

Empire State Mines
2025 NI 43-101 Technical Report

Gouverneur, New York, USA

Prepared for:
Titan Mining Corporation

Effective Date: December 1, 2025
Signature Date: December 15, 2025

Prepared by the following Qualified Persons:

- Donald R. Taylor, MSc., PG..... Titan Mining Corporation
- Todd McCracken, P.Geo..... BBA USA Inc.
- Bahareh Asi, P.Eng. ..... BBA USA Inc.
- David Willock, P.Eng. ..... BBA USA Inc.
- Deepak Malhotra, P.Eng..... Forte Dynamics, Inc.
- Oliver Peters, MSc, P.Eng., MBA..... Metpro Management Inc.
- Derick de Wit, FAusIMM..... Dorfner Anzaplan UK Limited
- Steven M. Trader, PG, CPG..... Alpha Geoscience

Date and Signature Page

This technical report is effective as of the 1st day of December 2025.

"Original signed (Donald R. Taylor)"	*December 15, 2025*
Donald R. Taylor, MSc, PG Titan Mining Corporation	Date

"Original signed and sealed (Todd McCracken)"	*December 15, 2025*
Todd McCracken, P.Geo. BBA USA Inc.	Date

"Original signed and sealed (Bahareh Asi)"	*December 15, 2025*
Bahareh Asi, P.Eng. BBA USA Inc.	Date

"Original signed and sealed (David Willock)"	*December 15, 2025*
David Willock, P.Eng. BBA USA Inc.	Date

"Original signed and sealed (Deepak Malhotra)"	*December 15, 2025*
Deepak Malhotra, P.Eng. Forte Dynamics, Inc.	Date

"Original signed (Oliver Peters)"	*December 15, 2025*
Oliver Peters, MSc, P.Eng., MBA Metpro Management Inc.	Date

"Original signed (Derick de Wit)"	*December 15, 2025*
Derick de Wit, FAusIMM Dorfner Anzaplan UK Limited	Date

"Original signed and sealed (Steven M. Trade)"	*December 15, 2025*
Steven M. Trader, PG, CPG Alpha Geoscience	Date

CERTIFICATE OF QUALIFIED PERSON

Donald R. Taylor, MSc, PG

This certificate applies to the NI 43-101 Technical Report titled "*Empire State Mines 2025 NI 43-101 Technical Report, Gouverneur, New York*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of December 15, 2025, and an effective date of December 1, 2025.

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

1. I am the Vice Chair 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 most recently served as President and CEO of Augusta Gold before its acquisition by AngloGold Ashanti 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 Vice Chair of the Company.
7. I am author and responsible for the preparation of Items 1 (except 1.6, 1.7.2, 1.9.2, 1.10, 1.11.2, 1.12, 1.13.2, 1.14.2), 3, 4, 5, 6, 7, 8, 9, 10, 11, 12 (except 12.1.3, 12.1.4, 12.1.5, 12.2), 14 (except 14.2), 15 (except 15.2), 16 (except 16.3), 18.1, 19, 21.1.1, 21.2.1, 23, 24, 25.1, and 26.1. I am also co-author of Item 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 was Chief Executive Officer of the Company from June 21, 2018 to September 8, 2025, and Vice Chair of the Company since September 8, 2025.
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 December 2025.

"Original signed (Donald R. Taylor)"

Donald R. Taylor, MSc, PG

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CERTIFICATE OF QUALIFIED PERSON

Todd McCracken, P.Geo.

This certificate applies to the NI 43-101 Technical Report titled "*Empire State Mines 2025 NI 43-101 Technical Report, Gouverneur, New York*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of December 15, 2025, and an effective date of December 1, 2025.

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 Items 1.7.2, 1.14.2, 12.2.1, 14.2, 21.1.2.7, 21.1.2.8, 22, 25.2.1, 25.2.8, and 26 (except 26.1, 26.2.1.2, 26.2.1.3, 26.2.2, 26.2.3). I am also co-author of Items 25.2.9, 25.2.10, and 27 of the Technical Report.
8. I have visited the Property that is the subject of the Technical Report on August 26-27, 2024, and on July 23-24, 2025, as part of this current mandate.
9. I have prior involvement with the Property that is the subject of the Technical Report as I participated as QP for the previous report titled "Empire State Mines 2024 NI 43-101 Technical Report Update" dated January 15, 2025.
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 December 2025.

"Original signed and sealed (Todd McCracken)"

Todd McCracken, P.Geo.

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CERTIFICATE OF QUALIFIED PERSON

Bahareh Asi, P.Eng.

This certificate applies to the NI 43-101 Technical Report titled "*Empire State Mines 2025 NI 43-101 Technical Report, Gouverneur, New York*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of December 15, 2025, and an effective date of December 1, 2025.

I, Bahareh Asi, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am Senior Mining Engineer with the firm BBA USA Inc., located at 10 Carlson Court, Suite 100, Toronto, ON, M9W 7K6, Canada.
1. I am a graduate in Mining from the Bahonar University of Kerman in 2001, with a Bachelor of Engineering, and from Tarbiat Modares University in 2004 with a Master of Engineering. I have been employed in consulting engineering and mining operations since my graduation and practiced my profession continuously.
2. I am a member in good standing of the Professional Engineers of Ontario (PEO No: 100203076).
3. My relevant experience includes over 20 years in the area of open pit mine engineering for the design, planning and estimation in technical studies and mine operations for numerous mining projects.
4. 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.
5. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
6. I am author and responsible for the preparation of Items 1.8, 1.9.2, 1.13.2, 2, 12.2 (except 12.2.1, 12.2.3, 12.2.4, 12.2.5, 12.2.6, 12.2.7), 15.2, 16.3, 21 (except 21.1.1, 21.2.1, 21.1.2.2.3, 21.1.2.2.4, 21.1.2.2.5, 21.1.2.4, 21.1.2.5, 21.1.2.6, 21.1.2.7,21.1.2.8, 21.2.2.4, 21.2.2.5, 21.2.2.6, 21.2.2.7), 25 (except 25.1, 25.2.1, 25.2.3 to 25.2.8), and 26.2.1.2. I am also co-author of Items 25.2.9, 25.2.10, and 27.
7. I have not visited the Property that is the subject of the Technical Report.
8. I have no prior involvement with the Property that is the subject of the Technical Report.
9. 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.
10. 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 December 2025.

"Original signed and sealed (Bahareh Asi)"

Bahareh Asi, P.Eng.

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CERTIFICATE OF QUALIFIED PERSON

David Willock, P.Eng.

This certificate applies to the NI 43--101 Technical Report titled "*Empire State Mines 2025 NI 43-101 Technical Report, Gouverneur, New York*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of December 15, 2025, and an effective date of December 1, 2025.

I, David Willock, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a Mining Engineer with the firm BBA USA Inc., located at 144 Pine Street, Unit 501, Sudbury, ON, P3C 1X3, Canada.
2. I graduated from the Laurentian University in 2000 with a Bachelor of Engineering, Mining. I have practiced my profession continuously since my graduation.
3. I am a member in good standing of the Professional Engineers of Ontario (No: 100113931).
4. I have been employed in mining engineering, operations and projects for over 20 years. My relevant experience includes underground hard-rock production planning, mine studies, operations supervision and project execution. I have worked as a senior project engineer for numerous North American base metal projects.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("N 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 an author and responsible for the preparation of Items 1.11.2, 12.2.5, 18 (except 18.1, 18.2.8 to 18.2.10), 21.1.2.2.3, 21.1.2.4, 21.2.2.4, 25.2.3, and 26.2.1.3. I am also co-author of Items 25.2.9, 25.2.10, and 27 of the Technical Report.
8. I have visited the Property that is the subject of the Technical Report on July 23-24, 2025 as part of this current 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 December 2025.

"Original signed and sealed (David Willock)"

David Willock, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Deepak Malhotra, SME-RM

This certificate applies to the NI 43-101 Technical Report titled "*Empire State Mines 2025 NI 43-101 Technical Report, Gouverneur, New York*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of December 15, 2025, with an effective date of December 1, 2025.

I, Deepak Malhotra, SME-RM, 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 Items 1.6.1, 1.10.1, 12.1.3, 13 (except 13.2), and 17 (except 17.2). I am also co-author of Item 27 of the Technical Report.
8. I have visited the Property that is the subject of the Technical Report for 2 days 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 Technical Reports titled, "Empire State Mines 2021 NI 43-101 Technical Report (Amended)" with an effective date of February 24, 2021, and "Empire State Mines 2024 NI 43-101 Technical Report Update" dated January 15, 2025 (effective date of December 3, 2024). 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 December 2025.

"Original signed and sealed (Deepak Malhotra)"

Deepak Malhotra, SME-RM

Metpro Management Inc.

CERTIFICATE OF QUALIFIED PERSON

Oliver Peters, MSc, P.Eng., MBA

This certificate applies to the NI 43-101 Technical Report titled "*Empire State Mines 2025 NI 43-101 Technical Report, Gouverneur, New York*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of December 15, 2025, and an effective date of December 1, 2025.

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

1. I am Principal Metallurgist & 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 14 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 Items 1.6.2, 1.10.2 (except 1.10.2.2), 12.2.3, 13.2, 17.2 (except 17.2.3), 18.2.8, 18.2.9, 21.1.2.2.4, 21.1.2.5, 21.2.2.5, 21.2.2.6, 25.2.4, 25.2.5, and 25.2.6. I am also co-author of Items 25.2.9, 25.2.10, and 27 of the Technical Report.
8. I have visited the Property that is the subject of the Technical Report on October 20, 2024.
9. I have prior involvement with the Property that is the subject of the Technical Report as I have participated as QP on the Company's report titled "Empire State Mines 2024 NI 43-101 Technical Report Update" dated January 15, 2025 (effective date of December 3, 2024).
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 December 2025.

"Original signed (Oliver Peters)"

Oliver Peters, MSc, P.Eng., MBA

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Peterborough, ON, K9H7T2, Canada

oliver@metpro.ca
T: +1 (705) 761-7276

CERTIFICATE OF QUALIFIED PERSON

Derick de Wit, FAusIMM

This certificate applies to the NI 43-101 Technical Report titled "*Empire State Mines 2025 NI 43-101 Technical Report, Gouverneur, New York*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of December 15, 2025, and an effective date of December 1, 2025.

I, Derick de Wit, FAusIMM, as a co-author of the Technical Report, do hereby certify that:

1. I am a Director and Principal Chemical Engineer with Dorfner Anzaplan UK Limited., located at 1st Floor, Prospect House, Rouen Road, Norwich, NR1 1RE, United Kingdom.
2. I hold the following academic qualifications: MBA, B. Tech (Chem. Eng.), PMP (PMI®).
3. I am a Fellow in good standing of the Australian Institute of Mining and Metallurgy under membership number 301519.
4. My relevant experience includes working as a Chemical Engineer continuously within the mineral resources and chemical industries since 1998. My relevant experience includes engineering and design of mineral and chemical processes and development of projects worldwide in accordance with the major reporting codes, including this Instrument, from geological exploration, through the different feasibility phases, receipt of legislative permits and licenses and implementation, including graphite process flow development, overseeing of metallurgical test work, engineering, design and cost estimation.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("N 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 an author and responsible for the preparation of Items 1.10.2.2, 17.2.3, 18.2.10, 20.2.6, 21.1.2.2.5, 21.1.2.6, 21.2.2.7, 25.2.7, and 26.2.2. I am also co-author of Items 25.2.9, 25.2.10, and 27 of the Technical Report.
8. I have not visited the Property that is the subject of the Technical Report.
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 December 2025.

"Original signed (Derick de Wit)"

Derick de Wit, FAusIMM

CERTIFICATE OF QUALIFIED PERSON

Steven M. Trader, PG, CPG

This certificate applies to the NI 43-101 Technical Report titled "*Empire State Mines 2025 NI 43-101 Technical Report, Gouverneur, New York*" (the "Technical Report"), prepared for Titan Mining Corporation (the "Company"), with a signing date of December 15, 2025, and an effective date of December 1, 2025.

I, Steven M. Trader, PG, CPG, as a co-author of the Technical Report, do hereby certify that:

1. I am a Senior Geologist consulting with Alpha Geoscience, located at 679 Plank Road, Clifton Park, NY 12065, USA.
2. I graduated from Virginia Polytechnic Institute and State University.
3. I am a member in good standing and a Certified Professional Geologist with the American Institute of Professional Geologists (#11452). I am also a Licensed Professional Geologist in New York (#231), a Licensed Professional Geologist in New Hampshire, and a Certified Professional Geologist in Virginia.
4. My relevant experience includes over 30 years of managing and conducting geological, hydrogeological, groundwater and surface water investigations, mine permitting, and evaluating the environmental impacts of mining.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("N 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 an author and responsible for the preparation of Items 1.12, 12.1.4, 12.1.5, 12.2.4, 12.2.6, 20 (except 20.2.6), and 26.2.3. I am also co-author of Item 27 of the Technical Report.
8. I have visited the Property that is the subject of the Technical Report most recently on October 1 and 2, 2025, and numerous other times since 2011.
9. I have prior involvement with the Property that is the subject of the Technical Report as I have provided consulting services for renewing ESM's NYS SPDES permit; helped obtain NYS MLR permits for ESM's Turnpike East and West open pit mines; and helped permit the Kilbourne Graphite Demonstration Pit.
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 December 2025.

"Original signed and sealed (Steven M. Trade)"

Steven M. Trader, PG, CPG

TABLE OF CONTENTS

1. Summary 1-1
1.1 Introduction 1-1
1.2 Project Description 1-1
1.3 Location, Access, and Ownership 1-2
1.4 History, Exploration, and Drilling 1-2
1.4.1 Zinc 1-2
1.4.2 Graphite 1-3
1.5 Geology and Mineralization 1-3
1.5.1 Zinc 1-3
1.5.2 Graphite 1-3
1.6 Metallurgical Testing and Mineral Processing 1-4
1.6.1 Zinc 1-4
1.6.2 Graphite 1-4
1.7 Mineral Resource Estimates 1-5
1.7.1 Zinc 1-5
1.7.2 Graphite 1-9
1.8 Mineral Reserve Estimates 1-11
1.9 Mining 1-11
1.9.1 Zinc 1-11
1.9.2 Graphite 1-13
1.10 Recovery Methods 1-14
1.10.1 Zinc 1-14
1.10.2 Graphite 1-15
1.11 Infrastructure 1-16
1.11.1 Zinc 1-16
1.11.2 Graphite 1-18
1.12 Environment and Permitting 1-20
1.12.1 Zinc 1-20
1.12.2 Graphite 1-20
1.13 Operating and Capital Cost Estimates 1-23
1.13.1 Zinc 1-23
1.13.2 Graphite 1-25
1.14 Economic Analysis 1-27
1.14.1 Zinc 1-27
1.14.2 Graphite 1-27

1.15 Adjacent Properties 1-29
1.16 Other Relevant Data and Information 1-29
1.17 Interpretation and Conclusions 1-29
1.17.1 Zinc 1-29
1.17.2 Graphite 1-30
1.18 Recommendations 1-30
1.18.1 Zinc 1-30
1.18.2 Graphite 1-31
1.19 References 1-31
2. Introduction 2-1
2.1 Basis of the Technical Report 2-2
2.2 Units, Currency, and Rounding 2-4
3. Reliance on Other Experts 3-1
4. Property Description and Location 4-1
4.1 Location 4-1
4.2 Mineral Tenure 4-4
4.3 Mining Rights 4-12
4.4 Project Agreements 4-12
4.5 Environmental Liabilities and Considerations 4-12
4.6 Permit Requirements 4-13
4.7 Risks 4-14
5. Accessibility, Climate, Local Resources, Infrastructure, and Physiography 5-1
5.1 Accessibility 5-1
5.2 Local Resources and Infrastructure 5-2
5.3 Climate 5-2
5.4 Vegetation and Wildlife 5-2
5.5 Physiography 5-2
5.6 Surface Facilities and Rights 5-3
6. History 6-1
6.1 Empire State Mines History 6-1
6.1.1 Management and Ownership 6-1
6.1.2 Exploration History 6-1
6.1.3 Production History 6-2
6.1.4 Historical Mineral Resource and Mineral Reserve Estimates 6-4

6.2 Kilbourne History 6-4
6.2.1 Kilbourne Management and Ownership 6-5
6.2.2 Kilbourne Exploration History 6-5
6.2.3 Kilbourne Production History 6-5
6.2.4 Kilbourne Historical Mineral Reserves and Mineral Resources 6-5
7. Geologic Setting and Mineralization 7-1
7.1 Geological Setting 7-1
7.2 Regional Geology 7-4
7.3 Property Geology 7-5
7.4 Mineralization 7-7
7.4.1 ESM Mineralization 7-7
7.4.2 Kilbourne Mineralization 7-10
8. Deposit Types 8-1
8.1 Zinc 8-1
8.2 Graphite 8-4
9. Exploration 9-1
9.1 Zinc 9-1
9.1.1 Historic Data Review 9-1
9.1.2 Surface Geochemical Sampling 9-3
9.1.3 Hydrogeochemistry 9-13
9.1.4 Airborne Geophysics 9-17
9.1.5 Exploration Potential and Targeting 9-19
9.2 Graphite 9-22
9.2.1 Kilbourne Historic Data Review 9-22
9.2.2 Airborne Geophysics 9-22
9.2.3 Surface Channel Sampling 9-23
9.2.4 Exploration Potential and Targeting 9-23
10. Drilling 10-1
10.1 ESM Drilling 10-1
10.1.1 Drilling Summary 10-2
10.1.2 Drilling Procedures 10-6
10.1.3 Core Handling and Sampling 10-6
10.1.4 Downhole Surveying 10-8
10.2 Graphite 10-8
10.2.1 Core Re-sampling 10-8
10.2.2 Core Drilling Summary 10-9
10.2.3 Drilling Procedure 10-11

10.2.4 Core Handling and Sampling 10-12
10.2.5 Downhole Surveying 10-13
11. Sample Preparation, Analyses, and Security 11-1
11.1 Zinc Historical Assaying 11-1
11.1.1 Pre Hudbay and Checks 11-1
11.1.2 Hudbay 2005-2010 Assaying 11-2
11.2 2017 to 2025 Zinc Sample Preparation and Assaying 11-4
11.2.1 Sample Preparation and Analysis 11-4
11.2.2 Security 11-6
11.2.3 Quality Assurance / Quality Control 11-7
11.3 Graphite – Kilbourne Sample Preparation and Assaying 11-13
11.3.1 Sample Preparation and Analysis 11-13
11.3.2 Security 11-15
11.3.3 Quality Assurance / Quality Control 11-15
12. Data Verification 12-1
12.1 Zinc 12-1
12.1.1 Geology 12-1
12.1.2 Geotechnical 12-1
12.1.3 Metallurgical 12-1
12.1.4 Hydrological / Hydrogeological 12-2
12.1.5 Environmental 12-2
12.1.6 Marketing 12-2
12.1.7 Cost Estimates 12-3
12.2 Graphite 12-3
12.2.1 Geology 12-3
12.2.2 Geotechnical – Kilbourne Open Pit 12-3
12.2.3 Metallurgical 12-4
12.2.4 Hydrogeological 12-4
12.2.5 Kilbourne Site Infrastructure 12-4
12.2.6 Environmental 12-5
12.2.7 Marketing 12-5
12.2.8 Cost Estimates 12-5
13. Mineral Processing and Metallurgical Testing 13-1
13.1 Zinc 13-1
13.1.1 Processing 2018–2024 13-1
13.1.2 Turnpike and Hoist House Metallurgical Testwork 13-3
13.1.3 Conclusions 13-9

13.2 Graphite 13-10
13.2.1 Concentrate Plant 13-10
13.2.2 Micronization and Purification 13-34
14. Mineral Resource Estimates 14-1
14.1 Zinc 14-1
14.1.1 Drillhole Database 14-1
14.1.2 Density 14-3
14.1.3 Topography Data 14-6
14.1.4 Geological Interpretation 14-7
14.1.5 Voids Model 14-11
14.1.6 Exploratory Data Analysis 14-12
14.1.7 Resource Block Model 14-19
14.1.8 Resource Classification 14-26
14.1.9 Mineral Resource Tabulation 14-28
14.1.10 Model Validation 14-46
14.1.11 Relevant Factors 14-48
14.1.12 Comparison to Previous Estimate 14-48
14.2 Graphite 14-49
14.2.1 Deposit Database 14-49
14.2.2 Density 14-51
14.2.3 Topography Data 14-52
14.2.4 Geological Interpretation 14-52
14.2.5 Exploratory Data Analysis 14-54
14.2.6 Resource Block Model 14-60
14.2.7 Resource Classification 14-61
14.2.8 Mineral Resource Tabulation 14-62
14.2.9 Model Validation 14-63
14.2.10 Previous Estimates 14-67
15. Mineral Reserve Estimates 15-1
15.1 Zinc 15-1
15.2 Graphite 15-1
16. Mining Methods 16-1
16.1 Zinc Underground 16-1
16.1.1 Deposit Characteristics 16-3
16.1.2 Mineral Resources Within the PEA Mine Plan – Estimation Process 16-6
16.1.3 Mining Method Selection 16-6
16.1.4 Geotechnical Parameters 16-12
16.1.5 Stope Design Parameters 16-14

16.1.6 Mine Dilution and Recovery 16-15
16.1.7 Cut-off Grade Criteria 16-15
16.1.8 Mine Plan Tons and Grade 16-16
16.1.9 Mine Design Criteria 16-17
16.1.10 Production Rate Selection 16-18
16.1.11 Production Sequencing 16-19
16.1.12 Underground Mine Development 16-19
16.1.13 Unit Operations 16-20
16.1.14 Mine Services 16-22
16.1.15 Underground Mine Equipment 16-27
16.1.16 Mine Personnel 16-28
16.1.17 Mine Production Schedule 16-30
16.1.18 Mine Development Schedule 16-31
16.1.19 Projected Tailings Production 16-31
16.2 Zinc Open Pit 16-31
16.2.1 Hydrological Parameters 16-31
16.2.2 Open Pit Geotechnical Considerations 16-32
16.2.3 Cut-off Value 16-34
16.2.4 Dilution and Mining Recovery Factors 16-35
16.2.5 Pit Limit Analysis 16-35
16.2.6 Pit Design 16-38
16.2.7 Mining Method 16-42
16.2.8 Open Pit Equipment 16-44
16.2.9 Open Pit Labor and Staff 16-44
16.2.10 Proposed Open Pit Production Schedule 16-45
16.2.11 Projected Tailings Production 16-46
16.3 Graphite Open Pit 16-46
16.3.1 General Parameters Used to Estimate In-pit Mineable Resources 16-46
16.3.2 Mining Dilution and Mining Loss Factors 16-48
16.3.3 Pit Limit Analysis 16-48
16.3.4 Pit Design 16-53
16.3.5 Mine Plan 16-61
16.3.6 Open Pit Mine Equipment Fleet 16-72
16.3.7 Open Pit Workforce 16-79
17. Recovery Methods 17-1
17.1 Zinc 17-1
17.1.1 Introduction 17-1
17.1.2 Plant Design Criteria 17-1
17.1.3 Metallurgical Balance 17-10

17.1.4 Energy and Process Material Requirements....17-10
17.1.5 Water Balance....17-11
17.1.6 Opportunities for Metallurgical Improvement....17-12
17.1.7 Assumptions....17-12
17.1.8 Conclusions....17-12
17.2 Graphite....17-13
17.2.1 Concentrate Plant....17-13
17.2.2 Micronization Plant....17-19
17.2.3 Secondary Transformation Site....17-20
18. Project Infrastructure....18-1
18.1 Zinc....18-1
18.1.1 General Site Arrangement....18-1
18.1.2 Buildings and Structures....18-2
18.1.3 Power....18-6
18.1.4 Water Control Infrastructure....18-7
18.1.5 Waste Rock Management....18-8
18.1.6 Tailings Management Facility....18-9
18.1.7 Concentrate Transportation....18-10
18.2 Graphite....18-10
18.2.1 General Overview....18-10
18.2.2 Infrastructure....18-12
18.2.3 Power Supply and Distribution....18-14
18.2.4 Utilities and Services....18-15
18.2.5 Water Management....18-15
18.2.6 Tailings Management Facilities....18-18
18.2.7 Kilbourne Off-site Infrastructure....18-29
18.2.8 Concentrate Plant....18-29
18.2.9 Micronization Plant....18-30
18.2.10 Secondary Transformation Site....18-31
19. Market Studies and Contracts....19-1
19.1 Zinc....19-1
19.1.1 Smelter Market....19-1
19.1.2 Zinc Price and Concentrate Terms....19-2
19.2 Graphite....19-2
19.2.1 Market Information....19-2
19.2.2 Study Price & Sales Terms....19-3
19.2.3 Contracts....19-5

20. Environmental Studies, Permitting and Social or Community Impact 20-1
20.1 Zinc 20-1
20.1.1 Environmental 20-1
20.1.2 Permitting 20-1
20.1.3 Groundwater 20-2
20.1.4 Closure 20-3
20.1.5 Social and Community Factors 20-6
20.2 Graphite 20-6
20.2.1 Environmental 20-6
20.2.2 Permitting 20-7
20.2.3 Closure 20-9
20.2.4 Social and Community Factors 20-10
20.2.5 Concentrate Plant 20-10
20.2.6 Purification and CSPG Plant 20-11
21. Capital and Operating Costs 21-1
21.1 Capital Cost Estimate 21-1
21.1.1 Zinc 21-1
21.1.2 Graphite 21-7
21.2 Operating Cost Estimate 21-25
21.2.1 Zinc 21-25
21.2.2 Graphite 21-28
22. Economic Analysis 22-1
22.1 Methodology Used 22-1
22.2 Financial Model Parameters 22-1
22.3 Pricing 22-2
22.4 Royalties 22-2
22.5 Taxes 22-2
22.6 Working Capital 22-3
22.7 Economic Analysis 22-3
22.8 Sensitivities 22-7
23. Adjacent Properties 23-1
24. Other Relevant Data and Information 24-1
25. Interpretation and Conclusions 25-1
25.1 Zinc 25-1
25.1.1 Risks 25-3
25.1.2 Opportunities 25-4

25.2 Graphite 25-5
25.2.1 Geology 25-5
25.2.2 Open Pit 25-6
25.2.3 Infrastructure 25-7
25.2.4 Mineral Processing and Metallurgical Testing 25-8
25.2.5 Concentrate Plant 25-8
25.2.6 Micronization Plant 25-9
25.2.7 Secondary Transformation Site 25-9
25.2.8 Financials 25-10
25.2.9 Risks 25-11
25.2.10 Opportunities 25-15
26. Recommendations 26-1
26.1 Zinc 26-1
26.1.1 Operations 26-1
26.1.2 Exploration 26-2
26.2 Graphite 26-3
26.2.1 Kilbourne Site 26-4
26.2.2 Secondary Transformation Site 26-6
26.2.3 Environmental Permitting and Community Impacts 26-8
26.2.4 Economics 26-9
27. References 27-1

LIST OF TABLES

Table 1-1: Ownership history....1-2
Table 1-2: Update periods, model methodology, and volumes....1-6
Table 1-3: Underground Mineral Resource Estimate as of June 9, 2025....1-7
Table 1-4: Turnpike Open Pit Mineral Resource Estimate as of October 17, 2024....1-8
Table 1-5: Kilbourne Graphite Mineral Resource summary and in situ metal within pit shells....1-10
Table 1-6: Mine production schedule....1-12
Table 1-7: Turnpike open pit conceptual schedule....1-12
Table 1-8: Capital cost summary....1-23
Table 1-9: Breakdown of estimated site operating costs....1-24
Table 1-10: Initial, expansion and sustaining capital costs....1-26
Table 1-11: Project All-in Operating Costs....1-27
Table 1-12: Operating results summary....1-28
Table 1-13: Study input pricing....1-29
Table 1-14: Financial analysis summary....1-29
Table 1-15: Project recommendations and estimated cost....1-30
Table 1-16: Cost estimate for recommended exploration activities....1-31
Table 1-17: Project recommendations and estimated cost....1-31
Table 2-1: QP Responsibilities and date of last site visit....2-3
Table 4-1: Lease list with expiration dates....4-6
Table 4-2: Mineral tenure information....4-8
Table 4-3: Environmental permits for operation of ESM #4 Mine....4-14
Table 6-1: History of ownership....6-1
Table 6-2: Historic production totals by region....6-3
Table 6-3: Empire State Mines annual production totals....6-3
Table 6-4: Historical Mineral Reserves....6-4
Table 7-1: Upper Marble stratigraphic sequence....7-5
Table 9-1: Occurrences highlighting lead and zinc occurrences within the district....9-2
Table 9-2: 2022 Soil sampling totals and high Zn (%) values....9-6
Table 9-3: Summary of assay methods....9-9
Table 9-4: Upper and lower limits for aqua regia ICP method....9-9
Table 9-5: Upper and lower limits for MS89L super trace analysis method....9-9
Table 9-6: 2019 to 2025 rock samples by target with highest zinc values....9-11
Table 9-7: Near-mine exploration targets....9-19
Table 9-8: Exploration targets....9-21

Table 9-9: ESM outcrop channel samples 9-23
Table 10-1: Area drilling by year since 2020 10-5
Table 10-2: Table of company owned and operated core drills 10-6
Table 10-3: ESM surface holes re-sampled for graphite 10-9
Table 10-4: Kilbourne drilling by year 10-9
Table 10-5: 2025 Kilbourne exploration drilling 10-10
Table 10-6: Significant intercepts from 2025 Kilbourne exploration program 10-11
Table 11-1: Hudbay QA/QC standards certified by OREAS Hudbay 11-3
Table 11-2: ESM QA/QC certified standards supplied by OREAS June 2008 11-3
Table 11-3: Summary of assay methods 11-4
Table 11-4: Upper and lower limits for aqua regia ICP method 11-5
Table 11-5: Upper and lower limits for MS89L super trace analysis method 11-6
Table 11-6: Blank failure threshold 11-7
Table 11-7: Summary tables of results for reference materials 11-9
Table 11-8: Summary of assay methods 11-13
Table 11-9: Upper and lower limits for aqua regia GE-ICP21B20 method 11-14
Table 11-10: Certified reference material expected values 11-15
Table 11-11: Blank failure threshold 11-16
Table 11-12: Summary of results for reference materials 11-16
Table 13-1: ESM mill statistics 2018–2024 13-3
Table 13-2: Head analyses of composite samples including ICP 13-4
Table 13-3: Bond's ball mill work index 13-5
Table 13-4: Sequential rougher flotation results - Turnpike 13-5
Table 13-5: Sequential rougher flotation results - Hoist House 13-6
Table 13-6: Cleaner flotation results - Turnpike 13-7
Table 13-7: Cleaner flotation results - Hoist House 13-8
Table 13-8: Results from the XRD analysis 13-10
Table 13-9: Chemical analysis of Kilbourne composite 13-12
Table 13-10: Flash & rougher flotation tests (F3 to F6 and F8) 13-14
Table 13-11: Primary cleaner flotation tests (F7 and F9) 13-15
Table 13-12: Results of full cleaner test F10 13-16
Table 13-13: Size fraction analysis of F10 9th cleaner concentrate 13-17
Table 13-14: Bulk concentrate production results 13-18
Table 13-15: 6th Cleaner tests of flash flotation concentrate 13-19
Table 13-16: 6th Cleaner tests of rougher flotation concentrate 13-20
Table 13-17: Size fraction analysis – 6th Cleaner flash concentrate Batch 1 13-22

Table 13-18: Carbon speciation of master and variability composites 13-22
Table 13-19: Chemical characterization of the master composite 13-23
Table 13-20: Primary grind size evaluation 13-24
Table 13-21: Cleaner optimization tests 13-25
Table 13-22: Variability flotation tests 13-29
Table 13-23: Average flake size distribution 13-32
Table 13-24: Detailed concentrate analysis 13-32
Table 13-25: Concentrate size analysis 13-34
Table 13-26: Concentrate characterization 13-35
Table 13-27: Concentrate thermogravimetry and crystallography 13-36
Table 13-28: X-ray fluorescence analysis 13-36
Table 13-29: Micronization results 13-37
Table 13-30: XRF analysis of pre- and post-purification micronized products 13-38
Table 13-31: Characterization of pre- and post-purified micronized graphite 13-39
Table 13-32: Characterization of pre- and post-purified spherical graphite 13-41
Table 14-1: Core holes used in estimation of each zone 14-2
Table 14-2: Density by zone and material type 14-5
Table 14-3: Turnpike indicator RBF interpolant performance statistics 14-8
Table 14-4: Update periods, model methodology, and volumes 14-9
Table 14-5: ESM assay summary statistics by domain 14-12
Table 14-6: ESM capping summary by domain 14-14
Table 14-7: Compositing method by domain 14-16
Table 14-8: Block model size and location by zone 14-20
Table 14-9: Estimation method, ellipse parameters, and outlier restrictions 14-23
Table 14-10: Underground Mineral Resource Estimate as of June 9, 2025 14-29
Table 14-11: Turnpike pit constraint parameters 14-30
Table 14-12: Open Pit Mineral Resource Estimate as of October 17, 2024 14-31
Table 14-13: Change of underground Mineral Resources from previous estimate 14-49
Table 14-14: Kilbourne deposit geological domains 14-50
Table 14-15: Kilbourne deposit specific gravity and tonnage factor summary 14-52
Table 14-16: Kilbourne deposit wireframe volume to block model volume summary 14-53
Table 14-17: Kilbourne deposit drillhole basic “raw” statistics by domain 14-55
Table 14-18: Kilbourne deposit grade capping summary 14-56
Table 14-19: Kilbourne deposit drillhole composited statistics by domain 14-58
Table 14-20: Variogram parameters 14-59
Table 14-21: Block model parameters 14-60

Table 14-22: Search ellipse and rotations 14-61
Table 14-23: Interpolation parameters 14-61
Table 14-24: Kilbourne deposit pit constraint parameters 14-62
Table 14-25: Kilbourne Graphite Mineral Resource summary and in situ metal within pit shell . 14-63
Table 14-26: Kilbourne global composite to block model statistics comparison 14-65
Table 16-1: Mineral Resources for the LOM by mining method 16-3
Table 16-2: Rock mass characterizations for Lower Mahler 16-13
Table 16-3: Production stope design criteria 16-14
Table 16-4: Overbreak dilution parameters 16-15
Table 16-5: Cut-off grade parameters 16-16
Table 16-6: Tons contained in the LOM plan by zone 16-17
Table 16-7: Rates used for mine scheduling 16-18
Table 16-8: Existing mobile mine equipment fleet 16-27
Table 16-9: Mine personnel summary 16-29
Table 16-10: Annual mineralized material 16-30
Table 16-11: Projected production for 2025 16-30
Table 16-12: Annual development schedule 16-31
Table 16-13 Annual tailings production 16-31
Table 16-14: Knight Piésold pit slope recommendations 16-33
Table 16-15: Generalized slope angles for pit optimization and design 16-34
Table 16-16: Cut-off value assumptions 16-34
Table 16-17: Pit shell optimization results 16-38
Table 16-18: Open pit projected tons and grades 16-41
Table 16-19: Open pit drilling parameters 16-43
Table 16-20: Equipment estimate 16-44
Table 16-21: Open pit labor and supervision 16-45
Table 16-22: Conceptual open pit production schedule 16-45
Table 16-23 Annual Tailings Production 16-46
Table 16-24: Impact of regularization of block model 16-48
Table 16-25 Pit limit analysis parameters 16-49
Table 16-26: Nested pit shell results 16-52
Table 16-27: Pit design characteristics 16-54
Table 16-28: Pit design configuration 16-54
Table 16-29: Haul ramp design 16-55
Table 16-30: Ultimate pit material inventory 16-60
Table 16-31: Material inventory by phase 16-60

Table 16-32: Mining quantities by period 16-62
Table 16-33: Mine equipment KPIs 16-74
Table 16-34: Drill and blast parameters 16-74
Table 16-35: Loading productivity 16-76
Table 16-36: Haulage speed assumptions 16-77
Table 16-37: Truck parameters 16-78
Table 16-38: Auxiliary equipment used 16-79
Table 16-39 Workforce positions considered 16-80
Table 17-1: Crushing circuit design criteria 17-3
Table 17-2: Grinding circuit design criteria 17-5
Table 17-3: Zinc rougher / scavenger flotation circuit design criteria 17-7
Table 17-4: Zinc first cleaners design criteria 17-8
Table 17-5: Zinc second cleaners 17-8
Table 17-6: Concentrator mass balance 17-10
Table 17-7: ESM water balance, plant operating 17-11
Table 17-8: ESM water balance, plant not operating 17-11
Table 17-9: Key process design criteria 17-14
Table 17-10: List of major mechanical equipment 17-17
Table 17-11: Process design criteria for the Micronization Plant 17-19
Table 17-12: Process design criteria for the Purification Plant 17-22
Table 17-13: Mass balance for the Purification Plant 17-24
Table 17-14: Process design criteria for purification stage of CSPG Plant 17-27
Table 17-15: Process design criteria for spheroidization stage of CSPG Plant 17-28
Table 17-16: Process design criteria for coating stage of CSPG Plant 17-28
Table 17-17: Mass balance for the CSPG Plant 17-29
Table 18-1: #4 Shaft availability 18-4
Table 18-2: Facility building list 18-6
Table 18-3: Tailings characteristics 18-20
Table 18-4: Volume of tailings to be relocated and produced tailings 18-22
Table 18-5: Required factor of safety for stability analysis 18-23
Table 18-6: Details of staged Extended TMF 18-25
Table 18-7: Tailings management planning during LOM 18-28
Table 19-1: North American zinc smelters 19-1
Table 19-2: International zinc smelters (partial list) 19-1
Table 19-3: Natural Flake Concentrate: The base target purity is 95.0% - 96.0% LOI 19-4
Table 19-4: STD Purity Micronized Grades: A minimum purity of 95.0% LOI 19-4

Table 19-5: High Purity Micronized Grades: A minimum purity of 99.9% LOI 19-4
Table 19-6: CSPG Anode Grades: A minimum purity of 99.95% LOI 19-5
Table 19-7: Recommended study price 19-5
Table 20-1: Environmental permits 20-2
Table 20-2: Post-closure water quality monitoring frequency 20-5
Table 20-3: Schedule of closure activities 20-5
Table 21-1: Capital cost summary 21-1
Table 21-2: Distribution of #4 Mining capital equipment costs 21-2
Table 21-3: #4 Mine capital waste development cost 21-3
Table 21-4: Distribution of #4 Mine infrastructure and process costs 21-4
Table 21-5: Distribution of N2D and expansionary capital costs 21-4
Table 21-6: Closure cost summary 21-6
Table 21-7: Estimate scope division 21-8
Table 21-8: Initial capital costs by pre-production years 21-9
Table 21-9: Initial, expansion and sustaining capital costs 21-9
Table 21-10: Currency exchange rates on April 1, 2025 21-11
Table 21-11: Major and support mining equipment required and purchase price assumptions 21-17
Table 21-12: Direct capital costs – Kilbourne open pit mine 21-17
Table 21-13: Direct capital cost estimate – Site infrastructure 21-18
Table 21-14: Indirect capital cost estimate – Mine infrastructure 21-18
Table 21-15: Direct capital cost estimate – TMF 21-19
Table 21-16: Direct capital costs – Concentrate and Micronization plants 21-20
Table 21-17: Initial capital costs by pre-production year – Concentrate Plant 21-20
Table 21-18: Initial capital costs by year – Micronization Plant 21-20
Table 21-19: Direct costs – Initial capital and expansion costs by year – Micronization Plant 21-21
Table 21-20: Direct costs – Concentrate Plant mechanical breakdown 21-21
Table 21-21: Direct costs – Micronization Plant mechanical breakdown 21-21
Table 21-22: Indirect costs – Concentrate Plant 21-22
Table 21-23: Indirect costs – Micronization Plant 21-22
Table 21-24: Direct and indirect costs by phase – Purification and CSPG Plants 21-22
Table 21-25: Direct costs – Purification 21-23
Table 21-26: Indirect costs – Purification 21-23
Table 21-27: Direct expansion costs – CSPG Plant 21-24
Table 21-28: Indirect costs – CSPG Plant 21-24
Table 21-29: Contingency breakdown 21-25
Table 21-30: Summary of underground operating cost 21-26

Table 21-31: Summary of open pit operating costs 21-27
Table 21-32: Summary of site personnel 21-27
Table 21-33: Project all-in operating costs 21-29
Table 21-34: Breakdown of average LOM mining operating costs 21-30
Table 21-35: Mining operating cost by activity by year 21-31
Table 21-36: Annual compensation per mining position 21-33
Table 21-37: Other cost assumptions 21-34
Table 21-38: Summary of site infrastructure 21-34
Table 21-39: Summary of average Concentrate Plant operating costs 21-35
Table 21-40: Annual compensation per Concentrate Plant position 21-36
Table 21-41: Costs of comminution circuit consumables 21-37
Table 21-42: OPEX Summary – Micronization Plant 21-38
Table 21-43: OPEX Summary – Micronization Plant per saleable product 21-39
Table 21-44: OPEX Summary – Secondary Transformation Site 21-41
Table 21-45: OPEX Summary – Purification Plant (Phase 1 only) 21-41
Table 21-46: OPEX Summary – CSPG Plant 21-41
Table 21-47: Reagent cost basis for the Purification Plant 21-42
Table 21-48: Reagent cost basis for the CSPG Plant 21-43
Table 21-49: Annual compensation per position for the Purification Plant and CSPG Plant 21-43
Table 21-50: Laboratory cost inputs for the Purification Plant OPEX 21-44
Table 21-51: Laboratory cost inputs for the CSPG Plant OPEX 21-44
Table 22-1: Working capital assumptions 22-3
Table 22-2: Summary of the economic analysis results 22-4
Table 22-3: Annual and cumulative cash flow 22-6
Table 22-4: Pre-tax and post-tax sensitivity analysis 22-7
Table 22-5: Post-tax NPV sensitivity analysis 22-7
Table 25-1: Zinc Operation risks 25-3
Table 25-2: Zinc Operation identified opportunities 25-4
Table 25-3: Graphite Study risks 25-11
Table 25-4: Graphite Study opportunities 25-15
Table 26-1: Project recommendations and cost 26-1
Table 26-2: Cost estimate for recommended exploration activities 26-3
Table 26-3: Recommendation and estimated budget for Kilbourne and Secondary Transformation sites 26-4
Table 26-4: Kilbourne Site recommendations and estimated cost 26-4
Table 26-5: Secondary Transformation Site recommendations and estimated cost 26-6

LIST OF FIGURES

Figure 4-1: Regional project location 4-2
Figure 4-2: Local project location 4-3
Figure 4-3: Mineral tenure map 4-11
Figure 5-1: Site accessibility 5-1
Figure 5-2: Empire State Mines aerial view 5-4
Figure 5-3: Empire State Mine, Turnpike, and Kilbourne 5-5
Figure 7-1: Regional geology setting 7-2
Figure 7-2: Local geologic setting 7-6
Figure 7-3: Section through the Sylvia Lake Syncline 7-7
Figure 7-4: Plan view showing assay Pb (%) grade variation within the Sylvia Lake Syncline 7-9
Figure 7-5: Upper Marble 2 mapped surface expression 7-11
Figure 8-1: Illustration of the process of formation of Sedex deposits 8-3
Figure 9-1: Historic surface geochemical samples by type relative ESM and past operations 9-4
Figure 9-2: Location of 2022 soil sampling programs relative to ESM 9-7
Figure 9-3: Location of rock samples by target 9-12
Figure 9-4: 2023 Water sampling sites by area 9-14
Figure 9-5: 2025 Water sampling sites by area 9-16
Figure 9-6: Geophysical survey area 9-18
Figure 9-7: Near-mine exploration targets shown in green, mine workings in grey 9-20
Figure 9-8: Kilbourne exploration target 9-24
Figure 10-1: Diamond drilling targeting zinc since the 2024 technical report 10-2
Figure 10-2: Map showing Balmat underground drilling colored by drill date 10-3
Figure 10-3: Drillhole database areas; drillholes outside the shaded areas are stored in District 10-4
Figure 10-4: Underground core storage crate staged outside the on-site logging facility 10-7
Figure 10-5: Kilbourne drilling with collars colored by year; RPEEE constraining pit footprint in green 10-10
Figure 10-6: Diamec #2 on the surface drilling for graphite 10-11
Figure 10-7: Example of photographed AWJ size graphitic core 10-12
Figure 11-1: Hudbay Flin Flon Lab check assays of ESM 1995 to 2000 pulps 11-2
Figure 11-2: Zinc in blank control chart 11-8
Figure 11-3: Control chart for Zn in reference material H-5 11-11
Figure 11-4: Control chart for Zn in reference material G-5 11-12
Figure 11-5: Graphitic carbon in blank control chart 11-17
Figure 11-6: Control chart for graphitic carbon in reference material OREAS-722 11-18

Figure 11-7: Control chart for graphitic carbon in reference material OREAS-724 11-19
Figure 11-8: Control chart for graphitic carbon in reference material OREAS-725 11-20
Figure 13-1: ESM mill flowsheet 13-2
Figure 13-2: Photomicrographs from the optical microscope from F03225 13-11
Figure 13-3: Flowsheet test F10 13-16
Figure 13-4: Integrated Forte Analytical flowsheet 13-21
Figure 13-5: Concentrate size fraction analysis – Mass distribution 13-27
Figure 13-6: Concentrate size fraction analysis – Grade distribution 13-27
Figure 13-7: VAR Concentrate size fraction analysis – Mass distribution 13-31
Figure 13-8: VAR Concentrate size fraction analysis – Grade distribution 13-31
Figure 14-1: Scatterplot of specific gravity vs assay zinc (%) for Mahler 14-4
Figure 14-2: Zones relative to topographic surface 14-6
Figure 14-3: Mud Pond – Main vein model 14-7
Figure 14-4: Locations of each zone 14-10
Figure 14-5: ESM 3D voids model 14-11
Figure 14-6: Plan view of block model extents 14-19
Figure 14-7: Classification for New Fold, view looking SE (Az 135) 14-26
Figure 14-8: Classification for all ESM zones 14-27
Figure 14-9: American grade tonnage graph 14-33
Figure 14-10: Cal Marble grade tonnage graph 14-34
Figure 14-11: Fowler grade tonnage graph 14-35
Figure 14-12: Lower Mahler grade tonnage graph 14-36
Figure 14-13: Upper Mahler grade tonnage graph 14-37
Figure 14-14: Mud Pond Apron grade tonnage graph 14-38
Figure 14-15: Mud Pond - Main grade tonnage graph 14-39
Figure 14-16: N2D grade tonnage graph 14-40
Figure 14-17: New Fold grade tonnage graph 14-41
Figure 14-18: Northeast Fowler grade tonnage graph 14-42
Figure 14-19: Sylvia Lake grade tonnage graph 14-43
Figure 14-20: Turnpike UG grade tonnage graph; SO constrained 14-44
Figure 14-21: Turnpike Open Pit grade tonnage graph 14-45
Figure 14-22: New Fold model and composite values for zinc 14-47
Figure 14-23: Swath plot Zn% - Turnpike area 14-48
Figure 14-24: Interpretation of Kilbourne Domains 14-54
Figure 14-25: Parrish decile analysis for domain 210 14-56
Figure 14-26: Global top cut analysis for domain 210 using Snowden Supervisor 14-57

Figure 14-27: Variography for Domain 210 using Snowden Supervisor 14-59
Figure 14-28: Surface plan showing the optimized pit shell for the Kilbourne deposit 14-64
Figure 14-29: Kilbourne deposit visual validation through A-A' 14-64
Figure 14-30: Kilbourne deposit swath plot, 300 ft slice - easting (X) 14-66
Figure 14-31: Kilbourne deposit swath plot, 300 ft slice - northing (Y) 14-67
Figure 16-1: Mine production by method 16-2
Figure 16-2: Mining zones in the LOM 16-5
Figure 16-3: Plan view of Panel Mining 16-7
Figure 16-4: Isometric view of Panel Mining 16-8
Figure 16-5: Typical LRS with sill pillar 16-10
Figure 16-6: Typical C&F 16-11
Figure 16-7: 2500 level workshop back conditions 16-12
Figure 16-8: Ground support for typical ground 16-14
Figure 16-9: Typical development cross-sections 16-20
Figure 16-10: LOM ventilation installations (conceptual, not to scale); view from above to the SE 16-23
Figure 16-11: Site elementary electrical one-line diagram 16-24
Figure 16-12: Plan view optimization shells (with cross-section locations) 16-36
Figure 16-13: Cross-section views 16-37
Figure 16-14: Open pit designs 16-39
Figure 16-15: Cross-section of design and shell 16-40
Figure 16-16: Layout of open pit 16-42
Figure 16-17 Pit-by-pit graph 16-52
Figure 16-18: Proposed ultimate pit design 16-53
Figure 16-19: Access ramp 16-56
Figure 16-20: Minimum setback requirements by DEC 16-57
Figure 16-21: Minimum pushback width 16-58
Figure 16-22: Phase definition 16-59
Figure 16-23: LOM plan material mined by year 16-63
Figure 16-24: Average daily mining production rate 16-63
Figure 16-25: Mill feed throughput by year 16-64
Figure 16-26: Concentrate production by year 16-65
Figure 16-27: Pit phases relative to existing tailings and perimeter containment dikes 16-66
Figure 16-28: Material mined by phase by year 16-67
Figure 16-29: LOM sequence, Year 1 to Year 2 inclined view looking north, not to scale 16-68
Figure 16-30: LOM sequence, Year 3 to Year 4 inclined view looking north, not to scale 16-69
Figure 16-31: LOM sequence, Year 5 to Year 6 inclined view looking north, not to scale 16-70

Figure 16-32: LOM Sequence, Year 7 to Year 8 inclined view looking north, not to scale 16-71
Figure 16-33: LOM Sequence, Year 9 to Year 13 inclined view looking north, not to scale 16-71
Figure 16-34: Equipment utilization model 16-73
Figure 16-35: Major mine equipment by period - Drilling 16-75
Figure 16-36: Major mine equipment by period - Loading 16-77
Figure 16-37: Major mine equipment by period - Hauling 16-78
Figure 16-38: Open pit mine workforce by year 16-81
Figure 17-1: Concentrator flowsheet current state 17-2
Figure 17-2: Concentrate Plant flowsheet 17-15
Figure 17-3: Main processing steps in the Purification Plant 17-20
Figure 17-4: Main processing steps in the CSPG Plant 17-25
Figure 18-1: ESM general site arrangement 18-1
Figure 18-2: Kilbourne Graphite Study Site 18-11
Figure 18-3: General water flow diagram 18-16
Figure 18-4: Water balance (flow rates in GPM) 18-18
Figure 18-5: Raised TMF layout 18-24
Figure 18-6: Proposed Extended TMF layout 18-26
Figure 18-7: Proposed containment dike around the Historic Arnold Pit 18-27
Figure 22-1: Annual and cumulative cash flows 22-5
Figure 22-2: Post-tax NPV and IRR sensitivity analysis 22-8

List of Abbreviations and Units of Measurement

Abbreviation	Description
$, US$	United States dollar
$/t	US dollar per metric tonne
$/ton	US dollar per short ton
%	percent
°	angular degree
°C	degrees Celsius (e.g., 22 °C)
°F	degrees Fahrenheit (e.g., 72 °F)
µm	micron
‰	parts per thousand
3D	three dimensional
a.k.a.	also known as
AACE	Association for the Advancement of Cost Estimation
ABA	acid base accounting
AC	alternating current
AFR	Air Facility Registration
Ag	silver
AI	Artificial Intelligence
Ai	Bond Abrasion Index
AISC	All-in Sustaining Cost
Al	aluminum
Al_2O_3	aluminum oxide
ALS	ALS Limited (laboratories)
amsl	above mean sea level
ANFO	ammonium nitrate fuel oil (explosive)
ANT	Ambient Noise Tomography
Anzaplan	Dorfner Anzaplan UK Limited
AQE	Adaptive Query Execution
As	arsenic
ASL	Analytical Solutions Ltd.
ASP	average sales prices
ASTM	American Society for Testing and Materials
Au	gold
Azi	Azimuth
Ba	barium
BBA	BBA USA Inc.

Abbreviation	Description
BESS	Battery Energy Storage System
BET	Brunauer–Emmett–Teller (BET) theory
BFA	bench face angle
Bi	bismuth
Bt	billion tonnes
Bton	billion tons
BWi	Bond's Ball Mill Work Index
C	carbon
C&F	cut and fill
Ca	calcium
Calc.	calculation
CaO	calcium oxide
CAPEX	capital cost estimate
Cd	cadmium
CDA	Canadian Dam Association
Cg	graphitic carbon
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
Cl	chlorine
CLCPA	*Climate Leadership and Community Protection Act*
Clnr	cleaner
cm	centimeter
cm^3	cubic centimeter
Co	cobalt
COG	cut-off grade
Conc	concentrate
CPG	Certified Professional Geologist
CPT	cone penetration testing
Cr	chromium
CRMs	certified reference materials
CSPG	Coated Spherical Purified Graphite
Cu	copper
d	day (24 hours)
d/y	days per year
DBA	Dam Breach Analysis
DCF	discounted cash flow
DDH	diamond drillhole
DEC	Department of Environmental Conservation

Abbreviation	Description
DEF	diesel exhaust fluid
DEIS	Draft Environmental Impact Statement
dmt	dry metric tonne
DOW	Department of War
DSR	Dam Safety Review
DTH	down-the-hole
EA	Environmental Assessment
EIS	Environmental Impact Statement
EM	electromagnetic
EPCM	Engineering, Procurement and Construction Management
ESM	Empire State Mines
et al.	and others
EUR	Euro
FAST	Fixing America's Surface Transportation Act.
FAusIMM	Fellow Australasian Institute of Mining and Metallurgy
FC	fixed carbon
Fe	iron
Fe_2O_3	ferric oxide
FEIS	Final Environmental Impact Statement
FOG	fall of ground
Form 43-101F1	Form 43-101F1 Technical Report and Related Consequential Amendments
FoS	factor of safety
ft	feet
ft^3	cubic feet
FW	footwall
g	gram
G&A	General and Administrative
g/cm^3	grams per cubic centimeter
g/t	gram per tonne
gal	gallon
gal/d	gallon per day
gal/min	gallon per minute
gal/s	gallon per second
GBP	pound sterling
Ge	germanium
GET	ground engaging tools
GOH	Gross Operating Hours

Abbreviation	Description
Gr	graphite
GT	grade/tonnage
h	hour (60 minutes)
h/d	hours per day
h/y	hours per year
H_2S	hydrogen sulfide
HCl	hydrochloric acid
HDPE	High-Density Polyethylene
HF	hydrofluoric acid
hp	horsepower
Hudbay	Hudbay Minerals Inc.
HVAC	heating, ventilation and air conditioning
HW	hanging wall
IBC	Intermediate Bulk Containers
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma Atomic Emission Spectrometry
ICP-MS	Inductively Coupled Plasma Optical Mass Spectrometry
ICP-OES	Inductively Coupled Plasma Optical Emission Spectrometry
ID	identification
ID2	inverse distance squared
ID3	inverse distance cubed
IDW	inverse distance weighted
in	inch
IRA	inter-ramp angle
Ja	Joint Alteration Number (rock mass parameter)
Jn	Joint Set Number (rock mass parameter)
Jr	Joint Roughness Number (rock mass parameter)
K	potassium
K	thousand ($)
K_2O	potassium oxide
kcfm	kilo-cubic feet per minute
kg	kilogram
km	kilometer
km^3	cubic kilometer
KPI	key performance indicators
kt, ktonne	kilotonne (thousand tonnes)
kton	kiloton (thousand tons)

Abbreviation	Description
kV	kilovolt
kVA	kilovolt amperes
kW	kilowatt
kWh	kilowatt per hour
kWh/t	kilowatt per hour/tonne
L	liter
lb	pound(s)
LGS	Lower Graphitic Schist
LHD	load haul dump (loaders)
LIB	lithium-ion batteries
LLD	lower limit of detection
LME	London Metal Exchange
LOI	Loss of Ignition
LOM	life of mine
LRS	longitudinal retreat stoping
LSTM	Lone Star Tech Minerals
LVA	Locally Varying Anisotropy
m	meter
M	million
m^3	cubic meter
Ma	mega annum
MACRS	Modified Accelerated Cost Recovery System
Max.	maximum
mesh	US mesh
Metpro	Metpro Management Inc.
Mg	magnesium
mg	milligram
MgO	magnesium oxide
mi	mile
MIBC	methyl isobutyl carbinol
min	minute
Min.	minimum
MLUP	Mined Land Use Plan
mm	millimeter
Mn	manganese
Mo	molybdenum
MPSO	MinePlan Schedule Optimizer

Abbreviation	Description
MRDS	Mineral Resources Data System
MRE	Mineral Resource Estimate
Mt	million (metric) tonnes
Mton	million (short) tons
MVT	Mississippi Valley-type
MW	megawatt
N2D	Number 2 Deep (deposit)
Na	sodium
Na_2O	sodium oxide
NAG	net acid generation
NEPA	National Environmental Policy Act
NFG	natural flake graphite
NI	National Instrument
Ni	nickel
NI 43-101	Canadian National Instrument 43-101
NN	nearest neighbor
No.	number
NOH	net operating hours
NONEL	non-electric
non-PAG	non-Potentially Acid Generating
NPV	net present value
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
P	phosphorus
PAP & PAS	Panel Mining – Primary and Secondary
Pb	lead
PbS	lead sulfide

Abbreviation	Description
pcf	pounds per cubic foot
PDC	process design criteria
PEA	Preliminary Economic Assessment
PEG	Pegmatite Intrusion
PEO	Professional Engineers Ontario
PFG	purity flake grade
PG	Professional Geologist
PGA	peak ground acceleration
PGO	Professional Geoscientists Ontario
PGS	Phlogopitic Garnet Schist
PHG	Popple Hill Gneiss
PM	Panel Mining
PMF	potential mill feed
PMG	purified micronized graphite
PPE	personal protective equipment
ppm	parts per million
psi	pound per square inch
PSS	Pathway-Specific Standards
PV	present value
Q1, Q2, etc.	first quarter, second quarter, etc.
QA/QC	quality assurance / quality control
QP(s)	qualified person(s)
RBF	radial basis function
RDi	Resource Development Inc.
RF	revenue factor
RMR	rock mass rating
ROM	run of mine
RP	Recommended Practice
RPEEE	Reasonable Prospects for Eventual Economic Extraction
s	second
S	sulfur
SD	standard deviations
SEDAR+	System for Electronic Document Analysis and Retrieval
Sedex	sedimentary exhalative
SEQR	State Environmental Quality Review Act
SG	specific gravity
SGS	SGS Canada Inc.

Abbreviation	Description
SHA	Seismic Hazard Analysis
SHPO	State Historic Preservation Office
Si	silicon
SiO_2	silicon dioxide
SLZ	St. Lawrence Zinc Company, LLC
SME	Society for Mining, Metallurgy and Exploration (Registered Member)
SMM	stirred media mill
SO	stope optimization
SO_4	sulfate
SPDES	State Pollutant Discharge Elimination System
SPG	spherical purified graphite
SPT	Standard Penetration Tests
Sr	strontium
SRK	SRK Consulting Ltd.
t	metric ton (tonne) (approximately 2,204.62 lb)
t/d	tonne per day
t/h	tonne per hour
t/t	tonne per tonne
$t/t_{graphite}$	tonne per tonne of natural flake graphite concentrate
t/w	tonne per week
t/y	tonne per year
TC	total carbon
TF	tonnage factor
TGA	thermogravimetry
Ti	titanium
TI/RE	Toxicity Identification/Reduction Evaluation
TIC	total inorganic carbon
TiO_2	titanium dioxide
Titan or the Company	Titan Mining Corporation
TMF	Tailings Management Facility
TOC	total organic carbon
ton	short ton (2,000 lb)
ton/d	ton per day
ton/h	ton per hour
ton/w	ton per week
ton/y	ton per year
TP1, TP2, etc.	Tailings Pond #1, #2, etc.

Abbreviation	Description
TSF	tailings storage facility
U.S.	United States
UG	underground
UGS	Upper Garnet Schist
UM	Upper Marble
UM1	Unit 1 of the Upper Marble
UM2	Unit 2 of the Upper Marble
UM3	Unit 3 of the Upper Marble
USA	United States of America
USACE	United States Army Corps of Engineers
USDA	US Department of Agriculture
USGS	United States Geological Survey
V	vanadium
V	volt
VO	Variable Orientation
VTEM	versatile time domain electromagnetic
W	tungsten
w/w	weight in weight
WET	Whole Effluent Toxicity
WQC	Water Quality Certification
wt.%	percent by weight
WWP	Water Withdrawal Permit
XRD	X-ray Diffraction
XRF	X-ray Fluorescence
y	year
yd	yard
yd^3	cubic yard
ZCA	Zinc Corporation of America
Zn	zinc
Zr	Zirconium

1. Summary

1.1 Introduction

BBA USA Inc. (BBA) has been engaged by Titan Mining Corporation (Titan or the Company) to complete a Preliminary Economic Assessment (PEA) for the Kilbourne Graphite Project (Graphite Study) and update the National Instrument 43-101 (NI 43-101) Technical Report for the Empire State Mines (ESM) operation (Zinc Operation). This Technical Report summarizes the results of the PEA and was prepared following the guidelines of NI 43-101.

The Kilbourne resource represents a significant diversification of ESM's resource base beyond zinc. In addition to the graphite evaluation, the report also provides an updated zinc Mineral Resource Estimate (MRE), incorporating new data from recent diamond drilling and underground exposures since the previous technical report.

The currency in this report is United States dollars (US$), unless stated otherwise. Zinc-related operations and reporting are presented in imperial units throughout the Technical Report. For graphite-related operations, imperial units are used up to the concentrator level, consistent with site conventions. Metric units are used for graphite concentrate reporting, including product specifications and downstream metrics. Readers should be aware of this unit transition when reviewing graphite-related sections.

1.2 Project Description

Empire State Mines, owned by Titan Mining Corporation, is located in the Balmat–Edwards–Pierrepont mining district of northern New York State, near Gouverneur and is 25 miles (mi) south of the Port of Ogdensburg. The site includes a complex of mines, including the fully developed #4 underground zinc mine and associated surface infrastructure, including a concentrator, tailings facility, and rail access.

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. ESM resumed underground zinc production in 2018 and currently operates at approximately 2,275 tons per day (ton/d), with plans to ramp up to 2,800 ton/d by 2028.

The Kilbourne Graphite Project is located within 4,000 ft of the existing mill and infrastructure underlying the Zinc Operations Tailings Management Facility (TMF).

The zinc 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 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.

1.3 Location, Access, and Ownership

ESM's underground Zinc Operation and Kilbourne Graphite resource are co-located on the same property approximately 7 miles southeast of Gouverneur, New York State, in St. Lawrence County. The 2,715 acres of surface rights owned by Titan are divided among the townships of Fowler, Edwards and Pierrepont, containing 1,769, 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.

1.4 History, Exploration, and Drilling

1.4.1 Zinc

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 continually produced since then.

Drilling in the district has been dominantly core drilling either with contract drillers such as Cabo, Major, and Boart Longyear, or by company owned and operated drills. The drillhole database contains 12,105 surface and underground diamond drillholes.

The Balmat mine as of December 31, 2024 produced a total of 36.4 Mton grading 8.6% zinc. A history of property ownership is listed in Table 1-1.

Table 1-1: 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: Taylor et al., 2024

1.4.2 Graphite

There is no history of graphite mining on the Property.

Graphite mineralization has long been documented within the Balmat-Edwards-Pierrepont district. Its occurrence has been logged as a curiosity, or defining mineralogical characteristic of geologic units within the region. Exploration activity targeting graphite began in 2022 with the sampling of five historic drillholes from the now Kilbourne Project, and three drillholes from the Company's Bostwick Creek target for graphitic carbon. In 2023, Company drilling at Kilbourne totaled 39 holes with 11,917 ft drilled.

1.5 Geology and Mineralization

1.5.1 Zinc

Zinc sulfide mineralization occurs within the Upper Marble, a stratigraphic unit of the Grenville Supergroup, composed of metamorphosed and complexly folded siliceous dolomitic marbles. Mineralization is concentrated primarily in the hinges of large fold structures.

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.

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 veins can display considerable geometrical variability depending on the degree of folding.

1.5.2 Graphite

Graphite mineralization occurs as 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. The Kilbourne graphite deposit footprints are up to 500 ft wide and 9,000 ft long.

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 56% for the UG operations. Currently, the concentrator is producing zinc concentrate at an average of 60% zinc with 3% iron and 0.50% magnesium.

While lead occurs at low grades in the historic #2 Mine, economic lead recovery is not viable, and ESM does not recover lead.

1.6.2 Graphite

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

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% Cg culminated in a flowsheet and conditions that produced a final concentrate grading 97.4% TC. 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% TC.

Forte Analytical conducted a testwork program on two composites grading between 2.4% and 2.5% Cg. The focus of the test program was to produce a concentrate grading at least 95% TC while minimizing flake degradation. The optimized flowsheet and conditions produced an upgraded flash concentrate grading 98.3% TC 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.

A second test program conducted by SGS subjected a Master composite and four variability composites to high-level optimization work. The test program produced final concentrate grades of at least 97.6% TC with open circuit graphite recoveries between 85.6% and 94.6%. Closed-circuit recovery will increase due to cycling of intermediate tailings streams. For design purposes, a closed-circuit graphite recovery of 90% is recommended.

While the execution of the test programs conducted by SGS and Forte Analytical 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% TC.

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

As of November 7 2025, the complete database for ESM consists of 12,105 diamond drillholes. 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 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 1-2). The 2025 model updates were completed in version 2024.1.3. Each zone has been analyzed and divided where appropriate to facilitate a more accurate estimation of the grade. In some cases, this has resulted in splitting of domains based on morphology or orientation for the purposes of estimation.

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

Zone	Modeling Method	Years Modeled and Updated	Model Volumes (ft^3)
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, Annually 2019 - 2025	19,400,000
Mud Pond	Implicit vein system model	2008, 2009, 2017, Annually 2019 - 2025	15,463,500
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, Annually 2020 - 2025	9,553,100
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

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 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 grades 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 (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 and LVA parameters were defined within the estimator based on the modeled vein surfaces.

The Underground and Open Pit Zinc Mineral Resources were modeled using Leapfrog Geo™ (version 2024.1.3) and estimated in Leapfrog Edge. The QP, Don Taylor, has reviewed the geological models and estimation results through site visits and remote sessions, assessed the methodologies and outcomes for consistency with industry standards, and is satisfied that the work is reasonable and suitable for reporting Mineral Resources.

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 June 9, 2025

Category	Tons (000's US short tons)	Zn (%)	Contained Pounds (M lb)
Measured	282	17.3	97
Indicated	1,133	16.0	362
Measured + Indicated	1,415	16.2	459
Inferred	4,512	12.1	1,088

Notes:

1. The qualified person for the 2025 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 2025 underground MRE encompasses 41 vein domains and 6 indicator RBF interpolant shells totaling 45 individual wireframes.
4. Geological and block models for the underground MRE used data from a total of 1,153 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 11 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-economic, marketing, or political issues or any other relevant issues that could materially affect this MRE.

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). The open pit is considered an accretive project with no G&A costs, and selling costs are incorporated into the selling price. The QP has reviewed these assumptions and considers them reasonable for the purposes of this Mineral Resource Estimate. The pit-constrained Mineral Resource and in situ metal for Turnpike is summarized in (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: Taylor et al., 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. The 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-economic, marketing or political issues or any other relevant issues that could materially affect this MRE.

1.7.2 Graphite

Drill Database

The Kilbourne Graphite Study 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 fill the volumes.

Drillhole sample 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 shells

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: Taylor et al., 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 one 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).
11. The QP is unaware of any known environmental, permitting, legal, title-related, taxation, socio-economic, marketing or political issues or any other relevant issues that could materially affect this MRE.

1.8 Mineral Reserve Estimates

There are no Mineral Reserves for the ESM Zinc Project or the Kilborne Graphite Project.

1.9 Mining

1.9.1 Zinc

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 are also used in the design where applicable. The mine plan scales up slightly from the current production rate of 2,275 ton/d to 2,800 ton/d by 2028 continuing through 2030 winding down in 2031. The current mine life is projected to be 6 years. During the period of production scale-up, mine tons will be supplemented by a small open pit operation called Turnpike.

The ESM deposit will be accessed from surface via the #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. As depth increases, a ventilation upgrade of a new raise to surface will be required to support the increasing fleet size.

Measured, Indicated, and Inferred Mineral Resources were included in the mine design and schedule optimization process. The mine plan is based on the Mineral Resource stated as of June 9, 2025 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 August 2025 with the production from 2025 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 good geotechnically, so overbreak is considered to be minimal. For LRS and back-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% of mine recovery.
- Panel mining assumes 75% of mine recovery after losses from pillars left behind.

For the open pit mine, mining dilution and mining loss have been incorporated into the mining block model by reblocking and regularizing the geological resource block model.

The production schedule for the underground LOM is provided in Table 1-6. The schedule for open pit extraction is in Table 1-7.

Table 1-6: Mine production schedule

Item	Unit	Total	2026	2027	2028	2029	2030	2031	2032
Underground Mill Feed	kton	4,161	511	558	552	658	690	663	0
Zinc Grade	%	7.4	8.4	7.9	7.3	7.1	7.1	7.1	0.0
Contained Zinc	M lb	620	85	88	81	94	98	94	0

Table 1-7: Turnpike open pit conceptual schedule

Item	Unit	Total	2026	2027	2028	2029
Open Pit Mill Feed	kton	**399**	30	88	221	60
Total Open Pit Waste	kton	**1,060**	114	235	622	89
Stripping Ratio	W:O	**2.7**	3.8	2.7	2.8	1.5
Total Material moved	kton	**1,459**	144	323	843	149
Zinc Grade	%	**3.2**	2.3	2.9	3.2	3.8
Contained Zinc	000s lb	**25**	1.4	5.1	14.3	4.5

Source: Asi & McCracken, 2024

1.9.2 Graphite

The PEA mine design and mine plan are based on Inferred Mineral Resources to support the scoping level mine planning work for the Kilbourne Graphite Study. Pit analyses were completed on the Kilbourne deposit to identify the potential in-pit mineable Mineral Resources. These in-pit mineable Mineral Resources, referred to as Potential Mill Feed (PMF), are not Mineral Reserves and do not have demonstrated economic viability.

Conventional owner-operated open pit mining methods will be used to mine the material within the designed open pit of the Kilbourne deposit. This method was selected considering the deposit's proximity to the surface.

The reference point at which in-pit mineable resources are defined is where the mill feed is delivered to the concentrator plant facility, which includes the run-of-mine stockpiles. It incorporates mining dilution and mining loss assumptions for the open pit mining method.

Changes in the following factors and assumptions may affect the in-pit mineable Mineral Resources estimate:

- Concentrate price;
- Interpretations of mineralization geometry and continuity of mineralization zones; grade and geology estimation assumptions;
- Geomechanical assumptions;
- Ability of the mining operation to meet the annual production rate;
- Operating cost assumptions;
- Process plant recoveries;
- Mining loss and dilution;
- Ability to meet and maintain permitting and environmental licence conditions.

Open pit mining will include drilling and blasting with a combination of a backhoe-type excavator and front-end loader-type excavator loading blasted material into haul trucks, which will haul the material from the bench to the designated destination of the crusher, run-of-mine stockpile, or mine waste stockpiles, depending on the material type. Support equipment includes dozers, graders, utility loaders, water truck, and service vehicles.

The operation scenario for the Kilbourne deposit involves:

- Mining starts without pre-production stripping period requirement in Year 1.
- Average annual mining rate of PMF and waste over the life of mine (LOM) is approximately 4.83 M tons, with a peak of approximately 8.0 M tons at Year 8.

- The total PMF is estimated at 19.95 M tons, with an average grade of 2.84% Cg, and a cut-off grade of 1.5% Cg over a 13-year mine life of the open pit mines, with an overall strip ratio of 2.15 to 1.
- Total material mined over the LOM is 62.77 M tons, consisting of 19.95 M tons of PMF, 30.82 M tons of waste rock, 6.44 M tons of overburden, and 5.56 M tons of existing zinc tailings material;
- An average of 1.53 M tons per year of PMF will be sent to the concentrate plant to produce graphite concentrates.
- The mine plan ramp-up targets for concentrate plant feed throughput are set at 1.2 M tons for Year 1, 1.4 M tons for Years 2 and 3, and increasing to 1.7 M tons from Year 4 onward. These throughput levels align with the concentrate production targets of 22.5 k tonnes in Year 1, 27.5 k tonnes in Year 2, 38 k tonnes in Year 3 and 40 k tonnes from Year 4 onward, reflecting production and processing constraints.

During full production, the mine equipment fleet requirements were calculated to be 10 haul trucks, two hydraulic excavators, one wheel loader, and three production drills, in addition to the fleet of support and service equipment. All mining mobile equipment will be leased. The total mine workforce will reach a peak of 62 employees for the mining segment only.

To manage water that collects in the open pits, sumps will be developed on the pit floor as mining progresses, and a series of pumps will be used to pump the water to settling ponds located at surface. It has been assumed that in general, a total of two pumps should be adequate to serve the needs of the open pit.

1.10 Recovery Methods

1.10.1 Zinc

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 in 2018. The concentrator flowsheet includes crushing, grinding, sequential lead and zinc flotation circuits, concentrate dewatering circuits, and loadout facilities.

Throughout the history of the Balmat operation (now ESM), the concentrator's design capacity of 5,000 tons per day has consistently exceeded the mines' production rate. The operating strategy is to operate the concentrator at its rated hourly throughput of 200 ton/h to 220 ton/h, but for only as many hours as necessary to suit mine production. It is currently processing between 10,124 ton/w and 11,375 ton/w 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 60% zinc with 3% iron and 0.5% magnesium.

Lead values in the underground mineralization are expected to be low, and no lead concentrate production is planned from this source. Although lead values in the Turnpike open pit are anticipated to be higher and could support lead concentrate production, no lead concentrate is planned to be produced.

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

1.10.2 Graphite

1.10.2.1 Concentrate and Micronization Plants

The graphite Concentrate Plant was designed to maximize graphite recovery and minimize flake degradation, while minimizing capital expenditure and operating costs.

The process consists of a crushing and grinding circuit, rougher and cleaner flotation, and final graphite concentrate dewatering and handling circuit. The cleaning circuit consists of one stage of polish grinding and three stages of stirred media milling followed by cleaner flotation after each grinding stage to separate the liberated graphite from the gangue minerals.

The proposed reagents are consistent with other graphite projects, namely diesel as the graphite collector and Methyl Isobutyl Carbinol as the frother (MIBC). No gangue depressant or pH modifier is required. The only other reagent is a yet to be determined flocculant that aids in the settling of the tailings.

The process design is based on metallurgical testwork that was conducted by two independent laboratories. The expected overall graphite concentrate grade and recovery are 95.0% TC and 90.0%, respectively. The average feed grade to the mill is 2.84% Cg based on the mine design.

The plant will produce up to 44,500 tonnes (t) of graphite per year and 37,500 tonnes per year (t/y) on average, with a mill feed range between 1,226,000 and 1,864,000 short tons (ton).

Only a small percentage of the natural flake graphite (NFG) concentrate is sold as-is and, instead, most of the product is upgraded further to achieve higher sales prices. The first upgrading step consists of milling the flakes into a micronized product for sale or as the feed material for the Purification Plant.

The Micronization Plant has been designed for a processing rate of up to 44,500 t/y (dry basis) using a phased approach. The expected availability of the Micronization Plant is 90% for a total annual operating time of 7,183 hours or 6.2 t/h. It is estimated that 97% of the graphite concentrate feeding the air classifier mills is converted to a micronized product with the balance captured in the ultra-fines product.

The dried flotation concentrate is micronized in air swept classifier mills to produce micronized graphite with two different size specifications.

1.10.2.2 Secondary Transformation Site

Micronized NFG concentrate will be processed at both the Purification Plant and CSPG Plant.

The Purification Plant comprises an acid leach process designed to process 10,670 t/y (dry basis) of micronized NFG concentrate with a fixed carbon (FC) content of ≥95 wt.%. This process purifies the material to produce PMG with a FC content of ≥99.90 wt.%. Based on an assumed yield of 94.11 wt.%, the Purification Plant is expected to produce approximately 10,042 t/y (dry basis) of PMG.

The CSPG Plant is designed to process 21,340 t/y (dry basis) of micronized NFG through a dedicated acid purification circuit, followed by spheroidization and coating. The resulting CSPG will have a FC content of ≥99.95 wt.%. Assuming a yield of 94.11 wt.% for purification, 70 wt.% for spheroidization, and 105 wt.% for coating, the CSPG Plant is expected to produce 14,761 t/y (dry basis) of CSPG.

Wastewater from the Purification Plant and CSPG Plant will be managed through the wastewater treatment plant, though this will be addressed during the next phase when the site location is finalized.

1.11 Infrastructure

1.11.1 Zinc

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 a transmission line from Niagara Mohawk's substation at Battle Hill-ESM #5 circuit. On-site power is distributed to the plant and mine. ESM 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 that will run the #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-inch (in) shaft water line to the various mine levels.

Tailings are placed in a permitted 260-acre conventional impoundment located approximately 4,000 ft north of the mill. The TMF is categorized as a low-risk dam by the New York State Bureau of Flood Protection and Dam Safety. Water from the tailings flows through a series of retention ponds before being discharged into Turnpike Creek under the New York State Department of Environmental Conservation (NYSDEC) 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 runoff will drain to the concentrator pond. The capacity of this stockpile area is sufficient for the tonnages in the contained mine schedule.

The ultimate capacity of the 260-acre footprint has been estimated at 20 million tons (Mton), with immediate capacity of 1.7 Mton, before further embankment construction is 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.

1.11.2 Graphite

The Kilbourne Site will be located within the boundaries of the existing ESM site and will include the open pit mine, tailings management facilities, a Concentrate Plant, and a Micronization Plant. As part of this vertically integrated model, PMF extracted from Kilbourne open pit mine will be processed at the Concentrate Plant and subsequently micronized at Kilbourne Site. The micronized NFG concentrate will then be transported to the Secondary Transformation Site.

The Secondary Transformation Site will accommodate both the Purification Plant and the Coated Spherical Purified Graphite (CSPG) Plant. These facilities will be situated in an industrial hub within New York State to leverage existing infrastructure and utilities. The transported micronized NFG concentrate will be further upgraded at the Purification and CSPG Plants, which aim to produce purified micronized graphite (PMG) and CSPG, respectively.

Delivering approximately 1.53 million tons (1.39 million tonnes) of mill feed annually, the Kilbourne open pit will produce around 40,000 tonnes of graphite concentrate per year. Mining activities will generate roughly 30.8 M tons of waste rock and 6.4 M tons of overburden, which will be stored in engineered stockpiles equipped with runoff collection systems. The site layout includes the open pit, concentrate plant, stockpiles, and tailings facilities, all strategically positioned to optimize operational efficiency and minimize environmental impact.

Infrastructure development at the Kilbourne Site will include new and upgraded roads totaling approximately 7.8 miles to facilitate haulage and plant access. Construction of the extended TMF will result in the closure of the western access to Sylvia Lake Road; however, local traffic will have unobstructed access through the eastern entrance from Balmat-Fowler Road.

A diesel fueling station will be installed with an initial capacity of 5,300 gal, expanding to 10,600 gal in Year 5 to meet growing demand. Maintenance facilities will feature a steel structure measuring 105 ft by 70 ft, equipped with three heavy equipment bays with overhead cranes, a light vehicle bay and a dedicated wash bay housed in a separate structure.

Utilities will require an estimated 12–15 MVA of electrical power, supplied through a new skid-mounted substation integrated with the existing ESM infrastructure. Preliminary discussions with National Grid indicate support for the additional electrical load. Process water will be sourced from treatment ponds and Sylvia Lake, potable water will be provided in bottled form, and wastewater will be managed via a buried tank serviced by local providers. Fire protection systems will include a 300-horsepower pump, one kilometer of buried piping, and 20 hydrants strategically placed around the process plant site.

Water management strategies for the Kilbourne Project are designed to integrate the Turnpike Creek and Sawyer Creek watersheds, employing a closed-loop system to minimize freshwater consumption and control discharges. Stormwater will be managed through segregation of clean and contact water flows, erosion control measures, and sediment basins. A preliminary water balance model has been developed to account for precipitation, runoff, groundwater inflows, and operational phases, ensuring adaptive management throughout the LOM. This model supports compliance with regulatory standards and anticipates future environmental requirements.

Tailings management is a critical component of the Project, involving a multi-stage plan that includes the Extended TMF, Raised TMF, Kilbourne Pit backfilling, and deposition in the Historic Arnold Pit. Over the LOM, approximately 484.6 million cubic feet of tailings will be managed, including both existing zinc tailings and new graphite tailings. TMF designs incorporate staged expansions, containment dikes, and seismic stability measures based on a peak ground acceleration of 0.34 g. The Historic Arnold Pit will require full dewatering before tailings deposition can begin and construction of containment dikes to provide the required capacity.

Processing infrastructure at the Kilbourne Site will include a Concentrate Plant equipped with offices, workshops, laboratories, reagent storage, and change rooms. The Micronization Plant will be collocated with the concentrator and will utilize turnkey air-swept classifier milling systems requiring electricity and compressed air. Maintenance for micronization equipment will be minimal, with periodic replacement of wear parts. One-ton bulk bags of finished products will be stored in the concentrator warehouse.

The Secondary Transformation Site will house the Purification and CSPG Plants. It is assumed to be located in an established prime chemical industrial area, providing strategic advantages such as access to developed plots, bulk utilities, and cost-effective service solutions. These plants will require additional infrastructure, including internal roads, parking bays, step-down transformers, and specialized facilities for handling hazardous materials such as hydrofluoric acid. Onsite utilities will include water purification systems, steam generation units, compressed air systems, and integrated power and water distribution networks. The location of this site is under evaluation, with preference given to industrial hubs within New York State to ensure proximity to transportation infrastructure and international markets. ESM is in discussions with counterparties regarding a few site locations that meet these requirements.

The Kilbourne Graphite Study emphasizes leveraging existing ESM infrastructure while introducing new facilities to support graphite production. The Project integrates robust water and tailings management strategies and comprehensive utility systems to ensure operational efficiency, environmental compliance, and long-term sustainability.

1.12 Environment and Permitting

1.12.1 Zinc

All permits required to operate the ESM #4 Mine are active and in place. Additionally, there are no 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 #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 the length of time elapsed since the 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 ESM's SPDES permit and is expected to remain so for operating, closure, and post-closure periods.

The ESM #2 Mine site has been partially reclaimed. ESM #2 Shaft serves as secondary access to the underground operations at the #4 Mine and will be included in the final reclamation of the #4 Mine and concentrator complex. Reclamation of the ESM #4 Mine and tailings is assured with a $2,701,000 surety bond.

1.12.2 Graphite

1.12.2.1 Mine, Concentrator, and Micronization Plant

New York State Permitting

A major modification to ESM's mining permit will need to be approved by the NYSDEC. The proposed Kilbourne Pit extends beyond the currently permitted LOM boundary; consequently, the mining permit application (for modification) will necessitate a *State Environmental Quality Review Act* (SEQR) review due to the potential for significant environmental impacts. In accordance with NYSDEC requirements, a comprehensive suite of environmental and technical studies must be completed prior to submitting a permit modification application. These include wetlands delineation, visual and noise impact assessments, residential well surveys, pre-blast building inspections, traffic analyses, and a hydrogeologic impact evaluation. Although it is not anticipated, the large area of the Kilbourne Project's footprint is such that an archaeological or

cultural resources survey may be required. The necessity of such studies would be determined by the NYSDEC, in consultation with the State Historic Preservation Office (SHPO). If SHPO identifies a potential sensitivity, it will require a Phase 1A archaeological assessment, and a subsequent Phase 1B field survey if an archaeological site is identified. The above studies will all be incorporated into a revised Mined Land Use Plan (MLUP) to accompany ESM's mining permit application. ESM is already advancing discussions with the regulators to kick-start the required permitting.

Upon completion of the required studies and MLUP revisions, the mining permit application will be submitted to the NYSDEC. It is anticipated that the Project will be classified as a Type I action under SEQR, which typically requires a Draft Environmental Impact Statement (DEIS) when a positive declaration is issued by NYSDEC. To ensure compliance with applicable standards and operational requirements, the following will likely require resubmission for review and approval: Air Registration/Permit, Water Withdrawal Permit (WWP), Building Permits, and a State Pollution Discharge Elimination System (SPDES) permit modification.

The graphite at the Kilbourne Site comes from a similar host rock as the zinc that ESM currently mines. As a result, the tailings from graphite processing are expected to be non-acid-generating. Water used in graphite processing, along with quarry pump-out water from the Kilbourne Pit and the Historic Arnold Pit, will be recycled as much as possible. Any water not recycled will be released into ESM's treatment system, together with the zinc process water.

The area of the Kilbourne Graphite requires removing parts of the existing tailings ponds and serpentine ponds, as well as relocating the SPDES discharge point further downstream. ESM's SPDES permit will need to be updated to reflect these changes, including the addition of graphite tailings and process water, a revised TMF, and a new settling pond.

ESM's WWP will be modified to include any new water sources, pumps, or increased water usage due to the addition of the graphite processing.

Federal Permitting

Given the likelihood of federal funding for the Kilbourne Project, its development may require a small number of well-defined federal permits and reviews, consistent with other large mining projects in the US. In this respect, key authorizations are outlined below:

- ***Clean Water Act* Section 404 Permit (U.S. Army Corps of Engineers):** Required where the mine footprint intersects federally regulated wetlands or streams. This is a standard permit for any US mining project having these impacts. Current wetlands delineation is ongoing to determine the applicability of this.

- **Section 401 Water Quality Certification (WQC):** Section 401 of the *Clean Water Act* requires that any applicant for a federal permit that may result in a discharge to waters of the United States must obtain a WQC. In New York, the 401 WQC is issued by NYSDEC, but it is tied to the Army Corps permit. The 404 permit cannot be obtained without NYSDEC's 401 WQC.
- ***National Environmental Policy Act* (NEPA) Environmental Impact Statement:** Provides a comprehensive review of environmental, cultural, and community impacts, led by the United States Army Corps of Engineers (USACE) in coordination with other federal agencies. To avoid duplication, ESM will coordinate with the NYSDEC and the USACE to ensure that the preparation of the DEIS will also serve the needs of the NEPA EIS for USACE purposes. USACE has indicated that it may require the less comprehensive Environmental Assessment (EA) for the Kilbourne Project, rather than an EIS; however, this decision is only preliminary at this time.
- ***Endangered Species Act* (Section 7):** To confirm there are no impacts to federally protected species. Based on a preliminary review of the Project, current species potentially present include Northern long-eared bat (Endangered), tricolored bat (proposed endangered), and the monarch butterfly (proposed threatened). It could restrict tree clearing to certain times of the year.
- ***National Historic Preservation Act* (Section 106):** To review potential cultural or historic resources. ESM does not expect anything specific in this regard to the Kilbourne Project (see above under New York State Permitting).

Given the aspect of federal funding associated with Kilbourne, the Project has a FAST-41 designation. This federal program coordinates permitting schedules across agencies and sets clear deadlines. FAST-41 designation ensures transparency, accountability, and limits legal challenges, giving investors confidence that the Project can be permitted on a predictable timeline. ESM is working with federal and state agencies to align permitting schedules given the strategic nature of the Kilbourne Project and has received indication of the prospect of alignment and completion of any federal permits within a 16-month timeline, which is aligned with the projected state permitting.

1.12.2.2 Secondary Transformation Site

State Permitting

Regarding Secondary Transformation Site (Purification and CSPG Plants), these components are not included in the current Graphite Study due to the early stage of the Graphite Project. Once a final location for the proposed Purification Plant and CSPG Plant is selected, the Project will undergo a full SEQR review, and it is likely that a Draft Environmental Impact Statement (DEIS) will be required due to the scale and nature of the facility. The DEIS will evaluate all potential environmental, socio-economic, and physical impacts associated with the plant's construction

and operation, including air and water emissions, land disturbance, noise, traffic, community effects, and any other relevant considerations, ensuring that these issues are fully identified and addressed.

Federal Permitting

It is not anticipated that any federal permitting will be required, however, this is dependent on site selection and may change.

1.13 Operating and Capital Cost Estimates

1.13.1 Zinc

Estimated Project capital costs (including closure costs) total $68.6M, consisting of the following distinct areas:

- #4 infrastructure and process capital;
- #4 mining capital equipment;
- #4 mining capital development;
- N2D and expansionary 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 2025 US$ with no escalation.

Table 1-8: Capital cost summary

Area	Cost Estimate ($M)
#4 Infrastructure & Process Capital	5.4
#4 Mining Capital Equipment	8.0
#4 Mining Capital Development	9.6
N2D and Expansionary Capital	35.5
Total Capital Cost	**58.5**
Closure Costs	15.2
Salvage Value	(5.1)
Total Capital Cost (including closure costs)	**68.6**

Capital costs for the #4 Mine are estimated to be $23M. This includes replacements for two mechanical bolters, three LHDs, two haul trucks, and underground support equipment. Additional equipment including five LHDs, two haul trucks, an additional bolter, two additional jumbo drills, transformers required for electrical expansions, and ventilation fans and doors. The costs of additional equipment are applied by the expected area of use.

ESM has assumed that due to the short life of the pits (4 years), labor resources used to mine the open pit will eventually be shared with the graphite pit.

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 $5.1M 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 Item 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 2025 US$ on a calendar year basis. No escalation or inflation is included.

Table 1-9: Breakdown of estimated site operating costs

Underground	Fixed Cost ($K/y)	Variable Cost ($/ton milled)	LOM Cost ($M)
#4 Mine			
Mining – Mineralized Material	351.00	53.00	165.25
Mining – Waste	-	17.00	6.88
Processing	854.00	10.00	38.25
G&A	9,256.00	-	64.66
Concentrate Transportation	8.23	8.00	31.01
Royalties	0.09	0.18	0.17
Subtotal	**10,469.32**	**88.18**	**306.22**

Underground	Fixed Cost ($K/y)	Variable Cost ($/ton milled)	LOM Cost ($M)
#2 Mine			
Mining – Mineralized Material	-	33.00	37.21
Mining – Waste	-	17.00	4.85
Processing	-	10.00	11.28
G&A	-	2.00	2.26
Concentrate Transportation	-	-	-
Royalties	**-**	**-**	**-**
Subtotal	**-**	**62.00**	**55.60**
Total	**10,469.32**	**150.18**	**361.82**

1.13.2 Graphite

The capital and operating estimate is classified as a Class 5 estimate, as defined in the Association for the Advancement of Cost Estimation (AACE) Recommended Practice No. 47R-11, typically used for preliminary evaluations. For the purpose of this study, an accuracy range of approximately +/- 40% has been assumed.

All capital and operating estimate are expressed in United States dollars ($), and the base date of estimate and currency exchange rates were obtained on Q1 2025.

The total capital cost for the Kilbourne Study is estimated at $431.7M, which includes initial, expansion, and sustaining capital requirements. The Kilbourne Project involves two sites. The first is the Kilbourne Site, which includes the open pit, site infrastructure and TMF, the Concentrate Plant, and the Micronization Plant. The second is the Secondary Transformation Site, which includes the Purification Plant and the CSPG Plant. The Micronization, Purification, and CSPG facilities are constructed in two phases—initial and expansion—as reflected in the capital cost schedule in the cash flow model.

A summary of the capital costs, including initial, expansion, and sustaining capital, is presented in Table 1-10

Table 1-10: Initial, expansion and sustaining capital costs

Project Area	Total ($K)	Initial Costs ($K)	Expansion Costs ($K)	Sustaining Costs ($K)
Open Pit Mine	41,641	-	-	41,641
Site Infrastructure and TMF	46,150	27,241	-	18,909
Concentrate Plant	115,922	72,610	-	43,312
Micronization Plant	22,362	11,497	10,865	-
Purification Plant	13,311	5,291	8,020	-
CSPG Plant	99,477	-	99,477	-
Closure and Salvage	-4,065	-	-	-4,065
Direct costs	**334,799**	**116,639**	**118,362**	**99,798**
Owner's Cost and Indirects	49,939	15,792	33,900	247
Contingency	47,000	23,328	23,672	-
Total	**431,738**	**155,759**	**175,934**	**100,045**

The operating cost estimate, similar to capital cost estimate, includes all activities at both the Kilbourne Site and the Secondary Transformation Site. Operating costs cover the open pit mine, site infrastructure, TMF, and the Concentrate, Micronization, Purification, and CSPG Plants, as well as the transportation of micronized NFG from the Kilbourne Site to the Secondary Transformation Site.

Over the 13-year life of mine, operations are based on 62.77 M tons of material mined, 18.10 M tonnes (19.95 M tons) milled, and total production of 486.7 k tonnes of graphite concentrate, resulting in saleable products of 157.6 k tonnes of micronized NFG, 121.4 k tonnes of PMG, 110.8 k tonnes of CSPG, and 26 k tonnes of remaining concentrate.

Total life of mine operating costs are estimated at $886M presented in Table 1-11.

Table 1-11: Project All-in Operating Costs

Project Area		LOM Total	Remaining Concentrate	NFG Micronized	PMG	CSPG
		($K)	($/t Concentrate)	($/t Micronized NFG)	($/t Saleable PMG)	($/t Saleable CSPG)
Open Pit	Kilbourne Pit Mining	183,832	378	389	414	563
Site Infrastructure	G&A	47,077	97	100	106	144
	TMF	11,178	23	24	25	34
Concentrate and Micronization Plants	Concentrate Plant	239,677	492	508	539	734
	Micronization Plant	75,647	-	176	186	228
Purification and CSPG Plants	Transport[1]	21,694	-	-	80	108
	Purification Plant	107,222	-	-	883	-
	CSPG Plant	199,610	-	-	-	1,801
Operating Costs		**885,936**	**990**	**1,197**	**2,233**	**3,612**

Notes:

[1] Micronized NFG concentrate transportation cost from Kilbourne Site to Secondary Transformation Site.

Numbers may not add up due to rounding.

1.14 Economic Analysis

1.14.1 Zinc

Item 22 of Form 43-101F1 permits producing issuers to exclude the information required under Item 22 for technical reports on properties currently in production, provided there is no material expansion of those operations. As no material expansion of ESM's Zinc Operations is planned, the information required under Item 22 related to ESM's Zinc Operations has been excluded from this report.

1.14.2 Graphite

A 7% discount rate was applied to the cash flow to derive the Kilbourne Study project's net present value on a pre-tax and post-tax basis. Cash flows have been discounted annually starting in the second year with an end of year period under the assumption that major project financing would be carried out at this time.

Table 1-12 presents the anticipated operating results for the potential future mining operations at the Kilbourne Project. Based on the expected product distribution, the weighted average sales price listed in Table 1-13 has been considered for the Project.

The summary of the financial evaluation for the base case of the Project is presented in Table 1-14.

Table 1-12: Operating results summary

Parameters	Unit	Value
Physicals		
Mine Life	year	12.8
Total Material Mined	ton	62,769,000
Total Waste Mined	ton	42,818,000
Total ROM Mined	ton	19,951,000
ROM Head Grade	% Cg	2.84
Mill Recovery	%	89.7
Total Metal Tonnage Recovered	ton	509,670
	tonne	462,364
Concentrate Grade	%	95%
Total Concentrate Produced	tonne	486,699
Operating Costs		
Mining	$/ton milled	9.21
Concentrate Processing	$/ton milled	12.01
G&A	$/ton milled	2.36
Secondary Processing	$/ton milled	20.26
Tailings Relocation	$/ton milled	0.56
Total Operating	$/ton milled	44.41
	$/tonne milled	48.95
Operating Costs of Salable Products		
STD Purity Flake Concentrate	$/tonne	990
STD Purity Micronized Flake Grades	$/tonne	1,197
High Purity Micronized Flake Grades	$/tonne	2,233
CSPG Anode Grades	$/tonne	3,612
Capital Costs		
Initial Capital	$M	155.8
Expansion Capital	$M	175.9
Sustaining Capital	$M	100.0

Table 1-13: Study input pricing

Product	Weighted Average Sale Price ($/Mt)
STD Purity Flake Concentrate (95.0% LOI MIN)	1,575
STD Purity Micronized Flake Grades (95.0% LOI MIN)	3,770
High Purity Micronized Flake Grade (99.9% LOI MIN)	5,185
CSPG Anode Grades (99.95% LOI MIN)	11,193

Table 1-14: Financial analysis summary

Parameters	Unit	Value
Pre-Tax Cash Flow	$M	1,187.7
Pre-Tax NPV	$M	580.6
Pre-Tax IRR	%	38.9%
Pre-Tax Payback Period	year	2.66
Taxes	$M	134.1
Post-Tax Cash Flow	$M	1,053.6
Post-Tax NPV	$M	513.2
Post-Tax IRR	%	37.0%
Post-Tax Payback Period	year	2.69

1.15 Adjacent Properties

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

1.16 Other Relevant Data and Information

To the best of the authors' knowledge, there is no other relevant data, additional information or explanation necessary to make the report understandable and not misleading.

1.17 Interpretation and Conclusions

1.17.1 Zinc

The ESM operation has a long history of successful mining and mineral processing, with over a century of production and a strong record of replacing Mineral Resources through ongoing

exploration. The current Mineral Resource estimates are supported by extensive drilling, mapping, and geological modeling using Leapfrog™ Geo and Edge software. Mining will employ a combination of underground methods such as LRS, Cut and Fill, Panel Mining, longhole stoping and a small open pit, with material processed at the existing ESM concentrator, which has demonstrated reliable performance and requires no major modifications. All key permits remain active and in good standing, and no significant access or title risks have been identified. While economic outcomes remain sensitive to commodity prices, dilution control, recoveries, and ventilation constraints, these risks are typical of comparable operations and can be mitigated through proper engineering and planning.

1.17.2 Graphite

Based on the assumptions and constraints outlined, it is the conclusion of the QPs that this Preliminary Economic Assessment demonstrates reasonable technical and economic viability and is considered suitable to advance to the next stage of development. Several opportunities and risks were identified during the study and should be addressed in the next phase.

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.

1.18 Recommendations

1.18.1 Zinc

The items shown in Table 1-15 are recommended for ESM to improve confidence and performance of the PEA mine plan and economics.

Table 1-15: Project recommendations and estimated cost

Item	Cost ($)
Infill and Exploration Drilling	1,230,000
Ventilation Trade-off Study	50,000
Sorting Testwork and Integration Study	100,000
Total Estimate	**1,380,000**

The items shown in Table 1-16 are recommended exploration activities for ESM to advance systematic district-scale exploration and prioritize targets within historically productive stratigraphies.

Table 1-16: Cost estimate for recommended exploration activities

Item	Estimated Cost ($)
Surface Geochemical Sampling	200,000
Near Mine – Exploration Drilling	670,000
Exploration Drilling	1,130,000
Geophysics	115,000
Land Acquisition and Management	-
Estimate for 2026	**2,115,000**
Annual Estimate	**2,000,000**

1.18.2 Graphite

The items listed in Table 1-17 are proposed for Titan to proceed with the Project advancement for the Kilbourne Site and Secondary Transformation Site. The recommended Graphite programs for sites are not successive.

Table 1-17: Project recommendations and estimated cost

Recommended Items	Estimated Cost ($)
Kilbourne Site	18,745,000
Secondary Transformation Site	7,851,600
Total Estimate	**26,596,600**

1.19 References

All references in this report can be found in Item 27.

2. Introduction

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

This Technical Report titled *"Empire State Mines 2025 NI 43-101 Technical Report"* provides an update to the ESM Zinc Mineral Resource Estimate (MRE) and mine plan, and provides a Preliminary Economic Assessment (PEA) for the Kilbourne Graphite Project (the "Graphite Study").

The ESM Zinc Mine update and the Graphite Study are included in the same technical report as the projects are contiguous and development of the Kilbourne Graphite Project would share certain infrastructure used in the current ESM Zinc Mine.

ESM is an existing 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 2,715 acres of fee simple surface rights in three towns in St. Lawrence County. The majority of the Property consists of the 1,769 acres in the town of Fowler where the ESM shafts, mill, and tailings management facilities (TMF) 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. The mineral rights of these parcels are part of the 51,428 acres owned by ESM. The Company has lease and option to lease agreements on an additional 72,960 acres. In total, ESM has 124,388 acres within its mineral tenure.

ESM is currently comprised of a group of high-grade zinc mines, including two active underground operations: the ESM #4 Mine, which resumed mining in January 2018 and began producing zinc concentrate in March 2018, and the ESM #2 Mine, which is currently in production in the N2D (Number 2 Deep) zone. The ESM #1, #3, Hyatt, Pierrepont and Edwards mines are all within a 30-mile radius of the 5,000 t/d mill. Near-surface mineralization with open pit potential has been identified at the historic #1 and #2 mines, collectively referred to as the Turnpike zone.

The development of the Kilbourne Graphite Project including Secondary Transformation Site, which consists of a Purification Plant and CSPG Plant, forms a core part of Titan's strategy to establish a secure end to end domestic source of supply of natural flake graphite (NFG) for core US industries such as the defense, battery and energy sectors. This strategy includes mining NFG from the Kilbourne graphite deposit located in St Lawrence County, NY State, USA. The NFG is concentrated and micronized at a dedicated facility to be constructed within the existing ESM permit boundary in Gouverneur, NY State. The resulting micronized concentrate will be transported to another site in NY State for further processing at either the Purification Plant or the

CSPG Plant, where it will be upgraded into PMG or transformed into CSPG. These advanced materials serve critical applications across high-performance industrial, energy and defense sectors.

The Purification Plant and CSPG Plant will be located on the same property; however, the exact location within NY State has not yet been finalized. For the PEA, it is assumed that the plants will be located in an established prime chemical industrial estate or such other industrial location that has existing bulk supply services, access road and connection for gas, water, electricity, effluent and sewage disposal and treatment, stormwater management, and waste removal. ESM is in active discussions on a few sites that meet these requirements.

2.1 Basis of the Technical Report

The following companies contributed to the preparation of this Technical Report and provided qualified person (QP) sign-off for their respective items.

Empire State Mines

- Zinc geology, Mineral Resource Estimates, mining, infrastructure, and operations.

BBA USA Inc. (BBA)

- Overall report integrator;
- Kilbourne Mineral Resource Estimate;
- Kilbourne minable resource plan and production schedule;
- Kilborne surface infrastructure excluding the mineral processing plant;
- Tailing management facility, related to Kilborne Graphite Project.

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

- Zinc metallurgical testwork and mineral processing.

Metpro Management Inc. (Metpro)

- Graphite metallurgical testwork and mineral processing.

Dorfner Anzaplan UK Limited (Anzaplan)

- Graphite Secondary Transformation.

Alpha Geoscience

- Environmental studies, permitting and social or community impact.

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

The key information used in this report is listed in Item 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 1, 2025.

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**	**Items of the Report**
Donald R. Taylor Vice Chair	Titan	No	August 20-22, 2024	SME Registered Member	Items 1 (except 1.6, 1.7.2, 1.9.2, 1.10, 1.11.2, 1.12, 1.13.2, 1.14.2), 3 to 11, 12 (except 12.1.3, 12.1.4, 12.1.5, 12.2), 14 (except 14.2), 15 (except 15.2), 16 (except 16.3), 18.1, 19, 21.1.1, 21.2.1, 23, 24, 25.1, 26.1. Co-author of Item 27.
Todd McCracken Director – Mining & Geology – Central Canada	BBA	Yes	August 26-27, 2024 July 23-24, 2025	PGO	Items 1.7.2, 1.14.2, 12.2.1, 14.2, 21.1.2.7, 21.1.2.8, 22, 25.2.1, 25.2.8, 26 (except 26.1, 26.2.1.2, 26.2.1.3, 26.2.2, 26.2.3). Co-author of Items 25.2.9, 25.2.10, 27.
Bahareh Asi Principal Mining Engineer / National Practice Leader, Geology, Mining and Process	BBA	Yes	None	PEO	Items 1.8, 1.9.2, 1.13.2, 2, 12.2 (except 12.2.1, 12.2.3, 12.2.4, 12.2.5, 12.2.6, 12.2.7), 15.2, 16.3, 21 (except 21.1.1, 21.2.1, 21.1.2.2.3, 21.1.2.2.4, 21.1.2.2.5, 21.1.2.4, 21.1.2.5, 21.1.2.6, 21.1.2.7,21.1.2.8, 21.2.2.4, 21.2.2.5, 21.2.2.6, 21.2.2.7), 25 (except 25.1, 25.2.1, 25.2.3 to 25.2.8), 26.2.1.2. Co-author of Items 25.2.9, 25.2.10, 27.
David Willock Senior Mining Engineer	BBA	Yes	July 23-24, 2025	PEO	Items 1.11.2, 12.2.5, 18 (except 18.1, 18.2.8 to 18.2.10), 21.1.2.2.3, 21.1.2.4, 21.2.2.4, 25.2.3, 26.2.1.3. Co-author of Items 25.2.9, 25.2.10, 27.
Deepak Malhotra Director of Metallurgy	Forte Dynamics	Yes	2016	SME Registered Member	Items 1.6.1, 1.10.1, 12.1.3, 13 (except 13.2), 17 (except 17.2). Co-author of Item 27.

Qualified Persons Responsible for the Preparation of this Technical Report					
Qualified Person	**Employer**	**Independent of Titan?**	**Date of Last Site Visit**	**Professional Designation**	**Items of the Report**
Oliver Peters Mineral Processing Engineer / President	Metpro	Yes	October 30, 2024	MSc, P.Eng., MBA	Items 1.6.2, 1.10.2 (except 1.10.2.2), 12.2.3, 13.2, 17.2 (except 17.2.3), 18.2.8, 18.2.9, 21.1.2.2.4, 21.1.2.5, 21.2.2.5, 21.2.2.6, 25.2.4 to 25.2.6. Co-author of Items 25.2.9, 25.2.10, 27.
Derick de Wit Managing Director	Anzaplan	Yes	None	FAusIMM	Items 1.10.2.2, 17.2.3, 18.2.10, 20.2.6, 21.1.2.2.5, 21.1.2.6, 21.2.2.7, 25.2.7, 26.2.2. Co-author of Items 25.2.9, 25.2.10, 27.
Steven M. Trader Senior Geologist	Alpha Geoscience	Yes	October 1-2, 2025	PG, CPG	Items 1.12, 12.1.4, 12.1.5, 12.2.4, 12.2.6, 20 (except 20.2.6), 26.2.3. Co-author of Item 27.

Note: Where a QP is noted as not having conducted a personal inspection at the Property, it was determined that such a personal inspection was not required for the QP to complete their scope of work for this Technical Report.

2.2 Units, Currency, and Rounding

The units of measurement used in this Technical Report follow the Imperial system, unless otherwise noted. All dollar figures quoted in this report refer to US dollars (US$, $) 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, entirely for matters pertaining to legal ownership of mineral concessions, surface rights and mining leases, as disclosed in Item 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 Demetrius Stavros (Senior Consultant), Thomas Seguljic (Principal), and Jesse Zahn (Principal & Regional Manager), HRP Associates, Inc., entirely for matters pertaining to environmental factors, as disclosed in Item 20.2.4, pursuant to statements made by HRP that were confirmed to be current as of the effective date of the Technical Report.

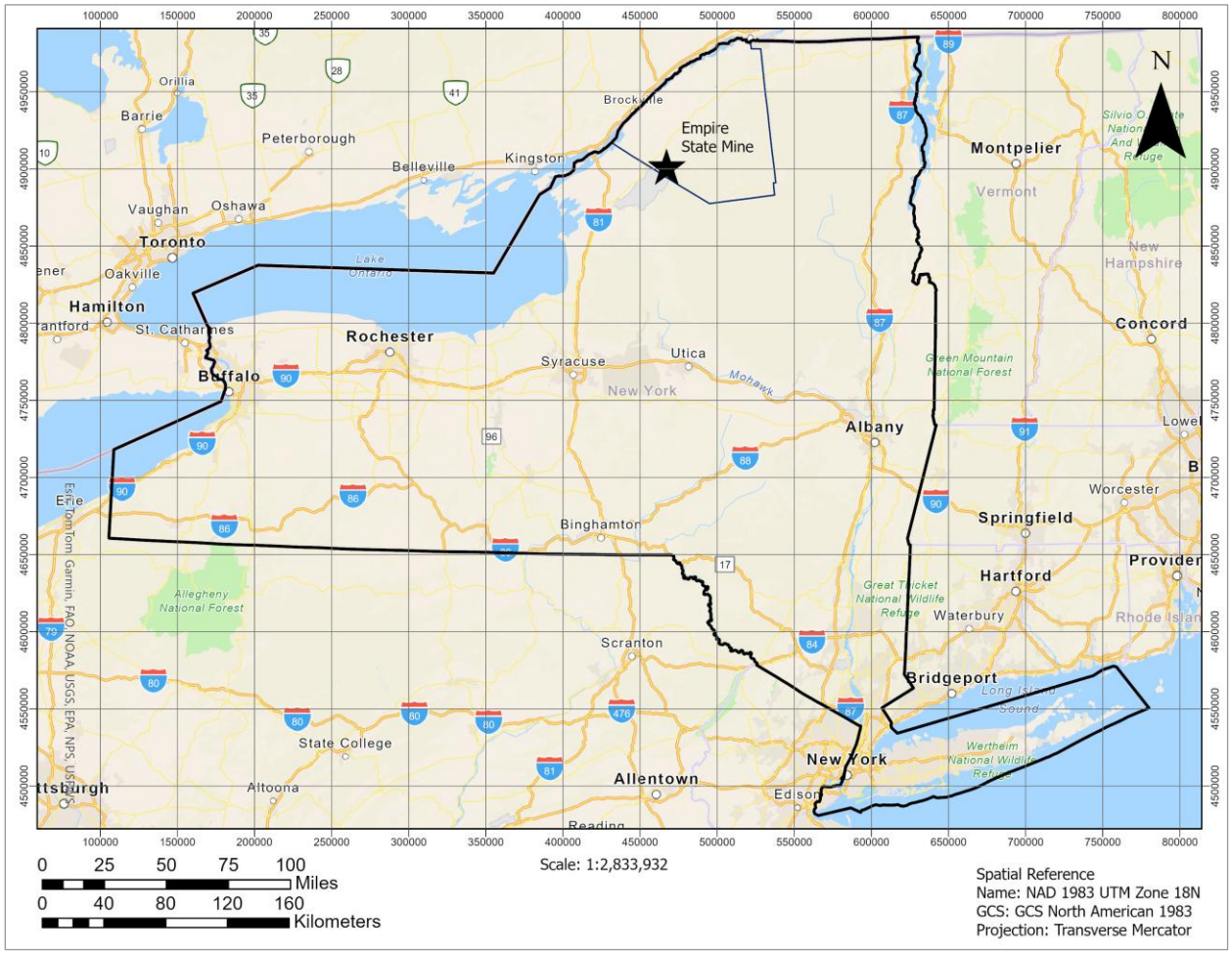
The QPs relied on Kevin Hart, Chief Financial Officer of Titan Mining Corporation, entirely for matters pertaining to taxation on the Property, as disclosed in Item 22, pursuant to statements made by Mr. Hart that were confirmed to be current as of the effective date of the Technical Report.

4. Property Description and Location

4.1 Location

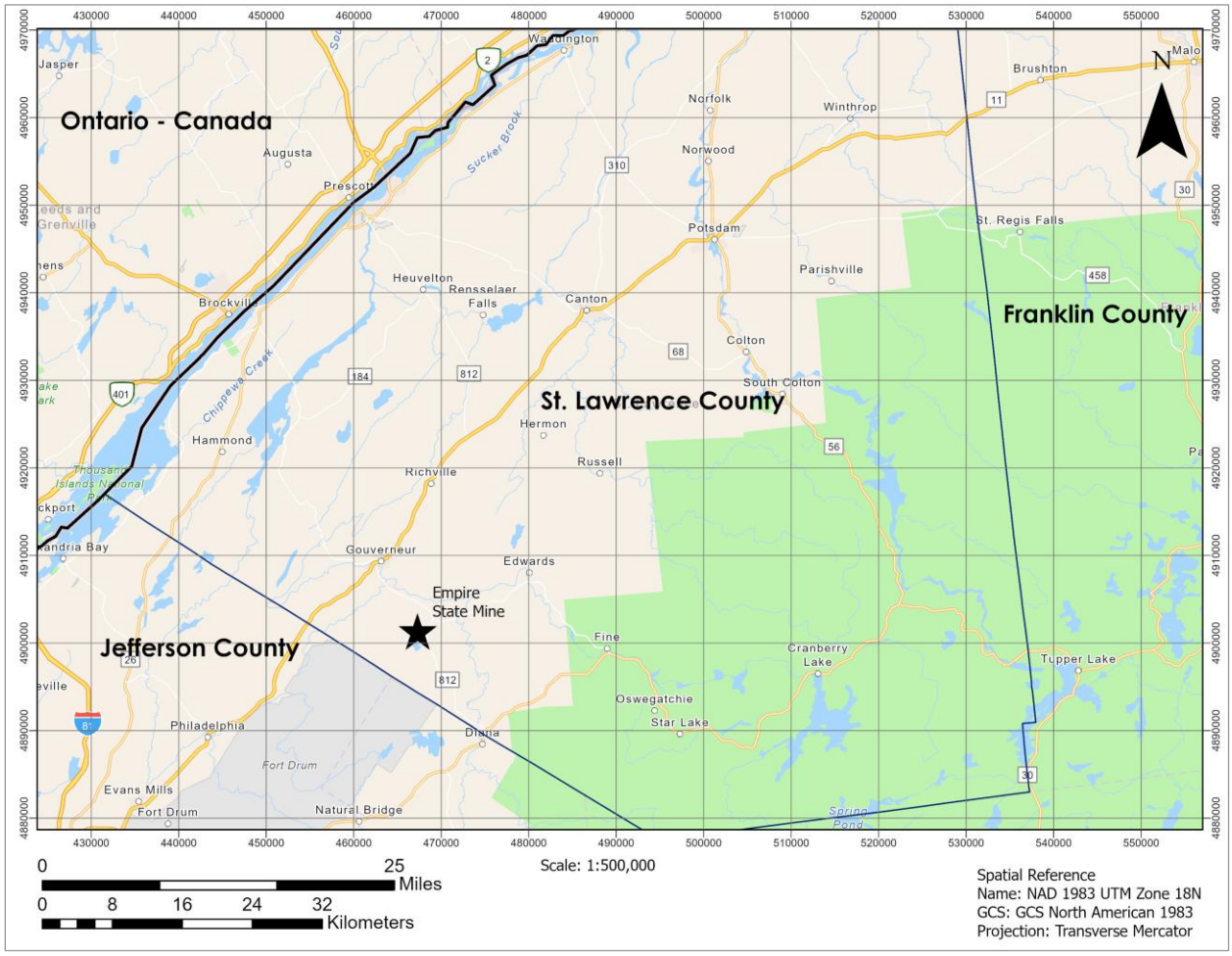
ESM's underground zinc mine and Kilbourne Graphite Project are co-located on the same property approximately 7 miles southeast of Gouverneur, New York State, at latitude 44°14'51" N, longitude 75°23'50" W, and an elevation of 710 feet above mean sea level (amsl). The site is accessible via State Road #812, located 38 miles from the St. Lawrence Seaway at Ogdensburg, New York (Figure 4-1 and Figure 4-2). The Kilbourne Graphite Project is approximately 4,000 feet northwest of the existing ESM mill and mine infrastructure.

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



Source: ESM 2025, modified from ESRI base map

Figure 4-1: Regional project location



Source: ESM 2025, modified from ESRI base map

Figure 4-2: Local project location

4.2 Mineral Tenure

The 2,715 acres of surface rights owned by Titan are divided among the townships of Fowler, Edwards and Pierrepont, containing 1,769, 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 72,960 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. The Company entered into a lease and option to lease agreement with St. Lawrence County in May 2025. The lease agreement is for an initial period of 25 years, and renewable for an additional 15 years through three 5-year extensions. The option to lease agreement is for an initial 5 years, and renewable for 20 years through four 5 -year extensions. In lieu of an annual payment based on acreage, the Company will pay a value equal to the annual property and school taxes of the included parcels.

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 is provided in Table 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 parcels historically included within the ESM mineral tenure had been claimed through tax sale 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*	135	20 years: NOT renewable	4%
Whitman Lease	Lease	10/02/2018*	30	20 years: renewable for additional 20 years	4%
Brian Tripp Lease (90Ac)	Lease	22/03/2021*	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*	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%
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%

Name	Type	Expiration Date	Acres	Term	NSR
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%
St. Lawrence County	Lease	05/05/2050	37,867	25 years; renewable for additional 15 years	N/A
St. Lawrence County	Option	05/05/2030	6,075	5 years; renewable for additional 20 years	N/A

*Refer to Item 4-2 for additional information on the expired leases.

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 shows surface and mineral parcels assessed to the Company.

Table 4-2: Mineral tenure information

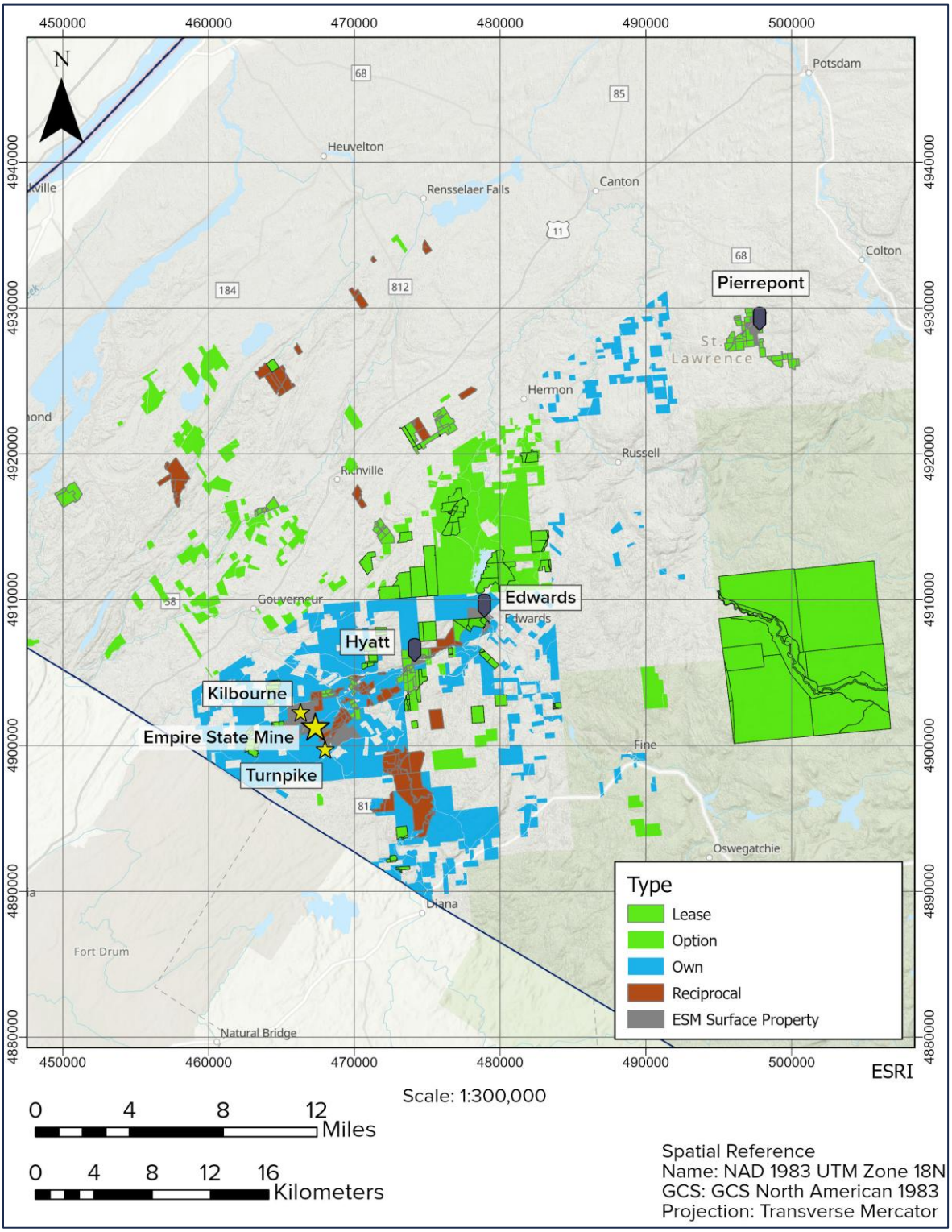
Assessor Parcel Number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2025 Taxes ($)
119.001-1-8	Pierrepont	80.4	-	-	322	410.22
119.001-1-10	Pierrepont	102.1	-	-	330	520.87
119.001-1-11	Pierrepont	0.52	-	-	720	1.70
119.001-1-12	Pierrepont	59.3	-	-	720	353.63
119.001-1-18./1	Pierrepont	-	1.4	-	720	42.56
174.004-3-2	Edwards	0.85	-	-	314	48.03
174.004-4-2	Edwards	10.37	-	-	720	199.00
174.004-4-1	Edwards	1.35	-	-	314	86.92
175.003-3-1.1	Edwards	71.6	-	-	720	617.59
175.003-3-19.1	Edwards	3.4	-	-	720	118.95
175.002-1-5.1	Edwards	370.2	-	-	323	2,667.08
175.002-1-33	Edwards	161.7	-	-	323	1,237.47
175.002-1-34.1	Edwards	72.2	-	-	330	622.16
175.002-1-32.1	Edwards	11.7	-	-	330	208.14
175.002-1-34./1	Edwards	-	74	-	720	162.40
1.044-18	Edwards	-	100	-	720	160.11
175.002-1-25./1	Edwards	-	92.2	-	720	150.97
175.001-1-4./1	Edwards	-	165	-	720	162.40
175.002-1-5./1	Edwards	-	1,044	-	314	599.29

Assessor Parcel Number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2025 Taxes ($)
175.003-1-1./2	Edwards	-	72	-	720	150.97
175.003-1-1./4	Edwards	-	18.8	-	720	150.97
175.003-3-1.1/1	Edwards	-	70	-	720	473.48
175.003-3-1.1/4	Edwards	-	-	Electrical	720	1,326.67
175.003-3-10./1	Edwards	-	115	-	720	150.97
175.003-3-13./2	Edwards	-	53.1	-	720	150.97
175.004-1-3./1	Edwards	-	58	-	720	150.97
175.004-1-6./1	Edwards	-	20	-	720	150.97
175.004-1-7./1	Edwards	-	63.8	-	720	150.97
175.004-1-11./1	Edwards	-	97.4	-	720	242.46
175.004-1-14./2	Edwards	-	62	-	720	150.97
187.002-2-1./1	Edwards	-	30	-	720	150.97
187.002-2-1./2	Edwards	-	80.9	-	720	150.97
188.001-1-15./2	Edwards	-	25	-	720	150.97
188.001-1-15./3	Edwards	-	169.1	-	720	150.97
188.001-1-17./1	Edwards	-	65.6	-	720	150.97
188.001-1-27./1	Edwards	-	73.8	-	720	150.97
188.002-1-2./1	Edwards	-	36	-	720	150.97
174.004-1-18	Fowler	89.3	89.3	-	720	406.20
187.001-1-5	Fowler	2.5	-	-	720	135.39
187.001-1-21.2	Fowler	44.49	-	-	720	280.28
186.004-1-44	Fowler	705.3	-	-	720	1,354.02
186.004-1-33.11	Fowler	86.5	-	-	720	1,373.37
186.004-1-31	Fowler	61.6	-	-	720	1,252.46
187.003-1-2	Fowler	82.3	-	-	720	270.80
187.003-1-1	Fowler	1.6	-	-	720	4,671.33
187.069-1-38	Fowler	0.7	-	-	720	1,751.82
187.003-1-4.11	Fowler	63.8	-	-	720	786.15
187.003-1-4.121	Fowler	124.7	-	-	720	473.91
187.003-2-1.1	Fowler	45.2	-	-	720	270.80
199.001-2-52	Fowler	445	-	-	720	1,354.02
186.002-1-14.11/3	Fowler	-	146.6	-	720	13.54

Assessor Parcel Number	Town	Surface (acres)	Mineral (acres)	Structure	Class	2025 Taxes ($)
186.002-1-14.11/4	Fowler	-	144	-	720	13.54
187.003-1-3./1	Fowler	-	0.01	-	720	135.39
187.003-1-4.11/2	Fowler	-	-	Shaft 4	720	22,002.69
187.003-1-4.11/3	Fowler	-	0.01	-	720	11,678.34
187.003-1-4.11/5	Fowler	-	-	Shop	720	2,957.16
187.003-1-4.11/7	Fowler	-	-	Electric	720	19,362.35
187.003-1-4.11/9	Fowler	-	-	Buildings	720	52,806.43
187.003-1-4.11/11	Fowler	-	-	Paint, oil storage building	720	2,615.94
187.003-1-4.11/12	Fowler	-	-	Timber storage	720	2,802.80
187.003-1-4.11/17	Fowler	-	-	Railroad #4	720	7,007.01
187.003-1-4.11/18	Fowler	-	-	Mill	720	95,051.56
187.003-1-4.11/20	Fowler	-	-	Storage buildings	720	16,369.99
187.003-1-4.11/21	Fowler	-	-	Storage	720	6,600.80
187.003-1-50.2	Fowler	16.10	-	-	720	541.61
199.001-2-43.1/2	Fowler	-	-	Pipe shop 2	720	373.71
142.004-2-7.12/1	Macomb	-	60.30	-	720	13.24
Owned Fee Parcels	**-**	**2,715**	**3,027**	**-**	**-**	**267,403.30**

Source: St. Lawrence County Government 2025

All properties listed in Table 4-2 matches the St. Lawrence County 2025 tax rolls and are fully paid and current as of November 1, 2025.



Source: ESM 2025, modified ESRI base map

Figure 4-3: Mineral tenure map

Where mineral rights are denoted as "reciprocal" in Figure 4-3, this denotes that ESM and a third party have entered into reciprocal lease agreements pursuant to which ESM has granted certain mineral rights over minerals of which its business is not focused in exchange for mineral rights pertinent to its business.

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. In certain limited cases, Titan has ownership title or leasehold rights only to the minerals of certain properties (a.k.a. the mineral estate), while other parties own the non-mineral or surface rights of the properties (a.k.a. the surface estate). In such cases, Titan has the legal right to access the Property, and conduct activities on the Property to the fullest extent reasonably necessary to realize its rights to the mineral estate, subject only to an obligation to avoid unreasonably impacting the surface estate and the activities of owners of the surface estate, and with due regard to their physical safety. See, e.g., Drake v. Fox, 894 N.Y.S.2d 306, 307 (App. Div. 2010); Allen v. Gouverneur Talc Co., 668 N.Y.S.2d 755, 755 (App. Div. 1998); Schlueter v. Shawnee Operating Co., 535 N.Y.S.2d 867, 867 (Sup. Ct. 1988). Without limiting the foregoing, it is ESM's practice to attempt to enter with the owners of the surface estate into an agreement that coordinates the timing and other parameters of ESM's activities in furtherance of its rights to the mineral estate.

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 with a $2,701,000 surety bond. With the addition of the Kilbourne Graphite Project, this surety bond will increase to cover the cost of the additional reclamation required, as described in Item 20.2.2.

The mining activity in the Balmat region has not created any known long-term liabilities, beyond those described in Item 20 of this report, because 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 Zinc Mine are active and in place. See Item 20 for additional detail (including permits required for the Kilbourne Graphite Project).

Major environmental permits required for operation of the ESM 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[3]	6-4038-00024/00006	31 May 2030
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 2026

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.

(3) Permit modification allowing Kilbourne demonstration pit approved May 7, 2025.

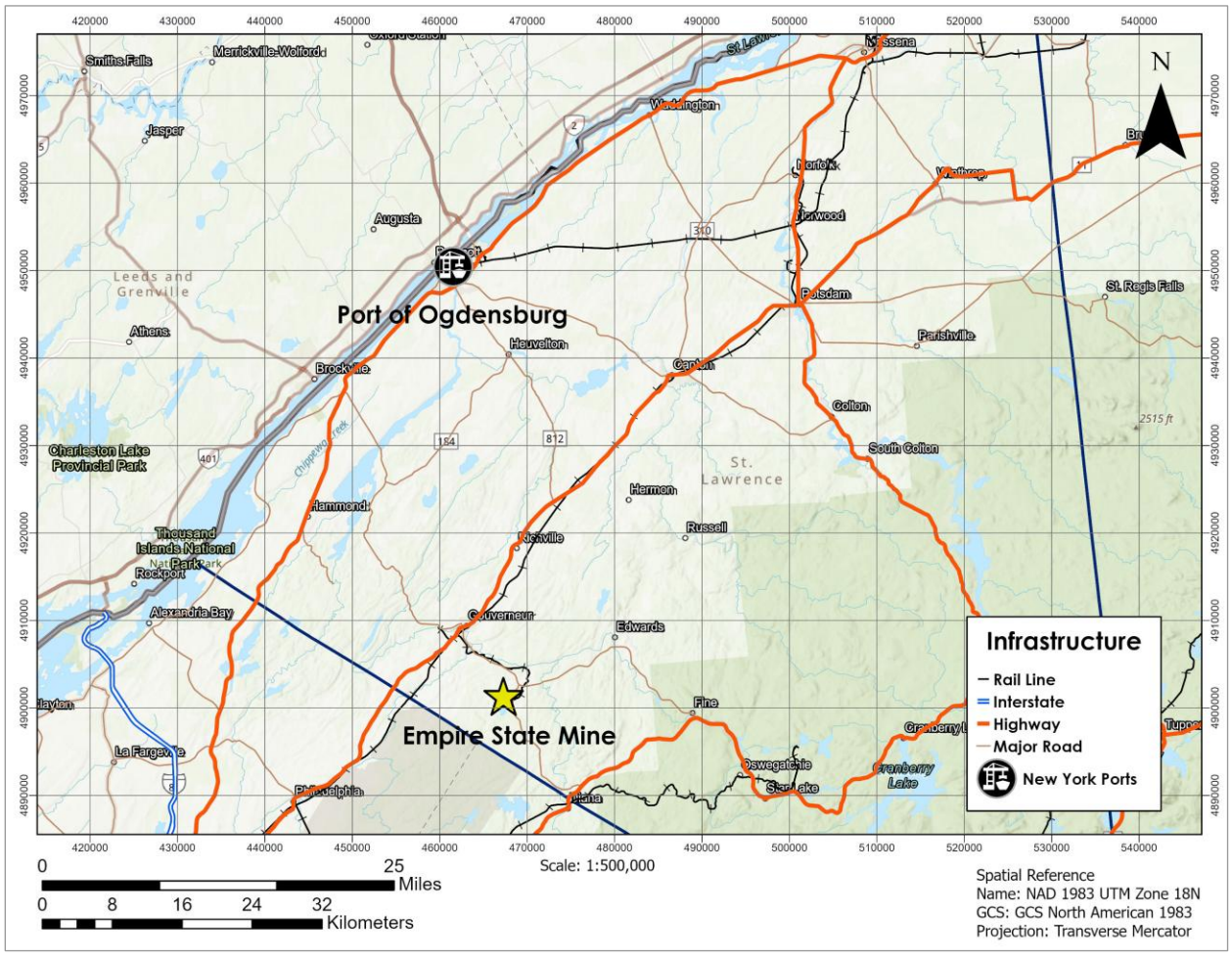
4.7 Risks

There are no significant factors or risks that may affect access, title or the right or ability to perform work on the ESM properties.

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

5.1 Accessibility

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



Source: Taylor et al., 2024

Figure 5-1: Site accessibility

5.2 Local Resources and Infrastructure

The nearest population center is the village of Gouverneur with an estimated population of 3,600. 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, and 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 monthly mean temperatures range from approximately 19°F in January to 69°F in July. Summer highs may reach 85°F and winter lows may reach -20°F. Annual average number of frost-free days is 115. Annual average precipitation is approximately 40 in, 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

ESM 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. The elevation of the Kilbourne Site ranges between 610 ft amsl and 650 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.

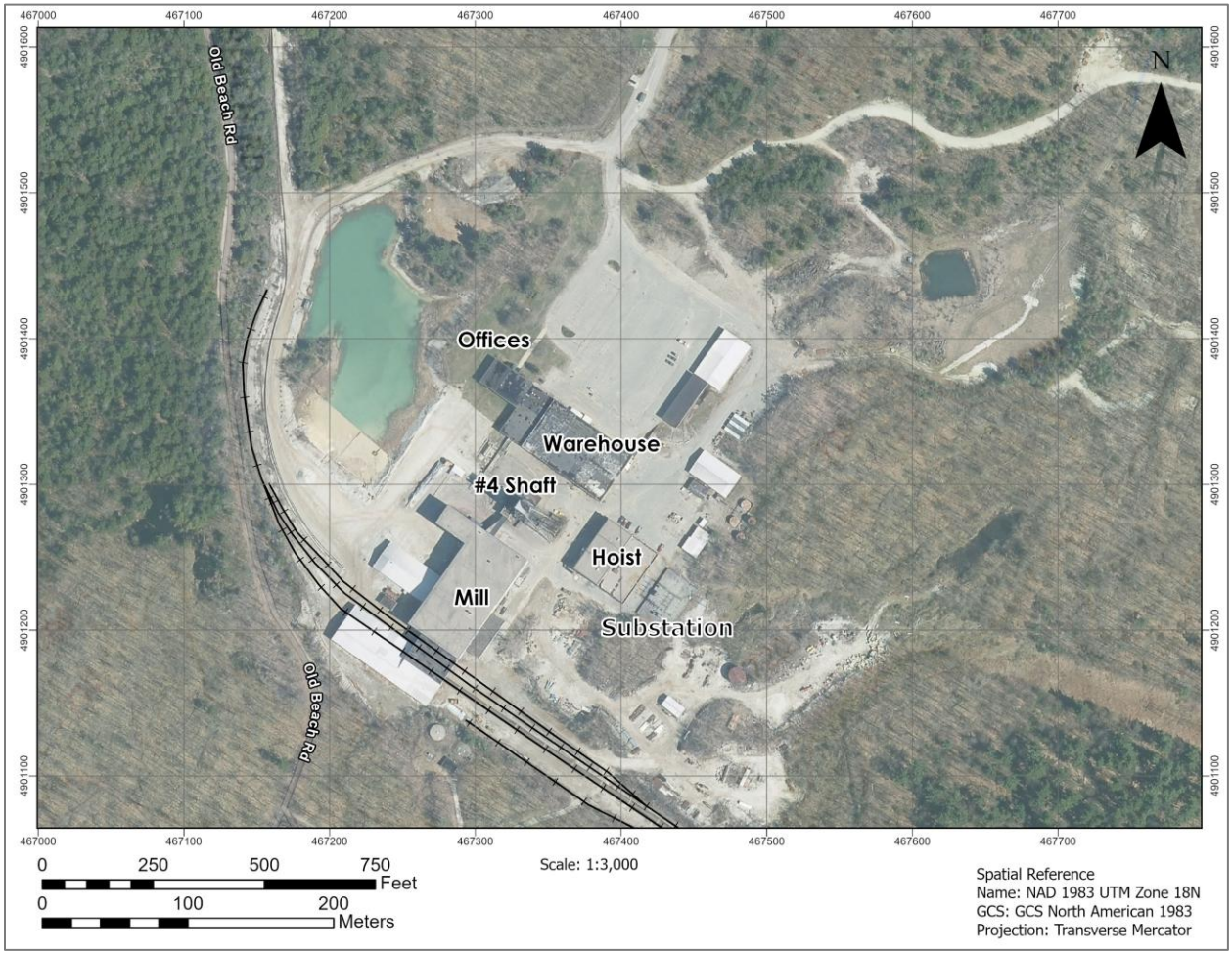
5.6 Surface Facilities and Rights

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 established surface infrastructure including office buildings, maintenance shops, mill, headframe, tailings, and ventilation systems (Figure 5-2). During the start-up of the mine, labor that was not available locally was sourced from outside of the region. At present, local labor is hired as needed and trained on site to meet operational requirements.

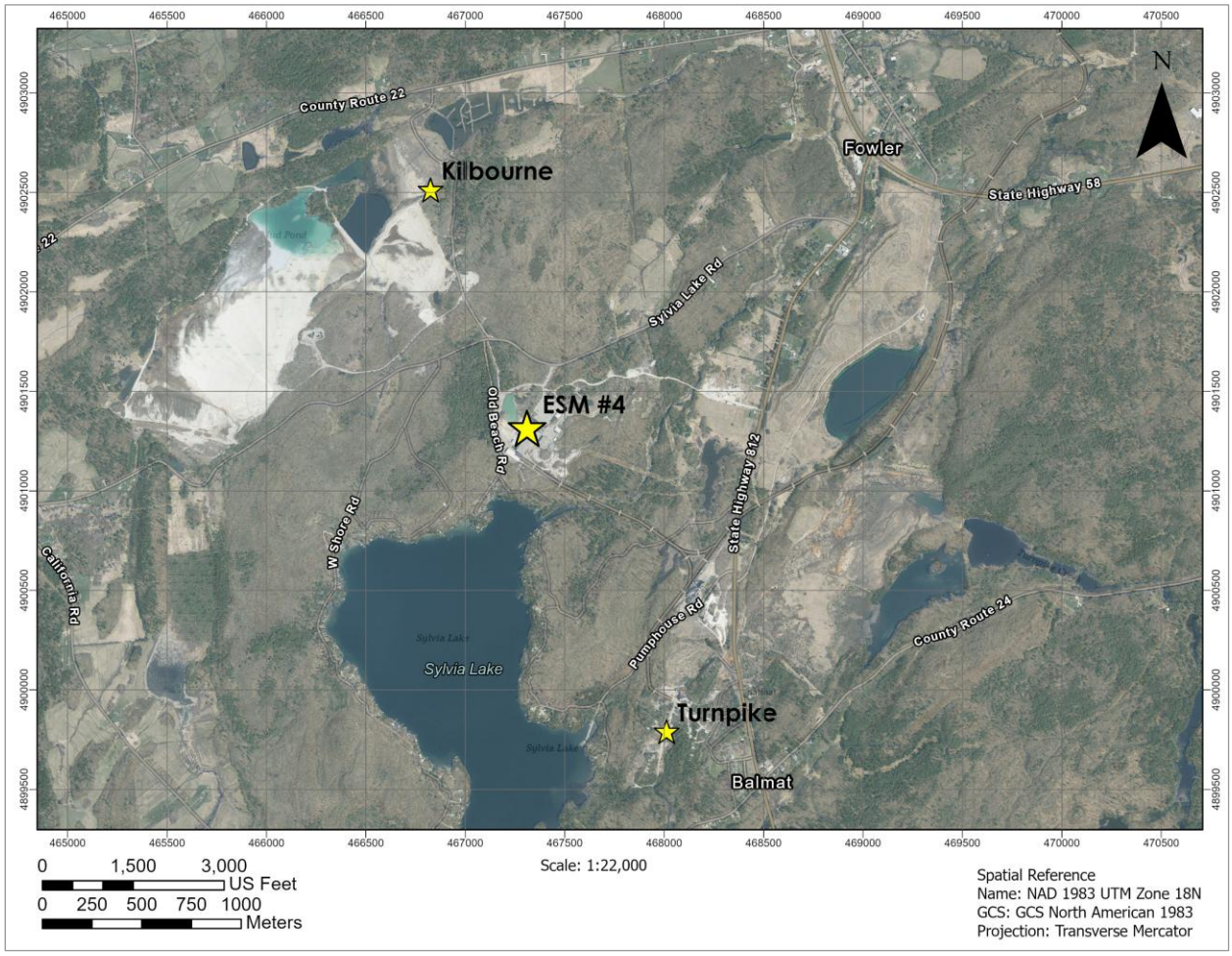
The Company's Turnpike Project is located within ESM's surface and mineral rights, located roughly 5,000 ft from the ESM #4 Shaft and mill. The Project area is adjacent to the #2 Shaft (Figure 5-3).

The Kilbourne Project is roughly 4,000 ft from the Company's #4 Shaft and mill and is within ESM's surface and mineral rights (Figure 5-3). The Project currently has no associated facilities (Figure 5-3); however, the Kilbourne Demonstration Plant currently occupies floor space within the existing ESM Mill facility (Figure 5-2).



Source: Taylor et al., 2024

Figure 5-2: Empire State Mines aerial view



Source: Taylor et al., 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 December 30, 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 Corporation	Purchased Balmat shares from Star Mountain and renamed Balmat mine to ESM

Source: Taylor et al., 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 (1903) and Hyatt mine (1917). 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 1903 (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) 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 36.4 million tons (Mton) at 8.6% Zn since operations began in 1930. The greater Balmat-Edwards-Pierrepont district has produced more than 46 Mton of 9.3% Zinc since mining began in 1915 at the Edwards mine (Table 6-2).

The existing Balmat mill was constructed in 1971 by St. Joe and has a nameplate capacity of 5,000 ton/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 6-3 presents the annual production totals at ESM from 2018 to 2024.

Table 6-2: Historic production totals by region

Region	Years Active	Tons	Grade (Zn%)	Zinc (lb)*
Balmat	1930-2001 2006-2008 2018-2024	36,440,115	8.59	6,257,918,531
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	**-**	**46,870,828**	**9.32**	**8,735,869,403**

* 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	7.9	23,932	58.2
2019	218,823	8.3	29,925	58.7
2020	323,414	8.6	45,161	59.3
2021	387,438	7.5	47,066	59.3
2022	425,022	7.5	52,547	58.8
2023	445,803	8.4	60,145	59.6
2024	410,869	8.7	57,334	60.0

6.1.4 Historical Mineral Resource and Mineral Reserve Estimates

A list of the most recent Mineral Reserve Estimates is presented in Table 6-4. Hudbay's Mineral Reserve Estimates concluded in 2008, with the 2015 reserves prepared by Star Mountain Resources. Titan is not treating these historical estimates as a current Mineral Reserve and a qualified person has not done sufficient work to classify the historical estimates as a current Mineral Reserve. The QP is unaware of the methods, parameters or assumptions used to generate these historic estimates and cannot comment to their accuracy. The historic estimates are not considered to be relevant or reliable as the ESM Zinc Operations have had ordinary course mining depletion and Mineral Resource replacement. The QP has prepared a current Mineral Resource Estimate for the ESM Zinc Operations, which supersedes all prior resource estimates and is detailed in Item 14. There are no historical estimates of Mineral Resources that are considered significant.

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: Hudbay 2005-2009

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 or leased by ESM; these mineral rights are subject to the same history as the Empire State Mines. As such, Item 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 and Mineral Resources

There are no historic estimates of Mineral Reserves or Mineral Resources on the Kilbourne Project.

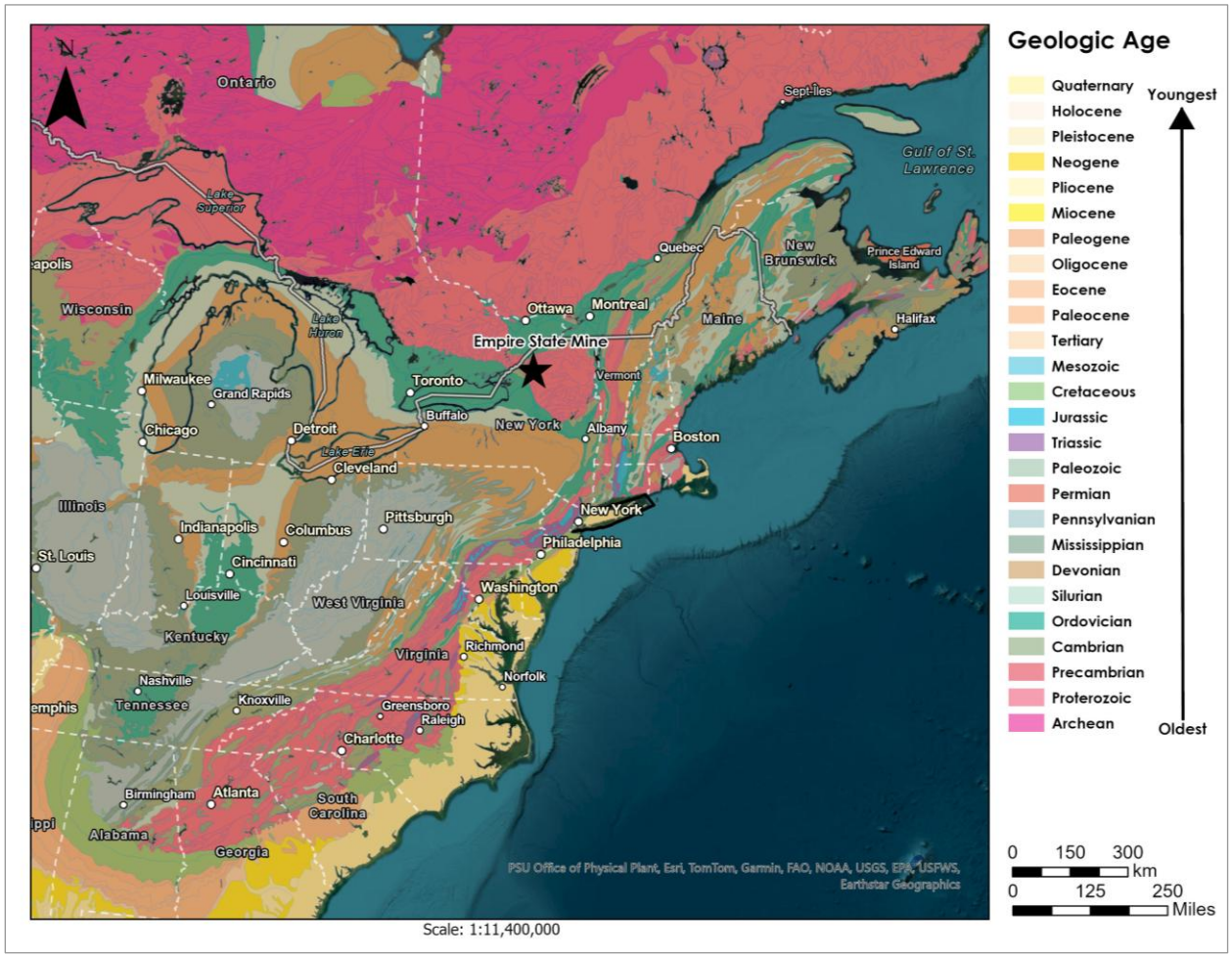
7. Geologic Setting and Mineralization

7.1 Geological Setting

The host rocks at ESM were deposited during the mid-Proterozoic era between roughly 1,300 Ma to 1,000 Ma (mega-annum, millions of years before present), near the edge of the North American craton. The region experienced episodic tectonic activity spanning over a billion years, with distinct orogenic and rifting phases contributing to its complex geological evolution.

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 events. The current day stratiform graphite mineralization in the region likely formed as a result of repeated metamorphism of syndepositional organic carbon reaching suitable metamorphic temperature and pressure conditions 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: Taylor et al., 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 Adirondack Lowlands, 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 Allegheneian 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 Allegheneian orogeny (Share, 2012). By the end of the Allegheneian 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.

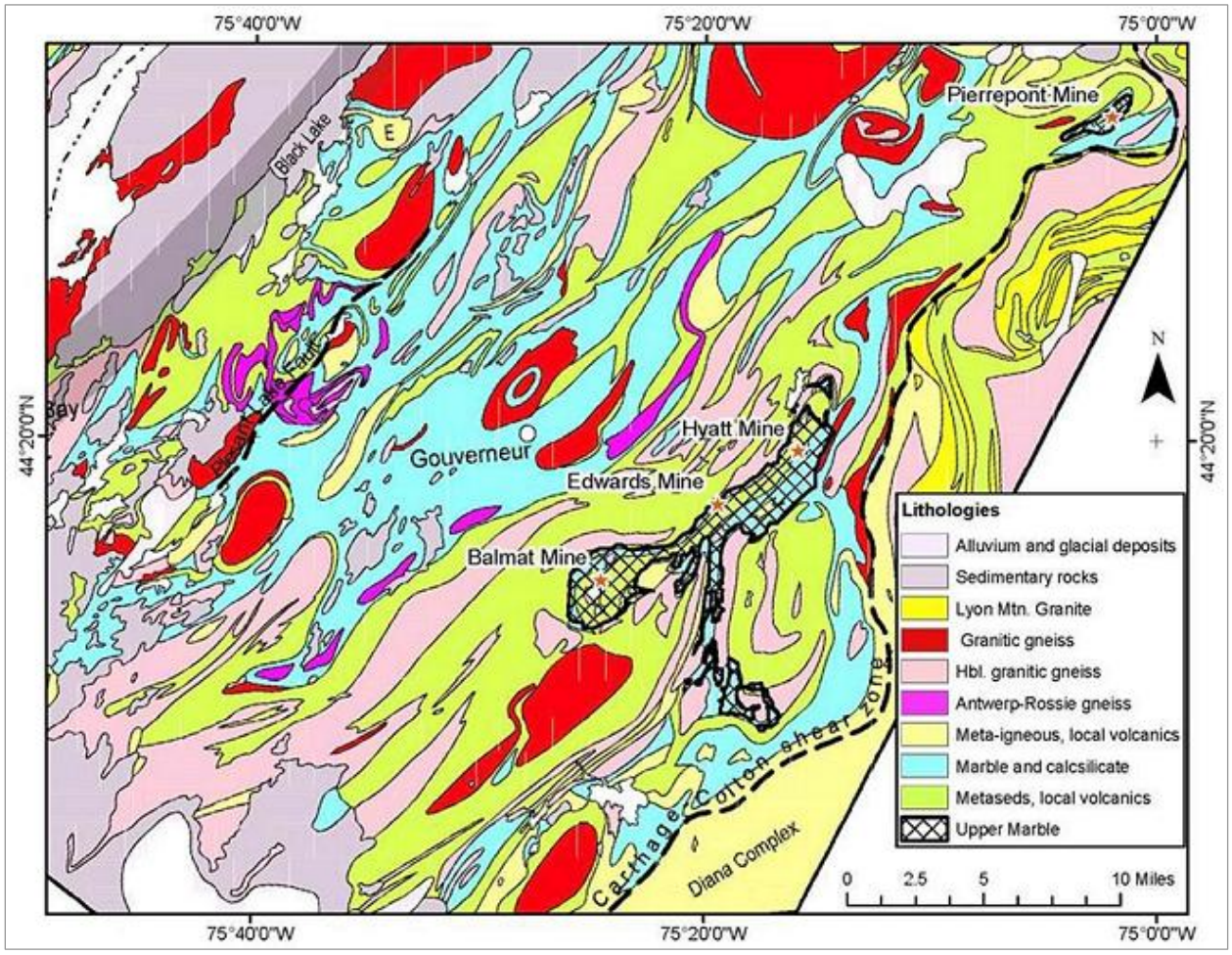
Table 7-1: Upper Marble stratigraphic sequence

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: Taylor et al., 2024

7.3 Property Geology

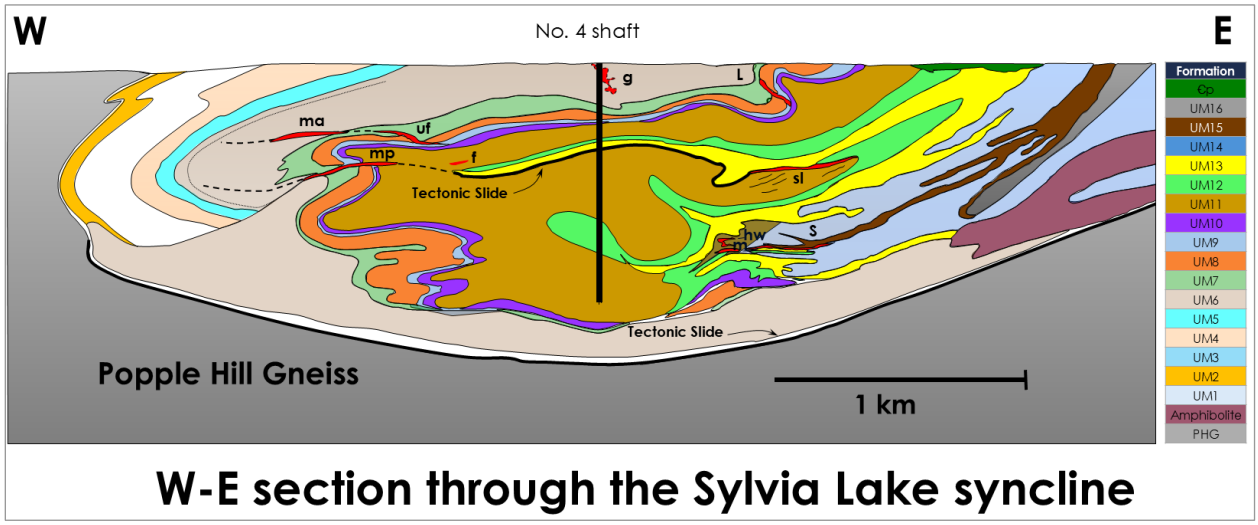
As a result of intense tectonism, the Upper Marble Formation is extensively deformed. The predominant structure is the Sylvia Lake Syncline, a major south-west to north-east trending fold lying between the original Balmat mine and the Edwards mine. Aerial exposure of the Upper Marble Formation is limited, 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: Taylor et al., 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 Item 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: Modified from Taylor et al., 2024

Figure 7-3: Section through the Sylvia Lake Syncline

7.4 Mineralization

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

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.

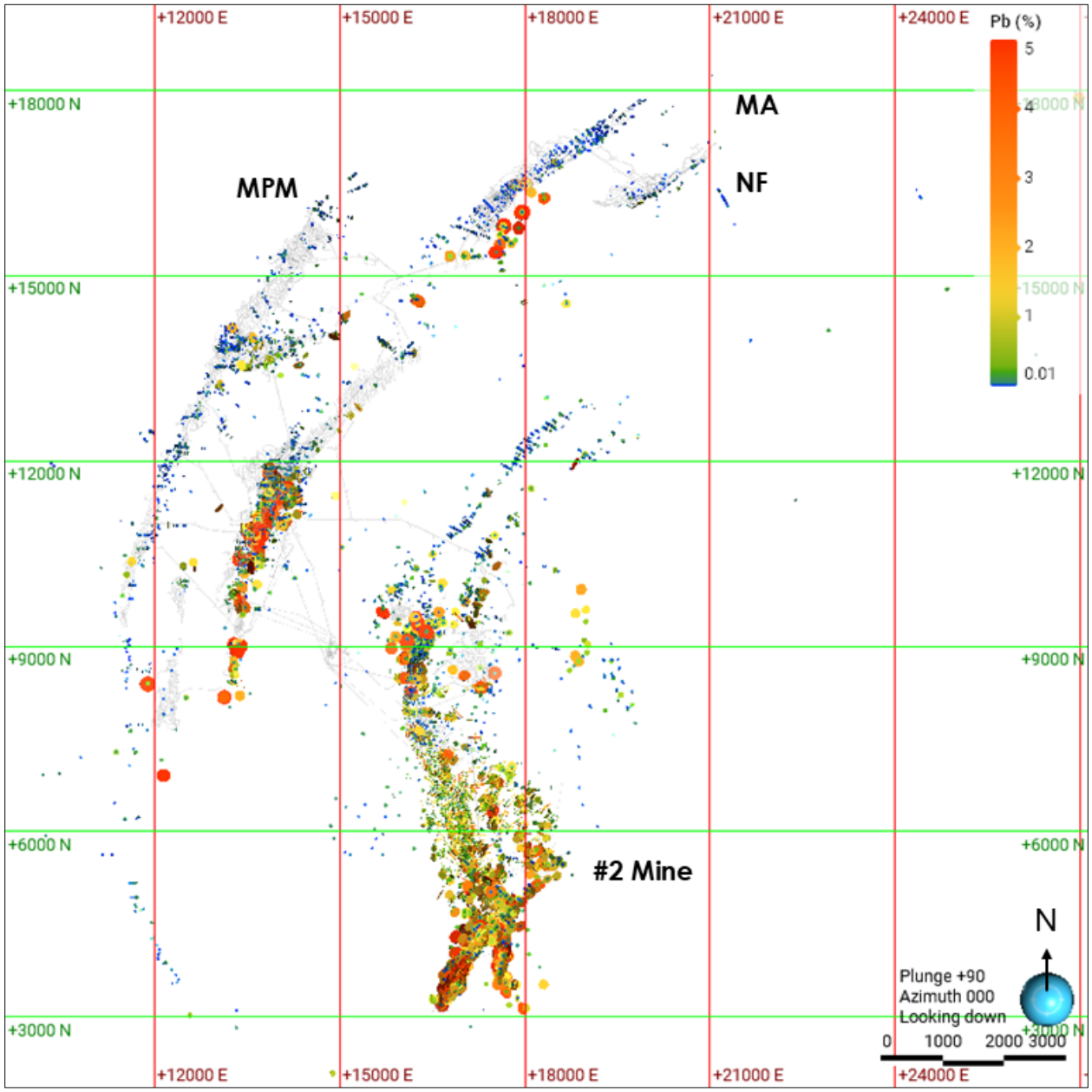
The Property contains 14 known zones of sphalerite mineralization. Three clusters have been defined consisting of three to five deposits each. The zinc mineralization extends from the surface down to a depth of 5,700 ft below surface. The zones are aerially scattered and all zones except NE Fowler and 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.



Source: Taylor et al., 2024

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, likely resulted from the metamorphism of organic-rich sediments deposited contemporaneously with the host rocks, with tectonic burial providing the necessary pressure-temperature conditions for graphitization. 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.

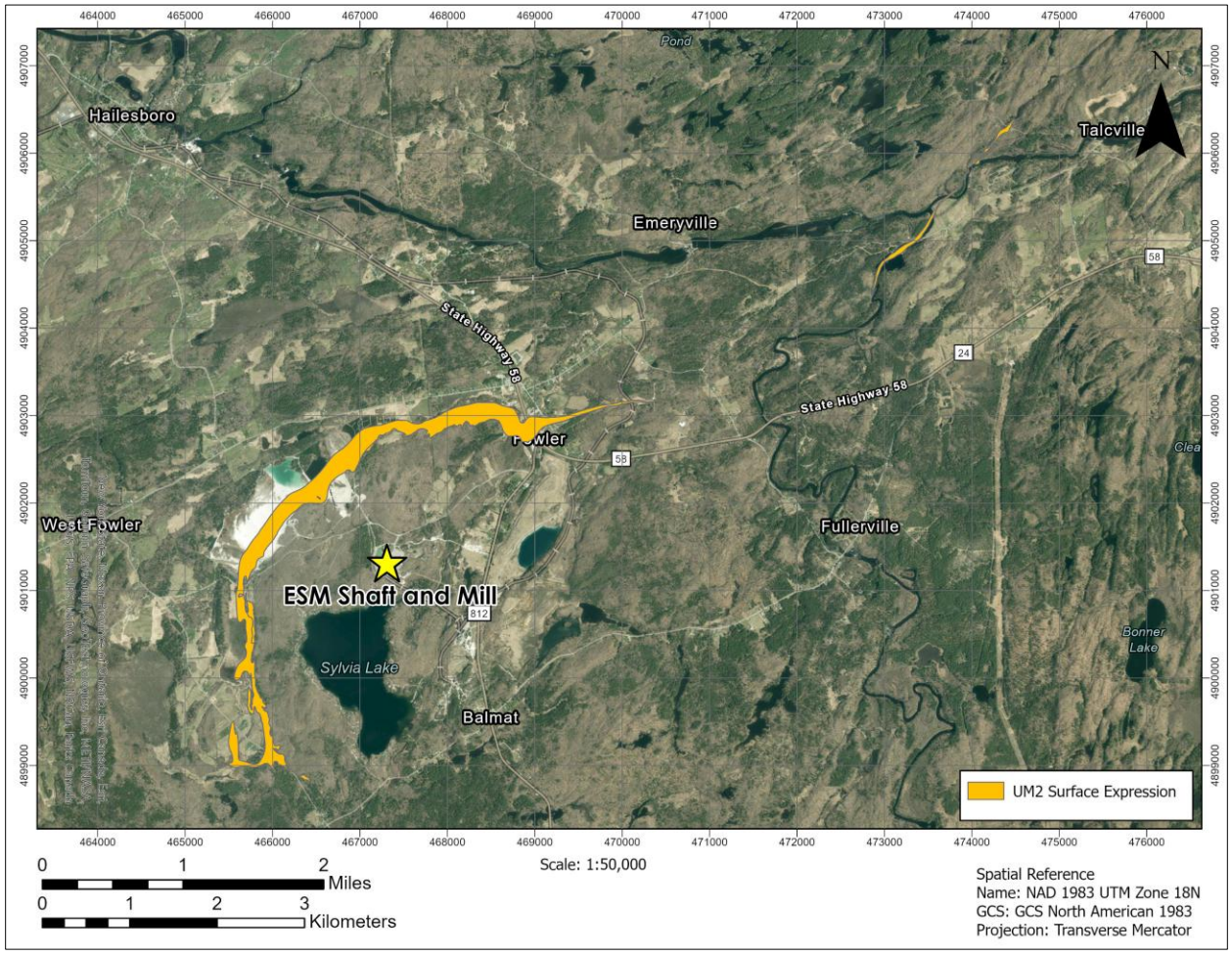
B) **The Phlogopitic Garnet Schist (PGS)** is a visually distinguishable phlogopite/biotite schist with a strong wavy fabric and pegmatitic boudins/inclusions. The sub-unit is dominated by a dark ferromagnesian mica with quartz-sillimanite-garnet-graphite with less common pyrite/pyrrhotite. Graphite 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 ft.

Unit 2, as defined by drilling, has an approximate strike length of 32,800 ft and extends to a vertical depth of 3,450 ft below surface.

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.



Source: Taylor et al., 2024

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 "sedimentary exhalative", referring to the process by which metal-bearing brines are expelled into sedimentary basins. 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 form in saltwater sedimentary basins within extensional tectonic domains, such as the Trans-Adirondack back-arc basin (Figure 8-1). These environments are characterized by restricted flow to the open ocean, which promotes extensive and rapid seawater evaporation—particularly in low latitudes—on large evaporative carbonate platforms. This process generates dense brines that sink and migrate through porous terrigenous sedimentary layers toward the basin's deepest parts, leaching metals along the way. As the brines descend, they heat up due to increasing temperature with depth. Upon encountering extensional fault surfaces, the brines may ascend and be exhaled onto the basin floor, where they interact with distal basin facies conducive to H_2S generation, precipitating zinc and lead sulfides.

Evidence of these evaporative conditions is well recorded at ESM, including periodic anhydrite beds and dolomitization of the Upper Marble. Paleolatitude reconstructions (Cocks et al., 2005) place the region at a latitude favorable for rapid evaporation during deposition. The carbonate platform at ESM represents the proximal facies of the sedimentary basin (Chiarenzelli et al., 2015), supporting the interpretation of a brine-driven metal transport system essential for Sedex deposit formation.

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).

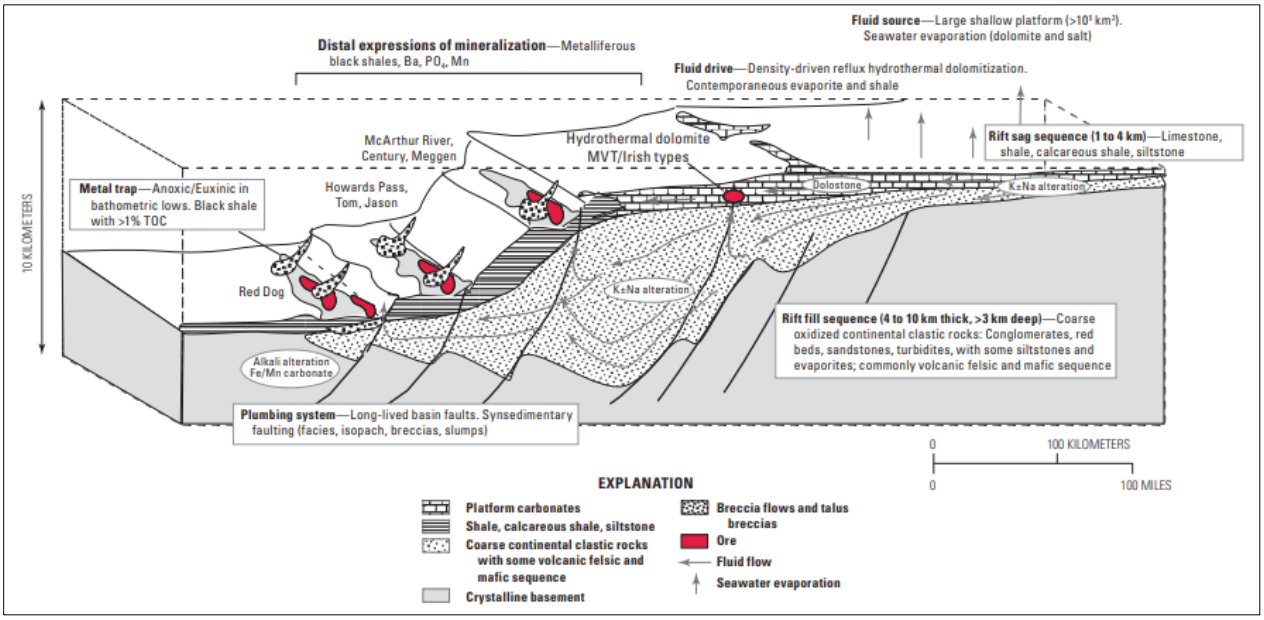
Mass balance studies indicate that large volumes (thousands of km^3) of clastic sediments are required to generate enough metals to form a Sedex deposit. Fluid inclusion studies indicate that Sedex deposits are formed from brines with temperatures between 100 °C to 200 °C. Metals are most soluble in 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. Works 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. While the ESM deposits occur in more proximal facies than typical Sedex models suggest, their genesis remains consistent with Sedex processes modified by metamorphism.

A pyritic schist unit underlying the Upper Marble may represent a remnant of the original distal facies, though its current metamorphic state obscures primary features. 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. The ESM deposits were 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: Emsbo et al., 2016

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 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. ESM has also completed surface and underground exploration drilling; this is expanded upon in the drilling section (Item 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/mineralized zones 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 the 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

ESM 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 Pork Creek targets, 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 (Table 9-1). 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; Taylor et al., 2024

9.1.2 Surface Geochemical Sampling

9.1.2.1 Historic Geochemical Sampling

Within the region there have been multiple pulses of surface geochemical sampling. These began at scale during the mid twentieth century and have been implemented periodically to the present. Programs have generally focused on calc-silicate rocks within areas having known zinc occurrences, with additional samples scattered throughout the region targeting prospective areas. The Company currently has complete or partial records of 16,656 historic surface geochemical samples. Within this historic data set there are12,100 soil samples, 18 rock samples, 296 stream sediment samples, and 4,242 samples of an unverified type. The validation of the historic surface geochemical samples is an ongoing exercise. These samples have been mapped using historically recorded coordinates and available maps that have been georeferenced (Figure 9-1).

The QP cautions that certain data referenced in this report are historic in nature and were collected prior to the implementation of current industry-standard QA/QC protocols. As such, the reliability, representativity, and overall quality of these samples cannot be verified. There is insufficient information available to assess the adequacy of sample preparation, analytical procedures, or security measures associated with these historic datasets. Consequently, these data may be subject to unknown sampling or analytical bias and should be considered non-compliant with current CIM best practices. While the historic data provide context and support broader geological interpretations, they have not been used in the estimation of current Mineral Resources without appropriate validation.

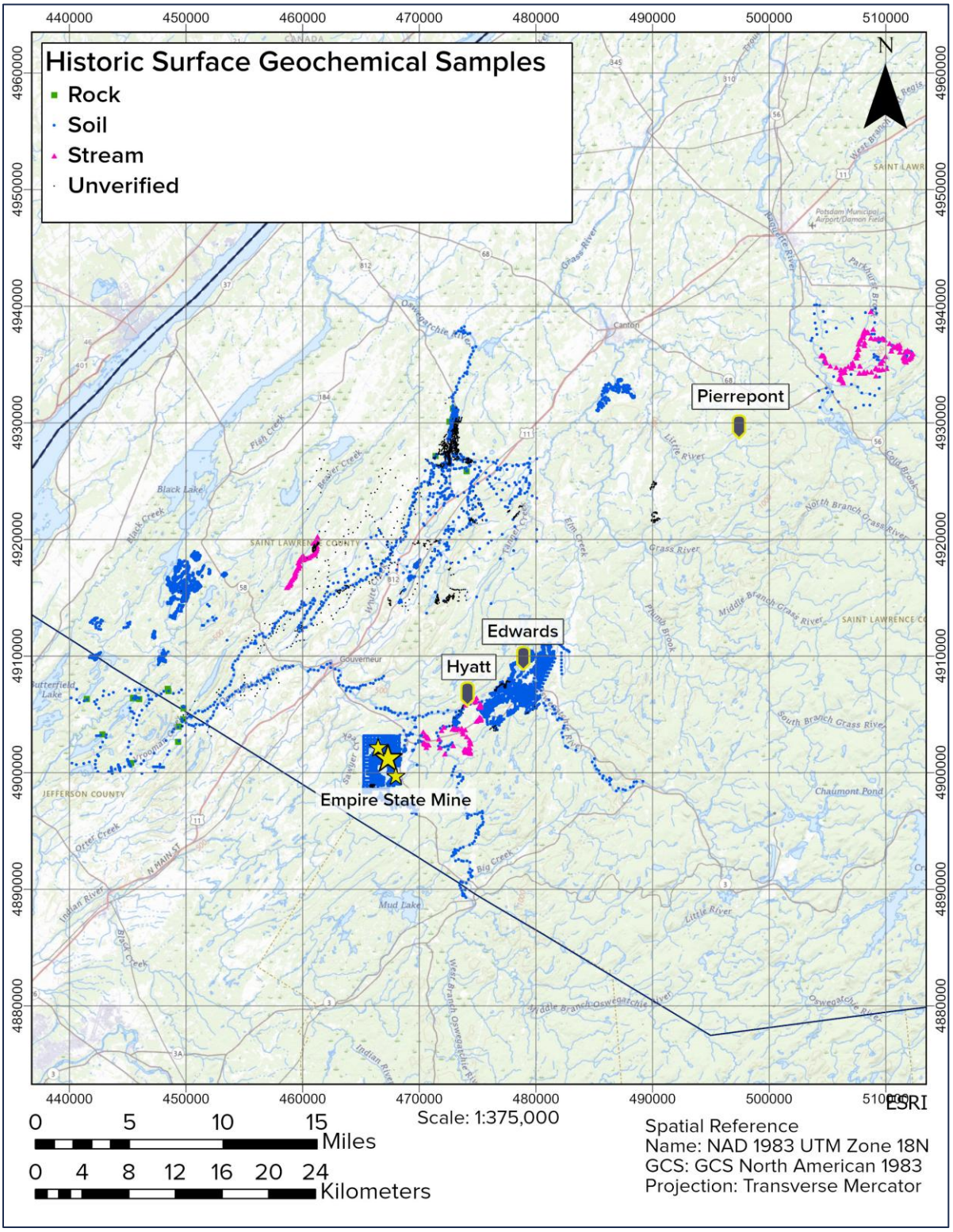


Figure 9-1: Historic surface geochemical samples by type relative ESM and past operations

The most recent program of a notable scale was completed in 2022 with 1,961 soil samples collected from district and regional targets by ESM. The Company has targeted zinc in soil anomalies generated by this program for rock sampling, resulting in the collection of 13 rock samples between 2022 and 2024.

9.1.2.2 Soil Sampling

In 2022, ESM 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 9-2). 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 trend (210 samples total).

Grids were generally laid out in a 328 ft (100 m) by 656 ft (200 m) spacing, and designed to ensure a higher sample frequency, tighter spacing, and cross cutting the orientation of the geologic units. A tighter spacing was employed on targets with verified historic zinc in soil anomalies. Samplers assigned each sample with a unique identification number and recorded the sample location with handheld GPS paired tablets. In addition to the location, data samplers recorded the slope, vegetation, and indicators of human activity at each site. Descriptions of each sample included the sampling method, sample depth, soil horizon, and its physical characteristics. Samples were not collected in wetlands, or in areas where human activity may impact the sample confidence.

There are no documented factors that would have had a negative impact on sample quality. Grid sampling, with geologic, topographic, hydrologic, and cultural influences taken into account was implemented to help ensure sample site selection remained unbiased. Sampling targeted the C horizon of the soil profile; the majority of the samples were representative of this horizon. There are no known factors that may have resulted in sample biases. 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. Both facilities are ISO/IEC 17025:2017 certified and are independent of Titan. The analytical method used was AQ200, an aqua regia digest followed by ICP-OES/MS 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 highlights the highest zinc value recorded from each of the 2022 target areas.

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

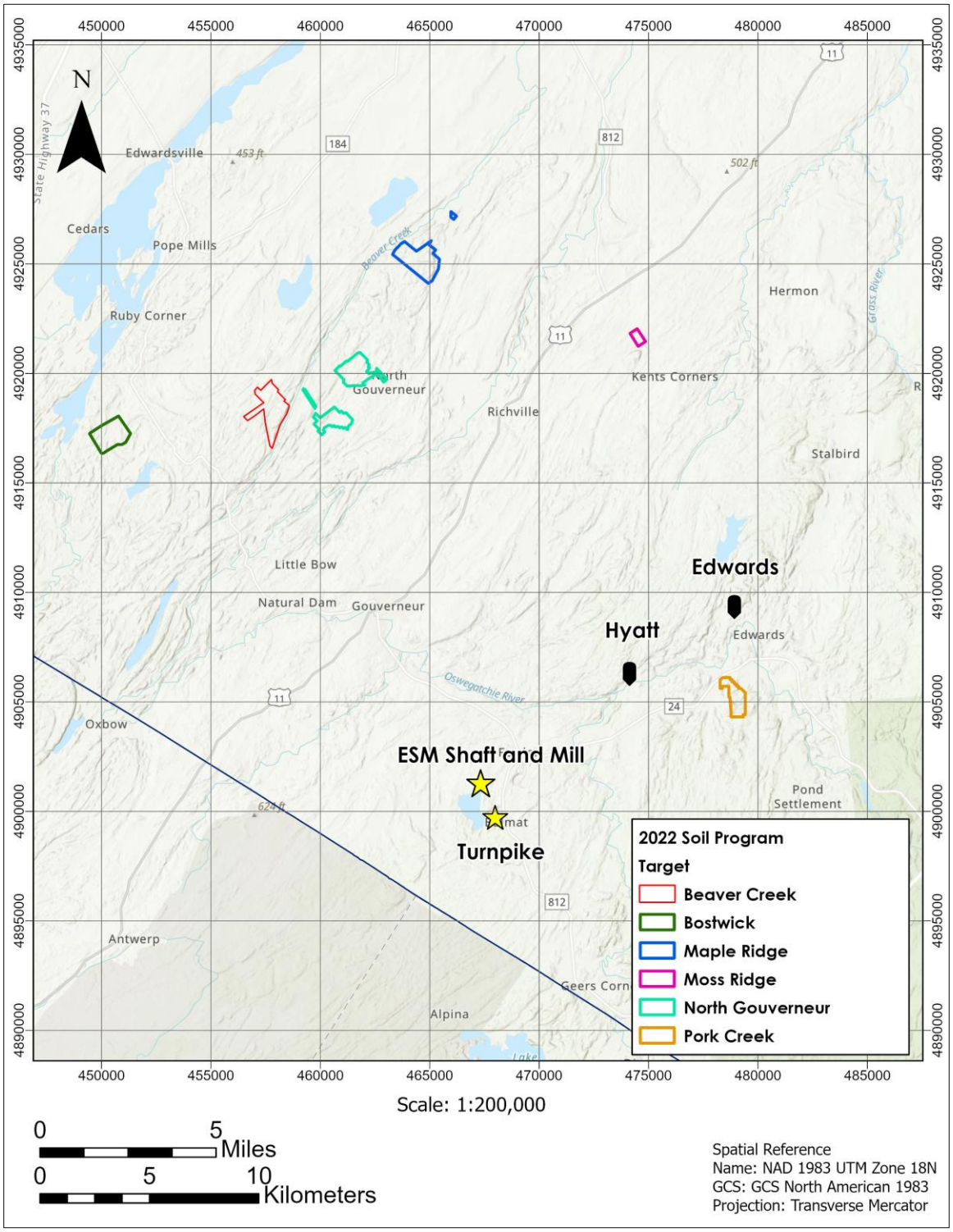
Target	Total Samples	Highest Zinc Value (%)
District	**1,751**	**1.67**
Beaver Creek	531	0.49
Bostwick	197	0.04
Maple Ridge	135	0.03
Moss Ridge	206	1.67
North Gouverneur	681	0.46
Trend	**210**	**0.33**
Pork Creek	210	0.33

Source: Taylor et al., 2024

Samples with values above 250 ppm Zn have historically been viewed as anomalous within the rocks of St. Lawrence County, NY. All of the soil targets from the 2022 program had samples above this threshold.

Within the district, the results from Beaver Creek, Moss Ridge, and North Gouverneur returned values well above this threshold. In addition to Zn values, Pb, Cd, Ag, and Hg were evaluated to help make a determination on mineralization style. For the samples collected at Beaver Creek and North Gouverneur, elevated zinc corresponded to elevated lead, with an apparent east-west orientation to the anomalies. The positive relationship between Zn and Pb, along with the east-west orientation suggest a relationship to the historically exploited Rossie–Macomb veins (Neuman, 1952). The Moss Ridge sample that ran 1.67% Zn in soil also reported elevated cadmium. This relationship may indicate system of mineralization other than late, fracture filling polymetallic carbonate veins, and shares similarities to the mineralized bodies of Hyatt, ESM, Edwards, and Pierrepont. Sampling at Bostwick Creek and Maple Ridge failed to return results within the elevated ranges of the other district targets.

The one target tested in trend was Pork Creek, the highest zinc value recorded from the program was 0.33% Zn. Samples with elevated zinc have corresponding elevated values of cadmium. This may suggest that the zinc in soil anomalies at Pork Creek are related to ESM style mineralization, rather than late fracture filling systems.



Source: Taylor et al., 2024

Figure 9-2: Location of 2022 soil sampling programs relative to ESM

9.1.2.3 Rock Sampling

Between 2019 and 2025, the Company has collected 46 samples within St. Lawrence and Jefferson counties. Sampling efforts have been classified as prospecting or follow-up rock sampling. Prospecting samples are generally first pass samples taken at targets identified through historic data review, or recent geologic observation. Follow-up sampling targets areas with previously identified geochemical anomalies, known areas of near-surface mineralization, or areas with historically documented mineral production.

Rock samples were collected by ESM personnel or contractors using industry standard practices. Sample locations are recorded in handheld GPS devices and labeled with a unique sample identification number. The XY coordinates of the sample are recorded in field notebooks or within the sample tag book containing the corresponding sample ID. A brief description of the rock is recorded, often including the exposure type, visible mineralogical composition, and indications of mineralization. The sample is placed into a cloth or polyethylene sample bag, the sample ID is recorded on the bag, and a sample tag is inserted. A representative piece of the sample is retained for future reference.

There are no documented factors that would have had a negative impact on sample quality. Sample selection during these programs is often made based on outward and visible signs of mineralization, sample size and methodology are modified to ensure representative and unbiased sample are collected. However, sample bias may be experienced due to factors such as limited geologic exposure, subconscious selection of visually appealing samples, geologic misinterpretation, sampling of historically worked or transported material.

All samples collected by the Company are packaged and shipped by Company personnel and have been sent to ALS for multielement analysis. Sample preparation (crushing and pulverizing) has been performed at ALS, an ISO/IEC 17025 accredited laboratory 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. ALS is independent of Titan.

All samples were prepared using ALS Method 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.

The analytical methods are summarized in Table 9-3.

Table 9-3: Summary of assay methods

Analyte	Method Code	Detection Limit	Digest	Instrumentation
35 elements, see Table 9-4	ME-ICP41	Varies; see Table 9-4	0.25 g two-acid: HNO_3 + HCl digest plus HCl leach	ICP-AES
52 elements, see Table 9-5	ME-MS89L	Varies; see Table 9-5	0.2 g Sodium Peroxide Fusion, HCl leach	ICP-AES/ICP-MS
Au	Au-AA23	0.005 ppm	30 g fire assay	FA-AAS
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.

Table 9-4: 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			

Table 9-5: Upper and lower limits for MS89L super trace analysis method

Analyte	Lower Limit	Upper Limit	Analyte	Lower Limit	Upper Limit	Analyte	Lower Limit	Upper Limit
Ag (ppm)	5	12,500	Ho (ppm)	0.01	25,000	Sm (ppm)	0.04	25,000
As (ppm)	4	25,000	In (ppm)	0.3	25,000	Sn (ppm)	3	25,000
Ba (ppm)	2	25,000	K (%)	0.05	25	Sr (ppm)	20	25,000
Be (ppm)	0.4	25,000	La (ppm)	0.08	25,000	Ta (ppm)	0.04	25,000

Analyte	Lower Limit	Upper Limit	Analyte	Lower Limit	Upper Limit	Analyte	Lower Limit	Upper Limit
Bi (ppm)	0.1	25,000	Li (ppm)	2	25,000	Tb (ppm)	0.01	25,000
Ca (%)	0.1	25	Lu (ppm)	0.05	25,000	Te (ppm)	0.5	25,000
Cd (ppm)	0.8	25,000	Mg (%)	0.01	30	Th (ppm)	0.1	25,000
Ce (ppm)	0.2	25,000	Mn (ppm)	10	25,000	Ti (%)	0.005	25
Co (ppm)	0.5	25,000	Mo (ppm)	2	25,000	Tl (ppm)	0.02	25,000
Cs (ppm)	0.1	25,000	Nb (ppm)	0.8	25,000	Tm (ppm)	0.01	25,000
Cu (ppm)	20	25,000	Nd (ppm)	0.07	25,000	U (ppm)	0.2	25,000
Dy (ppm)	0.03	25,000	Ni (ppm)	10	25,000	V (ppm)	1	25,000
Er (ppm)	0.02	25,000	Pb (ppm)	0.5	25,000	W (ppm)	0.3	25,000
Eu (ppm)	0.03	25,000	Pr (ppm)	0.03	25,000	Y (ppm)	0.2	25,000
Fe(%)	0.01	25	Rb (ppm)	0.5	25,000	Yb (ppm)	0.02	25,000
Ga (ppm)	0.5	25,000	Re (ppm)	0.01	25,000	Zn (ppm)	10	25,000
Gd (ppm)	0.03	25,000	Sb (ppm)	0.3	25,000			
Ge (ppm)	0.5	25,000	Se (ppm)	3	25,000			

2019 Sampling

In 2019, 11 samples were collected. Of these, 10 can be classified as prospecting samples. One sample was collected from the historic Bostwick Pit, a documented zinc prospect within the Company's mineral tenure. This sample returned the highest values for Zn (0.18%), Cu (0.64%), and Au (2.71 ppm) confirming historic documentation of copper mineralization associated with the prospect. The polymetallic nature of the mineralization at Bostwick suggests a separate deposit type from those associated with the Balmat–Pierrepont trend, or may represent an evolution of the mineralizing system.

2023 Sampling

All rock samples collected in 2023 were from areas with zinc in soil anomalies identified during the 2022 soil sampling program. A total of13 samples were collected, four from the Pork Creek target, and nine from the Company's Moss Ridge target. The Moss Ridge sampling targeted a 1.67% Zn in soil anomaly, a rock sample collected from a nearby outcrop returned an assayed grade of 4.53% Zn. Multielement data from this sample shows a positive relationship between Zn and Cd, further supporting observations from the 2022 soil program that Moss Ridge may share similarities with mineralization in the Balmat–Pierrepont trend.

2024 Sampling

A total of four samples were collected in 2024, all were prospecting samples collected within the historic Adirondack Magnetite belt, with a focus on the Parish Prospect. Samples were anomalous for gold but lacked copper mineralization. Of the four samples collected, three returned anomalous gold results with grades ranging from 0.11 ppm Au to 0.41 ppm Au. In addition to elevated Au, these samples returned elevated Fe, Ba, Mn, and P. This program demonstrated a positive relationship between Fe and Au mineralization.

2025 Sampling

A total of 18 samples were collected in 2025, 11 of these were from the Bend Zn target within the trend, and seven follow-up samples were from the Company's Parish target.

Of the 11 samples collected at Bend, four returned anomalous zinc results ranging from 0.31% Zn to 1.7% Zn. The samples with elevated zinc were sourced from the wall rock of a historic prospect pit. The rock was heavily oxidized and may represent a gossan or weathered cap of a larger body of zinc mineralization. Multielement data from these samples shows a positive relationship between Zn and Cd.

Of the seven samples collected at Parish, two returned gold grades greater than or equal to 0.1 ppm Au, ranging from 0.1 ppm to 0.15 ppm Au. The sample that ran 0.1 ppm Au also returned a copper grade of 0.45% Cu. Sampling at Parish in 2025 continues to demonstrate the positive relationship between Fe and Au.

Table 9-6 shows the highest zinc values associated with the 36 samples collected from existing targets, the 10 samples collected during the 2019 prospecting effort returned zinc grades below or slightly above detection. Figure 9-3 shows the samples by year.

Table 9-6: 2019 to 2025 rock samples by target with highest zinc values

Target	Total Samples	Highest Zinc Value (%)
District	**21**	**4.53**
Moss Ridge	9	4.53
Bostwick	1	0.18
Parish	11	0.08
Trend	**15**	**1.7**
Pork Creek	4	0.02
Bend	11	1.7

Source: Modified from Taylor et al., 2024

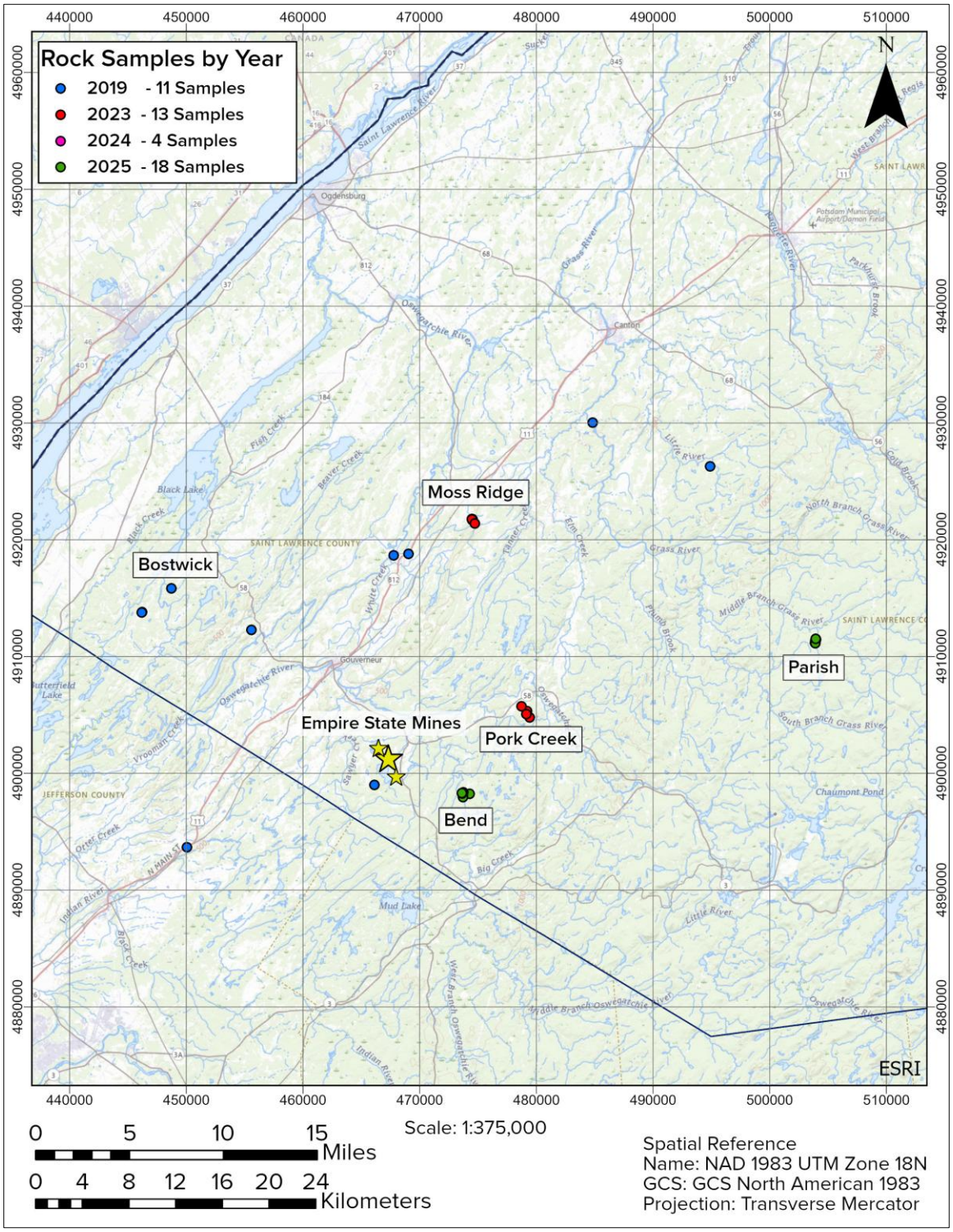


Figure 9-3: Location of rock samples by target

9.1.3 Hydrogeochemistry

During the 2023 and 2025 field seasons, 253 samples were collected from surface and mine water sources in St. Lawrence County, NY.

All sampling locations were recorded with an ESRI Arrow in the field and positions accuracy and precision was less than 20 cm for all points. Waters were filtered in the field with a 0.45-micron filtered and acidified. Field water data such as pH, TDS, and temperature were recorded. Next, the acidified water samples were analyzed in the lab. Samples are representative of available groundwater sources within areas of known mineral occurrence or mineral prospectivity. There were no known factors that would impact sample quality. All reasonable measures were taken to ensure an unbiased sample was collected.

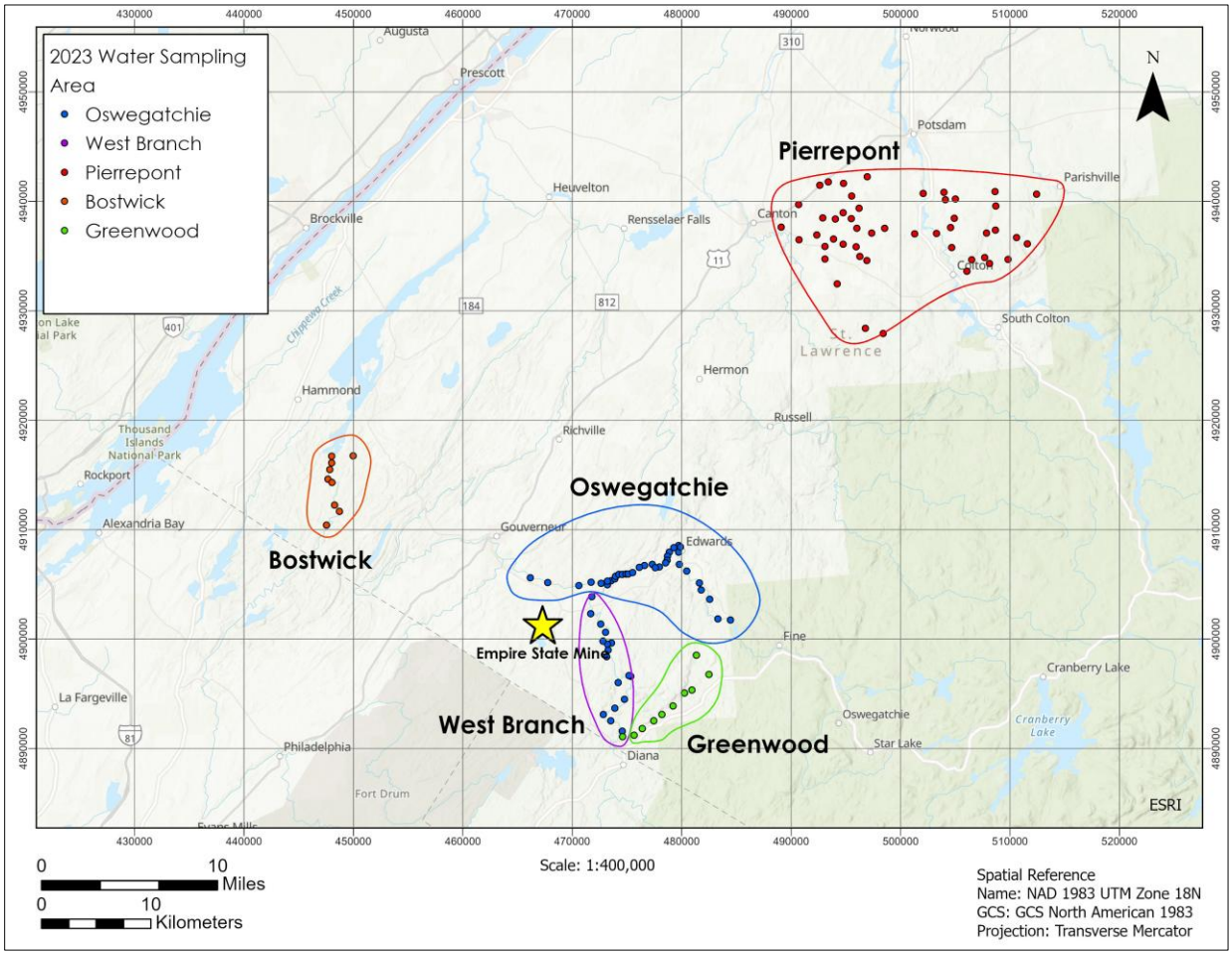
For the 2023 program, 5 ml of the sample was used for major ion (Ca, Mg, Fe, Al, etc.) on the Agilent 5900 ICP-OES at Juniata College. Data reported were corrected internally with Y, all data were better than 5% error. A volume of 3 ml from each sample was used for trace element ion (Cu, Zn, Cd, Sn, etc.) on the Thermo iCAP at Rutgers University, with reported errors of less than 3%. For Cu and Zn isotope analysis, 120 ml of sample was dried, and Cu and Zn were purified from these salts. The purification required a two-stage ion exchange chromatography (Mathur et al., 2005; Mathur et al., 2009) technique at Juniata College. The samples were measured on the Thermo Neptune Plus ICP-MS for Cu and Zn isotope values. The copper isotope data are reported relative to the NIST 976 standard and zinc isotope data are reported relative to the ETH Zn standard (Archer et al., 2017). The errors for these analyses were determined using both internal and external standards and are 0.13‰ and 0.07‰ respectively (2s). An internal copper isotope standard and international USGS rock standards were measured along with all samples and reproduced the Cu and Zn isotope values for BVHO-2 and AGV-2.

For the 2025 program, 5 ml of the sample was used for major ion (Ca, Mg, Fe, Al, etc.) on the Agilent 5900 ICP-OES at Juniata College. Data reported were corrected internally with Y, all data were better than 5% error. A volume of 3 ml from each sample was used for trace element ion (Cu, Zn, Cd, Sn, etc.) on the Thermo iCAP at Rutgers University, with reported errors of less than 3%. For copper isotope analysis, 120 ml of sample was dried, and copper was purified from these salts. The purification required a two-stage ion exchange chromatography (Mathur et al., 2005; Mathur et al., 2009) technique at Juniata College. The samples were measured on the Thermo Neptune Plus ICP-MS for copper isotope values. The copper isotope data are reported relative to the NIST 976 standard (Archer et al., 2017). The errors for this analysis were determined using both internal and external standards and are 0.13‰. An internal copper isotope standard and international USGS rock standards were measured along with all samples and reproduced the copper isotope values for BVHO-2 and AGV-2.

2023

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 9-4.



Source: Taylor et al., 2024

Figure 9-4: 2023 Water sampling sites by area

Samples from all five targets returned $\delta^{65}Cu:\delta^{63}Cu$ ratio of ≥1 per mil. This threshold is viewed as indicative of copper sulfide weathering at depth (Mathur, 2023). Of the 120 surface water samples collected in 2023, 44 were above the $\delta^{65}Cu:\delta^{63}Cu$ 1 per mil ratio. There was no relationship between recorded concentration values, and the copper isotope ratio. The 2023 study also measured the isotopic ratio of $\delta^{68}Zn:\delta^{66}Zn$. These measurements proved less than useful in the identification of potential zinc sulfides at depth, in part due to the high fractionation rate of zinc. Consistently, samples taken downstream from known bodies of mineralization (Bostwick, Hyatt, ESM, and Bend) demonstrated $\delta^{65}Cu:\delta^{63}Cu$ above the 1 per mil threshold.

Although copper mineralization does not occur at economic concentrations within the Balmat-Pierrepont trend, copper sulfide minerals have been documented within the zinc ore bodies. The results of the 2023 program support the use of copper isotope sampling as a prospecting tool for sulfide mineralization within the Balmat–Pierrepont trend and district.

2025

In August 2025, an additional 121 samples were collected in partnership with Juniata College of Pennsylvania from surface water sources within the St. Lawrence County. These samples were collected to evaluate copper isotope ratios within prospective areas. Figure 9-5 shows the sample locations. Four areas were targeted during this program, Pierrepont Follow-Up, Trout Lake Marbles, Adirondack Magnetite, and Colton Cover.

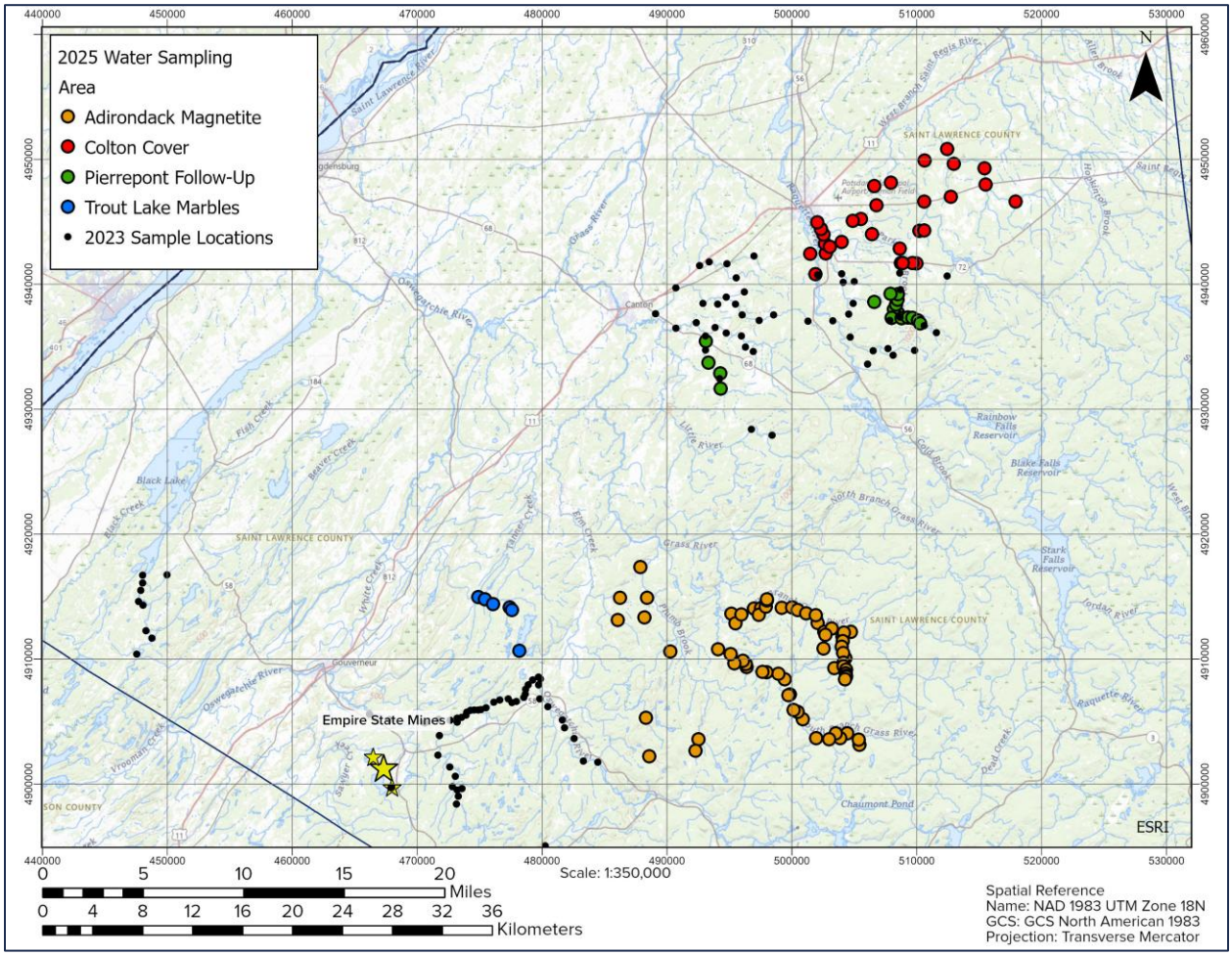


Figure 9-5: 2025 Water sampling sites by area

The 2025 program focused primarily on Parish, with follow up sampling at Pierrepont, and prospective sampling at the Trout Lake Marble and Colton Cover targets. Of the 121 samples collected, eight had copper isotope ratios above the $\delta^{65}Cu$:$\delta^{63}Cu$ 1 per mil threshold. Five of the eight samples were within areas believed to be underlain by prospective marbles of the Adirondack Lowlands, the remaining three were collected from within the Adirondack Magnetite Belt. The samples collected from the Adirondack Lowlands with $\delta^{65}Cu$:$\delta^{63}Cu$ ratios ≥1 per mil have helped confirm anomalies from the 2023 program, as well as identify new target areas in the Colton Cover and Trout Lake Marlbe sample sets. The samples within the Adirondack Magnetite belt further support the prospectivity of the area for copper mineralization.

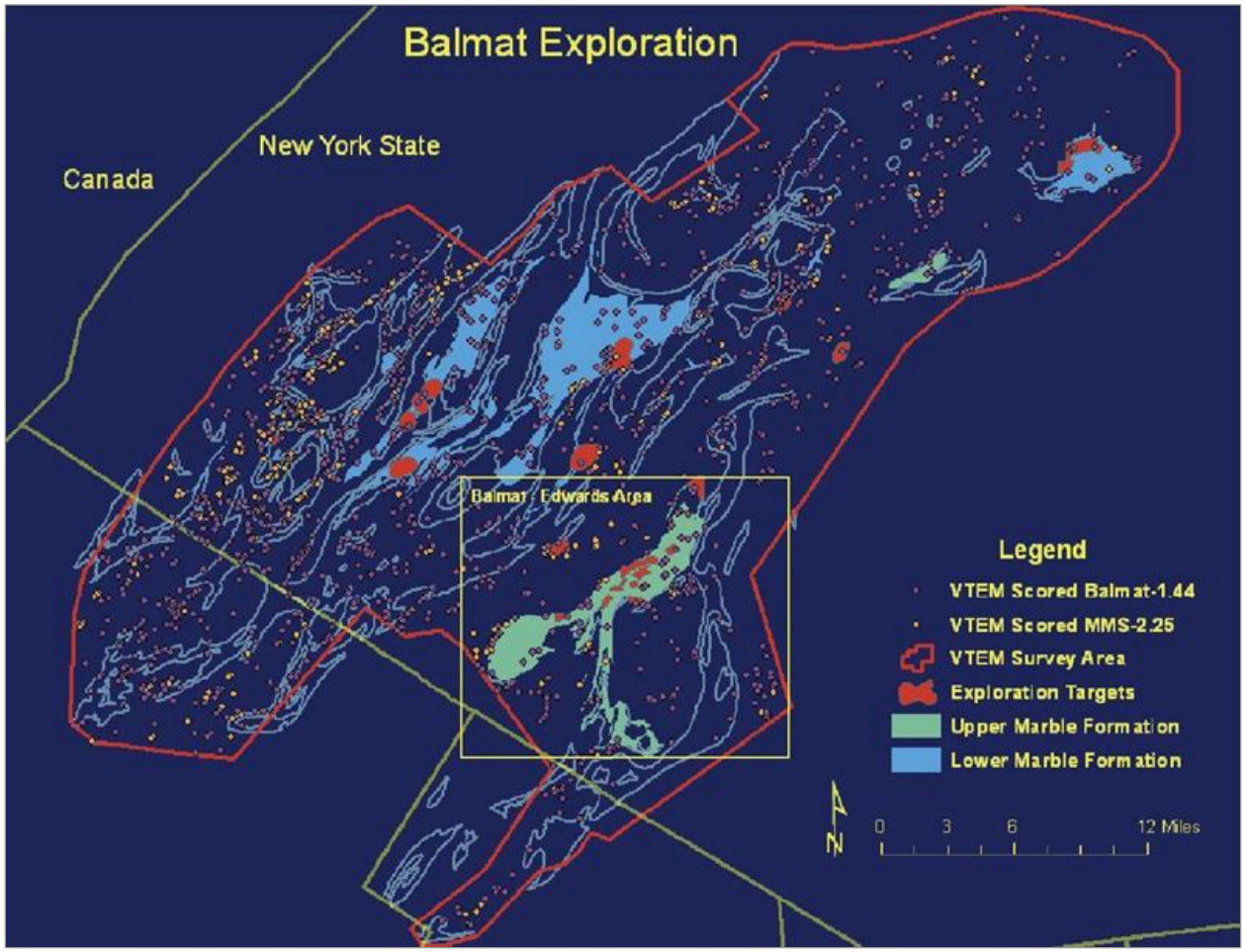
9.1.4 Airborne Geophysics

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 9-6 shows the area covered by the geophysical survey and areas where low resistivity was recorded (Rivard and Stephens, 2013).



Source: Macdonald et al., 2017

Figure 9-6: 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 southern two thirds of the 2013 survey area has been re-evaluated.

9.1.5 Exploration Potential and Targeting

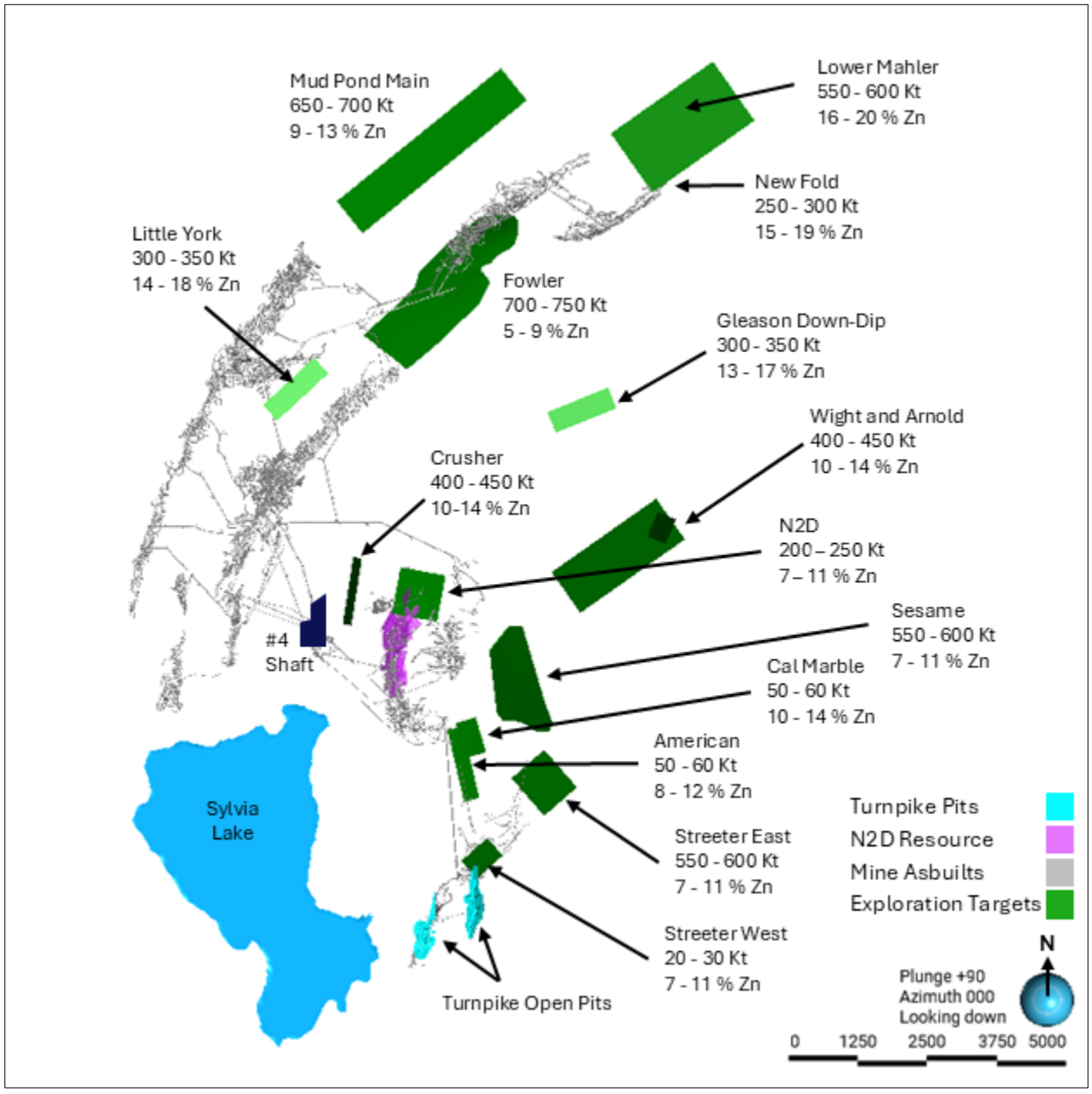
9.1.5.1 Near-Mine Exploration Targets

Several exploration targets at ESM have been identified based on possible extensions of open mineralized horizons at various depths and promising historic hits in the drillhole database. The targets shown in Table 9-7 and Figure 9-7 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 9-7. This list is not exhaustive and subject to change as new drilling information is available.

Table 9-7: Near-mine exploration targets

Target	Tons (kton)	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
N2D	200–250	7–11
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: Taylor et al., 2024



Source: Modified from Taylor et al., 2024

Figure 9-7: 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 eleven drill targets within the Balmat-Pierrepont trend that are within current mineral rights ownership. In addition to the targets on trend, the Company has six regional drill targets with mineral rights access. These can be found in Table 9-8.

In addition to drilling surface geochemical sampling, hydrogeochemical sampling, geologic mapping are all tools to be employed on the over 120,000 acres of mineral rights within St. Lawrence County.

Table 9-8: Exploration targets

Drill Targets	Target Type
Balmat–Pierrepont Trend	
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
Pleasant Valley	Testing extensions of known mineral occurrence
Pork Creek	Testing favorable stratigraphy and structure
Regional	
Moss Ridge	Testing geochemical anomaly, and mineralized breccia at surface
Bostwick	Testing historic mineralization, and geophysical anomaly
Greenwood	Testing favorable stratigraphy
Maple Ridge	Testing stratigraphy and structure
Beaver Creek	Testing geophysical and geochemical anomaly
Parish	Testing historically reported base metal occurrence

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 graphitic drill intercepts from past exploration programs. The Company has also completed 65 drillholes testing the graphite mineralization at Kilbourne. This includes the 39 holes completed during the 2023–2024 exploration program, as well as 26 holes completed in 2025. Five of the holes completed during the 2025 program were exploratory in nature and are detailed in this Item. The remaining drilling, along with the resampling, is featured in Item 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 (Item 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.

9.2.2 Airborne Geophysics

The airborne geophysical survey described in Item 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 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 9-9. The channel was cut with a Husqvarna K 770 demo saw to a depth between 4 in and 6 in. Sample lengths varied, ranging between 2.5 ft and 5 ft. Samples were chiseled out between two cuts spaced 2 in 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 Item 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. KT23-002A is considered representative of the mineralized zone with no known sampling biases, and its assay results confirmed continuity of mineralization consistent with subsequent core drilling.

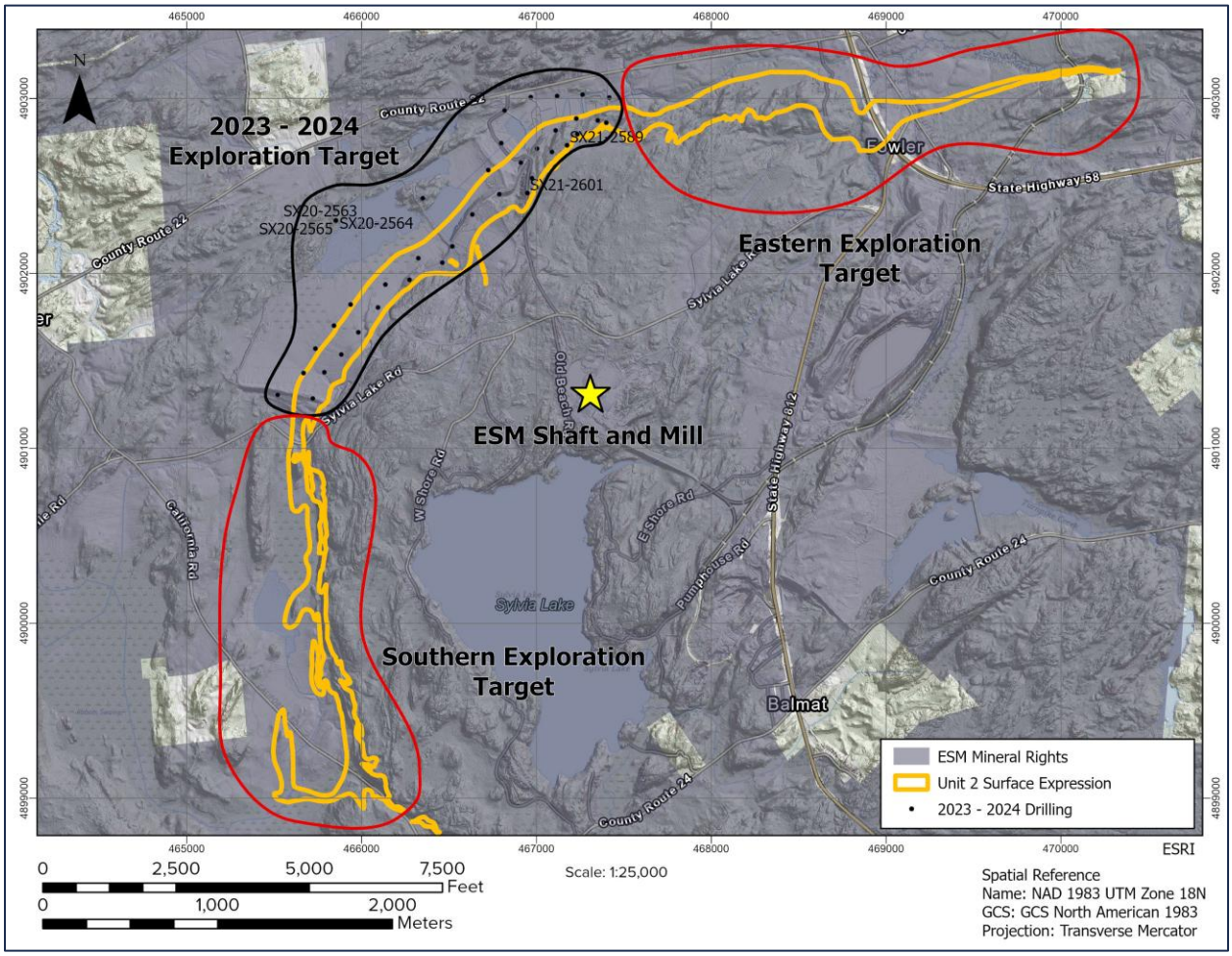
Table 9-9: ESM outcrop channel samples

Channel ID	Length (ft)	UTM NAD83			Azimuth	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: Taylor et al., 2024

9.2.4 Exploration Potential and Targeting

The Company has tested roughly 12,500 ft of near surface mineralization at Kilbourne. The mapped surface expression of UM2 continues an additional ~7,500 ft to the south, and ~5,300 ft to the east. Historic drill intercepts along strike from Kilbourne in both directions have documented graphite mineralization. Figure 9-8 shows the mapped extensions of Kilbourne's strike length. Half 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: Modified from Taylor et al., 2024

Figure 9-8: 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 60% 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 Item of the report provides an update of ESM's drilling in two sections. Item 10.1 describes the overall ESM drillhole database inclusive of the Kilbourne graphite drilling that overlies the zinc deposits. Item 10.2 breaks out the Kilbourne graphite drilling in more detail.

10.1 ESM Drilling

Since the last technical report (Taylor et al., 2024) ESM has completed 109 diamond drillholes totaling 39,410 ft across multiple zinc-bearing areas (Figure 10-1). The 2024 program focused on definition and exploration drilling in the Lower Mahler zone and initial infill drillholes in the New Fold horizon, while the 2025 program expanded drilling in both zones and initiated exploration in the No. 2 Deep and Little York areas. A single deep hole (UX24-036) tested the Mud Pond Main zone for potential extensions.

Drilling in the Mahler zone confirmed the continuity of mineralization and improved confidence in modeled lenses, supporting conversion of Inferred resources to Indicated and expanding the total Inferred Mineral Resources. Overall, model updates driven by drilling results since 2024 balanced mining-related depletions and support continued development of Mahler and New Fold.

Exploration drilling in N2D, Little York, and Mud Pond Main is ongoing; assays for the Little York program are pending, and the N2D program will be completed by year-end, with resource updates expected thereafter. The Mud Pond Main exploration hole confirmed the presence of mineralization, intersecting 11.6 ft grading 13.6% Zn approximately 1,760 ft down-dip and 650 ft deeper than the current Mineral Resource model limit. Additional drilling will be required to classify this material as Inferred, but the zone remains promising for future potential Mineral Resource growth.

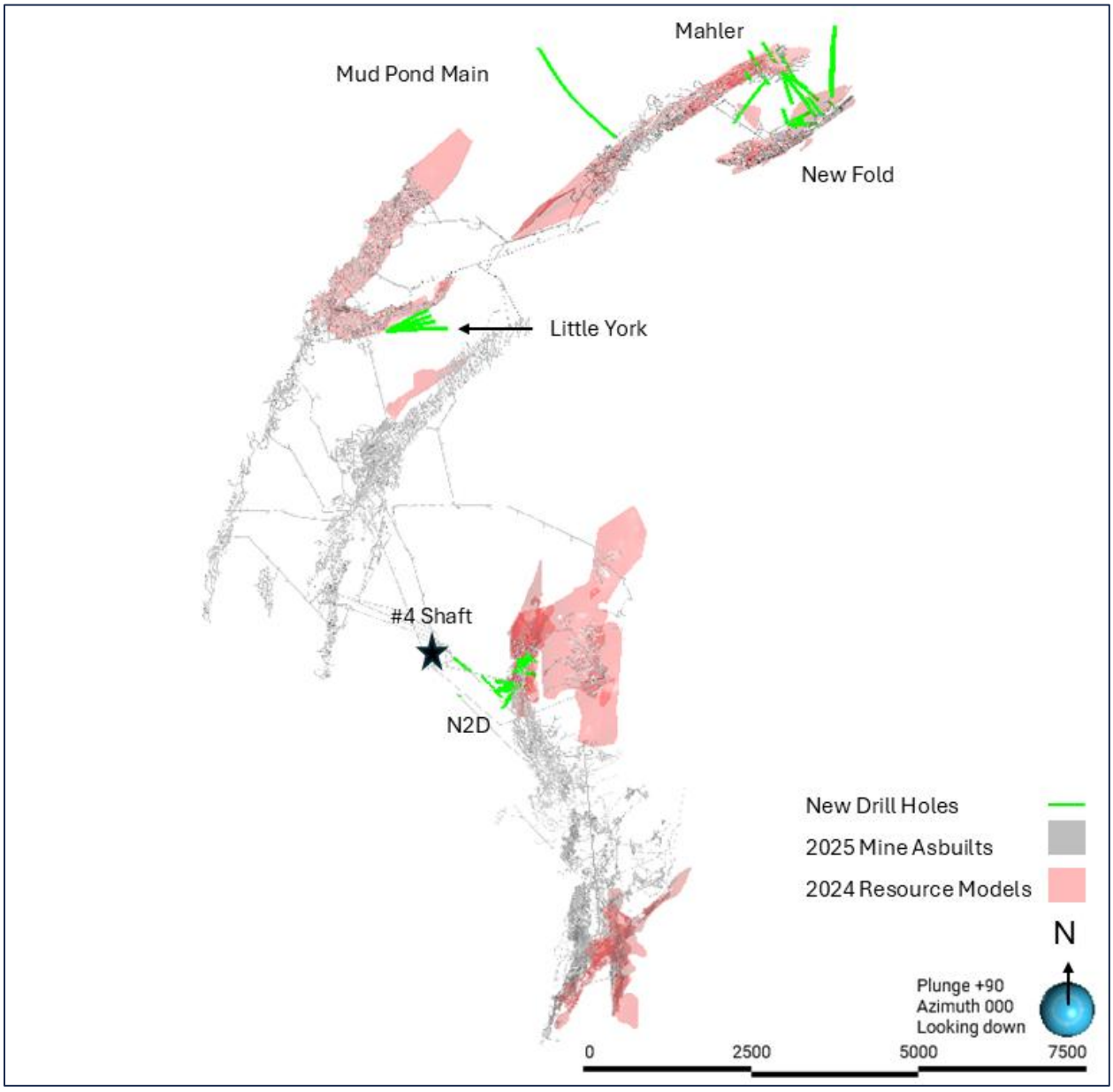
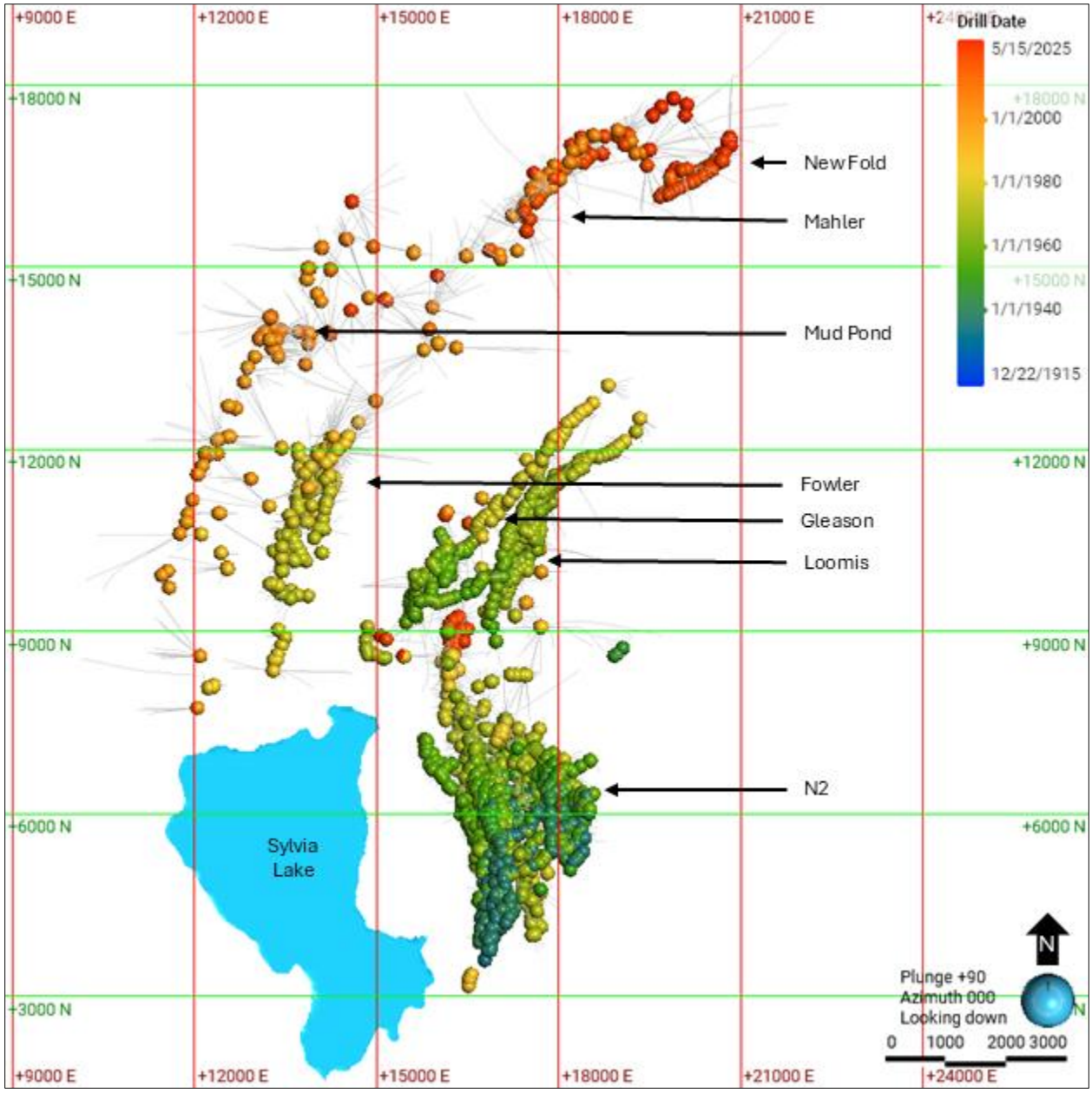


Figure 10-1: Diamond drilling targeting zinc since the 2024 technical report

10.1.1 Drilling Summary

As of November 7, 2025, a total of 12,105 diamond drillholes have been completed at ESM, totaling 4,450,353 ft, as shown in Table 10-1. All drilling since 2017 has been conducted by or on behalf of ESM. Historic drilling is continually digitized and incorporated into the digital database as records are discovered; consequently, the numbers in this document may be greater than those

in earlier reports (Macdonald et al., 2017; Warren et al., 2021, Taylor et al., 2024). As far as ESM is aware, no additional significant groups of historical drilling remain to be digitized. Figure 10-2 displays underground core drilling with collars colored by date drilled to highlight the mining progression for each zone from south to north at increasing depths from the surface.



Source: Modified from Taylor et al., 2024

Figure 10-2: Map showing Balmat underground drilling colored by drill date

The drillhole database was sub-divided into geographic "Areas" that can be extracted individually (Figure 10-3). Any drillholes beyond the extents of the shaded domains in the figure are stored in District. Drill footages for each area are presented in Table 10-1.

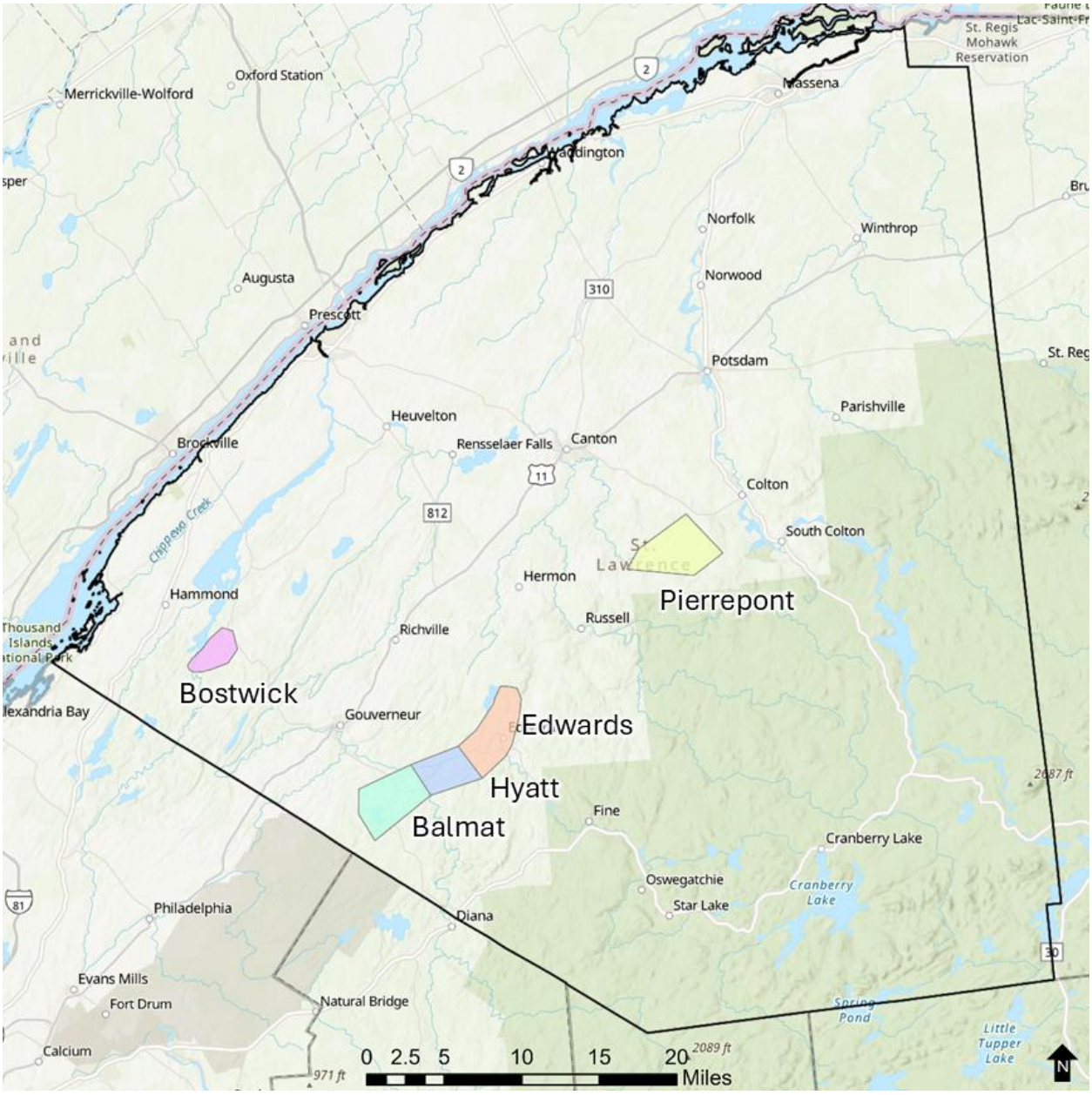


Figure 10-3: Drillhole database areas; drillholes outside the shaded areas are stored in District

Table 10-1: Area drilling by year since 2020

Area	Year	Surface Core		UG Core		Total Holes	Total Length (ft)	Total Length (m)
		Holes	Length (ft)	Holes	Length (ft)			
Balmat	pre-2020	1,169	1,133,215	6,685	1,757,744	7,854	2,890,959	881,164
	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	41	13,520	44	17,465	85	30,985	9,444
	2025	27	9696	76	29,187	103	38,883	11,852
	Total	**1,346**	**1,209,099**	**7,143**	**1,934,198**	**8,489**	**3,143,298**	**958,077**
Hyatt	pre-2020	403	380,087	318	42,980	721	423,067	128,951
	2020	1	307	-	-	1	307	93.5736
	2021	15	4,428	-	-	15	4,428	1,350
	2025	5	7,025	-	-	5	7,025	2,141
	Total	**424**	**391,847**	**318**	**42,980**	**742**	**434,827**	**132,535**
Edwards	**Total**	**246**	**217,695**	**2030**	**332,111**	**2276**	**549,806**	**167,581**
Pierrepont	pre-2020	280	162,915	97	10,431	377	173,346	52,836
	2021	8	2,707	-	-	8	2,707	825.0936
	Total	**288**	**165,622**	**97**	**10,431**	**385**	**176,053**	**53,661**
Bostwick	pre-2020	57	24,206	-	-	57	24,206	7,378
	2022	3	3,690	-	-	3	3,690	1,125
	Total	**60**	**27,896**	**-**	**-**	**60**	**27,896**	**8,503**
District	pre-2020	136	103,212	-	-	136	103,212	31,459
	2022	14	12,996	-	-	14	12,996	3,961
	2025	3	2,267	-	-	3	2,267	691
	Total	**153**	**118,475**	**-**	**-**	**153**	**118,475**	**36,111**
Grand Total		**2,517**	**2,130,633**	**9,588**	**2,319,720**	**12,105**	**4,450,353**	**1,356,468**

Source: Modified from Taylor et al., 2024

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

10.1.2 Drilling Procedures

Drilling in the Balmat-Edwards District has primarily been conducted using core drilling methods. At ESM, the mine currently owns and operates several drill rigs for both underground and surface exploration. These include Diamec 262 drills for definition drilling, an Epiroc U-6 for underground exploration, and JKS Boyle B20 rigs for surface exploration. Table 10-2 summarizes the currently owned and operated drills.

Table 10-2: Table of company owned and operated core drills

Drill Model	Type	Core Size(s)	Primary Use
Diamec 262 (×2)	Underground	AWJ	Definition drilling
Epiroc U-6	Underground	BQ	Underground exploration
JKS Boyle B20 (×2)	Surface	HQ, NQ, BQ	Surface exploration

10.1.3 Core Handling and Sampling

Since 2017, underground core has been managed using the following procedure. Prior to ESM operations, core handling was generally similar, except that geological logging occurred underground and only mineralized intervals selected for assaying were transported to surface in wooden boxes. The current on-site practice is as follows:

After drilling, the driller removes the core from the drill string and places it 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 10-4.



Source: Taylor et al., 2024

Figure 10-4: 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. The 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 on site in the 1930's 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 QP is 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 Graphite

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 10-3. While the holes were used in the MRE they form a minor component of the overall Kilbourne drillhole database. The red and green collar points in Figure 10-5 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: Taylor et al., 2024

10.2.2 Core Drilling Summary

As of November 7, 2025, a total of 66 surface diamond drillholes targeting graphite were completed by ESM, totaling 21,613 ft. Kilbourne core footage totals by year are presented in Table 10-4. Drillholes KX23-001 to KX24-039 were completed in 2024 and used for the initial Mineral Resource Estimate. KX25-040 to KX25-044 were exploration holes outside the limits of the initial resource; the results are discussed in Item 9. KX25-045 to KX25-065 are definition holes within the footprint of the initial resource and assays are pending. Logged lithologies are consistent with the initial drilling and are not expected to materially impact the resource. Drillhole collars colored by year are shown in Figure 10-5.

Table 10-4: Kilbourne drilling by year

Year	Holes	Length (ft)	Length (m)
2023	1	270	82
2024	38	11,647	3,550
2025	26	9,696	2,955
Total	**65**	**21,613**	**6,588**

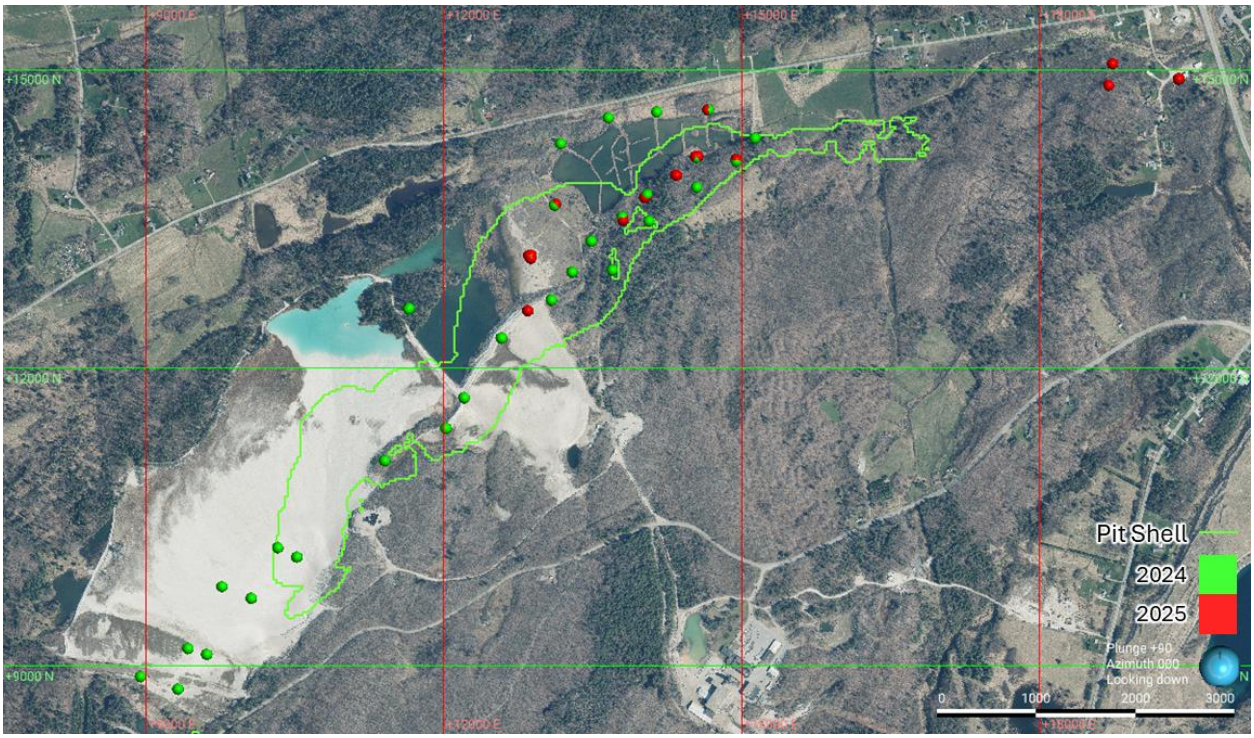


Figure 10-5: Kilbourne drilling with collars colored by year; RPEEE constraining pit footprint in green

The 2025 Kilbourne East exploration program has intercepted near surface graphite mineralization 4,278 ft east of the easternmost drillholes of the 2023-2024 program. The Company has completed five holes, totaling 1,738 ft drilled of a planned 4,500 ft program. This program will continue to target near surface mineralization to the east of the current Kilbourne resource. Graphite intercepts have been consistent in grade and nature to those recorded in the Kilbourne deposit. Table 10-5 lists the collar information for the exploration holes completed to date during the 2025 exploration program. Table 10-6 lists the significant intercepts of graphite mineralization from the completed holes.

Table 10-5: 2025 Kilbourne exploration drilling

Hole ID	Length (ft)	UTM NAD83			Azimuth	Dip	Start Date	End Date
		Easting (m)	Northing (m)	Elevation (m)				
KX25-040	317	468715.4	40903110.1	180.7	180	-50	8/26/2025	9/1/2025
KX25-041	348	468715.3	4903111.3	180.4	180	-90	9/4/2025	9/5/2025
KX25-042	307	468500.7	4903091.0	179.4	180	-45	9/9/2025	9/15/2025
KX25-043	358	468500.6	4903092.6	179.4	180	-90	9/17/2025	9/18/2025
KX25-044	408	468512.5	4903158.0	179.4	180	-90	9/19/2025	9/24/2025

Table 10-6: Significant intercepts from 2025 Kilbourne exploration program

Hole ID	From (ft)	To (ft)	Interval (ft)	From (m)	To (m)	Interval (m)	Cg%	Zone
KX25-040	195.7	215.0	19.3	59.6	65.5	5.9	2.9	Upper
KX25-041	186.6	298.3	111.7	56.9	90.9	34.0	2.8	Combined
KX25-042	50.0	65.0	15.0	15.2	19.8	4.6	2.6	Upper
	225.0	240.2	15.2	68.6	73.2	4.6	2.6	Lower
KX25-043	No Significant Intercepts							
KX25-044	162.2	214.9	52.7	49.4	65.5	16.1	2.5	Upper

10.2.3 Drilling Procedure

The 2023-2024 drilling program was completed using an ESM owned and operated Diamec 262 underground drill mounted on a skid plate specifically for the Project as shown in Figure 10-6. All core samples were AWJ size. The 2025 exploration program was completed using an ESM owned and operated JKS Boyles B-20. All core samples in 2025 were NQ size.



Source: Taylor et al., 2024

Figure 10-6: Diamec #2 on the surface drilling for graphite

10.2.4 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 10-7), 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: Taylor et al., 2024

Figure 10-7: 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.

There are no documented factors that would have had a negative impact on sample quality. Sample selection during this program is often made based on outward and visible signs of mineralization.

10.2.5 Downhole Surveying

Surface exploration drilling used an ESM owned DeviShot.

The QP is not aware of any drilling, sampling, or recovery factors that would negatively impact the accuracy and reliability of drill sample results at the Kilborne Project.

11. Sample Preparation, Analyses, and Security

11.1 Zinc 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 located in the mill on the mine site near Balmat, New York. The certification status is unknown and the laboratory was not independent of the operator at the time.

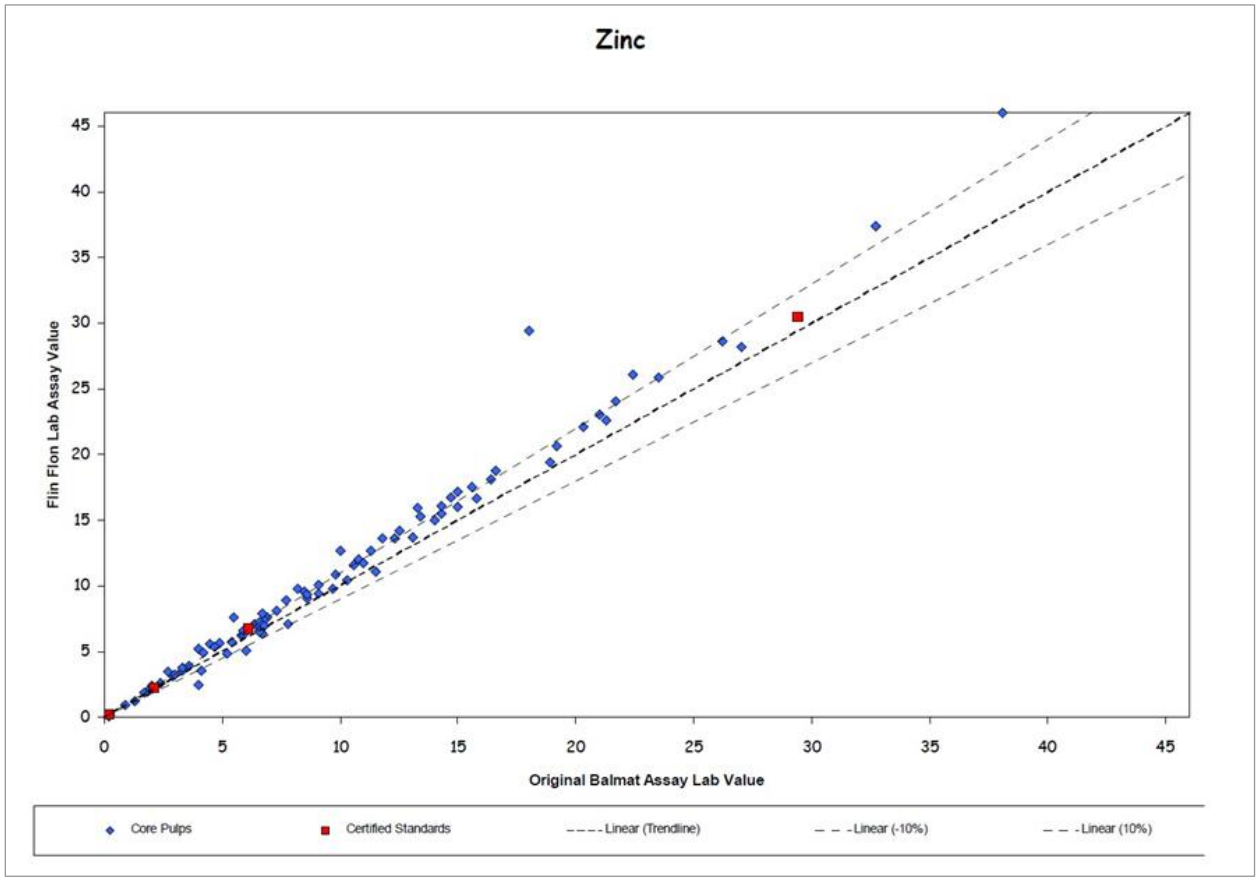
Historic Verifications

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 in-house Flin Flon, Manitoba, assay laboratory for check analyses. The certification status of the Flin Flon laboratory is unknown and was not independent of the operator at the time. 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.

The pulps are no longer available for testing and further verification of historic assays drilled prior to 2000 is not possible.



Source: Macdonald et al., 2017

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 Flin Flon laboratory; these indicate that the ESM assays prior to 2003 may underestimate zinc concentrations.

11.1.2 Hudbay 2005-2010 Assaying

All drillhole core samples from the 2005 to 2010 diamond drilling programs were sent to the ALS Chemex Laboratory (ISO/IEC 17025 accredited) 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: Taylor et al., 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: Taylor et al., 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.

The pulps are no longer available for testing and further verification of historic assays drilled between 2005 and 2010 is not possible.

11.2 2017 to 2025 Zinc Sample Preparation and Assaying

A total of 42,646 drill core samples were submitted to ALS Geochemistry between April 2017 and June 2025. The quality control data for these sample submittals are discussed in Item 11.2 for zinc, lead, copper, silver, gold, and iron.

11.2.1 Sample Preparation and Analysis

For the 2017 to 2025 drilling campaigns, sample preparation (crushing and pulverizing) has been performed at the ALS prep-laboratory located in Sudbury, Ontario, Canada. A pulp of each sample was prepared, and a portion (typically 100 grams) was forwarded to the ALS laboratory in Vancouver, British Columbia, Canada, for analysis. Both the Sudbury and the Vancouver ALS facilities are ISO/IEC 17025:2017 certified and are independent of Titan.

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.

Beginning in May 2025, select drillholes and samples were analyzed for 52 super trace elements (Method Code ME-MS89L) in addition to aqua regia analysis. Samples are digested by sodium peroxide fusion and HCl leach and then analyzed by ICP-AES and ICP-MS to obtain lower detection limits.

The lower and upper limits for the aqua regia digest method (ME-ICP41) are shown in Table 11-4. The lower and upper limits for the super trace multi-element analysis (ME-MS89L) are shown in Table 11-5.

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			

Table 11-5: Upper and lower limits for MS89L super trace analysis method

Analyte	Lower Limit	Upper Limit	Analyte	Lower Limit	Upper Limit	Analyte	Lower Limit	Upper Limit
Ag (ppm)	5	12,500	Ho (ppm)	0.01	25,000	Sm (ppm)	0.04	25,000
As (ppm)	4	25,000	In (ppm)	0.3	25,000	Sn (ppm)	3	25,000
Ba (ppm)	2	25,000	K (%)	0.05	25	Sr (ppm)	20	25,000
Be (ppm)	0.4	25,000	La (ppm)	0.08	25,000	Ta (ppm)	0.04	25,000
Bi (ppm)	0.1	25,000	Li (ppm)	2	25,000	Tb (ppm)	0.01	25,000
Ca (%)	0.1	25	Lu (ppm)	0.05	25,000	Te (ppm)	0.5	25,000
Cd (ppm)	0.8	25,000	Mg (%)	0.01	30	Th (ppm)	0.1	25,000
Ce (ppm)	0.2	25,000	Mn (ppm)	10	25,000	Ti (%)	0.005	25
Co (ppm)	0.5	25,000	Mo (ppm)	2	25,000	Tl (ppm)	0.02	25,000
Cs (ppm)	0.1	25,000	Nb (ppm)	0.8	25,000	Tm (ppm)	0.01	25,000
Cu (ppm)	20	25,000	Nd (ppm)	0.07	25,000	U (ppm)	0.2	25,000
Dy (ppm)	0.03	25,000	Ni (ppm)	10	25,000	V (ppm)	1	25,000
Er (ppm)	0.02	25,000	Pb (ppm)	0.5	25,000	W (ppm)	0.3	25,000
Eu (ppm)	0.03	25,000	Pr (ppm)	0.03	25,000	Y (ppm)	0.2	25,000
Fe(%)	0.01	25	Rb (ppm)	0.5	25,000	Yb (ppm)	0.02	25,000
Ga (ppm)	0.5	25,000	Re (ppm)	0.01	25,000	Zn (ppm)	10	25,000
Gd (ppm)	0.03	25,000	Sb (ppm)	0.3	25,000			
Ge (ppm)	0.5	25,000	Se (ppm)	3	25,000			

11.2.2 Security

The whole core is photographed. Underground definition drilling is submitted to the ALS laboratory 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 ALS laboratory. Failures are identified on a batch basis and followed up as required. Quarterly QA/QC reports are prepared internally to monitor overall ALS 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. (ASL) 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 11-6.

Table 11-6: Blank failure threshold

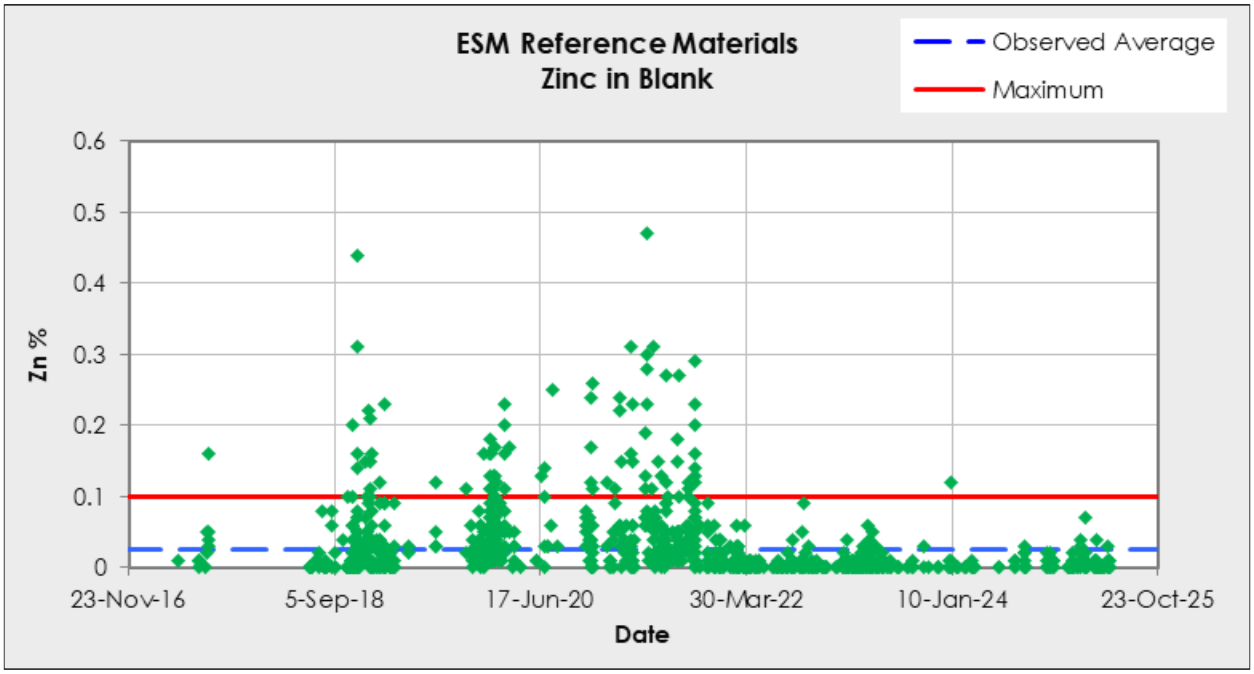
Blank	Zinc (ppm)	Lead (ppm)	Silver (ppm)	Copper (ppm)	Iron (%)
Blank	1,000	400	5	400	0.7

Source: Taylor et al., 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 1,207 blanks inserted with samples, all silver values were less than 1 ppm with the exception of two cases where silver reported at 12.7 ppm and 1.02 ppm. Copper values were less than 100 ppm, with the exception of one case where copper reported at 172 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 zinc sample cross-contamination in eight samples received after Q3 2021 where ore-grade samples were separated out. Of the 1,207 cases recorded since 2017, 77 have reported zinc levels over 0.1%. 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 was a period in January and February 2020 when zinc values in blank samples were reporting over 0.04% zinc. In October 2021, there was a decrease in cross-contamination after ore-grade batches were analyzed at ALS laboratory separately. After raising the zinc failure threshold, one blank exceeded the threshold in the period of December 2022 to June 2025.



Summary Statistics

Expected Values	
Mean	0.000
Maximum	0.100

Observed Values	
Number of Samples	1207
Mean	0.025
Percent of Maximum	25.21%

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 ALS 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 11-7.

Table 11-7: 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	397	2	5	24.6	0.799	24.430	0.542	99.31%
OREAS-G5	838	3	14	10.3	0.22	10.310	0.180	100.10%
OREAS-135b	83	-	-	2.73	0.075	2.674	0.043	97.93%
OREAS-135	488	-	1	2.8	0.104	2.764	0.055	98.72%
Total	**1,806**					**Weighted Average**		**99.45%**

Reference Material	Qty	Outliers Excluded	Failures Excluded	Cu %		Observed Cu %		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-H5	403	1	-	0.0433	0.002	0.045	0.002	103.29%
OREAS-G5	852	3	-	0.0601	0.0038	0.061	0.002	101.11%
OREAS-135b	82	1	-	0.0116	0.0006	0.012	0.000	99.21%
OREAS-135	488	1	-	0.0282	0.0012	0.029	0.001	101.26%
Total	**1,825**					**Weighted Average**		**101.54%**

Reference Material	Qty	Outliers Excluded	Failures Excluded	Pb %		Observed Pb %		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-H5	402	1	1	0.0753	0.0061	0.080	0.003	106.25%
OREAS-G5	852	3	-	0.0759	0.0058	0.078	0.003	103.25%
OREAS-135b	83	-	-	1.69	0.037	1.695	0.033	100.28%
OREAS-135	469	18	2	1.7	0.062	1.726	0.040	101.50%
Total	**1,806**					**Weighted Average**		**103.33%**

Reference Material	Qty	Outliers Excluded	Failures Excluded	Ag ppm		Observed Ag ppm		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-H5	400	-	-	3.81	0.51	4.285	0.181	112.46%
OREAS-G5	845	-	-	3.5	0.55	3.807	0.155	108.76%
OREAS-135b	82	1	-	53.5	1.34	53.688	1.623	100.35%
OREAS-135	473	2	-	54.9	2.17	55.529	2.222	101.14%
Total	**1,800**					**Weighted Average**		**107.20%**

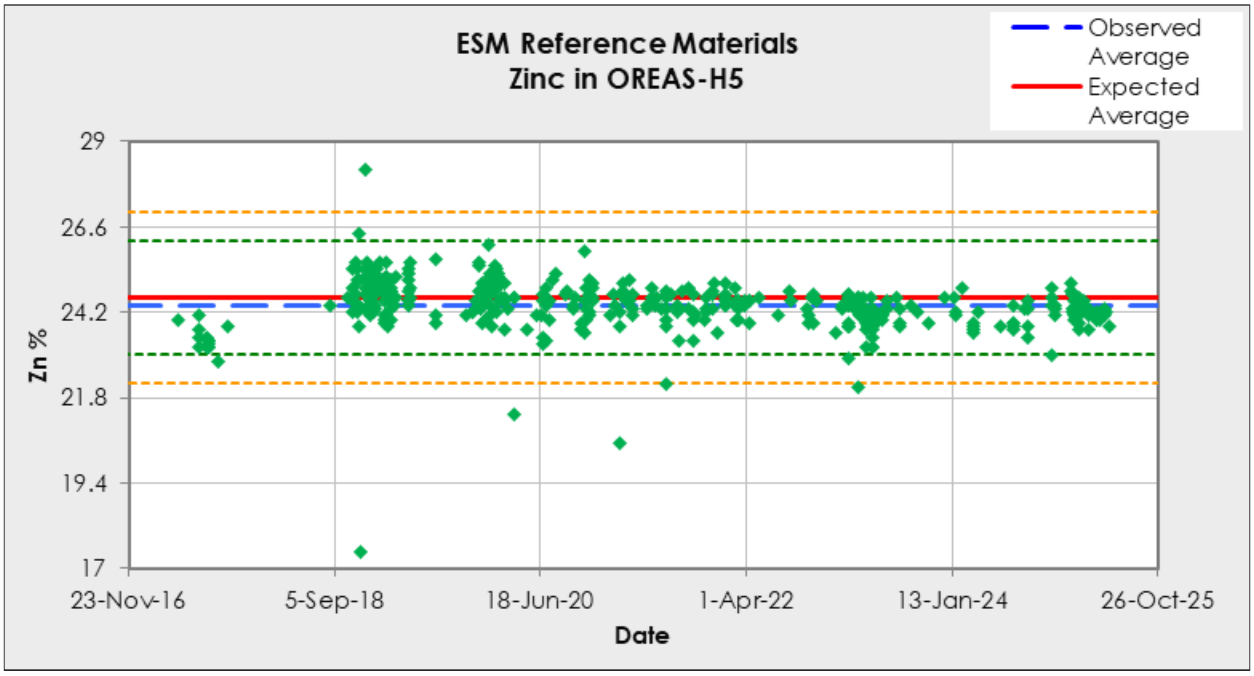
Reference Material	Qty	Outliers Excluded	Failures Excluded	Fe %		Observed Fe %		Percent of Accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS-H5	403	-	-	1.59	0.1	1.584	0.049	99.59%
OREAS-G5	852	-	-	1.49	0.09	1.473	0.040	98.85%
OREAS-135b	82	1	-	5.1	0.201	5.022	0.096	98.48%
OREAS-135	482	1	-	8.97	0.363	8.790	0.251	97.99%
Total	**1,819**					**Weighted Average**		**98.77%**

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.



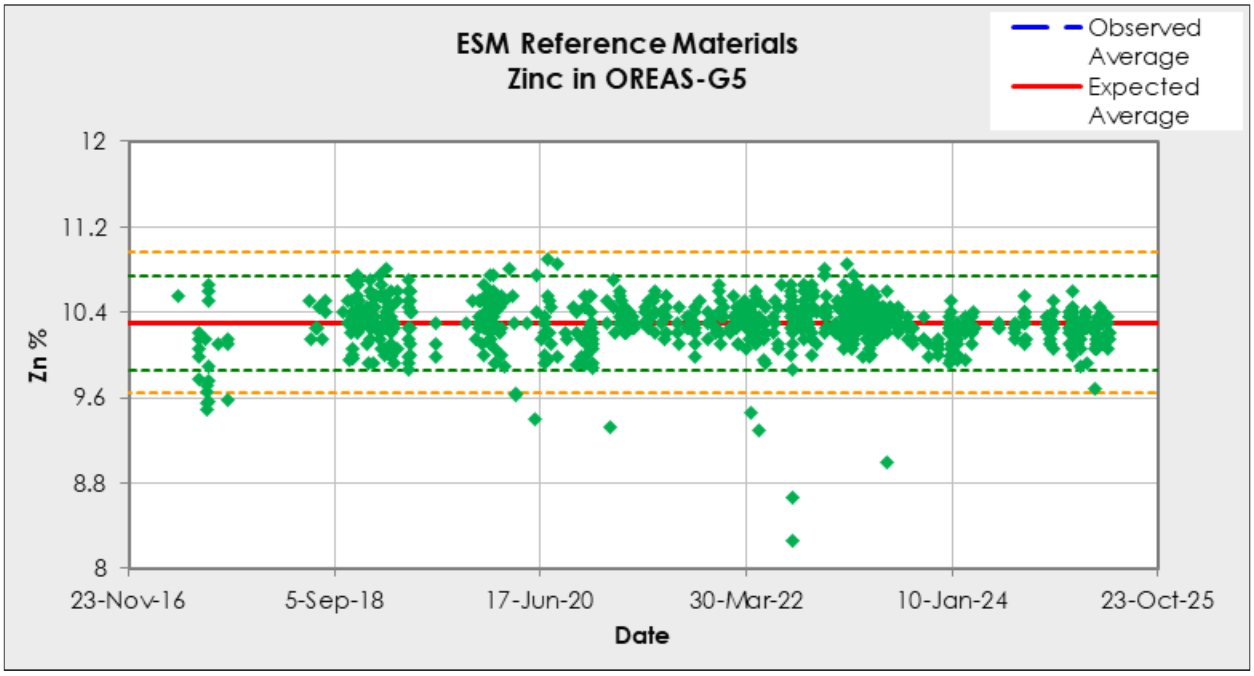
Summary Statistics

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

Observed Values	
Number of Samples	404
Mean	24.390
Standard Deviation	0.734
2 x RSD	6.02%
Falls Within 3 SD of Certified Mean	99%
Falls Within 2 SD of Certified Mean	98%
Falls Within 1 SD of Certified Mean	80%

Source: Graph generated in QC Mine Software

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	855
Mean	10.294
Standard Deviation	0.226
2 x RSD	4.39%
Falls Within 3 SD of Certified Mean	99%
Falls Within 2 SD of Certified Mean	96%
Falls Within 1 SD of Certified Mean	77%

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 for the zinc operations meet or exceed industry standards and are, therefore, acceptable for the estimation of Mineral Resources.

11.3 Graphite – Kilbourne Sample Preparation and Assaying

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

11.3.1 Sample Preparation and Analysis

Sample preparation (crushing and pulverizing) has been performed at SGS Lakefield laboratory 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. Both the SGS Lakefield and SGS Burnaby laboratories are ISO/IEC 17025 accredited and are independent of the Company.

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 11-8: 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 11-9.

Table 11-9: 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 SGS laboratory. Failures are identified on a batch basis and followed up as required. Drilling program QA/QC reports are prepared internally to monitor overall SGS 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 11-10.

Table 11-10: 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 11-11.

Table 11-11: 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 0.025% allowable carry-over within SGS laboratory 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.

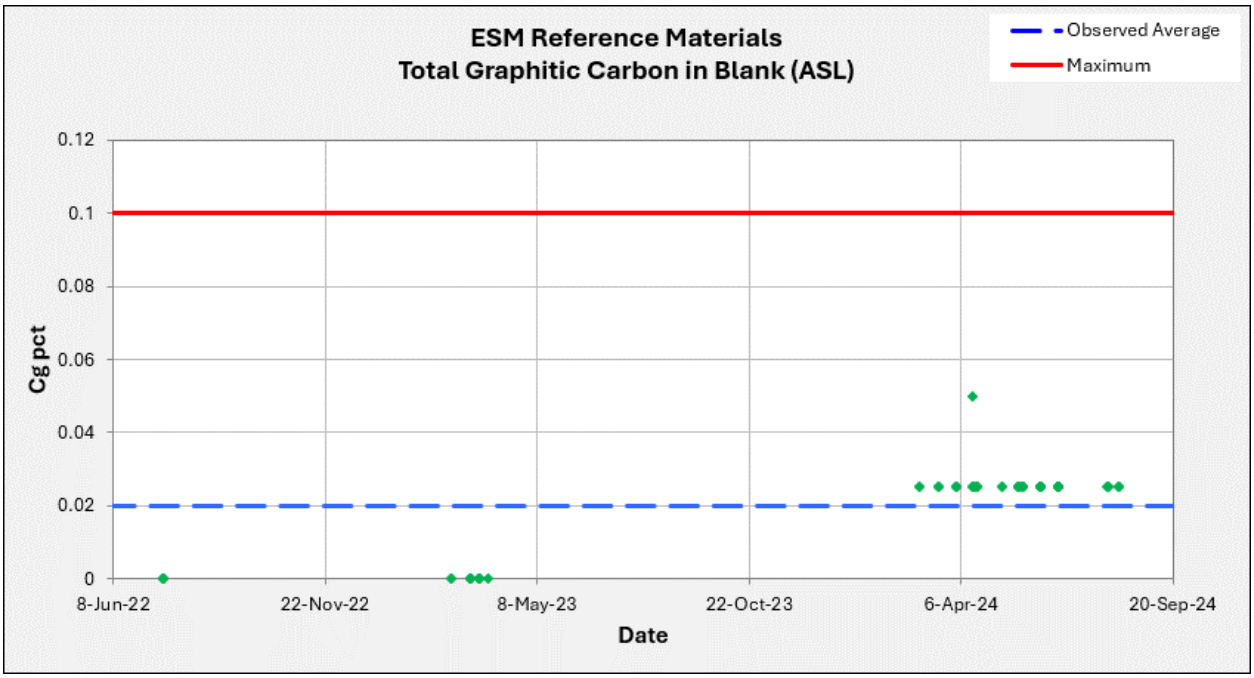
Summary statistics for OREAS-722, 724, 725 graphite performed well and report on average within 101.2–104.4% of the expected values. The results for the certified reference materials are summarized in Table 11-12.

Table 11-12: 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.



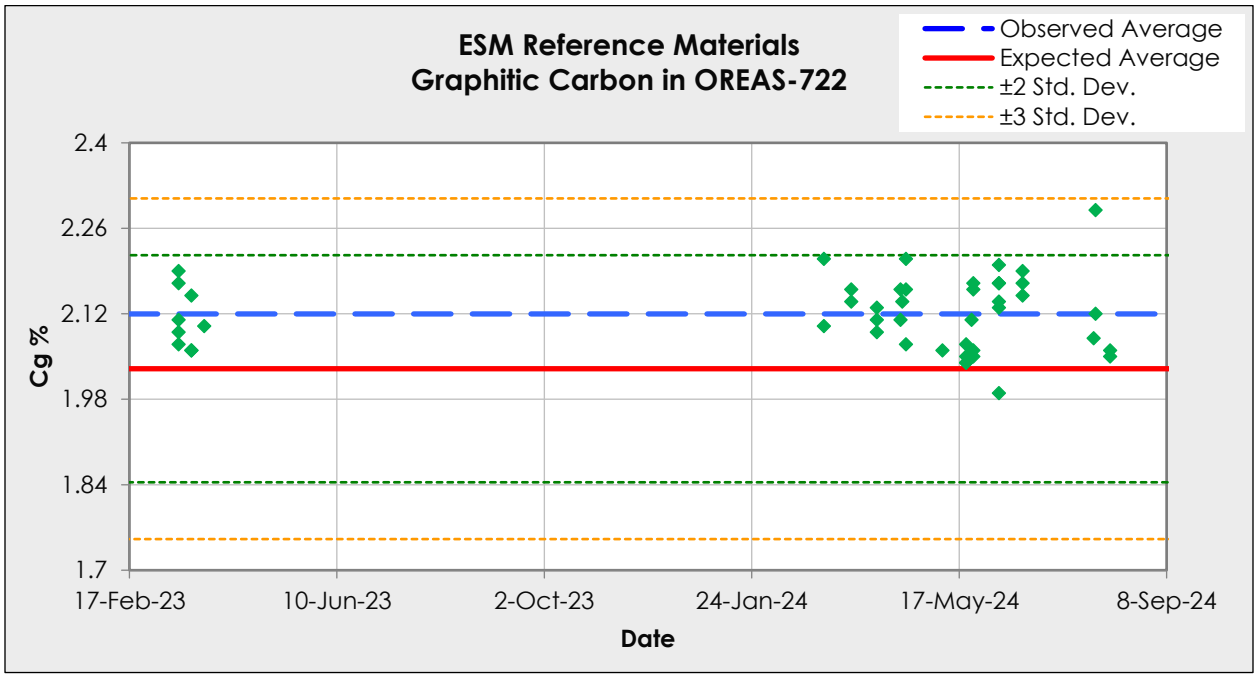
Summary Statistics

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



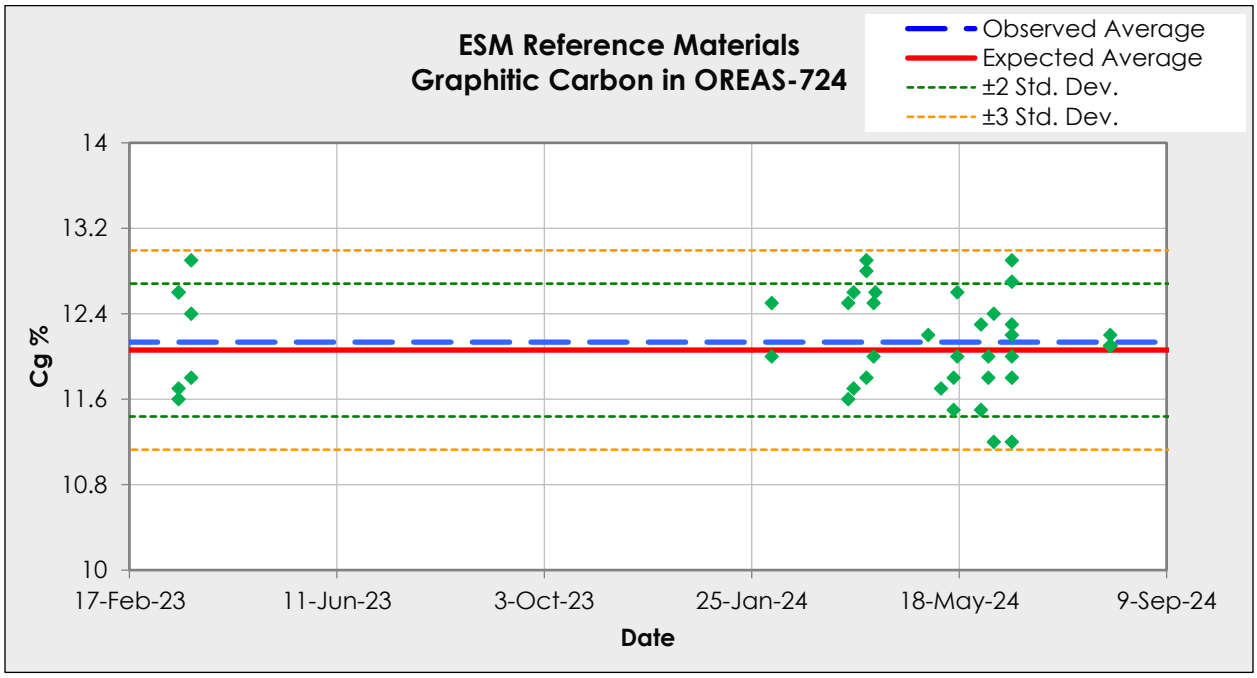
Summary Statistics

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



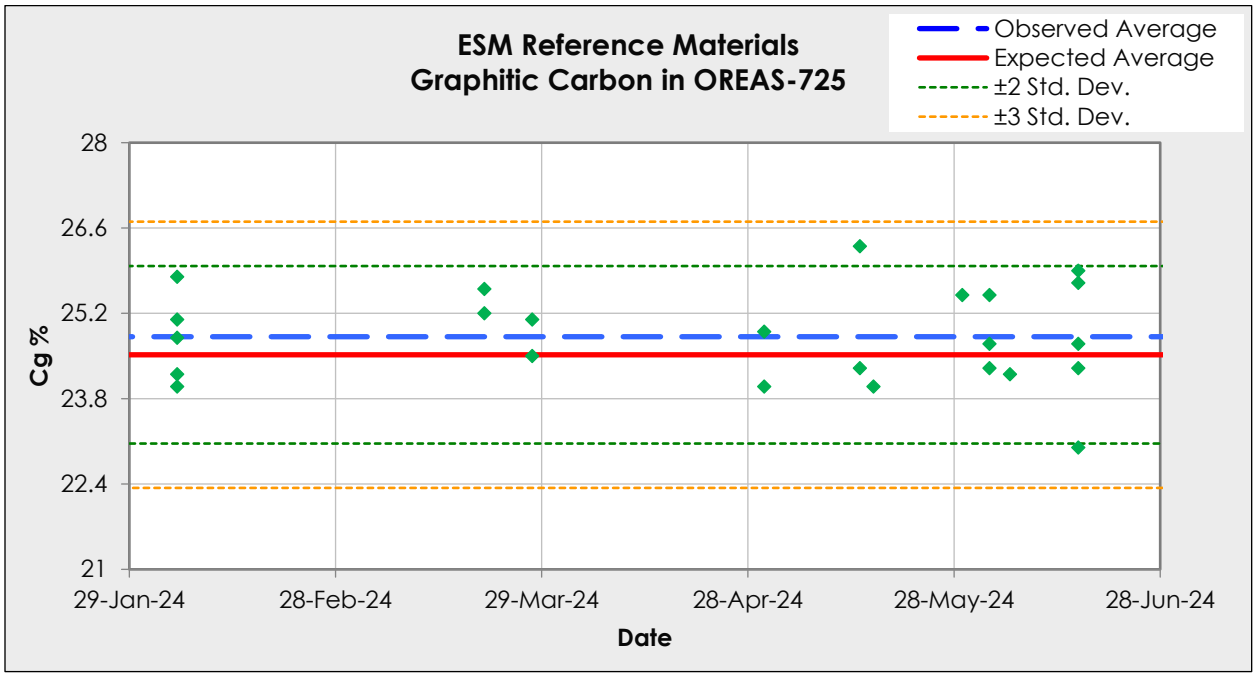
Summary Statistics

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

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

12. Data Verification

12.1 Zinc

12.1.1 Geology

The QP, Mr. Don Taylor, visited the site most recently from August 20 to 22, 2024, and has made multiple site visits since 2016. During these visits, the QP examined drift headings, drill collar locations, and diamond drill core. The QP reviewed geology, logging procedures, QA/QC protocols, and modeling procedures with ESM staff. Drillhole information used in the resource models was checked against original source documents, including historical geologic logs. Independent consultants assisted in cleaning and verifying the database prior to resource modeling in 2017.

The QP did not rerun laboratory analyses or collect third-party samples as the QP reviewed the consultant data validation procedures and considered them consistent with industry standards.

In the QP's opinion, the geology data and supporting documentation are acceptable to support the Preliminary Economic Assessment (PEA).

12.1.2 Geotechnical

The QP, Mr. Don Taylor, visited the site most recently from August 20 to 22, 2024, and has made multiple site visits since 2016. During these visits, the QP examined active workings and observed ground support procedures. The QP reviewed geotechnical reports, core logging, mapping, smart cable readings, extensometer data, pull testing, and damage tracking. In addition to internal reviews, ESM's underground geotechnical procedures have been subject to third-party reviews by independent consultants to confirm compliance with industry standards.

The QP did not perform independent laboratory testing or collect third-party samples as the QP reviewed the methodologies and considered them consistent with industry standards.

In the QP's opinion, the geotechnical data and supporting documentation are acceptable to support the PEA.

12.1.3 Metallurgical

The QP, Mr. Deepak Malhotra, last visited the site prior to start-up in 2016, where he inspected the concentrator laboratory, all process circuits, and concentrate storage facility. The QP supervised metallurgical testwork conducted in 2020 targeting the Turnpike Open Pit resource. Metallurgical data reviewed by the QP includes historical and recent testwork as well as current operating mill performance data.

The QP did not perform independent laboratory analyses or collect third-party samples as the QP reviewed testwork methodologies, QA/QC procedures, and laboratory certifications, and considered them consistent with industry standards.

In the QP's opinion, the metallurgical data and supporting documentation are acceptable to support the PEA.

12.1.4 Hydrological / Hydrogeological

The QP, Mr. Steve Trader, visited the site most recently on October 1 and 2, 2025, and has made numerous site visits since 2011. During these visits, the QP personally measured water levels, collected water quality samples from wells and mine inflows, prepared water budgets, and authored hydrogeological assessment reports for ESM and its predecessors. The QP reviewed groundwater inflow assessments, water table elevation models, and water quality data provided by ESM. In the QP's opinion, the available hydrogeological data and supporting documentation are acceptable to support the PEA.

12.1.5 Environmental

The QP, Mr. Steve Trader, visited the site most recently on October 1 and 2, 2025, and has made multiple site visits since 2011. The QP reviewed environmental baseline studies, permitting documentation, and ongoing monitoring results. The QP was actively involved in water level monitoring, monitoring well installations, residential well surveys, and water quality sampling.

In the QP's opinion, the environmental data and supporting documentation are acceptable to support the PEA.

12.1.6 Marketing

Marketing data includes internal assessments of concentrate quality, historical treatment charges, and smelter capacity. The QP, Mr. Don Taylor, reviewed the offtake agreement, benchmarked treatment charges and payables against market data, assessed concentrate specifications, and confirmed that smelter capacity and historical sales support the marketing assumptions used in this report. The QP reviewed internal marketing documentation, concentrate specifications, and offtake terms, and is of the opinion that the data is considered acceptable to support the PEA.

12.1.7 Cost Estimates

Cost estimates were developed using current operating data, vendor quotes, and engineering inputs provided by ESM. The QP, Mr. Don Taylor, reviewed the cost estimate reports, supporting documentation, and methodologies for consistency and reasonableness. The QP did not independently verify vendor quotations or operating cost data but confirmed that the sources and assumptions align with industry practice for Preliminary Economic Assessments.

The QP did not identify limitations in the data verification for cost estimates.

In the QP's opinion, the cost estimates and supporting documentation are acceptable to support the PEA.

12.2 Graphite

12.2.1 Geology

The QP, Mr. Todd McCracken, P.Geo., visited the Property on August 26 and 27, 2024 and on August 23 and 24, 2025. While on site, Mr. McCracken examined the outcrops, drill collar location, channel samples and diamond drill core samples. Mr. McCracken reviewed the geology, logging procedures and the QA/QC procedures with ESM.

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.

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. The QP did not collect independent samples. In the QP's opinion, 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.

12.2.2 Geotechnical – Kilbourne Open Pit

BBA's rock mechanic team, under the supervision of the QP, Ms. Bahareh Asi, P.Eng., analyzed the geological information compiled from diamond drilling. The geotechnical design basis for this PEA study used available RQD data and core photos from existing Kilbourne Graphite diamond drilling to establish estimates of rock mass characteristics for the hanging wall (Popple Hill Gneiss) and Footwall (UM2 and UM3 units). These rock mass assumptions were used with two empirical design

approaches (and using Design acceptance criteria of FS = 1.2) to establish the scoping study level slope design criteria. Slope design criteria were verified by comparison to other similar open pit projects and experience of the QP.

In the QP's opinion, the available data is adequate to support the preliminary pit slope design for this PEA at scoping level, though additional geotechnical site investigations will be required for the next study level.

12.2.3 Metallurgical

Metallurgical sample selection for the Phase II SGS program was performed to include 118 samples with a wide spatial distribution. All assay results were generated by commercial labs that are ISO certified with good QA/QC protocol. The results were validated with additional QA/QC steps such as ensuring that direct and reconciled grades fell within an acceptable range. It is the QP's opinion that the methods and procedures employed in the metallurgical test programs reflect standard industry practice and that the generated data is sufficient for a process plant design that meets Preliminary Economic Assessment requirements.

12.2.4 Hydrogeological

The QP, Mr. Steve Trader, reviewed available hydrogeological data relevant to the Kilbourne Project. The QP has been actively involved in hydrogeological work at the ESM site, including water level monitoring, water table elevation modeling, monitoring well installations, residential well surveys, and water quality sampling. The QP examined groundwater inflow assessments and water table elevation models provided by Titan.

In the QP's opinion, the available hydrogeological data and supporting documentation are acceptable to support the PEA.

12.2.5 Kilbourne Site Infrastructure

The QP, Mr. David Willock, P.Eng., visited the Property on August 23 and 24, 2025. While on site, Mr. Willock examined the existing site infrastructure including electrical installations, process and wastewater management facilities, ESM surface site facilities, roadways, potential tailings management sites, as well as potential locations for future graphite infrastructure.

All data regarding existing equipment and facilities was provided by Titan. The QP is of the opinion that the available data is acceptable to support the PEA.

12.2.6 Environmental

The QP, Mr. Steve Trader, reviewed available environmental data relevant to the Kilbourne Project. The QP has been actively involved in environmental work at the ESM site, including wetland surveying, governmental permitting, and ongoing discharge monitoring. The QP examined baseline studies and permitting documentation provided by Titan.

In the QP's opinion, the environmental data and supporting documentation are acceptable to support the PEA.

12.2.7 Marketing

The QP, Mr. Don Taylor, reviewed relevant marketing documents and compared the information to publicly disclosed information. The QP is of the opinion that the data is considered acceptable to support the PEA.

12.2.8 Cost Estimates

The cost estimates were developed based on the engineering inputs, assumptions, current ESM operating data, vendor quotes, benchmark data, and industry-standard cost models. The QP, Ms. Bahareh Asi, P.Eng., compiled and reviewed these estimates for accuracy, consistency and applicability. The QP did not validate the technical data or estimates prepared by other QPs.

The QP is of the opinion that the cost estimate is acceptable to support the PEA and is not aware of any mining, processing, infrastructure, permitting, or other factors that would materially affect the estimate beyond the expected accuracy range at this level of study.

13. Mineral Processing and Metallurgical Testing

13.1 Zinc

Empire State Mines is an active underground mine that processes mineralized material to produce zinc concentrate. Two new zones of near-surface mineralization near the existing operation were recently discovered. Metallurgical testwork 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 testwork is reviewed in Item 13.1.2.

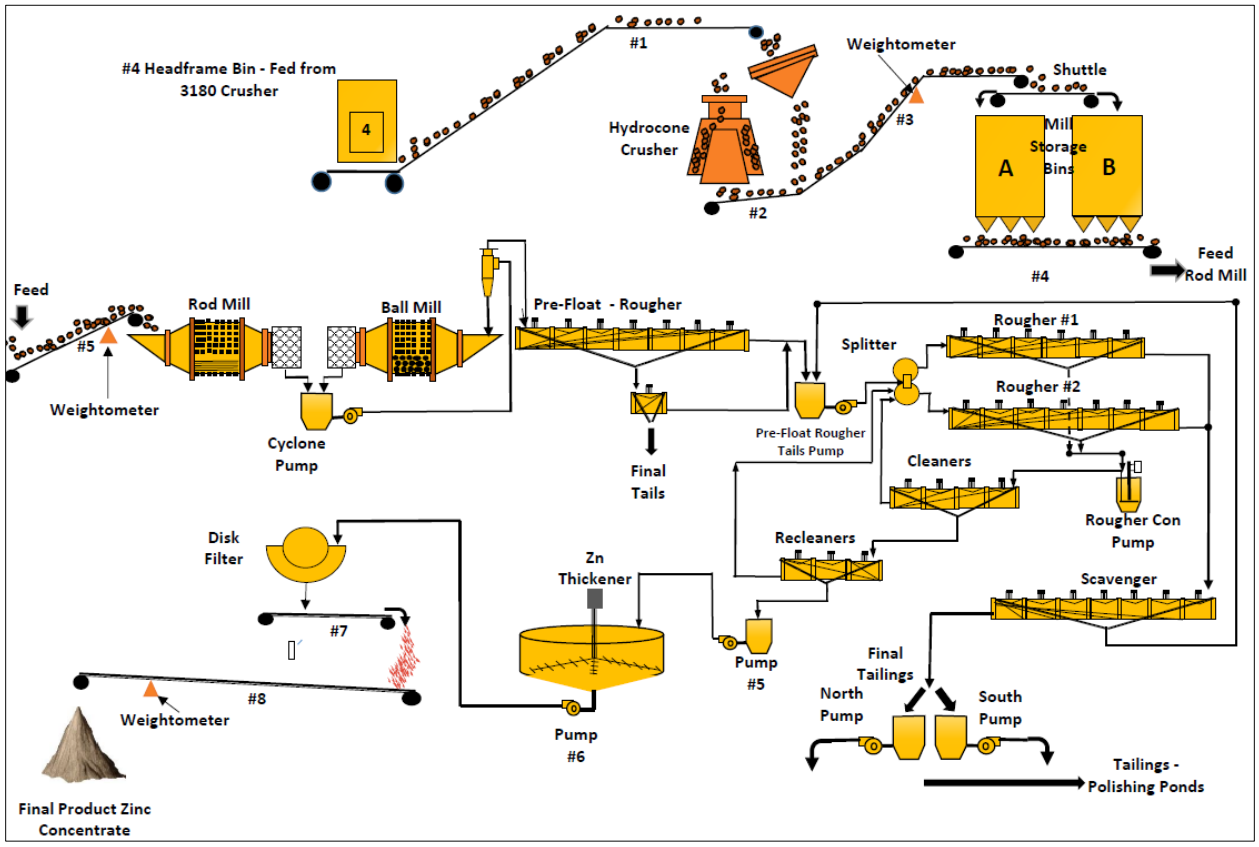
13.1.1 Processing 2018–2024

A test program was undertaken by Hudbay in 2005 (Hudbay, 2005b) to confirm the processing requirements of selected mineralized material zones from ESM. 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 ESM since 1984. SGS Lakefield Research performed site reviews and assisted with the 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: Taylor et al., 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 2.2-3.5% 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 2018–2024

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
2024	410,869	8.7	96.3	60.0	57,334

Source: ESM 2025

13.1.2 Turnpike and Hoist House Metallurgical Testwork

The primary objective of the testwork 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 testwork, which consisted of Bond's Ball Mill Work Index (BWi) and Bond Abrasion Index (Ai) determination and flotation testwork. Reagents, currently employed in the milling circuit at the mine, were also sent for the study.

13.1.2.1 Sample Preparation and Characterization

Half-core samples from Turnpike and Hoist House received for comminution testing were crushed to minus ¾ inch and submitted for Ai testing. The comminution samples were then crushed to P_{100} passing 6 mesh for 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/t Ag compared to 0.36% Pb and 11.7 g/t Ag). Both samples contained trace amounts of gold.

Table 13-2: Head analyses of composite samples including ICP

	Turnpike	Hoist House
Au, g/t	0.022	0.010
Ag, g/t	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 Ai 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 kg 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/t)	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**

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

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/t 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/t)	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**

Product	Recovery %				Product Grade		
	Wt.	Ag	Pb	Zn	Ag (g/t)	Pb (%)	Zn (%)
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/t)	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**
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 testwork:

- 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/t to 450 g/t 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.

13.1.3 Conclusions

ESM has been in continuous production for the past 7 years, during which time the milling performance has been based on actual plant data rather than historic metallurgical testwork. The operational results from 2018 through 2024 encompass the full range of mineralization types currently mined and processed and therefore provide a more representative basis for metallurgical performance than earlier laboratory test programs. These results demonstrate consistent zinc recovery of approximately 96% and concentrate grades near 60%, confirming that the processing response of the deposit is well understood under commercial operating conditions.

Processing factors and deleterious elements are well understood based on 7 years of continuous production data. Operational data indicate that mercury averages about 730 ppm in the zinc concentrate, magnesium averages around 0.5%, and iron averages approximately 3.0%. These levels have been managed effectively within the existing circuit and do not present any material risk to economic extraction under current operating conditions.

13.2 Graphite

13.2.1 Concentrate Plant

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

13.2.1.1 SGS Mineralogical Characterization

Seven drill core samples were received by the SGS Advanced Mineralogy Facility from ESM 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% Cg to 9.53% Cg and total sulfur ranged from 0.39% S to 3.87% S.

The results of the XRD 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

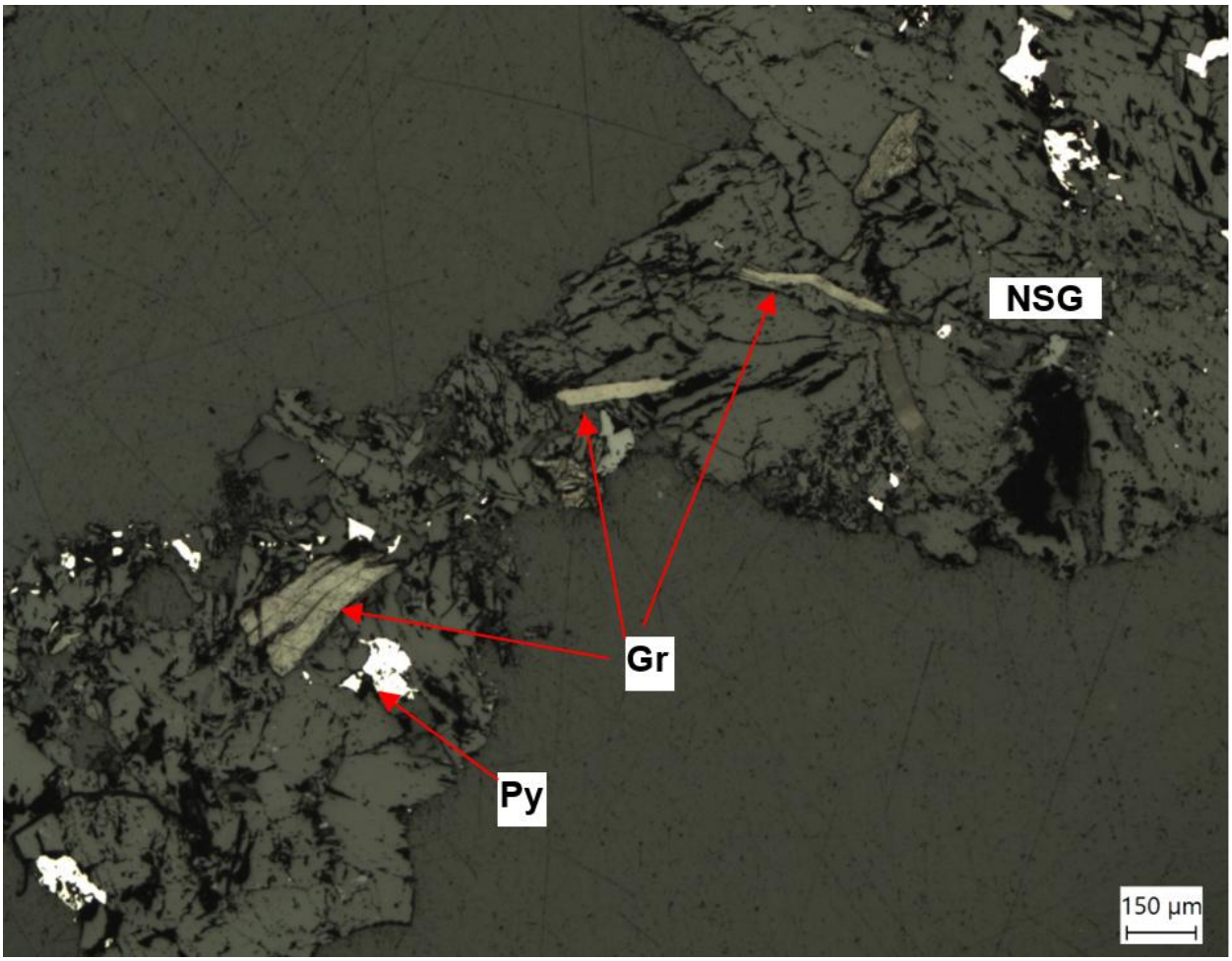
Mineral	F03225	F031913	F031995	F031911	F031518	F032222	F032245
Quartz	2.3	4.8	2.9	18.2	6.0	2.1	8.4
Calcite	2.0	63.0	77.3	17.3	3.3	1.6	46.3
Plagioclase	56.6	1.5	0.2	3.1	30.0	49.9	1.6
Chlorite	6.1	11.1	0.4	17.1	10.8	4.7	2.5
Diopside	7.9	0.9	9.2	1.9	20.1	13.7	25.0
Pyrite	0.6	3.5	1.2	4.8	4.8	0.8	0.4
Meionite	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
Graphite	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

13.2.1.2 SGS Phase I Metallurgical Program

Assay rejects were submitted to SGS Canada in October 2023 to produce two composites for metallurgical testing. The primary objectives of the test program were to assess the metallurgical response of the mineralized material and to produce a graphite concentrate grading at least 95% TC. 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% Cg 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
TC %	1.96
Cg %	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 degraded. For this reason, flash flotation is generally incorporated into the primary grinding circuit. During the first two tests, it was noted that the Kilbourne graphite flakes were relatively small, thus lessening the benefit of a coarse flotation stage. Hence, the remaining three tests eliminated flash flotation and, instead, the entire sample was ground to the final grind size target.

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

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

The three rougher only tests explored different grind sizes, namely P_{80} = 120 µm in test F6, P_{80} = 86 µm in test F5, and P_{80} = 53 µm in test F8. Test F8 with the finest primary grind size produced a rougher concentrate grading 26.6% TC at 97.4% graphite recovery. The two other rougher tests yielded near identical grades and recoveries of 17.7-17.8% TC and 97.2-97.3%, respectively.

Table 13-10: Flash & rougher flotation tests (F3 to F6 and F8)

Test	Product	Weight (%)	Assays % TC, Cg	% Distribution TC
F3 Flash & Rougher Flash P_{80} = 1,850 µm Rougher Tails P_{80} = 167 µm	Flash Conc 1	3.3	25.5	48.0
	Flash Conc 1-2	4.4	22.3	56.3
	Flash Conc 1-2 & Ro Conc 1	11.8	13.6	92.6
	Flash Conc 1-2 & Ro Conc 1-2	14.2	11.8	96.5
	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 & Rougher Flash P_{80} ~ 1,000 µm Rougher Tails P_{80} = 100 µm	Flash 1	7.0	16.9	75.7
	Flash 1 + Flash 2	11.3	12.1	87.3
	Flash 1+2 Ro Conc 1	12.6	11.6	93.3
	Flash 1+2 Ro Conc 1 + 2	14.2	10.6	96.0
	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 Rougher Only 33-minute grind Rougher Tails P_{80} = 86 µm	Ro Conc 1	3.5	38.6	80.6
	Ro Conc 1-2	5.7	27.1	93.1
	Ro Conc 1-3	6.7	23.8	95.2
	Ro Conc 1-4	7.8	20.7	96.3
	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 Rougher Only 21-minute grind Rougher Tails P_{80} = 120 µm	Ro Conc 1	4.9	19.8	58.7
	Ro Conc 1-2	7.3	19.3	86.2
	Ro Conc 1-3	8.2	19.0	94.3
	Ro Conc 1-4	8.4	18.8	96.0
	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 Rougher Only 40-minute grind Rougher Tails P_{80} = 53 µm	Ro Conc 1	3.0	28.4	48.7
	Ro Conc 1-2	5.5	27.4	85.1
	Ro Conc 1-3	6.1	27.4	95.3
	Ro Conc 1-4	6.3	27.1	96.7
	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
Cg				

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 µm in test F7 and P_{80} = 53 µm in test F9. Both tests then reground the rougher concentrate in a polishing mill for 30 minutes 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 (F7 and F9)

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

Source: SGS 2024

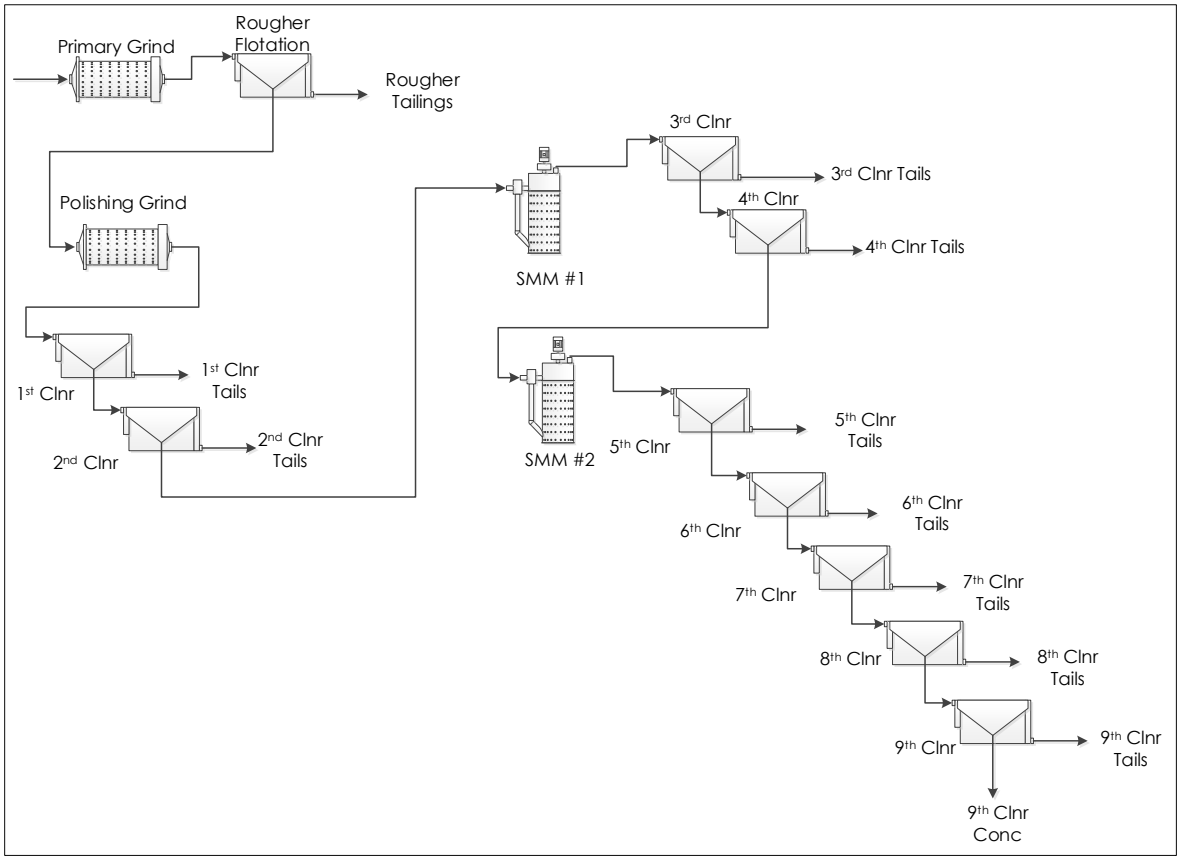
The two tests produced similar 3rd cleaner concentrate grades of 76.6% TC in test F7 and 72.2% TC in test F9. Despite the finer primary grind, test F9 yielded higher rougher tailings losses of 7.2% compared to 3.5% in test F7 with the coarser primary grind. The open circuit total graphite recovery of 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 chose 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% TC at an open circuit graphite recovery of 72.1%. Even the 5th cleaner concentrate still yielded an acceptable concentrate grade of 95.9% TC at a higher open circuit graphite recovery of 86.5%. It should be noted that open circuit tests always understate recoveries since intermediate streams that would circulate in a continuous operation are instead treated as final tailings.

Table 13-12: Results of full cleaner test F10

Product	Weight (%)	Assays % TC, Cg	% Distribution TC
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 contained a very high total carbon content of 97.4% TC.

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

Concentrate Size Fraction	Wt. %	Assays % TC	% Distribution TC
+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

13.2.1.3 Forte Analytical Phase II Metallurgical Program

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

Forte Analytical received approximately 75 kg of two mineralized samples for the Graphite 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% Cg and 2.39% Cg, 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 using -10 mesh (2.0 mm) test charges. 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 µm 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% Cg for Batch 1 and 11.4% TC 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/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 Cg (%)	Rec Cg (%)
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 Analytical, 2024

The flash and rougher concentrates were then 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 cleaning of the flash flotation concentrate recovered 89.3%–94.4% and 96.1%–98.1% of graphite assaying 18.16%–24.16% Cg and 21.75%–21.95% Cg 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% Cg and 27.34% Cg 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% Cg. Four-stage cleaning of the flash flotation concentrate with polishing grinds recovered 90.2% and 90.5% of graphite assaying 37.53% Cg and 34.61% Cg 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% Cg and containing 48.3% of the graphite. Batch 2 responded inferiorly with a grade of only 85.6% Cg and 55.1% of graphite recovery. Note that the stated graphite recovery only considers the flash flotation circuit and that global recovery will increase once the rougher flotation performance is taken into account. The results of the test with six stages of cleaner flotation and three polishing regrinds are presented in Table 13-15

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

Product	Mass (%)	Grade Cg (%)	Stage Rec Cg (%)	Total Rec Cg (%)
Batch 1 - Flash 6th Cleaner 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 Cleaner 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 Analytical, 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% Cg and 42.19% Cg for Batch 1 and 2, respectively.

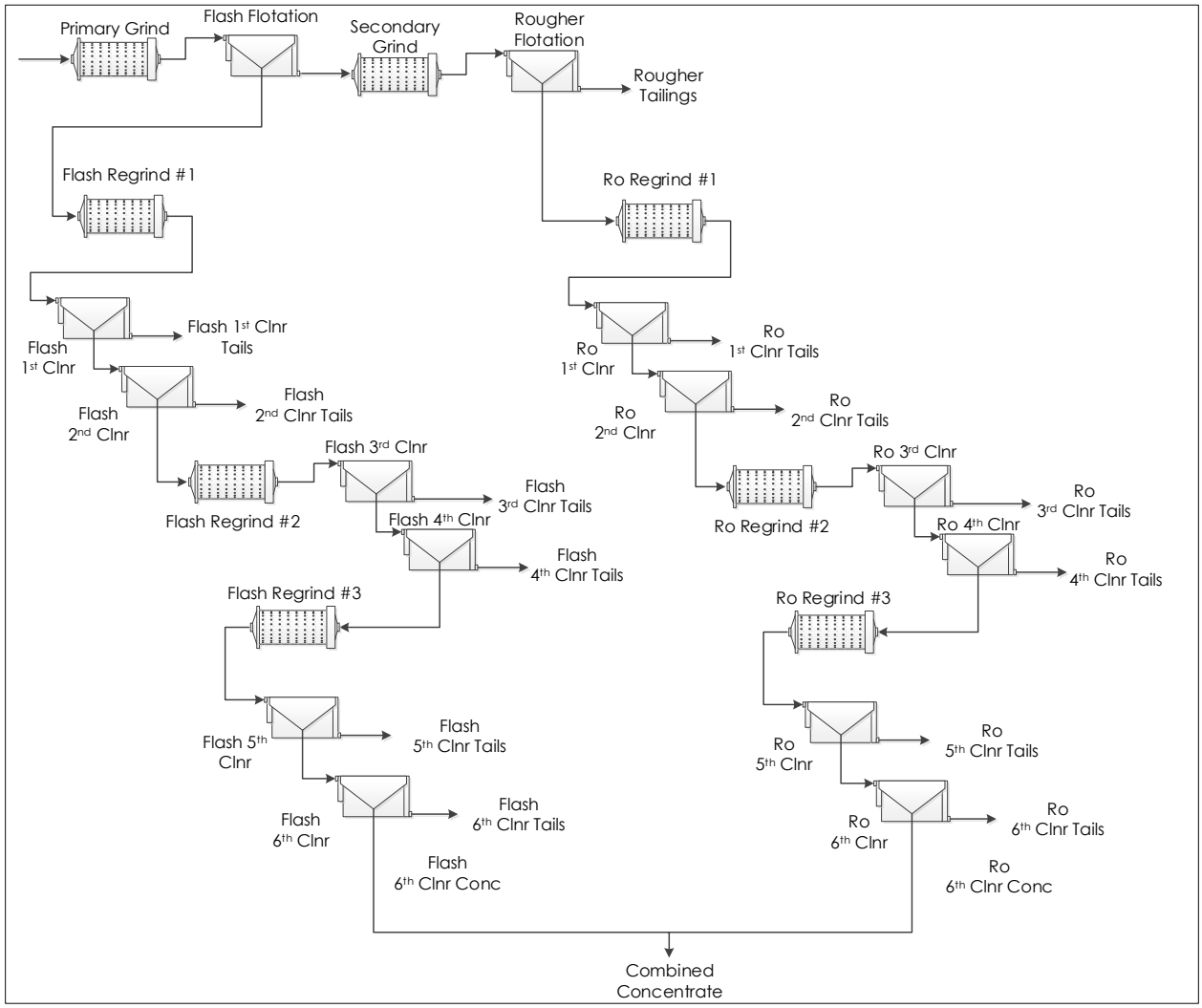
As expected, treating the rougher flotation concentrate with three stages of polishing followed by cleaner flotation produced the best overall results, which are summarized in Table 13-16. Batch 1 produced a 6th cleaner concentrate grading 98.34% Cg and containing 26.8% of the graphite. Batch 2 responded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

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

Product	Mass (%)	Grade Cg (%)	Stage Rec Cg (%)	Total Rec Cg (%)
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 Analytical, 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% Cg. The graphite recovery for Batch 2 was identical at 81.9% but at a lower concentrate grade of 85.9% Cg. The integrated Forte Analytical flowsheet is presented in Figure 13-4.



Source: Metpro 2024

Figure 13-4: Integrated Forte Analytical 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. However, no sizing was performed on the upgraded rougher concentrate and, therefore, an overall concentrate mass distribution cannot be determined from these tests.

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 (% Cg)
+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 Analytical, 2024

13.2.1.4 SGS Phase I Metallurgical Program

In October 2024, a total of 118 assay reject samples were blended to prepare one master composite and four variability composites.

All composites were submitted for total and graphitic carbon analyses by Elemental Combustion analysis (Leco). The master composite was also submitted for inorganic and organic carbon analyses, sulfur and sulfide analyses, whole rock analysis by X-ray Fluorescence (XRF), a multi-element scan by Inductively Coupled Plasma Optical Emission Spectroscopy (ICP-OES) and specific gravity by pycnometer. The results of the carbon speciation analysis are present in Table 13-18. The balance of the analysis results is provided in Table 13-19.

Table 13-18: Carbon speciation of master and variability composites

Element	Assay (%)				
	Master	North Shallow	South Shallow	North Deep	South Deep
Total Carbon (TC)	3.48	3.38	3.51	2.87	3.57
Carbon Graphite (Cg)	3.14	3.21	3.28	2.68	3.25
Total Inorganic Cabon	**0.08**				
Total Organic Carbon	**0.26**				

Table 13-19: Chemical characterization of the master composite

Element	Master Composite (%)	Element	Master Composite (g/t)
S	7.00	Ag	< 0.8
S^{2-}	6.64	As	< 30
SiO_2	60.8	Ba	397
Al_2O_3	9.85	Be	0.72
Fe_2O_3	11.6	Bi	< 10
MgO	1.68	Co	25
CaO	0.72	Cu	95
Na_2O	0.15	Li	34
K_2O	4.27	Mo	25
TiO_2	0.48	Ni	165
P_2O_5	0.34	Pb	85
MnO	0.03	Sb	< 10
Cr_2O_3	0.03	Se	< 30
V_2O_3	0.09	Sn	< 20
LOI	9.77	Sr	32.1
Sum	99.8	Tl	< 30
		Y	33.2
SG	2.84	Zn	999

A series of eight flotation tests was completed on the master composite to optimize the flowsheet and conditions that were developed in the first SGS scoping program. The MF2 (mill-flotation-mill-flotation) flowsheet was applied using fuel oil (Diesel) as the collector and Methyl Isobutyl Carbinol (MIBC) as a frothing agent.

The first three tests evaluated primary grind sizes between P_{80} = 86 µm and P_{80} = 135 µm. The combined flash and rougher concentrates were polished and then upgraded in three stages of cleaner flotation. The pertinent results of the three tests are summarized in Table 13-20. As the primary grind size was coarsened, the graphite losses to the rougher tailings gradually increased from 2.1% to 3.8%. Also, the 3rd cleaner concentrate grade improved from 52.6% TC for the test with the finest grind to 40.0% TC for the test with the coarsest grind. As expected, the 3rd cleaner concentrate size distribution shifted towards a finer product from P_{80} = 101 µm to P_{80} = 95 µm as the primary grind size was reduced. Based on these results, a decision was made to proceed with a primary grind size of approximately P_{80} = 100 µm.

Table 13-20: Primary grind size evaluation

Test	Product	Weight (%)	Assays % TC	% Distribution TC
F1 Rougher P_{80} = 86 µm 3rd Cleaner Conc P_{80} = 95 µm	3rd Clnr Conc	6.1	52.6	95.6
	2nd Clnr Conc	7.5	43.1	96.4
	1st Clnr Conc	11.9	27.5	97.1
	Rougher Conc	30.2	10.9	97.9
	Rougher Tails	69.8	0.10	2.1
	Head (calc.)	100.0	3.36	100.0
F2 Rougher P_{80} = 108 µm 3rd Cleaner Conc P_{80} = 96 µm	3rd Clnr Conc	6.6	45.2	93.3
	2nd Clnr Conc	7.7	38.8	93.5
	1st Clnr Conc	11.1	27.6	95.5
	Rougher Conc	29.4	10.5	96.7
	Rougher Tails	70.6	0.15	3.3
	Head (calc.)	100.0	3.19	100.0
F3 Rougher P_{80} = 135 µm 3rd Cleaner Conc P_{80} = 101 µm	3rd Clnr Conc	7.6	40.0	93.2
	2nd Clnr Conc	9.2	33.3	94.0
	1st Clnr Conc	12.9	23.9	94.6
	Rougher Conc	34.8	8.98	96.2
	Rougher Tails	65.2	0.19	3.8
	Head (calc.)	100.0	3.25	100.0

The following five cleaner flotation tests investigated different regrind technologies, number of regrind stages, and regrind times. Given the finer flake size distribution of the Kilbourne mineralization, only a single stream cleaner flowsheet design was evaluated.

A summary of the flotation results is presented in Table 13-21. The use of only polishing mill failed to produce an acceptable minimum grade of 95.0% TC in the final concentrate and at least two stages of SMM were required to achieve this target. The best test results were achieved with one polishing stage and three stages of SMM, producing a concentrate grading 99.3% TC at an open circuit graphite recovery of 89.0%. The corresponding final concentrate yielded a P_{80} of 106 µm.

The final concentrates were submitted for a size fraction analysis and the mass and grade distribution for the five tests are presented in Figure 13-5 and Figure 13-6, respectively.

Table 13-21: Cleaner optimization tests

Test	Product	Weight (%)	Assays % TC	% Distribution TC
F4 3 x Polish Rougher P_{80} = 96 µm 7th Cleaner Conc P_{80} = 98 µm	7th Clnr Conc	3.4	91.9	93.2
	6th Clnr Conc	3.5	90.0	93.9
	5th Clnr Conc	3.7	84.8	94.5
	4th Clnr Conc	4.0	78.1	94.9
	3rd Clnr Conc	4.7	66.6	95.2
	2nd Clnr Conc	7.0	45.0	95.5
	1st Clnr Conc	10.3	31.0	95.9
	Rougher Conc	25.0	12.8	96.8
	Rougher Tails	75.0	0.14	3.2
	Head (calc.)	100.0	3.32	100.0
F5 2 x Polish & 1 x SMM Rougher P_{80} = 103 µm 7th Cleaner Conc P_{80} = 108 µm	7th Clnr Conc	3.4	93.3	92.9
	6th Clnr Conc	3.5	91.1	93.4
	5th Clnr Conc	3.7	85.9	94.0
	4th Clnr Conc	4.0	79.1	94.4
	3rd Clnr Conc	4.7	67.4	94.6
	2nd Clnr Conc	7.0	45.5	94.8
	1st Clnr Conc	10.3	31.4	95.4
	Rougher Conc	25.0	13.1	96.7
	Rougher Tails	75.0	0.15	3.3
	Head (calc.)	100.0	3.38	100.0
F6 1x Polish & 2 x SMM Rougher P_{80} = 107 µm 7th Cleaner Conc P_{80} = 96 µm	7th Clnr Conc	3.1	95.7	90.5
	6th Clnr Conc	3.2	93.3	91.4
	5th Clnr Conc	3.5	87.6	93.1
	4th Clnr Conc	4.1	75.5	93.9
	3rd Clnr Conc	5.0	61.8	94.5
	2nd Clnr Conc	7.5	42.1	94.9
	1st Clnr Conc	11.0	28.6	95.5
	Rougher Conc	28.4	11.2	96.7
	Rougher Tails	71.6	0.15	3.3
	Head (calc.)	100.0	3.30	100.0

Test	Product	Weight (%)	Assays % TC	% Distribution TC
F7 1x Polish & 2 x SMM & Longer Regrind Times Rougher P_{80} = 101 µm 7th Cleaner Conc P_{80} = 86 µm	7th Clnr Conc	3.1	94.4	91.3
	6th Clnr Conc	3.2	91.8	92.2
	5th Clnr Conc	3.5	86.7	93.0
	4th Clnr Conc	3.9	76.7	93.5
	3rd Clnr Conc	4.8	63.6	94.9
	2nd Clnr Conc	6.9	44.7	95.7
	1st Clnr Conc	10.4	29.8	96.1
	Rougher Conc	24.2	13.0	96.9
	Rougher Tails	75.8	0.13	3.1
	Head (calc.)	100.0	3.23	100.0
F8 1x Polish & 3 x SMM Rougher P_{80} = 95 µm 7th Cleaner Conc P_{80} = 106 µm	9th Clnr Conc	2.9	99.3	89.0
	8th Clnr Conc	3.0	99.1	90.3
	7th Clnr Conc	3.0	98.2	91.4
	6th Clnr Conc	3.1	95.8	92.0
	5th Clnr Conc	3.4	90.3	92.7
	4th Clnr Conc	3.7	81.7	93.2
	3rd Clnr Conc	4.5	69.5	94.4
	2nd Clnr Conc	6.5	47.8	95.5
	1st Clnr Conc	9.0	35.0	96.0
	Rougher Conc	24.0	13.2	97.0
	Rougher Tails	76.0	0.13	3.0
	Head (calc.)	100.0	3.28	100.0

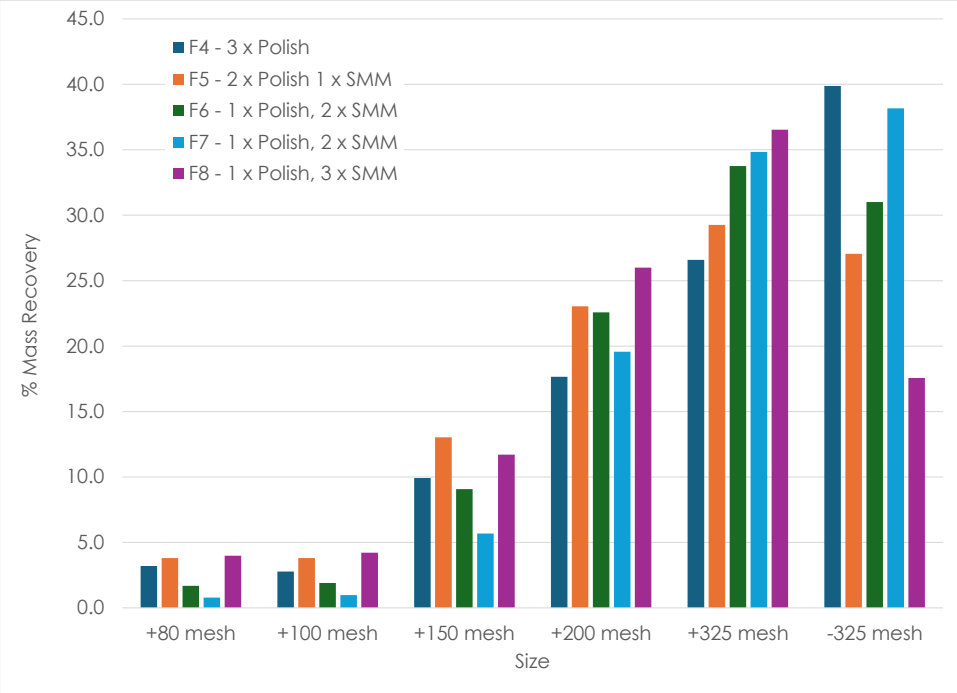


Figure 13-5: Concentrate size fraction analysis – Mass distribution

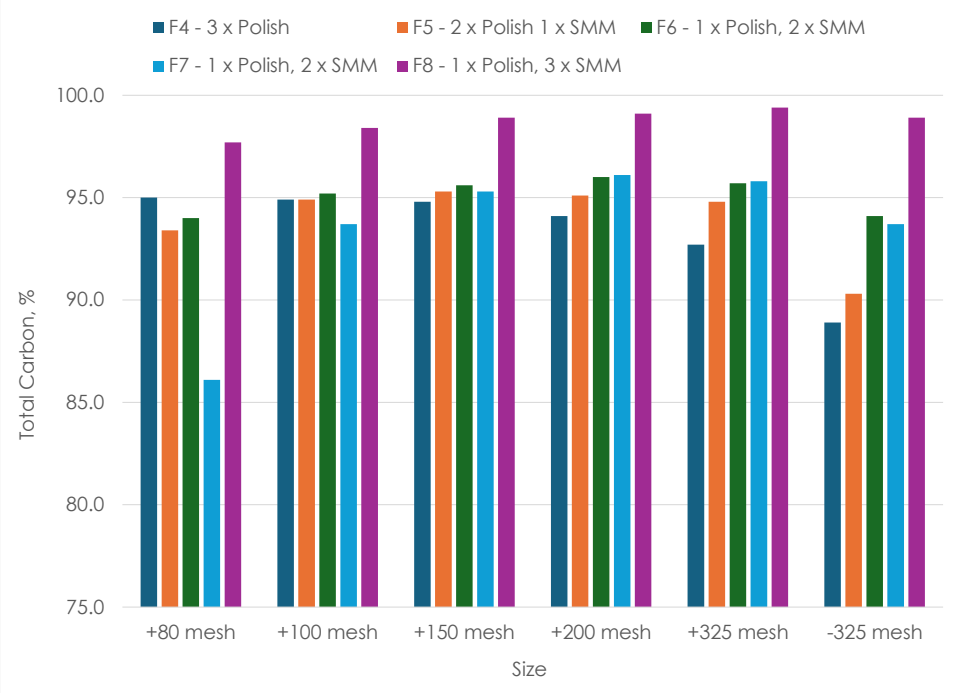


Figure 13-6: Concentrate size fraction analysis – Grade distribution

The size fraction analysis results illustrate that a circuit with only polishing mills struggle to achieve acceptable total carbon grades in the smaller size fractions. Only the circuit with three stages of SMM produced total carbon grades of over 95% TC in all size fractions. Hence, the flowsheet and conditions of test F8 with one polishing stage and three stages of SMM were adopted for the variability flotation tests.

The four variability composites represented different locations and depths of the deposit, namely North Shallow, North Deep, South Shallow, and South Deep. The results of the four cleaner tests are summarized in Table 13-22. All four composites responded well to the flowsheet and conditions, producing final concentrate grades between 97.6% TC for the North Shallow composite and 99.3% TC for the South Deep composite. The open circuit graphite recoveries were slightly lower for the Shallow composites (85.6% and 88.9%) compared to the North composites (94.6% and 93.3%).

It appears that the shallow mineralization is slightly softer compared to the deep material since P_{80} of the rougher tailings of the shallow samples was finer compared to the deep samples using the same primary grind times. Also, the final concentrate P_{80} value was on average 12 µm higher in the deep samples.

The combined concentrates were submitted for a size fraction analysis. The mass recovery into the different size fractions and their corresponding total carbon grades are presented in Figure 13-7 and Figure 13-8, respectively. The mass distribution chart clearly shows the increased mass recovery into the +80 mesh and +100 mesh size fractions for the deep composites. The two deep composites also contained the lowest mass in -400 mesh fines.

In terms of concentrate grades, the North Shallow composite produced consistently lower total carbon grades compared to the other three variability samples, while the two deep composites outperformed the South Shallow composite for most size fractions.

Table 13-22: Variability flotation tests

Test	Product	Weight (%)	Assays % TC	% Distribution TC
VAR-1 North Shallow Rougher Tails P_{80} = 95 µm 9th Cleaner Conc P_{80} = 91 µm	9th Clnr Conc	2.9	97.6	85.6
	8th Clnr Conc	2.9	96.9	87.1
	7th Clnr Conc	3.1	95.1	89.7
	6th Clnr Conc	3.3	91.4	91.8
	5th Clnr Conc	3.6	84.6	93.0
	4th Clnr Conc	4.3	71.7	94.3
	3rd Clnr Conc	5.3	58.5	95.4
	2nd Clnr Conc	8.0	39.2	96.6
	1st Clnr Conc	11.1	28.5	97.0
	Rougher Conc	27.5	11.6	98.0
	Rougher Tails	72.5	0.09	2.0
	Head (calc.)	100.0	3.26	100.0
VAR-2 South Shallow Rougher Tails P_{80} = 96 µm 9th Cleaner Conc P_{80} = 97 µm	9th Clnr Conc	3.0	98.5	88.9
	8th Clnr Conc	3.1	98.2	89.8
	7th Clnr Conc	3.1	97.3	90.8
	6th Clnr Conc	3.3	94.6	92.5
	5th Clnr Conc	3.5	89.9	93.0
	4th Clnr Conc	3.9	80.5	93.5
	3rd Clnr Conc	4.7	67.1	94.2
	2nd Clnr Conc	7.1	44.9	95.0
	1st Clnr Conc	10.4	30.9	95.5
	Rougher Conc	25.6	12.6	96.4
	Rougher Tails	74.4	0.16	3.6
	Head (calc.)	100.0	3.35	100.0

Test	Product	Weight (%)	Assays % TC	% Distribution TC
VAR-3 North Deep Rougher Tails P_{80} = 107 µm 9th Cleaner Conc P_{80} = 108 µm	9th Clnr Conc	2.6	98.8	94.6
	8th Clnr Conc	2.6	97.5	94.6
	7th Clnr Conc	2.7	96.0	94.6
	6th Clnr Conc	2.7	94.3	94.6
	5th Clnr Conc	2.8	92.2	95.2
	4th Clnr Conc	3.0	85.9	95.6
	3rd Clnr Conc	3.3	78.5	96.0
	2nd Clnr Conc	4.2	61.5	96.4
	1st Clnr Conc	6.2	42.3	96.6
	Rougher Conc	19.3	13.6	97.0
	Rougher Tails	80.7	0.10	3.0
	Head (calc.)	100.0	2.71	100.0
VAR-4 South Deep Rougher Tails P_{80} = 104 µm 9th Cleaner Conc P_{80} = 105 µm	9th Clnr Conc	3.2	99.3	93.3
	8th Clnr Conc	3.2	99.2	94.0
	7th Clnr Conc	3.2	98.7	94.9
	6th Clnr Conc	3.3	97.0	94.9
	5th Clnr Conc	3.4	95.4	95.4
	4th Clnr Conc	3.6	89.7	96.0
	3rd Clnr Conc	4.1	79.2	96.4
	2nd Clnr Conc	5.0	64.6	96.7
	1st Clnr Conc	6.3	52.1	97.0
	Rougher Conc	21.5	15.3	97.7
	Rougher Tails	78.5	0.10	2.3
	Head (calc.)	100.0	3.37	100.0

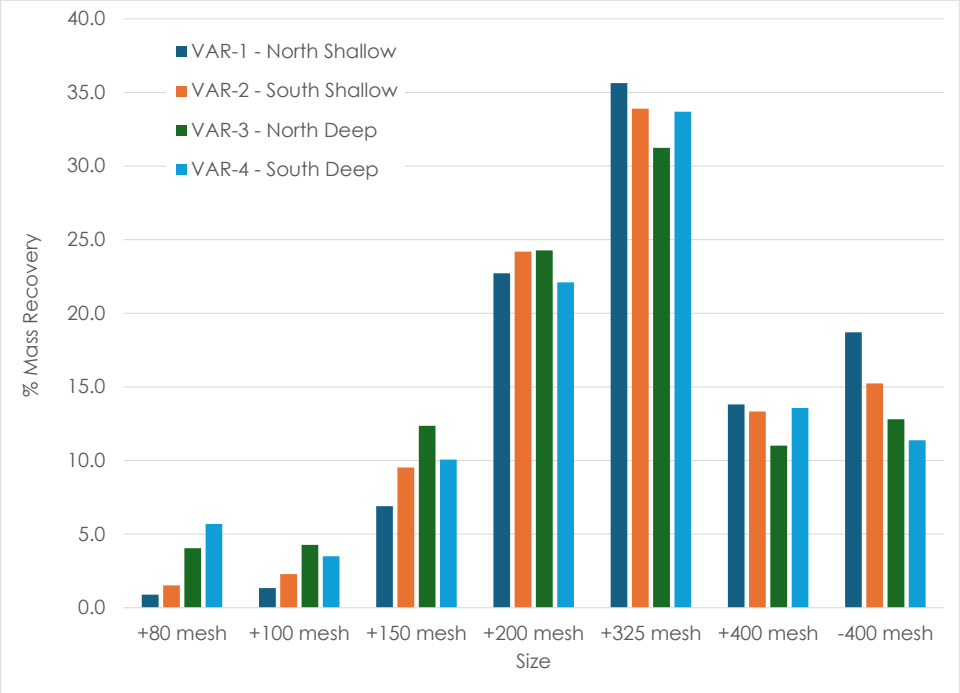


Figure 13-7: VAR Concentrate size fraction analysis – Mass distribution

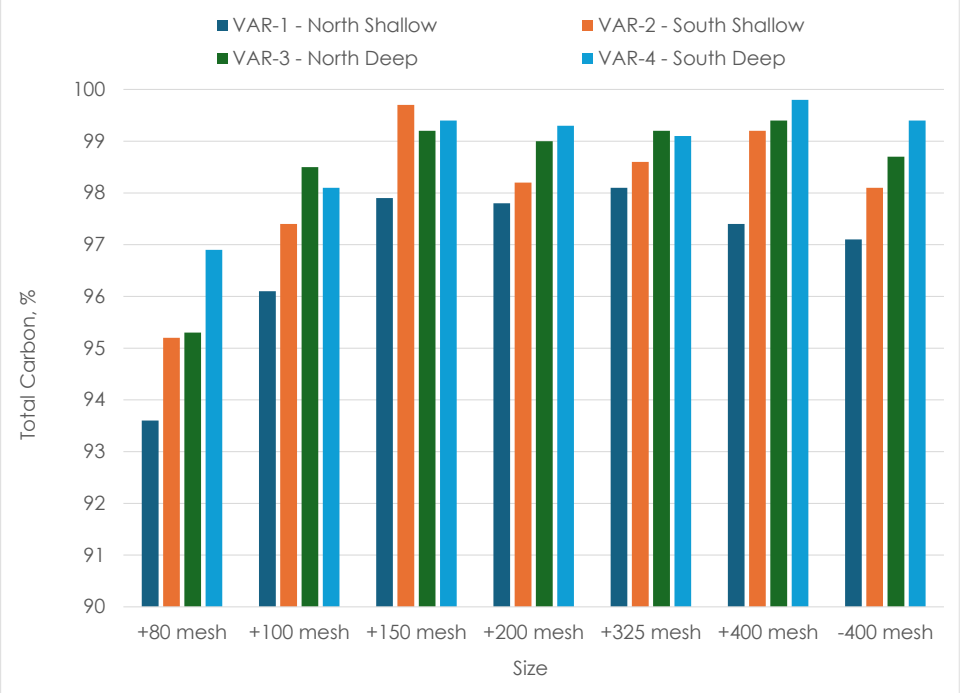


Figure 13-8: VAR Concentrate size fraction analysis – Grade distribution

The average flake size distribution of the master composite and the four variability composites is shown in Table 13-23.

Table 13-23: Average flake size distribution

Size		% Mass
Microns	Mesh	
+150	+100	6.3
+74	+200	34.0
-74	-200	59.7

The final concentrates of the master composite test F8 and the four variability flotation tests were combined and submitted for a detailed concentrate analysis. The results are presented in Table 13-24. The most abundant impurities included silica, aluminum, and iron.

Table 13-24: Detailed concentrate analysis

Sample ID	Ash	Elements (g/t)												TC
	(%)	Si	Al	Fe	Mg	Ca	Na	K	Ti	P	Mn	Cr	V	(%)
+100 mesh Conc	2.80	7,050	2,330	3,170	797	237	103	970	61.8	33	14.4	31	29	94.3
+150 mesh Conc	2.13	5,440	2,040	2,170	618	83	81	729	17.4	< 20	14.2	35	19	97.7
+200 mesh Conc	1.80	4,240	1,900	2,140	575	75	31	708	13.4	< 20	9.6	46	18	97.9
-200 mesh Conc	2.00	4,440	2,180	2,520	579	104	33	848	14.7	< 20	12	74	20	98.3

A graphite rougher tailings sample and a weighted composite of the tailings from the existing Zinc Operation and the graphite rougher tailings were submitted for net acid generation (NAG) and modified acid base accounting (ABA) tests.

The graphite tailings in isolation produced a net neutralizing potential of -216 t $CaCO_3$ per 1,000 tonnes of tailings and a net acid generation potential of 108 kg H_2SO_4 per tonne of tailings at a pH of 7.0. In contrast, the combined graphite and zinc tailings yielded significantly improved results with a net neutralizing potential of -16.8 t $CaCO_3$ per 1,000 tonnes of tailings and a net acid generation potential of 2.4 kg H_2SO_4 per tonne of tailings at a pH of 7.0 due to the neutralizing properties of the zinc tailings. Further investigation is required to characterize the potential tailings stream produced by the combined operation.

13.2.1.5 Conclusions

Three 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 that the flake size distribution in the Kilbourne mineralization is relatively fine. The SGS program showed that a graphite concentrate grading well over 95% TC can be generated with a relatively simple process.

The proposed Forte Analytical 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 programs have shown that high purities can be achieved even for the small size fractions. The ability to produce high-grade fines differentiates the Kilbourne mineralization from many other graphite projects and may present excellent marketing opportunities.

Since no clear benefit of two separate cleaning circuits is apparent, future testing will focus on the optimization 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.

The various composites that were evaluated in the three test programs produced a consistent metallurgical response. The variability tests provided a preliminary confirmation that the proposed flowsheet and conditions are suitable to produce a high-grade graphite concentrate using mineralized samples from various areas of the deposit.

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

The second SGS program culminated in the flowsheet and conditions that were used for the engineering design. The composites that were generated for the program included a total of 118 drill core intervals from two different drillholes. One drillhole was on the north side of the resource, and the second drillhole was on the south side. Both drillholes intersected both the upper and lower bands of the mineralization. Based on current knowledge, the Kilbourne mineralization appears to be reasonably homogenous and, therefore, the representativeness of the selected samples for metallurgical testing is considered sufficient for a PEA level study.

Preliminary concentrate characterization did not identify any generic deleterious elements although concentration of individual elements may limit the concentrate suitability for specific graphite applications. This is consistent with other graphite deposits since each graphite concentrate has a unique signature that may impact its suitability for specific applications.

13.2.2 Micronization and Purification

Blue Coast Research on Vancouver Island processed approximately 100 kg of a Kilbourne bulk sample to produce an intermediate concentrate. This intermediated concentrate was then upgraded by SGS Québec to generate 1.7 kg of graphite concentrate grading at least 95% TC.

The bulk flotation concentrate was shipped to ProGraphite in Germany in July 2025 to perform a comprehensive characterization of the product. The combined concentrate was submitted for a size analysis before it was screened into four separate size fractions, namely +150 µm, -150 µm, -150/+71 µm and -71 µm. The sizing results for the as-received concentrate are shown in Table 13-25. The amount of -40-micron material was noticeably higher than in the master composite. It is concluded that this was the result of additional regrind stages due to non-optimized scale-up at Blue Coast Research and SGS Québec, which led to higher flake degradation.

Table 13-25: Concentrate size analysis

Screening	% Retained
180 µm	2.9
150 µm	3.3
100 µm	13
71 µm	16.9
40 µm	32.1
-40 µm	31.9

The combined concentrate and the four size fractions were submitted for a comprehensive characterization and the results are summarized in Table 13-26. The combined concentrate grade was 95.0% LOI and gradually decreased from 96.8% LOI in the +150-micron product to 93.6% LOI in the -71-micron product. Most of the results were comparable with data from other graphite projects. The tap and bulk density fell into the medium to low range.

Table 13-26: Concentrate characterization

Grade	Unit	Concentrate	+150 µm	-150 µm	-150 +71 µm	-71 µm
Sample-ID	-	**S#3633**	**S#3634**	**S#3635**	**S#3652**	**S#3636**
Carbon Content (LOI) and Ash						
Loss of Ignition (LOI)	%	95.0	96.8	94.8	96.4	93.6
Ash	%	5.0	3.2	5.2	3.6	6.4
Mass Distribution after Screening						
Proportion	%	100.0	6.2	93.8	29.9	63.9
Particle Size Distribution by Laser Diffraction						
d10	µm	17	-	16	17	15
d50	µm	54	-	50	64	41
d90	µm	110	-	101	125	76
Density						
Bulk Density	g/L	368	558	358	-	-
Tap Density	kg/L	0.50	-	0.51	-	-
Specific Surface Area (BET)	m^2/g	-	1.3	2.4	1.75	3.1
Springback (@ 0,43 MPa)	%	-	10.1%	12.3%	-	-
Electrical Conductivity						
@1.56 MPa	S/cm	-	120	82	-	-
@3.12 MPa	S/cm	-	119	80	-	-
pH Value	-	6.5	6.4	6.6	6.7	6.4

The results of the thermogravimetric and crystallographic analysis are presented in Table 13-27. The volatiles ranged between 0.6% for the +150 micron size fraction and 0.9% for the -71 micron product. The resulting minimum and maximum fixed carbon content was 92.7% FC and 96.2% FC. The oxidation rate was somewhat elevated while the degree of graphitization is considered average.

Table 13-27: Concentrate thermogravimetry and crystallography

Grade	Unit	Concentrate	+150 µm	-150 µm	-150 +71 µm	-71 µm
Sample-ID	-	**S#3633**	**S#3634**	**S#3635**	**S#3652**	**S#3636**
Thermogravimetry (TGA)						
Moisture (105 °C)	%	0.09	0.05	0.12	0.07	0.12
Volatiles (400 °C)	%	0.8	0.6	0.8	0.8	0.9
LOI (950 °C)	%	95.0	96.8	94.8	96.4	93.6
Fixed Carbon	%	94.2	96.2	94.0	95.6	92.7
Oxidation at 650 °C						
Weight Loss in 120 Minutes	%	18.3	7.9	18.7	11.9	24.8
Crystallographic Data via XRD						
Degree of Graphitization	%	-	97	96	-	-
d002 Spacing	nm	-	0.33566	0.33575	-	-
Lc	nm	-	173.9	218.6	-	-
Lattice Deformation (c-direction)	%	-	0.176	0.151	-	-
La	nm	-	221.9	272.5	-	-

The as-received flotation concentrate and the four size fractions were submitted for an impurity analysis using X-ray fluorescence analysis. The results are presented in Table 13-28 and reveal aluminium, iron, potassium, silica, and zinc as the most abundant impurities. Most impurities are consistent with other projects, but potassium, magnesium, and zinc levels are considered elevated. The higher zinc concentration may be a function of the proximity of the zinc sulphide resource.

Table 13-28: X-ray fluorescence analysis

Grade	Unit	Concentrate	+150 µm	-150 µm	-150 +71 µm	-71 µm
Sample-ID	-	**S#3633**	**S#3634**	**S#3635**	**S#3652**	**S#3636**
XRF						
Tablet- ID	-	T#154	T#157	T#163	T#158	T#159
Al	ppm	4,690	2,858	4,762	3,651	5,454
Ca	ppm	227	99	241	117	356
Cl	ppm	120	N/A	234	226	403
Cr	ppm	25	14	17	15	35
Cu	ppm	82	32	85	59	145
Fe	ppm	5,844	4,563	5,946	5,252	6,704

Grade	Unit	Concentrate	+150 µm	-150 µm	-150 +71 µm	-71 µm
Sample-ID	-	**S#3633**	**S#3634**	**S#3635**	**S#3652**	**S#3636**
K	ppm	2,330	936	2,382	1,363	3,383
Mg	ppm	1,348	1,056	1,371	1,220	1,438
Mn	ppm	44	23	37	33	45
Mo	ppm	92	28	101	49	144
Ni	ppm	31	31	26	25	33
P	ppm	25	15	31	18	32
Pb	ppm	92	70	108	71	111
S	ppm	1,333	566	1,379	746	1,802
Si	ppm	7,064	4,087	7,258	5,074	8,668
Ti	ppm	N/A	21	20	N/A	32
V	ppm	43	43	61	55	84
Zn	ppm	2,099	481	2,230	683	3,575
Zr	ppm	61	64	62	45	76

Micronizing was completed using a continuous impact mill with an internal classifier. The objective was to produce two micronized graphite specifications, namely D90 = 45 µm and D90 = 15 µm. The results of the micronization tests are presented in Table 13-29 and show that the achieved particle sizes were very close to target.

Table 13-29: Micronization results

Test	Unit	PMG D90-45 Micron (pre-purification)	PMG D90-15 Micron (pre-purification)
Sample-ID	**-**	**S#3664**	**S#3665**
Particle Size Distribution by Laser Diffraction			
d10	µm	6.1	2.9
d50	µm	20.4	7.8
d90	µm	47.2	16.7

The micronized samples were purified using hydrofluoric acid (HF) with the objective of reaching a minimum carbon content of 99.99% LOI. The hydrofluoric acid route was selected since it is considered the industry standard and almost all graphite purification conducted on a commercial scale is performed using HF. The purified products were submitted for a XRF analysis, and the results are compared with the pre-purification values in Table 13-30. The results show that the purification worked very well for both micronized products and levels of remaining impurities were low in both samples, especially when considering that only a single purification test could be performed on each size fraction due to limited sample availability.

Table 13-30: XRF analysis of pre- and post-purification micronized products

	Unit	PMG D90-45 Micron (pre-purification)	PMG D90-15 Micron (pre-purification)	PMG D90-45 Micron (post-purification)	PMG D90-15 Micron (post-purification)
Sample-ID	**-**	**S#3664**	**S#3665**	**S#3691**	**S#3692**
XRF Results - Oxides					
SiO_2	%	1.696	1.736	-	-
Al_2O_3	%	1.008	1.016	-	-
Fe_2O_3	%	0.82	0.984	-	-
MgO	%	0.274	0.295	-	-
K_2O	%	0.258	0.225	-	-
Na_2O	ppm	437	512	-	-
CaO	ppm	338	341	-	-
TiO_2	ppm	83	82	-	-
P_2O_5	ppm	55	64	-	-
MnO	ppm	61	62	-	-
S	ppm	1,513	1,673	-	-
LOI	**%**	**94.84**	**95.02**	**-**	**-**
XRF Results - Elements					
Al	ppm	-	-	32.0	29.7
As	ppm	-	-	n.d.	n.d.
Ca	ppm	-	-	3.4	6.7
Co	ppm	-	-	0.2	0.8
Cr	ppm	-	-	1.1	1.0
Cu	ppm	-	-	2.3	2.6
Fe	ppm	-	-	26.0	18.9
Mg	ppm	-	-	8.8	10.7
Mo	ppm	-	-	0.9	0.7
Na	ppm	-	-	-1*	-1*

	Unit	PMG D90-45 Micron (pre-purification)	PMG D90-15 Micron (pre-purification)	PMG D90-45 Micron (post-purification)	PMG D90-15 Micron (post-purification)
Sample-ID	**-**	**S#3664**	**S#3665**	**S#3691**	**S#3692**
S	ppm	-	-	-1*	-1*
Si	ppm	-	-	23.0	37.0
Ti	ppm	-	-	14.5	10.2
V	ppm	-	-	n.d.	0.5
Zn	ppm	-	-	1.8	1.4
Zr	ppm	-	-	17.3	15.7
LOI	**%**	**-**	**-**	**99.97**	**99.95**

* -1: no valid data obtained

The micronized and purified concentrates were also submitted for a comprehensive characterization, which is presented in Table 13-31. The loss in ignition values of the coarser and fine micronized material yielded 99.97% LOI and 99.95% LOI, respectively, and were therefore well above the minimum target of 99.9% LOI.

Table 13-31: Characterization of pre- and post-purified micronized graphite

Test	Unit	PMG D90-45 Micron (pre-purification)	PMG D90-15 Micron (pre-purification)	PMG D90-45 Micron (post-purification)	PMG D90-15 Micron (post-purification)
Sample-ID	**-**	**S#3664**	**S#3665**	**S#3691**	**S#3692**
Particle Size Distribution via Laser Diffraction					
d10	µm	6.1	2.9	6.7	3.4
d50	µm	20.4	7.8	20.7	8.4
d90	µm	47.2	16.7	47.0	17.5
Density					
Bulk Density	g/cm³	251.8	191.7	292.3	276.1
Tap Density	g/cm³	0.40	0.35	0.43	0.44
Spring Back (Post Tap)	%	12.00	13.15	6.42	6.41
pH value	-	6.32	6.21	6.3	6.0
Specific Surface Area (BET)	m²/g	5.5	6.1	8.1	9.4

Test	Unit	PMG D90-45 Micron (pre-purification)	PMG D90-15 Micron (pre-purification)	PMG D90-45 Micron (post-purification)	PMG D90-15 Micron (post-purification)
Sample-ID	**-**	**S#3664**	**S#3665**	**S#3691**	**S#3692**
Thermogravimetry (TGA)					
Moisture	%	0.15	0.18	0.04	0.10
Loss of Ignition	%	94.84	95.02	99.97	99.95
Ash	%	5.16	4.98	0.03	0.05
Oxidation Threshold					
Weight Loss % @ 450C Holding Time 4h	%	0.6	1.1	0.5	0.6
Weight Loss % @ 650C Holding Time 4h	%	68.0	81.0	67.1	71.5

A final test explored the spheroidization of the micronized material prior to purification. Only a single test was completed due to the limited sample availability. The test was performed in a special batch spheroidization mill and the yield was approximately 50%. While the yield is within a normal range, it may be improved with additional optimization. The spherical graphite was then subjected to purification using HF to produce spherical purified graphite (SPG). The tap density of the pre- and post purified spherical graphite was high at 0.97 kg/L and 0.96 kg/L, respectively. The specific surface area (BET) yielded a good 6.5 m^2/g to 6.6 m^2/g. The ratio of D90 to D10 was approximately 3 and could be reduced further with optimization. This should also lead to even higher tap densities.

The results of the pre- and post-purification XRF analysis are presented in Table 13-32. The loss on ignition value increased to a very high 99.99% after purification and the concentration of the various impurities were consistently lower compared to the purified micronized graphite.

Table 13-32: Characterization of pre- and post-purified spherical graphite

	Unit	uSPG 18 (pre-purification)	SPG 18 99.95 (post-purification)
Sample-ID	-	**S#3666**	**S#3702**
LOI	%	95.40	99.99
XRF Results - Elements			
Al	ppm	3,520	13.3
As	ppm	n.d.	n.d.
Ca	ppm	270	2.2
Co	ppm	n.d.	0.3
Cr	ppm	36	0.7
Cu	ppm	61	0.9
Fe	ppm	3,613	11.4
Mg	ppm	812	2.8
Mo	ppm	23	1.3
Na	ppm	177	6.0
S	ppm	778	-1*
Si	ppm	5,837	11.3
Ti	ppm	38	10.1
V	ppm	46	0.3
Zn	ppm	1,358	1.1
Zr	ppm	37	9.9

* -1: no valid data obtained

The test program conducted by ProGraphite has demonstrated that the flotation concentrate produced from the Kilbourne mineralized material is suitable to produce micronized, on-spec products. It has also been demonstrated that the material can be purified to very high purities even though no process optimization was completed due to the limited sample mass of 1.7 kg. Furthermore, it was shown that a good spherical graphite can be produced and that the Kilbourne graphite would qualify for use as battery anode material based on the various analyses.

14. Mineral Resource Estimates

This Item of the report describes updates of the geologic and grade block models for the ESM deposits. Item 14.1 updates the zinc resources following additional drilling and mining exposure. Item 14.2 provides the resource estimate for the Kilbourne Graphite Project.

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 composited drillhole samples, 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

The ESM zinc deposits are comprised of multiple zones in and around Fowler, NY and include the following: American, Cal Marble, Fowler, Mahler, Mud Pond, Number 2 Deep (N2D), Northeast Fowler, New Fold, Sylvia Lake, and Turnpike. Site convention splits the Mahler, Mud Pond, and Number 2 (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 known as N2 Pits in prior technical documents.

This Mineral Resource report update has been supervised by Donald Taylor. All geological modeling and grade estimation since 2020 used Leapfrog Geo and Edge software. The models updated for this report used Leapfrog Geo™ version 2024.1.3 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 the 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 the ESM by Geospark. The database is 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 on June 9, 2025 for the annual in-house resource updates and included collar, downhole survey, lithology, assay, and density data. Assays and associated composites were extracted from 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 the 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.

A total of 122 records were flagged in the database as "Do Not Export" due to insufficient confidence for use in grade estimation or geological interpretation. These include drillholes with uncertain collar locations, abandoned and twinned holes that contribute minimal additional data, and channel samples with questionable sampling practices.

The extracted drillhole database consisted of 8,381 surface or UG core holes. There are 89 sets of channel samples, 1,304 surface core holes, 7,077 UG core holes and 225 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	229
Upper Mahler	113
Mud Pond - Main	135
Mud Pond - Apron	122
N2D	209
New Fold	122
Northeast Fowler	24
Sylvia Lake	98
Turnpike	254

Source: Modified from Warren et al., 2021

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 $tons/ft^3$.

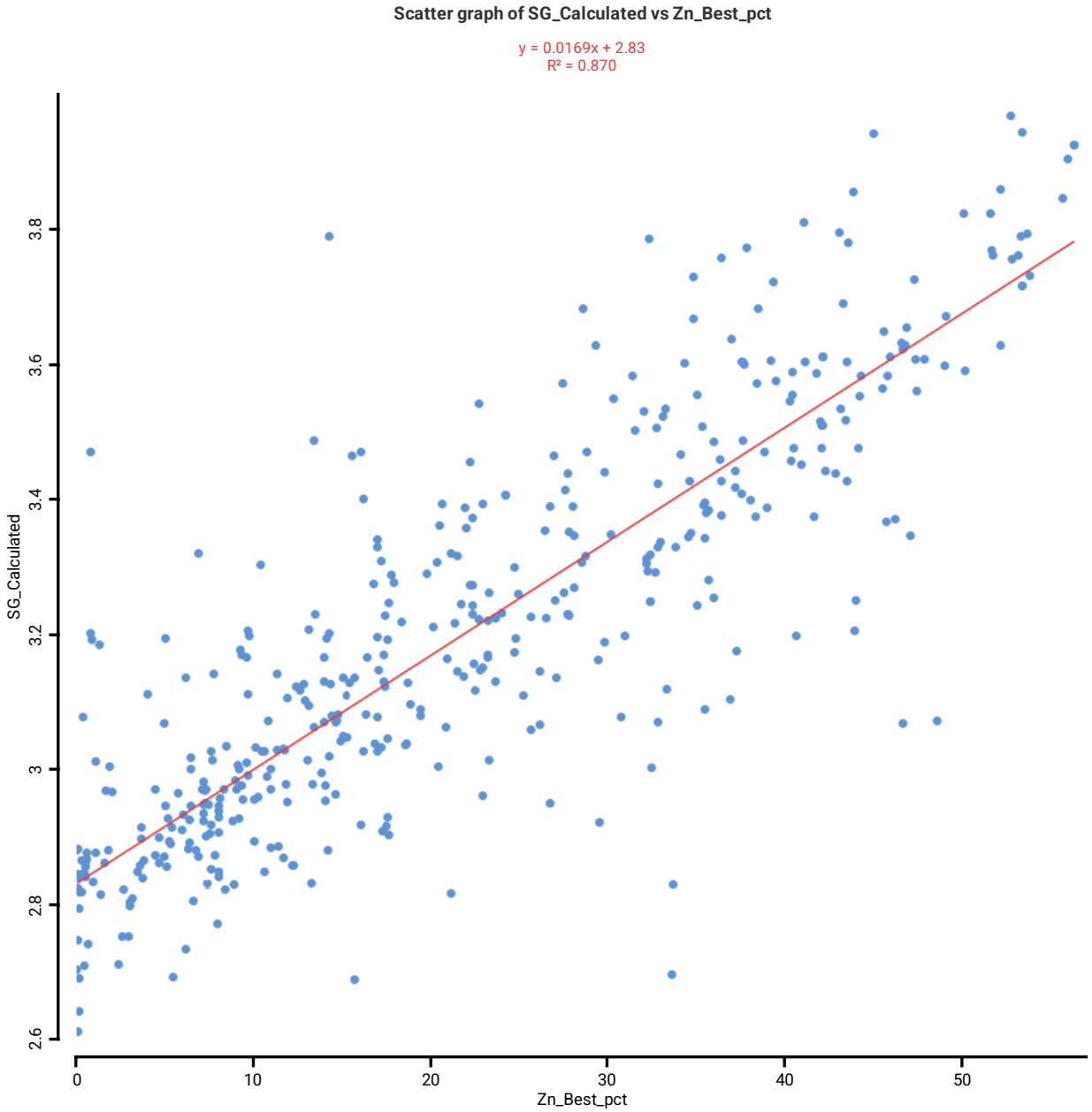


Figure 14-1: Scatterplot of specific gravity vs assay zinc (%) for Mahler

Table 14-2: Density by zone and material type

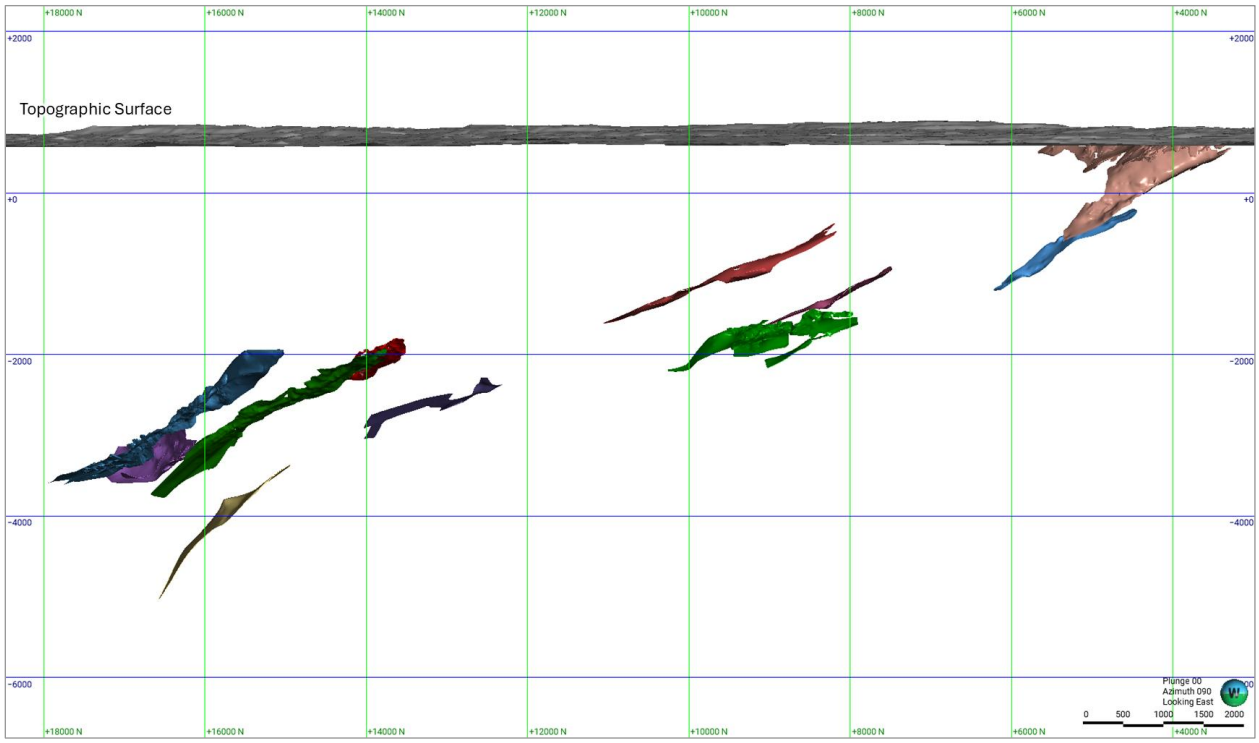
Zone	Specific Gravity	Tonnage Factor (t/ft^3)
American	3.123	0.0975
Cal Marble	3.123	0.0975
Fowler	3.123	0.0975
Lower Mahler	3.204	0.1000
Upper Mahler	3.214	0.1003
Mud Pond - Apron	3.139	0.0979
Mud Pond - Main	3.055	0.0953
N2D	3.061	0.0955
N2D – Waste	2.930	0.0915
New Fold	3.131	0.0977
Northeast Fowler	3.137	0.0979
Sylvia Lake	3.123	0.0975
Turnpike - UM14	3.271	0.1021
Turnpike - UM11	3.171	0.0989
Turnpike - Waste	2.845	0.0888
Waste	2.800	0.0874

Source: Modified from Warren et al., 2021

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 intersects the topographic surface.

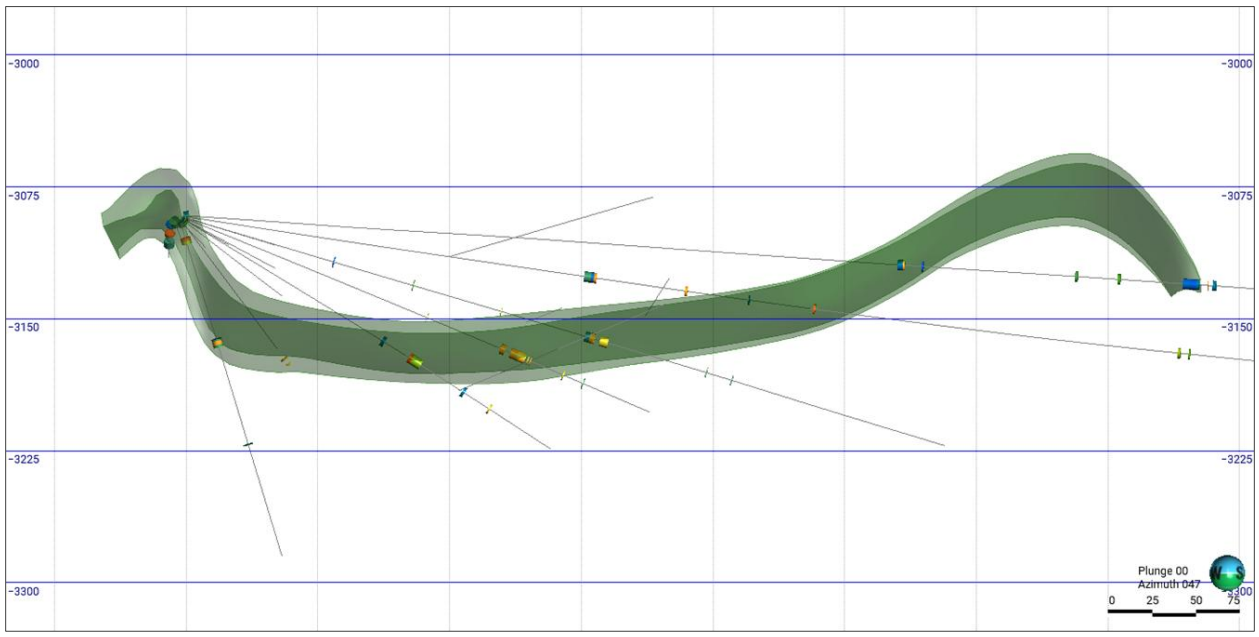


Source: Modified from Taylor et al., 2024

Figure 14-2: Zones relative to topographic surface

14.1.4 Geological Interpretation

All zones were defined and modeled by the 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 modeled within a single mineralized envelope as shown in Figure 14-3.



Source: Taylor et al., 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%	-

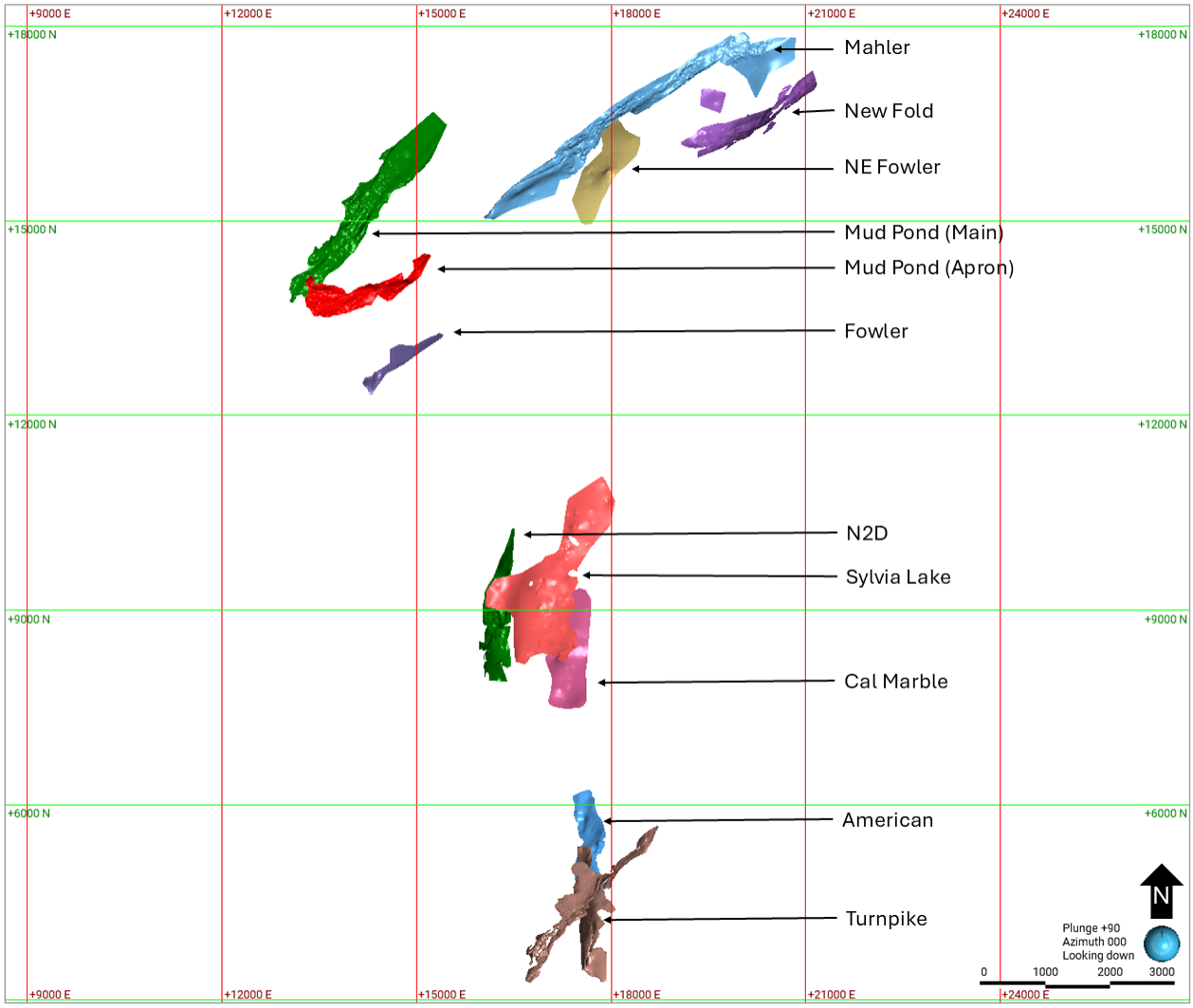
Source: Taylor et al., 2024

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 Items 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 2025 model updates were completed in version 2024.1.3. The American and Northeast Fowler deposits were modeled in Leapfrog Geo™ version 6.0.1 and estimated in MapTech Vulcan as described in the 2021 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 Volume (ft^3)
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, Annually 2019–2025	19,400,000
Mud Pond	Implicit vein system model	2008, 2009, 2017, Annually 2019–2025	15,463,500
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, Annually 2020– 2025	9,553,100
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: Modified from Warren et al., 2021



Source: Modified from Warren et al., 2021

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 3D modeled in Deswik (most recently version 2024.2.1611). 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. As-built wireframes were updated and utilized to deplete tonnage within the block models. The wireframes are shown in Figure 14-5.

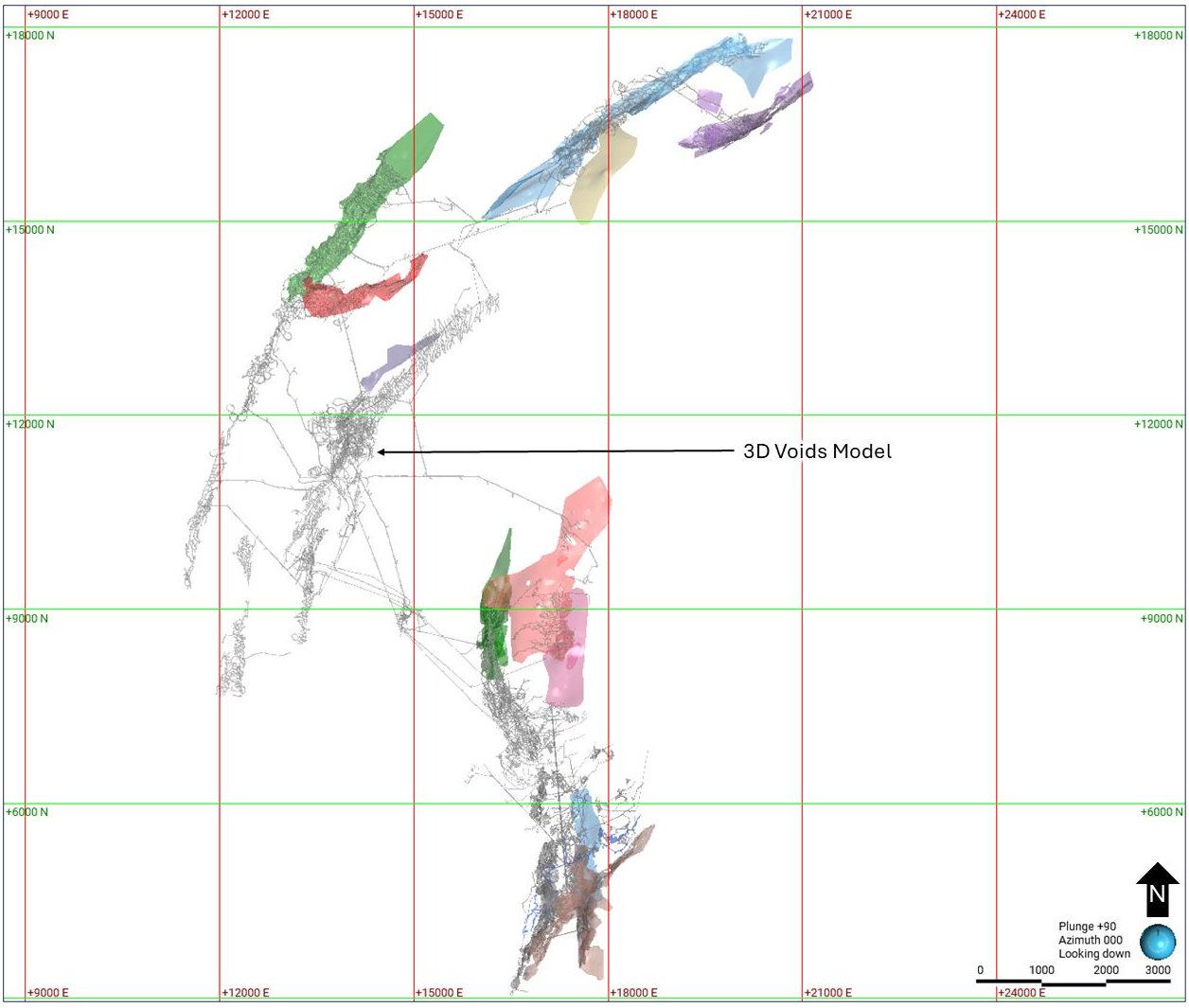


Figure 14-5: ESM 3D voids model

14.1.6 Exploratory Data Analysis

14.1.6.1 Assays

The ten zones are subdivided into 57 estimation domains that are included in the Mineral Resource. A total of 10,433 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	0.00	23.80
Cal Marble	CM	29	145.5	12.14	5.9	0.5	35.0	0.00	25.00
Cal Marble	CM2	8	16.5	8.29	6.8	0.8	47.2	0.00	20.00
Fowler	XC1	23	232.5	7.57	4.8	0.6	23.0	0.00	15.90
Mahler - Upper	UMA	492	2,028.1	17.39	15.7	0.9	247.1	0.00	59.72
Mahler - Upper	HW Interpolant	210	792.8	5.03	7.4	1.4	54.5	0.00	53.32
Mahler - Lower	LMA	593	2,531.4	16.30	14.7	0.9	216.3	0.00	59.15
Mahler - Lower	LMA2	9	13.2	13.15	11.7	0.8	138.6	2.30	38.97
Mahler - Lower	MWD4	311	1,185.3	20.82	17.7	0.8	316.7	0.00	59.34
Mahler - Lower	MWD4B	110	287.6	14.65	12.4	0.8	154.2	0.00	45.77
Mahler - Lower	MWD5	24	67.8	9.54	9.4	0.9	89.4	0.00	26.70
Mahler - Lower	MWD6	103	244.6	25.64	17.4	0.6	303.7	0.01	56.82
Mahler - Lower	FW Interpolant	127	335.7	3.95	7.0	1.7	50.1	0.00	47.71
Mahler - Lower	HW Interpolant	672	1,773.7	3.23	5.6	1.7	32.2	0.00	54.24
Mud Pond - Apron	MPA	325	1,624.7	12.70	11.0	0.9	122.6	0.00	52.74
Mud Pond - Apron	MPA2	10	25.9	13.07	12.2	0.9	150.7	0.49	54.34
Mud Pond - Main	MPM	411	2,419.8	12.05	8.8	0.7	77.4	0.00	51.37
N2D	UM14 HW1	382	1,479.0	12.80	8.7	0.7	76.7	0.00	37.33
N2D	UM14 HW2	8	58.0	13.08	14.8	1.1	220.1	0.00	42.00
N2D	UM14 FW1	125	341.9	7.44	5.7	0.8	33.1	0.00	25.70
N2D	UM14 FW2	38	248.8	6.81	5.7	0.8	33.0	0.00	20.10
N2D	UM14 FW3	11	37.6	8.82	5.5	0.6	30.2	1.63	18.50
N2D	UM14 FW4	16	78.2	9.93	4.6	0.5	21.8	0.00	18.60
N2D	UM14 FW5	8	33.3	8.55	5.9	0.7	34.9	0.88	19.25
N2D	UM11A vein	30	287.5	6.20	3.1	0.5	9.7	0.00	16.25
N2D	UM13 HW Anhy Zn Interpolant	191	677.4	5.96	7.8	1.3	61.1	0.00	42.90

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	0.00	39.33
New Fold	Vein 1	170	640.5	20.58	12.8	0.6	165.0	0.00	54.00
New Fold	Vein 2	119	474.1	15.18	11.2	0.7	126.2	0.00	54.00
New Fold	Vein 3	109	305.9	17.42	14.5	0.8	210.2	0.01	52.37
New Fold	Vein 4	13	38.9	13.87	8.9	0.6	79.9	5.15	34.50
New Fold	Vein 5	10	37.7	16.96	13.4	0.8	179.5	0.73	37.19
New Fold	Vein 6	5	17.7	13.43	5.7	0.4	33.1	8.05	23.30
New Fold	Vein 7	9	28.2	12.60	7.2	0.6	52.9	0.00	21.90
New Fold	Vein 8	19	62.4	16.63	16.2	1.0	263.5	0.00	44.50
New Fold	Vein 9	11	38.7	20.79	12.3	0.6	152.2	1.49	37.90
New Fold	Vein 10	20	79.5	11.29	9.3	0.8	87.1	0.11	44.20
New Fold	Vein 11	5	5.8	22.79	8.8	0.4	78.4	12.50	33.40
New Fold	Vein 12	11	24.5	17.75	10.1	0.5	103.9	0.90	41.96
New Fold	Vein 13	24	53.4	27.86	16.6	0.6	276.1	1.04	50.28
New Fold	Vein 14	37	110.1	23.34	17.8	0.8	319.2	0.08	51.59
New Fold	Interpolant	188	777.9	4.78	7.3	1.5	54.7	0.00	39.62
Northeast Fowler	Northeast Fowler	63	161.1	7.84	8.2	1.0	67.3	0.00	38.10
Sylvia Lake	SL	131	657.9	12.52	8.7	0.7	75.2	0.00	46.14
Sylvia Lake	SL LL	10	71.0	10.49	7.6	0.7	57.5	0.22	22.20
Turnpike	Hoist House FW	581	2,115.9	2.87	4.5	1.6	20.4	0.00	35.80
Turnpike	Hoist House HW	299	966.9	3.74	5.3	1.4	27.8	0.00	29.80
Turnpike	Pump House Lens A	271	980.7	3.47	5.3	1.5	27.9	0.00	26.20
Turnpike	Pump House Lens B	47	160.1	4.36	6.7	1.5	44.8	0.00	20.70
Turnpike	Pump House Vein 1	8	15.0	4.20	4.0	1.0	16.0	0.75	10.65
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	16.60
Turnpike	Streeter Lens A	106	273.1	3.79	6.0	1.6	35.8	0.00	27.70
Turnpike	Streeter Lens B	108	330.8	3.99	5.7	1.4	33.0	0.00	40.64
Turnpike	Streeter Lens C	27	85.9	4.54	5.9	1.3	35.3	0.00	22.40
Turnpike	Turnpike	1,099	5,977.0	4.26	5.0	1.2	25.0	0.00	24.90
Turnpike	West Ridge	254	810.8	6.25	7.6	1.2	57.6	0.00	40.84

Source: Modified from Taylor et al., 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	50	8	17.39	17.36
Mahler - Upper	HW Interpolant	26	9	5.03	4.84
Mahler - Lower	LMA	48.9	14	16.3	16.23
Mahler - Lower	LMA2	None	0	13.15	-
Mahler - Lower	MWD4	51.6	10	20.82	20.68
Mahler - Lower	MWD4B	38	4	14.65	14.51
Mahler - Lower	MWD5	None	0	11.88	-
Mahler - Lower	MWD6	53.7	3	25.64	25.59
Mahler - Lower	FW Interpolant	30	4	3.95	3.82
Mahler - Lower	HW Interpolant	34.6	11	3.23	3.15
Mud Pond - Main	MPM	28	20	12.05	11.81
Mud Pond - Apron	MPA	39	10	12.7	12.55
Mud Pond - Apron	MPA2	None	0	13.07	-
N2D	UM14 HW1	30	15	12.8	12.65
N2D	UM14 HW2	22	3	13.08	10.33
N2D	UM14 FW1	22	3	7.44	7.39
N2D	UM14 FW2	17	4	6.81	6.78
N2D	UM14 FW3	12	2	8.82	7.78
N2D	UM14 FW4	17	2	9.93	9.87
N2D	UM14 FW5	17	2	8.55	8.24
N2D	UM11A vein	13.5	4	6.2	6.15

Zone	Domain	Capping Value (Zn %)	Quantity Capped	Uncapped Mean (Zn %)	Capped Mean (Zn %)
N2D	UM13 HW Anhy Zn Interpolant	22	14	5.96	5.53
N2D	UM14 Serp Dol Zn Interpolant	24	27	3.48	3.44
New Fold	Vein 1	44.5	5	20.58	20.51
New Fold	Vein 2	41.8	4	15.18	15.09
New Fold	Vein 3	48	2	16.85	16.74
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	12.6	-
New Fold	Vein 8	None	0	16.63	-
New Fold	Vein 9	None	0	20.79	-
New Fold	Vein 10	26.2	3	11.29	10.62
New Fold	Vein 11	None	0	22.79	-
New Fold	Vein 12	None	0	17.75	-
New Fold	Vein 13	None	0	27.86	-
New Fold	Vein 14	None	0	23.34	-
New Fold	Interpolant	11	23	4.65	3.53
Northeast Fowler	Northeast Fowler	None	0	7.84	-
Sylvia Lake	SL	38.6	1	12.52	12.47
Sylvia Lake	SL LL	17	1	10.49	9.49
Turnpike	Hoist House FW	22	8	2.84	2.8
Turnpike	Hoist House HW	22	5	3.74	3.73
Turnpike	Pump House Lens A	24	2	3.47	3.45
Turnpike	Pump House Lens B	19	2	4.36	3.98
Turnpike	Pump House Vein 1	None	0	4.2	-
Turnpike	Pump House Vein 2	None	0	1.48	-
Turnpike	Pump House Vein 3	None	0	2.74	-
Turnpike	Streeter Lens A	16	5	3.79	3.49
Turnpike	Streeter Lens B	17	8	3.99	3.84
Turnpike	Streeter Lens C	17	3	4.54	4.41
Turnpike	Turnpike	19	18	4.26	4.23
Turnpike	West Ridge	19	13	6.25	5.97

Source: Modified from Taylor et al., 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 14-18.

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	182	18.99	18.96
Mahler - Upper	HW Interpolant	10 ft run length with residuals < 5 ft added to prior interval	89	4.73	4.54
Mahler - Lower	LMA	Vein Length	217	18.5	18.46
Mahler - Lower	LMA2	Vein Length	3	13.8	-
Mahler - Lower	MWD4	Vein Length	79	19.17	19.08
Mahler - Lower	MWD4B	Vein Length	24	16.32	15.91
Mahler - Lower	MWD5	Vein Length	6	9.59	
Mahler - Lower	MWD6	Vein Length	30	24.23	24.18
Mahler - Lower	FW Interpolant	10 ft run length with residuals < 5 ft added to prior interval	42	3.95	3.82
Mahler - Lower	HW Interpolant	10 ft run length with residuals < 5 ft added to prior interval	216	3.23	3.15
Mud Pond - Main	MPM	Vein Length	167	11.66	11.56
Mud Pond - Apron	MPA	Vein Length	119	13.04	12.96
Mud Pond - Apron	MPA2	Vein Length	6	13.07	n/a
N2D	UM14 HW1	Vein Length	136	12.71	12.56
N2D	UM14 HW2	Vein Length	4	13.08	10.33

Zone	Domain	Method	Composite Count	Un-capped, Composite Mean (Zn %)	Capped, Composite Mean (Zn%)
N2D	UM14 FW1	Vein Length	70	7.44	7.39
N2D	UM14 FW2	Vein Length	21	6.81	6.78
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	75	21.26	21.21
New Fold	Vein 2	Vein Length	49	16.3	16.23
New Fold	Vein 3	Vein Length	22	16.85	16.74
New Fold	Vein 4	Vein Length	8	14.31	n/a
New Fold	Vein 5	Vein Length	2	11.54	n/a
New Fold	Vein 6	Vein Length	2	9.3	n/a
New Fold	Vein 7	Vein Length	4	14.22	n/a
New Fold	Vein 8	Vein Length	11	12.94	n/a
New Fold	Vein 9	Vein Length	2	14.94	n/a
New Fold	Vein 10	Vein Length	13	15.82	14.23
New Fold	Vein 11	Vein Length	5	23.82	n/a
New Fold	Vein 12	Vein Length	6	18.77	n/a
New Fold	Vein 13	Vein Length	5	28.65	n/a
New Fold	Vein 14	Vein Length	5	18.3	n/a
New Fold	Interpolant	10 ft run length with residuals < 5 ft added to prior interval	107	4.54	3.35
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

Zone	Domain	Method	Composite Count	Un-capped, Composite Mean (Zn %)	Capped, Composite Mean (Zn%)
Turnpike	Pump House Lens A	10ft run length with residuals < 5 ft added to prior interval	99	3.44	3.42
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	10ft 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: Modified from Taylor et al., 2024

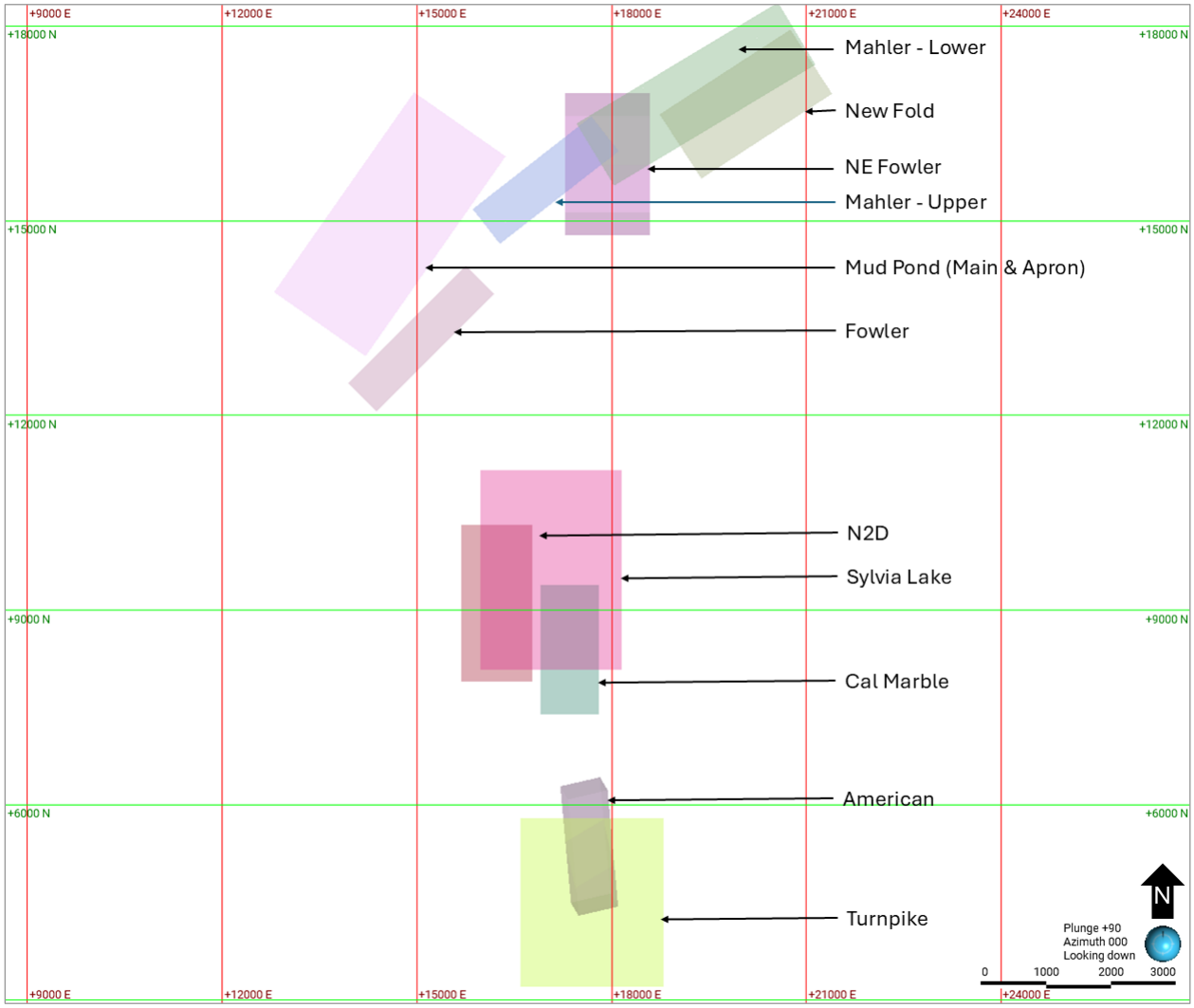
14.1.6.4 Variogram Analysis and Modeling

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: Modified from Warren et al., 2021

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 (ft):	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 (ft):	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 (ft):	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:	59
Sub-block count (ft):	8	8	8	Dip:	0
Minimum size (ft):	2.5	2.5	2.5	Pitch:	0
Base point (ft):	17,460	16,500	-2,840	-	-
Boundary size (ft):	1,120	3,620	900	-	-
Mahler - Upper					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	20	20	20	Azimuth:	52
Sub-block count (ft):	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 (ft):	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):	1,720	3,760	2,080	-	-
N2D					
Blocks	**X**	**Y**	**Z**	**Orientation**	**Degrees**
Parent block size (ft):	20	20	20	Azimuth:	0
Sub-block count (ft):	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 (ft):	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 (ft):	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 (ft):	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 (ft):	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: Modified from Taylor et al., 2024

14.1.7.2 Estimate Parameters

Due to the high variability of the ESM deposits and the absence of robust variography, inverse distance squared (ID2) and cubed (ID3) interpolation methods, consistent with site standard practice, were used to estimate grade into parent blocks within the block model. Declustering was applied and used for most of the ID estimates. The majority of the estimates were designed for a single pass; however, some domains required multiple passes. 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	1000	500	250	Variable			3	5	1	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	-	-
Mahler - Lower	LMA	1	ID2	700	300	300	Variable			2	5	1	Clamp	20	17
Mahler - Lower	LMA2	1	ID2	300	300	300	0	0	0	3	3	1	None	-	-
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	MWD4B	1	ID2	700	350	350	Variable			2	7	1	None	-	-
Mahler - Lower	MWD5	1	ID2	500	250	250	15	38	117	3	5	1	Clamp	11.5	20
Mahler - Lower	MWD6	1	ID2	500	250	250	Variable			2	7	1	Clamp	23.4	30
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	-	-
Mud Pond - Main	MPM	1	ID2	700	350	350	Variable			2	12	None	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	-	-
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	110	110	110	0	0	0	5	5	1	None	-	-
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	700	350	350	Variable			3	15	1	Clamp	17	40
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	-	-

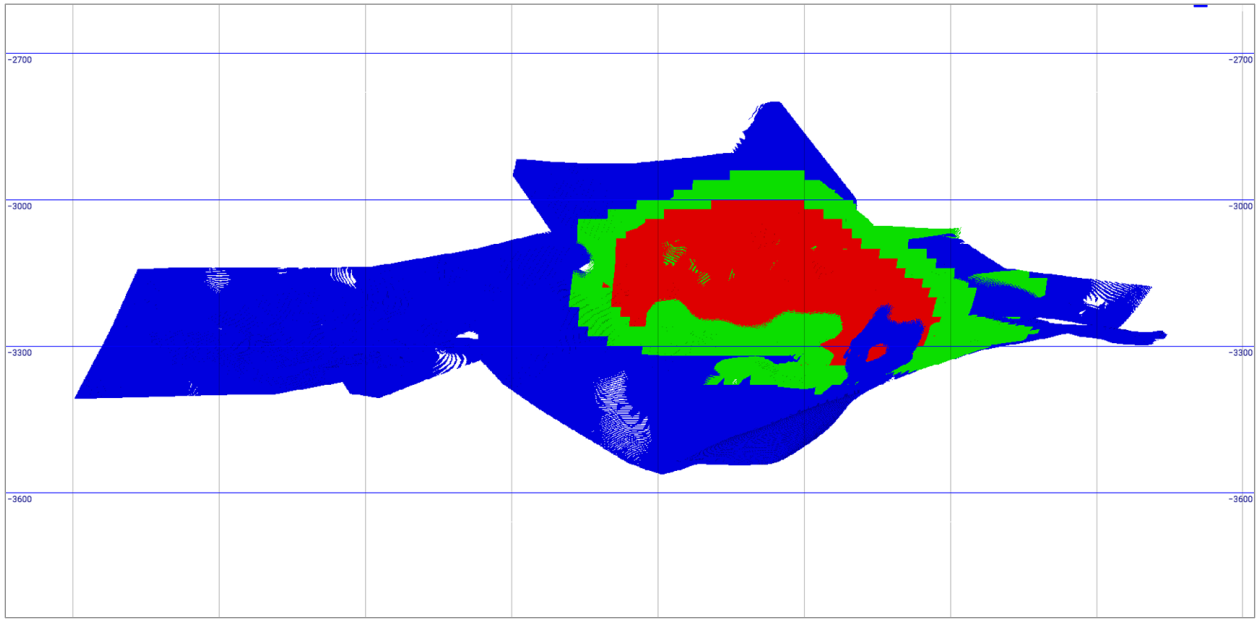
Zone	Domain	Pass	Est.	Search Radius			Search Orientation (Leapfrog)			Sample Selection			Outlier Restriction		
				Maj.	Semi	Min.	Dip	Dip Azi	Pitch	Min	Max	Max / hole	Type	Threshold (Zn %)	Distance (% of search)
New Fold	Vein 10	1	ID3	300	200	200	Variable			3	5	1	None	-	-
New Fold	Vein 11	1	ID2	100	100	100	0	0	0	3	5	-	None	-	-
New Fold	Vein 12	1	ID2	600	300	300	0	104	0	3	5	1	None	-	-
New Fold	Vein 13	1	ID2	200	125	125	12	112	0	2	5	1	Clamp	30	50
New Fold	Vein 14	1	ID2	400	300	300	55	325	110	3	5	-	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: Modified from Taylor et al., 2024

14.1.8 Resource Classification

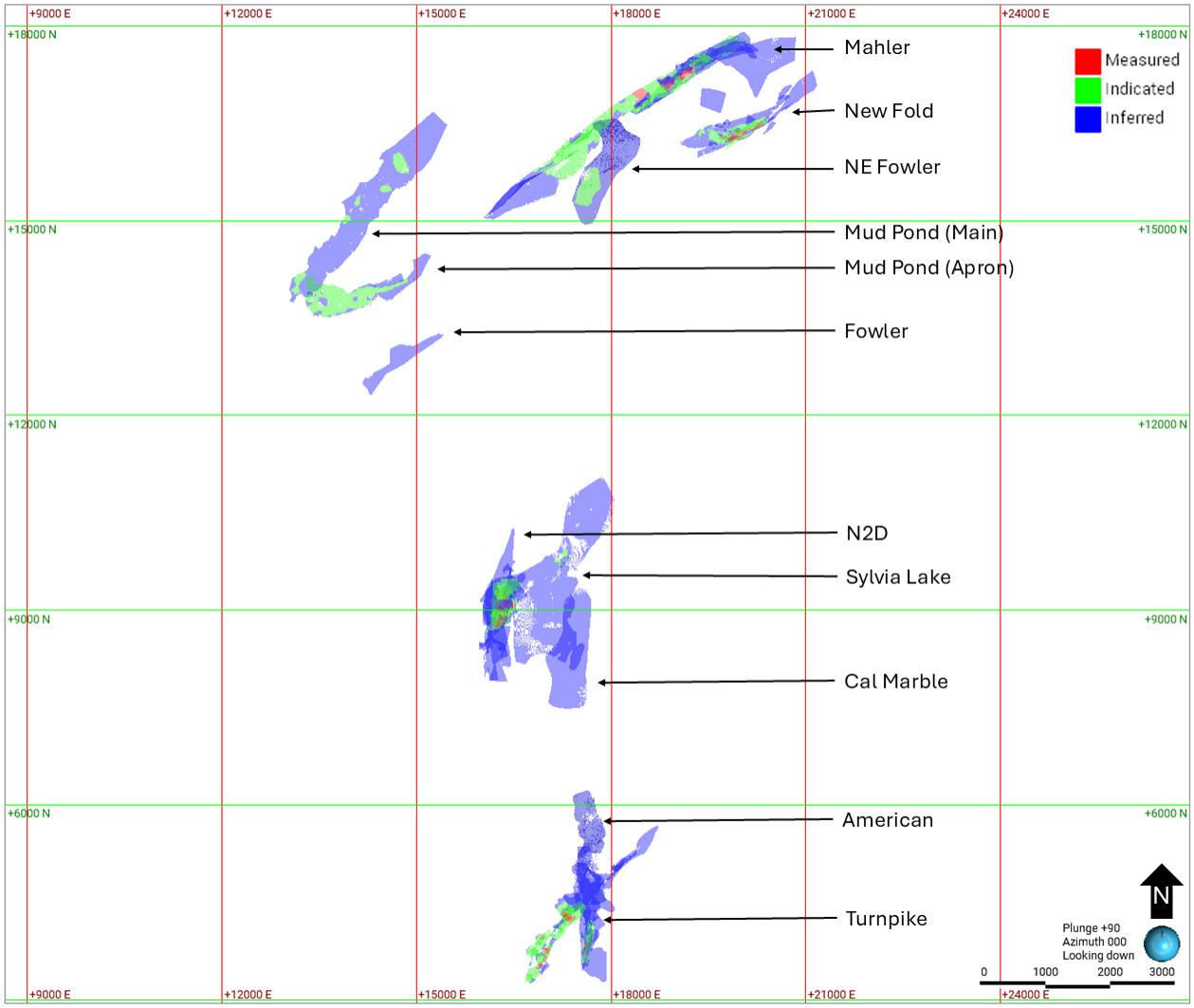
The ESM zinc deposits have been classified according to the CIM Definition Standard for Mineral Resources and Mineral Reserves. The resource classification considered the quality, quantity and distance to the data informing blocks in the model, as well as the geological continuity of the mineralized 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 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: Modified from Taylor et al., 2024
Note: Red=Measured, green=Indicated, blue=Inferred.

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



Source: Modified from Taylor et al., 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, political, 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. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.1.9 Mineral Resource Tabulation

14.1.9.1 Underground Mineral Resource

The underground Mineral Resource reported is effective as of June 9, 2025, and have been compiled from eleven separate block models including the American, Cal Marble, Fowler, Mahler – Lower, Mahler – Upper, Mud Pond, N2D, New Fold, Northeast Fowler, Silvia Lake, and Turnpike.

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, QP, considers the mineralized envelopes as modeled to have sufficient continuity and grade to have reasonable prospects for eventual economic extraction given the long site history (95 years) of successful planning and profitably mining the Balmat massive sulfides.

The Turnpike UG resource was constrained by stope optimization (SO) shapes that were generated in Deswik.SO to guide the classification of the resource.

The Underground and Open Pit Zinc Mineral Resources were modeled using Leapfrog Geo™ (version 2024.1.3) and estimated in Leapfrog Edge. The QP, Don Taylor, has reviewed the geological models and estimation results through site visits and remote sessions, assessed the methodologies and outcomes for consistency with industry standards, and is satisfied that the work is reasonable and suitable for reporting Mineral Resources.

Table 14-10: Underground Mineral Resource Estimate as of June 9, 2025

Classification	Tons (000's US short tons)	Zn (%)	Contained Pounds (M lb)
Measured	282	17.3	97
Indicated	1,133	16.0	362
Measured + Indicated	1,415	16.2	459
Inferred	4,512	12.1	1,088

Notes:

1. The qualified person for the 2025 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 2025 underground MRE encompasses 41 vein domains and 6 indicator RBF interpolant shells totaling 45 individual wireframes.
4. Geological and block models for the underground MRE used data from a total of 1,153 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 11 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-economic, marketing, or political issues or any other relevant issues that could materially affect this MRE.

14.1.9.2 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). The open pit is considered an accretive project with no G&A costs, and selling costs are incorporated into the selling price. The QP has reviewed these assumptions and considers them reasonable for the purposes of this Mineral Resource Estimate.

Table 14-11: Turnpike pit constraint parameters

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

Source: Taylor et al., 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: Taylor et al., 2024

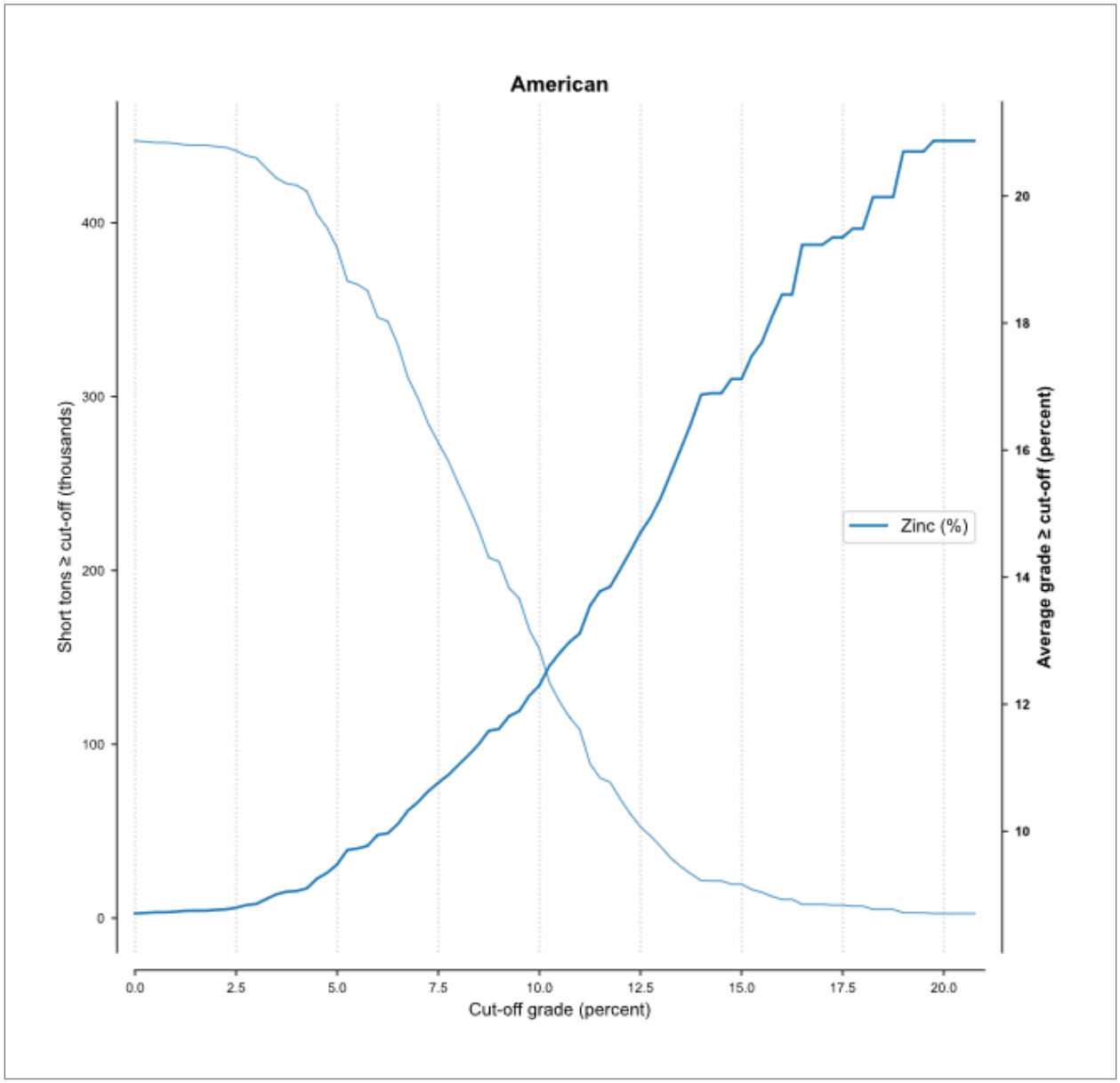
Notes:

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 the Project is 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-economic, marketing, or political issues, or any other relevant issues that could materially affect this MRE.

14.1.9.3 Mineral Resource Sensitivity

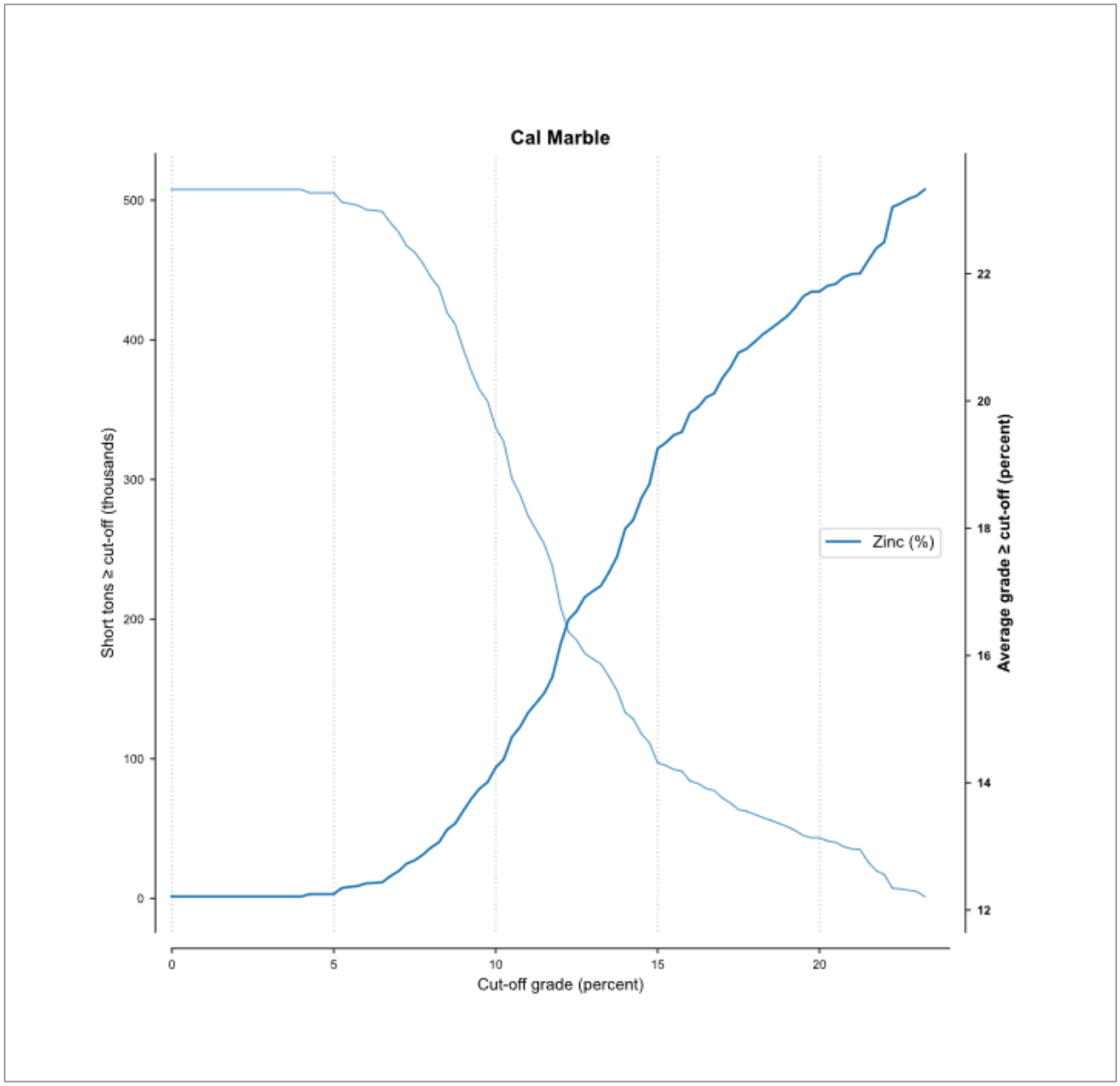
ESM Underground

Grade/tonnage (GT) graphs for each area as a function of movement in cut-off grade are presented to document the sensitivity of the Mineral Resources to a variety of factors. 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. The Turnpike UG graph shows blocks only within the SO shapes. These graphs are shown for each area in Figure 14-9 through Figure 14-21. Due to the variances in grade and mineralization within each area, sensitivities to COG differ for each.



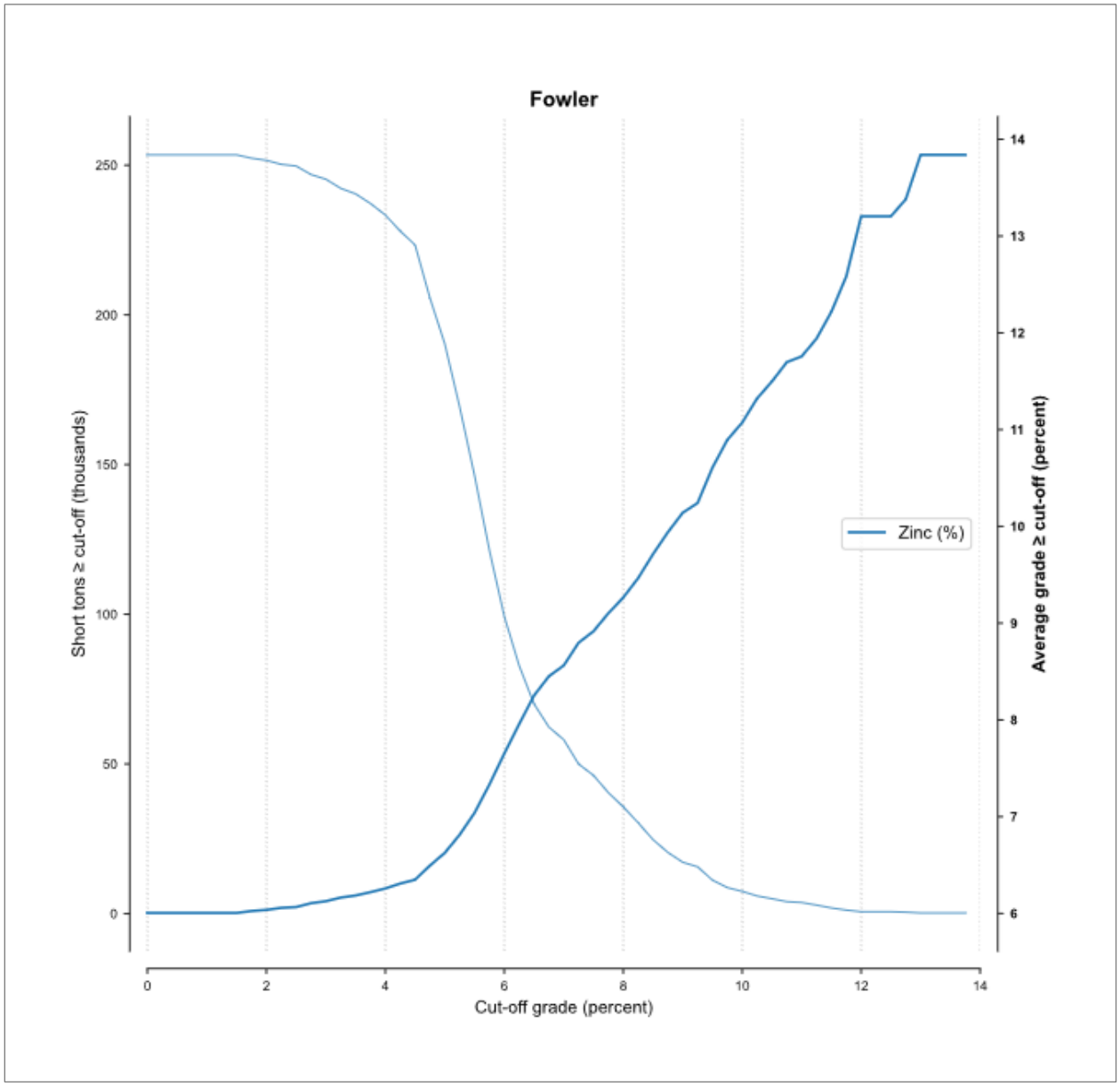
Source: Taylor et al., 2024

Figure 14-9: American grade tonnage graph



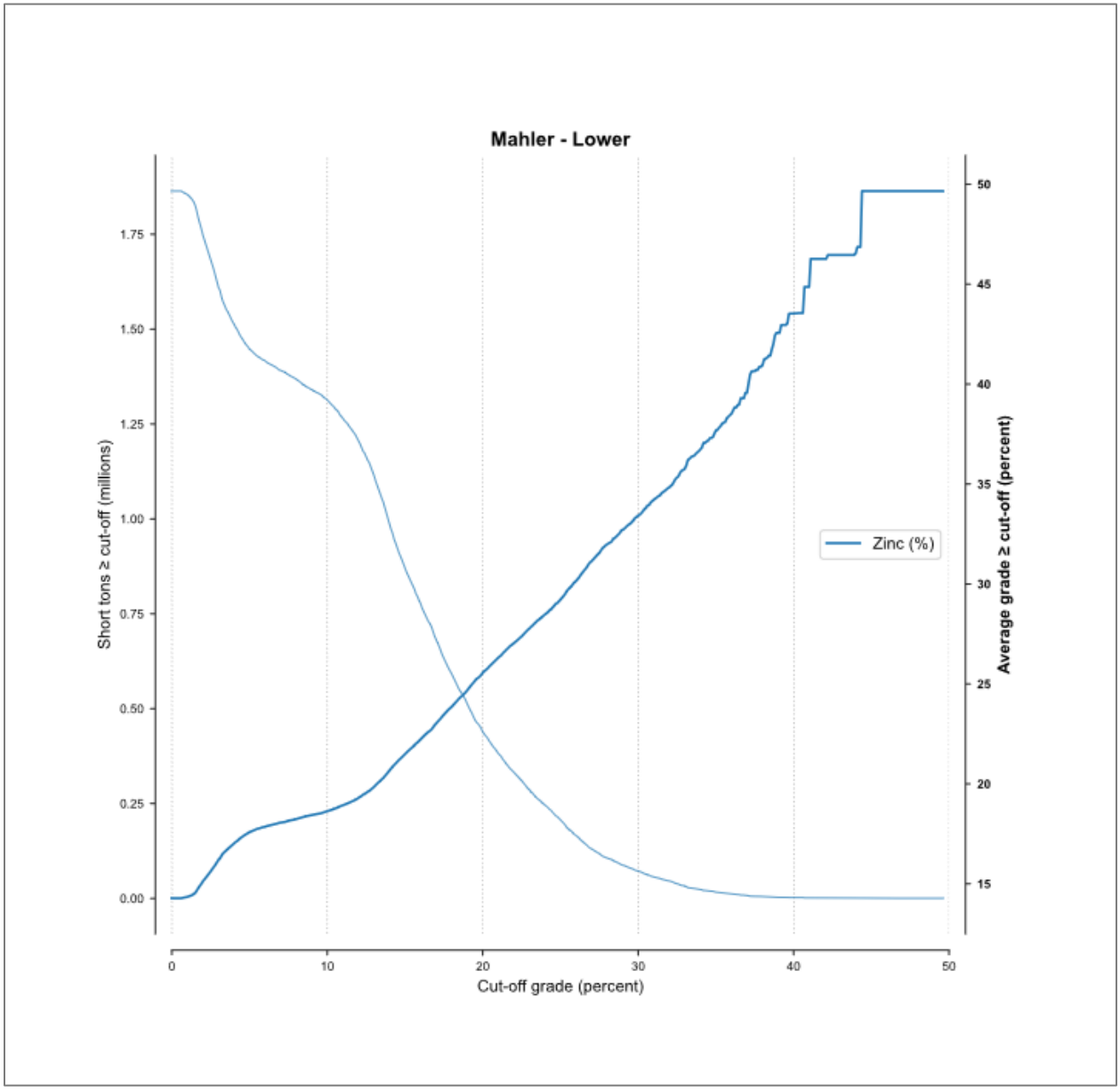
Source: Modified from Taylor et al., 2024

Figure 14-10: Cal Marble grade tonnage graph



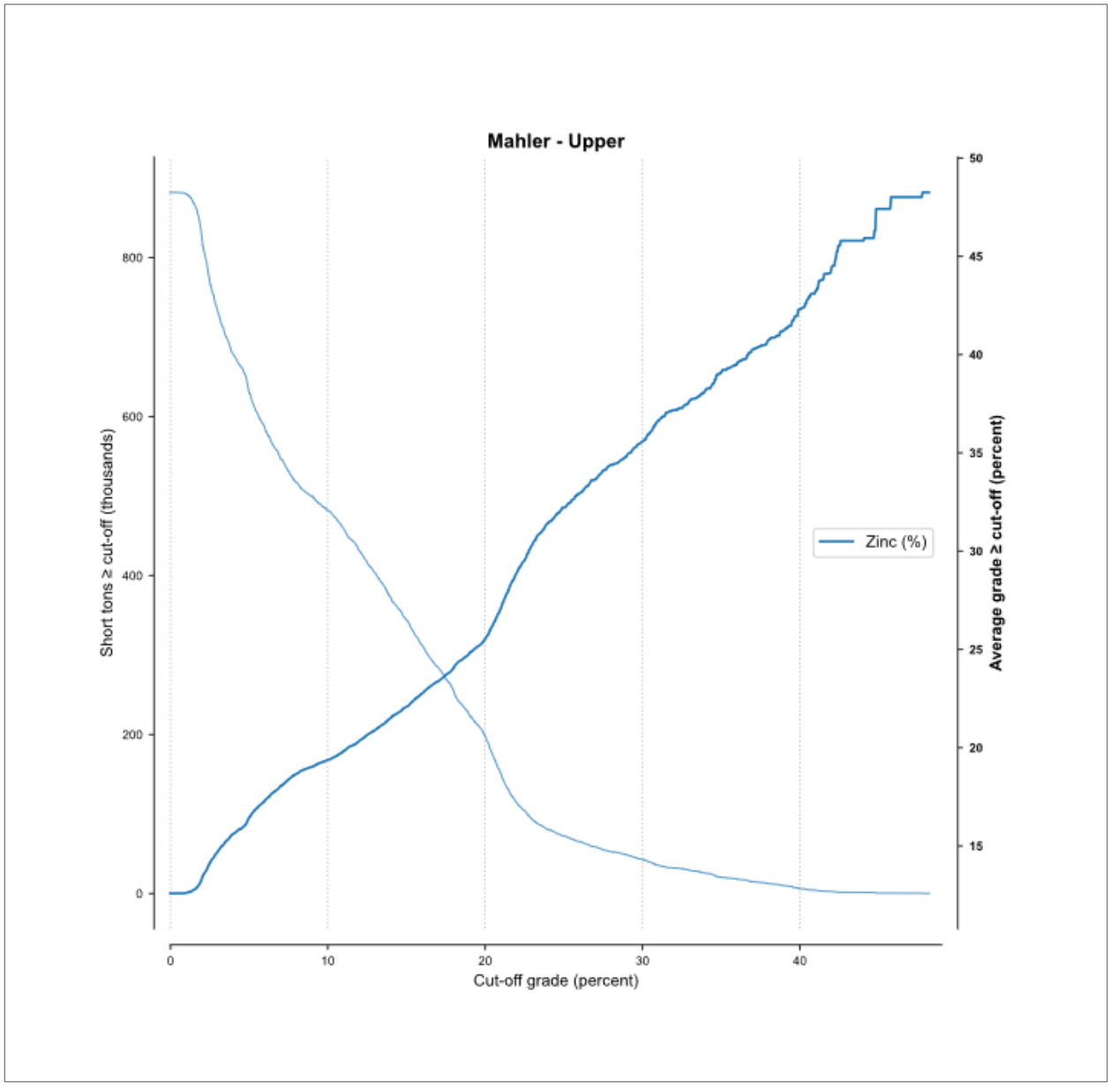
Source: Taylor et al., 2024

Figure 14-11: Fowler grade tonnage graph



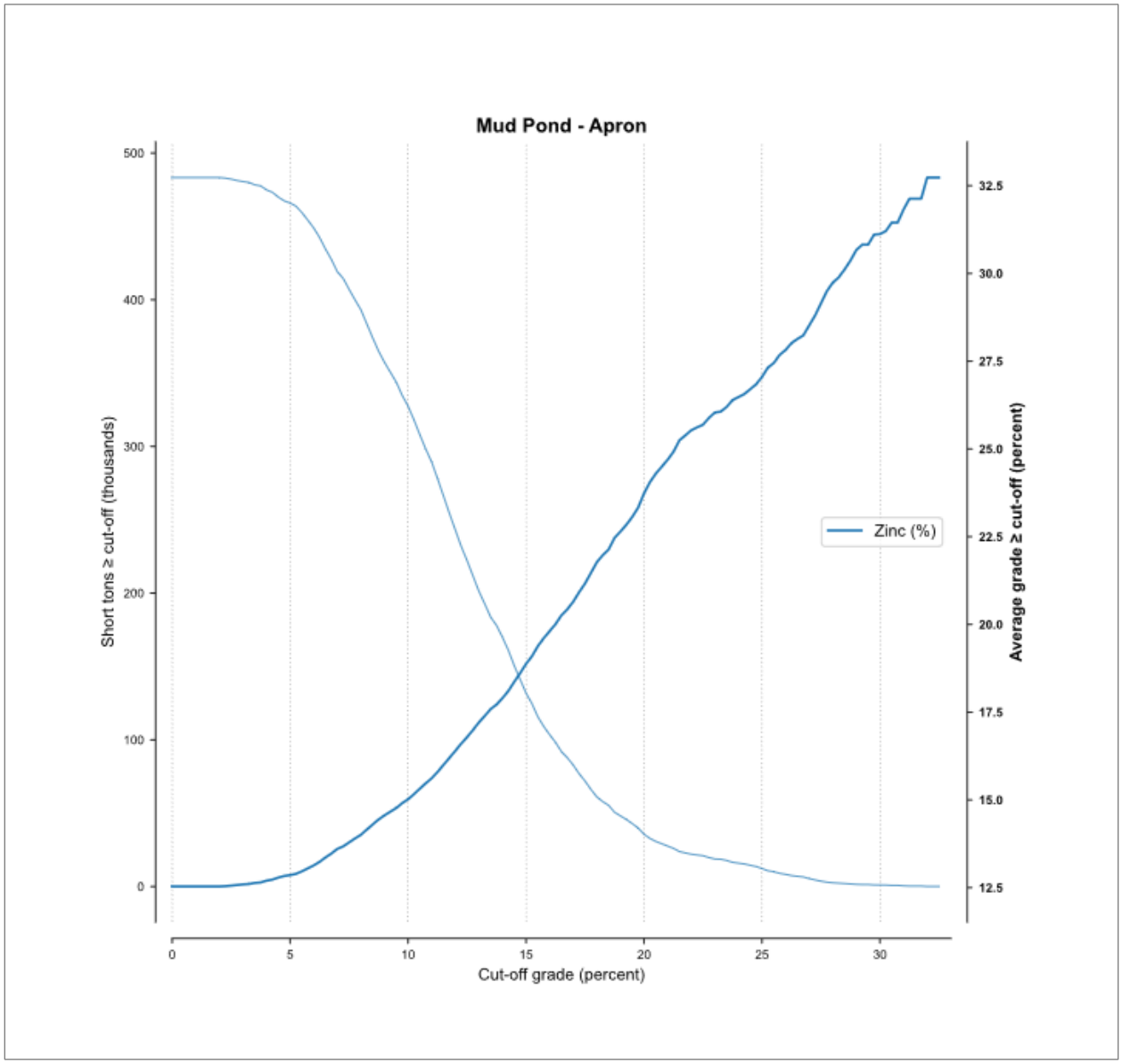
Source: Modified from Taylor et al., 2024

Figure 14-12: Lower Mahler grade tonnage graph



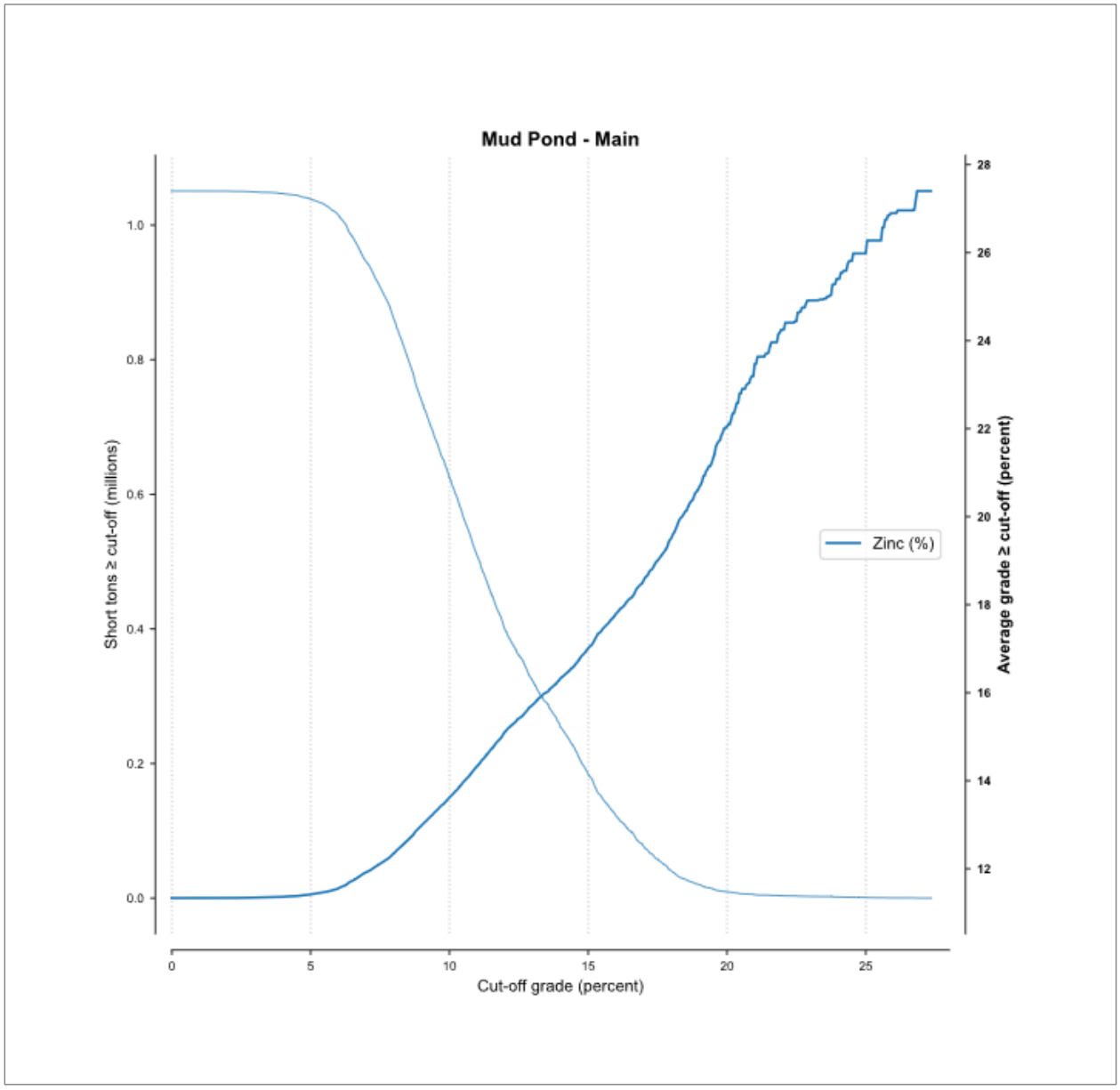
Source: Modified from Taylor et al., 2024

Figure 14-13: Upper Mahler grade tonnage graph



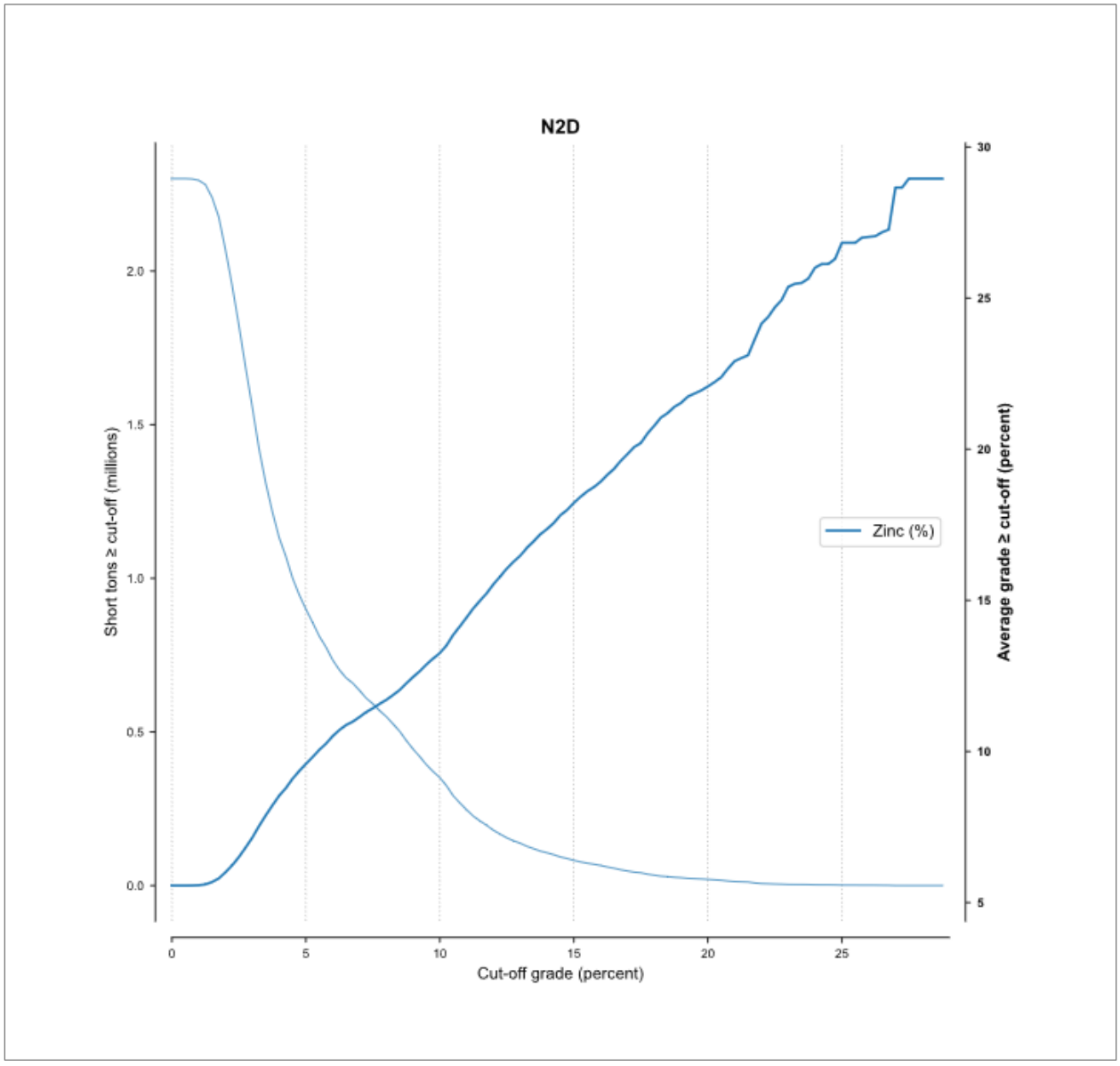
Source: Modified from Taylor et al., 2024

Figure 14-14: Mud Pond Apron grade tonnage graph



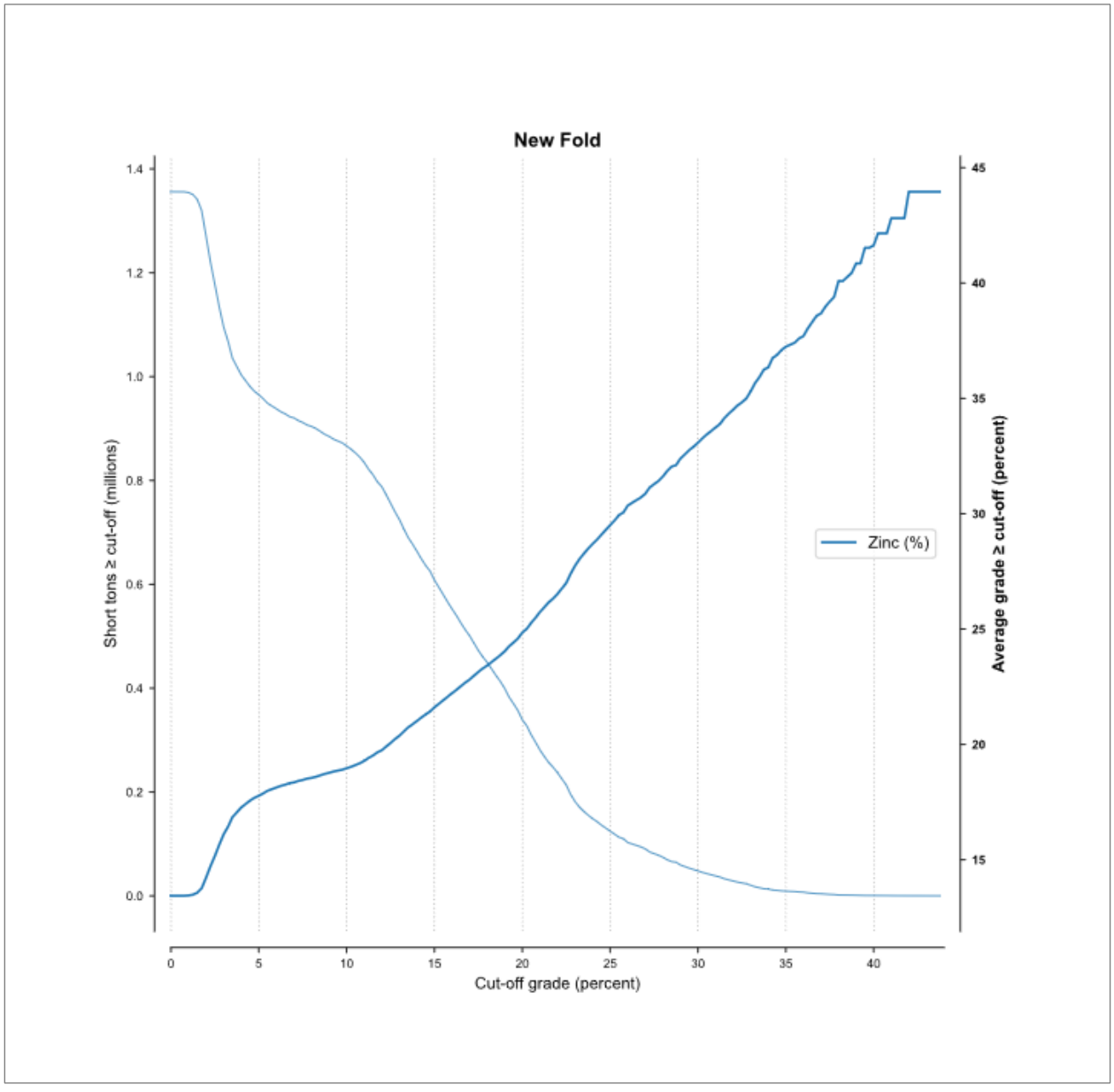
Source: Modified from Taylor et al., 2024

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



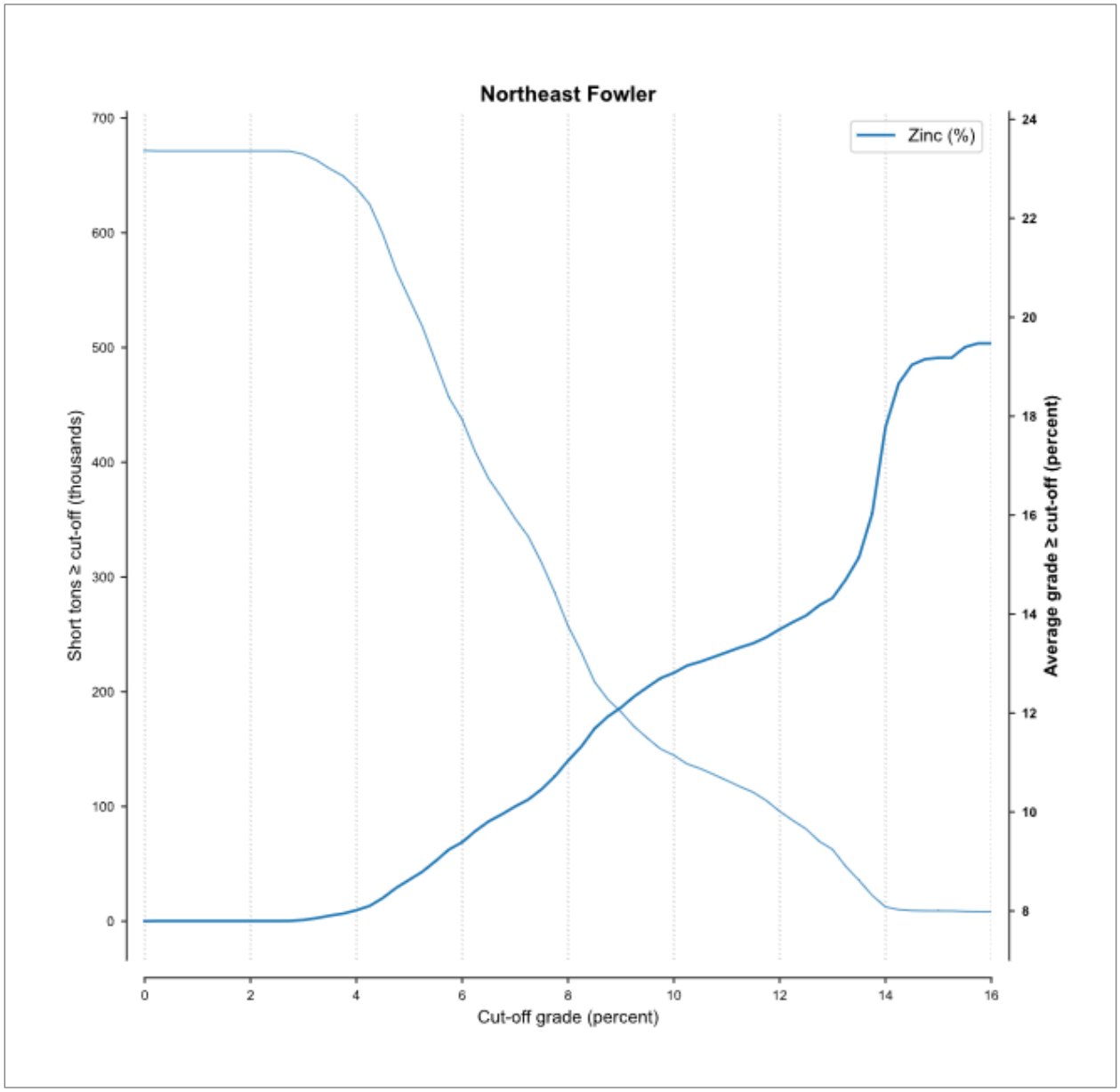
Source: Taylor et al., 2024

Figure 14-16: N2D grade tonnage graph



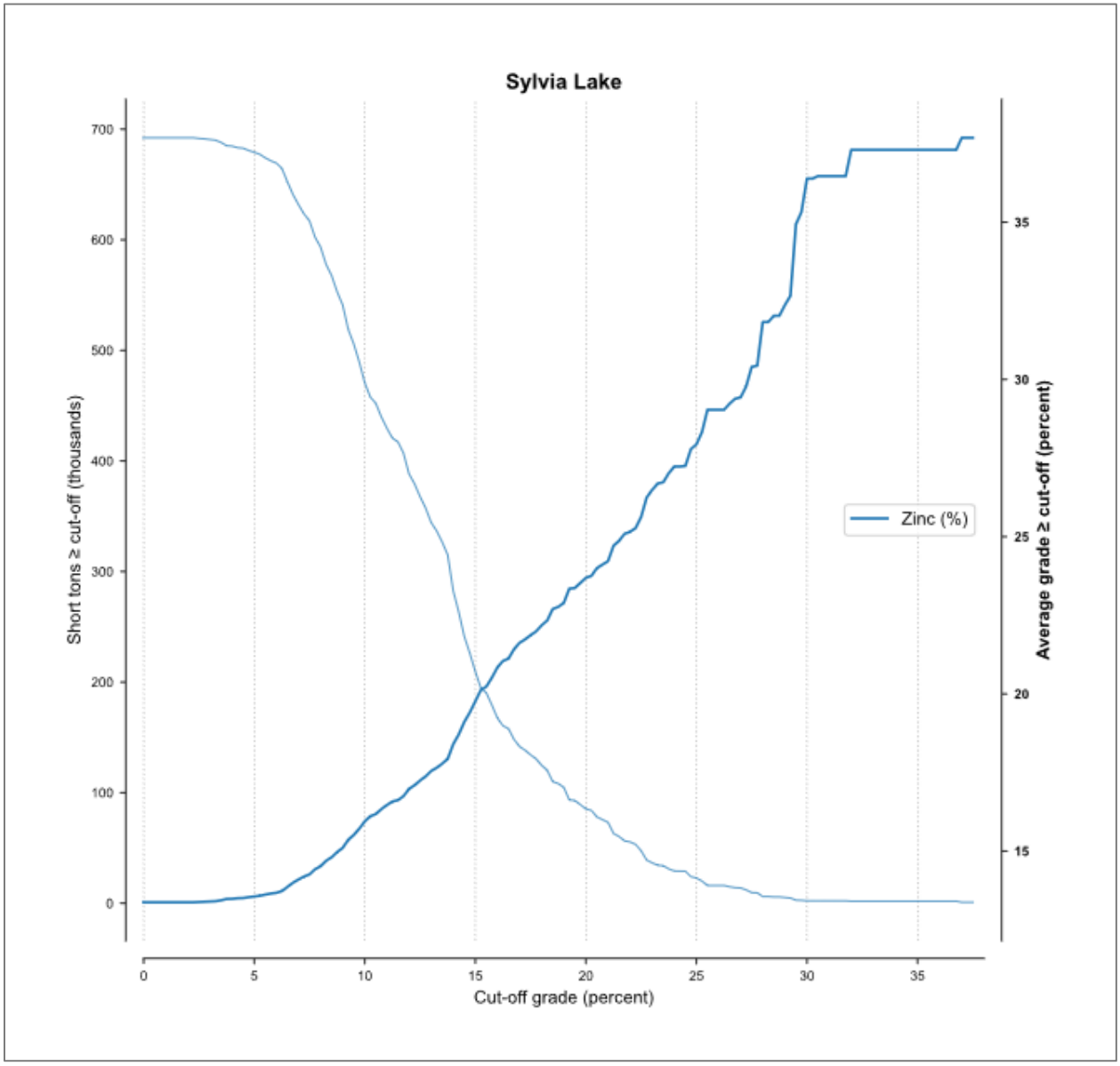
Source: Modified from Taylor et al., 2024

Figure 14-17: New Fold grade tonnage graph



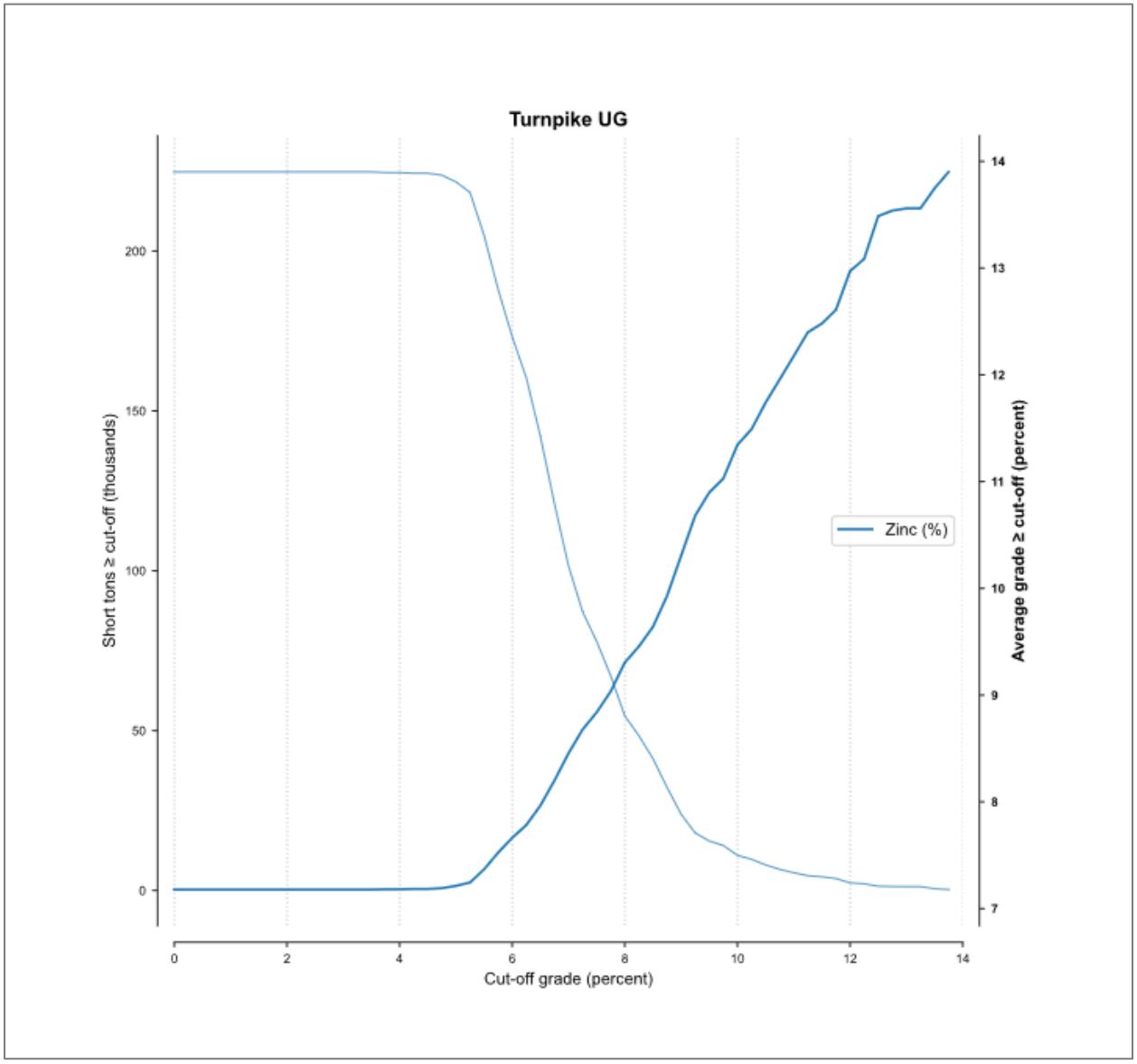
Source: Taylor et al., 2024

Figure 14-18: Northeast Fowler grade tonnage graph



Source: Taylor et al., 2024

Figure 14-19: Sylvia Lake grade tonnage graph

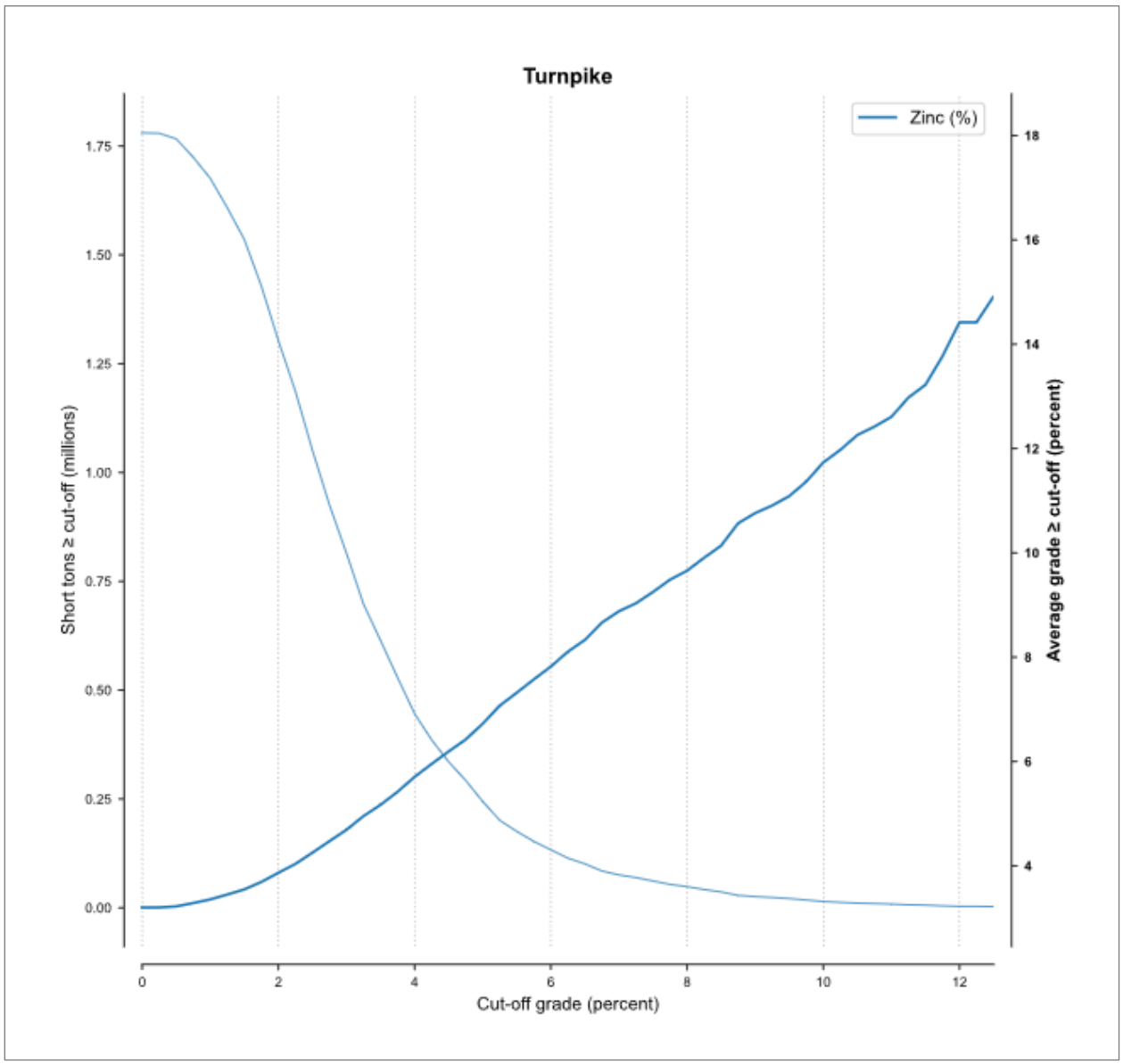


Source: Modified from Taylor et al., 2024

Figure 14-20: Turnpike UG grade tonnage graph; SO constrained

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 Item 14.1.9.2. The resources for the pit areas are more sensitive to COG than the underground resources, primarily due to the lower average grades.



Source: Taylor et al., 2024

Figure 14-21: Turnpike Open Pit grade tonnage graph

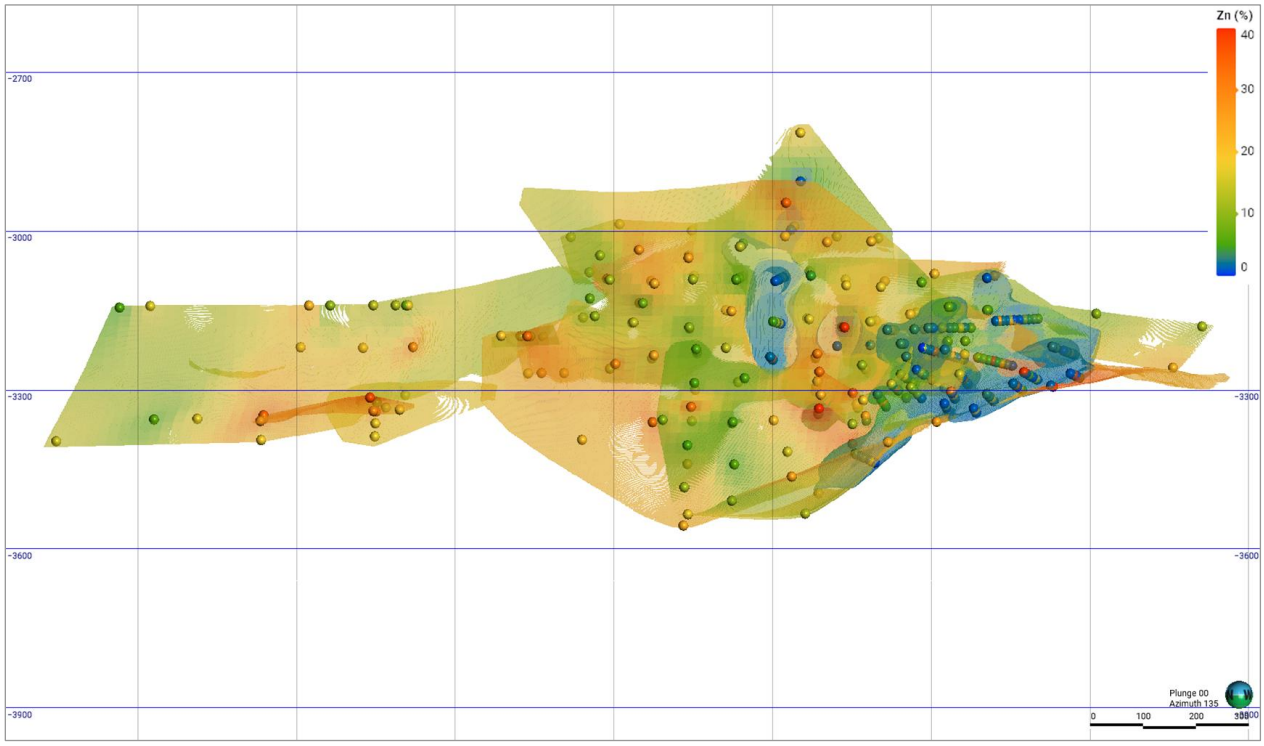
14.1.10 Model Validation

Model validation consisted of visual comparison of block estimates to informing composites and the use of swath plots to confirm spatial grade distribution and appropriate smoothing. The QP has reviewed the validation procedures and results and considers them reasonable and appropriate.

14.1.10.1 Visual Comparison

Visual validation of the block estimates for both the underground and open pit resources was conducted as part of the estimation process. 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 14-22 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.

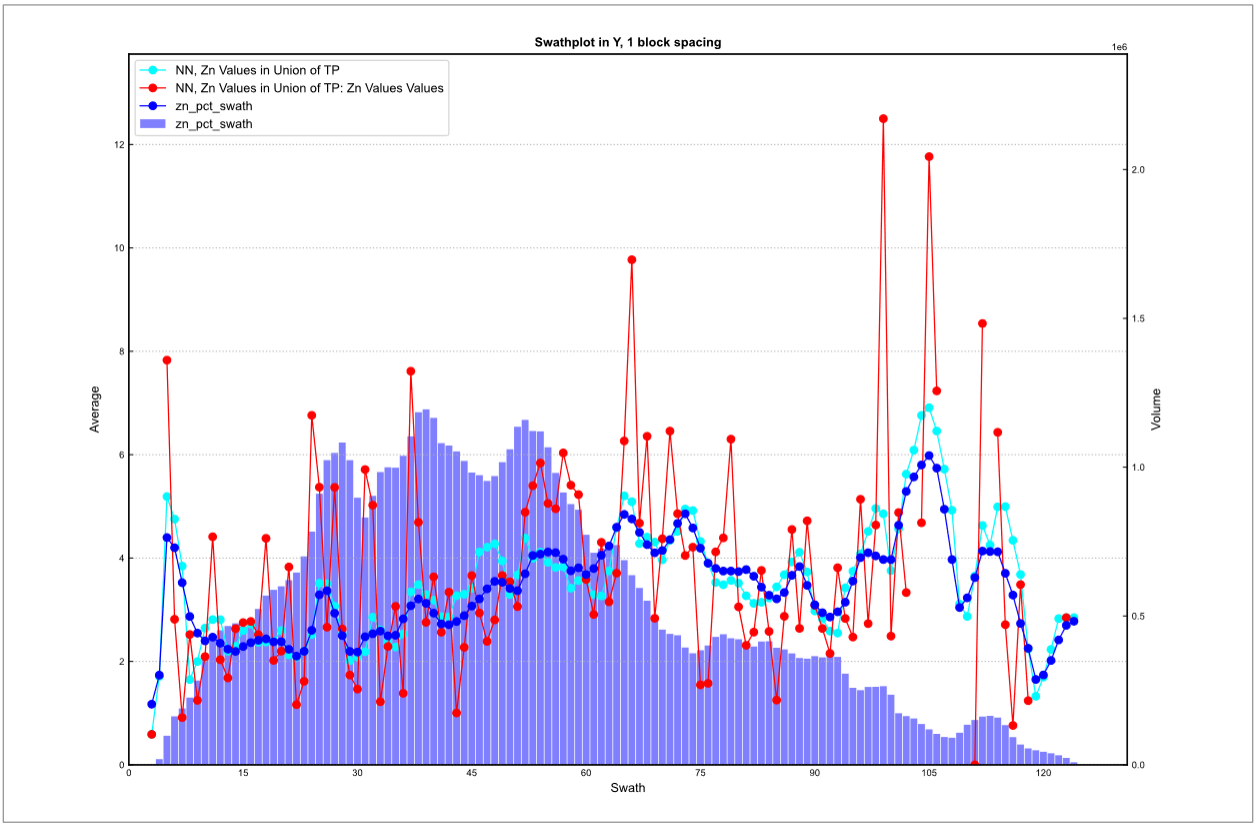


Source: Modified from Taylor et al., 2024
Note: View looking southeast.

Figure 14-22: New Fold model and composite values for zinc

14.1.10.2 Swath plots

Swath plots were used 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 14-23 for Turnpike. Swath plots generally show agreement of the estimate to the composites, with an appropriate degree of smoothing.



Source: Taylor et al., 2024

Figure 14-23: Swath plot Zn% - Turnpike area

14.1.11 Relevant Factors

The QP is not aware of any other material factors that may influence the disclosure of Mineral Resources. The underground areas are currently being mined and all requisite permits are in hand for open pit mining at Turnpike.

14.1.12 Comparison to Previous Estimate

The current underground Mineral Resource Estimate, effective June 9, 2025, supersedes the previous estimate dated July 16, 2024 (Taylor, 2024). Table 14-13 summarizes the changes in tonnage, grade, and contained zinc pounds across resource categories.

Table 14-13: Change of underground Mineral Resources from previous estimate

Category	Tons (000's)	Zn (%)	Pounds (M lb)	Δ Tons (000's)	Δ Pounds (M lb)	Δ (%)
Measured	282	17.3	97	-13	-4	-4.0%
Indicated	1,133	16.0	362	-25	-2	-0.5%
Measured + Indicated	1,415	16.2	459	-38	-6	-1.3%
Inferred	4,512	12.1	1,088	185	39	3.7%

Some mining activity during the period occurred in areas that were at the down-dip limits of the 2024 resource models, resulting in extraction from zones that were previously underrepresented. As a result, while depletion occurred within the defined resource, updated vein models incorporating material just outside those limits led to additions that partially offset the losses. These updates reflect improved geological understanding and more refined modeling of mineralized zones.

The Measured and Indicated categories show net depletions totalling 38,000 tons and 6 million pounds following updated geological modeling and refined estimation parameters. The Inferred category increased by 185,000 tons and 39 million pounds reflecting additional drilling and inclusion of Turnpike UG resources. This zone was added following the completion of engineering studies using Deswick.SO and pseudoflow methodologies which confirmed it's mineability and supported its classification within the current resource model. Continued growth in Inferred resources offsets depletion in producing areas indicating continued long term zinc mining at ESM.

This comparison is provided for context only. The July 2024 estimate is no longer considered current and should not be relied upon.

14.2 Graphite

14.2.1 Deposit Database

As of December 3, 2024, the Kilbourne Study database included 39 surface-collared diamond drillholes (DDH) completed in 2023 and 2024 totaling 11,917 ft, six surface-collared historic DDH partially re-assayed for graphite totaling 17,698 ft, and one 84-ft surface channel sample. In total, 29,699 ft of drilling and sampling were used for geological modeling of the Kilbourne deposit. Of the historic drilling, 1,378 ft were re-assayed for graphite. The database contains 3,396 assay records, of which 2,088 report graphite content (%Cg). Drillholes KX23-001 to KX24-039, completed in 2023 and 2024, form the basis of the initial Mineral Resource Estimate.

As of November 7, 2025, an additional five exploration drillholes totaling 1,738 ft and 21 definition drillholes totaling 7,958 ft have been completed. Assay results for the definition drilling are pending; however, logged lithologies are consistent with earlier drilling and are not expected to materially impact the resource.

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

Table 14-14: 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: Taylor et al., 2024

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

ESM 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 14-15 summarizes the results of the SG and TF measurements by domain.

Table 14-15: 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: Taylor et al., 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

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.

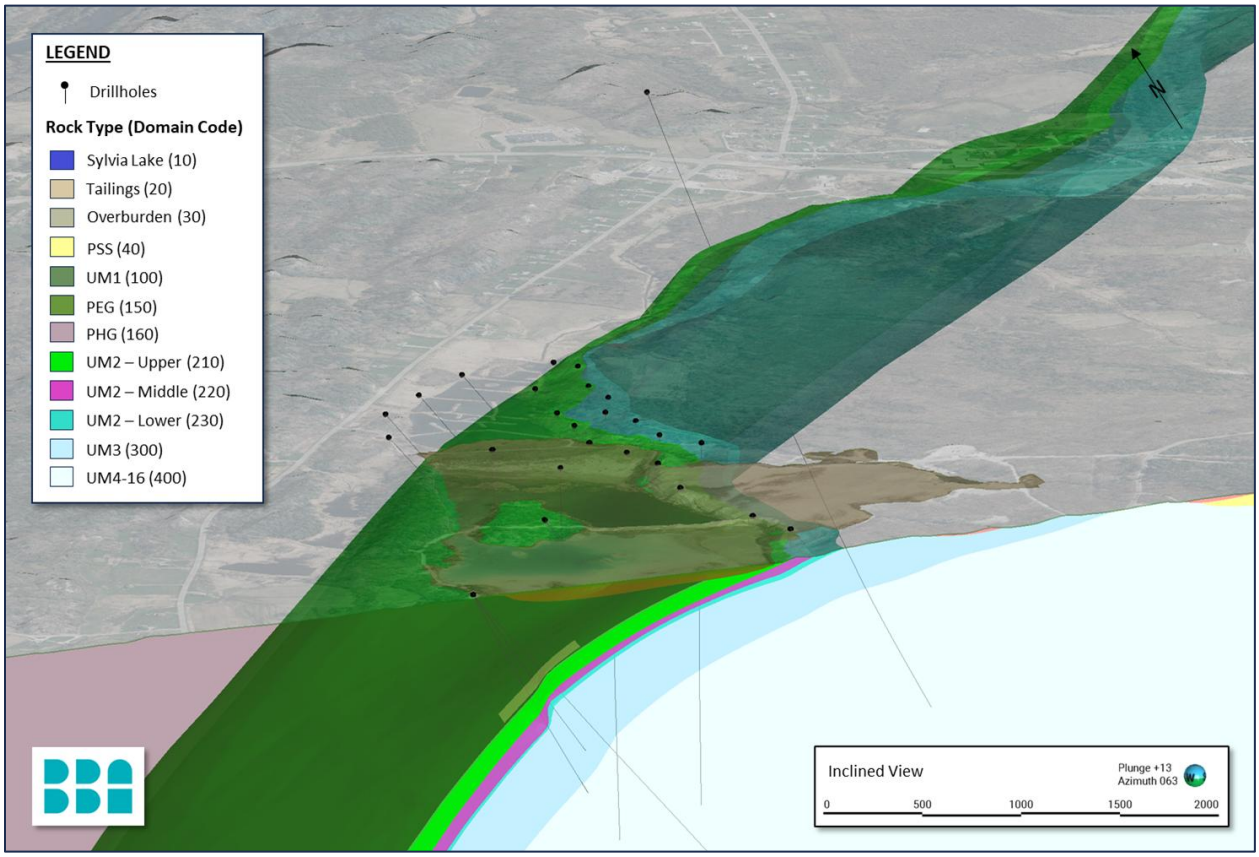
Table 14-16 summarizes the wireframe solids and associated volumes by domain. Figure 14-24 illustrates the model wireframes for each of the domains.

Table 14-16: Kilbourne deposit wireframe volume to block model volume summary

Domain	Rock Type	Wireframe Volume (ft^3)	Block Model Volume (ft^3)
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: Taylor et al., 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: Taylor et al., 2024

Figure 14-24: 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 14-17 summarizes the basic statistics for the assay intervals for each of the mineral domains on the Property.

Table 14-17: 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: Taylor et al., 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.

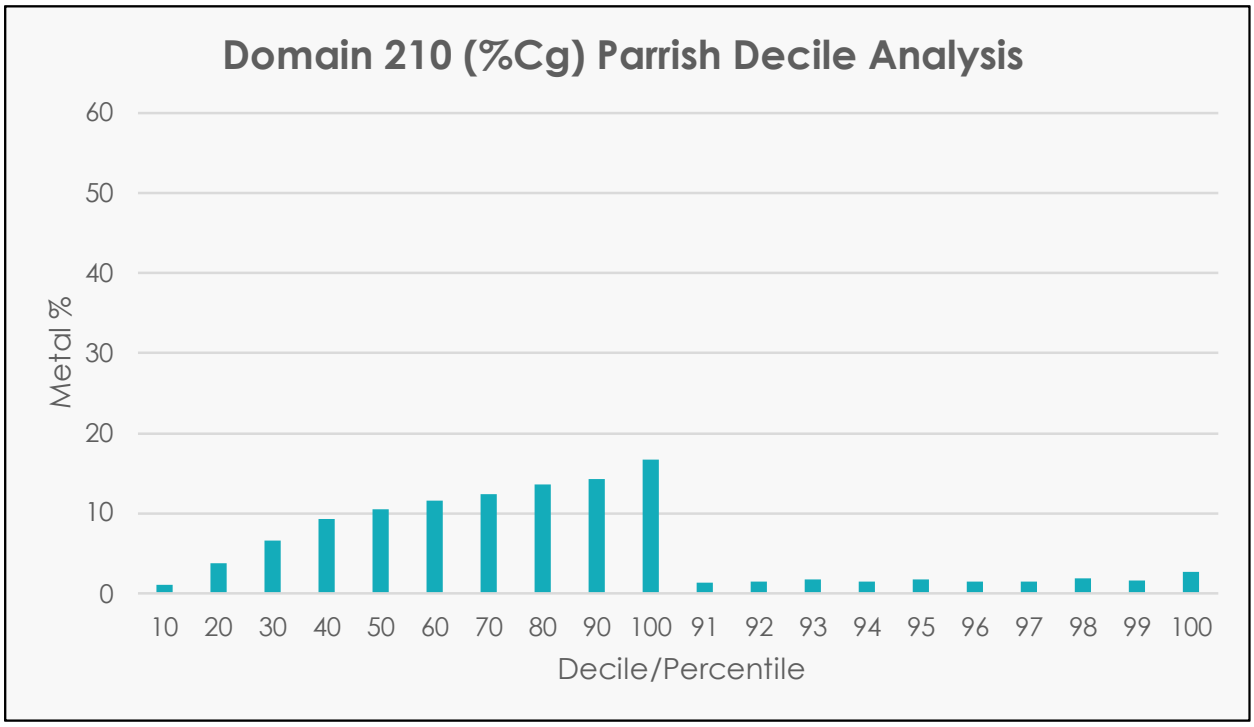
A review of the 3D spatial distribution of the capped samples was completed to determine if the samples were spatially close and if there was potential of a higher-grade sub-domain. This was not observed in any of the domains on the deposit.

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 14-18 summarizes the capping applied to each domain by the QP. Figure 14-25 and Figure 14-26 show the decile analysis and global top cut analysis performed by the QP, using domain 210 as an example.

Table 14-18: Kilbourne deposit grade capping summary

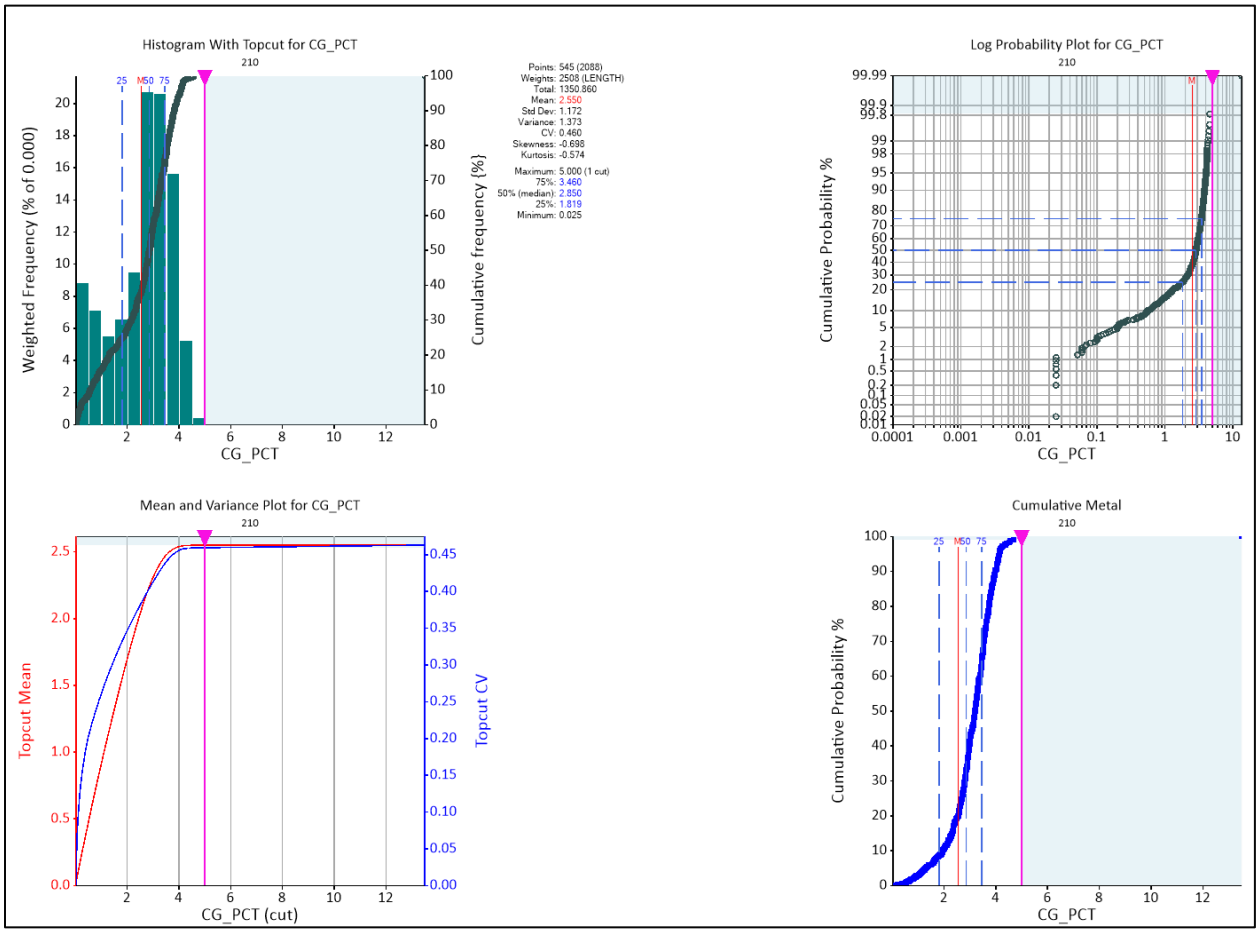
Domain	Element	Capping Value (%Cg)	Capped No. Samples	Uncapped Mean (%Cg)	Capped Mean (%Cg)	Metal Loss (%)
210	Cg (%)	5.00	1	2.55	2.55	0.1
220	Cg (%)	1.20	16	0.36	0.30	15.7
230	Cg (%)	6.00	2	2.49	2.47	0.5

Source: Taylor et al., 2024



Source: Taylor et al., 2024

Figure 14-25: Parrish decile analysis for domain 210



Source: Taylor et al., 2024

Figure 14-26: 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 14-19 summarizes the statistics for the drillholes after compositing.

Table 14-19: 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: Taylor et al., 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.

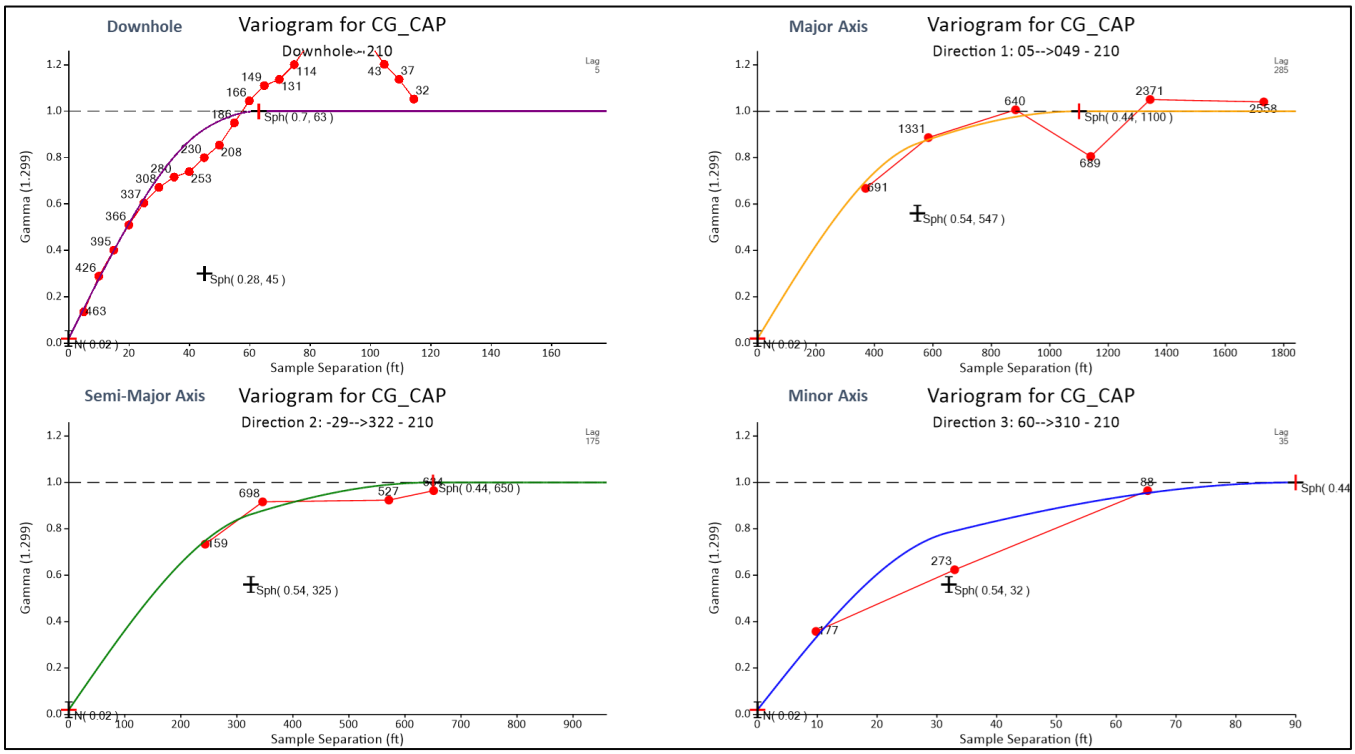
The variography for Kilbourne was determined using Snowden Supervisor software. Each 200 series domain was modeled using a downhole variogram to determine the nugget effect, and then a spherical pair-wise variogram was used to determine spatial continuity in the domain.

Table 14-20 summarizes the results of the variogram models for graphite. Figure 14-27 shows an example of the variography for domain 210.

Table 14-20: 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: Taylor et al., 2024



Source: Taylor et al., 2024

Figure 14-27: 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 14-21 summarizes details of the parent block model.

Table 14-21: 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: Taylor et al., 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 14-22 summarizes the search ellipse and rotations and Table 14-23 summarizes the interpolation criteria.

Table 14-22: 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: Taylor et al., 2024

Table 14-23: 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: Taylor et al., 2024

14.2.7 Resource Classification

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

- Prepared in accordance with NI 43-101;
- 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, political, 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. Mineral Resources that are not Mineral Reserves do not have demonstrated 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 14-24 summarizes the parameters used to develop the Kilbourne Pit constraints for a reasonable prospect of economic extraction.

Table 14-24: Kilbourne deposit pit constraint parameters

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

Source: Taylor et al., 2024

The pit-constrained Mineral Resource and in situ metal for the Kilbourne deposit is summarized in Table 14-25. Quantities and grades in the MRE are rounded to an appropriate number of significant figures to reflect that they are estimations.

Table 14-25: 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: Taylor et al., 2024

A Mineral Resource was reported 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, socio-economic, marketing, political, 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.

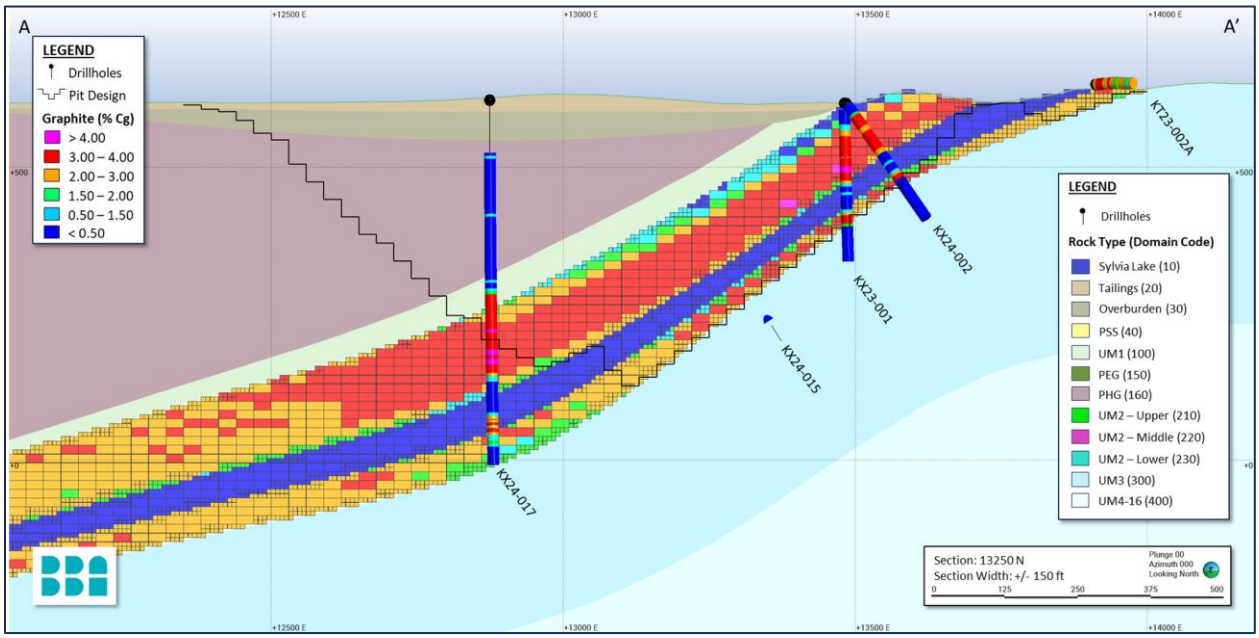
14.2.9.1 Visual Validation

The visual comparisons of ordinary kriging block model grades and composite drillholes show a reasonable correlation between the values (Figure 14-28 and Figure 14-29). 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: Taylor et al., 2024

Figure 14-28: Surface plan showing the optimized pit shell for the Kilbourne deposit



Source: Taylor et al., 2024

Figure 14-29: 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 ID², and NN models. Table 14-26 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 14-26: Kilbourne global composite to block model statistics comparison

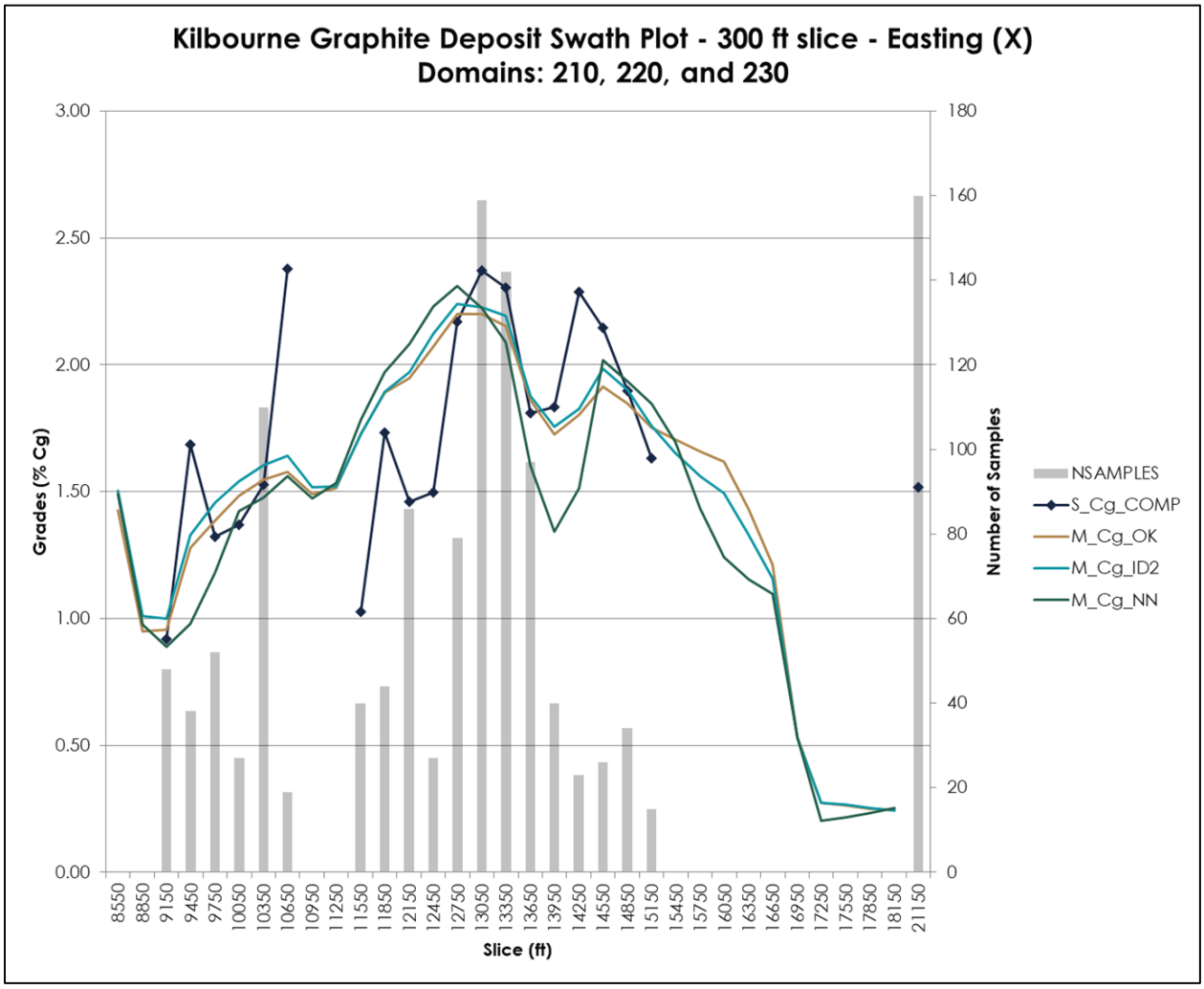
Domain	Element	Composite Mean	OK Mean	ID² Mean	NN Mean
210	Cg (%)	2.55	2.09	2.16	2.04
220	Cg (%)	0.30	0.23	0.24	0.26
230	Cg (%)	2.47	2.19	2.17	2.03

Source: Taylor et al., 2024

14.2.9.3 Swath Plots

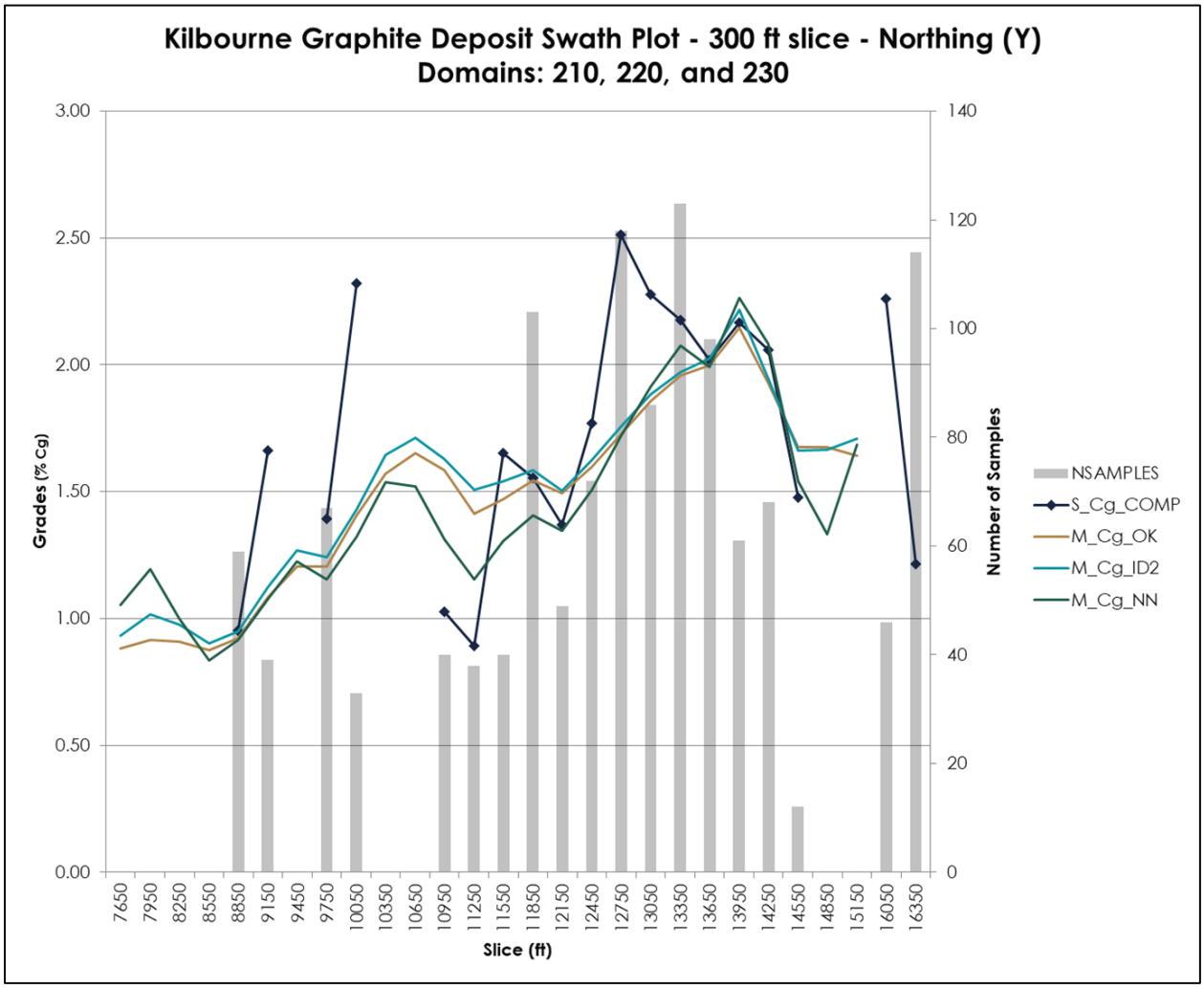
Figure 14-30 and Figure 14-31 display the comparison between the drillhole composites grades and the OK, ID² and NN estimates in a swath plot format. Comparisons were made using all blocks greater than 0.025% Cg for the 200 series domains.

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



Source: Taylor et al., 2024

Figure 14-30: Kilbourne deposit swath plot, 300 ft slice - easting (X)



Source: Taylor et al., 2024

Figure 14-31: 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

NI 43-101 provides that the terms "Mineral Reserve", "Probable Mineral Reserve" and "Proven Mineral Reserve" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended.

A Mineral Reserve is the economically mineable portion of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses that may occur when the material is mined or extracted. Mineral Reserves are defined by studies at Pre-Feasibility or Feasibility level studies, as appropriate, which include the application of modifying factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

15.1 Zinc

As noted above, Mineral Reserves must be defined by studies at Pre-Feasibility or Feasibility level. As the ESM Zinc Operation portion of this Technical Report has been prepared to a PEA level only, no Mineral Reserves are reported for the ESM Zinc Operations.

This PEA is based on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that this PEA will be realized.

15.2 Graphite

As noted above, Mineral Reserves must be defined by studies at Pre-Feasibility or Feasibility level. As the Kilbourne Graphite Mine Plan portion of this Technical Report has been prepared to a PEA level only, no Mineral Reserves are reported for the Kilbourne Graphite Project.

This PEA is based on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that this PEA will be realized.

16. Mining Methods

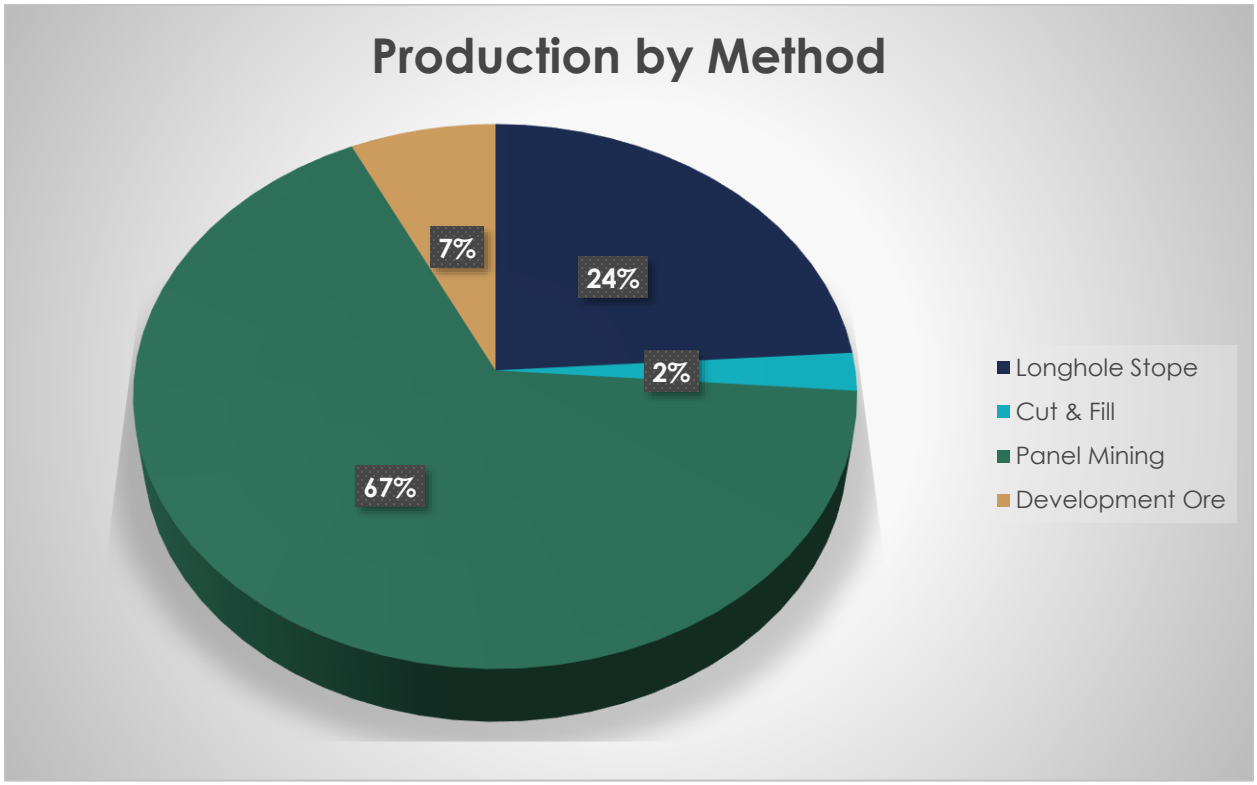
ESM expects to have three zinc production areas over the life of mine:

- #4 Mine;
- #2 Mine;
- Turnpike East & Turnpike West (open pits).

16.1 Zinc Underground

The Turnpike open pits and #2 Mine production are included as part of the overall mine plan. Surface operations utilize a separate surface fleet and crew. The #2 and #4 Mines are fully connected underground sharing staff, mining crews, utilities, ventilation circuit, and production fleet. The mine plan was balanced for both underground mines and the surface open pit all sending mineralized material to the mill in tandem despite being operationally distinct areas.

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 2025, the overall mine life is 6 years. Figure 16-1 outlines a summary of underground mining methods used at ESM.



Source: Modified from Warren et al., 2021

Figure 16-1: Mine production by method

The ESM Zinc Operations are accessed from surface via #4 Shaft, and all mineralized material and some waste rock are 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 taken as panel retreat. The Mineral Resources for the LOM are based on the Mineral Resource Estimate as stated in Item 14 of this report. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that this mine design and schedule optimization process will be realized.

For the purposes of this report, the LOM, as designed, starts in January of 2026. The remainder of production for 2025 is based on short-range projections. All tables in this Item reflect the combination of the short-range projection and long-range plan.

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

Mining Method	Diluted Tons (kt)	Percent of LOM Plan
Development Muck	297	7%
PM	2,774	67%
C&F	93	2%
LRS	996	24%
Total	**4,160**	**100%**

Source: Modified from Taylor et al., 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;
- N2D.

Deswik version 2024.2 Stope Optimizer shapes and development designs were created for the remaining mining zones:

- American;
- Cal-Marble;
- Fowler;
- NE Fowler;
- Sylvia Lake;
- Turnpike (Underground).

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

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.

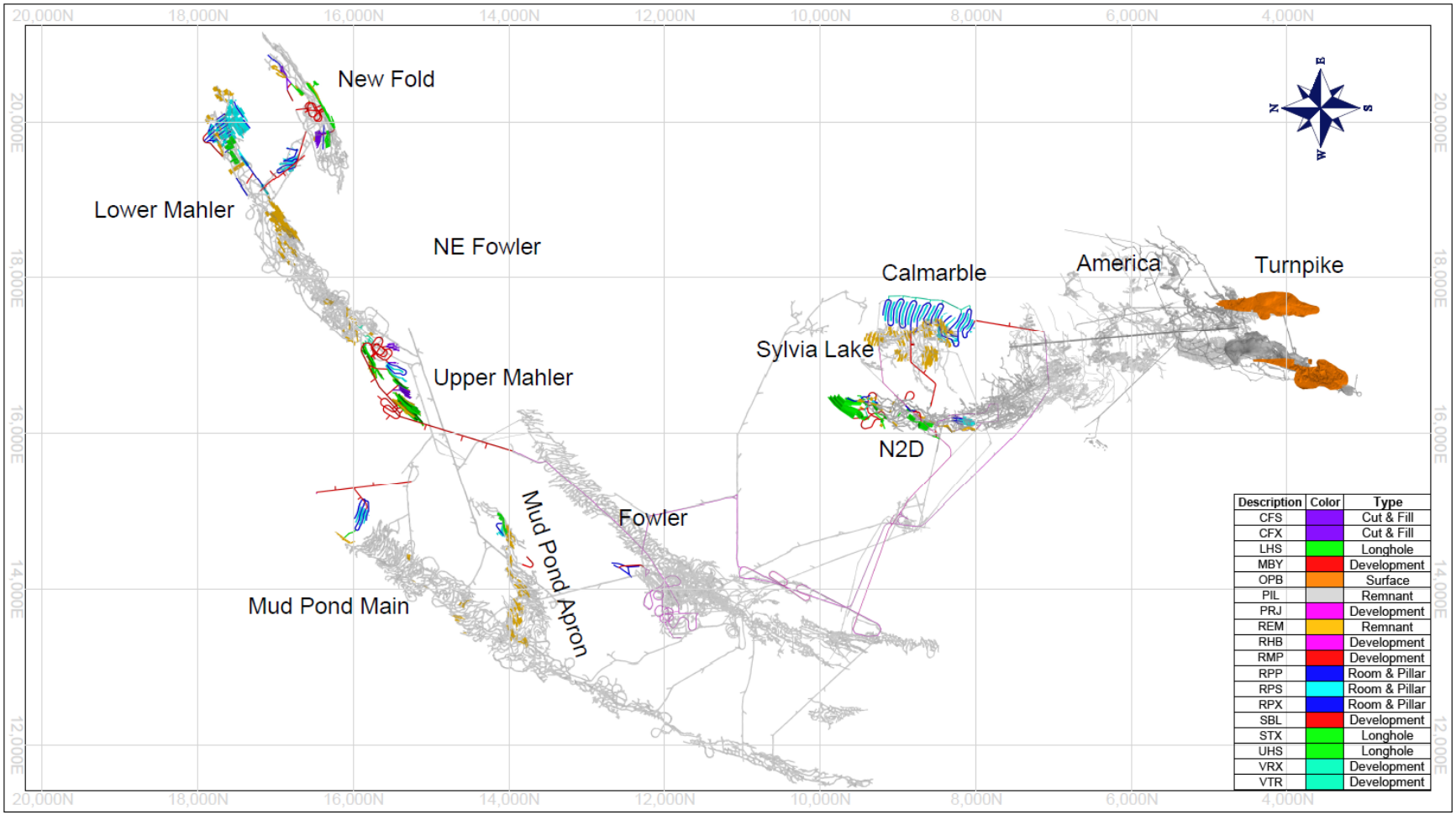


Figure 16-2: Mining zones in the LOM

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 that include dilution around these blocks.
- Query the production stope shapes for in situ tonnage and grade data and compare 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.

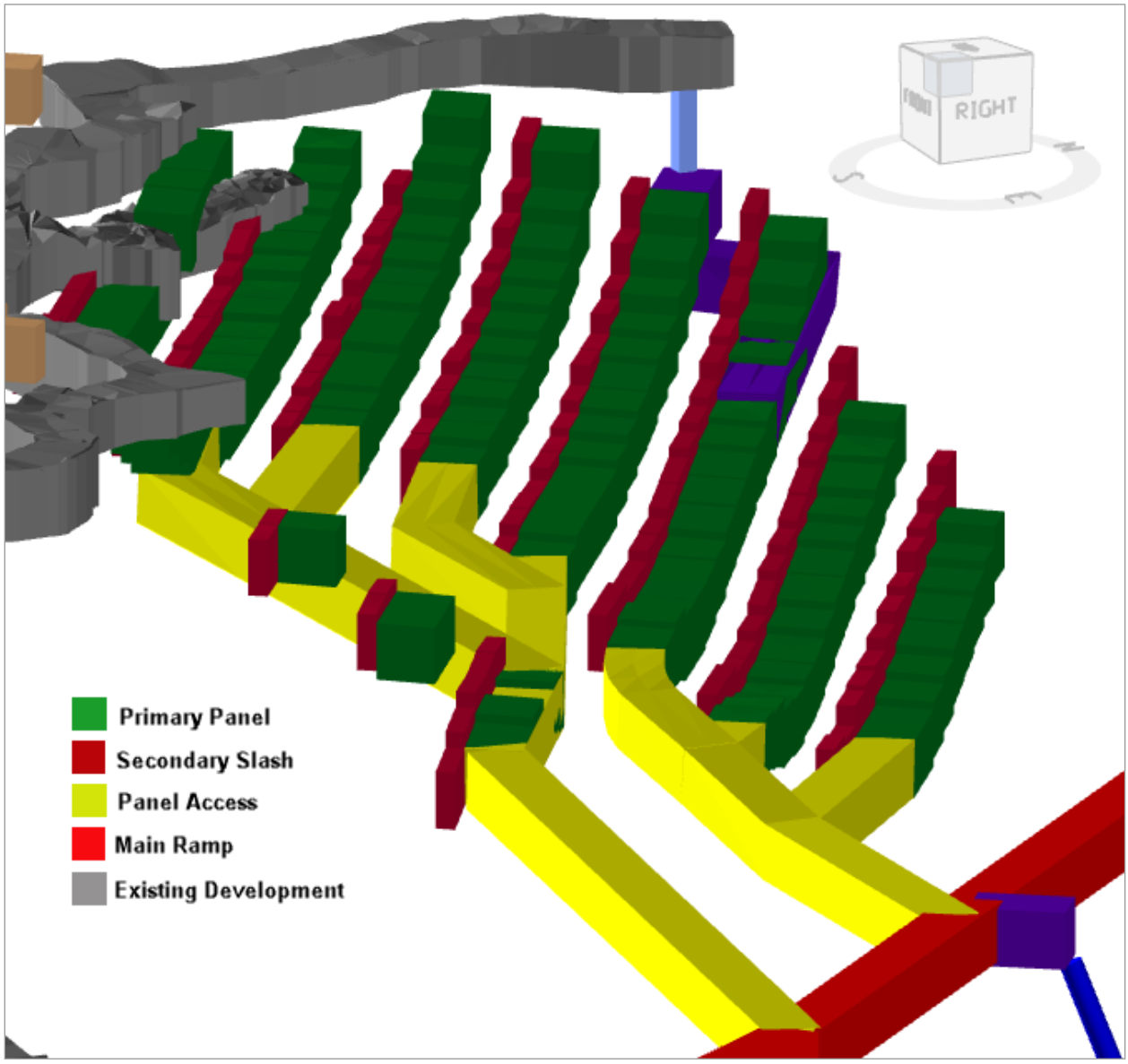
PM, with Primary and secondary cuts, is the principal mining method used at ESM. The second most common method is LRS. C&F is used where conditions are not suitable for LRS. In areas where the geometry of mineralization is simple and directional, normal development activities will be designed in mineralization.

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



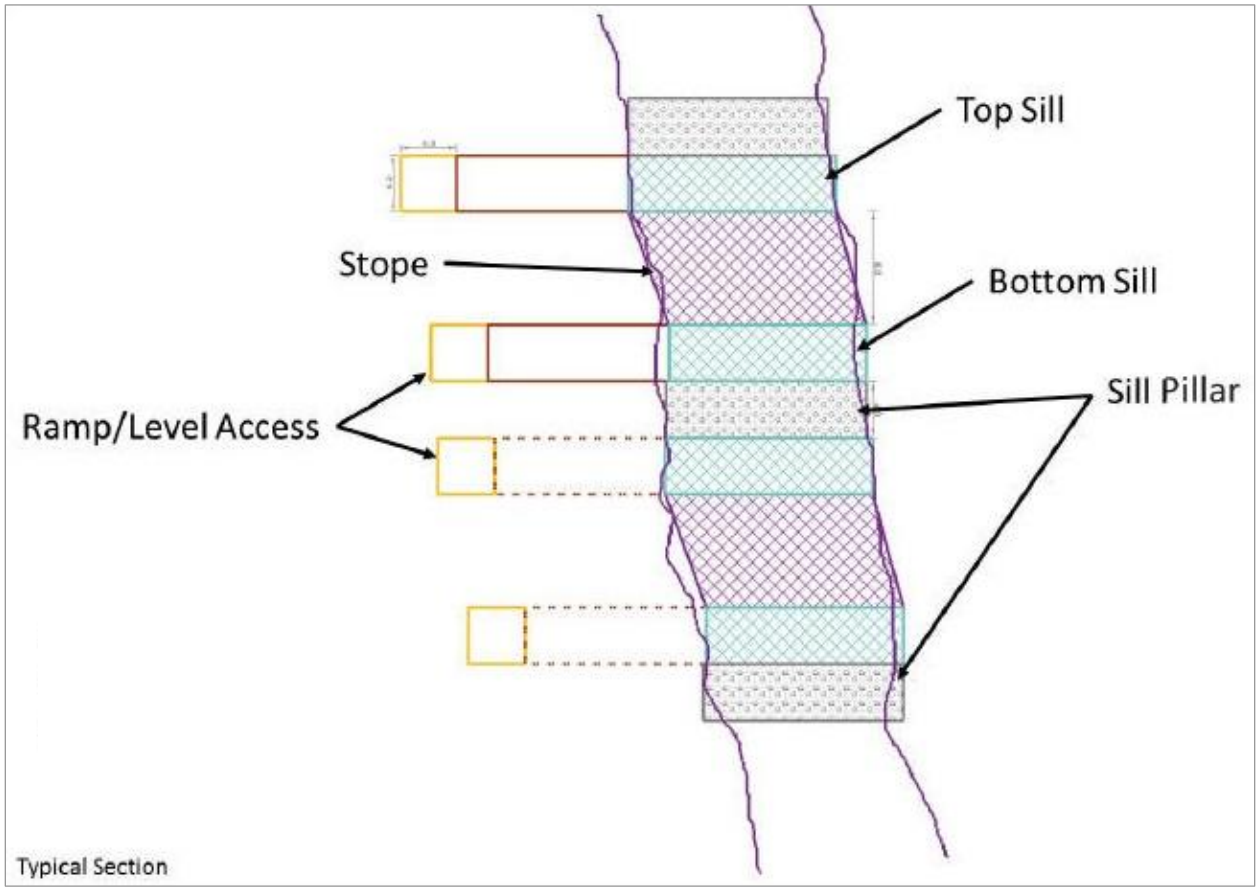
Source: Jackleg Consultants 2020

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.

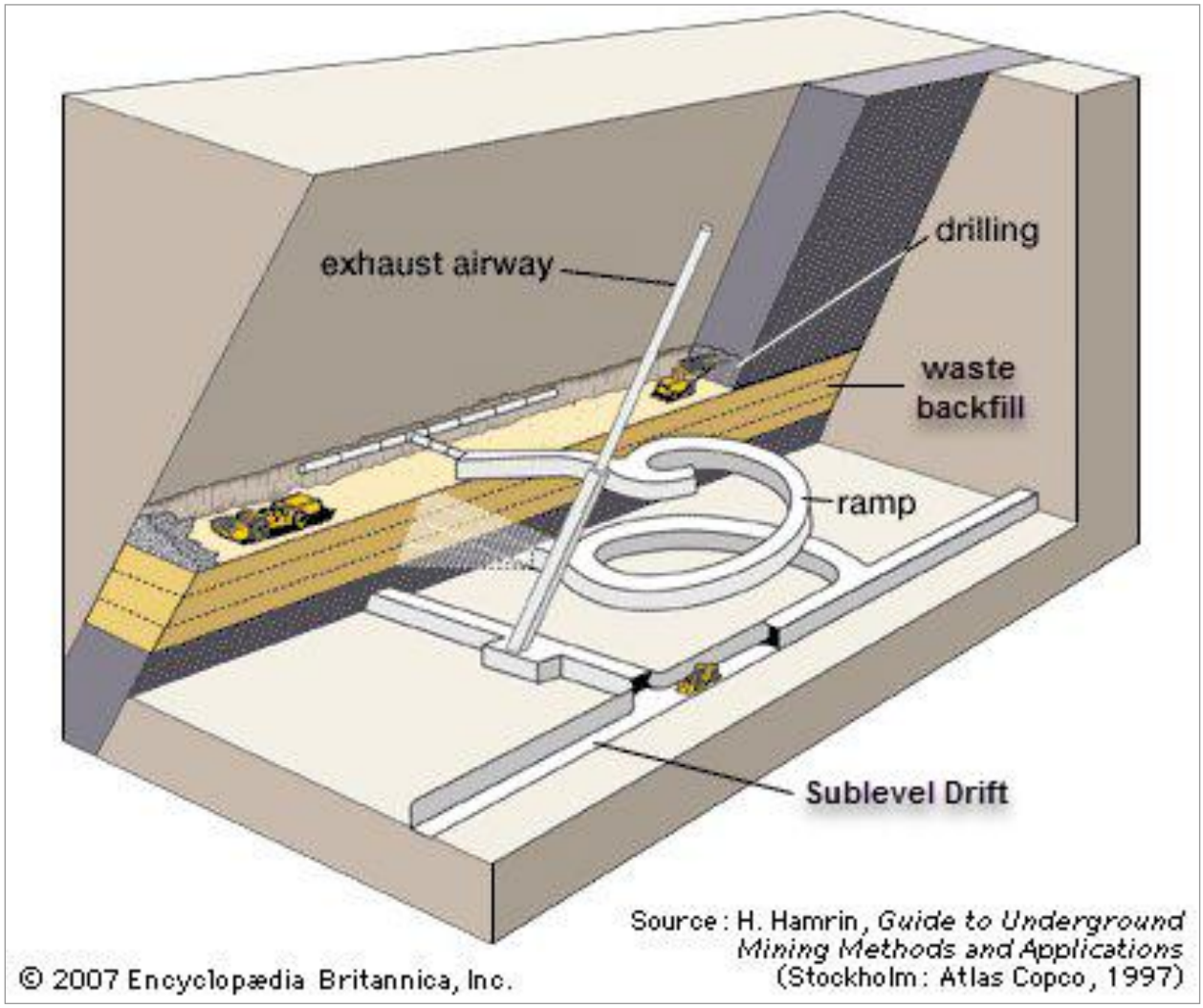


Source: Jackleg Consulting 2020

Figure 16-5: Typical LRS with sill pillar

LRS is used in Mahler, New Fold, N2D, 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.

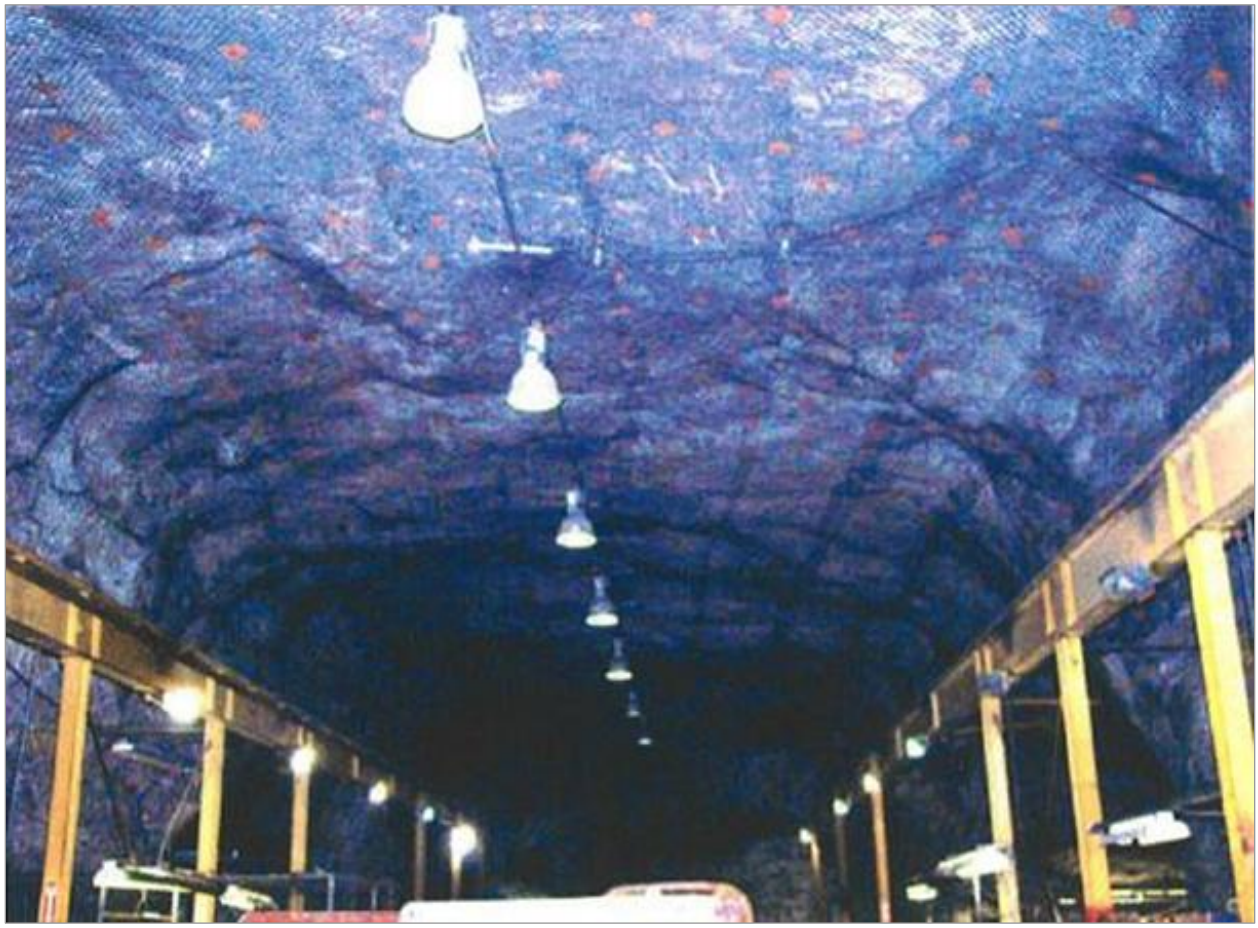


Source: Atlas Copco 1997

Figure 16-6: Typical C&F

16.1.4 Geotechnical Parameters

Rock quality at ESM is generally considered to be fair 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). More recently, observations by Chase Barnard (Barnard & Kalenchuk, 2025) provided the basis for the rock mass characterizations for Lower Mahler (Table 16-2).



Source: Itasca 2005

Figure 16-7: 2500 level workshop back conditions

Table 16-2: Rock mass characterizations for Lower Mahler

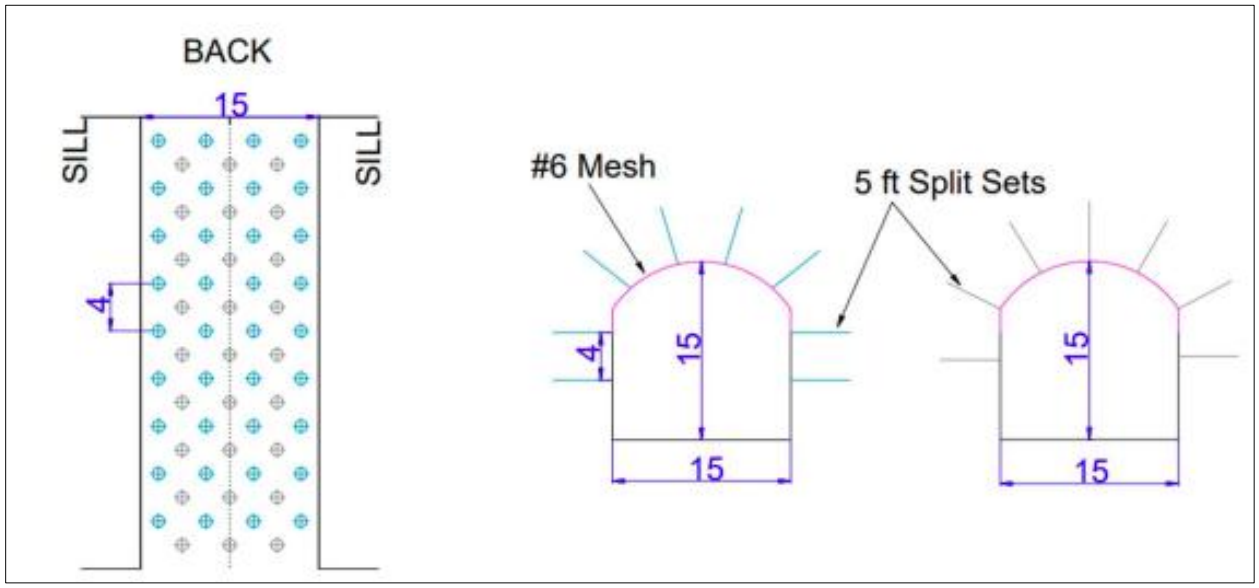
Unit	RQD (%)	Jn	Jr	Ja	Q'
Dolomite Marble	90–100	3	2–3	1–2	57.1
Sulfides (Upper Bound)	100	4	4.5	6	18.8
Sulfides (Lower Bound)	100	4	1.5	2	6.3

Source: Barnard & Kalenchuk 2025

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 2025, FS-39 split set bolts with a nominal diameter of 1.5 in (39 mm) in lengths of 60 in or 72 in, along with 6-gauge welded wire mesh, are the recommended primary ground support in all deposits (except New Fold) with support extending across the back (Figure 16-8). In New Fold, the recommended primary ground support is 8 ft long all thread bolts on a 4-ft 4-dice pattern across the back and down the ribs. Where necessary, secondary support typically consists of 8 ft or 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.



Source: Modified from Taylor et al., 2024

Figure 16-8: Ground support for typical ground

16.1.5 Stope Design Parameters

Deswik.SO version 2024.2 software was used to create all the mineable stope shapes in the LOM design. Stope design criteria are summarized in Table 16-3.

Table 16-3: Production stope design criteria

Mine Method	Minimum Stope Width (ft)	Stope Height (ft)	Stope Length (ft)	Dip (°)
C&F	15	16	N/A	40–90
PM - Primary	15	16	N/A	N/A
PM - Secondary	5	15	N/A	N/A
LRS	15	60	Max 150	50–90

Source: Modified from Taylor et al., 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 of 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. This was done as part of the stope optimization process. 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 16-4.

Table 16-4: Overbreak dilution parameters

Typical Profiles	Unit	C&F	PM - Primary	PM - Secondary	LRS w/Crown Pillar	Back-stope
Height	ft	15	15	15	50	60
Width (minimum)	ft	15	15	5	10	10
Footwall Overbreak	ft	0.5	0.5	0	2	2
Hanging Wall Overbreak	ft	0.5	0	0.5	2	2

Source: Modified from Taylor et al., 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 16-5.

Table 16-5: Cut-off grade parameters

Parameter	Unit	Value
Zn Price	$/lb	1.30
Mill Recovery	%	96.4
TC / RC / Transport	$/ton milled	35.13
Payable Metal	%	85
Royalties	%	0.3
Operating Costs	$/ton milled	86.73
Cut-off	% Zn	5.5
Incremental Cut-off	% Zn	2.0

Source: Taylor et al., 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 16-6 and Table 16-1 outline the diluted and recoverable mine plan mineralized material tons used for mine planning purposes by zone and extraction method.

Table 16-6: Tons contained in the LOM plan by zone

Zone	Diluted Tons (kton)	Diluted Grade (% Zn)
American	243	6.2
Cal-Marble	232	6.5
Fowler	60	6.1
Lower Mahler	1,074	8.9
Mud Pond Apron	160	7.1
Mud Pond Main	409	6.5
N2D	389	6.6
Northeast Fowler	265	5.7
New Fold	528	9.5
Sylvia Lake	200	5.9
Turnpike	123	6.0
Upper Mahler	476	6.5
Total	**4,160**	**7.4**

Source: Modified from Taylor et al., 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 16 ft by 16 ft profile.

16.1.10 Production Rate Selection

The ESM mine plan had been designed to ramp up to 1,400 tons/d from the #4 Mine in Year 1 of production and then to a sustained maximum of 1,700 tons/d from the #4 Mine. Ramp up was successfully completed. In 2025, ESM began a ramp up of the #2 Mine starting with 250 tons/d and moving to 1,000 tons/d by 2028. 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 2024.2 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		
Description	**Unit**	**Rate**
Lateral Development		
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
Stoping		
LRS	ton/day	350
Back-stope – Longhole	ton/day	350
C&F	ton/day	150
PM – Primary	ton/day	250
PM – Secondary	ton/day	100

Source: Taylor et al., 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.

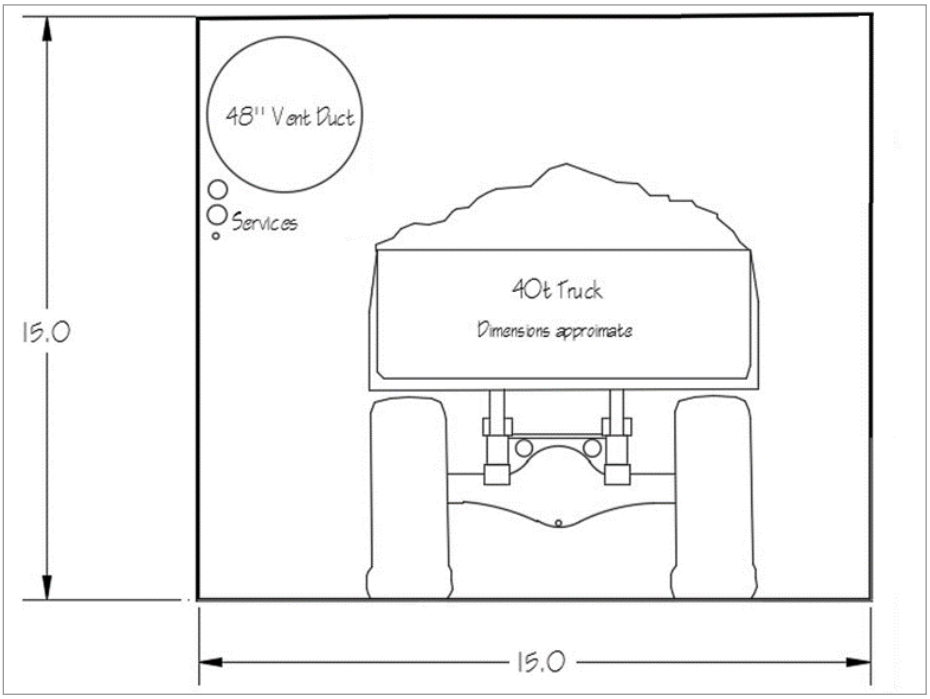
PM Primary and Secondary zones are planned from a top-down or bottom-up fashion depending on the direction of development in the zone. Access drifts are driven from the main ramp to the start of each Primary panel drift. A Primary drift is driven at full size to the end of the deposit. A Secondary slash 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 16 ft x 16 ft square profile to accommodate fully loaded 40-ton and 45-ton haul trucks and 48-in 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-in 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

Once mucking and scaling are complete, ground support is installed mechanically using a bolter or manually by experienced operators with jacklegs. In access development, typical ground support includes 5 ft or 6 ft split set bolts installed in the back and walls at a spacing of 4 ft x 4 ft. Welded wire mesh is installed in all ground conditions. In larger intersections, additional support may be provided using #7 all thread bolts, cable bolts or a combination of both.

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

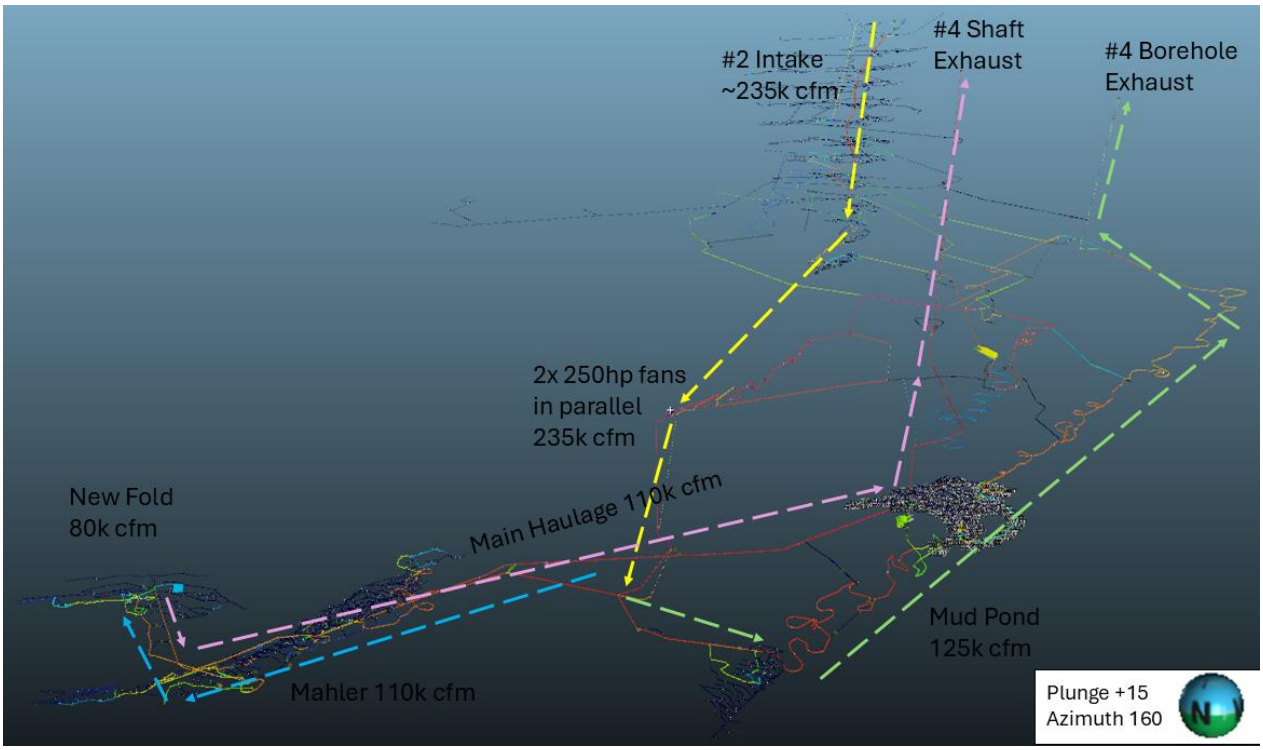
In 2016, the ESM ventilation network was modeled using Ventsim® Visual software by Practical Mining LLC (Practical Mining). The ventilation simulation model is routinely calibrated, verified and updated as mine activity changes.

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 to support the current circuit (Figure 16-10). After the installation of the ventilation doors, an intake borehole will need to be added in the future in order to support fleet requirements. Trade-off studies are expected to be run in 2026 to determine the best course of action for ventilation upgrades.



Source: Taylor et al., 2024

Figure 16-10: LOM ventilation installations (conceptual, not to scale); view from above to the SE

Ventilation Expansion

The current ventilation system will have to be augmented with an additional large borehole for fresh air intake to support the increased production at depth from the #2 and #4 Mines. To meet future equipment requirements and improve de-gassing an 18 ft diameter borehole from surface to the Mahler/New Fold zones is currently assumed—it would increase the available air volume to an estimated 458 kcfm. This will support the larger fleet required to maintain production at depth through end of mine life. Trade-off studies are expected to be run in 2026 to determine the best course of action for ventilation upgrades.

16.1.14.2 Mine Air Heating

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

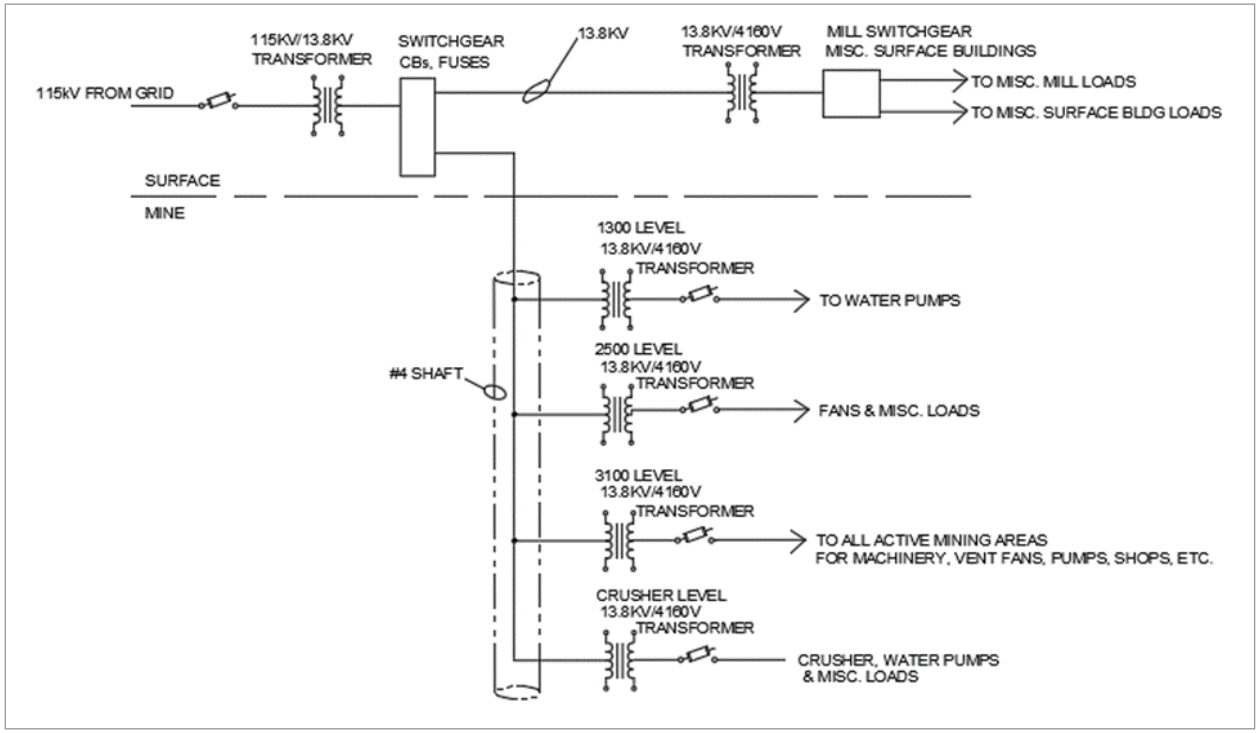
16.1.14.3 Electrical Power

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: Taylor et al., 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-inch 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-in stainless steel 314 pipe within the #4 Shaft and distributed in 2-in 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's 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 of 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 in to 6 in 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/ton mined for development and 0.7 lb/ton mined 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 efficiency 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 tons/6 yd) Atlas Copco ST 1030	1
LHD (10 tons/6 yd) Epiroc ST 1030	4
LHD (10 tons/6 yd) Sandvik LH 410	1
LHD (7 tons/4 yd) MTI 650	1
LHD (3 tons/2.5 yd) MTI 270	1
Haulage Truck – 40 tonne – Tamrock 40 D	3
Haulage Truck – 42 tonne – Epiroc MT 42	3
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: Modified from Taylor et al., 2024

Haulage requirements for LHDs and trucks were estimated for mineralized material, waste and backfill. Mineralized material is hauled to a stockpile, loaded into trucks or dropped into ore 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.

To support planned production increases at increasing depth, additional 42-tonne haul trucks and LHDs will need to be added as the mine progresses. An additional four trucks and six LHDs are expected to be needed over the next 5 years as #2 areas become active and #4 haulage distances increase. An additional jumbo drill and an additional bolter will be needed as well.

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 87 people at the current full production rate for underground of 2,250 tons/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	12
Miner 3 (Loader & Truck Operators)	Hourly	5 x 2	19
Miner 4 (Services, Equipment Operators)	Hourly	5 x 2	14
Subtotal – Mining Operations			**67**
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**
Mine Maintenance			
Maintenance Manager	Staff	5 x 2	1
Maintenance General Foreman	Staff	5 x 2	1
Electrical General Foreman	Staff	5 x 2	1
Maintenance Clerk	Staff	5 x 2	1
Maintenance Supervisor	Hourly	5 x 2	2
Heavy Duty Mechanic	Hourly	5 x 2	11
Electrician	Hourly	5 x 2	3
Subtotal – Mine Maintenance			**20**
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	2
Surveyor	Staff	5 x 2	2
Chief Geologist/Engineer	Staff	5 x 2	1
Geologist	Staff	5 x 2	2
Subtotal Technical Services			**11**
Grand Total			**107**

Source: Modified from Taylor et al., 2024

16.1.17 Mine Production Schedule

Mine scheduling for the ESM Project was done internally. The schedule seeks to produce consistent pounds 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 underground mine production statistics from 2026 are provided in Table 16-10. Annual production statistics for 2025 are included in Table 16-11.

Table 16-10: Annual mineralized material

Item	Total	2026	2027	2028	2029	2030	2031	2032
Mineralized Material Tons Mined (kton)	**3,632**	511	558	552	658	690	663	0
Zinc Grade	**7.4**	8.4	7.9	7.3	7.1	7.1	7.1	0.0
Waste Tons Mined (kton)	**676**	213	116	118	75	79	76	0
Contained Zinc (M lb)	**540**	85	88	81	94	98	94	0

Source: Modified from Taylor et al., 2024

The 2025 mine production was estimated from short-range projections. There was no open pit production in 2025.

Table 16-11: Projected production for 2025

Item	Unit	Total	2025
Mineralized Material Tons Mined	kton	**529**	529
Zinc Grade	%	**7.6**	7.6
Waste Tons Mined	kton	**64**	64
Contained Zinc	M lb	**80**	80

Source: Modified from Taylor et al., 2024

16.1.18 Mine Development Schedule

The development schedule is based on estimated equivalent footage for projected waste tons.

Annual development footage is summarized in Table 16-12.

Table 16-12: Annual development schedule

Development Schedule	Unit	Total	2026	2027	2028	2029	2030	2031	2032
Internal Waste Development	ft	**31,061**	9,771	5,337	5,404	3,452	3,619	3,478	0
Capital Development	ft	**34,204**	8,516	3,497	6,023	5,291	5,547	5,331	0
Total Waste Development	ft	**65,265**	18,287	8,834	11,427	8,743	9,165	8,809	0

Source: Modified from Taylor et al., 2024

16.1.19 Projected Tailings Production

Annual estimated tailings produced from underground mineralized material tons processed.

Table 16-13 Annual tailings production

Item	Unit	Total	2025	2026	2027	2028	2029	2030	2031
Tailings Tons Produced	kton	**3,666**	464	443	488	488	583	611	589

16.2 Zinc Open Pit

The Turnpike resource will be mined via two small open pits, Turnpike East and Turnpike West. Both pits will be mined at the same time.

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 Geoscience 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 16-14 and Table 16-15). The pit designs, in the Blackwell & Peacock (2020) report, are based on a previous block model. The parameters derived from the Knight Piésold study (Blackwell & Peacock, 2020) have been reviewed, verified and determined appropriate for use in this study by the QP. The generalized slope angles by modeled lithology, as presented in Table 16-15, have been adopted pending further geotechnical work.

Table 16-14: 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
						Bench Face Angle (BFA) (°)	Effective Bench Height (ft)(3)	Bench Width (ft)	Inter-ramp Angle (IRA)			Max. Inter-ramp Slope Height (ft)	Expected OSA Performance Based on Precedent Practice	
									From Bench Configuration (°)	Achievable Based on Kinematics	Achievable Based on LE			
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 16-15: 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: Taylor et al., 2024

The QP has reviewed the Knight Piesold report (Blackwell & Peacock, 2020), which is a scoping level study, and agrees with its recommendation for additional data collection to improve characterization of structural features.

16.2.3 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 16-16.

Table 16-16: Cut-off value assumptions

Mining Factors	Unit	Open Pit
Mining Dilution	%	10
Mining Recovery	%	100
Operating Costs		
Mining Cost for Mineralization	$/ton	4.60
Mining Cost for Waste	$/ton	3.50
Mining Cost for Overburden	$/ton	2.00
Processing Cost for Mineralization	$/ton	11.00
G&A Cost for Mineralization	$/ton	0.00
Processing Recovery		
Zinc	%	96

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

Source: Taylor et al., 2024

16.2.4 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.

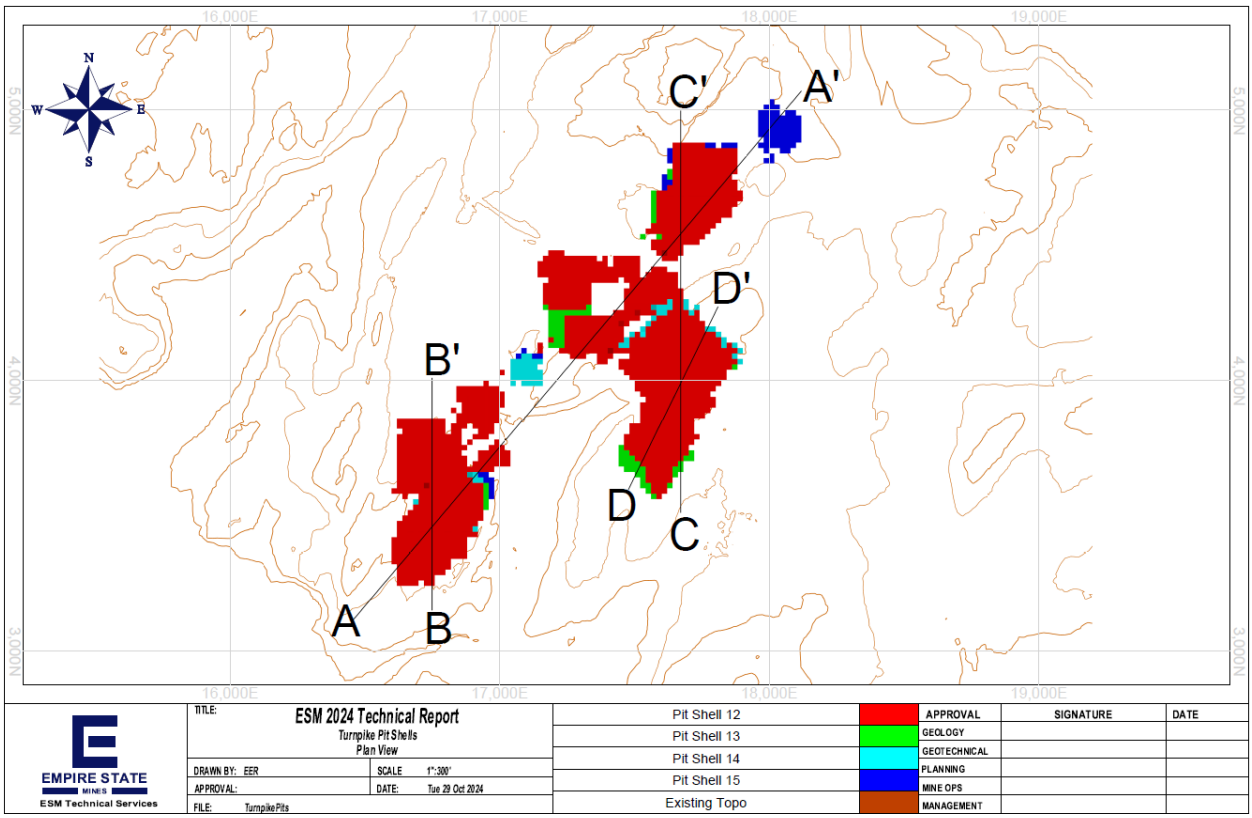
Mining recovery and dilution were accounted for by using a regularized block model. The estimated 10% dilution was not applied within the pit optimization, only to the cut-off grade calculation.

16.2.5 Pit Limit Analysis

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 shell analysis, pit design, and mine schedule based on inputs from ESM. The QP has reviewed these procedures and results and determined they are appropriate for use in this Technical Report.

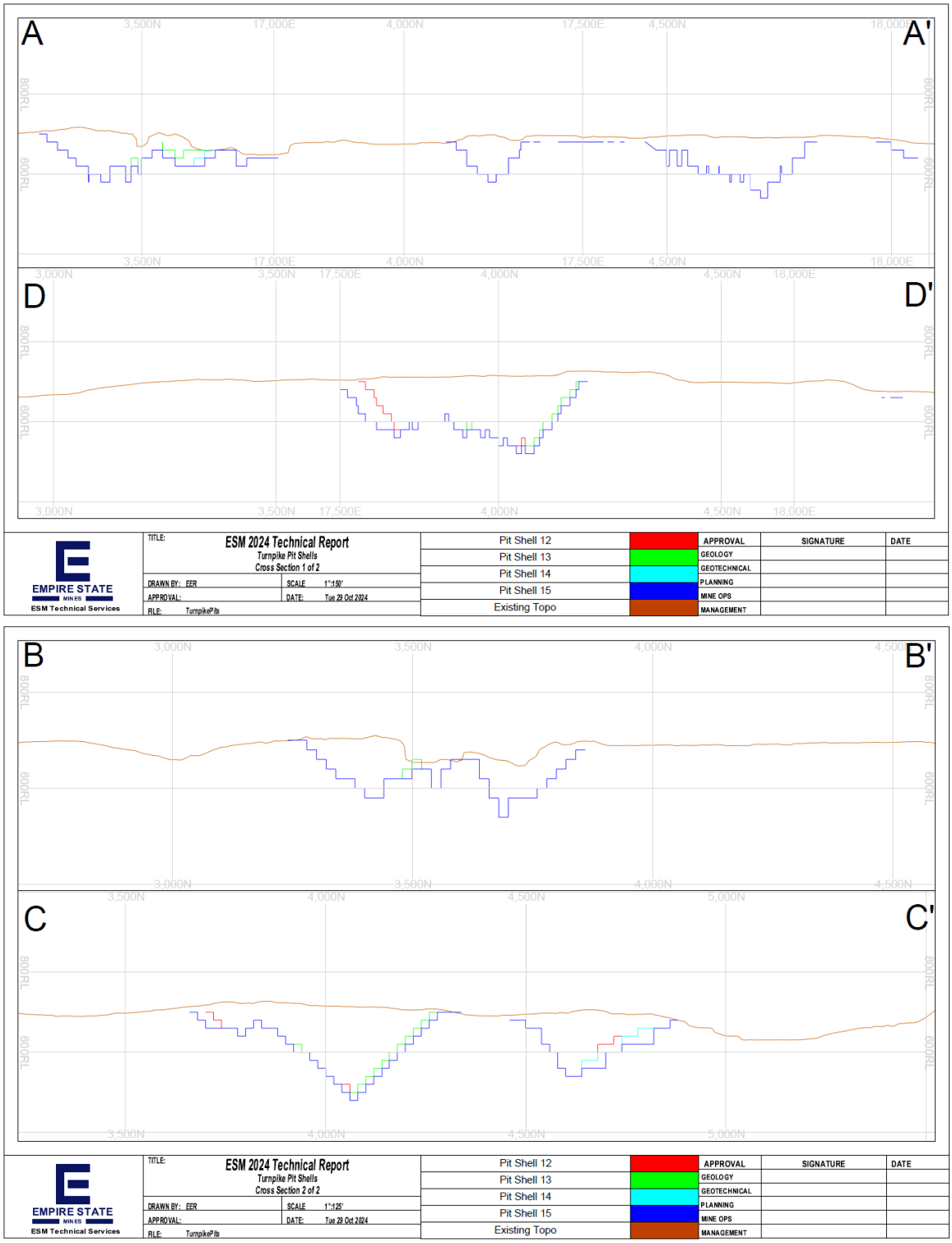
Further UG mining is not planned under the open pit zones. The block model was depleted of the existing UG workings. Therefore, the pit shell analysis did not consider any further influence from UG mining (Figure 16-12 and Figure 16-13).

The economic inputs required to run the pit limit analysis 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: Taylor et al., 2024

Figure 16-12: Plan view optimization shells (with cross-section locations)



Source: Taylor et al., 2024

Figure 16-13: 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 16-17: Pit shell optimization results

											Percent of Max NPV		
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	Best Case	Worst Case	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:

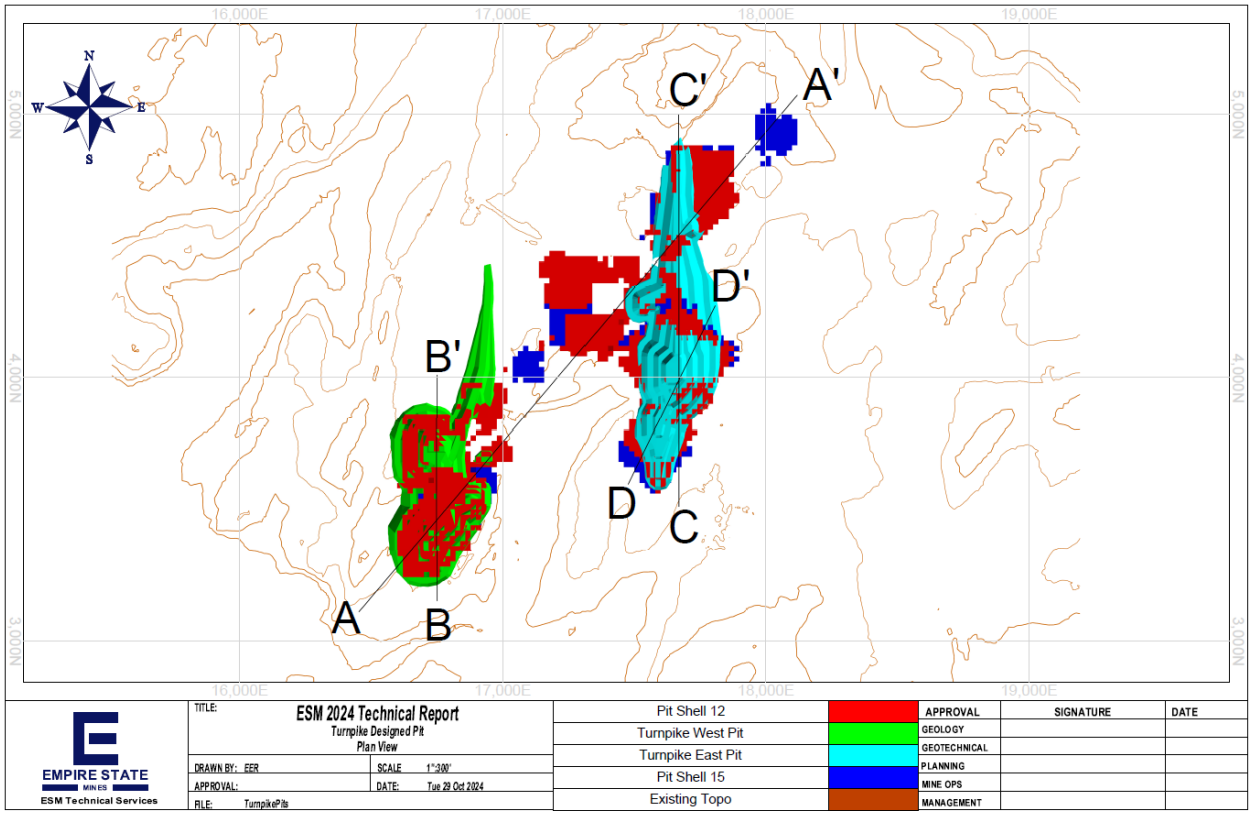
Values are expressed in US dollars
Numbers may not add due to rounding.
Inferred resources above 0.6% Zn.
Revenue factor is applied to base case price starting 0.3 to1.2 (0.05RF intervals).
No extra mining dilution and mining recovery are included as regularized BM is used.
Based on the resource block 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: Taylor et al., 2024

16.2.6 Pit Design

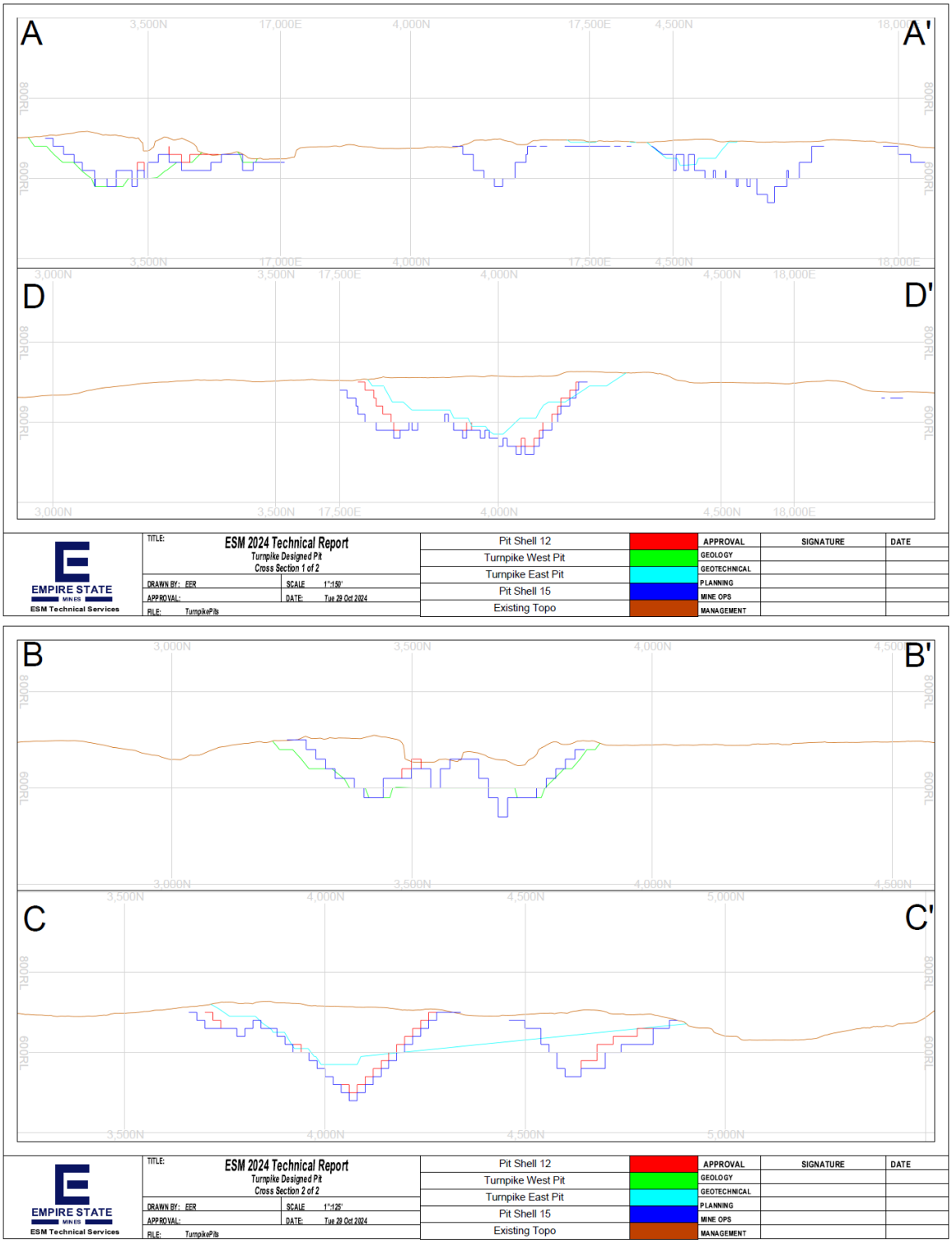
Conceptual pits were designed based on the selected pit optimization shell as described above. Design criteria were (Figure 16-14 and Figure 16-15):

- 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: Taylor et al., 2024

Figure 16-14: Open pit designs



Source: Taylor et al., 2024

Figure 16-15: Cross-section of design and shell

Indicative tons and diluted grades contained within the conceptual pit designs (Figure 16-16) are presented in Table 16-18.

Table 16-18: Open pit projected tons and grades

Zone	Mineralized Material (kton)	Zn (%)	Strip Ratio
Turnpike West Pit	200	3.37	3.1
Turnpike East Pit	199	2.97	3.8
Total	**399**	**3.17**	**3.4**
Pre-strip Waste	305	-	-
Internal Waste	1,059	-	-
Total Waste	**1,364**	-	-
Turnpike West Pit	**200**	**3.37**	**3.1**

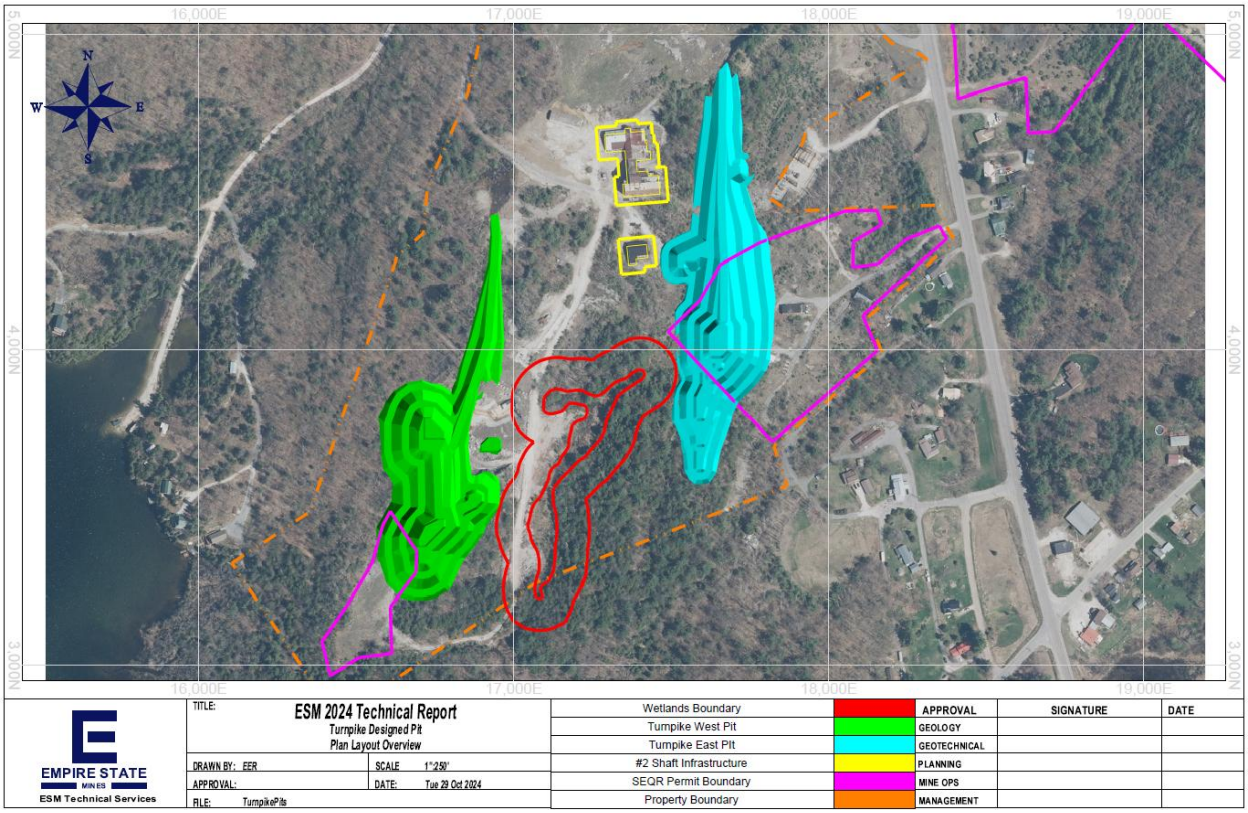
Source: Modified from Taylor et al., 2024

16.2.6.1 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: Taylor et al., 2024

Figure 16-16: Layout of open pit

16.2.7 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 to balance waste development in the underground zones. The pits will alternate production on a short-range basis as required.

16.2.7.1 Drill and Blast

The proposed drilling parameters for 20 ft bench heights are presented in Table 16-19. 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 16-19: 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	ton/ft^3	0.09
Powder Factor	lb/ton mined	0.46

Source: Taylor et al., 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 2,600 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.

16.2.7.2 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-ton 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 tons/h and trucks at 130 tons/h in mineralization and 170 tons/h in waste. Front-end loaders and trucks have been estimated to operate 3,130 h/y. Two trucks should be adequate to meet production. One front-end loader with a truck could stay permanently in waste. The second front-end loader with one truck could work exclusively in mineralization.

16.2.7.3 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.

16.2.8 Open Pit Equipment

The open pit contractor operations are projected to work on a 5-day, 10 h/d roster. One shift (day) is planned. Therefore 50 h/week are scheduled over 52 weeks per year for 2,600 h/year.

Based on the production schedule (Table 16-22), roster schedule, and equipment productivity estimates, the required equipment list is as shown in Table 16-20.

Table 16-20: Equipment estimate

Equipment	Y1	Y2	Y3	Y4
Trucks	1	2	2	2
Loaders	1	2	2	2
Drills	1	1	1	1
Graders	1	1	1	1
Water trucks	1	1	1	1
Dozers	1	1	1	1
Pickups	1	1	1	-

Source: Modified from Taylor et al., 2024

16.2.8.1 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.

16.2.9 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 16-21.

Table 16-21: Open pit labor and supervision

Labor	Y1	Y2	Y3	Y4
Mine Foreman	1	1	1	1
Drill Operator	1	1	1	1
Drill Helper	1	1	1	1
Blaster	1	1	1	1
Blaster Helper	2	2	1	1
Loader Operator	1	2	2	2
Haul Truck Operator	1	3	3	3
Dozer Operator	1	1	1	1
Water Truck Operator	1	1	1	1
Grader Operator	1	1	1	1
Mine Laborer	2	2	1	1
Mine Maintenance Foreman	1	1	1	1
Mechanic	1	1	1	1
Mechanic Heavy Equipment	2	2	2	2
Electrician	1	1	1	2
Serviceman	1	1	1	2
Maintenance Laborer	2	2	1	1
Total	**21**	**25**	**25**	**25**

Source: Modified from Taylor et al., 2024

16.2.10 Proposed Open Pit Production Schedule

The proposed open pit production schedule extends over a 4-year period in tandem with the underground operations. The expected production is summarized in Table 16-22. The pre-stripping is assumed to be completed before June of 2026.

Table 16-22: Conceptual open pit production schedule

Item	Total	2026	2027	2028	2029	2030
Mineralized Material Tons Mined (kton)	**399**	30	88	221	60	0
Zinc Grade	**3.2**	2.3	2.9	3.2	3.8	0
Waste Tons Mined (kton)	**1,059**	114	235	622	89	0
Contained Zinc (M lb)	**25**	1.4	5.1	14.3	4.5	0
Stripping Ratio	**2.7**	3.8	2.7	2.8	1.5	0

16.2.11 Projected Tailings Production

Annual estimated tailings produced from surface tons processed.

Table 16-23 Annual Tailings Production

Item	Unit	Total	2025	2026	2027	2028	2029	2030	2031
Tailings Tons Produced	kton	**379**	0	29	83	210	56	0	0

16.3 Graphite Open Pit

Industry standard open pit mining methods will be used to extract the material from the Kilbourne Pit. This method was selected considering the deposits' size, shape, orientation, and proximity to the surface.

Open pit mining will include conventional drilling and blasting with a combination of a backhoe type excavator and front-end loader type excavator loading material into haul trucks, which will haul the material from the bench to the crusher, ROM stockpile, overburden or waste stockpiling areas depending on the material type. Ancillary equipment includes dozers, graders, and various maintenance, support, service and utility vehicles.

The following sections outline the parameters and procedures used to perform the scoping level mine planning work for the Kilbourne Graphite Study at a proposed Concentrate Plant feed production rate of 1,700,000 t/y.

The Mine Plan presented in the following section is based on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that this Graphite Study will be realized.

16.3.1 General Parameters Used to Estimate In-pit Mineable Resources

The following section discusses the geological information used for the mine design. This information includes the topographic surface, the geological resource block model, and the material properties for mineralization, waste rock, and overburden.

16.3.1.1 Topographical Data

The mine design for the Graphite Study was carried out using a topographic surface derived from publicly available New York State LiDAR data and locally updated by Titan. The coverage of the DTM is sufficient to cover the entire Kilbourne Site area.

16.3.1.2 Geological Resource Block Model

The mine design for the Graphite Study is based on the Mineral Resource block model prepared for Kilbourne deposit, as presented in Item 14 of this report. The block model extents, dimensions, and rotation angles are also provided in that Item.

The 3D sub-blocked model is composed of parent blocks that are 30 ft x 30 ft x 15 ft high, sub-celled on a 7.5 ft x 7.5 ft x 7.5 ft pattern and contain only Inferred Mineral Resources. The Kilbourne deposit block model considered in this Graphite Study was imported into the Deswik software and regularized to 30 ft x 30 ft x 15 ft high.

16.3.1.3 Material Properties

In Situ Specific Gravity by Lithology

Specific Gravity is an important measurement that converts volumes modeled by the geologists into tonnages and contained tonnage of graphite. It is also used to estimate mine equipment requirements. The methodology used to estimate average specific gravity for each mineral domain to the block model is presented in Item 14.

Moisture Content

The moisture content reflects the amount of water present within the rock formation. It affects the estimation of haul truck requirements and must be considered during the payload calculations. The moisture content is also a contributing factor for the process water balance. No testwork data was available for validation of the moisture content.

The assumptions used for moisture content for mine planning are based on typical conditions for similar project types, as follows:

- 5% for rock and overburden material;
- 10% for the existing tailings material.

Swell Factor and Compaction

The swell factor reflects the increase in volume of the material from its in situ state to its state after it has been blasted and loaded into the haul trucks. The swell factor is an important parameter that is used to determine the loading and hauling equipment requirements, as well as the rock pile and stockpile designs.

The assumptions for the swell factor for mine planning are based on typical conditions for similar project types, as follows:

- 25% for overburden;
- 40% for rock.

No testwork data was available for validation of this number.

Material compaction is assumed to be 10% on waste dumps and 5% for overburden.

16.3.2 Mining Dilution and Mining Loss Factors

The Mineral Resources are based on the resource model with a 30 ft x 30 ft x 15 ft parent block size with sub-celling to 7.5 ft x 7.5 ft x 7.5 ft block size. For mine planning, these blocks have been regularized to a mining unit size of 30 ft x 30 ft x 15 ft, which accounts for planned open pit mine operating conditions. The regularized block size was selected to align with mining equipment, operating bench height, and selectivity.

By way of reblocking and regularizing the geological resource block model, mining dilution and mining loss have been incorporated into the mining block model. Table 16-24 tabulates the results.

Table 16-24: Impact of regularization of block model

Cut-off Grade	Dilution (%)	Dilution Grade (%Cg)	Mining Recovery (%)	Mining Loss (%)
1.5% Cg	5.3	0.26	86	14

16.3.3 Pit Limit Analysis

Potential economic pit limits were determined using Deswik mining planning software, which uses the pseudoflow algorithm. The algorithm progressively identifies potential economic blocks, taking into account waste stripping that results in a highest possible total value mined within the open pit shell, subject to the specified pit slope constraints.

The pit limit analysis was evaluated on the Graphite Study using Inferred Mineral Resources.

Input Parameters

A 3D geological block model and other economical and operational variables are used as inputs in the software. These variables include overall pit slope angle, mining costs, processing costs, concentrate sales price, metallurgical recovery, and other variables listed in Table 16-25. Although these parameters are not necessarily final, a reasonable degree of accuracy is required since the analysis is an iterative process. The input parameters used at the time of the pit limit analysis may not necessarily conform to those stated in the financial analysis.

Table 16-25 Pit limit analysis parameters

Parameter	Unit	Value
Resource Classification	n/a	Inferred only
Mineralization Domain	n/a	210: UM2 – Upper (High-Grade Cg) 220: UM2 – Middle (Low-Grade Cg) 230: UM2 – Lower (High-Grade Cg)
Property Boundary Constraint	n/a	No boundary constraint
Commodity	%Cg	Cg
Product	ton/y	Concentrate
Mining Cost – Mill Feed	$/ton mined	4.6
Mining Cost – Waste Rock	$/ton mined	3.5
Mining Cost – Overburden	$/ton mined	2.0
Mining Cost – Existing Tailings	$/ton mined	4.5
Incremental bench Cost Under Reference Level (610 mZ)	$/ton mined/bench level	0.03
Processing Cost and G&A	$/ton milled	14.0
Concentrate Price	$/ton Conc	1 090
Operating Parameters		
Dilution	%	7
Mining Recovery	%	95
Mill Recovery	%	89
Concentrate Grade	%	95
Mine Production Rate	Mton/y	1.7
Discount Factor	%	8
Cut-Off-Grade:		
Study Cut-off Grade (Cg)	%	1.50
Overall Slope Angle		
Rock - Hanging Wall (PHG lithology)	deg	45
Rock - Foot Wall (UM2 lithology)	deg	40
Overburden/Tailings	deg	23

Operating Costs

The operating costs are preliminary and are used for pit limit analysis and mine planning. Detailed operating costs are developed based on a detailed mine design and plan and discussed in Item 21.

The operating costs for the pit limit analysis are based on current and historic zinc operation on site.

Metallurgical Recovery

The assumptions for the metallurgical recoveries are based on constant recovery assumptions provided by the metallurgical QP for Item 13.2.1.

Commodity Price

The selling price used for the pit optimization work is preliminary and based on bench marking on similar graphite projects.

Boundary Constraints

No property limit or other boundary constraints were applied for the pit limit analysis.

16.3.3.1 Cut-off Value

The cut-off grade (COG) is calculated to determine if material within the pit should be sent to the mill for processing or to the waste rock pile. The marginal COG, referred to as the "Open Pit Discard COG" in the CIM Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines, differs from the breakeven COG since mining costs are excluded from the calculation. The reason for excluding mining costs is that material already defined to be within the limits of the open pit must be mined, regardless of whether it is classified as mineralized material or waste, to access the bench below. The only exception where a mining cost would be included in the marginal COG calculation is if there is an incremental cost for mining mineralized material relative to mining waste, which is not the case in the Graphite Study. To calculate the marginal COG for the Graphite Study, the following calculation was used to cover the costs of processing, general and administrative costs, and selling costs using the economic and technical parameters listed in Table 16-25.

$$Marginal\ COG\ = \frac{Processing\ Cost\ x\ Concentrate\ Grade}{(Sales\ Price\ -\ Selling\ Cost\ -\ G\&A\ Cost)x\ Mill\ Recovery}$$

Inferred Mineral Resource material contained within the pit design and above the cut-off grade is classified as potential mill feed (PMF), while resource material below the cut-off grade is classified as waste.

The marginal open pit cut-off grade using parameters listed in Table 16-25 was calculated to be 1.44% Cg (accounting for 7% dilution with zero grade). The cut-off grade was elevated to 1.5% Cg for the purpose of the Graphite Study.

16.3.3.2 Pit Limit Analysis Results

The pit limit analysis process results in a series of 19 nested pit shells, each corresponding to a revenue factor (RF) ranging from 0.70 to 1.15. The RF scales the commodity prices only, ranging from $763 to $1,254 per ton of concentrate, and no costs are factored by the RF. The RF1 corresponds to the selling price of $1,090 per ton of concentrate.

Table 16-26 and Figure 16- summarize the nested pit shell results for the Graphite Study at a selection of revenue factors.

The discounted cash flow (DCF) of each shell was calculated assuming a selling price of $1,090/ton of graphite concentrate, a discount rate of 8% and an annual production rate of 1.7 Mton/y of PMF. It is important to note that the DCF's presented do not include initial and sustaining capital costs and are therefore not indicative of the Project's DCF. The DCF value shown in the pit limit analysis tables and figures is used only as a guide in pit shell selection. The actual net present value (NPV) of the Project is estimated in the Economic Analysis Item of this report (Item 22).

Table 16-26: Nested pit shell results

Revenue Factor	Selling Price	Mineralization - Potential Mill Feed (PMF)			Waste	Overall Stripping Ratio	Incremental Stripping Ratio	Best Case DCF	Worst Case DCF	Average Case DCF
		Tonnage	Cg Grade	Concentrate	Tonnage					
	USD/Ton Conc.	('000 Ton)	(%Cg)	('000 Ton)	('000 Ton)			(M$)	(M$)	(M$)
0.700	763	4,192	3.24	127.07	3,646	0.9	1.4	52.96	51.22	52.09
0.725	790	4,863	3.20	146.01	4,629	1.0	1.5	58.35	55.89	57.12
0.750	818	5,393	3.17	160.31	5,363	1.0	1.4	62.00	58.91	60.46
0.775	845	6,121	3.13	179.77	6,507	1.1	1.6	66.32	62.23	64.28
0.800	872	7,062	3.07	202.81	7,525	1.1	1.1	70.71	65.29	68.00
0.825	899	8,210	3.02	232.51	9,539	1.2	1.8	75.56	68.25	71.90
0.850	927	9,974	3.00	279.85	13,532	1.4	2.3	81.47	70.70	76.09
0.875	954	11,549	2.97	321.21	17,169	1.5	2.3	85.38	71.26	78.32
0.900	**981**	**13,922**	**2.93**	**382.24**	**22,424**	**1.6**	**2.2**	**89.86**	**68.95**	**79.41**
0.925	1,008	15,630	2.88	421.79	25,419	1.6	1.8	91.80	66.09	78.95
0.950	1,036	18,194	2.80	477.38	29,161	1.6	1.5	93.55	60.92	77.23
0.975	**1,063**	**21,361**	**2.75**	**550.11**	**35,690**	**1.7**	**2.1**	**94.60**	**54.08**	**74.34**
1.000	**1,090**	**22,745**	**2.72**	**579.32**	**37,982**	**1.7**	**1.7**	**94.72**	**50.77**	**72.75**
1.025	1,117	24,126	2.69	608.40	40,493	1.7	1.8	94.52	47.07	70.79
1.050	1,145	25,563	2.67	639.27	43,554	1.7	2.1	93.99	42.92	68.45
1.075	1,172	27,184	2.64	673.16	47,006	1.7	2.1	93.12	36.96	65.04
1.100	1,199	28,920	2.62	710.16	51,206	1.8	2.4	91.97	30.65	61.31
1.125	1,226	30,887	2.60	751.22	56,017	1.8	2.4	90.44	23.66	57.05
1.150	1,254	32,591	2.58	786.72	60,447	1.9	2.6	89.03	16.93	52.98

Highest DCF pit shown in **BLUE**, RF = 1.0 pit shown in **OLIVE**, Selected pits shown in **RED**.

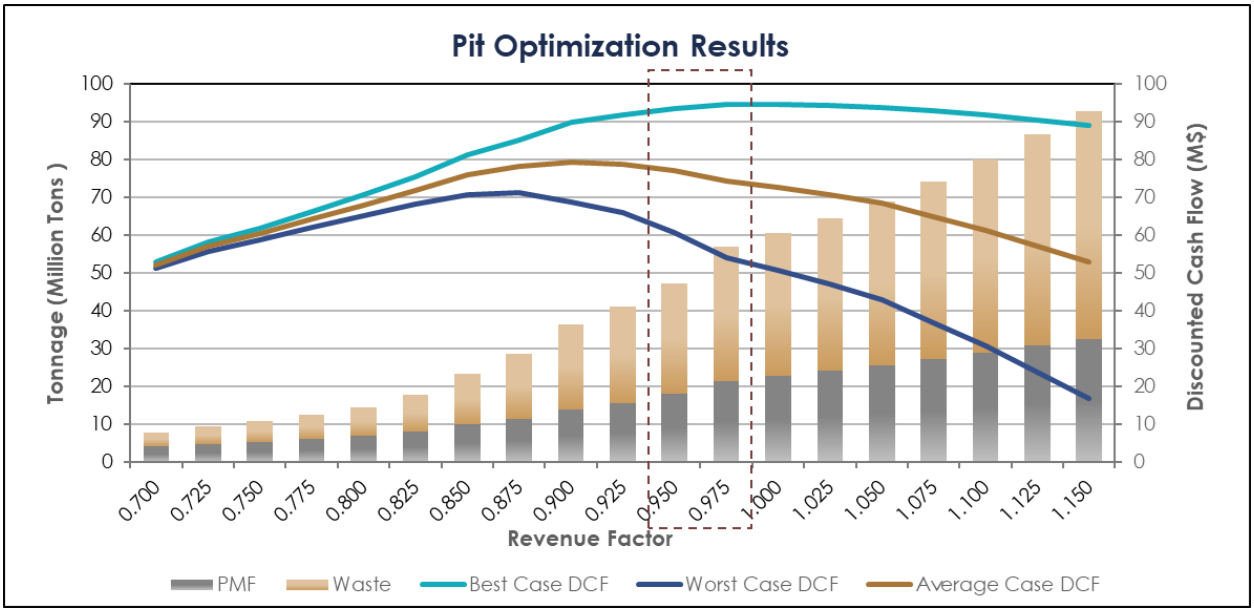


Figure 16-17 Pit-by-pit graph

Observations and Recommendations from Pit Limit Analysis

The recommended pit shell to be used as a guide for ultimate pit layout is RF0.950 and RF0.975. These pit shells provide a balanced outcome—capturing most of the value at RF0.90, which is the highest value pit shell on the Average DCF curve, while reducing reliance on optimistic inputs of

an RF1.00 pit shell—and define a potential concentrate tonnage target of greater than 450,000 tons.

The difference between the Best Case and Worst Case DCF curves in the RF0.900 and RF1.000 pit shells range is approximately 20–40%, indicating that pit phasing and sequencing have an impact on the pit's NPV.

16.3.4 Pit Design

The following section presents the design criteria used for the open pit design. The design was guided by the two selected shells from the pit limit analysis. It includes smoothing of the pit walls, and the addition of haulage ramps to access the pit bottom. Figure 16-18 shows the proposed ultimate pit design. Table 16-27 summarizes the pit design characteristics.

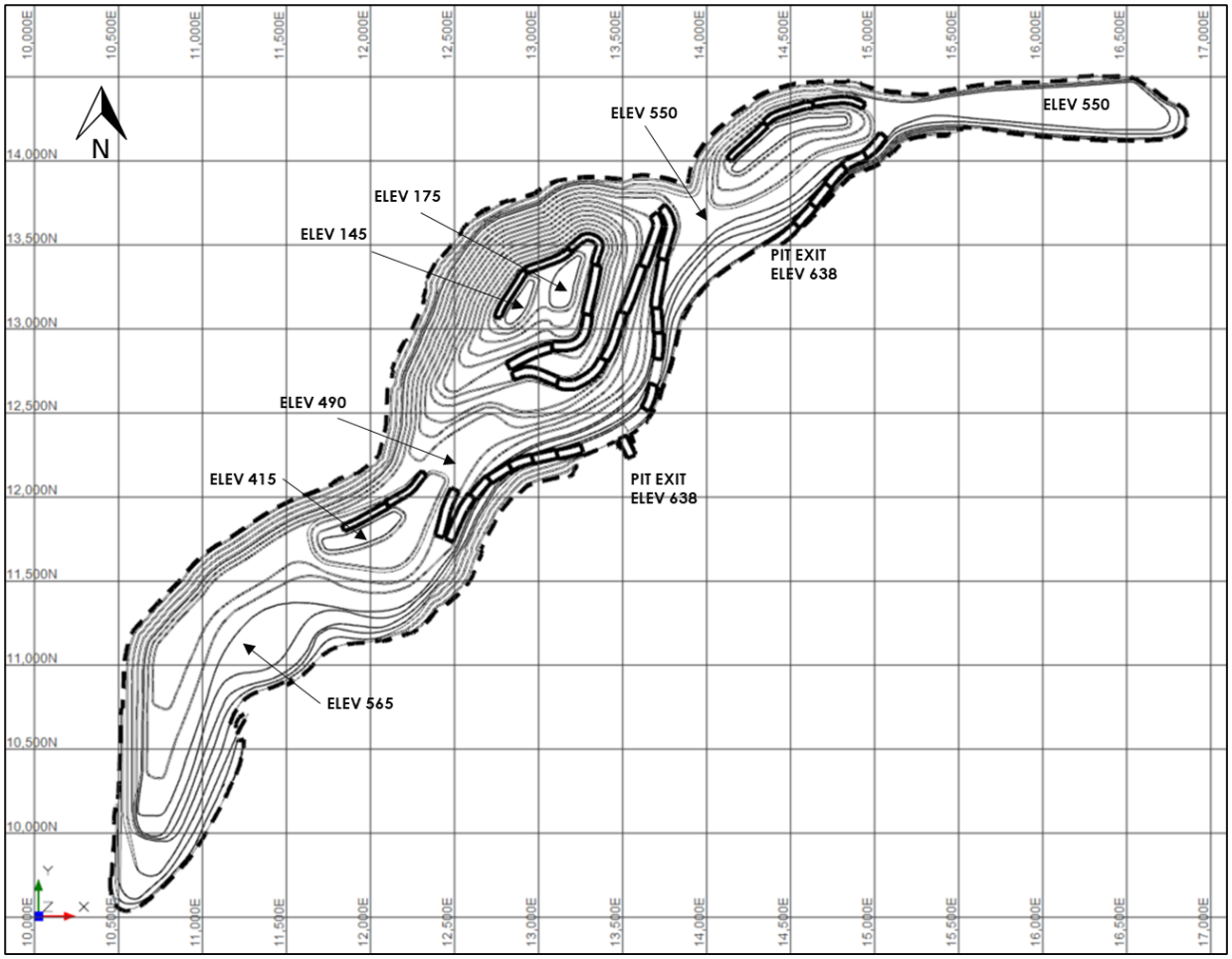


Figure 16-18: Proposed ultimate pit design

Table 16-27: Pit design characteristics

Item	Unit	Value
Pit Top Elevation	ft	680
Pit Bottom Elevation	ft	145
Pit Depth	ft	535
Volume of Pit	Mft^3	760
Area of Pit (surface boundary)	ha	61.6
Perimeter at the Top of the Pit	ft	19,700
Length from East to West	ft	6,320
Length from North to South	ft	5,000

16.3.4.1 Pit Wall Configuration

The benching parameters were established following the geotechnical parameter recommendations presented in the report titled "Kilbourne Graphite Preliminary Geotechnical Design Basis" (Henning and Asi, 2025).

Table 16-28 tabulates the parameters used for the pit design.

Table 16-28: Pit design configuration

Geotechnical Design Criteria	Hanging Wall (PHG lithology)	Footwall (UM2 lithology)	Overburden
Bench Height, BH	Single Bench: 15 ft Double Bench: 30 ft	Single Bench: 15 ft Double Bench: 30 ft	Single Bench: 15 ft
Bench Face Angle, BFA	75°	75°	35°
Catch Bench Width, CBW	Single Bench: 9.5 ft Double Bench: 19.0 ft	Single Bench: 8.6 ft Double Bench: 17.4 ft	Single Bench: 9.5 ft
Inter-ramp Angle, IRA	48°	50°	26°
Geotechnical Bench	Not anticipated due to indicated pit ultimate depth of 500 ft		-
Rock/Overburden contact	Offset the toe of the overburden slope from the pit crest by a minimum rock bench width of 16 ft.		

For this design, the same geometry recommended for the overburden was applied to the existing tailings. However, proper tailings removal with a suitable setback must be considered during mining operations.

16.3.4.2 Haul Ramp Design

The haul ramp design presented in Table 16-29 is based on a 40-ton class truck, including allowances for a safety berm and drainage ditch.

Table 16-29: Haul ramp design

Design Criteria	Width
Single Lane	40 ft
Double Lane	54 ft
Ramp Gradient	10%

16.3.4.3 Minimum Mining Width

A minimum mining width of 100 ft was considered for the pit design. This width must be respected to ensure that a 40-t haul truck can safely enter the mining area and make a 180° turn to be positioned for loading.

16.3.4.4 Final Bench Access

In order to reduce the stripping ratio as much as is feasibly safe and efficient, the access ramp will not be designed to the bottom of the lowest benches. When mining the final bench, the haul trucks will be positioned on the bench crest rather than on the bench toe. Figure 16-19 illustrates this operating scenario, commonly referred to in the industry as a goodbye cut. This final bench will be 15 ft high.

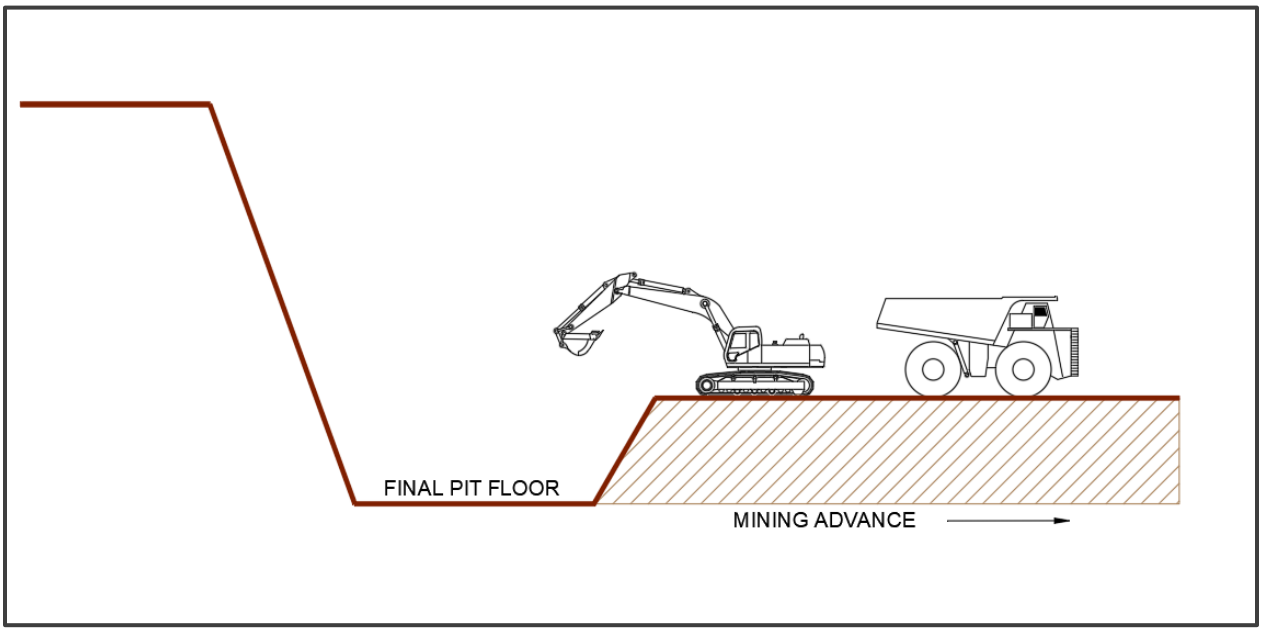


Figure 16-19: Access ramp

16.3.4.5 Department of Environmental Conservation setback requirements

The Department of Environmental Conservation (DEC) of New York State requires a minimum setback for all adjacent properties regardless of whether they are disturbed, built on or not (NYSDEC, 1996):

- Unconsolidated Material (tails, overburden):
  - 25 ft + 1.5 * Depth of pit.
- Consolidated Material (rock):
  - 25 ft + 1.25 * Depth of pit.

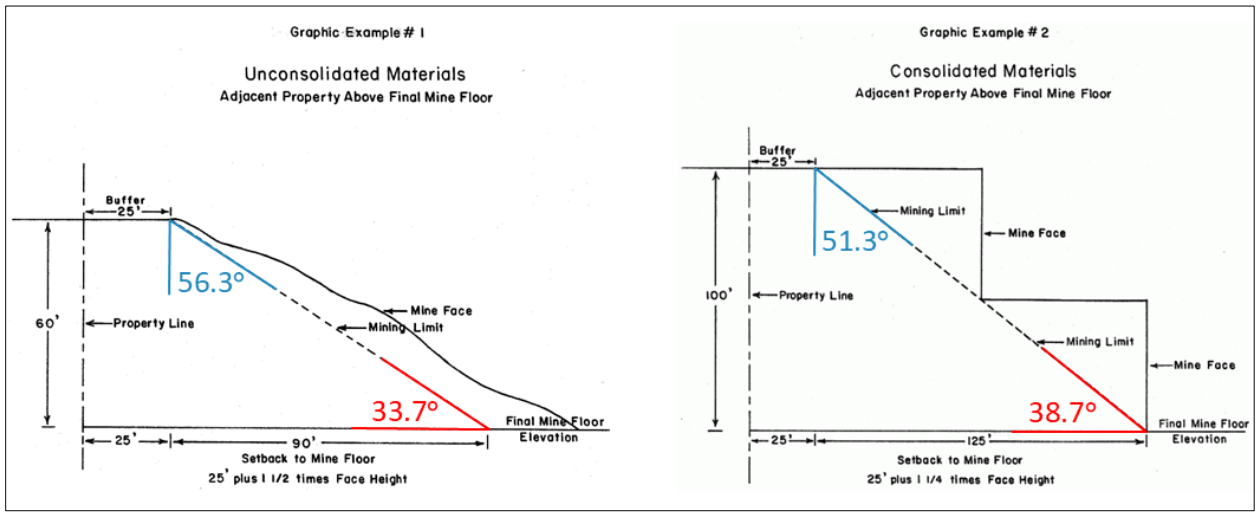


Figure 16-20: Minimum setback requirements by DEC

The setback requirement from the mine floor is respected with the mine plan.

16.3.4.6 Phase Selection

The phase selection for the Graphite Study is based on smaller revenue nested pit shells from the Pit Limit Analysis task. The number of phases or pushbacks was dependent on the pit limit analysis results and the space available between selections. A minimum pushback width of 170 ft is estimated. Figure 16-21 illustrates the method of estimation.

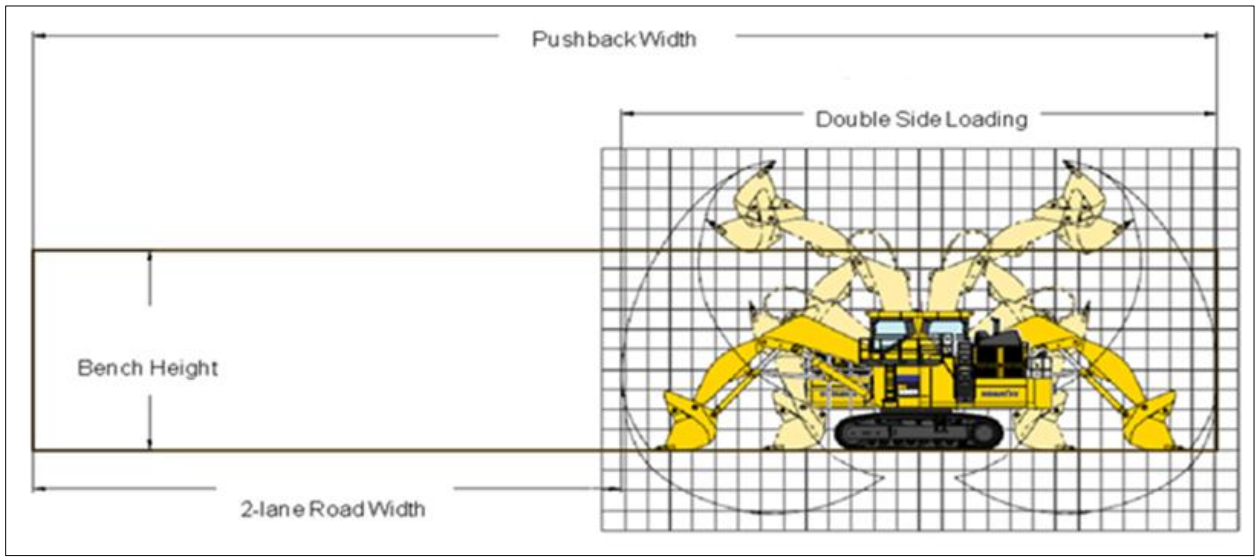


Figure 16-21: Minimum pushback width

Based on the pit limit analysis results:

- Recommended pit shell to be used as a guide for Initial starter pit (Phase 1) is RF0.625. This pit shell targeted the approximate mineralized tonnage of 2.33 Mton, which corresponds to roughly two years of Mill Production Plan.
- Recommended pit shell for use as a guide for subsequent phase is RF0.825. This shell targets an approximate mineralized tonnage of 8.21 Mton, with an overall stripping ratio of 1.2.

Further phase definition was applied by defining the northern limb as a phase and the southern limb as a phase. For the level of study, these phases were delineated by vertical splicing. It should be noted that the phase labeling may not necessarily correspond to the order in which they are started to be mined.

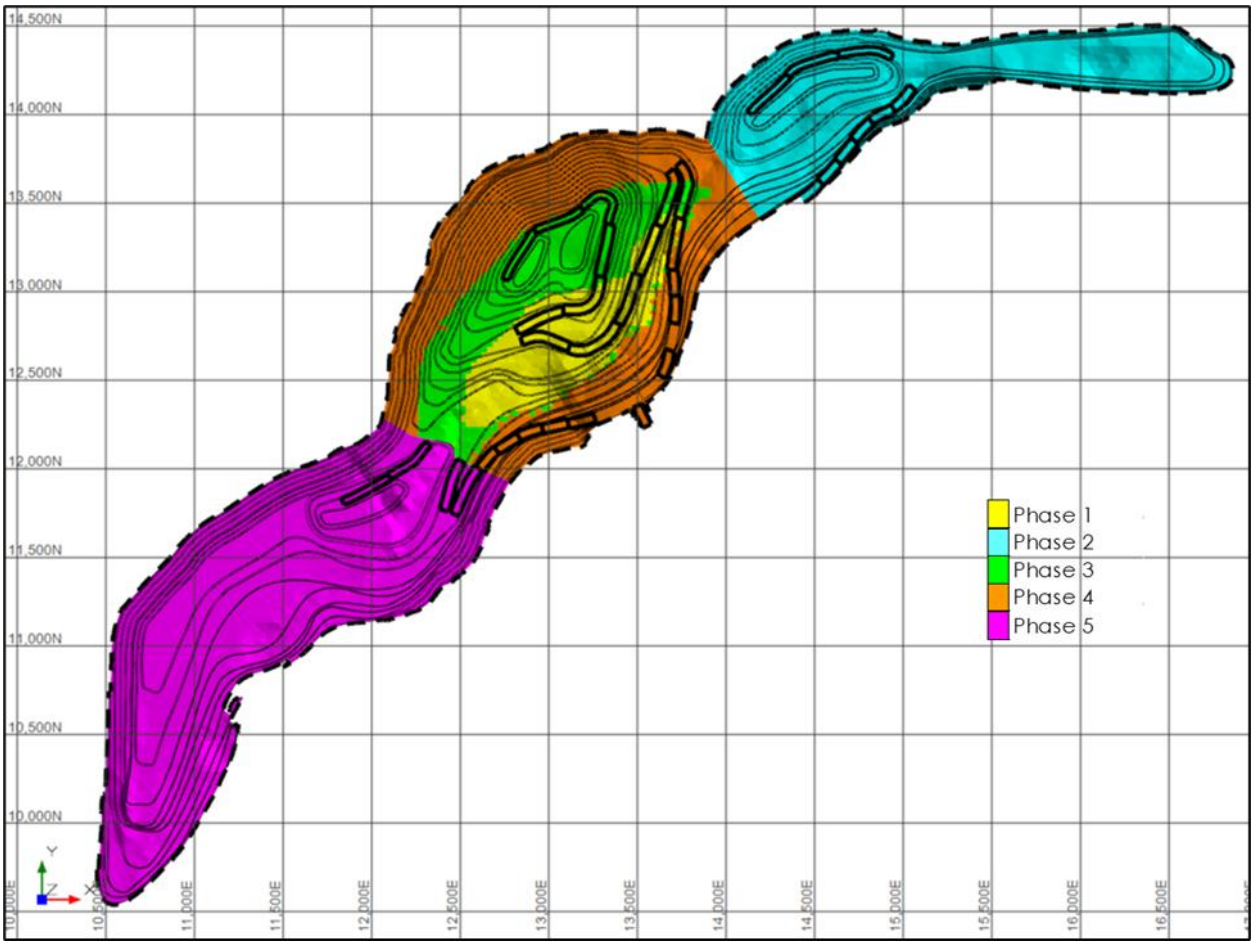


Figure 16-22: Phase definition

16.3.4.7 Pit and Phase Inventory

Table 16-30 and Table 16-31 tabulate the Ultimate Pit and Phase material inventories.

Table 16-30: Ultimate pit material inventory

Material	Total	
	kton	Cg %
Total PMF Material	**19,951**	**2.84**
Zone 210	14,910	2.95
Zone 220	66	1.63
Zone 230	4,975	2.53
Total Waste Material	**42,818**	
Tailings	5,561	
Overburden	6,436	
Waste Rock	30,821	
Total Material	**62,769**	
Strip Ratio	**2.1**	

Table 16-31: Material inventory by phase

Material	Phase 1		Phase 2		Phase 3		Phase 4		Phase 5	
	kton	Cg %	kton	Cg %	kton	Cg %	kton	Cg %	kton	Cg %
Total PMF Material	**2,177**	**3.50**	**2,958**	**2.28**	**3,650**	**3.25**	**5,935**	**2.97**	**5,231**	**2.46**
Zone 210	2,177	3.50	1,172	2.12	3,638	3.26	4,048	3.16	3,875	2.40
Zone 220	0	0.00	18	1.62	12	1.59	14	1.61	22	1.67
Zone 230	0	0.00	1,768	2.40	0	0.00	1,873	2.58	1,334	2.64
Total Waste Material	**1,328**	**-**	**3,743**	**-**	**6,172**	**-**	**20,904**	**-**	**10,671**	**-**
Tailings	484	-	0	-	842	-	1,004	-	3,231	-
Overburden	37	-	911	-	1,391	-	1,894	-	2,203	-
Waste Rock	807	-	2,832	-	3,939	-	18,006	-	5,237	-
Total Material	**3,505**	**-**	**6,701**	**-**	**9,822**	**-**	**26,839**	**-**	**15,902**	**-**
Strip Ratio	0.6	-	1.3	-	1.7	-	3.5	-	2.0	-

16.3.5 Mine Plan

16.3.5.1 Life of Mine Scheduling

The LOM production schedule for the open pit area has been prepared using the MinePlan Schedule Optimizer (MPSO) tool in the Hexagon™ MinePlan 3D software. Provided with economic input parameters and operational constraints such as phase sequencing, maximum bench sink rates, and mining and milling capacities, the software determines the optimal mining sequence and stockpiling strategy, which maximizes the present value (PV) of the mine production plan.

The overall objective of the mine scheduling and planning process is to maximize Project PV while achieving the processing plant objectives and targets. Generally, this is done by delaying the overburden and waste rock removal activities, e.g., costs for as long as possible. This objective is taken into consideration during all phases of the mine design and mine planning.

A portion of the open pit is overlaid with existing tailings material. This material will be required to be removed and perimeter dikes are required around portions of the open pit to keep the remaining in-place existing tailings from migrating into the open pit. A staged approach of removing the existing tailings and construction of the perimeter dikes has been undertaken.

The mine scheduling for the open pit incorporates the following General Assumptions or criteria. It should be noted that the units of measurement are imperial short tons for material movement to the Concentrate Plant. After the Concentrate Plant, the units of measurement are metric tonnes.

Concentrate Plant Feed Throughput Ramp up Targets:

- 1,228,000 tons for Y1
- 1,400,000 tons for Y2
- 1,400,000 tons for Y3
- 1,700,000 tons for Y4 onwards

Concentrate Production Target Ramp up:

- 22,500 tonnes for Y1
- 27,500 tonnes for Y2
- 38,000 tonnes for Y3
- 40,000 tonnes for Y4 onwards
- 100% Concentrate Plant utilization assumed to be 1,700,000 tons mill feed throughput and 40,000 metric tonnes of concentrate produced.
- A maximum Concentrate Plant Throughput Rate of +10%
- Maximum material movement of 8,000,000 tons

- Maximum stockpile inventory 35,000 tons
- LOM plan sequenced and scheduled on an annual basis

The annual mine plan has been developed in order to meet Concentrate Plant feed requirements according to general best open pit mine practices such as equipment fleet smoothing and maximizing PV. Table 16-32 tabulates the LOM plan quantities by material by period.

Table 16-32: Mining quantities by period

Period	Potential Mill Feed	Cg Diluted	Waste Rock	Overburden	Tailings Material	Total Material Mined
	kton	%	kton	kton	kton	kton
Year -1	0	0.00	0	0	0	**0**
Year 1	1,226	2.18	1,372	646	0	**3,244**
Year 2	1,400	2.33	1,269	265	1,133	**4,068**
Year 3	1,398	3.21	747	37	1,905	**4,087**
Year 4	1,355	3.39	706	1,175	2,252	**5,488**
Year 5	1,530	3.21	3,046	216	271	**5,064**
Year 6	1,695	3.06	2,204	1,067	0	**4,966**
Year 7	1,864	2.78	3,803	1,799	0	**7,466**
Year 8	1,850	2.42	4,944	1,222	0	**8,016**
Year 9	1,844	2.44	4,146	9	0	**5,999**
Year 10	1,617	2.77	4,343	0	0	**5,960**
Year 11	1,444	2.96	2,444	0	0	**3,887**
Year 12	1,328	3.22	1,212	0	0	**2,540**
Year 13	1,400	3.20	583	0	0	**1,983**
Total	**19,951**	**2.84**	**30,821**	**6,436**	**5,561**	**62,769**

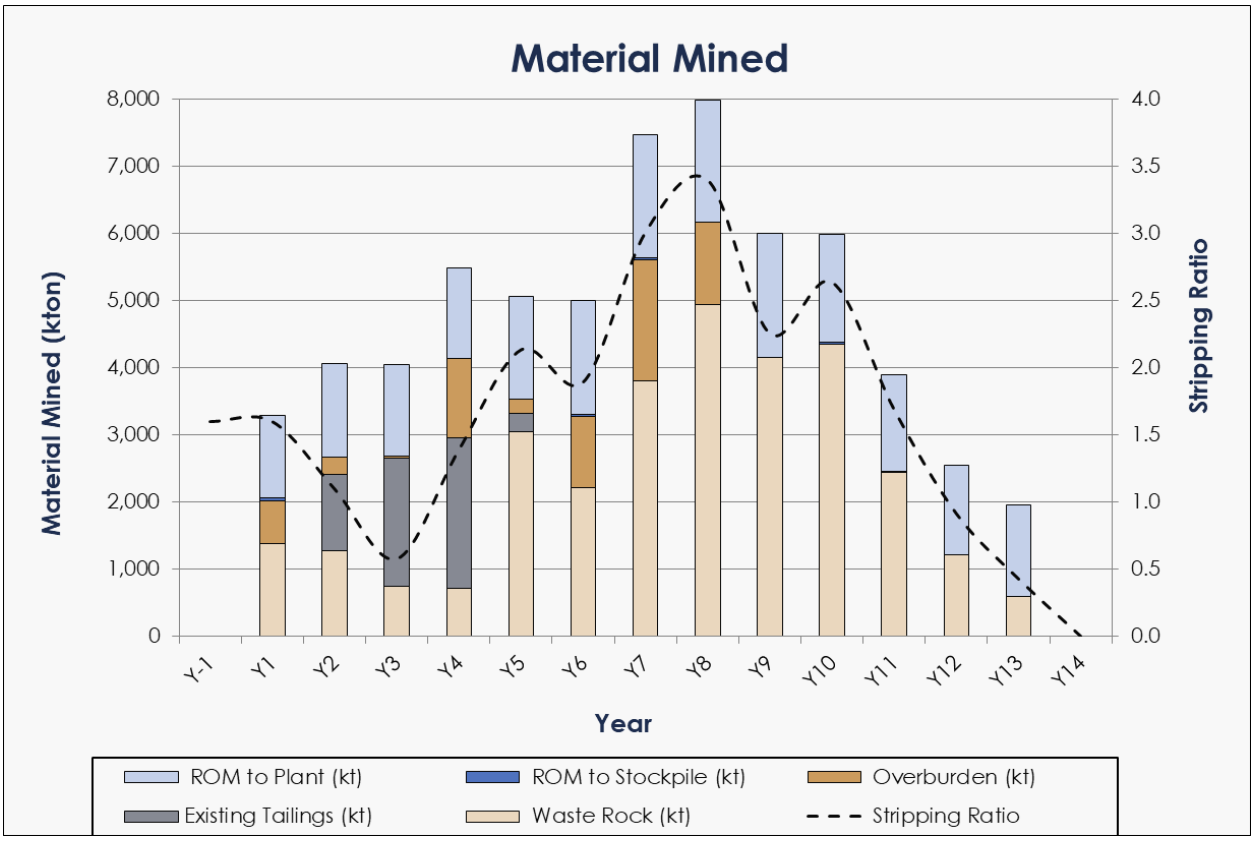


Figure 16-23: LOM plan material mined by year

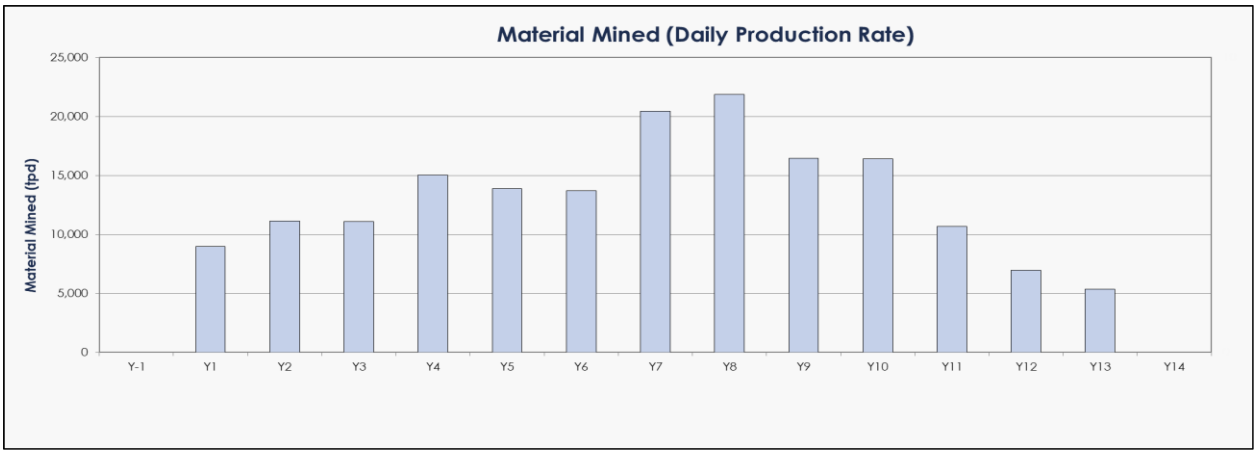


Figure 16-24: Average daily mining production rate

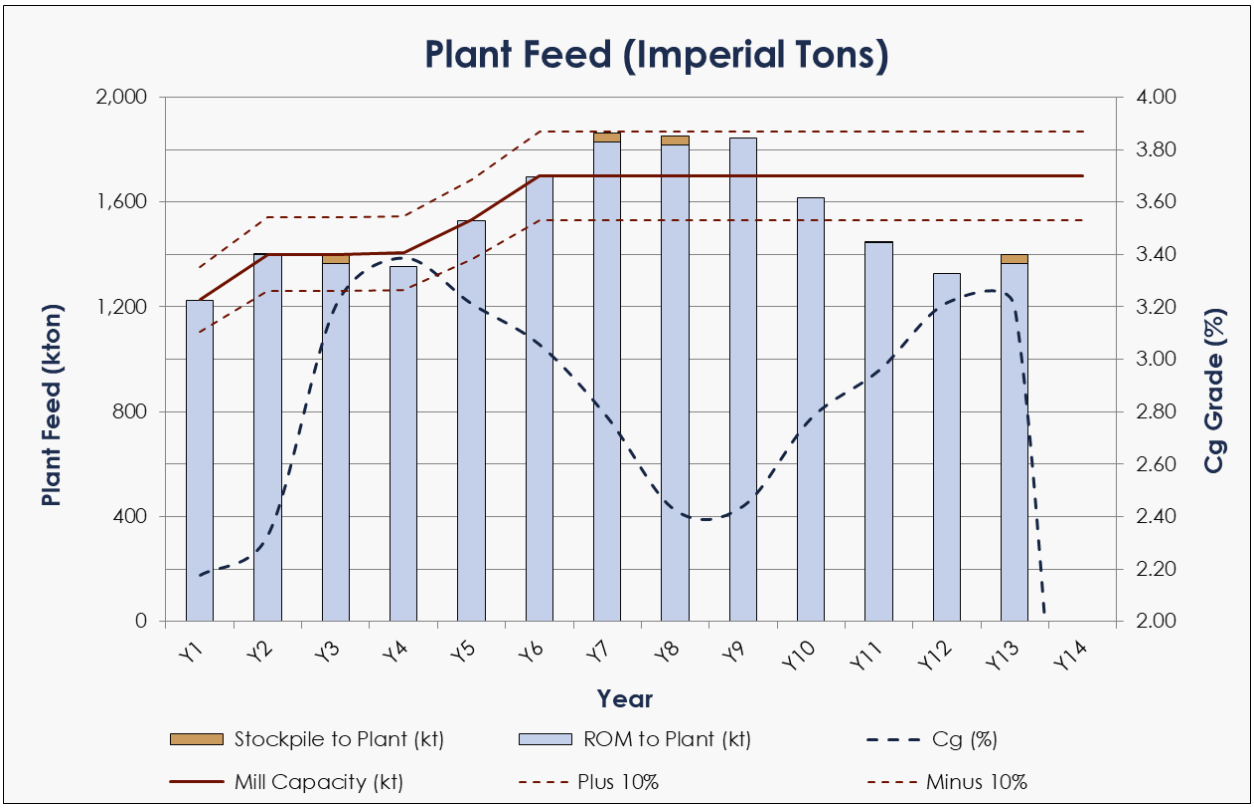


Figure 16-25: Mill feed throughput by year

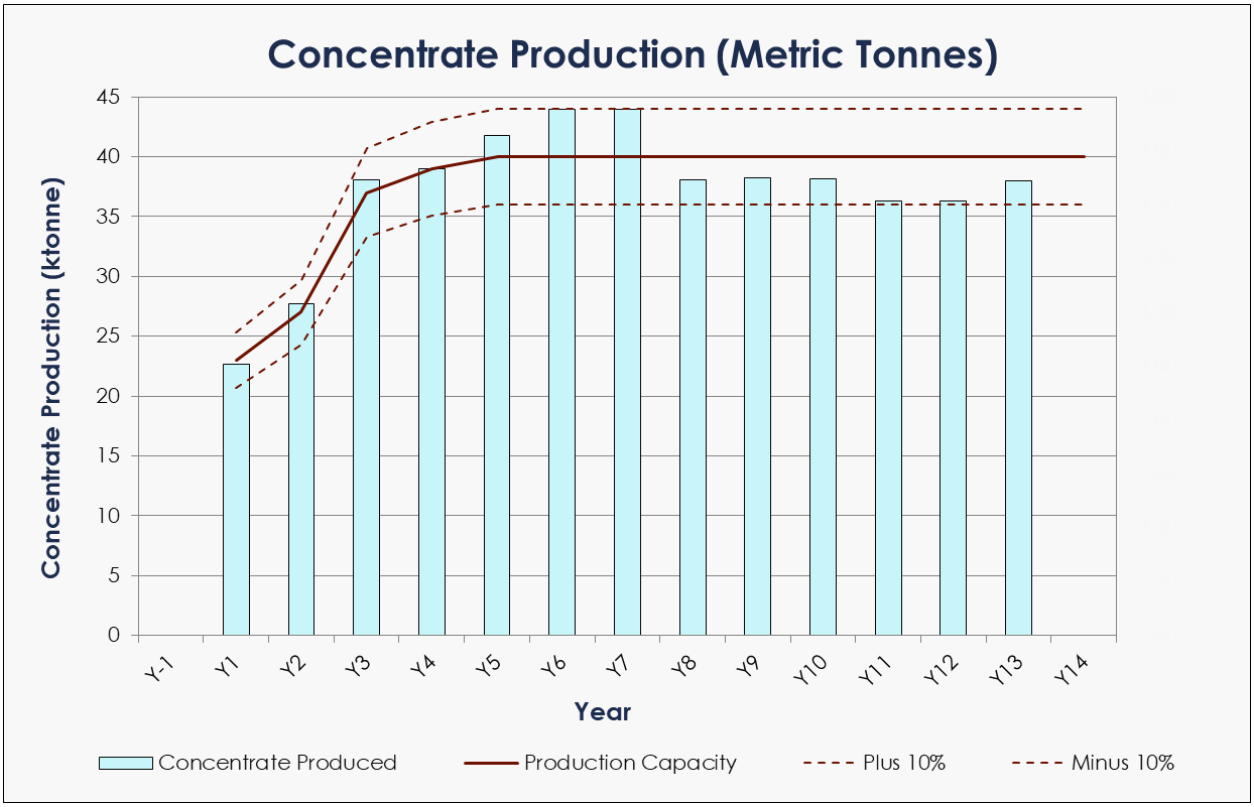


Figure 16-26: Concentrate production by year

Figure 16-27 shows the pit phases relative to the existing tailings and perimeter containment dikes. Figure 16-28 shows the material mined by phase by period.

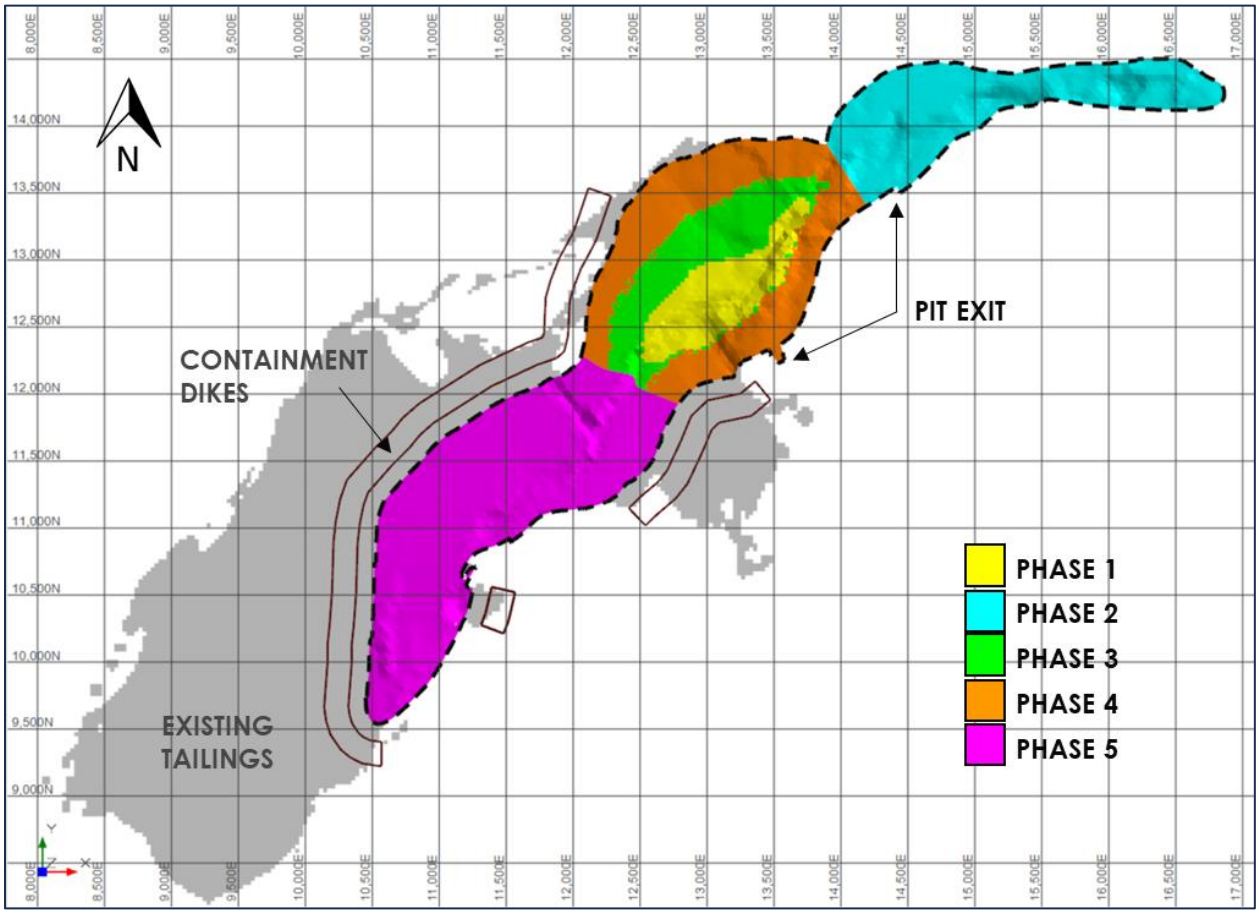


Figure 16-27: Pit phases relative to existing tailings and perimeter containment dikes

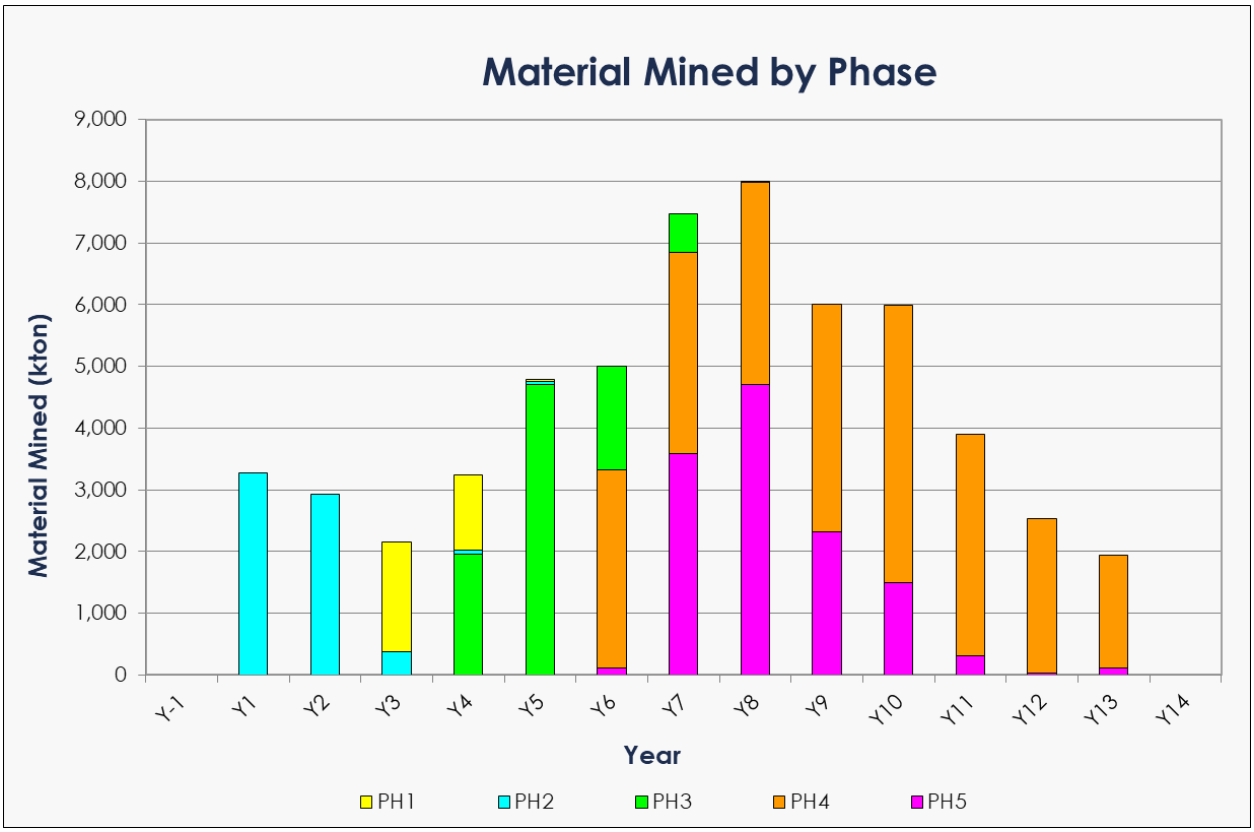


Figure 16-28: Material mined by phase by year

16.3.5.2 LOM Sequence

Figure 16-29 through Figure 16-31 depict the estimated progression of the open pit, on an annual basis for select years.

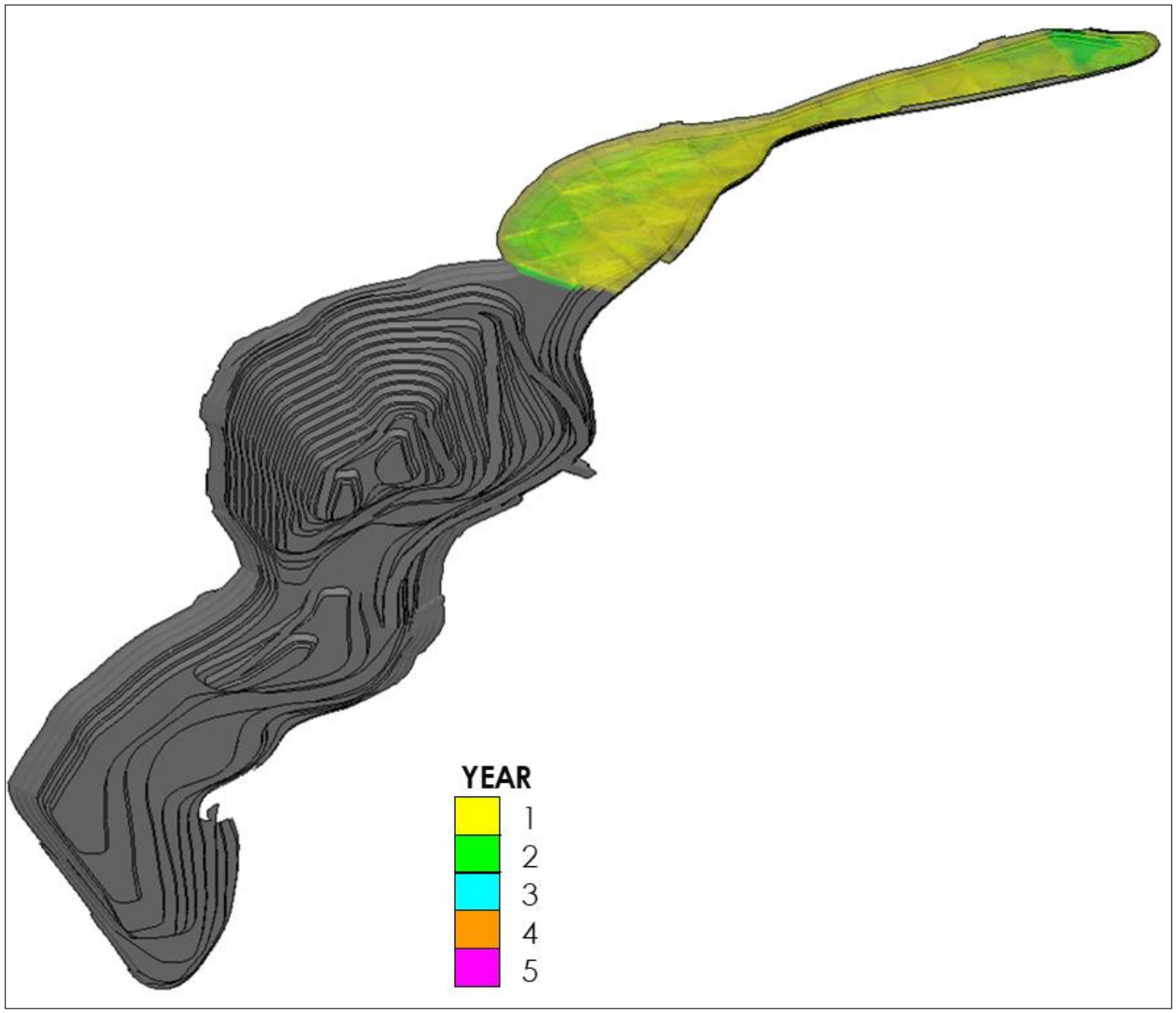


Figure 16-29: LOM sequence, Year 1 to Year 2 inclined view looking north, not to scale

Beginning the mine plan with mining in the northern limb area, where there are no existing tailings to remove, allows for deferring the removal of the existing tailings and the construction of the perimeter dikes. This approach prioritizes starting with Phase 2 instead of Phase 1, which has the highest DCF value.

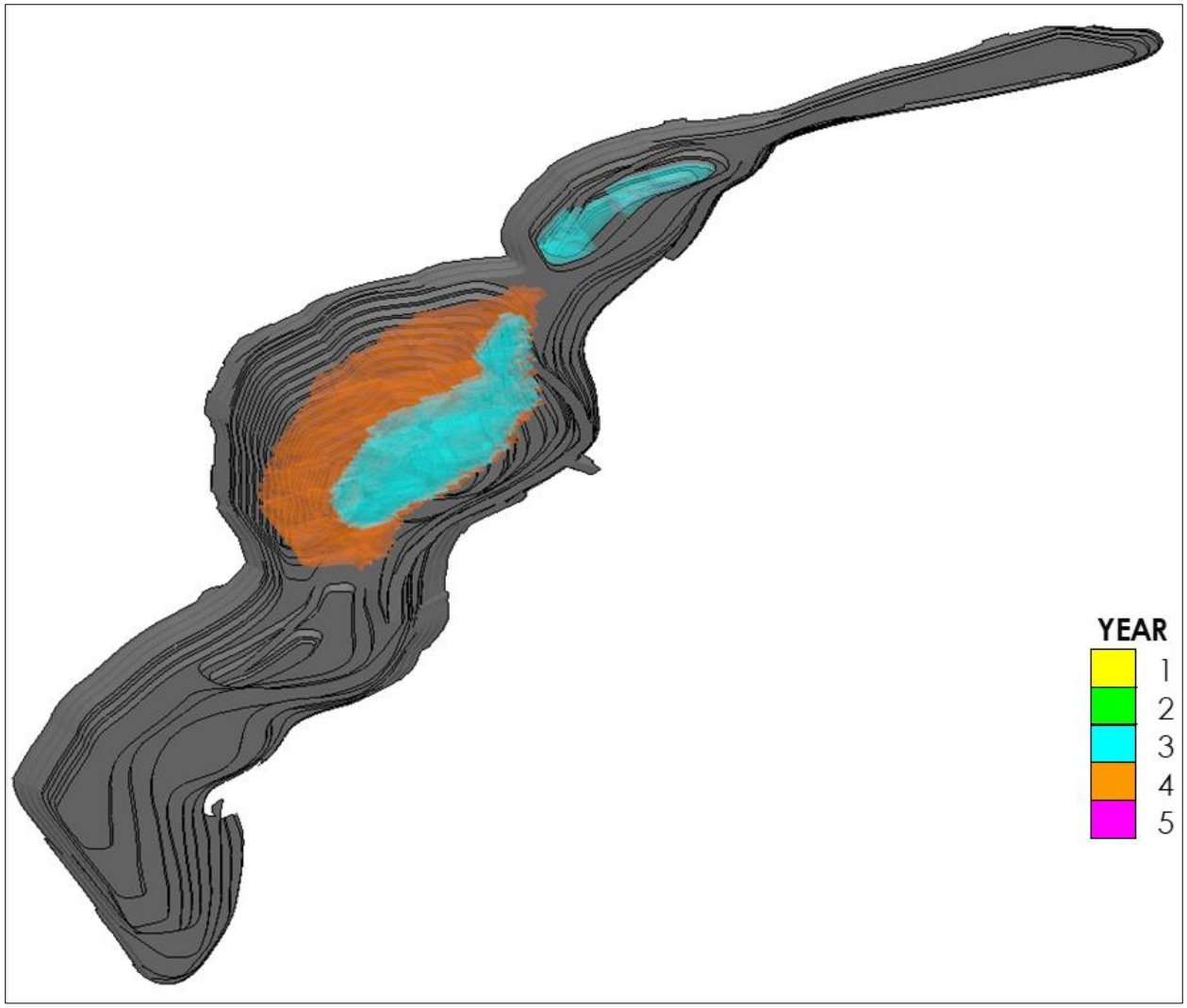


Figure 16-30: LOM sequence, Year 3 to Year 4 inclined view looking north, not to scale

Mining progresses to the central area of the pit and the deposit. At this point, tailings removal and construction of the perimeter dikes are required.

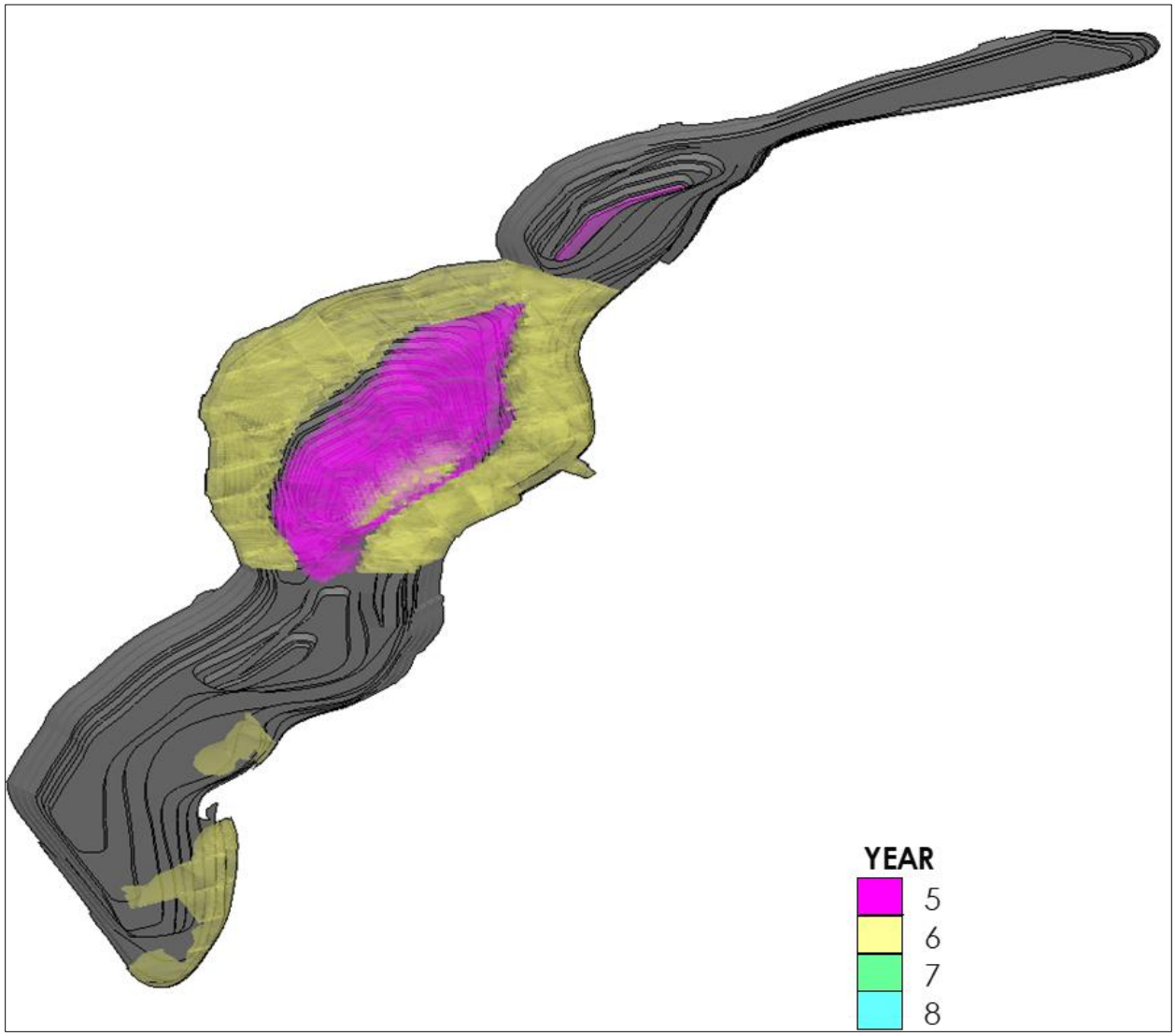


Figure 16-31: LOM sequence, Year 5 to Year 6 inclined view looking north, not to scale

Mining continues within the central area of the pit and the deposit. The second stage of tailings removal and perimeter dike construction are required to be completed by Year 6.

Figure 16-32 and Figure 16-33 illustrate the remaining progression of the mining sequence in the central and southern areas of the open pit.

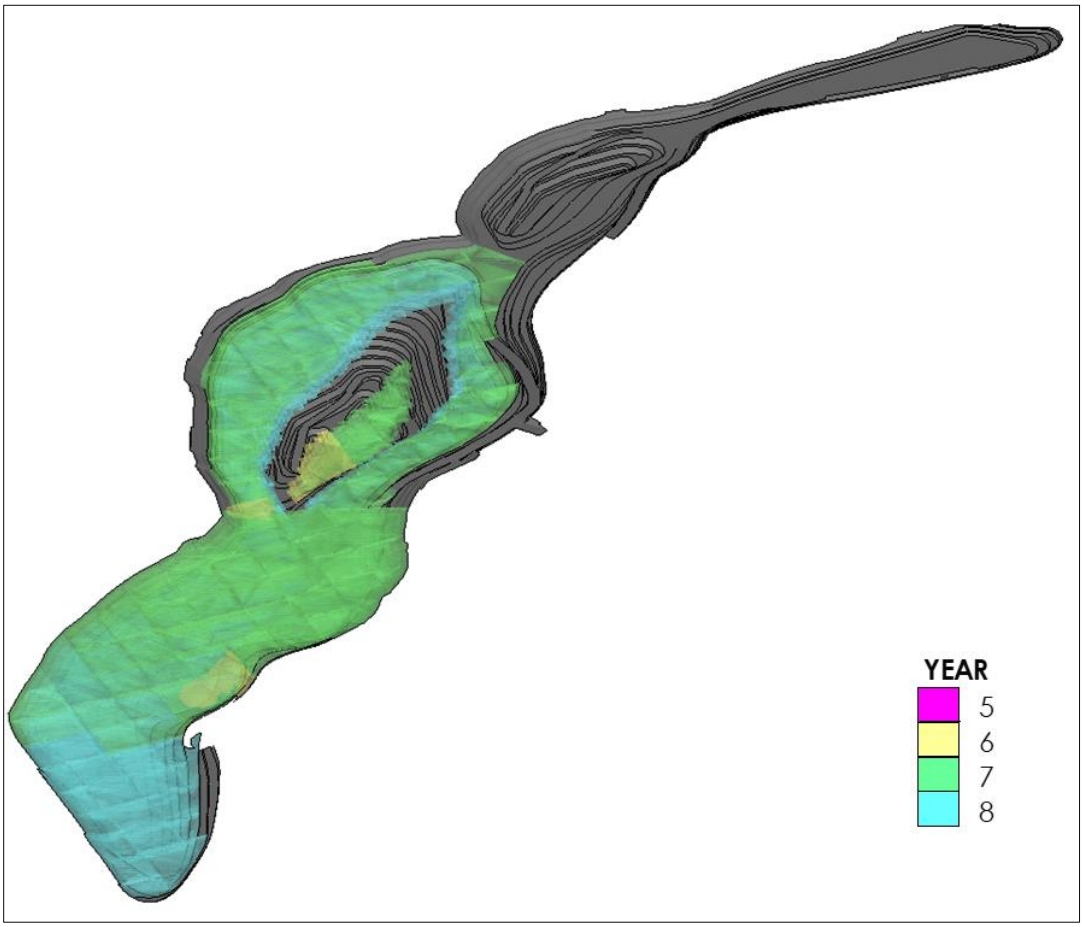


Figure 16-32: LOM Sequence, Year 7 to Year 8 inclined view looking north, not to scale

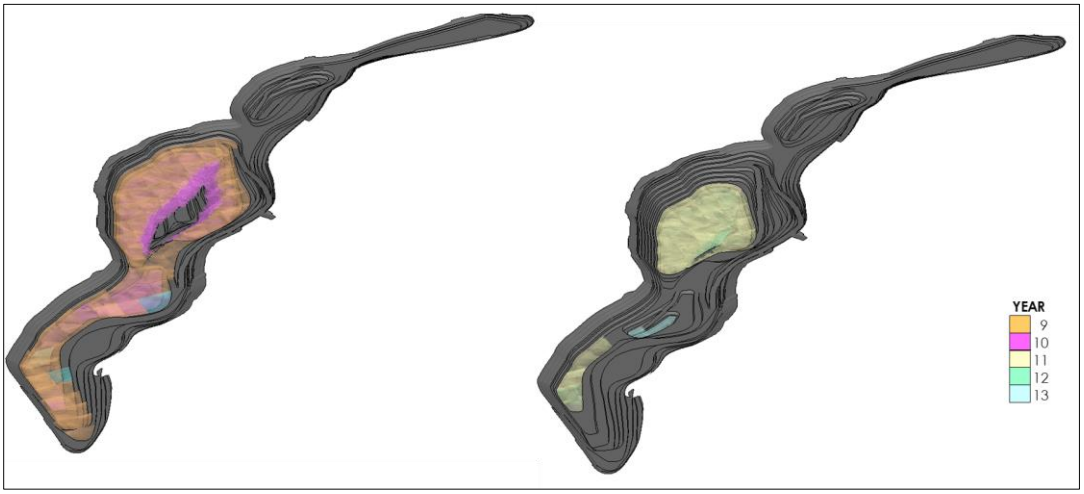


Figure 16-33: LOM Sequence, Year 9 to Year 13 inclined view looking north, not to scale

16.3.5.3 Mine Rock and Overburden Stockpiles

Preliminary designs for waste rock and overburden storage were assessed considering an overall reclaimed angle of 26 degrees and 21 degrees, respectively.

The targeted volume for the waste rock stockpile design, based on the pit design excludes the tonnage that will be required to construct the dikes planned around the pit and other tailings storage facilities. The estimated quantities requiring storage within the mine rock and overburden stockpiles are 356 million ft^3 and 67 million ft^3, respectively.

16.3.5.4 ROM Stockpile

The ROM stockpile is located next to the crusher and is planned to accommodate up to 40,000 tons of material, a 1-week capacity. A 37-degree angle of repose is assumed with a height of 15 ft.

16.3.6 Open Pit Mine Equipment Fleet

16.3.6.1 Equipment Parameters

The mine design and planning are proposed to be executed using the following major equipment fleet class:

- 45-ton class haul truck matched with a 163-lb class mining excavator,
- 5-inch down-the-hole (DTH) hammer drill, 11-ton bucket capacity front-end loader, and
- 43-ton track dozer and 14 ft blade road grader.

The remaining fleet of support, service, and ancillary equipment will be considered for capital and operating cost estimates (CAPEX, OPEX).

16.3.6.2 Equipment Utilization Model

Figure 16-34 presents the equipment utilization model, which is used to understand the key performance indicators (KPI) that govern the fleet requirements. The definitions for each time component are presented in Figure 16-34.

	Scheduled Time						
	Available Time					Down Time	
	Utilized Time (GOH)				Standby Time	Planned Loss	Breakdown Loss
	Operating Time (NOH)		Operating Delay				
Examples for Trucks & Shovels	Shovels	Trucks	Shovels	Trucks	Shovels & Trucks	Shovels & Trucks	Shovels & Trucks
	Spot	Travel empty	Wait onTruck	Wait on Shovel	Shift Change	Scheduled Maint.	Breakdown
	Fill	Spot @ Shovel	Relocating	Positioning	Lunch & Coffee	Inspections / PM's	Wait for Parts
	Swing	Loading	Face Prep.	Crusher Down	Refueling		Repair Time
	Dump	Travel Full	Highwall Scaling	Loader Moving	Pre-start Check		
	Return	Spot @ Dump	Positioning	Queuing	No Operator		
		Dumping			Weather Delays		

Figure 16-34: Equipment utilization model

- Scheduled Time – full calendar year less unplanned shutdowns;
- Down Time – the unit is inoperable due to either a scheduled maintenance or an unplanned breakdown;
- Available Time – scheduled time less down time;
- Standby Time – the unit is available mechanically but not being used (the engine will typically be shut off while the unit is on standby;
- Utilized Time – available time less standby time. This time is also referred to as the Gross Operating Hours (GOH);
- Operating Delays – the unit is available and not on standby but not effectively producing (the engine will be running during the operating delays);
- Operating Time – utilized time minus operating delays. This time is also referred to as the Net Operating Hours (NOH).

The following KPI's can be calculated from the different time components using the formulas below:

- Availability – (NOH + Op. Delays + Standby) / (NOH + Op. Delays + Standby + Down);
- Use of Availability – (NOH + Op. Delays) / (NOH + Op. Delays + Standby);
- Machine Utilization – (NOH + Op. Delays) / (Scheduled Time);
- Operating Efficiency – (NOH) / (NOH + Op. Delays);
- Effective Utilization – (NOH) / (Scheduled Time).

Table 16-33 presents the KPIs and time assumptions that will be used for the fleet of shovels, trucks and drills.

Table 16-33: Mine equipment KPIs

Description	Unit	Loading	Hauling	Drilling
Availability	%	85	85	75
Use of Availability	%	83.9	68.5	58.5
Machine Utilization	%	71.3	56.8	48.8
Operating Efficiency	%	85.7	86.7	83.5
Effective Utilization	%	61.1	49.3	40.4
Calendar Time	h/y	5,200	5,200	5,200
Scheduled Time	h/y	5,160	5,160	5,160
Down Time	h/y	780	780	1,300
Standby Time	h/y	712	748	638
Operating Delays	h/y	530	490	562
Utilized Time (GOH)	h/y	3,708	3,672	3,262
Operating Time (NOH)	h/y	3,178	3,182	2,700

16.3.6.3 Drilling and Blasting

For planning purposes, a 5-in drill was assumed. Table 16-34 shows the parameters assumed for mine planning.

Table 16-34: Drill and blast parameters

Description	Unit	PMF and Waste
Emulsion Usage	%	100
Emulsion Density	g/cm^3	1.20
Hole Diameter	inch	5
Bench Height	ft	15
Subdrilling	ft	2.3
Burden	ft	12.5
Spacing	ft	14.4
Retracting Rate	ft/min	78
Powder Factor (mass)	lb/ton	0.61

A 5-in drillhole size was selected to achieve a Scaled Depth of Burial factor below 1. Due to the open pit's proximity to essential surface infrastructure, blasting on the top benches will require extra precautions.

Blasting will be executed under contract with an explosives supplier that will supply the blasting materials and technology, as well as the equipment to store and deliver the explosives products. The explosives ingredients will be delivered to the mine site in containers by the explosives supplier. Based on an assumed 23,000-ton blast pattern, this would require approximately 2–3 blasts per week. This would peak in Years 7 to 9, with 5–6 blasts per week required due to the increased mining rate during those periods.

The average annual explosive consumption is estimated at 1,025,000 tons per year, with a peak of 1,707,000 tons in Year 8.

It is assumed that wall control drill and blast methods will be required for final walls.

For the purposes of estimating the requirements, it has been assumed that the overburden and existing tailings material will be free digging.

The drilling and blasting plan will be optimized during further studies.

Figure 16-35 shows the estimated drill fleet size by period.

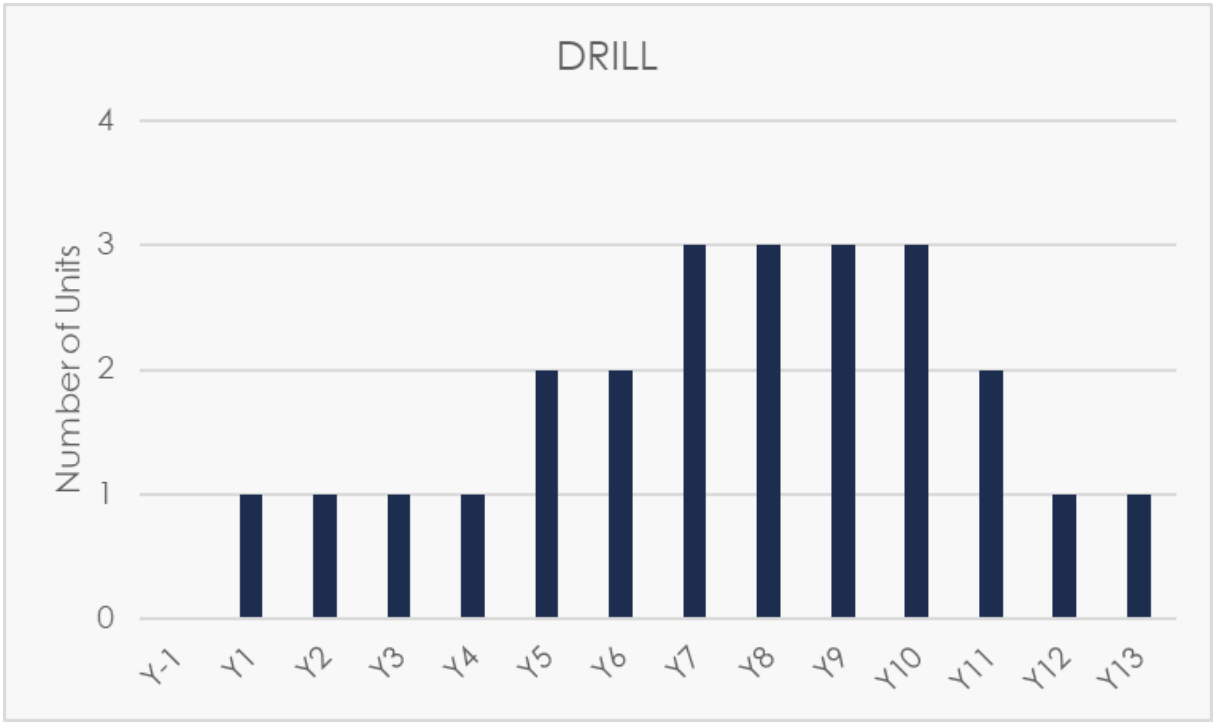


Figure 16-35: Major mine equipment by period - Drilling

16.3.6.4 Loading

A 7.5-ton bucket capacity mining hydraulic excavator is used in diesel configuration for loading of all rock types into the haul trucks. A 11-ton bucket capacity front-end loader is used for stockpile rehandle, pit clean up, and back-up loading. Effective capacity is adjusted by using an 85–90% fill factor. Table 16-35 shows the theoretical loading unit productivity.

Table 16-35: Loading productivity

Description	Unit	Excavator	Loader
Nominal Payload	ton	7.5	11.0
Nominal Capacity	yd^3	5.0	6.5
Load / Bucket	min	0.45	0.75
ROM Material			
# of Passes (Rounded)	#	6	5
Spot & Load Time	min	2.30	3.05
Theoretical Productivity	ton/h	1,100	830
Overburden			
# of Passes (Rounded)	#	6	4
Spot & Load Time	min	2.30	2.30
Theoretical Productivity	ton/h	1,067	1,067
Waste Rock			
# of Passes (Rounded)	#	6	5
Spot & Load Time	min	2.30	3.05
Theoretical Productivity	ton/h	1,100	830

Figure 16-36 shows the estimated loading fleet size by period.

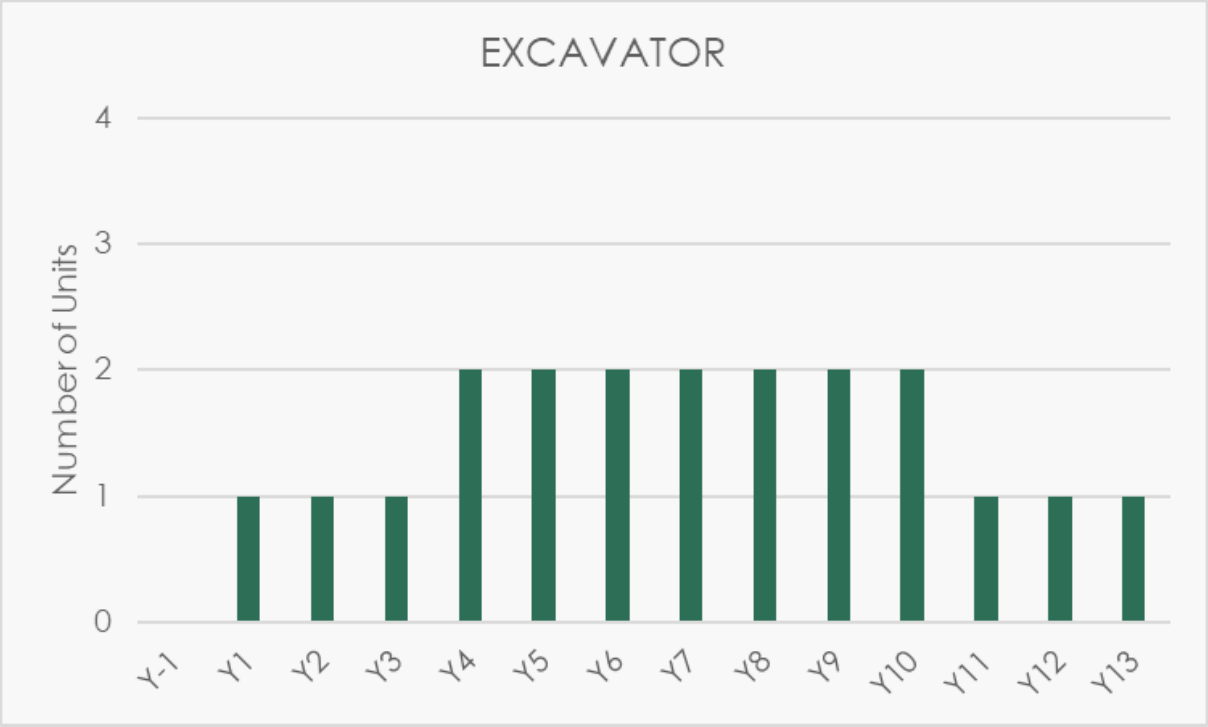


Figure 16-36: Major mine equipment by period - Loading

16.3.6.5 Hauling

A 45-ton capacity articulated haul truck is used for planning with 2% carry back. A representative haul cycle from each phase was used to estimate haul cycle times, accounting for depth at each bench. Various average speeds were applied as provided in Table 16-36.

Table 16-36: Haulage speed assumptions

Description	Loaded Travel		Empty Travel	
	ft/min	km/h	ft/min	km/h
Approach to Loading Unit	820	15	820	15
Bench (flat)	1,094	20	1,367	25
Flat	2,187	40	2,187	40
Incline	820	15	1,367	25
Decline	1,367	25	1,094	20

Effective truck capacity is shown in Table 16-37.

Table 16-37: Truck parameters

Description	Unit	Truck
		45-ton Articulated
Nominal Payload	ton	45
Nominal Capacity	yd^3	34
Spot @ Shovel	min	0.70
First Bucket Dump	min	0.05
Spot @ Dump	min	0.50
Dump Time	min	0.70
Effective Dry Payload – ROM Material	ton	42.2
Effective Dry Payload – Waste Rock Material	ton	42.2
Effective Dry Payload – Overburden Material	ton	40.9
Effective Dry Payload – Existing Tailings Material	ton	35.4

Figure 16-37 shows the estimated hauling fleet size by period.

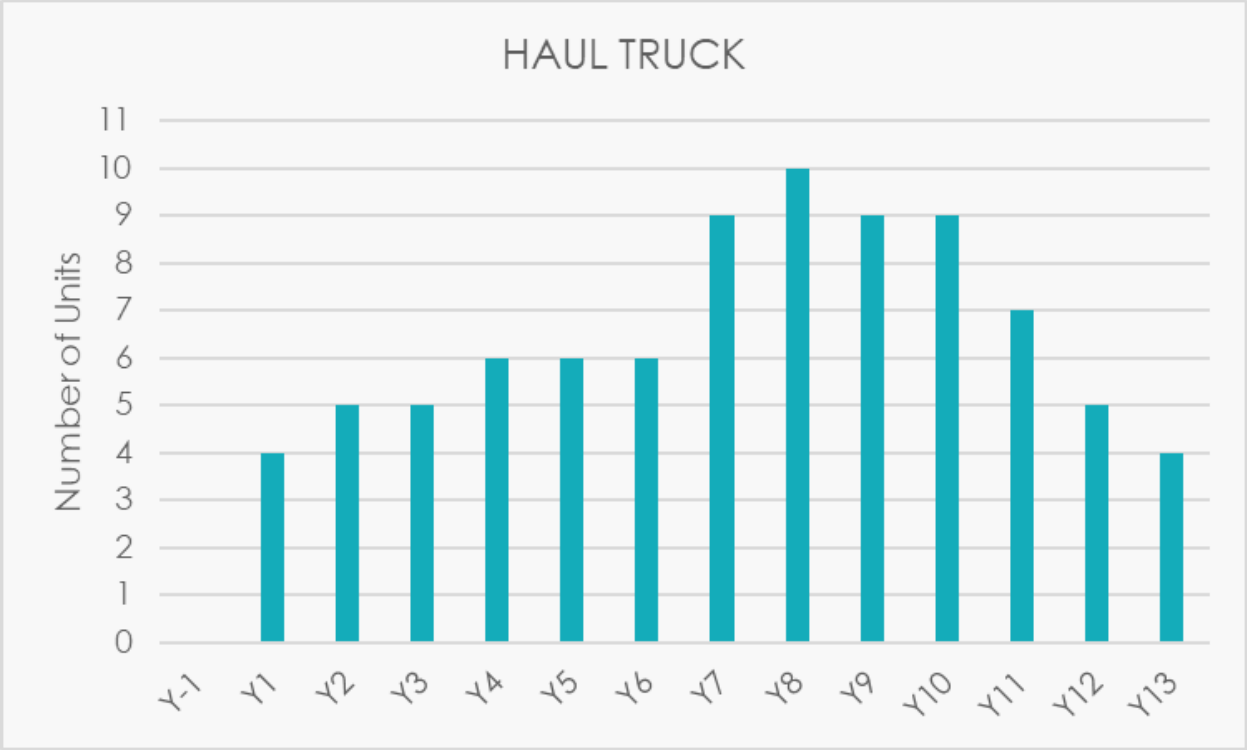


Figure 16-37: Major mine equipment by period - Hauling

16.3.6.6 Auxiliary Equipment

Support equipment is scheduled in order to support the primary equipment. Table 16-38 shows the types of equipment considered.

Table 16-38: Auxiliary equipment used

Description	Quantity
Support Equipment	
Track Dozer	Up to 4
Road Grader	1
Wheel Loader	1
Utility Excavator	1
Water Truck / Sand Spreader	1
Powder Truck	1
Lighting Plant	5
Service Equipment	
Fuel & Lube	1
Mechanic Service Vehicles	2
Equipment Transport Vehicle	1
Crew Transport Vehicle	1
Light-Duty Vehicles	4
Dewatering In-pit Sump Pump	2

16.3.6.7 Mine Dewatering

An allowance for two mobile 220 kW in-pit sump pumps is considered for in-pit dewatering.

16.3.7 Open Pit Workforce

The mine workforce has been calculated to total 40 employees during the first year of operation and will reach a peak of 62 employees in Year 8. The personnel requirement for the open pit mine includes the hourly staff working in open pit operations that are required for the operation and maintenance of the equipment involved with or supporting mining activities, as well as the salaried engineering, geology and supervisory staff.

The number of operators required for the major mining equipment (loading and hauling) was determined according to the number of operating units and number of rotations during which the equipment is in operation. Most of the operators for the major mine equipment are based on a two-crew rotation. Mine maintenance personnel requirements were determined based on a ratio to the pit equipment.

The mine workforce is based on operating two crews on two shifts per day, 5 days per week.

The workforce is separated between three groups: Mine Operations, Mine Maintenance, and Technical Services. Table 16-39 shows the list of positions assumed for the mining operation, while Figure 16-38 shows the estimated workforce levels by year. It has been assumed that some positions will be covered by the zinc operation for the first 5 years of graphite operations. The costs associated with those positions from Year 6 onward are captured under G&A.

Table 16-39 Workforce positions considered

Description	Number of Positions
Mine Operations	
Mine Manager	1
Mine Superintendent	0[1]
Mine Crew Supervisor	2
Drill Operator Excavator Operator Truck Operator Loader Operator	Based on equipment needs 2 per unit
Dozer Operator Grader Operator	Max 4
Water Truck / Excavator Operator	1 per crew
Laborer	1 per crew
Drill Helper	1 per crew
Blaster	1
Blast Crew	1
Utility Crew	2 per crew
Mine Maintenance	
Maintenance Manager	0[1]
Maintenance Superintendent	0[1]
Maintenance Crew Supervisor	1
Maintenance Planner	1
Maintenance Crew	4-8

Description	Number of Positions
Technical Services	
Technical Services Superintendent	0[1]
Senior Mining Engineer	0[1]
Mining Engineer	1
Mine Technician	1
Senior Geologist	0[1]
Geologist	1
Geology Technician	1
Mine Clerk	0[1]

[1] It has been assumed that some positions will be covered by the Zinc Operation.

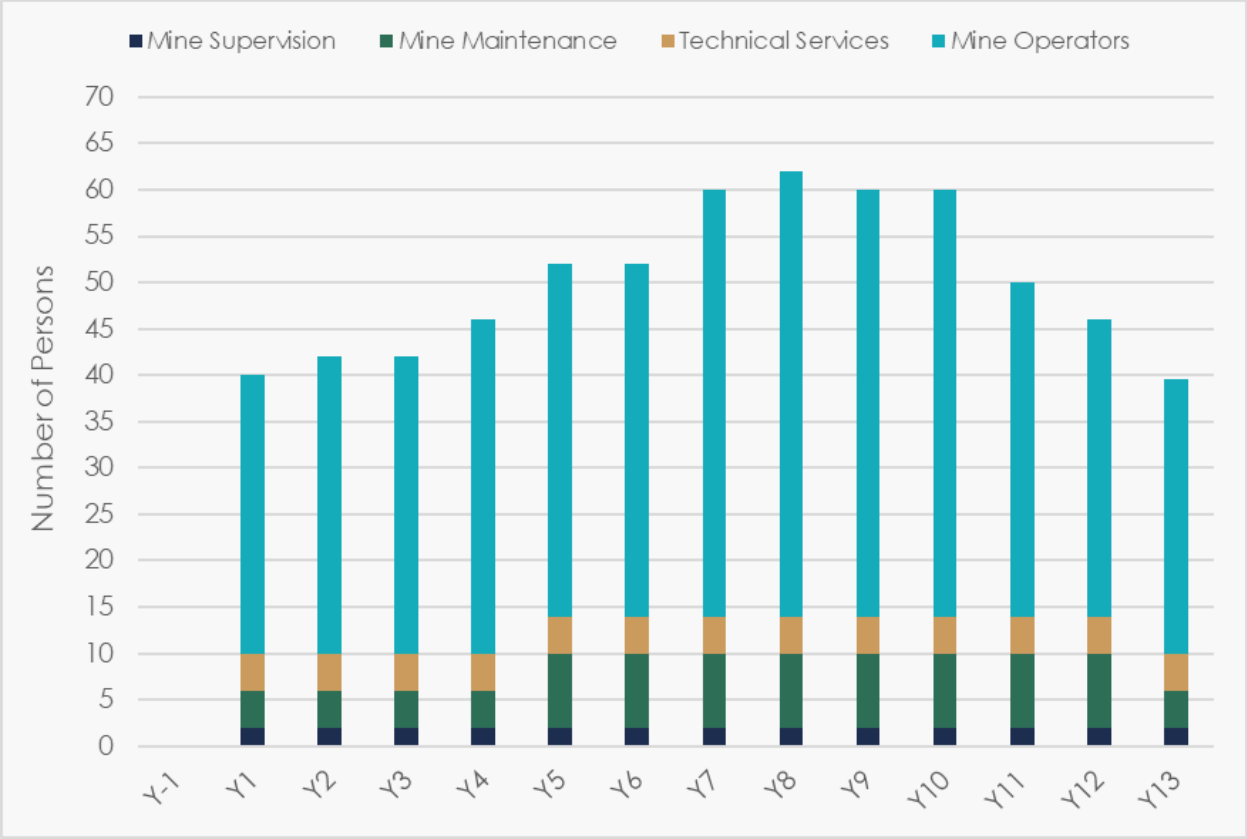


Figure 16-38: Open pit mine workforce by year

17. Recovery Methods

17.1 Zinc

17.1.1 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 design capacity of the concentrator is 5,000 short tons per day (ton/d). Throughout the history of the Balmat operation (now ESM), the capacity of the concentrator has exceeded that of the mines' capacity. The concentrator is currently processing between 10,124 tons and 11,375 tons per week operating on a schedule of one shift per day, 4 days per week. At peak LOM production of 14,000 tons per week, the operating strategy will be to operate the concentrator at its rated hourly throughput of 200 ton/h to 220 ton/h, for only as many hours as necessary to suit production.

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

17.1.2 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 is designed to handle up to 5,000 ton/d, underground mining operations typically produce no more than 2,275 ton/d, with a projected peak of 2,800 ton/d in 2028 as outlined in the LOM plan. The mill is operated for 10 to 14 hours per day and the concentrator suffers no notable losses from intermittent operation.

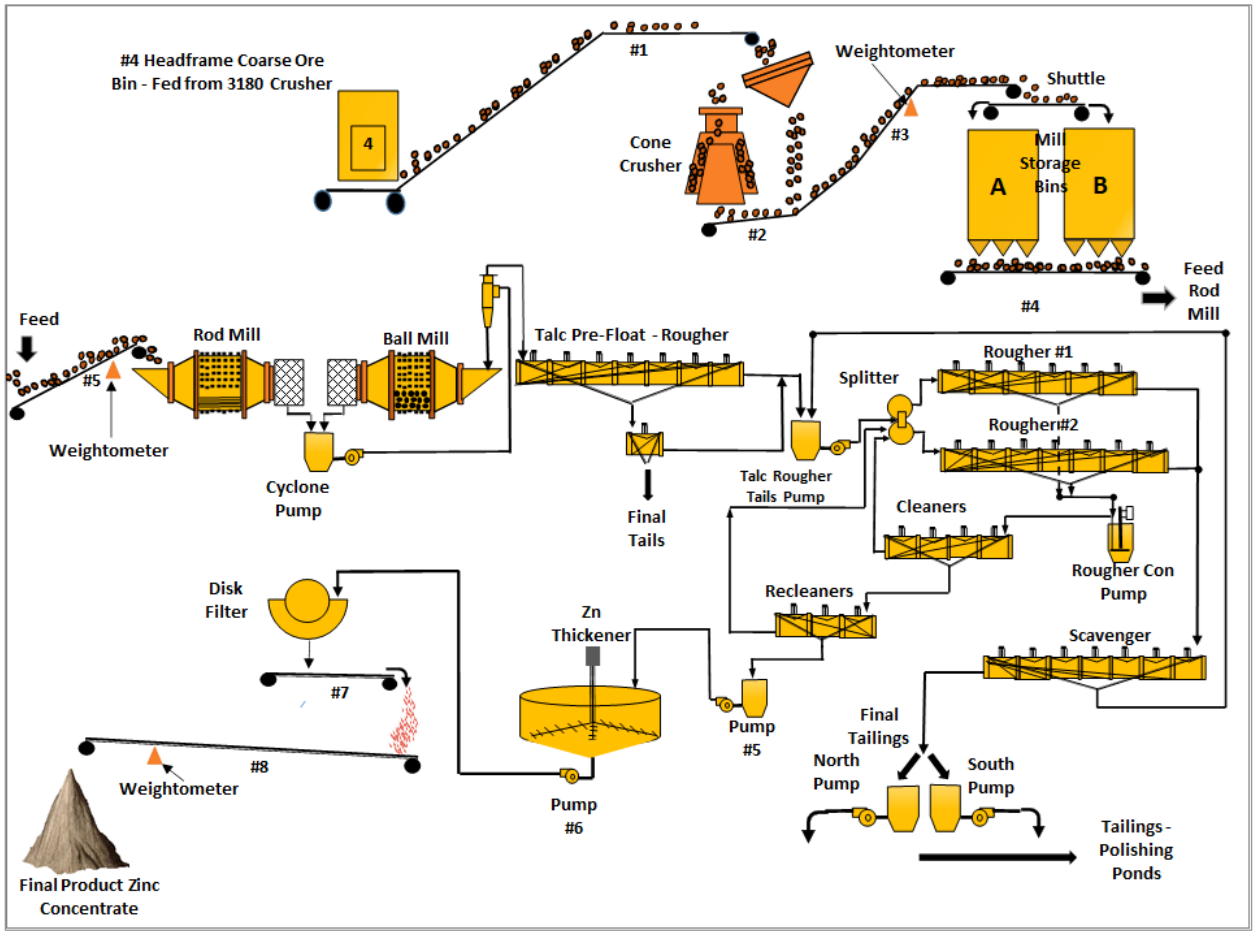


Figure 17-1: Concentrator flowsheet current state

17.1.2.1 Crushing Circuit

Primary crushing is done underground by a 36 in x 48 in jaw crusher, or on surface by a 30 in x 42 in 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 in 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 ft by 14 ft 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 in.

The crusher is an Allis Chalmers Hydrocone, Model 1084 EHD (84 in 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 in and 2 in with corresponding capacities in the order of 275 ton/h to 400 ton/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 tons.

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 ton/h–220 ton/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 in 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	ton/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 2025

17.1.2.2 Fine Mineralized Material Bin

There are two bins with a nominal capacity of 2,000 tons 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.

17.1.2.3 Grinding Circuit

Fine crushed mill feed is conveyed to the rod mill on a 36 in 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 in 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 in diameter balls and operated in closed circuit with two Warman 26-in cyclones.

Typical mill feed rates were in the range of 200 ton/h to 220 ton/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 (Table 17-2). Laboratory testwork 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 testwork 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	ton/h	200
ESM Mill Feed Material Work Index	kWh/ton	8.3
Rod Mill Diameter	ft	11.5
Rod Mill Length	ft	16
Installed Motor on Rod Mill	hp	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 2025

17.1.2.4 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 common 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.

17.1.2.5 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	ton/h	200
Zinc 1st Cleaner Tails to Zinc Roughers	ton/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	ton/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 2025

Table 17-4: Zinc first cleaners design criteria

Design Criteria – Zinc First Cleaners	Unit	Value
Solids Feed Rate into Zinc First Cleaners	ton/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	ton/h	49
▪ % Solids	% w/w	25
▪ Volume	gal/min	640

Source: ESM operating data 2025

Table 17-5: Zinc second cleaners

Design Criteria – Zinc Second Cleaners	Unit	Value
Solids Feed Rate into Zinc Second Cleaners	ton/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	ton/h	41
▪ % Solids	% w/w	36
▪ Flowrate	gal/min	326

Source: ESM operating data 2025

17.1.2.6 Lead Dewatering Circuit

The lead thickener is 40 ft 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 ft 10 in 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.

17.1.2.7 Zinc Dewatering Circuit

The zinc thickener is a 50 ft 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 ft 10 in 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 ft 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.

17.1.2.8 Ancillary Equipment

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.

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.

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.

17.1.3 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

17.1.4 Energy and Process Material Requirements

The existing zinc concentrator and associated mine infrastructure historically draw approximately 6.4 MW at peak load, based on utility data. The concentrator itself accounts for ~1.7 MW. Electrical supply is described in Item 18.

Materials currently used in the zinc plant per ton processed:

- Sodium sulfide: 0.14 lb;
- Sodium cyanide: 0.07 lb;
- Collector C100/3894: 0.06 lb;
- MIBC: 0.05 lb;
- Copper sulfate: 0.45 lb;
- Flocculant: 0.025 lb;
- Caustic (NaOH): 0.035 lb;
- Grinding rods: ~0.00025 ton;
- Grinding balls: ~0.00025 ton.

17.1.5 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	**Total Outflow**	**1,587,000**	**1,726,000**
Run-off and Drain Water	345,000	334,000			
Total Inflow	**1,587,000**	**1,726,000**			

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	**Total Outflow**	**426,000**	**483,000**
Run-off and Drain Water	102,000	75,000			
Total Inflow	**426,000**	**483,000**			

17.1.6 Opportunities for Metallurgical Improvement

The ESM concentrator will be required to operate at approximately 56% of its capacity to process the proposed peak mining rate of 2,800 ton/d. If mine production can be increased, the plant has sufficient capacity to handle the additional tonnage without requiring modifications. 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 60% zinc with 3% iron and 0.5% 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.

17.1.7 Assumptions

- The samples used for the metallurgical testwork are representative of the mineralized material planned to be mined in the Mud Pond and Mahler deposits.
- The results of the metallurgical testwork 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 10.45% based on recent operating data.

17.1.8 Conclusions

Despite its age, the concentrator remains in good operational condition and runs efficiently. No modifications are required to continue processing underground mineralization sources and minimal operational adjustments 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 7.5 hp and 15 hp Ingersoll Rand 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.

17.2 Graphite

17.2.1 Concentrate Plant

This Item outlines the process design basis that was adopted to develop the mass and water balance, process design criteria, and major equipment list for the Graphite Study. The process design is aligned with the mining design described in Item 16.

The process was designed to maximize graphite recovery and minimize flake degradation, while also minimizing capital expenditure and operating costs.

The process consists of a crushing and grinding circuit, rougher and cleaner flotation, and final graphite concentrate dewatering and handling circuit.

The process design is based on metallurgical testwork that was conducted by two independent laboratories. The expected overall graphite concentrate grade and recovery is 95% TC and 90%, respectively. The average feed grade to the mill is 2.84% Cg based on the mine design.

The plant will produce up to 44,500 tonnes of graphite per year and 37,500 tonnes per year on average, with a mill feed range between 1,226,000 and 1,864,000 short tons.

17.2.1.1 Process Design Criteria

Key process design criteria for the graphite Concentrate Plant are provided in Table 17-9.

Table 17-9: Key process design criteria

Description	Unit	Value
Operating Days per Annum	d	365
Operating Hours per Day	h	24
Daily Throughput	ton/d	4,657
Design Annual Throughput	ton/y	1,700,000
Crusher Circuit Availability	%	70
Processing Plant Availability	%	92
Filtration Plant Availability	%	80
Annual Crushing Operating Hours	h/y	6,132
Annual Concentrate Plant Operating Hours	h/y	8,059
Annual Filtration Operation Hours	h/y	7,008
Head Grade	% Cg	2.84
Average Concentrate Grade	% LOI	95
Average Graphite Recovery	%	90
Average Product Mix +100 mesh +200 mesh -200 mesh	 % % %	 6.3 34.0 59.7

17.2.1.2 Block Flow Diagram

The block flow diagram for the Concentrate Plant is presented in Figure 17-2.

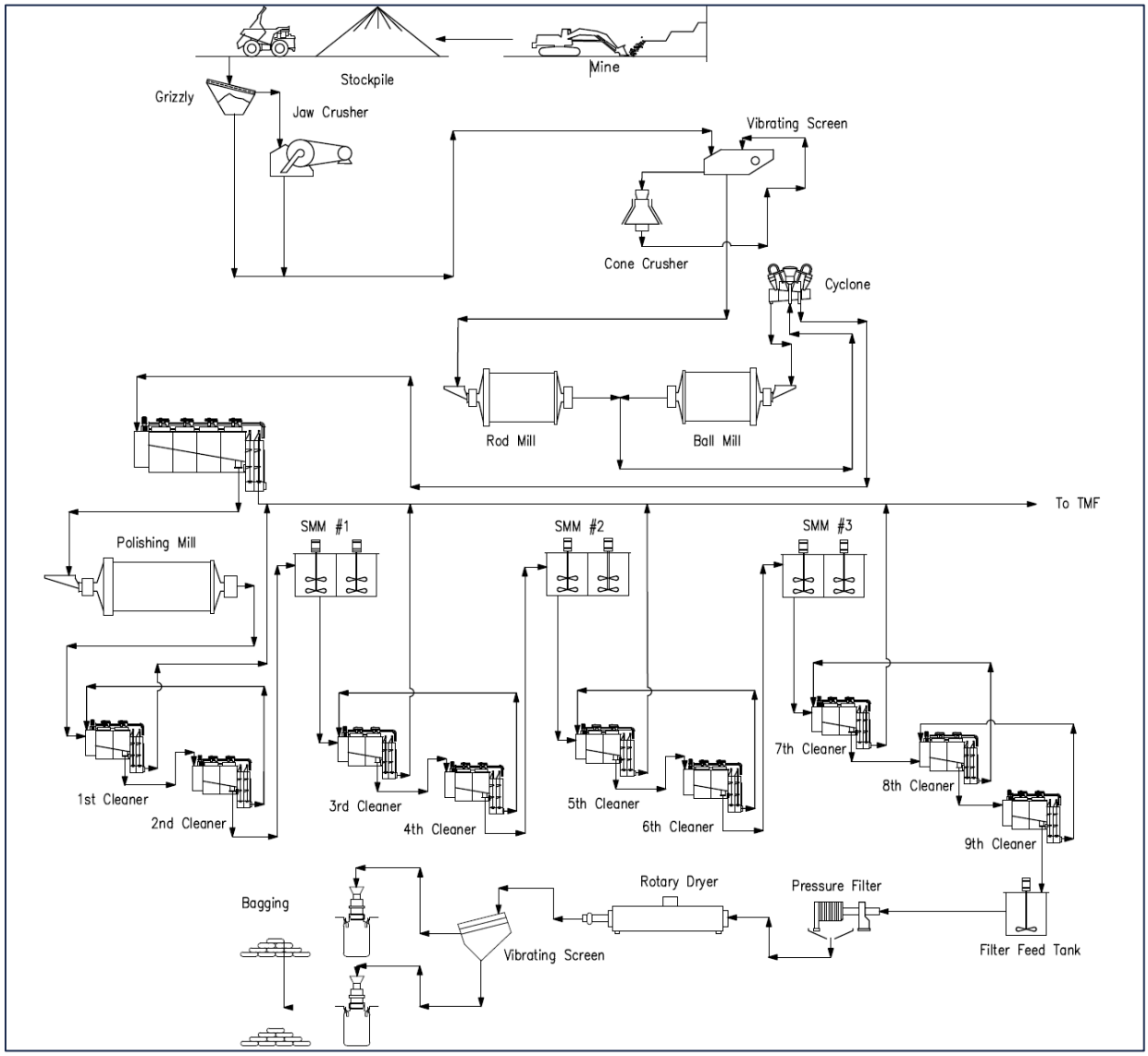


Figure 17-2: Concentrate Plant flowsheet

17.2.1.3 Process Description

Crushing Plant

The crushing plant consists of a jaw crusher operating in open circuit and a cone crusher that operates in closed circuit with a vibrating screen. A front-end loader dumps the mineralized material into a feed hopper with a grizzly feeder. The oversized material is directed to the jaw crusher while the grizzly feeder undersize is conveyed to the vibrating screen together with the jaw crusher product. The screen is equipped with a 25 mm aperture, and any oversized material is conveyed to the cone crusher. The screen undersize is transported to the mill feed bin.

Grinding and Rougher Flotation

Grinding is performed in a rod mill and a ball mill to produce a flotation feed with a P_{80} of 100 microns. The mineralized material is conveyed from the mill feed bin to the rod mill and the rod mill discharge is pumped to a cluster of cyclones. The cyclone underflow gravitates to the ball mill, which discharges into the same cyclone pump box as the rod mill.

The cyclone overflow feeds the graphite rougher flotation cells. The objective of the rougher circuit is to recover most of the graphite in the mineralized material into an intermediate concentrate stream, which is then upgraded to a final concentrate in the cleaning circuit.

Cleaning Circuit

The cleaning circuit upgrades the rougher concentrate from a grade of approximately 11% TC to a final concentrate grading 95% TC using a series of regrind mills and cleaner flotation stages.

The rougher concentrate is pumped to a polishing mill, which uses ceramic media to improve the liberation of the graphite flakes while minimizing flake degradation. The polishing mill discharge is transferred to two stages of cleaner flotation. The concentrate of the first cleaner stage is pumped to the second cleaner and the 1st cleaner tailings are combined with the rougher tailings. The concentrate of the 2nd cleaner feeds the 1st stirred media mill (SMM), while the 2nd cleaner tailings are cycled back to the 1st cleaner feed.

The 1st SMM operates in open circuit and the mill discharge is upgraded in two additional stages of cleaner flotation similar to the 1st and 2nd cleaner. The 3rd cleaner tailings are combined with the rougher tailings and the 4th cleaner concentrate is directed to the second SMM. The regrind and cleaning steps are repeated for the second and third SMM. The only difference in the final cleaning circuit is the addition of a 3rd cleaner stage to maximize concentrate grade. This 9th cleaner concentrate represents the final product of the cleaning circuit and yields a concentrate grade of 95% TC.

Concentrate Dewatering, Dewatering, and Bagging

The final graphite flotation concentrate is pumped to a filter feed tank that feeds a plate and frame pressure filter. The filter cake is conveyed to a dryer to reduce the moisture from approximately 15% w/w in the cake to less than 0.5% w/w. The dried concentrate is transferred to the screening plant that produces +100 mesh, -100 mesh, -100/+200 mesh, and -200 mesh products as required by the market. The products are bagged into bulk bags and smaller 25 kg bags. A certain allotment of the production will be shipped to external customers, and the balance of the material will be forwarded to the Micronization Plant.

Tailings

No dewatering is performed on the tailings. Instead, the graphite tailings slurry is combined with the tailings of the Zinc Operation and pumped to the tailings management facility.

Reagents

The Graphite Study requires a range of reagents that are consistent with other graphite projects, namely diesel and Methyl Isobutyl Carbinol (MIBC).

Diesel is used as the collector for graphite flotation and is transferred from the tanks for the mining fleet to a plant storage tank. Distribution from the plant storage tank to the various addition points is achieved with multiple variable speed peristaltic pumps.

MIBC is used as the frother for graphite flotation and is delivered to site in 1,000-L Intermediate Bulk Containers (IBC) or 45-gallon steel drums. The MIBC is transferred into a plant storage tank and distribution from this tank to the various addition points is achieved with multiple variable speed peristaltic pumps.

Air, Energy, Water, and Process Consumables

Plant and instrument air will be supplied by air compressors. The air used for instrumentation will be dried before distribution. A series of blowers will provide blower air to the flotation cells.

The Concentrate Plant will use three types of water, namely process water, potable water, and raw water. The source of water for the graphite Concentrate Plant will be the same as for the existing zinc concentrator.

17.2.1.4 Equipment List

A high-level equipment list is provided in Table 17-10. This equipment formed the basis for developing the capital cost estimate. The total connected power for plant is 8.7 MW with 7.4 MW drawn.

Table 17-10: List of major mechanical equipment

Description	Specification
Crushing Circuit	
Primary Crusher	Single toggle jaw, 34 ft x 47 ft
Secondary Crusher	Cone HP200
Crusher Closed-Circuit Screen	–

Description	Specification
Milling & Flotation	
Primary Mill	13 ft x 19 ft, 1,100 kW
Secondary Mill	14 ft x 20 ft, 2,000 kW
Rougher Flotation	6 x 30 m^3 cells
Polishing Mill	Heavy duty scrubber
1st Cleaner Flotation	3 x 10 m^3 cells
2nd Cleaner Flotation	2 x 5.0 m^3 cells
Stirred Media Mill #1	Stirred media mill – 10 m^3
3rd Cleaner Flotation	2 x 5.0 m^3 cells
4th Cleaner Flotation	2 x 5.0 m^3 cells
Stirred Media Mill #2	Stirred media mill – 10 m^3
5th Cleaner Flotation	2 x 5.0 m^3 cells
6th Cleaner Flotation	2 x 5.0 m^3 cells
Stirred Media Mill #3	Stirred media mill – 10 m^3
7th Cleaner Flotation	2 x 5.0 m^3 cells
8th Cleaner Flotation	2 x 5.0 m^3 cells
9th Cleaner Flotation	2 x 5.0 m^3 cells
Concentrate Handling	
Concentrate Filter Feed Tank	230 m^3 Agitated Tank
Concentrate Pressure Filter	Plate and frame
Concentrate Dryer	2.25 MMBTU/h rotary dryer
Concentrate Screening Plant	100 and 200 mesh screen cuts, TruBalance style sifters
Concentrate Bagging Plant	Vendor package for three size fractions and 3 bags/hour per system
Reagents	
Collector Mixing System	Vendor Package
Frother Mixing System	Vendor Package
Flocculant Mixing System	Vendor Package

17.2.2 Micronization Plant

Introduction

Only a small percentage of the NFG concentrate is sold as-is and, instead, most of the product is upgraded further to achieve higher sales prices. The first upgrading step consists of grinding the flakes into a micronized product for sale or as the feed material for the Purification Plant.

Process Description

The dried flotation concentrate is micronized in air swept classifier mills to produce micronized graphite with two different size specifications. The graphite flakes are discharged into a hopper and are then transferred into the mill chamber using a pneumatic or auger feed system. The mill is equipped with pin or bar hammers. The particle size is controlled by a rotating classifier by decreasing or increasing the classifier speed. Particles that meet size specifications are removed from the mill using negative pressure air flow. The micronized graphite is captured by a high-efficiency cyclone that discharges directly into bulk bags. Ultra fine graphite in the cyclone overflow is captured by a filter receiver and represents a secondary product.

Design Criteria

The Micronization Plant has been designed for a processing rate of up to 44,500 t/y (dry basis), using a phased approach. The expected availability of the Micronization Plant is 90% for a total annual operating time of 7,183 hours or 6.2 t/h. It is estimated that 97% of the graphite concentrate feeding the air classifier mills is converted to a micronized product with the balance captured in the ultra-fines product.

The process design criteria for the Micronization Plant are shown in Table 17-11.

Table 17-11: Process design criteria for the Micronization Plant

Criteria	Unit	Value
General		
Annual Feed (dry)	t/y	44,500
Annual Production Hours	h/y	7,183
Plant Availability	%	90.0
Hourly Flake Graphite Feed (dry)	t/h	3.34
Micronized Graphite Yield	%	97.0

17.2.3 Secondary Transformation Site

17.2.3.1 Purification Plant

NFG from the Kilbourne deposit is concentrated and micronized at the Kilbourne Site and subsequently shipped and processed in two processing facilities, namely the Purification Plant and the Coated Spherical Purified Graphite (CSPG) Plant, at the Secondary Transformation Site.

The Purification Plant is designed to process 10,670 t/y of micronized NFG concentrate (dry basis) at a fixed carbon (FC) content of ≥95 wt.% and median particle size (d_{50}) of 20 µm to 25 µm. It utilizes conventional acid leaching to produce purified micronized graphite (PMG) with a FC content of ≥99.90 wt.%. The Graphite Study assumes a yield of 94.11 wt.% across the Purification Plant, resulting in a production rate and sale of approximately 10,042 t/y PMG (dry basis).

Figure 17-3 illustrates the main processing steps for the Purification Plant, which comprises a two-stage acid leach, belt filter, dispersion dryer, packaging plant, and all necessary ancillary equipment (conveyors, pumps, etc.).

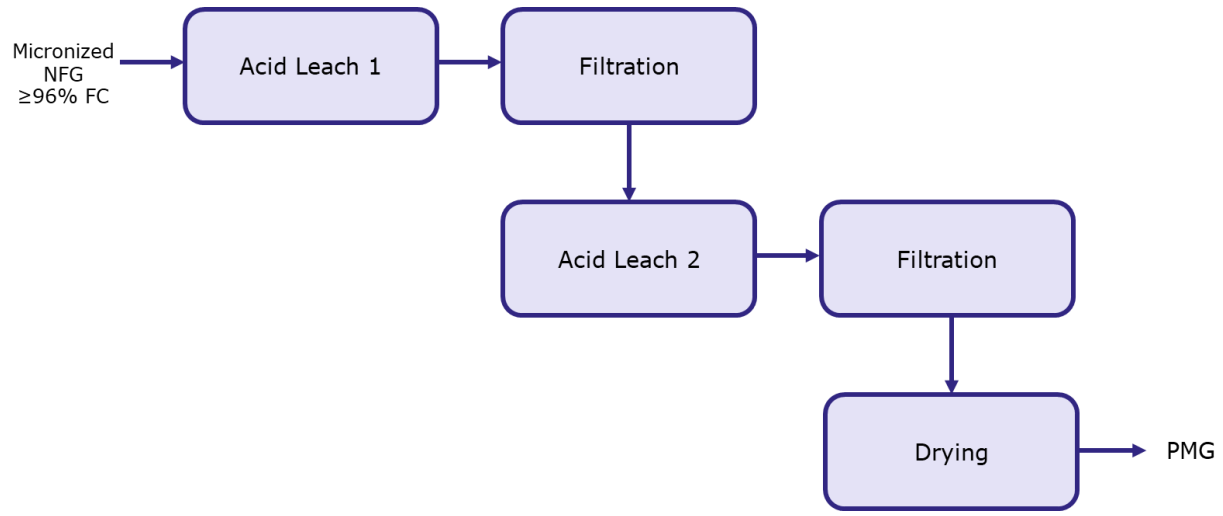


Figure 17-3: Main processing steps in the Purification Plant

17.2.3.1.1 Process Description

Purification Plant

Micronized NFG concentrate is treated in the Purification Plant to remove key impurities, including SiO_2, Al_2O_3 and Fe_2O_3, etc. The process involves a two-stage acid leach using a mixture of hydrofluoric (HF) and hydrochloric (HCl) acid.

During the first acid leaching stage, micronized NFG concentrate is transferred to a series of reaction vessels, where it is suspended in a HF and HCl mixture. The mixture is continuously agitated and heated using steam, allowing impurities within the graphite matrix to dissolve in solution (leach liquor).

The resulting slurry is directed to a belt filter, which separates the graphite solids from leach liquor, producing a filter cake and filtrate. The filter cake is washed in multiple stages with deionized water along the belt filter to remove residual acids and soluble impurities.

The belt filter is equipped with vacuum chambers that facilitate the separation and collection of both leach liquor and wash water. These filtrates are collected and stored for further treatment.

The washed filter cake is fed to the second acid leaching stage, which enables further removal of residual impurities. The second acid leaching stage is identical to the first stage, including the reaction vessels and belt filter.

As in the first stage, the leach liquor and wash water are collected and stored for future treatment. The final purified filter cake is conveyed to the drying stage, to remove residual moisture.

The drying stage utilizes a dispersion dryer, where the purified filter cake is dried by hot air generated by a natural gas-fired furnace. The hot air stream conveys the graphite upwards in a spiral flow within the dispersion dryer, facilitating drying. The dried graphite is fed to a cyclone separator, where the material is classified. Coarse particles are discharged through a rotary valve and collected in a product bin, where fine particles are captured by a dust collector and similarly discharged via a rotary valve into the same product bin.

The final dried PMG is conveyed to a packaging plant where it is sealed into metric tonne (t) bags for storage and sale.

Wastewater Treatment Plant

Wastewater generated from the Purification Plant will be treated in a dedicated wastewater treatment plant. The primary objective of the wastewater treatment plant is to neutralize and remove the acids and other chemical reagents, making it safe for discharge or recirculation (reuse).

Chemical neutralization is one of the key principles utilized in the system. This process involves adding chemical agents, such as calcium hydroxide, sodium hydroxide, or sodium carbonate, to the wastewater. These chemicals react with acids, forming less harmful and more environmentally friendly compounds. Through the neutralization process, the pH of the wastewater is adjusted to a safe level, ensuring compliance with regulatory standards for discharge or further treatment. In addition to chemical neutralization, the wastewater system may incorporate advanced treatment technologies to purify the wastewater further. These technologies include

sedimentation, filtration, activated carbon adsorption, or membrane processes, depending on the specific requirements and contaminants present in the wastewater. These processes remove any remaining impurities, ensuring the wastewater meets stringent environmental and regulatory standards. The final effluent stream should be suitable for disposal to an industrial wastewater treatment facility. However, this will be dependent on the final site selection and the associated environmental and disposal requirements. Wastewater treatment for the Purification Plant will be addressed during the next phase after site location and the subsequent disposal requirements have been concluded.

17.2.3.1.2 Design Criteria

Purification Plant

The Purification Plant processes micronized NFG concentrate with a median particle size (d_{50}) of 20–25 µm and a FC content of ≥95 wt.%, at a processing rate of 10,670 t/y concentrate (dry basis). Operating at 7,500 hours annually, this corresponds to an hourly feed rate of 1.42 t/h.

An average mass yield of 94.11 wt.% is assumed in the Purification Plant, with losses primarily attributed to dissolution during impurity removal, and subsequent losses to the leach liquor (mother liquor) and wash filtrates.

PMG is produced at a rate of approximately 10,042 t/y (dry basis), with a final FC content of 99.90 wt.%.

The PDC for the Purification Plant, summarized in Table 17-12, is based on criteria drawn from Anzaplan's database for plants of similar size and requirements.

Table 17-12: Process design criteria for the Purification Plant

Criteria	Unit	Value
General		
Annual Micronized NFG Feed (dry)	t/y	10,670
Annual Production Hours	h/y	7,500
Plant Availability	%	85.62
Hourly Micronized NFG Feed (dry)	t/h	1.42
Feed (micronized NFG) FC	wt.%	≥95
Average Mass Yield	wt.%	94.11
Product (PMG) FC	wt.%	99.90

Criteria	Unit	Value
Acid Leaching 1		
HF Addition (100 wt.%)	$t/t_{graphite\ feed}$	0.046
HF Concentration in Leach Solution	wt.%	40
HCl Addition (100 wt.%)	$t/t_{graphite\ feed}$	0.050
HCl Concentration in Leach Solution	wt.%	32
Mass Loss Due to Handling	%	0.26
Mass Loss in Purification	%	0.74
FC in Solids After Leaching	wt.%	97.00
Solid Content After Acid Leaching	wt.%	30
Solid Content in Filter Cake	wt.%	75
Wash Ratio	$t/t_{graphite\ feed}$	1
FC Loss in Filtrate	wt.%	1.03
Acid Leaching 2		
HF Addition (100 wt.%)	$t/t_{graphite\ feed}$	0.046
HF Concentration in Leach Solution	wt.%	40
HCl Addition (100 wt.%)	$t/t_{graphite\ feed}$	0.05
HCl Concentration in Leach Solution	wt.%	32
Mass Loss Due to Handling	%	0.81
Mass Loss in Purification	%	2.14
FC in Solids After Leaching	wt.%	99.90
Solid Content After Acid Leaching	wt.%	30
Solid Content in Filter Cake	wt.%	75
Wash Ratio	$t/t_{graphite\ feed}$	1
FC Loss in Filtrates	wt.%	1.03
Drying		
Moisture Content Product	wt.%	0.5

17.2.3.1.3 Mass Balance

Purification Plant

The mass balance for the Purification Plant is based on the PDC (Item 17.2.3.1.2 and is summarized in Table 17-13.

Table 17-13: Mass balance for the Purification Plant

Streams	Total Mass	Solids Content	FC
	(t/y)	(wt.%)	(wt.%)
Acid Leach 1			
Graphite Feed	10,702	99.70	95.00
HF Addition	1,227	0.00	0.00
HCl Addition	1,667	0.00	0.00
Water Addition	21,615	0.00	0.00
Leached Slurry	35,211	30.00	97.00
Acid Leach 1 Filtration			
Feed	35,211	30.00%	97.00
Wash Water	10,670	0.00%	0.00
Filter Cake	13,939	75.00%	97.00
Mother Liquor	21,272	0.51%	97.00
Wash Discharge	10,670	0.00%	0.00
Acid Leach 2			
Filter Cake from Acid Leach 1	13,939	75.00	97.00
HF Addition	1,202	0.00	0.00
HCl Addition	1,634	0.00	0.00
Water Addition	17,045	0.00	0.00
Leach Slurry	33,820	30.00	99.90
Acid Leach 2 Filtration			
Feed	33,820	30.00	99.90
Wash Water	10,455	0.00	0.00
Filter Cake	13,389	75.00	99.90
Mother Liquor	20,432	0.51	99.90
Wash Discharge	10,455	0.00	0.00
Drying			
Feed	13,389	75.00	99.90
Evaporated Water	3,297	0.00	0.00
Total Dry Product	10,092	99.50	99.90
Wastewater			
Total Wastewater to be Treated	62,828	0.34	98.42

17.2.3.2 CSPG Plant

Micronized NFG concentrate is also processed in an independent CSPG Plant at the Secondary Transformation Site, which has a design capacity of 21,340 t/y. The CSPG Plant includes a dedicated purification circuit that follows the same process flow as the Purification Plant described in Item 17.2.3. Within the CSPG Plant, the purification stage serves as an intermediary step, producing PMG at a FC content of ≥99.95 wt.%, which is the primary feedstock for CSPG production.

Following purification, PMG is directed to the spheroidization stage for size reduction and shaping. Based on the Graphite Study, a 70 wt.% yield is assumed for spheroidization, producing approximately 14,058 t/y of SPG (dry basis). The SPG is coated with pitch tar, producing 14,761 t/y of CSPG (dry basis).

Figure 17-4 presents the main processing steps for the CSPG Plant.

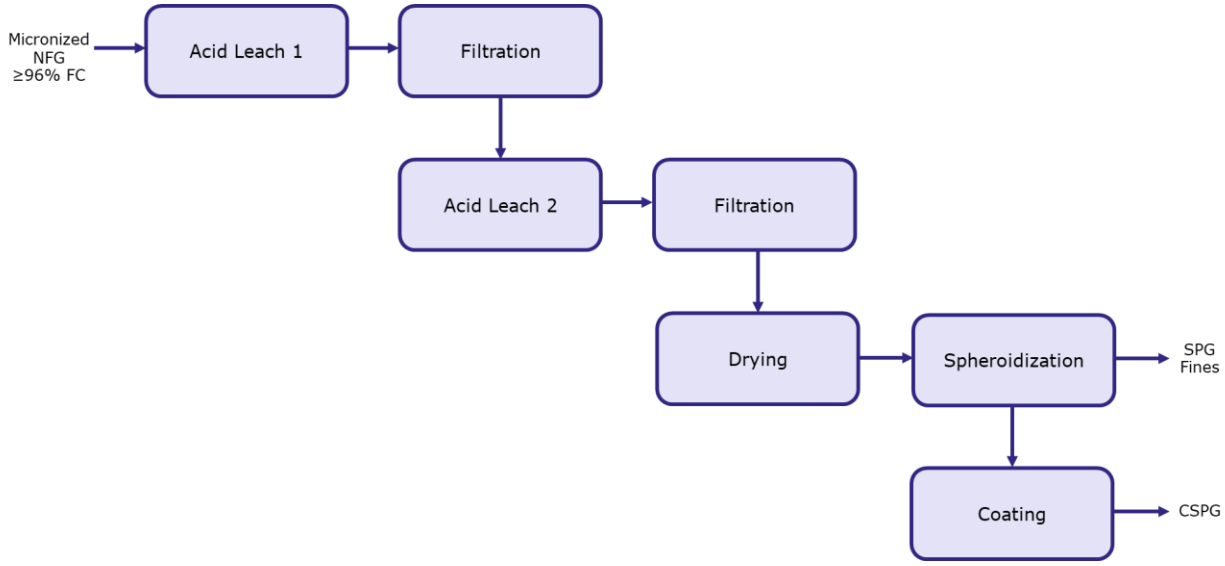


Figure 17-4: Main processing steps in the CSPG Plant

17.2.3.2.1 Process Description

Purification

The purification stage of the CSPG Plant follows an identical process to that of the Purification Plant described in Item 17.2.3.1.1.

Spheroidization

After purification, the resulting PMG is spheroidized in specialized mills to produce SPG. This mechanical shaping creates smooth, rounded particles that improve packing density and conductivity, both of which are key properties for battery performance.

Any fines are collected in a packaging plant and sealed into metric tonne (t) bags for storage.

Coating

The SPG product is coated with pitch tar to enhance its rate of performance, particularly in lithium-ion batteries (LIB) applications. The coating process includes milling the pitch tar, mixing it with SPG, and thermally applying the mixture in a pusher furnace.

Pitch tar is first conveyed to a jet mill, where it is reduced into fine particles. These fine particles are mixed with SPG in a mixing unit and the resulting mixture is conveyed into saggars, which are automatically stacked onto a conveyor leading to the pusher furnace. Inside the furnace, the temperature is gradually raised to 1,200 °C, causing the pitch tar to decompose and deposit a carbon layer onto the SPG surface, forming CSPG.

At the end of the furnace line, saggars are de-stacked, emptied, and cleaned before being returned for refilling. The CSPG is loosened, deagglomerated on a screen, and demagnetized, and finally conveyed to a packaging plant where it is bagged for storage and sale.

Wastewater Treatment Plant

Wastewater generated in the CSPG Plant will be treated in the wastewater treatment plant, which will be addressed in the next phase of work following final site selection.

17.2.3.2.2 Design Criteria

Purification

The purification stage of the CSPG Plant processes 21,340 t/y (dry basis) of micronized NFG concentrate at a FC content of ≥95 wt.%. Operating at 7,500 hours annually, this corresponds to an hourly feed rate of 2.85 t/h.

Similarly to the Purification Plant in Item 17.2.3, an average mass yield of 94.11 wt.% is assumed for the purification stage of the CSPG Plant. PMG is thus produced at a rate of approximately 20,083 t/y (dry basis), with a final FC content of 99.95 wt.%.

The PDC for the purification stage is summarized in Table 17-14, which is based on criteria drawn from Anzaplan's database for plants of similar size and requirements.

Table 17-14: Process design criteria for purification stage of CSPG Plant

Criteria	Unit	Value
General		
Annual Micronized NFG Feed (dry)	t/y	21,340
Annual Production Hours	h/y	7,500
Plant Availability	%	85.62
Hourly Micronized NFG Feed (dry)	t/h	2.85
Feed (micronized NFG) FC	wt.%	≥95.00
Average Mass Yield for Purification	wt.%	94.11
Product (PMG) FC	wt.%	99.95
Acid Leaching 1		
HF Addition (100 wt.%)	$t/t_{graphite\ feed}$	0.046
HF Concentration in Leach Solution	wt.%	40
HCl Addition (100 wt.%)	$t/t_{graphite\ feed}$	0.050
HCl Concentration in Leach Solution	wt.%	32
Mass Loss Due to Handling	%	0.26
Mass Loss in Purification	%	0.74
FC in Solids After Leaching	wt.%	97.00
Solid Content After Acid Leaching	wt.%	30
Solid Content in Filter Cake	wt.%	75
Wash Ratio	$t/t_{graphite\ feed}$	1
FC Loss in Filtrate	wt.%	1.03
Acid Leaching 2		
HF Addition (100 wt.%)	$t/t_{graphite\ feed}$	0.046
HF Concentration in Leach Solution	wt.%	40
HCl Addition (100 wt.%)	$t/t_{graphite\ feed}$	0.05
HCl Concentration in Leach Solution	wt.%	32
Mass Loss Due to Handling	%	0.81
Mass Loss in Purification	%	2.14
FC in Solids After Leaching	wt.%	99.95
Solid Content After Acid Leaching	wt.%	30
Solid Content in Filter Cake	wt.%	75
Wash Ratio	$t/t_{graphite\ feed}$	1
FC Loss in Filtrates	wt.%	1.03
Drying		
Moisture Content Product	wt.%	0.5

Spheroidization

Following purification, PMG is spheroidized at a rate of 20,083 t/y, corresponding to an hourly feed rate of 2.68 t/h based on 7,500 annual operating hours. Assuming an overall spheroidization yield of 70 wt.%, this process produces approximately 14,058 t/y SPG (dry basis).

The PDC for spheroidization is outlined in Table 17-15.

Table 17-15: Process design criteria for spheroidization stage of CSPG Plant

Criteria	Unit	Value
General		
Annual Feed Rate (dry)	t/y	20,083
Annual Production Hours	h/y	7,500
Plant Availability	%	85.62
Hourly Feed Rate (dry)	t/h	2.68
Feed (PMG) FC	wt.-%	99.95
Product (SPG) FC	wt.-%	99.95
Average Mass Yield for Spheroidization	wt.%	70

Coating

The coating section is designed to process SPG at a rate of 14,058 t/y, while operating at 7,500 h/y. Pitch tar is added at 0.1 $t/t_{graphite\ feed}$ of dry feed, producing approximately 14,761 t/y CSPG (dry basis).

The PDC for the coating section is outlined in Table 17-16.

Table 17-16: Process design criteria for coating stage of CSPG Plant

Criteria	Unit	Value
General		
Annual Feed Rate (dry)	t/y	14,058
Annual Production Hours	h/y	7,500
Plant Availability	%	85.62
Feed (PMG) FC	wt.-%	99.95
Product (CSPG) FC	wt.-%	99.95
Average Mass Yield for Coating	wt.-%	105
Coating		
Pitch Tar Addition	$t/t_{graphite\ feed}$	0.10
Percentage of Pitch Tar Lost to Volatiles	%	50.00

17.2.3.2.3 Mass Balance

The mass balance for the CSPG Plant is based on the PDC (Item 17.2.3.2.2) and is summarized in Table 17-17.

Table 17-17: Mass balance for the CSPG Plant

Streams	Total Mass (t/y)	Solids Content (wt.%)	FC (wt.%)
Purification			
Acid Leach 1			
Graphite Feed	21,404	99.70	95.00
HF Addition	2,454	0.00	0.00
HCl Addition	3,334	0.00	0.00
Water Addition	43,229	0.00	0.00
Leached Slurry	70,422	30.00	97.00
Acid Leach 1 Filtration			
Feed	70,422	30.00	97.00
Wash Water	21,340	0.00	0.00
Filter Cake	27,879	75.00	97.00
Mother Liquor	42,543	0.51	97.00
Wash Discharge	21,340	0.00	0.00
Acid Leach 2			
Filter Cake from Acid Leach 1	27,879	75.00	97.00
HF Addition	2,405	0.00	0.00
HCl Addition	3,267	0.00	0.00
Water Addition	34,090	0.00	0.00
Leach Slurry	67,641	30.00	99.95
Acid Leach 2 Filtration			
Feed	67,641	30.00	99.95
Wash Water	20,909	0.00	0.00
Filter Cake	26,778	75.00	99.95
Mother Liquor	40,863	0.51	99.95
Wash Discharge	20,909	0.00	0.00
Drying			
Feed	26,778	75.00	99.95
Evaporated Water	6,594	0.00	0.00
Total dry Product	20,184	99.50	99.95

Streams	Total Mass (t/y)	Solids Content (wt.%)	FC (wt.%)
Wastewater			
Total Wastewater to be Treated	125,655	0.34	98.45
Spheroidization			
Feed SPG Production	20,184	99.50	99.95
SPG Product	14,129	99.50	99.95
SPG Fines Byproduct	6,055	99.50	99.95
Coating			
Feed Coating	14,129	99.50	99.95
Pitch Tar	1,406	100.00	0.00
CSPG Product	14,832	99.52	99.95

17.2.3.3 Energy and Process Materials

The energy and process materials requirements for the Graphite processing streams are identified in Item 21.

18. Project Infrastructure

18.1 Zinc

18.1.1 General Site Arrangement

The general site arrangement is depicted in Figure 18-1. No modifications to the site layout have been made since mine closure by the previous mine operator in 2008.

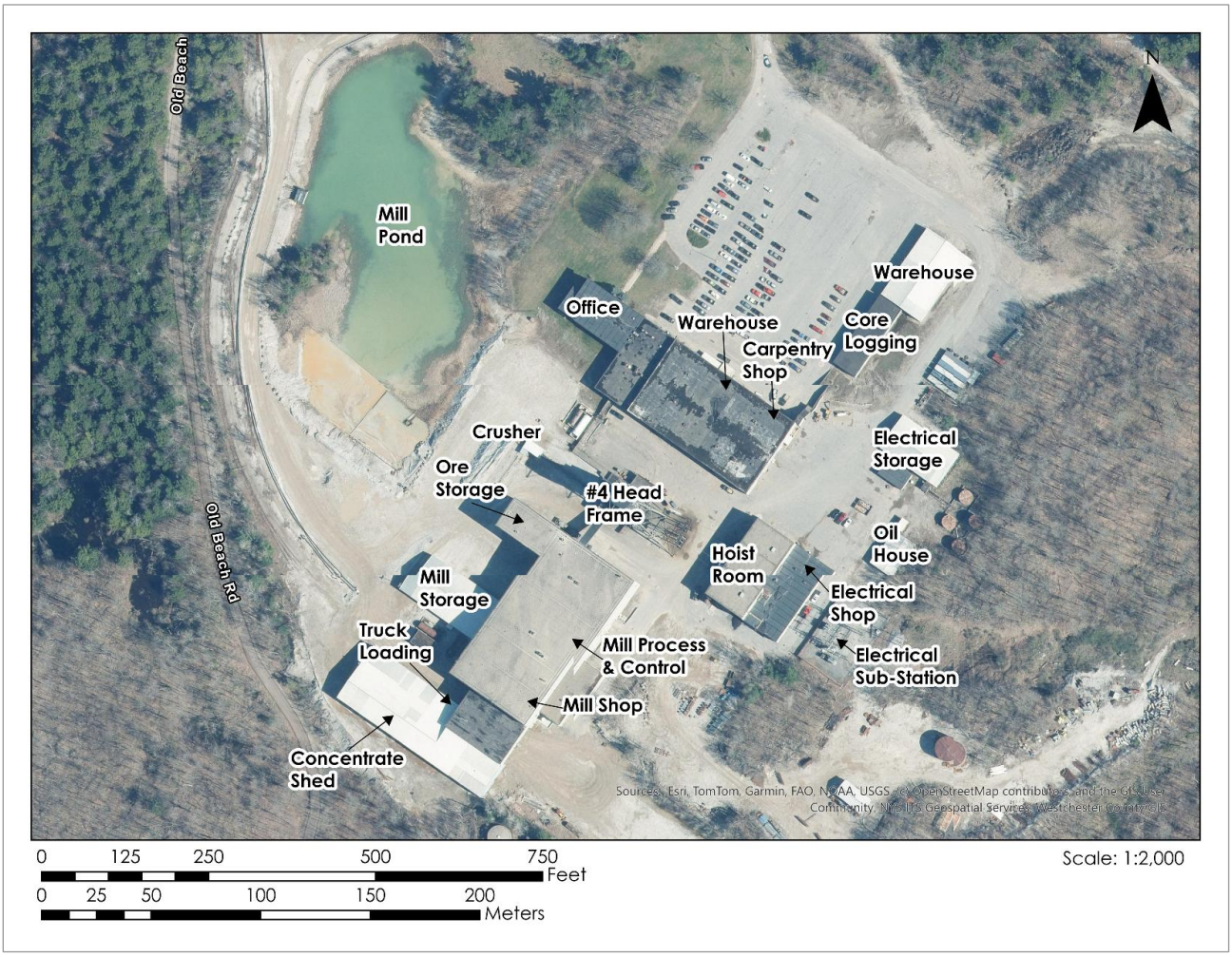


Figure 18-1: ESM general site arrangement

18.1.2 Buildings and Structures

Northeast Construction Company 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 starting in April 1970 until its completion in August 1971. The new mill started operations in the spring of 1972.

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.

18.1.2.1 Office Complex

The existing mine office complex is a two-story steel frame and concrete block structure with galbestos siding and a 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 personnel with individual lockers for clean clothes and hanging baskets for working clothes, as well as sufficient sanitary facilities.

A foreman's locker room, located near the front of the floor, accommodates 25 supervisors and visitors. An additional locker room near the main lobby accommodates 15 people.

The ground floor also contains mine offices, a boiler room and a lamp room. The boiler room houses two Cleaver Brooks 250 hp boilers. Hot water for sanitary purposes is provided by a 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 is equipped with a 15-ton overhead crane and the machine shop features a 25-ton crane.

For the ESM operation, shipping/receiving is handled through the surface warehouse. A second warehouse, located on the 2500 level underground, is part of the mine maintenance shop complex and is used to store equipment parts. A third warehouse facility is located in the 3100 level shop, providing additional storage capacity and logistical support for underground operations. One warehouse employee is assigned to the underground warehouse operations, with plans to increase to two shortly.

The first and second floors of the north-western brick-faced extension of the building (64 ft x 103 ft each floor) is used as office space and currently is organized to accommodate the following personnel and requirements:

- President & CEO;
- Vice President of Operations;
- Vice President of Commercial & Sales;
- Project & Surface Manager;
- Mine Manager;
- Mine Clerk and Surveying;
- Engineering and Geology Personnel;
- Conference Room;
- Accounting, Purchasing, and Human Resources.

18.1.2.2 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 stands 145 ft high and is fully clad. The hoistroom (135 ft x 138 ft) is equipped with 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 which is equipped with a 5-ton Shaw Box crane.

#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 in wire rope which services the production skip compartment. The lower sheave deck supports two 12 ft diameter head sheaves grooved for 1 ¾ in 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 4 ft wide by 14.5 ft long 20 hp Portec apron feeder.

Production Hoisting Plant

The production hoist is a Nordberg double-drum, double clutch mine hoist with Lebus grooving. The production hoist features two 15 ft diameter by 8 ft wide drums each with capacity to handle 3,990 ft of 2 ¼ in 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 feet per minute. The resultant hoisting rate is 200 t/h. Shaft and hoist related maintenance tasks that impact daily hoisting capacity are shown Table 18-1.

Table 18-1: #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: Modified from Warren et al., 2021

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 in wide drums each holding 3,990 ft of 1 ¾ in head rope and driven by a single 900 hp 400 rpm DC motor. The maximum hoisting speed is 1,190 feet 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.

#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.

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 in diameter by 76 in wide drums each holding 3,990 ft of 1¼ in head rope and driven by a single 700 hp 514 rpm wound rotor induction motor. The maximum hoisting speed is 1,150 feet 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.

18.1.2.3 Concentrator and Support Facilities

The 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 ft x 267 ft + 46 ft x 80 ft + 67 ft x 97 ft) complete with a raised mill control room, physical and analytical labs, offices, and X-ray room.

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

18.1.2.4 #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.

18.1.2.5 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 (ft x ft)
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: Macdonald et al., 2017

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

18.1.3 Power

The primary feed for 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 #4 main ventilation fan location, there is a 1,000 kVA 4,160 volt to 480-volt stepdown 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: a 125 kVA unit designated for operating the No. 4 service hoist, and a 100 kVA unit intended to power the No. 2 emergency egress hoist.

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

18.1.4 Water Control Infrastructure

18.1.4.1 Water Supply

The current non-potable water supply system is adequate to supply the ESM Project for showers, boiler make up, toilet facilities, etc. with no modifications envisaged at this time. Non-potable water is 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 runs through an underground 2 in Sclairpipe high-density polyethylene (HDPE) pipe to the vehicle storage building where it is treated by a Magnum CY 962 water softener before it enters 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 delivers 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) which is used for toilets and showers.

The chlorine residual is monitored on a daily basis and the result recorded as per NYS Dept. of Health code 360. The Department of Health reviews this report on a monthly basis. A monthly water sample is submitted for a coliform bacteria test.

Mill process and cooling water (non-potable) for the site is 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 constitutes excess capacity and the other two cycles off and on. Pump discharge is through a 10 in 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 is pumped from the booster pump house via the 4-in shaft water line to the various mine levels.

Grey water from the surface facilities, surface runoff, water from the facility catch basins, and overflow from the reservoir tank is directed to the mill holding pond. Wastewater from the holding pond is either recycled in the mill or pumped to the tailings dam through a 5,000-ft pipeline consisting of 14 in diameter HDPE pipe. From the tailings area, it will flow northeast through a series of settling and polishing ponds before it will be discharged to the environment.

18.1.4.2 Water Treatment

The current concentrator operation utilizes the addition of lime to the process tailings at the rate of approximately two tons per day. This facilitates the precipitation of metals and solids from the tailings slurry. During the colder months, it is necessary to continue the lime addition when the concentrator is not operating and the facility is discharging only mine dewater to aid in maintaining zinc and iron levels at the outfall within permitted ranges. During warmer months, this practice is normally discontinued. Additional treatment is available should it be required based on water chemistry observed at the outfall. A sodium sulfide dosing system is maintained near the start of the serpentine ponds. Sodium sulfide has been shown to be very effective in the reduction of zinc and iron levels in the effluent. The treated water is discharged by gravity through SPDES Discharge Point #001, consistently meeting regulatory standards. Since January 2009, mine water treatment has been successfully conducted using lime. Based on this operational experience, it is anticipated that the contact water treatment approach for the Graphite Study will be similar, involving sedimentation through a designated pond system, followed by pH adjustment prior to any controlled environmental discharge.

18.1.4.3 Water Balance

Mine water balances are calculated seasonally for May to October (summer) and November to April (winter). The average total flow for summer is 212,490,000 gal. During the winter months the average total flow increases to 232,744,500 gal. The average daily discharge through SPDES Outfall Point #001 is 1,219,820 gal/day.

18.1.5 Waste Rock Management

Waste rock generated during mining operations remains underground and is not hoisted to the surface. The waste rock is used as backfill for drift and fill mining or deposited in completed longhole stopes.

18.1.6 Tailings Management Facility

The ESM TMF is a fully permitted, 260-acre conventional impoundment currently used for permanent storage of tailings from zinc operations. The TMF includes four contiguous areas: Tailings Pond #1 (TP1), Tailings Pond #2 (TP2), the Reclaimed Tails Area, and a series of polishing ponds. TP1 is the active deposition area, while TP2 and the polishing ponds provide clarification and passive treatment prior to discharge through SPDES Outfall #0001.

Tailings and mine-impacted water are pumped to the TMF at approximately 1,600 gal/min. Water quality is managed through seasonal dosing of slaked lime and sodium sulfide. The TMF is classified as low-risk by the New York State Bureau of Flood Protection and Dam Safety and as "Low Hazard" by MSHA. However, no as-built records exist, any future embankment raises will require geotechnical validation. The estimated remaining capacity of TP1 is approximately 3.5 years at 450,000 tons per year.

To support the Kilbourne Graphite Study, the TMF will be expanded and reconfigured to accommodate both zinc and graphite tailings over the LOM. This includes:

- **Raised TMF near South Dam**: Adds 28.6 million ft^3 (M ft^3) of capacity for relocated tailings from the Kilbourne Pit footprint.
- **Extended TMF downstream of South Dam**: Adds 194.9 M ft^3 of capacity through staged construction with containment dikes.
- **Historic Arnold Pit Backfilling**: Repurposed for tailings deposition following dewatering and containment dike construction, with up to 168.8 M ft^3 capacity.

Tailings deposition will follow a phased strategy:

- Initial deposition in the Extended TMF (Years 1–5);
- Transition to Historic Arnold Pit (Years 6–13);
- Raised TMF used for relocated tailings from pit development.

The reader is referred to Item 18.2.6 for additional details.

Tailings and waste rock materials at the existing 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.

18.1.7 Concentrate Transportation

18.1.7.1 Roads

A well-maintained system of paved state and county roads surrounds ESM, providing a year-round option to transport concentrate to a port or smelter by truck if required. The concentrate loading shed at 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 in Canada following Highways 401 and 201.

18.2 Graphite

18.2.1 General Overview

The Kilbourne Graphite Study covers all major facilities and infrastructure required to support mining, processing, and concentrate transformation. The Graphite Study involves two primary sites:

- **Kilbourne Site:** Located adjacent to the existing ESM location approximately 107 miles north of Syracuse, NY. This site includes the open pit mine operations, supporting site infrastructure and tailings management facilities, as well as the Concentrate and Micronization plants;
- **Secondary Transformation Site:** A separate location dedicated to the construction of the Purification Plant and CSPG Plant, where micronized NFG will be further processed to produce market-ready material.

Kilbourne Site infrastructure will support the open pit mine, delivering on average 1,534,700 tons of mill feed to the on-site graphite Concentrate Plant, resulting in approximately 40,000 metric tonnes of graphite concentrate per year. The overall site layout showing the location of the open pit, Concentrate Plant, material stockpiles and tailings facilities is provided in Figure 18-2.

With this Project being developed near the well-established Empire State Mines, the infrastructure required to support graphite production at the Kilbourne Project will take advantage of existing facilities wherever possible.

While new facilities will be established to support the graphite operation, the following items will be accommodated with existing site:

- Offices for management, engineering, operations and administrative personnel;
- Dry facility;
- Training, first aid and emergency response;
- Core shack/logging and core storage;

- Garbage/waste management;
- Potable water supply;
- Explosives magazine;
- Parking for personnel.

The following section describes all new infrastructure and any required modifications to existing facilities.

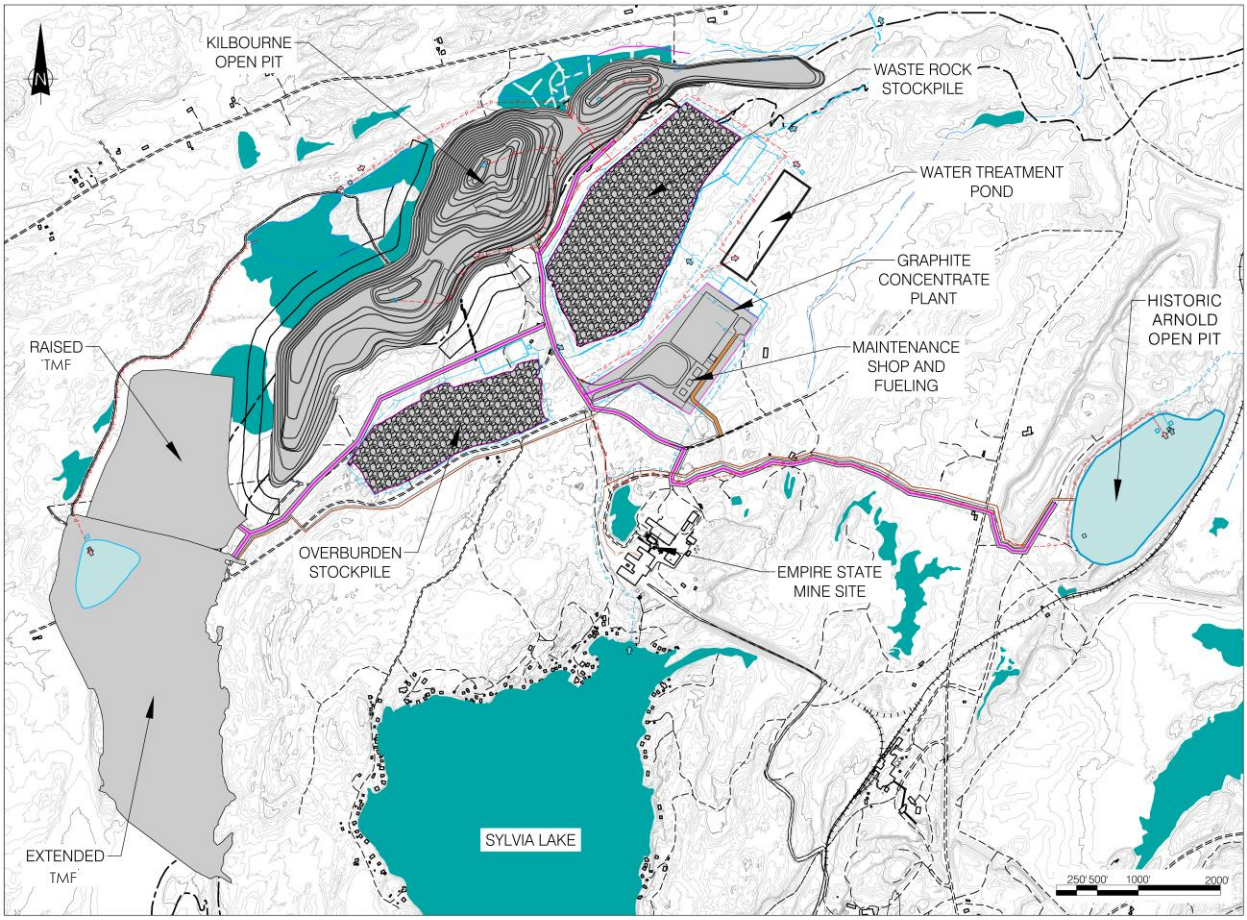


Figure 18-2: Kilbourne Graphite Study Site

18.2.2 Infrastructure

Site Access

The site is readily accessible by road and the entrance to the site is only 1.1 mi from New York Route 812. The public roads in the area are in very good condition.

A new road will be constructed to provide access to the Concentrate Plant site for on-highway trucks and light vehicles. This road will begin on Sylvia Lake public road, near the ESM entrance, and extend approximately 1.0 mi along the southeast edge of the Concentrate Plant site.

Approximately 3.7 mi of haulage roads will be required to connect the open pit to the Concentrate Plant site. These roads will be a combination of new roads and upgrades to existing roads that currently provide access to the ESM tailings storage area.

A new road of approximately 3.1 mi will be constructed to provide access to the extended and raised TMF areas.

Care has been taken to ensure that on-highway traffic does not interact with off-highway vehicles. At no time do production haulage trucks travel on roadways that are directly accessible by on-highway vehicles.

It should be noted that the Extended TMF will result in the closure of the western access to Sylvia Lake Road from California Road. Members of the public who require access to Sylvia Lake via West Shore Road will enter from the east along Sylvia Lake Road. This change in access is not expected to be problematic, as the eastern approach via Sylvia Lake Road is well-maintained, already used by local Sylvia Lake residential traffic, and provides a safe and efficient alternative route. The rerouting has been reviewed to ensure minimal disruption to public access, and signage will be installed to clearly direct traffic to the new access point.

Fuel Supply

A diesel tank and dispensing system will be constructed in the southwest portion of the Concentrate Plant site adjacent to the mobile maintenance shop. This will provide easy access to fuel for haulage trucks while fuel will be delivered by truck to in-pit equipment. The rear of the fueling station is accessible from the on-highway site access road allowing for fuel deliveries to site without any chance of interaction between on-highway and off-highway vehicles.

Initially, a single 5,300-gallon (20 k liters) diesel fuel tank with both splash fill and fast-fill dispensing will be installed. A second identical tank will be added in Year 5, bringing the capacity up to 10,600 gallons (40 k liters) to accommodate increased demand.

The Project has included a concrete pad sized to accommodate both tanks, with bollards to protect the installation from mobile equipment.

Mobile Equipment Maintenance Shop

The mobile maintenance shop will be a prefabricated steel structure with a concrete floor, measuring 105 ft by 70 ft with a ceiling height of 40 ft. It will have three bays for large equipment, each with a 16 ft by 30 ft roll up door, and a single light vehicle bay, with a 16 ft by 18 ft door, to service smaller equipment and light vehicles.

Each heavy equipment bay will be equipped with a 7.5-ton overhead crane.

Space within this shop will be allocated for both industrial mechanics and electrical personnel to store tools and equipment as well as to perform maintenance and repairs.

It is anticipated that on-road light vehicles such as pickup trucks and buses will be serviced and maintained off-site by existing local service providers.

Wash Bay

A mobile equipment wash bay will be situated beside the maintenance shop.

It will be housed in a fabric structure measuring 70 ft by 60 ft. An allowance has been included in the capital estimate for equipment including pressure washers and water treatment.

Waste Rock and Overburden Stockpiles

Mining of the Kilbourne open pit will generate 30.8 M tons of waste rock and 6.4 M tons of overburden material.

The waste rock stockpile will be located between the Concentrate Plant site and the Kilbourne open pit. With a footprint of just over 65 acres, this stockpile will have a maximum height of 240 ft and contain 354 M ft^3 of material.

The overburden stockpile will be located south of the Kilbourne open pit and west of the Concentrate Plant site. This pile will have a footprint of 27 acres, a maximum height of 100 ft and a total volume of 71 M ft^3.

The material contained in these piles is currently assumed to be non-acid generating, therefore, no membranes are required to contain runoff water. A perimeter ditch will collect drainage/runoff water and direct it into a pond dedicated for each pile. Water from these ponds will be pumped and/or flow by gravity to a central collection pond from which it will be released to the environment. Water quality monitoring has been included, but no formal treatment system is deemed necessary.

Fire Protection

The Project includes a 300 hp fire pump, 1 km of buried piping and 20 dry-barrel fire hydrants to provide fire protection for the Concentrate Plant site, including the mobile equipment shop and fuelling station. Water for this system will be sourced from the existing fire protection reservoir fed directly from the existing process water pumping station at Sylvia Lake.

Truck Scale

A drive-on truck scale with a 70 ft by 11 ft raised deck and a capacity of 80,000 lb is included to allow for weighing of material deliveries as well as concentrate shipments. This scale will be located adjacent to the on-highway access road and will not be accessible by off-highway haulage trucks.

18.2.3 Power Supply and Distribution

The electrical power requirements for the open pit mine and related infrastructure will be minimal in comparison to those of the graphite processing facilities, which is anticipated to be approximately 3 MVA whereas the graphite Concentrate and Micronization plants will require between 9–11 MVA totaling approximately 12–15 MVA.

Based on the estimated power requirements for the Kilbourne Graphite Study and the condition of the existing substation, a new 15 MVA ONAN/ONAF transformer with related equipment will be added to the existing substation, along with necessary supporting infrastructure. It is assumed that the existing overhead service line to the ESM site has sufficient capacity.

Due to space constraints and limited shutdown windows, a new skid-mounted substation - including breakers, transformer, and e-house - is proposed at the existing ESM substation site. The high bay will require modifications to integrate the additional infrastructure, including cabling, isolating devices (disconnect or recloser depending on utility provider requirements), insulators, and other necessary components. Consultation with the utility provider will be required as the Project progresses to validate substation requirements and to verify the acceptance of selected electrical equipment in the next phase of study. New York Power Authority has indicated its willingness to meet the required power needs, and discussions with respect to a purchase power agreement are ongoing.

The self-contained substation has been selected because it reduces the additional space requirements due to its prefabricated nature and minimizes the duration of the connection and commissioning outage.

The 5 kV transmission lines from this new transformer to the Concentrate and Micronization plants will be buried in a conduit until a safe distance is reached; beyond that point, pole-mounted lines will carry power for the remaining 2,600 ft required. On-site power distribution will be achieved with overhead lines as required.

Finally, the power required for pumping at the Historic Arnold Pit site, both for initial dewatering and during tailings deposition, will be met by existing electrical infrastructure that is currently still energized.

18.2.4 Utilities and Services

Process Water

Process water for the graphite Concentrate Plant will be supplied primarily from the treatment pond, which is in turn fed by contact water from the open pit dewatering pumps and the TMF decant water pumps. In situations where adequate water volumes are not available from this pond, such as during first-fills, water will be sourced from Sylvia Lake via the existing pumphouse.

Potable Water

Bottled water will be used to meet the potable water requirements of the Kilbourne Site, including the open pit, Concentrate Plant, and maintenance facility. Personnel located at the ESM site will consume water from the existing potable water system that feeds the plant site.

Wastewater

Wastewater (sewage and domestic/grey water) at the Kilbourne Site will be collected in a 30,000-gallon, buried sewage tank. This tank will be pumped out and waste material disposed of off-site by local service provider.

18.2.5 Water Management

The Kilbourne Site is situated primarily within the Turnpike Creek–Oswegatchie River watershed (HUC 041503020801), which encompasses approximately 75 km². Sylvia Lake, located entirely within this watershed, serves as the principal surface water source for the Project. While the majority of the site falls within this basin, the Extended TMF is located in the adjacent Sawyer Creek watershed (HUC 041503020701), which spans approximately 82 km².

Despite crossing watershed boundaries, all water flows associated with the Extended TMF will be integrated into a unified management strategy centered on the Turnpike Creek system.

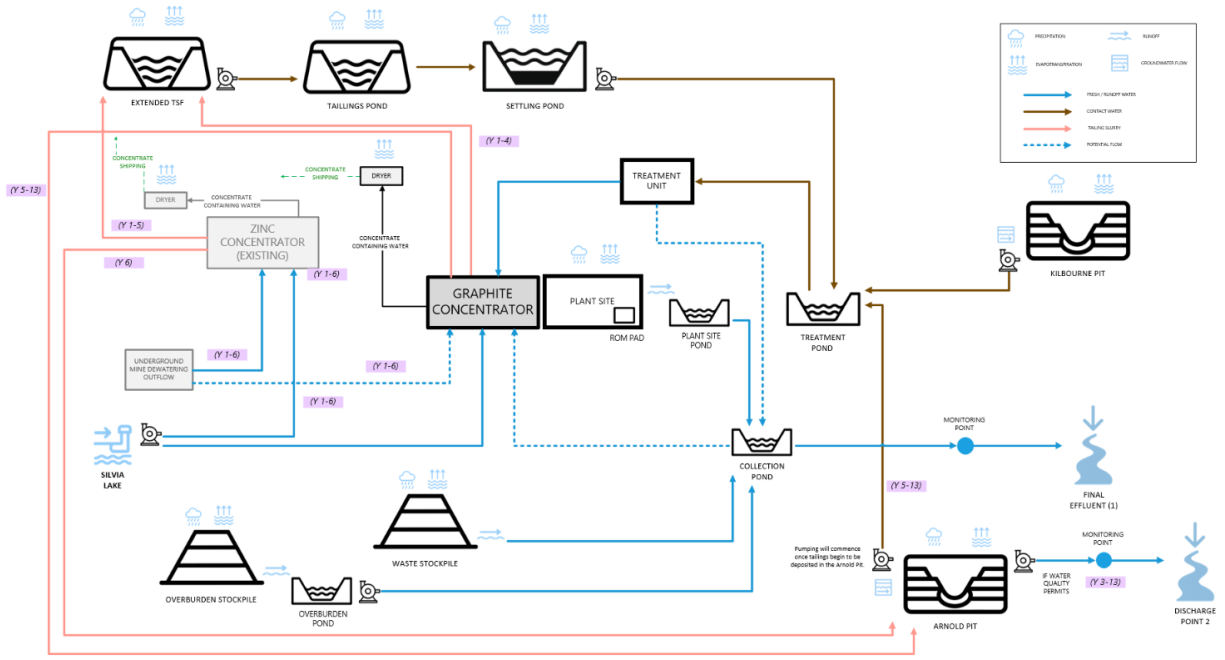
The regional hydrology includes multiple headwater tributaries and a central drainage channel—Turnpike Creek—that collectively shape the site's surface runoff dynamics and inform the overall water balance framework.

18.2.5.1 Water Management Strategy

A preliminary water balance was established, incorporating key inputs such as precipitation, runoff, and groundwater inflow, alongside operational water demands and discharge pathways (Rodriguez et al., 2025). Drawing from national best practices, the strategy promotes modular infrastructure design, contingency planning, and treatment based on sedimentation and pH adjustment, in anticipation of future regulatory and environmental requirements.

Water handling within the graphite concentrator circuit is organized around a closed-loop system designed to minimize freshwater consumption and control discharges. Freshwater is primarily sourced from Sylvia Lake, with supplementary inputs from underground dewatering, the collection pond, and treated water reservoirs. Contact water from processing and tailings storage is routed through a sequence of clarification and treatment ponds before reuse or discharge. In parallel, clean runoff from stockpiles and undisturbed areas is collected separately and monitored prior to any environmental release, ensuring hydraulic segregation and reduced treatment volumes.

Operational phases drive progressive adjustments to the site's water flow configuration. Initially, all tailings are directed to the Extended TMF. As operations advance, the Historic Arnold Pit is dewatered and later repurposed for tailings deposition, altering water routing and treatment flows accordingly. Continuous monitoring and iterative updates to the water balance support an adaptive system that evolves with the Project and safeguards compliance across its life cycle. Figure 18-3 presents the Project general waterflow diagram.



Note: Parentheses indicate years of operation for non-permanent facilities (e.g., Y3–5).

Figure 18-3: General water flow diagram

18.2.5.2 Water Balance

The water balance developed for the Kilbourne Project provides a preliminary framework for assessing site-wide hydrology under varying operational and climatic conditions. It quantifies key inputs and outputs—including precipitation, runoff, groundwater inflows, evapotranspiration, process water demand, and discharges—supporting evaluation of storage requirements and flow routing. Sylvia Lake has been identified as the primary freshwater source; however, no availability assessment has yet been completed, and supply adequacy is assumed pending future validation.

The model was developed in a spreadsheet platform using surface-specific runoff coefficients and historical climate data under normal, dry, and wet scenarios. It incorporates water demand from the zinc and graphite concentrators, pit dewatering, and seepage, and adapts to changing infrastructure across mine phases. Precipitation recurrence intervals (1/5- and 1/10-year monthly events) were applied to size treatment facilities, designed for wet-year conditions with a 1/10-year recurrence.

Precipitation data were sourced from the Gouverneur 3 NW station, with rainfall, snowmelt, and evapotranspiration derived using the USGS Thornthwaite method and adjusted by 15% to reflect projected climate change. Surface runoff was estimated from tailored coefficients for each sub-watershed, while groundwater inflows were calculated for the Kilbourne and Historic Arnold pits based on historical data and hydrogeologic conditions. Process water demands assume full-capacity concentrator operation, with underground dewatering included as a supplementary source.

The model was prepared for four operational periods reflecting mine development (Rodriguez et al., 2025):

- **Period 1:** Zinc and graphite tailings to Extended TMF; Historic Arnold Pit dewatering begins in Year 4.
- **Period 2:** Graphite tailings to Historic Arnold Pit; Zinc tailings remain in Extended TMF; Historic Arnold Pit water pumped to treatment pond.
- **Period 3:** Both tailings streams to Historic Arnold Pit; Extended TMF inactive.
- **Period 4:** Zinc operations cease; Only graphite tailings to Historic Arnold Pit, reducing overall water volume.

Figure 18-4 presents the water balance flow diagram.

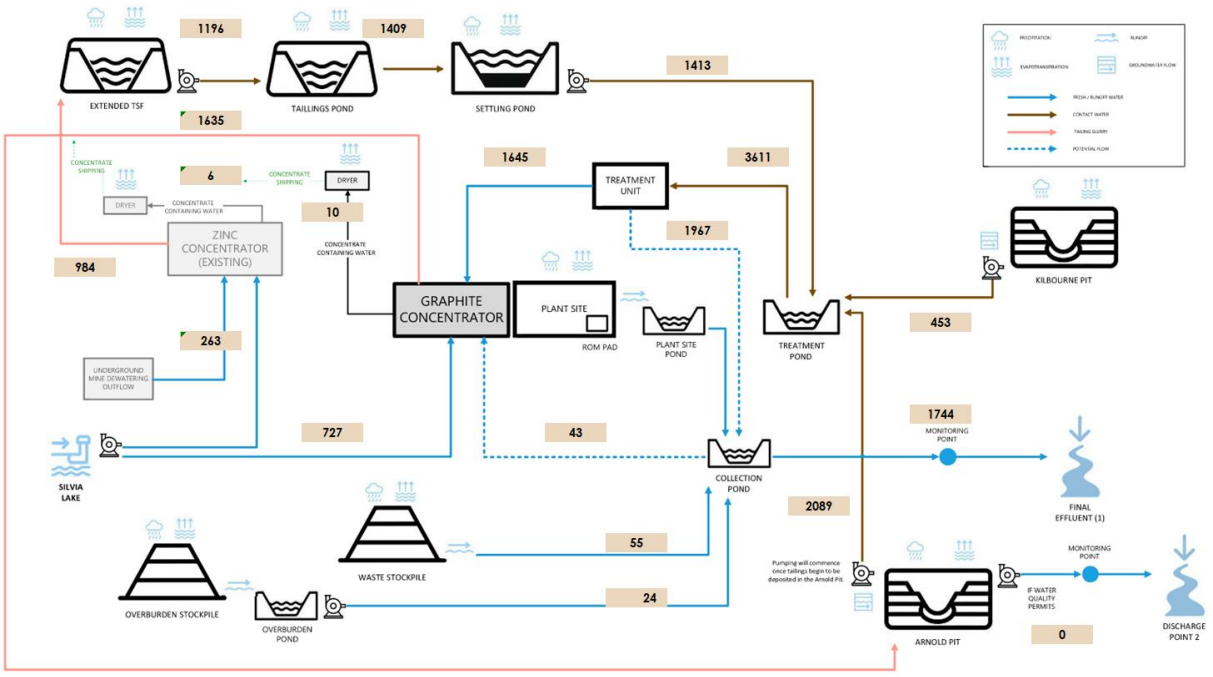


Figure 18-4: Water balance (flow rates in GPM)

18.2.5.3 Storm Water Management

The stormwater management strategy for the Kilbourne Project is designed to minimize off-site discharge of contact water, control runoff volumes and peak flows, mitigate erosion and sediment transport, and ensure compliance with regulatory water quality standards. Key design principles include the segregation of clean and potentially impacted flows, the use of gravity-driven conveyance where feasible, and the integration of low-impact development practices. Drainage areas were delineated based on topographic and infrastructure mapping, distinguishing sub-catchments by land use and flow direction to support targeted management. Future design phases will incorporate erosion control measures, such as stabilization, sediment basins, and protective structures, particularly around disturbed areas like the Extended TMF (Rodriguez et al., 2025).

18.2.6 Tailings Management Facilities

This Item has been prepared based on the following supporting documents related to the ESM TMF:

- Operations, Maintenance, and Surveillance (OMS) Manual for the ESM TSF (Tierra Group, 2023a);
- Geotechnical investigation of the TSF (Dietzko, 2022);

- Slope stability modeling results and tailings storage capacity calculations; Capacity Evaluation Report ISSD 20230804. (Tierra Group, 2023b);
- Dam Breach Study. Analysis, methods, and results for dam breach scenarios at the ESM TSF (Tierra Group, 2022a);
- Model assumptions, methods, analysis results, and conclusions for slope stability and seismic analyses of the TSF dams (Tierra Group, 2022b);
- Trade-off study, including evaluation criteria and scoring for different tailings management alternatives (Tierra Group, 2024);
- Empire State Mine Seismic Hazard Analysis (Tierra Group, 2022c);
- Empire State Mines 2024 NI 43–101 Technical Report Update (Taylor et al., 2024).

18.2.6.1 Subsurface Conditions at the Location of TMF

No site-specific geotechnical inspections have been conducted outside the existing TMF area (i.e., within the Extended TMF and Historic Arnold Pit area), the same subsurface conditions as those of the existing TMF have been assumed for the preliminary high-level assessment. Material properties were assigned based on laboratory data and analyses described in ESM Geotechnical Report (Dietzko, 2022).

18.2.6.2 Site Seismicity

Seismic hazard was assessed using both the CENA and NGA-West2 ground motion models for return periods ranging from 475 to 10,000 years. Results were consistent; the CENA model was selected for its regional relevance and conservative PGA estimates.

A peak ground acceleration (PGA) of 0.34 g, corresponding to the 10,000-year return period, was adopted for design. Although classified as "No Hazard" under NYSDEC criteria, the TMF is considered "Extreme Consequence" under CDA guidelines due to potential downstream impacts.

18.2.6.3 Graphite Tailings Characteristics

The key physical and geochemical characteristics of the graphite tailings that inform material handling, storage design, and environmental performance assessments are summarized in Table 18-3. These values are based on current test results and operational assumptions, with additional data pending from ongoing characterization studies.

Table 18-3: Tailings characteristics

Parameter	Value	Reference
Specific Gravity (SG)	2.79	Titan Mining Corp
Bulk Density	136 pcf	
Recovery Rate	~3–4%	Titan Mining Corp
Mineralogy Note	Comparable to ore	Titan Mining Corp
Preliminary Tests	NAG/ABA (blended tails)	Titan Mining Corp

18.2.6.4 TMF Development Strategy

Several development options for the TMF have been evaluated to identify a long-term tailings management strategy for the Kilbourne Graphite Study and future Zinc Operation. This assessment considered the presence of existing zinc tailings over the Kilbourne deposit and the limited capacity of the current tailings storage facilities to accommodate graphite tailings. The preferred approach over the life of the mine includes a partial southward expansion of the TMF, raising the existing TMF near the South Dam, and backfilling both the Kilbourne Pit and the Historic Arnold Pit, as illustrated in Figure 18-2.

There are several sources of tailings that need to be managed over the LOM, including both existing zinc tailings to be relocated and new tailings produced from ongoing Zinc Operation and Kilbourne Graphite Study.

- **Existing Tailings** are divided into two categories:
  - *Existing Tailings within Pit*: Material currently located within the footprint of the Kilbourne Pit.
  - *Existing Tailings within Embankment*: Material stored within the containment dikes of the existing tailings facility.

  These two sources together make up the Total Existing Tailings to be moved, totaling 147.9 M ft^3 over the LOM.
- **Future Tailings** are generated by:
  - *Graphite Study:* Tailings produced from the new graphite operations.
  - *Zinc Operation:* Tailings generated from the existing zinc operations.

  Together, these amount to 285.5 M ft^3 from graphite and 51.2 M ft^3 from zinc over the LOM.

Table 18-4 presents the total tailings to be managed over the LOM, including the sum of all existing and future tailings to be handled each year. The LOM total tailings is 484.6 M tonnes, with annual quantities varying depending on the mining and relocation schedule. To reduce the amount of existing tailings relocation, a containment dike system is proposed around the Kilbourne Pit to support pit development. Associated works for the construction of the Kilbourne Pit are outlined in the following sections. The overall TMF development layout is presented in Figure 18-2.

Table 18-4: Volume of tailings to be relocated and produced tailings

Description		Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Total
Existing Tailings within the footprint of the Kilbourne Pit and the containment dikes	Existing Tailings within Pit (M ft³)	-	-	16.7	28.0	33.1	4.0	-	-	-	-	-	-	-	-	**81.8**
	Existing Tailings within Embankment (M ft³)	-	-	4.3	4.3	16.5	16.5	-	-	-	-	-	-	-	-	**41.5**
	Total Existing Tailings[1] (M ft³)	**-**	**-**	**25.1**	**38.7**	**59.5**	**24.6**	-	-	-	-	-	-	-	-	**147.9**
Produced Tailings	Graphite Study (M ft³)	-	17.7	20.1	19.9	19.3	21.8	24.2	26.7	26.6	26.5	23.2	20.6	18.9	20.0	**285.5**
	Zinc Operation (M ft³)	9.1	9.1	9.1	9.1	9.1	5.7	-	-	-	-	-	-	-	-	**51.2**
Total Tailings to Manage (M ft³)		**9.1**	**26.8**	**54.4**	**67.8**	**87.9**	**52.1**	**24.2**	**26.7**	**26.6**	**26.5**	**23.2**	**20.6**	**18.9**	**20.0**	**484.6**

(1) An expansion factor of 1.2 is considered for the total existing tailings to be relocated.

18.2.6.5 Containment Dike for Existing Tailings

To optimize the volume of existing tailings that need to be relocated, setbacks to the graphite open pit and/or containment structures have been proposed. These measures aim at supporting the mining of Kilbourne graphite deposits under the current TMF while minimizing the volume of tailings that must be displaced and relocated.

The consequence classification of the proposed containment dikes is estimated as “Class "B" or "Intermediate Hazard" dam based on the New York State Department of Environmental Conservation (NYSDEC) and “High” based on the Canadian Dam Association (CDA). The assumption is that in the event of a breach, the runout would be retained within the open pit.

The liquefaction analysis using results of Standard Penetration Tests (SPT) and Cone Penetration Tests (CPT) shows that the tailings are susceptible to liquefaction. The foundation soil (sandy silts and clays) is not liquefiable or marginally liquefiable below an elevation of approximately 590 ft to 600 ft.

The factors of safety considered for the analysis are summarized in Table 18-5.

Table 18-5: Required factor of safety for stability analysis

Loading condition	Minimum Factor of Safety	Reference
Static (Long-term)	1.5	CDA, 2014
Seismic	1.0	
Post-Seismic	1.2	

The dike is to be constructed in two stages using waste rock:

- **Stage 1**: Excavation to bedrock with side slopes no steeper than 2H:1V. The lower section is constructed to the elevation of foundation soil, with a 1H:1V upstream slope, 2H:1V downstream slope, and 20 ft crest width. The upstream face is backfilled with overburden.
- **Stage 2**: The upper section extends to the top elevation of the tailings, with 2H:1V slopes on both sides and a 20 ft crest. Tailings are used as upstream backfill.

Stability analysis confirms compliance with required factors of safety under all loading conditions. Preliminary runout modeling indicates a potential runout distance of 40–44 m under failure conditions.

18.2.6.6 Raised Tailings Management Facility

To optimize storage capacity and reduce haulage distances, the existing TMF will be raised near the South Dam. This area will serve as the relocation site for the consolidated tailings currently occupying the Kilbourne Pit footprint, which must be cleared to enable open pit development.

The preliminary design parameters for the raised storage area are as follows:

- **Tailings storage elevation:** 650 ft;
- **Storage capacity**: 28.6 million ft^3;
- **Setback distance**: 150 ft from perimeter dikes, based on preliminary runout analysis;
- **Side slopes:** 20H:1V.

Excavated tailings will be placed in this area without rehydration. The geometry is designed to maintain long-term stability while adhering to setback requirements and minimizing downstream risk.

Figure 18-5 shows the location and configuration of the raised TMF.

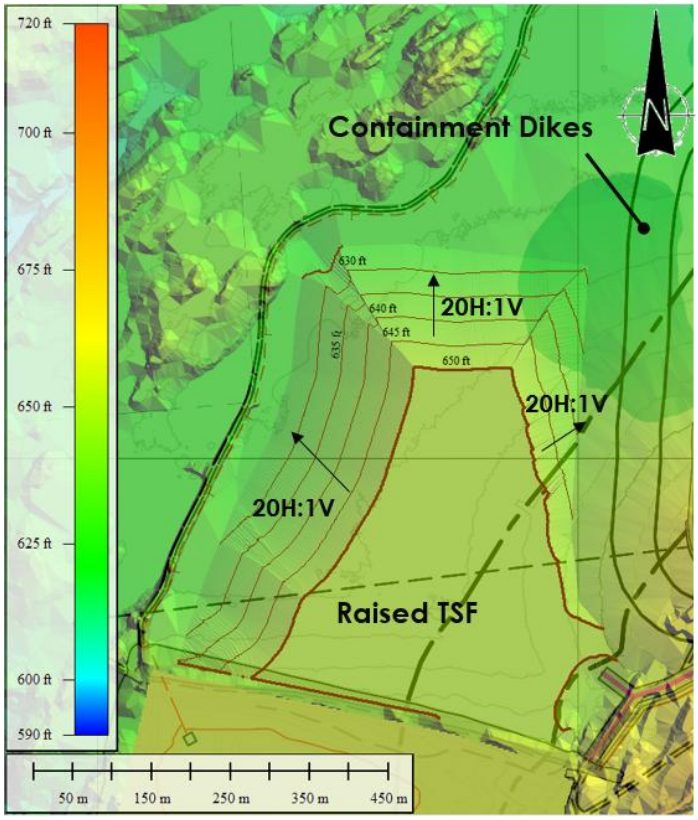


Figure 18-5: Raised TMF layout

18.2.6.7 Extended Tailings Management Facility

To accommodate future graphite and zinc tailings, land located downstream of the South Dam has been identified as available for acquisition, enabling a potential extension of the existing TMF footprint (Figure 18-2).

In the absence of site-specific inundation studies for the proposed embankments, the preliminary consequence classification has been assumed to match that of the existing containment dikes.

Preliminary assessments at the PEA level have considered storage capacity, slope stability, and associated geotechnical risks for the Kilbourne Graphite Study within this potential extension area.

The proposed Extended TMF has the following basis and assumptions:

- Storage capacity elevation at 657 ft;
- Dam crest elevation at 665 ft providing a freeboard of 8 ft. The freeboard depth is based on 2023 OMS Manual, 6 ft IDF + 2 ft dry freeboard) as per NYSDEC 1989;
- Dam crest width: 30 ft;
- Dam upstream slope: 2H:1V;
- Dam downstream slope: 3H:1V;
- Bedrock elevation is assumed based on the provided geological model (Wireframe Geological Block Model);
- Foundation soil is sandy silts and clays, similar to the existing south dam (Geotechnical Analyses, ESM Tailings Storage Facility, 2023);
- Groundwater assumed to be from the edge of the model, through the middle of the dam upstream face to the toe.

The extended TMF is planned to be constructed in multiple stages (Table 18-6).

Table 18-6: Details of staged Extended TMF

Stage	TMF Elevation	Dam Crest Elevation	Storage Capacity Volume	Dam Fill Volume
	ft	ft	M ft³	M ft³
Stage 1	615	623	34.2	4
Stage 2	632	640	91.4	7
Stage 3	657	665	70.3	23
Total	**657**	**665**	**194.9**	**34**

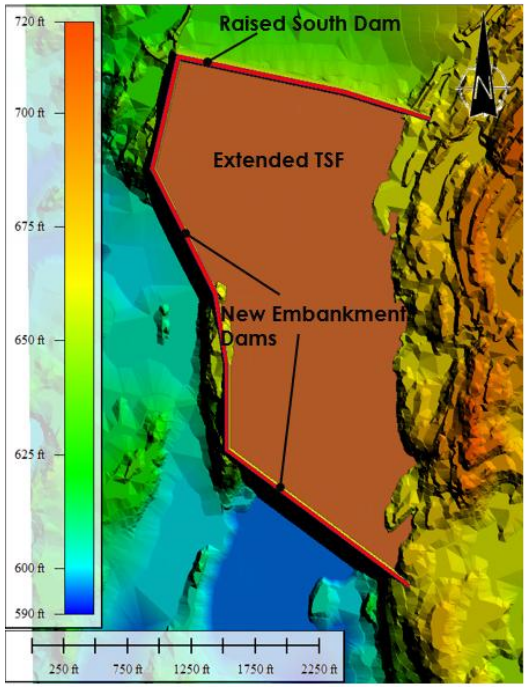


Figure 18-6: Proposed Extended TMF layout

18.2.6.8 Historic Arnold Pit Backfilling

The Historic Arnold Pit has been identified as a potential area for additional tailings deposition. Existing haul roads provide access to the site; however, some segments require regrading and possible filling to restore suitability for haul traffic. Alternatively, tailings slurry transport via pipeline may be considered.

The pit is currently flooded and will require full dewatering prior to deposition.

To maximize the storage of the Historic Arnold Pit, containment dikes around the Historic Arnold Pit will be constructed (Figure 18-7). For the proposed dike, the dam crest elevation was considered to be at an elevation of 700 ft. The freeboard of 8 ft considered (Based on 2023 OMS Manual, 6 ft IDF and 2 ft dry freeboard as per NYSDEC 1989). The following assumptions are considered for the stability analysis:

- Due to the lack of geotechnical information around the Historic Arnold Pit, the foundation soil beneath the proposed containment dike was assumed to be 50 ft thick (according to previous drilling, an overburden thickness of approximately 9 ft to 50 ft was observed near Historic Arnold Pit).
- Foundation soil is assumed to be sandy silts and clays, similar to the existing south dam (Geotechnical Analyses, ESM Tailings Storage Facility, 2023).

- Soil parameters are assumed to be consistent with previous analysis.
- Containment dike geometry:
  - Height varies;
  - Crest width 30 ft;
  - Upstream slope 2H:1V;
  - Downstream slope 3H:1V.
- A good drainage system was assumed to be present in the containment dike to lower the water table within the dike.
- The consequence classification of the proposed dike is considered as high, and 1:2475 year event was considered for the seismic slope stability analysis.

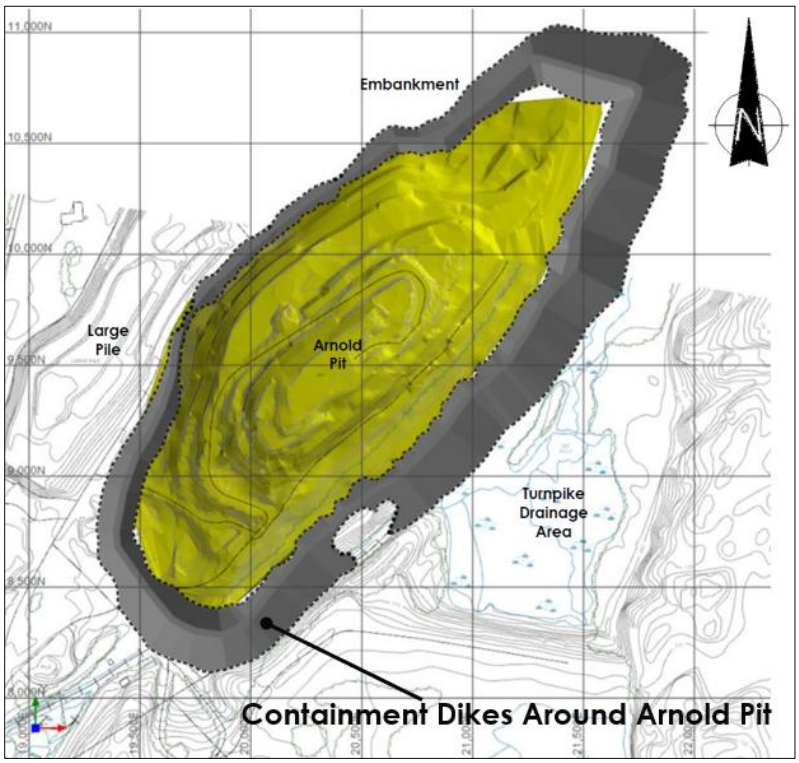


Figure 18-7: Proposed containment dike around the Historic Arnold Pit

18.2.6.9 Tailings Deposition and Relocation Strategy

Based on the provided TMF structures, the tailings deposition sequence will be according to Table 18-7 to accommodate the existing tailings (within the footprint of Kilbourne Pit), and the produced Zinc to year 6 LOM and Graphite tailings to 13 years LOM.

Table 18-7: Tailings management planning during LOM

Description	Unit	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Total
Kilbourne Pit Back-fill	M ft³	-	-	-	-	29.7	-	-	-	-	-	-	-	-	-	**29.7**
Raised TMF	M ft³	-	-	25.1	3.5	0.0	-	-	-	-	-	-	-	-	-	**28.6**
Extended TMF	M ft³	9.1	26.8	29.2	64.3	59.8	5.7	-	-	-	-	-	-	-	-	**194.9**
Historic Arnold Pit - As is	M ft³	-	-	-	-	28.1	46.4	24.2	26.7	26.6	16.9	-	-	-	-	**168.8**
Historic Arnold Pit - with Extension	M ft³	-	-	-	-	-	-	-	-	-	9.6	23.2	20.6	18.9	20.0	**92.3**
Total Storage Capacity	**M ft³**	**9.1**	**26.8**	**54.4**	**67.8**	**117.6**	**52.1**	**24.2**	**26.7**	**26.6**	**26.5**	**23.2**	**20.6**	**18.9**	**20.0**	**514.3**

18.2.7 Kilbourne Off-site Infrastructure

It is expected that off-site high-voltage transmission lines are of suitable capacity and in a good state of repair to provide power to the site for the duration of the mine life. Further review and analysis of this system is recommended for future stages of the Project.

With its proximity to several well-established communities, it is anticipated that local vendors and service providers will be able to deliver services and materials as required to support the additional activities at the ESM site.

18.2.8 Concentrate Plant

The infrastructure to support the concentrator plant operation includes:

- Plant office;
- Workshop;
- Laboratory;
- Reagent storage;
- Change room and shower facilities.

18.2.8.1 Plant Office

A plant office will accommodate up to 20 people in a combination of single offices and an open area arrangement. The plant office will also provide bathroom facilities, a reception area, kitchenette, and boardroom for up to 15 people.

18.2.8.2 Workshop

The workshop will support the maintenance and repair of mechanical, electrical, and mobile equipment of the concentrator. The workshop will be in a separate building adjacent to the concentrator. Installation of the workshop will be completed as part of the overall civil and earthworks package.

The workshop features the following:

- Heavy-duty workbenches and tool storage systems;
- Hoists for lifting equipment like pumps or motors;
- Welding, cutting, and fabrication tools;
- Lubrication and oil dispensing systems;
- Spare parts storage;
- Workstations for maintenance management;
- Offices and washrooms.

18.2.8.3 Laboratory

The laboratory will be part of the concentrator building. It will consist of two separate areas. One area will contain sample preparation equipment and a flotation setup to support ongoing plant optimization. The second area will be dedicated to analytical equipment to support the generation of certificates and analysis and support plant optimization activities.

18.2.8.4 Reagent Storage

The reagent storage facility is located adjacent to the concentrator and will hold the reagents required for the processing plant, which are diesel as the collector and MIBC as the frother. No other reagent will be required since no pH adjustment is performed and no thickeners are included in the process design.

18.2.8.5 Change Room and Shower Facilities

The change room and shower facilities are connected to the concentrator building and will be sized to accommodate the work force at the plant.

18.2.9 Micronization Plant

The Micronization Plant will be collocated with the concentrator. The selected micronization technology consists of a turnkey system and only requires electricity and compressed air. A separate dust collection system at the feed and discharge ends only requires electricity.

The air swept classifier milling systems used for micronization are low maintenance, which consists primarily of daily, weekly, and monthly checks and lubrication. Every 6–12 months comprehensive service is performed to replace wear parts like hammers and classifier vanes as needed. Due to the small space requirement for spare parts, the concentrator warehouse will stock all micronization equipment spares.

The only consumables required by the Micronization Plant are 1-tonne bulk bags for the finished product. The continuous feed to the micronization lines will be achieved with screw feeders from the dry concentrate bins. Reusable bulk containers will be used for surge capacity. Skids with bulk bags will be stored at the concentrator warehouse.

Change room and shower facilities are shared with the concentrator given that only two people are required to operate each of the two micronization lines. Further, all analyses are performed by the concentrator laboratory.

18.2.10 Secondary Transformation Site

The Purification Plant and CSPG Plant are akin to a chemical factory and, therefore, can be situated independently from the Kilbourne Graphite Study's Mineral Resources and Mineral Reserves.

While the location of the Purification Plant and CSPG Plant have not been selected, their development will require adequate transportation infrastructure and access to international markets. For the purposes of the PEA, it is assumed that both plants will be located at a site in a chemical processing hub or in an industrial site within New York State. Titan is currently evaluating a number of industrial sites with existing infrastructure, including those in close proximity to the mine.

The Purification Plant and CSPG Plant are planned to be constructed on flat topography within a prime chemical industrial estate. Locating the plants within an industrial estate provides several strategic advantages, including access to developed plots, established bulk infrastructure and supply services, and cost-effective solutions for additional service requirements.

To support the operational requirements of both plants, a range of essential infrastructure and bulk services will be required. These include electricity, water, natural gas, stormwater drainage, industrial effluent removal, waste management facilities, access roads, and internet connectivity.

Additional infrastructure, not generally supplied as part of conventional industrial utility services, will need to be developed. These include vehicle and trucking parking bays, internal roads, stepdown transformers from bulk power supply, and integrated systems for internal power and water distribution. Furthermore, on-site utilities will include water purification, steam generation units, compressed air systems, and specialized facilities for handling and storing HF, given its hazardous nature and regulatory requirements.

19. Market Studies and Contracts

19.1 Zinc

19.1.1 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: Taylor et al., 2024

Table 19-2: International zinc smelters (partial list)

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: Taylor et al., 2024

19.1.2 Zinc Price and Concentrate Terms

The zinc price assumptions used in this Technical Report were based on the London Metal Exchange (LME) average cash-settlement prices. The LME is recognized as the global benchmark for zinc pricing, and assumptions reflected the prevailing market conditions at the time.

Although efforts have been made 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. Treatment charges are typically based on global benchmark terms.

The zinc concentrates produced by the Company at ESM are 100% sold to Glencore Ltd. pursuant to an off-take agreement between the Company and Glencore Ltd. dated February 26, 2018. In Titan's view, the terms, rates and charges provided in the off-take agreement are within industry norms.

The QP has reviewed the price assumptions and off-take agreement and deemed them appropriate for use in this Technical Report.

19.2 Graphite

There are no standard, spot or future prices for graphite. Market and pricing information was sourced from a commissioned study for the Kilbourne Graphite Study by an independent graphite consultancy, Lone Star Tech Minerals (LSTM), based in Texas, USA (LSTM, 2025). The QP has reviewed these studies and analyses and the results support the assumptions used in this Technical Report.

19.2.1 Market Information

Owing to the unique combination of metallurgical and chemical properties, both traditional and advanced graphite powders are used across a wide range of applications. While a comprehensive list of markets and uses encompasses numerous industries and end-use scenarios, the potential markets and applications for Kilbourne natural flake graphite products are extensive.

Graphite products are categorized into four distinct groups based on their processing and end-use application:

- **Natural Flake Concentrate:** This is the raw, mined product that has undergone initial flotation and concentration. It is a commodity primarily differentiated by its flake size (mesh) and purity (LOI). It serves as the feedstock for both traditional industrial markets and advanced downstream processing and is hence referred to as upstream grade. Target Upstream

market groups include thermal management, lubricants and dispersions, engineered products, plastics-polymers-rubber, and energy storage.

- **Standard (STD) Purity Micronized Grades:** Micronized Natural Flake Graphite, commonly referred to as "Micronized NFG," is the product of the Micronization Plant. This is a value-added product created by post processing of the upstream grade and is hence referred to as downstream grade. Market groups include energy storage, aerospace / specialized industrial and semi-conductor.
- **High Purity Micronized Grades:** Purified Micronized Graphite (PMG), commonly referred to as "PMG," is the product of the Purification Plant. This grade is obtained through additional purification of micronized NFG. PMG targets advanced downstream applications, including high purity micronized flake graphite for advanced applications as well as primary batteries and cathode conductive additives for secondary battery applications for defense, energy storage, aerospace, and semiconductor.
- **High-Purity Anode Materials:** Coated Spherical Purified Graphite (CSPG)s is the most highly refined product, produced by purifying, spheronizing, and often coating natural flake graphite. CSPG is primarily used in secondary battery applications, including aerospace, industrial, consumer goods, and U.S. Department of War (DOW) sectors, and is more extensively applied globally in stationary battery systems.

The Kilbourne natural graphite signature is classified as a fine to medium natural flake deposit (≤ 100 mesh) and is production ready for both upstream and downstream grades. Kilbourne graphite grades include **Natural Flake Concentrate**, **STD Purity and High Purity Micronized Graphite** products with planned production of up to 15,000 Mt **CSPG Anode** graphite products beginning 5-7 years after commencement of full production based on market conditions.

Located in New York state, the Project is designed to become a local supplier of graphite product that is production-ready for both upstream and downstream grades. The Project is currently undertaking the development of a demonstration plant on the existing property to qualify the product for the domestic market.

19.2.2 Study Price & Sales Terms

Pricing assumptions for the Graphite Study are based on North American regional prices, which are higher than global average sales prices (ASP) due to supply chain premiums for non-Chinese material. The Graphite Study takes into account these premiums for all planned upstream and downstream grades. The FOB Port pricing data is sourced from quarterly reports developed by Lone Star Tech Minerals-USA, based on data points from a wide range of contacts across various markets.

Table 19-3 to Table 19-5 summarize the graphite product categories, purities, and indicative pricing. Table 19-3 presents Natural Flake Concentrates (95.0%–96.0% Loss on Ignition (LOI)), Table 19-4 shows Standard Purity Micronized Grades (95.0% LOI), Table 19-5 outlines High Purity Micronized Grades (99.9% LOI) and Table 19-6 presents CSPG Anode Grades.

Pricing varies based on product type, particle size, and purity level.

All monetary amounts in this chapter are expressed in United States dollars, unless stated otherwise.

Table 19-3: Natural Flake Concentrate: The base target purity is 95.0% - 96.0% LOI

Product Description	ASTM Mesh Grade	Purity (LOI %)	Price Weight Factor (%)	Price ($/Mt)
-100 Mesh Concentrate	80% Min. Pass; -100 mesh	95.0% Min.	50	1,699
-150 Mesh Concentrate	80% Min. Pass; -150 mesh	95.0% Min.	25	1,477
+200 Mesh Concentrate	(80% Min. Ret;+200 mesh) / (100 x 200 mesh)	95.0% Min.	15	1,501
-200 Mesh Concentrate	60% Min. Pass; -200 mesh	95.0% Min.	10	1,311
Weighted Average Sale Price			**100**	**1,575**

Table 19-4: STD Purity Micronized Grades: A minimum purity of 95.0% LOI

Product Description	Micron Size (D90)	Purity (LOI %)	Price Weight Factor (%)	Price ($/Mt)
D90 – 45 Micron	45 µm	95.0% Min.	50	2,439
D90 – 15 Micron	15 µm	95.0% Min.	45	4,822
D90 – 5 Micron	5 µm	95.0% Min.	5	7,614
Weighted Average Sale Price			**100**	**3,770**

Table 19-5: High Purity Micronized Grades: A minimum purity of 99.9% LOI

Product Description	Micron Size (D90)	Purity (LOI %)	Price Weight Factor (%)	Price ($/Mt)
D90 – 45 Micron	45 µm	99.9% Min.	30	3,199
D90 – 15 Micron	15 µm	99.9% Min.	65	5,770
D90 – 5 Micron	5 µm	99.9% Min.	5	9,497
Weighted Average Sale Price			**100**	**5,185**

Table 19-6: CSPG Anode Grades: A minimum purity of 99.95% LOI

Product Description	Micron Size (D90)	Purity (LOI %)	Price Weight Factor (%)	Price ($/Mt)
D90 – 50 Micron	50 µm	99.95% Min.	35	9,930
D90 – 30 Micron	30 µm	99.95% Min.	40	11,152
D90 – 15 Micron	15 µm	99.95% Min.	25	13,028
Weighted Average Sale Price			**100**	**11,193**

Based on the expected product distribution, the weighted average sales price listed in Table 19-7 has been considered for the Project.

Table 19-7: Recommended study price

Product	Weighted Average Sale Price ($/Mt)
STD Purity Flake Grades Concentrate (95.0% LOI MIN)	1,575
STD Purity Micronized Flake Grades (95.0% LOI MIN)	3,770
High Purity Micronized Flake Grade (99.9% LOI MIN)	5,185
CSPG Anode Grades (99.95% LOI MIN)	11,193

The pricing assumptions for the Graphite Study aligns to a strategic analysis of the market, including its location, product quality, and the overall demand for a secure, North American supply.

- **Geopolitical Advantage:** Located in the United States, the Kilbourne Study offers a stable and secure supply chain, mitigating the risks associated with global supply chain dependencies.
- **Logistical Edge:** The Project's proximity to major industrial hubs in North America significantly reduces transportation costs and enhances supply chain reliability.
- **Stable Market Base:** The Project's initial product offerings will allow flexibility in meeting the demand across stable industrial, specialty and high-tech applications.
- **Market Expansion:** As the demand in high-growth critical markets increases in defense, aerospace, energy storage and battery applications, the Project's expected product evolution will allow it to capture the demand in graphite that is expected to follow.

19.2.3 Contracts

At this time, no sales agreements or contracts have been executed with vendors, contractors or manufactures.

20. Environmental Studies, Permitting and Social or Community Impact

20.1 Zinc

20.1.1 Environmental

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 TMF has been completed. In accordance with the Canadian Dam Association (CDA) Standards, both a Dam Breach Analysis (DBA) and a Seismic Hazard Analysis (SHA) have been completed. The Operation, Maintenance and Surveillance (OMS) Manual has been developed and published (Tierra Group, 2023a). The facility completed its last Dam Safety Review (DSR) on 15 October 2024 with the next review scheduled for April 2026. The TMF is classified as low-risk by the New York State Bureau of Flood Protection and Dam Safety and as "Low Hazard" by MSHA.

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.

20.1.2 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 the 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 the 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 May 2030
Storage	NYDEC Petroleum Bulk Storage	PBS#6-451770[2]	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	1425550087H	30 Jun 2026

Source: ESM 2025

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

(2) The NYSDEC Petroleum Bulk Storage Facility ID# is technically not a permit, but a legally required authorization to operate under the PBS program.

Tailings storage and management is discussed in detail in Item 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.

20.1.3 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 flows from #2 and #3 mines, plus flow from the Vanderbilt Minerals LLC - Gouverneur Minerals Division's 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.

20.1.4 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 $2,701,000 surety bond. With the addition of the Kilbourne Graphite Project, this surety bond will increase to cover the cost of the additional reclamation required, as described in Item 20.2.2.

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. 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 buildings 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 it can be demonstrated that site conditions, reclamation and water chemistry are stable and no further monitoring is required. Any remaining financial assurances not used for closure and reclamation costs would be released back to the owner at that time. In the case of ESM, this final financial assurance release would likely occur after a 5 to 10-year successful post-closure monitoring period.

A Closure Plan and Cost Estimate update was completed by SRK Consulting in 2011 (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

20.1.5 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.

20.2 Graphite

20.2.1 Environmental

Waste and Tailings Management

Waste rock stockpiling is covered in Item 18.2.2 under the subheading "Waste Rock and Overburden Stockpiles." Waste rock generated during the Kilbourne pit development will be stockpiled in designated areas between the pit and Concentrate Plant, with perimeter ditches and collection ponds to manage runoff.

Tailings management facilities are detailed in Item 18.2.6. Tailings will be deposited in a phased manner within the Extended TMF (Years 1–5), Historic Arnold Pit (Years 4–13), and Raised TMF for relocated material from the pit footprint. The northern portion of the Kilbourne Pit will be backfilled with tailings in Year 4. All tailings and waste rock are assumed non-acid generating, allowing conventional reclamation methods. The TMF design incorporates staged embankment raises, containment dikes, and stability factors consistent with NYSDEC standards.

Water Management

The water management strategy is detailed in Item 18.2.5 and will utilize a closed-loop system to minimize freshwater use and control discharges. Contact water from pit dewatering and tailings storage will be routed through clarification and treatment ponds prior to reuse or controlled discharge under SPDES permit conditions. Clean runoff will be segregated and monitored before release. Post-closure, water handling will transition to passive treatment and periodic monitoring until compliance with regulatory standards is demonstrated.

Monitoring and Compliance

Operational monitoring will follow the OMS Manual protocols currently applied at ESM, including daily inspections of tailings deposition areas, monthly water quality sampling at treatment ponds, and quarterly geotechnical reviews of containment structures. Post-closure monitoring will continue for 5–10 years, focusing on water quality, vegetation establishment, and structural stability of reclaimed areas. Monitoring frequency and parameters will align with NYSDEC requirements and industry best practices.

20.2.2 Permitting

New York State Permitting

The proposed Kilbourne Pit extends beyond the currently permitted LOM boundary, necessitating a *State Environmental Quality Review Act* (SEQR) review due to the potential for significant environmental impacts. In accordance with NYSDEC requirements, a comprehensive suite of environmental and technical studies must be completed prior to submitting a mining permit modification application. These potentially include wetlands delineation, visual and noise impact assessments, residential well surveys, pre-blast building inspections, traffic analyses, and a hydrogeologic impact evaluation. ESM has already conducted residential well surveys and pre-blast building surveys around the Kilbourne Project area. Wetlands delineation is nearing completion. Although it is not anticipated, the large size of the Kilbourne Project's footprint is such that an archaeological or cultural resources survey may be required. The necessity of such studies would be determined by the NYSDEC, in consultation with the State Historic Preservation Office (SHPO). If SHPO identifies a potential sensitivity, it will require a Phase 1A archaeological assessment, and a subsequent Phase 1B field survey if an archaeological site is identified. ESM has already commenced work on this.

In contrast to prior permitting efforts for the Turnpike pits (previously referred to as Hoist House and Pump House pits), the studies required for the Kilbourne Pit must be conducted independently, as the proposed site encompasses a substantially larger and distinct area that was not previously evaluated. Consequently, all relevant assessments must be newly developed to reflect the unique environmental and geological conditions of the Kilbourne site.

The hydrogeologic evaluation will require the installation of new monitoring wells and the collection of baseline monthly water level data. Baseline water quality sampling will also be necessary, both from selected new wells and residential wells. These data will be integrated into the revised Mined Land Use Plan (MLUP), which must also include updated mining and reclamation plan maps, reclamation cross-sections, environmental assessments, and operational details such as water and dust management, traffic and blasting protocols, and post-mining land use objectives. ESM is already in the process of initiating new monitoring wells.

Upon completion of the required studies and MLUP revisions, the mining permit application will be submitted to the NYSDEC. It is anticipated that the Project will be classified as a Type I action under SEQR, which typically requires a Draft Environmental Impact Statement (DEIS) if a positive declaration is issued by NYSDEC. If a positive declaration is issued by NYSDEC, the SEQR process could take up to 310 days. This timeline assumes that the State will take all its allotted time during each step of the process, including agency coordination, preparation of a scoping document for the DEIS, DEIS preparation, public hearings, and the development of a Final Environmental Impact Statement (FEIS) and Findings Statement. Key variables that could extend this timeline include, but are not limited to, delays in agency responses; public comments; the need for supplemental environmental studies; multiple rounds of DEIS revisions with the State; public hearings; and contested issues.

To ensure compliance with applicable standards and operational requirements, the following will likely require resubmission for review and approval: Air Registration/Permit, Water Withdrawal Permit, SPDES Permit, and Building Permits. The NYSDEC will require an Article 24 Wetlands Permit for any State-regulated wetlands or their 100-ft wetland buffer zones disturbed by the Kilbourne Project. This ensures that all wetland impacts are reviewed and mitigated under state law.

Federal Permitting

Given the likelihood of federal funding for the Kilbourne Project, its development may require a small number of well-defined federal permits and reviews, consistent with other large mining projects in the U.S. In this respect, key authorizations are outlined below:

- ***Clean Water Act* Section 404 Permit (U.S. Army Corps of Engineers):** Required where the mine footprint intersects federally regulated wetlands or streams. This is a standard permit for any U.S. mining project having these impacts. Current wetlands delineation is ongoing to determine the applicability of this.
- **Section 401 Water Quality Certification (WQC):** Section 401 of the *Clean Water Act* requires that any applicant for a federal permit that may result in a discharge to waters of the United States must obtain a WQC. In New York, the 401 WQC is issued by NYSDEC, but it is tied to the Army Corps permit. The 404 permit cannot be obtained without NYSDEC's 401 WQC.
- ***National Environmental Policy Act* (NEPA) Environmental Impact Statement:** Provides a comprehensive review of environmental, cultural, and community impacts, led by the U.S. Army Corps of Engineers in coordination with other federal agencies. To avoid duplication, ESM will coordinate with the NYSDEC and the USACE to ensure that the preparation of the DEIS will also serve the needs of the NEPA EIS for USACE purposes. USACE has indicated to ESM that it may require the less comprehensive Environmental Assessment (EA) for the Kilbourne Project, rather than an EIS; however, this decision is only preliminary at this time.

- ***Endangered Species Act* (Section 7):** To confirm there are no impacts to federally protected species. Based on a preliminary review of the Project, current species potentially present include Northern long-eared bat (Endangered), tricolored bat (proposed endangered), and the monarch butterfly (proposed threatened). It present, tree clearing will be restricted to certain times of the year.
- ***National Historic Preservation Act* (Section 106):** To review potential cultural or historic resources. ESM does not expect anything specific in this regard to the Kilbourne Project (see above under New York State Permitting).

Given the aspect of federal funding associated with Kilbourne, the Project has a FAST-41 designation. This federal program coordinates permitting schedules across agencies and sets clear deadlines. FAST-41 designation ensures transparency, accountability, and limits legal challenges, giving investors the confidence that the Project can be permitted on a predictable timeline. ESM is working with federal and state agencies to align permitting schedules given the strategic nature of the Kilbourne Project and has received indication of the prospect of alignment.

20.2.3 Closure

A reclamation plan for the graphite pit and associated infrastructure will be developed and will be included in the forthcoming MLUP submission to the NYSDEC. Once the NYSDEC has determined that the mining permit modification application is complete, including the revised reclamation plan for Kilbourne, the Department will calculate the updated cost of reclamation. DEC will then issue a letter to ESM requesting the appropriate financial security, typically in the form of a surety bond or other acceptable surety. The modified mining permit will be formally issued only after the required financial security has been posted, ensuring that funds are available to cover reclamation obligations.

Reclamation activities are expected to include backfilling of excavated areas with overburden and waste rock, followed by grading to achieve stable slopes, typically at a 3H:1V ratio. Stripped topsoil will be retained and redistributed across reclaimed surfaces to support revegetation. Final grading will be designed to ensure proper drainage, and vegetative cover will be established to promote long-term stability and integration with the surrounding landscape. These measures will be implemented to restore the site to an environmentally sound condition consistent with post-mining land use objectives.

All closure and reclamation activities will be conducted in accordance with applicable regulatory requirements and will be subject to review and approval by the NYSDEC to ensure compliance with environmental standards.

The closure cost is estimated at $16.5M and is the subject of Item 21.1.2.7.

20.2.4 Social and Community Factors

There are no known social or community relations issues at this time that would impede or influence operations.

20.2.5 Concentrate Plant

The proposed graphite concentrate facility is planned to be co-located with ESM's existing Zinc Operation in St. Lawrence County, New York. This strategic siting offers notable advantages, including access to established infrastructure and the benefit of prior environmental assessments conducted for the existing operation. Nevertheless, the addition of new mineral processing activities will necessitate a comprehensive environmental planning process and impact evaluation.

Key regulatory updates will include amendments to the existing Mined Land Reclamation Permit to reflect the scope of the new facility, incorporate appropriate environmental safeguards, and define reclamation strategies. Additional permits, such as those governing air emissions, water discharge under the State Pollutant Discharge Elimination System (SPDES), and water withdrawal, will be assessed in detail during the next phase of study to ensure full compliance with applicable state and federal regulations.

The permitting timeline for a facility of this nature is variable and influenced by several factors, including the complexity of the environmental review under the state and federal permitting requirements the completeness and quality of submitted documentation, and the extent of public engagement. While co-location is expected to streamline certain aspects of development, the Project will still be subject to rigorous environmental review to evaluate potential physical, biological, and socio-economic impacts and to ensure alignment with relevant environmental standards.

20.2.5.1 Air Permit

Detailed emissions calculations, including process flow rates, emission factors, and operational hours will be critical in the next phase of the Project as they will determine the air permitting pathway. Should particulate matter emissions double with the addition of the graphite processing unit, the existing Air Facility Registration (AFR) will need to be modified to reflect the updated emissions profile. This modification process is expected to take approximately 8 months. Should total particulate matter emissions exceed 50 tons per year (ton/y), the facility will be required to obtain a State Facility Permit, which typically involves a more rigorous review and may take up to 12 months. Should total particulate matter emissions exceed 100 ton/y, a Title V Permit will be required and may exceed 12 months.

Regardless of the permitting pathway, the facility will be required to submit air permit applications to the NYSDEC, which may involve iterative rounds of review and comment. Notably, the facility is located more than 3 miles from the nearest disadvantaged community, and therefore a co-pollutant evaluation under the *Climate Leadership and Community Protection Act* (CLCPA) will not be required, should a CLCPA demonstration be necessary. The CLCPA demonstrations are not applicable to AFR applications. However, all permitting scenarios will require a review of air toxics to demonstrate compliance with Title 6 NYCRR Part 212.

20.2.5.2 Water Withdrawal Permit

To accommodate the graphite Concentrate Plant, ESM's Water Withdrawal Permit (WWP) will need to be amended to include any new pump installations. As part of the modification process, a 72-hour constant-rate pumping test will be required at each new well pump location to establish appropriate withdrawal limits. The quarry dewatering will also need to be incorporated into the water withdrawal permit. The rate of dewatering anticipated will be estimated during the hydrogeological evaluation. The WWP modification may take approximately 9 to 12 months, depending on completeness of the application and regional workload of NYSDEC staff.

20.2.5.3 SPDES Permit

The facility will need to submit a modification application for its existing State Pollutant Discharge Elimination System (SPDES) permit to account for any new wastewater discharges associated with the graphite Concentrate Plant. The mine is already conducting a water quality optimization program to improve discharge performance under its current permit. This ongoing work may introduce additional complexity to the modification process and is being addressed.

20.2.6 Purification and CSPG Plant

An Environmental Impact Statement (EIS) is a key step in evaluating, predicting and addressing potential environmental and social impacts, and the provision of a regulatory framework for a proposed development at an early stage. The EIS of the proposed Purification Plant and CSPG Plant have not commenced since the site location has not been finalized. The required EIS will be undertaken during the next phase to ensure both plants comply with all applicable environmental regulations in NY State and to evaluate their potential physical, biological and socio-economic impacts.

21. Capital and Operating Costs

This Item of the Technical Report presents the costs individually for the Zinc Operation and the Kilbourne Graphite Project.

21.1 Capital Cost Estimate

21.1.1 Zinc

21.1.1.1 Basis of Estimate

The estimated capital costs (CAPEX) for Zinc Operations come from previous project costs from the site, quotes, and industry reference guides.

21.1.1.2 Capital Cost Summary and Estimate Results

Estimated Project capital costs (including closures costs) across the LOM total $68.6M, consisting of the following distinct areas:

- #4 infrastructure and process capital;
- #4 mining capital equipment;
- #4 mining capital development;
- N2D and expansionary capital.

Table 21-1 presents the capital estimate summary for each area in Q1 2025 US$ with no escalation.

Table 21-1: Capital cost summary

Area	Cost Estimate ($M)
#4 Infrastructure & Process Capital	5.4
#4 Mining Capital Equipment	8.0
#4 Mining Capital Development	9.6
N2D and Expansionary Capital	35.5
Total Capital Cost	**58.5**
Closure Costs	15.2
Salvage Value	(5.1)
Total Capital Cost (including closure costs)	**68.6**

21.1.1.3 Zinc

Mining Capital Equipment

Replacement equipment includes:

- Three 6-yd LHDs;
- Two UG haul trucks;
- Two bolters;
- One jumbo drill;
- On powder tractor;
- Two telehandlers;
- One scissor lift.

Additional equipment includes:

- Five 6-yd LHDs;
- Two UG haul trucks;
- One bolter;
- Two jumbo drills;
- Seven transformers.

Service vehicles less than $25,000 are expensed and not capitalized. Rebuilds and other sustaining equipment requirements of fleet equipment are also expensed. One electric locomotive will be refurbished and put back into service. The utility vehicle is part of the support fleet, not in active production.

Table 21-2 presents the #4 Mine capital equipment costs estimate. Underground equipment for #2 Mine is included under N2D and Expansionary Capital.

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

#4 Mining Capital Equipment	Cost Estimate ($000)
Amalgamated Mining - Sandvik Robolt	1,170
ST1030 Replacement Loader	1,600
MT42	2,030
Telehandlers	120
Locomotive - Refurb	155
#4 Sustaining Capital	1,200
Transformers	1,120

#4 Mining Capital Equipment	Cost Estimate ($000)
Utility Truck (exploration)	75
Vent Fans	125
Forklift - Surface Warehouse	80
Roll-up Ventilation Door Replacement	282
Total	**7,957**

Underground Mining Capital Development

Capital development encompasses ramps to new extraction areas.

Table 21-3: #4 Mine capital waste development cost

#4 Mining Capital Development	Cost Estimate ($000)
Waste Development	9,641

Over the life of mine, #4 Mine will require 26,060 ft of capital development, and #2 Mine will require 8,140 ft of capital development. This is separate from the planned operational waste development.

Infrastructure and Processing Cost Estimate

Total infrastructure and processing capital costs are estimated to be $5.4M.

Processing capital costs include some equipment repairs, inspections and relining of the ball and rod mill, replacement of the surface loader for loading concentrate, and mill sustaining costs.

Infrastructure capital costs include ore skip rail replacement, a new engine for the fire water pump, rebuild of the waste transfer wall at the UG crusher, installation of a grizzly at the 3100 level ore transfer, a new spare ore skip, replacement of the shaft telehandler, replacement of the surface loader, installation of an OSA Head Stream, and roof repairs.

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

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

#4 Infrastructure & Process Capital	Cost Estimate ($000)
Ball Mill Reline	324
Rod Mill Reline	600
Roof Repair	900
Fire Pump (new engine)	75
Refurbish Waste Side Transfer Wall in 3180 Crusher Room	360
Install Grizzly at 3100 Ore Transfer	290
Crusher Recovery	50
New Ore Skip - Spare	396
Replace JD 544 Loader	200
Mill and Surface Sustaining	450
OSA Head Stream	425
IR telehandler - Ingersoll Rand	125
Ore Shaft Rails	1,229
Total	**5,424**

N2D and Expansion Capital

Total open pit capital costs for Turnpike are estimated to be $1.4M. This will cover pre-stripping of the pits. The equipment fleet will be leased at an expected total of $4.8M spread over the 4 years of mine life for the pits.

The capital cost for #2 has been combined with the overall expansion capital for both the underground zinc mines (#2 and #4) unless the capital is solely for #4 Mine. The zinc open pit costs are included in the expansion capital.

Table 21-5: Distribution of N2D and expansionary capital costs

N2D and Expansionary Capital	Cost Estimate ($000)
Waste Development	3,011
Transformers	370
EPIROC ST1030 1	1,235
Epiroc 2-Boom Jumbo	1,170
John Deere 310 Powder Tractor	150

N2D and Expansionary Capital	Cost Estimate ($000)
Sandvik Bolter	575
Bobcat Telehandler	150
Getman Scissorlift	220
Electrical Disconnect	1,500
EPIROC ST1030 2	435
Graphite/Open Pit Haul Truck and Excavator	400
Lower Mahler Ventilation	20,000
Turnpike Prestrip	1,435
Turnpike CAPEX	4,832
Total	**35,483**

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 Q1 2025 US$ and totaled $15.2M. 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 21-6.

Table 21-6: 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	4,838	4,838	-	-
Tailings	6,464	647	5,817	-
Surface Water Diversions	1,321	1,321	-	-
Contaminated Soils	160	160	-	-
Landfills	95	47	47	-
Closure Project Management Administration and Environmental Management Costs	902	451	451	-
Subtotal	**13,780**	**7,464**	**6,315**	**-**
Post-closure Costs				
Earthworks Inspection and Maintenance	373	-	-	373
Environmental Management	1,093	-	-	1,093
Subtotal	**1,466**	**-**	**-**	**1,466**
Total	**15,246**	**7,464**	**6,315**	**1,466**

Source: ESM, from Fennema & Sollner 2011 in 2025 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 $5.1M.

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.

21.1.1.4 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 Q1 2025. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- **Units of measure:** Short ton (ton), which is equivalent to 2,000 pounds;
- **Currency:** All capital costs are estimated in US$.

21.1.2 Graphite

The capital cost estimate for the Kilbourne Graphite Project encompasses all major facilities and infrastructure required to support mining, concentrate and Secondary Transformation. The Kilbourne Project involves two sites:

- **Kilbourne Site:** Located adjacent to the existing ESM location, it includes the open pit operations, supporting site infrastructure, tailings management facilities, Concentrate Plant, and Micronization Plant.
- **Secondary Transformation Site:** A separate location dedicated to the construction of the Purification Plant and CSPG Plant, where micronized natural flake graphite (NFG) will be further processed to produce enriched market-ready material.

The estimate covers four main areas of development:

- **Open Pit:** Includes the mobile equipment fleet, support and auxiliary equipment, and associated miscellaneous items required to mine and manage material from the Kilbourne open pit mine.
- **Site Infrastructure:** Includes site preparation, on-site access roads, mine rock and overburden storages, ROM stockpile, buildings and supporting infrastructure (truck shops, fuel facilities, etc.), mine water management systems, tailings management facilities, water supply and distribution system, transmission lines and substation, and other essential services required to sustain mine operation over the LOM.
- **Concentrate and Micronization Plants:** Covers the construction of the Concentrate Plant, the Micronization Plant, and related systems. The Concentrate Plant includes crushing, grinding, flotation, dewatering, and material handling to produce flake graphite concentrate. The flake graphite concentrate will then be sent to the Micronization Plant, located adjacent to the Concentrate Plant at Kilbourne Site, where it will be further processed to produce micronized NFG. The Micronization Plant consists of an air swept classification mill plus auxiliary equipment. The Micronization Plant reduces the NFG to two different product sizes, namely a D90 = 45-micron product and D90 = 15-micron product.

  It also covers the micronized NFG transportation from Kilbourne Site to the Secondary Transformation Site;
- **Secondary Transformation Site:** Consists of the Purification Plant and CSPG Plant designed to create PMG and CSPG to meet market specifications, including all related processing systems and supporting infrastructure.

This estimate consolidates input from specialized contributors for each Project area. Subsequent sections provide detailed descriptions of the scope and supporting assumptions for each component.

Responsibility for the cost estimates is divided among the study contributors, as defined in Item 2. Table 21-7 summarizes the scope divisions for the Graphite Study. The QP consolidated the cost information from the various QPs to develop the Kilbourne Graphite Project's capital cost estimates.

Table 21-7: Estimate scope division

Site	Scope
Kilbourne Site	**Open Pit Mining** ▪ Capital Costs ▪ Operating Costs
	Concentrate and Micronization Plants ▪ Capital Costs ▪ Operating Costs
	Site Infrastructure ▪ Capital Costs ▪ Operating Costs
	Site Closure & Reclamation Capital Costs
Secondary Transformation Site	**Purification Plant and CSPG Plant** ▪ Capital Costs ▪ Operating Costs

21.1.2.1 Summary of Results

A summary of the capital costs is presented in Table 21-8 and Table 21-9. The total capital cost for the Project is estimated at $431.7M, which includes initial, expansion, and sustaining capital. Of this total, direct costs amount to $334.8M, while owner's and indirect costs are $49.9M, and contingency is $47M.

Initial capital costs total $155.8M, primarily associated with the Concentrate Plant ($72.6M), Site Infrastructure and TMF ($27.2M), Micronization Plant ($11.5M), and Purification Plant ($5.3M).

Expansion capital is estimated at $175.9M, largely driven by the CSPG Plant ($99.5M) and further investments in the Micronization ($10.9M) and Purification ($8.0M) facilities.

Sustaining capital totals $100.0M, mainly related to ongoing costs in the Open Pit Mine ($41.6M), Concentrate Plant ($43.3M), and Site Infrastructure ($18.9M).

The closure costs were estimated at $16.5M with a total salvage value of $20.5M resulting in a net post-production cost of -$4M.

Table 21-8: Initial capital costs by pre-production years

Project Area		Total ($K)	Y-2 ($K)	Y-1 ($K)	Y1 ($K)
Kilbourne Site	Open Pit Mine	-	-	-	-
	Site Infrastructure and TMF	27,241	9,186	18,055	-
	Concentrate Plant	72,610	7,261	65,349	-
	Micronization Plant	11,497	-	10,348	1,150
Secondary Transformation Site	Purification Plant	5,291	1,058	3,704	529
	CSPG Plant	-	-	-	-
Direct Costs		**116,639**	**17,505**	**97,456**	**1,679**
Indirect Costs		15,792	3,990	11,508	294
Contingency		23,383	3,501	19,491	336
Total		**155,759**	**24,996**	**128,455**	**2,309**

Table 21-9: Initial, expansion and sustaining capital costs

Project Area	Total ($K)	Initial costs ($K)	Expansion Costs ($K)	Sustaining Costs ($K)
Open Pit Mine	41,641	-	-	41,641
Site Infrastructure and TMF	46,150	27,241	-	18,909
Concentrate Plant	115,922	72,610	-	43,312
Micronization Plant	22,362	11,497	10,865	-
Purification Plant	13,311	5,291	8,020	-
CSPG Plant	99,477	-	99,477	-
Closure and Salvage	-4,065	-	-	-4,065
Direct Costs	**334,799**	**116,639**	**118,362**	**99,798**
Owner's Cost and Indirects	49,939	15,792	33,900	247
Contingency	47,000	23,328	23,672	-
Total	**431,738**	**155,759**	**175,934**	**100,045**

21.1.2.2 Graphite Basis of Estimate

21.1.2.2.1 General

Accurate estimation of capital and operating costs is fundamental to assessing the economic viability of a proposed project. Together with projected revenues and other anticipated expenses, these cost estimates provide the foundation for the financial analysis detailed in Item 22.

For the Kilbourne Graphite Project, capital and operating costs were determined based on the mine plan and various plant designs. The estimation process incorporated assessments of material and labor requirements derived from the design, analysis of the process flowsheet, and anticipated consumption of power and supplies.

The CAPEX estimate for the Graphite Study is based on a scoping-level engineering assessment. Direct costs cover the supply and installation of equipment including associated direct installation workforce, as well as costs of buildings and site preparation. Indirect costs include expenses not directly involved in process operations, such as engineering, procurement and construction management services, development studies, legal fees, other temporary contractor services and fees.

Sources on capital costs include vendor budget quotations, historical data, similar projects, and factors.

Key Estimate Parameters

The following key parameters apply to the capital cost estimates:

- **Estimate class:** The CAPEX estimate is classified as a Class 5 estimate, as defined in the American Association of Cost Engineering (AACE) Recommended Practice No. 47R-11, typically used for preliminary evaluations. It carries an expected accuracy range of –20% to –50% on the low side and +30% to +100% on the high side, consistent with Class 5 estimate guidelines. For the purpose of this study, an accuracy range of approximately +/- 40% has been assumed;
- **Estimate base date:** The base date of the estimate is Q1, 2025. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- **Currency:** All capital costs are estimated in US$.

Assumptions

The following assumptions have been made:

- The estimate is based on imperial units for mining, site infrastructure, and Concentrate Plant inputs, and metric units for the Secondary Transformation Site;
- All mining mobile equipment will be leased;

- All mechanical and process equipment will be purchased new;
- Equipment costs are based on information and testwork available at the time of study;
- All required earthworks materials such as fill, sand, gravel, crushed rock, etc. can be sourced locally to the site;
- Construction is possible throughout the year;
- Numbers presented in the tables may not add up precisely due to rounding.

Currency and Exchange Rates

Cost information was received in multiple currencies and converted to US$ using the exchange rates provided in Table 21-10.

Table 21-10: Currency exchange rates on April 1, 2025

Currency	Code	Rate
United States Dollar	US$	1.0000
Euro	EUR	0.9267
Pound Sterling	GBP	0.7738

Major Exclusions

- No allowance was made in the CAPEX estimate for:
- Cost escalation and/or inflation beyond the Base Date;
- Currency fluctuations;
- Financing and interest charges during construction;
- Required permits, licenses or legal and administrative costs associated with government environmental and other relevant regulations. This includes reporting requirements during operation and related administrative costs;
- Federal and state sales tax, customs charges, excises;
- Costs related to force majeure and major strikes;
- Costs incurred before final investment decision;
- Financing costs;
- Insurance;
- 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.).

Direct and Indirect Costs

Direct cost allocations are captured within each associated area, including the open pit, site infrastructure, Concentrate Plant, Micronization Plant, Purification Plant and CSPG Plant, as required. Indirect costs related to the open pit, Kilbourne Site infrastructure, and Concentrate Plant are factors of the direct cost to cover the costs of the following:

- Engineering, procurement and construction management;
- Construction equipment and temporary facilities;
- Commissioning spares;
- Initial fills;
- Freight;
- Vendor representatives;
- Commissioning costs including quality control and other consultants.

Details regarding the indirect cost basis for the Purification and CSPG Plants are provided in Item 21.1.2.2.5.

Owner's Costs and Contingency

Costs incurred by the owner are not included in the capital cost estimates, as it is assumed that any owner-related expenses will be addressed through the ongoing Zinc Operation.

A contingency of 20% has been added on direct costs of the initial CAPEX and expansion CAPEX.

21.1.2.2.2 Open Pit

The capital estimate for the mining equipment includes items related to the purchase of the mining mobile equipment fleet for the open pit operations, maintenance, and the mine support equipment.

Initial capital cost estimate for the mining estimate includes:

- The initial purchase of the equipment fleet required for Year 1;
- Pre-production mining operating costs. However, for the proposed mine plan, a pre-production period was not required.

Sustaining capital costs for the mining estimate includes:

- Equipment purchases after Year 1 (fleet expansion);
- Equipment replacement;
- Equipment overhaul.

Estimates for mining equipment are based on the mining fleet equipment schedules and the equipment pricing from BBA equipment database.

The costs of overhauling larger types of major equipment were calculated separately from the distributed hourly cost so that major repair costs are forecasted in the year they occur, rather than this cost being averaged over many years. This method provides a more representative cash flow. For equipment replacement costs, individual equipment units cumulative operating hours are tracked up to a set limit, and then a replacement is estimated as sustaining capital costs incurred in that year.

21.1.2.2.3 Site Infrastructure

The following is a summary of the site infrastructure developed as part of the Kilbourne Graphite Project to support the open pit mining operation and the Concentrate and Micronization Plants at the Kilbourne Site. The CAPEX estimate covers the required equipment, workforce and materials to construct the infrastructure, as well as associated indirect costs for:

- **Site Preparation** – Site grading around the Concentrate Plant area.
- **Access Roads** – New primary and secondary access routes, as well as expansion and upgrading of existing roads within the Kilbourne on-site area.
- **Truck Facilities** – Multi-bay mobile equipment maintenance shop and truck wash bay.
- **Fuel Bay** – Dedicated facility to support mobile mining equipment.
- **Site Water Management Systems** – Drainage, ditches, and distribution systems.
- **Concentrate Plant Site Services** – Piping and pumps from existing treatment station for initial fill and make-up water and electrical allowances.
- **Tailings Management Facilities** – Placement and spreading of material for tailings facilities across the Raised TMF, extended TMF, and Historical Arnold pit. The required overburden and waste rock material will be sourced from the open pit or rehandled from the stockpiles.
- **General Services** – Sewage tanks, fire suppression system, weigh stations and telecommunications infrastructure.

No additional off-site infrastructure is anticipated for the Kilbourne Project Site. The existing high-voltage transmission lines, as well as other regional utilities and service networks, are expected to be sufficient to support the planned operations for the life of the mine. Consequently, no capital cost estimate has been included in this study for off-site infrastructure upgrades or modifications.

21.1.2.2.4 Concentrate and Micronization Plants

Concentrate Plant

The CAPEX estimate for the Concentrate Plant represents the costs to construct the milling and grinding circuit and associate indirect costs, and the supporting infrastructure required to service the facility. This includes the following:

- Jaw crusher;
- Magnets;
- Screens;
- Cone crusher;
- Rod mill;
- Ball mill;
- Flotation circuit with regrind mills;
- Concentrate dewatering and bagging equipment;
- Other miscellaneous equipment such as tanks, conveyor belts, hoppers, feeders, compressors, blowers and laboratory equipment.

The CAPEX was compiled using budgetary quotes of the major mechanical equipment. Other direct costs were factored using an industry standard approach. Other direct costs include:

- Reagents;
- Water services;
- Air services;
- Concrete;
- Structural steel;
- Architectural and unit building;
- Mechanical platework and tanks;
- Mechanical install;
- Piping;
- Electrical & instrumentation.

The sustaining capital costs for the concentrator have been estimated on the basis of $2.5/ton of mill feed.

Micronization Plant

The CAPEX of the Micronization Plant represents the costs to construct the milling and grinding circuit and associate indirect costs, and the supporting infrastructure required to service the facility. This includes the following:

- Air swept classifying mill including auxiliary equipment (vendor package);
- Dust extraction system;
- Other miscellaneous equipment such as hoppers, feeders and lab equipment.

The CAPEX was compiled using budgetary quotes of the major mechanical equipment. Other direct costs were factored using an industry standard approach. Other direct costs include:

- Air services;
- Concrete;
- Structural steel;
- Architectural and unit building;
- Mechanical install;
- Electrical & instrumentation.

Wear and spare parts are included in micronization operating costs; accordingly, no allowances are required for sustaining capital.

21.1.2.2.5 Secondary Transformation

The CAPEX estimate for the Secondary Transformation Site, consisting of the Purification Plant and CSPG Plant is based on the conceptual engineering design developed for the Kilbourne Graphite Project, including the PDC and mass balance as described in Item 17.2.3. The estimate includes both direct and indirect costs.

The direct cost components for the Purification Plant and CSPG Plant include the following:

- Mechanical equipment, installation and freight to the Secondary Transformation Site;
- Total installed costs for the following disciplines:
  - Electrical;
  - Piping;
  - Instrumentation;
  - Control;
  - Civil works;
  - Earthworks.

- Buildings and building services, including materials and supplies for process plants and ancillary buildings, such as offices and maintenance shops, as well as plumbing, heating, lighting and ventilation of these buildings;
- Site development, including fencing, grading, roads and parking areas, sidewalks, landscaping and similar items;
- Service facilities, including the cost of supplying steam, water, power, compressed air, fuel, waste disposal, fire protection, and other miscellaneous service items;
- Workforce required for construction, installation and management activities on site.

Indirect costs for the Purification Plant and CSPG Plant include engineering and construction management, overhead and legal costs.

Budgetary quotations for major mechanical equipment were obtained and used as the basis of the CAPEX. All remaining direct and indirect costs were derived using factors from Table 6-9 in Plant Design and Economics for Chemical Engineers by Peters, Timmerhaus and West (2002). These factors have been benchmarked against comparable projects and are supported by both internal data and commercial cost databases.

While the Purification Plant and CSPG Plant location has not yet been finalized, it is assumed that both plants will be located on the same property in an established prime chemical industrial estate with existing bulk supply services, access road and connection for gas, water, electricity, effluent and sewage disposal and treatment, stormwater management and waste removal.

The wastewater treatment plant has not been included in the Purification Plant and CSPG Plant CAPEX, as this will be addressed in the next phase of work following final site selection. This is possible as the wastewater treatment plant may be located within an industrial site where wastewater treatment services are already available.

For the Purification Plant and CSPG Plant, there is no equipment in the process flow that will require replacement and/or major refurbishment. As such, sustaining capital has not been included in the CAPEX.

21.1.2.3 Open Pit

Table 21-11 tabulates the major and support equipment purchase price assumptions used in the mining cost estimate. The costs are assumed to include purchase price, freight, and assembly & commissioning.

Table 21-11: Major and support mining equipment required and purchase price assumptions

Area	Example Model	Initial Units (#)	Peak Units (#)	Replacement Units (#)	Unit Purchase Price ($K)
Major Equipment					
Hauling	CAT745	4	10	0	847
Loading	EX CAT374	1	2	1	1,066
Drilling	5-inch	1	3	0	1,170
Support Equipment					
Track Dozer	CAT D8	3	4	1	975
Grader	CAT14	1	1	1	792
Wheel Loader	CAT 980	1	1	0	609
Utility Excavator	CAT 352	1	1	1	690
Water Truck / Sand Spreader	--	1	1	1	651
Powder Truck	--	1	1	0	90

Table 21-12 summarizes the initial and sustaining costs related to the open pit mine. A total of $41.6M has been estimated, of which $15.7M is for initial costs and $26.0M for sustaining capital. Equipment procurement is assumed to be conducted through a leasing arrangement, and the associated costs are captured under the operating cost estimate.

The lease term is 5 years, with a downpayment of 20% and an annual amortization rate of 20%. An interest rate of 4.75% has been applied to the outstanding lease balance, and the total leasing cost includes both principal repayments and interest charges over the term.

Table 21-12: Direct capital costs – Kilbourne open pit mine

Project Area	Total ($K)	Initial ($K)	Sustaining ($K)
Mine Equipment	39,753	-	39,753
Leasing Payments Interest Cost	1,888	-	1,888
Direct Costs	**41,641**	**-**	**41,641**

21.1.2.4 Site Infrastructure

Table 21-13 summarizes the Site Infrastructure capital cost estimate, covering initial capital and sustaining capital. Table 21-14 details the Indirect capital costs. Table 21-15 summarizes the costs related to the tailings. The capital costs of the site infrastructure are estimated at $46.1M, with $27.2M as initial capital and $18.9M for sustaining capital. Additionally, the indirect costs associated with the mine infrastructure are estimated to be $2.9M.

Indirect costs in this estimate are calculated as a function of direct costs. Because the site is already operational, some facilities, services, and workforce requirements are shared with existing operations, resulting in lower direct capital costs. As a result, the total indirect costs are reduced accordingly, even though the applied indirect percentage remains unchanged.

Table 21-13: Direct capital cost estimate – Site infrastructure

Project Area	Total ($K)	Initial ($K)	Sustaining ($K)
General	1,645	-	1,645
Site Preparation	3,156	2,861	296
Access Roads	2,787	708	2,079
Truck Shop	2,256	2,256	-
Truck Wash	1,074	1,074	-
Fuel Bay	460	-	460
Site Water Management	15,868	9,709	6,158
Process Plant Site Services	9,048	8,489	560
TMF	8,427	877	7,550
Land Purchase*	1,430	1,268	162
Direct Costs	**46,150**	**27,241**	**18,909**

* Provided by Titan

Table 21-14: Indirect capital cost estimate – Mine infrastructure

Description	Total ($K)	Initial ($K)	Sustaining ($K)
EPCM Services	1,818	1,818	-
Commissioning Services	287	228	58
Overhead Expenditures	202	140	62
Common Distributable	624	497	127
Indirect Costs	**2,931**	**2,683**	**247**

Table 21-15 tabulates the estimated TMF construction costs of $8.4M, with $0.9M allocated to the pre-production period at Year -1 and the remaining $7.6M allocated to the production phase. These costs cover three main study areas over the LOM, including the construction of the Extended TSF embankment, the development of the Kilbourne pit containment dike, and the construction of the Historical Arnold Pit containment dike.

Table 21-15: Direct capital cost estimate – TMF

Project Area	Total ($K)	Pre-production (Y-1) ($K)	Production ($K)
Extended TSF Embankment	4,167	877	3,291
Kilbourne Containment Dike	1,982	-	1,982
Historical Arnold Pit Containment Dike	2,278	-	2,278
Direct Costs	**8,427**	**877**	**7,550**

21.1.2.5 Concentrate and Micronization Plants

The direct capital expenditures for the graphite Concentrate and Micronization plants are presented in Table 21-16. The total direct capital cost for the Concentrate and Micronization plants is $138.3M, consisting of $95.0M in initial cost and $43.3M in sustaining capital cost. The Concentrate Plant accounts for $115.9M, while the Micronization Plant represents $22.4M of the total.

Details of the initial capital costs, direct mechanical costs, and indirect costs for both the Concentrate and Micronization plants are presented in Table 21-17 through Table 21-23.

The sustaining capital costs for the concentrator have been estimated on the basis of $2.5/ton of mill feed, which corresponds to a total of $43.3M for the LOM. Wear and spare parts are included in Micronization OPEX accordingly, no allowance for sustaining CAPEX has been considered in the Micronization Plant CAPEX estimate.

Table 21-16: Direct capital costs – Concentrate and Micronization plants

Concentrate Plant Area	Total ($K)	Concentrate Plant ($K)	Micronization Plant ($K)
Concrete	7,395	5,098	2,298
Structural Steel	9,548	8,974	574
Architectural and Building	11,793	8,921	2,872
Piping	3,823	3,823	-
Mechanical	56,740	40,696	16,044
Electrical and Instrumentation	5,672	5,098	574
Initial Costs	**94,972**	**72,610**	**22,362**
Sustaining Costs	**43,312**	**43,312**	**-**
Direct Costs	**138,284**	**115,922**	**22,362**

Table 21-17: Initial capital costs by pre-production year – Concentrate Plant

Concentrate Plant Area	Total ($K)	Y-2 ($K)	Y-1 ($K)
Concrete	5,098	510	4,588
Structural Steel	8,974	897	8,076
Architectural and Building	8,921	892	8,029
Piping	3,823	382	3,441
Mechanical	40,696	4,070	36,626
Electrical and Instrumentation	5,098	510	4,588
Initial Costs	**72,610**	**7,261**	**65,349**

Table 21-18: Initial capital costs by year – Micronization Plant

Micronization Plant Area	Total ($K)	Y-1 ($K)	Y1 ($K)
Concrete	1,181	1,063	118
Structural Steel	295	266	30
Architectural and Building	1,477	1,329	148
Mechanical	8,249	7,424	825
Electrical and Instrumentation	295	266	30
Initial Costs	**11,497**	**10,348**	**1,150**

Table 21-19: Direct costs – Initial capital and expansion costs by year – Micronization Plant

Micronization Plant Area	Total ($K)	Initial Capital (Y-1 to Y1) ($K)	Expansion (Y2 to Y6) ($K)
Concrete	2,298	1,181	1,116
Structural Steel	574	295	279
Architectural and Building	2,872	1,477	1,395
Mechanical	16,044	8,249	7,795
Electrical and Instrumentation	574	295	279
Initial and Expansion Costs	**22,362**	**11,497**	**10,865**

Table 21-20: Direct costs – Concentrate Plant mechanical breakdown

Concentrate Plant Area	Cost ($K)
Mechanical Equipment	**25,489**
Crushing	1,369
Grinding	6,510
Flotation	5,220
Concentrate Dewatering, Drying and Bagging	2,708
Misc. Mechanical Equipment	9,683
Reagents	**2,549**
Water Services	**2,549**
Air Services	**1,274**
Installation Costs	**8,834**
Direct Mechanical Costs	**40,696**

Table 21-21: Direct costs – Micronization Plant mechanical breakdown

Micronization Plant Area	Cost ($K)
Mechanical Equipment	**11,488**
Air Swept Milling System (turnkey)	11,163
Dust Collection System	325
Air Services	**1,723**
Installation Costs	**2,833**
Direct Mechanical Costs	**16,044**

Table 21-22: Indirect costs – Concentrate Plant

Description	Total ($K)
EPCM Services	4,830
Commissioning Services	1,685
Overhead Expenditures	3,655
Indirect Costs	**10,170**

Table 21-23: Indirect costs – Micronization Plant

Description	Total ($K)
EPCM Services	265
Commissioning Services	166
Overhead Expenditures	377
Indirect Costs	**1,308**

21.1.2.6 Secondary Transformation

The total estimated cost for the Secondary Transformation Site is $148.3M, consisting of $112.8M in direct costs and $35.5M in indirect costs presented in Table 21-24.

The Secondary Transformation Site consists of two processing facilities: the Purification Plant and CSPG Plant. Construction of the Secondary Transformation Site will occur in two phases, with the Purification Plant constructed in Phase 1, followed by the CSPG Plant in Phase 2 (including dedicated purification circuit).

Table 21-24: Direct and indirect costs by phase – Purification and CSPG Plants

Project Area	Total ($K)	Phase 1	Phase 2
		Initial Capital ($K)	Expansion Cost ($K)
Purification	13,311	5,291	8,020
Spheroidization	24,585	-	24,585
Coating	74,892	-	74,892
Direct Costs	**112,788**	**5,291**	**107,497**
Indirect Costs	**35,530**	**1,631**	**33,900**
Total Costs	**148,318**	**6,922**	**141,397**

Construction of the Phase 1 Purification Plant is planned for Years -2, -1 and 1 with a 20%, 70% and 10% expenditure split, respectively. The CSPG Plant comprises a dedicated purification circuit designed to produce PMG for subsequent spheroidization and coating. The dedicated purification circuit for the CSPG Plant is considered an expansion of Phase 1 Purification Plant and as such, is constructed in Phase 2, which is planned for Years 4, 5 and 6, with a 10%, 80% and 10% split, respectively.

Details of the direct costs, and indirect costs for the Purification Plant (Phase 1) and dedicated purification circuit for CSPG Plant (Phase 2) are presented in Table 21-25 and Table 21-26, respectively.

Table 21-25: Direct costs – Purification

Description	Total ($K)	Phase 1	Phase 2
		Initial Capital ($K)	Expansion Cost ($K)
Site Development and Building	3,692	1,468	2,225
Piping	1,413	562	851
Mechanical	6,564	2,609	3,955
Electrical and Instrumentation	1,641	652	989
Direct Costs	**13,311**	**5,291**	**8,020**

Table 21-26: Indirect costs – Purification

Description	Total ($K)	Phase 1	Phase 2
		Initial Capital ($K)	Expansion Cost ($K)
EPCM Services	3,054	1,214	1,840
Overhead Expenditures	1,048	417	632
Indirect Costs	**4,103**	**1,631**	**2,472**

Phase 2 also entails construction of the spheroidization and coating sections of the CSPG Plant. These areas are to be constructed after the production starts (Years 4, 5 and 6 with 10%, 80% and 10% split, respectively) and therefore are considered as expansion capital costs. Table 21-27 and Table 21-28 provide the details of the direct and indirect costs for the CSPG Plant (spheroidization and coating sections only), respectively.

Table 21-27: Direct expansion costs – CSPG Plant

Description	Total ($K)	Spheroidization ($K)	Coating ($K)
Site Development and Buildings	27,953	7,178	20,775
Piping	9,386	1,436	7,951
Mechanical	50,392	13,459	36,933
Electrical and Instrumentation	11,746	2,512	9,233
Direct Costs	**99,477**	**24,585**	**74,892**

Table 21-28: Indirect costs – CSPG Plant

Description	Total ($K)	Spheroidization ($K)	Coating ($K)
EPCM Services	23,644	6,460	17,184
Overhead Expenditures	7,783	1,884	5,899
Indirect Costs	**31,428**	**8,345**	**23,083**

21.1.2.7 Site Closure and Reclamation

The closure cost estimate for the Kilbourne Site at ESM is based on the closure plan (Fennema, 2011) prepared for the existing Zinc Operation. The graphite site is expected to have a similar footprint, infrastructure complexity, and environmental management requirements as the Zinc concentrator. Accordingly, the closure activities such as demolition, decontamination, water management, and post-closure monitoring are assumed to be comparable in scope. The original 2011 plan estimated a total closure cost of $10M, which has been adjusted to $16.5M (2025 US$). This adjusted figure provides a reasonable basis for estimating the closure liability associated with the graphite addition, pending further engineering and regulatory review.

The Purification Plant and CSPG Plant are akin to a chemical factory and as such, are not linked to a LOM. Both plants can continue to process micronized NFG concentrate from other third-party suppliers after closure of the Kilbourne mine. As a result, site closure has not been included in the Purification Plant and CSPG Plant CAPEX, as it is not foreseen this expenditure will be incurred.

21.1.2.8 Contingency

A 20% contingency, representing $23.6M is carried on all direct initial and expansion capital costs. Table 21-29 presents the contingency breakdown by Project area.

No contingency is added to sustaining capital and indirect capital costs since the site is currently in operation.

Table 21-29: Contingency breakdown

Project Area	Contingency ($K)
Site Infrastructure	5,448
Concentrate Plant	14,522
Micronization Plant	4,472
Purification Plant	2,662
CSPG Plant	19,895
Direct Costs	**47,000**

21.2 Operating Cost Estimate

21.2.1 Zinc

21.2.1.1 Site Operating Cost Summary

Estimated Project operating costs total $373M for all Zinc Operations. $362M is for the #2 and #4 mines. $11M is for the Turnpike surface pits.

21.2.1.1.1 Underground

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 Item of the report is broken into four major sections, which include mining, processing, General and Administrative (G&A), and concentrate transportation costs. As the #2 Mine is considered unburdened ore, the cost of transportation is carried by #4 Mine material and the G&A is only what applies solely to #2 Mine.

Site operating costs are presented in 2025 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 and 2025, so is considered highly accurate. Mining, milling, G&A, and transportation costs for 2025 are considered to be representative of operating costs going forward. Site operating costs for the underground are summarized in Table 21-30.

Table 21-30: Summary of underground operating cost

Underground	Fixed Cost ($K/y)	Variable Cost ($/ton milled)	LOM Cost ($M)
#4 Mine			
Mining – Mineralized Material	351.00	53.00	165.25
Mining – Waste	-	17.00	6.88
Processing	854.00	10.00	38.25
G&A	9,256.00	-	64.66
Concentrate Transportation	8.23	8.00	31.01
Royalties	0.09	0.18	0.17
Subtotal	**10,469.32**	**88.18**	**306.22**
#2 Mine			
Mining – Mineralized Material	-	33.00	37.21
Mining – Waste	-	17.00	4.85
Processing	-	10.00	11.28
G&A	-	2.00	2.26
Concentrate Transportation	-	-	-
Royalties	**-**	**-**	**-**
Subtotal	**-**	**62.00**	**55.60**
Total	**10,469.32**	**150.18**	**361.82**

21.2.1.1.2 Open Pit

Surface site operating costs were prepared from first principles and industry estimation references. The site operating cost is based on leased but Owner operated mining / services fleets, and minimal use of permanent contractors.

Site operating costs in this Item of the report is broken into three major sections, which include mining, processing, and G&A. As Turnpike open pits are considered unburdened ore, only direct costs are considered. Administrative support for the surface pits will be provided by the staff already in place for #2 and #4 mines. So minimal G&A is expected.

Site operating costs are presented in 2025 US$ on a calendar year basis. No escalation or inflation is included. Mining, milling, G&A, and transportation costs for 2025 are considered to be representative of operating costs going forward. Site operating costs for the zinc open pits are summarized in Table 21-31.

Table 21-31: Summary of open pit operating costs

Open Pit	Unit Cost ($/ton milled)	LOM Cost ($M)
Mining - Ore	$4.70	$1.88
Mining - Waste	$4.70	$4.98
Processing	$10.00	$3.99
G&A	$2.00	$0.00
Concentrate Transportation	N/A	N/A
Total	**$21.40**	**$10.85**

21.2.1.2 Summary of Site Personnel

Table 21-32: Summary of site personnel

Position	Staff/Hourly	Total
Underground Mining		
Mine Management	1/0	**1**
Mine Operations	0/67	**67**
Mine Maintenance	1/19	**20**
Crush, Hoist, Shaft	0/9	**9**
Processing		
Process Management	1/0	**1**
Process Operations	0/14	**14**
Process and Surface Maintenance	0/6	**6**
G&A		
General Management	1/0	**1**
Accounting	3/0	**3**
Technical Services	11/0	**11**
Warehouse	3/3	**6**
Human Resources	3/0	**3**
Safety and Environment	3/0	**3**
Site Total	**27/118**	**145**

Site personnel is based on current staffing levels. The site is currently operating with 141 full time employees.

21.2.2 Graphite

21.2.2.1 Summary of Results

This Item summarizes the operating cost estimate developed for the Kilbourne Site and Secondary Transformation Site. Similar to the capital cost summary, the operating costs encompass the open pit mine, the associated site infrastructure, and the facilities for the Concentrate, Micronization, Purification and CSPG Plants. In addition to the on-site operating costs, the off-site transportation costs for micronized NFG concentrate from the Kilbourne Site to the Secondary Transformation Site are included.

The total operating costs over 13-year life of mine are based on the following:

- Kilbourne graphite pit mining operations commencing in Year 1;
- Total material mined is 62.77 M tons;
- Total milled tonnage is 18.10 M tonnes (equivalent to 19.95 M tons);
- Total graphite concentrate is 486.7 M tonnes;
- Total micronized NFG is 157.6 k tonnes;
- Total PMG is 121.4 k tonnes;
- Total CSPG is 110.8 k tonnes.

The Kilbourne Graphite Study operating costs are presented in Table 21-33. The total LOM operating costs are estimated at $886M. The largest cost contributors are:

- Open Pit Mining: $183.8M (21%);
- Concentrate Plant: $239.7M (27%);
- Micronization Plant: $75.6M (9%);
- Purification Plant: $107.2M (12%);
- CSPG Plant: $199.6M (23%).

Other costs include G&A ($47.1M, 5%), TMF ($11.2M, 1%), and transportation ($21.7M, 2%).

The unit operating cost for the Project is estimated at $44.4 per ton milled at the Concentrate Plant. Further breakdowns of each cost component are provided in the following sections.

Table 21-33: Project all-in operating costs

Project area		LOM Total	Remaining Concentrate	NFG Micronized	PMG	CSPG
		($K)	($/t Concentrate)	($/t Micronized NFG)	($/t Saleable PMG)	($/t Saleable CSPG)
Open Pit	Kilbourne Pit Mining	183,832	377.71	389.39	413.70	562.99
Site Infrastructure	G&A	47,077	96.73	99.72	105.94	144.17
	TMF	11,178	22.97	23.68	25.15	34.23
Concentrate and Micronization Plants	Concentrate Plant	239,677	492.45	507.68	539.38	734.02
	Micronization Plant	75,647	-	176.42	186.18	227.65
Purification and CSPG Plants	Transport[1]	21,694	-	-	79.68	108.44
	Purification Plant	107,222	-	-	883.00	-
	CSPG Plant	199,610	-	-	-	1,801.00
Operating Costs		**885,936**	**989.86**	**1,196.89**	**2,233.05**	**3,612.50**

(1) Micronized NFG concentrate transportation cost from Kilbourne Site to Secondary Transformation Site.

21.2.2.2 Major Assumptions

- A diesel price of $0.79 per liter delivered to the site was considered;
- Diesel exhaust fluid (DEF) is estimated at 3%;
- Electricity price of $0.07 per kWh was used;
- Natural gas price of $0.02 per Nm^3 was used;
- Water price of $1.61 per m^3 was used;
- The mine workforce is based on operating two crews on two shifts per day, 5 days per week;
- The Concentrate, Micronization,Purification and CSPG Plants will operate in four operational shifts to ensure continuous production of the plants;
- No expats are being considered for key positions;
- The workforce rates for salaried and hourly personnel were provided by Titan, which account for assumptions related to bonuses and overtime;
- All estimated salaries and wages also include the entire workforce burden (e.g., benefits, insurance, etc.).

21.2.2.3 Open Pit

Mining operating costs have been developed based on the Kilbourne pit mining plan.

Mine operating costs are built up from first principles. Based on the level of study, inputs are derived from BBA database of equipment pricing, as well as inputs provided by Titan. This includes database cost and consumption rates for such inputs as fuel, fluids, tires, undercarriage, ground engaging tools (GET), machine parts, machine major components, and operating and maintenance workforce.

Mine operating costs averaged over the life of the operation are estimated at $3.11/ton of material mined, equal to $9.77/ton of material milled. The $3.11/ton includes $2.93/ton for the Kilbourne open pit mining and $0.18/ton for TMF relocation costs.

A breakdown of the mining operating cost estimate is provided in Table 21-34.

Table 21-35 shows the mining operating cost by activity by year.

Table 21-34: Breakdown of average LOM mining operating costs

Project Area	LOM Total ($K)	Unit Cost ($/ton Mined)	Percentage (%)
Drilling	11,690	0.19	6%
Blasting	40,395	0.64	21%
Loading	7,946	0.13	4%
Hauling	22,457	0.36	12%
Support	22,425	0.36	11%
Service	9,357	0.15	5%
Workforce	71,865	1.14	37%
General	8,875	0.14	5%
Direct Costs	**195,009**	**3.11**	**100%**

Note: Includes $11.2M for TMF relocation costs.

Table 21-35: Mining operating cost by activity by year

Item	Unit	LOM	Year												
		Total	1	2	3	4	5	6	7	8	9	10	11	12	13
Drilling	$M	**11.69**	0.58	0.59	0.46	0.45	1.01	0.87	1.25	1.49	1.32	1.32	0.86	0.84	0.64
Blasting	$M	**40.39**	2.09	2.12	1.67	1.63	3.64	3.12	4.50	5.38	4.76	4.76	3.09	2.06	1.57
Loading	$M	**7.95**	0.41	0.52	0.53	0.71	0.63	0.63	0.95	1.02	0.75	0.75	0.49	0.32	0.25
Hauling	$M	**22.46**	1.12	1.39	1.17	1.63	1.66	1.62	2.44	2.68	2.18	2.44	1.77	1.29	1.09
Support	$M	**22.43**	1.48	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.09
Service	$M	**9.36**	0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.69
Workforce	$M	**71.87**	4.41	4.60	4.60	5.05	5.78	5.78	6.68	6.87	6.68	6.68	5.51	5.06	4.17
General	$M	**8.87**	0.62	0.65	0.64	0.64	0.68	0.73	0.77	0.75	0.76	0.71	0.66	0.63	0.63
Total Mine OPEX	**$M**	**195.01**	**11.42**	**12.38**	**11.60**	**12.64**	**15.93**	**15.27**	**19.12**	**20.72**	**18.98**	**19.20**	**14.91**	**12.72**	**10.13**
Unit Mining Rate	$/ton mined	**3.11**	3.48	3.05	2.86	2.30	3.15	3.05	2.56	2.60	3.16	3.20	3.83	5.01	5.20
Unit Mining Rate	$/ton milled	**9.77**	9.32	8.84	8.30	9.33	10.41	9.01	10.26	11.20	10.30	11.87	10.32	9.57	7.24

Notes: Includes $11.2M for tailings relocation costs.

21.2.2.3.1 Equipment

Equipment costs have been estimated based on the database cost from similar projects.

Equipment costs include:

- Fuel;
- Lube/fluids:
  - Fluid's estimate does not include consumables (fuel/DEF/grease). All other fluids for regular maintenance are included (oils/coolant/etc.);
  - Allowance for grease consumption.
- Tires/undercarriage:
  - Tires estimate is based on MSRP tire pricing, without the installation workforce.
- Undercarriage estimate is based on parts for regular undercarriage maintenance and replacement;
- GET/ bucket/truck body includes reconditioning, supplies, and wear item replacement at regular intervals. Based on an average hard rock application;
- Parts;
- Maintenance & repair includes parts for planned maintenance (no fluids), and includes all major component replacements at planned intervals;
- Overhaul costs have been estimated based upon 35–45% of the purchase price and expended at approximately ½ the life (typically 18,000 hours) for major equipment. Overhaul costs are categorized as sustaining capital.

Items 21.2.2.3.1 to 21.2.2.3.5 summarize the equipment operating cost basis used in estimating the mining operating cost.

21.2.2.3.2 Equipment Fuel

Diesel fuel is used to operate mine trucks, shovels, loaders, drills, dozers and other mine equipment. Fuel consumption was estimated for each year of operation based on equipment specifications and equipment utilization.

21.2.2.3.3 Drilling and Blasting

Unit costs are estimated to include emulsion product, initiation systems which include detonation cord, boosters, and non-electric detonators, delivery of the product to the site, site storage, delivery of the product to the hole, hole loading and shooting blasts, necessary equipment, and facilities (pickup trucks, blasting trucks, stemming loader, storage facilities, magazine, garage, trailers, and fencing).

The unit costs exclude wall control. Wall control drilling and blasting was estimated at 15% of production costs.

21.2.2.3.4 Workforce

The workforce requirements were estimated to support the mine plan. The mine workforce has been calculated to total 40 employees during the first year of operation and will reach a peak of 62 employees in Year 8. The workforce rates for salaried and hourly personnel were provided by Titan, which included assumptions for bonuses and overtime. Additional allowances were included to account for the entire workforce burden (e.g., benefits, insurance, etc.).

Table 21-36 tabulates the assumptions used to build up the annual compensation per position.

Table 21-36: Annual compensation per mining position

Description	Base Salary ($/y)	Bonus ($/y)	Overtime ($/y)	Burden (%)	Burden ($/y)	Total Cost ($/y)
Manager	140,000	-	-	20%	28,000	168,000
Superintendent	130,000	-	-	20%	26,000	156,000
Supervisor	100,000	-	25,000	20%	20,000	145,000
Engineer	100,000	-	-	20%	20,000	120,000
Geologist	82,000	-	-	20%	16,400	98,400
Technician	55,000	-	12,000	20%	11,000	78,000
Planner	66,000	50,000	20,000	20%	13,200	149,200
Skilled Operator	60,000	42,000	18,000	20%	12,000	132,000
Semi-Skilled Operator	52,000	35,000	15,000	20%	10,400	112,400
Lower-Skilled Operator	48,000	25,000	12,000	20%	9,600	94,600
Mechanic	80,000	-	20,000	20%	16,000	116,000

21.2.2.3.5 General Services and Miscellaneous

This item covers allowances for costs related to technical services consulting, specialized mining software, equipment rental, grade control, and pit dewatering.

Table 21-37: Other cost assumptions

Description	Unit	Value
Pit Dewatering Miscellaneous	$/year	100,000
Ore Grade Control	$/ton of material	0.25
Miscellaneous Allowance	$/year	200,000

21.2.2.4 Site Infrastructure

Site infrastructure accounts for an estimated $58.3M of the total operating costs, including general and administrative (G&A) activities and the operation of the removing existing zinc tailings located on the Kilbourne graphite deposit. These costs cover essential site services and support functions required to sustain mining and processing operations, such as administration, maintenance, and tailings management.

The G&A total of $47.1M is based on an estimated 8% of the Zinc Operations G&A, for $765K annually allocated from Year 1 to Year 5 of the Kilbourne Graphite Project. Starting in Year 6, when the Zinc Operation ceases, the G&A increases to $5.1M annually. Included in the G&A is $2.45M for site water pumping.

A total of approximately 5.6 M tons of existing zinc tailings material is planned to be relocated over the life of the Kilbourne open pit mine, primarily during Years 2 to 5 operations. The relocation activities, estimated at a unit rate of $2.01 per ton, represent a total cost of about $11.2M. This work includes the loading, transport, and placement of tailings within the proposed TMF area.

Table 21-38: Summary of site infrastructure

Description	LOM Total ($K)
G&A	47,077
TMF	11,178
Operating Costs	**58,254**

21.2.2.5 Concentrate Plant

The operating cost (OPEX) of the Concentrate Plant is based on an average annual throughput of 1,534,700 ton/y of potential mill feed over the LOM. The following availabilities are assumed for the different parts of the plant:

- Crushing Plant 70% (6,132 h/y);
- Process Plant 92% (8,029 h/y);
- Filtration plant 80% (7,008 h/y).

The operating cost for the Concentrate Plant is $239.7M or $12.01/ton milled. A summary of the operating breakdown is provided in Table 21-39.

Table 21-39: Summary of average Concentrate Plant operating costs

Description	LOM Total ($K)	Unit Cost ($/ton milled)	Percentage (%)
Workforce	65,540	3.29	27%
Fuel (Diesel)	1,591	0.08	1%
Reagents (MIBC)	10,600	0.53	4%
Comminution Circuit Consumables	58,559	2.94	24%
Bagging	23,941	1.20	10%
Technical Support	1,170	0.06	0%
Electrical Power	78,277	3.92	33%
Operating Costs	**239,677**	**12.01**	**100%**

The operating cost estimate was compiled from various sources including workforce and materials cost from the operating zinc plant, metallurgical testwork, supplier pricing, and first-principle calculations.

The following sections detail the OPEX basis for the Concentrate Plant.

21.2.2.5.1 Power

The entire concentrator operates on electrical power only including the concentrate dryer. The total connected power is 9.3 MW and drawn power is 7.9 MW. The power consumption of the entire concentrator including dryer amounts to 41.3 kWh/ton or $3.92/ton milled.

21.2.2.5.2 Reagents

Only two reagents are required for the graphite concentrator, namely diesel and MIBC. The expected reagent costs for the concentrator are presented in Table 21-39. The lab reagent dosages were used as the basis for the reagent consumption calculation. These dosages were adjusted to take into account the continuous operation of the circuit with recycling streams.

21.2.2.5.3 Workforce

Total staffing of the Concentrate Plant is projected to be 58 people in plant operations, plant maintenance, and technical services.

Staffing levels assume a high degree of plant automation including real-time process monitoring and control using Bluecube or similar technology. This reduces the amount of manual process adjustments and sampling required. Some technical services are shared with the Zinc Operation.

Table 21-40: Annual compensation per Concentrate Plant position

Description	Base Salary ($/y)	Bonus ($/y)	Burden ($/y)	Total Cost ($/y)
Plant Operations				
Plant Superintendent - Ops & Maintenance	150,800	15,080	45,240	211,150
General Foreman	124,800	9,360	37,440	171,630
Plant Operations Shift Supervisor	83,200	4,160	24,960	112,350
Control Room Operator	68,536	1,371	20,561	90,498
Crushing / Screening / Grinding Operator	68,536	1,371	20,561	90,498
Polishing / Flotation Operator	68,536	1,371	20,561	90,498
Product Screening & Bagging Operator	48,880	978	14,664	64,552
Plant helper / cleaner / Laborer	48,880	978	14,664	64,552
Plant Maintenance				
General Foreman - Plant Maintenance / Planning	124,800	9,360	37,440	171,630
Planner - Mechanical	87,360	4,368	26,208	117,966
Shift Supervisor - Mechanical / Electrical	87,360	4,368	26,208	117,966
Mechanical Trades	79,040	1,581	23,712	104,363
Electrical & IE&C Trades	79,040	1,581	23,712	104,363

Description	Base Salary ($/y)	Bonus ($/y)	Burden ($/y)	Total Cost ($/y)
Technical Services				
Mechanical Engineer	145,600	7,280	43,680	196,590
Senior Metallurgist	145,600	7,280	43,680	196,590
Lab Chemist / Lead	87,360	4,368	26,208	117,966
Lab Technician	68,536	1,371	20,561	90,498

21.2.2.5.4 Operating Consumables

The operating consumables include items such as liners, lifters, grinding media, concentrate bagging, concentrate filter cloths, dust collector bags, and lab consumables.

Table 21-41: Costs of comminution circuit consumables

Description	Unit Cost ($/ton milled)
Crusher Liners	0.16
Rod Mill Liner/Lifters	0.10
Ball Mill Liner/Lifters	0.10
Steel Grinding Media	1.43
Ceramic Grinding Media	0.97
Misc Consumables	0.19
Operating Costs	**2.94**

21.2.2.5.5 Advanced Plant Control

The Kilbourne graphite Concentrate Plant will be designed with advanced process control to reduce the amount of manual sampling and process adjustment. This advanced process control strategy includes real-time analysis of the various process streams, which is analyzed by Artificial Intelligence (AI) supported algorithms to make automatic adjustments to process variables such as reagent dosages, air flow rates, pulp densities, or froth levels.

21.2.2.6 Micronization Plant

The Micronization Plant is planned to produce 157.6 kt of micronized NFG over the LOM. OPEX are estimated on an annual and per dry tonne of graphite concentrate feed. The costs are summarized in Table 21-42. The unit cost per tonne of concentrate feed of $155.4 is primarily driven by workforce cost (32%), electrical power (24%), and overhead (21%). Table 21-43 presents the breakdown of operating costs associated with the micronization process for each product stream. The first section corresponds to graphite that is sold directly after micronization without further processing, while the subsequent sections include the portion of micronized graphite that undergoes additional processing to produce purified and coated-spheroidized graphite (PMG and CSPG).

Table 21-42: OPEX Summary - Micronization Plant

Description	LOM Total ($K)	Unit Cost ($/t concentrate feed)	Percentage (%)
Workforce	24,488	50.3	32%
Electrical Power	18,009	37.0	24%
Consumables	11,436	23.5	15%
Overhead	16,178	33.2	21%
Wear Parts	5,236	10.8	7%
Spare Parts	300	0.6	0%
Operating Costs	**75,647**	**155.4**	**100%**

Table 21-43: OPEX Summary - Micronization Plant per saleable product

Description	LOM Total	Micronized NFG		PMG		CSPG	
		Total	Unit Cost	Total	Unit Cost	Total	Unit Cost
	($K)	($K)	($/t Micronized NFG)	($K)	($/t Saleable PMG)	($K)	($/t Saleable CSPG)
Workforce	**24,488**	9,002	57.1	7,319	60.3	8,168	73.7
Electrical Power	**18,009**	6,620	42.0	5,382	44.3	6,007	54.2
Consumables	**11,436**	4,204	26.7	3,418	28.1	3,814	34.4
Overhead	**16,178**	5,947	37.7	4,835	39.8	5,396	48.7
Wear Parts	**5,236**	1,925	12.2	1,565	12.9	1,746	15.8
Spare Parts	**300**	110	0.7	90	0.7	100	0.9
Operating Costs	**75,647**	**27,808**	**176.4**	**22,608**	**186.2**	**25,231**	**227.7**

21.2.2.6.1 Workforce

Each micronization system is operated by two lower-level skilled workers with a total annual cost of $90,500 including bonus and burden. A total of 16 workers is required to operate the two micronization systems 24/7.

21.2.2.6.2 Electrical Power

Each micronization system plus ventilation draws 900 kW for a total power draw of 1,800 kW.

21.2.2.6.3 Consumables

Consumables include lines bulk bags, safety equipment, workshop tools and office supplies.

21.2.2.6.4 Overhead

A standard industry practice overhead is applied to the direct processing costs.

21.2.2.6.5 Wear and Spare Parts

Wear and spare parts include stationary and moving parts that will require periodical replacement. The replacement frequency of wear and spare parts were provided by the supplier and are based on experience with other operations. Wear and spare parts costs include the maintenance workforce.

21.2.2.7 Secondary Transformation

The OPEX for the Secondary Transformation Site include transportation of micronized NFG concentrate to the Secondary Transformation Site, power (electricity and natural gas), water, workforce, reagents, maintenance, laboratory services and other miscellaneous costs. All unit costs in the Purification Plant and CSPG Plant OPEX are on a per metric tonne (t) basis.

Table 21-44 summarizes the OPEX for the Secondary Transformation Site, including the Purification Plant, CSPG Plant and transportation costs. The Purification Plant OPEX is estimated at $107.2M while the CSPG Plant OPEX is estimated at $199.6M. The transportation cost of $21.7M captures the cost of all NFG transported to the Secondary Transformation Site, including NFG that is purified and further coated and spheroidized.

The OPEX for the Purification Plant (Phase 1 only) and CSPG Plant, including costs associated with plant operations over the LOM, are summarized in Table 21-45 and Table 21-46, respectively.

Table 21-44: OPEX Summary – Secondary Transformation Site

Project Area	LOM Total ($K)	Percentage (%)
Transportation	21,694	7%
Purification Plant	107,222	33%
CSPG Plant	199,610	61%
Operating Costs	**328,526**	**100%**

Table 21-45: OPEX Summary – Purification Plant (Phase 1 only)

Project Area	LOM Total ($K)	Unit Cost ($/t PMG)	Percentage (%)
Electrical Power and Gas	5,464	45	5%
Water	1,214	10	1%
Reagents	43,957	362	41%
Workforce	44,079	363	41%
Maintenance	1,579	13	1%
Laboratory	3,886	32	4%
Miscellaneous	7,043	58	7%
Operating Costs	**107,222**	**883**	**100%**

Table 21-46: OPEX Summary – CSPG Plant

Project Area	LOM Total ($K)	Unit Cost ($/t CSPG)	Percentage (%)
Electrical Power and Gas	14,962	135	7%
Water	1,552	14	1%
Reagents	89,774	810	45%
Workforce	53,754	485	27%
Maintenance	20,615	186	10%
Laboratory	6,207	56	3%
Miscellaneous	12,746	115	6%
Operating Costs	**199,610**	**1,801**	**100%**

The following sections detail the OPEX basis for the Secondary Transformation Site.

21.2.2.7.1 Feed Transport

The cost of transporting micronized NFG graphite from Kilbourne Site in Gouverneur to the Secondary Transformation Site is estimated to cost approximately $75/t of micronized NFG concentrate. The material is trucked in loads with a capacity of 18 t.

21.2.2.7.2 Electrical Power and Gas

The total power cost for the Purification Plant and CSPG Plant is divided between electricity and natural gas.

The Purification Plant operates at a continuous electrical load of 732 kW, in addition to a 200-kW load for auxiliary systems, such as heating, ventilation and air conditioning (HVAC), plant lighting, workshops and administrative buildings.

The CSPG Plant has a continuous electrical load of 4,092 kW.

The Purification Plant and CSPG Plant have a natural gas (at standard conditions) consumption of 66 Nm³/h and 343 Nm³/h, respectively.

21.2.2.7.3 Water

The total water consumption for the Purification Plant and CSPG Plant is 8 m³/h and 17 m³/h, respectively.

21.2.2.7.4 Reagents

The total reagent costs for the Purification Plant and CSPG Plant were based on the unit costs and dosage rates as summarized in Table 21-47 and Table 21-48, respectively.

Table 21-47: Reagent cost basis for the Purification Plant

Reagent	Unit Cost ($/t reagent)	Dosage Rate ($t/t_{graphite\ feed}$ dry feed)
HF Acid (40 wt.%)	1,140	0.23
HCl Acid (32 wt.%)	249	0.31

Table 21-48: Reagent cost basis for the CSPG Plant

Reagent	Unit Cost ($/t reagent)	Dosage Rate ($t/t_{graphite\ feed}$ dry feed)
HF Acid (40 wt.%)	1,140	0.23
HCl Acid (32 wt.%)	249	0.31
Nitrogen	0.53 ($/m^3)	0.11 (m^3/t dry feed)
Pitch Tar	2,205	0.10

21.2.2.7.5 Workforce

The workforce at the Purification Plant and CSPG Plant are organized into four operational shifts to ensure continuous production of the plant. Plant operators and technical staff rotate through these shifts, while administration, marketing and management personnel work standard day shifts only. A total workforce of 40 employees is required for the Purification Plant to support this setup, while 88 employees are required for the CSPG Plant.

Workforce costs for the Purification Plant and CSPG Plant include base salaries, bonuses and benefits for all employees, which are outlined in Table 21-49.

Table 21-49: Annual compensation per position for the Purification Plant and CSPG Plant

Description	Base Salary ($/y)	Benefits ($/y)	Bonus ($/y)	Total Cost ($/y)
Highly skilled (Upper)	152,000	45,000	8,000	205,000
Skilled (Upper)	127,000	38,000	7,000	172,000
Skilled (Lower)	90,000	27,000	5,000	122,000
Trained (Upper)	81,000	24,000	2,000	107,000
Trained (Lower)	71,000	21,000	2,000	94,000
Minimum Wage	51,000	15,000	1,000	67,000

21.2.2.7.6 Maintenance

The projected maintenance cost for the Purification Plant and CSPG Plant is 7% of the total capital equipment.

The maintenance costs cover scheduled maintenance for critical equipment in all process areas, as well as the replacement of consumables and parts subject to wear-and-tear, such as pumps, bearings, etc.

21.2.2.7.7 Laboratory

Laboratory services are outsourced, with external analysis conducted on samples collected from the feed to the Purification Plant, as well as the leach solution, dried PMG product and wastewater discharge. Similar samples are taken for the CSPG Plant with the addition of sampling the pitch tar, SPG and CSPG product. Table 21-50 and Table 21-51 outlines the sample frequency as well as the corresponding cost per sample for the Purification Plant and CSPG Plant, respectively.

Table 21-50: Laboratory cost inputs for the Purification Plant OPEX

Sample Type	Frequency	Cost per Sample ($)
Feed	Once every 8 hours	34
Leach Solution	Once every 8 hours	20
PMG	Once every hour	34
Wastewater	Once every 8 hours	20

Table 21-51: Laboratory cost inputs for the CSPG Plant OPEX

Sample Type	Frequency	Cost per Sample ($)
Feed	Twice every 8 hours	34
Leach Solution	Twice every 8 hours	20
PMG	Twice every 8 hours	34
Wastewater	Twice every 8 hours	20
SPG	Twice every 8 hours	34
Pitch	Twice every 8 hours	34
CSPG	Twice every hour	34

21.2.2.7.8 Miscellaneous

Miscellaneous costs for the Purification Plant and CSPG Plant include items such as safety equipment (including surveillance systems), vehicles and their maintenance, workshop tools and consumables, office supplies, IT equipment and general overhead expenses. The miscellaneous costs are estimated to be the standard industry percentage of the total OPEX.

22. Economic Analysis

Item 22 of Form 43-101F1 permits producing issuers to exclude the required information under Item 22 for technical reports on properties currently in production, provided there is no material expansion of those operations. As no material expansion is planned for ESM's Zinc Operations, the information required under Item 22 related to ESM's Zinc Operations has been excluded from this report.

22.1 Methodology Used

A cash flow model was prepared to evaluate the Kilbourne Graphite Project on a real basis. This model was prepared on an annual basis. The model is based on the mine plan as outlined in Item 16. Capital and operating costs were developed in Item 21 and the build-ups and associated accuracy, and contingency can be found in those Items.

This Item presents the main assumptions used in the cash flow model and the resulting indicative economics. 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.

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.

All results, and technical and cost information are presented in this Item on a 100% basis reflective of Titan Mining's ownership, unless otherwise noted.

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

22.2 Financial Model Parameters

All costs incurred prior to the model start date are considered sunk costs. The potential impact of these costs on the economics of the Project is not evaluated. This includes exploration expenditures and working capital as these items are assumed to have a zero balance at model start.

The model continues one year beyond the mine life to incorporate closure costs in the cash flow analysis.

The discount rate selected is 7%. A discount of 5% was used previously for ESM, which is in operation. The increase in the discount rate to 7% reflects the history of the ESM operation in conjunction with the Kilbourne Graphite Project. Discounting was conducted annually on an end of year period. Discounting commenced in the second year.

22.3 Pricing

Modeled prices are based on prices developed in Item 19 of this report, including, Std flake graphite concentrate, Std Purity Micronized Flake Grades (95.0% LOI MIN) (NFG), High Purity Micronized Flake Grade (99.9% LOI MIN) (PFG), and Coated Spherical Purified Graphite (CSPG). Any other metals present are not considered in the model. All pricing is considered in Q3 2025 dollars. No inflation or escalation are considered in the model.

22.4 Royalties

The Kilbourne Graphite Project is not subject to any royalties.

22.5 Taxes

The Kilbourne Graphite Project is subject to a combined state and federal income tax rate of 27.5%. Taxable income is determined based on gross revenue, minus allowable deductions, including royalties, offsite costs, operating costs, tax depreciation, depletion, and net operating losses incurred, among other specific state and federal tax adjustments. Project capital costs are depreciated using the Modified Accelerated Cost Recovery System (MACRS) applicable to the specific categories of mine development and infrastructure. Project operating costs for Purified and CSPG materials are eligible for the U.S. Internal Revenue Code Section 45X tax credit for critical mineral components used in renewable power generation and energy storage. The critical minerals credit is equal to 10% of the cost incurred by the taxpayer to produce the critical mineral.

Property taxes are included in operating costs within G&A costs. Estimates are based on the actual costs incurred by ESM.

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;
- 10% Section 45X tax credit on production costs for Purified and CSPG materials.

22.6 Working Capital

The assumptions for working capital in this analysis are summarized in Table 22-1. The change in working capital over the life of the Project is zero.

Table 22-1: Working capital assumptions

Working Capital	Days
Days per Year	365
Days in Accounts Receivable	15
Days of Cost of Goods Sold in Inventory	10
Days in Accounts Payable	60

22.7 Economic Analysis

The Kilbourne Graphite Project economics for this report reflects only the Graphite Project. Table 22-2 summarizes the economic results. The annual and cumulative cash flows, presented on an annual basis, are illustrated in Figure 22-1 and shown in Table 22-3. The post-tax NPV is $513.2M, the post-tax IRR is 37.0%, and the post-tax pay-back period is 2.69 years.

All monetary amounts in this Item are expressed in United States dollars, unless stated otherwise.

Table 22-2: Summary of the economic analysis results

Parameters	Unit	Value
Physicals		
Mine Life	year	12.8
Total Material Mined	ton	62,769,000
Total Waste Mined	ton	42,818,000
Total ROM Mined	ton	19,951,000
ROM Head Grade	% Cg	2.84
Mill Recovery	%	89.7
Total Metal Tonnage Recovered	ton	509,670
	tonne	462,364
Concentrate Grade	%	95.0
Total Concentrate Produced	tonne	486,699
Operating Costs		
Mining	$/ton milled	9.21
Concentrate Processing	$/ton milled	12.01
G&A	$/ton milled	2.36
Secondary Processing	$/ton milled	20.26
Tailings Relocation	$/ton milled	0.56
Total Operating	$/ton milled	44.41
	$/tonne milled	48.95
Capital Costs		
Initial Capital	$M	155.8
Expansion Capital	$M	175.9
Sustaining Capital	$M	100.0
Discount Rate		
Discount Rate	%	7.0
Financials		
Pre-tax Cash Flow	$M	1,187.7
Pre-tax NPV	$M	580.6
Pre-tax IRR	%	38.9
Pre-tax Payback Period	year	2.66
Taxes	$M	134.1
Post-tax Cash Flow	$M	1,053.6
Post-tax NPV	$M	513.2
Post-tax IRR	%	37.0
Post-tax Payback Period	year	2.69

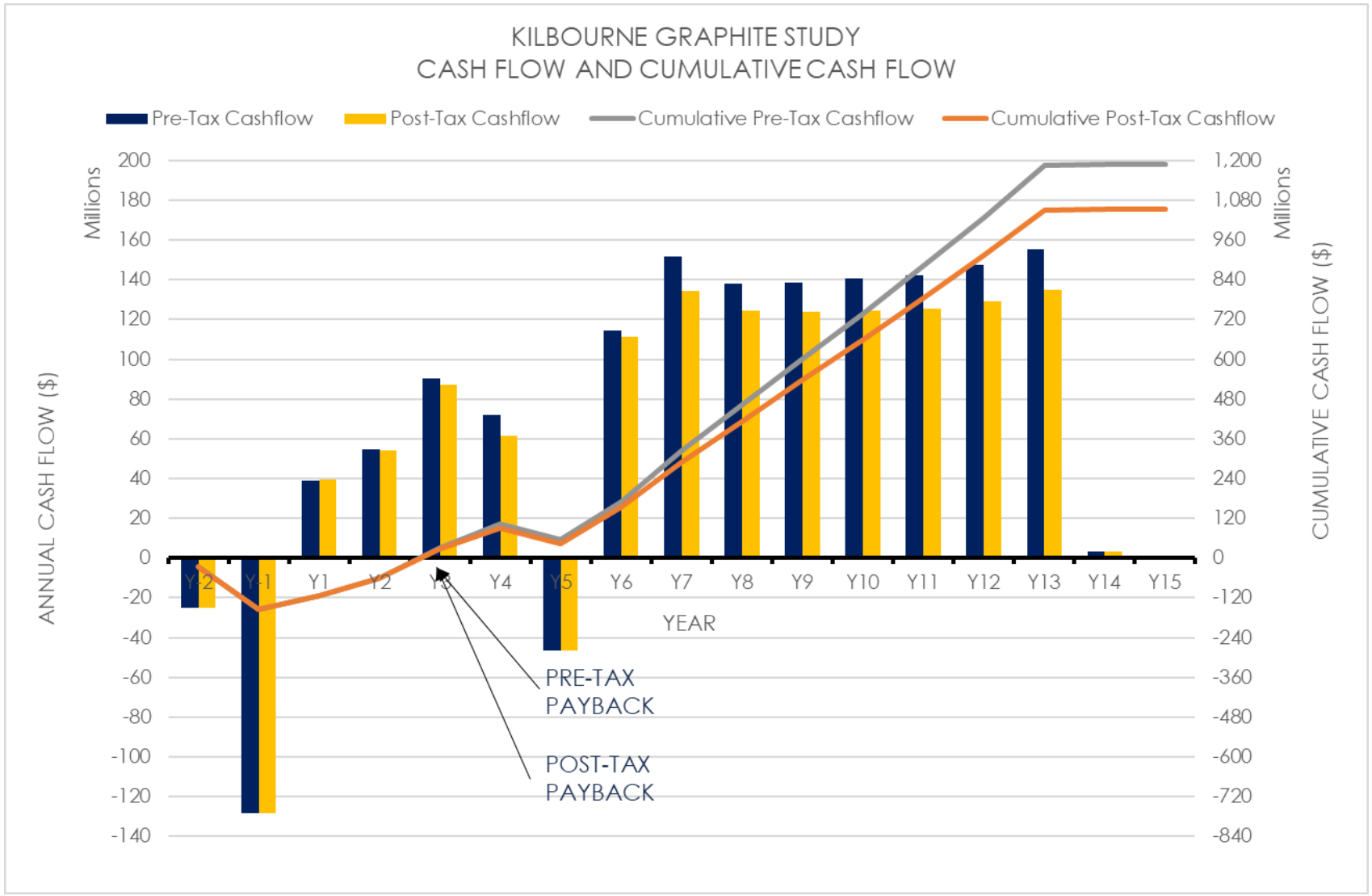


Figure 22-1: Annual and cumulative cash flows

Table 22-3: Annual and cumulative cash flow

Description	LOM Total	Unit	Construction		Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
			Y-2	Y-1														
Mill Feed Tons Processed	19,951	kton	-	-	1,226	1,400	1,398	1,355	1,530	1,695	1,864	1,850	1,844	1,617	1,444	1,328	1,400	-
Concentrate Tonnage	486.7	kt	-	-	22.2	27.7	38.5	39.4	42.2	44.5	44.5	38.5	38.7	38.6	36.7	36.7	38.5	-
Graphite Products:																		
Micronized NFG	157.6	kt	-	-	14.6	20.3	23.3	23.3	23.3	13.1	11.1	5.4	5.5	5.4	3.6	3.6	5.3	-
PMG	121.4	kt	-	-	5.5	5.5	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	-
CSPG	110.8	kt	-	-	-	-	-	-	-	8.9	13.4	14.8	14.8	14.8	14.8	14.8	14.8	-
Remaining Concentrate	26.0	kt	-	-	1.2	0.8	3.5	4.4	7.2	6.8	2.0	-	-	-	-	-	-	-
Revenue:																		
Revenue[1]	**2,505**	**$M**	**-**	**-**	**85.1**	**106.1**	**145.4**	**146.8**	**151.2**	**211.2**	**247.4**	**237.5**	**238.0**	**237.6**	**230.8**	**230.8**	**237.2**	**-**
Operating Cost Estimate:																		
Kilbourne Site	481.8	$M	-	-	27.3	30.2	29.4	29.9	35.3	40.9	46.8	48.2	46.4	43.9	37.5	34.0	32.2	-
Secondary Transformation Site[2]	404.2	$M	-	-	8.9	9.9	15.6	15.6	15.6	32.8	42.2	44.0	44.1	44.0	43.7	43.7	44.0	-
Total Operating Cost	**885.9**	**$M**	**-**	**-**	**36.1**	**40.0**	**45.0**	**45.5**	**50.9**	**73.7**	**88.9**	**92.2**	**90.4**	**87.9**	**81.2**	**77.7**	**76.3**	**-**
EBITDA	**1,619.4**	**$M**	**-**	**-**	**49.0**	**66.1**	**100.4**	**101.3**	**100.3**	**137.6**	**158.5**	**145.3**	**147.6**	**149.7**	**149.5**	**153.1**	**161.0**	**-**
Capital Cost Estimate:																		
Initial Capital Costs	116.6	$M	17.5	97.5	1.7													
Expansion Capital Costs	118.4	$M				5.2	0.3	10.7	91.2	11.0	-	-	-	-	-	-	-	-
Sustaining Capital Costs	103.9	$M	-	-	9.0	4.9	8.7	12.7	10.9	7.3	7.6	7.9	8.6	8.8	6.8	5.4	5.3	-
Indirect Capital Costs	49.9	$M	4.0	11.5	0.3	-	-	3.5	27.1	3.4	-	-	-	-	-	-	-	-
Contingency	47.0	$M	3.5	19.5	0.3	1.0	0.1	2.1	18.2	2.2	-	-	-	-	-	-	-	-
Closure	-4.1	$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-4.1
Total Capital Costs	**431.7**	**$M**	**25.0**	**128.5**	**11.3**	**11.1**	**9.0**	**29.2**	**147.4**	**24.0**	**7.6**	**7.9**	**8.6**	**8.8**	**6.8**	**5.4**	**5.3**	**-4.1**
Change in Working Capital	-	$M	-	-	-1.4	0.3	0.9	-	-0.6	-0.7	-0.6	-0.9	0.3	0.3	0.6	0.5	0.5	0.7
Taxes	134.1	$M	-	-	-0.3	0.7	3.1	10.4	-0.1	2.9	17.1	13.8	14.9	16.3	16.5	18.3	20.5	-
Cash Flow Results:																		
Pre-tax Cash Flow	**1,187.7**	**$M**	**-25.0**	**-128.5**	**39.1**	**54.7**	**90.5**	**72.2**	**-46.5**	**114.3**	**151.5**	**138.3**	**138.7**	**140.6**	**142.2**	**147.3**	**155.2**	**3.4**
Cumulative Pre-tax Cash Flow	-	$M	-25.0	-153.5	-114.3	-59.7	30.8	102.9	56.4	170.7	322.1	460.4	599.1	739.7	881.8	1,029.1	1,184.3	1,187.7
Post-tax Cash Flow	**1,053.6**	**$M**	**-25.0**	**-128.5**	**39.4**	**54.0**	**87.4**	**61.7**	**-46.4**	**111.3**	**134.4**	**124.5**	**123.7**	**124.3**	**125.6**	**129.0**	**134.7**	**3.4**
Cumulative Post-tax Cash Flow	-	$M	-25.0	-153.5	-114.1	-60.1	27.3	89.0	42.6	154.0	288.4	412.9	536.6	660.9	786.5	915.5	1,050.2	1,053.6

Table Notes:
[1] Price assumptions: NFG concentrate $1,575/tonne, Micronized NFG $3,770/tonne, PMG $5,185/tonne, CSPG $11,193/tonne.
[2] Secondary Transformation OPEX includes operating expenses for micronization associated with the PMG and CSPG material.

22.8 Sensitivities

A sensitivity analysis was completed to determine the relative sensitivity of the Project's NPV to a number of key parameters (Graphite Price, OPEX and CAPEX). This is accomplished by flexing each parameter upwards and downwards by 10% increments to a maximum of 20%. Within the constraints of this analysis, the Project appears to be most sensitive to graphite pricing. The results of the pre-tax and post-tax sensitivity are displayed in Table 22-4.

Table 22-4: Pre-tax and post-tax sensitivity analysis

Description	Unit	Pre-tax					Post-tax				
		-20%	-10%	Base	+10%	+20%	-20%	-10%	Base	+10%	+20%
NPV @ 7.0% Discount Rate											
Graphite Price, Blended ASP	$M	300	440	581	721	861	286	402	513	624	735
Operating Cost	$M	482	531	581	630	679	446	480	513	547	580
Total Capital Cost	$M	515	548	581	613	646	460	487	513	540	566

Table 22-5 and Figure 22-2 provide the post-tax NPV and IRR sensitivity analysis, showing the impact of varying graphite prices on the study economics at different discount rates.

Table 22-5: Post-tax NPV sensitivity analysis

Graphite Price	Unit	-20%	-10%	Base	+10%	+20%
Graphite Price, Blended ASP	$/tonne	4,118	4,633	5,148	5,662	6,177
NPV @ 5.0% Discount Rate	$M	363	497	628	758	887
NPV @ 7.0% Discount Rate	$M	286	402	513	624	735
NPV @ 8.0% Discount Rate	$M	253	361	464	567	671
NPV @ 10.0% Discount Rate	$M	198	291	381	470	559
IRR	%	25.2%	31.5%	37.0%	42.2%	47.2%
Payback Period	year	4.7	3.2	2.7	2.4	2.2

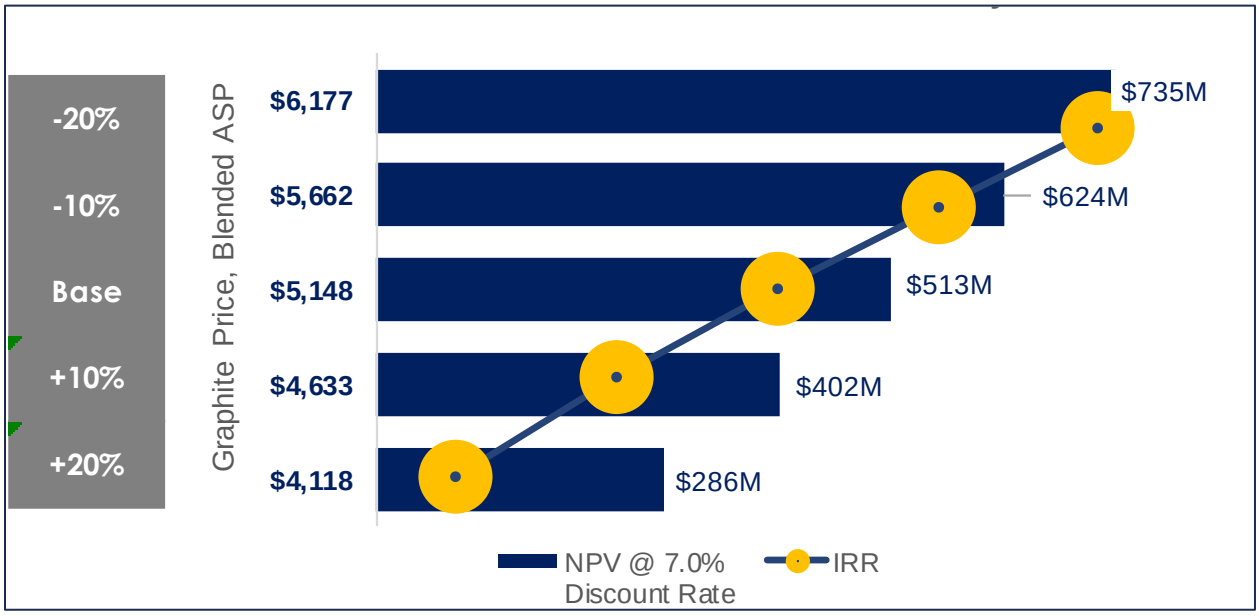


Figure 22-2: Post-tax NPV and IRR sensitivity analysis

23. Adjacent Properties

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

24. Other Relevant Data and Information

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

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 supporting the LOM plan. The mine is fully developed with shaft access and mobile equipment on-site.

ESM is comprised of multiple deposits in and around Fowler, NY. There are ten zones currently considered as viable economic targets. Historic mining at these locations has yielded a robust geological understanding of each, with supporting mapping, sampling, and drilling data. The ten zones were defined and modeled by ESM geologists. Each one is comprised of multiple veins designating variably oriented and spatially-distinct mineralized envelopes, 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 2024.1.3 and Edge software. 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. Mineral Resources for the underground areas have been compiled from separate block models including the American, Cal Marble, Fowler, Mahler, Mud Pond, N2D, New Fold, Northeast Fowler, Silvia Lake, and Turnpike 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.

Tons included in the mine plan at the ESM deposit will be 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. Open pit mining is included as part of the overall mine plan. Current mine production is targeted at 2,275 tons per day. The underground mine plan ramps up through 2028 to support annual production of 80 million payable pounds, with mill throughput at up to 2,800 ton/d. The mine life is currently projected at 6 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. 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, for 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 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: Zinc Operation 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 via an engineering trade-off study 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.

25.1.2 Opportunities

There are several opportunities to improve the Project through a combination of resource expansion, productivity enhancements and the use of new technology to lower mine operating costs.

Table 25-2: Zinc Operation identified 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, cash flows and increased Project NPV.
Mine Plan Expansion	Resource zones added may add significant mineable tons to the LOM plan.	Increased mine life and increased Project NPV.
By-Product	The plant feed contains 20-25 g/t of germanium (Ge), which is naturally enriched in some plant streams. It may be possible to upgrade the Ge in one or more process streams to concentrations that are attractive for third party processors. Metallurgical and mineralogical work has commenced to develop an understanding of the Ge deportment and identify suitable upgrading routes.	Produce a Ge pre-concentrate that can be sold to a refiner for by-product credits.
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.

25.2 Graphite

The Kilbourne Graphite Study shows potential for a viable graphite operation. The discovery of graphite in Unit 2 of the Upper Marble (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 spread out across Titan's nearby holdings. Although there has been no previous graphite production at the Kilbourne Site, historical exploration data shows the presence of other mineral occurrences, including a recorded iron and sulfur prospect dating back to 1917.

Overall, Kilbourne's graphite mineralization shows considerable exploration potential. The mapped surface extension of UM2 continues to the south, and east. Exploration drilling targeting the eastern extension of Kilbourne has intercepted graphite mineralization 4,278 ft from the easternmost drillholes of the 2024 exploration program. The graphite grades, and nature of the graphite mineralization are consistent with those reported at Kilbourne. This program is ongoing, with 1,738 ft completed of a planned 4,500 ft.

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 Kilbourne Site, resulting in increased mine life and significantly enhanced economics.

25.2.1 Geology

Graphite mineralization at Kilbourne, consistent with other deposits in the Grenville Province, is believed to result from metamorphic processes acting on thick accumulations of 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 the Kilbourne Site 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 Graphite Study was prepared using 12 geological domains and 45 surface drillholes with one surface channel, all totaling 29,699 ft. The Mineral Resource was completed by ordinary kriging and using a cut-off of 1.5% Cg. The pit-constrained Inferred Mineral Resources total 22,423,000 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.

25.2.2 Open Pit

The Graphite open pit study was completed using industry-standard methods, including conventional drilling, blasting, and loading with excavators and haul trucks at the scoping level.

The open pit design and planning is based on Inferred Mineral Resources, which are geologically speculative and cannot be classified as Mineral Reserves. There is no certainty the Project will be realized.

The estimate of in-pit mineable resources is based on a 3D block model, regularized for mining selectivity and equipment compatibility. At 1.5% Cg cut-off grade, mining dilution and recovery are incorporated into the block model, with 5.3% dilution at 0.26% Cg and 86% mining recovery assumed.

The pit limit analysis for the Graphite Study was carried out in Deswik using the pseudoflow algorithm to identify the most economic open pit shells based on Inferred Resources, geological block models, and a range of operational and economic parameters, with no property or boundary constraints applied. The recommended ultimate pit limits are RF0.95 and RF0.975, which balance value capture and input realism, targeting over 450,000 tons of concentrate.

Pit wall configuration basis follows best practices for safe equipment operation. Hydrological impacts are manageable, with pit drainage into existing workings and sufficient dewatering capacity.

The LOM production schedule for the open pit was developed using Hexagon's MPSO to determine the mining sequence and stockpiling strategy that maximizes the Project's present value while meeting processing plant targets at scoping level. Key constraints include sequencing, bench rates, capacities, and a staged approach to removing existing tailings and constructing perimeter dikes. The plan ramps up mill feed from 1.23 M tons in Year 1 to 1.7 M tons from Year 4 onwards, with concentrate production increasing from 22,500 tonnes to a steady 40,000 tonnes annually. Over the 13-year mine life, approximately 19.95 M tons of potential mill feed averaging 2.84% Cg, 30.82 M tons of waste rock, 6.44 M tons of overburden, and 5.56 M tons of tailings will be mined, totaling about 62.8 M tons of material movement, with an overall strip ratio of 2.1.

The major equipment fleet includes 45-ton haul trucks, large excavators, drills, loaders, dozers, and graders, with workforce peaking at 62 employees.

25.2.3 Infrastructure

The Graphite Study involves two primary sites. The first is the **Kilbourne Site**, located adjacent to the existing ESM complex, which includes the open pit operations, supporting site infrastructure, tailings management facilities, and the Concentrate and Micronization plants. The second is the **Secondary Transformation Site**, a separate location housing the Purification Plant and CSPG Plant, where micronized NFG will be further processed into market-ready material.

Infrastructure required to support the mining operations, Concentrate and Micronization plants was developed to minimize surface disturbance.

The infrastructure concepts for the Graphite Study are preliminary and intended to demonstrate the Project's viability at scoping level. Existing facilities at ESM (offices, dry facility, training, first aid, potable water, waste management, explosives magazine, and parking) are expected to be leveraged, which could reduce capital requirements and facilitate early development.

Site access is expected to be robust, with new and upgraded roads proposed to connect the Kilbourne Site facilities, shared ESM facilities, and off-site infrastructure. Separation of on-highway and off-highway traffic is considered to enhance safety and operational efficiency. It is anticipated that diesel fuel dispensing and mobile equipment maintenance are to be supported by dedicated, scalable, facilities.

Waste rock and overburden stockpiles are planned with environmental protection in mind, including perimeter ditches and water collection ponds. Water quality monitoring is proposed, and current data suggest no need for formal treatment systems due to non-acid generating nature of the material.

Power supply to the ESM Site is expected to be sufficient, with a new transformer and substation upgrades proposed to meet anticipated demand for mining and processing. Utilities, including process water, potable water, and wastewater, are expected to be managed through a combination of existing infrastructure and new installations, with adaptive strategies for water sourcing and discharge.

Water management is a key consideration, with a closed-loop system proposed to minimize freshwater consumption and control discharges. The preliminary water balance model incorporates climate scenarios, operational phases, and regulatory requirements, supporting adaptive management and compliance.

Tailings management is expected to involve relocation of some existing tailings, construction of containment dikes, expansion and raising of tailings facilities, and eventual backfilling of Kilbourne and historical Arnold pits. The staged approach is intended to optimize storage, minimize haulage, and address geotechnical and environmental risks. Stability analyses and consequence classifications have been performed at a high level, with ongoing monitoring and further site-studies recommended.

Processing infrastructure (Concentrate, Micronization, Purification, and CSPG plants) is anticipated to include modern facilities, efficient workflows, and integration of essential utilities. Off-site infrastructure is expected to leverage local vendors and service providers, with further review of transmission lines and industrial estate options recommended for future phases.

25.2.4 Mineral Processing and Metallurgical Testing

None withstanding the limited systematic process development work, the two SGS programs demonstrated that the Kilbourne graphite mineralization can be upgraded to a saleable concentrate with relative ease using traditional mineral processing technologies. The selected reagent regime consisting of diesel and MIBC has been proven as simple and effective.

Given that the flake size distribution is relatively fine, only limited flake size preservation strategies were deployed to keep the flowsheet as simple as possible. The proposed flowsheet includes a standard roughing stage followed by a single stream cleaning circuit. The robustness of this flowsheet was demonstrated using a larger composite sample and four variability samples. The final concentrates yielded grades over 97% TC for all five samples. Despite the relatively low head grade, open circuit graphite recoveries were very high between 82.3% and 92.4%. These numbers are expected to increase further during closed circuit operation.

It is expected that a systematic process development program will produce further improvements in terms of concentrate grade and/or flake size distribution. Also, the fact the Kilbourne graphite mineralization produced concentrate grades between 97.6% TC and 99.3% TC derisks the ramp up of the operation since even sub-optimal operating conditions should still produce a concentrate that meets the minimum grade requirement of 95% TC.

25.2.5 Concentrate Plant

The Concentrate Plant design is based on the flowsheet and conditions developed in the laboratory scale programs. Equipment selection and sizing is based on specific Kilbourne process design data and standard industry practices. The Concentrate Plant will be co-located with the mine.

The Concentrate Plant will treat up to 1,864 kilotons per year at an average feed grade of 2.84% Cg. It will produce up to 44.5 kilotonnes of graphite concentrate annually with a 90% graphite recovery.

The process design includes a conservative cleaning circuit design, which provides additional regrind mill and cleaner flotation capacity to ensure that the final concentrate product meets and exceeds the minimum product specifications. The use of traditional mineral processing technologies and equipment also reduces the technological risk.

25.2.6 Micronization Plant

The Micronization Plant will process the dried flake graphite concentrate into micronized graphite flake with product sizes of P_{90} = 45 microns and P_{90} = 15 microns. Micronization will be performed using an air swept classifier system. The equipment is provided turnkey as a fully integrated system. The battery limits of the system are the feed hopper receiving the flake concentrate and micronized product chutes. Given the turnkey and integrated nature of the equipment, the process risk is minimized, especially since comprehensive tests will be conducted by the vendor on commercial scale equipment using Kilbourne graphite concentrate.

25.2.7 Secondary Transformation Site

Titan Mining is developing a fully integrated graphite processing facility in USA, consisting of a Purification Plant and CSPG Plant. The facility will produce value-add graphite products, entirely sourced and processed domestically, for applications in the defense and energy markets. The location of the Purification Plant and CSPG Plant has not been finalized. For the Graphite Study, it is assumed that the Purification Plant and CSPG Plant will be located in an established prime chemical industrial estate in New York that has existing bulk supply services, access road and connection for gas, water, electricity, effluent and sewage disposal and treatment, stormwater management and waste removal. The Company is already evaluating a few proposed sites in this context.

Micronized NFG concentrate from the Kilbourne Site will be processed in the Purification Plant and CSPG Plant, which aim to produce PMG and CSPG, respectively.

The Purification Plant will process 10,670 t/y of micronized NFG concentrate (≥96 wt.% FC) to produce approximately 10,042 t/y of PMG (≥99.90 wt.% FC).

The CSPG Plant is designed to process 21,340 t/y of micronized NFG concentrate to produce approximately 14,761 t/y of CSPG (≥ 99.95 wt.%).

Wastewater treatment will be addressed in the next phase of work, pending final site selection.

25.2.8 Financials

The economic analysis demonstrates that the Graphite Study at this stage is financially robust under the assumptions used in this Technical Report. Over a mine life of approximately 13 years, the operation generates total post-tax cash flow of $1.05B, an NPV of $513.2M, a 37.0% IRR, and a 2.69-year payback period.

The total capital cost for the Graphite Study is $431.7M, which includes $156M in initial capital over 2 years, $176M in expansion capital, and $100M in sustaining capital.

Operating costs over the life of mine total approximately $886M, averaging $68M per year.

On a saleable-product basis for the four products considered in this study, operating costs are estimated at $990/t for flake concentrate, $1,197/t for micronized NFG, $2,233/t for PMG, and $3,612/t for CSPG.

Average EBITDA over the mine life is approximately $125M per year.

Overall, the Graphite Study indicates positive economic potential at the current level of study.

25.2.9 Risks

Table 25-3: Graphite Study risks

Area	Risk	Explanation / Potential Impact	Possible Risk Mitigation
Geology	Geological continuity	Mineral Resources are at an Inferred Mineral Resource classification using limited drill information. Tons and grades may not be achieved.	Additional exploration in the form of diamond drilling, geological and structural mapping and bulk sample will provide geological confidence to adjust the Mineral Resource classification.
Mining	Reliance on Inferred Resources	Geological uncertainty; Inferred resources are speculative and may not convert to reserves, risking project viability.	Conduct additional drilling and resource definition to upgrade resource confidence.
	Geotechnical assumptions	Pit slope, bench design, and stability are based on preliminary studies; unexpected ground conditions could cause failures or require redesign.	Undertake detailed geotechnical investigations and monitoring; adjust designs as new data becomes available.
	Structural geology model	Unknown structural condition around the open pit.	Develop a 3D geological fault model, interpretating orientations and inclinations of faulting encountered by diamond drilling at the Kilbourne Site.
	Existing tailings removal	Tailings overlying the pit must be removed before mining; delays or technical challenges could impact the schedule and increase costs.	Develop a detailed tailings removal plan; stage removal and dike construction; monitor progress and adjust as needed.
Infrastructure	Power supply and utility reliability	The adequacy of off-site transmission lines and substation upgrades is assumed but not confirmed. Power interruptions or insufficient capacity could disrupt operations.	Validate power supply and transmission capacity with utility providers; plan for backup power solutions and phased upgrades.
	No geotechnical site investigation	Geotechnical investigation in the proposals for the TMF, waste dumps, and plant infrastructure have not been completed.	Complete a geotechnical site investigation at all critical sites.
	Hydrological uncertainty	Dewatering and water management rely on current assumptions; unexpected inflows could disrupt operations or increase costs.	Implement robust water monitoring and contingency plans; reassess hydrological models as mining progresses.

Area	Risk	Explanation / Potential Impact	Possible Risk Mitigation
	Potential dam breach and inundation	The impact of a dam breach is unknown.	Analyses must be conducted for the extended TMF and the containment dikes around the historical Arnold Pit to support the consequence classification process.
	The tailings and waste rock are assumed to be NPAG	If the material is PAG, liners will be required to be installed. Current testing shows net neutralized material.	Conduct ARD and metal leach tests on tailings and waste rock to determined geochemistry.
	Embankment material is available and suitable	Embankments including filter and transition zones can be constructed using overburden material excavated from the Kilbourne Pit footprint.	Complete detailed staged designs of the embankments tied to the mine plan.
	Dike construction around the pit	Building perimeter dikes is required to contain remaining tailings; engineering or construction failures could lead to environmental incidents or operational delays.	Develop and follow a proper execution plan for dike construction: ▪ Conduct thorough geotechnical and engineering studies; ▪ Use proven construction methods and qualified contractors; ▪ Sequence dike construction with mining phases; ▪ Monitor dike integrity and stability throughout operations; ▪ Establish contingency plans for emergency repairs.
	Flood and Seepage Analysis	Emergency spillway was not included in the design at this stage for the extended TMF, due to limited available information.	Implement hydrology model, flood studies, and seepage analysis. Complete the emergency spillway design and seepage control strategies (any cut-off, clay core)
	Preliminary nature of infrastructure concepts	All infrastructure plans are conceptual; no detailed engineering or permitting has been completed. This may result in changes, delays, or increased costs during future project stages.	Advance to engineering and permitting early in the next phase; maintain flexibility in planning to accommodate changes.
	Community and public access	Closure requires highway superintendent and town board approvals, plus consent from adjacent landowners. Lack of consent may require alternate access, affecting schedule and community relations.	Engage early with officials and landowners; develop alternate access plans if consent is not granted.

Area	Risk	Explanation / Potential Impact	Possible Risk Mitigation
	Property boundary and land access	Some infrastructure (roads, stockpiles, plants, tailings facilities) including mine may extend beyond current property boundaries, requiring land acquisition or access agreements. Failure to secure land could delay or restrict development.	Review and update property boundaries; engage with landowners and authorities early to secure necessary land access and permits.
Concentrate Plant and Micronization	Metallurgical sample quality	Samples evaluated in metallurgical test programs were not fully representative of the average potential mill feed.	Extensive variability program will be conducted during feasibility study.
	Concentrate specs	The projected concentrate grade and/or graphite recovery may not be achieved.	Additional process development work will be conducted during the feasibility study and through the demonstration facility. A large set of variability samples will also confirm the robustness of the process and conditions.
	Labor force	The operation of a graphite concentrator requires a special skillset (only one graphite mine is in operation in Canada).	A 20 t/h demonstration plant will be commissioned in Q4 2025, which will serve as a training facility for operators.
	Equipment selection	The process technology employed in the Concentrate Plant is very mature. However, knowledge of scaleup for full-scale plants is limited.	The different unit operations in the demo plant will undergo DOE optimization to quantify key factors affecting the performance to facilitate proper sizing for the commercial plant.
	Ramp up period	The ramp up period for the Concentrate Plant and Micronization Plant may be longer than forecasted for financial modeling.	Risk mitigation strategies to be implemented include technical mitigation (e.g. root cause analysis), operational flexibility, and training.
	Concentrate acceptance	Graphite concentrates have distinct signature plots. Potential customers need to qualify the graphite concentrate or micronized product before committing to an off-take agreement.	The demonstration plant will be used to produce concentrate for qualification. A micronization module is planned for Q1 2026, a pilot facility for Purification and CSPG is also planned in 2026.

Area	Risk	Explanation / Potential Impact	Possible Risk Mitigation
Secondary Transformation Site	Bulk infrastructure requirement changes	Changes to bulk services and infrastructure may result following final site selection. All infrastructure is currently assumed to be developed and available within the prime chemical industrial estate, for which the Purification Plant and CSPG Plant will be located. Any deviations from this assumption may lead to increased CAPEX.	Conduct a site location study in the next phase to finalize bulk services and infrastructure.
	ESIA	Pending final site selection, an ESIA, which is to be completed in the next phase, may result in schedule delays and increased costs.	Initiate ESIA following site selection.
	Wastewater treatment plant	No allowance has been made for the wastewater treatment plant, as final site confirmation is pending; its inclusion in the next phase may result in increased CAPEX.	Prioritize a site location study and allow for the wastewater treatment plant in the next phase.
	Metallurgical testwork	The selected process equipment may not reproduce the laboratory results on a commercial scale.	Conduct additional metallurgical testwork and a pilot plant campaign to support both scale-up and vendor confirmatory testing for process equipment.
	Deviations in cost inputs	Fluctuations in OPEX inputs, such as reagent costs, electricity and natural gas, may result in increased costs.	Conduct price forecasting and investigate securing long-term supply contracts in the next phase.
	Market competition	Increased competition in the North American graphite market is anticipated to influence the market share.	Perform ongoing market analyses to anticipate competitor trends and market changes.

25.2.10 Opportunities

Table 25-4: Graphite Study opportunities

Area	Explanation	Possible Benefit
Geology	Drill additional exploration holes east of the existing resource. Surface mapping, historic and recent exploration drilling demonstrate additional mineral potential east of the initial resource.	Increases Mineral Resource and enhances operational flexibility.
Mining	Phased mining and flexible scheduling allow adjustment to market and operational changes and defer major activities like tailings removal and dike construction.	Optimizes cash flow, reduces upfront costs, and increases adaptability to changing conditions.
	Lack of rock mass characteristic results in conservative pit slope parameters.	Future DDH's should be systematically logged for geotechnical parameters to obtain factual rock mass characterization data (Jn, Jr, Ja) and rock hardness estimate Perform Televiewer inspections on select existing DDH's to capture rock mass fabric (jointing) trends for HW and FW lithologies.
	Application of proven open pit mining methods and equipment.	Enhances reliability, safety, and ease of workforce training.
Infrastructure	Consider the addition of a Battery Energy Storage System (BESS) to the on-site power distribution system.	There are many benefits to a BESS including: reduced energy costs, peak shaving to reduce demand charges, and backup power in the event of an outage to reduce the impact to process plant operation.
	Use of existing infrastructure (e.g. haul roads) from previous operations.	Reduces capital expenditure, accelerates start-up, improves efficiency, and minimizes initial site preparation time.
	Complete a site-specific geotechnical site investigation (or use any historical data available) for the extend TMF and containment dikes around Arnold historical pit.	Optimizes the required material for the dike construction and potentially reduces capital costs. Optimize the deposition sequences, and the method of dike raise (i.e. using centerline raise instead of downstream raise upon receipt of geotechnical site investigations)

Area	Explanation	Possible Benefit
Concentrate Plant and Micronization Plant	Process simplification.	The demonstration plant may reveal that the cleaning circuit with conservative regrind and cleaner flotation capacity is not required, thereby reducing the capital cost of the Project.
	Overstated comminution energy requirements.	To-date, only one grind test was completed, which places the Kilbourne mineralization into the hard to very hard category with high abrasivity. Additional comminution tests may lower grinding energy requirement and abrasivity numbers, which would result in reduced capital and operating costs.
	Maximum process automation.	The mill can be designed to maximize process automation and machine learning to reduce labor requirements and to maximize the metallurgical performance of the plant.
	Sourcing of external graphite concentrate to expand micronization capacity.	Externally sourced graphite concentrate could lead to a higher output of micronized material, thus increasing project revenue.
Secondary Transformation Plant	Expanding the Mineral Resource may extend the mine's operational life and potentially increase the Purification Plant and CSPG Plant throughput.	The Purification Plant and CSPG Plant could operate at a higher capacity, producing more PMG/CSPG and thereby increasing revenue.
	At the end of the mine's LOM, the Purification Plant and CSPG Plant could source external NFG concentrate feed, as its role as a chemical factory enables it to operate indefinitely, exceeding the mine's LOM.	The Purification Plant and CSPG Plant revenues will be extended past the mine's LOM.

26. Recommendations

26.1 Zinc

26.1.1 Operations

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:

- Conduct infill drilling of existing drillholes to improve resource resolution and accuracy, and upgrade the classification of the Inferred Mineral Resource. In parallel, continue exploration drilling programs to expand known resources and replace mined inventory, ensuring long-term sustainability of the operation.
- Advance the proposed ventilation trade-off study, including a desktop analysis of a 4,200-ft deep vent raise to deliver approximately 250 kcfm of fresh air to the underground workings. This expansion will support increased fleet size and access to deeper mineralization, enabling sustained higher production rates. The study includes updating the VentSim model, building an AQE database, and evaluating shaft diameter and development costs.
- Conduct optical sorting testwork 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.

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 and Exploration Drilling	1,230,000
Ventilation Trade-off Study	50,000
Sorting Testwork and Integration Study	100,000
Total Estimate	**1,380,000**

26.1.2 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.
- **Near Mine – Exploration Drilling:** Conduct a minimum of 17,000 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 20,000 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 one-third of the HudBay airborne geophysical survey to further identify prospective areas for both base metal, and graphite mineralization. Additional geophysical methods should be considered for deep near mine and regional exploration targeting. One such method, Ambient Noise Tomography (ANT), may prove useful in confirming historic stratigraphic models and identifying anomalies indicative of sulfide 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.

Further Test Areas of Known Metallogenic Significance: Within the region there are areas with demonstrated metalliferous sulfide and oxide mineralization. These areas have often been excluded from exploration activities due to their dissimilar nature from the Balmat-Pierrepont zinc deposits. These should be evaluated for mineral potential. One such area, the Adirondack Magnetite Belt, has seen little exploration since the mid-twentieth century with developments. This area should be considered for both base and precious metal potential, with shallow drill testing,

soil sampling, and geophysical surveying of known magnetite occurrences recommended. Additional drilling on the Parish Magnetite occurrence is recommended, subject to initial drilling results. 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	200,000
Near Mine – Exploration Drilling	670,000
Exploration Drilling	1,130,000
Geophysics	115,000
Land Acquisition and Management	-
Estimate for 2026	**2,115,000**
Annual Estimate	**2,000,000**

The recommended zinc work programs are not successive.

26.2 Graphite

ESM intends to advance the Kilbourne Project through a structured approach designed to manage risk and optimize value. The overarching objectives guiding this advancement are:

- Health, Safety, and Environment: Maintain the highest standards of safety and environmental stewardship throughout all phases of the Project.
- Cost and Schedule Control: Deliver the Project within defined budgets and timelines to maximize economic returns.
- Community Engagement: Foster positive relationships and contribute to local prosperity.
- Regulatory Compliance: Ensure full adherence to applicable legislative and permitting requirements.

The recommendation for the Graphite Study has been divided into the Kilbourne Site, and the Secondary Transformation Site, which includes the Purification and CSPG plants. The estimated budget for both sites is $26,596,600 and is summarized in Table 26-3. The recommended Graphite programs for the Kilbourne Site and Secondary Transformation Site are not successive.

Table 26-3: Recommendation and estimated budget for Kilbourne and Secondary Transformation sites

Recommended Item	Estimated Cost ($)
Kilbourne Site	18,745,000
Secondary Transformation Site	7,851,600
Total Estimate	**26,596,600**

26.2.1 Kilbourne Site

Based on the results of the Graphite Study, it is recommended that the Company proceed with the Kilbourne Site advancement. A single-phase budget of $18.7 million has been proposed for the mining and processing segments of the Project.

Table 26-4 shows the estimated costs of the recommended drilling, metallurgical, and engineering programs.

Table 26-4: Kilbourne Site recommendations and estimated cost

Recommended Study Item	Estimated Cost ($)
Resource Drilling, Modeling and Estimate	1,842,000
Geotechnical and Hydrogeology Drilling and Modeling	6,074,000
Metallurgical Testwork	385,000
Permitting	373,000
Engineering Studies (mine planning, infrastructure, process design, water management and closure)	3,805,000
Studies Cost	**12,479,000**
Land Acquisitions	2,501,000
Contingency (25%)	3,765,000
Total Estimate	**18,745,000**

26.2.1.1 Geology

The following key recommendations related to geology are provided to assist the Company with future, more detailed levels of study:

- Conduct infill diamond drilling to convert Inferred Mineral Resources to higher level resource classification.
- Conduct exploration diamond drilling to expand the Mineral Resource.
- Collect specific gravity data for the host rock material.
- Update geological and Mineral Resource models.

26.2.1.2 Open Pit

The following key recommendations related to mine design and mine planning are provided to assist the Company with future, more detailed levels of study:

- Generate Mineral Reserve statement.
- Conduct hydrology, hydrogeology and geotechnical studies.
- Obtain robust quotes for open pit mining equipment leasing option to improve the accuracy of cost estimates.
- Perform a blasting study to assess impact of the blasting in nearby infrastructures.

26.2.1.3 Mining Infrastructure

The following key recommendations relating to the site development and mine infrastructure are provided to assist the Company with future, more detailed levels of study:

- Collect geotechnical data for the following locations:
  - Extended TMF;
  - Overburden stockpile;
  - Waste rock stockpile;
  - Graphite processing site;
  - Arnold pit perimeter and containment dikes around it;
  - All new and existing site roads;
  - Extension of existing substation civil works.
- Conduct test on runoff water from waste dumps, overburden stockpiles and processing site for Acid Rock Drainage, Metal Leaching, humidity cells and other tests impacting environmental discharge requirements.
- Develop an execution plan for the removal of the existing tailings.
- Develop construction plans for Kilbourne open pit containment dike, and Historic Arnold Pit.
- A power service connection study should be completed, which includes, but is not limited to:
  - A power study;
  - Confirmation of the capacity and viability of the electrical transmission service to site as well as the ground grid requirements at the existing substation facility;
  - Assessment of the electrical assets to ensure the viability of electrical service to the facility throughout the life of the Kilbourne mine, particularly the existing main substation.

26.2.2 Secondary Transformation Site

Based on the results of the Graphite Study, it is recommended that the Company proceed with Secondary Transformation Site recommendations set out herein.

A single-phase budget has been proposed for the Purification and CSPG segments of the Graphite Study. A budget of approximately $7.9 million is proposed.

Table 26-5 shows the cost of the recommended study items for the Secondary Transformation Site.

Table 26-5: Secondary Transformation Site recommendations and estimated cost

Recommended Study Item	Estimated Cost ($)
Utility Supply Study	20,000
Metallurgical Testwork (bench-scale)	180,000
Wastewater Treatment Testwork (bench-scale)	100,000
CSPG Pilot Plant	5,593,000
Permitting and Environmental (including an EIS)	150,000
AACE Class 3 Study (Feasibility)	500,000
Studies Cost	**6,543,000**
Contingency (20%)	1,308,600
Total Estimate	**7,851,600**

The following activities are recommended to support planning and preparation for the next Study phase:

- Conduct the site location study for the Purification Plant and CSPG Plant to finalize layout, bulk infrastructure, and supply services;
- Perform geotechnical and detailed topographical surveys of site selected to assess ground conditions and to inform the next phase of engineering and design;
- Confirm suppliers of utility services (power, natural gas and water) and define the interface requirements once the site is selected;
- Investigate reversing the process sequence, by placing purification before micronization to improve filterability.

26.2.2.1 Mineral Processing and Metallurgical Testing

Additional bench-scale testwork is recommended to support process optimization and design in the next phase. This includes:

- Assessing the impact of NFG concentrate variability on product performance, quality, CAPEX and OPEX.
- Conducting locked cycle tests to confirm the grade, yield and recovery, and to evaluate the effects of recycling streams (wastewater) and the potential impurity build-up.
- Optimizing reagent consumption and recovery across all major unit operations.
- Investigating wastewater treatment options, given the significant volume of acidic wastewater produced by the Purification Plant and CSPG Plant.
- Performing filtration and drying testwork.

In addition, a CSPG pilot plant campaign (comprising Purification and CSPG) is recommended to:

- Develop metallurgical scale-up parameters necessary to support basic and detailed engineering, as well as sizing and specification of primary equipment (for purification, spheroidization and coating).
- Generate representative samples for equipment vendors to perform confirmatory testwork, verify equipment suitability to ensure process guarantees.
- Produce PMG and CSPG product samples for distribution to customers and off-takers for initial qualification and assessment.
- Refine process flowsheet based on feedback from off-takers and customer qualifications.

26.2.2.2 Recovery Methods

Engineering and design of the Purification Plant and CSPG Plant should be advanced to a more detailed level of study.

26.2.2.3 Capital and Operating Costs

The CAPEX, OPEX and Economic Analysis of the Purification Plant and CSPG Plant should be advanced to a more detailed level of study 3, following final site selection.

The Graphite Study currently excludes bulk services and infrastructure costs, as it is based on the assumption that the Purification Plant and CSPG Plant will be located in a prime chemical industrial estate, where developed plots with existing bulk infrastructure and supply services are available. However, if such infrastructure is unavailable at the selected site, additional engineering, design and cost estimation will be required with future, more detailed levels of study requirements.

26.2.3 Environmental Permitting and Community Impacts

Within the State Environmental Quality Review (SEQR) process in New York State, the Environmental Impact Statement (EIS) serves as the main vehicle for evaluating and disclosing the potential environmental and community impacts of proposed actions, such as the Kilbourne Site and the Purification and CSPG plants site. The EIS will provide detailed analyses of potential environmental impacts on areas such as land, surface water, groundwater, air, plants and animals, community character, and public health. The EIS also needs to provide reasonable alternatives, if any, and mitigation measures, if any. The SEQR and EIS processes promote transparency and public involvement by allowing for public review and comment before the NYSDEC makes final permitting decisions on a project. This ensures that environmental considerations are fully integrated into the planning and approval process. In some cases, such as with wetland impacts and wetlands-related permits, the EIS will serve as a joint document that satisfies both NEPA and SEQR requirements for any Federal and State approvals, respectively.

It is recommended that environmental baseline studies begin as soon as possible to support the EIS so it can be completed promptly within ESM's desired timelines. The results of these studies will assist ESM with future, more detailed studies that will be included in the EIS. These studies include, at a minimum:

- Wetland delineation;
- Noise impact analysis;
- Hydrogeologic evaluation of the potential impact of the open pit, tailings management facility, and the Purification Plant and CSPG Plant;
- Hydrological modeling of the planned discharge location(s);
- Evaluate the contaminants of concern within the tailings and within the planned tailings discharge and ascertain the availability of treatment technologies to meet the desired quality before discharge;
- Undertake multiple accounts analysis of the tailings management facility site to evaluate potential alternatives;
- Acquire land as needed to accommodate the potential expansion of the tailings management facility and the siting of the Plants, if necessary.

26.2.4 Economics

The economic analysis supports the proposition that the Graphite Study may have economic merit across a reasonably wide range of assumptions. Additional precision in terms of analysis is indicated and recommended if the study moves to a full Feasibility Study.

In the next stage of development, the engineering, design, cost estimate and economic analysis should be advanced aligned with future, more detailed level of study requirement following final site selection for the Purification and CSPG plants.

Costing inputs will be refined. Additional resolution in terms of production scheduling should be achieved by moving to more detailed scheduling for mine planning, processing and construction activities.

The Graphite Study currently excludes bulk services and infrastructure costs, as it is based on the assumption that the Purification Plant and CSPG Plant will be located in a prime chemical industrial estate, where developed plots with existing bulk infrastructure and supply services are available. However, if such infrastructure is unavailable at the selected site, additional engineering, design and cost estimation will be required to align with future, more detailed levels of study requirements.

The tax model should be updated to incorporate the existence of any opening balance for tax losses, prior expenditure and depreciation balances.

An updated market study needs to be completed during the next study to understand the available market on each concentrate.

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