Empire State Mines

2024 NI 43-101 Technical Report Update

Gouverneur, New York, USA

Prepared for:
Titan Mining Corporation

Effective Date: December 3, 2024
Signature Date: January 15, 2025

Prepared by the following Qualified Person:

- Donald R. Taylor, MSc., PG ....Titan Mining Corporation
- Todd McCracken, P.Geo.......................BBA USA Inc.
- Deepak Malhotra, P.Eng........................Forte Dynamics, Inc.
- Oliver Peters, MSc, P.Eng., MBA............Metpro Management Inc.

Date and Signature Page

This technical report is effective as of the 3rd day of December 2024.

Signed and Sealed	*January 15, 2025*
Donald R. Taylor, MSc, PG Titan Mining Corporation	Date
Signed and Sealed	*January 15, 2025*
Todd McCracken, P.Geo. BBA USA Inc.	Date
Signed and Sealed	*January 15, 2025*
Deepak Malhotra, P.Eng. Forte Dynamics, Inc.	Date
Signed and Sealed	*January 15, 2025*
Oliver Peters, MSc, P.Eng., MBA Metpro Management Inc.	Date

CERTIFICATE OF QUALIFIED PERSON

Donald R. Taylor, MSc, PG

This certificate applies to the NI 43101 Technical Report titled "*Empire State Mines 2024 NI 43101 Technical Report Update*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of January 15, 2025, and an effective date of December 3, 2024.

I, Donald R. Taylor, MSc, PG, as a co-author of the Technical Report, do hereby certify that:

1. I am the Chief Executive Officer of the Company, located at Suite 555 - 999 Canada Place, Vancouver, BC V6C 3E1.
2. I am a graduate of Southeast Missouri State University and the University of Missouri Rolla, where I received a Bachelor of Science degree in Geology and a Master of Science in Geology and Geophysics, respectively.
3. I am a member in good standing of the Society for Mining, Metallurgy & Exploration (SME Registered Member #4029597).
4. My relevant experience includes over 40 years of global mineral exploration and mining experience in the precious and base metal sectors. I have been responsible for many successful exploration and mine development programs, including several discoveries and deposit expansions. My experience includes positions with BHP Minerals, Bear Creek Mining, American Copper and Nickel, The Doe Run Resources Company and Westmont Mining. I serve as President and CEO of Augusta Gold and CEO of the Company, two mining companies listed on the Toronto Stock Exchange. Prior to this I was the Chief Operating Officer for Arizona Mining Inc., where I was credited with the discovery of the Hermosa Taylor Project in Arizona. The Hermosa Taylor Project is one of the world's largest undeveloped lead/zinc/silver deposits, which was purchased by Australia's South32. I was the 2017 winner of the SME's Ben F. Dickerson award; the recipient of PDAC's 2018 Thayer Lindsley award for the best global discovery and winner of the 2019 Robert M Dreyer award.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am not independent of the issuer applying all the tests in Section 1.5 of NI 43-101, as I am the Chief Executive Officer of the Company.
7. I am author and responsible for the preparation of Chapters 1 (except Sections 1.5.2, 1.6.2, 1.7.2 and 1.9), 2, 3, 4, 5, 6, 7, 8 (except Section 8.2), 9 (except Section 9.2), 10 (except Section 10.2), 11, 12, 14 (except Section 14.2), 15, 16, 18, 19, 20, 21, 22, 23, 24, 25 (except Section 25.2) and 26 (except Section 26.2). I am also co-author and responsible for the relevant portions of Chapter 27 of the Technical Report.
8. I have visited the Property that is the subject of the Technical Report, from August 20 to 22, 2024 as part of this current mandate.
9. I have had prior involvement with the Property that is the subject of the Technical Report as I have been Chief Executive Officer of the Company since June 21, 2018.
10. I have read NI 43-101, and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 15th day of January 2025.

Signed and Sealed

Donald R. Taylor, MSc, PG

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CERTIFICATE OF QUALIFIED PERSON

Todd McCracken, P.Geo.

This certificate applies to the NI 43101 Technical Report titled "*Empire State Mines 2024 NI 43101 Technical Report Update*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of January 15, 2025, and an effective date of December 3, 2024.

I, Todd McCracken, P.Geo., as author of the Technical Report, do hereby certify that:

1. I am Senior Geologist and Director of Mining and Geology at BBA USA Inc., located at 144 Pine Street, Unit 501, Sudbury, ON, P3C 1X3.
2. I am a graduate from University of Waterloo, Ontario, in 1992, with a bachelor's degree in Honors Applied Earth Sciences. I have practiced my profession continuously since my graduation.
3. I am a member in good standing of Association of Professional Geoscientists of Ontario (PGO No. 0631) and Ordre des Géologues du Québec (OGQ No. 02371).
4. My relevant experience includes 30 years in exploration, operations, and consulting, including resource estimation on graphite deposits.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Sections 1.5.2, 1.7.2, 8.2, 9.2, 10.2, 14.2, 25.2 and 26.2. I am also co-author and responsible for the relevant portions of Chapter 27 of the Technical Report.
8. I have visited the Property that is the subject of the Technical Report, on August 26-27, 2024, as part of this current mandate.
9. I have had no prior involvement with the Property that is the subject of the Technical Report.
10. I have read NI 43-101, and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 15th day of January 2025.

Signed and Sealed

Todd McCracken, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

Deepak Malhotra, P.Eng.

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I, Deepak Malhotra, P.Eng., as author of the Technical Report, do hereby certify that:

1. I am Principal / Director at Forte Dynamics, Inc., located at 12600 W Colfax Ave, Ste A-540, Lakewood, CO 80215.
2. I am a graduate of Colorado School of Mines with a M.Sc. degree in Metallurgical Engineering (1974), and PhD in Mineral Economics (1978).
3. I am a member in good standing of the Society for Mining, Metallurgy & Exploration (SME Registered Member #2006420).
4. My relevant experience includes having worked as a Metallurgist / Mineral economist for over 40 years, since my graduation from university; as an employee of several mining companies, an engineering company, a mine development and mine construction company, an exploration company, and as a consulting engineer.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Chapters 13 (except Section 13.2), 17, and Section 1.9. I am also co-author and responsible for the relevant portions of Chapter 27 of the Technical Report.
8. I have visited the Property that is the subject of the Technical Report in 2016 as part of another mandate. I did not visit the Property as part of this current mandate as it was not required for the purpose of this mandate.
9. I have had prior involvement with the Property that is the subject of the Technical Report as I have participated as a QP on the Company's most recent NI 43-101 technical report titled, "Empire State Mines 2021 NI 43-101 Technical Report (Amended)" with an effective date of February 24, 2021. I have also participated in due diligence examinations of the project as part of its purchase in 2017.
10. I have read NI 43-101, and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 15th day of January 2025.

Signed and Sealed

Deepak Malhotra, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Oliver Peters, MSc, P.Eng., MBA

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I, Oliver Peters, MSc, P.Eng., MBA, as author of the Technical Report, do hereby certify that:

1. I am Mineral Processing Engineer & President of Metpro Management Inc., located at 102 Milroy Drive, Peterborough, ON, K9H 7T2.
2. I am a graduate from RWTH Aachen, Germany.
3. I am a member in good standing of the Professional Engineers of Ontario (Member #100078050).
4. My relevant experience includes 26 years of mineral processing of which the last 13 years focused on graphite with over 40 projects completed.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Sections 1.6.2 and 13.2. I am also co-author and responsible for the relevant portions of Chapter 27 of the Technical Report.
8. I have visited the Property that is the subject of the Technical Report in 2024 as part of another mandate. I did not visit the Property as part of this current mandate as it was not required for the purpose of this mandate.
9. I have no prior involvement with the Property that is the subject of the Technical Report.
10. I have read NI 43-101, and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 15th day of January 2025.

Signed and Sealed

Oliver Peters, MSc, P.Eng., MBA

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Abbreviation	Description
$ or US$	United States dollar (examples of use: US$2.5M / $2.5M)
$/t	dollar per ton
%	percent
°	angular degree
°C	degrees Celsius (e.g., 22 °C)
°F	degrees Fahrenheit (e.g., 72 °F)
3D	three dimensional
AC	alternating current
ACM	AMC Mining Consultants (Canada) Ltd.
Ag	silver
Ai	Bond Abrasion Index
AIF	Annual Information Filing
Al	aluminum
ALS	ALS Limited (laboratories)
amsl	above mean sea level
ANFO	ammonium nitrate fuel oil (explosive)
As	arsenic
asl	above sea level
Au	gold
Azi	Azimuth
Ba	barium
BBA	BBA USA Inc.
BCK	back-stope
BFA	bench face angle
Bi	bismuth
BWi	Bond Ball Mill Work Index
C&F	cut and fill
Ca	calcium
CAPEX	Capital cost estimate
Cd	cadmium
CDA	Canadian Dam Association
Cg or C(g)	graphitic carbon
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimeter
cm^3	cubic centimeter
Co	cobalt
COG	cut-off grade
Conc	concentrate

Abbreviation	Description
Cr	chromium
CRMs	certified reference materials
CSX	CSX Corporation
C(t)	Total carbon
Cu	copper
d	day (24 hours)
d/y	days per year
DBA	Dam Breach Analysis
DDH	diamond drillhole
DEC	Department of Environmental Conservation
dmt	dry metric ton
EM	electromagnetic
ESM	Empire State Mines
et al.	and others
Fe	iron
FOG	fall of ground
FoS	factor of safety
ft	feet
ft^3	cubic feet
g	gram
G&A	General and Administration
gal	gallon
gal/d	gallons per day
gal/min	gallon per minute
gal/s	gallons per second
GT	grade/tonnage
h	hour (60 minutes)
h/d	hours per day
h/y	hours per year
H_2S	hydrogen sulfide
HDPE	High Density Polyethylene
hp	horsepower
Hudbay	Hudbay Minerals Inc.
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma Atomic Emission Spectrometry
ICP-OES	Inductively Coupled Plasma Optical Emission Spectrometry
ID	identification
ID2	inverse distance squared
ID3	inverse distance cubed
IDW	inverse distance weighted

Abbreviation	Description
in or "	inch
IRA	inter-ramp angle
JDS	JDS Energy and Mining Inc.
K	potassium
kcfm	kilo-cubic feet per minute
kg	kilogram
km	kilometer
km^3	cubic kilometer
kt	kiloton
kV	kilovolt
kVA	kilovolt amperes
kW	kilowatt
kWh	kilowatt per hour
L	liter
lb	pound(s)
LGS	Lower Graphitic Schist
LHD	load haul dump (loaders)
LLD	lower limit of detection
LOM	life of mine
LRS	longitudinal retreat stoping
LVA	Locally Varying Anisotropy
m	meter
M	million
m^3	cubic meter
Ma	mega annum
Max.	maximum
mesh	US mesh
Mg	magnesium
mg	milligram
mi	mile
min	minute
Min.	minimum
mm	millimeter
Mn	manganese
Mo	molybdenum
MRE	Mineral Resource Estimate
Mt	million tons
MVT	Mississippi Valley-type
MW	megawatt
Na	sodium

Abbreviation	Description
Ni	nickel
NI 43-101	Canadian National Instrument 43-101
NN	nearest neighbor
No.	number
NONEL	non-electric
NPV	net present value
NR	net revenue
NS	not sampled
NSG	non-sulfide gangue
NSR	net smelter royalty
NY	New York
NYSDEC	New York State Department of Environmental Conservation
OK	ordinary kriging
OMS	Operation, Maintenance and Surveillance Manual
ON	Ontario
OPEX	Operating cost estimate
OREAS	Ore Research and Exploration Pty. Ltd.
OSA	overall slope angle
OVB	overburden
oz	ounce
PAP & PAS	Panel Mining – Primary and Secondary
Pb	lead
PbS	lead sulfide
PEA	preliminary economic assessment
PEG	Pegmatite Intrusion
PGS	Phlogopitic Garnet Schist
PHG	Popple Hill Gneiss
PM	Panel Mining
ppm	parts per million
psi	pound per square inch
PSS	Pathway-Specific Standards
Q1, Q2, etc.	first quarter, second quarter, etc.
QA/QC	quality assurance / quality control
QP	qualified person
RBF	radial basis function
RDi	Resource Development Inc.
RMR	rock mass rating
ROM	run of mine
RPEEE	Reasonable Prospects for Eventual Economic Extraction
s	second

Abbreviation	Description
SD	standard deviations
SEDAR+	System for Electronic Document Analysis and Retrieval
Sedex	sedimentary exhalative
SG	specific gravity
SGS	SGS Canada Inc.
SHA	Seismic Hazard Analysis
SLZ	St. Lawrence Zinc Company, LLC
SO_4	sulfate
SPDES	State Pollutant Discharge Elimination System
Sr	strontium
SRK	SRK Consulting Ltd.
t	short ton (2,000 lb) (ton)
t/d	ton per day
t/h	ton per hour
t/w	ton per week
t/y	ton per year
TF	tonnage factor
Ti	titanium
Titan	Titan Mining Corporation
TMF	Tailings Management Facility
TP1, TP2, etc.	Tailings Pond #1, #2, etc.
TSF	tailings storage facility
UG	underground
UGS	Upper Garnet Schist
UM	Upper Marble
USDA	US Department of Agriculture
V	vanadium
V	volt
VFD	Variable Speed Drive
VTEM	versatile time domain electromagnetic
W	tungsten
w/w	weight in weight
XRF	X-ray fluorescence
y	year
yd	yard
yd^3	cubic yard
ZCA	Zinc Corporation of America
Zn	zinc

1. Summary

1.1 Introduction

BBA USA Inc. (BBA) has been engaged by Titan Mining Corporation (Titan or the Company) to update the current National Instrument 43-101 (NI 43-101) Technical Report for the Empire State Mines (ESM) operation. This technical report summarizes the results of this update and was prepared following the guidelines of NI 43-101.

ESM owns the Balmat No. 4 Underground Zinc Mine (the Mine), which is known as ESM No. 4 Mine or #4 Mine. The mine is located in the Balmat-Edwards-Pierrepont mining district in northern New York State, near Gouverneur and is 25 miles (mi) south of the Port of Ogdensburg.

The district is a mature zinc mining camp with production first recorded in 1915. Mining proceeded over the decades primarily as underground (UG) operations serviced by shafts and portals.

This revision to the technical report provides an update to ESM's zinc resources following additional drilling and mining exposure since the last technical report. Additionally, an initial resource estimate is provided for a graphite target called "Kilbourne".

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

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 uses a combination of longhole stoping, inclined room and pillar style panel mining, 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 mine plan scales up slightly from the current production rate of 1,750 t/d to 1,775 t/d in 2025. The current mine life is projected to be 9 years with the open pits being started depending on zinc price.

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. Tailings 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 southwest of Fowler, New York State, in St. Lawrence County. St. Lawrence Zinc Company, LLC (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 totaling 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 totaling 242 acres.

1.4 History, Exploration, and Drilling

The Balmat-Edwards-Pierrepont district consists of four mining regions (Balmat, Hyatt, Edwards, and Pierrepont) with production first recorded out of Edwards in 1915. Balmat 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. ESM resumed production in 2018 and has produced continually since then.

Drilling on site has been exclusively core drilling either with contract drillers such as Cabo, Major, and Boart Longyear, or by company owned and operated drills. The drillhole database contains 9,514 surface and underground drillholes, of which 513 holes totaling 219,095 ft have been drilled since 2020.

The Balmat mine (now ESM) has produced a total of 35.5 Mt grading 8.7% 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: ESM 2024

1.5 Geology and Mineralization

1.5.1 Zinc

The carbonate hosted ESM zinc deposits are comprised of multiple zones in and around Fowler, NY. There are ten deposits currently considered as viable economic targets; American, Cal Marble, Fowler, Mahler, Mud Pond, N2, Northeast Fowler, New Fold, Sylvia Lake, and Turnpike. Historic mining at these locations has provided a good geological understanding of each, with supporting mapping, sampling, and drilling data.

Mining and grade control experience by ESM geologists have supported that the implicit modeling of the mineralized zones as veins in Leapfrog Geo™ version 2023.2.3, results in more accurate geological wireframes.

The zinc mineralization extends from the surface down to a depth of 5,700 ft below surface. The zones are aerially scattered and all zones except NE Fowler and Cal Marble are connected by existing development to the shaft. The zones range in thickness from 2 ft to 50 ft with an overall plunge between 20° to 25° with local dips ranging from 0° to 90°. The deposit footprints are up to 500 ft wide and 9,000 ft long. The veins can display considerable geometrical variability depending on the degree of folding.

1.5.2 Graphite

Graphite mineralization occurs as weakly disseminated flakes within many of the marbles and dolomites, and occurs in the highest grades in the Upper Marble Unit 2 schists with graphitic carbon content averaging around 3% graphitic carbon.

1.6 Metallurgical Testing and Mineral Processing

1.6.1 Zinc

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 UG 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 (lb) or 55 kg 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 Ball 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 from 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.

1.6.2 Graphite

Mineralogical characterization and metallurgical testing were performed on samples from the Kilbourne Graphite Project (Kilbourne).

Optical microscopy of the samples showed that graphite was acicular to prismatic, and platy in habit. It ranged from <50 µm as individual flakes to 1.5 mm in size as polycrystalline clusters. Graphite was generally finer-grained in the low-grade samples and coarser in the higher-grade samples.

Flotation process development conducted at SGS on a sample grading 1.67% C(g) culminated in a flowsheet and conditions that produced a final concentrate grading 97.4% C(t). The graphite concentrate was classified as finer grained with less than 8% of the concentrate mass reporting to the +100 mesh size fractions. It is noteworthy that even the smallest size fraction of -200 mesh produced a very high total carbon content of 97.4% C(t).

Forte Analytical (Forte) conducted a test work program on two composites grading between 2.4% and 2.5% C(g). The focus of the test program was to produce a concentrate grading at least 95% C(t) while minimizing flake degradation. The optimized flowsheet and conditions produced an upgraded flash concentrate grading 98.3% C(t) with 21.4% of the concentrate mass reporting to the +100 mesh size fractions. The flash concentrate accounted for only 50-60% of the contained graphite and a global concentrate product including the upgraded rougher concentrate was not characterized.

While the execution of the test programs conducted by SGS and Forte varied significantly, the results are consistent. Both programs determined that the flake size distribution in the Kilbourne mineralization is relatively fine but upgraded readily to very high concentrate grades well above 95% C(t).

A review of the drillhole data revealed that the material between the upper and lower zones is almost barren. Sensor-based ore sorting may be an effective technology to reject the barren material, thus upgrading the average mill feed noticeably. Hence, ore sorting will be explored in the next phase of testing, which could significantly increase the mill head grade.

1.7 Mineral Resource Estimates

1.7.1 Zinc

Drillhole Database

The drillhole database was exported as CSV files for the resource updates. Assays and associated composites were extracted from drillholes that were used in estimation, of which there were 1,321 in total.

The complete database for ESM consists of 12,264 records. There are 89 sets of channel samples, 2,193 surface core holes, 7,321 UG core holes, and 2,661 holes identified as other (including monitoring wells, blast holes, and un-categorized historic drilling). Smaller subsets of this database were used for geologic modeling and/or estimation on a lithological unit basis. Each lithological group was modeled separately in isolated geological and estimation projects.

Geologic Model

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

All geological modeling was conducted in Leapfrog Geo™ version 2023.2.3. Each zone has been analyzed and divided where appropriate to facilitate a more accurate estimation of grade. In some cases, this has resulted in splitting of domains based on morphology or orientation for the purposes of estimation. Updates periods for modeling are summarized in Table 1-2.

Table 1-2: Update periods, model methodology, and volumes

Zone	Modeling Method	Years Modeled and Updated	Model Volumes
American	Implicit vein model	2019	4,586,000
Cal Marble	Implicit vein system model	2009, 2017, 2019, 2024	5,206,900
Fowler	Implicit vein system model	2019, 2023	2,598,000
Mahler	Implicit vein model; indicator RBF interpolant	2009, 2017, 2019, 2020, 2021, 2022, 2023, 2024	25,915,000
Mud Pond	Implicit vein system model	2008, 2009, 2017, 2019, 2020, 2021, 2022, 2023, 2024	14,875,000
N2D	Implicit vein system model; indicator RBF interpolant	2019, 2021, 2022, 2023	22,420,000
New Fold	Implicit vein system model; indicator RBF interpolant	2009, 2017, 2020, 2021, 2022, 2023, 2024	15,392,000

Zone	Modeling Method	Years Modeled and Updated	Model Volumes
Northeast Fowler	Implicit vein model	2017, 2019	6,852,600
Sylvia Lake	Implicit vein system model	2017, 2019, 2024	7,102,000
Turnpike	Indicator RBF interpolant	2019, 2021, 2022, 2023	65,041,000

Source: ESM 2024

Block Model

Separate block models were created for each zone. The parameters for each consist of origins, rotations (in Leapfrog rotation convention), parent block parameters and associated sub-block parameters. The American and Northeast Fowler block models were created in Vulcan and have parameters consistent with Vulcan conventions.

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 June 11, 2024, and used 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 Leapfrog EDGE called Variable Orientation or in Vulcan called Locally Varying Anisotropy (LVA). This oriented the search ellipse for each block down a plane which paralleled the modeled geologic continuity (i.e., the hanging wall or footwall of the ESM veins). The VO and LVA parameters were defined within the estimator based on the modeled vein surfaces.

Underground Mineral Resources have been modeled (Leapfrog Geo™ version 2023.2.3) and estimated (Leapfrog EDGE) by ESM geologists and reviewed for consistency with industry standards. Don Taylor of Titan Mining Corp. is the qualified person (QP) who has reviewed the geological models and estimates and has conducted multiple site inspections. Mineral Resources for the underground #4 Mine areas have been compiled from ten separate block models including the American, Cal Marble, Fowler, Mahler – Lower, Mahler - Upper, Mud Pond, N2D, New Fold, Northeast Fowler and Silvia Lake areas (Table 1-3).

Table 1-3: Underground Mineral Resource Estimate as of July 16, 2024

Category	Tons (000's US short tons)	Zn (%)	Contained Pounds (000's lb)
Measured	295	17.1	101,086
Indicated	1,158	15.7	363,825
Measured + Indicated	1,453	16.0	464,911
Inferred	4,327	12.1	1,048,630

Source: ESM 2024

Notes:

1. The qualified person for the 2024 MRE, as defined by the NI 43-101 guidelines, is Donald (Don) R. Taylor, of Titan Mining Corp., SME registered member (#4029597).
2. 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.
3. Three-dimensional (3D) wireframe models of mineralization were prepared in Leapfrog Geo based on the geological interpretation of the logged lithology on contiguous grade intervals defining mineralized sub-domains. The 2024 underground MRE encompasses 36 vein domains and six indicator RBF interpolant shells totaling 42 individual wireframes.
4. Geological and block models for the underground MRE used data from a total of 1,100 surface and underground diamond drillholes (core). The drillhole database was validated prior to resource estimation and QA/QC checks were made using industry-standard control charts for blanks and commercial certified reference material inserted into assay batches by Empire State Mines personnel.
5. High-grade capping was evaluated and implemented on the raw assay data on a per-zone basis using histograms and log-probability plots. Outliers were further evaluated during estimation and limited if necessary using the Leapfrog Edge clamping method.
6. The MRE was compiled from 10 individual block models that were prepared using Leapfrog Edge. Block models were sub-blocked at domain boundaries and samples were composited using vein length intervals where a single composite is generated for each complete vein intersection with a drillhole. Composites were generated within the indicator RBF interpolant models as 10-ft run-length composites with residuals less than 5 ft added to the prior interval, honoring the modeled geological boundaries. Grade estimation was carried out using inverse distance weighted (IDW) methods coupled with variably orientated search ellipses derived from modeled vein surfaces.
7. The specific gravity (SG) assessment was carried out for all domains using measurements collected during the core logging process. Where there is sufficient sampling, the SG is interpolated into model blocks using IDW techniques. If insufficient sampling exists, then density was assigned to models based on calculated means or by a regression formula.
8. Resources are reported using a 5.3% Zinc cut-off grade, based on actual break-even mining, processing, G&A costs, and smelter terms from the ESM operation at a zinc recovery of 96.4%.
9. Resources stated as in-situ grade at a Zinc price of $1.30/lb.
10. 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. Classification parameters vary slightly depending on the nature and continuity of the individual zones. Block classification was explicitly domained based on a calculation that used quality, quantity, and distance parameters.
11. Quantities and grades in the MRE are rounded to an appropriate number of significant figures to reflect that they are estimations.
12. The Mineral Resource Estimate was prepared following the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019).
13. CIM definitions and guidelines for Mineral Resource Estimates have been followed.
14. The QP is unaware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issues that could materially affect this MRE.

Open Pit #2 Mine (Turnpike) Mineral Resources have also been modeled (Leapfrog Geo™ version 2023.2.3) and estimated (Leapfrog EDGE) by ESM geologists and reviewed for consistency with industry standards. Don Taylor of Titan Mining Corp. is the QP who has reviewed the geological models and estimates and has conducted multiple site inspections. Mineral Resources for the Turnpike Open Pit area have been taken from a single block model (Table 1-4).

Table 1-4: Turnpike Open Pit Mineral Resource Estimate as of October 17, 2024

Category	Tons (000's US short tons)	Zn (%)	Contained pounds (000's lb)
Measured	251	3.1	15,679
Indicated	950	3.2	61,088
Measured + Indicated	1,201	3.2	76,767
Inferred	461	3.5	32,360

Source: ESM 2024

Notes:

1. The qualified person for the 2024 MRE, as defined by the NI 43-101 guidelines, is Donald (Don) R. Taylor, of Titan Mining Corp., SME registered member (#4029597).
2. 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.
3. Three-dimensional (3D) wireframe models of mineralization were prepared in Leapfrog Geo based on the geological interpretation of the logged lithology on contiguous grade intervals defining mineralized sub-domains. The 2024 Open Pit MRE encompasses three vein domains and nine indicator RBF interpolant shells totaling 12 individual wireframes.
4. Geological and block models for the Open Pit MRE used data from a total of 254 surface and underground diamond drillholes (core). The drillhole database was validated prior to resource estimation and QA/QC checks were made using industry-standard control charts for blanks and commercial certified reference material inserted into assay batches by Empire State Mines personnel.
5. High-grade capping was evaluated and implemented on the raw assay data on a per-zone basis using histograms and log-probability plots. Outliers were further evaluated during estimation and limited if necessary using the Leapfrog Edge clamping method.
6. The Open Pit MRE was compiled from a single block model that was prepared using Leapfrog Edge. The block model was sub-blocked at domain boundaries and samples were composited within the indicator RBF interpolant models as 10-ft run-length composites with residuals less than 5 ft added to the prior interval, honoring the modeled geological boundaries. Assays were composited within the vein models using vein length intervals where a single composite is generated for each complete vein intersection with a drillhole. Grade estimation was carried out using IDW methods coupled with variably orientated search ellipses derived from modeled trend surfaces.
7. The SG assessment was carried out for all domains using measurements collected during the core logging process. Where there is sufficient sampling, the SG is interpolated into model blocks using IDW techniques. If insufficient sampling exists, then density was assigned to models based on calculated means or by a regression formula.
8. Resources stated as internal to an optimized pit shell, above a cut-off grade of 0.6% Zn.
9. Cut-off is based on break-even economics at a Zinc price of $1.27/lb, with an assumed zinc recovery of 96%, and actual processing, mining, and transportation costs from the ESM operation. No G&A costs were applied as ESM considers the Project accretive. No extra mining dilution was added as a regularized block model was used.
10. 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. Classification parameters vary slightly depending on the nature and continuity of the individual zones. Block classification was explicitly domained based on a calculation that used quality, quantity, and distance parameters.
11. Quantities and grades in the MRE are rounded to an appropriate number of significant figures to reflect that they are estimations.

12. The Mineral Resource Estimate was prepared following the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019).
13. CIM definitions and guidelines for Mineral Resource Estimates have been followed.
14. The QP is unaware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issues that could materially affect this MRE.

1.7.2 Graphite

Drill Database

The Kilbourne Graphite Project database totals 45 surface-collared diamond drillholes (DDH) and one surface channel, totaling 29,699 ft used for modeling Kilbourne. There are a total of 3,396 assay records in the Kilbourne database, of which 2,088 assay records for graphite (%Cg).

Geology Model

Three-dimensional (3D) wireframe models of mineralization were developed in Leapfrog Geo™ version 2023.2.3 (Leapfrog) by ESM and reviewed by the QP. The wireframes were based on the geological interpretation of the logged lithology and sub-domained based on contiguous grade intervals greater than or less than 0.50 %Cg within the Upper Marble #2 (UM2) formation, defining the Upper, Middle, and Lower sub-domains of UM2 (210, 220, 230). Contiguous grade intervals greater than or equal to 0.50 %Cg were modeled within the higher-grade 210 and 230 sub-domains (UM2 – Upper and Lower, respectively), while contiguous grade intervals less than 0.50 %Cg were modeled as the 220 sub-domain (UM2 – Middle). These 200 series domains form the basis of the Kilbourne Mineral Resource Estimate.

The wireframe solids were imported from Leapfrog into Datamine Studio RM™ version 2.1.125.0 (Datamine) in .dwg format. The solids were validated within Datamine. The modeling is broken down into twelve separate geological domains based on lithology

The wireframes extend at depth, below the deepest DDH. This is to provide a target for future exploration. The block model extents did not encompass the entire wireframe extents to reduce block model and file sizes. As such the volumes related to the block model may significantly differ in comparison to the wireframe volumes. The volumes were validated with an initial block fill of the entire wireframes and no significant discrepancies were noted.

Block Model

Block modeling was completed in Datamine using industry accepted standard practices. The geological model wireframes were filled with parent block 30’ x 30’ x 15’ and sub-celled to filled the volumes.

Drillhole samples intervals were assigned to the appropriate mineral domain. Geostatistical analysis was completed on each mineral domain for grade capping, compositing, and spatial analysis.

Grades were estimated into the model using a three-pass estimation requiring a minimum and maximum number of samples to estimate a block. Table 1-5 summarizes the pit constrained Mineral Resource using a 1.5% Cg cut-off grade.

Table 1-5: Kilbourne Graphite Mineral Resource summary and in situ metal within pit shell

Classification	Deposit	Cut-Off Grade (% Cg)	Tonnage ('000 Ton)	Grade (% Cg)	Contained Graphite ('000 Ton)
Inferred	Kilbourne	1.50	22,423	2.91	653

Source: BBA 2024

Notes:

1. The independent qualified person for the 2024 MRE, as defined by NI 43-101, is Mr. Todd McCracken (PGO 0631) of BBA USA Inc. The effective date of this Mineral Resource Estimate is December 3, 2024.
2. Three-dimensional (3D) wireframe models of mineralization were based on the geological interpretation of the logged lithology and sub-domained based on contiguous grade intervals greater than or less than 0.50% Cg defining two mineralized sub-domains.
3. Geological and block models for the Mineral Resource Estimate used data from a total of 45 surface diamond drillholes (core) and 1 surface channel sample. The drillhole database was validated prior to resource estimation and QA/QC checks were made using industry-standard control charts for blanks and commercial certified reference material inserted into assay batches by Empire State Mines personnel.
4. Quantities and grades in the Mineral Resource Estimate are rounded to an appropriate number of significant figures to reflect that they are estimations.
5. The Mineral Resource Estimate was constrained using the following optimization parameters, as agreed upon by Empire State Mines and the QP. The parameters include mining costs of $4.60/ton for mineralized rock, $3.50/ton for unmineralized rock, and $2.00/ton for overburden and tailings, with a 5.0% dilution and 95.0% mining recovery. Processing costs are $14.00/ton milled, with a 91.0% processing recovery and a concentrate grade of 95.0%. No general and administrative (G&A) costs were applied. The selling price is $1,090/ton of concentrate, with transportation costs of $50/ton and no additional selling costs. The overall slope angles are 23 degrees for overburden and tailings, and 45 degrees for rock.
6. The resource reported has been tabulated in terms of a pit-constrained cut-off value of 1.50% Cg.
7. The block model was prepared using Datamine Studio RM™. A 30 ft x 30 ft x 15 ft block model was created, and samples were composited at 5 ft intervals. Grade estimation for graphite used data from drillhole data and was carried out using ordinary kriging (OK), inverse distance squared (ID2), and nearest neighbor (NN) methods. The OK methodology is the method used to report the mineral estimate statement.
8. Grade estimation was validated by comparison of the global mean block grades for OK, ID2, and NN by domain and composite mean grades by domain, swath plot analysis, and by visual inspection of the assay data, block model, and grade shells in cross-sections.
9. The SG assessment was carried out for all domains using measurements collected during the core logging process. The mean specific gravity value within the mineralized domains is 2.75.
10. The Mineral Resource Estimate was prepared following the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019).

1.8 Mining

The mine plan tons at the ESM deposit are extracted using a combination of longitudinal retreat stoping (LRS), Cut and Fill (C&F), Panel Mining (PM) – Primary and Secondary (PAP & PAS), and development drifting underground mining methods with rock backfill. Longhole back-stopes (BCK) are also used in the design where applicable. The mine plan scales up slightly from the current production rate of 1,750 t/d continuing through 2032 winding down in 2033. The current mine life is projected to be 9 years with the Turnpike open pits being dependent on zinc price. For the purposes of this report, the open pits are not included in the economic analysis. A conceptual schedule is included, but the tons are not included in the life of mine (LOM) mineable inventory.

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 July 2024 and is estimated at a 5.5% Zinc cut-off grade for the UG mine and 0.6% Zn for open pit mining. The LOM plan is considered to start January 2025 with the production from 2024 being calculated from actuals and short-range estimates.

For the underground mine, dilution was estimated based on typical stope dimensions to calculate unplanned overbreak experienced during mining operations. The rock quality at ESM is considered to be very good geotechnically, so overbreak is considered to be minimal. For LRS and BCK stopes, two sources of dilution were considered. Sloughing was estimated to be 2.0 ft on both the hanging wall and footwall of LRS 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.

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 back-stopes assume 95% mine recovery.
- Panel mining assumes 75% mine recovery after losses from pillars left behind.

Provided care is taking during blasting and rigorous ore control and monitoring systems are followed, BBA estimates that dilution and ore losses can be minimized for open pit mining. A mining recovery factor and dilution factor were not applied as a regularized block model was used for the mine design and scheduling.

The production schedule for the underground LOM is provided in Table 1-6. A proposed schedule for potential Turnpike Open Pit order of extraction is provided in Table 1-7.

Table 1-6: Mine production schedule

Item	Unit	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
Underground Ore Mined	kt	**4,467**	425	462	467	455	455	455	455	455	455	383
Zinc Grade	%	**7.41**	8.60	7.80	7.50	7.30	7.30	7.30	7.30	7.30	6.50	7.30
Contained Zinc	M lb	**662**	73	72	70	66	66	66	66	66	59	56

Source: ESM 2024

Table 1-7: Turnpike Open Pit conceptual schedule

Item	Unit	LOM	Y1	Y2	Y3
Open Pit Ore Mined	kt	399	120	195	84
Total Open Pit Waste	kt	1,364	580	627	158
Stripping Ratio	W:O	4.8	3.2	1.9	3.4
Total Material moved	kt	1,763	700	822	242
Zinc Grade	%	2.79	3.13	3.80	3.17
Contained Zinc	000s lb	6,695	12,210	6,390	25,292

Source: BBA 2024

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. Throughout 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, 4 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 people 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 ESM 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. ESM 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 $37.2M, consisting of the following distinct areas:

- No. 4 Mine capital equipment;
- No. 4 infrastructure and process capital.

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

Table 1-8 presents the capital estimate summary for each area in 2024 US$ with no escalation.

Table 1-8: Capital cost summary

Area	Cost Estimate ($M)
#4 Mine Capital Equipment	13.1
#4 Infrastructure and Process Capital	13.9
Total Capital Cost	**27.0**
Closure Costs	15.4
Salvage Value	5.2
Total Capital Cost (incl. closure costs)	**37.2**

Source: ESM 2024

Underground capital costs are estimated to be $13.1M. This includes an additional mechanical bolter as well as a replacement bolter, replacement of two 6 yd loaders, replacement of two UG haul trucks, replacement of a single boom jumbo, four additional 750 kVA transformers, ventilation fans and doors, a replacement locomotive, a surface exploration drill, and main dewatering pumps.

ESM has assumed that due to the short life of the pits (3 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 to a total of $15.4M, this will be offset by the estimated $5.2M 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 UG 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.

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

Site operating costs (Table 1-9) are presented in 2024 US$ on a calendar year basis. No escalation or inflation is included.

Table 1-9: Breakdown of estimated site operating costs

Site Operating Costs	Unit Cost ($/t milled)	LOM Cost ($M)
Mining	55	244
Processing	18	80
G&A	20	90
Concentrate Transportation	8	32
Total	**101**	**446**

Source: ESM 2024

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 5%;
- Nominal 2024 US dollars;
- Revenues, costs, and taxes are calculated for each period in which they occur;
- All costs and time prior to January 1, 2024, 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 $5.6M for the LOM.

The economic analysis incorporates royalties. A royalty of 0.3% is applied to the NSR for the zinc 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 $104M, with a pre-tax and post-tax net present value (NPV) at a discount rate of 5% of $88M and $83M, respectively. The results of the assessment are provided in Table 1-10.

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 1-11.

Table 1-10: Summary of the economic analysis results

Summary of Results	Unit	Value
Mine Life	year	9.0
Resource Mined	kt	4,469
LOM Throughput Rate	t/d	1,775
LOM Operating Days per Year	d/y	260
Average Head Zinc Grade	% Zn	7.4
LOM Recovered Zinc	M lb	636
LOM Payable Zinc	M lb	541
Total Revenue	$M	577
Total Offsite Charges	$M	107
Royalties	$M	0.2
NSR (net of royalties)	$M	577
Capital Costs (including sustaining)	$M	27
Operating Costs	$M	446
Operating Costs	$/t processed	101
Pre-tax Cash FLOW	$M	104
Taxes	$M	5.6
After-tax Cash Flow	$M	98
Pre-tax NPV (5% discount)	**$M**	**88**
After-tax NPV (5% discount)	**$M**	**83**

Source: ESM 2024

Table 1-11: Sensitivity analysis results

Variable	Pre-tax NPV @ 5% ($M)			Post-tax NPV @ 5% ($M)		
	-10% variance	0% variance	10% variance	-10% variance	0% variance	10% variance
Zinc Price	47	88	133	38	83	125
Zinc Grade	49	88	126	46	83	116
CAPEX	90	88	85	85	83	76
OPEX	116	88	55	109	83	44

Source: ESM 2024

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.

There is an opportunity to increase production and project NPV by accelerating the mining of the N2D zone. This would require the purchase $2.8M of additional mining equipment, a power upgrade of $2.6M and hiring additional miners and mechanics to add 500 t/d of incremental ore to the mill feed. The expansion would decrease the LOM by 1 year compared to the base case due to accelerated depletion of resources. It would also add $14M to the project pre-tax NPV calculation and 13 M payable zinc pounds per year during its 3.5-year life.

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

1.14.3.1 Zinc

The items shown in Table 1-12 are recommended for ESM to improve confidence and performance of the PEA mine plan and economics.

Table 1-12: Project recommendations and estimated cost

Item	Cost ($)
Infill Drilling and Conversion of Inferred Mineral Resources	150,000
Review Financing for Production Expansion from N2D Zone	5,400,000
Sorting Test Work and Integration Study	100,000
Contractor Quotes for Open Pit Cost Assumptions	15,000
Total Estimate	**5,665,000**

Source: ESM 2024

1.14.3.2 Graphite

The items shown in Table 1-13 and Table 1-14 are recommended for ESM to advance Kilbourne to a PEA level and ensure commercial viability.

Table 1-13: Project recommendations and estimated cost

Recommended Study Item	Estimated Cost ($)
Infill Drilling	950,000
Geotechnical Study	50,000
Phase III Metallurgical Study	47,000
Mining Study	250,000
Optical Sorting Study	30,000
Contractor Quotes	15,000
Permitting	130,000
PEA Technical Report Update	500,000
Preliminary Economic Assessment Subtotal	**1,972,000**
Contingency (25%)	493,000
Total Estimate	**2,465,000**

Source: ESM 2024

Table 1-14: Commercial recommendations and estimated cost

Recommended Study Item	Estimated Cost ($)
Commercial Scoping Study	150,000
Product Qualification Consulting	68,000
Demonstration Plant	6,110,000
Commercial Scoping Subtotal	**6,328,000**
Contingency (25%)	1,582,000
Total Estimate	**7,910,000**

Source: ESM 2024

2. Introduction

BBA USA Inc. (BBA) has been engaged by Titan Mining Corporation (Titan or the Company) to update the current National Instrument 43-101 (NI 43-101) Technical Report for the Empire State Mines (ESM or the Property) operation and was prepared following the guidelines of NI 43-101.

This Technical Report titled *"Empire State Mines 2024 NI 43-101 Technical Report Update"* provides an update to the ESM zinc Mineral Resource Estimate (MRE) and mine plan and describes a maiden MRE for the Kilbourne graphite deposit.

ESM is an underground zinc mine near the town of Gouverneur, New York State. It is located approximately 1.3 miles (mi) southwest 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 facilities are located. Nine parcels totaling 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 totaling 242 acres. Titan holds 100% ownership.

ESM is comprised of a group of high-grade zinc 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 t/d mill. Open pit potential has been identified at the surface of the historic #1 and #2 mines and is known as Turnpike.

2.1 Basis of the Technical Report

ESM technical staff provided updates for most of the individual chapters. The following companies contributed to this Technical Report and provided qualified person (QP) sign-off for their respective sections.

BBA USA Inc. (BBA)

- Overall report integrator;
- Kilbourne Mineral Resource Estimate.

Forte Dynamics Inc. (Forte) (recently merged with RDi Resource Development)

- Zinc metallurgical test work and mineral processing.

Metpro Management Inc. (Metpro)

- Graphite metallurgical test work and mineral processing.

The individuals listed in Table 2-1, by virtue of their education, experience, and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions.

The key information used in this report is listed in Chapter 27 - References.

This Technical Report has been produced following the Standards of Disclosure for Mineral Projects as contained in NI 43-101 and accompanying policies and documents. NI 43-101 uses 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) (CIM, 2014).

A draft of the Technical Report was provided to Titan to check for factual accuracy. The Technical Report is effective as of December 3,2024.

Table 2-1: QP Responsibilities and date of last site visit

Qualified Persons Responsible for the Preparation of this Technical Report					
Qualified Person	**Employer**	**Independent of Titan?**	**Date of Last Site Visit**	**Professional Designation**	**Sections of the Report**
Donald R. Taylor Chief Executive Officer	Titan	No	August 20-22, 2024	SME Registered Member	Chapters 1 (except 1.5.2, 1.6.2, 1.7.2, and 1.9), 2 to 8 (except 8.2), 9 (except 9.2), 10 (except 10.2), 11, 12, 14 (except 14.2), 15, 16, 18 to 25 (except 25.2) and 26 (except 26.2) Co-author of Chapter 27
Todd McCracken Director – Mining & Geology – Central Canada	BBA	Yes	August 26-27, 2024	PGO	Sections 1.5.2, 1.7.2, 8.2, 9.2, 10.2, 14.2, 25.2 and 26.2 Co-author of Chapter 27
Deepak Malhotra Principal / Director	Forte	Yes	2016	SME Registered Member	Chapters 13 (except 13.2) and 17 Section 1.9 Co-author of Chapter 27
Oliver Peters Mineral Processing Engineer & President	Metpro	Yes	October 30, 2024	MSc, P.Eng., MBA	Sections 1.6.2 and 13.2 Co-author of Chapter 27

Source: BBA

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 list of abbreviations and units of measurement after 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 relied on C. Connor Messler, Exploration Manager, Empire State Mines, for matters pertaining to mineral concessions, surface rights and mining leases, as disclosed in Chapter 4, pursuant to statements made by Mr. Messler that were confirmed to be current as of the effective date of the Technical Report.

The QPs relied on Ryan Schermerhorn, Production Manager, Empire State Mines, for matters pertaining to permitting, environmental, social, and community factors, as disclosed in Chapter 20, pursuant to statements made by Mr. Schermerhorn that were confirmed to be current as of the effective date of the Technical Report.

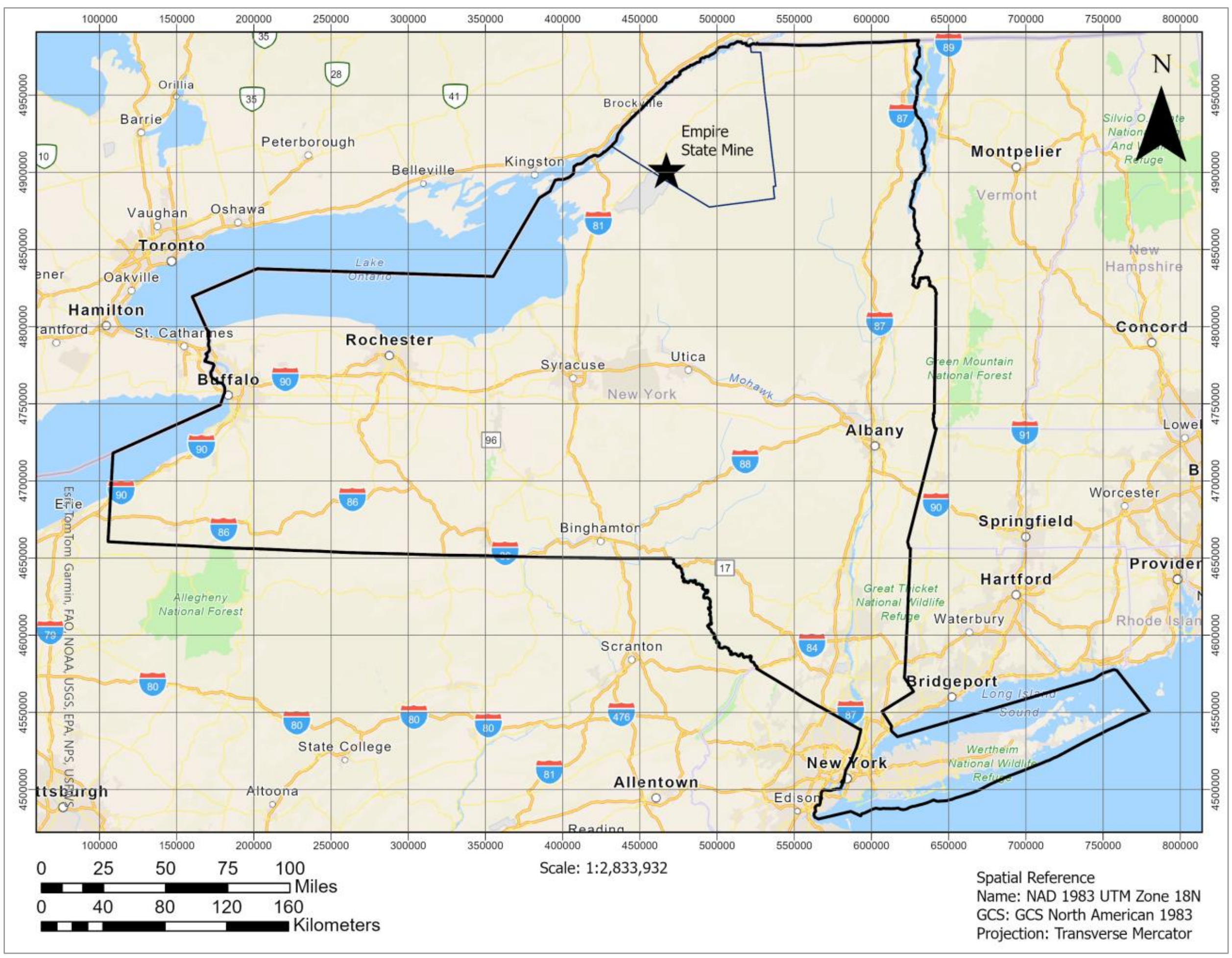
The QPs relied on Ty Minnick, Interim Chief Financial Officer of Titan Mining Corporation, for matters pertaining to taxation on the Property, as disclosed in Chapter 22, pursuant to statements made by Mr. Minnick that were confirmed to be current as of the effective date of the Technical Report.

4. Property Description and Location

4.1 Location

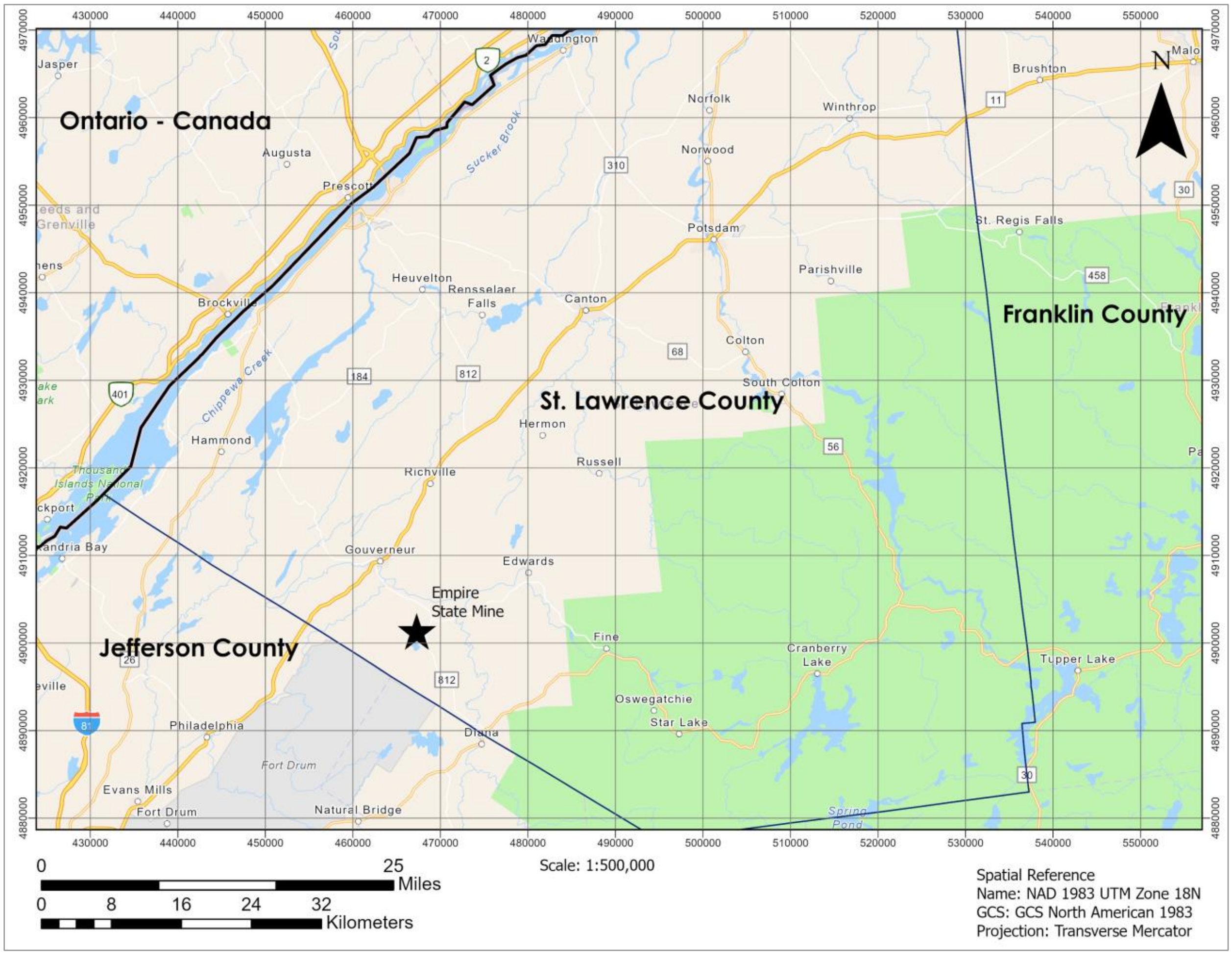
The ESM mine is located 7 mi southeast of Gouverneur, New York, at 44°14’51” N latitude, 75°23’50” W longitude, and 710’ above mean sea level (amsl). The site is 38 mi via State Road #812 from the St. Lawrence Seaway at Ogdensburg, NY (Figure 4-1 and Figure 4-2).

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



Source: ESM 2024, modified from ESRI base map

Figure 4-1: Regional project location



Source: ESM 2024, modified from ESRI base map

Figure 4-2: Local project location

4.2 Mineral Tenure

The 2,699 acres of surface rights owned by Titan are divided among the townships of Fowler, Edwards and Pierrepont, containing 1,753, 703 and 242 acres, respectively. 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 Company has an additional 29,017 acres of leased and optioned mineral rights targeting prospective exploration areas, and within proximity to the Balmat, Hyatt, and Pierrepont mine areas. Leases have an initial 20-year term, renewable for an additional 20 years, and are subject to a 4% net smelter return (NSR) royalty. Optioned mineral rights have a renewable 5-year initial term. Option payments amount to $4 per acre per year.

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.

A list of leases with expiration dates are provided in Table 4-1Table 4-1. In certain limited cases outside of the current mineral resource and subsequent anticipated mining areas, certain lease agreements have not been formally extended due to administrative challenges in signing official extension documents. In these limited cases, the Company has continued to make annual payments on such leases (which payments have been received), and the Company is of the view that these leases have been constructively extended. The current mineral resource and subsequent anticipated mining areas are not impacted in any way by the leases that have not yet been formally extended.

Differences from past acreage totals are in part attributable to discrepancies between surface parcel acreage and mineral rights acreage, with mineral rights often representing historic parcel geometries and locations. These historic parcel shapes do not always align with the current surface outline. Additionally, review of the Lansing-Dodge Agreement listed in Table 4-1 has shown that sections of this agreement had been claimed by St. Lawrence County in the mid-1900s. These issues have been recognized and recorded by ESM personnel during the course of property due diligence prior to exploration activities.

Table 4-1: Lease list with expiration dates

Name	Type	Expiration Date	Acres	Term	NSR
Warriner Lease	Lease	18/01/2031	80.82	20-year lease: renewable	4%
St. Lawrence Ore Lease	Lease	25/01/2010 [annual payments made and received]	135	20 years: NOT renewable	4%
Whitman Lease	Lease	10/02/2018 see note	30	20 years: renewable for additional 20 years	4%
Brian Tripp Lease (90Ac)	Lease	22/03/2021 [annual payments made and received]	90	20 years: renewable for additional 20 years	4%
Gilbert Lease	Lease	22/03/2031	96.4	20-year lease: renewable	4%
Jenne Lease	Lease	02/19/2041	111	20 years: renewable for additional 20 years	4%
Wells Lease	Lease	10/01/2029	178	40 years: NOT renewable	4% Zinc; 5% Lead
Hull Lease	Lease	30/04/2017	20	20 years: renewable for additional 20 years	4%
Kelly Freeman Lease	Lease	02/05/2015 [annual payments made and received]	310	20 years: renewable for additional 20 years	4%
Davis (Robert and Peggy) Lease (0.5 Ac)	Lease	26/05/2030	0.5	20 years: renewable for additional 20 years	4%
Edwards Lease	Lease	06/03/2039	96	20 years: renewable for additional 20 years	4%
Cole Lease	Lease	19/02/2041	94	20 years: renewable for additional 20 years	4%
Aleta Billings Heirs Leases	Lease	26/06/2039 (Gary E. Wight) 12/06/2039 (Joann A. Whitaker) 05/07/2039 (Lee H. Wight) 13/06/2039 (Linda M. Love)	157.5	20 years: renewable for additional 20 years	4%
Alan Latimer Lease	Lease	07/07/2043	20	20 years: renewable for additional 20 years	4%
Yerdon Lease	Lease	10/07/2027	0.3	20 years: renewable for additional 20 years	4%
Barrigar Lease (Larry P. & Elaine P.) (part of former Lloyd & Lillian Barrigar Lease)	Lease	02/07/2039	122.4	20 years: renewable for additional 20 years	4%

Name	Type	Expiration Date	Acres	Term	NSR
Pusateri-Linda, Etal Lease (part of former Lloyd & Lillian Barrigar Lease)	Lease	29/07/2039	158.4	20 years: renewable for additional 20 years	4%
Timothy J. Sweeney (Lease)	Lease	16/07/2030	1.91	20 years: renewable for additional 20 years	4%
Zira Lease	Lease	25/07/2027	0.93	20 years: renewable for additional 20 years	4%
Webb Lease	Lease	18/09/2039	46	20 years: renewable for additional 20 years	4%
Van Brocklin Lease	Lease	28/07/2042	100	20 years: renewable for additional 20 years	4%
Davis, Daniel Lease (formerly Barkley Lease)	Lease	25/07/2040	78	20 years: renewable for additional 20 years	4%
Brown Lease	Lease	09/09/2039	165	20 years: renewable for additional 20 years	4%
Bogardus Lease (Peter & Penny Bogardus)	Lease	11/12/2039	162.2	20 years, renewable in 20 years	4%
James Morrill Lease	Lease	08/09/2029	464	20 years: renewable for additional 20 years	4%
Stanley Morrill Lease	Lease	08/09/2029	266.22	20 years: renewable for additional 20 years	4%
Lansing-Dodge Lease	Lease	08/10/2039	19,230	20 years: renewable for additional 20 years	4%
Emery Webb Lease	Lease	22/09/2029	181.46	20 years: renewable for additional 20 years	4%
Hutchinson-Todd Lease	Lease	10/03/2042	37	20 years: renewable for additional 20 years	4%
Manning Lease	Lease	01/10/2027	0.65	20 years: renewable for additional 20 years	4%
Walter Planty Lease (64.39 Ac)	Lease	30/10/2039	64.39	20 years: renewable for additional 20 years	4%
Marjory Tyler Lease	Lease	06/11/2039	183	20 years: renewable for additional 20 years	4%
Brian Tripp Lease (0.79Ac)	Lease	06/12/2026	0.79	20 years: renewable for additional 20 years	4%
Brian Tripp (formerly Robert G., Sr. and Phyllis J. Tripp) Lease (19 Ac)	Lease	09/05/2039	19	20 years: renewable for additional 20 years	4%
Davis (Stanley and Carol) Lease (14.4 Ac)	Lease	06/11/2026	12.28 & 2.12	20 years: renewable for additional 20 years	4%
Gouverneur Talc Co Lease	Lease	28/06/2030	~5,900	20-year lease	4%
Bishop Lease	Lease	15/06/2037	0.50 0.69	20 years: renewable for additional 20 years	4%
Spellacy Lease	Lease	18/09/2040	360.67	20 years: renewable for additional 20 years	4%

When necessary, surface rights have been purchased from landowners; generally, these purchases have accommodated the construction and development of infrastructure related to mining and processing. Titan's surface rights include the lands where the surface facilities of the ESM mine, concentrator, tailings impoundment, and Kilbourne Project 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 a second party own the remaining 1/5 (20%) interest.

Table 4-2: Mineral tenure information

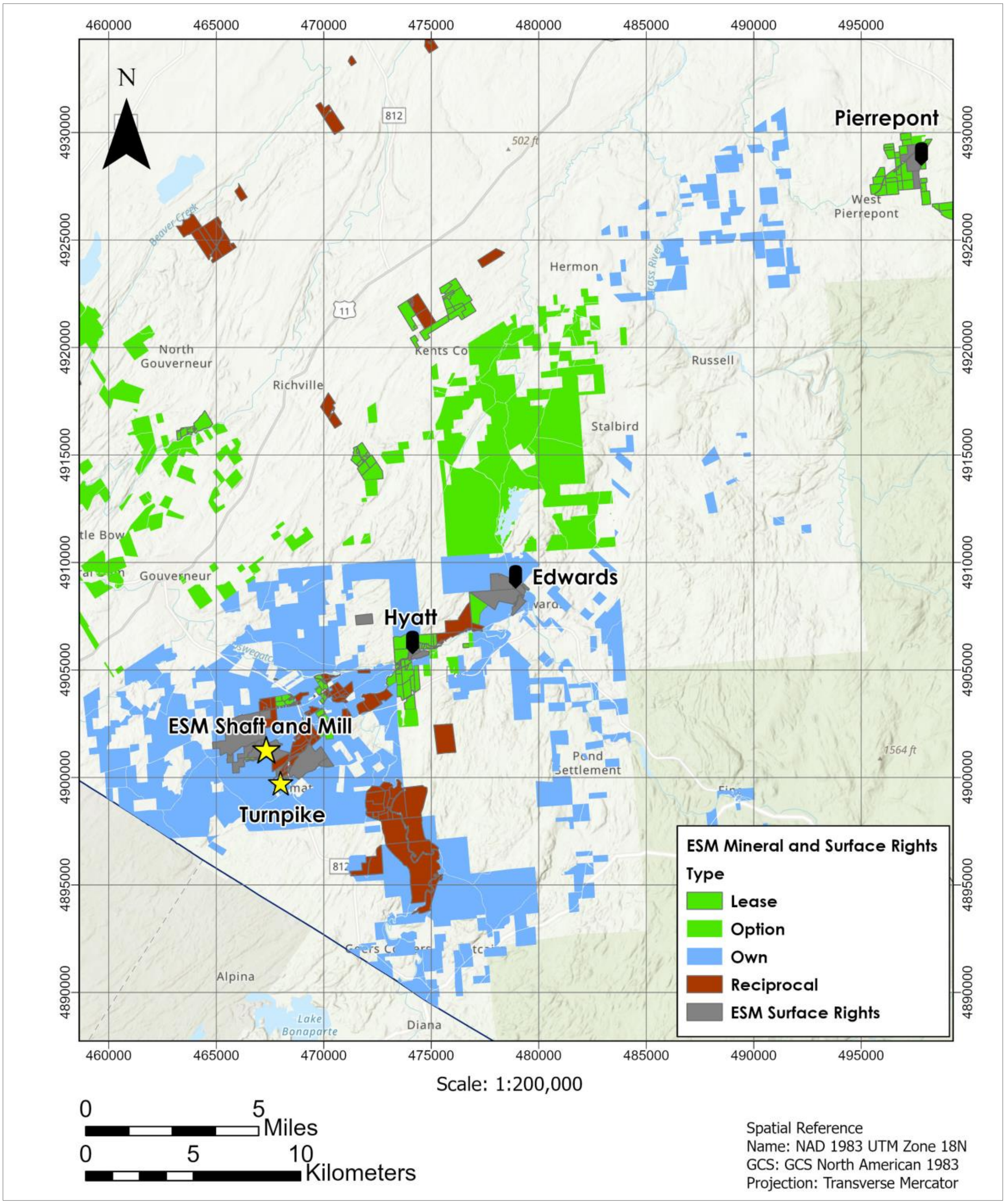
Assessor Parcel Number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2024 Taxes ($)
119.001-1-8	Pierrepont	80.4			322	414.46
119.001-1-10	Pierrepont	102.1			330	526.26
119.001-1-11	Pierrepont	0.52			720	1.72
119.001-1-12	Pierrepont	59.3			720	357.28
119.001-1-18./1	Pierrepont		1.4		720	43.00
174.004-3-2	Edwards	0.85			314	48.00
174.004-4-2	Edwards	10.37			720	198.83
174.004-4-1	Edwards	1.35			314	86.84
175.003-3-1.1	Edwards	71.6			720	617.06
175.003-3-19.1	Edwards	3.4			720	118.83
175.002-1-5.1	Edwards	370.2			323	2,664.77
175.002-1-33	Edwards	161.7			323	1,236.39
175.002-1-34.1	Edwards	72.2			330	621.63
175.002-1-32.1	Edwards	11.7			330	207.97
175.002-1-34./1	Edwards		74		720	162.26
1.044-18	Edwards		100		720	159.98
175.002-1-25./1	Edwards		92.2		720	150.84
175.001-1-4./1	Edwards		165		720	162.26
175.002-1-5./1	Edwards		1,044		314	598.77
175.003-1-1./2	Edwards		72		720	150.84

Assessor Parcel Number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2024 Taxes ($)
175.003-1-1./4	Edwards		18.8		720	150.84
175.003-3-1.1/1	Edwards		70		720	473.09
175.003-3-1.1/4	Edwards			Electrical	720	1,325.53
175.003-3-10./1	Edwards		115		720	150.84
175.003-3-13./2	Edwards		53.1		720	150.84
175.004-1-3./1	Edwards		58		720	150.84
175.004-1-6./1	Edwards		20		720	150.84
175.004-1-7./1	Edwards		63.8		720	150.84
175.004-1-11./1	Edwards		97.4		720	242.25
175.004-1-14./2	Edwards		62		720	150.84
187.002-2-1./1	Edwards		30		720	150.84
187.002-2-1./2	Edwards		80.9		720	150.84
188.001-1-15./2	Edwards		25		720	150.84
188.001-1-15./3	Edwards		169.1		720	150.84
188.001-1-17./1	Edwards		65.6		720	150.84
188.001-1-27./1	Edwards		73.8		720	150.84
188.002-1-2./1	Edwards		36		720	150.84
174.004-1-18	Fowler	89.3	89.3		720	382.85
187.001-1-5	Fowler	2.5			720	127.61
187.001-1-21.2	Fowler	44.49			720	264.18
186.004-1-44	Fowler	705.3			720	1,276.22
186.004-1-33.11	Fowler	86.5			720	1,294.46
186.004-1-31	Fowler	61.6			720	1,180.49
187.003-1-2	Fowler	82.3			720	255.25
187.003-1-1	Fowler	1.6			720	4,402.93
187.069-1-38	Fowler	0.7			720	1,651.16
187.003-1-4.11	Fowler	63.8			720	740.97
187.003-1-4.121	Fowler	124.7			720	446.68
187.003-2-1.1	Fowler	45.2			720	255.25
199.001-2-52	Fowler	445			720	1,276.22
186.002-1-14.11/3	Fowler		146.6		720	12.77

Assessor Parcel Number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2024 Taxes ($)
186.002-1-14.11/4	Fowler		144		720	12.77
187.003-1-3./1	Fowler		0.01		720	127.61
187.003-1-4.11/2	Fowler			Shaft 4	720	20,738.46
187.003-1-4.11/3	Fowler		0.01		720	11,007.34
187.003-1-4.11/5	Fowler			Shop	720	2,787.25
187.003-1-4.11/7	Fowler			Electric	720	18,249.86
187.003-1-4.11/9	Fowler			Buildings	720	49,772.31
187.003-1-4.11/11	Fowler			Paint, oil storage building	720	2,465.64
187.003-1-4.11/12	Fowler			Timber storage	720	2,641.76
187.003-1-4.11/17	Fowler			Railroad #4	720	6,604.40
187.003-1-4.11/18	Fowler			Mill	720	89,590.15
187.003-1-4.11/20	Fowler			Storage buildings	720	15,429.42
187.003-1-4.11/21	Fowler			Storage	720	6,221.54
199.001-2-43.1/2	Fowler			Pipe shop 2	720	352.24
142.004-2-7.12/1	Macomb		60.30		720	12.99
Owned Fee Parcels		**2,699**	**3,027**			**252,261.40**

Source: St. Lawrence County Government 2019

All properties listed in Table 4-2 matches the St. Lawrence County 2024 tax rolls and are fully paid and current as of November 1, 2024.



Source: ESM 2024, modified ESRI base map

Figure 4-3: Mineral tenure map

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 optioned to lease from property owners.

4.4 Project Agreements

Mineral rights may be acquired from the owner by lease, 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.

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 the ESM #4 Mine and ESM #2 Shaft area (which is still in use as an alternate exit route and ventilation shaft for ESM #4 Mine) and are the ongoing responsibility of Titan. Reclamation of the ESM #4 Mine and tailings is assured by a $1,662,870 deposit certificate.

The mining activity in the Balmat region has not created any known long-term liabilities, beyond those described in Chapter 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 sulfate or lime upstream from the water holding ponds.

4.6 Permit Requirements

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. 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 5 years and may be renewed upon application. Permit holders must submit annually to the DEC a fee based on 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 #4 Mine are listed in Table 4-3.

Table 4-3: Environmental permits for operation of ESM #4 Mine

Permit Type	Permit	Permit Number	Expiration
Air	Registration to Operate a Zinc Mining and Milling Complex (amended)	6-4038-00024/02001	28 April 2034
Water	SPDES(1) Water Discharge Permit	NY0001791	31 May 2019(2)
Water	Water Withdrawal Permit	6-4038-00024/02001	30 April 2031
Mining	Mining Permit	6-4038-00024/00006	31 July 2025
Storage	NYDEC Petroleum Bulk Storage	PBS#6-451770	26 September 2028
Radiation	Certificate of Registration for Radiation Installation - XRF	44023174	15 September 2026
Public Water Supply	No permit required, but regulated by NYS Dept. of Health	Registered ID #NY4430004	None
Hazardous Material Transport	US Department of Transportation Registration – Pipeline and Hazardous Material Safety Administration	052324550160G	30 June 2025

Notes:

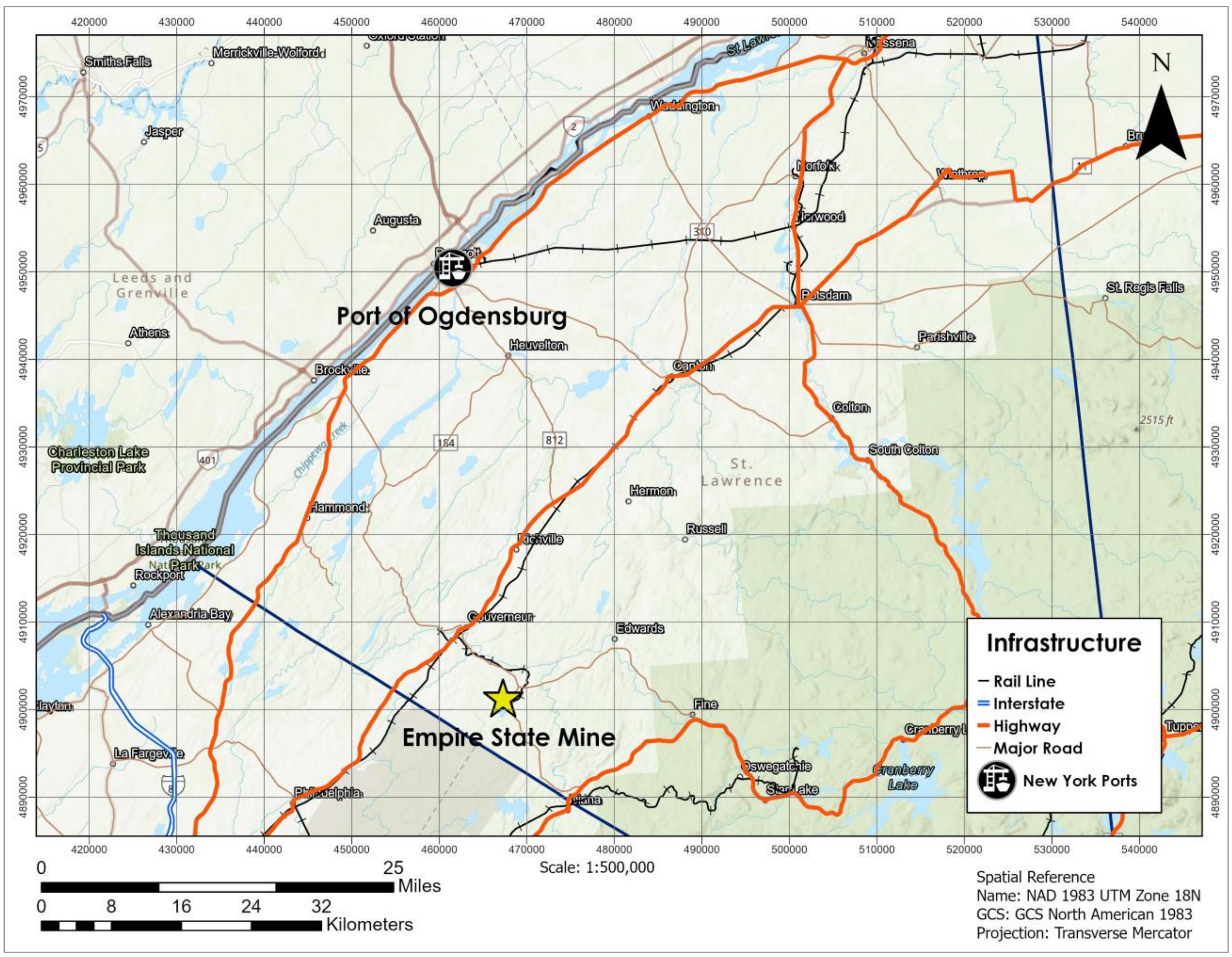
(1) SPDES = State Pollutant Discharge Elimination System.

(2) SPDES permits are under technical review by the New York State DEC and are still valid despite the expiration dates. Source: ESM 2024.

5. Accessibility, Climate, Local Resources, Infrastructure, and Physiography

5.1 Accessibility

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



Source: ESM 2024

Figure 5-1: Site accessibility

5.2 Local Resources and Infrastructure

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

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 number of frost-free days is 115. Annual average precipitation is approximately 40", 70% occurs as snow. The mine and process facilities operate year-round. Weather is not expected to frequently or significantly affect operations at any time of the year.

5.4 Vegetation and Wildlife

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

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

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

5.5 Physiography

The ESM Project is situated on the northwest flank of the Adirondack Mountains. The ESM Mine site lies within heavily forested bedrock ridges and interspersed low-lying marsh areas. Elevation at the mine site is 710 ft 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.

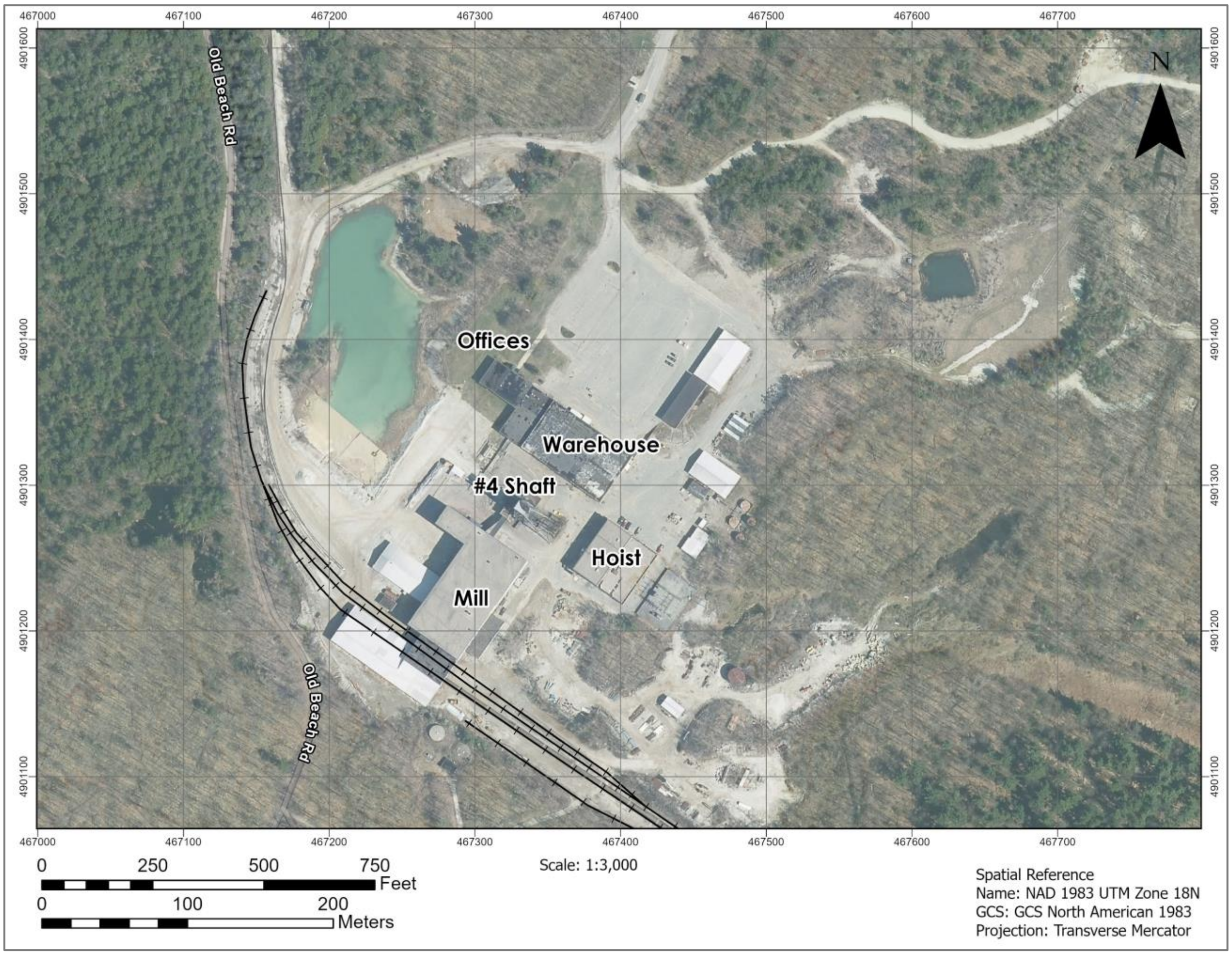
5.6 Surface Facilities and Rights

The existing operation is located on lands owned or leased by Titan. All utilities such as roads, rail, electricity, water, communications systems, tailings 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 (Figure 5-2). During the start-up of the mine, labor 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.

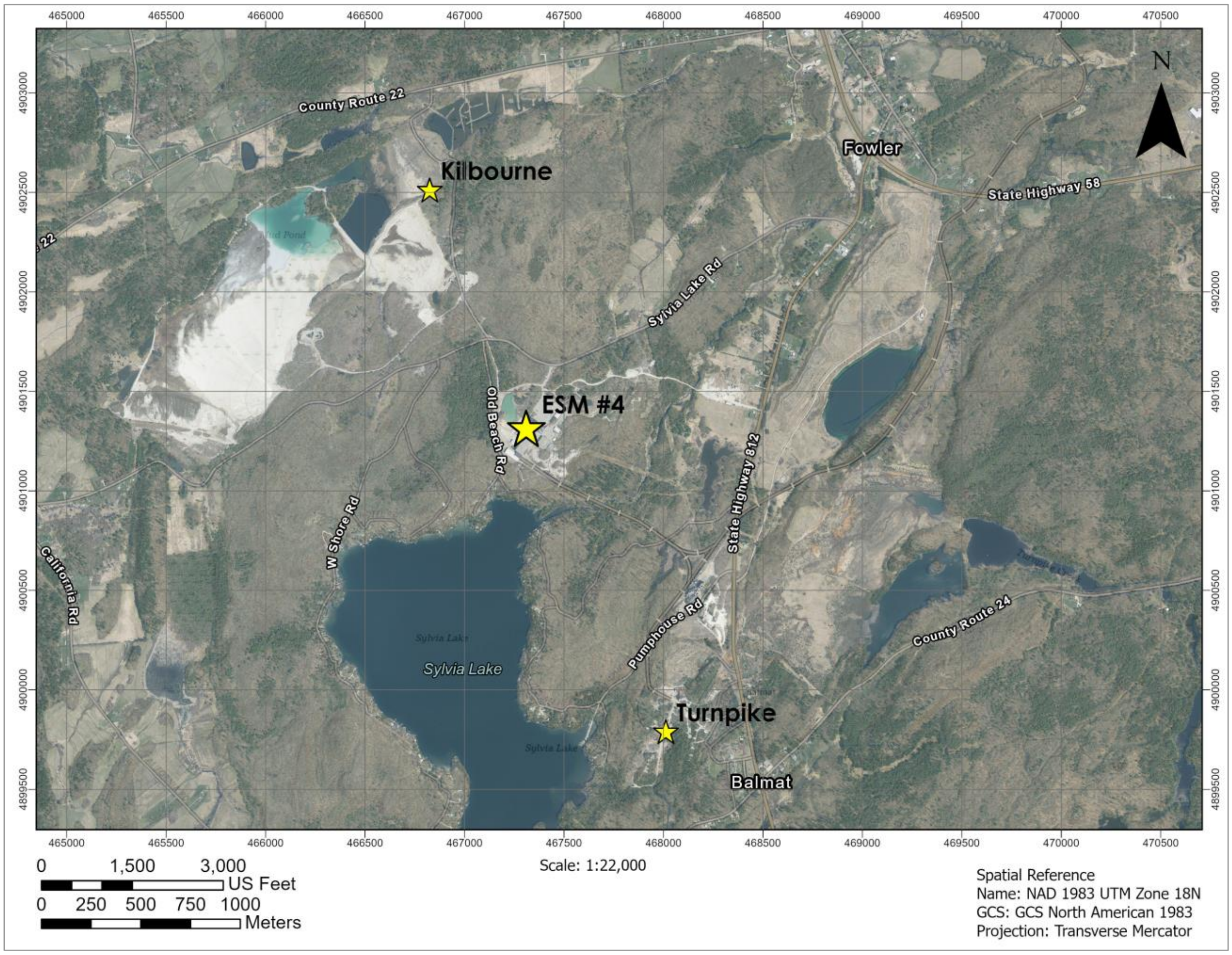
The Company's Turnpike Project is located within ESM's surface and mineral rights, located roughly 5,000 ft from the ESM #4 Shaft and mill. The Project area is adjacent to the #2 Shaft (Figure 5-3).

The Kilbourne Project is roughly 3,000 ft from the Company's #4 Shaft and mill and is within ESM's surface and mineral rights. The Project currently has no associated facilities (Figure 5-3).



Source: ESM 2024

Figure 5-2: Empire State Mines aerial view



Source: ESM 2024

Figure 5-3: Empire State Mine, Turnpike, and Kilbourne

6. History

6.1 Empire State Mines History

6.1.1 Management and Ownership

The ESM operation is wholly owned by Empire State Mines, LLC (formerly known as St. Lawrence Zinc Company, LLC), a subsidiary of Titan. A history of ownership is listed in Table 6-1.

Star Mountain Resources, Inc. purchased ESM 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 ESM. 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 ESM from Hudbay.
2016–Present	Titan Mining (US) Corporation	Purchased Balmat shares from Star Mountain and renamed Balmat mine to ESM.

Source: ESM 2024

6.1.2 Exploration History

In 1838, zinc was discovered in a prospect pit on the Balmat farm, which is located near the current location of Balmat #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 #2 Mine in 1928. In 1945, surface drilling, down-plunge from surface showings, intersected the Balmat #3 Mine Mineral Resources. A systematic fence-drilling program across the Sylvia Lake Syncline (perpendicular to the plunge) discovered the Mineral Resources of Balmat #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 #4 Mine in 1996, with the expansion of the Mud Pond zone. The New Fold and Mahler resources were later discovered in the #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.

6.1.3 Production History

Since 1915, several zinc mines have operated in the Balmat-Edwards-Pierrepont district, collectively now known as Empire State Mines, out of four mining camps. The mining camps are known as Balmat, Hyatt, Edwards, and Pierrepont. Mine access was primarily by shaft for both the Balmat and Edwards camps, and by portal for the Hyatt and Pierrepont camps. Shafts were added over the decades as mining deepened and additional discoveries were made. Zinc was first produced from the Edwards mine in 1915 and from the Balmat #2 Mine in 1930.

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 tons 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 Mt at 8.6% Zn since operations began in 1930. The greater Balmat-Edwards-Pierrepont district has produced more than 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 #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 #4 Shaft.

Table 6-2: Historic production totals by region

Region	Years Active	Tons	Grade (Zn%)	Zinc (lb)*
Balmat	1930-2001 2006-2008 2018-2023	36,029,247	8.59	6,186,537,996
Hyatt	1918-1922 1940-1949 1974-1983 1991-1998	1,205,526	8.24	198,695,031
Edwards	1915-1980	6,567,660	10.76	1,413,569,361
Pierrepont	1982-2001	2,657,527	16.29	865,686,479
Total	**-**	**46,459,960**	**9.32**	**8,664,488,867**

Source: ESM 2024

* Zinc pounds are theoretical pounds hoisted and not actual mill production totals.

Table 6-3: Empire State Mines annual production totals

Year	Balmat #4 Mine		Concentrate Produced	
	Tons	Grade (Zn%)	Tons	Grade (Zn%)
2018	187,854	7.94	23,932	58.19
2019	218,823	8.33	29,924	58.74
2020	323,414	8.58	45,161	59.35
2021	387,438	7.47	47,065	59.35
2022	425,022	7.54	52,547	58.79
2023	445,803	8.36	60,145	59.64

Source: ESM 2024

6.1.4 Historical Mineral Reserves

A list of the 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

6.2 Kilbourne History

The potential significance of the graphite mineralization at Kilbourne was first documented by ESM personnel in the second quarter of 2022. Surface exploration hole SX22-2621 drilled a 799.1 ft intercept of Unit 2 of the Upper Marbles (UM2) with elevated graphite mineralization observed. This mineralization was confirmed by assay prompting further review of historic drill records, where graphite had been commonly noted as a mineralogical component of UM2. During this preliminary data review, the Company reevaluated historical geophysical targets generated by Hudbay between 2009 and 2011. The previous exploration group had highlighted numerous electromagnetic highs. These anomalies correspond to the mapped surface expression of UM2.

Although there has been no historic graphite production or exploration on the property, the United States Geological Survey has a recorded iron and sulfur prospect pit on the property. The first documentation of this prospect was from Buddington in 1917 in his work on the pyrite and pyrrhotite deposits of St. Lawrence and Jefferson Counties. This was referred to as the Kilburn prospect, which has leant its name to Kilbourne.

6.2.1 Kilbourne Management and Ownership

The Kilbourne Project is within mineral rights owned by ESM, these mineral rights are subject to the same history as the Empire State Mines. As such, Section 6.1.1 is an accurate summary of the history of management and ownership for Kilbourne.

6.2.2 Kilbourne Exploration History

Graphite mineralization had not previously been targeted by Titan or its predecessors on the ESM properties. Review of historic drilling shows at graphite recorded as a mineralogical component of UM2.

6.2.3 Kilbourne Production History

There has been no historic graphite production at Kilbourne. Based on historic records, it appears that there was at least one small prospect pit for iron and sulfur in the early 20th century. A total of 800 tons is reported as being quarried (Buddington, 1917).

6.2.4 Kilbourne Historical Mineral Reserves

There are no historic mineral reserves on the Kilbourne Project.

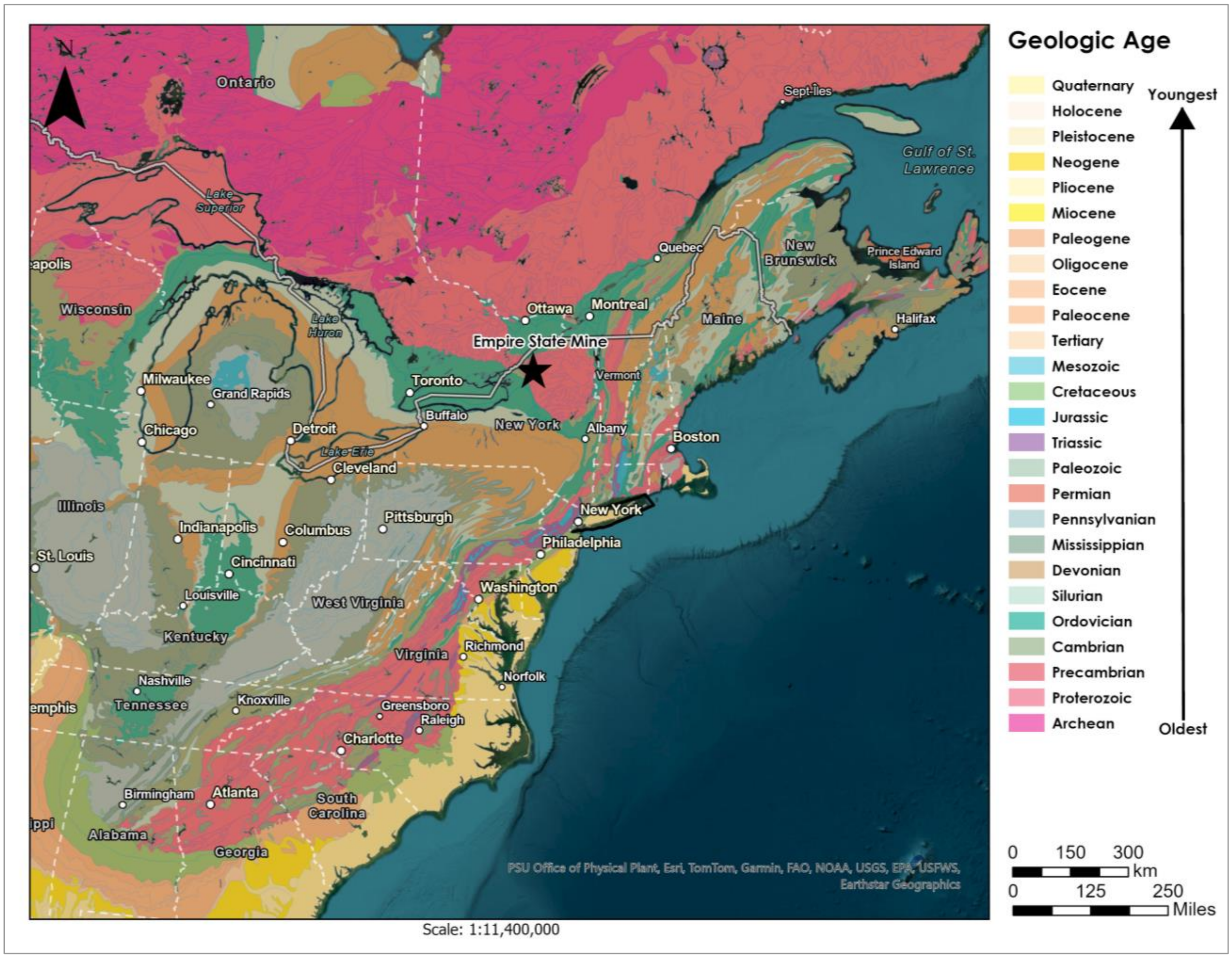
7. Geologic Setting and Mineralization

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.

Sulfide and graphite deposition is interpreted to have occurred contemporaneously with deposition of the rock units. The originally tabular sulfide deposits were intensely deformed and metamorphosed along with their host rocks through eons of varying tectonic forces. The stratiform graphite mineralization within the region likely formed as a result of these same tectonic forces, with syndepositional organic carbon reaching suitable metamorphic temperatures and pressures to form graphitic carbon. Historically, the primary mineral of interest in the district was sphalerite.

The mine is located near the eastern edge of the Canadian Shield, a vast expanse of very old, exposed bedrock that 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 eras, 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, 2009). In Figure 7-1, the Canadian Shield is the pink and red band encircling Hudson Bay.



Source: ESM 2024

Figure 7-1: Regional geology setting

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 that 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 et al., 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 sulfide and graphite 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 et al., 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 et al., 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 et al., 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 and Fitzgerald, 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 et al., 1992). The Lowlands have been metamorphosed to amphibolite grade, the Highlands to higher granulite grade (McLelland et al., 2010). ESM and Kilbourne are 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, 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, 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 sulfide and graphite mineralization are 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. Table 7-1 shows the mine stratigraphic column, which is divided into 16 units.

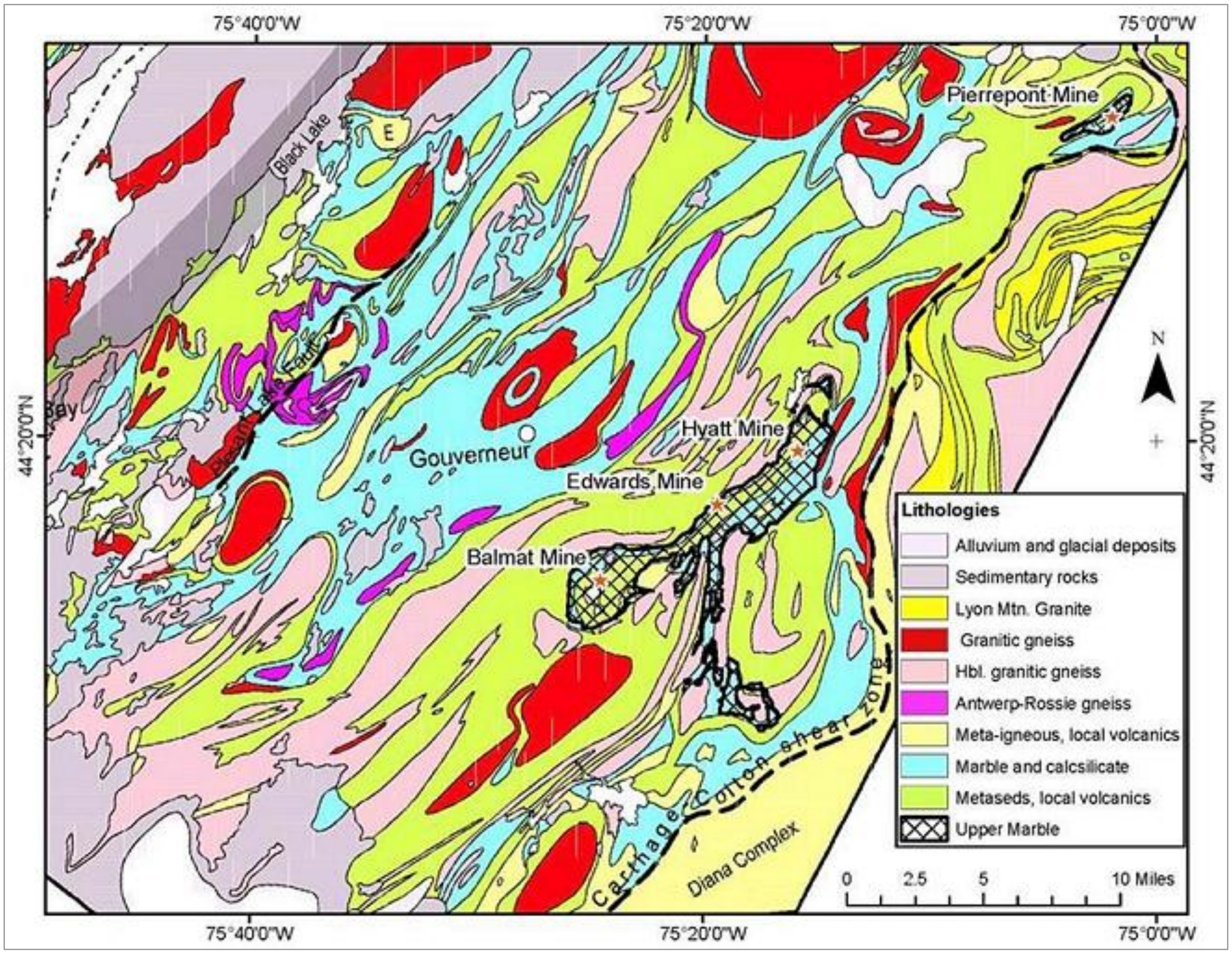
Table 7-1: Upper Marble stratigraphic sequence

Formation	Thickness (ft)	Lithology Description
€p	200	Potsdam Sandstone; siliceous hematitic breccia at base
UM16	200	"Median Gneiss"; quartz-biotite-diopside-scapolite
UM15	50	Phlogopitic calcitic marble, aka "Mica Hanging Wall"
UM14	360	Calcitic marble with diopsidic quartz layers
UM13	80	Talc-tremolite-anthophyllite schist; anhydrite
UM12	150	Medium to coarsely crystalline pale gray to white dolomite
UM11	300	Diopsidic quartz interlayered with anhydrite and marbles
UM10	50	Pea-green serpentinized calc-silicate ± anhydrite ± biotite ± tremolite
UM9	60	Medium to coarsely crystalline white dolomite
UM8	130	Diopsidic quartz interlayered with marbles ± tremolite
UM7	120	Distinctively fetid and dark gray crystalline dolomite
UM6	700	Silicated dolomite with distinct and persistent sub-units ± serpentine ± anhydrite
UM5	170	Medium to coarsely crystalline white dolomite
UM4	300	Diopsidic quartz interlayered with dolomitic marbles
UM3	400	Medium to coarsely crystalline white to gray dolomite
UM2	100	Graphitic pyritic schist ± quartz ± garnet ± silliminite ± feldspar
UM1	20	Medium to coarsely crystalline white to gray dolomite
HPG	unknown	Hermon Granite
PHG	unknown	Popple Hill Gneiss; migmatitic quartz-biotite-oligoclase gneiss

Source: ESM 2024

7.3 Property Geology

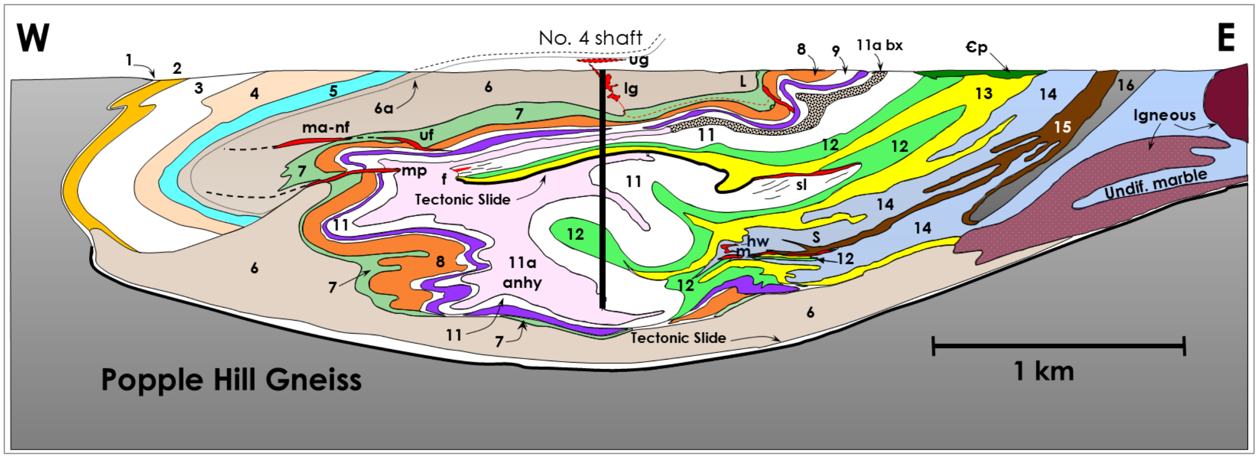
As a result of intense tectonism, the Upper Marble Formation is extensively deformed. The predominant structure is the Sylvia Lake Syncline, a major south-west to north-east trending fold lying between the original Balmat mine and the Edwards mine. Aerial exposure of the Upper Marble Formation is limited, 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. Graphite mineralization occurs as weakly disseminated flakes within many of the marbles and dolomites, it occurs in the highest grades in Unit 2 of the upper marble at Hyatt and ESM. In Figure 7-2, 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.



Source: ESM 2024

Figure 7-2: Local geologic setting

The sulfide deposits and graphite occurrences 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 sulfide 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 7-3 is a cross-section through the Sylvia Lake Syncline that illustrates the extent of deformation of the Upper Marble Formation.



Source: ESM 2024

Figure 7-3: Section through the Sylvia Lake Syncline

7.4 Mineralization

As the details of the Geologic Setting (Section 7.1), Regional Geology (Section 7.2) and Property Geology (Section 7.3) are shared by ESM's zinc operation and the Kilbourne Project, they have been grouped together prior to this section.

7.4.1 ESM Mineralization

The mineralization at ESM has been classified as sedimentary exhalative (Sedex) in origin. The composition is 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-sulfide 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. The zinc mineralization extends from the surface down to a depth of 5,700 ft below surface. The zones are aerially scattered and all zones except NE Fowler and Cal Marble are connected by existing development to the shaft. The zones range in thickness from 2 ft to 50 ft with an overall plunge between 20° to 25° with local dips ranging from 0° to 90°. The deposit footprints are up to 500 ft wide and 9,000 ft long. The veins can display considerable geometrical variability depending on the degree of folding. 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 Sylva Lake Syncline scale 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 contains highly variable amounts 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 majority of sphalerite mineralization at Balmat has been remobilized to some extent with most of the modeled mineralization categorized as secondary durch.

The average mined grade for the Balmat mines is 8.7% Zn, while the average for the greater Balmat-Pierrepont district is even higher at 9.3% Zn.

Galena is associated with all the deposits in very low concentrations and the mineralization style varies slightly between the orebodies. The secondary durch veins typically are surrounded by a low-grade aureole of galena enriched diopside, colloquially known and logged as “Pb-rock”. Visible galena is rare within the durch itself and is more characteristic of primary stratiform mineralization where it can grade up to 6% Pb. Galena is most prevalent in the #2 Mine and the Fowler deposit as shown in Figure 7-4.

Pyrite is also associated with all the deposits and zoned similarly as galena with the highest concentrations at the #2 Mine and lowest concentrations in the durch massive sulfides.

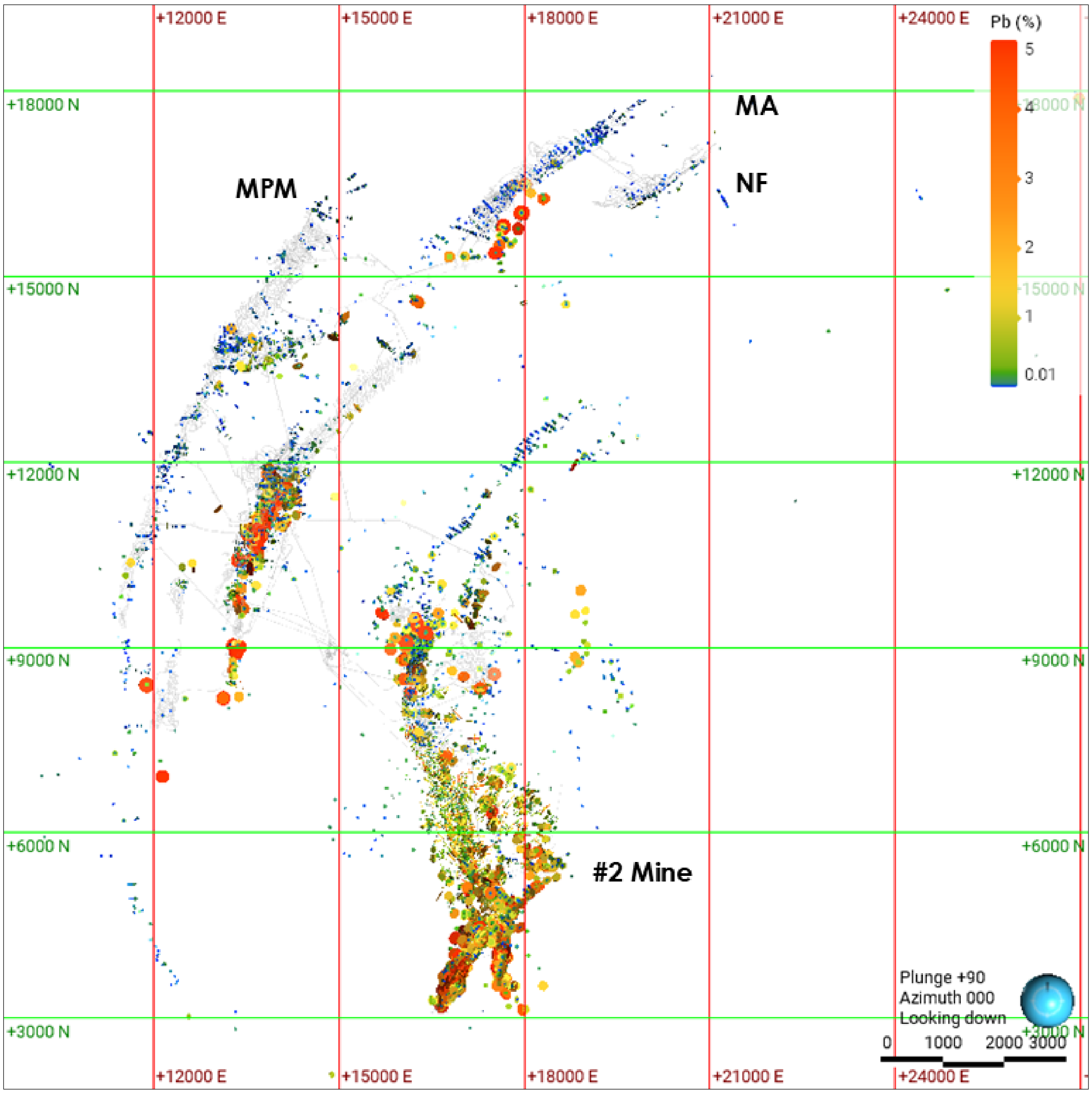


Figure 7-4: Plan view showing assay Pb (%) grade variation within the Sylvia Lake Syncline

7.4.2 Kilbourne Mineralization

Graphite mineralization at Kilbourne, and elsewhere within the Grenville Province, is believed to be the result of metamorphic processes on existing organic carbon found within the now metamorphosed sedimentary lithologies. This syndepositional source of carbon has resulted in stratiform graphite mineralization. In the nomenclature of the Balmat-Edwards District, the mineralized horizon is the Upper Marble Unit 2.

Unit 2 is currently divided into three sub-units, with transitional zones between each. The names assigned are based on their current relative positioning. The overall thickness of the unit varies substantially both along strike, and along dip. With the thinnest Unit 2 intercepts totaling 25 ft, and the thickest intercepts totaling 312 ft. These fluctuations are interpreted to be the result of the ductile behavior of the rocks during metamorphism, a behavior documented frequently in the units hosting the Company's zinc mineralization.

(A) The Upper Graphitic Schist (UGS) is a granulite composed of quartz-biotite/phlogopite-graphite-sillimanite-pyrite-pyrrhotite with rare garnet. The unit has a dark grey color, with discrete blebs of sillimanite often altered to clay. Graphite is generally coarser grained than in the lower mineralized unit and makes up an estimated 1.5%–3% Cg of the lithology. Grades as high as 13.5% Cg have been returned in assay. The average thickness of the Upper Unit is 57.1 ft.

The transitional zone leading into the middle unit is marked by an increase in clay/chlorite altered/replaced sillimanite, and garnets. A stronger fabric is also documented. Graphite remains present but is often at a lower percentage than in the upper mineralized unit. There is often a band of higher-grade graphite mineralization near the lower contact with the middle zone.

(B) The Phlogopitic Garnet Schist (PGS) is a visually distinguishable phlogopite/biotite schist with a strong wavy fabric and pegmatitic boudins/inclusions. The sub-unit is dominated by a dark ferromagnesian mica with quartz-sillimanite-garnet-graphite with less common pyrite/pyrrhotite. Graphite mineralization is present, often coarser grained than the upper and lower units, but is sparsely disseminated, contributing to <1% of rock groundmass. The average thickness of the Middle Unit is 61 ft.

The transitional zone between the middle and lower units is similar in appearance and composition to the transitional zone between the upper and middle units, marked by a weaker fabric, higher Cg, and discrete clay/chlorite altered/replaced sillimanite.

(C) The Lower Graphitic Schist (LGS) is a dark grey to black, massive granulite. The constituent minerals are likely ferromagnesian mica and quarts, with fine grained graphite and possibly fine-grained sulfide, contributing to the dark color of the rock. Graphite grades range from 1.5%–3% Cg, with samples as high as 11.3% Cg recorded. The average thickness of the Lower Unit is 29.0 ft.

In all of the Kilbourne sub-units iron sulfides (pyrite>pyrrhotite) are present. Trace sphalerite has also been documented in veinlets and rarely as disseminated mineralization.

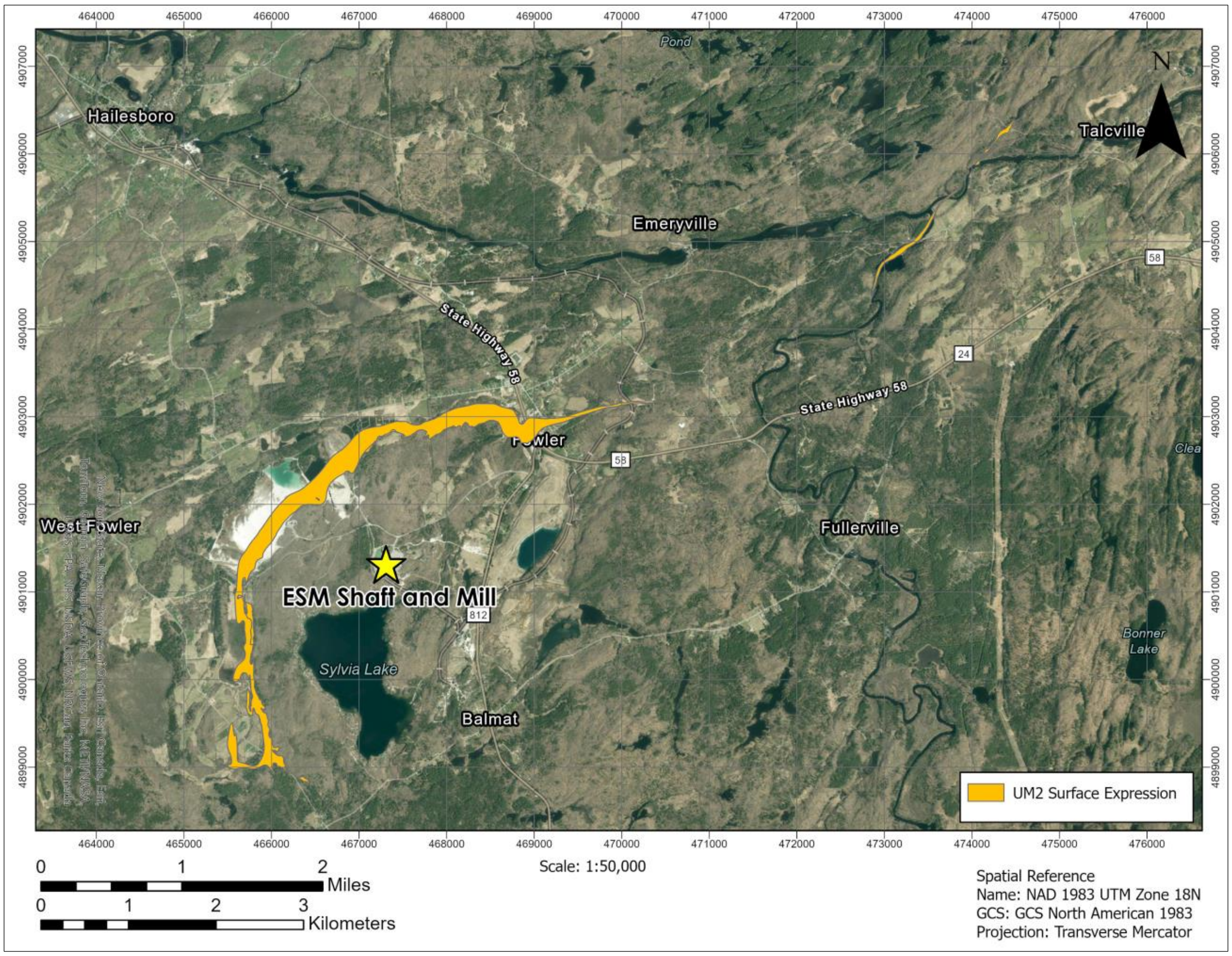


Figure 7-5: Upper Marble 2 mapped surface expression

8. Deposit Types

8.1 Zinc

Initially formed in a marine sequence of carbonates and evaporates, the ESM deposits are broadly classified as 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 United States Geological Survey (USGS) open-file report, Emsbo (2009) 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 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 (Figure 8-1). 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 sulfide.

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 et al., 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 et al., 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).

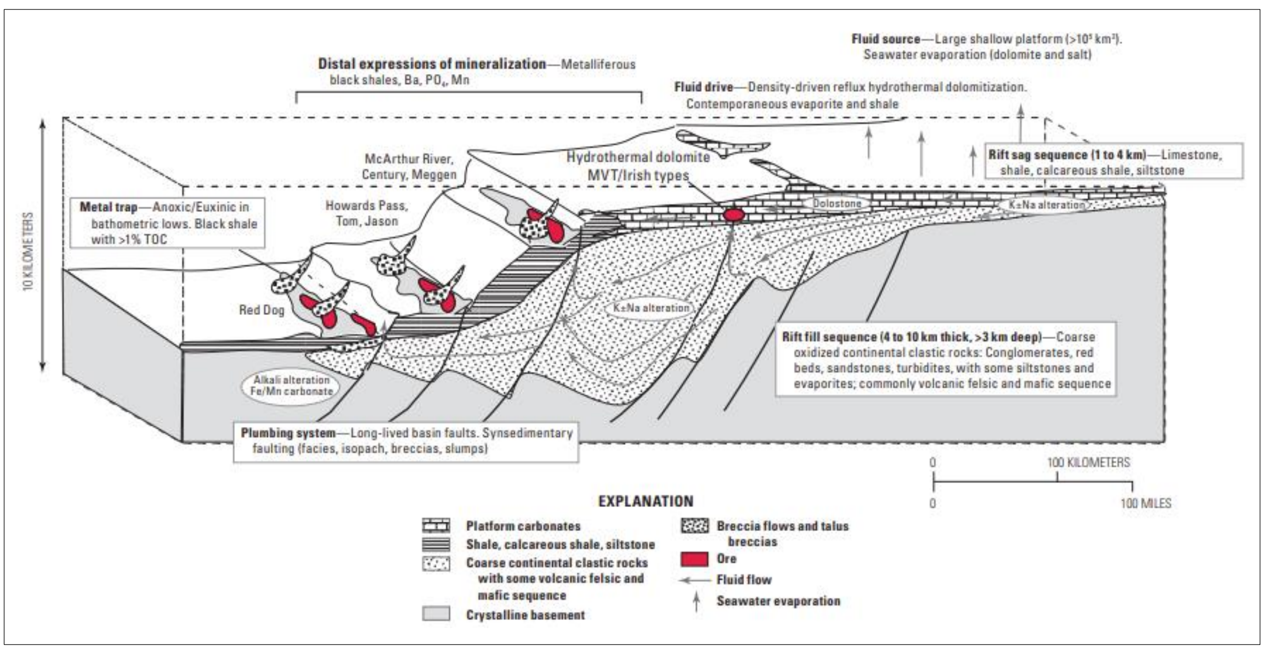
Mass balance studies indicate that large volumes (thousands of km^3) of clastic sediments are required to generate enough metals to form a Sedex deposit. Fluid inclusion studies indicate that Sedex deposits are formed from brines with temperatures between 100°C to 200°C. Metals are most soluble in this 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 allochthonous; 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 that 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 kilometers. 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 kilometers, represent bathymetric lows. Sedex deposits typically occur in third-order basinal areas within a few to tens of kilometers 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 sulfide (H_2S) is generated in this depositional environment by bacterial sulfate reduction. Bacteria living in the highly carbonaceous distal sediments or thermal vents oxidize the organic compounds in the shale while reducing sulfate (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 sulfide (sphalerite, (Zn, Fe)S)) and lead sulfide (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 sulfate (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.



Source: ESM 2018

Figure 8-1: Illustration of the process of formation of Sedex deposits

8.2 Graphite

Graphite is a naturally occurring form of pure carbon and is a common constituent mineral in metasedimentary, and sedimentary rocks. The mineral occurs as black crystal flakes and masses. It is chemically inert, thermally stable, has a high electrical conductivity, and lubricity. These properties have made it suitable for many industrial applications including electronics, lubricants, metallurgy, and steelmaking (Robinson et al., 2017). Natural graphite deposits are classified into three categories: 1) amorphous (microcrystalline) graphite deposits; 2) crystalline flake graphite deposits; and 3) lump (vein) graphite deposits. The mineralization at Kilbourne, and elsewhere in the Grenville rocks of North America, is largely classified within the crystalline flake category.

Flake graphite deposits make up a large proportion of worldwide graphite production (Robinson et al., 2017). These deposits are derived from carbonaceous sediments that undergo regional metamorphism and reach temperatures and pressures that allow for the crystallization of fully ordered graphite, and the recrystallization of the host rocks (Hoefs and Frey, 1976). These conditions are met at amphibolite facies metamorphic grades, where pressures are at or exceeding 2–10 kilobars and temperatures are at or exceeding 500–800°C. Most flake graphite deposits are in Precambrian crystalline metamorphic rocks that reached or exceeded the amphibolite facies (Robinson et al., 2017).

The carbon in these deposits was introduced during sedimentation as organic materials. The depositional environment of these sedimentary units includes sediment-starved basins with low-oxygen levels at depth allowing the accumulation of organic-rich sediments. This shares similarities to the genetic model of Sedex deposits are shown in Figure 8-1. As sea level rises relative to land during periods of marine transgression the carbonaceous sediments are buried with little to no erosion. These rocks are buried further as the basin develops, and later subjected to regional metamorphism. The primary host lithologies for flake graphite deposits are these metamorphosed sedimentary rocks (quartzite, aluminous paragneiss, and marble) (Simandl et al., 2015).

The syndepositional origin of the graphite creates stratiform bodies of mineralization that can be thousands of meters long, with thicknesses determined by initial basin geometries and later the ductile behavior of the host rocks. Economic deposits are generally tens of meters thick, and hundreds of meters long (Robinson et al., 2017). In addition to the dimensional variability documented in these deposits, grade can be quite variable as well. With graphitic carbon grades ranging from trace levels <1% graphitic carbon (Cg) to grades higher than 15% Cg and grades up to 60% Cg documented in rare instances globally (Robinson et al., 2017). Grade variability is also common on the deposit scale, with zones of higher and lower grade mineralization, possibly related to the total organic carbon variability in the protolith, possible on the meter scale. High-grade zones are also associated with structural controls such as lithologic contacts, lenses within fault zones, and segregations in fold crests suggesting that there may be carbon enrichment associated metamorphic fluids along these structural pathways (Robinson et al., 2017).

9. Exploration

9.1 Zinc

Exploration activities within the Balmat-Edwards district, and surrounding region include the digitization and review of historic exploration and mine data, surface geochemical sampling, surface hydrogeochemical sampling, and review of airborne geophysical data. Titan has also completed surface and underground exploration drilling; this is expanded upon in the drilling section (Chapter 10) of this report.

Regional zinc exploration in the Balmat-Edwards marble belt, as well as the northwest Adirondacks resulted in the discovery of five new mineralized bodies within the last 30 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.

9.1.1 Historic Data Review

Titan has access to over 100 years of data from past operators and explorers covering much of the Adirondack Lowlands. This includes records of drillholes, mine maps, surface geologic maps, geochemical samples, and geophysical data. An effort to digitize this data has been in action since the mine restarted and has generated multiple viable targets to date. This includes the Company's prospective Turnpike Project, and the reactivation of the #2 ore body at depth (N2D).

Historic regional and district geochemical data have also been used in the development of surface exploration programs. This includes ESM's drilling at the North Gouverneur, and the soil programs at Moss Ridge, Pork Creek, and North Gouverneur.

Based on public data and company records, a table of occurrences has been produced highlighting lead and zinc occurrences within the district. A total of 41 occurrences, prospects, and past producers have been identified in the district. Many of past producers for lead are individual operations located along the same mineralized feature or are part of a sheeted vein system. These areas have been grouped into the Macomb-Brown Farm, Rossie, and Bigelow School sites.

Table 9-1: Occurrences highlighting lead and zinc occurrences within the district

Site Name	Commodity	Status
Zinc		
Balmat (now ESM #2, #3, #4)	Zinc	Producer
Edwards	Zinc	Past Producer
Hyatt	Zinc	Past Producer
Pierrepont	Zinc	Past Producer
Pleasant Valley	Zinc	Prospect
Bostwick	Zinc, Copper	Prospect
Parker	Zinc	Occurrence
McGill (Pork Creek)	Zinc	Occurrence
Woodcock/Webb	Zinc	Occurrence
Lead		
Macomb – Brown Farm	Lead	Past Producer
Rossie (Coal Hill & Victoria)	Lead	Past Producer
Bigelow School	Lead	Past Producer
Mineral Point	Lead	Occurrence
Redwood	Lead	Occurrence
Nelson Farm	Lead	Occurrence
Wright Farm	Lead	Occurrence

Source: USGS MRDS[1]; ESM 2024

[1] MRDS: Mineral Resources Data System

9.1.2 Surface Geochemical Sampling

9.1.2.1 Soil Sampling

In 2022, Titan contracted GroundTruth Americas, a subsidiary of GroundTruth Exploration, of Dawson, YT, to implement a soil program developed by ESM personnel. The program targeted areas of historic mining activity and/or geochemical prospectivity. A total of 1,961 samples were collected in the fourth quarter of 2022 (Table 9-2 and Figure 9-1). The majority of these samples were collected from the regional targets Beaver Creek, Bostwick, Maple Ridge, Moss Ridge, and North Gouverneur (1,751 samples total). Pork Creek was the only soil target within the district (210 samples total).

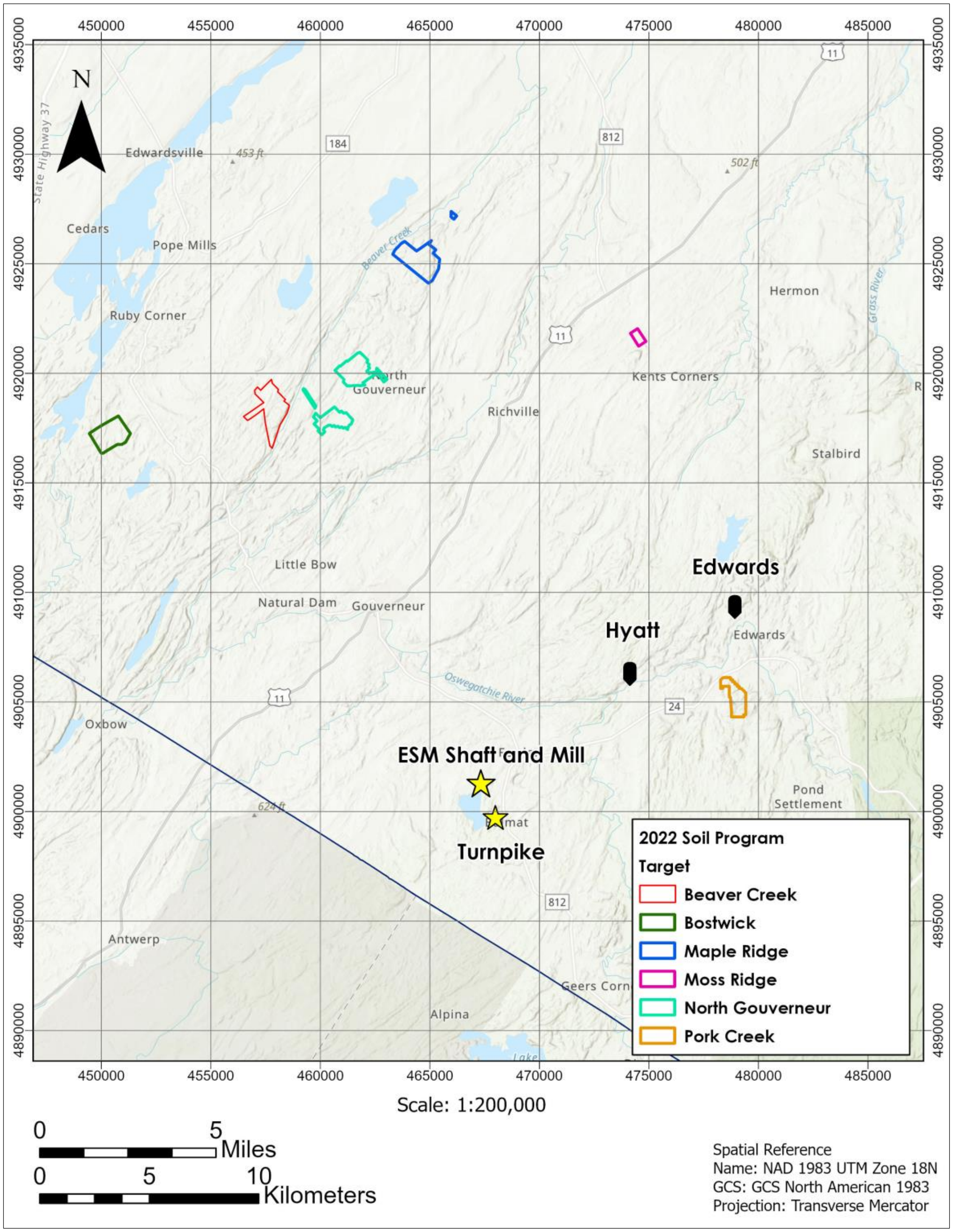
Samples were packaged at the mine site and sent to the Bureau Veritas laboratory in Reno, Nevada, where they were dried, and 100 g was sieved from the initial sample. The sieved sample was then sent to the Bureau Veritas facility in Vancouver, British Columbia for assay. The analytical method used was AQ200, an aqua regia digest followed by ICP-OES/MS[1] analysis testing for 37 elements (Ag, Al, As, Au, B, Ba, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Se, Sr, Te, Th, Ti, Tl, U, V, W, and Zn).

Table 9-2: 2022 Soil sampling totals and high Zn (%) values

Target	Total Samples	Highest Zinc Value (%)
District	**1,751**	**1.67**
Beaver Creek	531	0.49
Bostwick	197	0.04
Maple Ridge	135	0.03
Moss Ridge	206	1.67
North Gouverneur	681	0.46
Trend	**210**	**0.33**
Pork Creek	210	0.33

Source: ESM 2024

[1] ICP-ES/MS: Inductively Coupled Plasma – Optical Emission Spectrometry / Mass Spectrometry



Source: ESM 2024

Figure 9-1: Location of 2022 soil sampling programs relative to ESM

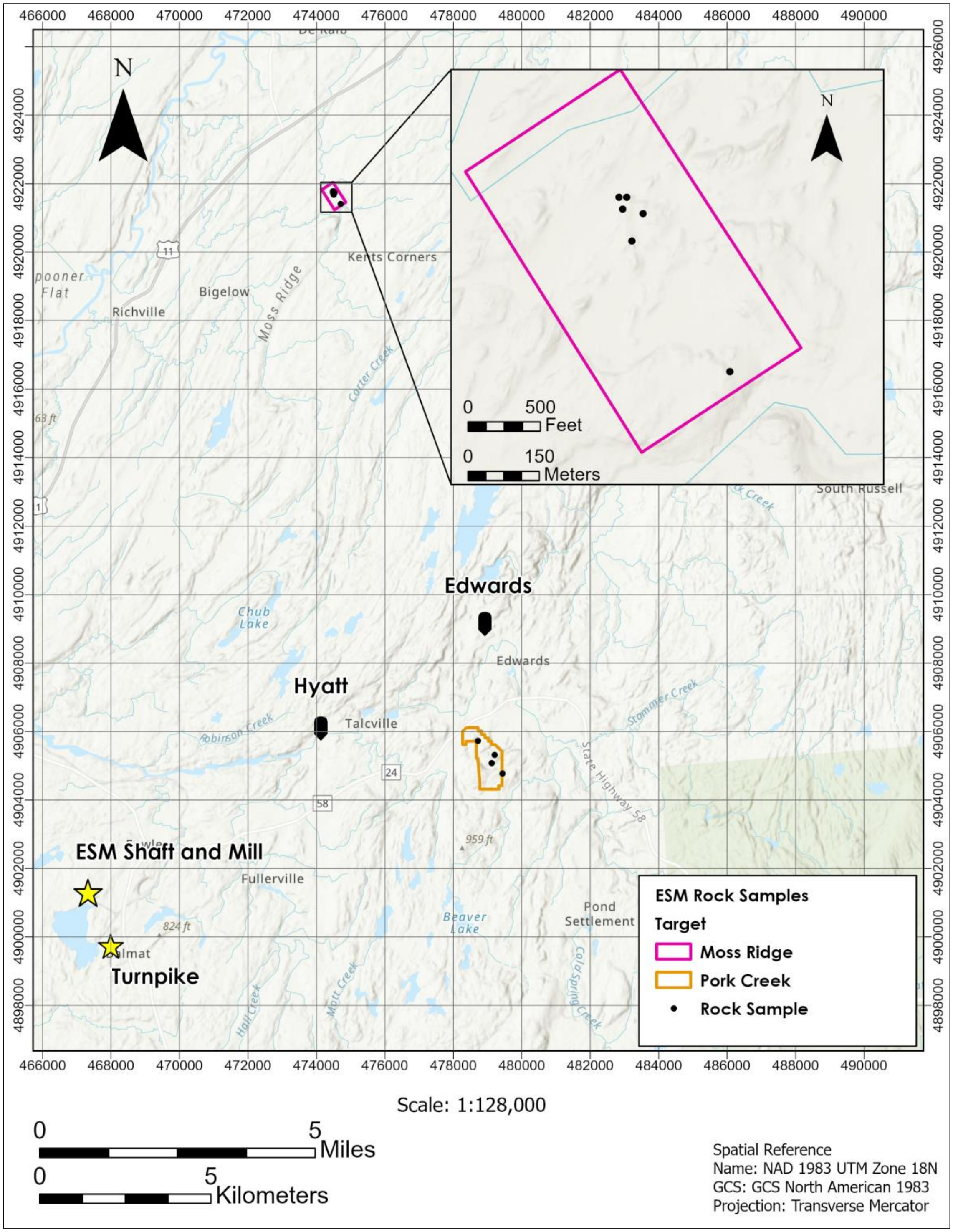
9.1.2.2 Rock Sampling and Prospecting

Prospecting targets have generally been generated to follow up on historic geochemical or geologic documentation of Zn occurrences and anomalies, investigate and ground truth anomalous soil samples from the 2022 program, and perform due diligence on potential property acquisitions. Between 2021 and 2024, 13 samples have been submitted for laboratory analysis, these were all collected while investigating soil anomalies identified through the 2022 soil program (Table 9-3 and Figure 9-2).

Table 9-3: Rock samples by target with highest zinc values

Target	Total Samples	Highest Zinc Value (%)
District	**9**	**4.53**
Moss Ridge	9	4.53
Trend	**4**	**0.02**
Pork Creek	4	0.02

Source: ESM 2024



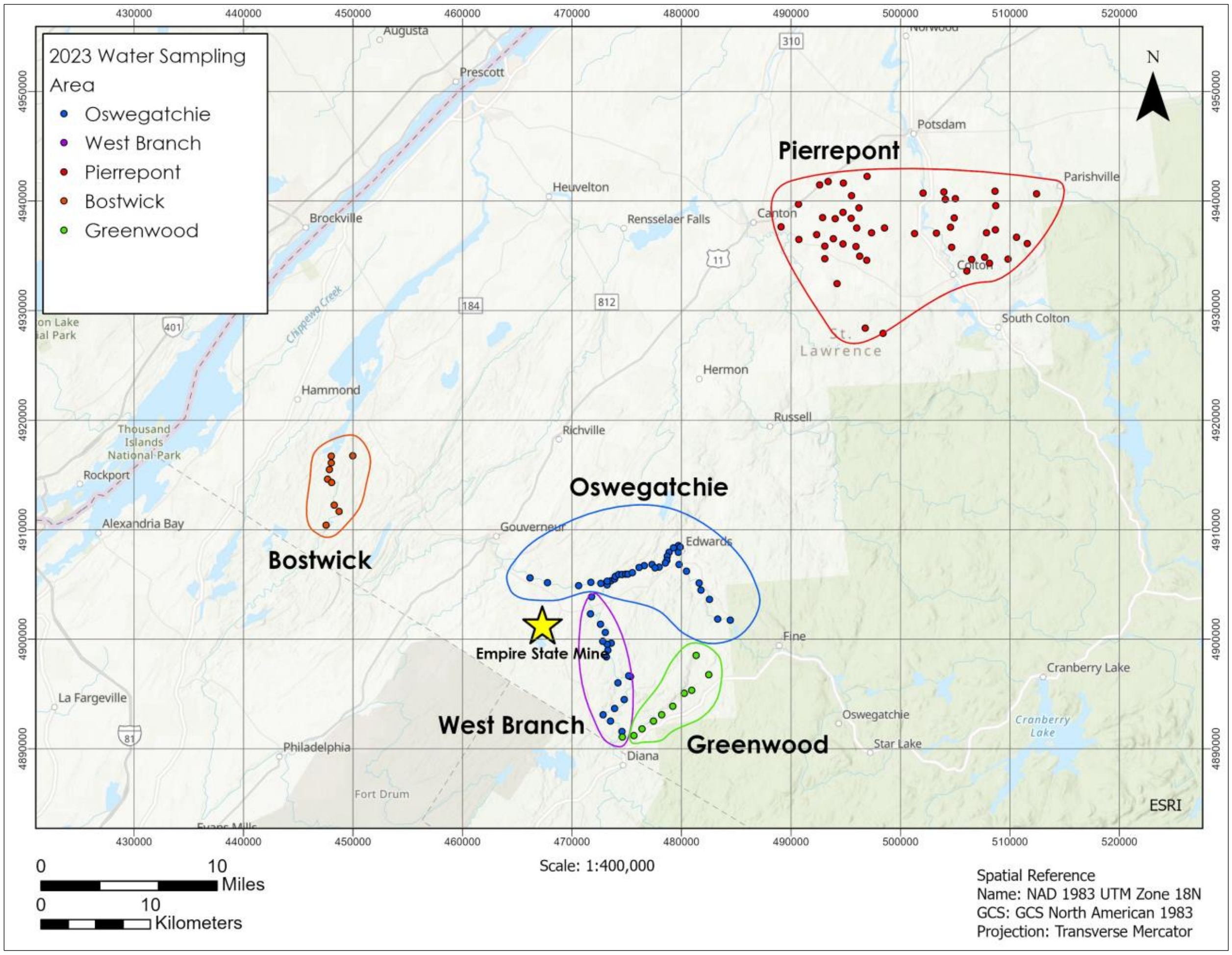
Source: ESM 2024

Figure 9-2: Location of rock samples by target

9.1.3 Hydrogeochemistry

In August 2023, ESM, in collaboration with Juniata University of Pennsylvania, collected 132 ground and mine water samples to evaluate the isotopic ratios of zinc and copper contained within the solution. This first phase of testing included an orientation study along the Oswegatchie river to determine if there were detectable signatures within close proximity to known mineralized occurrences. A total of 120 samples were taken upriver and downriver from the historic mines at Edwards and Hyatt, the known occurrence at Pleasant Valley, and along the river over projections of the ESM #4 stratigraphy. A total of 12 samples were also collected from surface and mine waters at Turnpike, and from mine water sources within the ESM #4 Mine.

Regional samples were collected along the West Branch of the Oswegatchie over favorable marbles, from surface waters in areas with under tested marbles at Greenwood, near the historic Bostwick deposit, and in areas with glacial or Paleozoic cover near the Pierrepont mine, as shown in Figure 9-3.



Source: ESM 2024

Figure 9-3: 2023 Water sampling sites by area

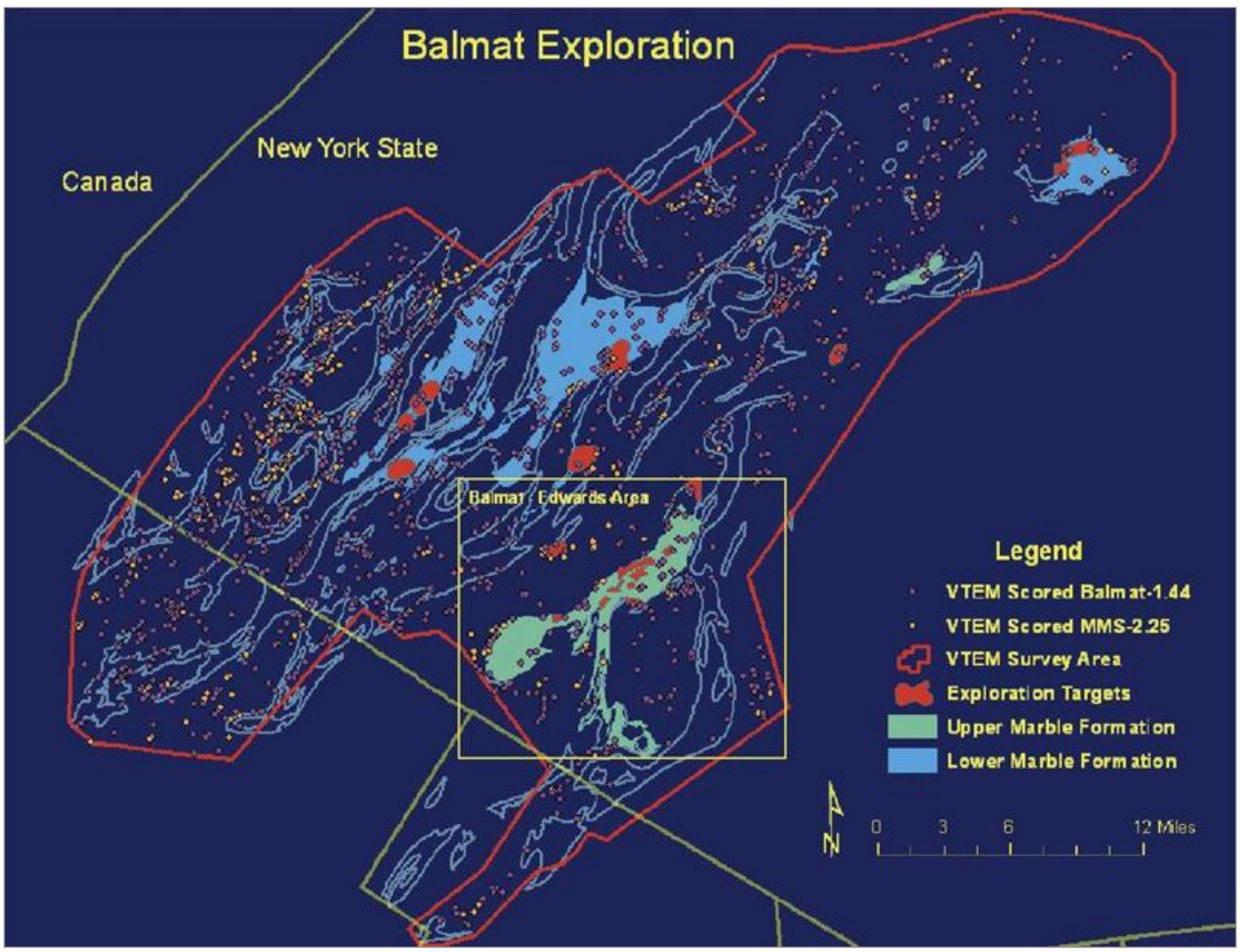
9.1.4 Airborne Geophysics

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 footprint of the mine. Figure 9-4 shows the area covered by the geophysical survey and areas where low resistivity was recorded (Rivard and Stephens, 2013).



Source: SLZ 2018

Figure 9-4: Geophysical survey area

In 2022, the Company began the re-evaluation of the raw data from this survey with the goal of filtering targets using two known bodies of unmined zinc mineralization. These are the Pleasant Valley Deposit, and the Bostwick Creek deposits. To date, the western third of the 2013 survey area has been re-evaluated.

9.1.5 Exploration Potential and Targeting

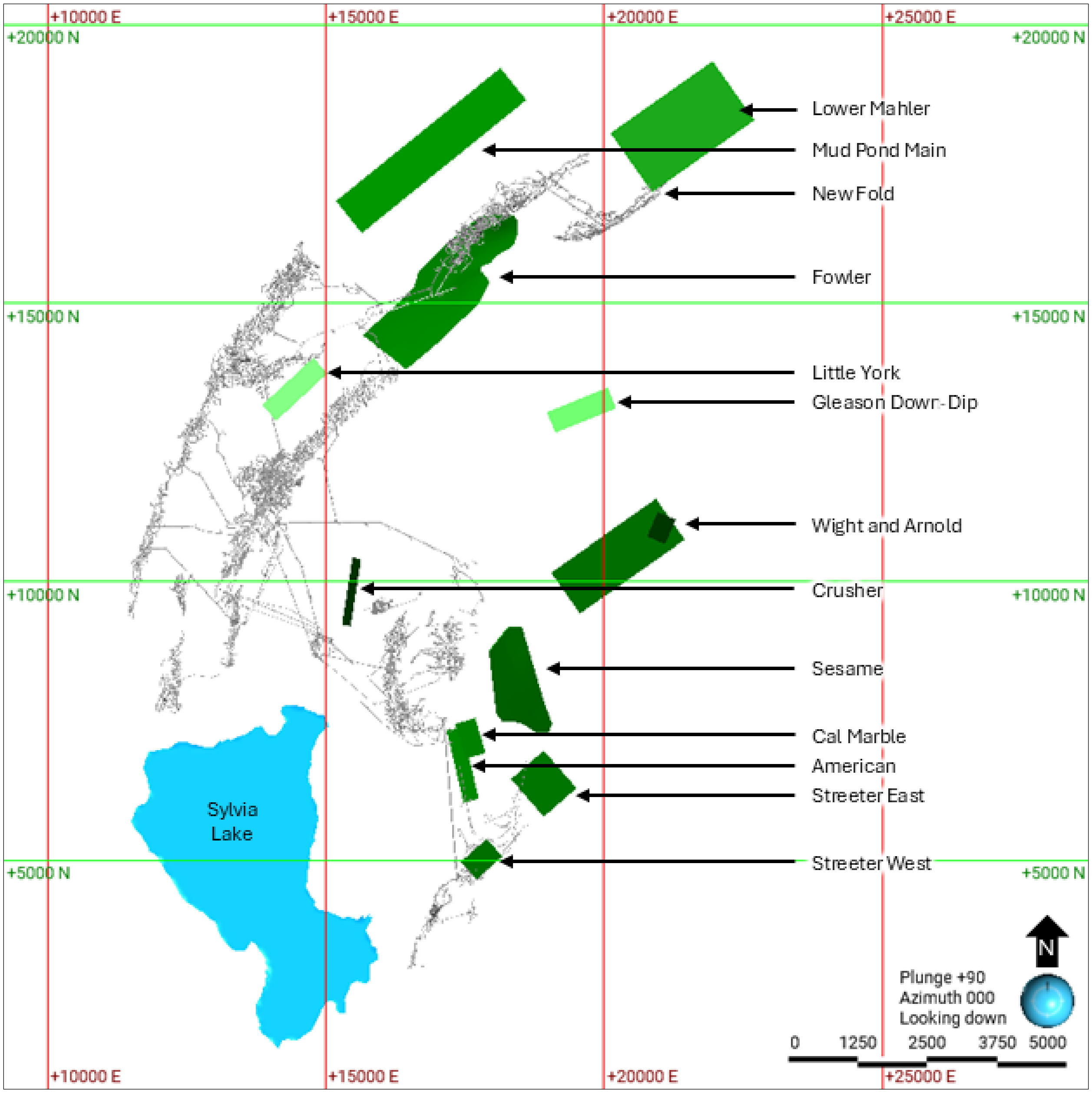
9.1.5.1 Near-Mine Exploration Targets

Several exploration targets at ESM have been identified based on extensions of open mineralized horizons at various depths and promising historic hits in the drillhole database. The targets shown in Figure 9-5 are conceptual in nature. There has been insufficient exploration to define a mineral resource in these areas and it is uncertain if further exploration will result in the targets being developed into mineral resources. The quantity and grades are based on past producing horizons of geological equivalence and are listed in Table 9-4. This list is not exhaustive and subject to change as new drilling information is available.

Table 9-4: Near-mine exploration targets

Target	Tons (kt)	Grade (Zn %)
American	50 - 60	8 - 12
Cal Marble	50 - 60	10 - 14
Crusher	400 - 450	10 - 14
Fowler	700 - 750	5 - 9
Gleason Down-Dip	300 - 350	13 - 17
Little York	300 - 350	14 - 18
Lower Mahler	550 - 600	16 - 20
Mud Pond Main	650 - 700	9 - 13
New Fold	250 - 300	15 - 19
Sesame	550 - 600	7 - 11
Streeter East	550 - 600	7 - 11
Streeter West	20 - 30	7 - 11
Wight and Arnold	400 - 450	10 - 14

Source: ESM 2024



Source: ESM 2024

Figure 9-5: Near-mine exploration targets shown in green, mine workings in grey

9.1.5.2 District Exploration Targets and Potential

Exploration targets within the district have been broken out into two groups: targets within the Balmat-Pierrepont trend, and targets within the greater district. In addition to geologic, geochemical, and historical data, mineral and surface ownership plays an important role in target generation. Currently the Company has ten drill targets within the Balmat-Pierrepont trend that are within current mineral rights ownership. In addition to the targets on trend, the Company has four regional drill targets with mineral rights access.

In addition to drilling surface geochemical sampling, hydrogeochemical sampling, geologic mapping are all tools to be employed on the over 80,000 acres of mineral rights within St. Lawrence County.

Table 9-5: Exploration targets

Drill Targets	Target Type
Balmat – Pierrepont Trend	
Pleasant Valley	Testing extensions of known mineral occurrence
Pork Creek	Testing historic mineralization along strike, and favorable stratigraphy
Bend	Testing historic mineralization along strike, and favorable stratigraphy
Sully	Testing favorable stratigraphy
Hydro Plant	Testing structure in UM14
58	Testing favorable stratigraphy
Hyatt	Testing UM14
Edwards	Testing mineralized extensions at depth, and stratigraphy
Bingo Road	Testing historic mineralization along strike, and favorable stratigraphy
Side Pocket	Testing UM14
R&G Club	Testing structure in UM14
District	
Moss Ridge	Testing geochemical anomaly, and mineralized breccia at surface
Greenwood	Testing favorable stratigraphy
Maple Ridge	Testing stratigraphy and structure
Beaver Creek	Testing geophysical and geochemical anomaly

9.2 Graphite

Exploration activities at Kilbourne include the review of historic exploration and mine data; collection of surface geochemical samples through trenching, the review of airborne geophysical data, and the sampling of retained UM2 drill intercepts from past exploration programs. The Company has also completed 39 drillholes testing the graphite mineralization at Kilbourne. This drilling, along with the resampling, and trenching are included in Chapter 10 of this report.

9.2.1 Kilbourne Historic Data Review

Nested in the same dataset that has generated the regional and near-mine zinc targets (Section 9.1.1), the Company has identified historic drill records, and geologic maps documenting graphite occurrences within the district. Chief among these is the Kilbourne Project, with graphitic intercepts recorded in historic drill logs and surface maps.

The Company continues to evaluate the potential of the district for additional graphite targets using historic drill logs, historic reports, geologic maps, and geophysical data to generate future targets for exploration.

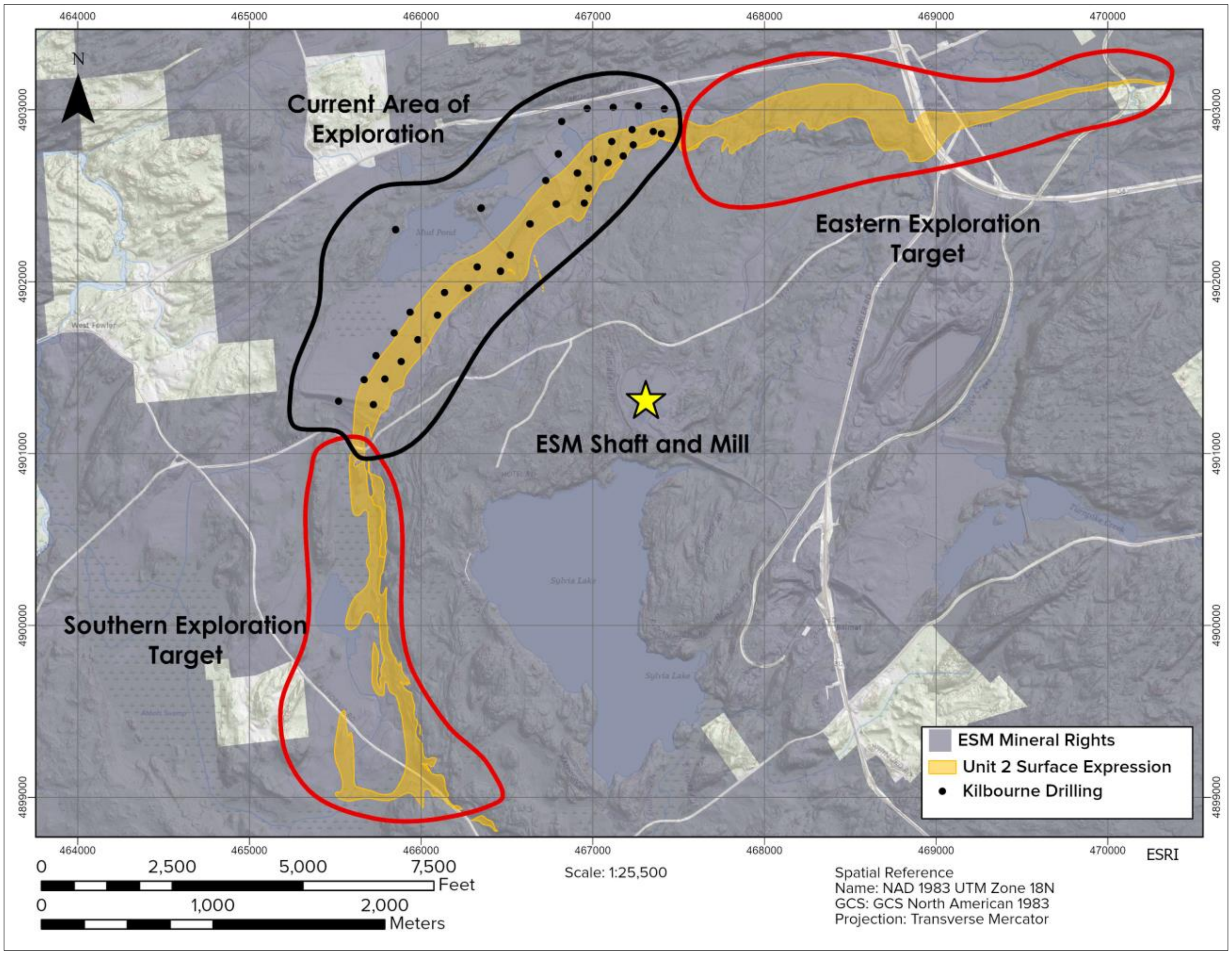
9.2.2 Airborne Geophysics

The airborne geophysical survey described in Section 9.1.4 has been evaluated for potential graphite occurrences, matching recorded electromagnetic (EM) highs with geologic units that have documented graphite mineralization. This overlap of high EM anomaly, and documented graphite mineralization is demonstrated at Kilbourne.

The re-evaluation of geophysical data that began in 2022 also aimed to highlight areas with electromagnetic signatures likely related to graphitic carbon mineralization. This effort used known occurrences and historic mines in the district to help identify additional graphite targets within the western most third of the district.

9.2.3 Exploration Potential and Targeting

The Company has tested roughly 8,250 ft of near surface mineralization at Kilbourne. The mapped surface expression of UM2 continues an additional ~7,500 ft to the south, and ~8,000 ft to the east. Historic drill intercepts along strike from Kilbourne in both directions have documented graphite mineralization. Figure 9-6 shows the mapped extensions of Kilbourne's strike length that warrant additional drilling. The remaining two thirds of the Kilbourne strike length remains open, and highly prospective. Graphite mineralization has also been documented within UM2 at the Company's Hyatt Mine.



Source: xxxxx

Figure 9-6: Kilbourne exploration target

In addition to the targets along strike from the Kilbourne Deposit, the Company has identified multiple areas of high prospectivity for additional graphite occurrences. These targets have been generated through the re-evaluation of the historic airborne geophysical data, and the digitization of historic geologic maps. Of the historic geophysical data, roughly 30% has been re-evaluated; as this process continues, the number of identified prospects is likely to increase. Areas currently identified as prospective that fall outside of the Company's ownership are undergoing lands research should future acquisition become a priority.

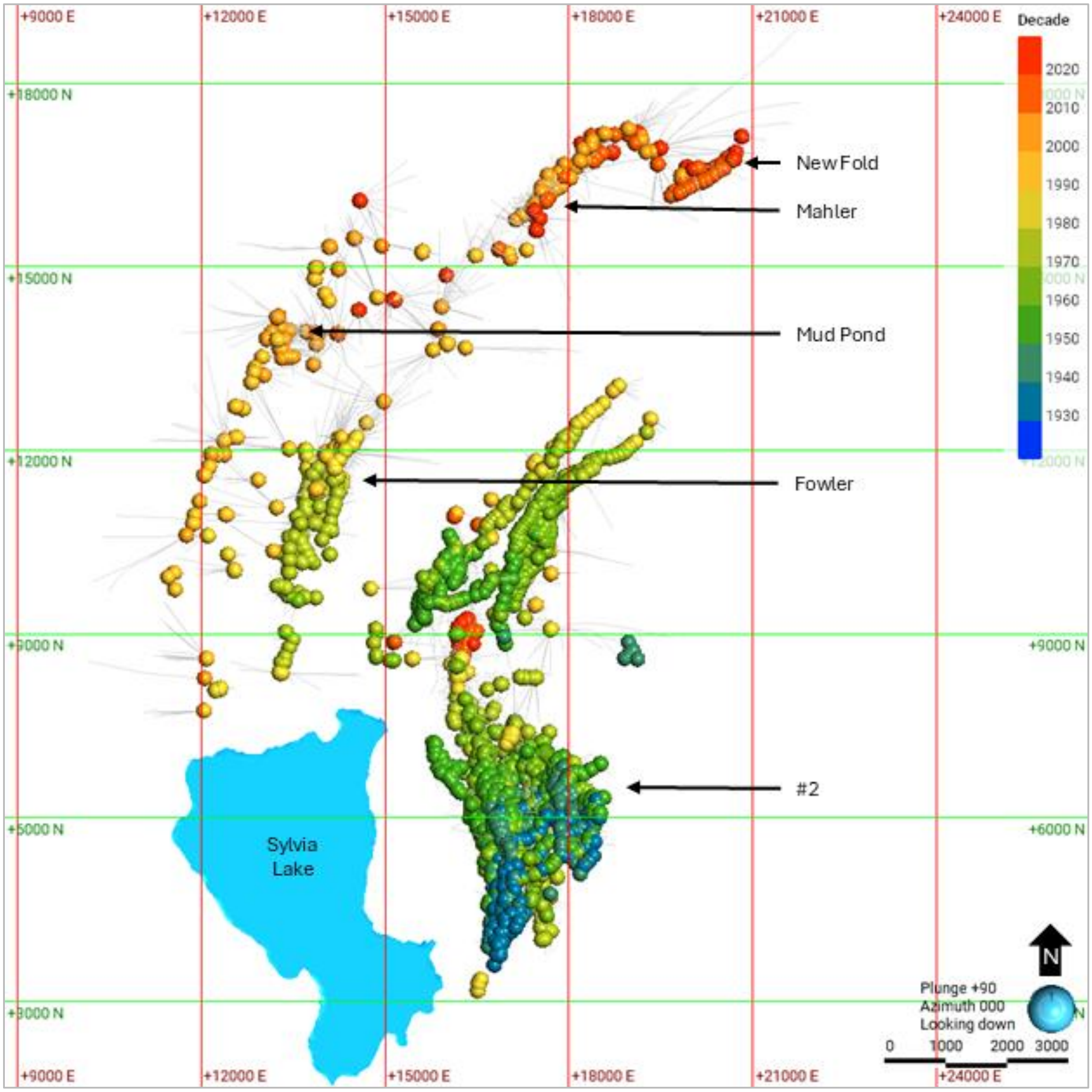
10. Drilling

This chapter of the report provides an update of ESM's drilling in two sections. Section 10.1 describes the overall ESM drillhole database inclusive of the Kilbourne graphite drilling that overlies the zinc deposits. Section 10.2 breaks out the Kilbourne graphite drilling in more detail.

10.1 ESM Drilling

10.1.1 Drilling Summary

As of August 20, 2024, a total of 11,570 diamond drillholes have been completed at ESM, totaling 4,223,857 ft, as shown in Table 10-1. Historic drilling is continually digitized and incorporated into the digital database as records are discovered, consequently the numbers in this document will be greater than those in earlier reports (Makarenko et al., 2018; Warren et al., 2021). As far as ESM is aware, no additional significant groups of historical drilling remain to be digitized. Figure 10-1 displays UG drilling with collars colored by decade drilled, which demonstrates the mining progression for each zone.



Source: ESM 2024

Figure 10-1: Map showing the distribution of Balmat underground drilling by decade

The drillhole database was sub-divided into geographic "Areas" that can be extracted individually. The Balmat Area covers deposits that are the subject of this report while the other projects cover drilling in other historic mining camps, which is summarized in the District Area in Table 10-1.

Table 10-1: Area drilling by year since 2017

Area	Year	Surface Core		UG Core		Total Holes	Total Length (ft)	Total Length (m)
		Holes	Length (ft)	Holes	Length (ft)			
Balmat	pre-2017	1,053	1,006,319	6,519	1,676,384	7,572	2,682,702	817,688
	2017	8	14,029	16	9,019	24	23,047	7,025
	2018	27	78,008	43	42,129	70	120,137	36,618
	2019	68	30,108	68	26,029	136	56,137	17,111
	2020	30	17,099	127	32,203	157	49,301	15,027
	2021	12	10,109	89	28,317	101	38,426	11,712
	2022	40	12,537	64	39,001	104	51,538	15,709
	2023	27	12,924	58	30,281	85	43,205	13,169
	2024	38	11,647	18	8,434	56	20,081	6,121
	Total	**1,303**	**1,192,778**	**7,002**	**1,891,796**	**8,305**	**3,084,574**	**940,178**
District	pre-2017	842	709,597	319	43,088	1,161	752,685	229,418
	2017	1	2,043			1	2,043	623
	2018	1	3,346			1	3,346	1,020
	2019	5	9,367			5	9,367	2,855
	2020	1	307			1	307	94
	2021	23	7,135			23	7,135	2,175
	2022	17	16,686			17	16,686	5,086
	Total	**890**	**748,481**	**319**	**43,088**	**1,209**	**791,569**	**241,270**
Total		**2,193**	**1,941,259**	**7,321**	**1,934,884**	**9,514**	**3,876,143**	**1,181,448**
District	pre-2017	un-categorized				2,056	347,714	105,983
Grand Total						**11,570**	**4,223,857**	**1,287,432**

Source: ESM 2024

Note: Table excludes blast holes, channel samples, and well holes.

The primary focus of the drilling programs since 2020 were further definition and exploration of the N2D, Fowler, Mahler (Lower and Upper), Mud Pond (Apron and Main), New Fold, and Turnpike zinc resources with additional near-mine and regional exploration of other zinc targets. Graphite was intersected in recent exploration drilling and specifically targeted in 2024. This program is discussed in more detail in Section 10.2. A categorical breakdown of drilling since the last technical report is presented in Table 10-2.

Table 10-2: Drilling by category and target commodity since the last technical report

Category	Commodity	Surface Core		UG Core		Total Holes	Total Length (ft)	Total Length (m)
		Holes	Length (ft)	Holes	Length (ft)			
Definition	Zinc			330	109,670	330	109,670	33,427
Exploration	Zinc	122	69,474	22	28,034	144	97,509	29,721
	Graphite	39	11,917			39	11,917	3,632
	Total	161	81,391	22	28,034	183	109,425	33,353
Grand Total		**161**	**81,391**	**352**	**137,704**	**513**	**219,095**	**66,780**

Source: ESM 2024

10.1.2 Drilling Procedures

Drilling at ESM has been exclusively core drilling. The mine owns two Diamec 262 underground drills which drill AWJ size core, which were utilized for the definition programs. The mine also owns an Epiroc U-6 that drills BQ size core and was primarily used for underground exploration programs. Three contract Longyear underground drills that drilled 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 from 2018–2020.

10.1.3 Core Handling and Sampling

Underground 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 filled core boxes were stored at the drill site until the end of shift where they were loaded in a vehicle and transported to the shaft station. At the station the core boxes were loaded into a custom wooden crate specifically designed for core box transportation up the shaft to the core logging facility. Full crates were typically brought to surface about once per week, but the frequency can vary depending on drill productivity. The shaft crew coordinates crate movement between the station staging area and the core shed's receiving bay. An example of the crate on the surface waiting to be unloaded is shown in Figure 10-2.

Source: ESM 2024

Figure 10-2: Underground core storage crate staged outside the on-site logging facility

Surface drill core is transferred from the core barrel to the core box. The core technician or logging geologist will pick up the core boxes from the site and return them to the on-site logging facility.

The core is washed, logged, photographed, and sampled. All exploration 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. Definition core from underground is whole-core sampled.

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, labeled 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.

10.1.4 Downhole Surveying

Downhole survey methodology on the Property evolved 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. 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 exploration drilling used the REFLEX EZ-SHOT instrument from 2017 to 2022 and an ESM owned DeviShot since 2023. Underground drilling has relied on the Devico DeviShot magnetic multishot survey tool since 2018.

Other than the downhole surveying in the historical drillholes, the QPs are not aware of any drilling, sampling, or recovery factors that would negatively impact the accuracy and reliability of drill sample results at ESM.

10.2 Kilbourne Graphite Drilling

10.2.1 Core Re-sampling

Core drilling by ESM targeting zinc intersected graphite in 2020–2022. These intervals were originally assayed for zinc and subsequently resampled in 2023 using quartered core to test the graphite content. A list of these holes that were used in the MRE are presented in Table 10-1. While the holes were used in the MRE they form a minor component of the overall Kilbourne drillhole database. The blue and green collar points in Figure 10-3 are the SX series holes.

Table 10-3: ESM surface holes re-sampled for graphite

Hole ID	Length (ft)	UTM NAD83			Azimuth	Dip	Core Size	Start Date	End Date
		Easting (m)	Northing (m)	Elevation (m)					
SX20-2563	3,153	465,846.0	4,902,302.0	186.0	120	-55	NQ	2020-09-28	2020-10-22
SX20-2564	3,487	46,584.0	4,902,302.0	186.0	125	-63	NQ	2020-10-22	2020-11-19
SX20-2565	3,407	46,584.0	4,902,302.0	186.1	125	-50	NQ	2020-11-19	2021-01-12
SX21-2589	2,287	467,176.0	4,902,744.0	186.0	0	-90	NQ	2021-05-04	2021-05-18
SX21-2601	1,877	466,948.0	4,902,442.0	193.0	0	-90	NQ	2021-12-02	2021-12-19
SX22-2621	3,487	469,182.2	4,903,659.5	183.1	150	-70	NQ	2022-04-04	2022-05-20

Source: ESM 2024

10.2.2 Surface Channel Sampling

Six channel samples were taken across exposed outcrop in 2023, and one of them was used in the MRE. The sample locations are listed in Table 10-4. The channel was cut with a Husqvarna K 770 demo saw to a depth between 4 and 6 inches. Sample lengths varied, ranging between 2.5 ft and 5 ft. Samples were chiseled out between two cuts spaced 2 inches apart and placed in a cloth sample bag, labeled 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. QA/QC procedures are like those used for core drilling as detailed in Chapter 11. Most of the channel samples were excluded from the MRE, as the subsequent core drilling provided significantly better assay information due to orientation and zone representativity.

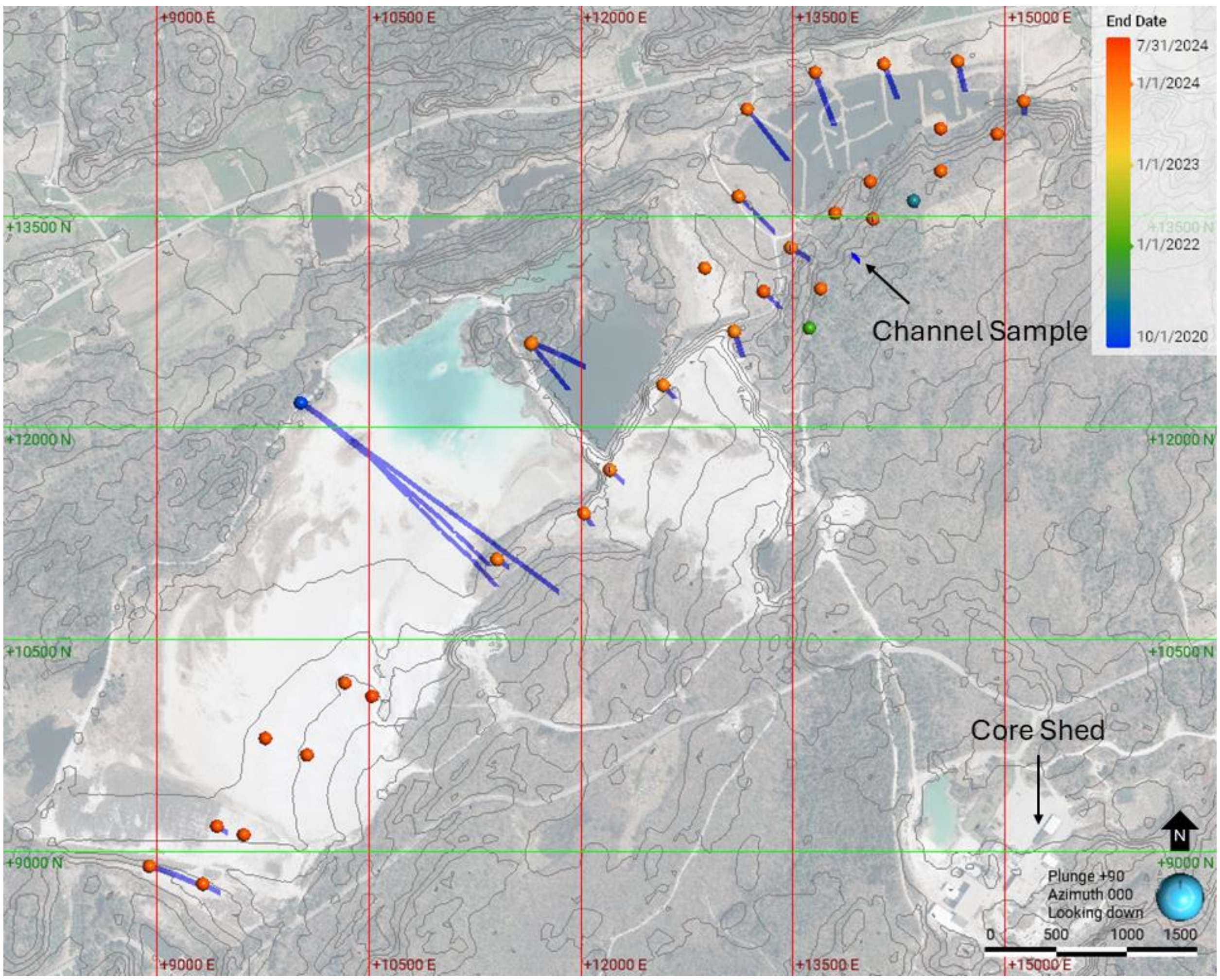
Table 10-4: ESM outcrop channel samples

Channel ID	Length (ft)	UTM NAD83			Azi	Dip	Start Date	End Date	Used in MRE
		Easting (m)	Northing (m)	Elevation (m)					
KT23-001A	55	466,794.8	4,902,439.7	194.1	255	0	12/12/2023	12/13/2023	no
KT23-001B	42	466,806.9	4,902,440.4	193.7	261	0	12/15/2023	12/15/2023	no
KT23-001C	42	466,817.2	4,902,445.5	194.4	240	0	12/15/2023	12/15/2023	no
KT23-001D	42	466,831.5	4,902,450.9	195.0	254	0	12/16/2023	12/16/2023	no
KT23-002A	84	467,040.4	4,902,607.2	195.7	129	0	12/30/2023	12/30/2023	yes
KT23-003B	35	467,065.7	4,902,667.7	191.3	154	0	unsampled		no

Source: ESM 2024

10.2.3 Core Drilling Summary

As of August 20, 2024, a total of 39 diamond drillholes have been completed by ESM targeting graphite, totaling 11,917 ft, as shown colored in orange and red in Figure 10-3 and listed in Table 10-5.



Source: ESM 2024

Figure 10-3: Kilbourne drilling with collars colored by end date, with 10 ft contours

Table 10-5: Kilbourne drilling by year since 2017

Hole ID	Length (ft)	UTM NAD83			Azimuth	Dip	Core Size	Start Date	End Date
		Easting (m)	Northing (m)	Elevation (m)					
KX23-001	270	466,909.3	4,902,622.3	185.8	0	-90	AWJ	2023-12-26	2024-01-03
KX24-002	256	466,910.2	4,902,621.8	185.7	120	-50	AWJ	2024-01-03	2024-01-05
KX24-003	315	466,786.4	4,902,442.8	195.5	160	-50	AWJ	2024-01-08	2024-01-12
KX24-004	356	466,786.8	4,902,441.7	195.5	0	-90	AWJ	2024-01-12	2024-01-18
KX24-005	241	466,632.6	4,902,327.1	199.3	0	-90	AWJ	2024-01-19	2024-01-23
KX24-006	209	466,633.3	4,902,326.3	199.2	140	-50	AWJ	2024-01-23	2024-01-25
KX24-007	209	466,517.0	4,902,144.6	196.3	0	-90	AWJ	2024-01-25	2024-01-29
KX24-008	221	466,517.7	4,902,143.9	196.3	130	-50	AWJ	2024-01-29	2024-01-30
KX24-009	644	466,347.9	4,902,416.9	195.8	140	-50	AWJ	2024-02-05	2024-02-15
KX24-010	636	466,349.9	4,902,419.2	195.8	110	-50	AWJ	2024-02-15	2024-02-20
KX24-011	165	466,273.2	4,901,952.0	194.3	140	-50	AWJ	2024-02-21	2024-02-22
KX24-012	157	466,272.2	4,901,952.6	194.2	140	-90	AWJ	2024-02-22	2024-02-22
KX24-013	126	466,460.7	4,902,051.3	195.8	0	-90	AWJ	2024-02-23	2024-02-25
KX24-014	167	466,461.4	4,902,020.3	195.8	140	-50	AWJ	2024-02-26	2024-02-27
KX24-015	544	466,798.3	4,902,732.7	186.0	130	-50	AWJ	2024-02-27	2024-03-03
KX24-016	607	466,797.4	4,902,733.5	185.9	130	-90	AWJ	2024-03-04	2024-03-11
KX24-017	623	466,724.0	4,902,578.8	187.5	0	-90	AWJ	2024-03-12	2024-03-17
KX24-018	719	466,817.1	4,902,921.2	182.9	140	-50	AWJ	2024-03-18	2024-03-24
KX24-019	408	467,112.8	4,903,017.1	181.1	160	-50	AWJ	2024-03-25	2024-03-27
KX24-020	332	467,271.7	4,903,022.4	181.1	170	-50	AWJ	2024-04-01	2024-04-02
KX24-021	152	467,413.3	4,902,935.4	181.1	180	-50	AWJ	2024-04-03	2024-04-03
KX24-022	611	466,964.0	4,903,000.2	182.4	160	-50	AWJ	2024-04-04	2024-04-10
KX24-023	106	466,973.9	4,902,533.1	194.3	0	-90	AWJ	2024-04-11	2024-04-11
KX24-024	67	467,087.2	4,902,683.0	192.4	0	-90	AWJ	2024-04-12	2024-04-15
KX24-025	95	467,234.1	4,902,785.7	194.1	0	-90	AWJ	2024-04-15	2024-04-15
KX24-026	612	465,519.9	4,901,294.3	191.3	105	-50	AWJ	2024-04-16	2024-04-30
KX24-027	179	467,005.5	4,902,696.0	186.2	0	-90	AWJ	2024-05-03	2024-05-06
KX24-028	149	467,081.2	4,902,763.5	184.4	0	-90	AWJ	2024-05-06	2024-05-07
KX24-029	119	467,233.9	4,902,877.2	180.8	0	-90	AWJ	2024-05-07	2024-05-07
KX24-030	117	467,356.0	4,902,864.8	193.3	0	-90	AWJ	2024-05-08	2024-05-08
KX24-031	286	466,850.6	4,902,527.8	191.5	140	-50	AWJ	2024-05-09	2024-05-12
KX24-032	226	465,635.1	4,901,255.4	194.1	110	-50	AWJ	2024-05-13	2024-05-14
KX24-033	178	465,634.5	4,901,255.6	194.0	110	-80	AWJ	2024-05-15	2024-05-15

Hole ID	Length (ft)	UTM NAD83			Azimuth	Dip	Core Size	Start Date	End Date
		Easting (m)	Northing (m)	Elevation (m)					
KX24-034	280	465,861.2	4,901,532.7	197.7	0	-90	AWJ	2024-05-16	2024-05-20
KX24-035	298	465,943.5	4,901,687.6	197.0	0	-90	AWJ	2024-05-21	2024-05-22
KX24-036	198	465,723.7	4,901,361.4	195.0	0	-90	AWJ	2024-05-23	2024-05-27
KX24-037	257	465,665.9	4,901,379.9	194.4	110	-70	AWJ	2024-05-28	2024-05-30
KX24-038	560	465,771.6	4,901,569.1	195.3	0	-90	AWJ	2024-05-30	2024-06-12
KX24-039	222	466,001.8	4,901,658.7	199.2	0	-90	AWJ	2024-06-13	2024-06-17

Source: ESM 2024

10.2.4 Drilling Procedure

Core drilling was completed using an ESM owned and operated Diamec 262 underground drill that was brought to the surface and mounted on a skid plate specifically for the Project as shown in Figure 10-4. All core samples were AWJ size.

Source: ESM 2024

Figure 10-4: Diamec #2 on the surface drilling for graphite

10.2.5 Core Handling and Sampling

The drill core was transferred from the core barrel to the core box. The core technician or logging geologist picked up the core boxes from the site and returned them to the on-site logging facility.

The core was washed, logged, photographed (Figure 10-5), and sampled. All core samples were cut in half, lengthwise, using a diamond saw with a diamond-impregnated blade and sampled on 5 ft intervals with adjustments made to match geological contacts.

Source: ESM 2024

Figure 10-5: Example of photographed AWJ size graphitic core

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 was placed in a plastic or cloth sample bag, labeled 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. Shipping boxes are placed onto pallets and shipped by freight to SGS Lakefield laboratory in Lakefield, ON, Canada for sample preparation and graphitic carbon analysis. Pulps are forwarded to SGS Burnaby laboratory in Burnaby, BC, Canada for multi-element analysis.

11. Sample Preparation, Analyses, and Security

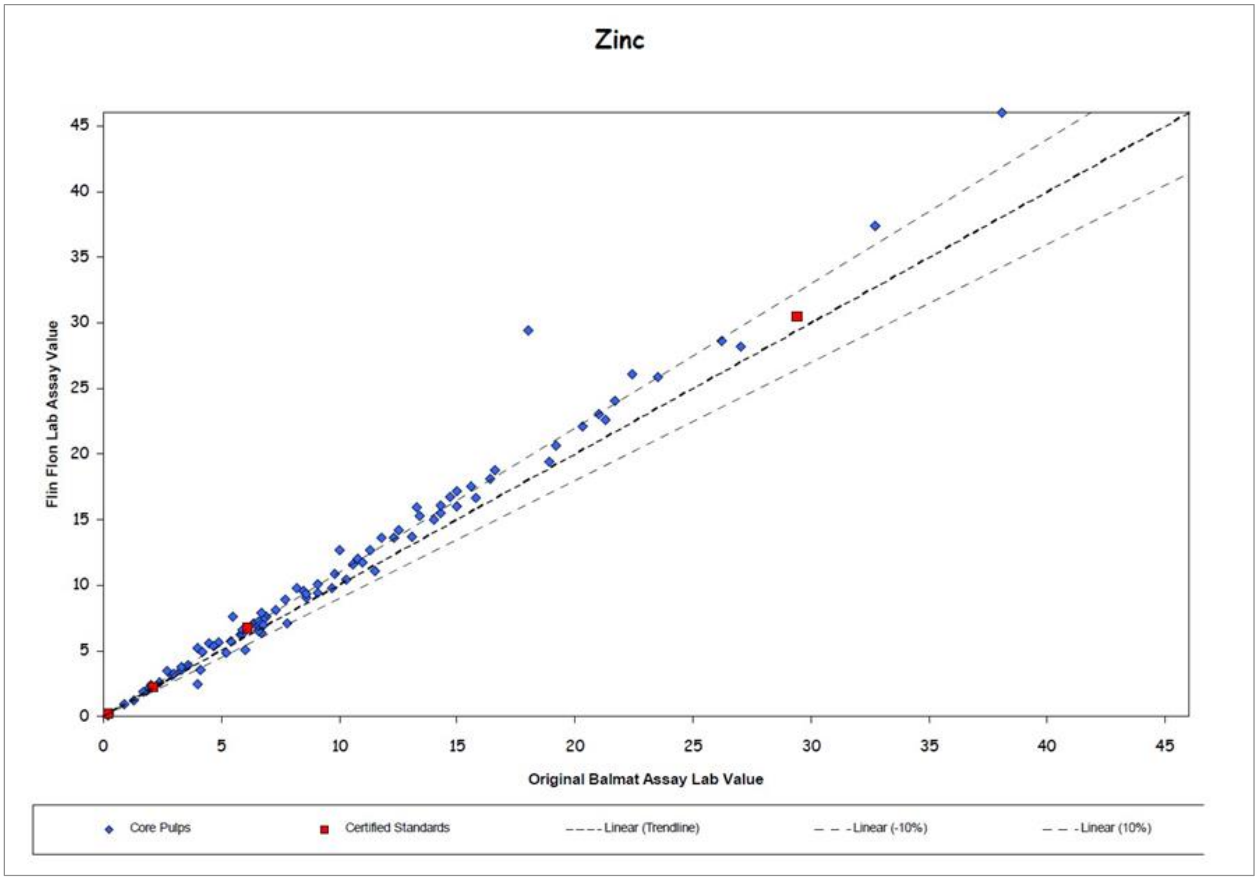
11.1 Historical Assaying

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 cores drilled between 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. For zinc assays greater than 25%, the Flin Flon laboratory reported zinc assays more than 10% higher. The certified reference standards were all within acceptable limits.



Source: SLZ 2018

Figure 11-1: Hudbay Flin Flon Lab check assays of ESM 1995 to 2000 pulps

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

Standard	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: ESM 2024

All standards came finely crushed in foil packages clearly labeled 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 sulfide material from the ESM Zinc Mine (ESM #4). The CRMs were certified with round robin assaying at 15 laboratories. All the laboratories performed analyses using an aqua regia digest and mostly ICP-OES instrumental finishes.

Table 11-2: ESM QA/QC certified standards supplied by OREAS June 2008

Standard	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: ESM 2024

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 2017 to 2024 Sample Preparation and Assaying

A total of 38,395 drill core samples were submitted to ALS Geochemistry between April 2017 and June 2024. The quality control data for these sample submittals are discussed in Section 11.2 for zinc, lead, copper, silver, gold, and iron.

11.2.1 Sample Preparation and Analysis

For the 2017 to 2024 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, which includes the following:

- Air dry if possible (maximum 120 °C if oven drying is necessary).
- Crush entire sample to at least 70% passing 0.1 in (2 mm).
- Riffle split 8 oz (250 g).
- 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 Table 11-4	ME-ICP41	Varies; see Table 11-4	0.25 g two-acid: HNO_3 + 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: HNO_3 + HCl	ICP-AES
Pb	Pb-OG46	0.001%	0.25 g two-acid: HNO_3 + HCl	ICP-AES
Zn	Zn-OG46	0.001%	0.25 g two-acid: HNO_3 + HCl	ICP-AES
Zn	Zn-VOL50	0.01%	1 g Titration	Titration

Reference to metric units of g = grams.

High grade samples, for silver greater than 100 ppm and base metals over 1%, 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			

11.2.2 Security

The whole core is photographed. Underground definition drilling is submitted to the lab whole with coarse rejects returned and retained after assaying has been completed. Exploration core is split in half with one-half retained for verification purposes.

Cores 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. The core is stored in wax cardboard boxes and organized in shipping containers by drillhole number.

11.2.3 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.

Until Q3 2021, 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.

Starting in Q3 2021 sample submissions were changed to separate out ore-grade zinc samples from low-grade samples. Ore-grade samples are flagged when estimated to be above 10% zinc and 20% visual sphalerite. Ore-grade batches include the insertion of high-grade CRMs only. Low-grade batches include blanks every 40th sample with the low and medium CRMs alternating every 20th sample. Results have minimized blank failures and potential for carry-over.

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

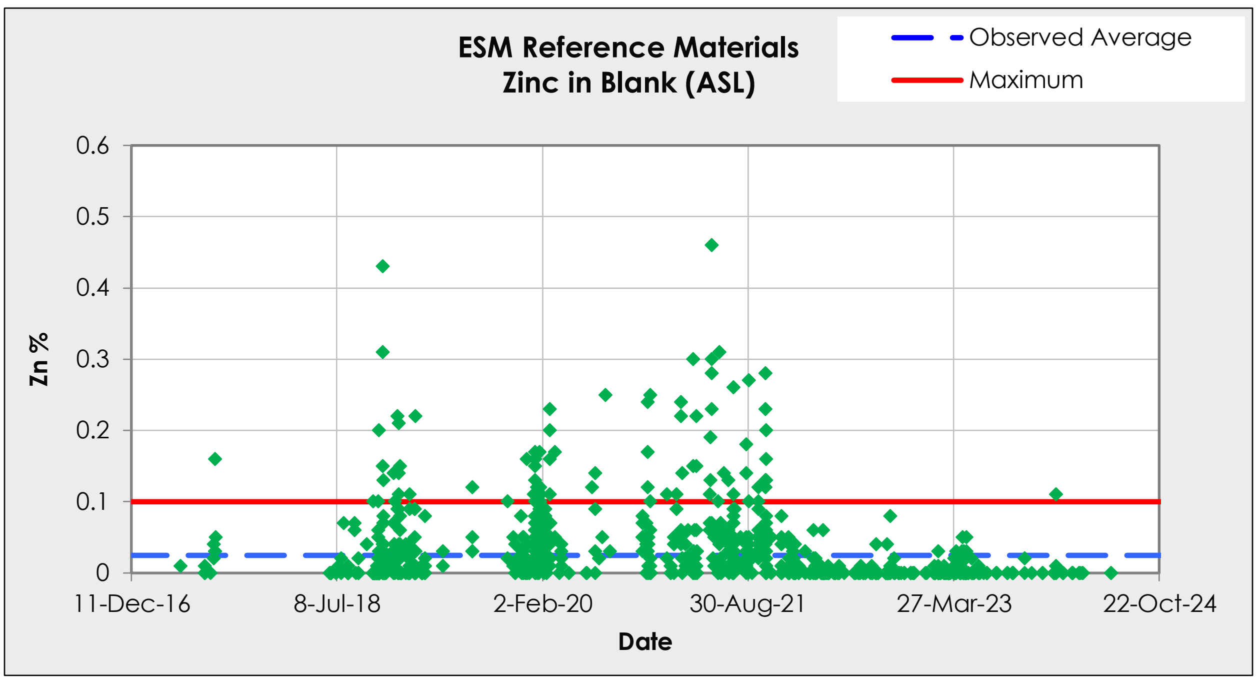
Blank	Zinc (ppm)	Lead (ppm)	Silver (ppm)	Copper (ppm)	Iron (%)
Blank (ASL)	1,000	400	5	400	0.7

Source: ESM 2024

The threshold levels were applied based on observations of past results and understanding of the risks to the Project. The threshold for zinc was adjusted in Q4 2022 from 400 ppm to 1,000 ppm, based on the allowable 1% carryover within ALS lab's method expectations. 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 1,087 blanks inserted with samples, all silver values were less than 1 ppm with the exception of one case where silver reported at 12.7 ppm. Copper values were less than 100 ppm. There was a total of six cases where lead values exceeded 400 ppm and reported up to 0.057% Pb.

Blanks are only inserted with low-grade batches approximately every 40th sample. As a result, there were cases of sample cross-contamination in 8 out of 736 samples received after Q3 2021 where ore-grade samples were separated out. Since 2017 there are a total of 71 out of 1,087 cases reporting over 0.1% 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. In October 2021 there is a decrease in cross-contamination after ore-grade batches were analyzed at ALS lab separately. After raising the zinc failure threshold, one blank exceeded the threshold in the period of December 2022 to June 2024.



Summary Statistics

Expected Values	
Mean	0.000
Maximum	0.100

Observed Values	
Number of Samples	1,087
Mean	0.025
Percent of Maximum	24.74%

Source: Graph generated in QC Mine Software

Figure 11-2: Zinc in blank control chart

The potential for zinc contamination is moderated by ESM's practice of flagging sulfide-rich samples placing the samples on a separate batch 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 0.1%, there are also reported cases of iron over 0.7%. The elevated iron values are also associated with high mineralized sulfide-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

Reference Material	Qty	Outliers Excluded	Failures Excluded	Zn %		Observed Zn %		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-H5	346	2	4	24.600	0.799	24.464	0.556	99.4%
OREAS-G5	714	3	11	10.300	0.220	10.321	0.185	100.2%
OREAS-135b	31	-	-	2.730	0.075	2.691	0.044	98.6%
OREAS-135	487	1	1	2.800	0.104	2.764	0.054	98.7%
Total	**1,578**					**Weighted Average**		**99.5%**

Reference Material	Qty	Outliers Excluded	Failures Excluded	Cu %		Observed Cu %		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-H5	350	-	2	0.043	0.002	0.045	0.002	103.3%
OREAS-G5	728	-	-	0.060	0.004	0.061	0.002	101.4%
OREAS-135b	30	1	-	0.012	0.001	0.011	0.000	98.0%
OREAS-135	480	2	1	0.028	0.001	0.029	0.001	101.3%
Total	**1,588**					**Weighted Average**		**101.7%**

Reference Material	Qty	Outliers Excluded	Failures Excluded	Pb %		Observed Pb %		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-H5	352	-	-	0.075	0.006	0.075	0.005	99.6%
OREAS-G5	728	-	-	0.076	0.006	0.073	0.004	95.8%
OREAS-135b	31	-	-	1.690	0.037	1.693	0.035	100.2%
OREAS-135	468	19	2	1.700	0.062	1.723	0.040	101.4%
Total	**1,579**					**Weighted Average**		**98.4%**

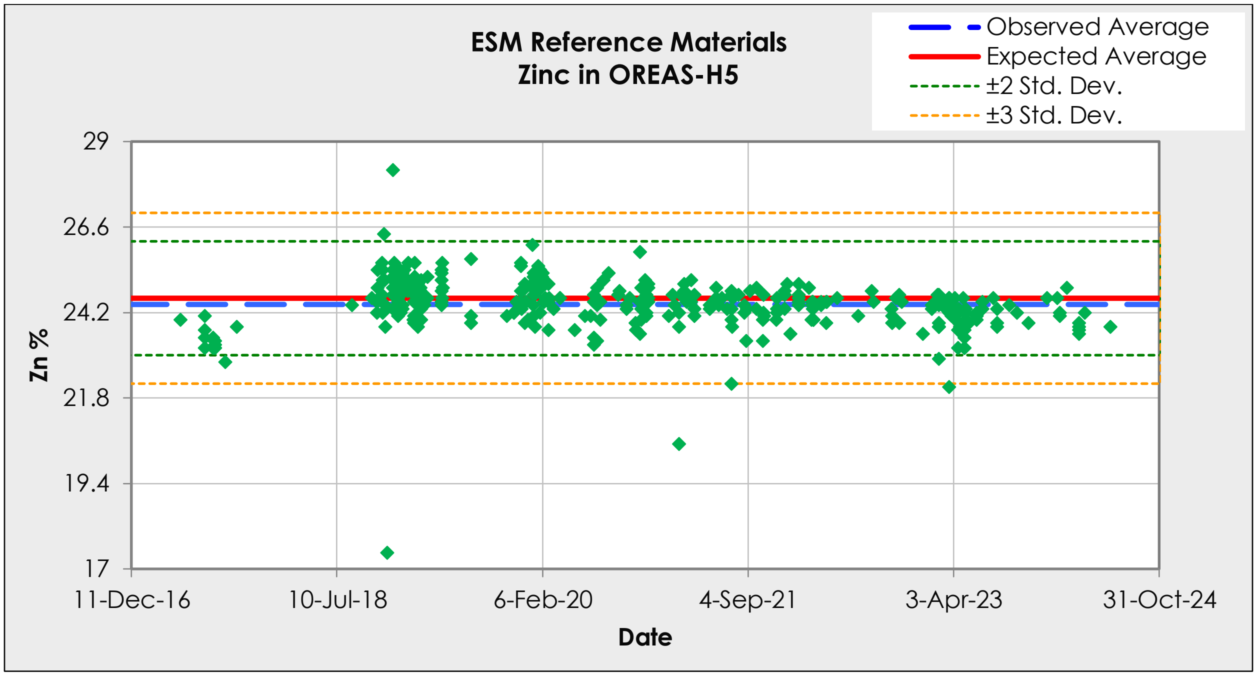
Reference Material	Qty	Outliers Excluded	Failures Excluded	Ag ppm		Observed Ag ppm		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-H5	349	-	-	3.810	0.510	4.286	0.186	112.5%
OREAS-G5	721	-	-	3.500	0.550	3.812	0.161	108.9%
OREAS-135b	31	-	-	53.500	1.340	53.171	1.342	99.4%
OREAS-135	464	3	8	54.900	2.170	55.445	2.015	101.0%
Total	**1,565**					**Weighted Average**		**107.2%**

Reference Material	Qty	Outliers Excluded	Failures Excluded	Fe %		Observed Fe %		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-H5	352	-	-	1.590	0.100	1.582	0.050	99.5%
OREAS-G5	728	-	-	1.490	0.090	1.473	0.041	98.8%
OREAS-135b	30	1	-	5.100	0.201	4.986	0.085	97.8%
OREAS-135	479	2	2	8.970	0.363	8.795	0.240	98.0%
Total	**1,589**					**Weighted Average**		**98.7%**

An Outlier is defined as being outside five standard deviations from the accepted value. These are cases that are most likely sample mis-labels or in the case of lead, reached the upper detection limit of the analysis method and was not requested for overlimits. 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. Fewer than 2% of the reference material insertions resulted in requests for repeat assays.

ALS performed well for all five metals for reference material OREAS-135 prepared by OREAS. OREAS-135 is a commercially available reference material created in 2017 and analyzed by 24 recognized laboratories. In 2022 ESM purchased the replacement standard OREAS-135b.

Figure 11-3 and Figure 11-4 show that the zinc results reported for OREAS-H5 and OREAS-G5 have been consistent and reported within a narrow range.

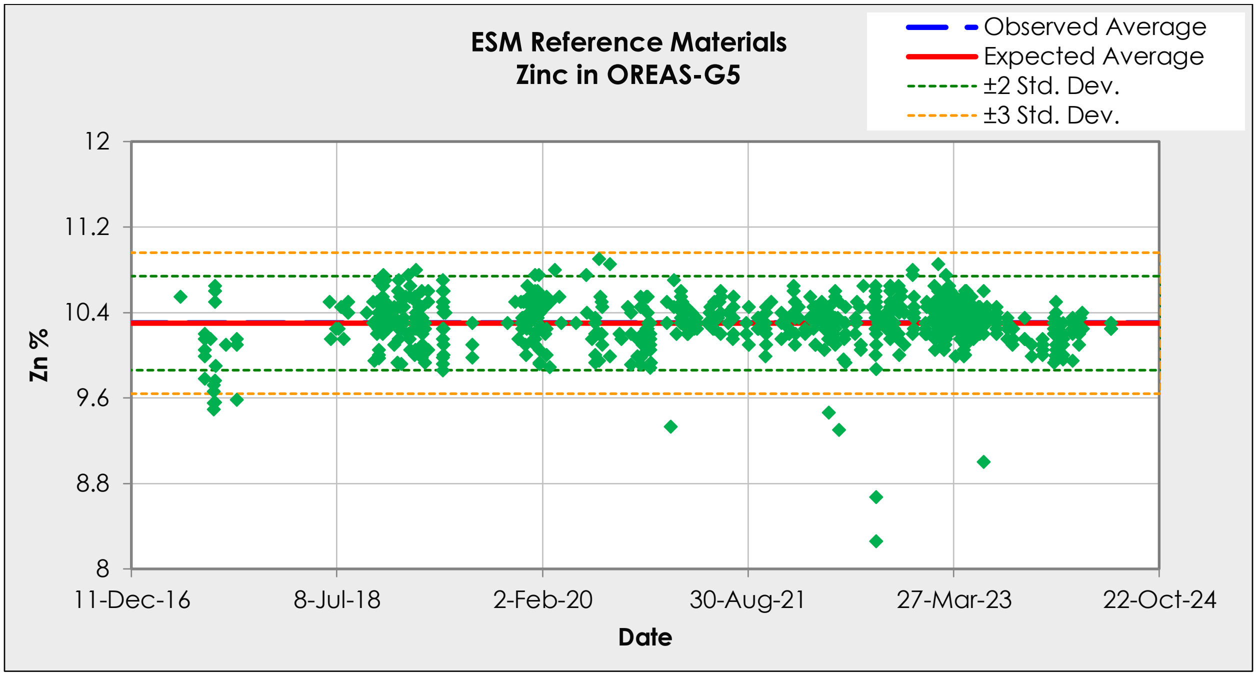


Summary Statistics

Expected Values	
Mean	24.600
Standard Deviation	0.799
2 x RSD	6.50%

Observed Values	
Number of Samples	352
Mean	24.426
Standard Deviation	0.752
2 x RSD	6.16%
Falls Within 3 SD of Certified Mean	99%
Falls Within 2 SD of Certified Mean	98%
Falls Within 1 SD of Certified Mean	80%

Figure 11-3: Control chart for Zn in reference material H-5



Summary Statistics

Expected Values	
Mean	10.300
Standard Deviation	0.220
2 x RSD	4.27%

Observed Values	
Number of Samples	728
Mean	10.305
Standard Deviation	0.233
2 x RSD	4.52%
Falls Within 3 SD of Certified Mean	99%
Falls Within 2 SD of Certified Mean	96%
Falls Within 1 SD of Certified Mean	75%

Source: Graph generated in QC Mine Software

Figure 11-4: Control chart for Zn in reference material G-5

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.

11.3 Kilbourne 2023 and 2024 Sample Preparation and Assaying

A total of 2,386 samples (including Quality Control "QC" samples) were obtained from February 2023 to June 2024. This section focuses on quality control data related to the Kilbourne Graphite Project.

11.3.1 Sample Preparation and Analysis

Sample preparation (crushing and pulverizing) has been performed at SGS Lakefield, an ISO/IEC 17025 accredited lab located in Lakefield, Ontario, Canada. SGS Lakefield prepares a pulp and runs graphite analysis, then ships the pulps to SGS Burnaby, British Columbia, Canada for multi-element analysis.

Sample Preparation Procedures (SGS Method Code G_CRU_KGCRU3_WT and G_CRU-CRU75):

1. Process, sort, and weigh samples;
2. Sample drying, 105 °C, <3 kg;
3. Crush entire sample to 3.36 mm (portion of coarse material used for metallurgical testing (G_CRU3);
4. Riffle split 250 g; crush to 75% passing 2 microns (G_CRU75);
5. Pulverize nominal 250 g to >85% passing 75 microns (pulps created for graphitic carbon and multi-element analysis).

Table 11-7: Summary of assay methods

Analyte	Method Code	Detection Limit	Digest	Instrumentation
34 elements, see below	GE-ICP21B20	Varies; see below	0.25 g two-acid: HNO_3 + HCl digest plus HCl leach	ICP-OES – Aqua Regia
Ag	ICP42Q100	0.01%	0.25 g two-acid: HNO_3 + HCl	ICP-OES-4 Acid
Ca	ICP42Q100	0.10%	0.25 g two-acid: HNO_3 + HCl	ICP-OES-4 Acid
Zn	ICP42Q100	0.01%	0.25 g two-acid: HNO_3 + HCl	ICP-OES-4 Acid
Mn	ICP42Q100	0.00%	0.25 g two-acid: HNO_3 + HCl	ICP-OES-4 Acid
Fe	ICP21B100	0.01%	0.25 g two-acid: HNO_3 + HCl digest plus HCl leach	ICP-OES-Aqua Regia
S	CSA06V	0.01%	0.1-0.03 g IR Combustion	IR Combustion
Cg (Graphitic Carbon)	CG-CSA06V	0.05%	0.1-0.03 g IR Combustion	IR Combustion

SGS Lakefield prepares the pulps and analyzes each sample for graphitic carbon (Cg-CSA06V) with a detection limit of >0.01%. Pulps are shipped to SGS Burnaby for multi-element analysis by aqua regia digestion (GE-ICP21B20 for 34 elements) with an ICP – OES finish. All samples in which silver, calcium, manganese, iron, zinc, and sulfur exceed their upper limit are re-run using methods of aqua regia digestion (Fe-ICP21B100), four acid digestion (Ag, Ca, Zn, and Mn-ICP42Q100) and infrared combustion (S-CSA06V) with the elements reported in percentage (%).

The lower and upper limits for the aqua regia digest method (GE-ICP21B20) are shown in Table 11-8.

Table 11-8: Upper and lower limits for aqua regia GE-ICP21B20 method

Analyte	Lower Limit	Upper Limit	Analyte	Lower Limit	Upper Limit	Analyte	Lower Limit	Upper Limit
Ag (ppm)	0.01	100	Hg (ppm)	0.01	100	Sc (ppm)	0.1	10,000
Al (%)	0.005	15	In (ppm)	0.005	500	Se (ppm)	1	1,000
As (ppm)	1	10,000	K (%)	0.05	10	Sn (ppm)	0.2	1,000
Ba (ppm)	2	10,000	La (ppm)	0.1	10,000	Sr (ppm)	0.5	10,000
Be (ppm)	0.05	100	Li (ppm)	0.5	10,000	Ta (ppm)	0.01	10,000
Bi (ppm)	0.01	10,000	Lu (ppm)	0.01	1,000	Tb (ppm)	0.02	10,000
Ca (%)	0.002	15	Mg (%)	0.001	15	Te (ppm)	0.05	10,000
Cd (ppm)	0.01	10,000	Mn (ppm)	2	10,000	Th (ppm)	0.05	10,000
Ce (ppm)	0.02	1,000	Mo (ppm)	0.05	10,000	Ti (%)	0.01	15
Co (ppm)	0.1	10,000	Na (%)	0.005	15	Tl (ppm)	0.02	10,000
Cr (ppm)	1	10,000	Nb (ppm)	0.05	1,000	U (ppm)	0.05	10,000
Cs (ppm)	0.05	10,000	Ni (ppm)	0.2	10,000	V (ppm)	1	10,000
Cu (ppm)	0.5	10,000	P (%)	0.003	15	W (ppm)	0.05	10,000
Fe (%)	1	15	Pb (ppm)	0.2	10,000	Y (ppm)	0.05	10,000
Ga (ppm)	0.1	10,000	Rb (ppm)	0.05	10,000	Yb (ppm)	0.1	100
Ge (ppm)	0.1	10,000	S (%)	0.01	5	Zn (ppm)	1	10,000
Hf (ppm)	0.05	500	Sb (ppm)	0.05	10,000	Zr (ppm)	0.5	10,000

11.3.2 Security

The whole core is photographed at the ESM mine site and is cut in half with one-half retained in a secured facility for verification purposes. The half-core samples are shipped to SGS Lakefield in Ontario, Canada.

The 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. The 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 barren coarse-grained blanks (blanks) and certified reference materials within each batch of samples. Blanks are inserted every 40th sample and CRMs are inserted every 20th sample, rotating a low, medium, and high grade (with respect to graphitic carbon) CRM.

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. Drilling program QA/QC reports are prepared internally to monitor overall laboratory performance.

CRMs and blanks are purchased from OREAS North America Inc. The reference material is high quality and was analyzed at more than fifteen laboratories to determine expected values and tolerances. The materials are sourced from Queens Graphite Mine in Matale/Kurunegala Project area in central Sri Lanka. It is prepared from crystalline vein graphite ore blended with granodiorite. The certified expected values are listed in Table 11-9.

Table 11-9: Certified reference material expected values

CRM	Graphite (%)
OREAS-722	2.03
OREAS-724	12.06
OREAS-725	24.52

Barren coarse-grained silica blanks were submitted with samples to determine if there has been contamination or sample cross-contamination during the preparation stage. Elevated values for blanks may also indicate sources of contamination in the analytical procedure (contaminated reagents or test tubes) or sample solution carry-over during instrumental finish.

The threshold levels for blanks are defined in Table 11-10.

Table 11-10: Blank failure threshold

Blank	Graphite (%)
Blank (ASL)	0.1

The blank threshold level was applied based on ore-grade values for graphitic carbon taking into account a 0.25% allowable carry-over within lab method expectations.

A total of 67 blanks were inserted with samples, all blanks reported at or below detection limit for graphitic carbon and does not constitute any risk of carry-over.

Figure 11-5 is the control chart for graphitic carbon in blanks.

A total of 114 CRMs were inserted with samples, six samples reported values outside of three standard deviations and were requested to be re-assayed. Re-assay results reported corrected values and the errors were determined to be isolated to the CRM samples.

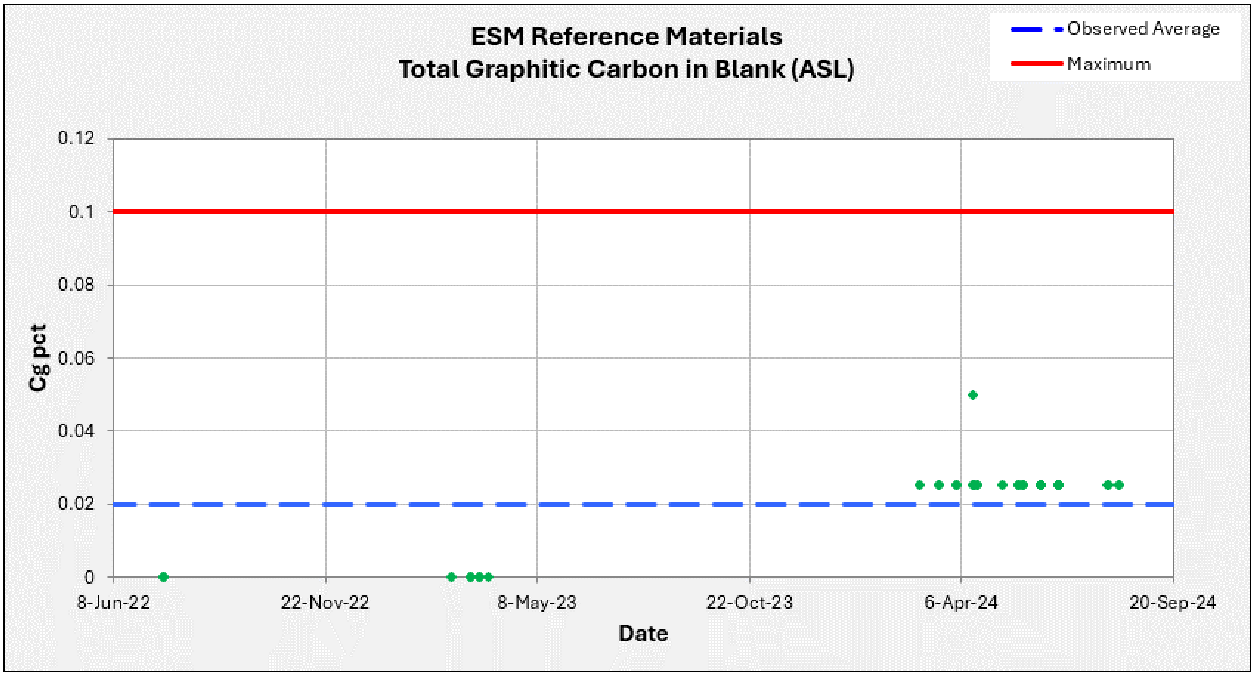
Summary statistics for OREAS-722, 724, 725 graphite performed well and report on average within 101.2–104.4% of the expected values. The results for the certified reference materials are summarized in Table 11-11.

Table 11-11: Summary of results for reference materials

Reference Material	Qty	Outliers Excluded	Failures Excluded	Cg %		Observed Cg %		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-725	24	-	-	24.52	0.728	24.817	0.779	101.2%
OREAS-724	44	-	-	12.06	0.311	12.134	0.446	100.6%
OREAS-722	46	-	-	2.03	0.093	2.120	0.059	104.4%
Total	**114**					**Weighted Average**		**102.3%**

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 ± 3 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.

Figure 11-6 through Figure 11-8 are the control charts for each CRM.



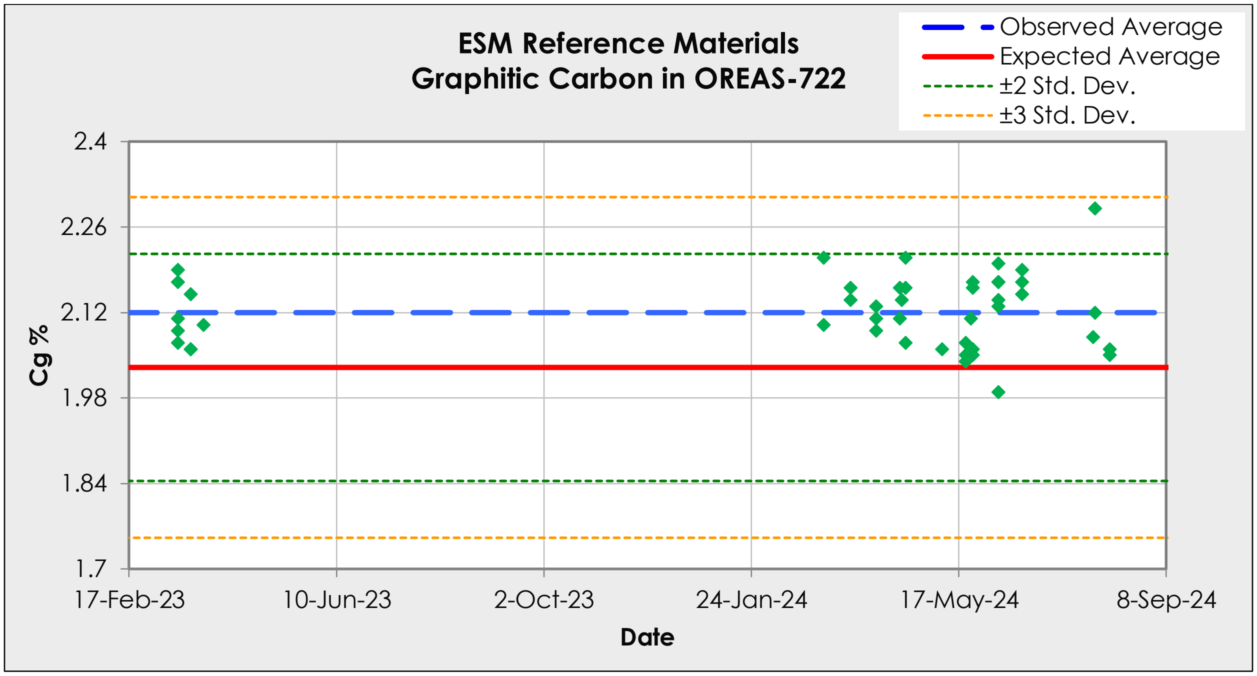
Summary Statistics

Expected Values	
Mean	0.000
Maximum	0.100

Observed Values	
Number of Samples	67
Mean	0.020
Percent of Maximum	19.78%

Source: Graph generated in QC Mine Software

Figure 11-5: Graphitic carbon in blank control chart



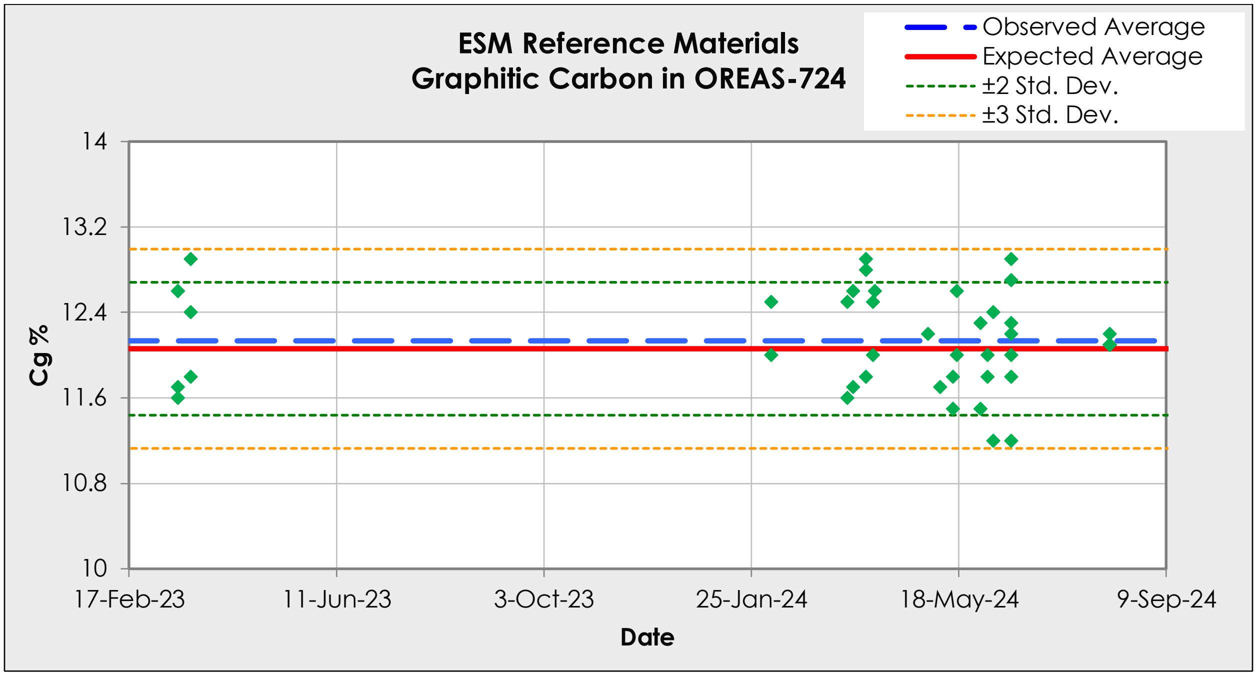
Summary Statistics

Expected Values	
Mean	2.030
Standard Deviation	0.093
2 x RSD	9.16%

Observed Values	
Number of Samples	46
Mean	2.120
Standard Deviation	0.059
2 x RSD	5.59%
Falls Within 3 SD of Certified Mean	100%
Falls Within 2 SD of Certified Mean	98%
Falls Within 1 SD of Certified Mean	52%

Source: Graph generated in QC Mine Software

Figure 11-6: Control chart for graphitic carbon in reference material OREAS-722



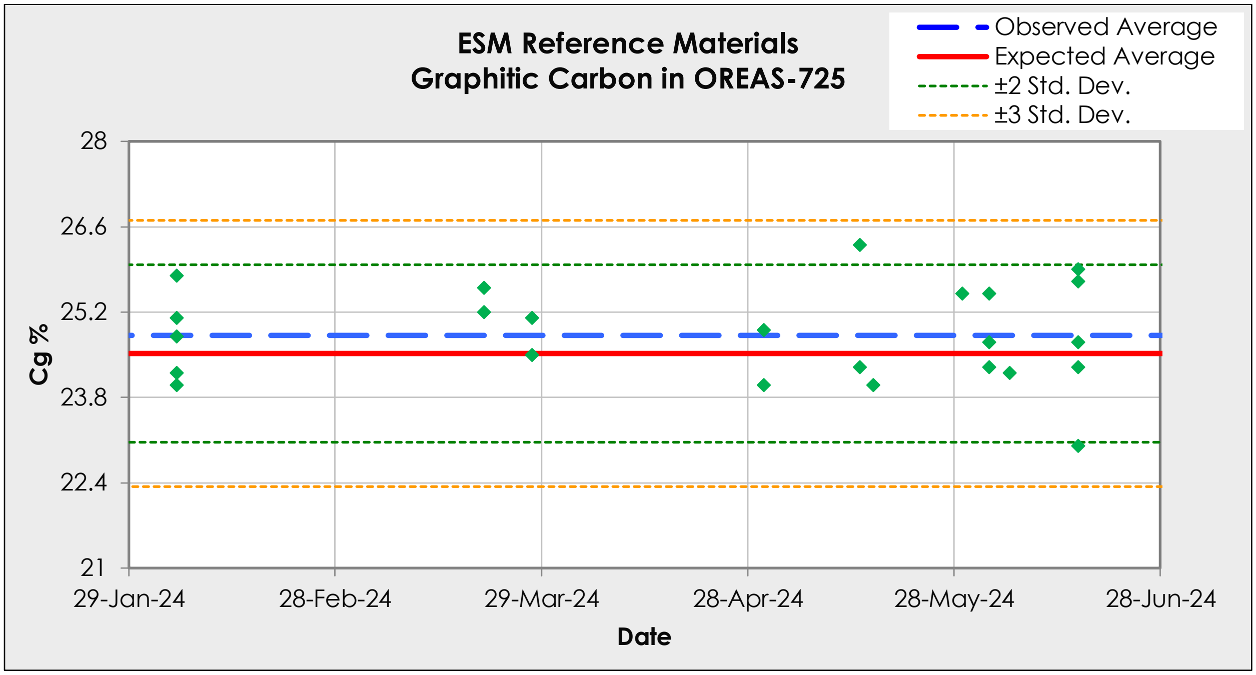
Summary Statistics

Expected Values	
Mean	12.060
Standard Deviation	0.311
2 x RSD	5.16%

Observed Values	
Number of Samples	44
Mean	12.134
Standard Deviation	0.446
2 x RSD	7.35%
Falls Within 3 SD of Certified Mean	100%
Falls Within 2 SD of Certified Mean	84%
Falls Within 1 SD of Certified Mean	45%

Source: Graph generated in QC Mine Software

Figure 11-7: Control chart for graphitic carbon in reference material OREAS-724



Summary Statistics

Expected Values	
Mean	24.520
Standard Deviation	0.728
2 x RSD	5.94%

Observed Values	
Number of Samples	24
Mean	24.817
Standard Deviation	0.779
2 x RSD	6.28%
Falls Within 3 SD of Certified Mean	100%
Falls Within 2 SD of Certified Mean	92%
Falls Within 1 SD of Certified Mean	67%

Source: Graph generated in QC Mine Software

Figure 11-8: Control chart for graphitic carbon in reference material OREAS-725

11.4 Qualified Person's Opinion

It is the opinion of the QP 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 (Makarenko et al., 2018) consisted of 4,317 holes from which a subset of 633 were used for the previous MRE. 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 5 ft 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 modeling. 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

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

The QP has a fulsome staff of experts on-site that thoroughly review and verify ESM technical data on a regular basis, as described above. For this reason, the QP has relied entirely on such verification procedures for verifying the data in this technical report.

The QP has not completed a 100% validation of the entire database to original source data. Focus has been placed 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. The QP is of the opinion that the verification process is appropriate, and that the drilling database is adequate for the purposes of this technical report.

12.5 Kilbourne Data Validation

12.5.1 Site Investigation

Mr. Todd McCracken, P.Geo., visited the Property on August 26 and 27, 2024.

While on site, Mr. McCracken examined the outcrops, drill collar location, channel samples and diamond drill core. Mr. McCracken reviewed the geology, logging procedures and the QA/QC procedures with ESM.

12.5.2 Drill Collar Validation

The QP confirmed the locations of 19 surface borehole collars during the site investigation in 2024. The QP collected the collar locations using a Garmin GPSMap 65 handheld GPS unit. All collar locations were located within the acceptable error limit of the handheld GPS unit.

12.5.3 Database Validation

For the purpose of this MRE, BBA's geological team, under the supervision of the QP, performed the validation on the Project's database. All the data was provided by Titan.

The Project contains 45 drillholes. No major errors were identified.

12.5.4 Independent Sampling

The QP did not collect independent samples.

12.5.5 Qualified Person's Opinion

The QP is of the opinion that the data is considered acceptable to support Mineral Resource if the Kilbourne Graphite Project.

13. Mineral Processing and Metallurgical Testing

13.1 Zinc

Empire State Mines is a currently operating mine, processing underground mineralization to produce zinc concentrate. Two new zones of near-surface mineralization near the existing operation were recently discovered. 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. That test work is reviewed in Section 13.1.2.

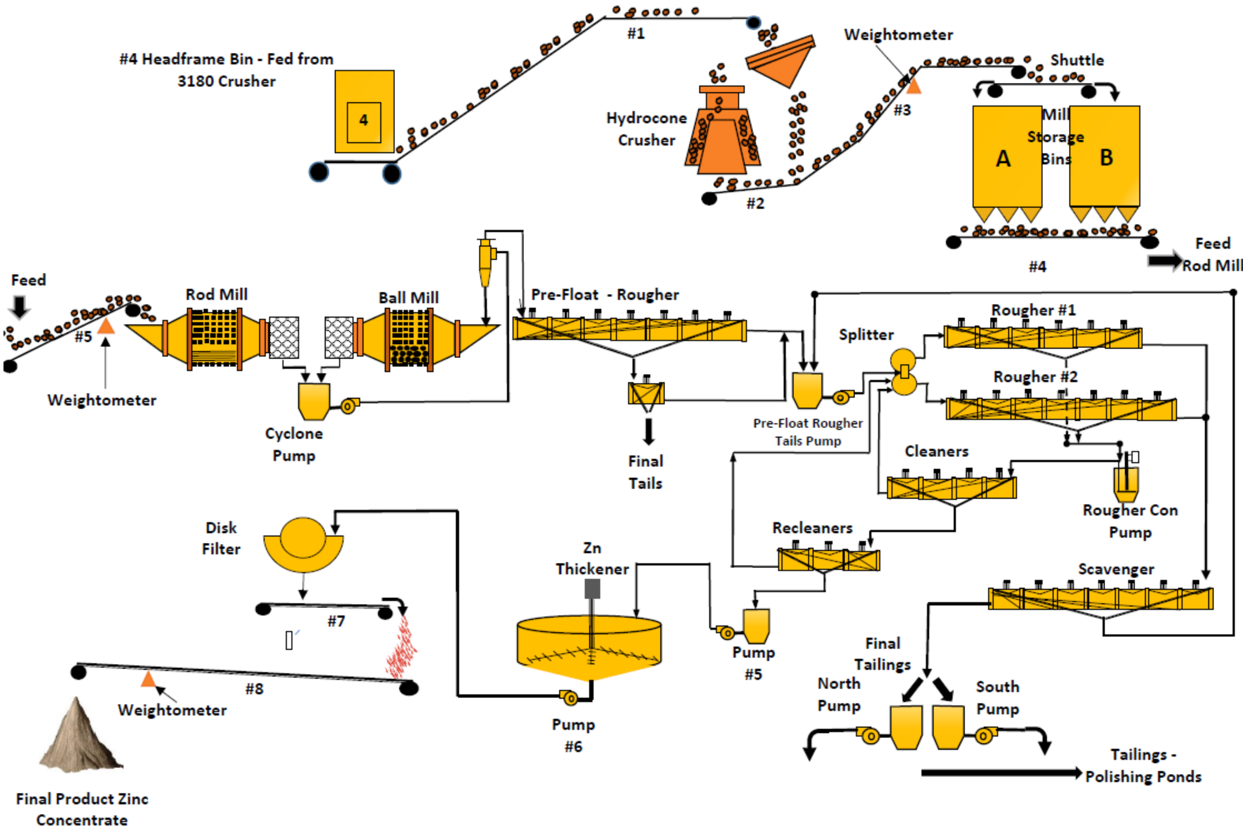
13.1.1 Processing 2018-2024

A test program was undertaken by Hudbay in 2005 (Hudbay, 2005b) 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 ESM 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.

The present flowsheet is shown in Figure 13-1. While the original design of the concentrator was as Pb-Zn, the present mine mineralization does not support the production of a saleable lead concentrate. The existing Lead Rougher has been re-purposed as a pre-float for light, deleterious materials. Using only a frother, this collector-less flotation has been instrumental in reducing the level of magnesium in the final concentrate to 0.50%.



Source: ESM 2024

Figure 13-1: ESM Mill flowsheet

The coarser grind has been beneficial in the form of efficient dewatering and improved recoveries. The concentrate dryer has not been in use since 2019 due to operational cost. Using only the vacuum disc filter, the moisture of the produced concentrate is maintained at an acceptable level for storage and/or shipment even during winter months. Pyrite depression is achieved with sodium sulfide and sodium cyanide in the grinding and cleaner circuits. This allows for the iron in the concentrate to be maintained in the 2.8-3.0% range which, in turn, will allow for zinc concentrate grades of 60% to be realized. This approach has shown to be effective with the milling up to a 50% addition rate of high pyrite mineralization.

The current process does not include any on-line or in-stream metallurgical analysis instrumentation, nor automated stream sampling. The operating crew utilizes 'panning' and visual monitoring of the froth to make process adjustments. Periodic samples are taken through the operating shift for analysis in the laboratory on the following day.

Table 13-1: ESM mill statistics 2018-2023

Year	Tons Milled	Head Grade (Zn %)	Recovery (%)	Concentrate Grade (Zn %)	Concentrate Tons
2018	187,854	7.9	93.4	58.2	23,932
2019	218,823	8.3	96.4	58.7	29,925
2020	323,414	8.6	96.6	59.3	45,161
2021	387,438	7.5	96.5	59.3	47,066
2022	425,022	7.5	96.4	58.8	52,547
2023	445,803	8.4	96.2	59.6	60,145

Source: ESM 2024

Six years of operational results from 2018-2024 have demonstrated 96% zinc recovery with a zinc concentrate grade of nearly 60%.

13.1.2 Turnpike and Hoist House Metallurgical Test Work

The primary objective of the test work undertaken at RDi in 2020 (RDi, 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 lb (55 kg) 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 Ball 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.

13.1.2.1 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 (Ai) 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 13-2.

The composite samples contained significant levels of zinc and sulfide sulfur. The Turnpike composite assayed 4.04% Zn and 5.4% $S_{sulfide}$, while the Hoist House assayed 2.86% Zn and 5.2% $S_{sulfide}$. 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 13-2: Head analyses of composite samples including ICP

	Turnpike	Hoist House
Au, g/tonne	0.022	0.010
Ag, g/tonne	20.2	11.7
Sulfide S %	5.37	5.22
Sulfate S %	3.74	2.38
Total S %	9.11	7.60
Percentage (%)		
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

Source: RDi 2020

13.1.2.2 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 13-3. 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 13-3: Bond's ball mill work index

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

Source: RDi 2020

13.1.2.3 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 sulfide. The zinc was depressed with a combination of sodium cyanide and zinc sulfate 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 sulfate 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 13-4 and Table 13-5.

Table 13-4: 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	106.0	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	106.0	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	126.0	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**

Source: RDi 2020

Table 13-5: 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**

Product	Recovery %				Product Grade		
	Wt.	Ag	Pb	Zn	Ag (g/tonne)	Pb (%)	Zn (%)
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**

Source: RDi 2020

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.

13.1.2.4 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 13-6 and Table 13-7.

Table 13-6: 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**

Source: RDi 2020

Table 13-7: 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	126.0	19.2	1.62
Pb Cleaner 2 Conc	17.2	67.5	86.0	12.4	103.0	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	142.0	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**

Product	Recovery %				Product Grade		
	Wt.	Ag	Pb	Zn	Ag (g/tonne)	Pb (%)	Zn (%)
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**

Source: RDi 2020

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.

13.1.2.5 Projected Lead Recovery and Process Flowsheet

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

- The lead rougher recovery would 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 would 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 would 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 would be similar to that obtained with the underground mineralization.

13.2 Graphite

One mineralogical characterization and two scoping level metallurgical test programs were completed at SGS Canada in Lakefield, Ontario and at Forte Analytical in Fort Collins, Colorado.

13.2.1 SGS Mineralogical Characterization

Seven drill core samples were received by the SGS Advanced Mineralogy Facility from Empire State Mines for mineralogical examination (Grammatikopoulos et al., 2023 – draft). The mineralogical examination was carried out using optical microscopy, X-Ray Diffraction (XRD), and geochemical assays.

The samples consisted mainly of SiO_2, Al_2O_3, CaO, Fe_2O_3, MgO, and lesser TiO_2, Na_2O, and K_2O. The graphitic carbon (Cg) content of the seven samples ranged from 1.97% C(g) to 9.53% C(g) and total sulfur ranged from 0.39% S to 3.87% S.

The results of the XRF analysis on the seven samples is presented in Table 13-8. The most abundant minerals were calcite, plagioclase, diopside, chlorite, quartz, and potassium feldspar. The mineralogical composition of the seven samples differed significantly.

Table 13-8: Results from the XRD Analysis

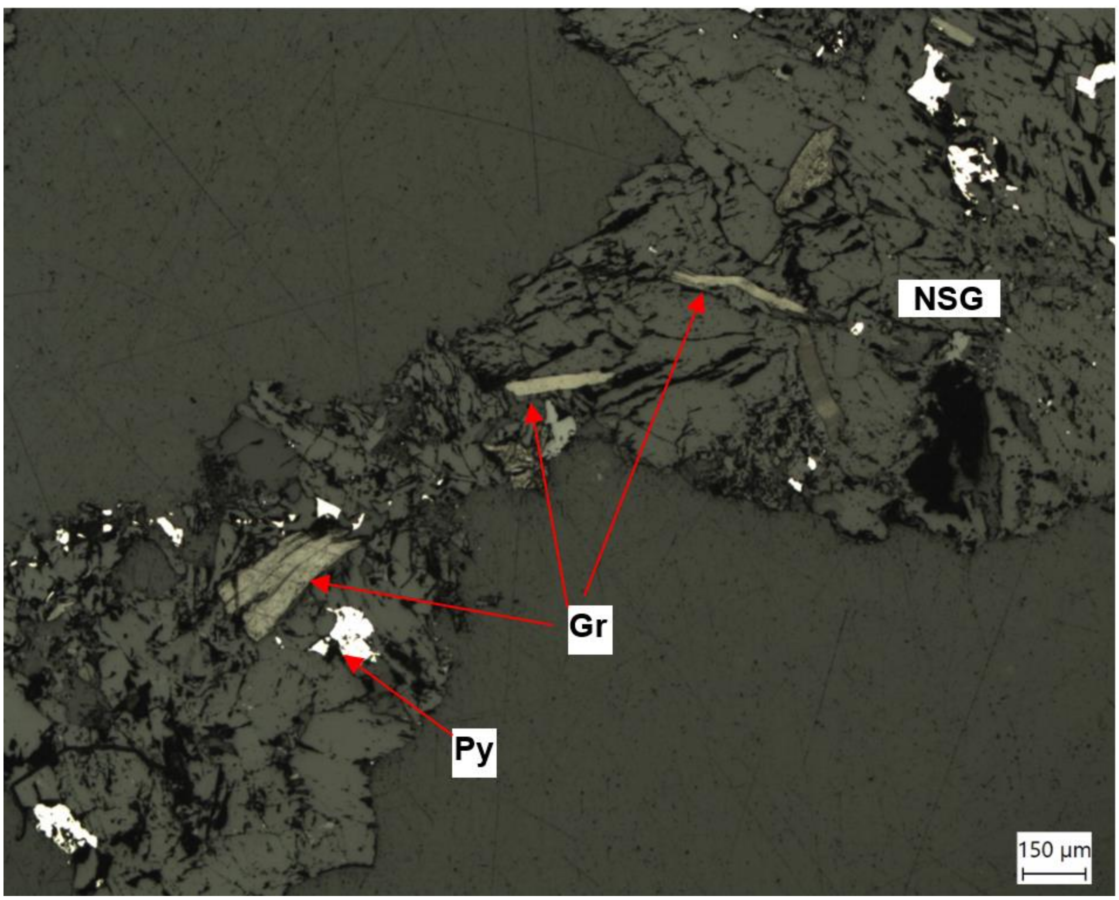
Mineral	F03225	F031913	F031995	F031911	F031518	F032222	F032245
Quartz	2.3	4.8	2.9	18.2	6.0	2.1	8.4
Calcite	2.0	63.0	77.3	17.3	3.3	1.6	46.3
Plagioclase	56.6	1.5	0.2	3.1	30.0	49.9	1.6
Chlorite	6.1	11.1	0.4	17.1	10.8	4.7	2.5
Diopside	7.9	0.9	9.2	1.9	20.1	13.7	25.0
Pyrite	0.6	3.5	1.2	4.8	4.8	0.8	0.4
Meionite	0.2	0.0	1.9	0.7	0.1	0.0	0.5
Mica	1.6	1.6	2.9	2.9	3.3	1.9	1.0
Potassium Feldspar	3.5	3.0	0.0	27.2	8.8	5.4	7.6
Pyrrhotite	1.1	1.6	0.6	1.3	1.5	0.8	1.4
Magnetite	1.3	1.0	0.8	0.8	0.6	0.0	0.7
Grpahite	7.0	6.3	2.5	4.7	9.5	9.2	4.6
Amphibole	9.8	0.7	0.0	0.0	1.2	9.9	0.0
Marcasite	0.0	1.1	0.0	0.0	0.0	0.0	0.0

Source: SGS 2023

Optical microscopy of the samples indicated that graphite was acicular to prismatic, and platy in habit. It ranged from <50 µm as individual flakes to 1.5 mm in size as polycrystalline clusters. Graphite was generally finer-grained in the low-grade samples and coarser in the higher-grade samples.

Graphite occurred disseminated in the matrix of rock fragments comprised mainly of non-sulfide gangue (NSG) (silicates, carbonates, and oxides), as intergrowths with NSG and sulfides, and interlayered with NSG and, less commonly, sulfides.

Most intergrowths of graphite displayed simple (i.e., straight to weakly curvilinear) contacts, locally moderately complex, with the NSG, and would be expected to liberate well upon grinding. However, fine-grained, interstitial, or locked graphite would require additional grinding to further liberate. An example of disseminated prismatic flakes of graphite (Gr)(red arrow) interstitially locked in NSG minerals is presented in Figure 13-2.



Source: SGS 2023

Figure 13-2: Photomicrographs from the optical microscope from F03225

13.2.2 SGS Phase I Metallurgical Program

Assay rejects were submitted to SGS Canada in October 2023 to form two composites for metallurgical testing. The primary objective of the test program was to assess the metallurgical response of the mineralized material. The two composites included samples from the Kilbourne and Bostwick Creek graphite targets. Only test results from the Kilbourne graphite prospect are included in this report.

The Kilbourne composite was submitted for detailed chemical characterization and the results are presented in Table 13-9. The lower head grade of 1.67% C(g) was the result of combining all intervals of four drillholes including bands of barren mineralization.

Table 13-9: Chemical analysis of Kilbourne composite

Element	Kilbourne Composite
C(t) %	1.96
C(g) %	1.67
TOC %	< 0.05
TIC %	0.32
SiO_2 %	61
Al_2O_3 %	11.1
Fe_2O_3 %	9.07
MgO %	2.89
CaO %	2.77
Na_2O %	0.19
K_2O %	4.37
TiO_2 %	0.55
P_2O_5 %	0.25
MnO %	0.1
Cr_2O_3 %	0.03
V_2O_5 %	0.06
S %	3.81
LOI %	5.93
Sum %	98.3

Source: SGS 2024

A total of eight rougher and cleaner flotation tests were carried out with the Kilbourne composite. Five flotation tests evaluated different flash and rougher circuit configurations and primary grind sizes. A summary of pertinent mass balance data of the five flash and/or rougher flotation tests is presented in Table 13-10.

Tests F3 and F4 employed a flash flotation stage followed by a regrind of the flash flotation tailings and rougher flotation. The objective of flash flotation is to recover any coarse graphite flakes as early as possible before they are overground. For this reason, flash flotation is generally incorporated into the grinding circuit. The remaining three tests eliminated flash flotation and, instead, the entire sample was ground to the final grind size target.

Test F3 processed the -6-mesh sample, which corresponded to a P_{80} of 1,850 microns, to flash flotation. The flash flotation tailings were reground to a grind size of approximately P_{80} = 170 microns prior to rougher flotation. The flash flotation stage recovered 56.3% of the contained graphite at a grade of 22.3% C(t). The rougher flotation stage recovered and additional 41.3% of the graphite and the resulting combined flash and rougher concentrate grade was 10.9% C(t).

In test F4, the sample was ground to a P_{80} of approximately 1,000 microns and the flash flotation tailings were reground to P_{80} ~100 microns. The graphite recovery into the flash flotation concentrate increased to 87.3% albeit at a lower grade of 12.1% C(t). The combined flash and rougher concentrate contained 97.3% of the graphite at a grade of 9.78% C(t). Performing the flash and rougher flotation at a finer grind size resulted in a slightly lower grade, but a comparable high graphite recovery of over 97%.

The three rougher tests explored different grind sizes, namely P80 = 120 microns in test F6, P80 = 86 microns in test F5, and P80 = 53 microns in test F8. Test F8 with the finest primary grind size produced a combined rougher concentrate grading 26.6% C(t) at 97.4% graphite recovery. The two other rougher tests produced near identical grades and recoveries of 17.7-17.8% C(t) and 97.2-97.3%, respectively.

Table 13-10: Flash & rougher flotation tests (F3 to F6)

Test	Product	Weight	Assays, %	% Distribution
		%	C(t, g)	C(t)
F3	Flash Conc 1	3.3	25.5	48.0
	Flash Conc 1-2	4.4	22.3	56.3
Flash & Rougher	Flash Conc 1-2 & Ro Conc 1	11.8	13.6	92.6
Flash P_{80} = 1,850 microns	Flash Conc 1-2 & Ro Conc 1-2	14.2	11.8	96.5
Rougher Tails P_{80} = 167 microns	Flash Conc 1-2 & Ro Conc 1-3	15.6	10.9	97.6
	Ro Tails	84.4	0.05	2.4
	Head (calc.)	100.0	1.74	100.0
F4	Flash 1	7.0	16.9	75.7
	Flash 1 + Flash 2	11.3	12.1	87.3
Flash & Rougher	Flash 1+2 Ro Conc 1	12.6	11.6	93.3
Flash P_{80} ~ 1,000 microns	Flash 1+2 Ro Conc 1 + 2	14.2	10.6	96.0
Rougher Tails P_{80} = 100 microns	Flash 1+2 Ro Conc 1+2+3	15.6	9.78	97.3
	Ro Tails	84.4	0.05	2.7
	Head (calc.)	100.0	1.56	100.0
F5	Ro Conc 1	3.5	38.6	80.6
	Ro Conc 1-2	5.7	27.1	93.1
Rougher Only	Ro Conc 1-3	6.7	23.8	95.2
33 min Grind	Ro Conc 1-4	7.8	20.7	96.3
Rougher Tails P_{80} = 86 microns	Ro Conc 1-5	9.2	17.7	97.3
	Ro Tails	90.8	0.05	2.7
	Head (calc.)	100.0	1.67	100.0
F6	Ro Conc 1	4.9	19.8	58.7
	Ro Conc 1-2	7.3	19.3	86.2
Rougher Only	Ro Conc 1-3	8.2	19.0	94.3
21 minute grind	Ro Conc 1-4	8.4	18.8	96.0
Rougher Tails P_{80} = 120 microns	Ro Conc 1-5	8.6	18.6	96.6
	Ro Conc 1-6	9.0	17.8	97.2
	Ro Tails	91.0	0.05	2.8
	Head (calc.)	100.0	1.64	100.0
F8	Ro Conc 1	3.0	28.4	48.7
	Ro Conc 1-2	5.5	27.4	85.1
Rougher Only	Ro Conc 1-3	6.1	27.4	95.3
40 minute grind	Ro Conc 1-4	6.3	27.1	96.7
Rougher Tails P_{80} = 53 microns	Ro Conc 1-5	6.5	26.6	97.4
	Ro Tails	93.5	0.05	2.6
	Head (calc.)	100.0	1.77	100.0
C(g)				

Source: SGS 2024

Based on the flash and rougher flotation test results, SGS decided to proceed with rougher flotation only and then subject the rougher concentrate to primary cleaning tests to upgrade the intermediate concentrate. Primary cleaner tests F7 and F9 were identical except for the primary grind size, which was P_{80} = 120 microns in test F7 and P_{80} = 53 microns in test F9. Both tests then subjected to rougher concentrate to 30 minutes of polishing followed by three stages of cleaner flotation. A summary of the mass balance for the two tests is presented in Table 13-11.

Table 13-11: Primary cleaner flotation tests

Test	Product	Weight	Assays, %	% Distribution
		%	C(t, g)	C(t)
F7	3rd Clnr Conc	1.6	76.6	78.4
	2nd Clnr Conc	2.0	72.4	89.6
Primary Cleaner	1st Clnr Conc	2.3	62.9	91.5
Rougher Tails P_{80} = 120 microns	Rougher Conc	8.2	18.8	96.5
30 min Polishing	Rougher Tails	91.8	0.06	3.5
	Head (calc.)	100.0	1.59	100.0
F9	3rd Clnr Conc	2.0	72.2	90.4
	2nd Clnr Conc	2.1	67.4	90.6
Primary Cleaner	1st Clnr Conc	2.6	55.0	91.4
Rougher Tails P_{80} = 53 microns	Rougher Conc	5.7	25.6	92.8
30 min Polishing	Rougher Tails	94.3	0.12	7.2
	Head (calc.)	100.0	1.57	100.0
C(g)				

Source: SGS 2024

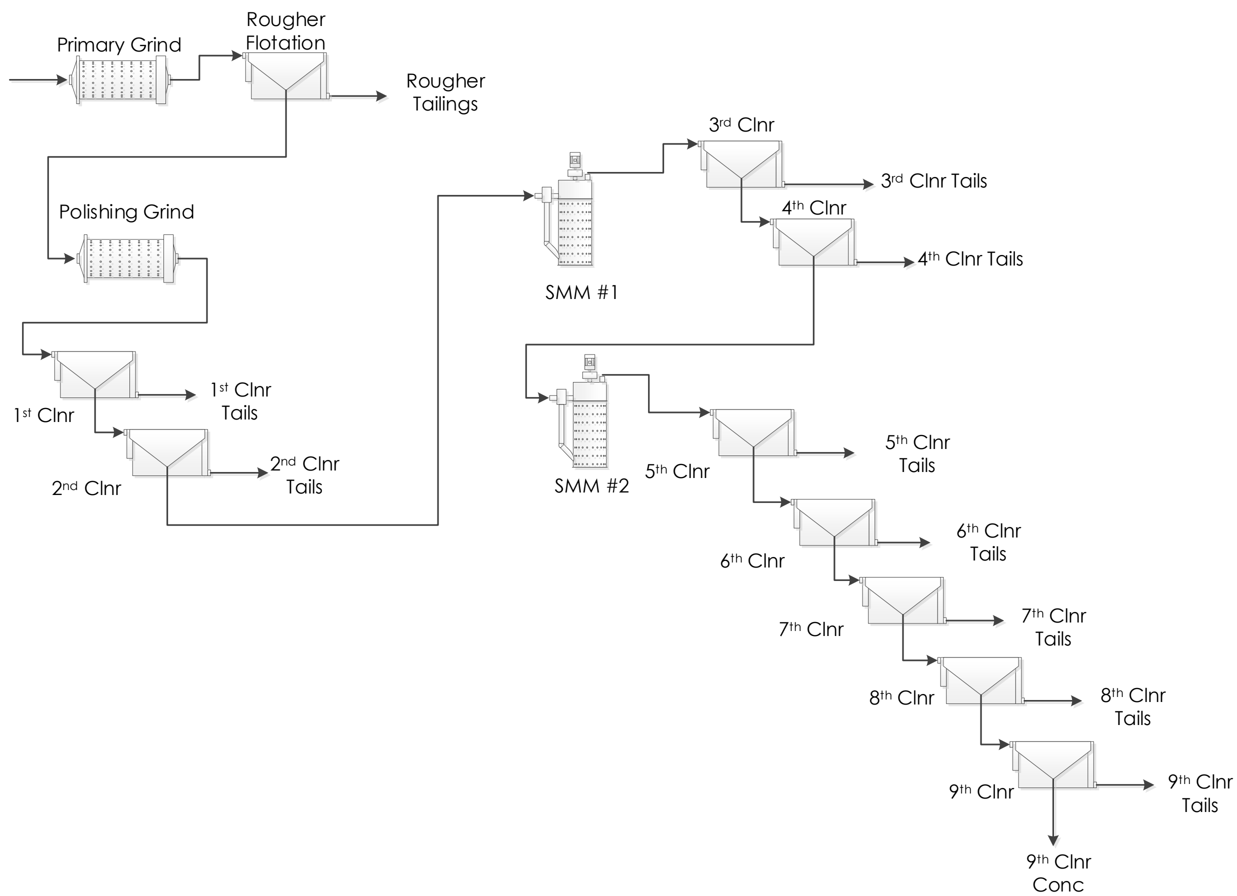
The two tests produced similar 3rd cleaner concentrate grades of 76.6% C(t) in test F7 and 72.2% C(t)in test F9. Despite the finer primary grind, test F9 yielded higher rougher tailings losses of 7.2% compared 3.5% in test F7 with the coarser primary grind. The open circuit total graphite recovery of the test with the finer grind size was noticeably higher at 90.4% compared to only 72.1% in test F7 with the coarser primary grind size.

SGS employed the conditions of test F7 followed by two stages of stirred media milling (SMM) and cleaner flotation for the final test F10. The flowsheet of test F10 is depicted in Figure 13-3 and a summary of the mass balance is shown in Table 13-12. The additional regrinding steps followed by cleaner flotation elevated the combined concentrate grade to 96.6% C(t) at an open circuit graphite recovery of 72.1%. Even the 5th cleaner concentrate still yielded an acceptable concentrate grade of 95.9% C(t) at a higher open circuit graphite recovery of 86.5%.

Table 13-12: Results of full cleaner test F10

Product	Weight	Assays, %	% Distribution
	%	C(t, g)	C(t)
9th Clnr Conc	1.3	96.6	72.1
8th Clnr Conc	1.4	96.5	78.2
7th Clnr Conc	1.5	96.3	82.9
6th Clnr Conc	1.5	95.9	86.5
5th Clnr Conc	1.6	94.5	90.0
4th Clnr Conc	1.7	89.7	91.8
3rd Clnr Conc	1.8	85.6	92.5
2nd Clnr Conc	2.5	63.2	94.4
1st Clnr Conc	3.1	52.9	95.1
Rougher Conc	10.0	16.5	97.4
Rougher Tails	90.0	0.05	2.6
Head (calc.)	*100.0*	*1.70*	*100.0*

Source: SGS 2024



Source: Metpro 2024

Figure 13-3: Flowsheet test F10

The 9th cleaner concentrate of test F10 was submitted for a size fraction analysis and the results are presented in Table 13-13. A total of 7.6% of the concentrate mass reported to the +100 mesh size fractions. Even the smallest size fraction of -200 mesh still produced a very higher total carbon content of 97.4% C(t).

Table 13-13: Size fraction analysis of F10 9th cleaner concentrate

Concentrate Size Fraction	Wt. %	Assays, % C(t)	% Distribution C(t)
+65 mesh	0.9	94.6	0.9
+80 mesh	1.8	95.0	1.7
+100 mesh	4.9	96.6	4.8
+150 mesh	18.2	99.2	18.6
+200 mesh	23.1	96.8	23.0
-200 mesh	51.1	97.3	51.0
Total Concentrate	100.0	97.4	100.0

Source: SGS 2024

13.2.3 Forte Analytical Phase II Metallurgical Program

Forte Analytical was requested to perform scoping level metallurgical test work with the primary objective of developing a preliminary process flowsheet to recover coarse graphite from the Kilbourne mineralization (Forte, 2024). The approach to recover coarse graphite entailed two parallel circuits for coarse graphite and fine graphite processing.

Forte received approximately 75 kg of two ore samples for the study. The samples were from the upper zone supposedly containing coarse graphite (designated Batch 1) and a deeper zone containing fine graphite (designated Batch 2). Batch 1 and Batch 2 graded 2.48% C(g) and 2.39% C(g), respectively. Sulfide sulfur concentrations were 5.13% $S^=$ for Batch 1 and 1.23% $S^=$ for Batch 2. A size-by-size analysis revealed that graphite was distributed in all size fractions in proportion to the weight of the sample so that sizing as a primary processing step does not provide an upgrading opportunity.

Flash flotation tests were performed on the two batches. After two minutes of flotation, the flash concentrate contained between 48.9% and 54.0% of the graphite in 7.4% to 7.9% of the mass. Rougher flotation tests were carried out on the flash flotation tailings to recover most of the remaining graphite. The flash flotation tailings were reground to a P_{80} of 100 microns and then subjected to 6 minutes of rougher flotation. The graphite recovery into the combined flash and rougher concentrate after 2 minutes of flash flotation and 2 minutes of rougher flotation was 95.0% for Batch 1 and 97.2% for Batch 2. The grades of the combined flash and rougher concentrate were12.8% C(g) for Batch 1 and 11.4% C(t) for Batch 2.

Cleaner tests were carried out on both Batch 1 and Batch 2 samples to evaluate the upgrading potential of the flash and rougher concentrates. The parameters of the cleaner test series included one to six cleaner stages for both the flash and rougher flotation concentrates, with and without attrition scrub/polishing regrinds.

To generate feed for the cleaner tests, five 2-kg flash and rougher flotation tests were carried out. A summary of the bulk concentrate production mass balance is presented in Table 13-14.

Table 13-14: Bulk concentrate production results

Product	Mass %	Grade C(g) %	Rec C(g) %
Batch 1, Test 5 A/B			
Flash Conc	6.9	18.1	51.6
Rougher Conc	17.7	5.68	42.9
Combined Conc	24.6	9.16	94.5
Batch 1, Test 7 A/B			
Flash Conc	11.7	11.7	58.6
Rougher Conc	13.3	7.36	37.7
Combined Conc	25	9.39	96.3
Batch 1, Test 9 A/B			
Flash Conc	11.1	11.5	58.4
Rougher Conc	13.2	6.73	37.8
Combined Conc	24.3	8.90	96.2
Batch 2, Test 6 A/B			
Flash Conc	8.7	16.9	56.0
Rougher Conc	18.9	5.50	39.5
Combined Conc	27.6	9.09	95.5
Batch 2, Test 8 A/B			
Flash Conc	9.2	14.1	57.2
Rougher Conc	18.4	5.77	39.8
Combined Conc	27.6	8.55	97.0
Batch 2, Test 10 A/B			
Flash Conc	9.5	15.1	60.6
Rougher Conc	21.1	3.66	35.5
Combined Conc	30.6	7.20	96.1

Source: Forte 2024.

The flash and rougher concentrates were upgraded in separate cleaning circuits. Cleaner tests with only one or two stages of regrind and cleaner flotation failed to produce acceptable results. One stage cleaner of the flash flotation concentrate recovered 89.3%-94.4% and 96.1%-98.1% of graphite assaying 18.16%-24.16% C(g) and 21.75%-21.95% C(g) for Batch 1 and 2, respectively. Two-stage cleaning of the flash flotation concentrate recovered 89.2% and 90.2% of graphite assaying 34.04% C(g) and 27.34% C(g) for Batch 1 and 2, respectively. Attrition scrubbing improved the initial concentrate grade as compared to non-attrition. This was more pronounced with Batch 1 (28.8% vs 35.96% C(g). Four stage cleaner of the flash flotation concentrate with polishing grinds recovered 90.2% and 90.5% of graphite assaying 37.53% C(g) and 34.61% C(g) for Batch 1 and 2, respectively.

Treating the flash flotation concentrate with three stages of polishing followed by cleaner flotation produced the best overall results, which are summarized in Table 13-15. Batch 1 produced a 6th cleaner concentrate grading 96.2 % C(g) and containing 48.3% of the graphite. Batch 2 responded inferior with a grade of only 85.6 % C(g) and 55.1% of graphite recovery. Note that the graphite recovery only considers the flash flotation circuit and that global recovery will increase once the rougher flotation performance is taken into account.

Table 13-15: 6th Cleaner tests of flash flotation concentrate

Product	Mass %	Grade C(g) %	Stage Rec C(g) %	Total Rec C(g) %
Batch 1 - Flash 6th Clenaer Kinetics - Polish grind prior to Cleaner 1, 3, and 5				
6th Clnr Conc 1	7.9	97.1	64.6	37.7
6th Clnr Conc 1+2	10.2	96.2	82.7	48.3
5th Clnr Conc	11.4	93.5	89.8	52.5
4th Clnr Conc	14.5	77.7	95.0	55.5
3rd Clnr Conc	16	71.3	96.1	56.1
2nd Clnr Conc	32.3	35.7	97.3	56.8
1st Clnr Conc	36.9	31.5	97.9	57.2
Batch 2 - Flash 6th Clenaer Kinetics - Polish grind prior to Cleaner 1, 3, and 5				
6th Clnr Conc 1	10.1	87.0	72.3	43.8
6th Clnr Conc 1+2	12.9	85.6	90.9	55.1
5th Clnr Conc	14.4	77.2	91.5	55.4
4th Clnr Conc	19.7	57.8	93.7	56.8
3rd Clnr Conc	22.9	50.2	94.7	57.4
2nd Clnr Conc	40.9	28.5	95.8	58.0
1st Clnr Conc	46.4	25.3	96.5	58.5

Source: Forte 2024.

The same upgrading circuits that were evaluated for the flash flotation concentrate were also tested for the rougher concentrate. Again, simple cleaner tests and up to two stages of polishing followed by cleaner flotation failed to produce acceptable concentrate grades. Four stage cleaners of the rougher flotation concentrate with two polishing grinds recovered 88.6% and 83.7% of graphite assaying 38.8% C(g) and 42.19% C(g) for Batch 1 and 2, respectively.

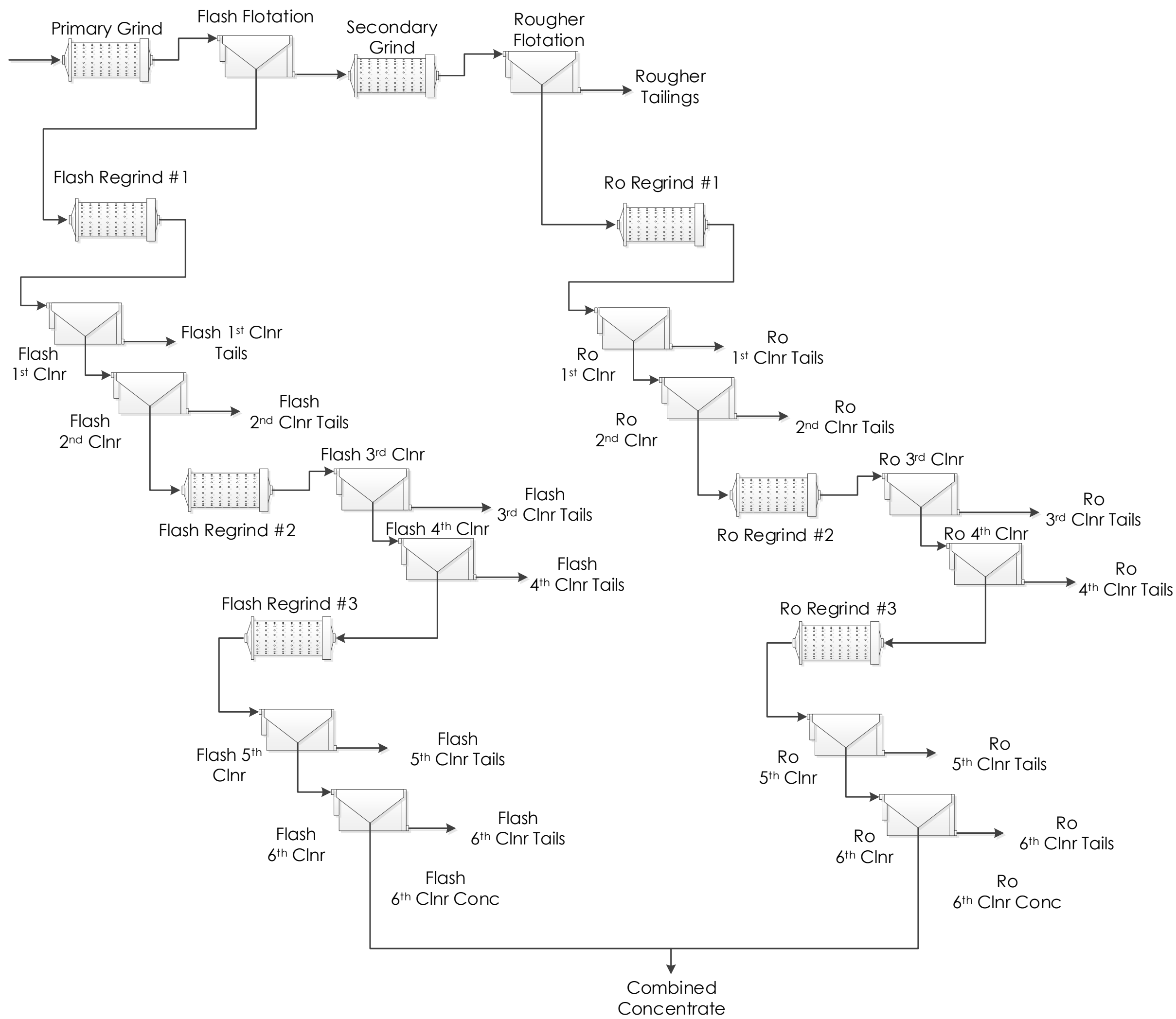
As expected, treating the rougher flotation concentrate with three stages of polishing followed by cleaner flotation produced the best overall results, which are summarized in Table 13-16. Batch 1 produced a 6th cleaner concentrate grading 98.34 % C(g) and containing 26.8% of the graphite. Batch 2 responded inferior with a grade of only 86.6 % C(g) and 26.8% of graphite recovery.

Table 13-16: 6th Cleaner tests of rougher flotation concentrate

Product	Mass %	Grade C(g) %	Stage Rec C(g) %	Total Rec C(g) %
Batch 1 - Flash 6th Cleaner Kinetics - Polish grind prior to Cleaner 1, 3, and 5				
6th Clnr Conc 1	7.9	98.3	71.0	26.8
6th Clnr Conc 1+2	9.9	98.1	88.8	33.6
5th Clnr Conc	10.6	95.5	92.6	35.0
4th Clnr Conc	12.6	82.2	94.8	35.8
3rd Clnr Conc	14	74.6	95.6	36.1
2nd Clnr Conc	28.1	37.5	96.5	36.5
1st Clnr Conc	32.5	32.6	96.9	36.6
Batch 2 - Flash 6th Cleaner Kinetics - Polish grind prior to Cleaner 1, 3, and 5				
6th Clnr Conc 1	3.3	87.4	59.8	21.2
6th Clnr Conc 1+2	4.2	86.6	75.5	26.8
5th Clnr Conc	5.1	79.8	84.4	30.0
4th Clnr Conc	7.4	58.3	89.5	31.8
3rd Clnr Conc	8.9	49.1	90.6	32.2
2nd Clnr Conc	19	23.4	92.3	32.8
1st Clnr Conc	21.9	20.6	93.4	33.1

Source: Forte 2024

Considering the combined performance of the flash and rougher flotation circuit, Batch 1 produced an overall open circuit graphite recovery of 81.9% and a concentrate grade of 97.1% C(g). The graphite recovery for Batch 2 was identical at 81.9% but at a lower concentrate grade of 85.9% C(g). The integrated Forte flowsheet is presented in Figure 13-4.



Source: Metpro 2024

Figure 13-4: Integrated Forte flowsheet

The cleaned flash flotation concentrate for Batch 1 was dry screened and size fractions were submitted for graphitic carbon analysis to quantify the amount of coarse graphite present. The results are shown in Table 13-17. A total of 21.4% of the concentrate mass reported to the +100 mesh size fractions. It should be noted that this size distribution only applies to the cleaning circuit of the flash concentrate of Batch 1 and that the 6th cleaner of the rougher concentrate has to be taken into account for a global concentrate flake size distribution.

The main impurities that were identified in the flotation concentrate included mica/Illite, quartz, serpentine, lepidocrocite, and goethite.

Table 13-17: Size fraction analysis - 6th Cleaner flash concentrate Batch 1

Size Mesh	Mass %	Grade % C(g)
+65	6.8	98
65 x 100	14.6	100
100 x 200	42.9	98
200 x 325	23.9	97
-325	12.0	98
	100.0	98

Source: Forte 2024

13.2.4 Conclusions

Two scoping level metallurgical test programs were completed by SGS Canada and Forte Analytical. While the execution of the test programs varied significantly, the results are consistent. Both programs determined the flake size distribution in the Kilbourne mineralization is relatively fine. The SGS program showed that a graphite recovery grading well over 95% can be generated with a relatively simple process.

The proposed Forte flowsheet is more complex since it includes separate upgrading of the flash and rougher concentrates. While the size fraction analysis of the 6th cleaner concentrate obtained in the flash cleaning circuit produced a higher mass recovery into the +100-mesh product compared to the SGS results, it disregarded the 6th cleaner concentrate of the rougher cleaning circuit, which is expected to be finer grained.

The SGS program has showed that high purities can be achieved even for the small size fractions with the -200-mesh product grading 97.3% C(t). The ability to produce high-grade fines differentiated the Kilbourne mineralization from many other graphite projects and may present excellent marketing opportunities.

Since no clear benefit of separate cleaning circuit is apparent, future testing will focus on the development of a single cleaning circuit treating the combined flash and rougher concentrate. This approach will lead to lower capital and operating costs to upgrade the graphite in the Kilbourne mineralization to a high-grade flotation concentrate. Although the Kilbourne material does not include large flakes, future process optimization work will still focus on minimizing flake degradation to avoid the generation of very fine flakes.

A review of the drillhole data revealed that the material between the upper and lower zones is almost barren. Sensor-based ore sorting may be an effective technology to reject the barren material, thus upgrading the average mill feed noticeably. Hence, ore sorting will be explored in the next phase of testing.

14. Mineral Resource Estimates

This chapter of the report describes updates of the geologic and grade block models for the ESM deposits. Section 14.1 reports updates to the zinc resources following additional drilling and mining exposure since the last technical report. Section 14.2 delivers a maiden resource estimate for a graphite deposit.

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.

14.1 Zinc Mineral Resource Estimate

The ESM zinc deposits are comprised of multiple zones in and around Fowler, NY. There are ten deposits currently considered as viable economic targets; American, Cal Marble, Fowler, Mahler, Mud Pond, N2, Northeast Fowler, New Fold, Sylvia Lake, and Turnpike. Site convention splits the Mahler, Mud Pond, and N2 deposits into two zones each, which is reflected in the models Lower Mahler, Upper Mahler, Mud Pond – Main, Mud Pond – Apron, N2D, and Turnpike. Turnpike was formerly known as N2 Pits in the last technical document. Historic mining at these locations has provided a good geological understanding of each, with supporting mapping, sampling, and drilling data.

This Mineral Resource report update has been prepared by Donald Taylor in accordance with Canadian NI 43-101 guidelines. All geological modeling and grade estimation since 2020 used Leapfrog Geo™ version 2023.2.3 and Edge software. The American and NE Fowler deposits were modeled in Leapfrog Geo™ version 6.0.1 and estimated in Maptek Vulcan in 2019. Mining and grade control experience by ESM geologists have supported that the implicit modeling of the mineralized zones as veins in Leapfrog Geo™ results in more accurate geological wireframes.

14.1.1 Drillhole Database

The drillhole database is stored as an industry standard SQL relational database with an Access interface customized for ESM by Geospark. The database was sub-divided into geographic "Areas" that can be extracted individually. The Balmat Area covers deposits that are the subject of this Mineral Resource report. The Balmat database was exported as CSV files for the annual in-house resource updates and included collar, downhole survey, lithology, assay, and density data. Assays and associated composites were extracted from a total of 1,321 drillholes that were used in estimation. The number of drillholes used for each zone is listed in Table 14-1.

This data has been continually checked for errors by ESM geologists and any errors that have been discovered were corrected in real time. There are historic drillholes with uncertainty in survey or analytical methodology as well as other drillholes that are drilled at low 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. Holes or samples deemed too low confidence for use in grade estimation or geological interpretation are flagged in the database and excluded from the export.

The drillhole database consisted of 8,295 surface or UG core holes. There are 89 sets of channel samples, 1,302 surface core holes, 6,993 UG core holes and 201 holes identified as other (including monitoring wells and blast holes). Smaller subsets of this database were used for geologic modeling and estimation and each zone was modeled separately in isolated geological and estimation projects.

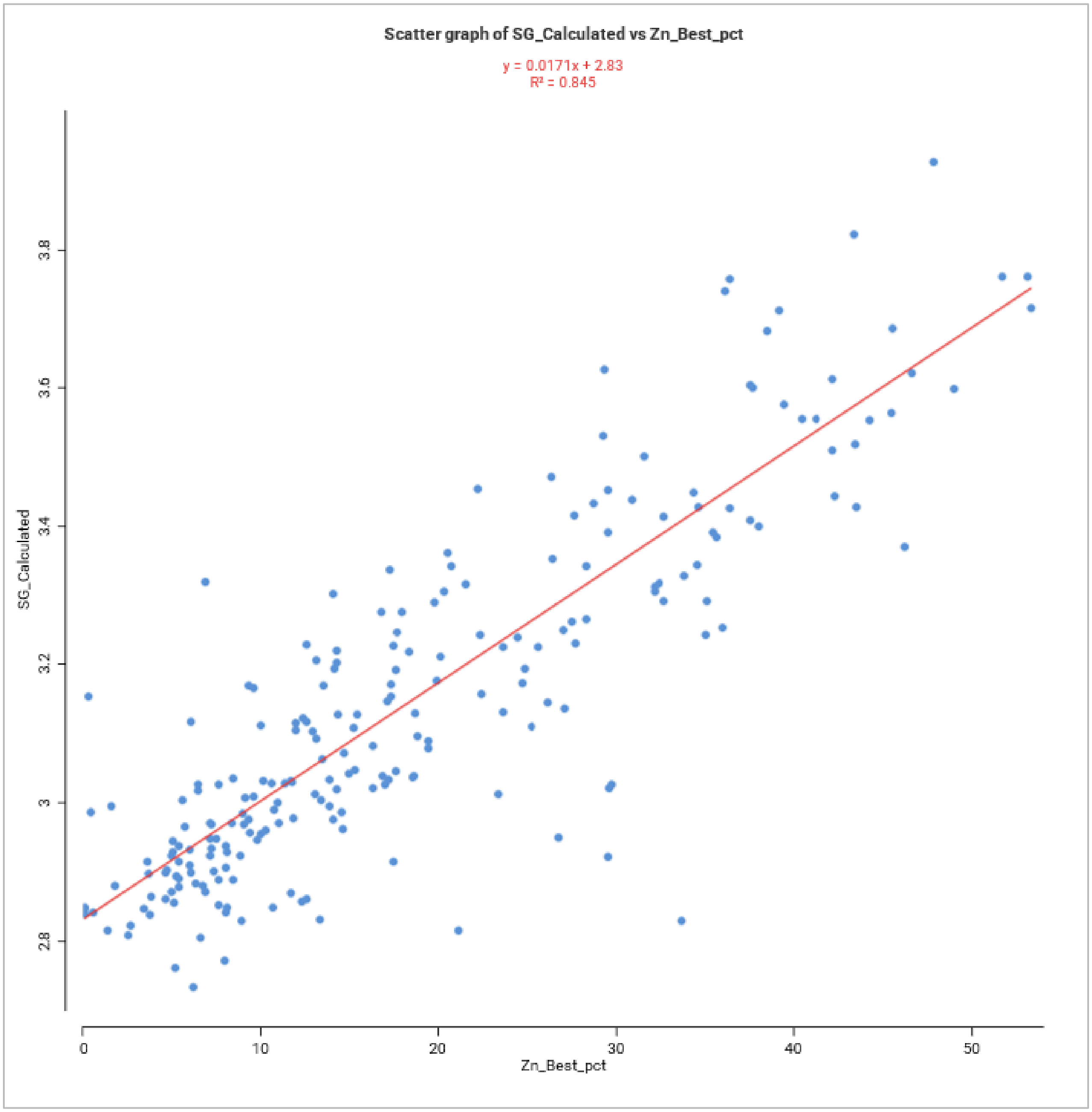
Table 14-1: Core holes used in estimation of each zone

Zone	Number of Core Holes Used
American	42
Cal Marble	25
Fowler	19
Lower Mahler	196
Upper Mahler	114
Mud Pond - Main	136
Mud Pond - Apron	123
N2D	209
New Fold	114
Northeast Fowler	24
Sylvia Lake	98
Turnpike	254

Source: ESM 2024

14.1.2 Density

Bulk density measurements are collected and entered into the drillhole database using the conventional Archimedes method as part site standard core processing in waste and mineralization since 2019. This technique involves weighing samples in air and in water and examining the displacement of the water in a controlled environment to calculate a SG. The values are stored in the drillhole database and each domain was evaluated separately. Where there is sufficient sampling the SG is interpolated into model blocks using inverse distance weighted (IDW) techniques. If insufficient sampling exists then density was assigned to models based on calculated means or by a regression formula. The simple sulfide mineralogy of the ESM #4 Mine resources results in a strongly positive correlative relationship between zinc grade and bulk density. An example of the relationship between zinc grade and SG for the Mahler resource is shown in Figure 14-1. The mean values for the primary zones are listed below in Table 14-2, but are not necessarily the values assigned in the block model. All SG measurements were converted to bulk density using an assumption of equal relationship of SG to grams per cubic centimeter (g/cm^3), and a unit conversion to a tonnage factor (TF) represented in short tons/ft^3.



Source: ESM 2024

Figure 14-1: Scatterplot of specific gravity vs assay zinc (%) for Mahler

Table 14-2: Density by zone and material type

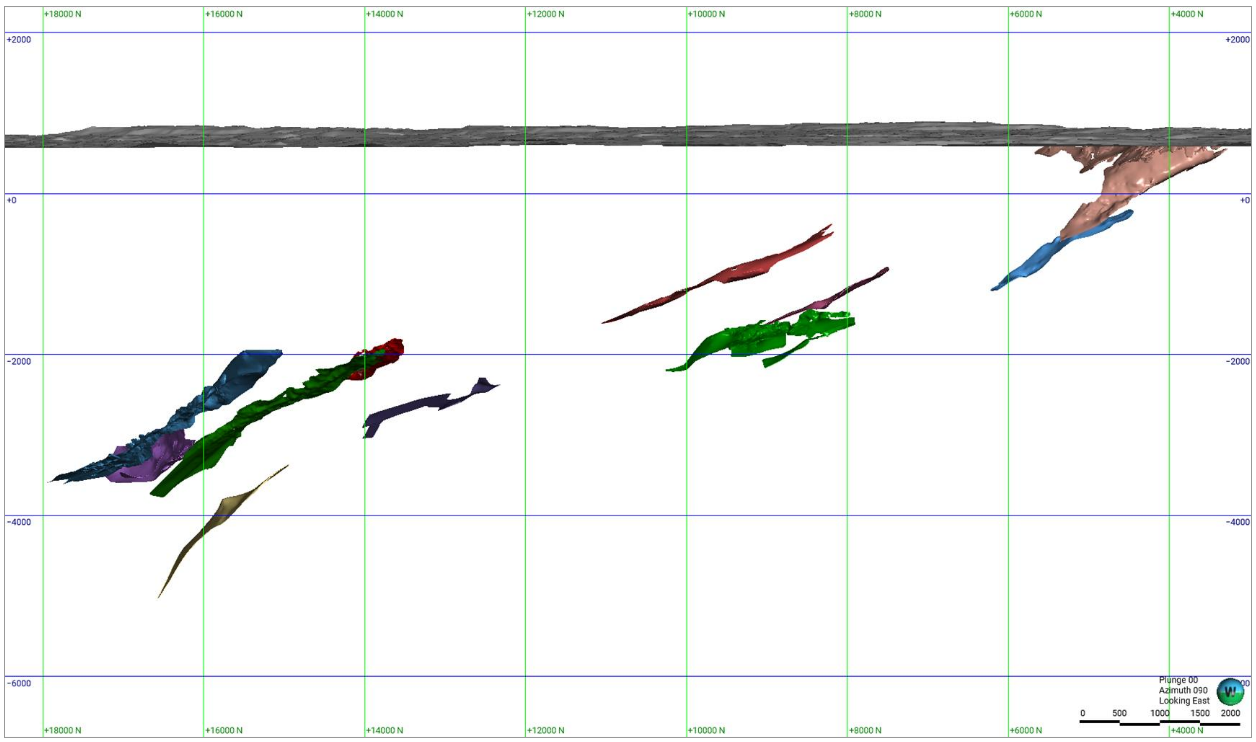
Zone	Specific Gravity	Tonnage Factor (t/ft^3)
American	3.123	0.0975
Cal Marble	3.123	0.0975
Fowler	3.123	0.0975
Mahler - Main	3.131	0.0977
Mahler - White Dolomite	3.243	0.1012
Mud Pond - Apron	3.205	0.1000
Mud Pond - Main	3.065	0.0957
N2D	3.061	0.0955
N2D – Waste	2.930	0.0915
New Fold	3.088	0.0963
Northeast Fowler	3.137	0.0979
Sylvia Lake	3.123	0.0975
Turnpike - UM14	3.266	0.1019
Turnpike - UM11	3.168	0.0989
Turnpike - Waste	2.845	0.0888
Waste	2.800	0.0874

Source: ESM 2024

14.1.3 Topography Data

Base topography is extracted from publicly available New York State LIDAR data. The topography is locally updated from photogrammetric data collected by an ESM owned and operated drone.

The majority of the models were considered below topography as seen in Figure 14-2 with the exception of the Turnpike model which crosses the topographic surface.

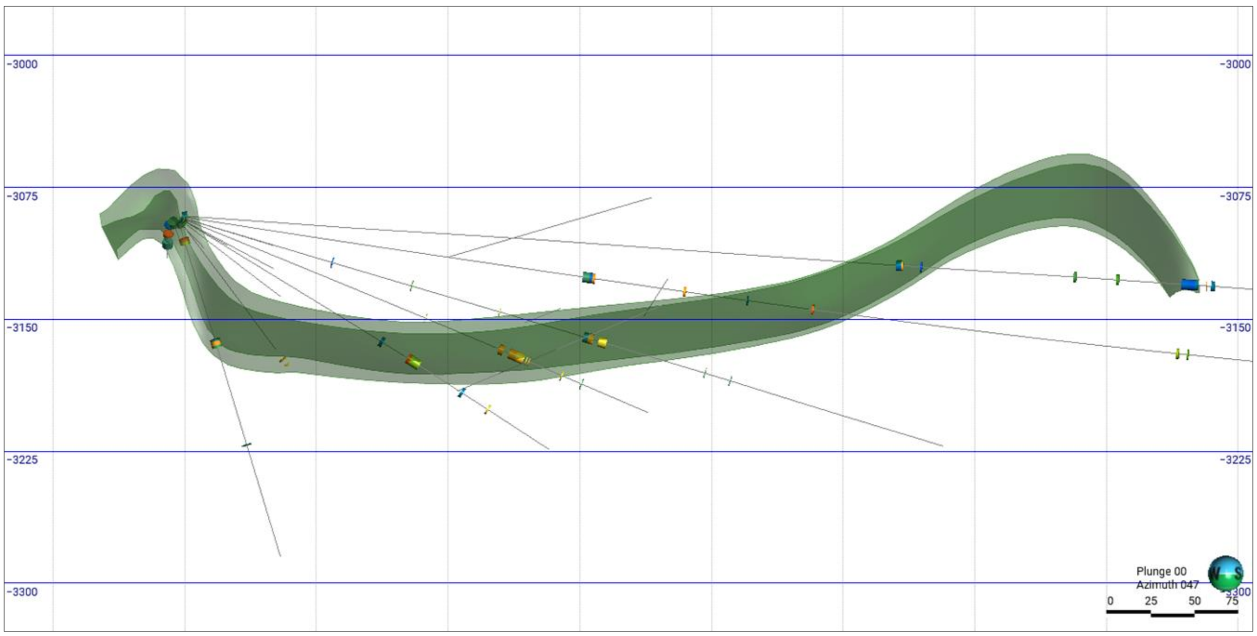


Source: ESM 2024

Figure 14-2: Zones relative to topographic surface

14.1.4 Geological Interpretation

All zones were defined and modeled by ESM geologists. The zones range in complexity and can be comprised of multiple veins designating variably oriented and spatially-distinct mineralized envelopes which were modeled using implicit hard boundary vein systems. Lower grade disseminated mineralization, stockworks, or highly folded systems are modeled using geology polyline guided indicator RBF (radial basis function) interpolant shells. The simplest deposits, such as Mud Pond – Main, can be modeling within a single mineralized envelope as shown in Figure 14-3.



Source: ESM 2024

Figure 14-3: Mud Pond – Main vein model

On the other end of the spectrum, Turnpike was modeled entirely using indicator RBF interpolants internal to modeled stratigraphic domains due to the highly folded and variable nature of the deposit. Statistics for all indicator interpolants are checked for performance and dilution. The performance statistics for Turnpike are shown in Table 14-3.

Table 14-3: Turnpike indicator RBF interpolant performance statistics

UM11 Interpolant			UM14 Interpolant		
Indicator Statistics			**Indicator Statistics**		
Total Number of Samples	**2,087**		**Total Number of Samples**	**5,637**	
Cut-off Value	**0.25**		**Cut-off Value**	**0.25**	
	≥ Cut-off	**< Cut-off**		**≥ Cut-off**	**< Cut-off**
Number of Points	1,162	925	Number of Points	1,337	4,300
Percentage	0.5568	0.4432	Percentage	0.2372	0.7628
Mean Value	4.759	0.0266873	Mean Value	4.03905	0.0136227
Minimum Value	0.25	0	Minimum Value	0.25	0
Maximum Value	24.5	0.248	Maximum Value	26.3	0.2496
Standard Deviation	4.43023	0.0554926	Standard Deviation	4.67031	0.0394753
Coefficient of Variance	0.930915	2.07936	Coefficient of Variance	1.15629	2.89775
Variance	19.6269	0.00307943	Variance	21.8118	0.0015583
Output Volume Statistics			**Output Volume Statistics**		
Resolution	5		Resolution	5	
Iso-value	0.35		Iso-value	0.5	
	Inside	**Outside**		**Inside**	**Outside**
≥ Cut-off			≥ Cut-off		
Number of Samples	1,138	24	Number of Samples	1,188	149
Percentage	0.5453	0.0115	Percentage	0.2108	0.0264
< Cut-off			< Cut-off		
Number of Samples	176	749	Number of Samples	57	4243
Percentage	0.0843	0.3589	Percentage	0.0101	0.7527
All Points			**All Points**		
Mean Value	4.16831	0.10025	Mean Value	3.91493	0.133129
Minimum Value	0	0	Minimum Value	0	0
Maximum Value	24.5	8.28	Maximum Value	26.3	26
Standard Deviation	4.44842	0.609218	Standard Deviation	4.4707	1.25686
Coefficient of Variance	1.0672	6.07702	Coefficient of Variance	1.14196	9.44097
Variance	19.7884	0.371147	Variance	19.9871	1.57971
Volume	39,503,000	30,462,000	Volume	26,439,000	534,940,000
Number of Parts	1	16	Number of Parts	7	12
ESM Calculated			**ESM Calculated**		
Dilution	13.4%		Dilution	4.6%	
Exclusion	2.1%		Exclusion	11.1%	

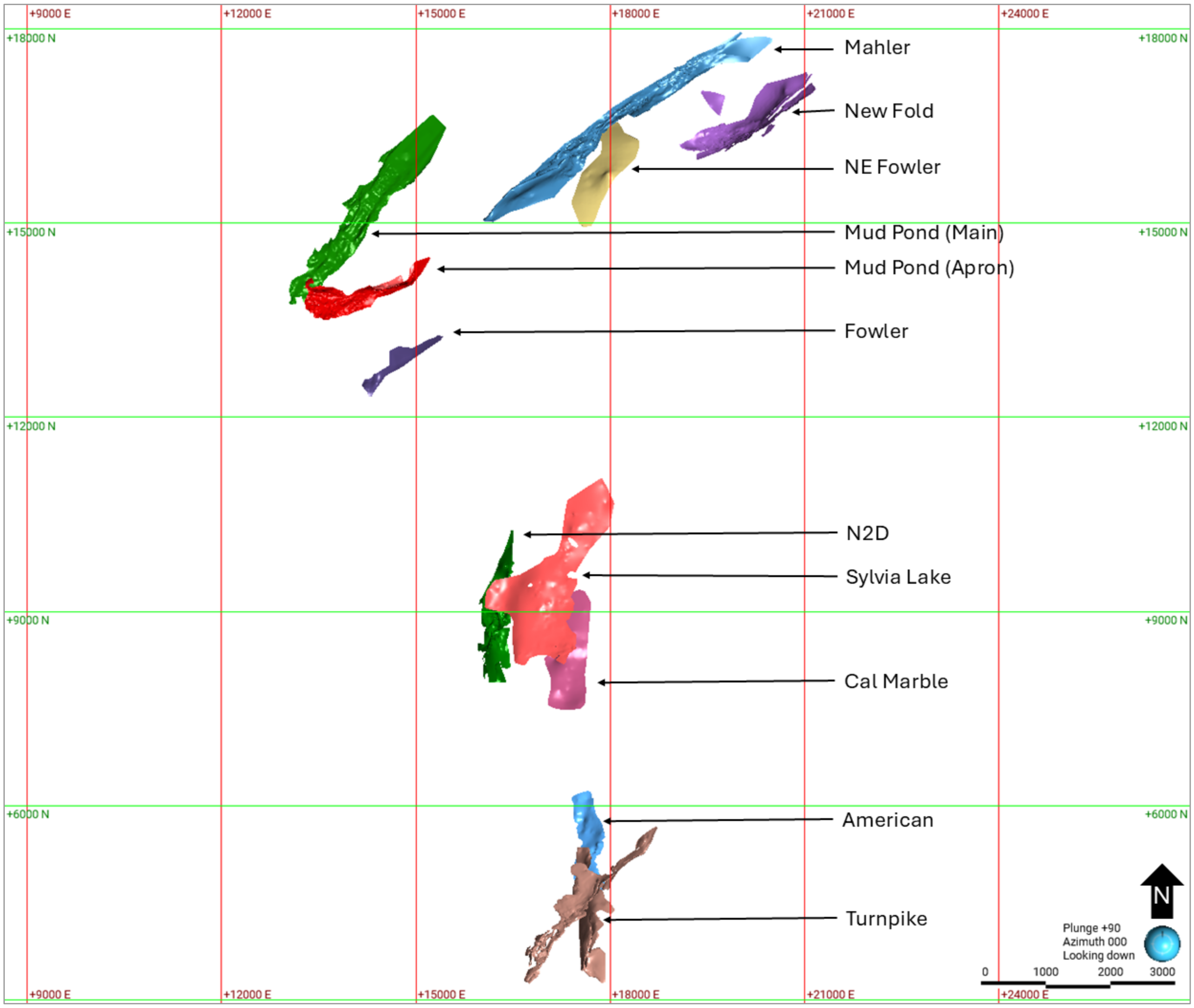
Source: ESM 2024

The resulting 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 modeled by ESM. 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. All modeling at ESM since 2019 has been conducted in Leapfrog Geo™ and updated as new information has become available as needed on an annual basis (Table 14-4). The 2024 model updates were completed in version 2023.2.3. The American and Northeast Fowler deposits were modeled in Leapfrog Geo™ version 6.0.1 and estimated in MapTech Vulcan as described in the prior Technical Report (Warren et al., 2021). Each zone has been analyzed and divided where appropriate to facilitate a more accurate estimation of grade. This has resulted in splitting of domains based on morphology or orientation for the purposes of estimation. Location and volume of each is demonstrated in Table 14-4 and Figure 14-4.

Table 14-4: Update periods, model methodology, and volumes

Zone	Modeling Method	Years Modeled and Updated	Model Volumes
American	Implicit vein model	2019	4,586,000
Cal Marble	Implicit vein system model	2009, 2017, 2019, 2024	5,206,900
Fowler	Implicit vein system model	2019, 2023	2,598,000
Mahler	Implicit vein model; indicator RBF interpolant	2009, 2017, 2019, 2020, 2021, 2022, 2023, 2024	25,915,000
Mud Pond	Implicit vein system model	2008, 2009, 2017, 2019, 2020, 2021, 2022, 2023, 2024	14,875,000
N2D	Implicit vein system model; indicator RBF interpolant	2019, 2021, 2022, 2023	22,420,000
New Fold	Implicit vein system model; indicator RBF interpolant	2009, 2017, 2020, 2021, 2022, 2023, 2024	15,392,000
Northeast Fowler	Implicit vein model	2017, 2019	6,852,600
Sylvia Lake	Implicit vein system model	2017, 2019, 2024	7,102,000
Turnpike	Indicator RBF interpolant	2019, 2021, 2022, 2023	65,041,000

Source: ESM 2024



Source: ESM 2024

Figure 14-4: Locations of each zone

14.1.5 Voids Model

Underground drifts are routinely surveyed with a Leica Total Station and irregular cavities such as stopes are LIDAR scanned with a Flyability ELIOS 3 drone. The survey data is compiled, validated, cleaned, and modeled in Deswik (most recently version 2023.1). Workings that intersect the resources are scanned and modeled in 3D. The 3D void model was 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 and utilized to deplete tonnage within the block models. This wireframe is shown in Figure 14-3.

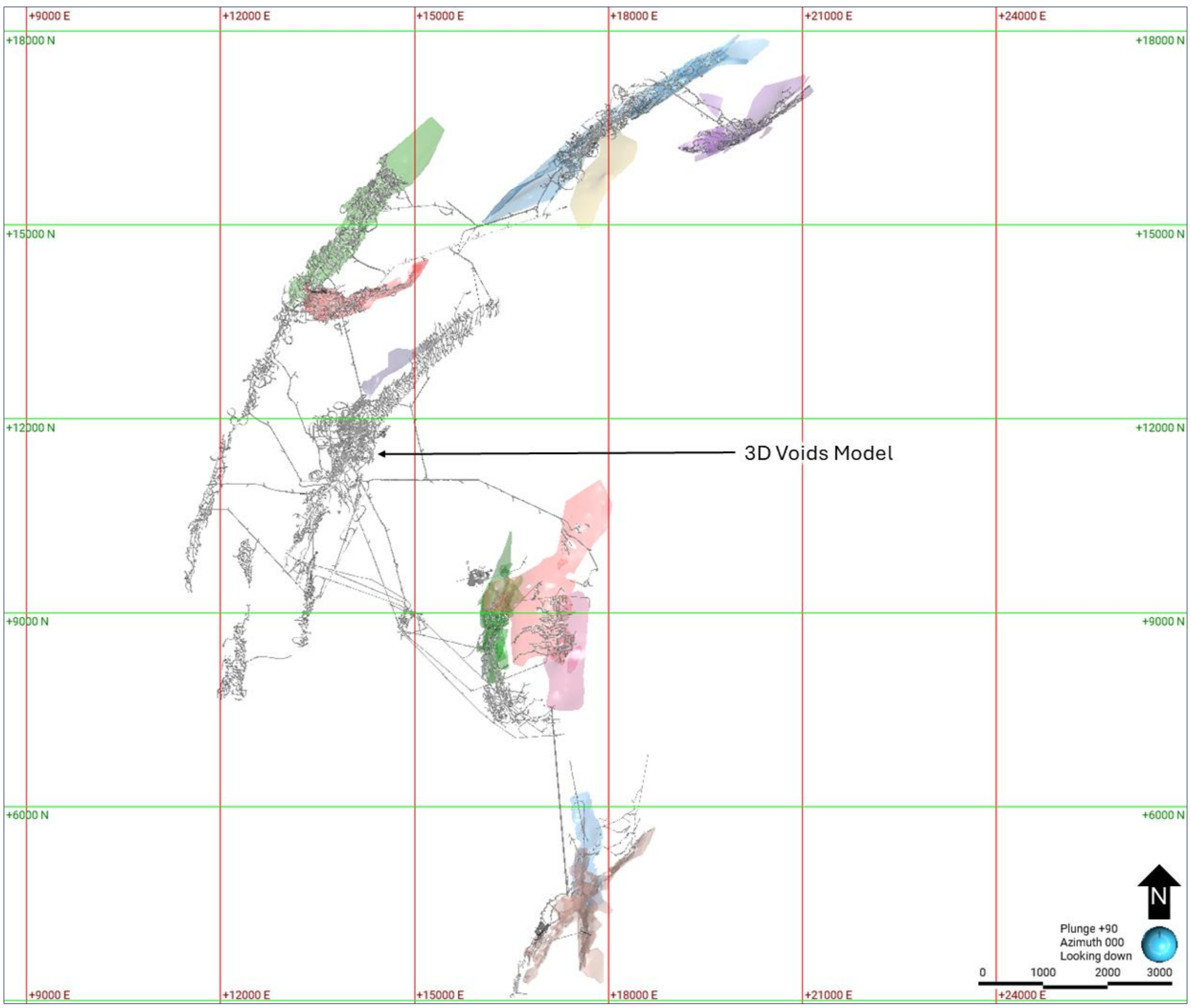


Figure 14-5: ESM 3D voids model

14.1.6 Exploratory Data Analysis

14.1.6.1 Assays

The ten zones are subdivided into 54 estimation domains that are included in the Mineral Resource. A total of 10,118 zinc (Zn%) samples were used for modeling purposes. Table 14-5 summarizes the basic zinc statistics for each domain. Historic site convention has been to assign zero to unsampled intervals.

Table 14-5: ESM assay summary statistics by domain

Zone	Domain	Count	Length (ft)	Mean (%)	SD	CV	Variance	Min	Max
American	American	85	488.0	9.22	7.0	0.8	48.9	0	23.8
Cal Marble	CM	29	145.5	12.14	5.9	0.5	35.0	0	25.0
Cal Marble	CM2	8	16.5	8.29	6.8	0.8	47.2	0	20.0
Fowler	XC1	23	232.5	7.57	4.8	0.6	23.0	0	15.9
Mahler - Upper	UMA	512	2,107	16.82	15.4	0.9	237.8	0	59.7
Mahler - Upper	HW Interpolant	227	830.0	4.80	7.0	1.5	49.2	0	41.8
Mahler - Lower	LMA	525	2,340.7	15.59	14.2	0.9	203.5	0	59.2
Mahler - Lower	MWD4	302	1,207.2	21.52	17.9	0.8	321.9	0	59.3
Mahler - Lower	MWD5	44	153.2	11.88	13.9	1.2	195.0	0	38.3
Mahler - Lower	MWD6	48	142.7	19.88	16.8	0.8	284.5	0	53.4
Mahler - Lower	FW Interpolant	128	357.2	3.77	6.9	1.8	47.9	0	47.7
Mahler - Lower	HW Interpolant	595	1,617.4	3.43	6.3	1.8	39.8	0	56.3
Mud Pond - Main	MPM	411	2,419.8	12.05	8.8	0.7	77.4	0	51.4
Mud Pond - Apron	MPA	326	1,623.7	12.72	11.0	0.9	122.3	0	52.7
Mud Pond - Apron	MPA2	10	25.9	13.07	12.2	0.9	150.7	0.49	54.3
Mud Pond - Apron	MPA3	8	24.0	7.78	5.7	0.7	33.2	2.95	21.4
N2D	UM14 HW1	382	1,479	12.80	8.7	0.7	76.7	0	37.3
N2D	UM14 HW2	8	58.0	13.08	14.8	1.1	220.1	0	42.0
N2D	UM14 FW1	125	341.9	7.44	5.7	0.8	33.1	0	25.7
N2D	UM14 FW2	38	248.8	6.81	5.7	0.8	33.0	0	20.1
N2D	UM14 FW3	11	37.6	8.82	5.5	0.6	30.2	1.63	18.5
N2D	UM14 FW4	16	78.2	9.93	4.6	0.5	21.8	0	18.6
N2D	UM14 FW5	8	33.3	8.55	5.9	0.7	34.9	0.88	19.3
N2D	UM11A vein	30	287.5	6.20	3.1	0.5	9.7	0	16.3
N2D	UM13 HW Anhy Zn Interpolant	191	677.4	5.96	7.8	1.3	61.1	0	42.9

Zone	Domain	Count	Length (ft)	Mean (%)	SD	CV	Variance	Min	Max
N2D	UM14 Serp Dol Zn Interpolant	2,314	8,480.5	3.48	4.7	1.4	22.6	0	39.3
New Fold	Vein 1	193	688.6	19.28	12.7	0.7	163.7	0	54.0
New Fold	Vein 2	134	530.7	14.79	11.1	0.7	122.8	0	54.0
New Fold	Vein 3	85	252.5	17.17	14.4	0.8	208.8	0.04	48.5
New Fold	Vein 4	13	38.9	13.87	8.9	0.6	79.9	5.15	34.5
New Fold	Vein 5	10	37.7	16.96	13.4	0.8	179.5	0.73	37.2
New Fold	Vein 6	5	17.7	13.43	5.7	0.4	33.1	8.05	23.3
New Fold	Vein 7	9	28.7	12.60	7.2	0.6	52.9	0	21.9
New Fold	Vein 8	19	62.4	16.63	16.6	1.0	263.5	0	44.5
New Fold	Vein 9	11	38.7	20.79	12.3	0.6	152.2	1.49	37.9
New Fold	Vein 10	20	79.5	11.33	9.3	0.8	87.6	0.11	44.2
New Fold	Vein 11	5	5.8	22.97	8.5	0.4	73.8	13.45	33.4
New Fold	Vein 12	7	18.7	16.14	8.9	0.6	79.2	5.18	37.5
New Fold	Interpolant	185	761.5	4.65	7.5	1.6	55.5	0	39.6
Northeast Fowler	Northeast Fowler	63	161.1	7.84	8.2	1.0	67.3	0	38.1
Sylvia Lake	SL	131	657.9	12.52	8.7	0.7	75.2	0	46.1
Sylvia Lake	SL LL	10	71.0	10.49	7.6	0.7	57.5	0.22	22.2
Turnpike	Hoist House FW	581	2,115.9	2.87	4.5	1.6	20.4	0	35.8
Turnpike	Hoist House HW	299	966.9	3.74	5.3	1.4	27.8	0	29.8
Turnpike	Pump House Lens A	271	980.7	3.47	5.3	1.5	27.9	0	26.2
Turnpike	Pump House Lens B	47	160.1	4.36	6.7	1.5	44.8	0	20.7
Turnpike	Pump House Vein 1	8	15.0	4.20	4.0	1.0	16.0	0.75	10.7
Turnpike	Pump House Vein 2	5	20.7	1.48	0.9	0.6	0.8	0.57	2.95
Turnpike	Pump House Vein 3	9	30.5	2.74	5.1	1.9	26.0	0.01	16.6
Turnpike	Streeter Lens A	106	273.1	3.79	6.0	1.6	35.8	0	27.7
Turnpike	Streeter Lens B	108	330.8	3.99	5.7	1.4	33.0	0	40.6
Turnpike	Streeter Lens C	27	85.9	4.54	5.9	1.3	35.3	0	22.4
Turnpike	Turnpike	1,099	5,977.0	4.26	5.0	1.2	25.0	0	24.9
Turnpike	West Ridge	254	810.8	6.25	7.6	1.2	57.6	0	40.8

Source: ESM 2024

14.1.6.2 Grade Capping

Assay capping was considered for each domain by analysis of histograms and log-probability plots. Additionally, higher-grade outlier samples were limited, when necessary, within grade estimation using the clamping method in Leapfrog Edge. The capping values are listed below in Table 14-6 and estimator high grade outlier threshold limits are listed in Table 14-9. The Threshold is the zinc (Zn) percent value limit, and the Distance is the distance as a percentage of the search ellipse size from the estimated block allowed for full unrestricted values. Beyond the distance specified, composite grades are still used but at the truncated Threshold value.

Table 14-6: ESM capping summary by domain

Zone	Domain	Capping Value (Zn %)	Quantity Capped	Uncapped Mean (Zn %)	Capped Mean (Zn %)
American	American	None	0	9.22	
Cal Marble	CM	23.4	2	12.14	12.12
Cal Marble	CM2	None	0	8.29	
Fowler	XC1	None	0	7.57	
Mahler - Upper	UMA	49.0	10	16.82	16.78
Mahler - Upper	HW Interpolant	26.0	7	4.80	4.66
Mahler - Lower	LMA	48.9	12	15.59	15.51
Mahler - Lower	MWD4	51.6	9	21.52	21.38
Mahler - Lower	MWD5	35.8	1	11.88	11.66
Mahler - Lower	MWD6	52.4	2	19.88	19.85
Mahler - Lower	FW Interpolant	26.2	4	3.77	3.58
Mahler - Lower	HW Interpolant	35.3	9	3.43	3.32
Mud Pond - Main	MPM	28.0	20	12.05	11.81
Mud Pond - Apron	MPA	38.3	12	12.72	12.55
Mud Pond - Apron	MPA2	None	0	13.07	
Mud Pond - Apron	MPA3	None	0	7.78	
N2D	UM14 HW1	30.0	15	12.80	12.65
N2D	UM14 HW2	22.0	3	13.08	10.33
N2D	UM14 FW1	22.0	3	7.44	7.39
N2D	UM14 FW2	17.0	4	6.81	6.78
N2D	UM14 FW3	12.0	2	8.82	7.78
N2D	UM14 FW4	17.0	2	9.93	9.87
N2D	UM14 FW5	17.0	2	8.55	8.24
N2D	UM11A vein	13.5	4	6.20	6.15

Zone	Domain	Capping Value (Zn %)	Quantity Capped	Uncapped Mean (Zn %)	Capped Mean (Zn %)
N2D	UM13 HW Anhy Zn Interpolant	22.0	14	5.96	5.53
N2D	UM14 Serp Dol Zn Interpolant	24.0	27	3.48	3.44
New Fold	Vein 1	44.5	5	19.28	19.22
New Fold	Vein 2	41.8	4	14.79	14.71
New Fold	Vein 3	44.5	2	17.17	17.12
New Fold	Vein 4	None	0	13.87	
New Fold	Vein 5	None	0	16.96	
New Fold	Vein 6	None	0	13.43	
New Fold	Vein 7	None	0	12.60	
New Fold	Vein 8	None	0	16.63	
New Fold	Vein 9	None	0	20.79	
New Fold	Vein 10	26.2	3	11.33	10.68
New Fold	Vein 11	None	0	22.97	
New Fold	Vein 12	None	0	16.14	
New Fold	Interpolant	11.0	23	4.65	3.37
Northeast Fowler	Northeast Fowler	None	0	7.84	
Sylvia Lake	SL	38.6	1	12.52	12.47
Sylvia Lake	SL LL	17.0	1	10.49	9.49
Turnpike	Hoist House FW	22.0	8	2.84	2.80
Turnpike	Hoist House HW	22.0	5	3.74	3.73
Turnpike	Pump House Lens A	24.0	2	3.47	3.45
Turnpike	Pump House Lens B	19.0	2	4.36	3.98
Turnpike	Pump House Vein 1	None	0	4.20	
Turnpike	Pump House Vein 2	None	0	1.48	
Turnpike	Pump House Vein 3	None	0	2.74	
Turnpike	Streeter Lens A	16.0	5	3.79	3.49
Turnpike	Streeter Lens B	17.0	8	3.99	3.84
Turnpike	Streeter Lens C	17.0	3	4.54	4.41
Turnpike	Turnpike	19.0	18	4.26	4.23
Turnpike	West Ridge	19.0	13	6.25	5.97

Source: ESM 2024

14.1.6.3 Compositing

In general, composites were generated using two different methodologies. For the discrete vein models, composites were created using vein length composites where a single composite is generated for each complete vein intersection with a drillhole. Composites were generated within the indicator RBF interpolant models as 10 ft run-length composites with residuals less than 5 ft added to the prior interval, honoring the modeled geological boundaries. Northeast Fowler is the exception which was estimated in 2019 before vein length compositing was the site standard for vein models. Compositing method and summary statistics are listed in Table 14-7.

Table 14-7: Compositing method by domain

Zone	Domain	Method	Composite Count	Un-capped, Composite Mean (Zn %)	Capped, Composite Mean (Zn%)
American	American	Vein Length	68	9.092	n/a
Cal Marble	CM	Vein Length	25	12.14	12.12
Cal Marble	CM2	Vein Length	4	8.29	n/a
Fowler	XC1	Vein Length	14	7.57	n/a
Mahler - Upper	UMA	Vein Length	183	16.71	16.66
Mahler - Upper	HW Interpolant	10 ft run length with residuals < 5 ft added to prior interval	90	4.8	4.66
Mahler - Lower	LMA	Vein Length	197	15.59	15.51
Mahler - Lower	MWD4	Vein Length	77	21.52	21.38
Mahler - Lower	MWD5	Vein Length	8	11.88	11.66
Mahler - Lower	MWD6	Vein Length	14	19.87	19.84
Mahler - Lower	FW Interpolant	10 ft run length with residuals < 5 ft added to prior interval	42	3.77	3.58
Mahler - Lower	HW Interpolant	10 ft run length with residuals < 5 ft added to prior interval	197	3.43	3.32
Mud Pond - Main	MPM	Vein Length	167	12.05	11.81
Mud Pond - Apron	MPA	Vein Length	120	12.72	12.55
Mud Pond - Apron	MPA2	Vein Length	6	13.07	n/a
Mud Pond - Apron	MPA3	Vein Length	3	7.78	n/a
N2D	UM14 HW1	Vein Length	136	12.71	12.56
N2D	UM14 HW2	Vein Length	4	13.08	10.33
N2D	UM14 FW1	Vein Length	70	7.44	7.39
N2D	UM14 FW2	Vein Length	21	6.81	6.78

Zone	Domain	Method	Composite Count	Un-capped, Composite Mean (Zn %)	Capped, Composite Mean (Zn%)
N2D	UM14 FW3	Vein Length	11	8.82	7.78
N2D	UM14 FW4	Vein Length	13	9.93	9.87
N2D	UM14 FW5	Vein Length	4	8.55	8.24
N2D	UM11A vein	Vein Length	23	6.2	6.15
N2D	UM13 HW Anhy Zn Interpolant	10 ft run length with residuals < 5 ft added to prior interval	86	5.61	5.21
N2D	UM14 Serp Dol Zn Interpolant	10 ft run length with residuals < 5 ft added to prior interval	882	3.43	3.39
New Fold	Vein 1	Vein Length	82	19.17	19.1
New Fold	Vein 2	Vein Length	52	14.82	14.74
New Fold	Vein 3	Vein Length	14	17.08	17.02
New Fold	Vein 4	Vein Length	8	13.27	n/a
New Fold	Vein 5	Vein Length	2	9.84	n/a
New Fold	Vein 6	Vein Length	2	9.4	n/a
New Fold	Vein 7	Vein Length	4	12.38	n/a
New Fold	Vein 8	Vein Length	11	13.42	n/a
New Fold	Vein 9	Vein Length	2	14.64	n/a
New Fold	Vein 10	Vein Length	13	11.33	10.68
New Fold	Vein 11	Vein Length	5	22.97	n/a
New Fold	Vein 12	Vein Length	5	15.4	n/a
New Fold	Interpolant	10 ft run length with residuals < 5 ft added to prior interval	102	4.4	3.19
Northeast Fowler	Northeast Fowler	5 ft run length with residuals distributed	38	8.05	n/a
Sylvia Lake	SL	Vein Length	96	12.52	12.47
Sylvia Lake	SL LL	Vein Length	6	10.49	9.49
Turnpike	Hoist House FW	10 ft run length with residuals < 5 ft added to prior interval	218	2.77	2.74
Turnpike	Hoist House HW	10 ft run length with residuals < 5 ft added to prior interval	103	3.51	3.49
Turnpike	Pump House Lens A	10 ft run length with residuals < 5 ft added to prior interval	99	3.44	3.42

Zone	Domain	Method	Composite Count	Un-capped, Composite Mean (Zn %)	Capped, Composite Mean (Zn%)
Turnpike	Pump House Lens B	10 ft run length with residuals < 5 ft added to prior interval	16	4.34	4.3
Turnpike	Pump House Vein 1	Vein Length	7	4.2	n/a
Turnpike	Pump House Vein 2	Vein Length	3	1.48	n/a
Turnpike	Pump House Vein 3	Vein Length	4	2.74	n/a
Turnpike	Streeter Lens A	10 ft run length with residuals < 5 ft added to prior interval	33	3.3	3.04
Turnpike	Streeter Lens B	10 ft run length with residuals < 5 ft added to prior interval	38	3.47	3.37
Turnpike	Streeter Lens C	10 ft run length with residuals < 5 ft added to prior interval	14	3.72	3.6
Turnpike	Turnpike	10 ft run length with residuals < 5 ft added to prior interval	643	4.08	4.05
Turnpike	West Ridge	10 ft run length with residuals < 5 ft added to prior interval	86	6.08	5.81

Source: ESM 2024

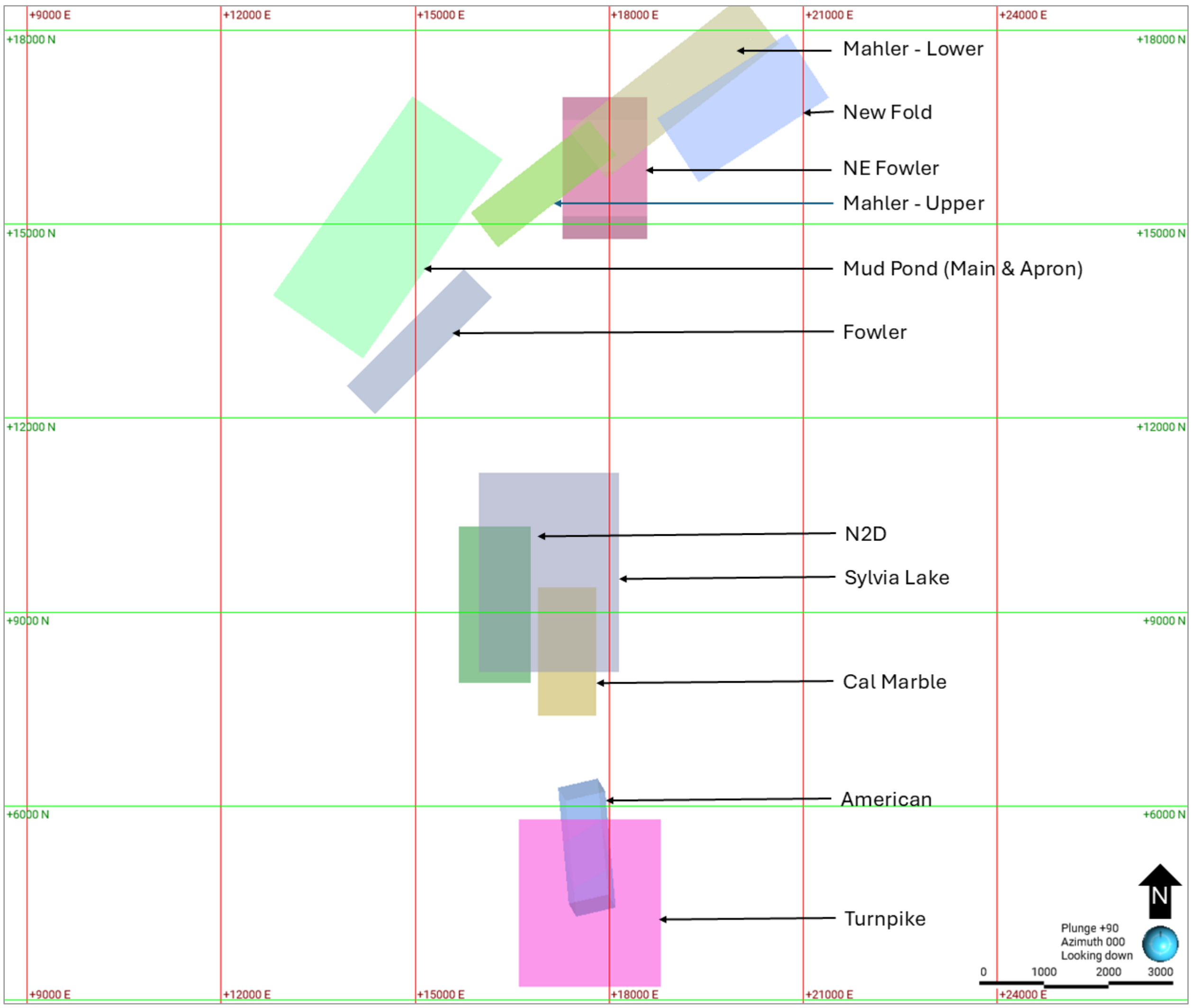
14.1.6.4 Variogram Analysis and Modeling

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

14.1.7 Resource Block Model

14.1.7.1 Parent Model

Separate block models were created for each zone. The parameters for each are listed in Table 14-8. They consist of origins, rotations (in Leapfrog rotation convention), parent block parameters and associated sub-block parameters. The American and Northeast Fowler block models were created in Vulcan and have parameters consistent with Vulcan conventions. A plan view of block model extents is shown in Figure 14-6 by zone.



Source: ESM 2024

Figure 14-6: Plan view of block model extents

Table 14-8: Block model size and location by zone

American					
Blocks	**X**	**Y**	**Z**	**Vulcan Rot.**	**Degrees**
Parent block size (ft):	20	20	20	Bearing:	77
Sub-block count:	2	2	8	Plunge:	12
Minimum size (ft):	10	10	2.5	Dip:	33.5
Base point (ft):	17,490	4,290	-335		
Boundary size (ft):	640	2,140	400		
Cal Marble					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	64	64	64	Azimuth:	0
Sub-block count:	32	32	32	Dip:	0
Minimum size (ft):	2	2	2	Pitch:	0
Base point (ft):	16,900	7,400	-820		
Boundary size (ft):	896	1,984	1,024		
Fowler					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	20	20	20	Azimuth:	45
Sub-block count:	8	8	8	Dip:	0
Minimum size (ft):	2.5	2.5	2.5	Pitch:	0
Base point (ft):	13,940	12,500	-2,230		
Boundary size (ft):	620	2,560	860		
Mahler - Lower					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	20	20	20	Azimuth:	52
Sub-block count:	8	8	8	Dip:	0
Minimum size (ft):	2.5	2.5	2.5	Pitch:	0
Base point (ft):	17,380	16,450	-2,840		
Boundary size (ft):	940	3,380	860		
Mahler - Upper					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	20	20	20	Azimuth:	52
Sub-block count:	8	8	8	Dip:	0
Minimum size (ft):	2.5	2.5	2.5	Pitch:	0
Base point (ft):	15,860	15,180	-1,880		
Boundary size (ft):	680	2,320	960		

Mud Pond (Main & Apron)					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	20	20	20	Azimuth:	35
Sub-block count:	8	8	8	Dip:	0
Minimum size (ft):	2.5	2.5	2.5	Pitch:	0
Base point (ft):	12,800	13,900	-1,760		
Boundary size (ft):	1,700	3,760	2,080		
N2D					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	20	20	20	Azimuth:	0
Sub-block count:	16	16	16	Dip:	0
Minimum size (ft):	1.25	1.25	1.25	Pitch:	0
Base point (ft):	15,680	7,900	-1,420		
Boundary size (ft):	1,100	2,420	820		
New Fold					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	20	20	20	Azimuth:	57
Sub-block count:	16	16	16	Dip:	0
Minimum size (ft):	1.25	1.25	1.25	Pitch:	0
Base point (ft):	18,740	16,640	-2,780		
Boundary size (ft):	1,180	2,400	880		
Northeast Fowler					
Blocks	**X**	**Y**	**Z**	**Vulcan Rot.**	**Degrees**
Parent block size (ft):	20	20	20	Bearing:	90
Sub-block count:	8	8	8	Plunge:	0
Minimum size (ft):	2.5	2.5	2.5	Dip:	45
Base point (ft):	17,285	14,775	-3,355		
Boundary size (ft):	1,300	2,600	500		
Sylvia Lake					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	64	64	64	Azimuth:	0
Sub-block count:	32	32	32	Dip:	0
Minimum size (ft):	2	2	2	Pitch:	0
Base point (ft):	15,980	8,080	-300		
Boundary size (ft):	2,176	3,072	1,408		

Turnpike					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	20	20	20	Azimuth:	0
Sub-block count:	8	8	8	Dip:	0
Minimum size (ft):	2.5	2.5	2.5	Pitch:	0
Base point (ft):	16,600	3,200	740		
Boundary size (ft):	2,200	2,600	1,360		

Source: ESM 2024

14.1.7.2 Estimate Parameters

Due to the high variability of the ESM deposits and the lack of robust variography, inverse distance squared (ID2) and cubed (ID3) estimates were used to estimate grade into parent blocks within the block model. Declustering was addressed and used for most of the ID estimates. The majority of the estimates were designed for a single pass, however, some domains necessitated multiple passes. ESM's experience with the deposits has determined that multipass estimations generate grade artifacts due to grade variability and sample clustering effects, and single pass estimates visually validate better when compared to the samples. 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. Sample selections for each domain are the result of an iterative validation process guided by first-hand experience with each deposit. 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 variable orientation algorithms either in Leapfrog called Variable Orientation (VO) or in Vulcan called Locally Varying Anisotropy (LVA). This oriented the search ellipse for each block down a plane which paralleled the modeled geologic continuity (i.e., the hanging wall or footwall of the ESM veins). The VO parameters were defined within the estimator based on the modeled vein surfaces. The American and Northeast Fowler domains were estimated in Vulcan in 2019 and the parameters listed in Table 14-9 are in Vulcan conventions.

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 in Table 14-9 for each domain.

Table 14-9: Estimation method, ellipse parameters, and outlier restrictions

Zone	Domain	Pass	Est.	Search Radius			Search Orientation (Leapfrog)			Sample Selection			Outlier Restriction		
				Maj.	Semi	Min.	Dip	Dip Azi	Pitch	Min	Max	Max / hole	Type	Threshold (Zn %)	Distance (% of search)
American	American	1	ID2	400	400	400	0	0	0	2	3	2	Exclude	19	12.5
Cal Marble	CM	1	ID3	500	500	500	0	0	0	2	9	1	None		
Cal Marble	CM2	1	ID2	600	200	200	64	93	26	3	4	1	Clamp	8.8	23
Fowler	XC1	1	ID2	1,000	500	250	Variable			3	5	1	None		
Mahler - Lower	LMA	1	ID2	700	300	300	Variable			2	5	1	Clamp	20	17
Mahler - Lower	MWD4	1	ID2	170	80	170	Variable			5	7	1	Clamp	28	20
Mahler - Lower	MWD4	2	ID2	700	350	350	Variable			2	7	1	Clamp	25	10
Mahler - Lower	MWD5	1	ID3	1,000	300	300	15	38	117	3	5	1	Clamp	8.5	17.5
Mahler - Lower	MWD6	1	ID2	800	800	200	90	150	10	2	7	1	Clamp	11.7	50
Mahler - Lower	FW Interpolant	1	ID2	450	150	150	19	14	134	5	15	2	Clamp	2.16	30
Mahler - Lower	HW Interpolant	1	ID2	250	250	125	Variable			5	15	2	None		
Mahler - Upper	UMA	1	ID2	150	75	75	Variable			5	9	1	None		
Mahler - Upper	UMA	2	ID3	500	250	250	Variable			2	5	1	Clamp	22.5	30
Mahler - Upper	HW Interpolant	1	ID3	300	150	150	Variable			3	10	2	None		
Mud Pond - Apron	MPA	1	ID2	300	150	150	Variable			3	7	1	Clamp	17	50
Mud Pond - Apron	MPA2	1	ID3	400	200	200	Variable			3	5	1	None		
Mud Pond - Apron	MPA3	1	ID2	400	200	200	27	80	55	3	3	1	Clamp	4.5	50
Mud Pond - Main	MPM	1	ID2	700	350	350	Variable			2	12	None	None		
N2D	UM14 HW1	1	ID2	350	175	175	Variable			2	7	1	None		
N2D	UM14 HW2	1	ID5	300	150	150	50	285	150	2	2	1	None		

Zone	Domain	Pass	Est.	Search Radius			Search Orientation (Leapfrog)			Sample Selection			Outlier Restriction		
				Maj.	Semi	Min.	Dip	Dip Azi	Pitch	Min	Max	Max / hole	Type	Threshold (Zn %)	Distance (% of search)
N2D	UM14 FW1	1	ID2	300	200	200	Variable			3	9	1	Clamp	8	20
N2D	UM14 FW2	1	ID3	400	400	200	85	50	25	2	5	1	Clamp	5	40
N2D	UM14 FW3	1	ID2	200	150	150	50	290	125	2	5	1	Clamp	5	45
N2D	UM14 FW4	1	ID3	300	300	300	0	0	0	3	7	1	Clamp	11	15
N2D	UM14 FW5	1	ID2	200	100	200	23	277.5	30	3	3	1	Clamp	7.5	23
N2D	UM11A vein	1	ID3	300	200	300	Variable			2	5	1	Clamp	6	30
N2D	UM13 HW Anhy Zn Interpolant	1	ID2	400	300	300	Variable			3	5	NA	None		
N2D	UM14 Serp Dol Zn Interpolant	1	ID2	60	60	30	Variable			9	18	2	None		
N2D	UM14 Serp Dol Zn Interpolant	2	ID2	120	120	60	Variable			9	18	2	Clamp	4.5	25
N2D	UM14 Serp Dol Zn Interpolant	3	ID2	300	300	150	Variable			4	18	2	Clamp	4.5	10
New Fold	Vein 1	1	ID2	300	210	80	Variable			7	7	1	Clamp	23.6	50
New Fold	Vein 1	2	ID2	900	450	450	Variable			2	12	1	Clamp	23.6	20
New Fold	Vein 2	1	ID2	650	650	325	Variable			2	5	1	None		
New Fold	Vein 3	1	ID2	500	500	500	0	0	0	5	15	1	Clamp	17	20
New Fold	Vein 4	1	ID2	300	300	300	0	0	0	3	5		Clamp	13.5	25
New Fold	Vein 5	1	ID2	300	300	300	0	0	0	2	2	1	Clamp	6	50
New Fold	Vein 6	1	ID2	300	300	300	0	0	0	2	2		None		
New Fold	Vein 7	1	ID3	300	200	200	60	320	30	3	3	1	None		
New Fold	Vein 8	1	ID2	300	200	200	54	330	155	3	5	1	None		
New Fold	Vein 9	1	ID2	300	300	300	0	0	0	2	2	1	None		
New Fold	Vein 10	1	ID3	300	200	200	Variable			3	5	1	None		
New Fold	Vein 11	1	ID2	100	100	100	0	0	0	3	5		None		

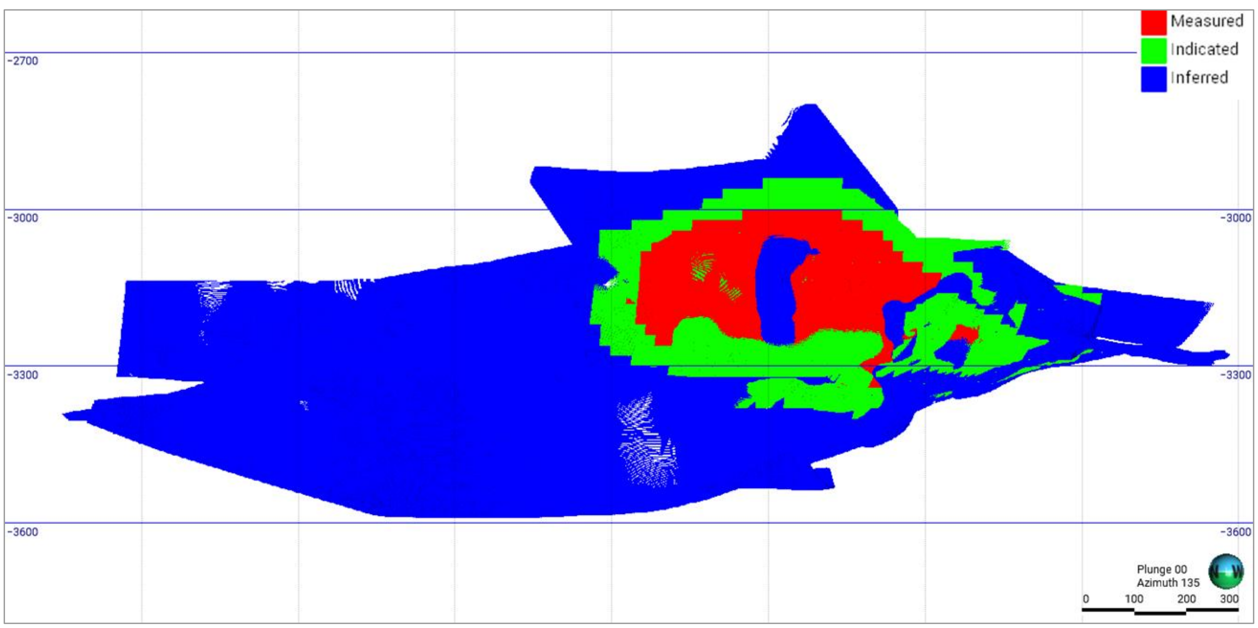
Zone	Domain	Pass	Est.	Search Radius			Search Orientation (Leapfrog)			Sample Selection			Outlier Restriction		
				Maj.	Semi	Min.	Dip	Dip Azi	Pitch	Min	Max	Max / hole	Type	Threshold (Zn %)	Distance (% of search)
New Fold	Vein 12	1	ID2	600	300	300	0	104	0	3	3	1	None		
New Fold	Interpolant	1	ID2	300	300	100	Variable			4	15	3	None		
Northeast Fowler	Northeast Fowler	1	ID2	425	425	425	Variable			2	30	2	Exclude	30	~12
Sylvia Lake	SL	1	ID3	500	375	375	Variable			3	20	1	Clamp	20	50
Sylvia Lake	SL LL	1	ID2	200	200	200	0	0	90	3	6	1	None		
Turnpike	Hoist House FW	1	ID2	300	300	150	Variable			4	15	3	NA	NA	NA
Turnpike	Hoist House HW	1	ID2	250	250	125	Variable			4	15	3	Clamp	5	75
Turnpike	Pump House Lens A	1	ID2	200	200	75	Variable			4	15	3	NA	NA	NA
Turnpike	Pump House Lens B	1	ID2	250	125	75	Variable			3	7	2	Clamp	3.3	20
Turnpike	Pump House Vein 1	1	ID2	200	200	100	52	292	107	3	3	1	NA	NA	NA
Turnpike	Pump House Vein 2	1	ID2	200	200	100	57	270	109	3	3	1	NA	NA	NA
Turnpike	Pump House Vein 3	1	ID2	200	200	200	0	0	0	3	3	1	Clamp	1	20
Turnpike	Streeter Lens A	1	ID2	300	300	150	Variable			4	15	3	Clamp	2.58	25
Turnpike	Streeter Lens B	1	ID2	300	225	100	50	315	9	4	15	3	Clamp	2.76	45
Turnpike	Streeter Lens C	1	ID2	250	175	50	63	316	9	4	10	3	NA	NA	NA
Turnpike	Turnpike	1	ID2	450	300	150	Variable			4	15	3	NA	NA	NA
Turnpike	West Ridge	1	ID2	200	200	100	58	305	165	4	10	3	Clamp	4.85	30

Source: ESM 2024

14.1.8 Resource Classification

The ESM 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 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.

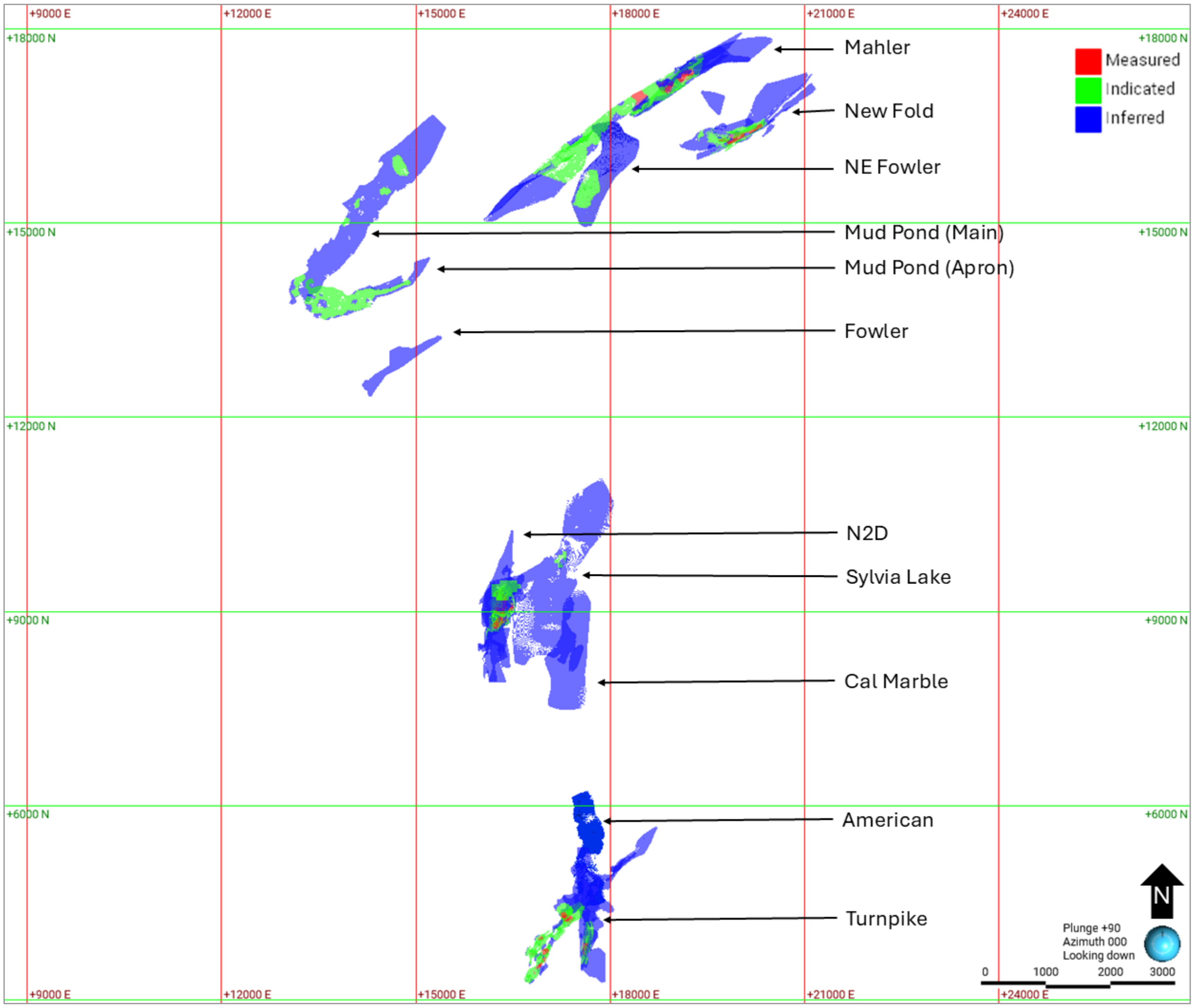
These model items were used as the basis of a calculation within the blocks. The scripted values were used as a guide to assign zones of confidence. The results of the calculation were then smoothed and encased in wireframes that facilitated the final model coding for classification. This allowed for removal of 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. In addition to estimation metadata, the ESM technical staff incorporate experience regarding geological continuity, mapping, and drilling data prior to assigning classification zones. An example vein is shown in Figure 14-7 and classification for all veins is demonstrated in Figure 14-8.



Source: ESM 2024

Note: Red=Measured, green=Indicated, blue=Inferred.

Figure 14-7: Classification for New Fold, view looking SE (Az 135)



Source: ESM 2024

Note: Red=Measured, green=Indicated, blue=Inferred.

Figure 14-8: Classification for all ESM zones

The zones that were classified as Measured exhibit excellent geological continuity that has been verified at dense sample spacing using reliable testing methods. Generally, these blocks were informed by a minimum of five drillholes at a spacing less than 75 ft and satisfy data quality and quantity requirements. 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. Generally, these blocks were informed by a minimum of five drillholes at a spacing less than 150 ft. 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.

No environmental, permitting, legal, title, taxation, socio-economic, marketing, or other relevant issues that may affect the estimate of Mineral Resources are known to the QP. Mineral Reserves can be estimated only on the basis of an economic evaluation that is used in a preliminary Feasibility Study or a Feasibility Study of a mineral project; thus, no reserves have been estimated. As per NI 43-101, Mineral Resources that are not Mineral Reserves do not have to demonstrate economic viability.

14.1.9 Mineral Resource Tabulation

14.1.9.1 Underground Mineral Resource

The underground Mineral Resource reported is effective as of July 16, 2024, and have been compiled from ten separate block models including the N2D, American, Cal Marble, Fowler, Mahler – Lower, Mahler - Upper, Mud Pond, Northeast Fowler, New Fold, and Silvia Lake.

The underground Mineral Resource reported has been tabulated at a COG of 5.3%. The COG was determined with a net smelter return (NSR) calculation that used mine actuals for inputs. Donald Taylor considers the mineralized envelopes as modeled to have more than sufficient continuity and grade to be considered potentially mineable given the long site history (95 years) of successful planning and profitably mining the Balmat massive sulfides.

ESM Underground and Open Pit Mineral Resources have been modeled (Leapfrog Geo™ version 2023.2.3) and estimated (Leapfrog Edge) by ESM geologists. Don Taylor is the QP who has reviewed the geological models and estimates and has conducted multiple site inspections.

Table 14-10: Underground Mineral Resource Estimate as of July 16, 2024

Classification	Tons (000's US short tons)	Zn (%)	Contained Pounds (M lb)
Measured	295	17.1	101
Indicated	1,158	15.7	364
Measured + Indicated	1,453	16.0	465
Inferred	4,327	12.1	1,049

Source: ESM 2024

Notes:

1. The qualified person for the 2024 MRE, as defined by the NI 43-101 guidelines, is Donald (Don) R. Taylor, of Titan Mining Corp., SME registered member (#4029597).
2. 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.
3. Three-dimensional (3D) wireframe models of mineralization were prepared in Leapfrog Geo based on the geological interpretation of the logged lithology on contiguous grade intervals defining mineralized sub-domains. The 2024 underground MRE encompasses 36 vein domains and six indicator RBF interpolant shells totaling 42 individual wireframes.
4. Geological and block models for the underground MRE used data from a total of 1,100 surface and underground diamond drillholes (core). The drillhole database was validated prior to resource estimation and QA/QC checks were made using industry-standard control charts for blanks and commercial certified reference material inserted into assay batches by Empire State Mines personnel.
5. High-grade capping was evaluated and implemented on the raw assay data on a per-zone basis using histograms and log-probability plots. Outliers were further evaluated during estimation and limited if necessary using the Leapfrog Edge clamping method.
6. The MRE was compiled from 10 individual block models that were prepared using Leapfrog Edge. Block models were sub-blocked at domain boundaries and samples were composited using vein length intervals where a single composite is generated for each complete vein intersection with a drillhole. Composites were generated within the indicator RBF interpolant models as 10-ft run-length composites with residuals less than 5 ft added to the prior interval, honoring the modeled geological boundaries. Grade estimation was carried out using IDW methods coupled with variably orientated search ellipses derived from modeled vein surfaces.
7. The SG assessment was carried out for all domains using measurements collected during the core logging process. Where there is sufficient sampling, the SG is interpolated into model blocks using IDW techniques. If insufficient sampling exists, then density was assigned to models based on calculated means or by a regression formula.
8. Resources are reported using a 5.3% Zinc cut-off grade, based on actual break-even mining, processing, G&A costs, and smelter terms from the ESM operation at a zinc recovery of 96.4%.
9. Resources stated as in-situ grade at a Zinc price of $1.30/lb.
10. 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. Classification parameters vary slightly depending on the nature and continuity of the individual zones. Block classification was explicitly domained based on a calculation that used quality, quantity, and distance parameters.
11. Quantities and grades in the MRE are rounded to an appropriate number of significant figures to reflect that they are estimations.
12. The Mineral Resource Estimate was prepared following the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019).
13. CIM definitions and guidelines for Mineral Resource Estimates have been followed.
14. The QP is unaware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issues that could materially affect this MRE.

14.1.9.2 Open Pit Mineral Resource

Turnpike is amenable to open pit and underground mining, however, while underground studies are ongoing the deposit presently only contributes to the Open Pit Mineral Resource.

The Open Pit Mineral Resource reported is effective as of October 17, 2024, and has been tabulated at a pit-constrained COG of 0.6%. Table 14-11 summarizes the parameters used to develop the constraining pit to determine a reasonable prospect for eventual economic extraction (RPEEE). ESM considers the open pit to be an accretive project with no G&A costs, and selling costs are incorporated into the selling price.

Table 14-11: Turnpike pit constraint parameters

Input	Unit	Variable
Mining		
Mining Cost - Ore	US$/t mined	4.6
Mining Cost - Waste	US$/t mined	3.5
Mining Cost - Overburden	US$/t mined	2
Processing		
Processing Cost	US$/t milled	11
G&A Cost	US$/t milled	-
Processing Recovery	%	96
Concentrate Grade	%	58
Other		
Selling Price	US$/t concentrate	1.27
Transportation Cost	US$/t concentrate	50
Payable Zinc	%	85
COG	Zn (%)	0.6
Overall slope angle	degrees	26-48
Discount Factor	%	10

Source: ESM 2024

The pit-constrained Mineral Resource and in situ metal for Turnpike is summarized in Table 14-12.

Table 14-12: Open Pit Mineral Resource Estimate as of October 17, 2024

Classification	Tons (000's US short tons)	Zn (%)	Contained Pounds (M lb)
Measured	251	3.1	16
Indicated	950	3.2	61
Measured + Indicated	1,201	3.2	77
Inferred	461	3.5	32

Source: ESM 2024

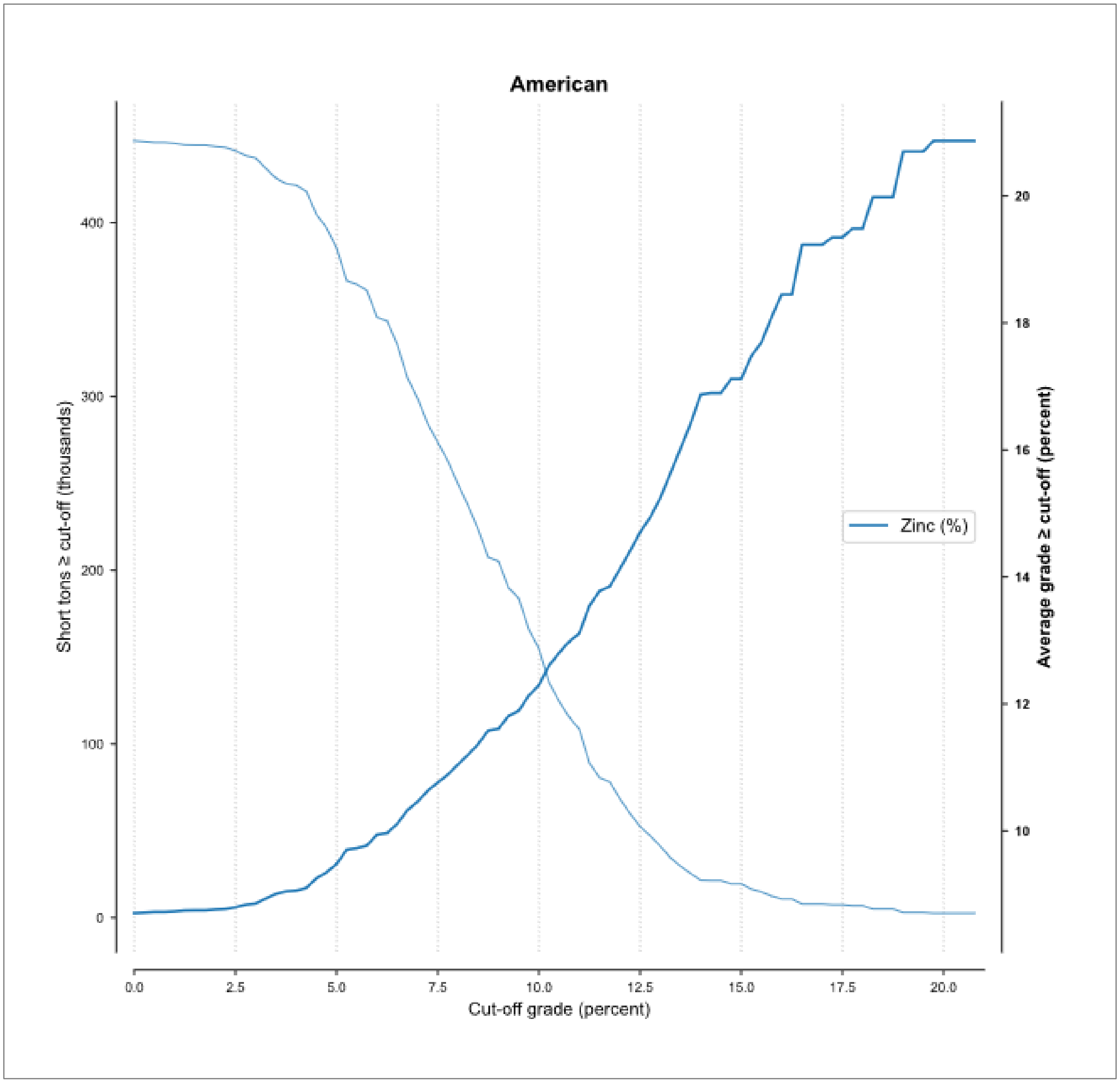
Notes:

1. The qualified person for the 2024 MRE, as defined by the NI 43-101 guidelines, is Donald (Don) R. Taylor, of Titan Mining Corp., SME registered member (#4029597).
2. 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.
3. Three-dimensional (3D) wireframe models of mineralization were prepared in Leapfrog Geo based on the geological interpretation of the logged lithology on contiguous grade intervals defining mineralized sub-domains. The 2024 Open Pit MRE encompasses three vein domains and nine indicator RBF interpolant shells totaling 12 individual wireframes.
4. Geological and block models for the Open Pit MRE used data from a total of 254 surface and underground diamond drillholes (core). The drillhole database was validated prior to resource estimation and QA/QC checks were made using industry-standard control charts for blanks and commercial certified reference material inserted into assay batches by Empire State Mines personnel.
5. High-grade capping was evaluated and implemented on the raw assay data on a per-zone basis using histograms and log-probability plots. Outliers were further evaluated during estimation and limited if necessary using the Leapfrog Edge clamping method.
6. The Open Pit MRE was compiled from a single block model that was prepared using Leapfrog Edge. The block model was sub-blocked at domain boundaries and samples were composited within the indicator RBF interpolant models as 10-ft run-length composites with residuals less than 5 ft added to the prior interval, honoring the modeled geological boundaries. Assays were composited within the vein models using vein length intervals where a single composite is generated for each complete vein intersection with a drillhole. Grade estimation was carried out using IDW methods coupled with variably orientated search ellipses derived from modeled trend surfaces.
7. The SG assessment was carried out for all domains using measurements collected during the core logging process. Where there is sufficient sampling, the SG is interpolated into model blocks using IDW techniques. If insufficient sampling exists, then density was assigned to models based on calculated means or by a regression formula.
8. Resources stated as internal to an optimized pit shell, above a cut-off grade of 0.6% Zn.
9. Cut-off is based on break-even economics at a Zinc price of $1.27/lb, with an assumed zinc recovery of 96%, and actual processing, mining, and transportation costs from the ESM operation. No G&A costs were applied as ESM considers the project accretive. No extra mining dilution was added as a regularized block model was used.
10. 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. Classification parameters vary slightly depending on the nature and continuity of the individual zones. Block classification was explicitly domained based on a calculation that used quality, quantity, and distance parameters.
11. Quantities and grades in the MRE are rounded to an appropriate number of significant figures to reflect that they are estimations.
12. The Mineral Resource Estimate was prepared following the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019).
13. CIM definitions and guidelines for Mineral Resource Estimates have been followed.
14. The QP is unaware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issues that could materially affect this MRE.

14.1.9.3 Mineral Resource Sensitivity

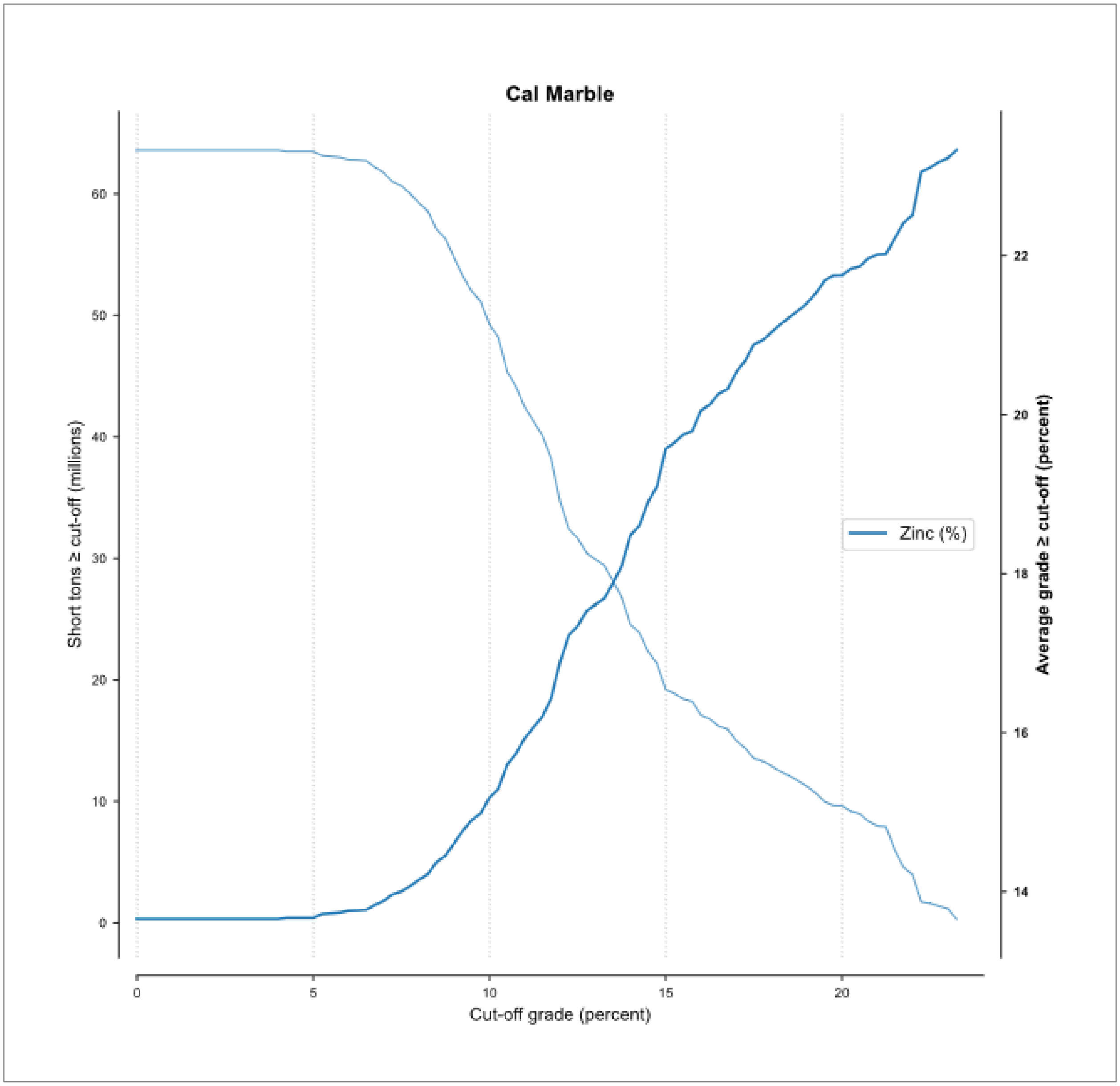
ESM Underground

To document the sensitivity of the Mineral Resources to a variety of factors, ESM 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 modeling or additional drilling factors) such as recovery, costs, pricing, etc. The graphs represent a range of tonnages and grades and are not intended to be construed as mineral resources. These graphs are shown for each area in Figure 14-9 through Figure 14-20. Due to the variances in grade and mineralization within each area, sensitivities to COG differ for each.



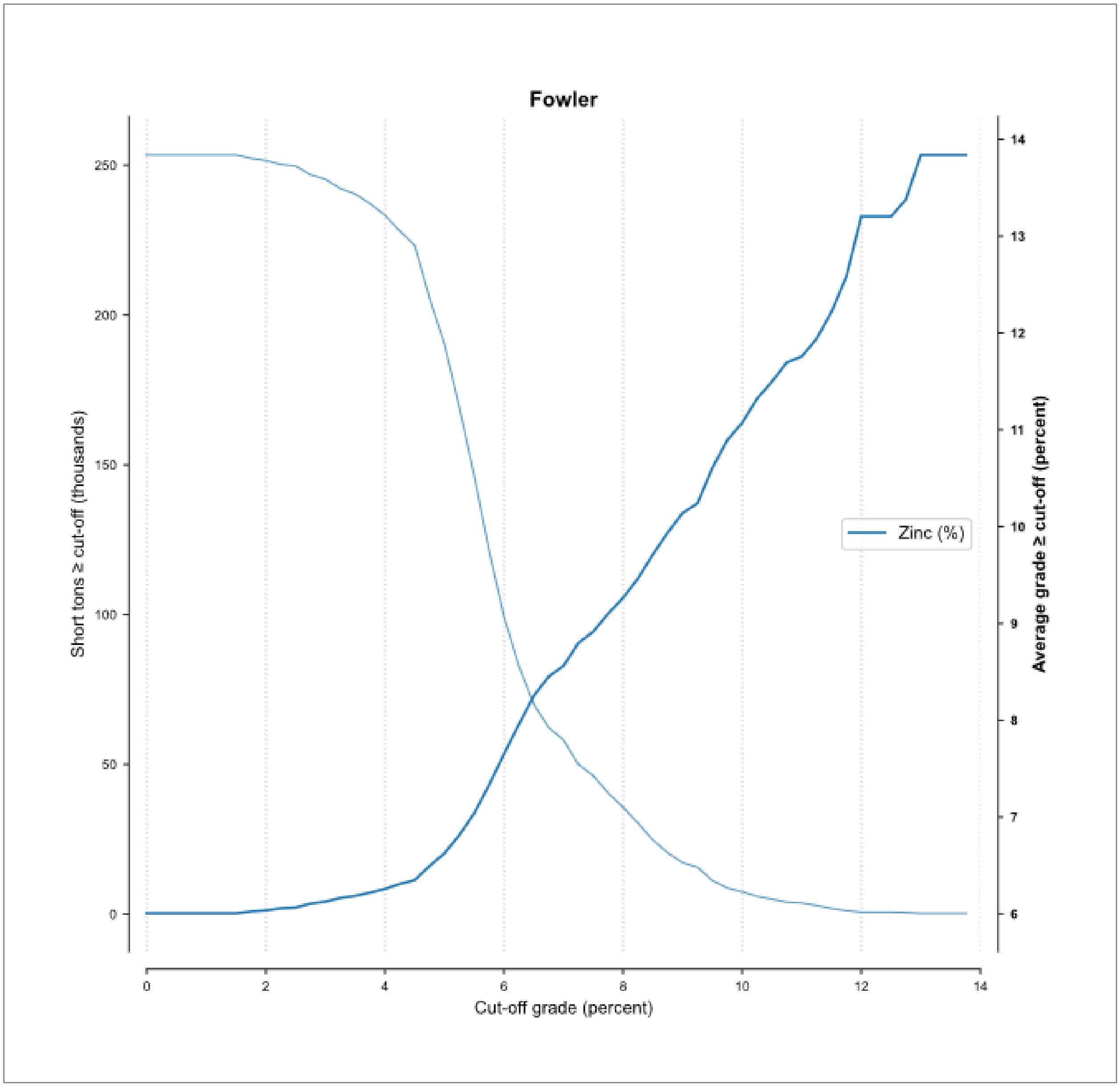
Source: ESM 2024

Figure 14-9: American grade tonnage graph



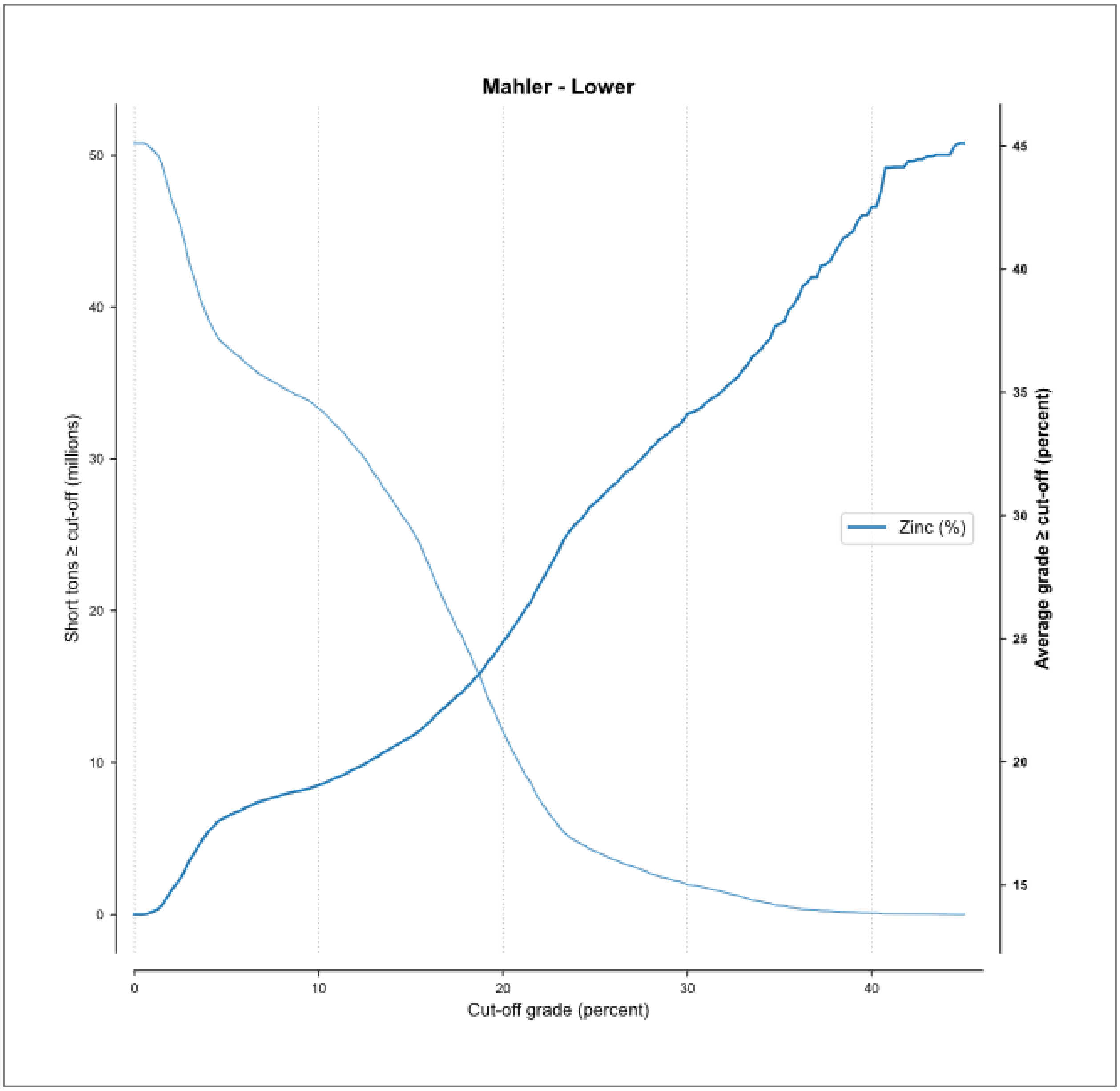
Source: ESM 2024

Figure 14-10: Cal Marble grade tonnage graph



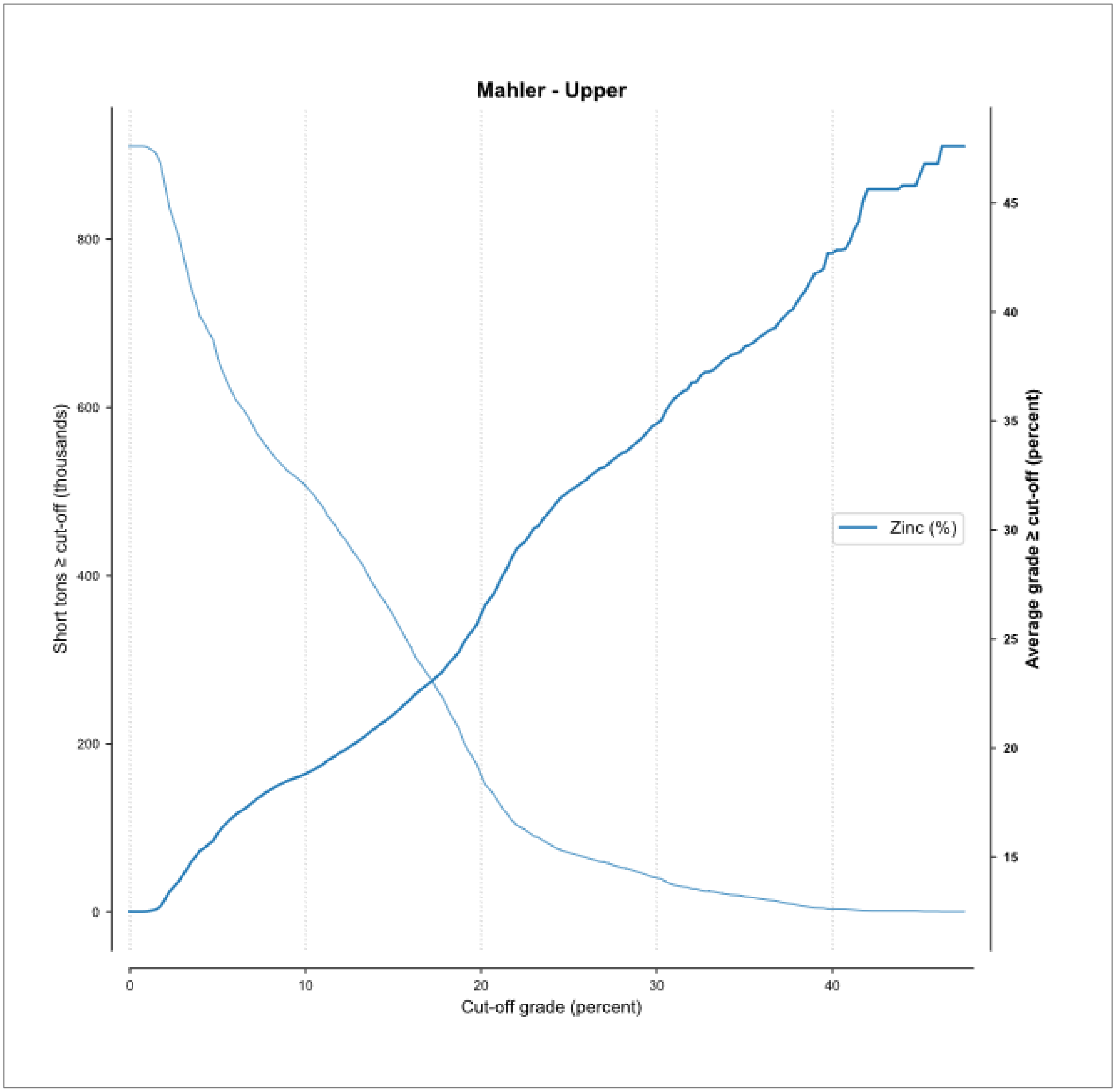
Source: ESM 2024

Figure 14-11: Fowler grade tonnage graph



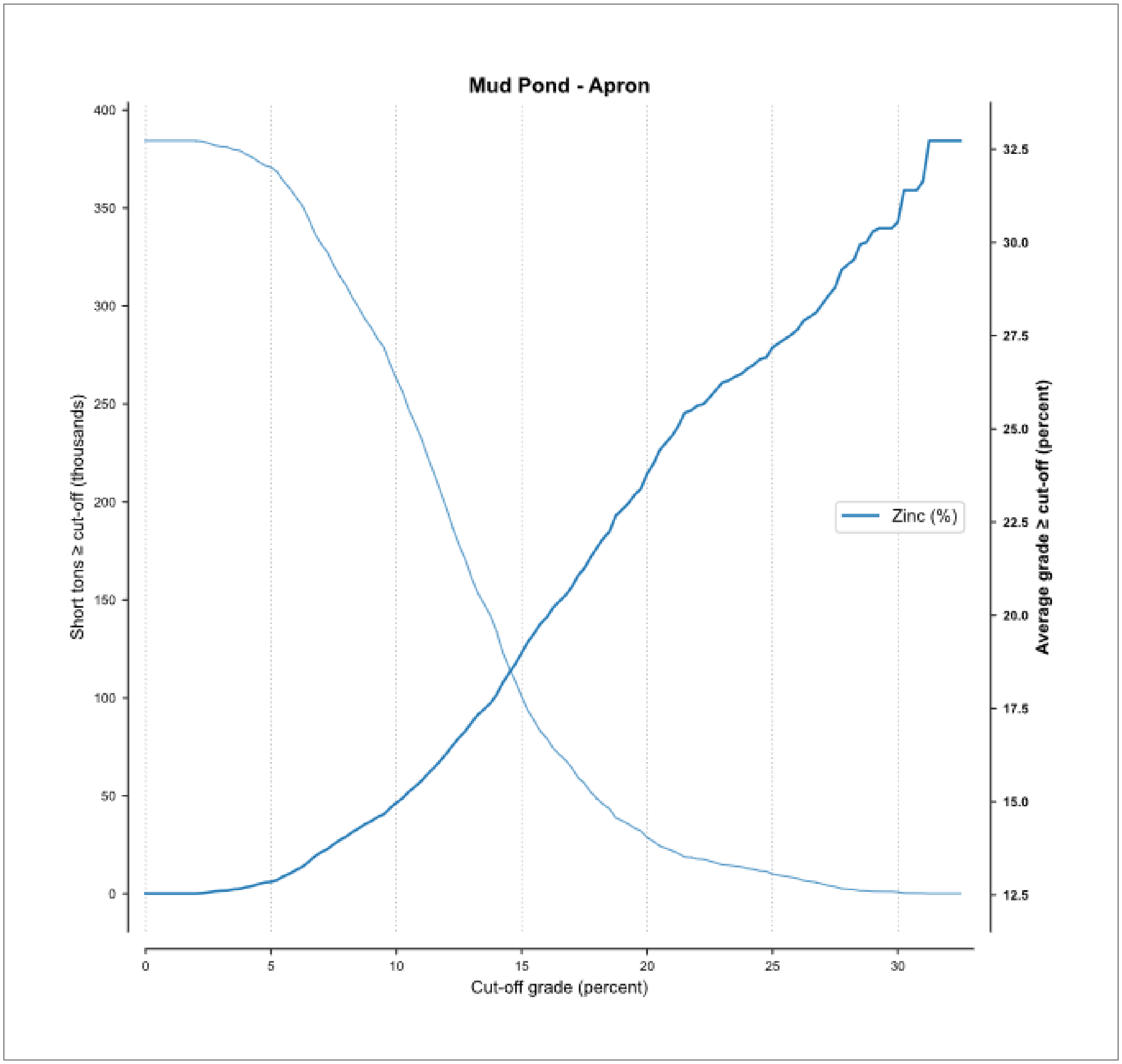
Source: ESM 2024

Figure 14-12: Lower Mahler grade tonnage graph



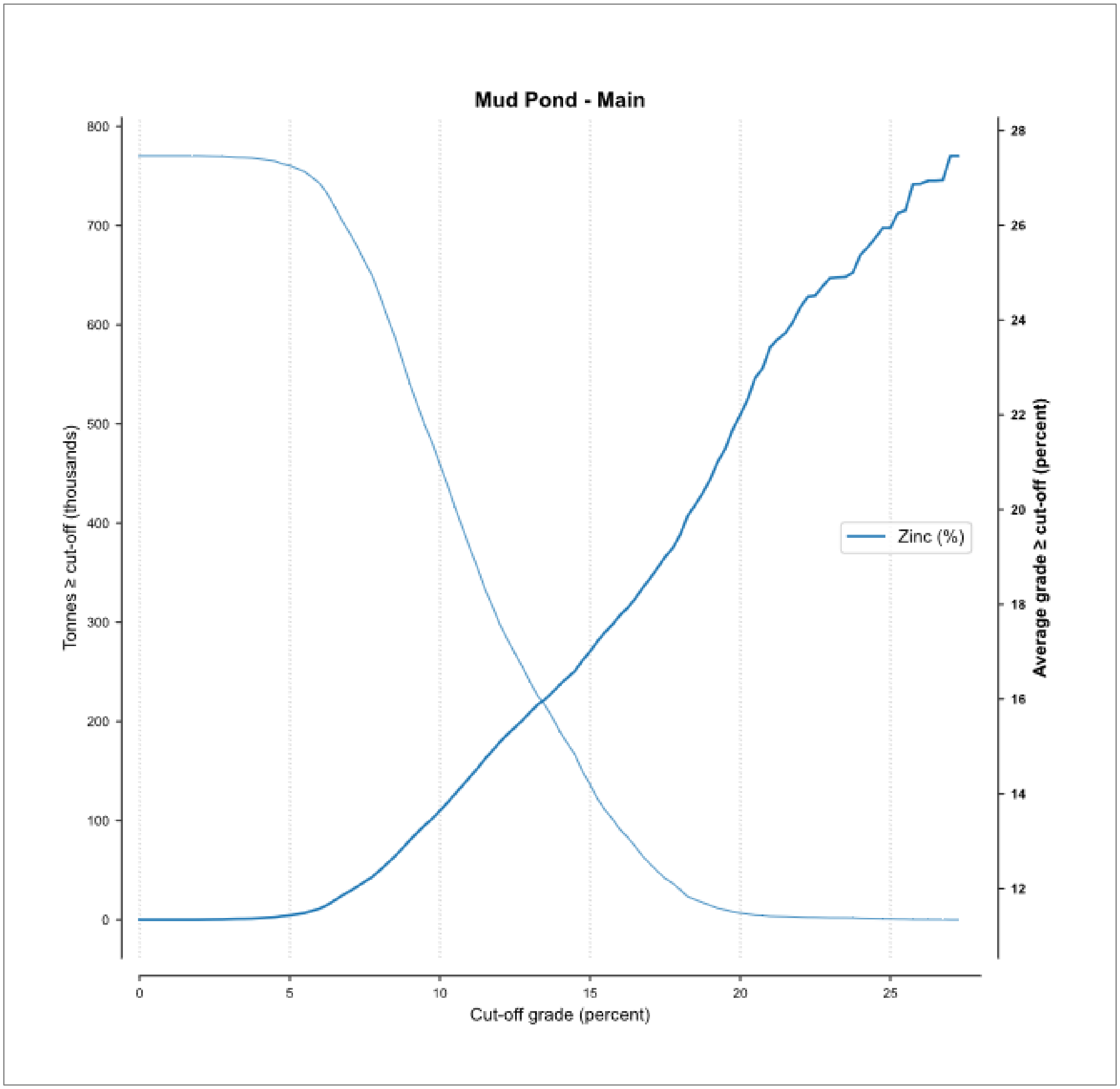
Source: ESM 2024

Figure 14-13: Upper Mahler grade tonnage graph



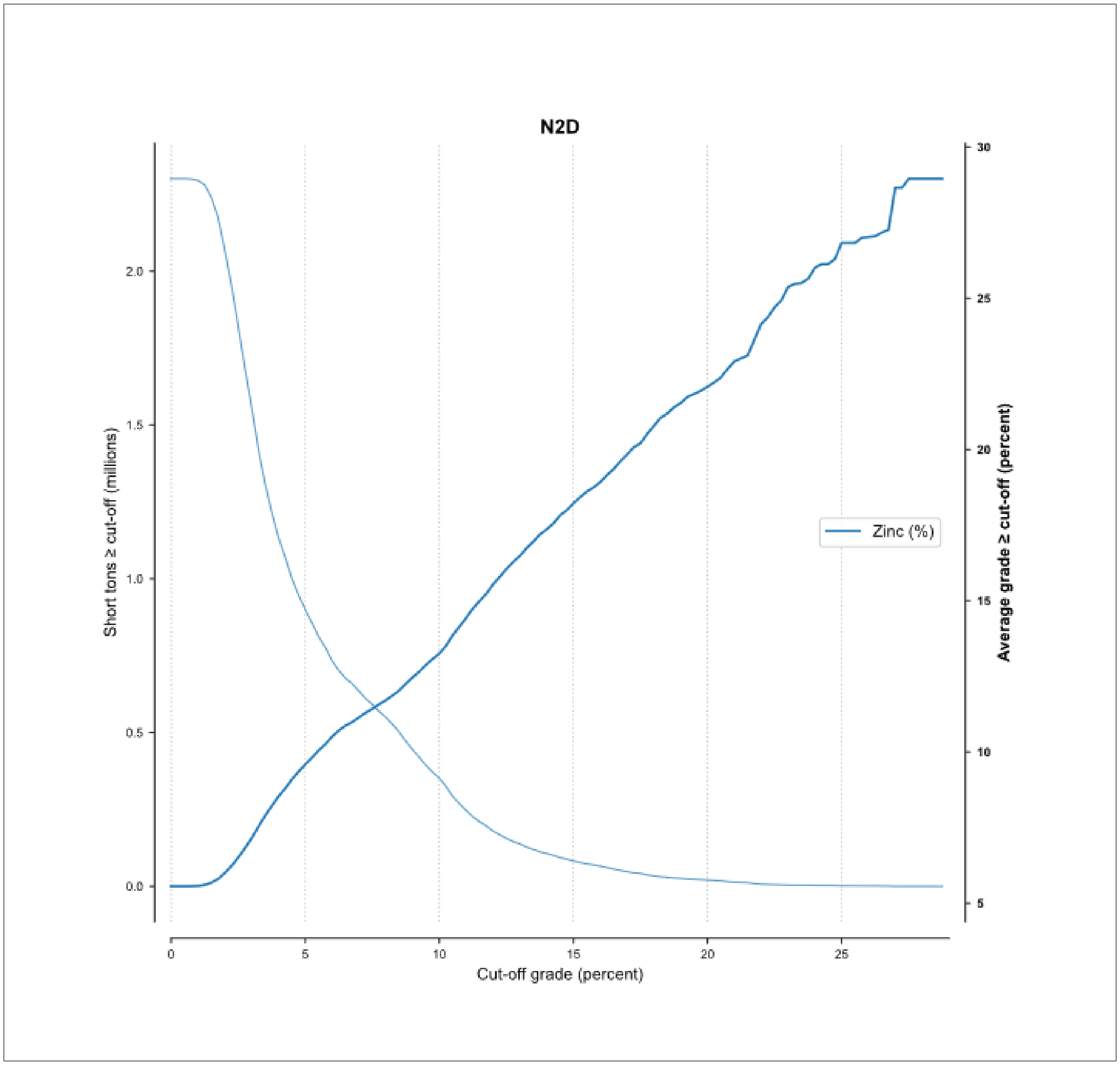
Source: ESM 2024

Figure 14-14: Mud Pond Apron grade tonnage graph



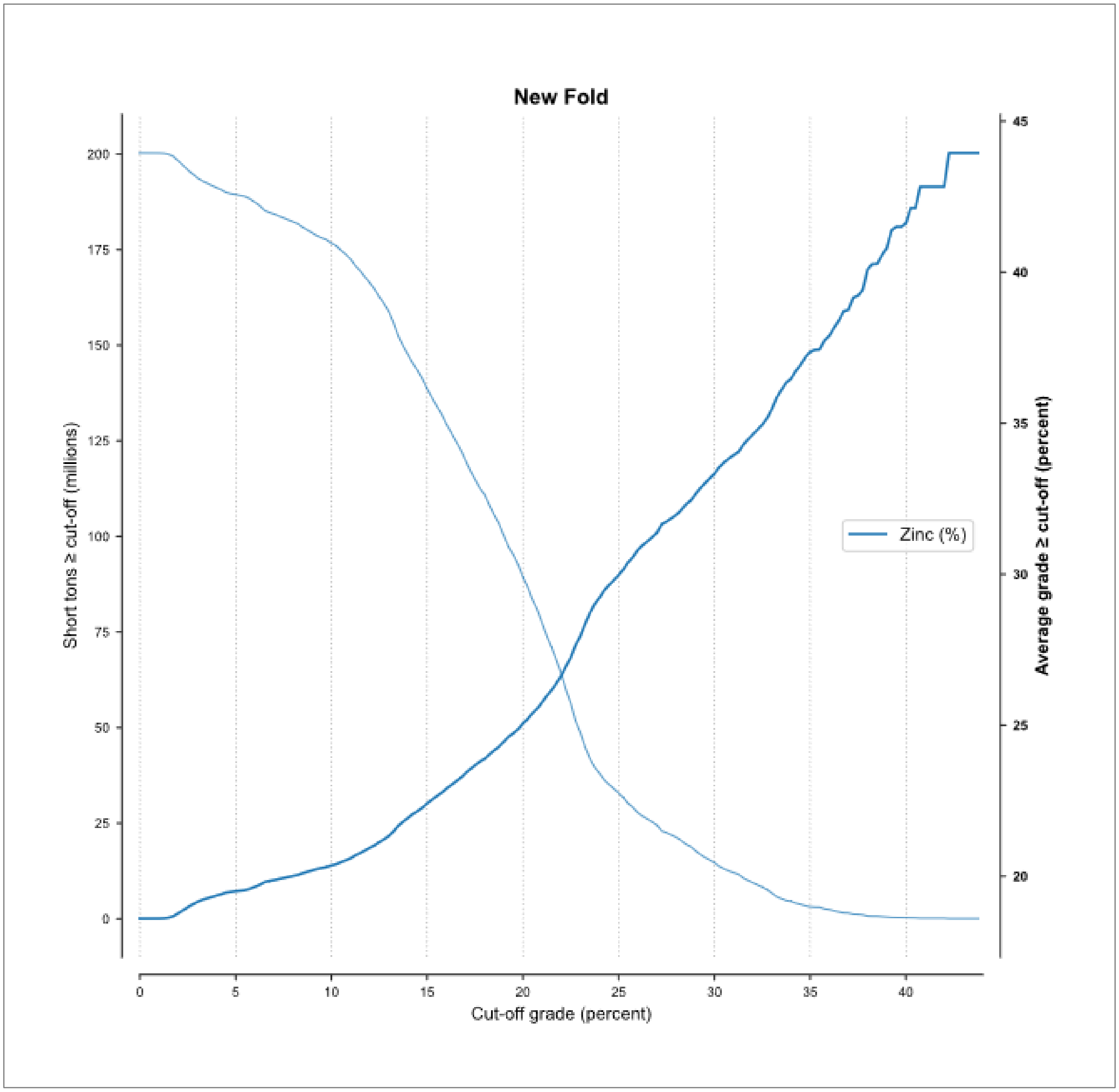
Source: ESM 2024

Figure 14-15: Mud Pond - Main grade tonnage graph



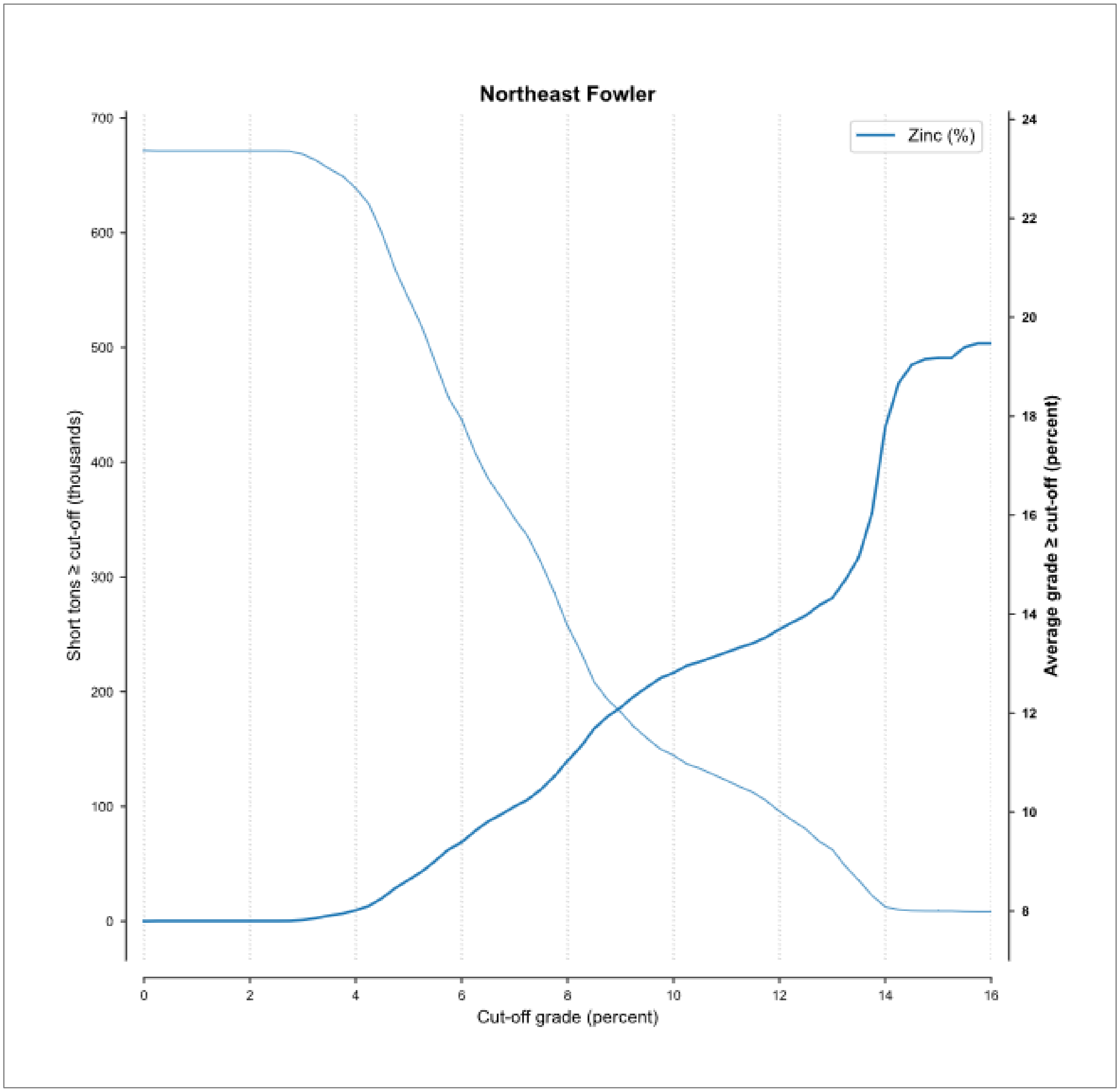
Source: ESM 2024

Figure 14-16: N2D grade tonnage graph



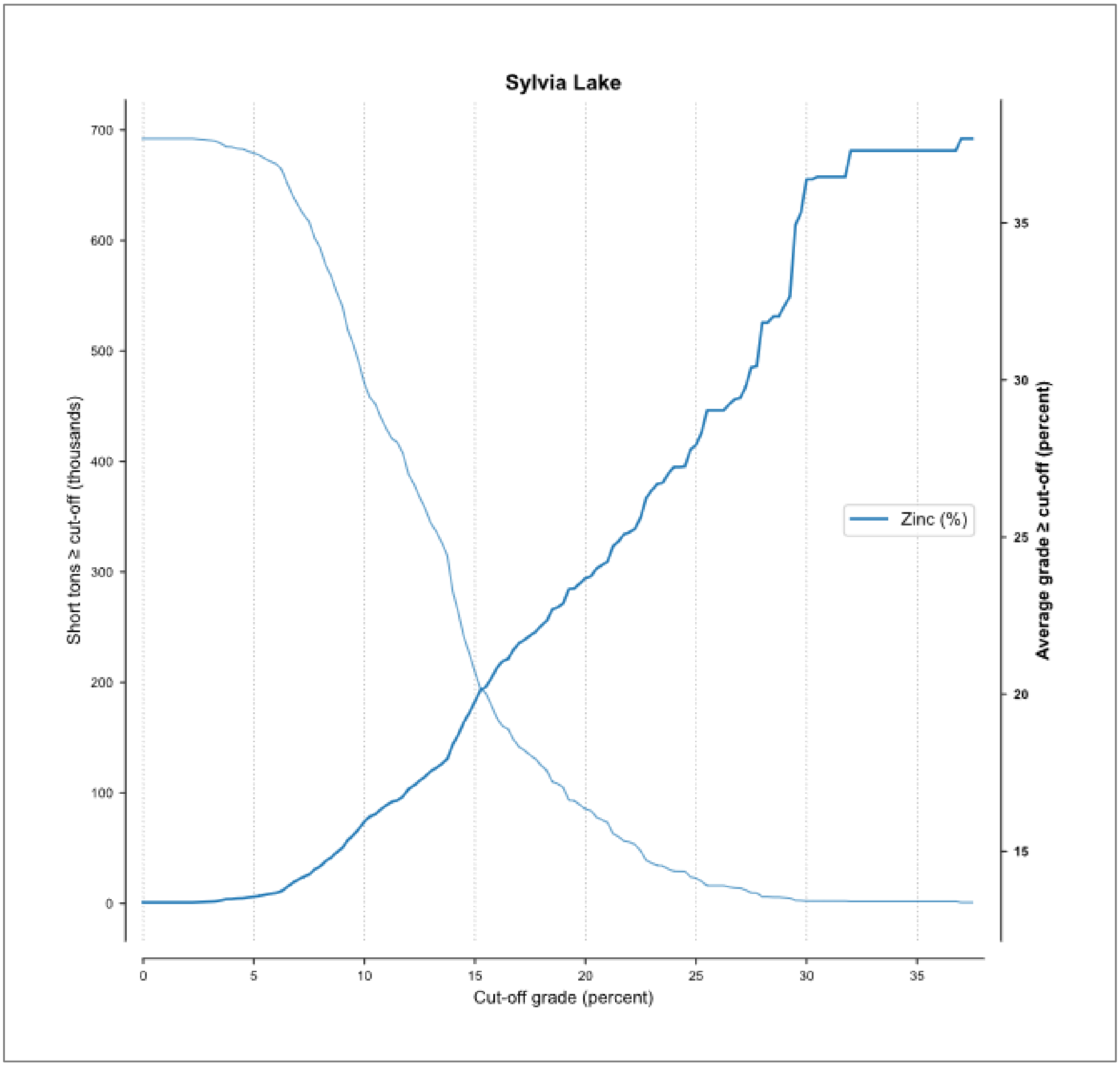
Source: ESM 2024

Figure 14-17: New Fold grade tonnage graph



Source: ESM 2024

Figure 14-18: Northeast Fowler grade tonnage graph

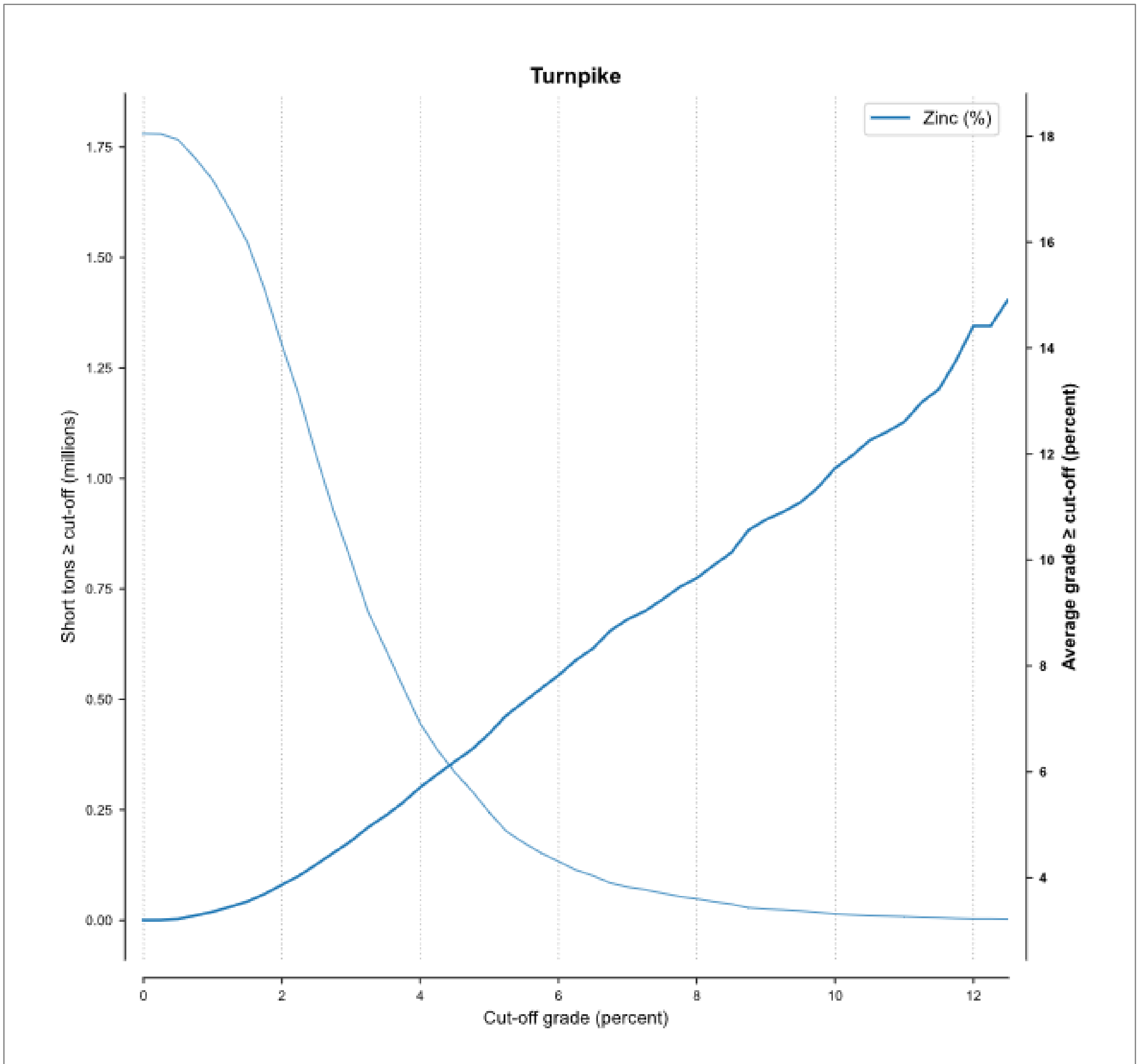


Source: ESM 2024

Figure 14-19: Sylvia Lake grade tonnage graph

Turnpike Open Pit

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 14.1.9.2. ESM 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.



Source: ESM 2024

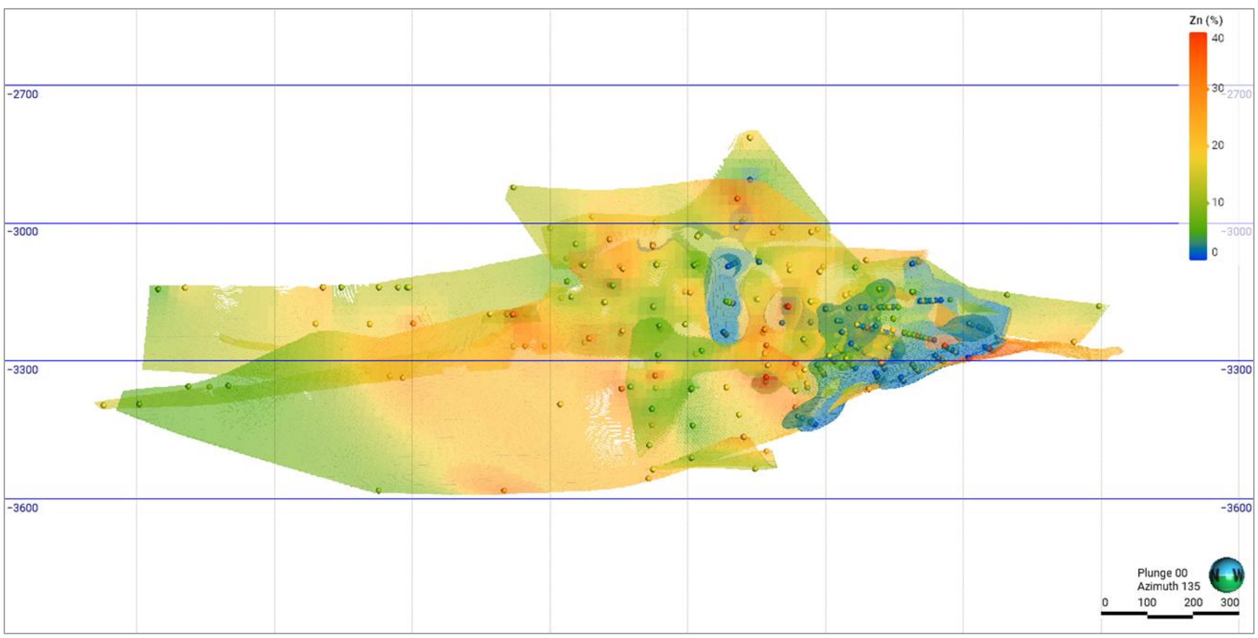
Figure 14-20: Turnpike Open Pit grade tonnage graph

14.1.10 Model Validation

14.1.10.1 Visual Comparison

ESM 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.

New Fold is provided as an example in Figure 14-21 for the underground. New Fold demonstrates both areas of excellent visual representation in the model and less ideal representation due to a range of sample clustering from tightly spaced to widely spaced sample data.

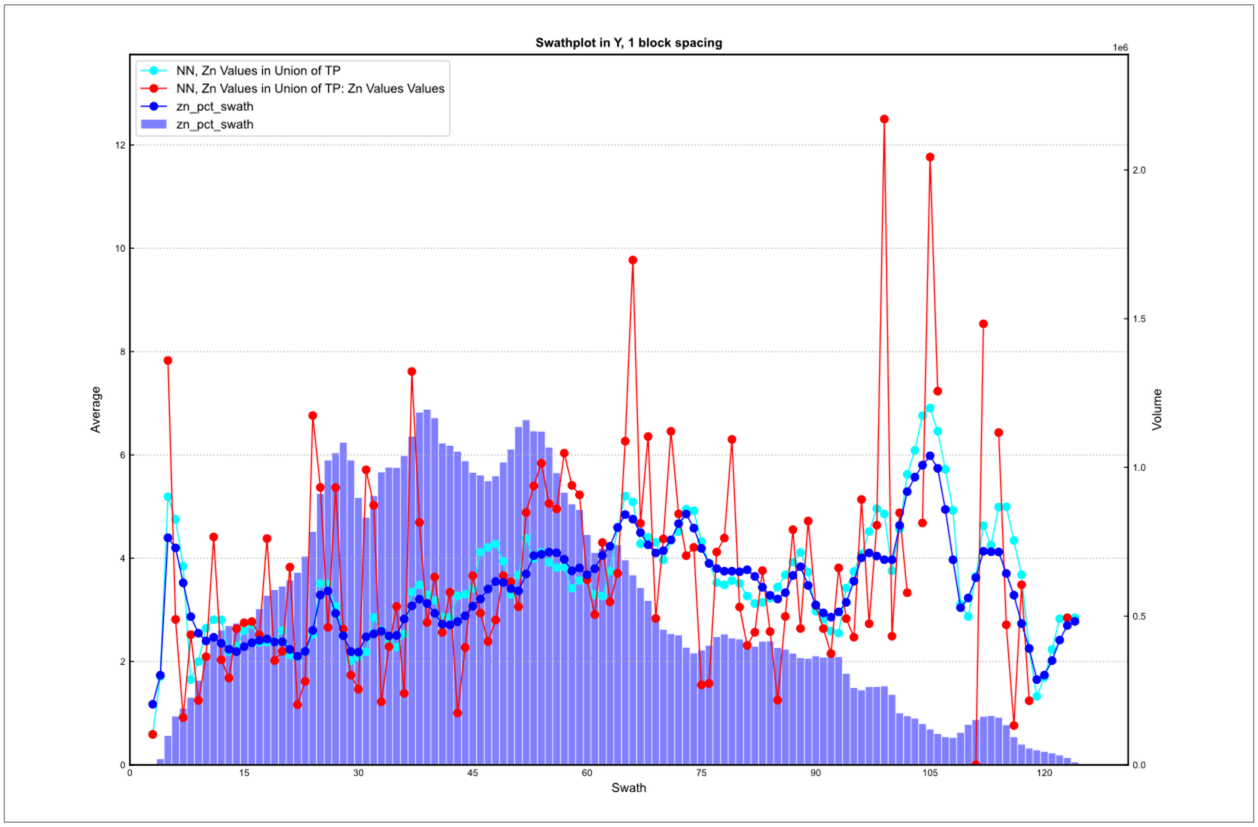


Source: ESM 2024
Note: View looking south-east.

Figure 14-21: New Fold model and composite values for zinc

14.1.10.2 Swath plots

ESM used swath plots to verify that the spatial distribution of grade in the composites is honored in the interpolated model by comparing the interpolated grade with composite values and nearest neighbor grades. An example is shown below in Figure 14-22 for Turnpike. Swath plots generally show agreement of the estimate to the composites, with an appropriate degree of smoothing.



Source: ESM 2024

Figure 14-22: Swath plot Zn% - Turnpike area

14.1.11 Relevant Factors

ESM is not aware of any other material factors that may influence the disclosure of Mineral Resources. The Turnpike Open Pit mining area is subject to permitting 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 underground areas are currently being mined.

14.2 Graphite Mineral Resource Estimate

14.2.1 Deposit Database

The Kilbourne Graphite Project (Kilbourne) database totals 45 surface-collared diamond drillholes (DDH) and one surface channel, totaling 29,699 ft used for modeling Kilbourne. There are a total of 3,396 assay records in the Kilbourne database, of which 2,088 assay records for graphite (%Cg).

The 12 geological domains at Kilbourne are summarized in Table 14-13. The domain naming convention is used consistently throughout this disclosure.

Table 14-13: Kilbourne Deposit geological domains

Domain	Rock Type
10	Sylvia Lake
20	Tailings
30	Overburden
40	Meta-sediments (PSS)
100	Upper Marble #1 Formation (UM1)
150	Pegmatite Intrusion (PEG)
160	Popple Hill Gneiss (PHG)
210	Upper Marble #2 Formation (UM2 – Upper)
220	Upper Marble #2 Formation (UM2 – Middle)
230	Upper Marble #2 Formation (UM2 – Lower)
300	Upper Marble #3 Formation (UM3)
400	Upper Marble Undifferentiated (UM4-16)

Source: BBA 2024

The drillhole database was validated before proceeding to the resource estimation phase, and the validation steps are detailed in Chapter 12.

Titan maintains all drillhole data in an industry standard SQL relational database called Geospark, with an Access interface customized for ESM.

Header, survey, assay, lithology, and specific gravity information were saved as individual tables within the database. A CSV format copy of the database was provided to the QP on July 24, 2024.

The unrecoverable intervals due to core loss were assigned void (-) value within the 200 series domains (domains 210, 220, 230). Essentially treating these intervals as potentially mineralized. The QP believes that non-assayed, unrecoverable material should not be assigned a zero value, as this does not reflect the true value of the material as the actual grades are unknown.

All negative and zero values within each domain were assigned to half the lower limit of detection (LLD) based on each elements reported LLD value. The LLD for graphite was 0.05%Cg.

The QP believes that the database is appropriate for the purposes of Mineral Resource estimation and the sample density allows a reliable estimate of the tonnage and grade of the mineralization in accordance with the level of confidence established by the Mineral Resource categories as defined in the CIM Guidelines.

14.2.2 Density

Titan collected a total of 7,487 samples from the diamond drillholes in the Kilbourne Deposit for SG measurements. A total of 4,599 measurements were used after outlier removal. Domain 400 used SG measurements where Zinc values were less than 0.50% Zn, reducing the measurements from 7,047 to 4,203 prior to outlier analysis and removal. This 0.50% Zn threshold removed values related to the ESM Zinc deposits and/or mineralized domains.

Titan used the following procedure to determine the average SG for each of the mineral domains:

- Sample selected for SG measurement;
- The Drillhole ID, row number, From, To and rock type were entered into a spreadsheet;
- The sample was weighted dry on the scale;
- The sample was then weighted, submerged and saturated in tap water at a constant 22 °C;
- The specific gravity is determined using the following equation:

$$SG = Wd/(Wd - Ws) * CF$$

Wd = Dry weight, Ws = Submerged weight, CF = Correction factor for water temperature

All SG measurements were converted to bulk density using an assumption of equal relationship of SG to grams per cubic centimeter (g/cm^3), and a unit conversion to a TF represented in short tons/ft^3. A constant SG and converted TF was assigned to each domain. A conversion of 1.00 g/cm^3 equal to 0.031214 US ton/ft^3 was used followed by rounding to 3 significant figures. The TF values were used in the block model.

Table 14-14 summarizes the results of the SG and TF measurements by domain.

Table 14-14: Kilbourne Deposit specific gravity and tonnage factor summary

Domain	Rock Type	Number of Samples	Minimum SG	Maximum SG	SG (Mean)	TF (Mean)	Comment
10	Sylvia Lake	-	-	-	1.00	0.0312	SG for Water
20	Tailings	1	2.62	2.62	2.62	0.0818	
30	Overburden	-	-	-	2.62	0.0818	Same as Tailings
40	PSS	-	-	-	2.62	0.0818	
100	UM1	22	2.72	3.06	2.87	0.0896	
150	PEG	5	2.57	2.68	2.63	0.0821	
160	PHG	110	2.60	2.76	2.68	0.0837	
210	UM2 - Upper	68	2.66	2.96	2.78	0.0868	
220	UM2 - Middle	74	2.65	2.83	2.72	0.0849	
230	UM2 - Lower	71	2.61	2.86	2.73	0.0852	
300	UM3	45	2.67	2.88	2.83	0.0883	
400	UM4-16	4,203	2.42	3.38	2.90	0.0905	<0.50% Zn

Source: BBA 2024

14.2.3 Topography Data

Base topography is extracted from publicly available New York State LIDAR data. The topography is locally updated from photogrammetric data collected by an ESM owned and operated drone. The area covered by the DTM is sufficient to cover the area defined by the current resource model.

14.2.4 Geological Interpretation

Three-dimensional (3D) wireframe models of mineralization were developed in Leapfrog Geo™ version 2023.2.3 (Leapfrog) by Titan and reviewed by the QP. The wireframes were based on the geological interpretation of the logged lithology and sub-domained based on contiguous grade intervals greater than or less than 0.50%Cg within the Upper Marble #2 (UM2) formation, defining the Upper, Middle, and Lower sub-domains of UM2 (210, 220, 230). Contiguous grade intervals greater than or equal to 0.50%Cg were modeled within the higher-grade 210 and 230 sub-domains (UM2 – Upper and Lower, respectively), while contiguous grade intervals less than 0.50%Cg were modeled as the 220 sub-domain (UM2 – Middle). These 200 series domains form the basis of the Kilbourne Mineral Resource Estimate.

The wireframe solids were imported from Leapfrog into Datamine Studio RM™ version 2.1.125.0 (Datamine) in .dwg format. The solids were validated within Datamine. The modeling is broken down into twelve separate geological domains based on lithology.

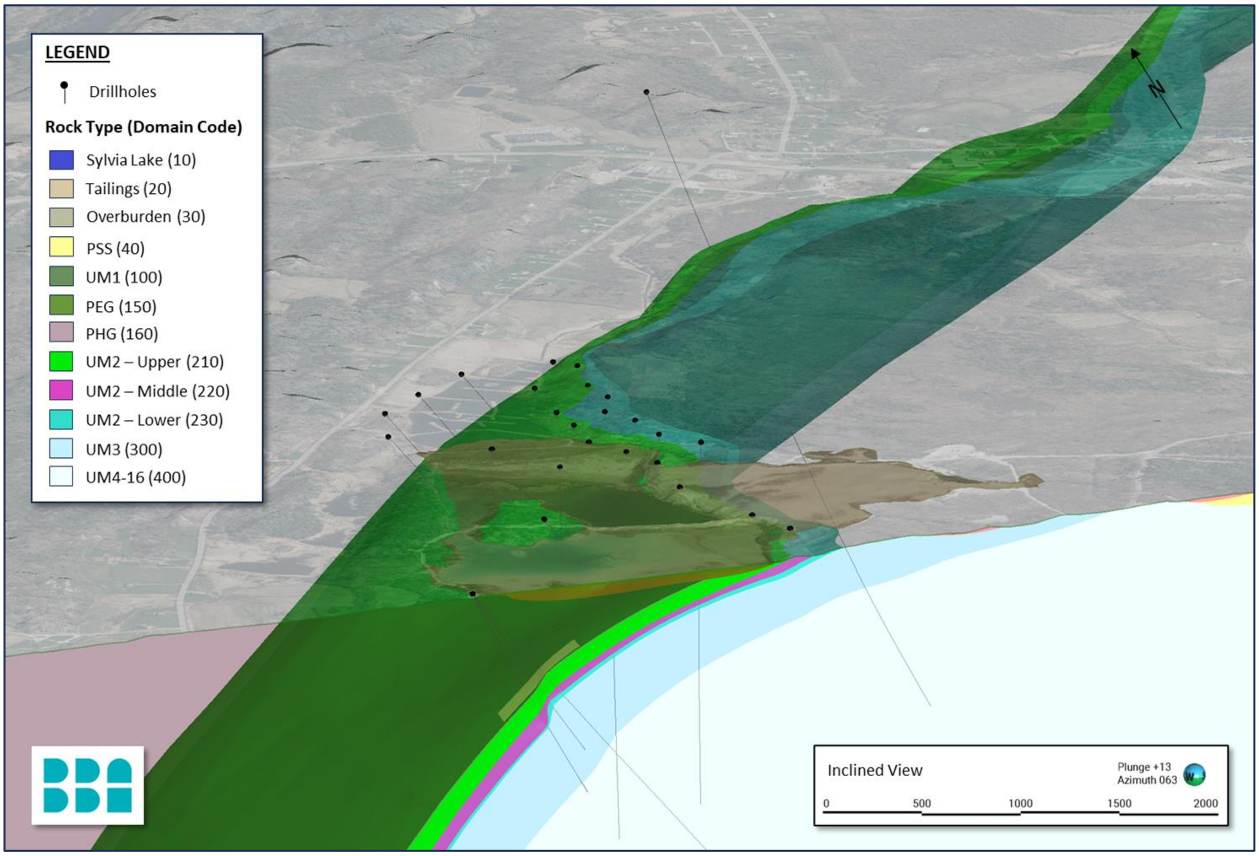
Table 14-15 summarizes the wireframe solids and associated volumes by domain. Figure 14-23 illustrates the model wireframes for each of the domains.

Table 14-15: Kilbourne Deposit wireframe volume to block model volume summary

Domain	Rock Type	Wireframe Volume (ft³)	Block Model Volume (ft³)
10	Sylvia Lake	1,112,463,783	3,283,031
20	Tailings	283,443,261	287,494,313
30	Overburden	796,788,068	536,716,125
40	PSS	3,255,495,868	3,105,833,203
100	UM1	2,675,882,913	1,082,348,578
150	PEG	94,518,988	94,532,063
160	PHG	378,108,715,672	166,716,125,016
210	UM2 - Upper	10,814,803,982	7,441,832,813
220	UM2 - Middle	5,256,559,823	3,201,850,688
230	UM2 - Lower	8,072,481,050	5,102,011,969
300	UM3	61,208,472,618	41,312,278,969
400	UM4-16	1,222,313,490,062	370,508,462,297

Source: BBA 2024

The wireframes extend at depth, below the deepest diamond drillholes. This is to provide a target for future exploration. The block model extents did not encompass the entire wireframe extents to reduce block model and file sizes. As such the volumes related to the block model may significantly differ in comparison to the wireframe volumes. The volumes were validated with an initial block fill of the entire wireframes and no significant discrepancies were noted.



Source: BBA 2024

Figure 14-23: Interpretation of Kilbourne Domains

14.2.5 Exploratory Data Analysis

14.2.5.1 Assays

The 12 domains included in the Mineral Resource were sampled for a total of 2,088 graphite (%Cg) samples, with eight additional elements modeled for internal project purposes. Not all domains were sampled for graphite, with primary graphite sampling focused on the UM2 formation (210, 220, and 230 domains). Some samples were only sampled for graphite and not the additional elements and vice versa.

The assay intervals within each mineral domain were captured using the Leapfrog evaluated column routine to flag the intercept into a new table in the database. These intervals were reviewed to ensure all the proper assay intervals were captured and no duplication or splitting of intervals occurred. Table 14-16 summarizes the basic statistics for the assay intervals for each of the mineral domains on the Property.

Table 14-16: Kilbourne Deposit drillhole basic "raw" statistics by domain

Domain	Element	Number of Samples	Missing Intervals	Minimum	Maximum	Mean	Variance
10	Cg (%)	-	-	-	-	-	-
	Length	-	-	-	-	-	-
20	Cg (%)	2	0	0.03	0.10	0.09	0.00
	Length	2	0	1.00	5.60	3.30	5.29
30	Cg (%)	-	-	-	-	-	-
	Length	-	-	-	-	-	-
40	Cg (%)	-	-	-	-	-	-
	Length	-	-	-	-	-	-
100	Cg (%)	85	1	0.03	1.28	0.12	0.04
	Length	86	0	0.60	6.00	3.85	1.63
150	Cg (%)	36	0	0.03	0.03	0.03	0.00
	Length	36	0	4.00	6.00	4.96	0.13
160	Cg (%)	252	1	0.03	2.74	0.17	0.10
	Length	253	0	0.50	6.70	4.58	0.90
210	Cg (%)	545	0	0.03	13.50	2.55	1.40
	Length	545	0	0.50	6.50	4.60	0.84
220	Cg (%)	451	0	0.02	5.39	0.36	0.26
	Length	451	0	0.40	6.10	4.57	0.96
230	Cg (%)	406	0	0.06	11.30	2.49	1.30
	Length	406	0	0.80	6.00	4.37	1.13
300	Cg (%)	311	91	0.03	2.06	0.07	0.02
	Length	402	0	0.60	7.00	4.61	0.76
400	Cg (%)	0	1,215	-	-	-	-
	Length	1,215	0	0.50	6.60	4.71	0.56

Source: BBA 2024

14.2.5.2 Grade Capping

The raw assay data for graphite within the 210, 220, and 230 domains were examined to assess the amount of metal that is bias from high-grade assays. A combination of reviewing decile analysis tables (Parrish,1997), histograms, Q-Q, and cumulative frequency plots was used to assist in determining if grade capping was required. The global top-cut analysis tool within the Snowden Supervisor™ version 9.0.3.0 software (Snowden Supervisor) was used in the capping process.

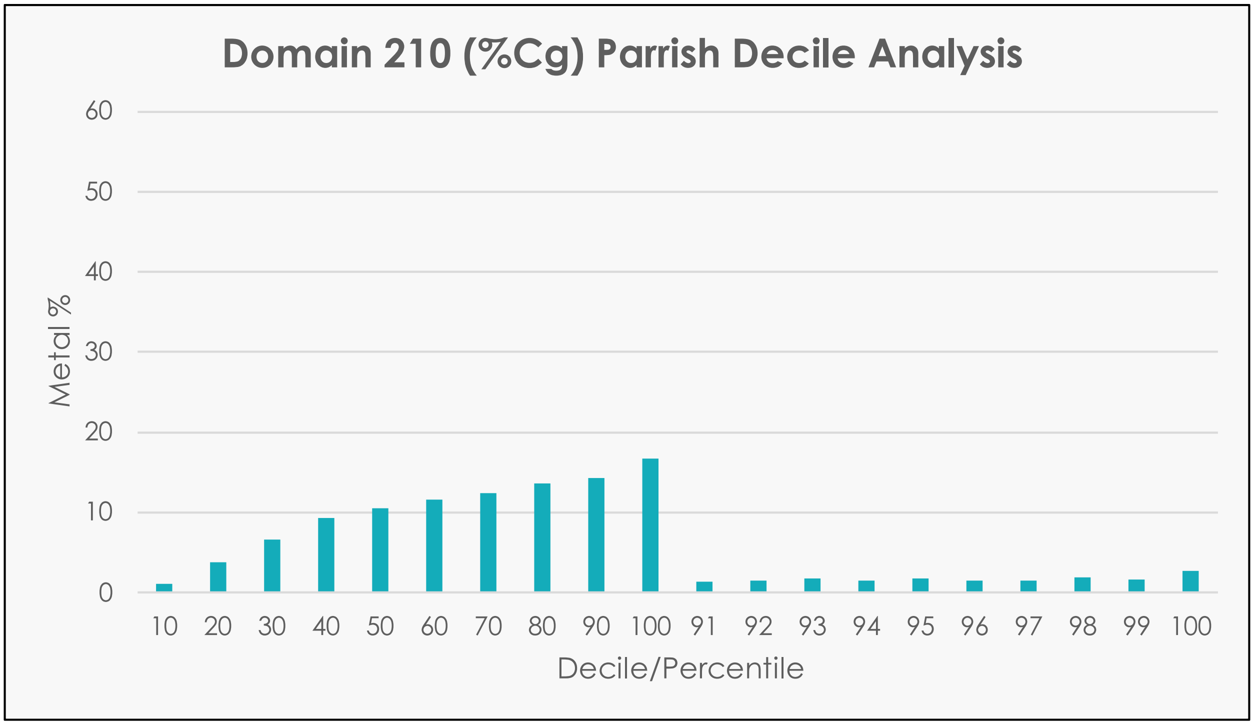
A review of the 3D spatial distribution of the capped samples was completed to determine if the samples were spatially close and if there was potential of a higher-grade sub-domain. This was not observed in any of the domains on the deposit.

This analysis concluded grade capping was required for domains 210, 220 and 230 individually. Thirteen of the samples capped in domain 220 were related to drillhole SX22-2621, approximately 6,300 ft from the nearest drillhole. The remaining three capped samples were greater than 1.50%Cg in drillhole KX24-038. Table 14-17 summarizes the capping applied to each domain by the QP. Figure 14-24 and Figure 14-25 show the decile analysis and global top cut analysis performed by the QP, using domain 210 as an example.

Table 14-17: Kilbourne Deposit grade capping summary

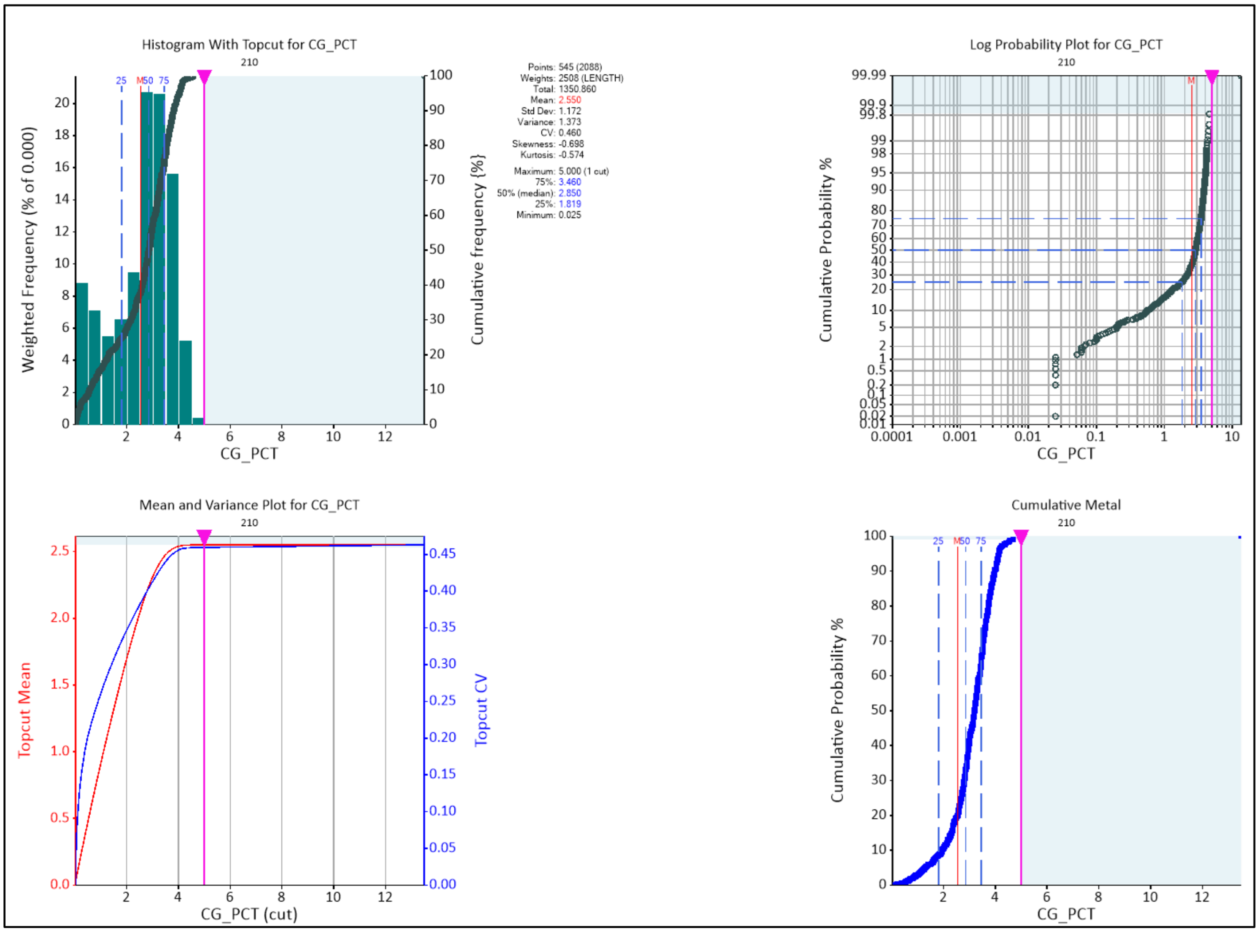
Domain	Element	Capping Value (%Cg)	Capped No. Samples	Uncapped Mean (%Cg)	Capped Mean (%Cg)	Metal Loss (%)
210	Cg (%)	5.00	1	2.55	2.55	0.1
220	Cg (%)	1.20	16	0.36	0.30	15.7
230	Cg (%)	6.00	2	2.49	2.47	0.5

Source: BBA 2024



Source: BBA 2024

Figure 14-24: Parrish decile analysis for domain 210



Source: BBA 2024

Figure 14-25: Global top cut analysis for domain 210 using Snowden Supervisor

14.2.5.3 Compositing

Compositing of all the capped assay data within each domain was completed on downhole intervals honoring the interpretation of the geological solids. Statistics indicate that a majority of the samples were collected at 5 ft intervals. Composites were generated at a 5 ft best-fit option, allowing all the material to be used in the compositing process. Datamine's backstitch option distributed the "tails" of the composite equally across all the composites in the hole to ensure all the sample material was used in the estimate. Table 14-18 summarizes the statistics for the drillholes after compositing.

Table 14-18: Kilbourne Deposit drillhole composited statistics by domain

Domain	Element	Number of Samples	Missing Intervals	Minimum	Maximum	Mean	Variance
10	Cg (%)	-	-	-	-	-	-
	Length	-	-	-	-	-	-
20	Cg (%)	1	0	0.10	0.10	0.10	-
	Length	1	0	5.60	5.60	5.60	-
30	Cg (%)	-	-	-	-	-	-
	Length	-	-	-	-	-	-
40	Cg (%)	-	-	-	-	-	-
	Length	-	-	-	-	-	-
100	Cg (%)	65	1	0.03	1.24	0.12	0.03
	Length	66	0	4.35	7.00	5.00	0.24
150	Cg (%)	36	0	0.03	0.03	0.03	-
	Length	36	0	4.90	5.01	4.96	0.00
160	Cg (%)	230	1	0.03	2.72	0.17	0.10
	Length	231	0	2.70	6.00	5.00	0.05
210	Cg (%)	504	0	0.03	4.48	2.55	1.30
	Length	504	0	3.60	6.10	4.96	0.03
220	Cg (%)	411	0	0.02	1.20	0.30	0.05
	Length	411	0	3.00	7.30	5.01	0.06
230	Cg (%)	351	0	0.17	5.16	2.47	1.07
	Length	351	0	3.80	6.90	5.05	0.08
300	Cg (%)	288	85	0.03	0.98	0.07	0.02
	Length	373	0	4.30	5.58	4.97	0.04
400	Cg (%)	0	1,144	-	-	-	-
	Length	1,144	0	4.83	5.13	5.00	0.00

Source: BBA 2024

14.2.5.4 Spatial Analysis

Variograms for graphite were created to inform the search ellipse dimensions for each 200 series domain. The variogram rotation and maximum range governed the search ellipse rotation and size. The variograms were also used to assign kriging weights during the estimation process.

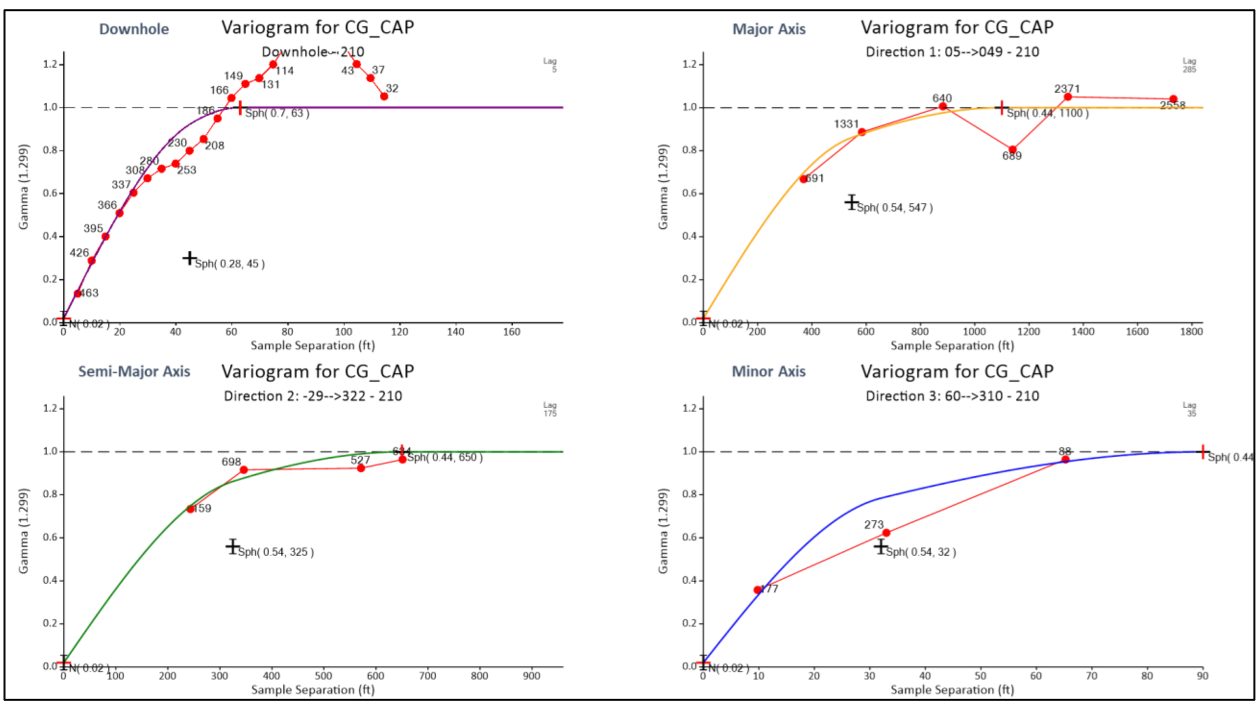
The variography for Kilbourne was determined using Snowden Supervisor software. Each 200 series domain was modeled using a downhole variogram to determine the nugget effect, and then a spherical pair-wise variogram was used to determine spatial continuity in the domain.

Table 14-19 summarizes the results of the variogram models for graphite. Figure 14-26 shows an example of the variography for domain 210.

Table 14-19: Variogram parameters

Domain	Element	Nugget (C_0)	First Structure (spherical)				Second Structure (spherical)			
			C_1	Range 1 (ft)	Range 2 (ft)	Range 3 (ft)	C_2	Range 1 (ft)	Range 2 (ft)	Range 3 (ft)
210	Cg	0.02	0.54	547	325	32	0.44	1,100	650	90
220	Cg	0.02	0.39	1,031	325	15	0.59	1,800	650	90
230	Cg	0.02	0.54	547	325	32	0.44	1,100	650	90

Source: BBA 2024



Source: BBA 2024

Figure 14-26: Variography for Domain 210 using Snowden Supervisor

14.2.6 Resource Block Model

14.2.6.1 Parent Model

A separate block model was established in Datamine for the Kilbourne Deposit. The model was not rotated.

A parent block size of 30 ft x 30 ft x 15 ft was selected to accommodate an open pit mining scenario. The block model was sub-celled on a 7.5 ft x 7.5 ft x 7.5 ft pattern, allowing the parent block to be split in each direction to fill the volume of the wireframes more accurately, and therefore more accurately estimate the tonnes in the Mineral Resource. Mineral estimation was completed on the parent blocks and the grades assigned to the sub-blocks.

Table 14-20 summarizes details of the parent block model.

Table 14-20: Block model parameters

Properties	X (column)	Y (row)	Z (level)
Origin Coordinates	7,520	7,500	-3,200
Number of Blocks	496	350	280
Block Size (ft)	30	30	15
Sub-block Size (ft)	7.5	7.5	7.5
Rotation	No Rotation		

Source: BBA 2024

14.2.6.2 Estimate Parameters

Only the 200 series domains were estimated and the remaining domains were assigned a waste value of half the lower limit of detection, as well as each corresponding tonnage factor per domain.

The interpolations of the domains were completed using the estimation methods ordinary kriging (OK), inverse distance squared (ID2), and nearest neighbor (NN). The estimations were designed for multiple passes. In each estimation pass, a minimum and maximum number of samples were required, as well as a maximum number of samples from a drillhole in order to satisfy the estimation criteria. All estimation passes used the capped and composited dataset for the appropriate domain being estimated. The third search pass was wide to fill blocks between drillholes at depth where mineralization would be expected. The OK methodology is the method used to report the mineral estimate statement.

An anisotropic search ellipse was used for the estimation. A hard boundary was used, only the samples within the domain wireframe were used in the estimation. The result is that the search ellipse will not locate samples outside the domain wireframe. Dynamic Anisotropy methodology was used for the three 200 series domains.

Table 14-21 summarizes the search ellipse and rotations and Table 14-22 summarizes the interpolation criteria.

Table 14-21: Search ellipse and rotations

Domain	Element	Major Axis (ft)	Semi-Major Axis (ft)	Minor Axis (ft)	Axis 3 Rotation Strike	Axis 1 Rotation Dip	Axis 3 Rotation Plunge
210	Cg	550	325	45	-50	30	10
220	Cg	900	325	45	-50	30	0
230	Cg	550	325	45	-50	30	10

Source: BBA 2024

Table 14-22: Interpolation parameters

Domain	Element	Pass 1				Pass 2				Pass 3			
		Min Comp	Max Comp	Max Comp /DDH	Search Ellipse Factor	Min Comp	Max Comp	Max Comp / DDH	Search Ellipse Factor	Min Comp	Max Comp	Max Comp/ DDH	Search Ellipse Factor
210	Cg	3	8	2	1	3	8	2	1.6	3	8	2	4
220	Cg	3	8	2	1	3	8	2	1.6	3	8	2	4
230	Cg	3	8	2	1	3	8	2	1.6	3	8	2	4

Source: BBA 2024

14.2.7 Resource Classification

Several factors are considered in the definition of a resource classification:

- NI 43-101 requirements;
- Canadian Institute of Mining, Metallurgy and Petroleum Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines (CIM, 2019);
- Author's experience with graphite deposits;
- Spatial continuity based on the assays within the drillholes;
- Understanding of the geology of the deposit;
- Drillhole spacing, data quality and the estimation runs required to estimate the grades in a block.

All blocks were classified as Inferred. No material in the block model was considered as Indicated or Measured.

No environmental, permitting, legal, title, taxation, socio-economic, marketing, or other relevant issues that may affect the estimate of Mineral Resources are known to the QP. Mineral Reserves can be estimated only on the basis of an economic evaluation that is used in a preliminary Feasibility Study or a Feasibility Study of a mineral project; thus, no reserves have been estimated. As per NI 43-101, Mineral Resources that are not Mineral Reserves do not have to demonstrate economic viability.

14.2.8 Mineral Resource Tabulation

The resource reported is effective as of October 21, 2024, and has been tabulated in terms of a pit-constrained cut-off value of 1.50%Cg.

Table 14-23 summarizes the parameters used to develop the Kilbourne pit constraints for a reasonable prospect of economic extraction.

Table 14-23: Kilbourne Deposit pit constraint parameters

Input	Unit	Variable
Mining		
Mining Cost - Ore	US$/t mined	4.60
Mining Cost - Waste	US$/t mined	3.50
Mining Cost - Overburden and Tailings	US$/t mined	2.00
Dilution	%	5.0
Mining Recovery	%	95.0
Processing		
Processing Cost	US$/t milled	14.00
G&A Cost	US$/t milled	-
Processing Recovery	%	91.0
Concentrate Grade	%	95.0
Other		
Selling Price	US$/t concentrate	1,090
Transportation Cost	US$/t concentrate	50
Selling Costs	US$/t concentrate	0
Overall Slope Angle	degrees	OVB and Tailings: 23 Rock: 45

Source: BBA 2024

The pit-constrained mineral resource and in-situ metal for the Kilbourne Deposit is summarized in Table 14-24.

Table 14-24: Kilbourne Graphite Mineral Resource summary and in situ metal within pit shell

Classification	Deposit	Cut-off Grade (% Cg)	Tonnage ('000 Ton)	Grade (% Cg)	Contained Graphite ('000 Ton)
Inferred	Kilbourne	1.50	22,423	2.91	653

Source: BBA 2024

A Mineral Resource was prepared in accordance with NI 43-101 and the CIM Definition Standards (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

14.2.9 Model Validation

The Kilbourne Graphite block model was validated by three methods:

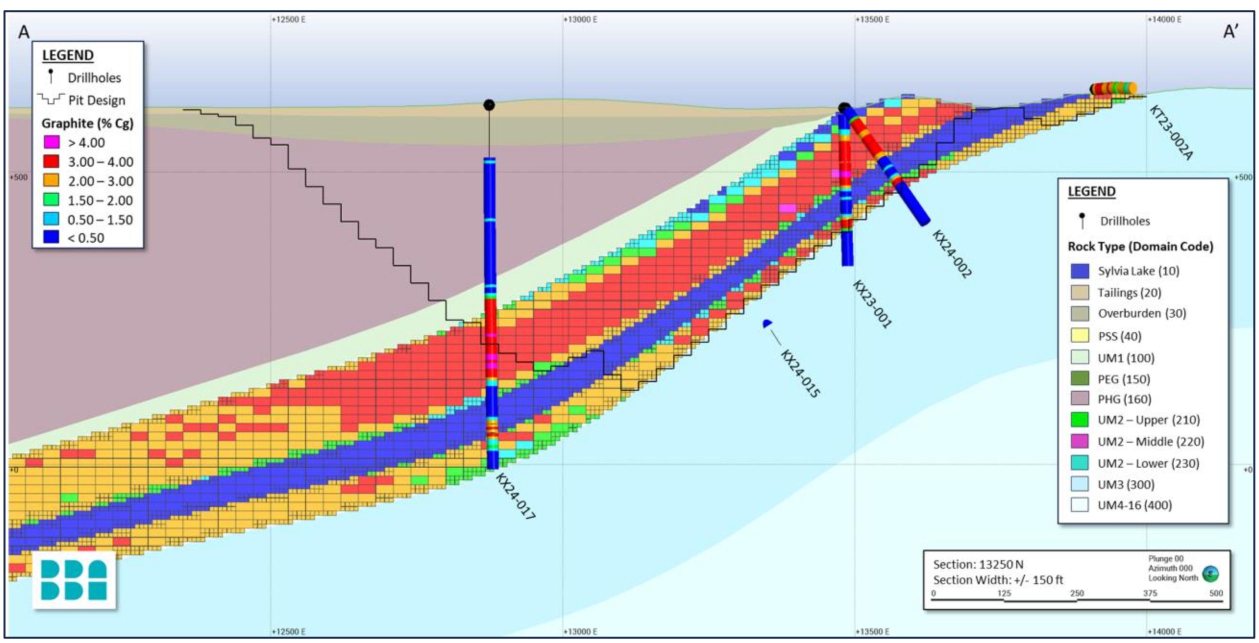
- Visual comparison of color-coded block model grades with composite grades on section;
- Comparison of the global mean block grades for OK, ID^2, and NN by domain and composite mean grades by domain;
- Swath plots.

14.2.9.1 Visual Validation

The visual comparisons of ordinary kriging block model grades and composite drillholes show a reasonable correlation between the values (Figure 14-27 and Figure 14-28). No significant discrepancies were apparent from the sections reviewed, yet grade smoothing was apparent in some of the lower elevations due to the distance between drill samples being broader in these regions.

Source: BBA 2024

Figure 14-27: Surface plan showing the optimized pit shell for the Kilbourne Deposit



Source: BBA 2024

Figure 14-28: Kilbourne Deposit visual validation through A-A'

14.2.9.2 Global Statistics

The global drillhole composite and block model statistics grouped by domain for the OK model were compared to the global ID^2, and NN models. Table 14-25 shows this comparison of the composite mean grades with the global estimates for the three estimation method calculations within the 200 series domains. Several optimization tests were conducted. It was determined the differences in estimated grades to the composite grades were related to data density and/or drillhole spacing. Comparisons were made using all blocks greater than 0.025%Cg.

Table 14-25: Kilbourne global composite to block model statistics comparison

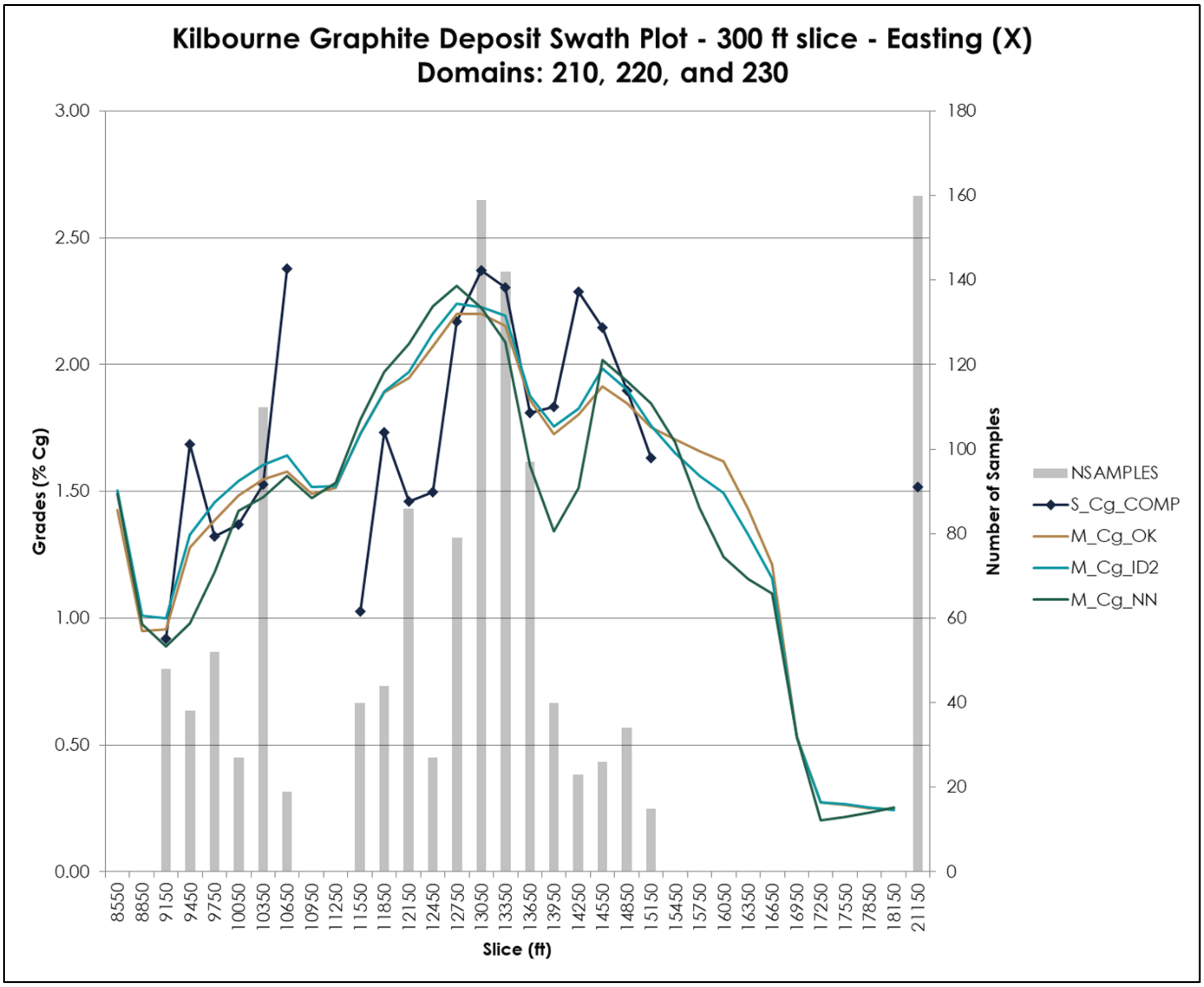
Domain	Element	Composite Mean	OK Mean	ID^2 Mean	NN Mean
210	Cg (%)	2.55	2.09	2.16	2.04
220	Cg (%)	0.30	0.23	0.24	0.26
230	Cg (%)	2.47	2.19	2.17	2.03

Source: BBA 2024

14.2.9.3 Swath Plots

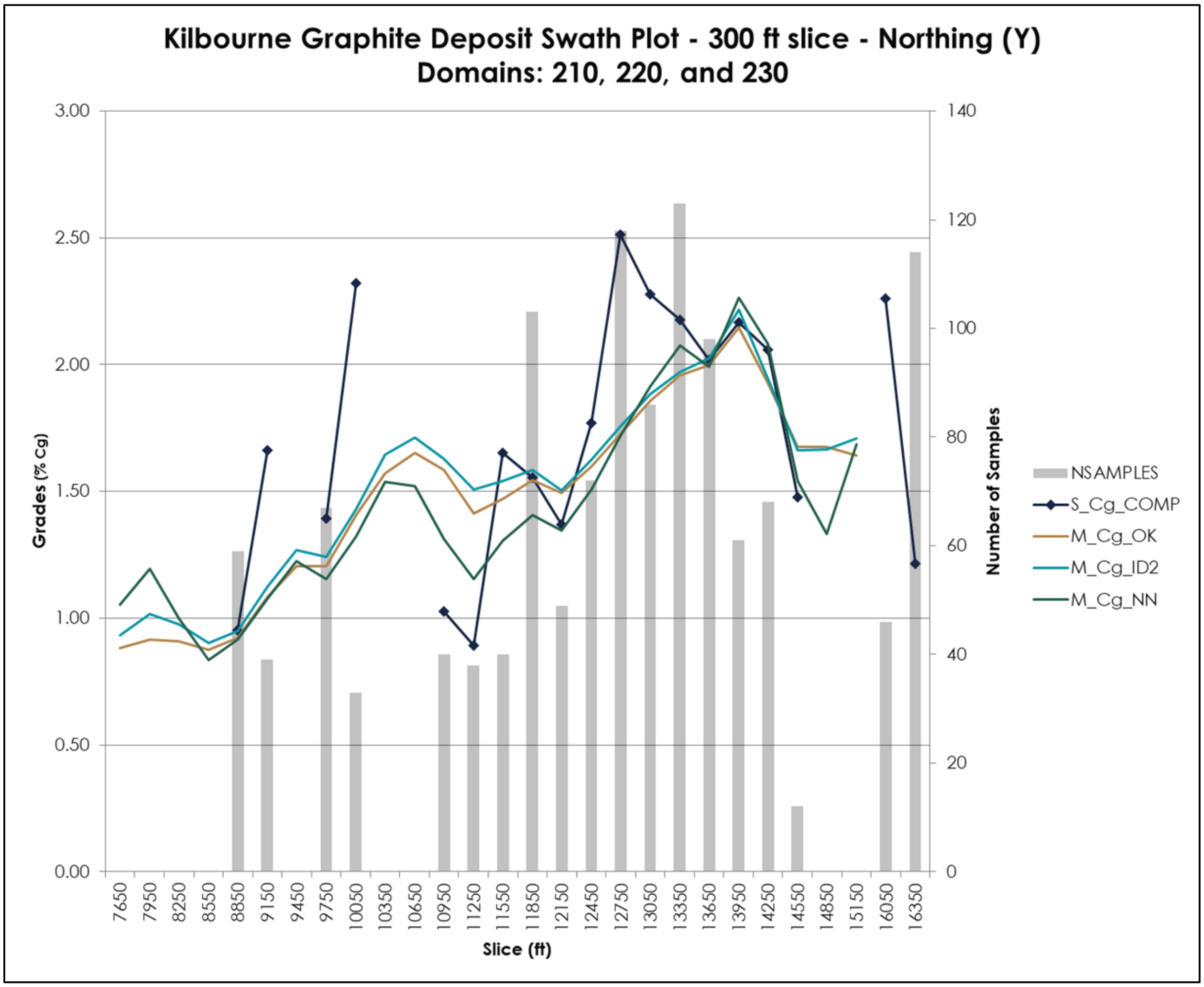
Figure 14-29 and Figure 14-30 display the comparison between the drillhole composites grades and the OK, ID^2 and NN estimates in a swath plot format. Comparisons were made using all blocks greater than 0.025%Cg for the 200 series domains.

As expected, there is a strong degree of grade smoothing with the OK methodology.



Source: BBA 2024

Figure 14-29: Kilbourne Deposit swath plot, 300 ft slice - easting (X)



Source: BBA 2024

Figure 14-30: Kilbourne Deposit swath plot, 300 ft slice - northing (Y)

14.2.10 Previous Estimates

The Kilbourne Graphite MRE is a maiden resource. There are no previous estimates to compare.

15. Mineral Reserve Estimates

There are no Mineral Reserves reported for ESM.

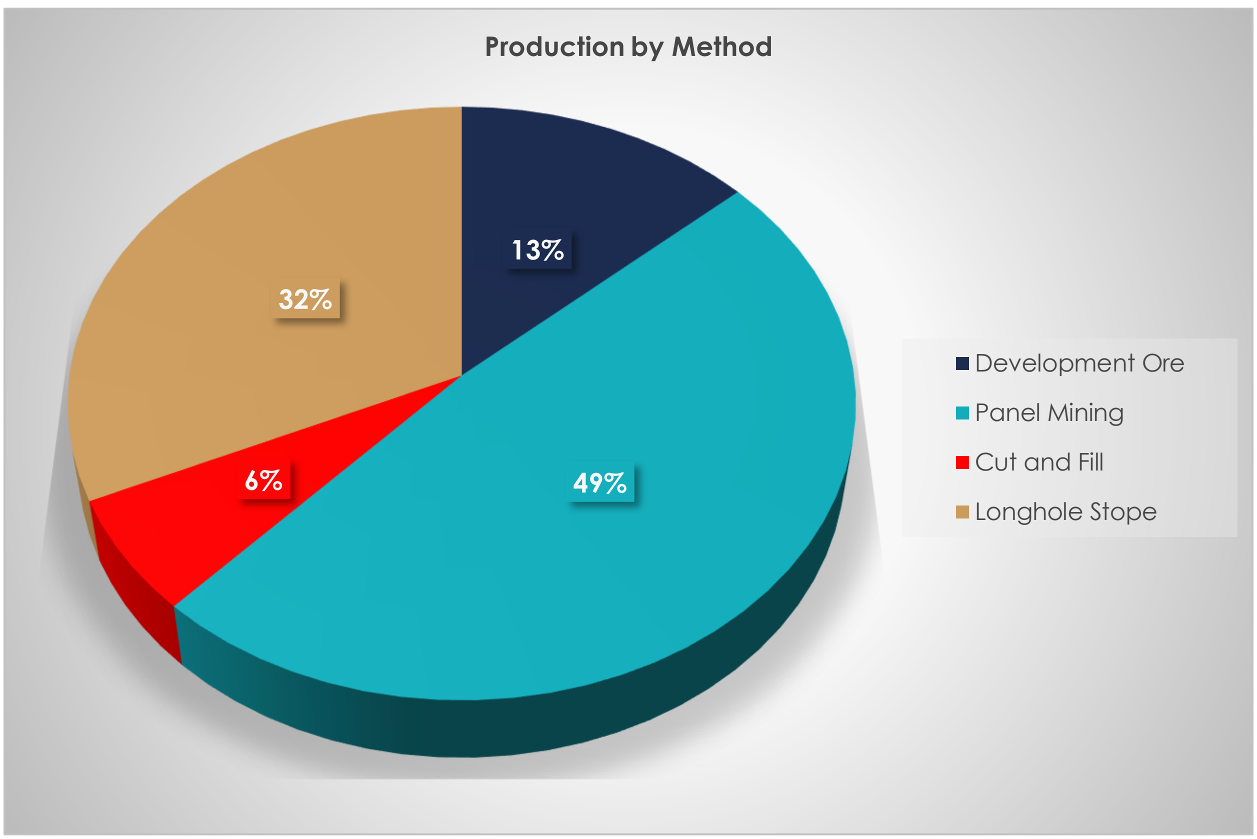
16. Mining Methods

Chapters 16 to 22 apply only to ESM's zinc operations. An economic analysis of ESM's graphite mineralization has not yet been completed.

The Turnpike open pits are not included in the LOM mineable resource or economic analysis. The inclusion into the LOM is dependent on zinc price. Conceptual pits with associated schedule are included in this report for permitting considerations.

16.1 Underground

The mine plan tons at the ESM deposit are extracted using a combination of longitudinal retreat stoping (LRS), Cut and Fill (C&F), Panel Mining (PM) – Primary and Secondary, and development drifting underground mining methods with rock backfill. Longhole back-stopes are also used in the design where applicable as part of LRS. As of 2024, the overall mine life is 9 years. Figure 16-1 outlines a summary of underground mining methods used at ESM.



Source: ESM 2024

Figure 16-1: Mine production by method

The ESM zinc operations are accessed from surface via the #4 Shaft, and all mineralized material and some waste rock is hoisted out of the mine via that same shaft. In addition to the existing development and raises, new lateral development and ramping is required to access new mineralized zones. To supplement the ventilation provided by the raises, as ramps are being driven, shorter internal ventilation drop raises ensure air delivery to the active development faces in areas where required.

Measured, Indicated, and Inferred Mineral Resources were included in the mine design and schedule optimization process. The proposed Mineral Resources for the life of mine (LOM) by mining method is shown in Table 16-1, which includes accessible remnants. The Mineral Resources for the LOM are based on the Mineral Resource Estimate as stated in Chapter 14 of this report.

For the purposes of this report, the LOM as designed starts in January of 2025.

Table 16-1: Mineral Resources for the LOM by mining method

Mining Method	Diluted Tons (kt)	Percent of LOM Plan
Development Ore	544	13%
PM	1,957	49%
C&F	255	6%
LRS	1,281	32%
Total	**4,037**	**100%**

Source: ESM 2024

Note: Totals may not compute exactly due to rounding.

16.1.1 Deposit Characteristics

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

- Upper Mahler;
- Lower Mahler;
- New Fold;
- Mud Pond Main;
- Mud Pond Apron.

Deswik version 2023.1 Stope Optimizer shapes and development designs were created for the remaining mining zones:

- America;
- Cal-Marble;
- Fowler;
- NE Fowler;
- N2D;
- Sylvia Lake.

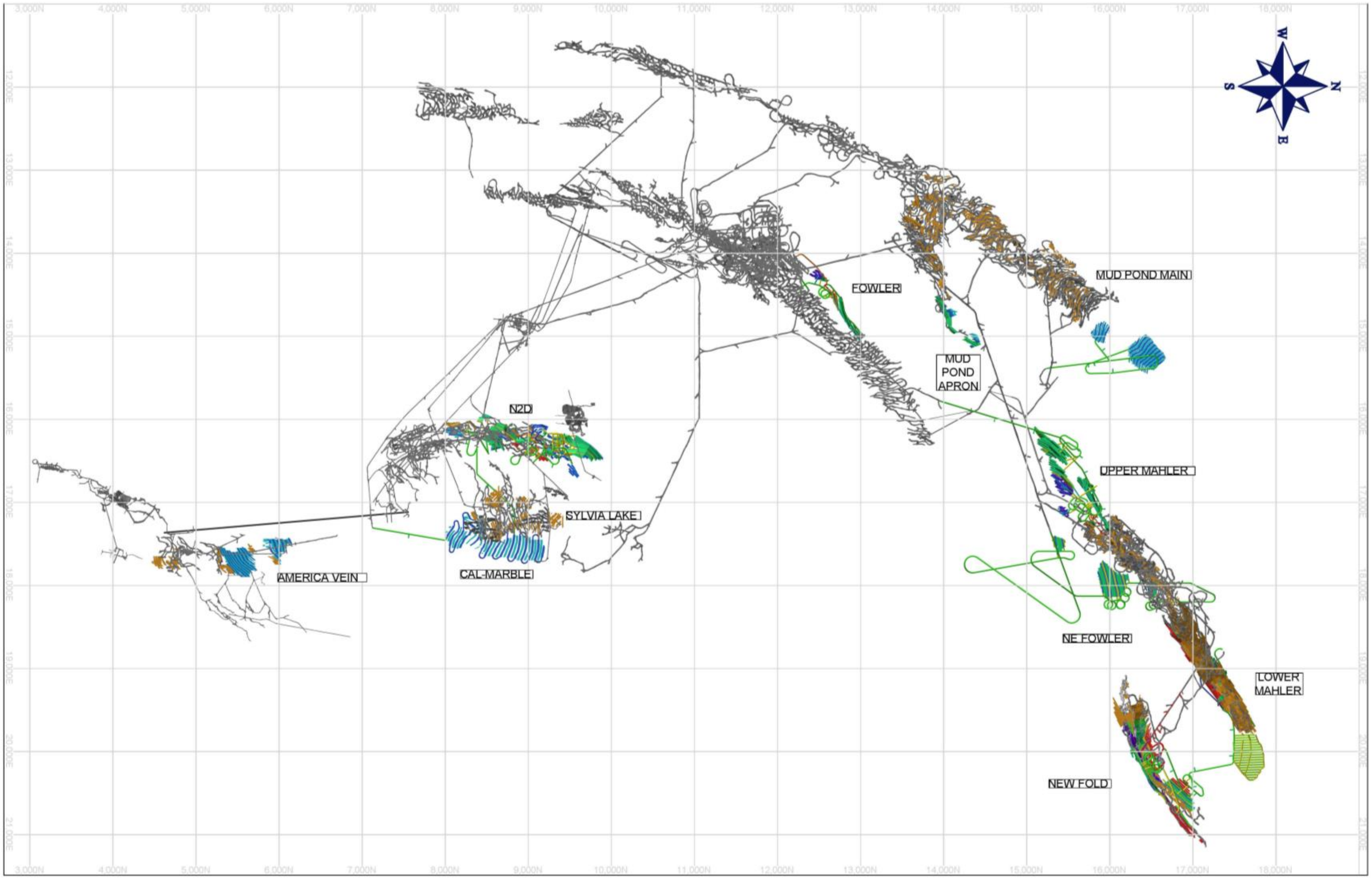
Figure 16-2 depicts the mining zones included in the LOM.

From Section 7.4:

> "The Property contains 14 known zones of sphalerite mineralization. Three clusters have been defined consisting of three to five deposits each. The zinc mineralization extends from the surface down to a depth of 5,700 ft below surface. The zones are aerially scattered and all zones except NE Fowler and Cal Marble are connected by existing development to the shaft. The zones range in thickness from 2 ft to 50 ft with an overall plunge between 20° to 25° with local dips ranging from 0° to 90°. The deposit footprints are up to 500 ft wide and 9,000 ft long. The veins can display considerable geometrical variability depending on the degree of folding."

Due to the complex geometry some local uncertainty is expected in areas with low density of exploration data. This leads to some deviation from designed plans but very rarely impacts the expected extraction.

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



Source: ESM 2024

Figure 16-2: Mining zones in the LOM

16.1.2 Mineral Resources Within the PEA Mine Plan – Estimation Process

To determine the Mineral Resources in the LOM, the following process was used:

- 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 economically viable production stopes and run economic models on various production scenarios.

16.1.3 Mining Method Selection

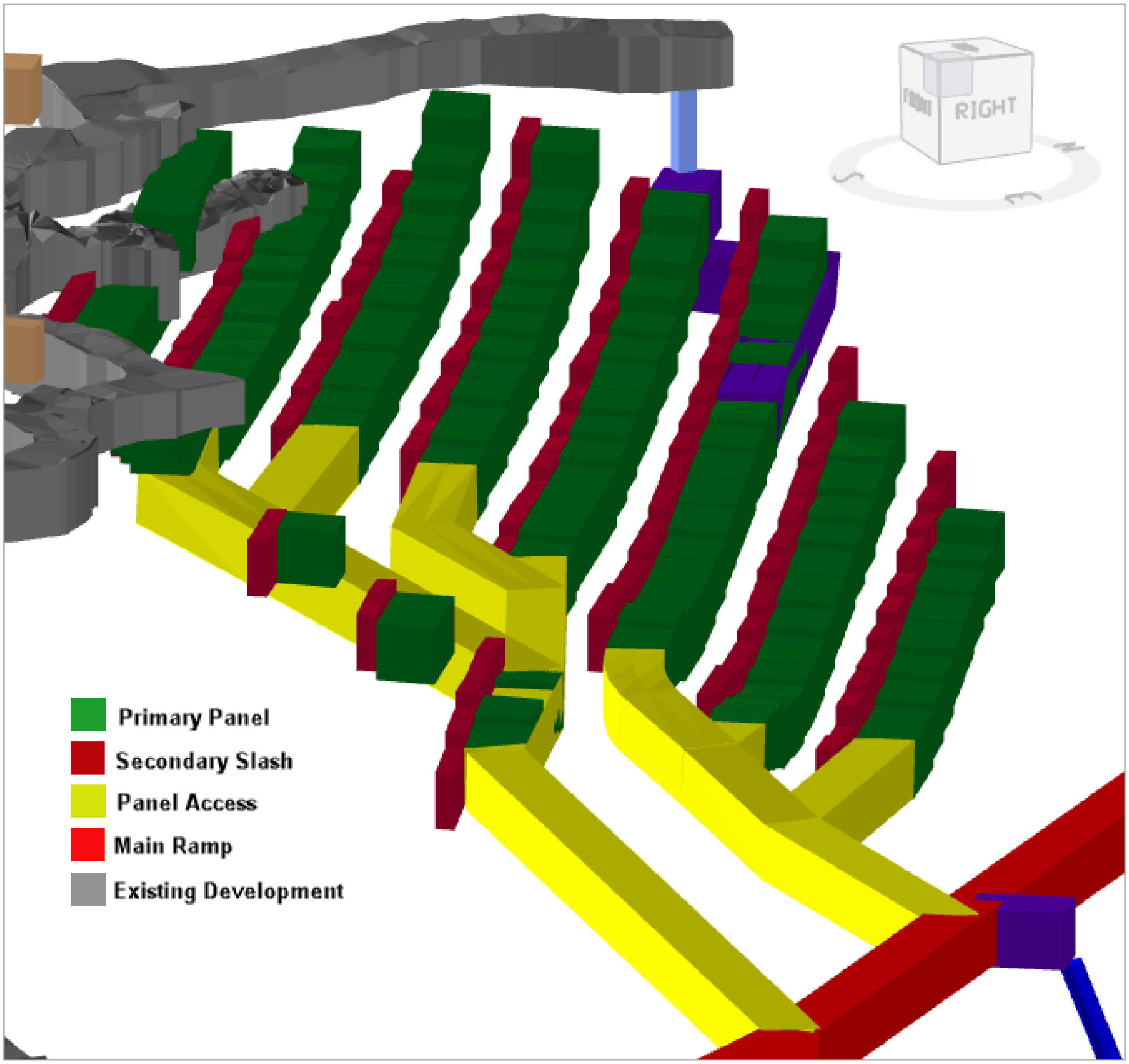
Given the locally variable resource geometries, several mining methods are in use at ESM.

PM, with Primary and secondary cuts, is the principal mining method used at ESM. The second most common method is LRS. C&F is used where conditions are not suitable for LRS. In areas where the geometry of mineralization is simple and directional, normal development activities will be designed in mineralization.

PM divides a mineralized area into three repeating sections, Primary panels, Secondary slashes, and pillars. The Primary panel or drift is mined to the deposit extents. A Secondary slash is mined in a retreating fashion up-dip, leaving a pillar between panels (Figure 16-3 and Figure 16-4). This method is suited for mineralization with a dip that is too shallow for LRS.

Source: Jackleg Consultants 2020

Figure 16-3: Plan view of Panel Mining



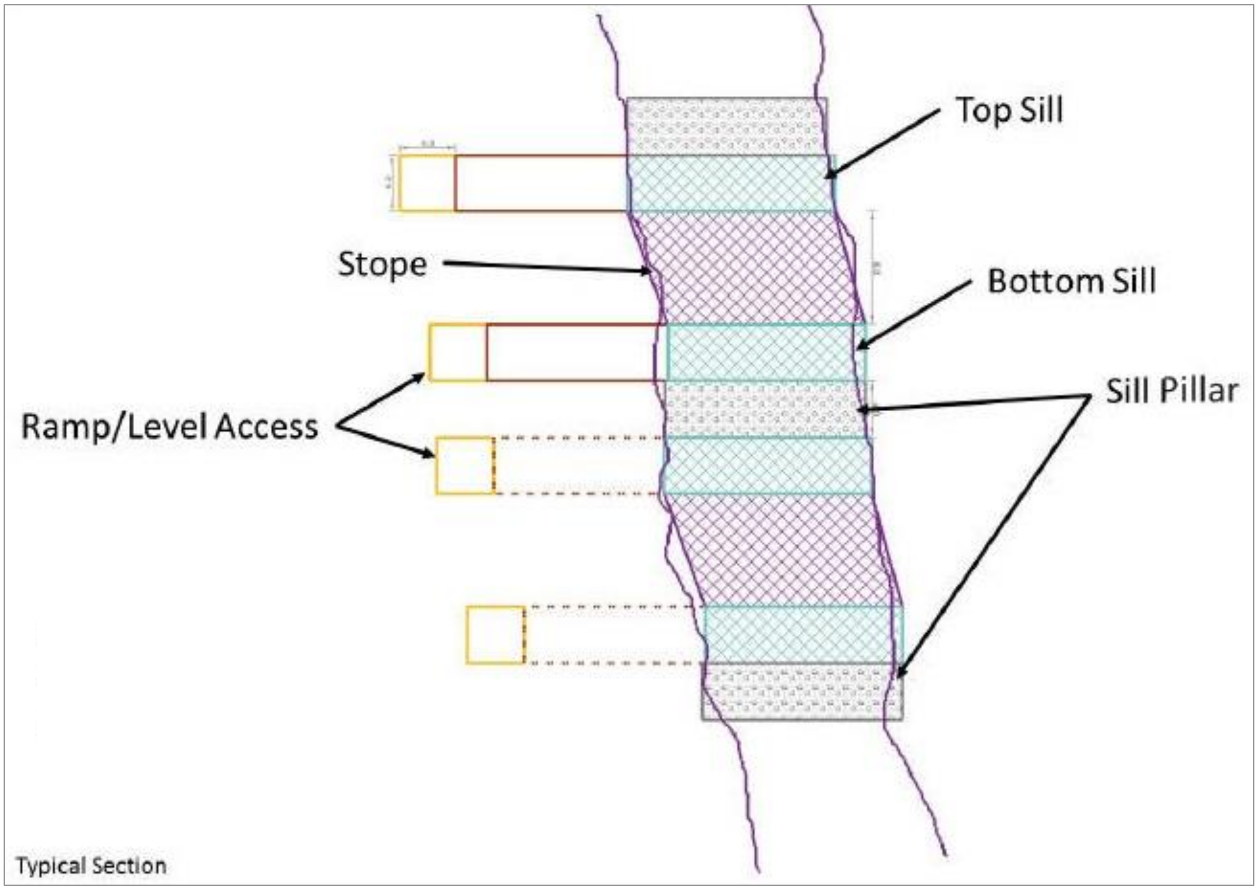
Source: Jackleg Consultants 2020

Figure 16-4: Isometric view of Panel Mining

LRS 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. 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 LRS, remote controlled load haul dump machines (LHD) are required to safely remove the blasted material from the stope.

One of the limitations with LRS is that the dimensions of the stope height should not exceed a longhole drilling machine's effective range. For the longhole drills in use at ESM, 80 ft is considered the uppermost limit. Another limitation with LRS is the stopes must remain open long enough to remove the mineralized material and then are filled with unconsolidated backfill material (where support pillars are not used). This mine plan assumes no backfill plant will be available, so sill pillars are left between levels, when longitudinal stoping is used.

The limitations discussed above, generally restrict level spacing at ESM to 60 ft. Back-stopes are designed to a height of 60 ft as there is no top cut (or level above). Back-stopes typically occur at the top of multi-level LRS areas. A typical cross section of an LRS with sill pillars is shown in Figure 16-5.

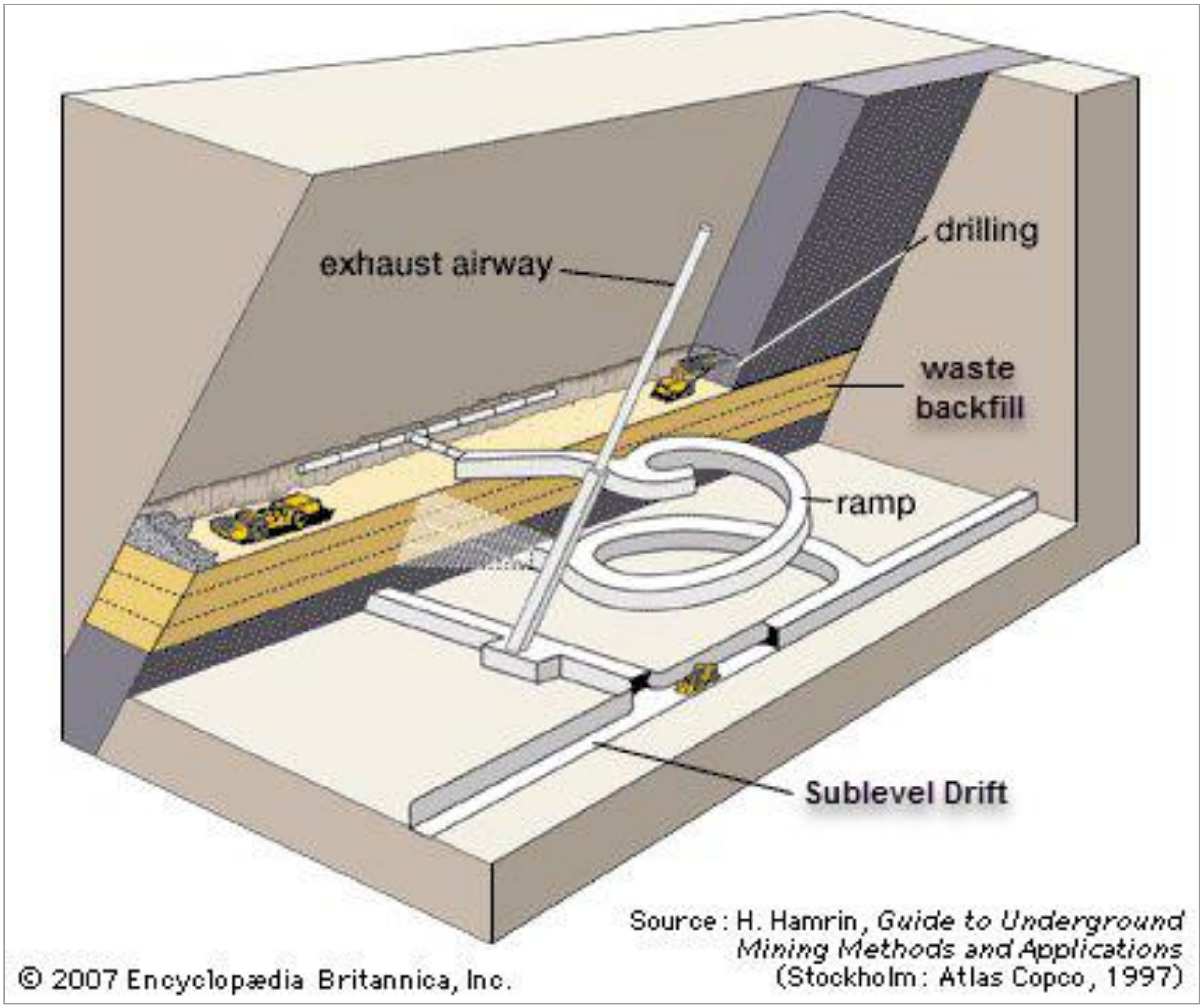


Source: Jackleg Consulting 2020

Figure 16-5: Typical LRS with sill pillar

LRS is used in Mahler, New Fold, and Mud Pond Apron with C&F and PM accessing the remaining mineralization that does not fit LRS design criteria.

C&F mining is used at ESM for areas of the deposit that fall below a practical dip for LRS, or where more selective mining is required. The method typically used is an overhand C&F whereby drifts are driven across strike and on level, backfilled with un-cemented fill, and then the next level above is mined. As there will not be a backfill plant, the un-cemented fill is waste rock from development headings. With the abundance of inactive areas, storage of waste material for C&F mining is not an issue. A typical layout for C&F is shown in Figure 16-6.

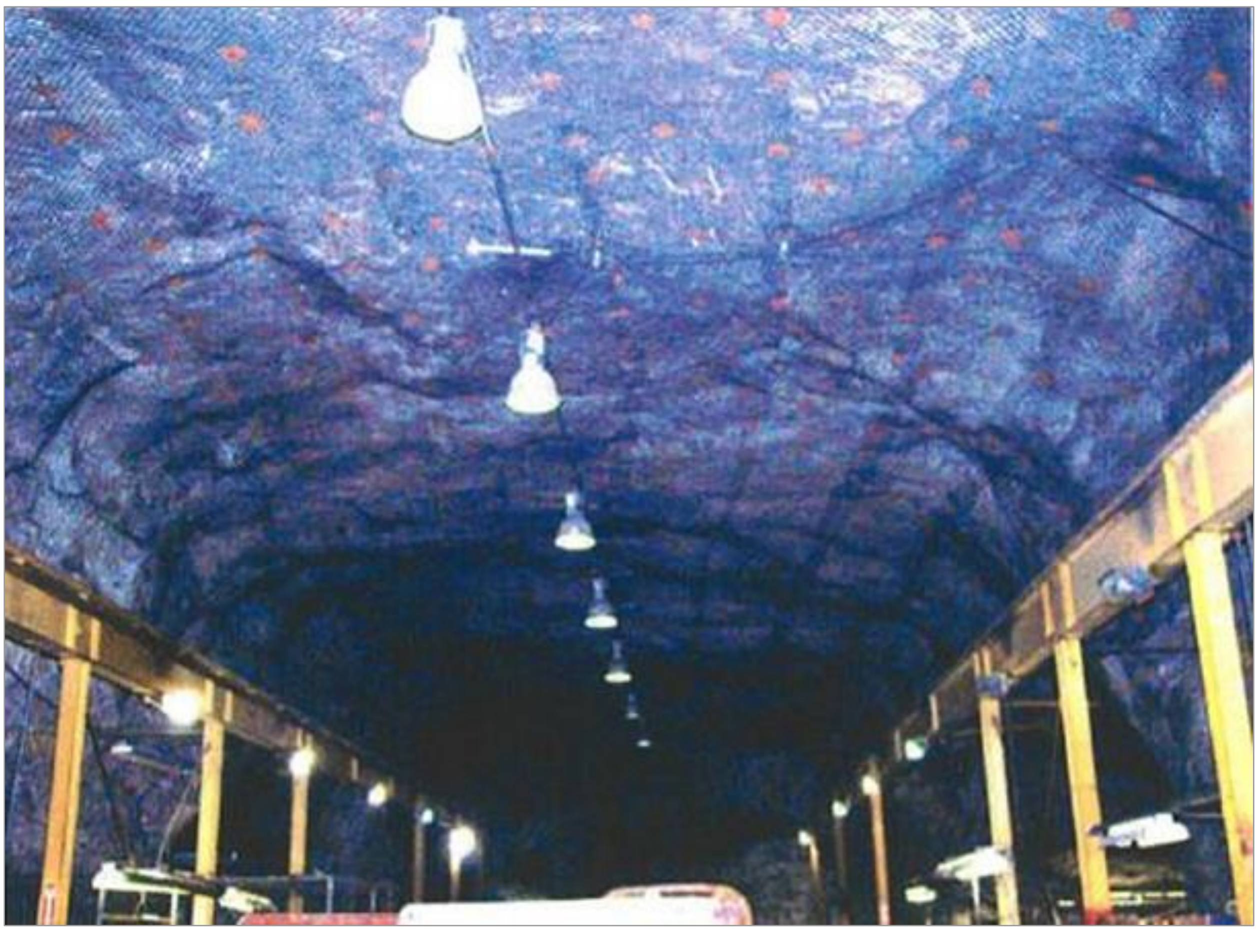


Source: Atlas Copco 1997

Figure 16-6: Typical C&F

16.1.4 Geotechnical Parameters

Rock quality at ESM is generally considered to be good to very good per internal site characterizations and third-party assessments. Dave West (West, 2018) reported that the rock mass is typically competent, consistent with a rock mass rating (RMR) of +85. Itasca Consulting (Brummer, 2005) reported that, in general, the rock would be rated as very good to excellent with RMR values of 80 or greater. Richard Brummer visited the 2500 level workshop, which is one of the largest openings at the mine, roughly 35 ft to 40 ft by 200 ft, and calculated an RMR of 87. The shop is supported by a combination of expansion shell bolts and Dywidag resin rebar (also known as Threadbar® in North America) (Figure 16-7).



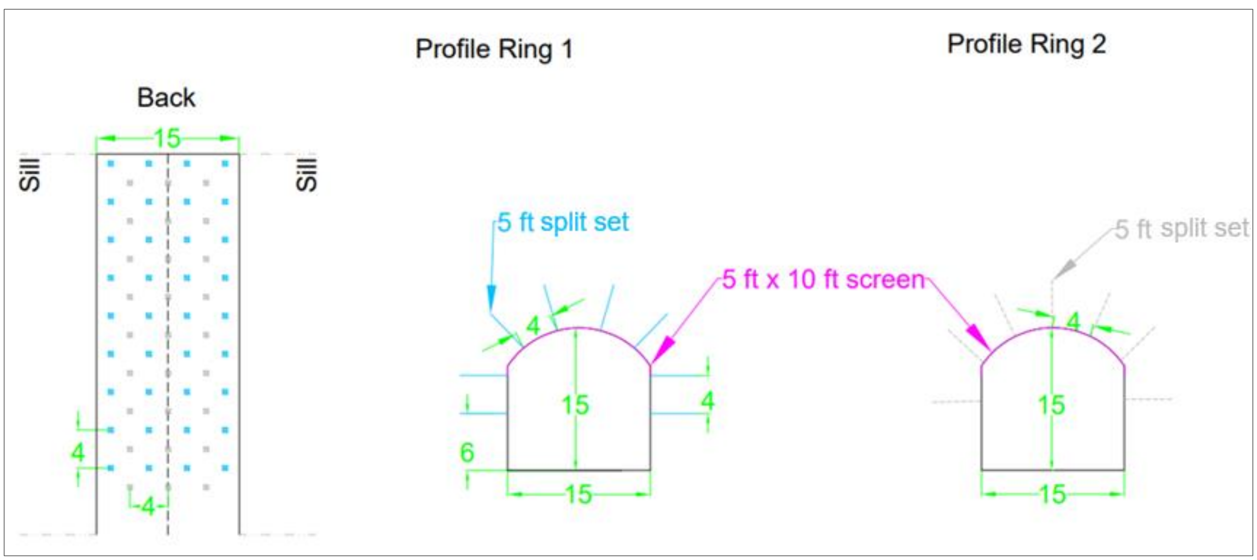
Source: Itasca 2005

Figure 16-7: 2500 level workshop back conditions

Prior to the 2001 shutdown, 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 totaled 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. Previous contractors were permitted to work under unsupported ground provided they deemed it safe, which is a practice no longer permitted nor 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. The standard included the use of SP33 split sets. Depending on the dimension of the drift and depth within the mine, split set lengths were increased and the application of welded wire mesh was incorporated.

As of 2024, 39 mm x 60 in (or 72 in) split sets and 6-gauge welded wire mesh are the primary ground support used at ESM with support extending across the back (Figure 16-8). Where necessary, secondary support typically consists of 8 ft to 12 ft long #7 all thread bolts and/or 20 ft long, 0.6 in diameter single strand cable bolts. Pull testing of the ground support is regularly done to assess ground support performance. The ground support systems and QA/QC is described in the Ground Control Management Plan (ESM, 2022).



Source: ESM 2022

Figure 16-8: Ground support for typical ground

16.1.5 Stope Design Parameters

Deswik.SO version 2023.1 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 (°)
C&F	13	15	N/A	40-90
PM - Primary	15	15	N/A	N/A
PM - Secondary	5	15	N/A	N/A
LRS	15	60	Max 150	50-90

Source: ESM 2024

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

LRS dimensions are variable to accommodate the geometry of the resource. A minimum 15 ft true width was used for stope design, along with a minimum overall 50° stope angle. 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. Back-stopes 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 overbreak experienced during mining operations. The rock quality at ESM is considered to be good geotechnically, so overbreak is considered to be minimal. For LRS, two sources of dilution were considered. Sloughing was estimated to be 2.0 ft on both the hanging wall and footwall of LRS. For C&F, planned overbreak 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.

Table 16-3: Overbreak dilution parameters

Typical Profiles	Unit	C&F	PM - Primary	PM - Secondary	LRS w/Crown Pillar	Back-stope
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
Hanging Wall Overbreak	ft	0.50	0	.50	2	2

Source: ESM 2024

Mine recovery was calculated under the following assumptions:

- C&F and waste development passing incremental cut-off, assumed 95% mine recovery after losses.
- LRS and back-stopes assumed 95% recovery.
- PM assumed 75% 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	1.30
Mill Recovery	%	96.4
TC / RC / Transport	$/t milled	35.13
Payable Metal	%	85
Royalties	%	0.3
Operating Costs	$/t milled	86.73
Cut-off	% Zn	5.5
Incremental Cut-off	% Zn	2.0

Source: ESM 2024

Incremental cut-off accounts for the cost of crushing, hoisting, milling, and general services incurred per ton of milled material. Incremental cut-off was applied to any waste development that crosses mineralization in order to access stopes designed with the primary cut-off of 5.5% Zn for all mining zones. Approximately 10% of all tons reporting to the mill are classified as incremental according to plan. Cut-off grade parameters may not reflect those used for economic modeling 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 met 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: Tons contained in the LOM plan by zone

Zone	Diluted Tons (kt)	Diluted Grade (% Zn)
America	280	6.5
Cal-Marble	303	6.2
Fowler	61	5.9
Lower Mahler	866	9.3
Mud Pond Apron	180	5.9
Mud Pond Main	367	6.0
N2D	457	6.5
Northeast Fowler	300	5.7
New Fold	515	10.0
Sylvia Lake	111	6.2
Upper Mahler	595	5.9
Total	**4,037**	**7.3**

Source: ESM 2024

Table 16-6: Tons contained in the LOM plan by mineral resource class

Mineral Resource Class	Diluted Tons (kt)	Diluted Grade (% Zn)
Measured	121	17.2
Indicated	429	17.1
Inferred	1,613	12.4
Unclassified	1,874	0.0
Total	**4,037**	**7.3**

Source: ESM 2024

16.1.9 Mine Design Criteria

16.1.9.1 Mine Access

The ESM deposit consists of a mineral resource extending nearly 4,200 vertical feet. Multiple shafts extend from surface to the existing underground workings. Extensive UG workings exist from previous mining operations. Digitized UG surveys suggest there are more than 50 mi of development in the #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 17 ft tall. The maximum gradient of the existing development is 20%.

ESM is situated on moderately flat lying terrain.

Existing workings are regularly 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.

16.1.10 Production Rate Selection

The ESM mine plan had been designed to ramp up to 1,400 t/d in Year 1 of production and then to a sustained maximum of 1,700 t/d. Ramp up was successfully completed. 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 version 2023.1 CAD and a manual scheduling method. The scheduling rates used are shown in Table 16-7.

Table 16-7: Rates used for mine scheduling

Scheduling Rates		
Lateral Development	**Unit**	**Rate**
Ramp	ft/day	4
Auxiliary	ft/day	4
Longitudinal Access – Waste	ft/day	4
Longitudinal Sill – Mineralization	ft/day	4
C&F Access – Waste	ft/day	4
PM Access – Waste	ft/day	4
Vertical		
Drop Raise	ft/day	5
Raiseboring	ft/day	9

Scheduling Rates		
Stoping		
LRS	t/day	350
Back-stope – Longhole	t/day	350
C&F	t/day	150
PM – Primary	t/day	250
PM – Secondary	t/day	100

Source: ESM 2024

16.1.11 Production Sequencing

Production in LRS zones is planned with a bottom-up sequence where necessary in situ sill pillars are left to separate mining horizons.

C&F zones are planned 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 crosscut slashed along the back and backfilled on the floor to allow access to the next level above, where the mining process is repeated.

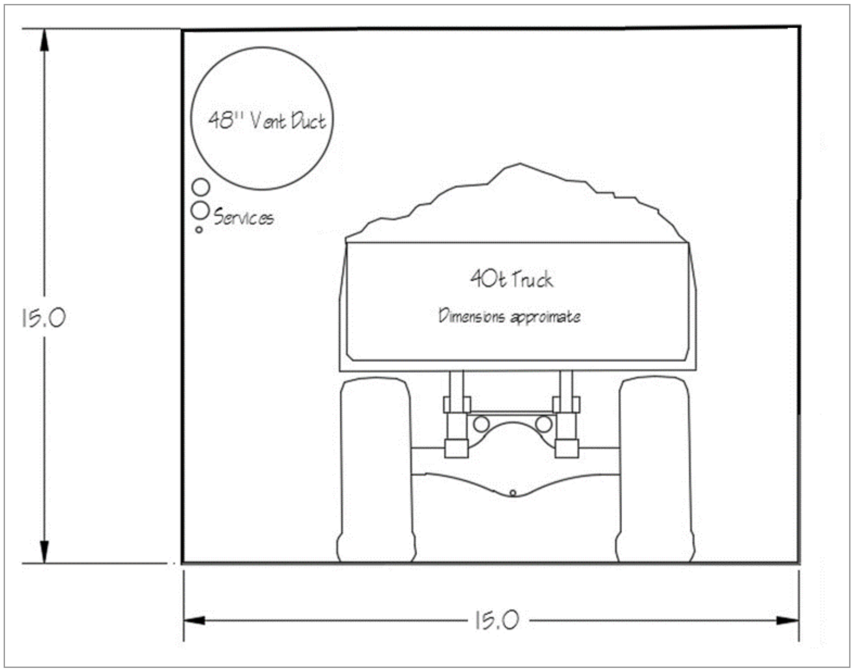
PM Primary and Secondary zones are planned from a top-down or bottom-up fashion depending on the direction of development in the zone. Access drifts are driven from the main ramp to the start of each Primary panel drift. A Primary drift is driven at full size to the end of the deposit. A Secondary slash in the hanging wall 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 and 45 t haul trucks and 48" round vent ducting. Crosscuts and sublevel development are driven at a 15 ft x 15 ft arched profile to accommodate truck access.

Figure 16-9 depicts a typical development section.



Source: Jackleg Consulting 2020

Figure 16-9: Typical development cross-sections

16.1.12.2 Vertical Development

Ventilation raises of varying lengths are used in the LOM design. For shorter, level to level connections, a 6 ft x 6 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 two 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 tractor for bulk explosives. Lifter holes are loaded with packaged emulsion for wet holes and prill ANFO for dry holes. Blasting is initiated by non-electric (NONEL) detonators.

For longhole production blasting, a combination of packaged emulsion and prill ANFO is used based on shot design with uni tronic™ detonators and 60 g boosters. Back-stopes are loaded using only packaged emulsion.

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. Typical ground support in access development is planned to consist of 5 ft or 6 ft split set bolts in the back and in the walls at a spacing of 4 ft x 4 ft. Welded wire mesh is installed in all ground conditions. In large intersections, cable bolts and/or #7 all thread bolts are installed, typically on a 6 ft x 6 ft pattern for deep ground support.

Cable bolts are installed into hanging walls prior to longhole stope firing as necessary.

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) LHDs directly to a haul truck, remuck bay, or material-pass. Broken material from LRS is mucked by remote control LHD.

16.1.13.5 Hauling

A fleet of 40 and 44 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. Some backfill is used in areas of LRS in place of sill pillars depending on geometry, grade, and geotechnical conditions. 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

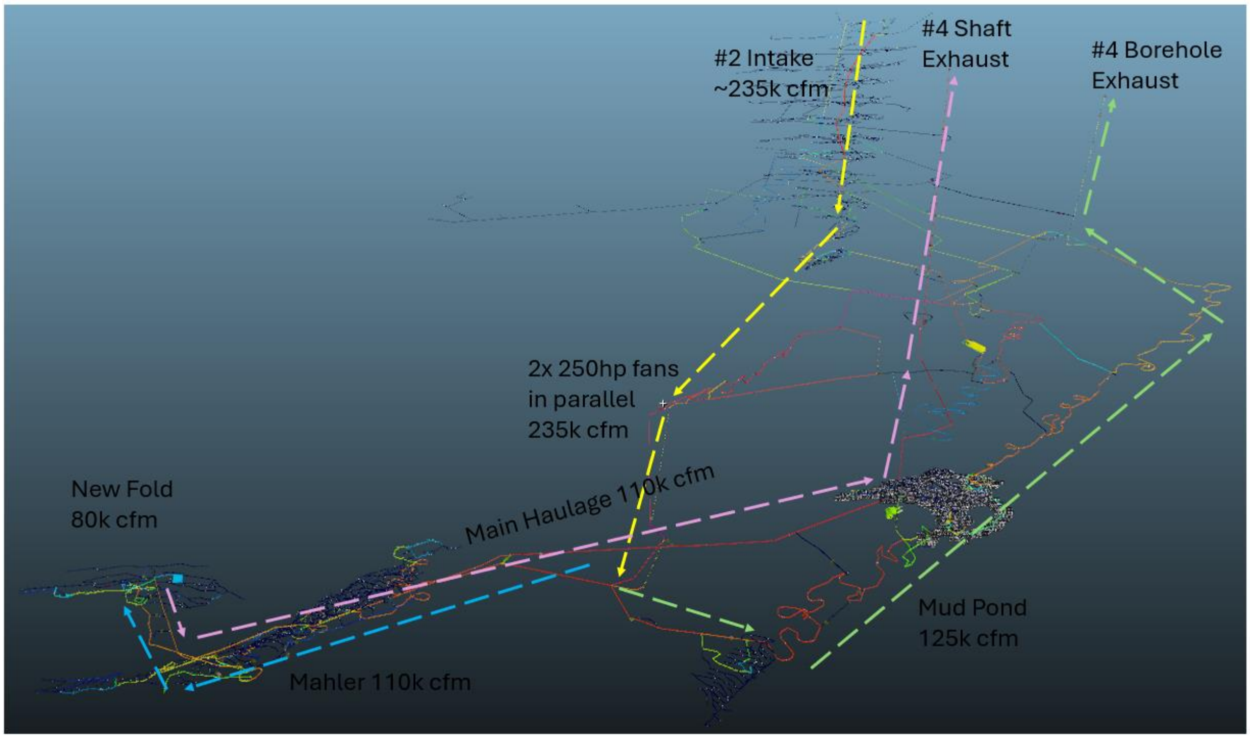
In 2016, the ESM ventilation network was modeled using Ventsim® Visual software by Practical Mining LLC (Practical Mining). The ventilation simulation model is routinely calibrated, verified and updated as mine activity changes.

Minimum airflow requirements are based on expected diesel emissions of the UG mining fleet required at peak mine production. Additional airflow is used underground to improve air quality. The power rating of each piece of equipment was determined, and the utilization factors representing the equipment in use at any time, were applied to estimate the amount of air required. The volume of air determined to ventilate the diesel emissions is 212 kcfm.

The generalized strategy for ventilating the ESM mine is to use the stopes and associated workings near the #2 Shaft as intake. Air is exhausted through the #4 Shaft and #4 Borehole. The #2 Shaft exhausts a minor amount for temperature control. Approximately 5% losses to unknown connections to surface through the #2 Mine are routinely measured.

On the 3500 level, two parallel 200 hp Alphair Primary fans draw air from the surface supply and send 235 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 50 hp to 150 hp ventilation fans in New Fold and Mahler (Figure 16-10).



Source: ESM 2024

Figure 16-10: LOM ventilation installations

16.1.14.2 Mine Air Heating

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

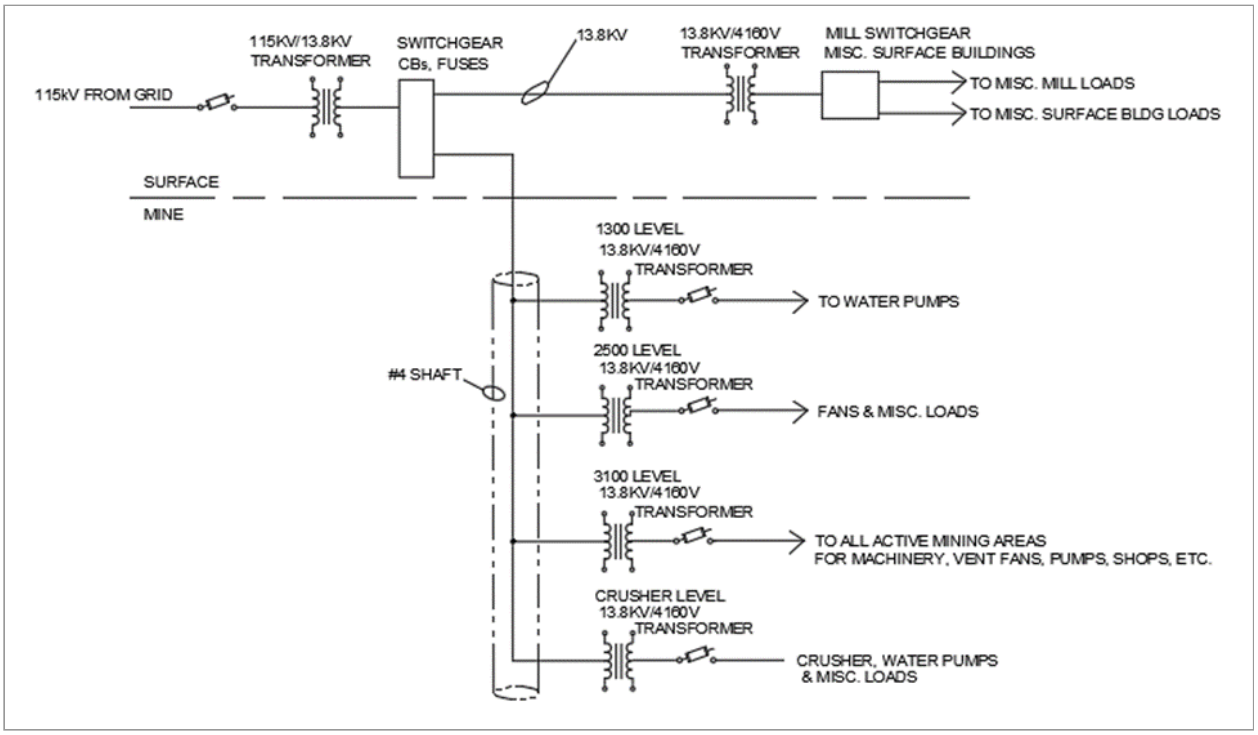
16.1.14.3 Electrical Power

Most of the electrical power consumption at the mine 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 substations near the mining zones. Power is delivered at 13.8 kV and reduced to 480 V at electrical substations.

Total electrical power consumption for UG mining is estimated at 2.4 MW during operations. The site elementary electrical one-line diagram is shown in Figure 16-11.



Source: ESM 2024

Figure 16-11: Site elementary electrical one-line diagram

16.1.14.4 Compressed Air

Compressed air is required for longhole drills, jacklegs, jumbos, bolters, bulk explosives tractor, 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 2" 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 #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 level in the connected #3 Mine, which prevents the majority of ground water from entering the mine and descending to the bottom at 3100 level (#3 Mine is the defunct sister mine to the #4 Mine and there are several points of where they join). Any small quantities encountered are picked up at the 1300 sump.

The mine neighbors a talc operation, which hosts a flooded pit, the Arnold Pit. There is an excavation connecting ESM Property and the Arnold Pit. ESM 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, 2005b).

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 have been 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 is 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 off site at the blasting contractor facility across the road from the mine entrance. Secondary magazines are located underground to provide explosives storage for up to 7 days. Explosives and detonators are stored in separate magazines 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, UG operations of this rock type require powder factors of approximately 1.9 lb/t for development and 0.7 lb/t for LRS with good fragmentation.

16.1.14.8 Fuel Storage and Distribution

Mobile equipment is re-fueled at UG fueling 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 use Kubota tractors and UTVs as transportation underground. A boom truck, flat deck truck and forklift are used to transport supplies and consumables from the #4 Shaft station to active UG workplaces.

16.1.15 Underground Mine Equipment

The required UG 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	Onsite
Drill Jumbo – 2-Boom – Sandvik Axera	1
Drill Jumbo – 2-Boom – Epiroc Boomer 282	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	2
Bolter – Epiroc Boltech S	1
LHD (10 t/ 6 yd) Atlas Copco ST 1030	1
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	1
LHD (3 t/2.5 yd) MTI 270	1
Haulage Truck – 40 Ton – Tamrock 40 D	3
Haulage Truck – 42 Ton – Epiroc MT 42	2
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
Utility Vehicles - Kubota RTV 900	3
Tractors – Kubota L2500/L2800/L3301	29
Jacklegs / Stopers	43

Source: ESM 2024

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 is performed with a Secoma Pluton bolter or an Epiroc Boltech bolter in addition to jacklegs working off muck piles and/or scissor decks.

Two Boart Longyear longhole drills are used for longhole production stoping.

16.1.15.1 Mine Equipment Maintenance

Mobile UG equipment is maintained at the existing UG mine shops. 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 the 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 1,700 t/d. The normal production schedule is two 10-hour shifts, 5 days per week, with no operations on Saturday and Sunday. This allows a 2-hour pause between shifts to clear blast gases from the mine. In general, blasting only occurs during day shift.

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 labor force quantities, and rotation schedules.

Table 16-9: Mine personnel summary

Position	Roster	Rotation	LOM Average
Mining Management			
Mine Superintendent	Salary	5 x 2	1
Subtotal – Mining Management			**1**
Mining Operations			
Shift Supervisor	Hourly	5 x 2	2
Lead Miner	Hourly	5 x 2	4
Miner 1 (Jumbos, Bolters)	Hourly	5 x 2	16
Miner 2 (Jackleg Bolters, LH drillers, Blasters)	Hourly	5 x 2	11
Miner 3 (Loader & Truck Operators)	Hourly	5 x 2	13
Miner 4 (Services, Equipment Operators)	Hourly	5 x 2	14
Subtotal – Mining Operations			**60**
Crushing and Hoisting			
Hoistman	Hourly	5 x 2	3
Lead Shaft Miner	Hourly	5 x 2	1
Shaft Miner	Hourly	5 x 2	5
Subtotal – Crushing & Hoisting			**9**

Position	Roster	Rotation	LOM Average
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	3
Heavy Duty Mechanic	Hourly	5 x 2	14
Electrician	Hourly	5 x 2	3
Subtotal – Mine Maintenance			**24**
Mining Technical Services			
Technical Services Manager	Staff	5 x 2	1
Mine Engineer	Staff	5 x 2	2
Junior Mine Engineer	Staff	5 x 2	1
Project Engineer	Staff	5 x 2	1
Surveyor	Staff	5 x 2	1
Chief Geologist/Engineer	Staff	5 x 2	1
Geologist	Staff	5 x 2	2
Subtotal Technical Services			**9**
Grand Total			**103**

Source: ESM 2024

16.1.17 Mine Production Schedule

Mine scheduling for the ESM project was done internally. The schedule seeks to produce consistent pound of zinc from 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. A swell factor of 30% is assumed for calculating loose waste rock volumes.

Annual mine production statistics from 2025 are provided in Table 16-10. Annual production statistics for 2024 are included in Table 16-11.

Table 16-10: Annual mineralized material by mining zone

Item	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033
Tons Mined (kt)	**4,042**	462	467	455	455	455	455	455	455	383
Zinc Grade	**7.3**	7.8	7.5	7.3	7.3	7.3	7.3	7.3	6.5	7.3
Contained Zinc (M lb)	**589.1**	72.3	70.3	66.3	66.3	66.3	66.3	66.3	59.2	55.8

Source: ESM 2024

The 2024 mine production was estimated from actual production and short-range projections.

Table 16-11: Projected production for 2024

Item	Total	2024
Tons Mined (kt)	**425**	425
Zinc Grade	**8.6**	8.6
Contained Zinc (M lb)	**73.2**	73.2

Source: ESM 2024

16.1.18 Mine Development Schedule

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

Annual development footages are summarized in Table 16-12.

Table 16-12: Annual development schedule

Development Schedule	Unit	Total	2025	2026	2027	2028	2029	2030	2031	2032
Total Waste Development	ft	**55,997**	7,754	12,202	10,241	9,147	4,163	4,163	4,163	4,163

Source: ESM 2024

16.2 Open Pit

16.2.1 Hydrological Parameters

In 2021, Alpha Geoscience investigated the potential hydrogeological impacts of the proposed Hoist House and Pump House pits. This work was part of a mining permit modification application. The study found the following:

- Anticipated drawdown will not impact nearby residential supply wells or wetlands.
- The pits will drain into existing workings and therefore no pumping should be necessary to maintain a dry floor.
- ESM's existing dewatering system can accommodate additional water flow from ground water and precipitation.
- ESM's existing Water Withdrawal Permit can accommodate the additional flow.

In 2025, Alpha will reassess the hydrogeological conditions in relation to the 2024 pit designs mentioned here-in and relative to the same criteria listed above.

16.2.2 Open Pit Geotechnical Considerations

Knight Piésold provided a study dated May 15, 2020, "Empire State Mine Scoping Level Pit Slope Design" (Blackwell & Peacock, 2020) in which the pit slope recommendations were given (Table 16-13 and Table 16-14). The pit designs, in the Blackwell & Peacock (2020) report, are based on a previous block model. For this study, the slope angle was generalized by modeled lithology from the Blackwell & Peacock (2020) report pending further geotechnical work (Table 16-14).

Table 16-13: Knight Piésold 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)[3]	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 raveling 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.

Source: Knight Piésold 2020 (Blackwell & Peacock, 2020)

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.

(3) Effective bench heights based on 20 ft benches in a double-bench configuration.

Table 16-14: Generalized slope angles for pit optimization and design

Open Pit Design Lithology	Nominal Inter-ramp Angle (°)
UM14, UM15	50
UM13	35
UM8, UM9, UM10	50
UM11, UM12	44
null	50
OVB, FILL	32

Source: ESM 2024

Knight Piesold's report (Blackwell & Peacock, 2020) is scoping level and recommends further data collection, particularly regarding characterizations of structural features.

16.2.3 Mineral Resource Model for Mining

The 2024 ESM internal block model estimate for Turnpike was used 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 ESM and are based on their current smelter contract. The NSR cut-off value is based on the assumptions shown in Table 16-15.

Table 16-15: Cut-off value assumptions

Mining Factors	Unit	Open Pit
Mining Dilution	%	10
Mining Recovery	%	100
Operating Costs		
Mining Cost for Mineralization	US$/t	4.60
Mining Cost for Waste	US$/t	3.50
Mining Cost for Overburden	US$/t	2.00
Processing Cost for Mineralization	US$/t	11.00
G&A Cost for Mineralization	US$/t	0.00

Mining Factors	Unit	Open Pit
Processing Recovery		
Zinc	%	96
Revenue		
Payable Zinc	%	85
Zinc Price	$/lb	1.27
Transportation Cost	US$/t con	50
Selling Cost	US$/t con	0
Cut-off Grade	% Zn	0.6

Source: BBA 2024 and ESM 2024

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 taken during blasting and rigorous mineralization control and monitoring systems are followed, it is estimated that dilution and mineralization losses can be minimized.

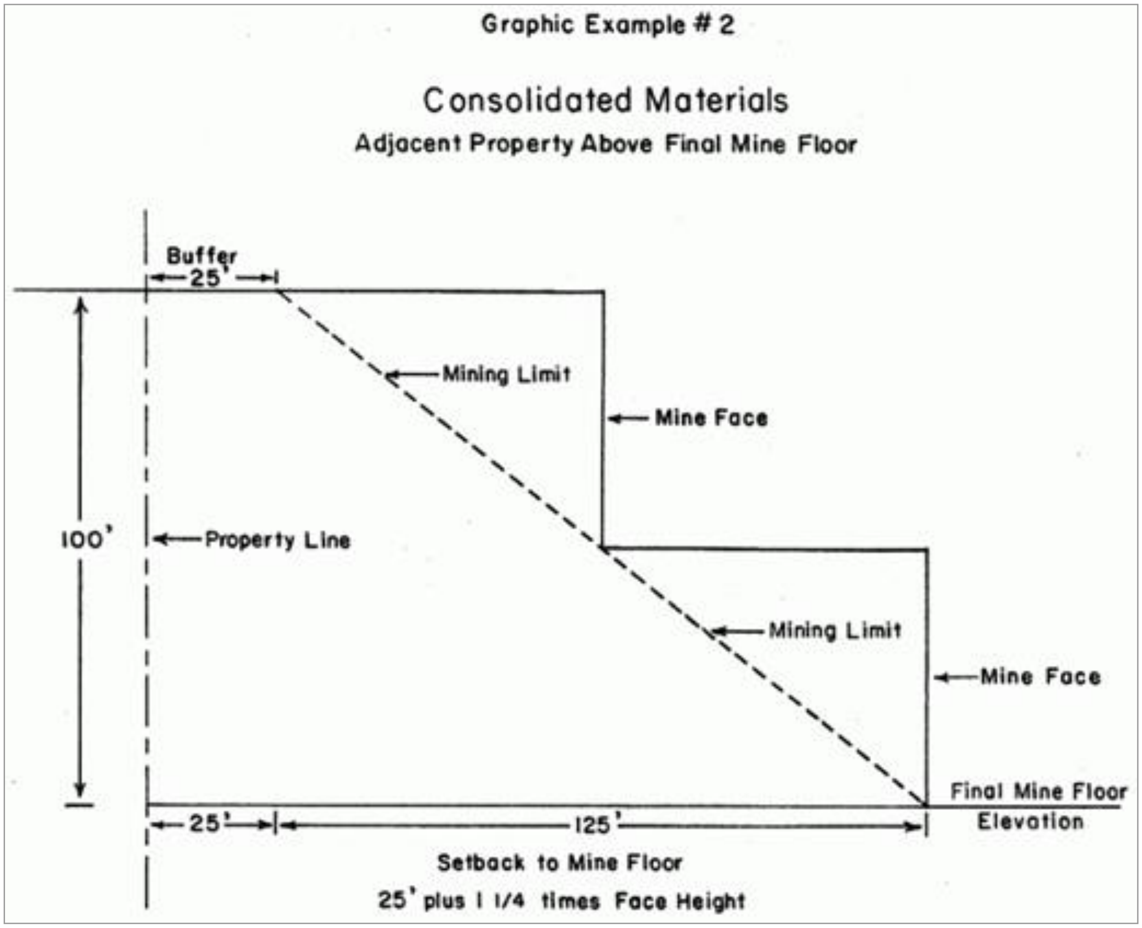
Mining recovery and dilution were accounted for by using a regularized block model. The estimated 10% dilution was not applied within the pit optimization, only to the cut-off grade calculation.

16.2.6 Pit Optimization and Selection

The Lerchs-Grossmann pit optimization algorithm was used to define the ultimate pit shell for the Turnpike area. The selected pit shells were then used to produce pit designs and the open pit mining schedule. BBA completed the pit optimization, pit design, and mine schedule based on inputs from ESM.

Further UG mining is not planned under the open pit zones. The block model was depleted of the existing UG workings. Pit optimization did not therefore consider any further influence from UG mining (Figure 16-13 and Figure 16-14).

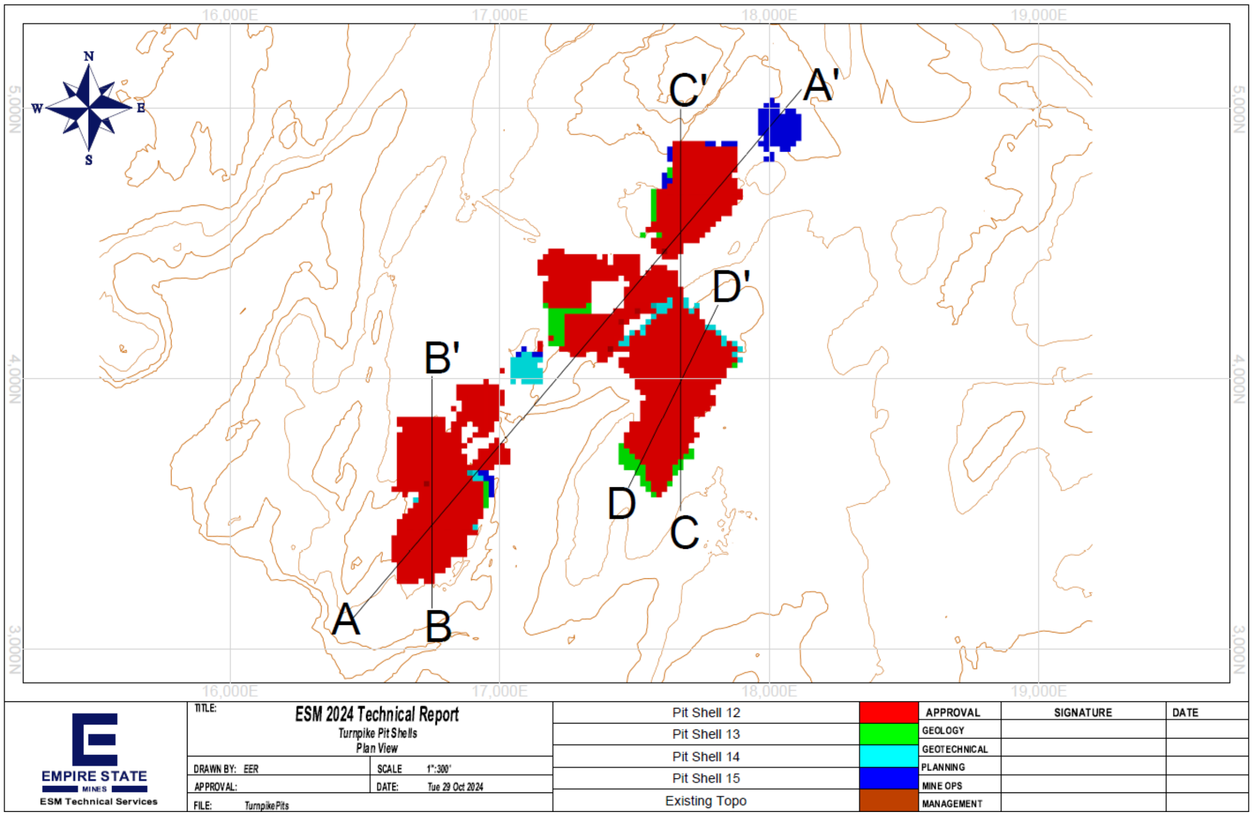
The pit area lies close to houses and residents along the east side of the Turnpike zone and outside of ESM's Property. As well, New York regulations stipulate an offset and slope cone from the Property boundary to the toe of any excavation. The offset is 25' and the slope from the pit crest to toe is 1:1.25 (38.66°). This 'no-go' limit was modeled in the software as an exclusion boundary past which the blocks could not be mined (Figure 16-12).



Source: ESM

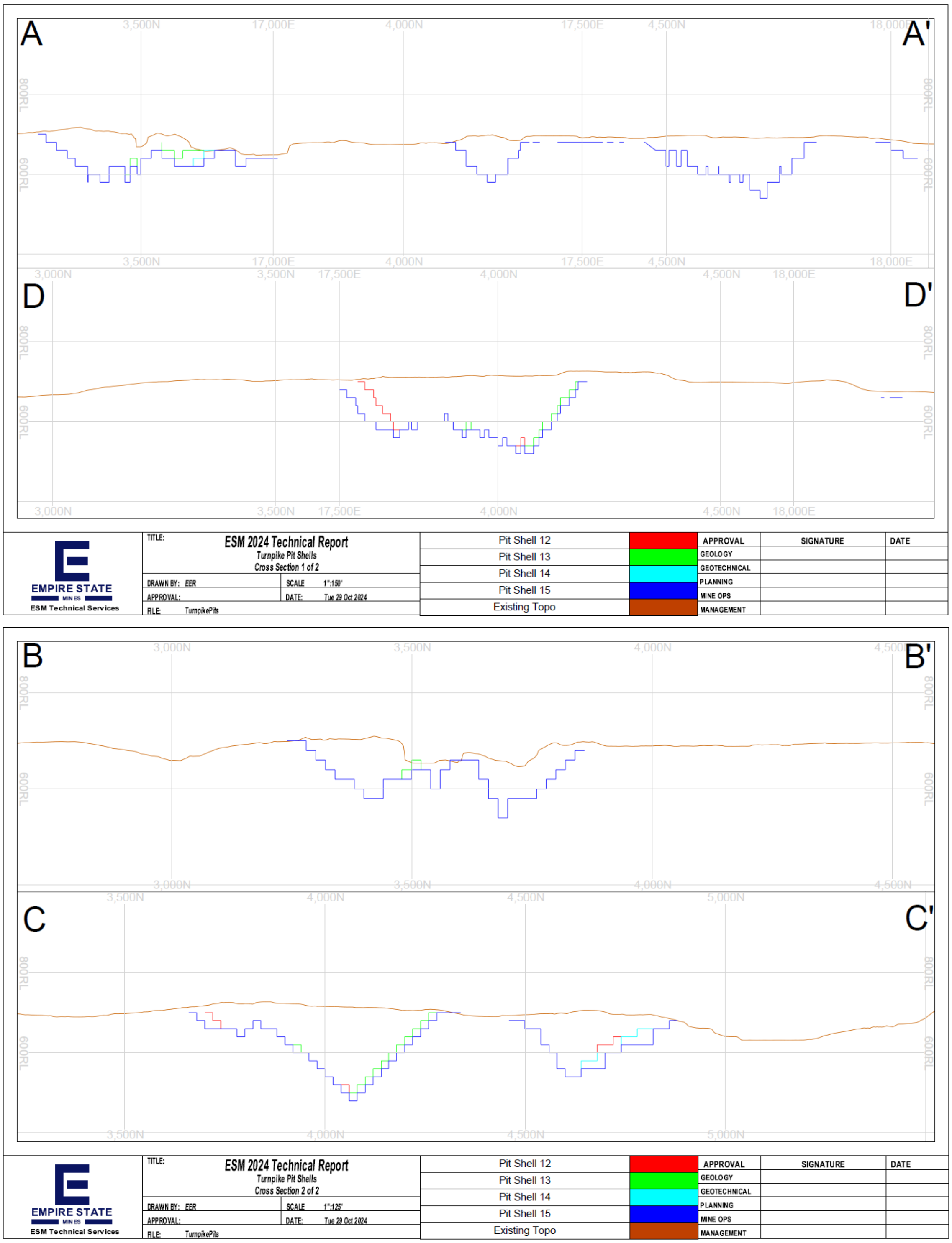
Figure 16-12: Permitting exclusion cone

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 ESM, various mineralization costs were considered to be covered by the current and future UG operations. Therefore, the applied costs did not include G&A, and the mineralized tonnage was treated as incremental for the purposes of processing costs.



Source: BBA 2024 and ESM 2024

Figure 16-13: Plan view optimization shells (with cross-section locations)



Source: BBA 2024 and ESM 2024

Figure 16-14: Cross-section views

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

Table 16-16: Pit shell optimization results

RF Pit Shell	Revenue Factor	Zn Price (USD/ lb)	Total Mill Feed (t)	Zn (%)	Total Waste (t)	Incremental Strip Ratio (W:O)	Overall Strip Ratio (W:O)	LOM (year)	Best Case DCF	Worst Case DCF	Average Case DCF	Percent of Max NPV: Best Case	Percent of Max NPV: Worst Case	Percent of Max NPV: Average Case
1	0.30	0.38	164,000	4.92	164,000	1.00	1.00	0.3	$ 12,705,649	$ 12,705,649	$ 12,705,649	49%	54%	52%
2	0.35	0.44	238,000	4.30	246,000	1.11	1.03	0.5	$ 15,527,446	$ 15,454,589	$ 15,491,018	60%	66%	63%
3	0.40	0.51	341,000	3.96	496,000	2.43	1.45	0.7	$ 19,382,567	$ 19,114,644	$ 19,248,606	76%	81%	79%
4	0.45	0.57	388,000	3.77	560,000	1.36	1.44	0.8	$ 20,599,615	$ 20,262,870	$ 20,431,242	80%	86%	83%
5	0.50	0.64	443,000	3.59	676,000	2.11	1.53	0.9	$ 21,862,385	$ 21,346,272	$ 21,604,328	85%	91%	88%
6	0.55	0.70	463,000	3.54	739,000	3.15	1.60	0.9	$ 22,276,608	$ 21,697,624	$ 21,987,116	87%	92%	90%
7	0.60	0.76	504,000	3.45	889,000	3.66	1.76	1.0	$ 23,038,970	$ 22,253,750	$ 22,646,360	90%	95%	92%
8	0.65	0.83	533,000	3.38	1,007,000	4.07	1.89	1.1	$ 23,476,771	$ 22,557,463	$ 23,017,117	91%	96%	94%
9	0.70	0.89	583,000	3.26	1,183,000	3.52	2.03	1.2	$ 24,050,864	$ 22,912,049	$ 23,481,456	94%	98%	96%
10	0.75	0.95	668,000	3.15	1,661,000	5.62	2.49	1.3	$ 24,939,339	$ 23,358,731	$ 24,149,035	97%	100%	99%
11	0.80	1.02	707,000	3.10	1,883,000	5.69	2.66	1.4	$ 25,299,582	$ 23,460,152	$ 24,379,867	99%	100%	100%
12	0.85	1.08	741,000	3.03	2,006,000	3.62	2.71	1.5	$ 25,455,937	$ 23,461,113	$ 24,458,525	99%	100%	100%
13	0.90	1.14	781,000	2.98	2,250,000	6.10	2.88	1.6	$ 25,620,174	$ 23,348,274	$ 24,484,224	100%	100%	100%
14	0.95	1.21	798,000	2.97	2,375,000	7.35	2.98	1.6	$ 25,661,416	$ 23,251,423	$ 24,456,419	100%	99%	100%
15	1.00	1.27	807,000	2.96	2,462,000	9.67	3.05	1.6	$ 25,670,246	$ 23,138,672	$ 24,404,459	100%	99%	100%
16	1.05	1.33	812,000	2.95	2,500,000	7.60	3.08	1.6	$ 25,666,382	$ 23,097,622	$ 24,382,002	100%	98%	100%
17	1.10	1.40	820,000	2.95	2,595,000	11.88	3.16	1.6	$ 25,636,672	$ 22,968,422	$ 24,302,547	100%	98%	99%
18	1.15	1.46	865,000	2.89	2,950,000	7.89	3.41	1.7	$ 25,472,400	$ 22,432,752	$ 23,952,576	99%	96%	98%
19	1.20	1.52	867,000	2.89	2,969,000	9.50	3.42	1.7	$ 25,461,871	$ 22,403,558	$ 23,932,715	99%	95%	98%

Notes:

Values are expressed in US dollars
Numbers may not add due to rounding.
Inferred resources above 0.6% Zn.
Revenue factor is applied to base case price starting 0.3 to1.2 (0.05RF intervals).
No extra mining dilution and mining recovery are included as regularized BM is used.
Based on the resource block model provided "'N2S_Whittle_Fin_Reg.dm (Deswik)".

Based on topography surface "N2S_20230905_TOPOGRAPHY_DXF.dm" .
Constraints:
- 25ft offset inside the property hard boundary provided "PROPERTY_25FT IN_BND.dm"
- 25ft offset outside the Shaft Building provided"'SHAFT_BUILDINGS_BUFFER25FT_BND.dm"
- Wetland boundary including buffer provided "WETLAND+BUFFER_BND.dm"

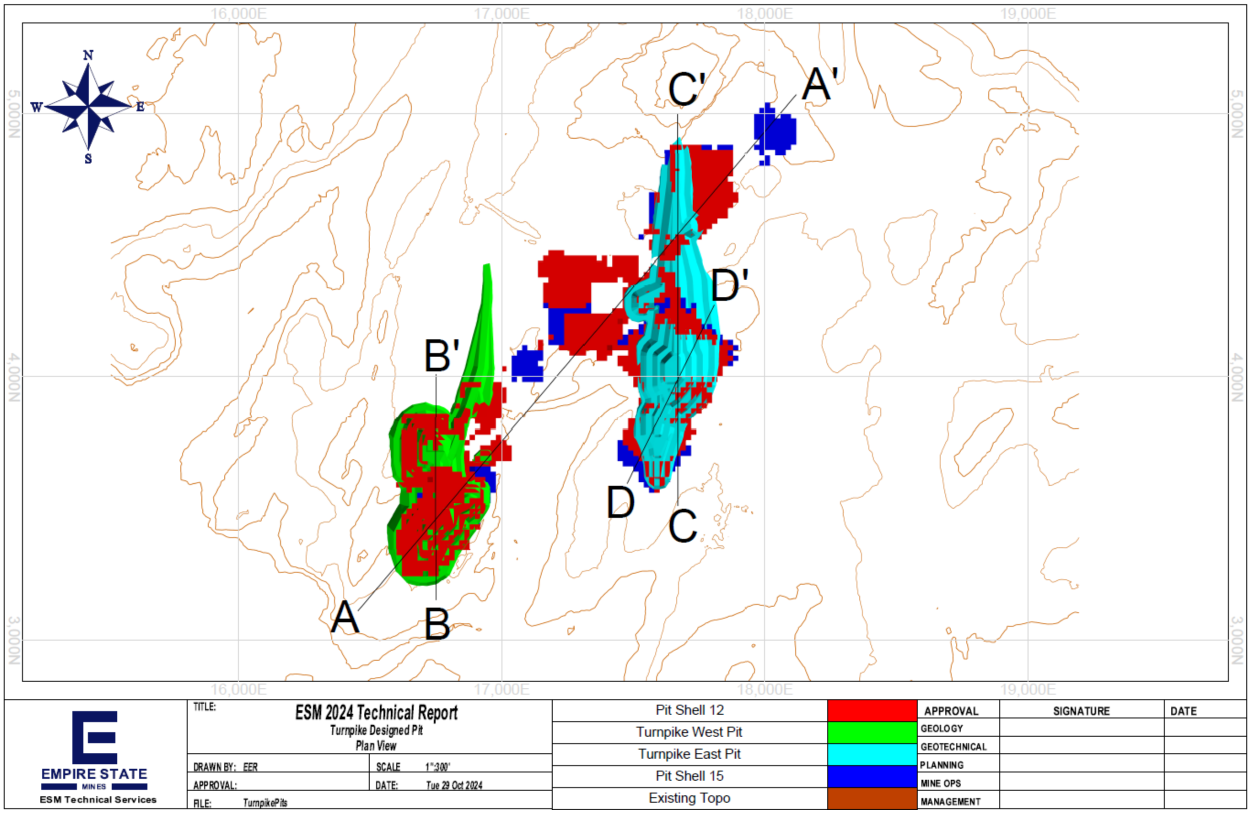
Pit shells are incremental. Deswik generates a solid per phase.

Source: BBA 2024

16.2.7 Pit Design

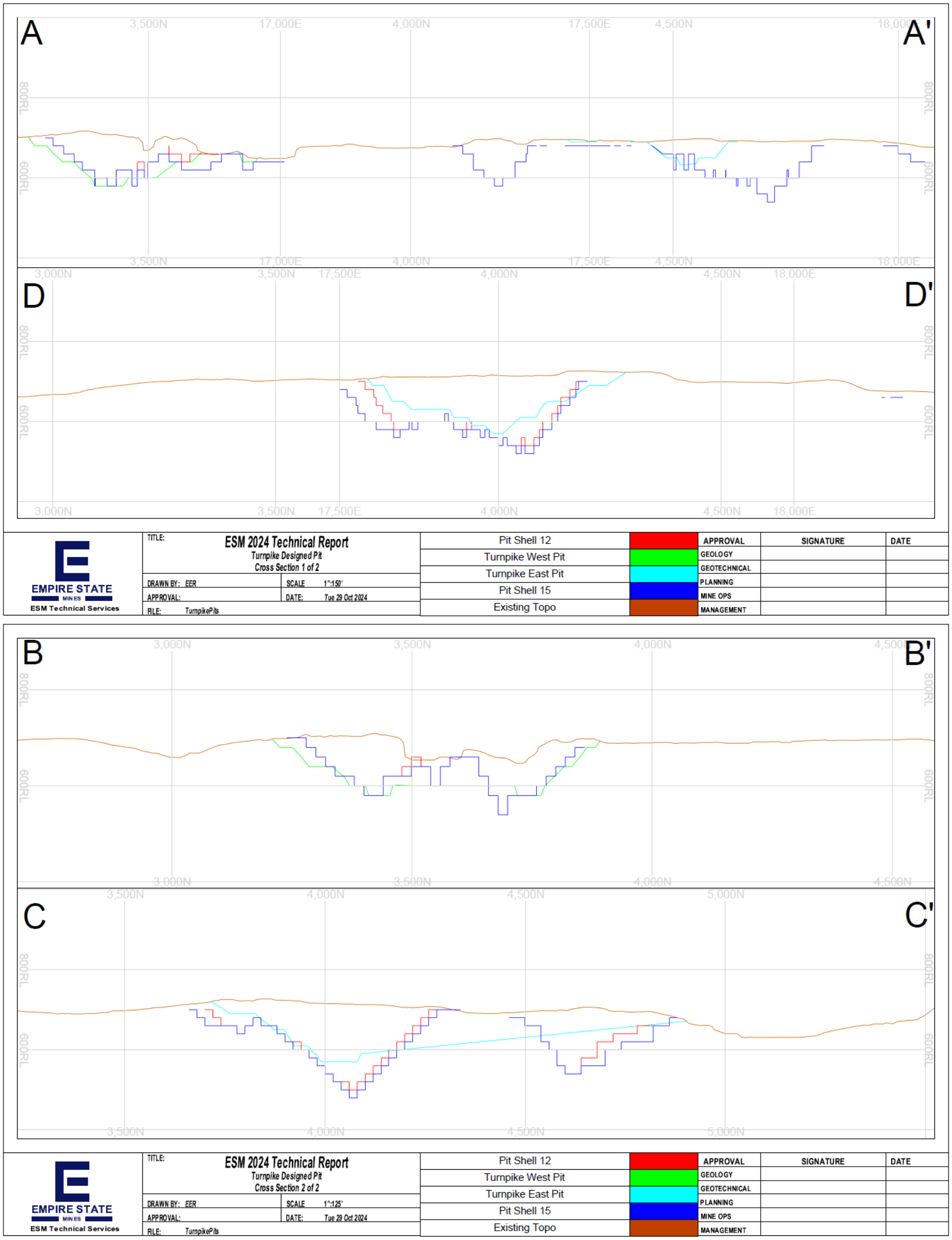
Conceptual pits were designed based on the selected pit optimization shell as described above. Design criteria were (Figure 16-15 and Figure 16-16):

- Single lane 25 ft wide up to 12% grade;
- Pit slopes as per geotechnical guidelines;
- Bench access maintained on one side of ramp (pits and dumps). i.e., benches not pinched off on both sides.



Source: BBA 2024 and ESM 2024

Figure 16-15: Open pit designs



Source: BBA 2024 and ESM 2024

Figure 16-16: Cross-section of design and shell

Indicative tons and diluted grades contained within the conceptual pit designs (Figure 16-17) are presented in Table 16-17.

Table 16-17: Open pit projected tons and grades

Zone	Mineralized Material (tons)	Zn (%)	Strip Ratio
Turnpike West Pit	199,621	3.37	3.10
Turnpike East Pit	199,310	2.97	3.80
Total	**398,931**	**3.17**	**3.40**
Total Waste	**1,364,423**	**-**	**-**

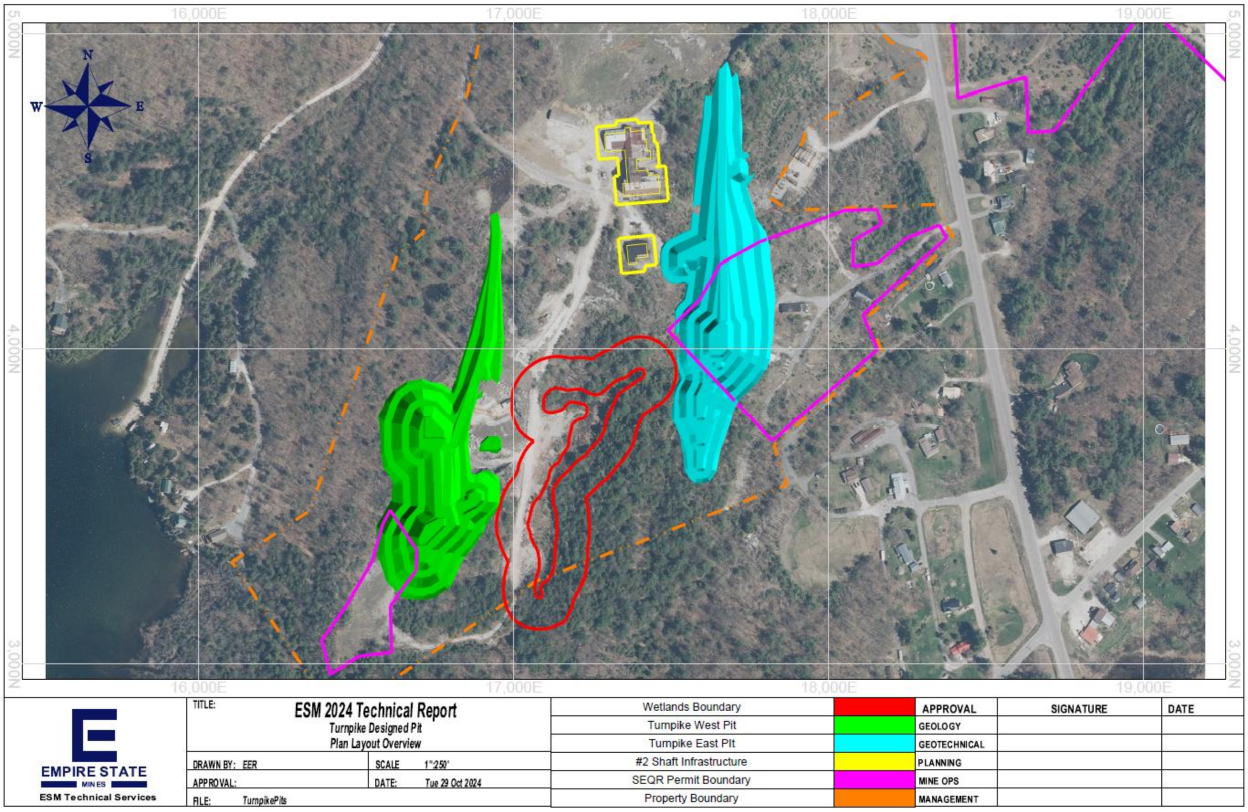
Source: BBA 2024

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 waste dump. ESM has acquired the Property and right of way. A short, direct haul road will connect the pits with the dump.

A portion of the haul road follows an existing rail line right of way. The line is no longer used for rail cars and was ideally located for hauling mineralization to the mill. The haul route crosses two public roads. ESM will install additional safety features in those locations to ensure safe access for the public.

The existing ESM underground mine uses the #2 Shaft as a secondary escape egress route for evacuation of personnel in an emergency. The collar of this shaft is located between the East Pit and West Pit. The head frame and other facilities at that location will not be impacted by the pit excavations. At this time, the surface accessible resource does not support constructing a second shaft to serve as an alternate escape route for ESM #4 Mine.



Source: ESM 2024

Figure 16-17: Layout of open pit

16.2.8 Mining Method

It is proposed to mine the open pits using conventional truck and loader mining methods. A mining contractor operation is presumed. 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 ft spacing). The loader could typically work on a temporary bench and load trucks on that same bench. Due to the small pit sizes, none of the pits are phased. The pits are sequenced in the schedule. The pits will alternate to manage total material movement in a 6-month period.

16.2.8.1 Drill and Blast

The proposed drilling parameters for 20 ft bench heights are presented in Table 16-18. 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 could be straight ANFO, emulsion, or ANFO blends. Drilling and explosive supply including loading and shooting, are assumed to be provided by contractors.

Table 16-18: Open pit drilling parameters

Parameter	Unit	Value
Bench Height	ft	20
Burden	ft	11.5
Spacing (Equilateral Triangle)	ft	13.3
Hole Size	inch	5.12
Collar	ft	7.25
Subdrill	ft	2.5
Explosive Density	g/cm^3	0.8
Rock Density	t/ft^3	0.09
Powder Factor	lb/t	0.46

Source: ESM 2024

Due to the projected short life of the open pit mines and the shallow mining depth, it is assumed that presplit blasting will not be required.

Assuming 10% redrill, 59 ft/h penetration rate, 75% mechanical availability and 90% utilization, and 3,130 h/y, one drill is required to meet production. The drill will be underutilized. Mechanical down time will not increase the requirement to two drills.

16.2.8.2 Load and Haul

Two front-end loaders equipped with 5.9 yd^3 (or 4.5 m^3) buckets (similar to CAT 930 machines) would be required to mine waste and mineralized material. They would load into a fleet of 40 t road trucks (such as Mercedes Actros) or articulated dump trucks (e.g., CAT 740 ADT). Waste hauls are short (approximately 0.65 mi) while hauls for mineralization are longer (approximately 1.5 mi). Overall, annual front-end loader productivity is estimated at approximately 350 t/h and trucks at 130 t/h in mineralization and 170 t/h in waste. Front-end loaders and trucks have been estimated to operate 3,130 h/y. Three trucks should be adequate to meet production. One front-end loader with two trucks could stay permanently in waste. The second front-end loader with one truck could work exclusively in mineralization.

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 run of mine (ROM) stockpile into the crusher. ESM 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 6-day, 10 h/d roster. One shift (day) is planned. Therefore 60 h/week are scheduled over 52 weeks per year for 3,130 hours/year.

Based on the production schedule (Table 16-21), roster schedule, and equipment productivity estimates, the required equipment list is as shown in Table 16-19.

Table 16-19: Equipment estimate

Equipment	Y1	Y2	Y3 (partial)
Trucks	3	3	3
Loaders	2	2	2
Drills	1	1	1
Graders	1	1	1
Water trucks	1	1	1
Dozers	1	1	1
Pickups	1	1	1

Source: ESM 2024

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 Labor 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, and survey. Numbers include a small supplement to account for redundancy in case of absenteeism, training etc.

The open pit contractor labor estimate is provided in Table 16-20.

Table 16-20: Open pit labor and supervision

Labor	Y1	Y2	Y3 (partial)
Mine Foreman	1	1	1
Drill Operator	1	1	1
Drill Helper	1	1	1
Blaster	1	1	1
Blaster Helper	2	2	1
Loader Operator	2	2	2
Haul Truck Operator	4	4	3
Dozer Operator	1	1	1
Water Truck Operator	1	1	
Grader Operator	1	1	1
Mine Laborer	2	2	1
Mine Maintenance Foreman	1	1	1
Mechanic	1	1	1
Mechanic Heavy Equipment	2	2	2
Electrician	1	1	1
Serviceman	1	1	1
Maintenance Laborer	2	2	1
Total	**25**	**25**	**20**

Source: ESM 2024

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 16-21.

Table 16-21: Conceptual open pit production schedule

Description	Unit	H1Y1	H2Y1	H1Y2	H2Y2	H1Y3	Total
East Pit							
Mill Feed	t	651	35,296	19,520	59,771	84,073	**199,310**
Zn	%	2.3	2.1	2.2	2.6	3.8	**3.0**
Overburden	t	27	42,206	31,927	43,589	2,708	**120,456**
Waste	t	312	151,169	95,753	231,556	154,909	**633,698**
Total Material Movement	t	990	228,671	147,199	334,915	241,689	**953,464**
Stripping Ratio	W:O	0.5	5.5	6.5	4.6	1.9	**3.8**
Zn recovered	lb	28,373	1,429,912	835,768	2,937,864	6,133,966	**11,365,453**
Zn payable	lb	24,117	1,215,425	710,403	2,497,185	5,213,871	**9,660,635**
West Pit							
Mill Feed	t	44,208	39,704	80,480	35,229	0	**199,621**
Zn	%	2.6	3.6	3.6	3.5	0.0	**3.4**
Overburden	t	31,320	10,003	316	0	0	**41,639**
Waste	t	223,482	121,623	172,005	51,521	0	**568,630**
Total Material Movement	t	299,010	171,329	252,801	86,750	0	**809,890**
Stripping Ratio	W:O	5.8	3.3	2.1	1.5	0.0	**3.1**
Zn recovered	lb	2,189,887	2,774,833	5,593,682	2,353,861	0	**12,916,277**
Zn payable	lb	1,861,404	2,358,608	4,754,630	2,000,782	0	**10,978,836**
Total							
Mill Feed	t	44,859	75,000	100,000	95,000	84,073	**398,931**
Zn	%	2.6	2.9	3.4	2.9	3.8	**3.2**
Overburden	t	31,347	52,208	32,242	43,589	2,708	**162,094**
Waste	t	223,794	272,792	267,758	283,077	154,909	**1,202,329**
Total Material Movement	t	300,000	400,000	400,000	421,665	241,689	**1,763,354**
Stripping Ratio	W:O	5.7	4.3	3.0	3.4	1.9	**3.4**
Zn Recovered	lb	2,222,135	4,204,800	6,432,000	5,289,600	6,133,966	**24,280,536**
Zn Payable	lb	1,888,815	3,574,080	5,467,200	4,496,160	5,213,871	**20,638,456**

Source: BBA 2024

17. Recovery Methods

Chapters 16 to 22 apply only to ESM's zinc operations. An economic analysis of ESM's graphite mineralization has not yet been completed.

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.

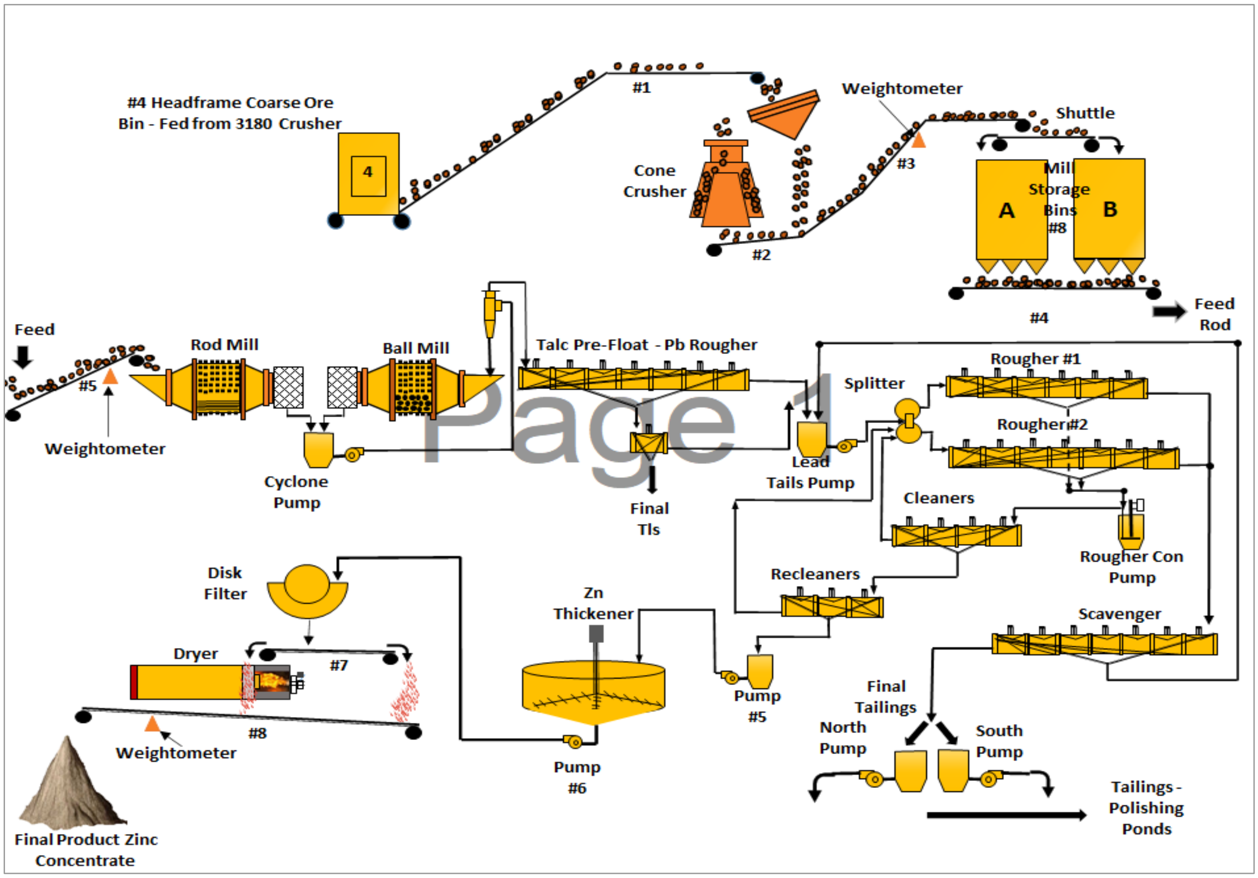
The design capacity of the concentrator is 5,000 t/d. Throughout the history of the Balmat operation (now ESM), the capacity of the concentrator has exceeded that of the mines' capacity. The operating strategy is to operate the concentrator at its rated hourly throughput of 200 t/h to 220 t/h, but for only as many hours as necessary to suit mine production. It currently is processing between 8,500 t and 8,750 t per week operating on a schedule of one shift per day, 4 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 restarting the concentrator follow below.

17.2 Plant Design Criteria

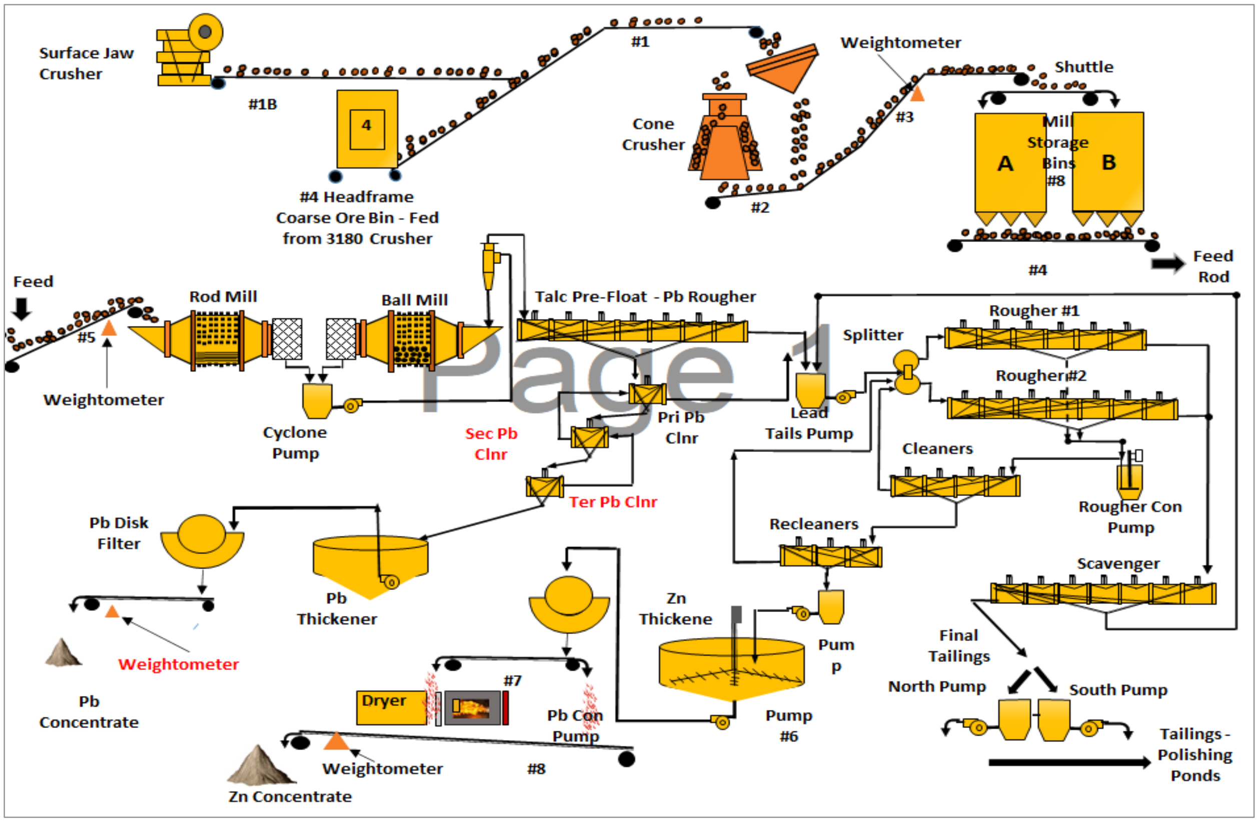
From a metallurgical perspective, the optimal way to operate a concentrator is on a continuous basis to minimize the usual occurrences of sub-standard metallurgy on start-up and product losses on shutdown.

While the mill has a capacity of 5,000 t/d, the underground mine production is typically no more than 1,750 t/d. The mill is operated for 10 to 12 hours per day. This inherently introduces some amount of instability during start-up and shutdown.



Source: ESM 2020

Figure 17-1: Concentrator flowsheet current state



Source: ESM 2020

Figure 17-2: Concentrator flowsheet with Pb circuit

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 mineralized material bins. An automatic sampler is installed on this belt. Discharge from the #3 conveyor is distributed between the two fine mineralized material bins by a shuttle conveyor. Each fine mineralized material 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	Unit	Value
Crushing Circuit Operating Time	h/d	10–12
Crushing Circuit Operating Time	d/w	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 Mineralized Material 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 mineralized material 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 was 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	Unit	Value
Grinding Circuit Operating Time	h/d	10–12
Grinding Circuit Operating Time	d/w	4–5
Design Throughput	t/h	200
ESM Mill Feed Material Work Index	kWh/t	8.3
Rod Mill Diameter	ft	11.5
Rod Mill Length	ft	16
Installed Motor on Rod Mill	hp	1,000
Required Power on Rod Mill	hp	1,000
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	1,000
Required Power on Ball Mill	hp	1,000
Grinding Ball Size	in	2
Estimated Charge Volume	%	34
Ball Mill Feed Size, 80% Passing	µm	1,000
Cyclone Diameter	in	26
Number of Operating Cyclones	qty	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 recommissioning by Hudson Bay Mining and Smelting Co.

The UG 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 17-4 and Table 17-5, respectively.

Table 17-3: Zinc rougher / scavenger flotation circuit design criteria

Design Criteria – Zinc Roughers	Unit	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/min	1,940
Existing Zinc Rougher Cells		
▪ Type (Wemco self-aspirated)		
▪ Individual Cell Size	ft^3	300
▪ Number of Cells	qty	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/min	640
Existing Zinc Scavenger Cells		
▪ Type (Wemco self-aspirated)		
▪ Individual Cell Size	ft^3	300
▪ Number of Cells	qty	7
▪ Installed Motor Size in each Cell	hp	30
Total Zinc Scavenger Flotation Retention Time	min	8

Source: ESM operating data 2020

Table 17-4: Zinc first cleaners design criteria

Design Criteria – Zinc First Cleaners	Unit	Value
Solids Feed Rate into Zinc First Cleaners	t/h	102
Feed Pulp Density	% w/w	31
Feed Flowrate into Zinc First Cleaners	gal/min	1,008
Existing Zinc First Cleaner Cells		
▪ Type (Denver forced air)		
▪ Individual Cell Size	ft^3	300
▪ Number of Cells	qty	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/min	640

Source: ESM operating data 2020

Table 17-5: Zinc second cleaners

Design Criteria – Zinc Second Cleaners	Unit	Value
Solids Feed Rate into Zinc Second Cleaners	t/h	49
Feed Pulp Density	% w/w	25
Feed flowrate into Zinc Second Cleaners	gal/min	640
Existing Zinc Second Cleaner Cells		
▪ Type (Denver)		
▪ Individual Cell Size	ft^3	300
▪ Number of Cells	qty	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/min	326

Source: ESM operating data 2020

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 sulfate, sodium cyanide, sodium sulfide 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 sulfate storage tanks on the bottom floor of the mill. All copper sulfate tanks have been removed from service.

A collection of diaphragms 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: ESM 2024

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 recommissioning, 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 restart, 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. The laboratory maintains a relationship with an outside contract laboratory for the purpose of running comparison and duplicate sample exercises.

18. Project Infrastructure

Chapters 16 to 22 apply only to ESM's zinc operations. An economic analysis of ESM's graphite mineralization has not yet been completed.

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 by the previous mine operator in 2008.

Source: JDS 2018

Figure 18-1: EMS general site arrangement

18.2 Roads, Barging, Airstrip, and 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 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.

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 in recent years.

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, boiler room and dry / lamp room form a 60 ft x 273 ft area. The dry room, located on the ground floor, accommodates 125 persons 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. An additional locker near the main lobby can accommodate15 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 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:

- Vice President of Operations;
- 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 counterweight.

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 3,990' of 2 ¼" head rope. The hoist system is driven by two 1,250 hp 500 rpm DC motors and is capable of hoisting at a speed of 1,750' per minute. The resultant hoisting rate is 200 t/h. Shaft and hoist related maintenance tasks that affect production hoisting (and hence daily capacity) are shown Table 18-1.

Table 18-1: No. 4 Shaft availability

Critical Tasks that Interfere with Skip Hoisting	Hours Per Week
Hoisting Compartment Maintenance	3
Cage & Counterweight Compartment Maintenance	1
Crusher Bin & Flop Gate Maintenance	1
Rope Maintenance	0.50
Headframe scrolls & Flop Gate Maintenance	2.0
Shaft Mucking	1.50
Hoist Inspections	3
Powder Delivery	4
Total non-hoist hours per week	16
Smoke time hours per week	10
Hours per week that hoist is not available	26
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.

Production ropes are inspected by x-ray every 5 months.

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 3,990 ft of 1 ¾” head rope and driven by a single 900 hp 400 rpm DC motor. The maximum hoisting speed is 1,190’ per minute. When the hoist is used for mine equipment moving operations, it can handle a maximum piece weight of 13 t. The cage rope and the counterweight rope are inspected by x-ray every 5 months.

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 3,990' of 1¼" 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 cage and counterweight ropes are inspected by x-ray every 5 months.

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 switchgear 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 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 3,750 kVA General Electric transformers (23,000-2,200) feeding the surface plant with secondary voltage of 2,300 V for sub-feeders.

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.

ESM owns two portable generators for emergency use. One is a 125 kVA portable used for operating No. 4 service hoist. 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 ESM 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 that is capable of 50 gallons per minute (gal/min) 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 liter (L) of water throughput. A Burks 5 hp pump will deliver 65 gal/min 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 three Worthington 14-135-2, 75 hp pumps rated at 1,500 gal/min. 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 northeast through a series of settling and polishing ponds before it will be discharged to the environment.

18.5.2 Water Treatment

During period of Care and Maintenance, 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 sulfide 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 SPDES 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 SPDES outfall of 2,640,000 gal/d.

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 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 tailings, mine impacted water is also pumped to the TMF at a rate approximately 1,600 gal/min. The TMF is permitted as a discharge facility and continuously operates within compliance limits. Slaked lime and/or sodium sulfide is added to achieve water quality discharge standards for an average of 5 months per year.

The ultimate capacity of the entire 260-acre TMF footprint has been estimated at 20 Mt of tailings 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 750,000 yd^3 of fill to be sourced from either mine waste or other local sources. Currently, the estimated remaining capacity within the active Tailings Pond #1 and without further embankment construction, will approximately be 3.5 years of production at 450,000 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:

- Tailings Pond #1 (TP1) 190 acres;
- Tailings Pond #2 (TP2) 30 acres;
- Reclaimed Tails Area 40 acres;
- Polishing Ponds 25 acres.

Tailings Pond 1 (TP1) is the active area for tailings placement. The South Dam is on the upstream side with a crest elevation of 650 ft 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.

TP2 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 tailings.

The polishing ponds allow additional time for solids to settle and for natural attenuation to improve water chemistry by flow through a passive wetlands 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 tailings and mine water flow. At times, sodium sulfide may be added to the flow at head of polishing ponds.

Tailings 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 Québec 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

Chapters 16 to 22 apply only to ESM's zinc operations. An economic analysis of ESM's graphite mineralization has not yet been completed.

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 19-2).

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: ESM 2024

19.1.1 International Zinc Smelters (partial list)

Table 19-2: International zinc smelters

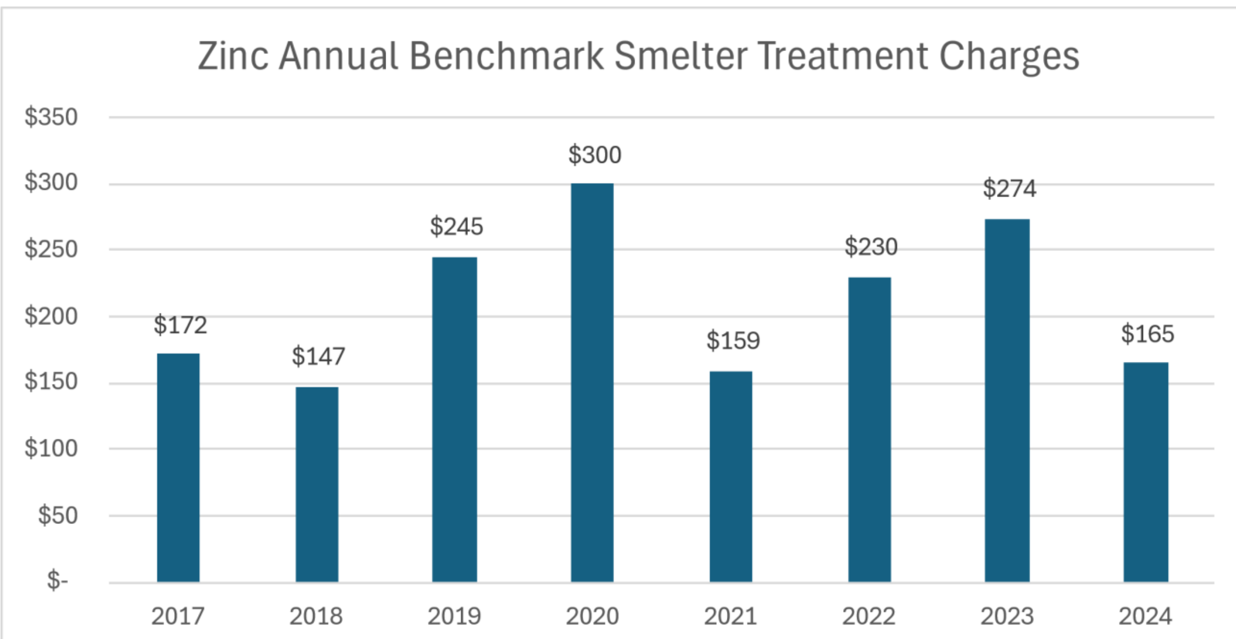
Company	Plant Name	Country	Zinc Capacity (kt)
Glencore	San Juan de Nieva	Spain	486
Glencore	Nordenham	Germany	150
Glencore	Portovesme	Italy	Not operating
Nyrstar	Balen	Belgium	260
Nyrstar	Budel	Netherlands	291
Nyrstar	Auby	France	172
Nyrstar	Hobart	Australia	271
Boliden	Kokkola	Finland	290
Boliden	Odda	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: ESM 2024

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, which is typically 56% for zinc. Additional payable by-products may include gold and silver when levels are sufficiently high. 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 2017 to 2024 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. Treatment charges are expected to drop below $150/dmt in 2025.



Source: Fastmarkets 2024

Figure 19-1: Zinc smelter treatment charges

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

Item	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
Zinc Treatment Charge ($/dmt)	165	140	150	165	165	165	165	165	165	165

Source: ESM 2024

20. Environmental Studies, Permitting and Social or Community Impact

Chapters 16 to 22 apply only to ESM's zinc operations. An economic analysis of ESM's graphite mineralization has not yet been completed.

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 #2 Mine in 1930. The other mines in the district are the Balmat #3, Balmat #4, Hyatt, and Pierrepont. The only remaining operating mine is ESM #4 (formerly known as Balmat). ESM #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. The geotechnical review of the tailings storage facilities (TSF) has been completed. Using Canadian Dam Association (CDA) Standards, a Dam Breach Analysis (DBA) and Seismic Hazard Analysis (SHA) have been completed. The Operation, Maintenance and Surveillance Manual (OMS) has been developed and published. The facility has completed its 2nd annual DSR on 15 October 2024.

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. There are no 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 the ESM #4 since the previous operating periods. No environmental studies are underway at this time, or required for this existing, fully permitted mine. The site is in compliance with all environmental regulatory requirements.

Environmental permits required for operation of the #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. The SPDES permit is on the Department's schedule for technical review due to length of time elapsed since previous review. The SPDES permit remains in force as written despite listed expiry date.

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	28 April 2034
Water	SPDES Water Discharge Permit	NY0001791	31 May 2019[(1)]
Water	Water Withdrawal Permit	6-4038-00024/02001	30 April 2031
Mining	Mining Permit	6-4038-00024/00006	31 Jul 2025
Storage	NYDEC Petroleum Bulk Storage	PBS#6-451770	26 Sep 2028
Radiation	Certificate of Registration for Radiation Installation - XRF	44023174	15 Sep 2026
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	052324550160G	30 Jun 2025

Source: ESM 2024

(1) The SPDES permit remains in effect as written despite listed expiry date.

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 ESM #3 underground mine has water seal plugs below the water table to minimize groundwater inflow to the lower levels of the mine. The static water level at #3 is approximately 30 ft below the surface collar elevation. Planned operation levels at the #4 Mine are currently dry. The #4 Mine receives water flow from #2 and #3 mines, plus flow from Gouverneur Minerals' abandoned underground workings.

Water quality sampling data from the ESM #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 ESM #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 (Hair, 2012). 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 #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.

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 #2 Mine site has been partially reclaimed. ESM #2 Shaft serves as secondary access to the UG operations at the #4 Mine and will be included in the final reclamation of the #4 Mine and concentrator complex. The ESM #4 Mine and mine tailings reclamation is assured with a $1,920,000 surety bond.

Final closure will commence when the Company has determined that the mine and plant will no longer support future economic recovery of any remaining or undiscovered resources. 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 tailings 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 5 to 10-year successful post-closure monitoring period.

A Closure Plan and Cost Estimate update was completed by R. Fennema and D. Sollner of SRK Consulting in 2011 (Fennema & Sollner, 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

Source: Fennema & Sollner 2011

Note: Five-year period including closure to monitor performance of new construction.

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: Fennema & Sollner 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 also contributes to the tax base in St. Lawrence County.

Over the years, housing development has increased in the area. Sylvia Lake, adjacent to the #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

Chapters 16 to 22 apply only to ESM's zinc operations. An economic analysis of ESM's graphite mineralization has not yet been completed.

21.1 Capital Cost Estimate

21.1.1 Capital Cost Summary and Estimate Results

Estimated project capital costs (including closures costs) total $37.2M, consisting of the following distinct areas:

- No. 4 Mine capital equipment;
- No. 4 infrastructure and process capital.

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

Table 21-1 presents the capital estimate summary for each area in Q4 2024 US$ with no escalation.

Table 21-1: Capital cost summary

Area	Cost Estimate ($M)
No. #4 Mine Capital	13.1
No. 4 Infrastructure and Process Capital	13.9
Total Capital Cost	**27.0**
Closure Costs	15.4
Salvage Value	5.2
Total Capital Cost (incl. closure costs)	**37.2**

Source: ESM 2024

21.1.2 Key Estimate Parameters

The following key parameters apply to the capital cost estimates:

- **Estimate class:** The capital cost estimates are considered AACE Class 3 estimates.
- **Estimate base date:** The base date of the estimate is June 30, 2024. No escalation has been applied to the capital cost estimate for costs occurring in the future.
- **Units of measure:** Short ton (t), which is equivalent to 2,000 pounds.
- **Currency:** All capital costs are estimated in US$.

21.1.3 Basis of Estimate

21.1.3.1 Underground Mine (#4 Mine)

Underground (UG) capital costs are estimated to be $13.1M. This includes an additional mechanical bolter as well as a replacement bolter, replacement of two 6-yd loaders, replacement of two UG haul trucks, replacement of a single boom jumbo, four additional 750 kVA transformers, ventilation fans and doors, a replacement locomotive, a surface exploration drill, and main dewatering pumps.

Service vehicles less than $25,000 are expensed and not capitalized. Rebuilds and other sustaining equipment requirements are also expensed.

Table 21-2 presents the capital cost distribution for the #4 Mine capital equipment.

Table 21-2: Distribution of #4 Mine capital equipment costs

Description	$ (x 1,000)
2 x Mechanical Bolters – (replacement)	1,948
2 x 6 yd Loaders – (replacement)	860
2 x 40 t Haul Trucks (replacement)	1,295
Single Boom Jumbo (replacement)	850
Telehandler	120
Rail Locomotive	155
4 x 750 kVA Transformers	935
Ventilation Fans and Doors	407
Mahler Ventilation Raise	3,500
Forklift	80
Main Dewatering Pumps	120
Diamond Drill	150
Outyear Sustaining	2,679
Total	**13,099**

Source: ESM 2024

21.1.3.2 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, tailings storage facility (TSF) lift, replacement of the surface loader for loading concentrate, and mill sustaining costs.

Infrastructure capital costs include ore skip rail replacement, a spare engine for the fire water pump, rebuild of the waste transfer at the UG crusher, installation of a grizzly at the 3100 level ore transfer, ore skip replacements, repairs to the UG crusher, replacement of the shaft telehandler, and roof repairs.

All costs are based on quotations. Table 21-3 presents the capital cost distribution for the #4 Mine infrastructure and process capital.

Table 21-3: Distribution of #4 Mine infrastructure and process costs

Description	$ (x 1,000)
TSF Lift	7,500
Ball Mill Reline (2)	324
Rod Mill Reline (2)	1,024
Surface Loader Replacement	175
Mill Sustaining - Outyears	875
Ore Skip Rail Replacement	1,229
Transfer Repair – Waste Side	360
Fire Pump New Engine	75
3100 Grizzly	290
Ore Skip Replacement (2)	535
UG Crusher Repairs	754
Telehandler	125
Roof Repair	600
Total	**13,866**

Source: ESM 2024

21.1.3.3 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 (Fennema & Sollner, 2011) adjusted for the Consumer Price Index from 2014 to 2024 US$ and totaled $15.4M. The majority of the physical closure work would occur over a 2-year period. Monitoring and environmental management costs would continue for another 23 years, as estimated by SRK, totaling $1.5M. The details of the closure costs are summarized in Table 21-4.

Table 21-4: Closure cost summary

Closure Costs	Total ($ x 1,000)	Closure Y1 ($ x 1,000)	Closure Y2 ($ x 1,000)	Closure Y3-Y26 ($ x 1,000)
Demolition and Miscellaneous Infrastructure	4,875	4,875		
Tailings	6,512	651	5,861	
Surface Water Diversions	1,331	1,331		
Contaminated Soils	161	161		
Landfills	95	48	48	
Closure Project Management Administration and Environmental Management Costs	909	454	454	
Subtotal	**13,883**	**7,520**	**6,363**	
Post-closure Costs				
Earthworks Inspection and Maintenance	376			376
Environmental Management	1,101			1,101
Subtotal	1,477			1,477
Total	**15,360**	**7,520**	**6,363**	**1,477**

Source: ESM, from Fennema & Sollner 2011 in 2024 US$

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 was estimated at $5.2M.

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.

21.1.3.4 Indirect, Owner's, and Contingency Costs

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

21.1.3.5 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 $446M or $101/t milled.

Preparation of the site operating cost estimate is based on current UG 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.

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

Site operating costs are presented in 2024 US$ on a calendar year basis. No escalation or inflation is included.

The operating cost estimate for the UG mine is based on actual operating data from 2024 so is considered highly accurate. Mining, milling, G&A, and transportation costs for 2024 are considered to be representative of operating costs going forward. Site operating costs for the underground are summarized in Table 21-5.

Table 21-5: Summary of underground operating cost

Underground	Unit Cost ($/t milled)	LOM Cost ($M)
Mining	55	244
Processing	18	80
G&A	20	90
Concentrate Transportation	8	32
Total	**101**	**446**

Source: ESM 2024

21.2.2 Summary of Site Personnel

Table 21-6: Summary of site personnel

Position	Staff/Hourly	Total
Mining		
Mine Management	1/0	**1**
Mine Operations	0/58	**58**
Mine Maintenance	1/19	**20**
Crush, Hoist, Shaft	0/9	**9**
Processing		
Process Management	1/0	**1**
Process Operations	0/12	**12**
Process and Surface Maintenance	0/5	**5**
G&A		
General Management	1/0	**1**
Accounting	3/0	**3**
Technical Services	9/0	**9**
Warehouse	3/2	**5**
Human Resources	3/0	**3**
Safety and Environment	3/0	**3**
Site Total	**22/108**	**130**

Source: ESM 2024

Site personnel is based on current staffing levels. The site is currently operating with 130 full time employees.

21.2.3 Underground Mining Operating Cost

The UG 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 $55 per ton milled. The UG mining cost is summarized in Table 21-7.

Table 21-7: Summary of underground mining cost

UG Mining	Unit Cost ($/t milled)	LOM Cost ($M)
Labor	26.59	118
Supplies	21.45	96
Energy	2.44	11
Services	3.26	15
Admin	1.25	6
Total	**55.00**	**245**

Source: ESM 2024

Note: Totals may not compute exactly due to rounding. Mining labor includes all production and UG maintenance labor as well as mine administration labor. 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 21-8. Mill labor includes all mill and surface maintenance labor as well as mill administration labor. 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 21-8: Summary of processing operating cost

Process	Unit Cost ($/t milled)	LOM Cost ($M)
Labor	5.10	23
Supplies	6.27	28
Energy	5.02	22
Services	1.53	7
Admin	0.09	0.4
Total	**18.00**	**80**

Source: ESM 2024

The G&A operating cost is summarized in Table 21-9. G&A labor includes all administration labor 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 21-9: Summary of G&A operating cost

G&A	Unit Cost ($/t milled)	LOM Cost ($M)
Labor	7.55	34
Supplies	0.14	1
Energy	0.00	0
Services	2.34	10
Admin	9.97	45
Total	**20.00**	**89**

Source: ESM 2024

Note: Totals may not compute exactly due to rounding.

22. Economic Analysis

Chapters 16 to 22 apply only to ESM's zinc operations. An economic analysis of ESM's graphite mineralization has not yet been completed.

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 Chapters 21 and 22 of this report. The economic analysis has been run with no inflation (constant US 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 9-year mine life that were used in the economic analysis.

Table 22-1: LOM plan summary

Parameter	Unit	Value
Mine Life	year	9.0
Underground Waste	kt	
Underground Mineralization	kt	
Total Plant Feed Material	kt	4,446
Throughput Rate	t/d	1,775
Operating Days per Year	d/y	260
Average Zinc Price	$/lb	1.25
Average Head Zinc Grade	%Zn	7.4

Source: ESM 2024.

Other economic factors include the following:

- Discount rate of 5%;
- Nominal 2024 US dollars;
- Revenues, costs, taxes are calculated for each period in which they occur;
- All costs and time prior to January 1, 2024, 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 into the international marketplace. Details regarding the terms used for the economic analysis can be found in the Market Studies (Chapter 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	d/y	260
Zinc Recovery from Process Plant (#4 Mine)	%	96

Source: ESM 2024

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 ESM 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 $5.6M.

22.5 Royalties

The economic analysis incorporates royalties. A royalty of 0.3% is applied to the NSR for the zinc concentrate.

22.6 Results

The Project economics for this report reflect only the UG mine, at this stage with an after-tax NPV of $83M at a 5% discount rate. The economics for the open pit continue to be evaluated. 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 the economic analysis results

Summary of Results	Unit	Value
Mine Life	year	9.0
Resource Mined	kt	4,469
LOM Throughput Rate	t/d	1,775
LOM Operating Days per Year	d/y	260
Average Head Zinc Grade	%Zn	7.4
LOM Recovered Zinc	M lb	636
LOM Payable Zinc	M lb	541
Total Revenue	$M	577
Total Offsite Charges	$M	107
Royalties	$M	0.2
NSR (net of royalties)	$M	577
Capital Costs (including sustaining)	$M	27
Operating Costs	$M	446
Operating Costs	$/t processed	101
Pre-tax Cash Flow	$M	104
Taxes	$M	5.6
After-tax Cash Flow	$M	98
Pre-tax NPV (5% discount)	**$M**	**88**
After-tax NPV (5% discount)	**$M**	**83**

Source: ESM 2024

Table 22-4: Cash flow model for ESM

Item	Unit	LOM	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
Zinc Price	$/lb	1.15	1.30	1.33	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25
Tons Mined UG	000s t	4,460	425	462	467	455	455	455	455	455	455	383
Zinc Grade	%	7.4	8.6	7.8	7.5	7.3	7.3	7.3	7.3	7.3	6.5	7.3
Contained Zinc	000,000s lb	663.0	73.2	72.3	70.3	66.3	66.3	66.3	66.3	66.3	59.2	55.8
Mineralization Processed	000s t	4,460	425	462	467	455	455	455	455	455	455	383
Zinc Grade	%	7.4	8.6	7.8	7.5	7.3	7.3	7.3	7.3	7.3	6.5	7.3
Contained Zinc	000s lb	663.0	73.2	72.3	70.3	66.3	66.3	66.3	66.3	66.3	59.2	55.8
Zinc Concentrate Produced	000s dry t	532.0	60.8	57.9	56.3	53.1	53.1	53.1	53.1	53.1	47.4	44.7
Shipping Weight	000s wet t	579.0	66.0	62.9	61.2	57.7	57.7	57.7	57.7	57.7	51.5	48.6
Zinc in Concentrate	000,000s lb	636.0	70.5	69.5	67.5	63.7	63.7	63.7	63.7	63.7	56.8	53.6
Payable Zinc	000,000s lb	541.0	60.0	59.1	57.4	54.1	54.1	54.1	54.1	54.1	48.3	45.6
Gross Metal Value - Zinc	000s $	803,838	90,925	92,393	84,414	79,607	79,607	79,607	79,607	79,607	71,058	67,014
Payable Zinc Value	000s $	683,406	77,286	78,534	71,752	67,666	67,666	67,666	67,666	67,666	60,399	56,961
Less Treatment Charges	000s $	91,048	11,416	8,718	8,985	9,196	9,196	9,196	9,196	9,196	8,208	7,741
Less Penalties	000s $	15,710	1,422	1,754	1,705	1,608	1,608	1,608	1,608	1,608	1,354	1,435
NSR Value	000s $	576,647	64,592	68,062	61,061	56,862	56,862	56,862	56,862	56,862	50,756	47,867
Revenue - Zinc	000s $	576,647	64,592	68,062	61,061	56,862	56,862	56,862	56,862	56,862	50,756	47,867
#4 Infrastructure & Process Capital	000s $	13,866	1,265	2,594	3,390	2,925	2,642	750	150	150	-	-
#4 Mining Capital Equipment	000s $	13,232	348	2,598	2,827	3,809	1,250	1,000	1,000	400	-	-
Total Capital Costs	**000s $**	**27,099**	**1,613**	**5,192**	**6,218**	**6,734**	**3,892**	**1,750**	**1,150**	**550**	**-**	**-**

Item	Unit	LOM	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
Mining	000s $	243,987	22,797	24,155	25,792	25,025	25,025	25,025	25,025	25,025	25,053	21,066
Processing	000s $	80,218	7,005	8,729	8,441	8,190	8,190	8,190	8,190	8,190	8,199	6,894
G&A	000s $	89,504	8,689	9,165	9,379	9,100	9,100	9,100	9,100	9,100	9,110	7,660
Concentrate Transport	000s $	31,735	3,544	3,461	3,364	3,173	3,173	3,173	3,173	3,173	2,832	2,671
Royalties	000s $	191	38	20	18	17	17	17	17	17	15	14
Total #4 Mine Operating Costs	000s $	445,635	42,071	45,531	46,994	45,505	45,505	45,505	45,505	45,505	45,209	38,360
Revenue	000s $	576,647	64,592	68,062	61,061	56,862	56,862	56,862	56,862	56,862	50,756	47,867
Capital Costs	000s $	27,099	1,612	5,192	6,218	6,734	3,892	1,750	1,150	550		
Operating Costs	000s $	445,635	42,071	45,531	46,994	45,505	45,505	45,505	45,505	45,505	45,209	38,360
Pre-tax Net Cash Flow	000s $	103,913	21,039	17,339	7,849	4,623	7,465	9,607	10,207	10,807	5,547	9,651
Cumulative pre-tax Net Cash Flow	000s $	103,913	21,039	38,378	46,227	50,850	58,316	67,923	78,130	88,937	98,498	104,044
Pre-tax Net Present Value (5%)	000s $	87,678	21,039	16,513	7,120	3,994	6,142	7,527	7,617	7,680	6,741	3,576
Net Income Before Tax	000s $			17,339	7,849	4,623	7,465	9,607	10,207	10,807	5,547	9,651
Corporate Tax	000s $	5,652	243	349	82	-	27	816	1,203	1,264	598	1,070
Post-tax Net Cash Flow	000s $	97,747	20,665	16,475	7,767	4,623	7,438	8,791	9,005	9,543	8,491	4,949
Cumulative Post-tax Net Cash Flow	000s $	97,747	20,665	37,140	44,907	49,530	56,968	65,759	74,764	84,307	92,798	97,747
Post-tax Net Present Value (5%)	000s $	83,330	20,665	16,180	7,045	3,994	6,119	6,888	6,719	6,782	3,190	5,747

Source: ESM 2024

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 5%.

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 @ 5% ($M)			Post-tax NPV @ 5% ($M)		
	-10% Variance	0% Variance	10% Variance	-10% Variance	0% Variance	10% Variance
Zinc Price	47	88	133	38	83	125
Zinc Grade	49	88	126	46	83	116
CAPEX	90	88	85	85	83	76
OPEX	116	88	55	109	83	44

Source: ESM 2024

Table 22-6: Discount rate sensitivities

Discount Rate (%)	Pre-tax NPV ($M)	After-tax NPV ($M)
0	104	98
5	88	83
8	80	76
10	76	73
12	**72**	**69**

Source: ESM 2024

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

25.1 Zinc

ESM began operating over 100 years ago (from 1915) and has a proven track record of replacing Mineral Resources 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 prepared by ESM under the Canadian NI 43-101 guidelines. A comprehensive re-modeling effort was undertaken by ESM in 2018 using Leapfrog™ Geo for all geological models. Mining and grade control experience by ESM geologists has supported that implicit modeling of mineralized zones as veins in Leapfrog™ Geo results in more accurate geological wireframes.

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

Underground Mineral Resources have been modeled and estimated using Leapfrog™ Geo 2023.2.3 and Edge software. Mineral Resources for the underground #4 Mine areas have been compiled from separate block models including the American, Cal Marble, Fowler, Mahler, Mud Pond Apron, Mud Pond Main, N2D, New Fold, Northeast Fowler, and Silvia Lake areas.

Open Pit Turnpike Mineral Resources have also been modeled and estimated using Leapfrog™ Geo and Edge software. Mineral Resources for Turnpike have been taken from a single block model.

The ESM deposit will be extracted using a combination of longitudinal retreat stoping (LRS), cut and fill (C&F), Panel Mining - Primary and Secondary, and development drifting underground mining methods with rock backfill as needed. Longhole back-stopes are also used in the design where applicable. The proposed UG plan is expected to produce 1,750 tons per day. Open pit mining will be completed independently from UG mining based on zinc price. The open pit is not included in life of mine considerations. The expected mine life of the underground is 9 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 the 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, zinc flotation circuits, concentrate dewatering circuits, and loadout facilities. The design capacity of the concentrator is 5,000 t/d. Throughout the history of the Balmat operation (now ESM), the capacity of the concentrator has exceeded that of the mines' 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.

While aged, the concentrator is in good working order and runs efficiently. No modifications are required to continue processing underground mineralization sources and no 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 no other significant factors or risks likely 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 complies with a SPDES permit and is expected to remain so during operation, 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 $104M, with a pre-tax and post-tax net present value (NPV) at a discount rate of 5% of $88M and $83M, 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.

25.1.1 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 Modeling	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 these areas. Losses from unknown sources as well as air leaks from door and bulkhead 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 is an important factor 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: ESM 2024

25.1.2 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 NPV.
Productivity Enhancement	Accelerate the mining of the N2D zone.	Increases project NPV over the life of the zone.
Mine Plan Expansion	Resource zones added may add significant mineable tons to the LOM plan.	Increased mine life and increased project NPV.
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: ESM 2024

25.2 Graphite

The Kilbourne Graphite Project has shown potential for significant graphite mineralization. The discovery of graphite in Unit 2 of the Upper Marbles (UM2) was first documented by ESM personnel in mid-2022, following surface exploration hole SX22-2621, which intercepted a 799.1 ft section of UM2. This initial discovery led to a review of historical drill data, revealing graphite mineralization in 130 records from adjacent properties, including Titan's adjacent property and the historic Hyatt mine. Although there has been no previous graphite production at Kilbourne, historical exploration data shows the presence of other mineral occurrences, including a recorded iron and sulfur prospect dating back to 1917.

Graphite mineralization at Kilbourne, consistent with other deposits in the Grenville Province, is believed to result from metamorphic processes acting on organic carbon in sedimentary lithologies. The mineralization occurs in a stratiform manner within UM2, which is divided into three sub-units with transitional zones between each: the Upper Graphitic Schist (UGS), the Phlogopitic Garnet Schist (PGS), and the Lower Graphitic Schist (LGS). These units exhibit variations in thickness and graphite content, with grades in the UGS and LGS ranging from 1.5% to 3% Cg, with higher grades of up to 13.5% Cg observed in some assays.

Exploration at Kilbourne has involved a combination of historic data review, geochemical sampling, airborne geophysical surveys, channel sampling, and exploration drillholes. The integration of historic geophysical data has helped identify additional graphite targets, and further exploration along strike from the known mineralization has been prioritized, especially where geophysical anomalies and documented graphite occurrences overlap.

The data has been validated by the QP by conducting site investigations, reviewing drill core logging, and sampling procedures and confirming drill collar locations.

The Mineral Resource Estimate for the Project was prepared using 12 geological domains and 45 surface drillholes with one surface channel, all totaling 29,699 ft. The Project's mineral resource was completed by ordinary kriging and using a cut off of 1.5% Cg. The pit-constrained Inferred Mineral Resources totaling 22,423 tons grading 2.91 % Cg. The MRE is supported by drilling, analysis, and specific gravity data. Geological controls were available and used to constrain the mineralization and reasonable parameters were used to constrain the mineralization within a pit shell.

Overall, Kilbourne's graphite mineralization shows considerable exploration potential. The mapped surface extension of UM2 continues in both directions, and additional drilling is warranted to evaluate the full extent of this prospective strike length. The continued re-evaluation of geophysical and geologic data may reveal further areas of prospectivity for graphite within the Project area.

26. Recommendations

26.1 Zinc

26.1.1 Zinc Operations

Based on the PEA results, it is recommended that ESM continues with project advancement. The following items are recommended for resource upgrade, project optimization, and confirmation of design parameters used in this study:

- Infill drilling of existing drillholes to improve resource resolution and accuracy, and upgrade the classification of the Inferred Mineral Resource.
- There is an opportunity to increase production and project NPV by accelerating the mining of the N2D zone. This would require the purchase of $2.8M of additional mining equipment pre-production, a power upgrade of $2.6M within 18 months of start of production and hiring additional miners and mechanics to add 500 tons per day of incremental ore to the mill feed. The expansion would decrease the life of mine by 1 year compared to the base case due to accelerated depletion of resources. It would also add $14M to the Project pre-tax NPV calculation and 13 M payable zinc pounds per year during its 3.5-year life.
- 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 Turnpike 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	150,000
Review Financing for Production Expansion from N2D Zone	5,400,000
Sorting Test Work and Integration Study	100,000
Contractor Quotes for Turnpike Open Pit Cost Assumptions	15,000
Total Estimate	**5,665,000**

Source: ESM 2024

26.1.2 Zinc Exploration

Based on the historic productivity of the Balmat-Pierrepont trend, available datasets owned by the Company, and the proven success of conventional exploration techniques, it is recommended that ESM engages in the systematic exploration of their current land package, while assessing the acquisition of additional prospective properties. Targets within the Balmat-Pierrepont trend, and the greater district should be explored, with priority given to those within the historically productive stratigraphies of the Balmat, Edwards, Hyatt, and Pierrepont mines. The following items are recommended as part of this effort:

- **Surface Geochemical Sampling:** Collect a minimum of 2,000 soil sample per year, with an initial focus on currently controlled lands within the Balmat-Pierrepont Trend, followed by properties with historic anomalous zinc samples. Hydrogeochemical sampling where access allows, focusing on areas with late geologic cover obscuring the productive Proterozoic marbles.
- **Near Mine - Exploration Drilling:** Conduct a minimum of 13,000 ft of drilling along strike from known mineralized horizons, and target favorable lithologies with limited historic data including areas where historic property access limited exploration.
- **Exploration Drilling:** Exploration drilling within the trend and district should be approached with the same systematic targeting as the surface geochemistry. Annual drilling should see a minimum of 18,000 ft drilled per year. Drilling should prioritize historic mineralized intercepts with enough space down dip and along strike to host a potentially significant zinc occurrence. Additional targets generated by surface geochemical sampling should be tested when access and timing allow.
- **Geophysics:** The reinterpretation of the remaining two-thirds of the HudBay airborne geophysical survey to further identify prospective areas for both base metal, and graphite mineralization.
- **Land Acquisition and Management:** Continued acquisition of land with historic mineral prospects and occurrences with a focus on the consolidation of prospective in trend geology. It is also recommended that the Company completes an in-depth review of current mineral rights, this review should extend to these neighboring properties within the trend.

With the exception of the geophysical reinterpretation, it is recommended that the above items be conducted annually.

Table 26-2: Cost estimate for recommended exploration activities

Item	Estimated Cost ($)
Surface Geochemical Sampling	246,000
Near Mine – Exploration Drilling	525,000
Exploration Drilling	570,000
Geophysics	90,000
Land Acquisition and Management	-
Estimate for 2025	**1,431,000**
Annual Estimate	**1,341,000**

Source: ESM 2024

26.2 Graphite

26.2.1 Preliminary Economic Assessment

Based on the exploration drilling results, it is recommended that ESM proceed with project advancement to a PEA level scoping study. The following items are recommended as part of this study:

- **Infill Drilling**: Conduct infill drilling of existing drillholes to improve the resolution and accuracy of the resource estimate and upgrade the classification of the Inferred Mineral Resource.
- **Geotechnical Study**: Conduct a geotechnical study to assess the stability and safety of the open pit resource. This should be incorporated into the next phase of infill drilling.
- **Phase III Metallurgical Study:** Initiate a Phase III metallurgical study to further evaluate the processing characteristics and recovery rates of the Mineral Resource. Dedicated metallurgical holes should be incorporated into the next phase of infill drilling.
- **Mining Study**: Conduct a detailed mining study to address the challenges of extracting the resource, with a particular focus on managing the overlying tailings from the active underground mine.
- **Optical Sorting Study**: Conduct a study to test optical sorting technology, evaluating its effectiveness in separating mineral from waste before entering the mill facility. This should include an integration study to assess the impact of this technology on the mine and the logistics of its application.

- **Contractor Quotes**: Obtain contractor quotes for open pit mining to improve the accuracy of cost estimates in the next level of study.
- **Permitting**: Engage with stakeholders and regulatory bodies to secure the necessary permits and approvals for the proposed mining activities.

Table 26-3 shows the cost of the recommended drilling, metallurgical, and engineering programs.

Table 26-3: Project recommendations and estimated cost

Recommended Study Item	Estimated Cost ($)
Infill Drilling	950,000
Geotechnical Study	50,000
Phase III Metallurgical Study	47,000
Mining Study	250,000
Optical Sorting Study	30,000
Contractor Quotes	15,000
Permitting	130,000
PEA Technical Report Update	500,000
Preliminary Economic Assessment Subtotal	**1,972,000**
Contingency (25%)	493,000
Total Estimate	**2,465,000**

Source: ESM 2024

26.2.2 Commercial Scoping and Demonstration Plant

To ensure the commercial viability and market readiness of the Project, the following steps are recommended:

- **Commercial Scoping Study**: Conduct a comprehensive commercial scoping study to evaluate the market potential and economic feasibility of the Project. This study should include a market analysis, competitive landscape assessment, and identification of potential customers and partners.

- **Demonstration Plant**: Construct and run a demonstration plant for a minimum of 12 to 18 months to validate the proposed extraction and processing methods. The demonstration plant will serve multiple purposes:
  - **Produce Concentrate**: Produce concentrate in sufficient quantities to deliver to potential consumers and obtain committed offtakes.
  - **Product Qualification Consulting**: Engage with product qualification consultants to ensure that the graphite product meets industry standards and customer specifications. This process will involve testing and validation of product quality, consistency, and performance.

By undertaking these steps, ESM will be well-positioned to advance the Project towards commercial production, ensuring that the product meets market demands and customer and industry qualification requirements.

Table 26-4: Commercial recommendations and estimated cost

Recommended Study Item	Estimated Cost ($)
Commercial Scoping Study	150,000
Product Qualification Consulting	68,000
Demonstration Plant (including working capital)	6,110,000
Commercial Scoping Subtotal	**6,328,000**
Contingency (25%)	1,582,000
Total Estimate	**7,910,000**

Source: ESM 2024

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