Preliminary Economic Assessment NI 43-101 Technical Report

Kay Mine Project
Arizona, USA

Prepared for:

ARIZONA METALS CORP.
66 Wellington Street West, Suite 4100
Toronto, ON, M5K 1B7

Prepared by:

G MINING SERVICES INC.
5025 Lapinière Blvd., Suite 1010
Brossard, QC, J4Z 0N5

Allan Armitage, PhD., P. Geo., SGS Canada Inc.
Ben Eggers, MAIG, P.Geo., SGS Canada Inc.
Carl Michaud, P.Eng., MBA, G Mining Services Inc.
Hind Zniber El Mouhabbis, P.Eng., M.Eng., G Mining Services Inc.
Sunil Koppalkar, P.Eng., G Mining Services Inc.
Nicolas Vanier-Larrivée, P.Eng., G Mining Services Inc.
Richard DeLong, P.Geo., WestLand Engineering & Environmental Services
Eric J. Mears, R.G., C.P.G., Haley & Aldrich, Inc.

Effective Date: April 30, 2026
Issue Date: June 12, 2026

IMPORTANT NOTE

General Conditions and Limitations

Use of the report and its contents

This report has been prepared for the exclusive use of the Client or its agents. The factual information, descriptions, interpretations, comments, recommendations and electronic files contained herein are specific to the projects described in this report and do not apply to any other project or site. Under no circumstances may this information be used for any other purposes than those specified in the scope of work unless explicitly stipulated in the text of this report of formally interpreted when taken individually or out-of-context. As well, the final version of this report and its content supersede any other text, opinion or preliminary version produced by G Mining Services Inc.

Preliminary Economic Assessment NI 43-101 Technical Report – Kay Mine Project

Arizona, USA

ARIZONA METALS CORP.

66 Wellington Street West, Suite 4100
Toronto, ON,
Canada, M5K 1B7
Tel: (647) 202-3904
E-mail: info@arizonametalscorp.com
Web Address: https://www.arizonametalscorp.com/

G MINING SERVICES INC.

5025 Lapinière Blvd., Suite 1010
Brossard, Québec
Canada J4Z 0N5
Tel: (450) 465-1950 • Fax: (450) 465-6344
E-mail: m.gignac@gmining.com
Web Address: www.gmining.com

June 12, 2026

Qualified Persons

Prepared by:

(signed and sealed) "Allan Armitage" Date: June 12, 2026

Allan Armitage, PhD., P. Geo.,
Technical Manager & Senior Resource Geologist
SGS Canada Inc. – Geological Services

(signed and sealed) "Ben Eggers" Date: June 12, 2026

Ben Eggers, MAIG, P.Geo.,
Senior Geologist
SGS Canada Inc. – Geological Services.

(signed and sealed) "Carl Michaud" Date: June 12, 2026

Carl Michaud, P.Eng., MBA,
Vice President of Mining Engineering
G Mining Services Inc.

(signed and sealed) "Hind Zniber El Mouhabbis" Date: June 12, 2026

Hind Zniber El Mouhabbis, P.Eng., M.Eng.,
Technical Services Director
G Mining Services Inc.

(signed and sealed) "Sunil Koppalkar" Date: June 12, 2026

Sunil Koppalkar, P.Eng.,
Senior Metallurgist
G Mining Services Inc.

(signed and sealed) "Nicolas Vanier-Larrivée" Date: June 12, 2026

Nicolas Vanier-Larrivée, P.Eng.,
Earthworks and Study Manager
G Mining Services Inc.

(signed and sealed) "Richard DeLong" Date: June 12, 2026

Richard DeLong, P.Geo.,
Senior Technical Advisor
WestLand Engineering & Environmental Services

(signed and sealed) "Eric J. Mears" Date: June 12, 2026

Eric J. Mears, R.G., C.P.G.,
Principal
Haley & Aldrich, Inc.

CERTIFICATE OF QUALIFIED PERSON – ALLAN ARMITAGE, MAIG, P.Geo.

To Accompany the Report entitled “Preliminary Economic Assessment NI 43-101 Technical Report – Kay Mine Project, Arizona, USA”, prepared for Arizona Metals Corp. effective as of April 30, 2026 (the “Technical Report”).

I, Allan E. Armitage, Ph. D., P. Geo., do hereby certify that:

1) I am a Senior Resource Geologist and Technical Manager with SGS Canada Inc., 2150 Cyrille-Duquet St., Unit 150, Quebec, QC, G1N 2G3, Canada.
2) I am a graduate of Acadia University having obtained the degree of Bachelor of Science - Honours in Geology in 1989, a graduate of Laurentian University having obtained the degree of Master of Science in Geology in 1992 and a graduate of the University of Western Ontario having obtained a Doctor of Philosophy in Geology in 1998.
3) I have been employed as a geologist for every field season (May - October) from 1987 to 1996. I have been continuously employed as a geologist since March of 1997.
4) I have been involved in mineral exploration and resource modeling at the early-stage exploration property to the advanced property, including producing mines, since 1991, including mineral resource estimation and mineral resource and mineral reserve auditing since 2006 in Canada and internationally. I have extensive experience in Archean and Proterozoic load gold deposits, volcanic and sediment hosted base metal massive sulphide deposits, porphyry copper-gold-silver deposits, low and intermediate sulphidation epithermal gold and silver deposits, magmatic Ni-Cu-PGE deposits, and unconformity- and sandstone-hosted uranium deposits. I have extensive experience in the preparation of NI 43-101 Technical Reports, including PEA, PFS and FS Technical Reports, and I have conducted numerous site visits to early-stage exploration and advanced projects, and operating mines (open pit and underground).
5) I am a member of: the Association of Professional Engineers, Geologists and Geophysicists of Alberta (P.Geol.) (License No. 64456; 1999), the Association of Professional Engineers and Geoscientists of British Columbia (P.Geo.) (Licence No. 38144; 2012), and the Professional Geoscientists Ontario (P.Geo.) (Licence No. 2829; 2017).
6) I have read the definition of "Qualified Person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects – (“NI 43-101"). I certify that by reason of my education, affiliation with a professional association (as defined in NI 43 101), and extensive work experience (including site visits), I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43 101, and I fulfill the necessary experience to take professional responsibility for certain sections of this Technical Report.
7) I am an author of the Technical Report and responsible for sections 1.4, 1.8, 1.12, 1.14, 1.23, 1.24, 4, 8, 12.3, 12.5, 14, 23, 24, 25.2, 25.10.1.1, 25.10.2.1, 26.1 and 27. I have reviewed these sections and accept professional responsibility for these sections of the Technical Report.
8) I have conducted two site visits to the Project. I conducted a site visit to the Project on October 25-26, 2023, and April 7-8, 2024.
9) I have had previous involvement with the Kay Project. I was an author of the previous NI 43-101 Technical Report titled “Mineral Resource Estimate for the Kay Deposit Cu-Au-Zn-Pb-Ag Project, Yavapai County, Arizona, USA” with an effective date of June 17, 2025.
10) I am independent of the Company as described in Section 1.5 of NI 43-101.
11) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12) I have read NI 43-101 and Form 43-101F1 (the “Form”), and the Technical Report has been prepared in compliance with NI 43-101 and the Form.

Dated this 12th day of June 2026

/signed and sealed/

Allan E. Armitage, Ph. D., P. Geo.,
Senior Resource Geologist and Technical Manager
SGS Geological Services – SGS Canada Inc.

CERTIFICATE OF QUALIFIED PERSON – BEN EGGERS, MAIG, P.Geo.

To Accompany the Report entitled "Preliminary Economic Assessment NI 43-101 Technical Report – Kay Mine Project, Arizona, USA", prepared for Arizona Metals Corp. effective as of April 30, 2026 (the "Technical Report").

I, Ben Eggers, MAIG, P.Geo., do hereby certify that:

1) At the time of the study, I was employed as Senior Geologist, at SGS Geological Services – GS Canada Inc. with an office at 150-2150 rue Cyrille-Duquet, Québec, QC, J7C 3V5, Canada;
2) I graduated from the University of Otago, New Zealand with a Bachelor of Science (Honours) in Geology in 2004;
3) I have practiced my profession continuously for 20 years and have been employed as a geologist since February of 2005. Since then, I have been involved in mineral exploration and resource modeling at the greenfield to advanced exploration stages, including at producing mines, in Canada, Australia, and internationally, and in mineral resource estimation since 2022 in Canada and internationally. I have experience in orogenic gold deposits, low, intermediate, and high sulphidation epithermal gold and silver deposits, porphyry copper-gold-silver deposits, volcanic and sediment hosted base metal massive sulphide deposits, metasomatite uranium deposits, and pegmatite lithium deposits;
4) I am a member of the Association of Professional Engineers and Geoscientists of British Columbia and use the designation (P.Geo.) (Licence No. 40384; 2014), I am a member of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG) and use the designation (P.Geo.) (Licence No. L5818, 2024), and I am a member of the Australian Institute of Geoscientists and use the designation (MAIG) (Licence No. 3824; 2013);
5) I have read the definition of "Qualified Person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects – ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101;
6) I am an author of the Technical Report and responsible for sections 1.5 to 1.7, 1.9 to 1.11, 5, 6, 7, 9, 10, 11, 12.1, 12.2, 12.4, and 27. I have reviewed these sections and accept professional responsibility for these sections of the Technical Report;
7) I conducted a site visit to the Property on May 30, 2025;
8) I have had previous involvement with the Kay Project. I was an author of the previous NI 43-101 Technical Report titled "Mineral Resource Estimate for the Kay Deposit Cu-Au-Zn-Pb-Ag Project, Yavapai County, Arizona, USA" with an effective date of June 17, 2025;
9) I am independent of the Company as described in Section 1.5 of NI 43-101;
10) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
11) I have read NI 43-101 and Form 43-101F1 (the "Form"), and the Technical Report has been prepared in compliance with NI 43-101 and the Form.

Dated this 12th day of June 2026

/signed and sealed/

____

Ben Eggers, MAIG, P.Geo.
Senior Geologist
SGS Geological Services – SGS Canada Inc.

CERTIFICATE OF QUALIFIED PERSON – CARL MICHAUD, P.ENG., MBA

To Accompany the Report entitled "Preliminary Economic Assessment NI 43-101 Technical Report – Kay Mine Project, Arizona, USA", prepared for Arizona Metals Corp. effective as of April 30, 2026 (the "Technical Report").

I, Carl Michaud P.Eng., MBA., do hereby certify that:

1) At the time of the study, I was employed as Vice President of Mining Engineering, at G Mining Services with an office at 5025, Lapinière Blvd, Suite 1010, Brossard, Québec , Canada, J4Z 0N5;
2) I have graduated from l'Université Laval with a B.Sc. (Mine Engineering) in 1996. In addition, I obtained an M.B.A. from the Université du Québec à Chicoutimi, in 2012;
3) I am a member of the "Ordre des ingénieurs du Québec " (OIQ, No. 117090);
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have experience in narrow-vein gold deposits, flat and steeply dipping, bulk and selective mining methods for base metals, mine infrastructure, design and planning, mine production and financial evaluation, reserve estimation, technical reviews, feasibility and pre-feasibility studies, project and construction management, contracts management and cost estimation. I have occupied different positions, both technical and operational, related to mining engineering, in Underground and Open Pit operation. This experience includes Kiena and Sigma Gold mine (Placer Dome), Éléonore Mine (Goldcorp) and Mont Wright Mine (Arcelor Mittal);
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 1.1, 1.15, 1.16, 1.21, 1.25, 1.26, 15, 16, 21.1.5, 21.2, 21.3.2, 21.4.1, 21.4.4,25.1, 25.3, 25.4, 25.8, 25.9, 25.10.1.2, 25.10.1.3, 25.10.2.2, 25.10.2.3, 26.2, 26.6, and 27;
7) I visited the Kay Project on October 6 and 7, 2025, for the purpose of inspecting general site conditions. I also reviewed drill cores and their locations and conducted aa review of the rock quality in the drill holes;
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101;
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 12th day of June 2026

/signed and sealed/

Carl Michaud, P.Eng., MBA.
Vice President of Mining Engineering
G Mining Services Inc

CERTIFICATE OF QUALIFIED PERSON – HIND ZNIBER EL MOUHABBIS, P.ENG., M.ENG.

To Accompany the Report entitled "Preliminary Economic Assessment NI 43-101 Technical Report – Kay Mine Project, Arizona, USA", prepared for Arizona Metals Corp. effective as of April 30, 2026 (the "Technical Report").

I, Hind Zniber El Mouhabbis, P.Eng., M.Eng. do hereby certify that:

1) At the time of the study, I was employed as Technical Services Director (Mining) by G Mining Services Inc, located at 5025, Lapinière Blvd, Suite 1010, Brossard, QC, J4X 0N5;
2) I graduated with a Bachelor's degree in Mining Engineering (B.Eng.) from École Polytechnique de Montréal (Montreal, Quebec) in 2008. In addition, I obtained a Master of Engineering in mining from McGill University (Montreal, Quebec) in 2013;
3) I am a member of the "Ordre des ingénieurs du Québec" (OIQ, No. 5007612);
4) My relevant experience includes a total of 16 years since graduating from university. I acquired my mining expertise in economic analysis through roles in project mining engineering and strategic planning since graduation;
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 1.2, 1.3, 1.19, 1.22, 2, 3, 19, 22;
7) I have not visited the site property that is the subject of this Technical Report;
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101;
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 12th day of June 2026

/signed and sealed/

Hind Zniber El Mouhabbis, P.Eng., M.Eng.
Technical Services Director
G Mining Services Inc

CERTIFICATE OF QUALIFIED PERSON – SUNIL KOPPALKAR, P.ENG. PhD.

To Accompany the Report entitled “Preliminary Economic Assessment NI 43-101 Technical Report – Kay Mine Project, Arizona, USA”, prepared for Arizona Metals Corp. effective as of April 30, 2026 (the “Technical Report”).

I, Sunil Koppalkar P.Eng. PhD, do hereby certify that:

1) At the time of the study, I was employed as Senior Metallurgist, at G Mining Services with an office at 5025, Lapinière Blvd, Suite 1010, Brossard, Québec, Canada, J4Z 0N5;
2) I have graduated from McGill University, Montreal with a doctorate degree in Mining and Materials Engineering in 2010, and a masters' degree in Mineral Engineering from the Indian Institute of Technology (IIT), Dhanbad, India in 1986.
3) I am a Professional Engineer registered in good standing with the Professional Engineers Ontario (PEO, No. 100190343).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in concentrator operations, mineral processing research & development, and engineering & consultancy. This experience includes work at Hindustan Copper Ltd. Copper concentrator operations, working with various engineering firms designing process flowsheets for a variety of mineral commodities.
5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 1.13, 1.17, 13, 17, 21.4.2, 25.5, 25.10.1.4, 25.10.2.4, 26.3, 27;
7) I have not visited the site that is the subject of this Technical Report;
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101;
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 12th day of June 2026

/signed and sealed/

____

Sunil Koppalkar, P.Eng. PhD.
Senior Metallurgist
G Mining Services Inc

CERTIFICATE OF QUALIFIED PERSON – NICOLAS VANIER-LARRIVÉE, P.ENG.

To Accompany the Report entitled "Preliminary Economic Assessment NI 43-101 Technical Report – Kay Mine Project, Arizona, USA", prepared for Arizona Metals Corp. effective as of April 30, 2026 (the "Technical Report").

I, Nicolas Vanier-Larrivée, P.Eng., do hereby certify that:

1) At the time of the study, I was employed as Earthworks & Study Manager, at G Mining Services with an office at 5025, Lapinière Blvd, Suite 1010, Brossard, Québec , Canada, J4Z 0N5;
2) I graduated with a bachelor's degree in civil engineering (B. ing.) from Université de Sherbrooke (Sherbrooke, Quebec) in 2006;
3) I am a member of the "Ordre des ingénieurs du Québec " (OIQ, No. 143023), of the Association of Professional Engineers and Geoscientists of Saskatchewan (APEGS, No. 78043) and of Engineers Yukon (EY, No. 4551);
4) My relevant experience includes a total of 20 years since graduating from university. This includes the design and construction of mineral processing plants for gold, lithium, and copper projects, as well as work on technical reviews, economic studies (PEA, PFS, FS), project and construction management, contracts, and cost estimation;
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 1.18, 18, 21.1 (excluding 21.1.5), 21.4.3, 25.6, 25.10.1.5, 25.10.2.5, 26.4;
7) I have not visited the site property that is the subject of this Technical Report;
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101;
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 10th day of June 2026

/signed and sealed/

____

Nicolas Vanier-Larrivée, P.Eng.
Earthworks & Study Manager
G Mining Services Inc

CERTIFICATE OF QUALIFIED PERSON – RICHARD DELONG, MMSA QP, P.G

To Accompany the Report entitled "Preliminary Economic Assessment NI 43-101 Technical Report – Kay Mine Project, Arizona, USA", prepared for Arizona Metals Corp. effective as of April 30, 2026 (the "Technical Report").

I, Richard DeLong, do hereby certify that:

1) At the time of the study, I was employed as Senior Technical Advisor, at WestLand Engineering & Environmental Services, a Trinity Consultants Team, with an office at 5401 Longley Lane, Suite 5, Reno, Nevada 89511;
2) I have graduated from California State University, Chico with a Bachelor's Degree in Geology in 1980, and from University of Idaho, Moscow, Idaho with a Master's Degree in Resource Management in 1984 and a Master's Degree in Geology in 1986;
3) I am a Qualified Member of the Mining and Metallurgical Society of America (01471QP) with special expertise in Environmental Permitting and Compliance, a Registered Geologist in the State of California (#5570), and a Professional Geologist in the State of Idaho (#727);
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operation permitting, and environmental evaluations for 40 years, including permit acquisition scheduling, surface facility layout, environmental audits and due diligence, and agency negotiations;
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 1.20, 4.5, 20.1, 20.2, 20.3, 20.4, 20.5, 20.6, 20.7, 25.7, and 26.5;
7) I have visited the site property that is the subject of this report on February 16, 2026. The purpose of the visit was to evaluate the site conditions, understand to physical layout of the proposed facilities, and understand the relationship between the project and the surrounding area;
8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101;
10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

Dated this 12th day of June 2026

/signed and sealed/

Richard DeLong, MMSA QP, P.G.
Senior Technical Advisor
WestLand Engineering & Environmental Services, a Trinity Consultants Team

HALEY & ALDRICH, INC.
201 East Washington Street
Suite 1795
Phoenix, AZ 85004
602.760.2450

CERTIFICATE OF QUALIFIED PERSON –Eric J. Mears, R.G., C.P.G.

To Accompany the Report entitled "Preliminary Economic Assessment NI 43-101 Technical Report – Kay Mine Project, Arizona, USA," prepared for Arizona Metals Corp. effective as of April 30, 2026 (the "Technical Report").

I, Eric J. Mears. R.G., C.P.G., do hereby certify that:

1) At the time of the study, I was employed as Vice President at Haley & Aldrich with an office at 201 East Washington Street, Suite 1795, Phoenix, Arizona, 85004;
2) I have graduated from Eastern Illinois University in Charleston, Illinois, with a B.S. in Geology in 1986, and from Eastern Illinois University with a BS in Physical Geography in 1986;
3) I am a member of the American Institute of Professional Geologists (AIPG) CPG-12335 and hold registration as a Professional Geologist in Arizona RG-28391.
4) I have practiced my profession continuously since my graduation from university. I have been involved in permitting, reclaiming, and closing numerous mining operations in the United States for over 35 years, including securing mine permitting and entitlements with federal, state, and local jurisdictions, performing reclamation and closure planning and closure cost estimating;
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43 101") and certify that by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43 101;
6) I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 20.8.3 and 21.3;
7) I have visited the Site on numerous occasions from 2024 to present. The purpose of the visits was to secure exploration entitlements and conduct background environmental studies;
8) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections and sub-sections of the Technical Report listed in item 6 above contain all scientific and technical information that is required to be disclosed to make these sections and sub-sections of the Technical Report not misleading;
9) I have read NI 43 101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43 101; and

10) I have read and understand NI 43 101, and I am considered independent of the issuer as defined in section 1.5 of NI 43 101 Rules and Policies.

Dated this 9th day of June 2026

/signed and sealed/

____

Eric J. Mears, R.G., C.P.G.
Vice President
Haley & Aldrich, Inc.

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1. SUMMARY

1.1 Introduction

Arizona Metals Corp. ("AMC" or the "Company") mandated G Mining Services Inc. ("GMS") as lead consultant along with other engineering consultants to prepare a Preliminary Economic Assessment ("PEA") under the supervision of the QPs for the Kay Mine Project ("USKM" or "Project"). The site lies next to the town of Black Canyon City, about 69 km (43 miles) north of Phoenix, in central Arizona, USA.

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this Report and the PEA is the evaluation of the potential technical feasibility and potential economic viability of the Project, notably the development of an underground mine, including processing facilities and related infrastructures. This Report provides operating and capital costs estimations and an economic analysis of the Project.

This Report includes the Mineral Resource Estimate ("MRE") statement announced on June 30, 2025, and documented in the technical report dated August 14, 2025. It is based on a drilling database dated June 17, 2025, and supporting geological information. The classification of the updated MRE is consistent with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definitions). In completing the MRE, the Author uses general procedures and methodologies that are consistent with industry standard practices, including those documented in the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019 CIM Guidelines). The Kay Mine Project does not contain Mineral Reserves.

The qualified persons ("QP") of this Technical Report are the following:

- Allan Armitage, P. Geo, SGS Canada Inc – Geological Services, Technical Manager and Senior Resource Geologist.
- Ben Eggers, P. Geo, SGS Canada Inc – Geological Services, Senior Geologist.
- Carl Michaud, P.Eng., MBA, G Mining Services, UG Mining, Vice President of Mining Engineering.
- Hind Zniber El Mouhabbis, P. Eng., M.Eng., G Mining Services, Technical Services Director.
- Sunil Koppalkar, P.Eng., PhD, G Mining Services, Senior Metallurgist.
- Nicolas Vanier-Larrivée, P.Eng., G Mining Services, Infrastructures, Earthworks and Study Manager.

- Richard DeLong, WestLand Engineering & Environmental Services, Senior Technical Advisor.
- Eric J. Mears, R.G., C.P.G, Haley & Aldrich Inc., Environment, Principle.

1.2 Terms of Reference

Unless otherwise stated, all information and data pertaining to the Mineral Resource Estimate contained herein or used in the preparation of this Report were prepared by SGS with an effective date of June 17, 2025. References, located at the end of this Technical Report, provides a complete list of the documents reviewed, all figures and tables cited, and other information sources used.

The units of measure presented in this Technical Report, unless noted otherwise, are in the metric system. Currency is expressed in United States dollars (“USD”), unless stated otherwise.

1.3 Reliance on Other Experts

This Technical Report, prepared by GMS under the supervision of Qualified Persons (QPs) for AMC is based on the following:

- Information and documentation available to GMS at the time of the preparation of this Report, which are listed in Section 27 -- References.
- Assumptions, conditions and qualifications as set forth in this Report.
- Data, reports, and opinions supplied by AMC and other third-party sources.

The QPs believe that the assumptions and interpretations in this report are factual and reasonable. They have relied on the data provided and believe no material facts have been withheld. The QPs have taken appropriate steps to ensure the reliability of the work and do not disclaim responsibility for the report.

The QPs have also relied upon the expertise of the following consultants for matters related to taxation:

- The taxation calculation was supplied by McGovern Hurley LLP staff and retained expert Shawn Lee, Chartered Professional Accountants.
- This information is used in support of Section 0, Section 22 and Section 25.

The results and opinions in this report depend on the accuracy and completeness of the experts' information as of the report's effective date. The QPs are only responsible for the sections of the report identified in their "Certificates of Qualified Persons" submitted to Canadian Securities Administrators. Any third-party use of this report beyond provincial securities laws is at the user's own risk.

1.4 Property Description and Location

The Kay Mine property is located immediately adjacent to the town of Black Canyon City, approximately 69 km (43 miles) north of the city of Phoenix, in central Arizona, USA. The Property is located in Township 8 North, Range 2 East (Gila and Salt River meridian), in the Tip Top mining district in Yavapai County, Arizona. The UTM coordinates of Shaft 1 on the eastern portion of the property are 392910E, 3769540N (WGS84 datum, Zone 12S). The property falls on the Black Canyon City 7.5-minute topographic map published by the United States Geological Survey.

The Kay Mine property consists of 88 unpatented lode mining claims covering approximately 645.2 ha (1,594.4 acres), six (6) patented mining claims covering approximately 30.4 ha (75.1 acres), and 78.0 ha (192.7 acres) of private land. The private land includes mineral rights, four (4) water wells, and housing for company staff. The company also owns two (2) unpatented placer mining claims totaling 16.2 ha (20.0 ac) co-located with unpatented lode mining claims.

1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access to the Kay Project is excellent by road on Interstate Highway 17, then by paved city streets in Black Canyon City to the banks of the Agua Fria River. Gravel drill and mine roads give access to the Kay Project. Vehicle access onto the Kay Project currently requires crossing Black Rock Creek, a small stream with intermittent flow highest in the winter months (January – March) and lowest in the spring and summer (May – July), with occasional storm-related high and turbulent flow.

The Kay Project is immediately adjacent to population in the town of Black Canyon City, population about 5,600, which offers basic services such as fuel, food, and housing. The Project is very well positioned relative to support infrastructure, with ready access to power and water in adjacent Black Canyon City, and excellent road access along Interstate Highway 17 and paved city streets.

The Kay Project lies in an area of moderate topography, reaching elevations of 683 m (2,240 feet) with relief of approximately 100 m (320 feet) from the streambed of the Agua Fria River to the summits of hills on the Kay Project. The terrain is accommodating to exploration activities, as evidenced by previous mine shafts and access roads.

1.6 History

The Kay Mine was discovered sometime before 1900 and mined on a small scale from the inclined No. 1 shaft, producing approximately 635 tonnes (700 short tons) of ore prior to 1916 or 1918.

Between 1918 and the late 1920s, the Property was owned by an eastern mining interest that became the Kay Copper Company in 1922. During this period, the owners deepened the No. 1 Shaft to 457 m (1,500 ft), sunk the No. 4 shaft to 366 m (1,200 ft), installed the No. 3 Shaft, and developed several thousand feet of underground workings on 11 levels, discovering the ore bodies above the 600 Level but apparently producing no ore. Judging by mine maps, the company drilled at least 89 underground drill holes (according to mine plan maps); assay data are plotted on mine plan maps, but no drill logs nor assay certificates are available. The Kay Copper Company failed in the late 1920s, and the project was dormant until 1949, apparently from a combination of low metals prices and litigation.

In the late 1940s the project was acquired by an unnamed owner for back taxes, and in 1949 leased to Black Canyon Copper Corporation, which opened the underground workings to the 500 Level and shipped about 907 tonnes (1,000 short tons) of ore.

In 1949 or 1950, Black Canyon Copper sub-leased the project to Shattuck-Denn Mining Company and New Jersey Zinc Company until 1952. These companies dewatered and rehabilitated the No. 4 Shaft at least to the 1,000 Level, and performed surface and underground exploration, including resampling and underground diamond drilling of at least 14 holes (according to mine plan maps). They shipped an uncertain amount of ore, reported to be 1,425 tonnes (1,571 short tons).

In 1955-1956, the project was leased to Republic Metals Company, which shipped 414 tonnes (456 short tons) of ore from above the 350 Level. A cave-in destroyed pumping operations, and the mine was allowed to flood. Following this, the project saw several unsuccessful attempts to revive operations until 1972.

The project was acquired by Exxon Minerals Company in 1972, which invested about $1.5M in exploration on the project. This work included geologic mapping; “mine mapping” (suggesting that Exxon re-opened the underground workings); relogging drill core and cuttings; petrographic studies; assaying 610 m (2,000 ft) of unassayed drill core; stream sediment and soil geochemistry surveys; reviewing historical assay data and incorporating into mine maps and cross sections; and geophysical surveys. Exxon drilled 23 core / rotary exploration holes totaling 8,094 m (26,554 ft), 14 of which were in the immediate vicinity of the Kay Mine and which total 6,807 m (22,333 ft). Fellows (1982) also mentions “ten (10) shallow air-track claim validation drill holes on various parts of the property,” but gives no specific locations. Exxon’s last reported work on its project was 1984.

The five (5) patented claims changed hands a number of times between 1990 and 2015, apparently without exploration work. In 1990 Exxon sold the five (5) patented claims to Rayrock Mines, which in turn sold them to American Copper and Nickel Company in 1995. Ownership was then conveyed to Shangri-La Development in 2000, to five (5) private individuals in 2002, and to Jodon Development in 2003. In 2015, Cedar Forest Inc. acquired the five (5) patented claims through foreclosure on Jodon Development. Cedar Forest did not appear to do any exploration work on the project.

In March 2017, Silver Spruce Resources Inc. acquired the five (5) patented mining claims from Cedar Forest and then staked 14 unpatented “KM” mining claims in April 2017. Together, these 19 claims comprise the property purchased by Arizona Metals. Silver Spruce took 39 samples on the project but did no other exploration work.

On September 26, 2018, Croesus Gold Corporation (now Arizona Metals) signed a letter of intent to acquire the five (5) patented and 14 unpatented “KM” claims from Silver Spruce Resources. To date, Arizona Metals has performed geologic, geochemical, and geophysical exploration and drilling on the project and staked additional unpatented mining claims.

The historical production record of the mine is scattered and almost certainly incomplete. Keith et al (1983) reported that the Kay Mine produced 2,600 short tons of ore containing 296,000 pounds Cu, 13,000 pounds Pb, 2,700 ounces Ag, and 150 ounces Au. Based on more detailed project-specific reports currently available, the total documented production from the Kay Mine is approximately 3,016 tonnes (3,325 short tons).

1.7 Geological Setting and Mineralization

The Kay Project is located in Precambrian metamorphic rocks in central Arizona. Central Arizona is characterized by basement rocks of Proterozoic age (1.8-1.6 Ga) with great stratigraphic complexity and pervasive yet variable deformation and metamorphism. The Proterozoic basement is well exposed in a broad 500-km-long NW-trending belt that transects the state from southeast to northwest known as the central volcanic belt. The Proterozoic basement is directly overlain in places by Tertiary volcanic and sedimentary rocks and by Quaternary surface deposits and has been intruded by widespread Laramide-age granitoids, many of which produced the large porphyry copper systems that have made Arizona famous for copper production. The Proterozoic basement rocks are the result of largely compressional tectonics active between 2.0 and 1.62 Ga, with several periods of subduction, accretion of numerous island arcs onto the ancestral Wyoming craton, and attendant volcanism, plutonism, deformation, and metamorphism (Smith, 2024, and references therein).

The Kay Project lies in an NNE-trending belt of schists and phyllites comprising metamorphosed volcanics and metasediments with minor chert and iron formation. In the property area, this belt of schists is bordered on the east by alluvium in the Agua Fria River drainage and Tertiary sediments and volcanics; and bordered on the west by the Proterozoic Crazy Basin monzogranite. The Shylock shear zone, a regional structural feature, runs to the west of the Property.

Host rocks on the Property consist of greenschist-metamorphosed volcanic, volcaniclastic, and sedimentary rocks of Proterozoic age. These rocks fall within the Townsend Butte facies of the Black Canyon Creek Group of the Yavapai Supergroup aged 1800-1740 Ma. The Property geology is divided into three (3) lithologic domains: the Hangingwall Mafic Sequence, the Hangingwall Felsic Sequence, and the Footwall Mafic Sequence. Hangingwall and footwall in this setting refer to above and below VMS mineralization, respectively.

Structure in the property area is complex. The host rocks on the Property are intensely deformed, characterized by steeply dipping bedding, foliation, lineations, and folds resulting from three (3) phases of deformation as recorded by SRK (2020a, 2020b, 2020c) and Baxter & Diekrup (2023). The first phase of deformation was the most intense and formed isoclinal folds with attenuated and sometimes separated fold limbs and a pervasive axial-planar S1 foliation that strikes 186-208 azimuth and dips 63-89 to the west. S1 fold axes have an average trend of 229 azimuth and plunge of 85. Geologic mapping by SRK (2020a) and Baxter & Diekrup (2023) shows that steeply dipping isoclinal S1 folding repeats the felsic and mafic schists across the property. SRK (2020a) noted that within this folding style, sulfide lenses are likely to be affected by steeply plunging tight folds, with thinned or boudined fold limbs and thickened fold hinges, and possible repetition of sulfide lenses through folding. Geologic modeling of the mineralization using drill data and historical underground mapping shows the nature of S1 folding.

In zones of strong to extreme strain in this region, primary features can be distorted into cigar shapes. This is reflected in the shape of the Kay deposit, which has a steeply dipping prolate shape parallel to the mineral stretching lineation. This is an important observation for exploration, and targets should be developed acknowledging that additional VMS bodies may be tubes or prolates rather than tabular bodies.

Mineralization on the property occurs principally near the historic Kay Mine workings. In this area, it consists of stratabound lensoid bodies of massive sulfide in a folded horizon that strikes generally north and dips from vertical to 75° west. Massive sulfide occurs along a strike length of approximately 430 m and a down-dip extent of over 950 m, as defined by Arizona Metals drilling combined with historical drilling and underground mapping. Drilled widths vary between <1 m and 125 m, with approximate true width of mineralization estimated to be 65-97% of reported core width, averaging 80%. Thinner portions are interpreted as fold limbs, and wider portions as thickened fold hinges, forming steeply dipping, generally cigar to tabular shapes that pinch and swell.

Kay Mine sulfide mineralization consists of massive, semi-massive, and stringer-like aggregates of pyrite, arsenopyrite, chalcopyrite, sphalerite, and galena. Petrographic studies reveal varying proportions of intergrown pyrite, arsenopyrite, chalcopyrite, sphalerite, tetrahedrite-tennantite, and galena. Rare boulangerite ($Pb_5Sb_4S_{11}$) is intergrown with galena; tellurobismuthite (Bi_2Te_3) and hessite (Ag_2Te) occur in chalcopyrite. Gangue minerals include chlorite, quartz, sericite, and dolomite; two (2) generations of carbonate have been observed, one older inclusion-rich, and a younger, clear more euhedral variety, typically occurring with mineralization. More recent analysis of carbonate trends indicates that ankerite signifies proximity to mineralization.

Exxon previously identified 18 massive sulfide bodies through drilling and underground mining, which they grouped into two (2) principal closely spaced zones, called the North Zone and South Zone. Recent drilling by Arizona Metals suggests greater continuity than proposed by Exxon, and it is now clear that what appeared to Exxon as separate sulfide bodies and separate North and South zones are more likely part of the same mineralized horizon.

1.8 Deposit Types

The Kay Deposit consists of structurally deformed and metamorphosed, stratabound, polymetallic massive, semi-massive and stringer sulphide mineralization. The sulphides contain copper, gold, zinc, lead and silver mineralization. Mineralization of the Kay Deposit show the geological, mineralogical and geochemical characteristics of Volcanogenic massive sulphide (VMS) deposits.

1.9 Exploration

Since 2019, Arizona Metals has performed the following exploration work:

- Staked 74 additional unpatented lode mining claims covering 566.8 ha (1,400.1 ac).
- Staked two (2) additional unpatented placer mining claims covering 16.2 ha (40 ac) co-located with unpatented lode mining claims.
- Purchased a total of 78.0 ha (192.7 ac) of private land in three (3) transactions.
- Collected and analyzed 30 due-diligence rock samples.
- Geologic reconnaissance to the west of the patented claims.
- Digitized all historical project data and conducted 3-dimensional modeling.
- Topographic survey by drone aircraft.
- VTEM geophysical survey followed by reprocessing and interpretation.

- Ground electromagnetic (EM) geophysical survey in three (3) areas of the project.
- Borehole electromagnetic (BHEM) geophysical survey in selected Arizona Metals drill holes.
- Geophysical gravity survey.
- Soil and rock sampling.
- Geologic mapping.
- Structural interpretation.
- Alteration and trace-element studies.
- Petrographic studies.

1.10 Drilling

Arizona Metals initiated drilling on the Property in January 2020 and has continued to explore and delineate the Kay deposit with a series of drill programs undertaken each year through to 2026. As of March 2026, Arizona Metals had completed 274 drill holes totalling 146,844 m and collected 12,329 assays.

Arizona Metals have continued to drill on the Project since the data cut-off for the current MRE. Exploration drilling completed in the period from June 2025 through March 2026 totals 20 drillholes for 12,932 m. Drilling during this period focused on exploration targets outside of the Kay MRE area targeting the prospective stratigraphy at the Kay North Extension and Northwest Target areas. Drilling within or proximal to the Kay MRE was limited to two (2) drillholes (KM-25-194 and KM-25-195). The exploration drillholes completed in the second half of 2025 and 2026 and their location relative to the MRE are not likely to materially change the current MRE for the Project.

Historical drilling on the Kay Mine Project was undertaken during the late 1910s and early 1920s (Kay Copper Company), in the early 1950s (New Jersey Zinc), between 1972 and 1984 (Exxon Minerals Company), and from 1991 to 1993 (Rayrock Mines) and collectively totals at least 139 holes. While partial documentation remains to support this historical drilling, these drillholes are utilized for exploration guidance only and not relied upon for the estimation of mineral resources.

Drilling by Arizona Metals within the Kay deposit has primarily been completed on 30 m to 60 m centres. Drilling to date has been completed from surface and comprises angled holes (collar dips range from -15° to -89°) completed predominantly from five (5) drill pad locations in a vertical and horizonal fan pattern. A significant proportion of the deep drilling has been completed using wedge holes and directional drilling. Holes are collared in the hanging wall of and as orthogonal as practical to target lenses.

Arizona Metals drilling of the Kay deposit sulphide lenses has delineated mineralization along a strike length of approximately 430 m, and a down-dip extent of over 950 m. Drilled widths vary between <1 m and 125 m, with approximate true width of mineralization estimated to be 65-97% of reported core width, averaging 80%.

Diamond drillholes are HQ diameter, with reduction to NQ diameter if necessitated by ground conditions. Drilling to date has been completed using surface drill rigs. Maximum drilling depths obtained to date are approximately 1,700 m. Drillhole collars positions have been obtained using handheld GPS for common drill pad locations. Downhole orientations of drillhole azimuth and inclination are recorded by a gyroscopic survey instrument every 30 m downhole or at 6 m intervals during directional drilling. Drillhole geology is recorded for lithology, alteration, mineralization, and structure. Drillhole recovery is recorded for sampled intervals and averages 96% within mineralized zones. Lab density measurements are collected by pycnometer on selected sampled intervals. Selective geochemical sampling is completed on intervals of potentially mineralized material. Logged mineralized intervals are sampled for geochemical assay at nominal 1.5 m intervals based on changes in lithology, alteration, mineralization, and structure.

1.11 Sample Preparation, Analyses and Security

Since initiating drilling on the Property in January 2020, Arizona Metals has maintained a consistent system for the sample preparation, analysis and security of all surface samples and drill core samples, including the implementation a QA/QC protocol. The current MRE is limited to drilling data collected by Arizona Metals since the acquisition of the Property until June 17, 2025.

Since 2020, all samples have been shipped to ALS Limited (ALS) in Tucson, Arizona, USA for sample preparation and transferred for analysis at the ALS laboratory in North Vancouver, BC, Canada. The ALS Tuscon and North Vancouver facilities are ISO/IEC 17025 certified. Samples are dried, weighed, and crushed to at least 70% passing 2 mm, and a 250 g split is pulverized to at least 85% passing 75 µm. Base metals and silver are analyzed using an intermediate level four-acid digestion with an inductively coupled plasma (ICP) finish. Over-limit analyses for copper, lead, zinc (>100,000 ppm), and silver (>200 ppm), are re-assayed using an ore-grade four-acid digestion with an ICP finish. Gold is assayed by 30-gram fire assay with atomic absorption (AA) spectroscopy finish. Over-limit analyses for gold (>10 ppm) are re-assayed using a 30-gram fire assay with a gravimetric finish. Control samples comprising certified reference samples, blank samples, and duplicates are systematically inserted into the sample stream and analyzed as part of the Company’s QA/QC protocol. ALS is independent of Arizona Metals, the QPs, and SGS Geological Services.

1.12 Data Verification

Data verification procedures that were carried out and completed and documented by the Authors for this technical report, include verification of all drill data collected by Arizona Metals during their 2020 to 2025 drill programs, as of the effective date of the MRE. Verifications were carried out on assay data in the drill sample database, drill hole locations, down hole surveys, lithology, SG and topography information. The database is considered of sufficient quality to be used for the current MRE.

Site visits to the Property were conducted to verify and validate the data collection protocols used and the geological interpretations related to the mineralization style, deposit type, lithology and mineralization modelling, and the mineral resource estimate.

1.13 Metallurgical Testing and Mineral Processing

A metallurgical test work program was completed in July 2025, at SGS Minerals Burnaby, British Columbia. An Early-Stage Metallurgical evaluation on composites from four (4) zones; K-MET-01, 02, 03 and 04 was completed in April 2023. Further metallurgical testwork was performed on a master composite that represented the overall mineralization of the orebody based on drilling completed up to May 2023. The scope of the test work included chemical and mineralogical assessment of the gold, sulphide and mercury content, optimization of the rougher-cleaner flotation scheme to upgrade the copper and zinc content into respective concentrates, rejection of the arsenic content present as arsenopyrite from the zinc concentrate through selective cleaner flotation and the recovery of gold to either copper-lead concentrate, and a separate pyrite arsenopyrite concentrate. The extraction of gold from the pyrite concentrate through cyanide leaching, and with or without Albion oxidative pretreatment. The relevant metallurgical data and findings from July 2025 programs are provided in this Technical Report.

Based on client's instruction, 5431 assays from holes KM-23-99 were created into three (3) metal clusters; Cu, Zn-Pb and Zn-Pb-Cu. From these assays, a total of 3,201 assays were selected for a specific metal cluster, and the resulting percentages were utilized to create the master composites for the test program. Two (2) master composites; MC-1A and 1B were produced using the three (3) composites (K-MET-01 to 03) representing copper, Zn-Pb and Zn-Pb-Cu based on a blend proportion of 27% Cu, 36.3% Zn-Pb and 36.7% Zn-Pb-Cu. The first master composite (MC-1A) was used until flotation test MC-F15 and the second master composite (MC-1B) was used for the remaining testwork. The head assays of the MC-1A and 1B blends were comparable suggesting reasonable continuity between the composite blends.

Modal mineralogy on the master composite 1A identified the sulphide gangue minerals as pyrite (23.5%) and arsenopyrite (3.9%). Non-sulphide minerals consisted primarily of silicates such as quartz (18.3%) and plagioclase (1.1%), carbonates such as dolomite / ankerite (17.3%) and siderite (5%), and phyllosilicates such as chlorites (13%), micas (2.6%), and clays (1.1%). Pyrite accounted for 72.1% of the sulphur content, followed by 12.8% as sphalerite, 9.9% as chalcopyrite and 4.5% in arsenopyrite. The primary copper bearing mineral was chalcopyrite (94.2%) with 4.6% total copper, arsenopyrite accounted for most of the arsenic content at 98.7% Zn was present entirely as sphalerite and lead as galena.

The visible gold deportment (grains >0.5 µm) within the master composite shows that 43.5% as native, 48.4% as electrum, with minor amounts of gold-tellurides in petzite (3.8%) and calaverite (3.3%).

Mercury was observed within a HgTe mineral, coloradoite at a grain size P80 of 14 µm. Coloradoite was found associated with tellurium phases (31%), pyrite phases (23%), and other sulphides such as chalcopyrite, galena, and arsenopyrite (12%). None of the scanned coloradoite was observed in the sphalerite mineral.

A series of open batch rougher and cleaner flotation tests were conducted on the master composite blend, followed by a single locked cycle test. A copper-lead concentrate recovered 88% of the copper at a grade of 27.1% Cu and a zinc concentrate recovered 76% of the zinc at 58.7% Zn grade in the locked cycle test. The copper-lead concentrate also recovered 50% Pb, 21% Au, and 67% Ag at grades of 3.3% Pb, 7.8 g/t Au, and 528 g/t Ag. Copper Lead separation testwork was not performed in this program. The arsenic and mercury grades of cleaner concentrates were high, assaying 0.98% As and 68 g/t Hg in the Cu-Pb concentrate and 1.31% As and 256 g/t Hg in the zinc concentrate. The pyrite concentrate recovered 62% of the gold into a mass yield of 31%. The gold was expectedly refractory and direct intensive cyanidation only recovered 14% of the gold.

Albion oxidative pretreatment of the pyrite concentrate produced positive results, with cyanidation of the Albion residue recovering 98% of the contained gold. Albion reagent consumptions were high as the test was of scoping nature and conditions were not optimized.

1.14 Mineral Resources Estimate

Completion of the current MRE involved the assessment of a drill hole database, which included all data for surface drilling completed through to June 17, 2025. Completion of the current MRE also included updated three-dimensional mineral resource models (resource domains), a 3D topographic surface model, 3D models of historical underground workings, and available written reports. The Inverse Distance Squared calculation method restricted to mineralized domains was used to interpolate grades for Au (g/t), Ag (g/t), Cu (ppm), Pb (ppm) and Zn (ppm) into a block model for the Kay Deposit. The MRE for the Kay Deposit takes into consideration that the Kay Deposit may be mined by underground mining methods.

The reporting of the current MRE complies with all disclosure requirements for Mineral Resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects. The classification of the MRE is consistent with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definitions). In completing the updated MRE, the Author uses procedures and methodologies that are generally consistent with industry standard practices, including those documented in the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019 CIM Guidelines).

To complete the current MRE for the Kay Deposit, a validated drill hole database comprising a series of comma delimited spreadsheets containing surface diamond drill hole information was provided by Arizona Metals. The database included hole location information, down-hole survey data, assay data for all metals of interest, lithology data and density data. The data in the geochemistry / assay tables included data for the elements of interest including Ag (g/t), Au (g/t), Pb (ppm) and Zn (ppm) and Cu (ppm). After review of the database, the data was then imported into GEOVIA GEMS version 6.8.3 software for statistical analysis, block modeling and resource estimation. No errors were identified when importing the data. The data was validated in GEMS, and no erroneous data, data overlaps or duplication of data was identified.

The updated database provided by Arizona Metals for the MRE included data for 234 surface diamond drill holes, completed on the Property, totalling 133,912 m. The database totals 11,533 assay intervals representing 14,066 m of drilling. The average assay sample length is 1.21 m.

For the current MRE, in collaboration with Arizona Metals, the authors constructed two (2) three-dimensional resource models and four (4) lithology models for the Kay deposit in Leapfrog Geo version 2025.1.0.

Host rock lithology models were constructed incorporating drilling data, surface mapping, and structural interpretations in addition to SGS field and drill core observations. Lithology models comprise the Hangingwall Mafic Sequence (MVS), Felsic Volcanic Sequence (FVS), Graphite-rich Horizon (GH), and the Mineralization Horizon (MIN-Horizon). The MIN-Horizon model was constructed using the Leapfrog Geo Vein tool from assays greater than 0.5% CuEq and was used to establish the bounding limits of the subsequently constructed resource models. The MIN-Horizon model is consistent with the interpretation that within the property-scale isoclinal folding the sulphide lenses are affected by steeply plunging tight folds (parasitic S-folds).

The Kay drillhole database and drill core was reviewed to evaluate the geological continuity and internal variability with respect to mineralization styles, metal zonation patterns, and density. The deposit displays complex internal variability of mineralization style, density, and relative metal distributions. Mineralization within the MIN-Horizon model was sub-domained using CuEq grade as a proxy for mineralization style and density. Two (2) resource models were constructed: a semi-massive to massive sulphide, high-grade domain (MIN-HG) and a stringer sulphide, low-grade domain (MIN-LG), to domain appropriate density and capping values in the estimation process.

The MIN-HG and MIN-LG resource models were constructed using the Leapfrog Geo Indicator RBF numerical modelling tool with a structural trend based on the folded MIN-Horizon model. The MIN-HG resource model was established from assay intervals above 1.5% CuEq constrained by the MIN-Horizon model. The MIN-LG resource model was established from assay intervals above 0.5% CuEq, outside of the MIN-HG model, and constrained by the MIN-Horizon model.

A digital elevation surface model (LiDAR) was provided for the Property area. All 3D resource models were clipped to topography and limited to the Property boundary.

Mineralization in the Kay sulphide lenses resource models extends for up to 400 m along strike and up to 850 m vertically (900 m down plunge). The mineralization horizon in general dips at 73º towards 260º (W) with local variations in strike and dip resulting from steeply plunging tight parasitic folds. The principal plunge direction of the sulphide lenses is 68º towards 300º (WNW) and appears to be influenced in part by steeply plunging tight parasitic folds.

The Author has reviewed the resource models on plan view and in section view and in the Author's' opinion the models are well constructed and appear to be representative of the mineralization identified on the Property and the distribution of the Cu-Au-Zn-Pb-Ag mineralization within these sulphide lenses. Models were reviewed by Arizona Metals during the modelling process and refined by SGS before final resource estimation. Models have been extended beyond the limits of the current drilling for the purpose of providing guidance for continued exploration. However, the extension of the mineral resource beyond the limits of drilling is limited by the search radius during the interpolation procedure (a maximum of 110 m in the plunge direction past drilling).

1.14.1 Mineral Resource Statement

Highlights of the Project Mineral Resource Estimate are as follows:

- The underground MRE includes 9.28 million tonnes grading 1.39 g/t Au, 27.6 g/t Ag, 0.97% Cu, 0.33% Pb, and 2.39% Zn in the Indicated category, and 0.86 million tonnes grading 1.06 g/t Au, 15.4 g/t Ag, 0.87% Cu, 0.20% Pb, and 1.68% Zn in the Inferred category, at a base-case cut-off grade of 1.00% CuEq.

Table 1.1: Kay Property Mineral Resource Estimate, June 17, 2025

Tonnes (Mt)	Average Grade						Contained Metal					
	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	CuEq (%)	Au (koz)	Ag (koz)	Cu (Mlbs)	Pb (Mlbs)	Zn (Mlbs)	CuEq (Mlbs)
Indicated												
9.28	1.39	27.6	0.97	0.33	2.39	3.18	415	8,253	197.9	67.3	490.1	650.6
Inferred												
0.86	1.06	15.4	0.87	0.20	1.68	2.44	29	423	16.4	3.8	31.8	46.1

Kay Deposit Mineral Resource Estimate Notes:

1. *The effective date of the Kay Project Mineral Resource Estimate (MRE) is June 17, 2025. This is the close-out date for the final mineral resource drilling database.*
2. *The mineral resource was estimated by Allan Armitage, Ph.D., P. Geo. of SGS Geological Services, an independent Qualified Person as defined by NI 43-101. Armitage conducted site visits to the Kay Deposit on two (2) occasions, on October 25-26, 2023, and April 7-8, 2024. The mineral resource was peer reviewed by Ben Eggers, MAIG, P.Geo. of SGS Geological Services, an independent Qualified Person as defined by NI 43-101. Eggers conducted a site visit to the Kay Property on May 30, 2025.*
3. *The classification of the current MRE into Indicated and Inferred mineral resources is consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves.*
4. *All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.*
5. *All mineral resources are presented undiluted and in situ, constrained by continuous 3D wireframe models (considered mineable shapes), and are considered to have reasonable prospects for eventual economic extraction.*
6. *Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that most Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
7. *The Kay Project MRE is based on a validated drill hole database which includes data from 234 surface diamond drill holes completed between 2020 and May 2025. The drilling totals 133,912 m (including wedge holes). The resource database totals 11,533 assay intervals representing 14,006 m of data.*
8. *Grades for Au, Ag, Cu, Pb and Zn are estimated for each mineralization domain using 1.50 m capped composites assigned to that domain. To generate grade within the blocks, the inverse distance squared (ID2) interpolation method was used for all domains.*
9. *Average density values were assigned to each domain based on a database of 2,307 samples.*
10. *Based on the size, shape, and orientation of the deposit, it is envisioned that the deposits may be mined using underground bulk mining methods such as Longhole Stoping. The MRE is reported at a base case cut-off grade of 1.00% CuEq. The mineral resource grade blocks are quantified above the base case cut-off grade and within the constraining mineralized wireframes (considered mineable shapes).*
11. *The underground base case cut-off grade of 1.00% CuEq considers metal prices of $4.10/lb Cu, $1.00/lb Pb, $1.35/lb Zn, $2,200/oz Au and $26/oz Ag, assumed metal recoveries of 92% for Cu, 76% for Pb, 85% for Zn, 76% for Au and 75% for Ag, a mining cost of US$49.00/t rock and processing, treatment and refining, transportation and G&A cost of US$29/t mineralized material.*
12. *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*

1.15 Mineral Reserve Estimate

This Preliminary Economic Assessment (PEA) of the Kay Mine Project is based on Indicated and Inferred Mineral Resources. Because of the inclusion of Inferred Resources, it is not applicable to determine Mineral Reserves at this stage of the project. Economic zones will be classified as mineralized material only.

1.16 Mining Methods

Kay Mine is planned as a mechanized long hole open stoping underground mine. The milling rate is planned at 0.7 Mtpa with a ramp-up period of nine (9) months during the first operational period. The mill will run for 10 years.

1.16.1 Underground

The underground operation consists of a single mine accessed via one surface portal located south of the surface infrastructure area. The selected mining method consists of long hole open stoping (LHOS), specifically sublevel transverse stoping and sublevel longitudinal stoping.

The life-of-mine (LOM) for the underground operation is estimated at 12.5 years, encompassing construction and development, pre-production, and full production phases. Of this total, the underground mine is expected to operate in production for approximately 10 years, including a nine (9) months ramp-up period. The pre-production phase is anticipated to last approximately two and a half (2½) years following portal construction, allowing sufficient underground development to be completed to support sustained full production.

The underground mine is planned to supply the mill feed at an average rate of 1,918 tonnes per day of mineralized material. The mine plan includes the excavation of approximately 39.3 km of lateral development and 4.0 km of vertical development.

A total of approximately 6.55 million tonnes (Mt) of mineralized material is expected to be mined at average diluted grades of 1.01% Cu, 2.67% Zn, 1.60 g/t Au, and 29.07 g/t Ag. The primary production fleet will consist of 15-t diesel-powered load–haul–dump (LHD) units in combination with 45-t underground haul trucks for the transport of all mined material.

1.16.2 Underground Geomechanics

A preliminary underground geomechanical assessment was carried out to support the selection of the mining method and related design parameters. The study was based on available geological data, historical structural information, and a targeted geotechnical photo-logging program of selected drill core.

Mineralization is primarily hosted within felsic metavolcanic rocks associated with a volcanogenic massive sulphide (VMS) system. Photo-logging and regional interpretation show a moderate to good quality rock mass characterized by a single dominant foliation-parallel joint set. Rock mass quality, assessed using the NGI Q′ classification system, generally ranges from about 6 to 15, depending on lithology and percentile considered.

No laboratory test results were available at this stage. Intact rock strength parameters were estimated from published values for comparable lithologies. These parameters are considered appropriate for a preliminary stability assessment.

Empirical stability and dilution analyses using the Mathews–Potvin stability method and the ELOS approach show that transverse longhole open stoping is suitable. Recommended stope dimensions are approximately 13 m of strike length, 5 to 15 m in width, and 25 m vertically. Most stope surfaces are in stable to transitional zones, and dilution is likely to be limited.

Ground support and backfill requirements were established using standard empirical design methods. Cemented rockfill is considered for most stopes.

1.16.3 Hydrogeology

A review of historical reports provided by Arizona Metals Corp indicates that no site-specific hydrogeological data, such as the hydraulic conductivity of the bedrock, is available to support calculations of groundwater inflow into the existing mine workings. To assess the presence of major aquifers in the project area, publicly available hydrogeological information was examined.

Regional hydrogeological information identifies two (2) principal aquifer systems in the area: basin-fill alluvial deposits associated with the Agua Fria River and sedimentary rock formations (conglomerates). While conglomerates are not present on the Kay Mine property, the site intersects the Agua Fria River alluvial plain, which consists of unconsolidated sand and gravel and may represent a potential source of groundwater inflow if hydraulically connected to the underground mine through permeable structures. At present, no data are available to confirm such connectivity, and this interaction remains conceptual.

No historical information was identified regarding dewatering pumping rates from the former Kay Mine operations. No information regarding the water quality of the flooded underground workings was identified during the review of historical reports.

The recommended path forward includes a comprehensive hydrogeological investigation during the next phase of engineering studies. This program should focus on characterizing groundwater flow conditions, defining bedrock permeability, particularly along geological structures, and assessing potential hydraulic connectivity with the Agua Fria River alluvium.

1.17 Recovery Methods

The proposed process plant design for the Kay Mine Project is based on a standard metallurgical flowsheet to treat Cu-Pb-Zn mineralized material. The flowsheet is based on metallurgical test work, industry standards and conventional unit operations.

The process plant is designed to nominally treat 0.7 Mtpa of fresh rock from the underground and includes the following unit operations:

- Comminution circuit (primary jaw crusher, secondary cone crusher and ball mill) to produce a primary grind size of P80 of 60 µm.

Flotation circuit consisting of roughers, rougher-regrind circuit followed by cleaner flotation for recovery of separate Cu-Pb and zinc concentrates. Rougher flotation is only considered for the pyrite concentrate recovery.

- Concentrates dewatering including thickening and filtration circuits.
- Albion oxidative leaching at a leach residence time of 72 hours for processing the refractory gold from pyrite concentrate.
- Carbon in Leach (CIL) is considered for gold recovery for the Albion treated product.
- Pre-leach thickening.
- Cyanide leaching and carbon adsorption via a Leach-CIP carousel circuit. Leach residence time of 48 hours to achieve optimal gold extraction.
- Carbon elution via 10-t Split Pressure Zadra circuit.
- Carbon handling and regeneration.
- Electrowinning and smelting to produce doré.

- Cyanide destruction of Leach-CIP tailings using SO_2 / O_2 process to produce weak acid dissociable (WAD) cyanide levels of less than 5 ppm.
- Tailings pumping to a tailings' storage facility.
- Air and oxygen circuits.
- Water systems (domestic water, raw water, gland seal water and process water).

1.18 Project Infrastructure

The Project infrastructure is designed to support the operation of an underground mine and mineral processing facility with a nominal throughput of 0.7 Mtpa operating on a continuous basis. The infrastructure layout has been developed in consideration of local site conditions, topography, operational requirements, environmental management objectives, and overall construction efficiency.

The proposed infrastructure includes water management infrastructure, mine infrastructure, process infrastructure, and supporting surface facilities required to sustain underground mining and mineral processing operations throughout the life of mine.

Water management infrastructure will include process water and fire water systems, potable water treatment facilities, contact water collection ponds, sewage treatment systems, and an effluent treatment plant. Surface water management systems, including diversion ditches, culverts, runoff conveyance infrastructure, and sediment control measures, will be implemented throughout the site to manage contact and non-contact water.

Mine infrastructure will include the underground mine access area, mine maintenance facility and warehouse, mine administration building, mine dry, explosives storage facilities, and the Dry Stack Tailings Storage Facility (DSTSF). The DSTSF will store filtered tailings generated from the process plant and will incorporate drainage and runoff management systems designed to support long-term operational stability and water management objectives.

Process infrastructure will include the mineral processing facility, assay and metallurgical laboratory, reagent storage facilities, compressor room, crushed mineralized material stockpile, and run-of-mine (ROM) pad. The process plant is designed to support the production of copper and zinc concentrates, as well as gold produced as doré.

Supporting infrastructure will include electrical substations and power distribution systems, emergency backup power generation systems, communications infrastructure, fuel storage and distribution facilities, kitchen and lunchroom facilities, and site security infrastructure with controlled site access.

The Project infrastructure will be connected through a network of access roads, service roads, haul roads, and DSTSF access roads designed to support the safe and efficient movement of personnel, materials, and mining equipment throughout the site.

1.19 Market Study and Contract

The Kay Mine Project is expected to produce gold and silver in doré bars, as well as copper and zinc concentrates containing payable copper, zinc, gold, and silver. Metal price assumptions for the PEA were established based on review of historical prices, long-term broker consensus forecast and pricing assumptions used by industry peers. The financial analysis for the Kay Mine Project considered a copper price of $4.70/lb, a zinc price of $1.27/lb, a gold price of $3,100/oz and a silver price of $38.00/oz. An exchange rate of 1.34 Canadian dollars per US dollar (1.34 CAD/USD) was used for this PEA.

There are no refining agreements or sales contracts currently in place for the Project that are relevant to this Technical Report.

1.20 Environmental Studies, Permitting and Social or Community Impact

Baseline studies conducted for the EPO and EA included a Class III cultural resources inventory and a biological evaluation (BE). The Class III inventory, completed between June and August 2025, identified one previously recorded site, 27 newly recorded sites, and 148 isolated occurrences. Of the 28 sites documented, six (6) are recommended eligible for listing on the NRHP, while the remaining 22 sites and all isolated occurrences are not recommended eligible due to a lack of significance.

The BE evaluated special-status species within the project area, including those protected under the ESA, the BGEPA, and BLM sensitive species. The analysis identified the threatened, yellow-billed cuckoo, as well as two (2) BLM sensitive species (the lowland leopard frog and Sonoran Desert tortoise) as present in the project area.

Additional studies required to support permit applications include groundwater characterization, geochemical characterization of tailings and waste rock, an air quality impact assessment, and a socioeconomic evaluation.

At present, AMC does not hold permits for the project. The following is a table of permits that will likely be required:

Table 1.2: Anticipated Permitting Requirements

Permit Type	Agency
BLM Plan of Operations	Bureau of Land Management (BLM)
Arizona Mined Land Reclamation Permit	Arizona State Mine Inspector (ASMI)
Aquifer Protection Permit	Arizona Department of Environmental Quality (ADEQ)
Air quality permit (Class II)	ADEQ Air Quality Permits Section
AZPDES Multi-Sector General Permit (MSGP) for mining	ADEQ Surface Water Section, Stormwater and General Permits Unit
AZPDES Individual Permit	ADEQ
Notification of Commencement of Operation	U.S. Department of Labor Mine Safety and Health Administration (MSHA)
MSHA Identification Number and MSHA Coordination	MSHA
Hazardous Waste Identification Number	U.S. Environmental Protection Agency (EPA)
Clean Water Act Section 404 Nationwide Permit	Army Corps of Engineers
Explosives User Permit	Bureau of Alcohol, Tobacco, Firearms and Explosives
Radio License	Federal Communications Commission
Notice of Start-up of Mine Operations	Arizona State Mine Inspector
Hazardous Waste, Treatment, Storage and Disposal Permit	ADEQ Hazardous Permits Unit
Special Waste Identification Number	ADEQ Solid Waste Unit
Notice of Intent to Drill, Deepen, or Modify a Monitor / Piezometer / Environmental Well	Arizona Department of Water Resources, Groundwater Permitting and Wells Section
Fire Safety Inspection and Permit	Arizona Office of the State Fire Marshal
Yavapai County Mining / Metallurgical Use Exemption	Yavapai County Chief Zoning Inspector

The Kay Mine Project is located adjacent to Black Canyon City in Yavapai County, approximately 69 km (43 miles) north of Phoenix. The nearby community has a population of approximately 5,600 and provides basic services such as fuel, food, and housing. Because many residences have direct views of the project area, the site is highly visible to the local community.

AMC has implemented community engagement efforts, including a public outreach webpage, sponsorship of local initiatives such as the Black Canyon City Heritage Park, and development of an External Relations Plan. This plan outlines strategies for consistent communication, stakeholder engagement, community partnerships, participation in community events, and social media outreach. AMC has also coordinated with federal, state, and county agencies, including the BLM, regarding exploration activities. Although no specific Tribal coordination has occurred to date, consultation would be expected as part of the federal review process.

Potential impacts to the local community are primarily related to the project's proximity to residential areas and may include visual effects, temporary disturbances from exploration and mining activities, and increased traffic along access routes. These impacts may be reduced through ongoing communication with residents, implementation of impact minimization measures, and careful siting of infrastructure away from populated areas. The project may also provide economic benefits, including local employment opportunities and support for the regional mining economy.

1.21 Capital and Operating Costs

Life-of-mine project capital costs are estimated to total USD 731 million consisting of the following three (3) distinct phases:

- Initial Capital Expenditure – This phase includes all costs to develop the property with a process plant designed to nominally treat 0.7 Mtpa of fresh rock. Initial capital costs total $609 million (including $84 million of contingency). The initial capital excludes pre-production net revenue of $18 million. The construction phase expends over 30-month design, construction, pre-production and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations and the underground mining development. Sustaining capital costs are estimated to be USD 87 million.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine for 25 years after operations. Closure costs are estimated to be a total of USD 28 million. In addition, an annual surety bond has been included as financial assurance, estimated at USD 7 million, bringing the total closure cost to USD 35 million.

The capital cost estimate has been developed to support the economic analysis of the project and is consistent with the level of accuracy expected at the current stage of study, typically within the 30% +50% range for a Preliminary Economic Assessment (PEA).

The Capital and sustaining expenditures are summarized in Table 1.3 according to the level 1 work breakdown structure (WBS). Expenditures are presented in US dollars.

Table 1.3: Capital Expenditure Summary

Capital Expenditures (k USD)	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
100 – Infrastructure	41,953	-	41,953
200 – Power and Electrical	23,192	-	23,192
300 – Water Management	20,619	-	20,619
400 – Surface Operations	21,691	-	21,691
500 – Mining	84,023	87,205	171,228
600 – Process Plant	185,059	-	185,059
700 – Construction Indirect	87,754	-	87,754
800 – General Services / Owner's Cost	15,131	-	15,131
900 – Pre-production, Start-up, Comm.	45,130	-	45,130
990 – Contingency	84,126	-	84,126
Total	**608,678**	**87,205**	**695,883**

A salvage value of USD 5.34 million was estimated for the major process plant equipment. This residual value is excluded from Capital Expenditure.

The operating costs include mining, processing, general services and administration ("G&A"), royalties, concentrates transportation and refining, and power cost which is included within each area. The average LOM operating cost is $138.47/t milled. Operating Costs are summarized in Table 1.4.

Table 1.4: Operating Cost

Item	Unit Cost ($/t milled)
Underground Mining Cost	60.24
Processing	48.36
General Services & Administration	11.49
Total Direct Cost	**120.09**
Royalty Cost	-

Item	Unit Cost ($/t milled)
Transport and Refining	18.38
Total OPEX Cost	**138.47**

1.22 Economic Analysis

The PEA is preliminary in nature and includes Inferred Mineral Resources, which are considered too geologically speculative to be categorized as Mineral Reserves with economic considerations. Therefore, there is no certainty that the PEA will be realized.

All economic figures are presented in real terms (i.e., excluding the effects of inflation) and are denominated in 2026 Q1 United States dollars (USD), unless otherwise indicated. The economic model excludes any Project debt or equipment financing.

The principal economic metrics used to evaluate the Project include net undiscounted after-tax cash flow, net discounted after-tax cash flow (NPV), internal rate of return (IRR), and payback period. The economic analysis was conducted using a discount rate of 5% and long-term metal price assumptions of copper at $4.70/lb Cu, zinc at $1.27/lb Zn, gold at $3,100/oz Au, and silver at $38.00/oz Ag. Cash flows were discounted from the start of construction, and all costs before this period were considered as sunk costs.

A summary of the Project economic results is presented in Table 1.5. The total after-tax cash flow over the Project life is $259M and NPV 5% is $40M pre-tax and $-6M after-tax. The after-tax Project cash flow results in a 7.5-year payback period from the commencement of commercial operations with an IRR of 6.0% pre-tax and 4.9% after-tax.

Table 1.5: Project Economic Results Summary

Assumptions	Unit	Base Case
Copper Price	$/lb	4.70
Zinc Price	$/lb	1.27
Gold Price	$/oz	3,100
Silver Price	$/oz	38.00
Exchange Rate	USD:CAD	1.34
Fuel Price	$/L	0.73

Assumptions	Unit	Base Case
Mine Life	yrs	10
Production Summary (Life-of-Mine)		
Underground		
Mineralized Material Mined	Mt	6.55
Copper Grade	%	1.01
Zinc Grade	%	2.67
Gold Grade	g/t	1.60
Silver Grade	g/t	29.07
Mill Feed		
Average Milling Throughput	Mtpa	0.7
Average Daily Throughput	tpd	1,918
Total Mill Feed Tonnes	Mt	6.55
Copper Head Grade	%	1.01
Zinc Head Grade	%	2.67
Gold Head Grade	g/t	1.60
Silver Head Grade	g/t	29.07
Contained Copper	Mlbs	146.1
Contained Zinc	Mlbs	385.9
Contained Gold	koz	336.8
Contained Silver	koz	6,119.2
Average Copper Recovery (%)	%	92
Average Zinc Recovery (%)	%	80
Average Gold Recovery (%)	%	86
Average Silver Recovery (%)	%	85
Payable Copper	Mlbs	127.4
Payable Zinc	Mlbs	292.7
Payable Gold	koz	258.1
Payable Silver	koz	4,712.2
Operating Costs (LOM average)		
Mining Cost - UG	$/t milled	60.24
Processing Cost	$/t milled	48.36

Assumptions	Unit	Base Case
G&A Cost	$/t milled	11.49
Total Direct Cost	**$/t milled**	**120.09**
Royalty	$/t milled	-
Refining & Transport Cost	$/t milled	18.38
Total Operating Cost	**$/t milled**	**138.47**
Capital Costs		
Initial Capital	M US$	609
Sustaining Capital	M US$	87
Closure Costs*	M US$	35
Total Capital Cost	M US$	**731**
Construction Working Capital	M US$	19
Salvage Value	M US$	5
Financial Evaluation Pre-Tax		
Free Cash Flow	M US$	332
Pre-Tax NPV 5%	M US$	40
Pre-Tax IRR	%	6.0
Payback	yrs	7.3
Financial Evaluation After-Tax		
Free Cash Flow	M US$	259
After-Tax NPV 5%	M US$	-6
After-Tax IRR	%	4.9
Payback	yrs	7.5

**Note: Include surety bond as financial assurance.*

A sensitivity analysis was conducted on the base case pre-tax and after-tax Cash Flow, NPV (5%), IRR, and payback of the project using the following variables:

- Metal prices.
- Operating cost.
- Initial capital cost.

Table 1.6 to Table 1.8 summarize pre-tax and after-tax sensitivity analysis. Figure 1.1 to Figure 1.4 show the sensitivity of the After-Tax Total Free Cashflow, NPV 5%, IRR and Payback.

Table 1.6: Metal Price Sensitivity

Metal Price	Pre-Tax				After-Tax			
	NPV 0% (M USD)	NPV 5% (M USD)	IRR (%)	Payback Period (yrs)	NPV 0% (M USD)	NPV 5% (M USD)	IRR (%)	Payback Period (yrs)
70%	-253	-239	-6.0%	-	-269	-369	-6.5%	-
80%	-58	-226	-1.3%	-	-81	-241	-1.8%	-
90%	137	-93	2.6%	8.5	105	-114	2.0%	8.7
Base Case	**332**	**40**	**6.0%**	**7.3**	**259**	**-6**	**4.9%**	**7.5**
110%	527	173	9.0%	6.3	405	97	7.4%	6.6
120%	722	306	11.7%	5.6	551	198	9.7%	5.9
130%	917	439	14.3%	5.1	696	299	11.9%	5.3

Table 1.7: OPEX Sensitivity

OPEX	Pre-Tax				After-Tax			
	NPV 0% (M USD)	NPV 5% (M USD)	IRR (%)	Payback Period (yr)	NPV 0% (M USD)	NPV 5% (M USD)	IRR (%)	Payback Period (yr)
70%	556	193	9.4%	6.1	427	113	7.8%	6.4
80%	480	142	8.3%	6.5	371	73	6.8%	6.8
90%	406	90	7.1%	6.9	315	34	5.8%	7.1
Base Case	**332**	**40**	**6.0%**	**7.3**	**259**	**-6**	**4.9%**	**7.5**
110%	258	-10	4.8%	7.7	203	-45	3.9%	7.9
120%	187	-59	3.5%	8.1	148	-85	2.8%	8.4
130%	115	-108	2.2%	8.7	84	-128	1.6%	8.9

Table 1.8: Initial CAPEX Sensitivity

Initial CAPEX	Pre-Tax				After-Tax			
	NPV 0% (M USD)	NPV 5% (M USD)	IRR (%)	Payback Period (yr)	NPV 0% (M USD)	NPV 5% (M USD)	IRR (%)	Payback Period (yr)
70%	514	211	11.3%	5.8	442	165	10.1%	6.0
80%	454	154	9.2%	6.3	381	108	8.1%	6.5
90%	393	97	7.5%	6.8	320	51	6.4%	7.0
Base Case	**332**	**40**	**6.0%**	**7.3**	**259**	**-6**	**4.9%**	**7.5**
110%	271	-17	4.6%	7.7	198	-63	3.5%	8.0
120%	210	-74	3.4%	8.1	137	-120	2.3%	8.5
130%	149	-131	2.3%	8.6	76	-177	1.2%	9.1

Figure 1.1: After-Tax Total Free Cash Flow Sensitivity (M$)

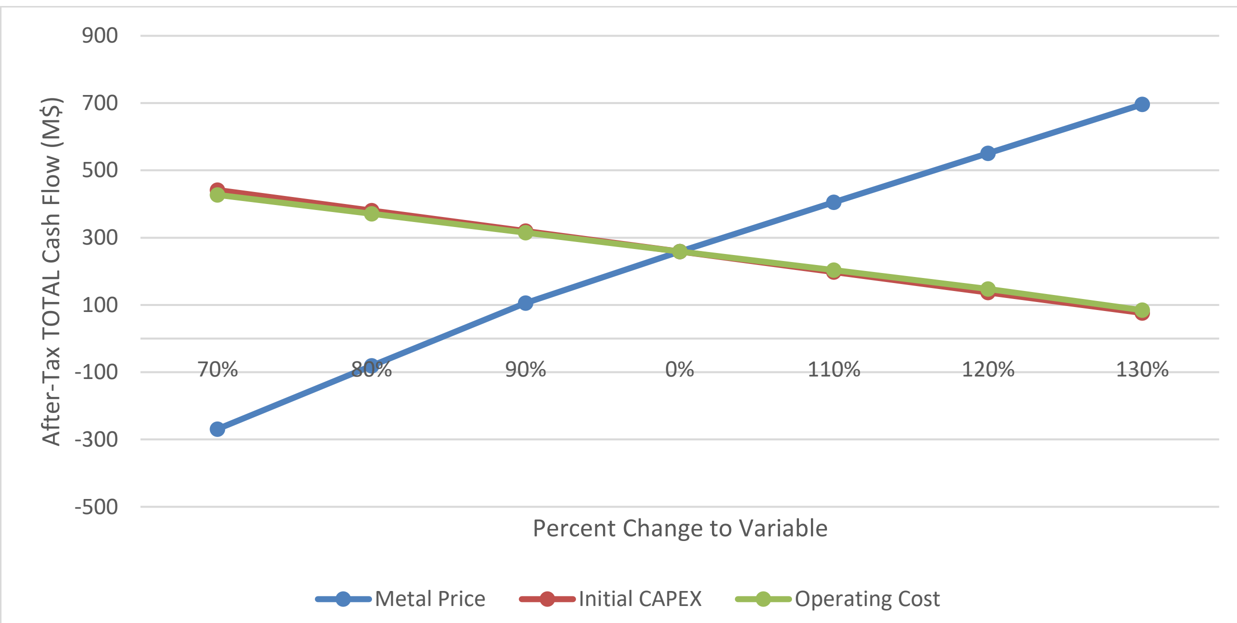


Figure 1.2: After-Tax NPV (5%) Sensitivity (M$)

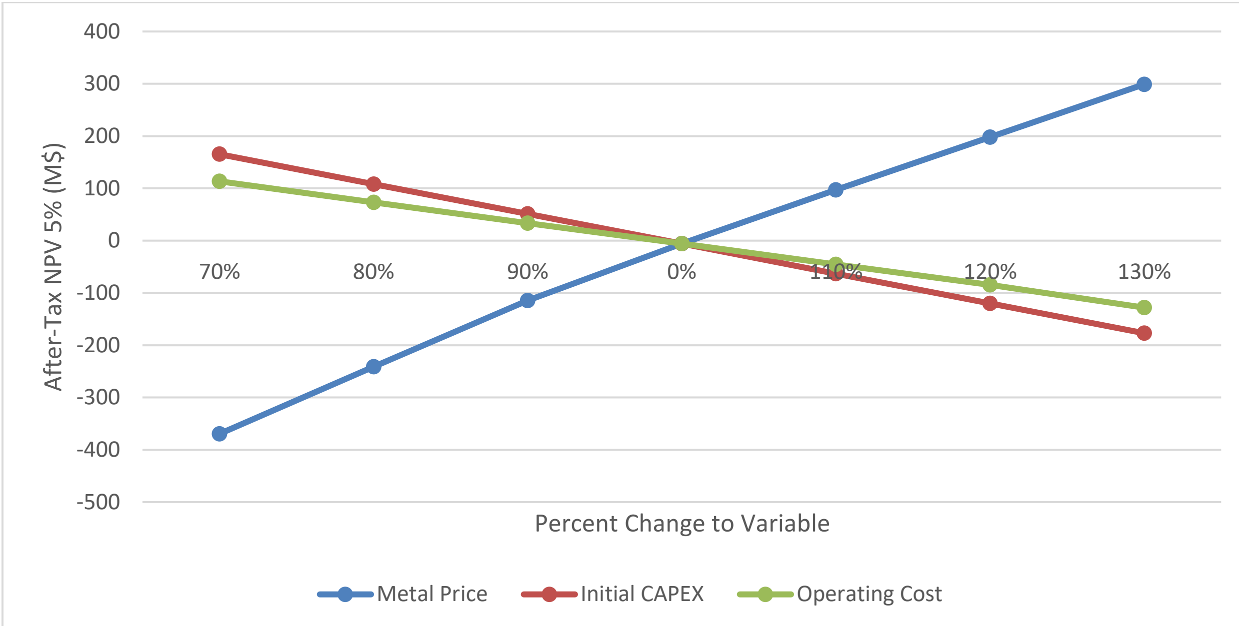


Figure 1.3: After-Tax Internal Rate of Return Sensitivity

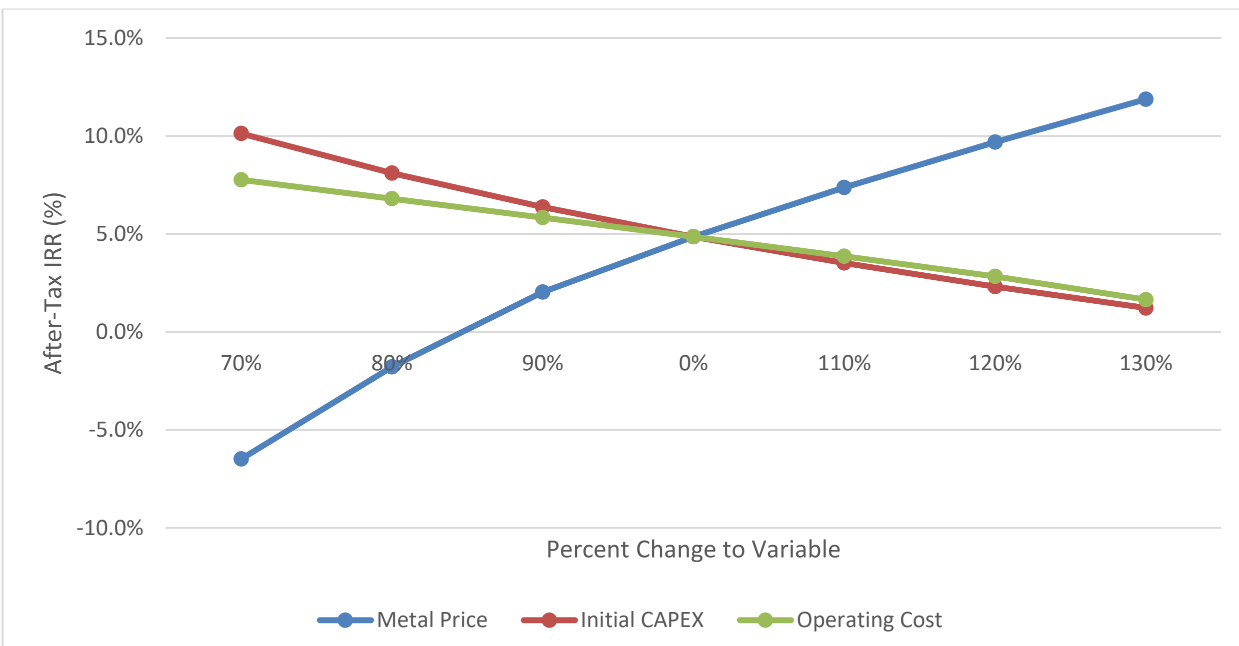
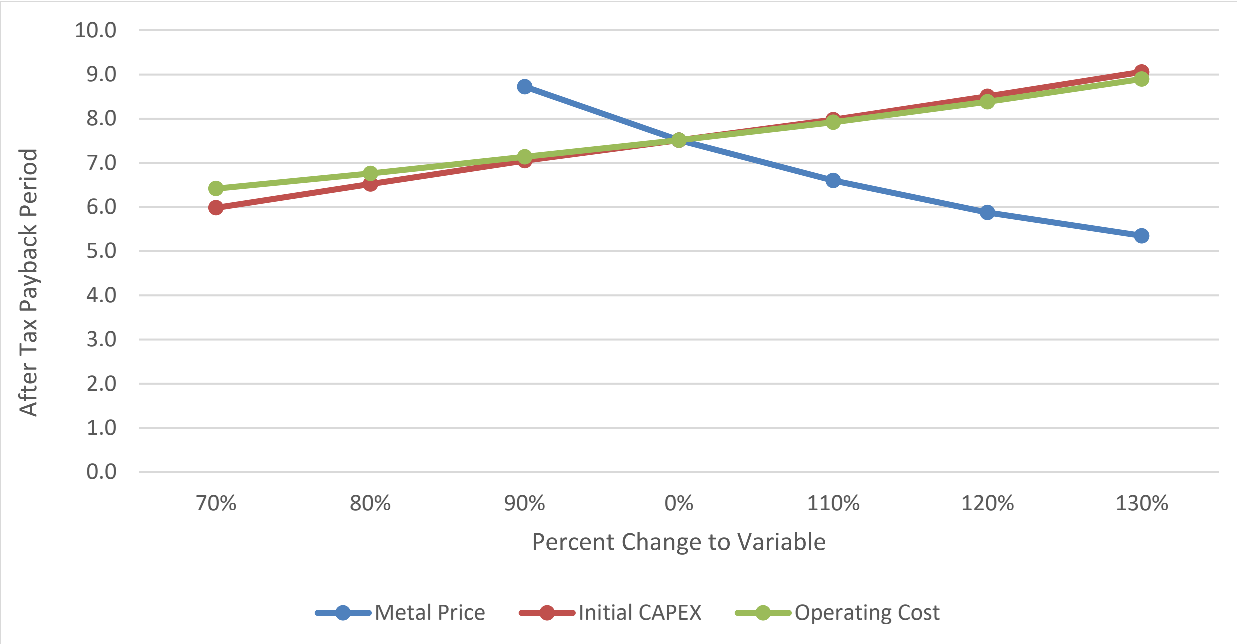


Figure 1.4: After-Tax Payback Period Sensitivity



1.23 Adjacent Properties

There is no information on properties adjacent to the Property necessary to make the technical report understandable and not misleading.

1.24 Other Relevant Data and Information

There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading. To the Authors' knowledge, there are no significant risks and uncertainties that could reasonably be expected to affect reliability or confidence in the exploration of information or MRE.

1.25 Interpretation and Conclusions

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this PEA Report is the evaluation of the potential technical feasibility and potential economic viability of the Project, notably the development of an underground mine, including processing facilities and related infrastructures.

This NI 43-101 Technical Report indicates that the Project demonstrates potential technical feasibility; however, it does not yet support robust economic viability under current assumptions. Additional work will be required to strengthen the resource base and improve project economics before advancing toward a future Pre-Feasibility Study, including expanded drilling to increase the overall mineralized inventory, as well as step-out and deeper exploration to identify extensions of higher-grade zones. Continued and expanded metallurgical test work, including variability testing, process optimization, and evaluation of alternative recovery methods, is also required to enhance recoveries, improve concentrate quality, and reduce processing costs.

1.26 Recommendations

The results of the financial analysis presented in this Preliminary Economic Assessment (PEA) demonstrate potential technical feasibility; however, it does not yet support robust economic viability under current assumptions. It is recommended to carry out additional work before advancing toward a future Prefeasibility Study ("PFS") for the project. The proposed budget total discussed in this section is $11.1M and is summarized in Table 1.9.

Table 1.9: Cost Estimate Associated with Recommendations

Description	k USD
Exploration and Drilling	9,132
Land and Property Fees	42
Metallurgical Testing Program	500
Climatic Study for Colling assessment	55
Geomechanical Drilling, Testing and engineering	808
Hydrogeology field investigation	600
Total	**11,137**

2. INTRODUCTION

Arizona Metals Corp. ("AMC" or "Arizona Metals" or the "Company") mandated G Mining Services Inc. ("GMS") as the lead consultant along with SGS Canada Inc. – Geological Services, WestLand Engineering & Environmental Services and HALEY & ALDRICH Inc to prepare a Preliminary Economic Assessment ("PEA"), under the supervision of the Qualified Persons ("QPs") for the Kay Mine Project ("Kay Project" or the "Property") located immediately adjacent to the town of Black Canyon City, approximately 69 km (43 miles) north of the city of Phoenix, in central Arizona, USA.

Arizona Metals is a mineral exploration company focusing on the exploration and development of mineral resource properties in Arizona. Based in Toronto, Ontario, AMC common shares trade on the Toronto Stock Exchange (TSX:AMC) and on the OTC Market (OTCQX:AZMCF).

The head office of Arizona Metals Corps. is located at:

66 Wellington Street, Suite 4100

Toronto, Ontario

M5K 1B7, Canada

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this report is to present the results of the Preliminary Economic Assessment (PEA) for the Kay Mine Project. The PEA focused on the evaluation of the technical feasibility and economic viability of the Project, particularly the development of an underground mine, including a processing facility and related infrastructures. The report contains all technical information relating to exploration of the Project, drilling methods, sampling and QA/QC protocols, data verification undertaken by the Qualified Person for this Technical Report, the Mineral Resource Estimate ("MRE") for the Project published in this report as of June 17, 2025. This Report provides estimates of operating and capital costs, as well as an economic analysis of the Project. The Kay Mine Project does not contain Mineral Reserves at this stage.

The intention of this Technical Report is to provide sufficient, clear and unambiguous technical and scientific information relating to the Project available at the effective date of the report. The qualified persons ("QP") understand that a copy of this report will be filed with the Canadian securities commissions and will be publicly available.

As of the Effective Date of this report, the QPs are not aware of any litigation potentially affecting the Project. The QPs did not verify the legality or terms of underlying agreements related to ownership, agreements, permits, licences, royalties or other contracts between Arizona Metals Corps. and third parties.

2.1 Scope of Work

The scope of work for this Preliminary Economic Assessment includes the compilation, review, and interpretation of geological, geomechanical, hydrogeological, metallurgical, environmental, field program, and infrastructure data relevant to the Kay Mine Project. GMS, SGS Canada Inc. – Geological Services, WestLand Engineering & Environmental Services and HALEY & ALDRICH Inc. contributed to this report, each within their respective areas of expertise. All consultants worked in close collaboration with the AMC technical team to ensure accuracy, consistency, and compliance with NI 43-101 standards. This report consolidated their findings to provide a comprehensive and current overview of the Kay Mine Project. The QPs involved in the mandate do not hold an interest in the issuer or its related entities.

All sections of this Technical Report were assembled by G Mining Services Inc., a mining consulting firm based in Brossard, Québec, Canada. The QPs are entirely independent of the issuer.

The QPs responsible for each section of the Technical Report are mentioned in Table 2.1.

Table 2.1: Summary of Qualified Persons

Qualified Person	Company	Title	Report Sections
Allan Armitage, PhD., P. Geo., (APEGA No. 64456, EGBC No. 38144, PGO No. 2829)	SGS Canada Inc. – Geological Services	Technical Manager & Senior Resource Geologist	1.4, 1.8, 1.12, 1.14, 1.23, 1.24, 4, 8, 12.3, 12.5, 14, 23, 24, 25.2, 25.10.1.1, 25.10.2.1, 26.1, 27.
Ben Eggers, MAIG, P.Geo., (EGBC No. 40384, NAPEG No. L5818)	SGS Canada Inc. – Geological Services	Senior Geologist	1.5 to 1.7, 1.9 to 1.11, 5, 6, 7, 9, 10, 11, 12.1, 12.2, 12.4, 27.
Carl Michaud, P.Eng., MBA, (OIQ No. 117090)	G Mining Services Inc.	Vice President of Mining Engineering	1.1, 1.15, 1.16, 1.21, 1.25, 1.26, 15, 16, 21.1.5, 21.2, 21.3.2, 21.4.1, 21.4.4,25.1, 25.3, 25.4, 25.8, 25.9, 25.10.1.2, 25.10.1.3, 25.10.2.2, 25.10.2.3, 26.2, 26.6, 27.
Hind Zniber El Mouhabbis, P.Eng., M.Eng., (OIQ No. 5007612)	G Mining Services Inc.	Technical Services Director	1.2, 1.3, 1.19, 1.22, 2, 3, 19, 22.
Sunil Koppalkar, P.Eng., (PEO No. 100190343)	G Mining Services Inc.	Senior Metallurgist	1.13, 1.17, 13, 17, 21.4.2, 25.5, 25.10.1.4, 25.10.2.4, 26.3, 27.

Qualified Person	Company	Title	Report Sections
Nicolas Vanier-Larrivée, P.Eng., (OIQ No. 143023, APEGS No. 78043)	G Mining Services Inc.	Earthworks and Study Manager	1.18, 18, 21.1 (excl 21.1.5), 21.4.3, 25.6, 25.10.1.5, 25.10.2.5, 26.4.
Richard DeLong, P.Geo., (MMSA No. 01471QP)	WestLand Engineering & Environmental Services	Senior Technical Advisor	1.20, 20.1, 20.2, 20.3, 20.4, 20.5, 20.6, 20.7, 25.7, 26.5.
Eric J. Mears, R.G., C.P.G., (AZ 28391 09/2026, CPG-12335 12/2026)	Haley & Aldrich, Inc.	Principal	20.8, 21.3, 27.

2.2 Sources of Information and Data

Unless otherwise stated, all the information and data relating to the Mineral Resource Estimate contained in the Report or used in its preparation have an effective date of June 17, 2025. The above-mentioned QPs have no reason to doubt the reliability of the information provided.

Sources of information include:

- Discussions with GMS, SGS, Westland, Haley & Aldrich Inc. and AMC personnel.
- Inspection of the Kay Mine Project area, including drill collars, drill core, and ground conditions by Allan Armitage and Ben Eggers.
- Drilling database and exploration information provided to the Authors by Arizona Metals.
- Previous technical reports written for the Property (see Section 2.5.1).
- Preliminary metallurgical testwork results.
- Technical and scientific reports by consultants.
- References, located at the end of this Technical Report in Section 27, provide a complete list of the documents reviewed, all figures and tables cited, and other information sources used.

Currency is expressed in United States dollars ("USD"), unless stated otherwise.

2.3 Site Visit

In accordance with NI 43-101 regulations, a current personal inspection was completed by the below-mentioned QP to the Kay Mine Project as part of the data validation process, as shown in Table 2.2.

Due to the early-stage nature of the Preliminary Economic Assessment, inspections were not conducted by all Qualified Persons, who instead relied on third-party data.

Table 2.2: Site Visit Dates of Qualified Person

Qualified Person	Company	Site Visit Scope	Dates
Allan Armitage	SGS Canada Inc. – Geological Services	Geology and Resources	October 25-26, 2023, and April 7-8, 2024
Ben Eggers	SGS Canada Inc. – Geological Services	Geology and Resources	May 30, 2025
Carl Michaud	G Mining Services Inc.	Mining Engineering	October 6 and 7, 2025
Richard DeLong	WestLand Engineering & Environmental Services	Environmental, Permitting & Social	February 16, 2026
Eric J. Mears	Haley & Aldrich	Mine Closure	January 14, 2022, and April 12, 2022

2.3.1 Site Inspection by Allan Armitage, P.Geo.

The Kay Project was visited by Allan Armitage on October 25-26, 2023, and April 7-8, 2024, for the purpose of:

- Inspection of selected drill sites and outcrops to review the drill and local geology.
- Inspection of the drill core logging, processing and storage facility.
- Reviewing current core sampling, QA/QC and core security procedures.
- Inspection of drill core, drill logs, and assay certificates to validate sampling, confirm the presence of mineralization in witness half-core samples, and review the local geology.

2.3.2 Site Inspection by Ben Eggers, P.Geo.

The Kay Project was visited by Ben Eggers on May 30, 2025, for the purpose of:

- Inspection of selected drill sites and outcrops to validate drill collar positions and review the drill and local geology.
- Inspection of the drill core logging, processing and storage facility.
- Reviewing current core sampling, QA/QC and core security procedures.
- Inspection of drill core, drill logs, and assay certificates to validate sampling, confirm the presence of mineralization in witness half-core samples, and review the local geology.

The site visit conducted by Eggers is considered the current site visit, per Section 6.2 of Companion Policy 43-101CP to NI 43-101.

2.3.3 Site Inspection by Carl Michaud, P.Eng., MBA

The Kay Project was visited by Carl Michaud on October 6 and 7, 2025, for the purpose of:

- Inspection of general site conditions, including visual assessments of site conditions, accessibility, road access, power supply, and existing infrastructure prior to the commencement of the assessment.
- Review drill cores and their locations.
- Conduct a review of the rock quality in the drill holes.

2.3.4 Site Inspection by Richard DeLong, P.Geo.

The Kay Project was visited by Richard DeLong on February 16, 2026, for the purpose of:

- Inspection of general site conditions, including visual assessments of site conditions, accessibility, road access, potential locations of project facilities, existing infrastructure, and proximity of the site to the local community prior to the commencement of the assessment.
- Discuss with site personnel ongoing activities at the site.

2.3.5 Site Inspection by Eric J. Mears

The Kay Project was visited by Eric J. Mears on January 14, 2022, and April 12, 2022, for the purpose of:

- Inspection of general site conditions.
- BLM Site Meeting.
- Discuss with site personnel ongoing exploration activities at the site.
- Development of permitting and reclamation strategies.

2.4 Effective Date

The PEA is derived using the Company's mineral resources (MRE) estimate, effective as of June 17, 2025. The effective date of the PEA is April 30, 2026. The issue date of the Technical Report is June 12, 2026.

2.5 Sources of Information

2.5.1 Previous Technical Reports

- Mineral Resource Estimate for the Kay Deposit Cu-Au-Zn-Pb-Ag Project, Yavapai County, Arizona, USA, Effective date June 17, 2025, Report date August 14, 2025.
- NI 43-101 Technical Report of the Kay Mine Project, Yavapai County, Arizona, USA, 2021, Effective date May 21, 2021, Report date June 23, 2021.

2.5.2 Agreements, Mineral Tenure, Surface Rights and Royalties

The issuer provided details regarding mining titles, royalty agreements, environmental liabilities, mineral agreements, and permits. The QPs are not qualified to offer any legal opinion on property titles, ownership, or potential litigation.

2.6 Units of Measure, Abbreviations and Nomenclature

The units of measure presented in this report, unless noted otherwise, are in the metric system. All dollar figures quoted in this report refer to United States dollars (USD or $) unless otherwise noted. A list of the main abbreviations and terms used throughout this report is presented in Table 2.3.

Table 2.3: List of Main Abbreviations

Abbreviations	Full Description
ac	Acres
Ag	Silver
As	Arsenic
Au	Gold
Bi	Bismuth
C	Carbon
CAD	Canadian Dollar
CIL	Carbon-in-leach
CoV	Cut-off Value
Cu	Copper
DD	Diamond Drilling

Abbreviations	Full Description
DGPS	Differential Global Positioning System
DSTSF	Dry Stack Tailings Storage Facility
ETP	Effluent Treatment Plant
F	Degrees Fahrenheit
ft	Feet
FA	Fire Assay
G	Giga – (000,000,000's)
g	Gram
gpt or g/t	Grams per tonne
g/L	Gram per Litre
G&A	General & Administration
GMS	G Mining Services Inc.
gpm	Gallons per minute (US)
GPS	Global Positioning System
ha	Hectares
h	Hour
h/d	Hours per day
h/y	Hours per year
h/wk	Hours per week
HDPE	High-Density Polyethylene
hp	Horsepower
Hz	Hertz
IRR	Internal Rate of Return
ISO	International Organization for Standardization
k	Kilo – (000's)
kg	Kilograms
kg/t	Kilograms per tonne
kV	Kilovolts
km	Kilometre
km/h	Kilometres per hour
kPa	Kilopascal

Abbreviations	Full Description
kW	Kilowatts
kWh	Kilowatts per hour
L	Litre
M	Mega or Millions (000,000's)
m	Metre
m/min	Metres per minute
m/s	Metres per second
m^2	Square metres
m^3	Cubic metres
Mg	Magnesium
mg	Milligram
mg/L	Milligrams per litre
mm	Millimeter
ml	Milliliter
min	Minute
MM	Mineralized Material
Mo	Month
Mt	Million tonnes
Mtpd	Metric tonnes per day
Mtpy	Metric tonnes per year
MVA	Megavolt-Ampere
MW	Megawatt
NI 43-101	National Instruments 43-101- Canadian Standards of Disclosure for Mineral Projects
NSR	Net Smelter Return
NPV	Net Present Value
NQ	Drill Core Diameter (47.6 mm)
Ø	Diameter
OK	Ordinary Kriging Methodology
OPEX	Operating Expenditures
oz	Troy Ounce (31.10348 grams)
OCR	Off-Channel Reservoir

Abbreviations	Full Description
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
Pb	Lead
PLC	Programmable Logic Controller
ppb	Parts per Billion
ppm	Parts per Million
psi	Pounds per Square Inch
PV	Present Value
RC	Reverse Circulation
RoM	Run-of-Mine
rpm	Revolutions per minute
S	Sulfur
Sec	Second (time)
STP	Sewage Treatment Plant
t	Tonnes (1,000 kg) (metric ton)
t/y or tpy	Tonnes per year
t/d or tpd	Tonnes per day
t/h or tph	Tonnes per hour
t/m^3	Tonnes per cubic metre
TRS	Tailings Reclaim Sump
TSF	Tailings Storage Facility
TTP	Thickened Tailings Plant
TWSP	Treated Water Storage Pond
URF	Uncemented Rockfill
USD	United States Dollar
V	Volt
wk	Week
XRF	X-ray Fluorescence
Y	Year
Zn	Zinc

3. RELIANCE ON OTHER EXPERTS

3.1 Introduction

This Technical Report has been prepared by GMS, under the supervision of the QPs, for AMC. The information, conclusions, opinions, and estimates contained herein are based on:

- Information and documentation available to GMS at the time of the preparation of this Report.
- Assumptions, conditions and qualifications as set forth in this Report.
- Data, reports, and opinions supplied by AMC and other third-party sources.

The QPs believe that the underlying assumptions in the information provided are factual and accurate and that the resulting interpretations are reasonable. To the extent applicable, the QPs have relied on such data and have no reason to believe that any material facts have been withheld. In their professional judgement, the QPs have taken appropriate steps to ensure that the information relied upon is sound and, accordingly, do not disclaim responsibility for the content of this Report.

The QPs have assumed that all information and technical documents referenced in Chapter 27 (References) are accurate and complete in all material respects. While the QPs have reviewed the available documentation, its accuracy and completeness cannot be guaranteed. The QPs reserve the right, but are not obligated, to revise this Report and its conclusions should additional information become available after the effective date.

The QPs relied on the following companies and consultants to prepare some aspects of this Report; their involvement is listed below:

- The Taxation calculation was supplied by McGovern Hurley LLP staff and retained expert Shawn Lee, Chartered Professional Accountants.
- This information is used in support of Section 1, Section 22 and Section 25.

The authors wish to emphasize that they are QPs only in respect of the areas in this Technical Report identified in their “Certificates of Qualified Persons” submitted with this Technical Report to the Canadian Securities Administrators. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of this Technical Report by any third party is at the party’s sole risk.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Kay Mine property is located immediately adjacent to the town of Black Canyon City, approximately 69 km (43 miles) north of the city of Phoenix, in central Arizona, USA (Figure 4.1 and Figure 4.2). The Property is located in Township 8 North, Range 2 East (Gila and Salt River meridian), in the Tip Top mining district in Yavapai County, Arizona. The UTM coordinates of Shaft 1 on the eastern portion of the property are 392910E, 3769540N (WGS84 datum, Zone 12S). The property falls on the Black Canyon City 7.5-minute topographic map published by the United States Geological Survey.

4.2 Land Tenure

The Kay Mine property consists of 88 unpatented lode mining claims covering approximately 645.2 ha (1,594.4 acres), six (6) patented mining claims covering approximately 30.4 ha (75.1 acres), and 78.0 ha (192.7 acres) of private land (Figure 4.1, Table 4.1). The private land includes mineral rights, four (4) water wells, and housing for company staff. The company also owns two (2) unpatented placer mining claims totalling 16.2 ha (40.0 ac) co-located with unpatented lode mining claims (Figure 4.1, Table 4.1).

Annual payments for the unpatented claims are due on or before August 31 to BLM and Yavapai County totaling approximately USD 18,000 per year. As of the effective date of this report, annual claim payments are current through August 31, 2026.

Annual Yavapai County tax for the patented claims in 2024 is approximately USD 5,841. The annual 2024 property tax for the currently owned private land is approximately USD 24,000. Yavapai County tax payments for the patented claims and currently owned private land are current as of the effective date of this report.

Figure 4.1: Kay Property Location Map and Claims Location Map

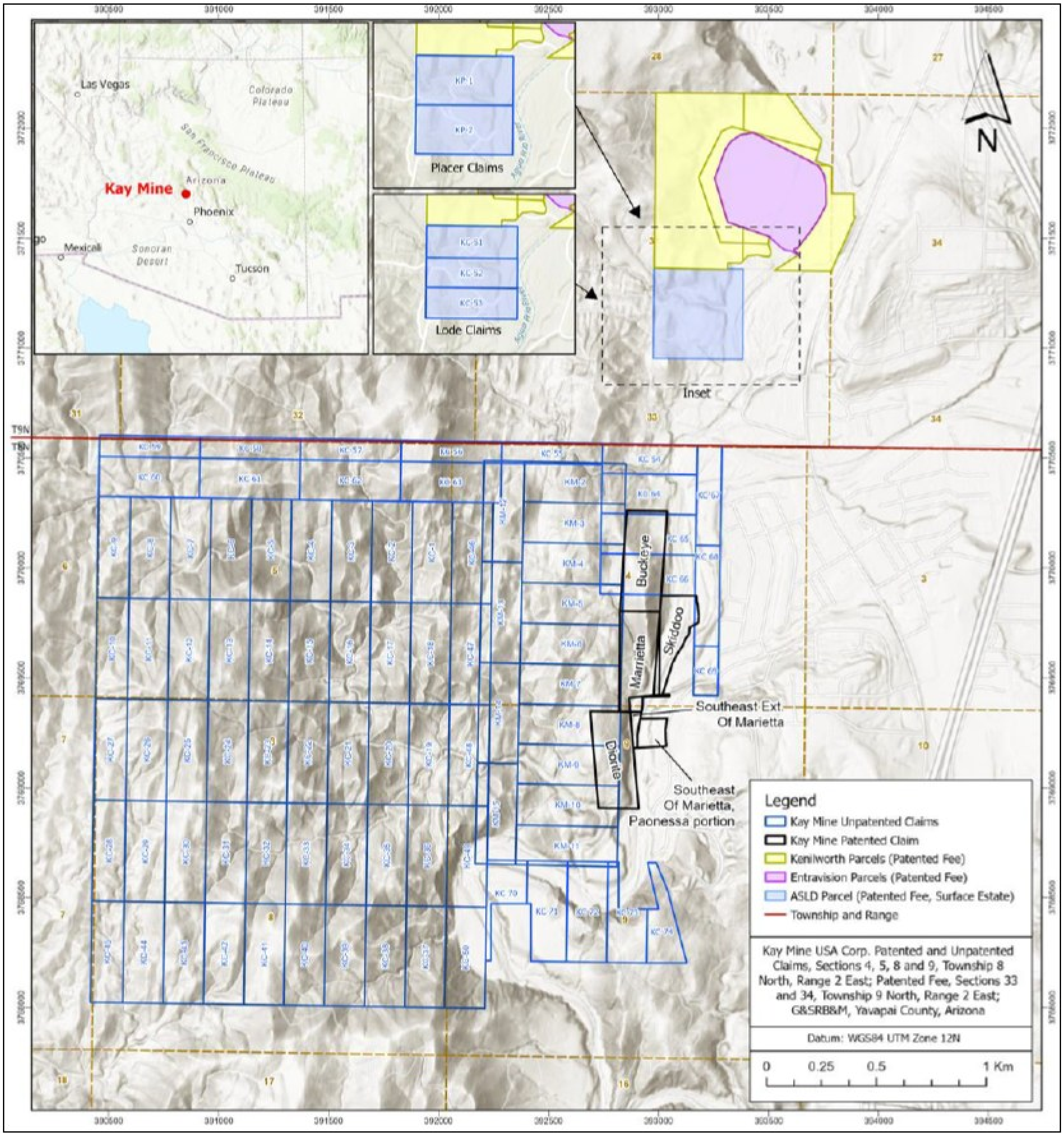


Figure 4.2: Kay Property Map

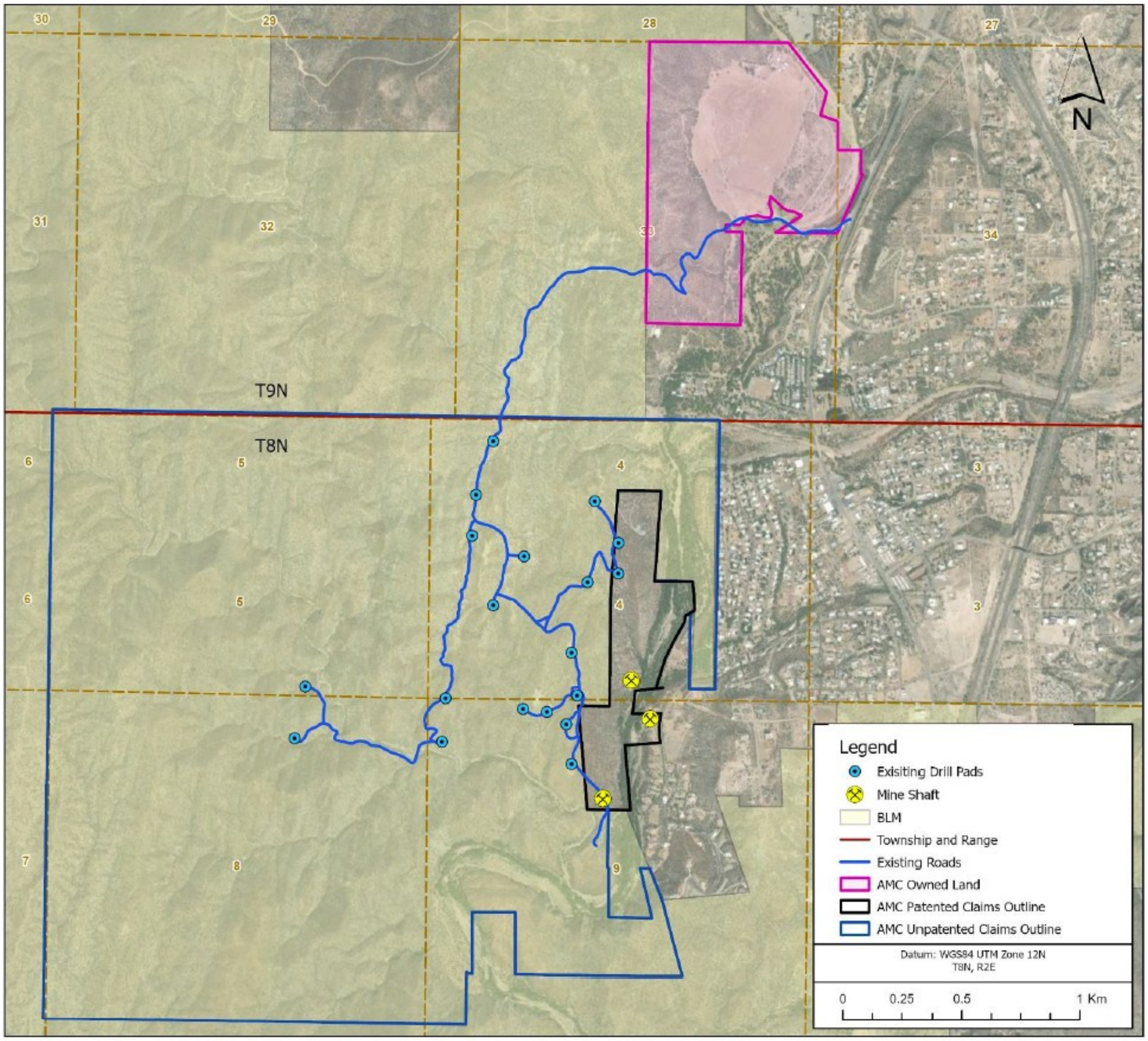


Table 4.1: List of Patented and Unpatented Mining Lode Claims and Unpatented Placer Mining Claims

Claim Name	Type	BLM Serial Number / Yavapai County Parcel Number	Approximate Area (ha)	Approximate Area (ac)
Buckeye	Patented lode	501-03-019B	28.7	70.9
Marietta	Patented lode	501-03-019B		
Southeast Extension of Marietta (western portion)	Patented lode	501-03-019B		
Skiddoo (western portion)	Patented lode	501-03-019B		
Diorite	Patented lode	501-03-019B		
Southeast Extension of Marietta (Paonessa portion)	Patented lode	501-03-019U, 501-03-019V	1.7	4.2
Total Patented Lode Claims			**30.4**	**75.1**
KM-2	Unpatented lode	AMC443132	8.1	20.0
KM-3	Unpatented lode	AMC443133	8.1	20.0
KM-4	Unpatented lode	AMC443134	8.1	20.0
KM-5	Unpatented lode	AMC443135	8.1	20.0
KM-6	Unpatented lode	AMC443136	8.1	20.0
KM-7	Unpatented lode	AMC443137	8.1	20.0
KM-8	Unpatented lode	AMC443138	6.3	15.4
KM-9	Unpatented lode	AMC443139	6.1	15.1
KM-10	Unpatented lode	AMC443140	7.4	18.3
KM-11	Unpatented lode	AMC443141	8.1	20.0
KM-12	Unpatented lode	AMC443142	8.1	20.0
KM-13	Unpatented lode	AMC443143	8.1	20.0

Claim Name	Type	BLM Serial Number / Yavapai County Parcel Number	Approximate Area (ha)	Approximate Area (ac)
KM-14	Unpatented lode	AMC443144	8.1	20.0
KM-15	Unpatented lode	AMC443145	8.1	20.0
KC-1	Unpatented lode	AMC454211	8.1	20.0
KC-2	Unpatented lode	AMC454212	8.1	20.0
KC-3	Unpatented lode	AMC454213	8.1	20.0
KC-4	Unpatented lode	AMC454214	8.1	20.0
KC-5	Unpatented lode	AMC454215	8.1	20.0
KC-6	Unpatented lode	AMC454216	8.1	20.0
KC-7	Unpatented lode	AMC454217	8.1	20.0
KC-8	Unpatented lode	AMC454218	8.1	20.0
KC-9	Unpatented lode	AMC454219	8.1	20.0
KC-10	Unpatented lode	AMC454220	8.1	20.0
KC-11	Unpatented lode	AMC454221	8.1	20.0
KC-12	Unpatented lode	AMC454222	8.1	20.0
KC-13	Unpatented lode	AMC454223	8.1	20.0
KC-14	Unpatented lode	AMC454224	8.1	20.0
KC-15	Unpatented lode	AMC454225	8.1	20.0
KC-16	Unpatented lode	AMC454226	8.1	20.0
KC-17	Unpatented lode	AMC454227	8.1	20.0
KC-18	Unpatented lode	AMC454228	8.1	20.0
KC-19	Unpatented lode	AMC454229	8.1	20.0

Claim Name	Type	BLM Serial Number / Yavapai County Parcel Number	Approximate Area (ha)	Approximate Area (ac)
KC-20	Unpatented lode	AMC454230	8.1	20.0
KC-21	Unpatented lode	AMC454231	8.1	20.0
KC-22	Unpatented lode	AMC454232	8.1	20.0
KC-23	Unpatented lode	AMC454233	8.1	20.0
KC-24	Unpatented lode	AMC454234	8.1	20.0
KC-25	Unpatented lode	AMC454235	8.1	20.0
KC-26	Unpatented lode	AMC454236	8.1	20.0
KC-27	Unpatented lode	AMC454237	8.1	20.0
KC-28	Unpatented lode	AMC454238	8.1	20.0
KC-29	Unpatented lode	AMC454239	8.1	20.0
KC-30	Unpatented lode	AMC454240	8.1	20.0
KC-31	Unpatented lode	AMC454241	8.1	20.0
KC-32	Unpatented lode	AMC454242	8.1	20.0
KC-33	Unpatented lode	AMC454243	8.1	20.0
KC-34	Unpatented lode	AMC454244	8.1	20.0
KC-35	Unpatented lode	AMC454245	8.1	20.0
KC-36	Unpatented lode	AMC454246	8.1	20.0
KC-37	Unpatented lode	AMC454247	8.1	20.0
KC-38	Unpatented lode	AMC454248	8.1	20.0
KC-39	Unpatented lode	AMC454249	8.1	20.0
KC-40	Unpatented lode	AMC454250	8.1	20.0

Claim Name	Type	BLM Serial Number / Yavapai County Parcel Number	Approximate Area (ha)	Approximate Area (ac)
KC-41	Unpatented lode	AMC454251	8.1	20.0
KC-42	Unpatented lode	AMC454252	8.1	20.0
KC-43	Unpatented lode	AMC454253	8.1	20.0
KC-44	Unpatented lode	AMC454254	8.1	20.0
KC-45	Unpatented lode	AMC454255	8.1	20.0
KC-46	Unpatented lode	AMC454256	7.0	17.3
KC-47	Unpatented lode	AMC454257	7.0	17.4
KC-48	Unpatented lode	AMC454258	7.0	17.4
KC-49	Unpatented lode	AMC454259	7.6	18.7
KC-50	Unpatented lode	AMC454260	8.1	20.0
KC-51	Unpatented lode	AZ105793702	5.4	13.3
KC-52	Unpatented lode	AZ105793703	5.4	13.3
KC-53	Unpatented lode	AZ105793704	5.4	13.3
KC 54	Unpatented lode	AZ106364103	5.4	13.3
KC 55	Unpatented lode	AZ106364104	4.0	10.0
KC 56	Unpatented lode	AZ106364105	4.4	10.8
KC 57	Unpatented lode	AZ106364106	4.4	10.9
KC 58	Unpatented lode	AZ106364107	4.4	10.9
KC 59	Unpatented lode	AZ106364108	4.4	10.9
KC 60	Unpatented lode	AZ106364109	8.4	20.7
KC 61	Unpatented lode	AZ106364110	8.4	20.7

Claim Name	Type	BLM Serial Number / Yavapai County Parcel Number	Approximate Area (ha)	Approximate Area (ac)
KC 62	Unpatented lode	AZ106364111	8.4	20.7
KC 63	Unpatented lode	AZ106364112	6.9	17.1
KC 64	Unpatented lode	AZ106364113	5.7	14.0
KC 65	Unpatented lode	AZ106364114	2.6	6.4
KC 66	Unpatented lode	AZ106364115	2.7	6.7
KC 67	Unpatented lode	AZ106364116	5.1	12.5
KC 68	Unpatented lode	AZ106364117	4.5	11.1
KC 69	Unpatented lode	AZ106364118	2.3	5.6
KC 70	Unpatented lode	AZ106364119	8.0	19.9
KC 71	Unpatented lode	AZ106364120	8.0	19.8
KC 72	Unpatented lode	AZ106364121	8.0	19.7
KC 73	Unpatented lode	AZ106364122	8.3	20.4
KC 74	Unpatented lode	AZ106364123	5.3	13.1
Total Unpatented Lode Claims:			**645.2**	**1,594.4**
KP-1	Unpatented placer	AZ105793705	8.1	20.0
KP-2	Unpatented placer	AZ105793706	8.1	20.0
Total Unpatented Placer Claims:			**16.2**	**40.0**

**Notes: Placer claims co-located with unpatented lode claims KC-51, 52, 53.*

4.3 Nature Of Arizona Metals' Interest

On January 30, 2019, Arizona Metals (under its previous name Croesus Gold Corp.) acquired 100% of the Kay Project from Silver Spruce Resources for a total cash consideration of $400,000. Arizona Metals also agreed to assume a USD 450,000 loan between Silver Spruce and a third-party lender, which matured on June 22, 2018; the company repaid this loan in full on March 12, 2019. This purchase consisted of 14 unpatented mining claims covering 108.8 ha (268.7 ac) and five (5) patented mining claims covering 28.7 ha (70.9 ac).

Following the initial project purchase described above, the Company acquired mineral rights to 74 additional unpatented lode claims and two (2) unpatented placer claims by staking claims, filing claim documents with BLM and Yavapai County, and making annual claim maintenance filings and payments to keep the claims, and therefore the Company's mineral rights to these claims, current. The Company acquired these additional unpatented mining claims in three (3) phases:

- 50 unpatented lode mining claims (400.8 ha, 989.9 ac) were staked in 2019.
- Three (3) unpatented lode claims (16.2 ha, 40 ac) and two (2) unpatented placer mining claims (16.2 ha, 40 ac) were staked in 2022. These five (5) claims cover private land purchased from the Arizona State Land Department purchased in 2024 (see below).
- 21 unpatented lode mining claims (119.5 ha, 295.1 ac) were staked in 2023.

In 2024, the Company purchased 1.7 ha (4.2 ac) of patented mining claims, acquiring the eastern portion of the Southeast Extension of Marietta claim for USD 325,000.

Arizona Metals has purchased a total of 78.0 ha (192.7 ac) of private land in three (3) transactions:

- Kenilworth purchase: 43.1 ha (106.5 ac) in 2021 for a purchase price of USD 2,250,000 from a private owner. This land includes mineral rights.
- Entravision purchase: 18.8 ha (46.4 ac) in 2024 for a purchase price of USD 2,500,000 from a private owner. This land includes mineral rights.
- Arizona State Land purchase: 16.1 ha (39.8 ac) in 2024 for a purchase price of USD 366,100, through an auction process with the Arizona State Land Department. This purchase did not include mineral rights, but the Company located unpatented lode and placer mining claims on this land in 2022.

The author is not aware of any underlying agreements or royalties on the Kay Project mining claims and private land.

4.4 Mineral Title and Mining Law

Mineral rights for economic minerals and metals on public lands in the United States are governed by the General Mining Act of 1872. This law allows for unpatented mining claims to be staked on public lands that are open to mineral entry and have not been designated for other specific uses. Unpatented mining claims confer mineral rights to the owner, while surface rights remain under the administration of the appropriate government agencies. Patented mining claims confer both mineral rights and surface rights to the owner and are private property. In the Kay Project area, mineral rights and permitting are administered by the Department of Interior, Bureau of Land Management (BLM), under the Federal Land Policy and Management Act of 1976.

According to Bureau of Land Management records, a recent legal title opinion, a mineral title report and Yavapai County tax documents, the mineral title appears to be valid for both the patented and unpatented mining claims on the property. Determination of secure mineral title is solely the responsibility of Arizona Metals.

4.5 Permitting and Environmental Consideration

No permitting is necessary for surface exploration work on the property, such as geologic mapping, surface sampling, and geophysics. Eighteen (18) drill sites and their access roads covering 5 acres on unpatented mining claims are currently permitted through a Notice that was submitted to and approved by the Bureau of Land Management (BLM). All work approved under the Notice is fully bonded with BLM (Figure 4.2).

Permitting for drilling on patented mining claims appears to be minimal, consisting of routine permitting through the Arizona Department of Water Resources.

Arizona Metals is pursuing an Exploration and Reclamation Plan of Operations to expand the scope of drill operations beyond what is currently permitted under existing permits; this plan was submitted to BLM in January 2026.

Because of the Project's proximity to Black Canyon City, Arizona Metals is taking extra care with community consultation during permitting and operation of drill programs by contracting the services of a community relations specialist.

Small historical mine dumps exist on the property at No. 1, No. 2, and No. 3 Shafts, and these are likely to contain sulfide minerals, particularly pyrite, which have the potential for producing acidic surface waters as they oxidize. The mineralization on the Project contains significant arsenic, above 10% in some recent Arizona Metals drill samples. Given the proximity of these mine dumps to the active Aqua Fria River, Arizona Metals will consult with a local environmental consultant to evaluate whether any environmental risk exists from these historic mine dumps.

4.6 Other Relevant Factors

To the Author's knowledge, the Property has no outstanding environmental liabilities from prior mining activities. The Author is unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform exploration work recommended for the Property.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility, Physiography, Vegetation and Wildlife

Access to the Kay Project is excellent by road on Interstate Highway 17, then by paved city streets in Black Canyon City to the banks of the Agua Fria River. Gravel drill and mine roads give access to the Kay Project. Vehicle access onto the Kay Project currently requires crossing Black Rock Creek, a small stream with intermittent flow, highest in the winter months (January – March) and lowest in the spring and summer (May – July), with occasional storm-related high and turbulent flow.

The Kay Project lies in an area of moderate topography, reaching elevations of 683 m (2,240 feet) with a relief of approximately 100 m (320 feet) from the streambed of the Agua Fria River to the summits of hills on the Kay Project. The terrain is accommodating to exploration activities, as evidenced by previous mine shafts and access roads. Vegetation is generally sparse, consisting of wide varieties of cactus and low brush, although the Agua Fria River channel is bordered by thicker underbrush and numerous trees.

Wildlife in the area can include a variety of large and small mammals, including black bears, mountain lions, mule deer, coyotes, bobcats, badgers, reptiles, including snakes and turtles, and a large variety of birds, including falcons, hawks, turkey vultures and golden eagles.

5.2 Local Resources and Infrastructure

The Kay Project is immediately adjacent to the population in the town of Black Canyon City, population about 5,600, which offers basic services such as fuel, food, and housing. Many private homes have views of the Property, so care is taken before and during exploration and mining operations to consult with and accommodate nearby residents.

Surface rights for mining on the unpatented claims are held by the United States government and are governed by the Federal Land Policy and Management Act of 1976 and General Mining Act of 1872, as described above and administered by the federal Bureau of Land Management. Surface rights for mining on the patented claims reside with the patented claim owners as private land.

The Project is very well positioned relative to support infrastructure, with ready access to power and water in adjacent Black Canyon City, and excellent road access along Interstate Highway 17 and paved city streets. Arizona has a long and rich mining history, and skilled miners and mining professionals reside throughout the state and are available for employment. Potential locations for tailings, waste disposal, and processing plants are numerous, particularly out of sight of town on the western portion of the Project.

5.3 Climate

The climate of the Project area is hot semi-arid, typified by very hot summers and mild winters. The area receives little precipitation, averaging about 254 mm (10 inches) per year, as heavy periodic rainstorms, generally in the winter months, and as late summer thunderstorms. Summers are very hot, often consisting of consecutive days over 38ºC (100ºF). Winter temperatures generally range from 6-22ºC (42-72ºF). Access and work can generally continue year-round. Average temperature and precipitation for Scottsdale, Arizona, located approximately 80 km southeast of the Project, are shown in Table 5.1.

The operating season is 12 months per year, with potential fire restrictions during the summer months that may limit advance exploration activities and drilling. It is expected that if the Project advances to development and mining operation, sufficient fire mitigation can be put in place to allow year-round operations.

Table 5.1: Average Monthly Temperature and Precipitation, Scottsdale, Arizona

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average high temperature (°C)	19	21	24	28	33	38	40	39	37	31	23	18
Average low temperature (°C)	6	8	10	14	19	24	27	39	23	17	9	6
Average precipitation (mm)	32	31	31	11	5	2	26	30	23	20	22	29

Source: U.S. Climate Data (2018).

6. HISTORY

Mineralization at the Kay Project was first discovered before 1900, and activity has continued intermittently since then (Smith, 2024).

6.1 Initial Discovery and Early Works

The Kay Mine was discovered sometime before 1900 and mined on a small scale from the inclined No. 1 shaft, producing approximately 635 tonnes (700 short tons) of ore prior to 1916 or 1918.

6.2 Kay Copper Company

Between 1918 and the late 1920s, the Property was owned by an eastern mining interest that became the Kay Copper Company in 1922. During this period, the owners deepened the No. 1 Shaft to 457 m (1,500 ft), sunk the No. 4 shaft to 366 m (1,200 ft), installed the No. 3 Shaft, and developed several thousand feet of underground workings on 11 levels, discovering the ore bodies above the 600 Level but apparently producing no ore. Judging by mine maps, the company drilled at least 89 underground drill holes (according to mine plan maps); assay data are plotted on mine plan maps, but no drill logs nor assay certificates are available. The Kay Copper Company failed in the late 1920s, and the Project was dormant until 1949, apparently due to a combination of low metals prices and litigation.

6.3 Mid-Century Operators

In the late 1940s, the Project was acquired by an unnamed owner for back taxes, and in 1949 leased to Black Canyon Copper Corporation, which opened the underground workings to the 500 Level and shipped about 907 tonnes (1,000 short tons) of ore.

In 1949 or 1950, Black Canyon Copper subleased the Project to Shattuck-Denn Mining Company and New Jersey Zinc Company until 1952. These companies dewatered and rehabilitated the No. 4 Shaft at least to the 1000 Level, and performed surface and underground exploration, including resampling and underground diamond drilling of at least 14 holes (according to mine plan maps). They shipped an uncertain amount of ore, reported to be 1,425 tonnes (1,571 short tons).

In 1955-1956, the Project was leased to Republic Metals Company, which shipped 414 tonnes (456 short tons) of ore from above the 350 Level. A cave-in destroyed pumping operations, and the mine was allowed to flood. Following this, the Project saw several unsuccessful attempts to revive operations until 1972.

6.4 Exxon Minerals

The Project was acquired by Exxon Minerals Company in 1972, which invested about $1.5M in exploration on the Project. This work included geologic mapping; "mine mapping" (suggesting that Exxon re-opened the underground workings); relogging drill core and cuttings; petrographic studies; assaying 610 m (2,000 ft) of unassayed drill core; stream sediment and soil geochemistry surveys; reviewing historical assay data and incorporating into mine maps and cross sections; and geophysical surveys. Exxon drilled 23 core / rotary exploration holes totalling 8,094 m (26,554 ft), 14 of which were in the immediate vicinity of the Kay Mine and which total 6,807 m (22,333 ft). Fellows (1982) also mentions "ten (10) shallow air-track claim validation drill holes on various parts of the property," but gives no specific locations. Exxon's last reported work on its project was in 1984.

6.5 Post-Exxon Multiple Owners

The five (5) patented claims changed hands a number of times between 1990 and 2015, apparently without exploration work. In 1990, Exxon sold the five (5) patented claims to Rayrock Mines, which in turn sold them to American Copper and Nickel Company in 1995. Ownership was then conveyed to Shangri-La Development in 2000, to five (5) private individuals in 2002, and to Jodon Development in 2003. In 2015, Cedar Forest Inc. acquired the five (5) patented claims through foreclosure on Jodon Development. Cedar Forest did not appear to do any exploration work on the Project.

6.6 Silver Spruce Resources

In March 2017, Silver Spruce Resources Inc. acquired the five (5) patented mining claims from Cedar Forest and then staked 14 unpatented "KM" mining claims in April 2017. Together, these 19 claims comprise the property purchased by Arizona Metals. Silver Spruce took 39 samples on the Project but did no other exploration work.

6.7 Arizona Metals Corporation

On September 26, 2018, Croesus Gold Corporation (now Arizona Metals) signed a letter of intent to acquire the five (5) patented and 14 unpatented "KM" claims from Silver Spruce Resources. To date, Arizona Metals has performed geologic, geochemical, and geophysical exploration and drilling on the Project and staked additional unpatented mining claims.

6.8 Historical Resources and Reserves

The historical mineral reserve estimate presented in this section is considered historical in nature, and Arizona Metals is not treating the historical reserve as current. The historical resources and reserves for the Kay Project are superseded by the Indicated and Inferred MRE for the deposits reported in Section 14 of this report.

A number of historical estimates of resources and reserves have been made over the years on the Project. In 1982, Exxon Minerals estimated a proven and probable reserve of 6.4 million short tons at a grade of 2.2% copper, 2.8 g/t gold, 3.0% zinc, and 55 g/t silver, using a cut-off grade of 2% copper-equivalent. This estimate has incorporated data from approximately seven (7) years of underground exploration by Exxon, as well as 7,000 m of surface drilling in the vicinity of the deposit.

The historical production record of the mine is scattered and almost certainly incomplete. Keith et al (1983) reported that the Kay Mine produced 2,600 short tons of ore containing 296,000 pounds Cu, 13,000 pounds Pb, 2,700 ounces Ag, and 150 ounces Au. The following production was reported in the more detailed project-specific reports currently available.

- 635 tonnes (700 short tons) grading 9.1% Cu, 36.3 g/t Ag, and 2.5 g/t Au (1.06 opt Ag and 0.072 opt Au) mined prior to 1916.
- 907 tonnes (1,000 short tons), no grade reported, shipped in 1949 by Black Canyon Copper Corp.
- 1,410 tonnes (1,554 short tons) with a weighted average grade of 5.62% Cu shipped between 1950 and 1953 by New Jersey Zinc / Shattuck-Denn Mining Company, Drake Mining Corp., and Republic Metals Company. This is likely the 1,425 tonnes (1,571 short tons) previously reported grading 5.67% Cu, 33.6 g/t Ag, and 2.0 g/t Au (0.98 opt Ag and 0.059 opt Au), and includes the 414 tonnes (456 short tons) grading 4.64% Cu, 17.1 g/t Ag, and 1.4 g/t Au (0.5 opt Ag and 0.04 opt Au) reported by Mattinen (1984b) as shipped by Republic Metals Company in 1955-1956.
- 64 tonnes (70 tons) grading 5.7% Cu selected from surface dumps and shipped by a private owner in 1966.

The total documented production from the Kay Mine is thus approximately 3,016 tonnes (3,325 short tons).

7. GEOLOGICAL HISTORY AND MINERALIZATION

7.1 Regional Geology

The Kay Project is located in Precambrian metamorphic rocks in central Arizona. Central Arizona is characterized by basement rocks of Proterozoic age (1.8-1.6 Ga) with great stratigraphic complexity and pervasive yet variable deformation and metamorphism. The Proterozoic basement is well exposed in a broad 500-km-long NW-trending belt that transects the state from southeast to northwest, known as the central volcanic belt. The Proterozoic basement is directly overlain in places by Tertiary volcanic and sedimentary rocks and by Quaternary surface deposits and has been intruded by widespread Laramide-age granitoids, many of which produced the large porphyry copper systems that have made Arizona famous for copper production. The Proterozoic basement rocks are the result of largely compressional tectonics active between 2.0 and 1.62 Ga, with several periods of subduction, accretion of numerous island arcs onto the ancestral Wyoming craton, and attendant volcanism, plutonism, deformation, and metamorphism (Smith, 2024, and references therein).

The Proterozoic basement in the region is divided into three (3) major blocks: Mojave on the west, Yavapai in the center (where the Kay Project is located) and Mazatzal to the east. The Yavapai block is further subdivided into several smaller blocks bordered by major shear zones, and the Kay Project is located in the Ash Creek block (Figure 7.1).

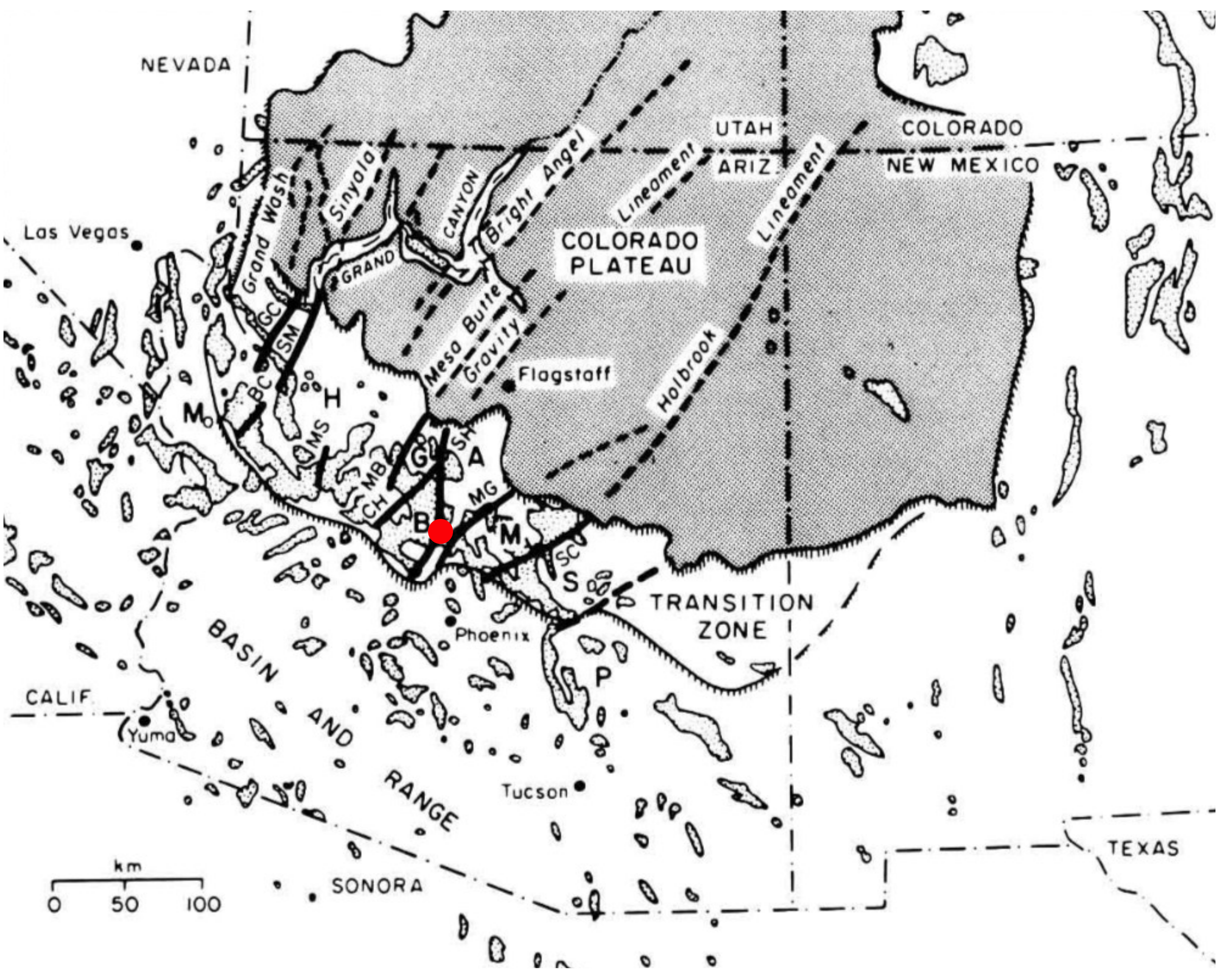
Proterozoic rocks in the Project region consist dominantly of metamorphosed bimodal volcanic and sedimentary rocks and large granitoid intrusive complexes. Host rocks in the Project area consist of the Townsend Butte facies within the Black Canyon Creek Group of the Yavapai Supergroup (Anderson, 1989b). This facies comprises a complex bimodal volcanic assemblage with related tuffaceous sediments, including felsic sediments and volcaniclastics interbedded with submarine basaltic-andesitic flows and dacite flows and tuffs, interpreted as having been formed in an intraoceanic island arc at 1800-1740 Ma. Pre- to syntectonic intrusive complexes crop out in the Project region, including the large Cherry Creek batholith to the northeast (1740-1720 Ma) and the Crazy Basin monzogranite west of the Project. The belt of Proterozoic rocks in which the Kay Project lies is referred to as the Black Canyon Belt (Figure 7.2).

All Proterozoic rocks in the area have been metamorphosed to greenschist to lower amphibolite grade between 1740-1720 Ma and 1699 Ma, likely during the Yavapai orogeny at 1700-1690 Ma, with peak metamorphism occurring at about 1700 Ma. The resulting rocks in the Kay area are now dominantly quartz-sericite-chlorite schists with smaller amounts of greenstone, calc-silicate schist, Fe-rich chert, and fine-grained quartzite.

These rocks show a pervasive NE to NNE foliation that dips steeply to the west and parallels the dominant fabrics and lithological breaks in the region. Two (2) major fault zones occur in the Project region: the N-trending Proterozoic-age Shylock shear zone west of the Project interpreted to be a major crustal boundary in Proterozoic time (Darrach et al, 1991; Leighty et al, 1991), and which now marks the western boundary of the Ash Creek tectonic block; and a younger N-trending left-lateral strike-slip fault zone with 3-5 km of offset that cuts Tertiary strata about 16 km east of the Project (Ferguson et al, 2008).

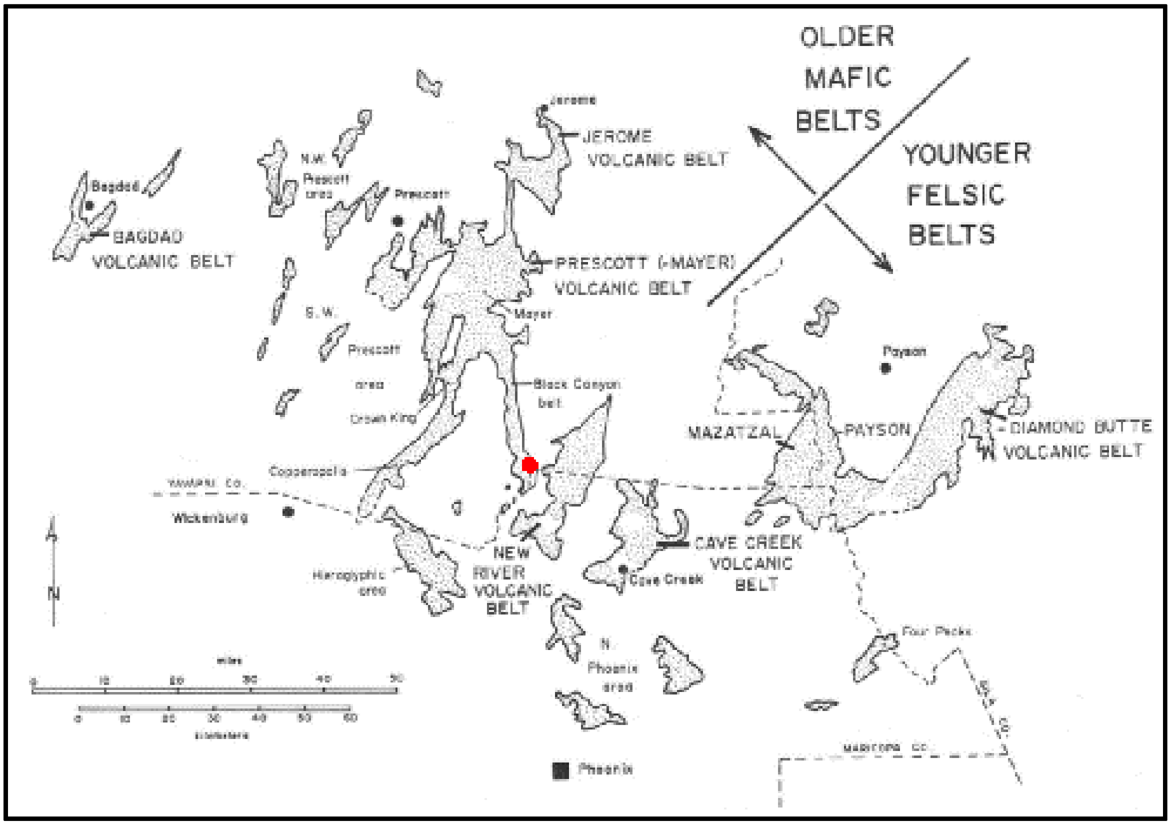
The Kay Mine is one of numerous Early Proterozoic volcanogenic massive sulfide deposits in the region (Figure 7.3). It is reported that 70 such deposits are known in Arizona that produced 50.2M tonnes (55.3 short tons) of ore with an average grade of 3.6% Cu containing 3.99B pounds Cu. The largest of these were the Verde and Big Bug districts northeast of the Kay Mine. VMS deposits near Kay include New River, Bronco Creek, and Gray's Gulch to the southeast; and Mayer, Agua Fria, Big Bug, and Verde to the north. The characteristics, geologic settings, ages, and enclosing host rocks are sufficiently similar among these deposits that they form a distinct metallogenic province and epoch in central Arizona.

Figure 7.1: Tectonic Blocks in Central Arizona



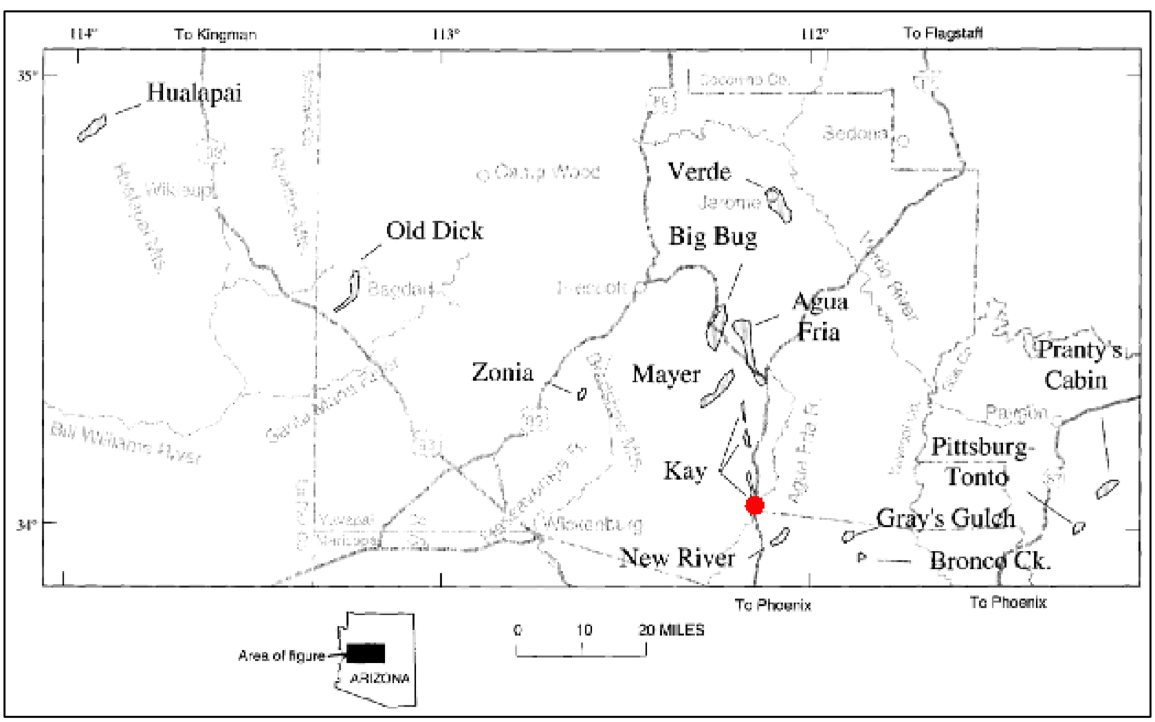
**Note: Kay Project (Red Dot), located in the Ash Creek Block (A) (Smith, 2024).*

Figure 7.2: General Map of Precambrian Basement Rocks of Central Arizona



Note: Kay Project (Red Dot), located in the Black Canyon Belt (Smith, 2024).

Figure 7.3: Map of Volcanogenic Massive Sulfide Districts in Central Arizona



Note: Kay Mine Property Shown as Red Dot (Smith, 2024).

7.2 Property Geology

The Kay Project lies in an NNE-trending belt of schists and phyllites comprising metamorphosed volcanics and metasediments with minor chert and iron formation (Figure 7.4, Figure 7.5). In the property area, this belt of schists is bordered on the east by alluvium in the Agua Fria River drainage and Tertiary sediments and volcanics and bordered on the west by the Proterozoic Crazy Basin monzogranite. The Shylock shear zone, a regional structural feature, runs to the west of the Property. The Property's host rocks and structure are described below.

Figure 7.4: Geologic Map of the Kay Project

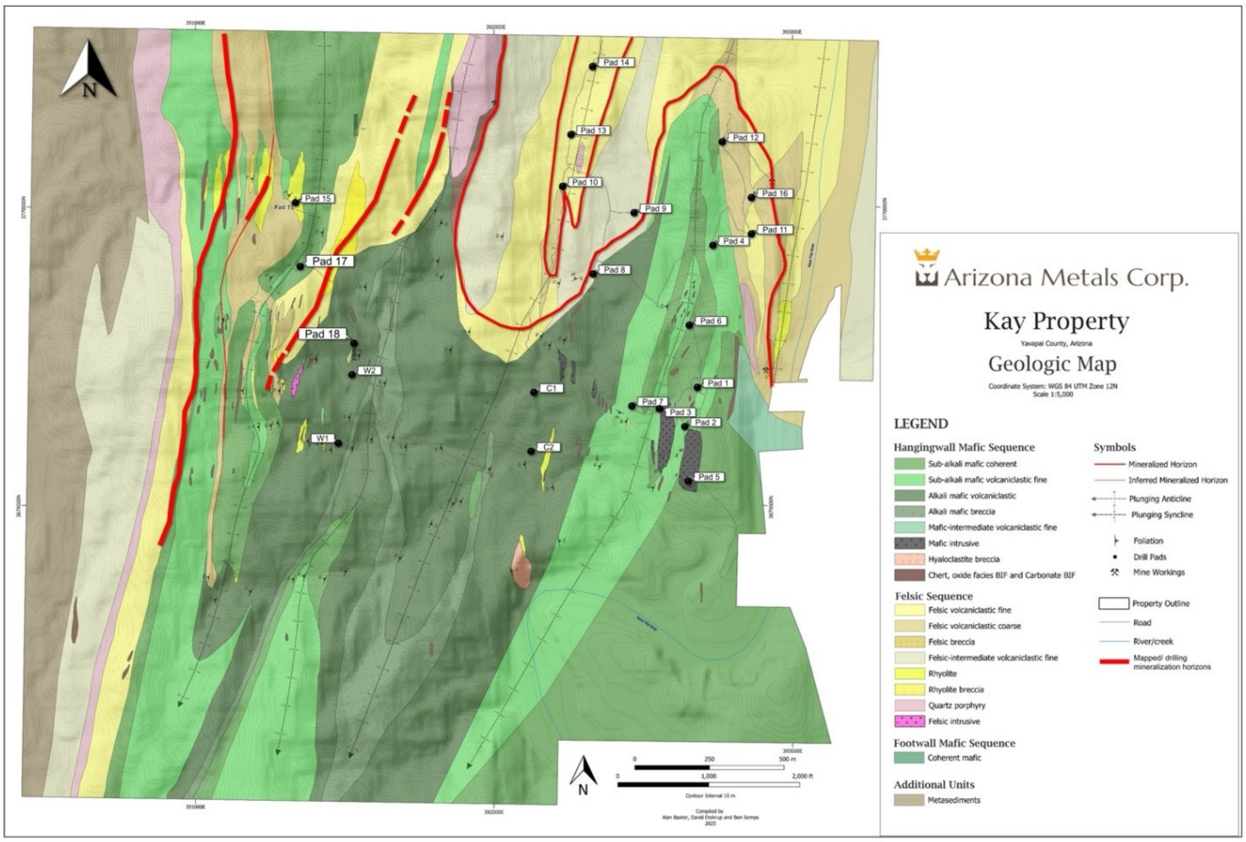
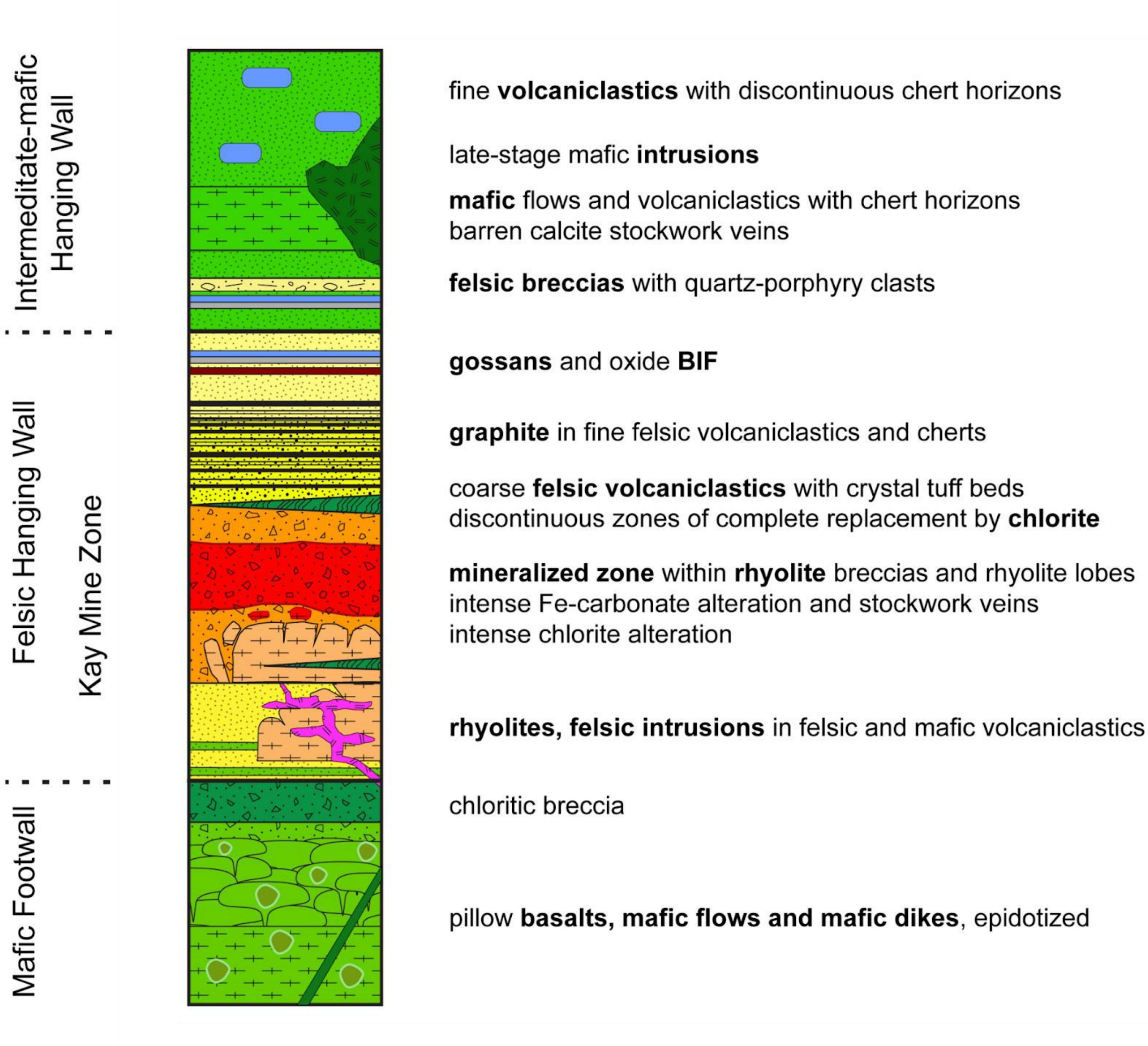


Figure 7.5: Stratigraphy of the Kay Project



7.2.1 Host Rocks

Host rocks on the Property consist of greenschist-metamorphosed volcanic, volcaniclastic, and sedimentary rocks of Proterozoic age. These rocks fall within the Townsend Butte facies of the Black Canyon Creek Group of the Yavapai Supergroup, aged 1800-1740 Ma. The Property geology is divided into three (3) lithologic domains: the Hangingwall Mafic Sequence, the Hangingwall Felsic Sequence, and the Footwall Mafic Sequence. Hangingwall and footwall in this setting refer to above and below VMS mineralization, respectively.

7.2.1.1 Hangingwall Mafic Sequence

The Hangingwall Mafic Sequence is characterized by volcaniclastic units that vary from fine to coarse mafic lapilli-tuff to matrix-supported conglomerates, with clasts ranging from 1 mm to 20 cm. These units exhibit notable metamorphic chloritization and frequent Fe-carbonate alteration. Intercalated within these volcaniclastics are chert horizons, massive quartz veins, and sporadic occurrences of oxide-banded iron formation. The sequence also comprises coherent basalts and andesites, and diorites which present as massive flows, some displaying pillow structures and quartz-amygdaloidal textures. Coherent Hangingwall mafic rock is especially prevalent in the vicinity of the Kay deposit.

7.2.1.2 Felsic Sequence

The Felsic Sequence features fine to coarse volcaniclastics, volcanic breccias, coherent rhyolites, and lesser intrusives. The Felsic Sequence is the direct host rock of VMS mineralization on the Project. The entire Felsic Sequence is considered prospective for VMS mineralization.

Felsic volcaniclastic rocks consist of very fine- to coarse-grained rhyolitic quartz-crystal tuffs, felsic breccias, and rare welded and non-welded lapilli tuffs, all of which are either strongly chloritized or sericitized. Graded bedding suggests that stratigraphic tops are to the west.

Coherent rhyolites present as lobes and dikes spanning 1-25 cm in thickness. Characterized by quartz-phyric to aphyric textures, these rhyolites display brecciated margins and form lozenge-shaped bodies embedded within the volcaniclastics. Their composition reflects varied silicification and sericitization and includes local hematite and Fe-carbonate alteration. Felsic breccias within the sequence are characterized by clast sizes between 2 mm and 40 cm, predominantly rhyolitic, with sporadic quartz-porphyry, chert, and intermediate volcanic constituents. The matrix is typically fine-grained and intensely chloritized. These breccias are deposited as wedges proximal to felsic centers, likely as accumulations of rhyolite flow breccias and mass wasting processes during volcanic slope collapse (Baxter & Diekrup, 2023). Less commonly, the sequence contains quartz porphyry and quartz-feldspar porphyritic intrusives, 1-15 m thick, intruding into the volcaniclastics.

The coherent rhyolites and rhyolite breccias have been interpreted as a metamorphosed rhyolite dome or cryptodome hosting the Kay mineralization, specifically where increased porosity and permeability are created through hyaloclastite brecciation or flow brecciation from dome or slope collapse. Within these rocks, SRK (2020a) pointed to a focus on massive rhyolite and zones of metamorphosed hydrothermal alteration as being most prospective, as they show evidence of volcanic centres and/or hydrothermal feeder zones.

Distinct chemical sediments in the Felsic Sequence encompass laminated cherts, with alternating light and darker bands, potentially indicative of Fe-carbonate content. Oxide-facies banded iron formation horizons are characterized by abundant magnetite and hematite, and form discontinuous horizons interpreted as products of intense boudinage. Accompanying these at surface are gossans, primarily appearing as finely laminated chert and carbonate-facies BIF with a distinct surficial jarosite.

7.2.1.3 Graphite-Rich Members

Graphite-rich members, evident in both felsic and mafic volcaniclastic rock, are intercalated sporadically within the sequence. At the Kay deposit, an extensive and consistent graphite unit lies 10-30 m stratigraphically above mineralization and serves as a dependable marker horizon in drilling. Within the middle to upper sections of the Felsic Sequence, graphite manifests as fine streaks in both felsic and mafic volcaniclastic rock. The graphite not only forms networks around clasts but is also observed as graphitic argillites, reaching up to 2 m in thickness, which contain diagenetic pyrite nodules. Other manifestations include graphite as silicified layers in exhalites and as 1-40 cm black chert clasts, which have likely undergone clastic transport or boudinage. These graphitic layers serve as significant stratigraphic markers in the Kay deposit sequence, suggesting deposition was likely influenced by increased biological activity, potentially linked to hydrothermal venting and accompanying elevated Zn levels.

7.2.1.4 Footwall Mafic Sequence

Coherent pillow basalt and andesite largely define the Footwall Mafic Sequence, often appearing as massive flows that span 0.1-2 m in thickness at the surface. Notable features include quartz-amygdaloidal and feldspar-phyric textures. Pillow structures, their remnants (pillow salvages), and flow breccias are especially prevalent in the property’s western region. These rocks exhibit pervasive silicification and chloritization, often accompanied by patchy olive green epidote alteration. Calcite and magnetite are common constituents, with localized occurrences of mm-scale euhedral pyrite either within the matrix or accompanying quartz amygdules. Due to their silicification, these rocks are potentially more prone to boudinage, contributing to greater outcrop occurrence, although their distribution can be discontinuous both laterally and at depth. Intercalated among these flows are fine to coarse mafic volcaniclastic rocks, integrating with the broader footwall sequence. Notably, the footwall pillow basalts serve as a key stratigraphic marker on the property. In the northern and western portions of the property, an intensely chloritized breccia (chlorite breccia) overlies the pillow basalts and andesites.

7.2.1.5 Metasedimentary Rocks

The western edge of the property hosts pelitic and tuffaceous volcaniclastic sedimentary rocks of the Cleator Formation, interpreted to lie unconformably above the Black Canyon Formation. These sediments are rich in carbonates and include chert beds and lenses, dolomite horizons, quartz-bearing meta-andesite, and chlorite-rich meta-tuff layers. Sequences of intermediate to mafic meta-volcanics comprising various interbedded dacitic tuffs, rhyodacite, rhyolite, and andesite have also been mapped. Post-metamorphic granophyre, lamprophyre dikes, and Tertiary sediments are also present in the Project area.

7.2.2 Structure

The structure in the Property area is complex. The host rocks on the Property are intensely deformed, characterized by steeply dipping bedding, foliation, lineations, and folds resulting from three (3) phases of deformation as recorded by SRK (2020a, 2020b, 2020c) and Baxter & Diekrup (2023). The first phase of deformation was the most intense and formed isoclinal folds with attenuated and sometimes separated fold limbs and a pervasive axial-planar S_1 foliation that strikes 186-208° azimuth and dips 63-89° to the west (Figure 7.6). S_1 fold axes have an average trend of 229° azimuth and plunge of 85°. Geologic mapping by SRK (2020a) and Baxter & Diekrup (2023) shows that steeply dipping isoclinal S_1 folding repeats the felsic and mafic schists across the property (Figure 7.4). SRK (2020a) noted that within this folding style, sulfide lenses are likely to be affected by steeply plunging tight folds, with thinned or boudined fold limbs and thickened fold hinges, and possible repetition of sulfide lenses through folding. Geologic modelling of the mineralization using drill data and historical underground mapping shows the nature of S_1 folding.

The second phase of deformation on the Project is shown as an azimuth 320° axial-planar cleavage formed by minor kink folds of 2.5-5 cm amplitude whose fold axes plunge steeply to the northwest and southeast within S_1 foliation. The third phase of deformation formed a shallowly dipping S_3 open cleavage.

Minor post-metamorphic and post-mineral faults that strike generally northwest are difficult to measure but apparently have minor offsets.

In zones of strong to extreme strain in this region, primary features can be distorted into cigar shapes. This is reflected in the shape of the Kay deposit, which has a steeply dipping prolate shape parallel to the mineral stretching lineation. This is an important observation for exploration, and targets should be developed acknowledging that additional VMS bodies may be tubes or prolates rather than tabular bodies.

Figure 7.6: Pervasive S_1 Foliation Axial Planar to Isoclinal Folding on the Property

7.3 Mineralization

7.3.1 Kay Deposit Mineralization

Mineralization on the property occurs principally near the historic Kay Mine workings. In this area, it consists of stratabound lensoid bodies of massive sulfide in a folded horizon that strikes generally north and dips from vertical to 75° west (Figure 7.7). Massive sulfide occurs along a strike length of approximately 430 m and a down-dip extent of over 950 m, as defined by Arizona Metals drilling combined with historical drilling and underground mapping. Drilled widths vary between < 1 m and 125 m, with an approximate true width of mineralization estimated to be 65-97% of reported core width, averaging 80%. Thinner portions are interpreted as fold limbs, and wider portions as thickened fold hinges, forming steeply dipping, generally cigar to tabular shapes that pinch and swell.

Figure 7.7 is a three-dimensional view of the mineralization intersected by Arizona Metals' drilling, showing historic mine workings and Arizona Metals drilling, looking to the northeast. Mineralization is open at depth, along strike to the north, and along strike to the south in some areas. In particular, the recently encountered Kay2 Zone (down plunge extension on the North zone) is open at depth and should be tested for extent. These locations provide good expansion targets for mineralization.

Figure 7.8 depicts a recent interpretation of mineralization and stratigraphy in the Kay deposit.

Exxon previously identified 18 massive sulfide bodies through drilling and underground mining, which they grouped into two (2) principal closely spaced zones, called the North Zone and South Zone. Recent drilling by Arizona Metals suggests greater continuity than proposed by Exxon, and it is now clear that what appeared to Exxon as separate sulfide bodies and separate North and South zones are more likely part of the same mineralized horizon, as shown in Figure 7.7.

Reported historic grades of mineralization are up to 16.6% Cu. Surface assays by Arizona Metals returned 16.4% Cu, and drill samples have assayed up to 20.7% Cu (drill hole KM-22-57B, 802.2-803.8 m), 273 g/t Au (drill hole KM-22-60, 634.3-635.5 m), and 30.0% Zn (drill hole KM-22-62, 645.6-646.2 m). Ratios of Zn/Cu increase as one moves outward from the centre of the massive sulfide bodies, and Zn/Cu ratios are therefore an important exploration vector. The ratio of Na to Zn is also a key mineralization vector: a decrease in Na (resulting from destruction of feldspar) coupled with elevated Zn (introduced by hydrothermal fluids) may signify proximity to mineralization.

The age of mineralization at Kay Deposit is between 1790 and 1740 Ma, the age of the enclosing strata, and likely within the tighter range of 1780-1760 Ma proposed for the majority of Proterozoic VMS deposits.

Prominent beds of iron formation and thin andesite flows at the top of the Townsend Butte facies demarcate the upper limit of felsic volcanism — and therefore the upper limit of prospective VMS stratigraphy.

Figure 7.7: Isometric View looking NE - Kay Deposit Models, Arizona Metals Drill Holes and Underground Workings

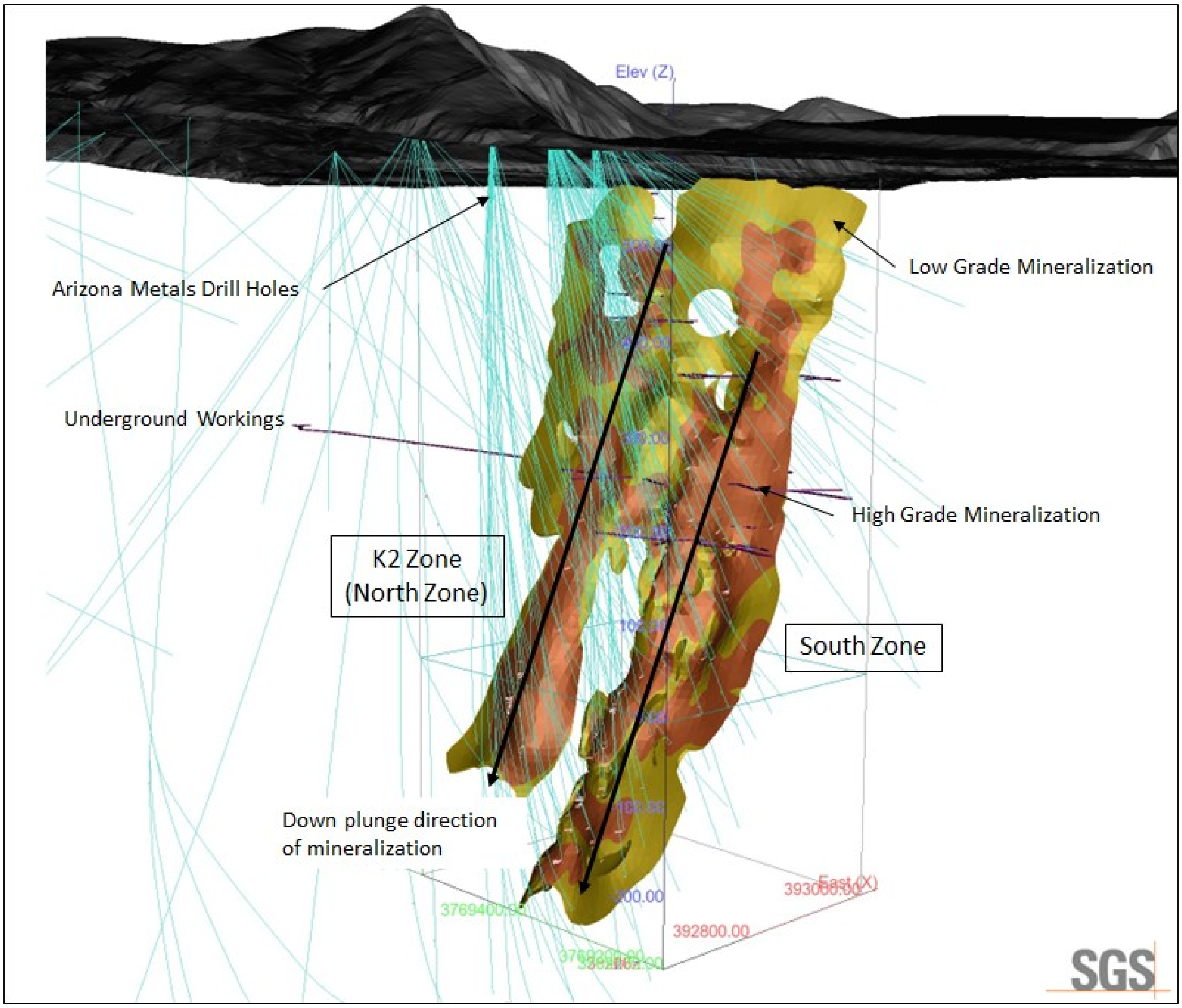
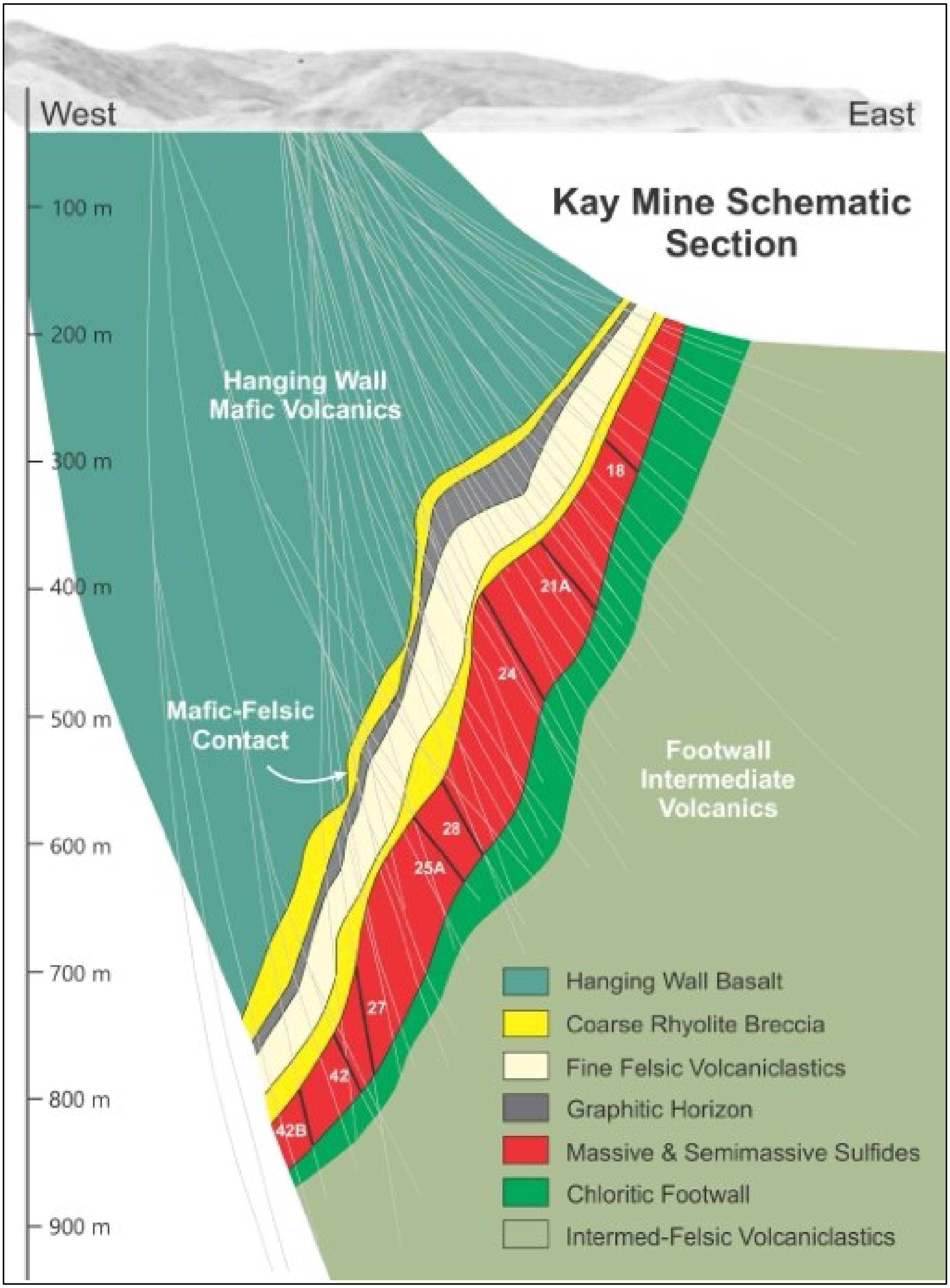


Figure 7.8: Schematic Cross-Section View of Mineralization



Source: Courtesy of Mark Hannington, 2022

Kay Mine sulfide mineralization consists of massive, semi-massive, and stringer-like aggregates of pyrite, arsenopyrite, chalcopyrite, sphalerite, and galena (Figure 7.9 and Figure 7.10). Petrographic studies reveal varying proportions of intergrown pyrite, arsenopyrite, chalcopyrite, sphalerite, tetrahedrite-tennantite, and galena (Figure 7.11). Rare boulangerite ($Pb_5Sb_4S_{11}$) is intergrown with galena; tellurobismuthite (Bi_2Te_3) and hessite (Ag_2Te) occur in chalcopyrite. Gangue minerals include chlorite, quartz, sericite, and dolomite; two (2) generations of carbonate have been observed, one older, inclusion-rich, and a younger, clear more euhedral variety, typically occurring with mineralization. More recent analysis of carbonate trends indicates that ankerite signifies proximity to mineralization.

Hannington (2020) provided an interpretation of the petrographic studies, as follows: “The studied samples are representative of the massive sulfides, stringer mineralization, and altered felsic and mafic volcanic rocks at Kay. The results confirm the strong similarity of the Kay mineralization to other bimodal mafic-felsic-hosted VMS deposits in the Jerome-Prescott area and in other Proterozoic VMS belts (e.g., Flin Flon-Snow Lake, Skellefte). The sulfide assemblage is mineralogically simple and typical of polymetallic ores in this type of deposit. Textures observed in thin section show that the mineralization and host rocks are strongly deformed, with locally intensive shearing and a strong penetrative fabric, but no significant metamorphic recrystallization or annealing of the sulfide minerals. The result is a fine granoblastic texture that should be amenable to conventional mineral processing.”

The sulfide- (and non-sulfide) assemblages confirm a low-temperature origin for the pyritic Zn-rich mineralization, indicated by low-Fe sphalerite and Mg-rich chlorite, and higher temperatures occurring with the chlorite stringer mineralization and Cu-rich sulfides. Possible meta-exhalite was identified in thin section, namely quartz-carbonate-graphite schist and the hematitic tuff that may serve as marker units. The abundant carbonate gangue and pervasive alteration of the felsic volcaniclastic host rocks suggest a subseafloor replacement origin for much of the mineralization.

Pyrite is the dominant sulfide mineral (30% modal abundance, on average), followed by sphalerite (10-15%), chalcopyrite (10-15%), and arsenopyrite (7%), with minor galena, tetrahedrite, and tennantite (all < 1%). Chalcopyrite is mainly interstitial to pyrite but locally more massive. It also occurs as disseminations in the chloritic stringers and with sphalerite and galena in polymetallic samples. Sphalerite is mainly intergrown with pyrite in polymetallic assemblages that also contain minor amounts of tennantite, tetrahedrite, galena, and chalcopyrite. The sphalerite is notably Fe-poor, evidenced by its translucence and pale red colour in transmitted light.

Arsenopyrite is most abundant in the Zn-rich mineralization from the South Zone (13% modal abundance), where it is intergrown with pyrite and sphalerite. Fine crystals of arsenopyrite occur individually and in aggregates in the pyrite-sphalerite assemblage. At the scale observed, the arsenopyrite is mostly inclusion-free. Arsenopyrite is less common in the Cu-rich massive sulfide and stringer mineralization (< 5% modal abundance on average).

Galena, tetrahedrite and tennantite are mainly in the Zn-rich samples, in polymetallic aggregates intergrown with sphalerite and pyrite. Tetrahedrite also occurs with chalcopyrite (sample 11-1860). Tellurobismuthite, altaite, and hessite were found in the Cu-rich samples as inclusions in pyrite and chalcopyrite. Though rare, these are typical accessory minerals in VMS deposits.

The mineralized samples all have a fine-grained, granoblastic texture typical of low-grade metamorphic recrystallization of VMS ores. The typical grain sizes of the sulfide minerals are between 25 and 250 microns. The sulfides exhibit complex intergrowths and intense fracturing of individual grains (especially pyrite), but they do not show extensive annealing or porphyroblastic growth that is common at higher grades of metamorphism (e.g., as in Snow Lake). Pyrite and arsenopyrite are the main brittle phases; all other sulfide minerals show limited deformation or remobilization. Interstitial carbonate, with lesser chlorite and muscovite, is present throughout the mineralized samples.

From the distribution of the samples, strong metal zonation can be inferred, with chloritic stringer mineralization at the base, through Cu-rich massive sulfide, to overlying or adjacent Zn-rich zones. Lower-temperature mineralization is generally in stratigraphically higher or outer zones, and pyrite-carbonate may cap the lenses, although carbonate is also present in the stringer zones. The inferred zonation is consistent with broad sheet-like lenses like the nearby Iron King deposit.

No free gold or electrum was observed in the thin sections. The gold grades are at the limit for easy detection of free gold by reflected light microscopy, so this is not surprising. However, the samples should be inspected more closely by SEM to confirm the siting of the gold. At least one (1) sample showed hessite and altaite locked in pyrite, where native gold or electrum also would be expected to occur. Four (4) other samples are identified in the recommendations for additional work.

Silver is most likely present in tetrahedrite and possibly in galena or tennantite; one (1) sample contained the Ag-telluride hessite. Silver is also possibly in solid solution in chalcopyrite, as at Kidd Creek, but this also needs to be tested. One (1) sample (B300190) with 2.2 wt.% Pb and 1,000 ppm Sb contains 350 ppm Ag, consistent with the presence of Ag-bearing tetrahedrite (freibergite). The Pb-Sb sulfosalt boulangerite was also identified in sample 15-1668 (B300573), which contains up to 192 ppm Ag in the drill core assays. SEM or microprobe analyses of the Ag-bearing minerals would provide the information needed for a full mineral balance.

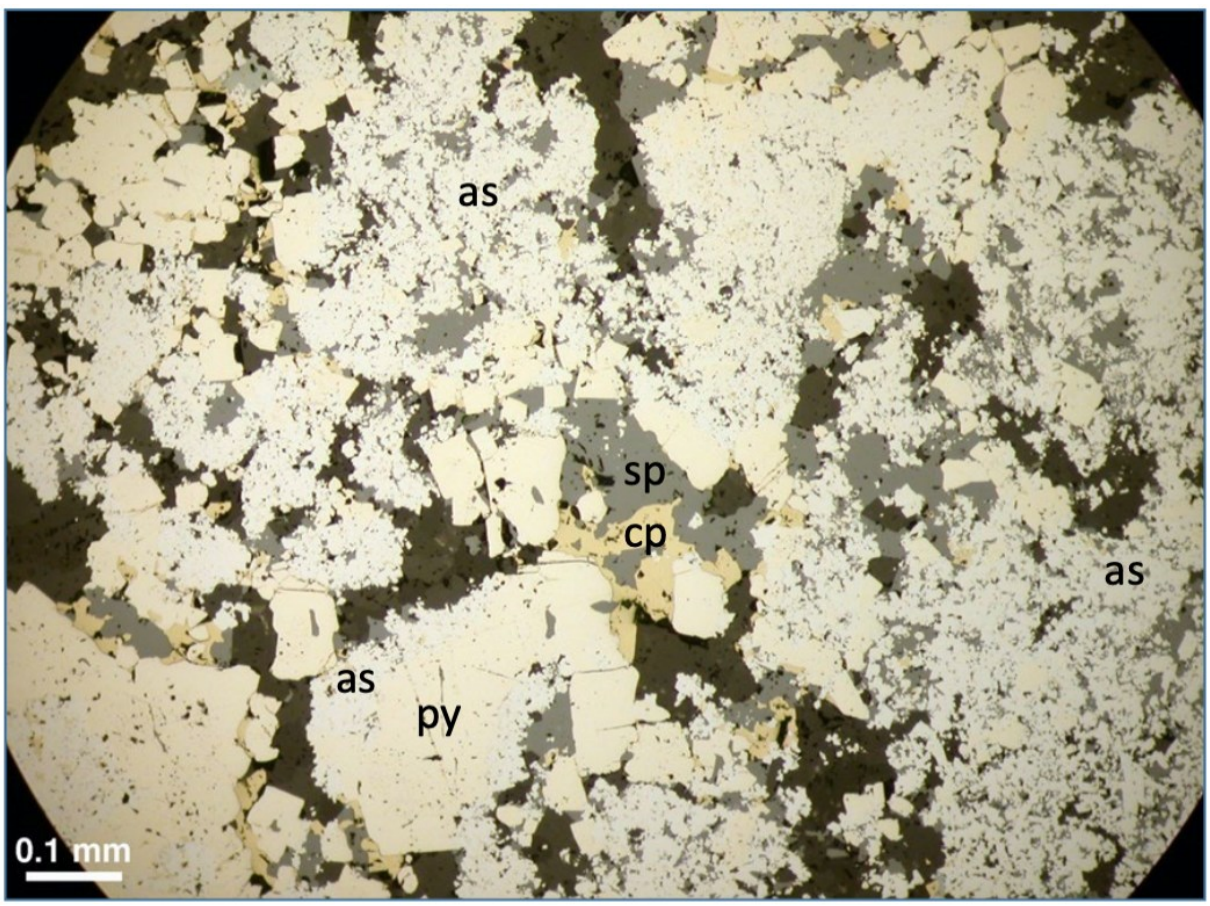
Multi-element analyses of drilled mineralization show a deposit dominant in Cu, Au, and Zn, with minor Pb and Ag. Elevated trace elements include, As, Cd, Co, and Sb. Statistical correlations between major metals of interest and trace elements are as follows (listed in decreasing order).

- Cu—Co, Bi.
- Au—As, Cd, Zn, Ag.
- Zn—Cd, Pb, Au, As.

Figure 7.9: Massive Sulfide Mineralization Collected by Author on Mine Dump at No. 1 Shaft (Smith, 2024)

Figure 7.10: Massive Chalcopyrite in Drill Core From a 1.2-M Sample Grading 9.8% Cu, 6.1 g/t Au (Drill Hole KM-21-26, 581.6-582.8 M) (Smith, 2024)

Figure 7.11: Photomicrograph of Mineralization Showing Intergrown Pyrite, Chalcopyrite, Sphalerite and Arsenopyrite (Smith, 2024)



**Note: Reflected Light, Drill Hole KM-20-11, 1,823 ft, 555.65 m.*

7.3.2 North Central Target Mineralization

Mineralization on the North Central Target is exposed in two (2) mineralized horizons as traced on the surface with geologic mapping and rock sampling and intersected at depth with drilling (Figure 7.12). Mineralization consists of sulfide minerals (pyrite, chalcopyrite, sphalerite) in disseminated, stringer, and semi-massive styles, with zones of anomalous gold, copper, and zinc accompanied by sodium depletion.

Stretching north from the Kay deposit and folding south along a syncline and then north along an anticline, the Kay mineralized horizon is exposed over a strike length of approximately 3 km, about 2 km of which has not been drilled (Figure 7.12). Surface assays from this horizon grade up to 9.6% Cu. Drill results from the Kay horizon on the North Central target include 2.7 m @ 0.5% CuEq (KM-22-95) and 3.2 m @ 0.36% CuEq (KM-24-161).

The recently discovered Pad 10 horizon is located stratigraphically above the Kay horizon, exposed along 1.7 km of strike length on the property, with just under 1 km remaining to be drill tested (Figure 7.12). Surface assays from this horizon grade up to 11.9% Cu. Drill results from the Pad 10 horizon on the North Central target include 0.5 m @ 11.3% CuEq (KM-24-153), 0.6 m @ 1.7% CuEq (KM-24-151), 0.6 m@ 1.2% CuEq (KM-24-157), 0.9 m @ 0.8% CUEq (KM-24-150), and 0.6 m @ 0.7% CuEq (KM-24-158).

7.3.3 West Target Mineralization

Mineralization also occurs on the West Target, as a north-trending mineralized horizon displaying sulfide minerals and anomalous trace elements intersected in eight (8) drill holes over 735 m of strike length, and sampled at surface over a strike length of approximately 385 m (Figure 7.13). The West Mineralized Horizon exhibits sulfide minerals (pyrite, pyrrhotite, sphalerite, and chalcopyrite) occurring in disseminated, stringer, and semi-massive styles, with broad zones of highly anomalous gold, copper, and zinc, accompanied by sodium depletion, a key indicator of hydrothermal activity in VMS systems. Mineralization appears to be strengthening to the north, where surface exposures of coherent rhyolite indicate a volcanic center and possible locations of massive sulfide mineralization.

Figure 7.12: Plan Map Showing North Central Target Mineralization (Smith, 2024)

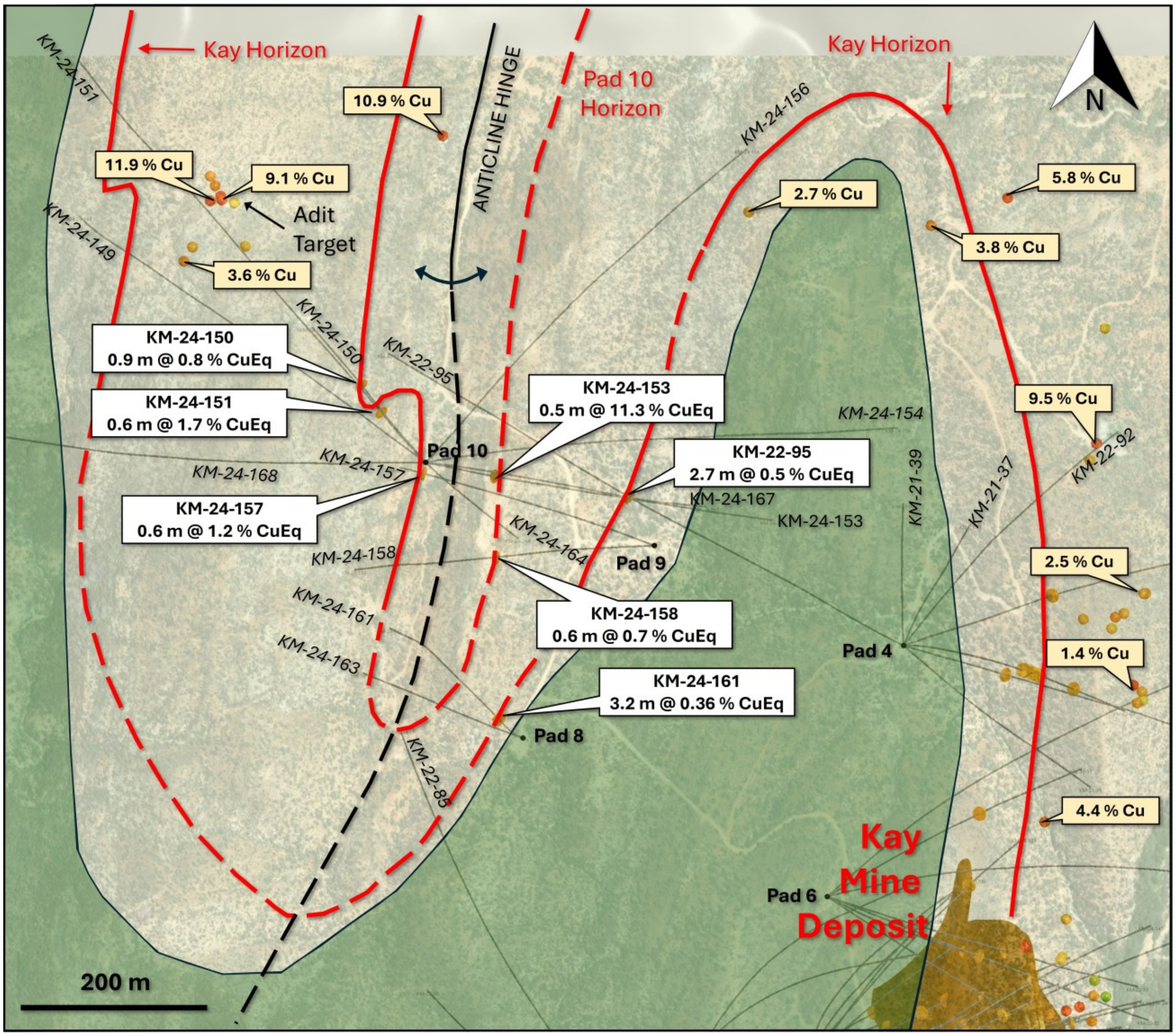
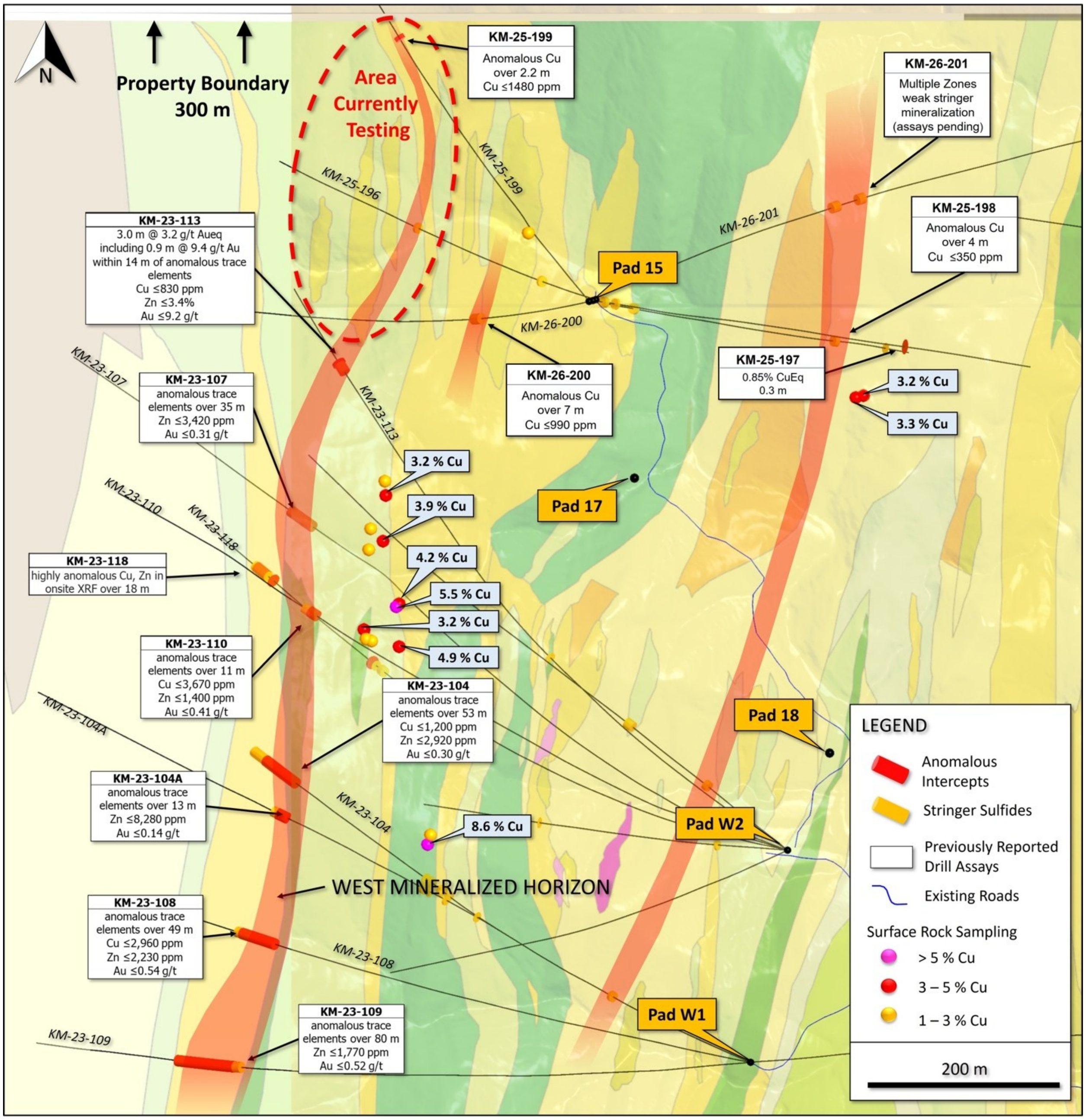


Figure 7.13: Plan Map Showing West Target Mineralization



7.4 Alteration

Historical descriptions of hydrothermal alteration on the Kay Project are limited, but consistent with that typical of volcanogenic massive sulfide deposits elsewhere. Chlorite, dolomite, and quartz alteration occur in the footwall to massive sulfide bodies on the property. This footwall alteration occurs in three (3) forms. First, widespread layers of black, Mg-rich chlorite occur in the footwall to mineralization in both the North and South zones, including zones below the North Zone 1000 level and the South Zone's “second” massive sulfide layer, presumably the 1200 level. Outcropping zones of this black chlorite mineralization are also shown on the summary project geology map. Second, silicification is present in rhyolite lapilli tuffs in the North Zone accompanied by minor pyrite and crosscutting dolomite-chalcopyrite veins; and in the footwall of the North Zone 1500 level as quartz-pyrite veins. Third, chlorite and dolomite alteration are present within “stringer ore” in the South Zone of mineralization. The increase in Mg in chlorite toward mineralization provides an excellent exploration vector. Footwall alteration shows strongly anomalous levels of Cu in the 60-90 metres below the mineralized horizon. Hangingwall alteration above the sulfide horizons consists of a 30-45 m thick section of silver-gray sericite phyllites immediately above sulfides in the North Zone, which is likely sericite alteration. Hangingwall alteration does not show anomalous levels of base metals.

Alteration studies by SRK (2020b) indicate that two (2) alteration indexes increase toward mineralization. The Ishikawa Index is a measure of K and Mg added to a rock by alteration, and the chlorite-carbonate-pyrite index (CCPI), measures the addition of Mg and Fe by alteration. Mapping of these indexes helped define the folding model of the deposit.

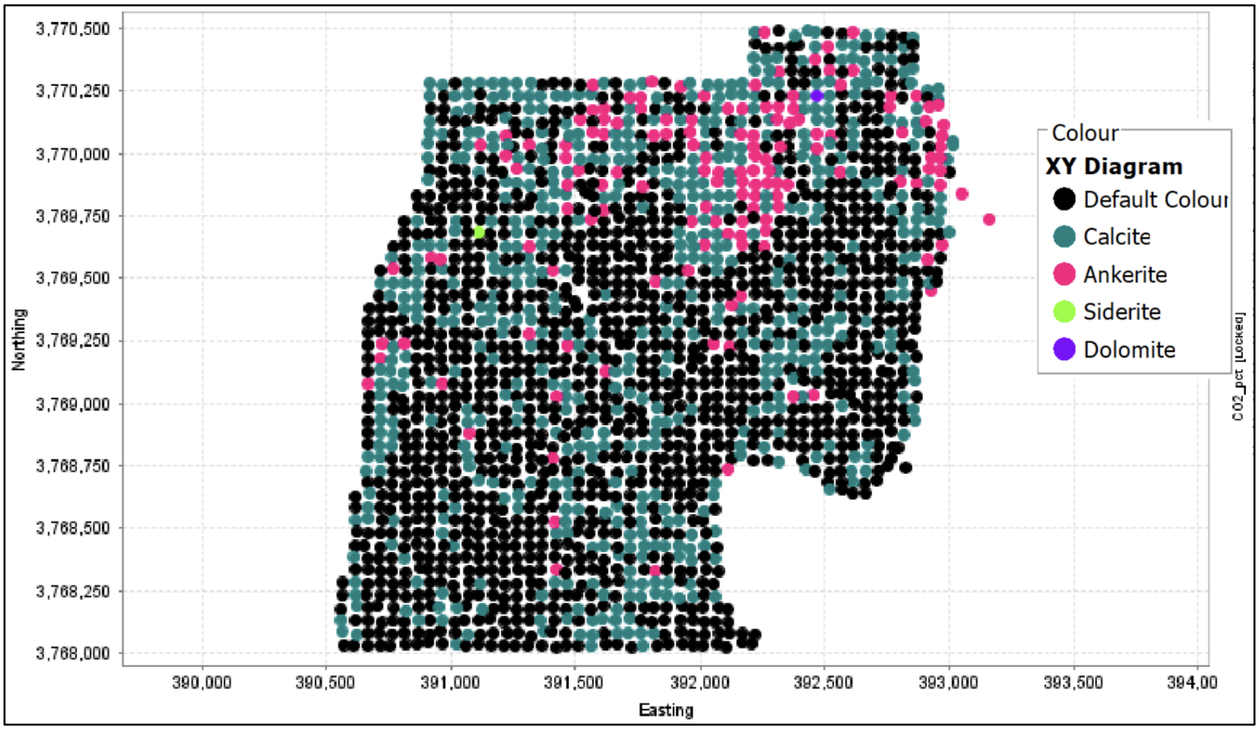
Petrography revealed abundant proximal carbonate and chlorite alteration, with more widespread sericite alteration. “Carbonate is the dominant alteration in unmineralized volcanic rocks (~30% modal abundance, on average), compared to 20% quartz, 20% muscovite, and 20% chlorite. Some banded carbonate may represent seafloor precipitation (i.e., exhalite), but most is in the matrix of the felsic volcaniclastics, consistent with subseafloor replacement. It is less abundant in the footwall quartz-sericite and quartz-chlorite schist, where it occurs as unreplaced clots. Muscovite is present throughout the mineralized samples and altered felsic volcanic units. Mg-rich chlorite is mostly restricted to the mineralization. The low Fe content of chlorite in the Zn-rich samples is consistent with the interpreted low temperature of formation of this assemblage. Chlorite appears more Fe-rich in the stringer mineralization, but this needs to be confirmed by microprobe or SEM analysis” (Smith, 2024).

Mineralization at Kay is accompanied by pervasive carbonate alteration “The most intense carbonate alteration occurs within the massive, semi-massive, and stringer sulfides, and within the footwall of the mineralization. The carbonate mineralogy, which includes dolomite, ankerite, and siderite, forms globular and nodular masses proximal to the massive sulfides and appears as finely disseminated anhedral constituents more distally. ‘Intense’ carbonate alteration is most commonly observed within the Kay Mine drill holes adjacent to and within the sulfide horizon. The most intense carbonate ‘alteration’ is characterized by a pervasive nodule-like texture, manifesting as globular masses that often appear as discrete, orbicular spots dispersed throughout the host rock. These spots, ranging from isolated inclusions to more confluent aggregates, vary in their distribution, transitioning from sparse pinpoint occurrences to more densely packed clusters. Additionally, carbonate alteration locally forms meandering, anastomosing ‘veins’, reminiscent of serpentine pathways” (Smith, 2024).

Laboratory carbon / carbonate analyses indicate that the abundance of inorganic carbon can be a vectoring tool toward mineralization in felsic host rocks on the Kay Project. Laboratory analyses show that carbonate is widespread in intermediate and mafic host rocks, identified in thin section as fine-grained anhedral ankerite and dolomite. However, carbonate is not widely distributed in the felsic host rocks, the most prospective host lithologies for VMS mineralization; thus, more focused discrete zones of carbonate within felsic host rocks are a first-order screening factor, since discrete carbonate zones may suggest that they are products of VMS related hydrothermal fluids and therefore prospective for mineralization.

However, “carbonate types must be strictly distinguished in order to determine significance on a property-wide scale. CO_2 concentration alone will not reveal vectoring significance” (Smith, 2024). Thus, a key alteration vectoring tool is the composition of carbonate minerals. Analysis of onsite portable x-ray fluorescent (pXRF) and laboratory hyperspectral measurements (Terraspec) indicate that dolomite and ankerite are characteristic of carbonate alteration proximal to mineralization, especially where they are Mn-rich. K means clustering analysis of pXRF data shows proximal additions of Ag, As, Bi, Ca, Cd, Cu, Mn, Pb, S, Sb, Se, Th, and Zn. This cluster contains the highest Mn concentration, suggesting manganiferous composition (ankerite). Somps interprets that the prospective carbonate alteration is “dolomite where iron commonly substitutes for some of the magnesium, in a complete series that likely extends between dolomite and ankerite.” Laboratory hyperspectral analyses of drill-core samples indicate that the presence of elevated FeOH values in combination with MgOH absorption features from 2,220-2,230 nm can be indicative of mineralization-proximal iron-bearing carbonates. Mapping of carbonate compositions derived from laboratory data indicate that felsic host rocks in the northern portion of the property contain relatively high concentrations of ankerite (Figure 7.14).

Figure 7.14: Carbonate Composition Map Derived from Laboratory Analyses of Rock-Grid Samples (Smith, 2024)



8. DEPOSIT TYPES

The Kay Deposit consists of structurally deformed and metamorphosed, stratabound, polymetallic massive, semi-massive and stringer sulfide mineralization. The sulfides contain copper, gold, zinc, lead and silver mineralization. Mineralization of the Kay Deposit shows the geological, mineralogical and geochemical characteristics of Volcanogenic massive sulfide (VMS) deposits.

8.1 Volcanogenic Massive Sulfide (VMS) Deposits

Volcanogenic massive sulfide (VMS) deposits are also known as volcanic-associated, volcanic-hosted, and volcano-sedimentary-hosted massive sulfide deposits (Galley et al. 2007, and references therein). They typically occur as lenses of polymetallic massive sulfide that form at or near the seafloor in submarine volcanic environments. They form from metal-enriched fluids associated with seafloor hydrothermal convection. Their immediate host rocks can be either volcanic or sedimentary. VMS deposits are major sources of Zn, Cu, Pb, Ag, and Au, and significant sources for Co, Sn, Se, Mn, Cd, In, Bi, Te, Ga, and Ge. Some also contain significant amounts of As, Sb, and Hg.

VMS deposits form at, or near, the seafloor through the focused discharge of hot, metal-rich hydrothermal fluids. For this reason, VMS deposits are classified under the general heading of “exhalative” deposits, which includes sedimentary exhalative (SEDEX) and sedimentary nickel deposits.

Most VMS deposits have two (2) components (Figure 8.1). There is typically a mound-shaped to tabular, stratabound body composed principally of massive (> 40%) sulfide, quartz and subordinate phyllosilicates, and iron oxide minerals and altered silicate wall-rock. These stratabound bodies are typically underlain by discordant to semiconcordant stockwork veins and disseminated sulfides. The stockwork vein systems, or “pipes”, are enveloped in distinctive alteration halos, which may extend into the hanging-wall strata above the VMS deposit

Figure 8.1: Schematic Diagram of the Modern TAG Sulfide Deposit on the Mid-Atlantic Ridge

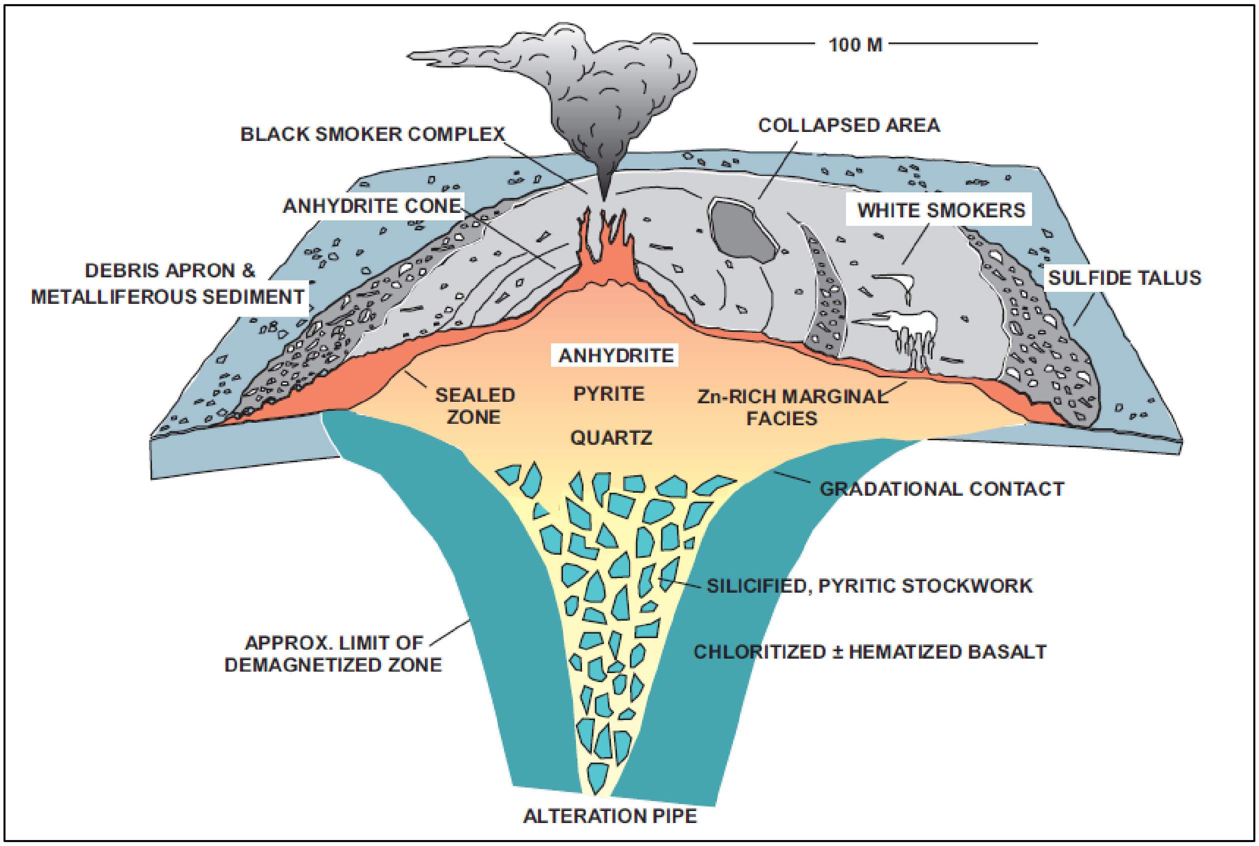
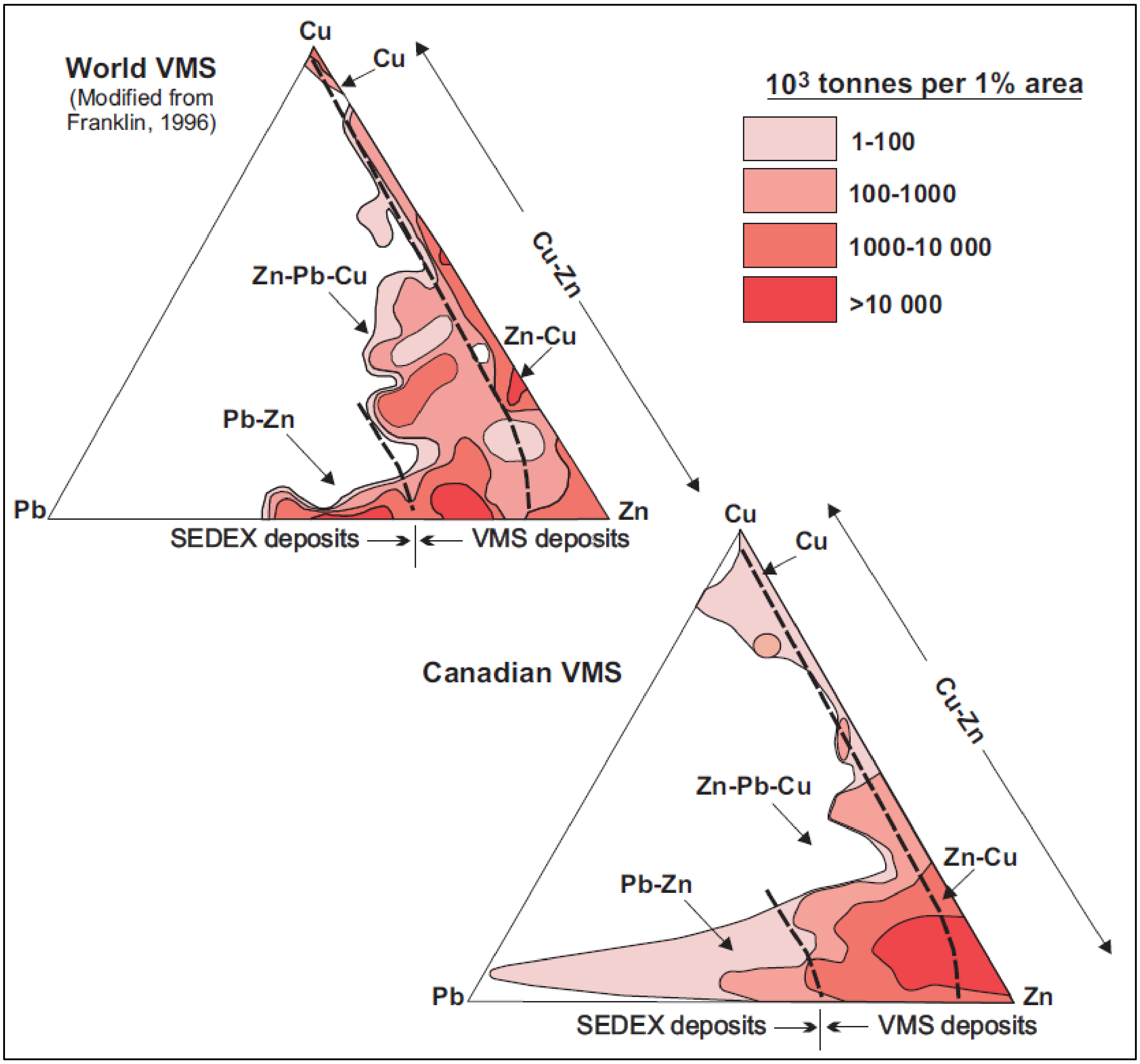


Figure 8.1 represents a classic cross-section of a VMS deposit, with a concordant semi-massive to massive sulfide lens underlain by a discordant stockwork vein system and associated alteration halo, or “pipe” (Galley et al. 2007).

VMS deposits are grouped according to base metal content, gold content, and host-rock lithology. The base metal classification is perhaps the most common. VMS deposits are divided into Cu-Zn, Zn-Cu, and Zn-Pb-Cu groups according to their contained ratios of these three (3) metals (Figure 8.2). The Cu-Zn and Zn-Cu categories for Canadian deposits were further refined into Noranda and Mattabi types, respectively, by including the character of their host rocks (mafic vs. felsic, effusive vs. volcaniclastic) and characteristic alteration mineral assemblages (chlorite-sericite dominated vs. sericite-quartz ± carbonate-rich). The Zn-Pb-Cu category was added in order to more fully represent the VMS deposits of Australia (Figure 8.2). A simple bimodal definition of “normal” versus “Au-rich” VMS deposits was also created (Figure 8.3). This originally was intended to identify deposits that are transitional between VMS and epithermal deposits (Figure 8.4). Further research has indicated a more complex spectrum of conditions for the generation of Au-rich VMS related to water depth, oxidation state, the temperature of the metal-depositing fluids, and possible magmatic contributions. Au-rich VMS deposits are arbitrarily defined as those in which the abundance of Au in ppm is numerically greater than the combined base metals (Zn+Cu+Pb in wt.%, Figure).

A third classification system that is gaining acceptance is a five-fold grouping. This system classifies VMS deposits by their host lithologies (Figure 8.4), which includes all strata within a host succession defining a distinctive time-stratigraphic event. These five (5) different groups are bimodal-mafic, mafic-backarc, pelitic-mafic, bimodal-felsic, and felsic-siliciclastic. To this is added a sixth group of a hybrid bimodal felsic, which represents a cross between VMS and shallow-water epithermal mineralization (Figure 8.4). These lithologic groupings generally correlate with different submarine tectonic settings. Their order here reflects a change from the most primitive VMS environments, represented by ophiolite settings, through oceanic rifted arc, evolved rifted arcs, continental back-arc, to sedimented back-arc.

Figure 8.2: Base Metal Classification Scheme of Worldwide and Canadian VMS Deposits to Include the Zn-Pb-Cu Class



The preponderance of Cu-Zn and Zn-Cu VMS deposits in Canada is due to the abundance of Precambrian primitive oceanic arc settings. Worldwide, there is a larger proportion of felsic-hosted, more Pb-rich continental rift and continent margin arc settings (Galley et al. 2007).

Figure 8.3: **Classification of VMS Deposits Based on their Relative Base Metal (Cu+Zn+Pb) Versus Precious Metal (Au, Ag) Contents (Galley Et Al. 2007)**

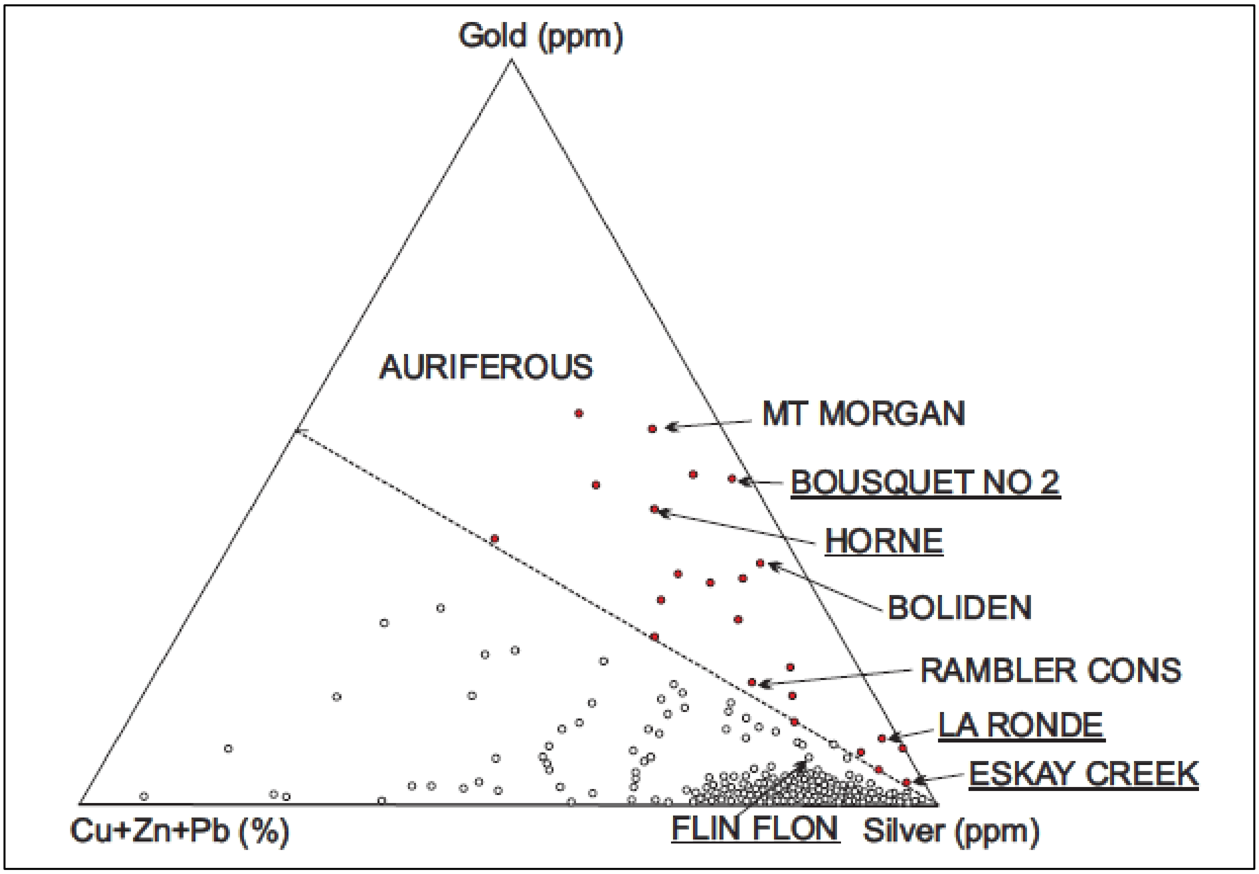
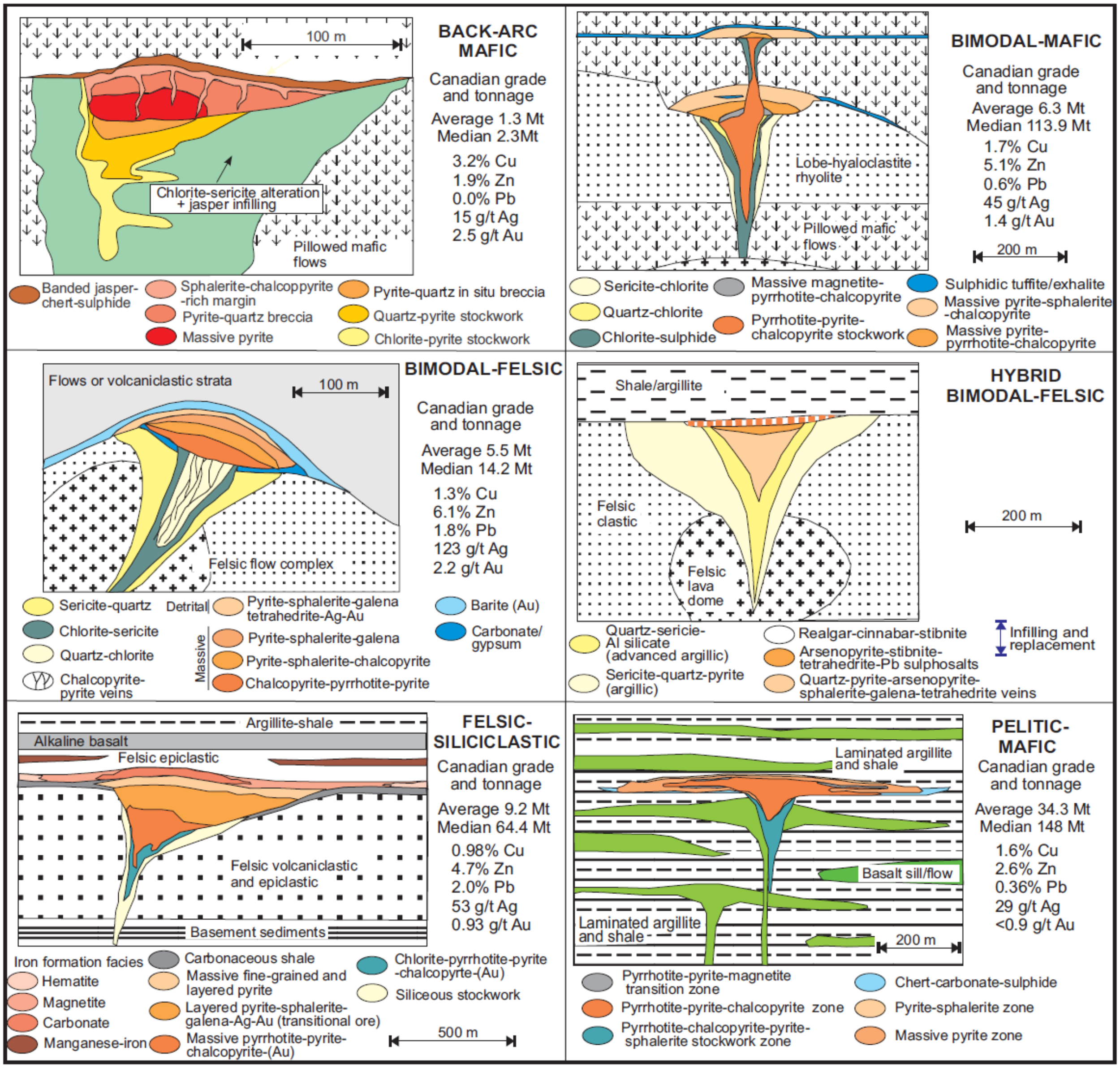


Figure 8.4: **Graphic Representation of Lithological Classifications with Addition of Hybrid Bimodal Felsic As a VMS-Epithermal Subtype of Bimodal-Felsic (Galley et al. 2007)**



9. EXPLORATION

9.1 Pre-Exxon Exploration

The only data that exists from the early, pre-Exxon exploration efforts on the property are mine plan maps and cross sections produced by the Kay Copper Company and New Jersey Zinc (Smith, 2024). These include the locations of underground workings and underground drill holes, and assay results from mine channel samples (including many sample widths) and drill assays. Mine plan maps indicate several hundred underground samples and at least 103 drill holes (89 by Kay Copper Company and 14 by New Jersey Zinc) with many plotted assay results. This is abundant data that, if verified with modern drilling and properly digitized into a 3D geologic model, could be integrated into a new resource estimate for the Project.

9.2 Exxon Minerals Exploration

Exxon Minerals explored the property between 1972 and the mid-1980s, reportedly spending over USD 1M. There are several gaps in the available reports, so the procedures, parameters, methods, quality, and other details of the exploration work are not completely available. Exxon's work is summarized here from available reports. Exploration work and results during 1977-1982 included the following.

- Mapping the area around the Kay Deposit at a scale of 1" = 200', resulting in a detailed understanding of the host rocks, structure, and geologic setting of the mineralization.
- Relogging drill core and cuttings.
- Examining 143 thin sections from surface and drill core.
- Splitting and assaying for Cu, Pb, and Zn 610 m (2,000 feet) of drill core from holes K-9, K-10A, and K-12; assays indicate that Zn / Cu ratios increase with distance from mineralization.
- A stream sediment sampling program, showing small base-metal anomalies immediately around the No. 1 Shaft.
- Geophysical surveys including complex resistivity (CR), CSAMT, Turam, and several generations of induced polarization (IP). There is a description of complex resistivity anomalies defining the Kay mineralized horizon over a strike length of 460-610 m (1,500-2,000 feet), which was possibly open to the south of the No. 4 Shaft.
- A soil sampling survey that included the Kay Deposit area, resulting in a mild Hg anomaly over the mine area. Soil grid geochemistry was "instrumental" in finding the Greyhound mineralized zone to the northwest of the Kay Deposit.

- Reviewing underground geology and assay data and including them on mine level plans and cross sections.

9.3 Rayrock Mines Exploration

In the late 1980s, Rayrock Mines Inc. optioned the property from Exxon Minerals and formed a joint venture with American Copper and Nickel Company. Rayrock conducted data review, induced polarization (IP) and electromagnetic (EM) geophysical surveys, geologic mapping, and rock sampling. Most of the data is not available. A draft map shows IP chargeability anomalies coincident with Arizona Metals' Central / MX-2 anomaly. Rayrock conducted two (2) drill campaigns: in 1991, consisting of six (6) reverse-circulation holes; and in 1993, comprising five (5) core holes. Hole depths are known only for K91-3 (244 m) and K93-1 (280 m).

9.4 Arizona Metals Exploration

Since 2019, Arizona Metals has performed the following exploration work:

- Staked 74 additional unpatented lode mining claims covering 566.8 ha (1,400.1 ac).
- Staked two (2) additional unpatented placer mining claims covering 16.2 ha (40 ac) co-located with unpatented lode mining claims.
- Purchased a total of 78.0 ha (192.7 ac) of private land in three (3) transactions.
- Collected and analyzed 30 due diligence rock samples.
- Geologic reconnaissance to the west of the patented claims.
- Digitized all historical Project data and conducted 3-dimensional modelling.
- Topographic survey by drone aircraft.
- VTEM geophysical survey followed by reprocessing and interpretation.
- Ground electromagnetic (EM) geophysical survey in three (3) areas of the Project.
- Borehole electromagnetic (BHEM) geophysical survey in selected Arizona Metals drill holes.
- Geophysical gravity survey.
- Soil and rock sampling.
- Geologic mapping.
- Structural interpretation.

- Alteration and trace-element studies.
- Petrographic studies.

9.4.1 Geologic Reconnaissance and Claim Staking

The company conducted initial geologic prospecting of the area west of the historic Kay Deposit, identifying the gossan outcrops near the VTEM anomaly (see below). Thirty (30) rock samples were collected and analyzed, as described in Data Verification, below. Based on prospecting results, Arizona Metals staked 50 additional new mining claims in 2019, followed by three (3) unpatented lode claims and two (2) unpatented placer mining claims in 2022, and 21 unpatented lode mining claims in 2023.

9.4.2 Data Digitizing and Drone Topography Survey

Arizona Metals commissioned the digitization of all the historical data on the Project, including historic drill data, underground workings, and underground samples. This data was incorporated into a three-dimensional computer model for exploration planning. Arizona Metals also commissioned several drone surveys to map the topography on the Project, which has been integrated into the 3D digital model.

9.4.3 VTEM Geophysical Survey

During March 2019, Geotech Ltd. of Aurora, Ontario, flew a helicopter airborne VTEM (versatile time domain electromagnetic) survey of the central portion of the property, totalling 107 line-km at 50-m spaced lines (Geotech, 2019a). The survey detected three (3) anomalies: over the existing Kay mineralization, a Central anomaly approximately 600 m to the east of the Kay mineralization, and a Western anomaly 1.6 km east of Kay.

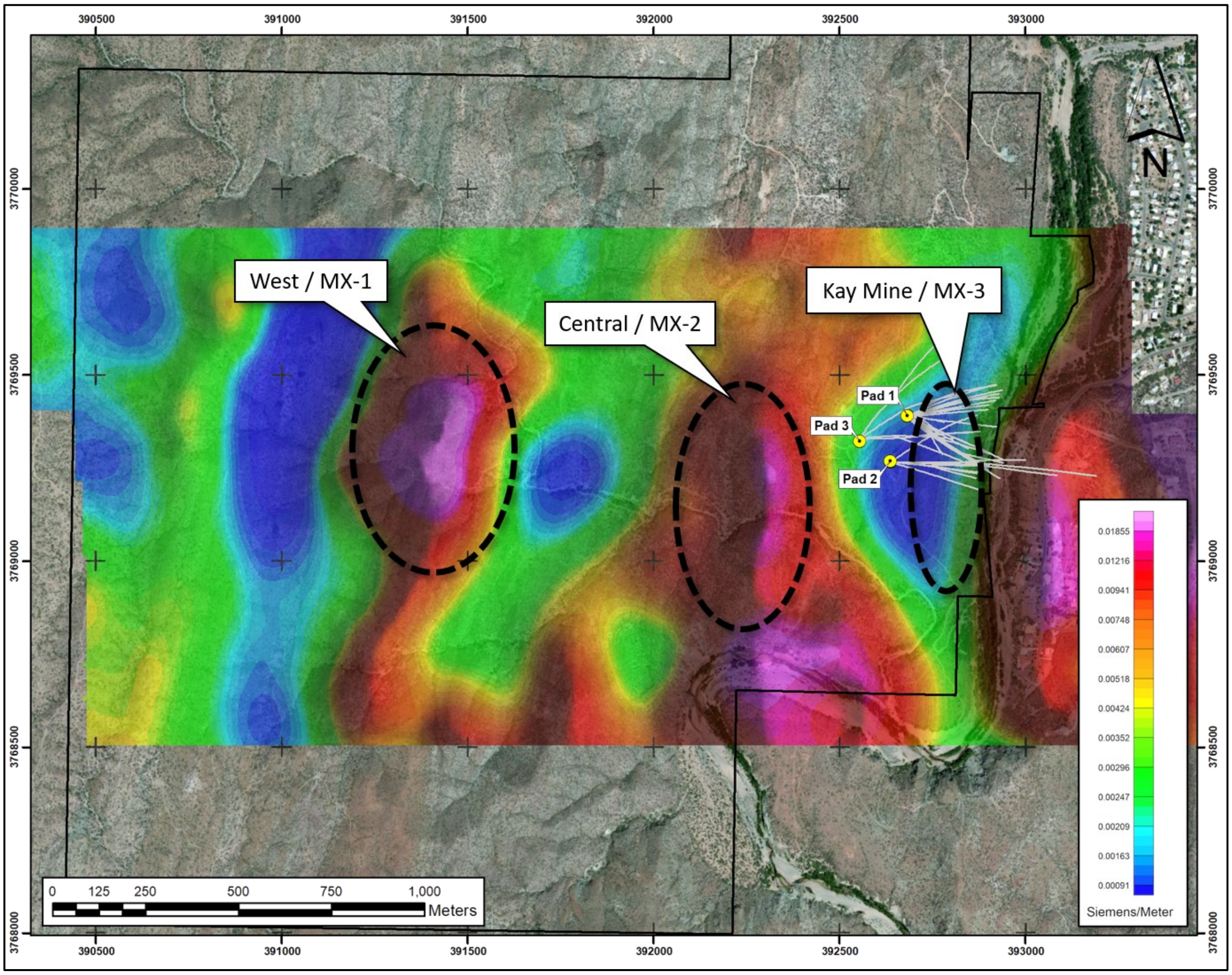
Following the VTEM survey, Geotech performed Maxwell plate modelling and interpretation (Geotech, 2019b). Maxwell plate modelling is a processing method that refines the VTEM anomalies by generating a series of rectangular plates to represent the possible causative geologic bodies. Geotech’s data was reviewed by consulting geophysicist Tom Weis (Weis, 2020a), who cautioned the use of Maxwell plate modelling alone, stating that the method can be useful but may be misleading, especially when “virtual” plates are used to influence the interpretation, as Geotech did on the West anomaly. Weis recommended detailed processing furthermore. This was subsequently performed by Computational Geosciences of Vancouver, B.C., who provided digital models directly to Weis, who interpreted them and prepared four (4) reports (Weis, 2020b, 2021a, 2021b, 2021c). Arizona Metals has imported the digital models into its 3D model and will use them for drill targeting.

The largest and most well-defined VTEM anomaly outside the historic Kay mineralization is the West anomaly, labelled MX-1 in Geotech's and Weis' reports. In his interpretation report, Weis (2021b) delineated this as a steeply dipping, north-trending, south-plunging zone of high conductivity approximately 150 m wide east-west by 450 m long north-south (Figure 9.1) and extending to approximately 500 m depth. Data shows evidence for multiple stacked conductor lenses within the anomaly. Weis defined eight (8) drill targets in this anomaly and recommended drilling of all high-conductivity features in the area.

The Central VTEM anomaly, also called MX-2, is a single north-south striking conductivity high anomaly of weak to moderate strength dipping steeply to the west (Weis, 2021c; Figure 9.1). The anomaly is approximately 150 m wide east-west, 500 m long, and extends to approximately 350 m depth. Weis outlined two (2) priority targets recommended for drilling.

The Kay Deposit anomaly (labelled MX-3) is coincident with the mineralization in the historic Kay Deposit as identified by underground workings, previous drilling, and Arizona Metals' drilling. This is a large and strong anomaly (Figure 9.1) and serves as an orientation anomaly because of the presence of known mineralization. Additional details of this anomaly are discussed below.

Figure 9.1: VTEM Anomalies

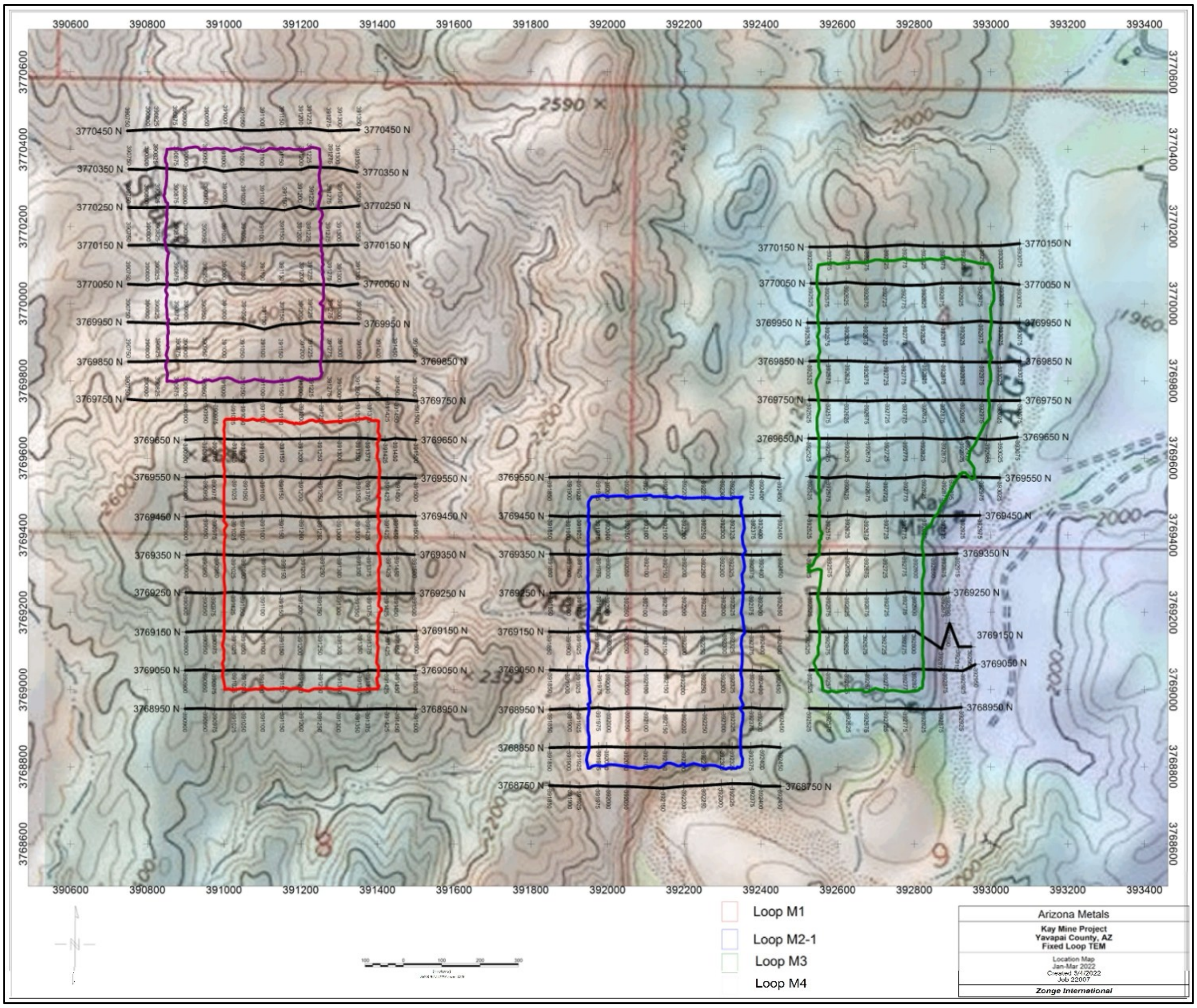


**Note: MX-3 is subtle and was further delineated with a borehole EM Survey; the large anomaly to the East of MX-3 is attributed to power lines.*

9.4.4 Ground EM Geophysical Survey

Between January and March 2022, Zonge International conducted a ground-based transient electromagnetic (ground EM) survey on three (3) areas of the Kay Project. The three (3) areas consisted of the following, with anomaly names retained from the airborne VTEM survey: 1) the Kay deposit and its northern extension (MX-3); 2) the Central anomaly (MX-2); and 3) the West anomaly (MX-1). The Kay Deposit and Central surveys were conducted with single fixed ground loops 400 x 1,100 and 400 x 700 m in extent, respectively. The West anomaly was surveyed with two (2) fixed ground loops, 400 x 600 and 400 x 700 m in extent (Figure 9.2). Surveys were conducted on stations spaced 50 m apart, on parallel lines spaced at 100 m. Data was processed by Computational Geosciences. The resulting 3D models were interpreted, and targets were generated by independent geophysicist Tom Weis.

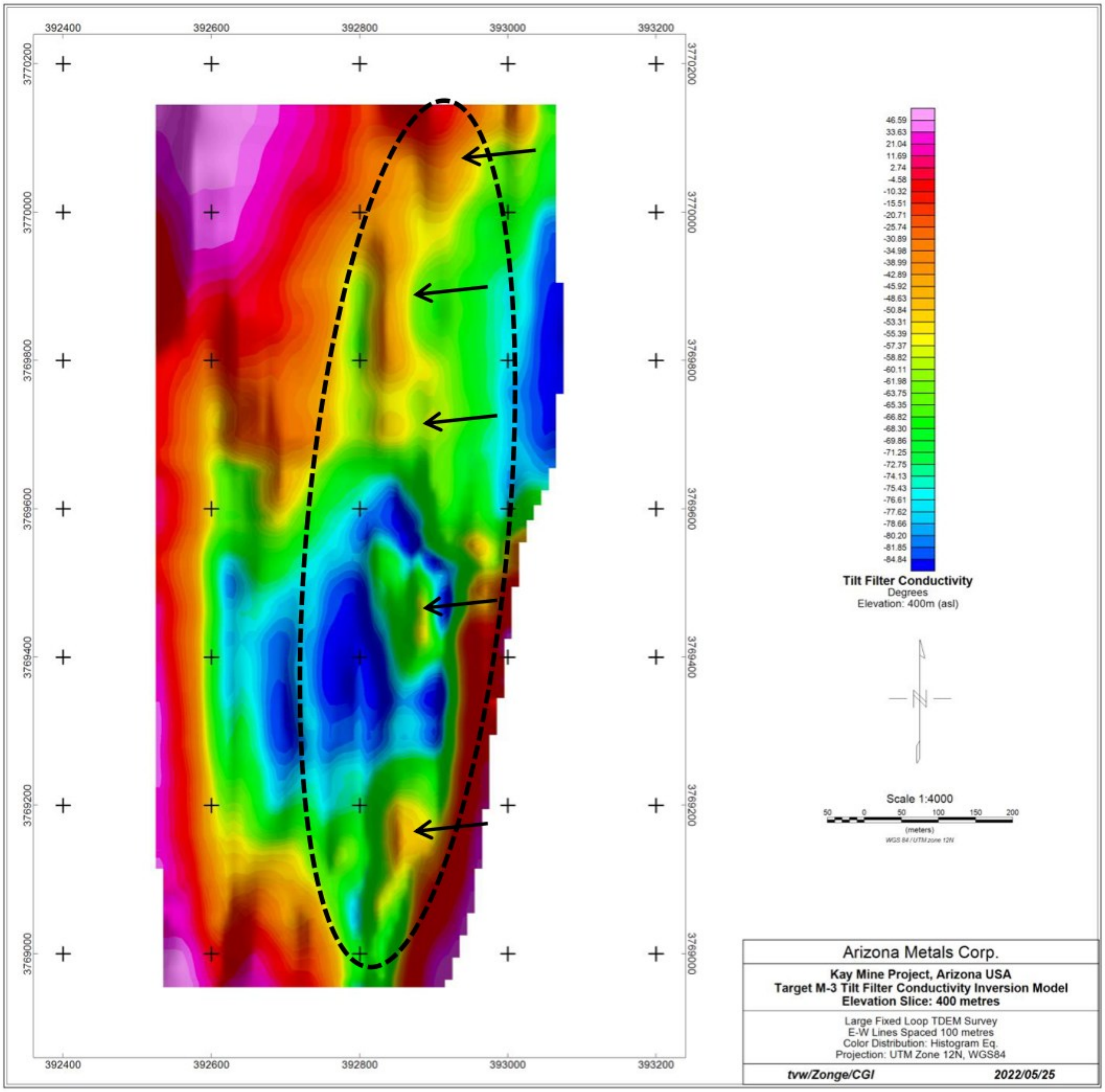
Figure 9.2: Ground EM Survey Loops and Lines



The intent of the Kay grid (M3) was to obtain an EM geophysical signature of the drilled Kay deposit and to track its extension to the north (Weis, 2022a). Initial interpretation indicated that the EM response to massive sulfide on the northern end of the grid was overwhelmed by the layer of carbonaceous sediments (graphite) that lies stratigraphically above (west) of Kay VMS mineralization. In order to de-emphasize the graphite EM responses, Weis employed a tilt angle filter to highlight features with conductivity lower than graphite. This showed possible conductive features at depth in the northern end of the M3 block (Figure 9.3). The northern three (3) of these features (A, B, C in Figure 9.3) have been drilled, with generally good results. Feature A was drilled in KM-22-91, returning 1.8 m grading 1.1% Cueq. Feature B was intersected in drill holes KM-21-30 (3 m @ 1.1% Cueq), KM-21-33 (1.2 m @ 4.2% Cueq), and KM-22-93 (multiple intervals, including 4.5 m @ 1.8% Cueq). Feature C was tested with hole KM-22-92, which showed no significant assays.

The Kay deposit itself shows a pronounced low conductivity in both the tilt-angle filter and unfiltered data (Figure 9.3). This is unexpected, given the large thicknesses of high-conductivity sulfide minerals drilled to date in the deposit. However, the low-conductivity response can be explained by the abundant carbonate alteration that accompanies the Kay mineralization. Thus, both EM conductivity low anomalies (possible abundant carbonate) and smaller conductivity high anomalies (thinner VMS lenses accompanied by less carbonate alteration) are of interest.

Figure 9.3: Kay Grid MX-3 Ground EM Conductivity Anomalies



**Note: Arrows Indicate Interpreted Features of Interest: Depth Slice at 400 metres Elevation, approximately 250 m Below Surface*

The survey on the Central grid (M2) showed a large high-conductivity anomaly with two (2) particular features of interest (Weis 2022b). This anomaly was drilled, and both features of interest were tested with seven (7) holes (KM-22-73, 76, 77, 83, 84, 85, 96). Drilling indicated that the EM anomaly is caused dominantly by graphitic sediments.

The southern grid on the West target (MX-1) showed an extensive, intense high conductivity anomaly (Weis 2022c). Subsequent drilling of the anomaly revealed that its source is graphitic sediments, although sulfide mineralization was encountered along the same stratigraphic horizon were repeated by folding to the west of the anomaly (see Drilling, below). No conductivity anomalies were detected in the northern grid on the West target (MX-4).

9.4.5 Borehole EM Geophysical Surveys

In August 2020, Arizona Metals commissioned a borehole electromagnetic (BHEM) survey, which measured electric conductivity downhole in portions of seven (7) selected Arizona Metals' drill holes within the Kay deposit. The survey was designed by geophysicist Tom Weis and performed by Zonge International (Zonge, 2020), which laid out three (3) surface transmitter loops: two (2) at approximately 400 x 400 m in extent, and one at about 100 x 100 m extent. Data was recorded at 10-metre intervals downhole over a total length of 1,415 m of drill hole. Data processing was performed by Computational Geosciences of Vancouver, B.C., who integrated the BHEM data with the VTEM data and ran several models with combinations of the two (2) data sets. Computational Geosciences provided digital models directly to Tom Weis, who interpreted them and prepared a report (Weis, 2021b). Weis eliminated the eastern portions of the Kay VTEM anomaly, which overwhelmed the conductive response in the area of drilling and is believed to be caused by powerlines running along a city street.

Weis outlined 20 drill targets within six (6) conductive zones of interest, some of which were combination BHEM-gravity anomalies (see below). Two (2) of these targets were tested by Arizona Metals drill holes. First, a combined BHEM-gravity anomaly (see discussion of gravity below) north of the area of current drilling was tested by KM-21-22 and KM-21-22A. Although no massive sulfide was intersected, the mineralized horizon was detected in KM-21-22, consisting of thin 0.3-1.2 m seams of pyrite, chalcopyrite, arsenopyrite, and probable tetrahedrite-tennantite grading up to 1.7% Cu and 2.9 g/t Au. Second, a deep anomaly to the east of the drilled area was tested by KM-21-17; this hole intersected no mineralization in the area of the anomaly. Arizona Metals has imported the BHEM digital models into its 3D model and will continue to use them to support drill targeting.

In 2023, Arizona Metals also conducted BHEM in four (4) drill holes on the West target in order to seek anomalies to refine drilling. The surveys were designed by geophysicist Tom Weis and performed by

SJ Geophysics using one (1) surface loop 550 x 550 m in extent. Data were recorded at variable intervals downhole over a total length of 2,478 m of drill hole in two (2) campaigns: 1) surveys of holes KM-23-104A and KM-23-107 during May 2023 (SJ Geophysics, 2023a); and 2) surveys in holes KM-23-109 and KM-23-110 in July 2023 (SJ Geophysics, 2023b). Data processing and interpretation were performed by Axiom Geophysics.

Interpretation confirmed that high conductivity anomalies present in hole KM-23-104A coincided with visible graphitic horizons in drill core. A weak off-hole anomaly was detected to the south of hole KM-23-107, which was drilled in hole KM-23-118; assay results are pending as of the effective date of this report. No anomalies were detected in the data from KM-23-109 or KM-23-110.

9.4.6 Gravity Geophysical Survey

The company commissioned a geophysical gravity survey on the Project that was completed in January and February 2021. The survey was designed by geophysicist Tom Weis and conducted by Magee Geophysical Services (Magee, 2021). The survey was conducted at 1,410 stations spaced at 25 to 50 metres along east-west lines spaced at 100 m. Data processing, interpretation, and reporting were done by Tom Weis (2021d), who integrated the gravity with VTEM and BHEM anomalies to look for correlations.

Weis delineated 23 drill targets, 11 of which were combined gravity-EM and 12 of which were standalone gravity targets. At the Kay Deposit area of historical and current drilling (MX-3), Weis outlined five (5) drill targets where EM and gravity were coincident (Figure 9.4), two (2) of which have been tested by drilling (see above). At the West anomaly (MX-1), Weis noted three (3) targets where VTEM and gravity agree very well (Figure 9.5), and these have been targeted for drilling. At the Central anomaly (MX-3), two (2) gravity features are coincident with VTEM conductivity highs and have been targeted for drilling (Figure 9.6). Weis also noted three (3) gravity-only features of interest in the northern part of the survey area that he recommended for field checking and ground EM surveys (Figure 9.7).

Figure 9.4: Combination Borehole EM-Gravity Anomalies (Dashed Ellipses) in Kay Drilling (MX-3) at 250 m Elevation, About 400 m Depth (Weis, 2020b)

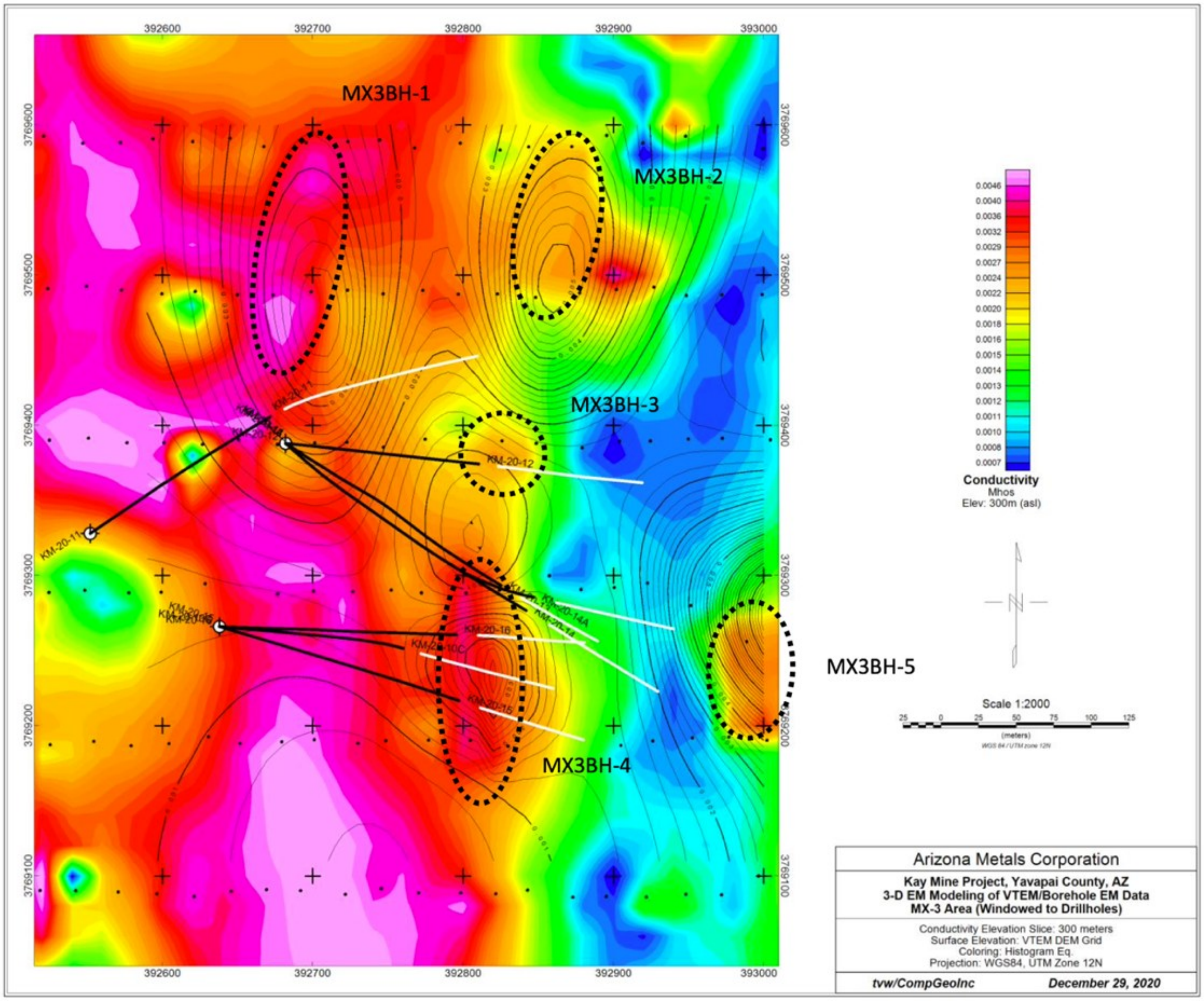


Figure 9.5: Combination Borehole EM-Gravity Anomalies (Dashed Ellipses) on West Anomaly (MX-1) at 400 m Elevation, About 300 m Depth (Weis, 2021d)

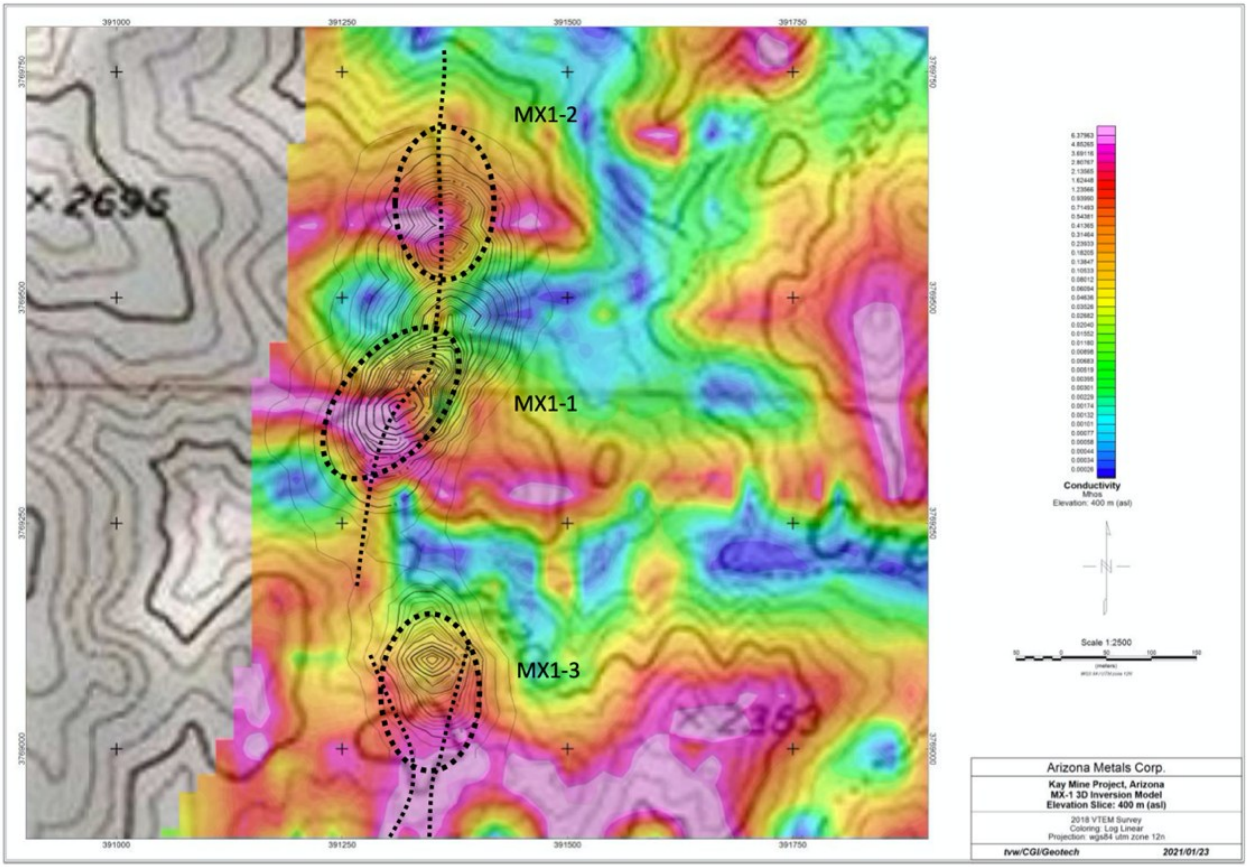
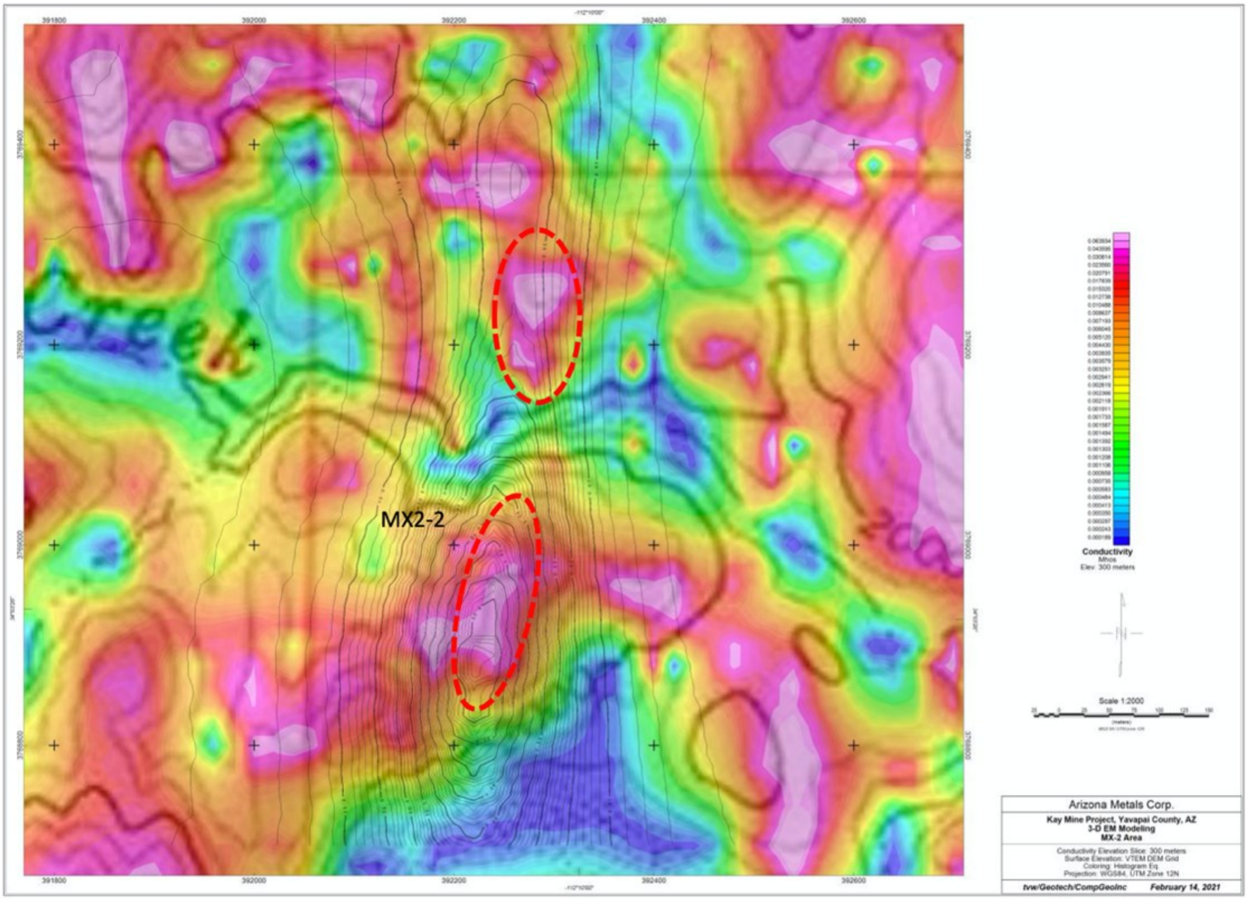
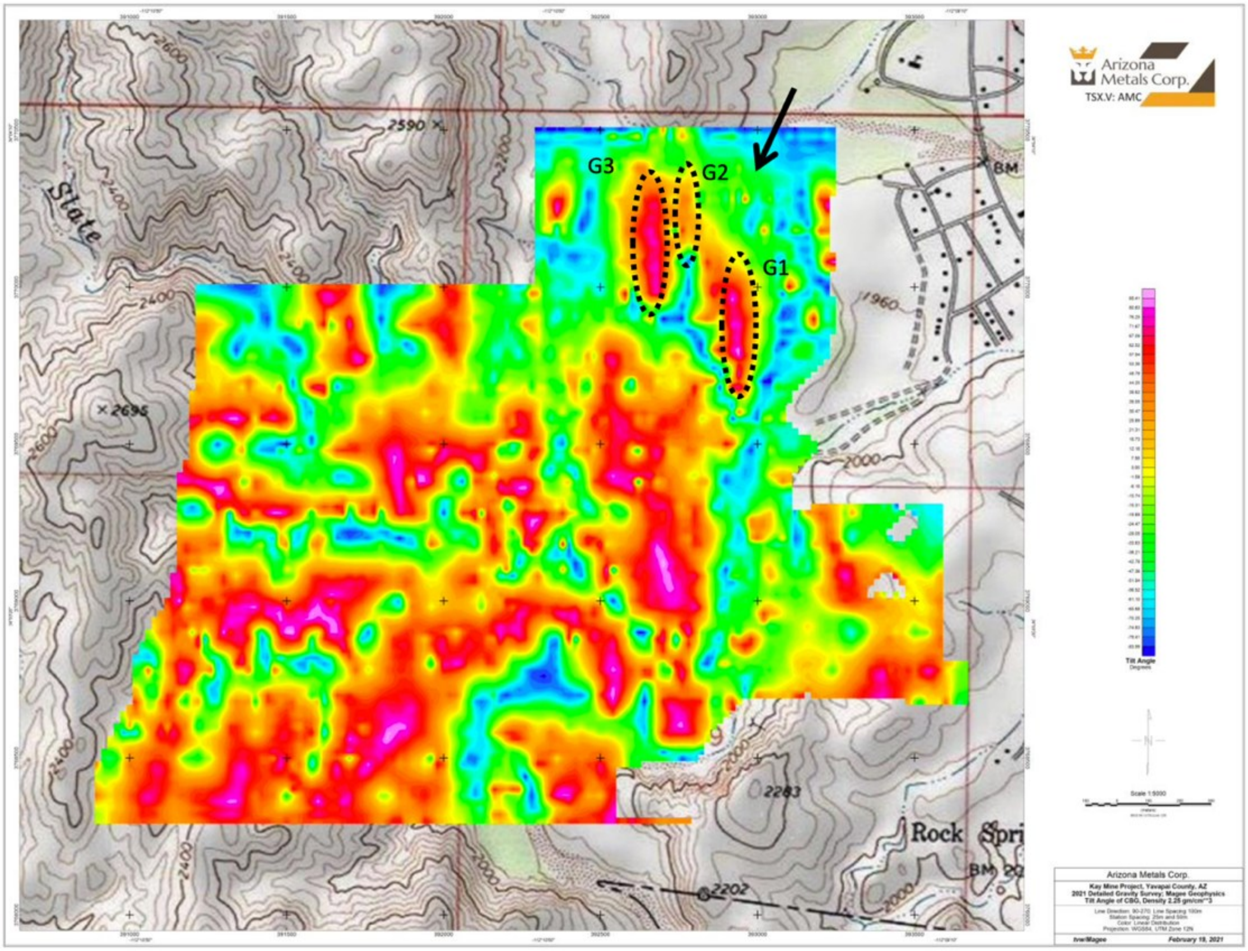


Figure 9.6: Combination VTEM-Gravity Anomalies (Dashed Ellipses) on Central Anomaly (MX-2) at 300 m Elevation, About 350 m Depth (Weis, 2021c)



**Note: Colours Represent Gravity, and Black Contour Lines Show Conductivity (VTEM).*

Figure 9.7: Standalone Gravity Targets (Dashed Ellipses) Recommended or Field Checking and Ground EM Surveys



9.4.7 Rock Sampling

A total of 2,416 rock samples have been taken on the Project by Arizona Metals. This includes due diligence and reconnaissance samples, samples collected during geologic mapping, and a grid of rock samples covering the full property. Rock-grid samples were collected at a spacing of approximately 50 m (Figure 9.8). Samples were submitted to ALS Minerals for Au and multi-element analysis.

9.4.7.1 Rock Geochemistry Results

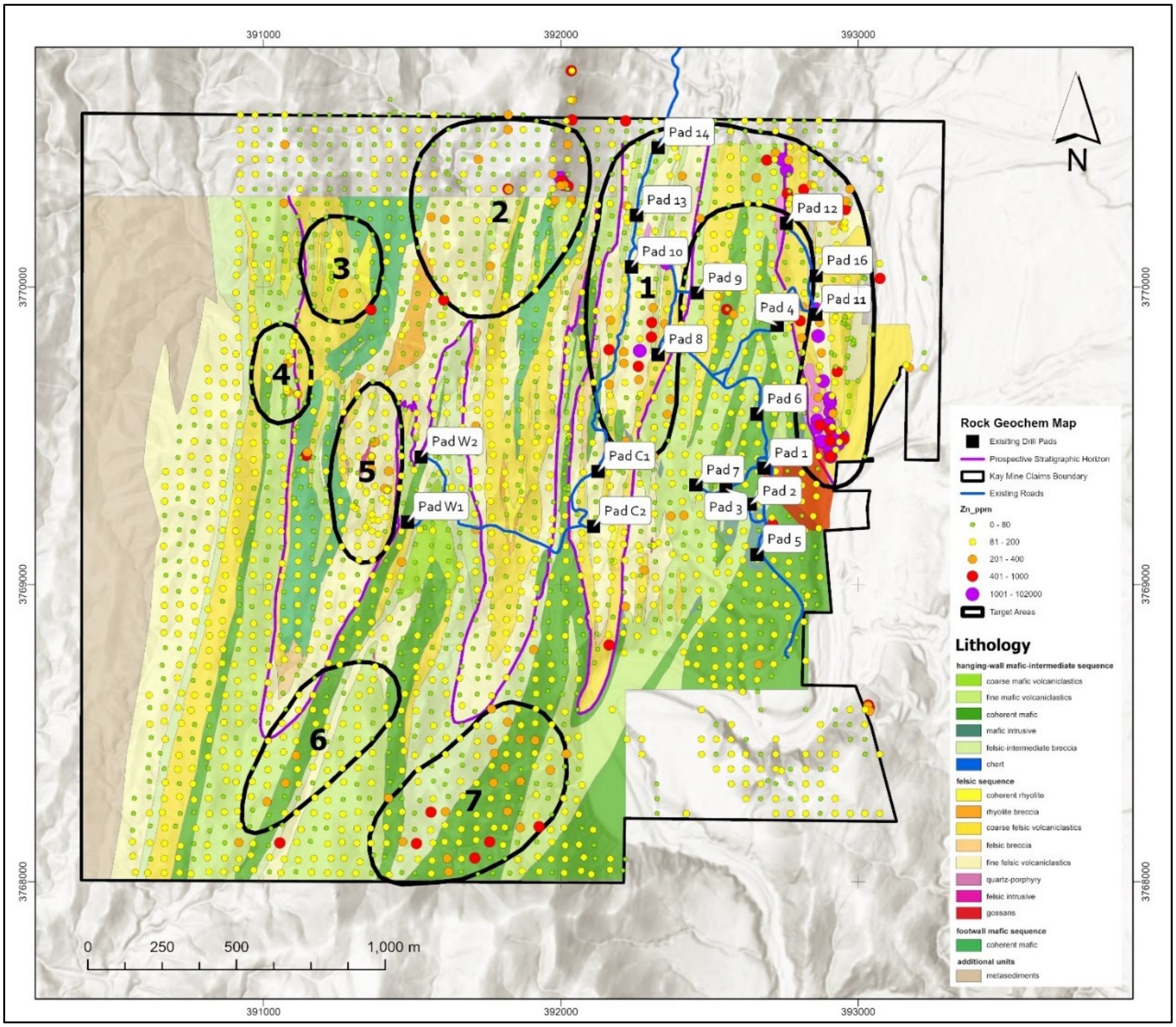
Rock sample results show numerous areas on the Project with anomalous major and trace elements. Figure 9.8 shows zinc in rocks and rock geochemistry anomalies as outlined below.

- Rock Anomaly 1: This is the strongest and most coherent rock-geochemistry anomaly on the Project, stretching north from the Kay Deposit along the mapped Kay mineralized horizon, then following favourable felsic stratigraphy and the Kay horizon around the nose of north-closing syncline, and curving southward toward the vicinity of drill pad C1. Rock Anomaly 1 is anomalous in Cu, Zn, Au, Ag, Bi, Hg, In, and Te.

- Rock Anomaly 2: A relatively strong anomaly of elevated Cu, Zn, Au, Ag, Bi, Hg, In, and Te centred around the Adit target, which returned 11.9% Cu on surface.
- Rock Anomaly 3: A focused anomaly Cu, Zn, Au, Ag, Mn within a mapped area of coherent rhyolite, suggesting a volcanic center and therefore a prospective target for VMS mineralization.
- Rock Anomaly 4: Anomalous Cu, Au, Ag, In, Te in favourable felsic stratigraphy, likely the surface expression of the deeper mineralized horizon intersected in drilling on the West target. This anomaly is especially high in Cu, returning up to 5.5% on surface.
- Rock Anomaly 5: Elevated Cu, Zn, Au, Ag, Mn on the West target, the outcrop of the shallower mineralized horizon encountered in West target drilling.
- Rock Anomaly 6: A somewhat diffuse anomaly of Cu, Zn, Au, Ag, Mn.
- Rock Anomaly 7: A broad but consistent anomaly of Cu, Zn, Au, Ag, and Mn. Although located in less-favourable mafic stratigraphy, this anomaly is large (about 750 m long) and relatively coherent.

These are among the elements shown to be anomalous in soils, and these rock anomalies are coincident with many of the soil geochemistry anomalies and central portions of the VTEM and EM anomalies.

Figure 9.8: Rock Geochemistry Anomalies and Zn Rock Geochemistry Results (Smith, 2024)



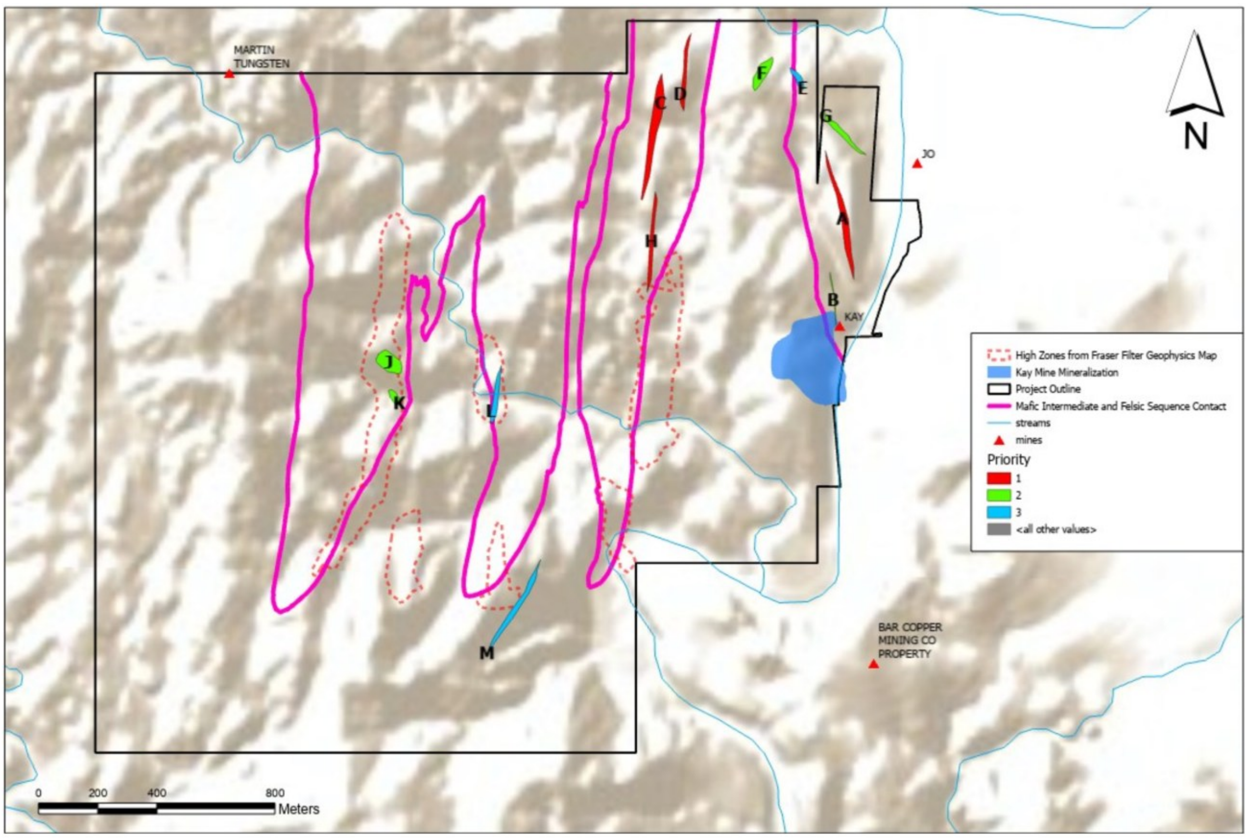
9.4.8 Soil Sampling

A total of 1,719 soil samples has been collected by Arizona Metals on the Project. Soil samples were collected in two (2) phases: 1) 287 samples on three (3) grids covering the Kay Deposit, Central, and West areas in 2020; and 2) 1,432 samples on an extended grid covering most of the property in 2022. All samples were collected at approximately 50-metre spacing, from the C soil horizon at depths of approximately 30-90 cm below surface. Samples were analyzed at ALS Minerals Labs by aqua regia methods for a suite of 51 elements. Field duplicate samples were analyzed by Ethos Geological for inverse difference hydrogen (IDH).

9.4.8.1 Soil Geochemistry Results

Interpretation of soil geochemistry resulted in 12 targets for follow-up defined by single-element patterns and multivariate methods such as summative indices and principal component analysis (Figure 9.9; Heberlein, 2022a). Priority 1 targets (targets A, C, D, H) are all located along the Kay North Extension or in the North Central Target, where geologic mapping has traced the Kay horizon and identified an additional mineralized horizon, the Pad 10 Horizon. Priority 1 soil targets are anomalous in Ag, Au, Bi, Cu, Hg, In, Se, Te and Zn. Priority 2 and 3 soil targets are located in the North Central Target, West Target, and South Target (Figure 9.9).

Figure 9.9: Soil Targets (Smith, 2024)



9.4.8.2 Soil IDH Results

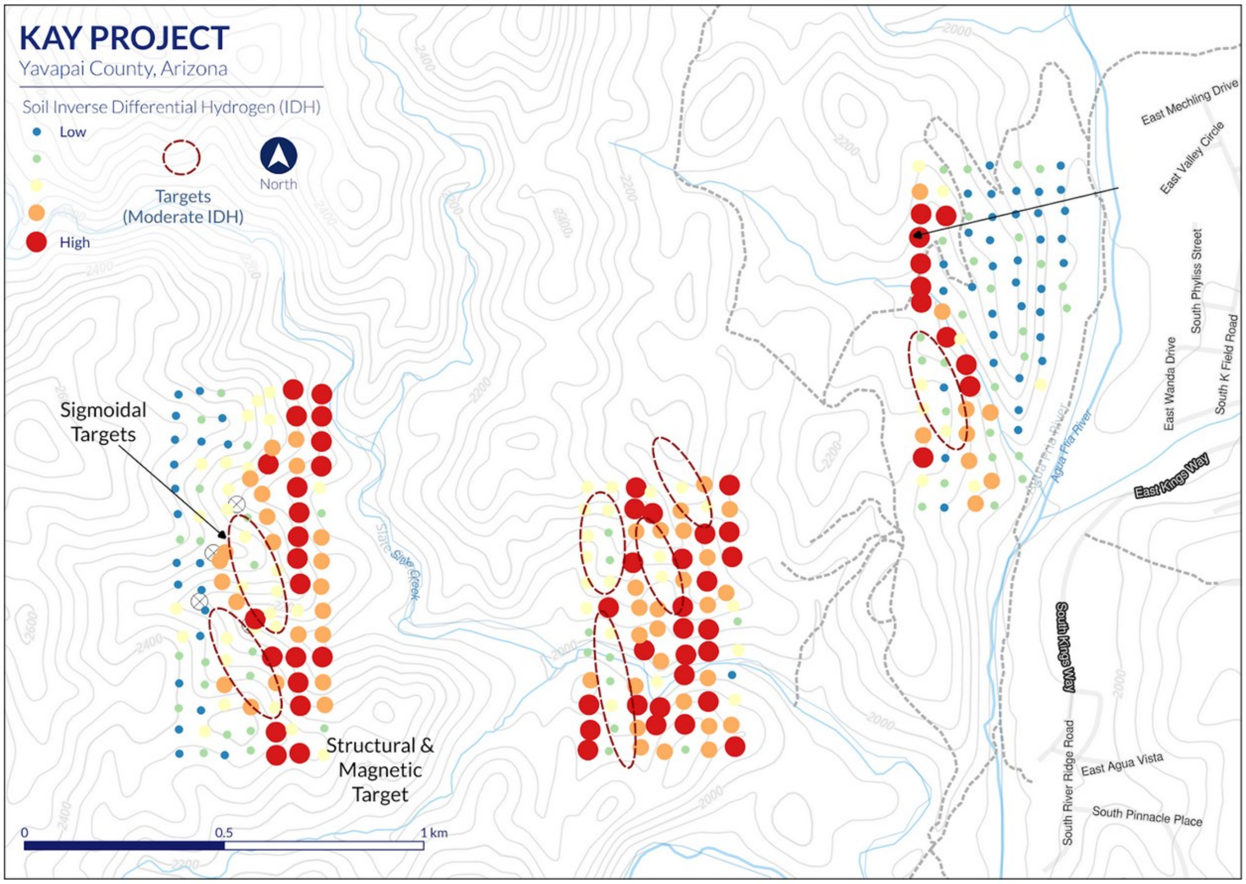
Inverse-difference hydrogen (IDH) analysis measures the amount of H+ and other changes in the soil that result from the decomposition of oxidizing sulfide minerals. Sulfide-bearing mineralization at depth creates zones of lower soil pH at the surface, caused by the release of H+ ions from oxidizing pyrite. These H+ ions appear to have a sufficiently high diffusion coefficient to cross appreciable thicknesses of unmineralized cover in short geological time spans. Within the low-pH zones, carbonates and other pH-sensitive elements become unstable and dissolve in pore waters. These waters move to the margins of the low-pH zones, where the dissolved elements are deposited in carbonate-stable conditions, creating haloes of elevated soil buffering capacity. Both the low-pH zones and the surrounding higher-buffering halo zones can be detected by simple pH measurements of soil samples. This is done by taking two (2) pH readings of a water-soil slurry, one without and one with dilute HCl or acetic acid. After converting the pH values to H+ concentrations, the inverse of the acidified minus non-acidified H+ values is calculated. This is IDH, or inverse difference hydrogen, which is a direct measure of the reactivity or acid buffering capacity of the soil. IDH is ideal for detecting the presence of sulfide mineralization at depth, below solid bedrock and/or transported cover. The method has been used to detect sulfide mineralization in many locations, including Oyu Tolgoi, Mongolia; the Marigold Mine, Nevada; and the Canadian Shield. The contrast and patterns are more important in IDH interpretation than the absolute values, and anomalies generally appear as low IDH zones surrounded by moderate to higher IDH values. Although quantitative, soil IDH analyses are not recommended for use alone, and are intended as a supporting layer of geochemical information in addition to more rigorously quantitative methods such as geophysics and laboratory geochemical analyses.

Soil IDH analyses were done on 287 samples collected from the initial three (3) soil grids in 2020. IDH analyses were performed by the independent consulting company Ethos Geological. Results on the three (3) grids (Figure 9.10) agree well with the soil and rock geochemical results and support the VTEM interpretation of sulfide-bearing zones at depth on the West and Central targets. On the Kay grid, a broad zone of low IDH values is present on the eastern majority of the grid, bordered by high IDH on the western edge. The broad eastern low-IDH area is difficult to interpret since it is open to the east, north, and south and would require a larger grid to close off. The high-IDH portion on the western edge, however, contains a low-IDH anomaly that is offset to the west from the linear soil anomalies on this grid, as expected from stratabound sulfides at depth in the west-dipping stratigraphy. This IDH anomaly is small, but it is confirmation of an IDH response above known mineralization.

IDH response on the Central grid is more broadly elevated but shows two (2) distinct IDH anomalies. These overlie the western portion of the VTEM anomaly and are offset to the west of the soil geochemical anomalies. This fits with the interpretation of the VTEM modelling dipping to the west and fits the known west stratigraphic dip in this area.

On the West grid, two (2) fairly clear soil IDH anomalies are present directly over and on the western edges of the VTEM anomaly and soil geochemical anomalies. This suggests a steeply west-dipping or near-vertical sulfide body, which is geophysicist Tom Weis' interpretation and geologist Ray Harris' observation in the field.

Figure 9.10: Soil IDH Results



9.4.9 Geologic Mapping

Arizona Metals has conducted geologic mapping on the majority of the Kay Deposit property. In 2020, the company contracted geologist Antoine Caté of SRK Consulting (Canada) to perform initial geologic mapping, followed by structural interpretation and alteration studies. Initial geologic mapping confirmed the intense nature of S1 folding and provided clarity on the nature of the pre-metamorphic host-rock protolith (SRK, 2020a). The report summarized, "Ductile deformation resulted in the repetition of the felsic schist and mafic schist on the property as the cores of anticline and syncline folds, respectively. The folded contact between the felsic and mafic schists and the felsic schist is interpreted as prospective for VMS mineralization. Massive rhyolite and zones of metamorphosed hydrothermal alteration are considered the most prospective zones within the felsic schist as they represent evidence of the proximity of volcanic and/or hydrothermal feeder zones. These prospective lithologies are interpreted to potentially extend beyond the current exploration property to the east, north and south. For these reasons, exploration for VMS mineralization should be extended regionally. Finally, the ductile deformation has strongly affected the geometry of geological features on the property. Sulfide lenses are likely to be affected by steep-plunging, tight folds, with the lenses being thinned and boudinaged in fold limbs and thickened in fold hinges. This geometry is leading to a high downdip continuity and to a lower lateral north-south continuity of the mineralization. Repetition of the sulfide lenses through folding is possible, and drilling should not stop immediately after intersecting a sulfide lens but rather should continue until the alteration halo of the deposit is excited.” Additional structural interpretation and alteration studies are discussed below in Drilling.

In 2021 and 2023, geologists Alan Baxter and David Diekrup mapped the majority of the property (Baxter & Diekrup, 2021). Their work delineated the overall stratigraphy of the Project, in particular, additional areas of coherent rhyolite that indicate volcanic centers with potential for mineralization. Their structural interpretation agreed with Caté (SRK, 2020a) and previous workers, and documented S1 foliation directions of 276-298° dipping 63-89° W. Primary bedding was generally parallel to S1 foliation; younging direction indicators included fining-upward sedimentary sequences and pillow basalts. Folds were observed throughout the property at all scales, from centimetre-sized small-scale folding to major kilometre-scale isoclinal folds. The dominant folding style was confirmed to be isoclinal with steeply south-plunging fold axes dipping south at 57-77°. Fold axial planes were consistently within the range 269-310°, dipping 60-86° W. A L1 stretching lineation dips south at 60-86°. No faults of major offset were encountered.

Baxter and Diekrup noted that the primary alteration suggesting mineralization was sericite and carbonate. They observed sericite primarily along with carbonate in strongly overprinted felsic lithologies, occurring as mm- to cm-scale domains of blue-gray sericite often surrounding light rust-brown-weathering carbonate-altered clasts. Carbonate is widespread on the property, and in particular, weathered iron carbonate lends an orange-brown cast to felsic stratigraphy on the property.

Baxter and Diekrup conducted additional, smaller-scale mapping on the West target during 2023 (Baxter and Diekrup, 2023). This work more fully delineated the Project stratigraphy, identified additional surface mineralization, noted two (2) new alteration styles (intense quartz-carbonate stockwork and Cr-rich mica), and discovered significantly more coherent rhyolite and felsic volcaniclastic rocks in this area of the property. These felsic rocks suggest a larger felsic center that hosts mineralization as drilled from the West drill pads, which is spatially distinct from the main Kay deposit but part of the same regional felsic volcanic event.

Since 2023, senior project geologist Ben Somps has conducted ongoing geologic mapping in order to refine work done by previous mappers and to more fully refine drill targets, structural understanding, and alteration vectors.

9.4.10 Petrographic Studies

Twenty-nine (29) polished thin sections were prepared and examined by consulting petrographer Ingrid Kjarsgaard (Kjarsgaard, 2021) and further interpreted by Arizona Metals technical advisor Mark Hannington (Hannington, 2021). Thin sections were spread throughout the deposit to cover a variety of depths, locations, mineralization styles, alteration assemblages, and host-rock types. Results are discussed in Mineralization, above.

9.5 Exploration Targets and Observations

As a result of the exploration work discussed above, numerous exploration targets are apparent on the Project as discussed below and shown in Figure 9.11.

9.5.1 Kay Expansion

Immediate expansions of the known mineralization in the Kay deposit are apparent to the north, to the south in some locations, and at depth. In particular, the Kay 2 Zone, located deep in the deposit and about 100 m north of the deeper portions of the South Zone, offers an excellent opportunity for expansion of the deposit.

9.5.2 North Central Target

The North Central target is the strongest and most appealing target on the Project (Figure 9.11). It displays a combination of geochemical, geophysical, lithological, and structural features prospective for VMS mineralization. It is located in the northeastern portion of the Project and covers a large syncline-anticline pair in a favourable felsic host rock where both the Kay mineralized horizon and the Pad 10 mineralized horizon crop out. Both horizons have been mapped and sampled on surface, returning Cu values up to 11.9% Cu. Both horizons have also been intersected in drilling, the most prominent result being 0.5 m @ 11.3% CuEq (KM-24-153) in the Pad 10 horizon (Figure 7.12). A total of approximately 3 km of strike length remains unexplored on these two (2) mineralized horizons (see North Central Target Mineralization, above). Rock geochemistry on the North Central target shows numerous individual anomalies in Cu, Zn, Au, Ag, Bi, Hg, In and Te. Soils are anomalous in Ag, Au, Bi, Cu, Hg, In, Se, Te and Zn. Alteration is present as elevated ankerite, low Na/Zn, and high CCPI. Gravity data shows prominent, standalone gravity high anomalies in this area (Figure 9.7).

9.5.3 Kay North Extension Target

The Kay mineralized horizon is a key exploration target on the Project, where it stretches north from the main Kay deposit within favourable felsic stratigraphy. This horizon has been traced on surface and in drill holes. It displays anomalous Zn, Cu, Au, Ag, Bi, Hg, Te in rocks, and an elevated principal component comprising Pb-Bi-Zn-Mo-Te-W-Hg-Ag-Cd. This area also shows several indications of hydrothermal alteration: low Na/Zn, high CCPI alteration index, and increased abundance of ankerite carbonate. Geophysics reveal EM high anomalies from the ground EM survey and modest gravity highs, both suggesting the presence of sulfide mineralization.

9.5.4 West Target

The West Target is a prominent linear target in the western part of the Project, stretching over 2 km south from the northern Project boundary that straddles a combination of favourable lithology, geochemistry, geophysics, and alteration. The target covers an anticline of felsic host rock, in particular a grouping of coherent rhyolite on the northern end that suggests a volcanic center, typical of heat sources that drive the formation of VMS deposits. The favourable lithology is anomalous in three (3) focus areas, Rock Anomalies 3, 4, and 5 (Figure 9.8), which show elevated Cu, Zn, Au, Ag, In, Te, and Mn. Soil samples returned muted anomalies in Ag, Cd, Cu, Fe, In, Mo, S, Se, Tl, and Zn. Alteration is present as scattered Na/Zn lows and somewhat elevated CCPI. Airborne geophysics initially identified an electromagnetic high anomaly; later borehole EM and gravity surveys revealed three (3) overlapping anomalies; drilling of these anomalies indicated that they were caused primarily by graphite. Several historic adits and one (1) shallow mine shaft indicate historic prospecting activity in this area. Exxon drilled one (1) hole into this target, a 30°-dipping hole to the WNW to 180 m depth; however, it appears to have missed the heart of the target as it only penetrated a vertical distance of about 90 m below surface. The West mineralized horizon crops out at surface, where it has been mapped and sampled, returning values up to 5.5% Cu.

9.5.5 South Target

The South target lies on a combination of mafic and felsic rock coincident with a large area of Na/Zn low and high CCPI alteration. Although dominantly in mafic rocks, the target shows compelling and relatively consistent geochemical anomalies: rock geochemistry shows anomalies in Cu, Zn, Au, Ag and Mn, and soil sampling shows anomalies in Ag, Cd, Hg, Tl and Zn. Near the nose of an anticline in the northeast part of this target, pervasive iron carbonate alteration with vent-proximal textures deserves exploration.

9.5.6 Target A

In the northern portion of the Project, a syncline in felsic stratigraphy shows rock anomalies in Cu, Zn, Au, Ag, Bi, Hg and Te accompanied by elevated ankerite.

9.5.7 Target B

In the center of the property, Target B contains minor Cu and Zn rock anomalies, soil anomalies in Ag, Tl and Zn, along with high CCPI and coherent rhyolite near an anticline fold hinge.

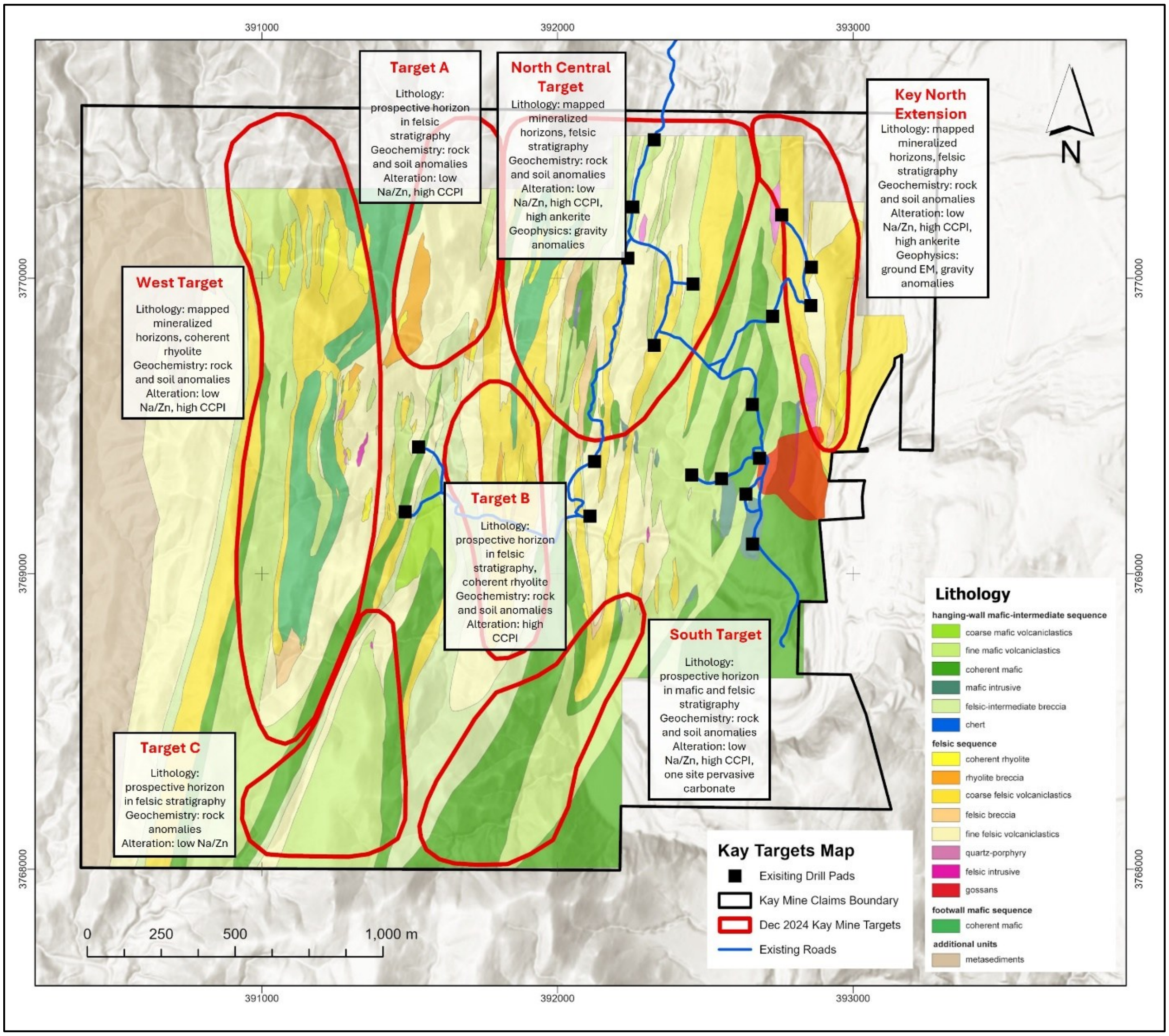
9.5.8 Target C

Target C is the southern extension of the West target, showing minor Cu, Zn, Ag, Au and Bi anomalies in rocks and Na/Zn low alteration.

9.5.9 Regional Potential

Exploration potential also exists for additional VMS targets in the surrounding region, including the Greyhound prospect about 3 km to the northeast of the property, a 1-km-long target previously drilled by Exxon.

Figure 9.11: Exploration Targets on the Project



10. DRILLING

10.1 Summary

Arizona Metals initiated drilling on the Property in January 2020 and has continued to explore and delineate the Kay deposit with a series of drill programs undertaken each year through to 2026. As of March 2026, Arizona Metals had completed 274 drill holes totalling 146,844 m and collected 12,329 assays (Table 10.1, Figure 10.1).

Arizona Metals has continued to drill on the Project since the data cut-off for the current MRE. Exploration drilling completed in the period from June 2025 through March 2026 totals 20 drillholes for 12,932 m. Drilling during this period focused on exploration targets outside of the Kay MRE area targeting the prospective stratigraphy at the Kay North Extension and Northwest Target areas. Drilling within or proximal to the Kay MRE was limited to two (2) drillholes (KM-25-194 and KM-25-195). The exploration drillholes completed in the second half of 2025 and 2026 and their location relative to the MRE are not likely to materially change the current MRE for the Project.

Historical drilling on the Kay Mine Project was undertaken during the late 1910s and early 1920s (Kay Copper Company), in the early 1950s (New Jersey Zinc), between 1972 and 1984 (Exxon Minerals Company), and from 1991 to 1993 (Rayrock Mines) and collectively totals at least 139 holes. While partial documentation remains to support this historical drilling, these drillholes are utilized for exploration guidance only and not relied upon for the estimation of mineral resources.

Drilling by Arizona Metals within the Kay deposit has primarily been completed on 30 m to 60 m centres. Drilling to date has been completed from surface and comprises angled holes (collar dips range from -15° to -89°) completed predominantly from five (5) drill pad locations in a vertical and horizontal fan pattern. A significant proportion of the deep drilling has been completed using wedge holes and directional drilling. Holes are collared in the hanging wall of and as orthogonal as practical to target lenses.

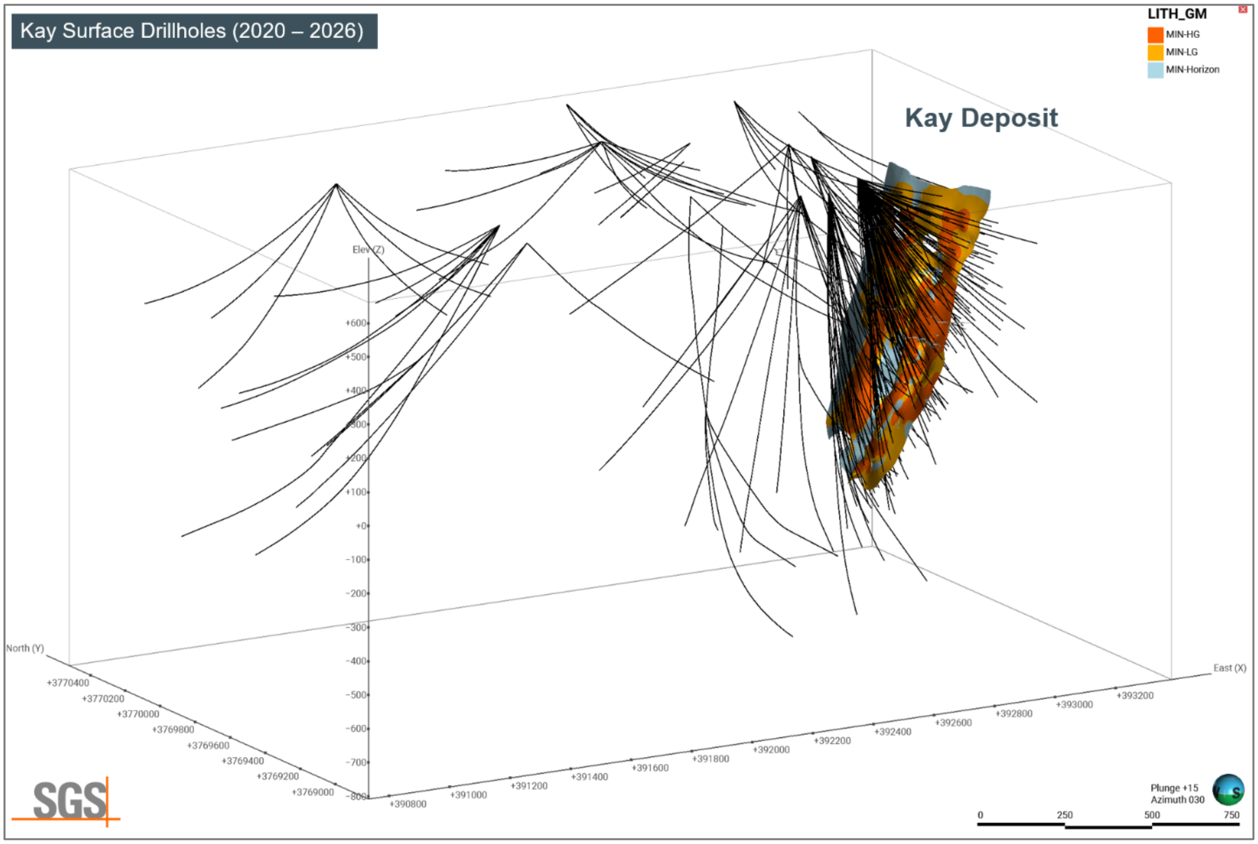
Arizona Metals' drilling of the Kay deposit sulfide lenses has delineated mineralization along a strike length of approximately 430 m, and a down-dip extent of over 950 m. Drilled widths vary between <1 m and 125 m, with an approximate true width of mineralization estimated to be 65-97% of reported core width, averaging 80%.

Diamond drillholes are HQ diameter, with reduction to NQ diameter if necessitated by ground conditions. Drilling to date has been completed using surface drill rigs. Maximum drilling depths obtained to date are approximately 1,700 m. Drillhole collar positions have been obtained using handheld GPS for common drill pad locations. Downhole orientations of drillhole azimuth and inclination are recorded by a gyroscopic survey instrument every 30 m downhole or at 6 m intervals during directional drilling. Drillhole geology is recorded for lithology, alteration, mineralization, and structure. Drillhole recovery is recorded for sampled intervals and averages 96% within mineralized zones. Lab density measurements are collected by a pycnometer on selected sampled intervals. Selective geochemical sampling is completed on intervals of potentially mineralized material. Logged mineralized intervals are sampled for geochemical assay at nominal 1.5 m intervals based on changes in lithology, alteration, mineralization, and structure.

Table 10.1: Summary of Drilling Completed by Arizona Metals on the Kay Project to March 2026

Year	Company	Hole Type	Drillhole Start	Drillhole Finish	Drillhole Count	Length Drilled (m)	Sample Count
2020	Arizona Metals Corp.	DDH	KM-20-01	KM-20-16	21	8,416.75	617
2021			KM-21-17	KM-21-59	60	33,924.24	2,681
2022			KM-22-57B	KM-22-96	53	32,543.50	2,147
2023			KM-23-97	KM-23-134	39	24,125.53	3,140
2024			KM-24-135	KM-24-94B	53	28,402.33	2,596
2025			KM-25-176	KM-25-181	7	18,037.76	1,148
2026			KM-26-200	KM-26-201	2	1,393.55	N/A
Total					**274**	**146,843.65**	**12,329**

Figure 10.1: Location of Kay Project Drillholes from January 2020 to March 2026 and Mineralization Models



10.2 Historical Drilling

Historical drilling on the Kay Mine Project was done by at least four (4) companies and totals at least 139 holes. In the late 1910s and early 1920s, the Kay Copper Company drilled 89 or more holes as detailed on mine level maps. In the early 1950s, New Jersey Zinc explored the property and drilled at least 14 underground drillholes. Some data for the Kay Copper Company and New Jersey Zinc assays are available on mine plan maps, but no drill logs exist.

The bulk of the documented drilling on the Project was done by Exxon Minerals Company between 1972 and 1984. Exxon drilled 28 core / rotary exploration holes totalling 9,565 m (31,380 ft). Eighteen (18) of these holes were in the immediate vicinity of the Kay Mine and totalled 7,525 m (23,793 ft); the remainder were in other parts of the Property and separate targets. Fellows (1982) also mentions “10 shallow air-track claim validation drill holes on various parts of the property,” which are plotted on a drillhole map as holes KA-1 through KA-10, but no location coordinates, logs, or assays are available. Details of the known Exxon drillholes are summarized in Table 10.1, with locations shown in Figure 10.2, and selected significant intersections are listed in Table 10.2.

Exxon sampled in variable interval lengths depending on geology, ranging from 0.3-3 m (1-10 ft). Core recovery is noted in drill logs; it is variable but appears to be good overall and shows mineralized zones to be very competent rock with consistent 98% recoveries. Other parameters of drilling are unknown. Exxon's drilling extended the size of the mineralized massive sulfide bodies previously discovered and mined from underground workings and outlined the mineralized bodies.

In 1991 and 1993, Rayrock Mines conducted two (2) drill programs totalling 11 holes: six (6) reverse-circulation holes in 1991; and five (5) core holes in 1993. Hole depths are known only for K91-3 (244 m) and K93-1 (280 m). Data for most Rayrock holes is not available, but one drill cross-section (Rayrock, 1992) includes assay data for hole K93-1, which returned two (2) intervals: 1.4 m grading 3.6% Cu, 0.63 g/t Au; and 0.8 m @ 1.8% Cu, 0.47 g/t Au. Details of the known Rayrock drillholes are summarized in Table 10.1, with locations shown in Figure 10.2, and selected significant intersections are listed in Table 10.2.

Table 10.2: Summary of Historical Drilling On and Proximal to the Kay Mine Project

Hole ID	East ACS	North ACS	East WGS84	North WGS84	Elev. (ft)	Azi	Inc	Depth (m)	Depth (ft)	Year	Type	Location
Exxon Minerals Company												
K-1	424,460	1,114,320	392,325	3,769,759	2,100	105	-45	155	510	1972	Core	Kay vicinity
K-2	421,665	1,112,500	391,467	3,769,200	2,100	285	-30	180	590	1972	Core	West of Kay
K-3	426,649	1,113,463	392,988	3,769,479	1,925	285	-45	202	663	1972	Core	Kay vicinity
K-4	426,649	1,113,463	392,988	3,769,479	1,925	285	-35	121	398	1973	Core	Kay vicinity
K-5	426,709	1,113,704	393,007	3,769,553	1,925	285	-45	137	450	1973	Core	Kay vicinity
K-6	425,758	1,113,164	392,716	3,769,391	2,084	89	-90	753	2,469	1973	Rotary / Core	Kay vicinity
K-7	425,758	1,113,164	392,716	3,769,391	2,084	124	-90	772	2,532	1973	Rotary / Core	Kay vicinity
K-8	425,758	1,113,164	392,716	3,769,391	2,084	140	-90	792	2,598	1974	Rotary / Core	Kay vicinity
K-9	425,758	1,113,164	392,716	3,769,391	2,084	61	-90	823	2,700	1974	Rotary / Core	Kay vicinity
K-10	425,080	1,112,450	392,507	3,769,175	2,000	152	-90	255	838	1974	Rotary	Kay vicinity
K-10A	425,325	1,113,287	392,584	3,769,429	2,086	108	-90	1,045	3,430	1975	Core	Kay vicinity
K-11	425,648	1,113,265	392,682	3,769,422	2,083	107	-67	507	1,663	1974	Core	Kay vicinity
K-12	425,684	1,113,477	392,694	3,769,486	2,109	106	-62	446	1,464	1974	Core	Kay vicinity
K-13	425,090	1,113,085	392,512	3,769,369	2,120	103	-90	413	1,355	1976	Rotary / Core	Kay vicinity
K-14	426,797	1,112,083	393,004	3,769,071	1,954	283	-56	248	813	1978	Core	Kay vicinity
K-15	425,670	1,106,328	392,670	3,767,308	1,940	114	-59	187	614	1978	Core	South of Kay
K-16	426,586	1,112,101	392,962	3,769,070	1,921	102	-60	293	960	1983	Core	Kay vicinity
K-17	425,720	1,116,570	393,040	3,770,283	2,000	121	-75	130	427	1983	Core	Kay vicinity

Hole ID	East ACS	North ACS	East WGS84	North WGS84	Elev. (ft)	Azi	Inc	Depth (m)	Depth (ft)	Year	Type	Location
K-18	--	--	--	--	--	NW	-53	183	600	1984	Core	Greyhound prospect
K-19	--	--	391,453	3,771,565	2,430	289	-65	219	720	1984	Core	Greyhound prospect
K-20	--	--	--	--	--	95	-75	385	1,263	1985	Rotary / Core?	Greyhound prospect
K-21	--	--	--	--	--	100	-65	554	1,816	1986	Core	Greyhound prospect
KV-1	423,890	1,111,020	392,141	3,768,742	1,900	105	-45	62	204	--	Core	Kay vicinity
KV-2	424,065	1,112,010	392,181	3,769,089	1,960	105	-45	97	319	--	Core	Kay vicinity
KV-3	422,490	1,112,440	391,717	3,769,194	2,050	--	-45	34	111	--	Core	West of Kay
EGH-1	420,820	1,122,560	391,237	3,772,268	2,640	109	-55	273	895	1979	Core	Greyhound prospect
EGH-2	421,070	1,121,430	391,310	3,771,923	2,590	100	-55	153	502	1980	Core	Greyhound prospect
EGH-3	421,000	1,124,080	391,453	3,772,690	2,390	89	-60	145	476	1981	Core	Greyhound prospect

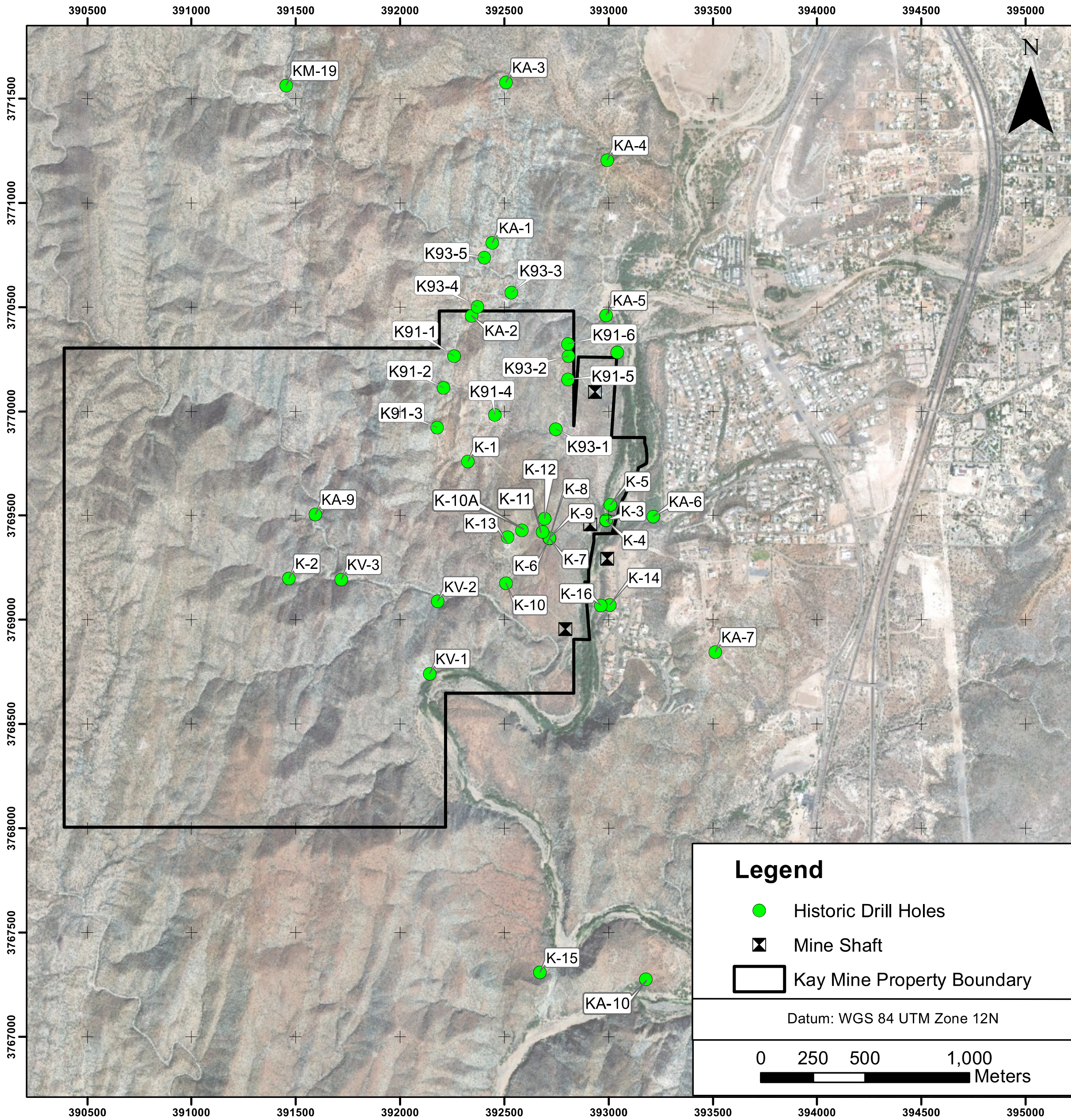
**Notes: ACS coordinates are feet, Arizona Coordinate System 1983; Rayrock hole locations are approximate, and most depths are not known.*

Table 10.3: Historical Drilling Significant Intersections from the Kay Mine Project

Company	Hole ID	From (ft)	To (ft)	Interval (ft)	True Thickness (ft)	True Thickness (m)	Cu %	Pb %	Zn %	Ag g/t	Au g/t
Exxon	K-6	2,013.0	2,020.0	7	4.9	1.49	1.14	0.05	0.22	12	0.29
Exxon	K-6	2,220.0	2,230.0	10	7.7	2.35	0.79	0.03	0.32	5	0.07
Exxon	K-6	2,244.0	2,259.0	15	11.5	3.51	3.06	0.05	0.06	12	0

Company	Hole ID	From (ft)	To (ft)	Interval (ft)	True Thickness (ft)	True Thickness (m)	Cu %	Pb %	Zn %	Ag g/t	Au g/t
Exxon	K-6	2,305.6	2,329.6	24	18.4	5.61	1.82	0.01	0.03	8	0.04
Exxon	K-6	2,371.6	2,381.6	10	7.1	2.16	2.11	0.06	0.25	9	0.34
Exxon	K-7	2,129.2	2,161.7	32.5	18.2	5.55	2.82	0.05	2.53	86	2.25
Exxon	K-7	2,200.0	2,223.6	23.6	16.7	5.09	1.04	0.71	4.8	38	0.93
Exxon	K-7	2,244.8	2,289.5	44.7	25.6	7.8	0.63	0.27	2.32	24	0.72
Exxon	K-7	2,335.6	2,365.8	30.2	17.2	5.24	0.13	0.29	2.19	21	1.45
Exxon	K-8	2,218.2	2,270.8	52.6	33.8	10.3	3.91	0.11	1.34	25	1.72
Exxon	K-8	2,298.5	2,434.0	135.5	95.8	29.2	0.21	0.41	2.67	35	0.82
Exxon	K-8	2,490.0	2,500.0	10	6.4	1.95	0.11	0.67	7.04	34	2.55
Exxon	K-9	2,165.5	2,174.0	8.5	4.9	1.49	1.28	0.07	0.28	7	0.08
Exxon	K-10A	2,890.0	2,896.7	6.7	3.6	1.1	5.03	0.04	0.09	15	0.33
Exxon	K-10A	2,916.4	2,925.0	8.6	5.5	1.68	0.53	0.03	0.38	12	1.14
Exxon	K-10A	2,948.5	2,955.0	6.5	3.6	1.1	2	0.01	0.22	6	0.26
Exxon	K-12	928.4	945	16.6	16.2	4.94	1.95	0.04	0.14	15	0.34
Exxon	K-12	968	978.3	10.3	9.5	2.9	0.34	0.2	1.17	24	0.42
Rayrock	K93-1	458.5	463	4.5	1.4	--	3.63	0.02	0.08	8.3	0.63
Rayrock	K93-1	491	493.5	2.5	0.8	--	1.8	0.01	0.02	4.3	0.47

Figure 10.2: Location of Historical Drillholes On and Proximal to the Kay Mine Project



10.3 Arizona Metals Drilling

10.3.1 2020 Drilling

Drilling of the Kay Mine deposit by Arizona Metals began in January 2020. Initial drilling sort to confirm and validate the results of historical drilling, underground mapping, and sampling data. The program successfully intersected mineralization within both the Kay South and North (Kay2) lenses at depths ranging from 120 m to 570 m below surface. Drilling information established an updated geological model for exploration targeting and paved the way for an expanded program in 2021.

Drilling in 2020 totalled 8,417 metres in 21 holes (Figure 10.3). Highlights of the 2020 drilling are presented in Table 10.4.

Figure 10.3: Location of 2020 Drillholes on the Kay Project and Mineralization Models

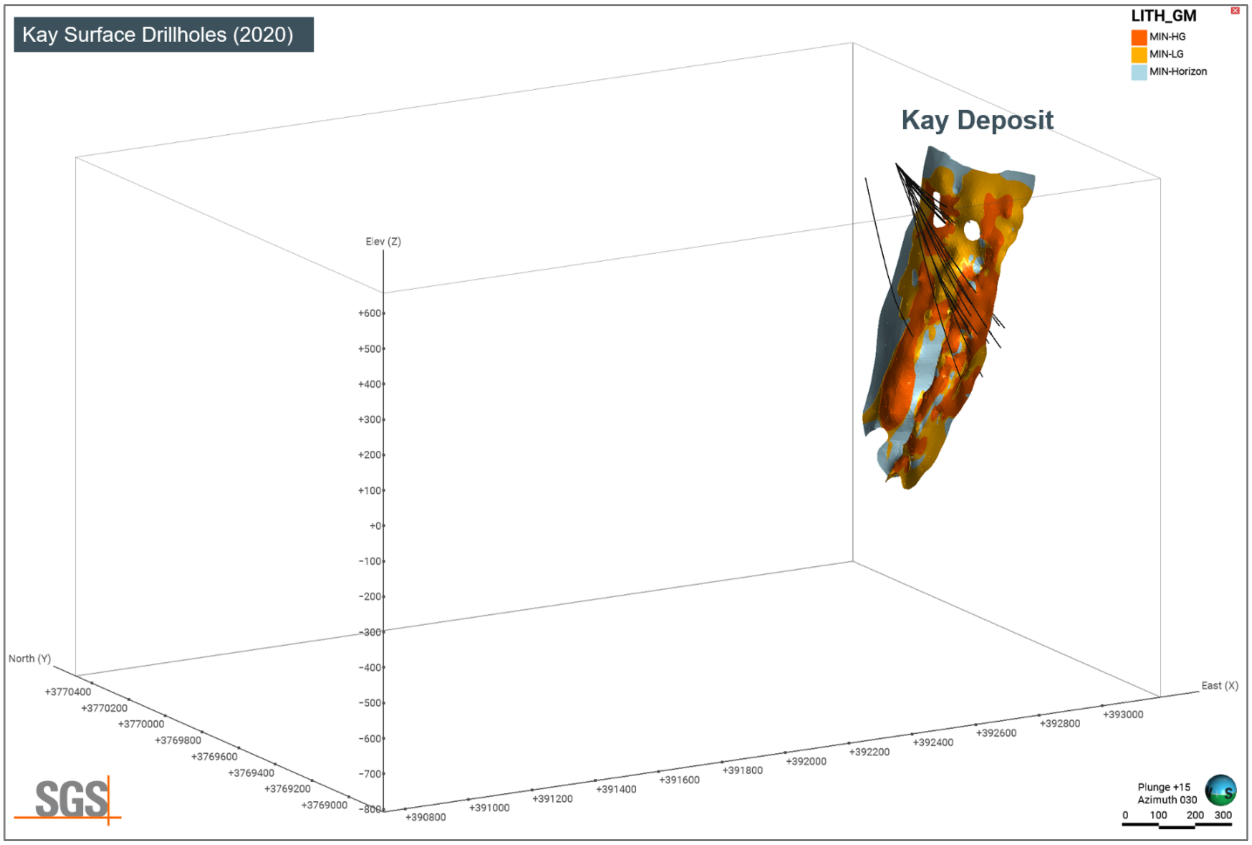


Table 10.4: Highlights of the 2020 Drilling

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
KM-20-06	267.9	281.5	**13.5**	**1.02**	**0.85**	**1.23**	45.6	0.30
including	267.9	268.4	0.5	1.54	2.20	6.10	31.0	0.81
including	276.6	281.5	4.9	1.86	0.87	1.96	92.1	0.42
including	280.0	281.0	1.1	3.22	1.03	0.64	340.0	0.04
KM-20-09	632.8	638.9	**6.1**	**0.12**	**4.18**	**8.02**	41.7	0.82
including	633.6	637.9	4.4	0.15	5.46	9.06	33.1	0.50
including	636.9	637.9	1.1	0.17	9.77	14.65	68.0	0.78

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
KM-20-10	563.6	568.5	**4.9**	**2.39**	**2.16**	**3.27**	24.9	0.31
including	563.6	566.6	3.0	3.66	2.42	3.16	28.2	0.32
including	567.2	568.5	1.2	0.33	2.52	5.10	28.4	0.43
KM-20-10B	503.0	530.7	**27.6**	**0.87**	**0.97**	**1.76**	21.3	0.32
including	503.0	509.6	6.6	1.78	1.55	2.55	29.8	0.37
including	513.9	518.3	4.4	1.08	1.89	4.05	47.4	0.68
including	527.2	530.7	3.5	1.91	2.32	3.93	52.9	0.99
KM-20-14	421.7	461.6	**39.9**	**1.47**	**1.00**	**1.67**	18.4	0.19
including	426.3	429.8	3.5	9.56	1.28	0.95	30.0	0.07
including	457.2	460.7	3.5	0.36	2.58	8.33	26.3	0.38
KM-20-16	480.4	518.8	**38.4**	**0.85**	**0.81**	**2.24**	24.3	0.25
including	480.4	492.9	12.5	1.63	1.98	4.23	48.5	0.50
including	480.4	483.4	3.0	2.40	4.74	7.49	77.9	0.91
including	489.8	492.9	3.0	3.61	2.59	6.90	100.7	0.92

10.3.2 2021 Drilling

Drilling in 2021 focused on delineation drilling of the Kay South lens with 50 drillholes at depths ranging from 150 m to 900 m below surface (800 m of down plunge extent). An additional five (5) drillholes targeted the North (Kay2) lens at depths ranging from 200 m to 540 m below surface. Exploration drilling on the Kay North Extension target was initiated with five (5) drillholes completed.

Drilling in 2021 totalled 33,924 metres in 60 holes (Figure 10.4). Highlights of the 2021 drilling are presented in Table 10.5.

Figure 10.4: Location of 2021 Drillholes on the Kay Project and Mineralization Models

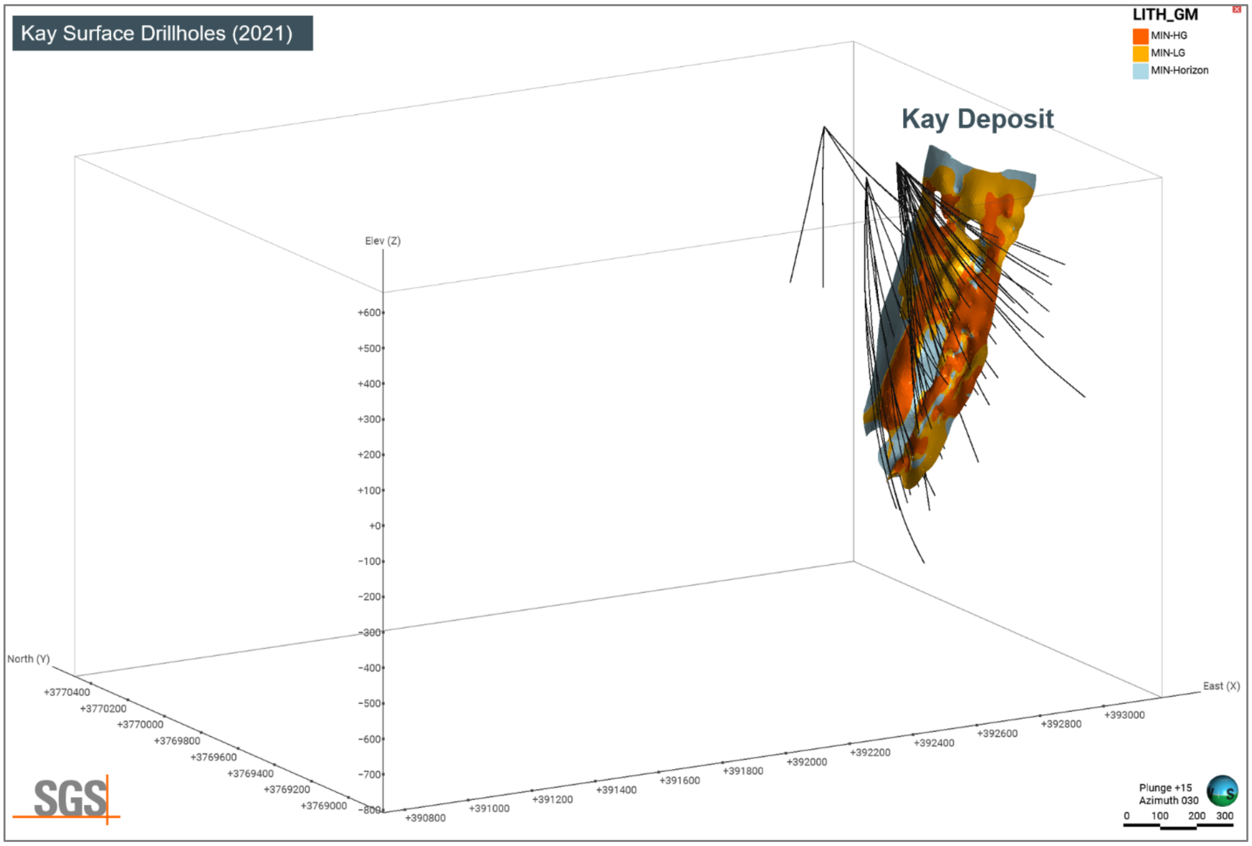


Table 10.5: Highlights of the 2021 Drilling

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
KM-21-17	429.5	449.9	**20.4**	**1.81**	**1.10**	**1.20**	21.2	0.17
including	429.5	434.0	4.6	4.61	1.73	1.91	29.1	0.24
including	432.7	434.0	1.4	0.52	6.81	8.29	40.0	1.10
KM-21-18A	391.4	423.8	**32.5**	**1.09**	**0.62**	**1.25**	17.7	0.15
including	393.3	395.8	2.4	9.57	2.83	2.72	40.9	0.28
KM-21-21	452.6	495.5	**42.8**	**0.80**	**0.78**	**1.52**	15.1	0.15
including	488.7	493.5	4.8	0.26	2.50	6.13	27.6	0.54
KM-21-21A	439.1	502.1	**63.0**	**0.45**	**1.28**	**3.14**	58.8	0.77
including	465.0	481.9	16.9	0.52	2.45	4.05	80.9	0.99
KM-21-24	501.2	592.1	**90.8**	**0.45**	**1.33**	**3.42**	44.6	0.41
including	501.2	521.7	20.4	1.34	1.70	6.35	113.1	0.66
including	520.9	521.7	0.8	1.75	16.50	9.55	574.0	1.22

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
including	575.9	592.1	16.2	0.16	2.50	6.00	44.4	0.79
including	588.7	590.4	1.7	0.47	9.98	23.70	18.2	0.13
KM-21-25	662.6	741.3	**78.6**	**1.41**	**2.33**	**2.79**	43.4	0.35
including	663.2	672.7	9.4	8.06	1.84	1.31	92.3	0.15
including	693.0	703.9	11.0	0.68	6.28	10.40	99.7	1.17
KM-21-25A	654.7	719.9	**65.2**	**1.04**	**1.94**	**2.15**	18.9	0.18
including	655.5	662.8	7.3	3.66	2.09	1.85	30.2	0.21
including	710.8	716.9	6.1	2.72	7.95	3.73	37.4	0.31
KM-21-26	506.7	582.8	**76.0**	**0.79**	**1.61**	**4.23**	32.7	0.54
including	511.1	526.1	14.9	0.73	1.78	9.68	43.3	0.77
including	573.8	582.8	9.0	4.02	6.06	3.32	18.2	0.19
KM-21-27A	666.3	769.4	**103.1**	**0.79**	**1.06**	**1.90**	35.8	0.42
including	666.3	687.0	20.7	3.21	1.39	1.26	19.4	0.20
including	706.4	724.6	18.3	0.69	2.69	4.70	92.2	1.21
including	752.9	763.8	11.0	0.07	1.07	4.68	95.3	0.98
KM-21-27B	665.8	762.9	**97.1**	**1.31**	**1.62**	**3.21**	31.7	0.40
including	702.0	723.0	21.0	0.87	4.56	9.03	81.5	1.10
including	723.0	738.2	15.2	4.97	0.36	0.42	18.7	0.05
KM-21-28	640.7	694.9	**54.3**	**1.87**	**2.85**	**5.03**	29.4	0.70
including	660.2	671.6	11.4	0.54	4.29	9.30	32.2	1.17
including	681.1	689.0	7.9	4.39	9.47	10.34	93.1	2.41
including	690.4	692.6	2.2	16.06	0.82	0.06	55.8	0.01
KM-21-40	627.9	680.8	**52.9**	**0.47**	**2.91**	**3.40**	35.7	0.40
including	641.1	648.3	7.2	1.15	7.66	8.27	88.5	0.92
including	670.3	674.1	3.8	1.53	10.89	9.47	24.6	0.61
KM-21-41	462.6	559.3	**96.7**	**1.04**	**1.54**	**2.66**	40.8	0.35
including	503.2	514.2	11.0	0.99	5.34	8.17	106.3	1.63
including	546.7	558.1	11.4	5.86	5.83	3.24	185.4	0.04
including	553.1	556.9	3.8	7.11	9.55	5.70	505.8	0.09
KM-21-42A	840.9	877.2	**36.3**	**0.55**	**0.62**	**1.35**	10.7	0.13
KM-21-42C	849.2	877.4	**28.2**	**3.81**	**0.47**	**0.29**	12.5	0.09

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
including	849.2	854.7	5.5	14.57	0.66	0.16	37.5	0.03
including	863.8	869.4	5.6	2.29	1.17	0.59	13.1	0.25
including	874.8	877.4	2.6	2.83	0.26	0.03	7.2	0.01
KM-21-50	489.5	501.9	**12.3**	**0.98**	**2.30**	**6.36**	111.9	1.24
including	489.5	493.0	3.4	2.64	3.59	9.49	207.7	1.65
KM-21-50	509.0	562.1	**53.1**	**0.44**	**0.84**	**1.28**	35.8	0.27
including	538.1	545.6	7.5	0.28	1.94	2.62	112.8	0.82
KM-21-52A	763.7	793.1	**29.4**	**0.25**	**1.12**	**1.36**	51.6	0.47
including	763.7	764.9	1.2	0.38	3.01	8.69	132.0	1.68
including	771.8	774.5	2.7	1.39	2.46	4.59	116.4	1.82
including	781.5	787.6	6.1	0.31	2.63	1.64	119.5	0.65
KM-21-58	614.2	682.6	**68.4**	**1.30**	**3.42**	**3.85**	47.2	0.50
including	640.7	648.0	7.3	0.79	4.34	10.20	51.9	0.56
including	668.1	678.6	10.5	5.30	12.19	6.67	194.7	1.88
including	668.1	669.6	1.5	2.55	43.20	7.76	856.0	0.80
KM-21-58A	569.4	641.8	**72.5**	**1.12**	**1.00**	**2.84**	18.1	0.33
including	584.3	591.9	7.6	0.29	1.19	6.23	4.4	0.40
including	602.3	613.3	11.0	4.02	0.11	1.38	12.6	0.40
including	630.3	630.9	0.7	1.14	6.35	11.20	356.0	0.65
including	633.5	641.8	8.3	1.53	2.33	5.12	26.5	0.36
KM-21-58A	665.5	676.0	**10.5**	**0.12**	**2.90**	**3.88**	167.5	1.92
including	672.5	676.0	3.5	0.12	6.89	6.40	332.0	3.81
including	673.6	674.5	0.9	0.28	19.65	12.65	844.0	10.20
KM-21-58B	543.2	627.6	**84.4**	**1.05**	**2.38**	**3.44**	23.8	0.55
including	571.2	582.5	11.3	0.51	5.27	9.96	35.4	1.52
including	605.3	622.7	17.4	3.20	6.19	4.18	40.9	0.22
including	609.6	612.0	2.4	1.45	17.73	7.97	82.5	0.44

10.3.3 2022 Drilling

Drilling in 2022 comprised continued delineation and exploration drilling of the Kay Mine lenses (40 drillholes) and exploration drilling on the Kay North Extension and West targets. Drilling was completed on the Kay South lens at depths ranging from 450 m to 1,050 m below surface and on the North (Kay2) lens at depths ranging from 140 m to 530 m below surface. Exploration drilling on the Kay North Extension target continued with six (6) drillholes completed, and drilling was initiated on the West target with seven (7) holes completed.

Drilling in 2022 totalled 32,544 metres in 53 holes (Figure 10.5). Highlights of the 2022 drilling are presented in Table 10.6.

Figure 10.5: Location of 2022 Drillholes on the Kay Project and Mineralization Models

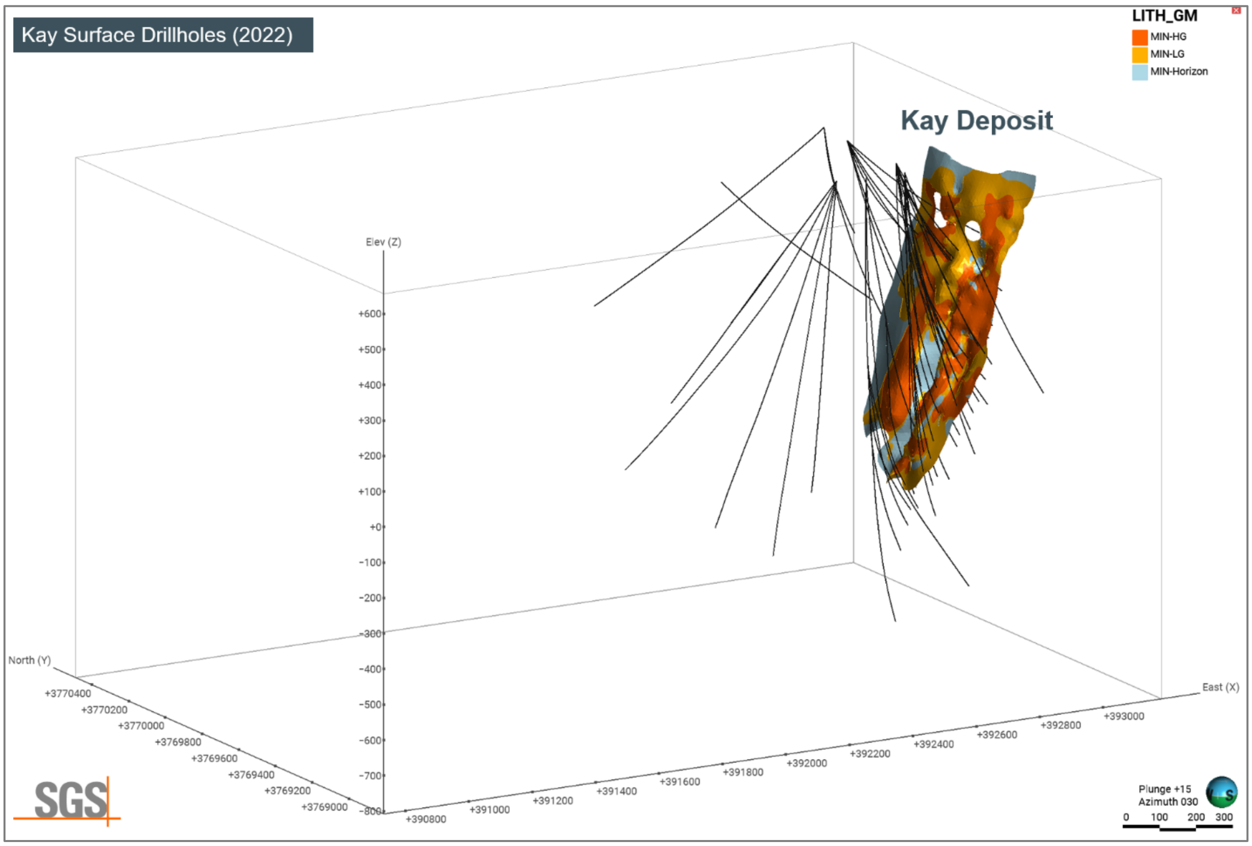


Table 10.6: Highlights of the 2022 Drilling

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
KM-22-57B	736.7	862.0	**125.3**	**1.41**	**0.83**	**1.27**	12.4	0.13
including	739.7	741.6	1.8	9.42	2.37	0.32	8.5	0.03
including	798.3	805.6	7.3	6.35	0.81	3.76	19.5	0.14
KM-22-57C	784.3	885.1	**100.9**	**1.24**	**1.54**	**1.56**	25.8	0.14
including	829.4	837.9	8.5	1.60	7.71	9.04	100.9	0.35

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
including	852.2	857.6	5.3	6.81	0.10	0.09	23.3	0.02
KM-22-60	554.7	648.0	**93.3**	**1.36**	**5.65**	**3.25**	32.6	0.34
including	591.6	597.7	6.1	0.58	5.62	12.00	56.3	1.40
including	627.0	644.5	17.5	5.22	25.37	4.71	100.6	0.59
including	634.3	635.5	1.2	5.63	273.00	0.18	715.0	0.28
KM-22-62	636.6	682.8	**46.2**	**0.22**	**1.47**	**3.22**	53.5	0.47
including	644.4	646.2	1.8	0.89	4.36	19.26	133.0	0.77
including	650.7	657.5	6.8	0.34	3.21	9.59	145.2	1.79
including	663.2	665.5	2.3	0.53	8.66	7.82	181.6	1.55
KM-22-62A	582.2	643.6	**61.4**	**0.31**	**1.27**	**2.65**	40.8	0.58
including	593.1	602.4	9.3	1.15	2.29	4.37	52.4	0.91
including	608.9	617.8	8.8	0.20	1.79	4.26	91.2	1.15
including	627.7	630.9	3.2	0.41	7.10	15.01	180.0	2.77
KM-22-71	657.8	668.6	**10.8**	**3.18**	**0.35**	**0.16**	22.6	0.01
including	657.8	661.4	3.7	6.75	0.28	0.09	30.9	0.02
KM-22-74	649.2	688.2	**39.0**	**0.40**	**1.77**	**3.39**	30.5	0.32
including	652.6	659.8	7.2	0.68	2.57	5.13	18.0	0.11
including	678.5	688.2	9.8	0.15	3.08	5.67	32.0	0.51
KM-22-81B	801.8	805.6	**3.8**	**9.60**	**1.81**	**1.83**	44.6	0.23
including	802.7	804.2	1.5	14.80	2.75	2.06	53.0	0.28

10.3.4 2023 Drilling

Drilling in 2023 comprised delineation and exploration drilling of the Kay Mine lenses and exploration drilling on the West and B targets. Drilling was completed with 30 drillholes on the Kay South and North (Kay2) lenses at depths ranging from 30 m to 480 m below surface. Shallowly dipping drillholes (-15° to -45°) were completed to test the up-dip mineralization extents of the Kay lenses close to surface. Exploration drilling on the West target continued with nine drillholes completed, and one (1) drillhole was completed into Target B, located midway between the West target and the Kay Mine deposit.

Drilling in 2023 totalled 24,126 metres in 39 holes (Figure 10.6). Highlights of the 2023 drilling are presented in Table 10.7.

Figure 10.6: Location of 2023 Drillholes on the Kay Project and Mineralization Models

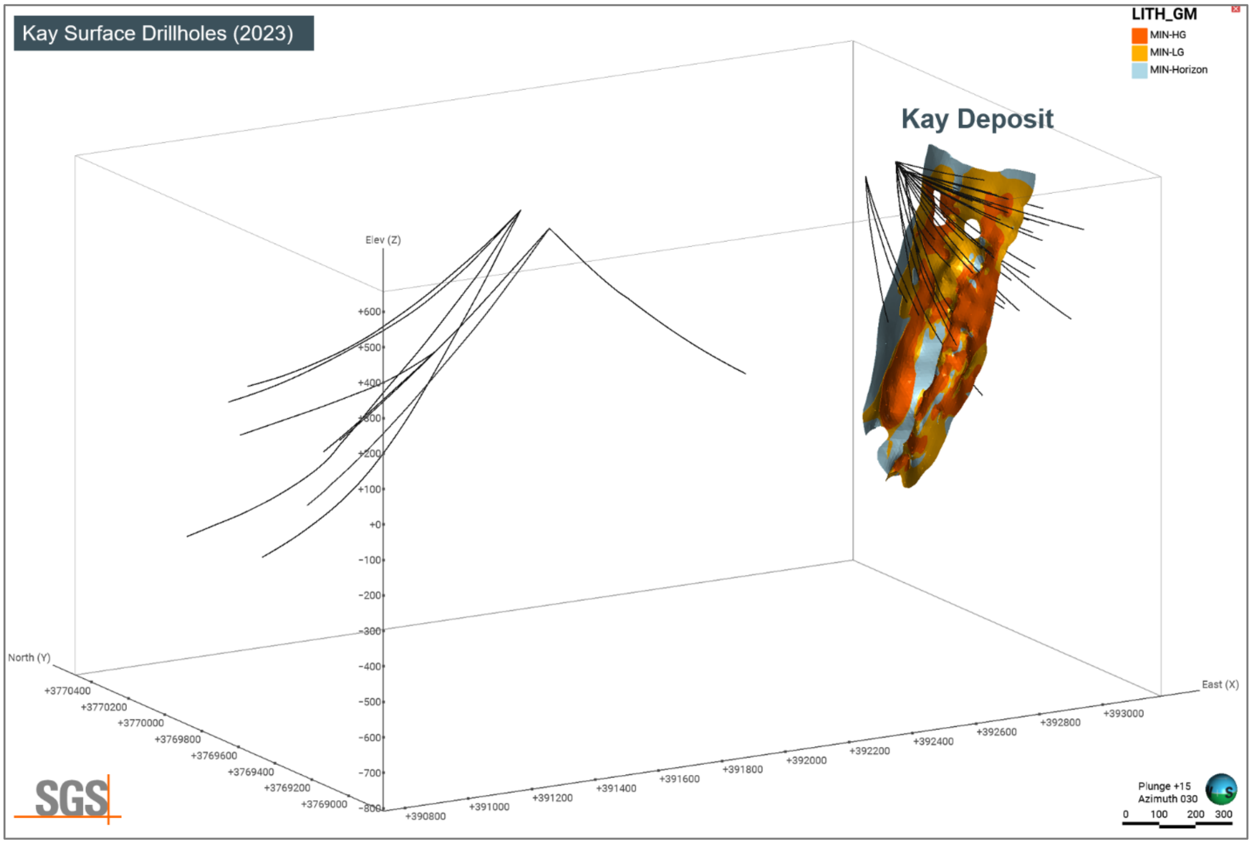


Table 10.7: Highlights of the 2023 Drilling

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
KM-23-97	512.2	521.0	**8.8**	**2.87**	**2.24**	**2.65**	27.7	0.31
including	516.1	517.7	1.6	8.12	3.67	2.33	61.2	0.14
including	516.8	517.2	0.4	17.10	4.59	0.40	59.0	0.08
KM-23-103	386.3	396.9	**10.5**	**2.40**	**3.25**	**6.09**	36.1	0.85
including	387.9	390.6	2.7	0.86	8.21	16.08	42.5	1.39
including	392.9	394.4	1.5	7.55	1.82	2.62	26.0	0.14
KM-23-106	517.4	566.6	**49.2**	**1.15**	**1.19**	**1.71**	14.4	0.44
including	556.3	566.6	10.4	5.10	3.05	0.47	22.6	0.01
KM-23-115	488.1	571.8	**83.7**	**0.38**	**1.19**	**3.00**	34.8	0.48
including	494.2	509.5	15.3	0.91	0.85	6.08	54.9	0.95
including	529.7	536.6	6.9	0.53	2.88	6.44	52.4	0.77
including	556.3	563.3	7.0	0.12	1.65	6.04	69.4	1.21

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
including	568.8	571.8	3.0	1.03	5.87	2.70	14.5	0.04
KM-23-117	539.2	604.8	**65.6**	**0.44**	**1.14**	**2.88**	24.7	0.43
including	574.4	580.1	5.7	0.53	2.42	6.36	29.2	0.51
including	588.4	591.6	3.2	0.50	8.14	12.58	97.4	1.77
including	602.6	604.3	1.7	0.24	3.96	11.36	135.3	1.78
KM-23-122	386.1	418.2	**32.1**	**0.69**	**0.60**	**0.84**	15.5	0.15
including	388.3	392.9	4.6	3.28	0.75	1.36	21.7	0.12
KM-23-132	378.1	404.5	**26.4**	**0.84**	**0.90**	**1.77**	12.1	0.22
including	389.6	392.0	2.4	3.18	1.09	1.39	18.6	0.10
including	398.7	401.5	2.7	2.12	2.72	3.04	25.2	0.37

10.3.5 2024 Drilling

Drilling in 2024 comprised delineation and exploration drilling of the Kay Mine lenses (37 drillholes) and exploration drilling on the West and North Central targets. Drilling on the Kay South lens was predominantly infill at depths ranging from 90 m to 780 m below surface. Drilling on the North (Kay2) lens included continued testing of the up-dip mineralization extents close to surface and, importantly, testing and discovery of a thickened zone of mineralization in the North (Kay2) lens between 600 m and 740 m below surface. Drilling depths on the North (Kay2) lens ranged from 50 m to 960 m below surface. Exploration drilling on the West target continued with three (3) drillholes completed, and drilling of the North Central target was initiated with 13 drillholes completed.

Drilling in 2024 totalled 28,402 metres in 53 holes (Figure 10.7). Highlights of the 2024 drilling are presented in Table 10.8.

Figure 10.7: Location of 2024 Drillholes on the Kay Project and Mineralization Models

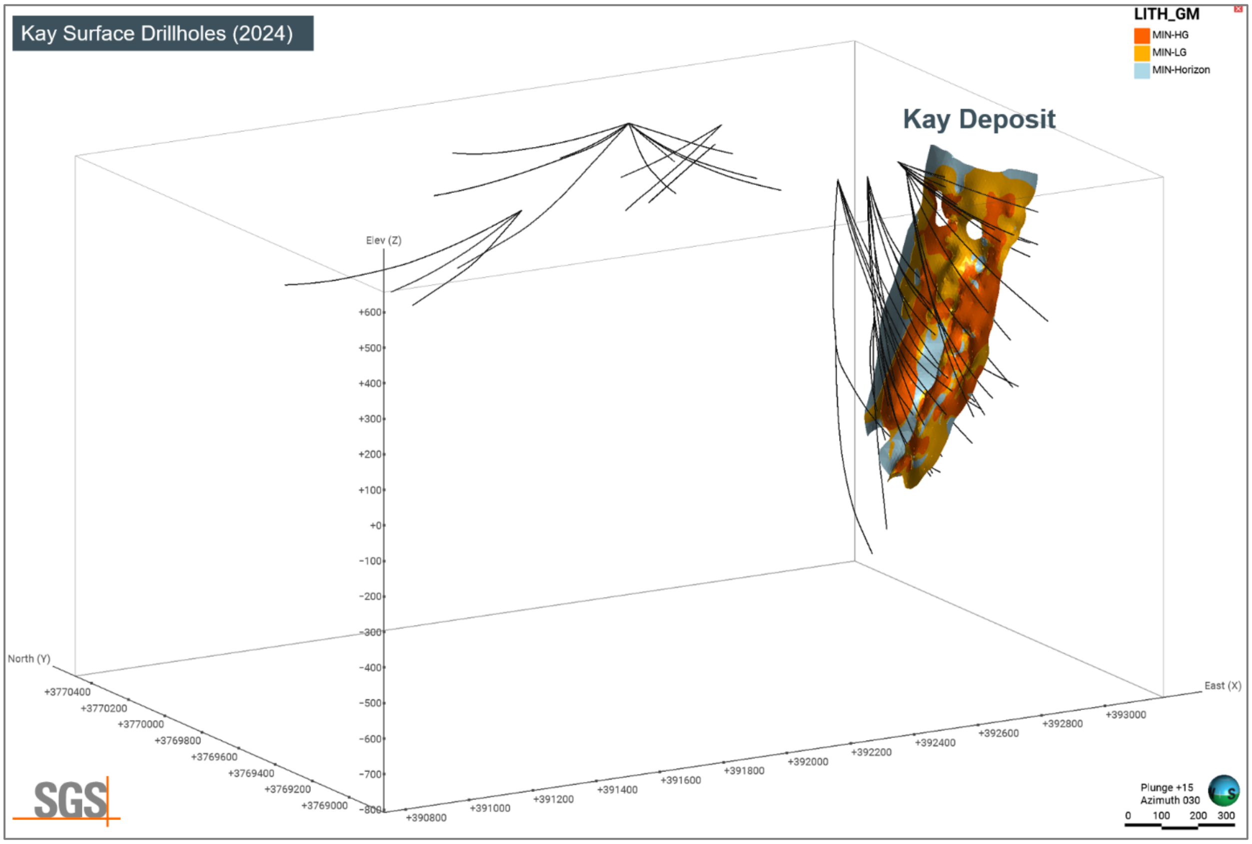


Table 10.8: Highlights of the 2024 Drilling

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
KM-24-94B	694.3	759.6	**65.2**	**1.37**	**2.48**	**3.82**	35.1	0.50
including	721.0	735.2	14.2	0.73	5.84	9.17	101.2	1.74
including	743.1	753.5	10.4	4.44	4.34	2.33	33.4	0.17
KM-24-139	525.9	563.9	**38.0**	**1.03**	**0.26**	**0.57**	13.6	0.09
including	553.1	557.5	4.4	6.57	0.63	1.64	23.5	0.13
KM-24-143	626.2	646.3	**20.1**	**1.88**	**1.05**	**2.05**	62.4	0.81
including	640.8	644.0	3.2	8.21	4.10	8.62	290.9	3.88
KM-24-146	830.3	857.7	**27.4**	**2.52**	**0.06**	**0.20**	6.1	0.01
including	851.0	854.2	3.2	7.51	0.09	0.06	12.5	0.00
KM-24-146A	790.7	851.8	**61.1**	**1.19**	**0.15**	**0.54**	4.6	0.03
including	820.1	821.6	1.5	9.94	0.07	0.08	22.0	0.04
including	820.1	824.6	4.6	5.19	0.08	0.04	11.1	0.02

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
including	834.2	835.5	1.2	8.08	0.12	0.07	19.0	0.03
KM-24-165	686.1	700.7	**14.6**	**0.47**	**1.08**	**4.18**	75.5	1.16
including	686.1	690.1	4.0	0.30	2.00	11.58	176.6	3.27
KM-24-166	663.2	713.2	**50.0**	**0.66**	**3.17**	**5.15**	30.5	0.49
including	676.2	683.1	6.9	0.49	5.76	11.14	92.7	1.79
KM-24-170	731.5	751.6	**20.1**	**0.55**	**1.59**	**2.64**	7.0	0.03
including	737.9	739.3	1.4	0.27	8.03	3.10	4.0	0.03
KM-24-170C	688.9	723.6	**34.8**	**0.75**	**6.04**	**8.47**	72.9	1.16
including	690.2	692.2	2.0	0.90	18.74	9.32	204.6	5.42
including	709.9	713.8	4.0	0.40	12.14	13.49	142.1	2.53

10.3.6 2025 Drilling (to June 2025 – MRE Data Cut-Off)

Drilling continued in 2025 and, as of June 17 (final hole included in the MRE), consisted of exploration drilling into the deeper portions of the North (Kay2) lens. Drilling targeting the North (Kay2) lens included four (4) holes testing the thickened zone of mineralization at depths of between 540 m and 690 m below surface and three (3) deep exploration holes targeting mineralization at depths of approximately 1,080 m to 1,250 m below surface.

Drilling in 2025 to June 17 totalled 6,500 metres in seven (7) holes (Figure 10.8). Highlights of the 2025 drilling are presented in Table 10.9.

Figure 10.8: Location of 2025 Drillholes (to June 17, 2025) on the Kay Project and Mineralization Models

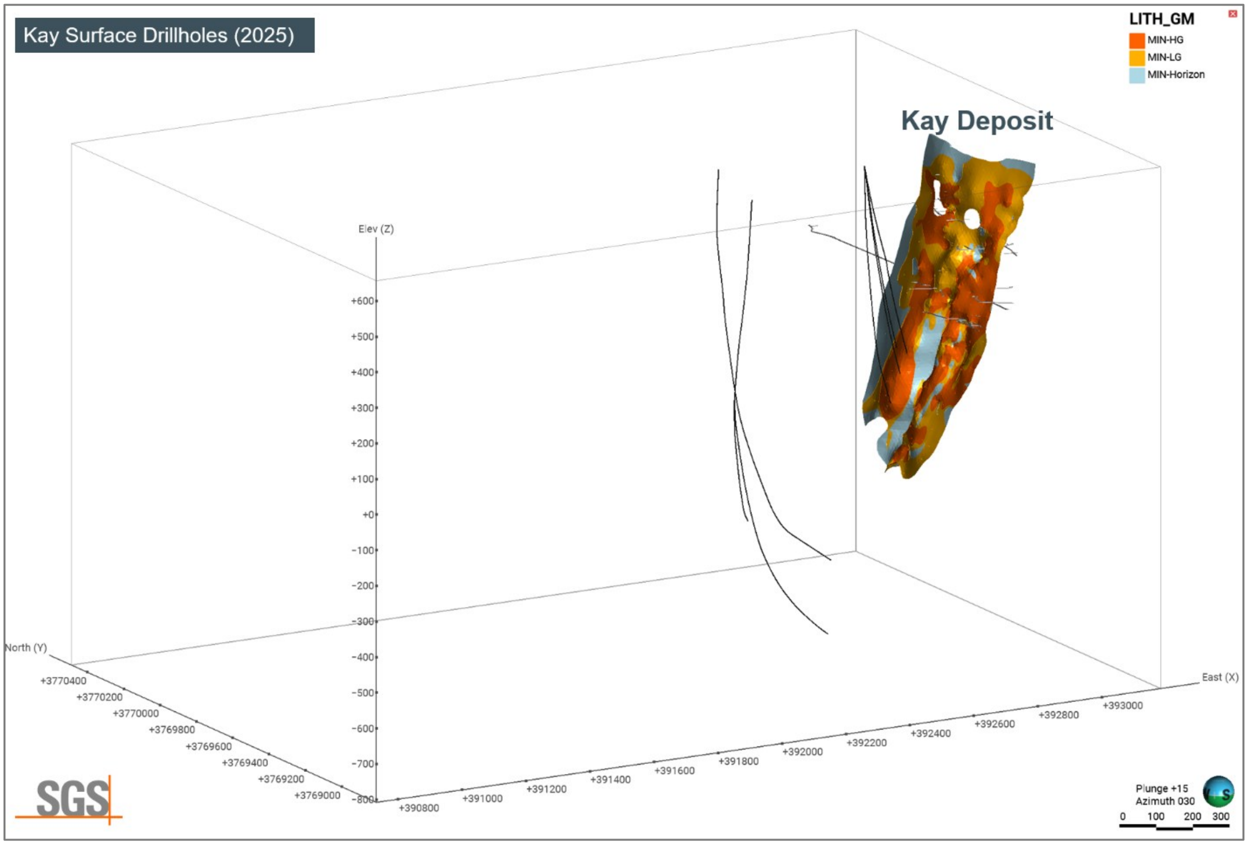


Table 10.9: Highlights of the 2025 Drilling (to June 17, 2025)

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
KM-25-178	614.2	632.8	**18.6**	**1.15**	**1.23**	**1.40**	4.8	0.10
including	623.6	626.5	2.9	0.28	3.29	6.42	7.4	0.50
KM-25-178	685.7	694.0	**8.4**	**1.67**	**0.65**	**0.05**	6.9	0.02
including	686.9	688.1	1.2	5.08	2.88	0.07	21.6	0.02
KM-25-179	607.2	639.2	**32.0**	**0.94**	**1.37**	**4.25**	27.2	0.56
including	609.5	611.7	2.3	0.43	5.44	12.10	41.1	0.30
including	619.8	625.9	6.1	0.65	2.73	12.19	35.9	1.86
KM-25-180	657.6	702.1	**44.5**	**0.67**	**1.68**	**2.78**	18.7	0.12
including	663.2	672.7	9.5	0.43	5.37	7.14	59.2	0.35
including	671.5	672.4	0.9	0.99	18.85	8.20	191.0	1.40
KM-25-181	734.7	764.3	**29.6**	**0.74**	**8.51**	**5.23**	47.0	0.50
including	750.7	764.3	13.6	1.46	13.88	8.79	38.7	0.47

10.3.7 Post-MRE Drilling (June 2025 to March 2026)

Arizona Metals has continued to drill on the Project since the data cut-off for the current MRE. Exploration drilling completed in the period from June 2025 through March 2026 totals 20 drillholes for 12,932 m. Drilling during this period focused on exploration targets outside of the Kay MRE area, targeting the prospective stratigraphy at the Kay North Extension and Northwest Target areas. Drilling within or proximal to the Kay MRE was limited to two (2) drillholes (KM-25-194 and KM-25-195). The exploration drillholes completed in the second half of 2025 and 2026 and their location relative to the MRE are not likely to materially change the current MRE for the Project.

Drilling in the Kay North Extension area intersected 2.4 m @ 0.7% Cu, 0.01 g/t Au, and 0.04% Zn along the northern extension of the Kay mineralized horizon in hole M-25-190. On the Northwest target, drill hole KM-25-197 intersected 0.3 m @ 0.27% Cu, 0.90 g/t Au, and 0.44% Zn. This hole was drilled to the east from Pad 15, into a previously untested portion of the prospective horizon on the Property. It targeted a surface sample within a mapped mineral horizon that returned 3.2% Cu. Although narrow, this is an encouraging result in an unexplored area of the Property.

Drilling from June 2025 to March 2026 totalled 12,932 metres in 20 holes (Figure 10.9). Highlights of the post-MRE drilling are presented in Table 10.10. Drilling results have been reported for hole KM-25-198.

Figure 10.9: Post-MRE Drillholes Location (July 2025 to March 2026) on Kay Project and Mineralization Models

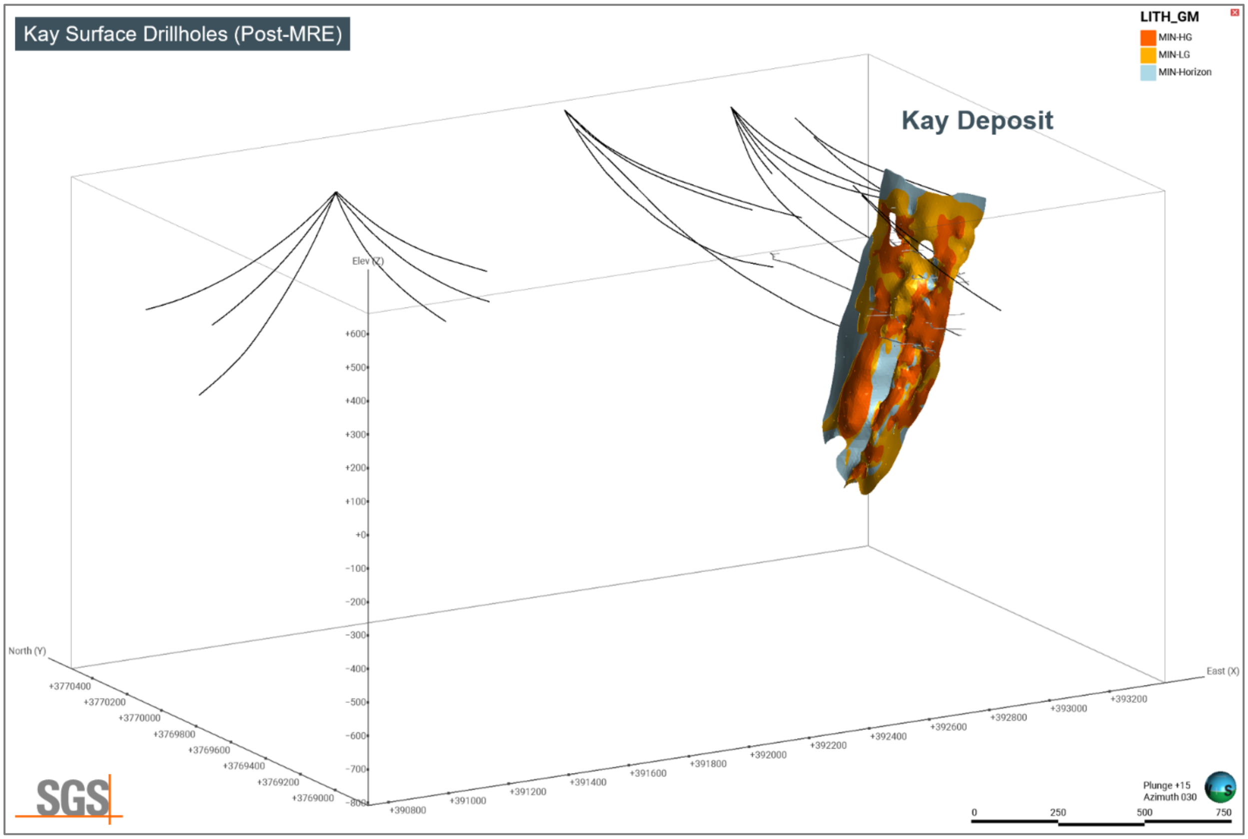


Table 10.10: Highlights of the Post-MRE Drilling (2025 to Hole KM-25-198)

Hole ID	From m	To m	Length m	Cu %	Au g/t	Zn %	Ag g/t	Pb %
KM-25-190	212.0	212.6	**0.6**	**0.16**	**0.01**	**1.76**	1.0	0.00
and	272.2	274.6	**2.4**	**0.70**	**0.01**	**0.04**	3.1	0.01
KM-25-195	265.3	287.7	**22.4**	**0.56**	**0.41**	**0.99**	5.1	0.07
KM-25-197	408.6	408.9	**0.3**	**0.27**	**0.90**	**0.44**	9.0	0.02

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Overview

Since initiating drilling on the Property in January 2020, Arizona Metals has maintained a consistent system for the sample preparation, analysis and security of all surface samples and drill core samples, including the implementation of a QA/QC protocol. The current MRE is limited to drilling data collected by Arizona Metals since the acquisition of the Property until June 17, 2025, as summarized in Table 11.1. The following describes sample preparation, analyses and security protocols implemented by Arizona Metals, with analytical labs and analysis methods summarized in Table 11.2.

Since 2020, all samples have been shipped to ALS Limited (ALS) in Tucson, Arizona, USA, for sample preparation and transferred for analysis at the ALS laboratory in North Vancouver, BC, Canada. The ALS Tucson and North Vancouver facilities are ISO/IEC 17025 certified. Samples are dried, weighed, and crushed to at least 70% passing 2 mm, and a 250 g split is pulverized to at least 85% passing 75 µm. Base metals and silver are analyzed using an intermediate level four-acid digestion with an inductively coupled plasma (ICP) finish. Over-limit analyses for copper, lead, zinc (> 100,000 ppm), and silver (> 200 ppm) are re-assayed using an ore-grade four-acid digestion with an ICP finish. Gold is assayed by a 30-gram fire assay with atomic absorption (AA) spectroscopy finish. Over-limit analyses for gold (> 10 ppm) are re-assayed using a 30-gram fire assay with a gravimetric finish. Control samples comprising certified reference samples, blank samples, and duplicates are systematically inserted into the sample stream and analyzed as part of the Company's QA/QC protocol. ALS is independent of Arizona Metals, the QPs, and SGS Geological Services.

Table 11.1: Summary of Drilling Samples from the Property by Year to June 17, 2025

Year	Company	Hole Type	Core Size	Drillhole Prefix	Drillhole Count	Length Drilled (m)	Sample Count
2020	Arizona Metals	DDH	HQ	KM-20	21	8,416.75	617
2021			HQ	KM-21	60	33,924.24	2,681
2022			HQ	KM-22	53	32,543.50	2,147
2023			HQ	KM-23	39	24,125.53	3,140
2024			HQ	KM-24	53	28,402.33	2,596
2025			HQ	KM-25	7	6,499.56	352
Total					**233**	**133,911.90**	**11,533**

Table 11.2: Summary of Drill Core Analytical Labs and Analysis Methods 2020 – 2025

Year	Company	Lab & Location	Prep Code	Fire Assay Method	Fire Assay Code	Multi-element Method	Multi-element Code
2020-2025	Arizona Metals	ALS Limited, Tucson, Arizona (prep.) & North Vancouver, British Columbia (analysis)	PREP-31	Au 30 g FA-AA finish, Overlimit Au 30 g FA-Gravimetric finish	Au-AA23, Au-GRA21	Intermediate Level Four Acid ICP-AES, Overlimit Ore Grade Four Acid ICP-AES	ME-ICP61a, ME-OG62

11.2 Sampling Methods

11.2.1 Rock Sampling

Surface rock samples collected from the Property include due diligence and reconnaissance samples, samples collected during geologic mapping, and a grid of rock samples covering the full property. Surface rock samples taken from potentially mineralized material are collected as in situ grab samples or as float samples. Rock-grid samples were collected at a spacing of approximately 50 m. Samples were placed in a bag with a unique sample ID tag and packed, together with other rock samples, into larger bags for shipment to the lab. Samples were submitted to ALS Minerals for Au and multi-element analysis with the same methods used for drill core samples.

11.2.2 Drill Core

Diamond drilling completed by Arizona Metals from 2020 to 2025 utilized conventional surface drills to produce predominantly HQ size (63.5 mm diameter) core and some NQ size (47.6 mm diameter) core.

Drill core is placed sequentially in core boxes with lids and marked with hole numbers at the drill by the drillers. A wooden block marker is inserted at the end of each core run, recording the down-hole depth and recovered interval. Core is transported to Arizona Metals logging facilities located in North Phoenix and back to the Property for cutting and sampling.

Core depth markers and box numbers are checked, and the drill core is cleaned prior to being logged and photographed. The core is logged geotechnically on a drill run-by-run basis for core recovery. Any void intervals associated with historical development are accounted for and recorded in the geology logs.

The drill core is logged for lithology, alteration, mineralization, and structure, prior to marking out sample intervals. Lithological and sample logging are done digitally using MS Excel software. Sample intervals are defined to honour mineralization, alteration, and lithology contacts. Suspect high-grade intervals are sampled separately. The nominal sample length is 1.5 m (5 ft) with a general maximum sample length of 1.5 m (5 ft) and a minimum sample length of 0.3 m (1 ft). The core is photographed after logging but prior to sampling.

The sampler saws the core in half, with half being submitted for analysis and half remaining in the core box as a record. Only one piece of core is removed from the core box at a time, and care is taken to replace the unsampled portion of the core in the core box in the original orientation. The drillhole number and sample intervals are clearly entered into a sample book to back up the digital logging files. The geologist staples the portion of the uniquely numbered sample ticket at the beginning of the corresponding sample interval in the core box, and the sampler places one portion of the ticket in the sample bag. The sample ticket book is archived. Certified reference materials, blanks, and duplicates are inserted into the sample stream. Cut samples and sample number sequences are checked for quality control prior to dispatch.

11.3 Sample Security and Storage

All exploration samples taken were collected by Arizona Metals staff or subcontractors to the Company. Chain of custody (COC) for samples was carefully maintained from collection at the drill rig to delivery at the laboratories to prevent inadvertent contamination or mixing of samples and render active tampering as difficult as possible.

At the core processing facility, the samples are bagged in sacks for transport. A control file, the laboratory sample dispatch form, includes the contained sample bag numbers in each submission. The laboratory sample dispatch form accompanies the sample shipment and is used to control and monitor the shipment. The control files are used to keep track of the time it takes for the samples to get to the lab, and the time taken to receive assay certificates, the turnaround time. The sample shipment is delivered to ALS in Tucson by Arizona Metals staff. ALS sends a confirmation email with detail of samples received upon delivery and signs a complete Chain of Custody form upon receipt of each sample submission.

Drill core is stored at the two (2) facilities, located on the Property and in North Phoenix, indoors to preserve its condition. The wax cardboard boxes containing the core are properly tagged with the corresponding drilling information and stored on pallets in an organized way and under acceptable conditions. All sample pulps are returned to the Property for storage.

11.4 Sample Preparation and Analyses

Sample preparation and reduction are carried out at ALS in Tucson, Arizona, USA and sample pulps are transferred to ALS in North Vancouver, BC, Canada for analysis. The ALS Tucson and North Vancouver facilities are ISO/IEC 17025 certified. Samples are dried, weighed, and crushed to at least 70% passing 2 mm, and a 250 g split is pulverized to at least 85% passing 75 µm (ALS Method Code PREP-31).

Base metals and silver are analyzed using an intermediate level four-acid digestion with an inductively coupled plasma (ICP) finish (ALS Method Code ME-ICP61a). Over-limit analyses for copper, lead, zinc (> 100,000 ppm), and silver (> 200 ppm) are re-assayed using an ore-grade four-acid digestion with an ICP finish (ALS Method Code OG62). Gold is assayed by 30-gram fire assay with atomic absorption (AA) spectroscopy finish (ALS Method Code Au-AA23). Over-limit analyses for gold (> 10 ppm) are re-assayed using a 30-gram fire assay with a gravimetric finish (ALS Method Code Au-GRA21).

11.5 Density

Specific gravity measurements obtained by Arizona Metals from 2020 to 2024 drill core were measured by ALS labs using the pycnometer with methanol method (ALS Method Code OA-GRA08b) on sample pulps. A prepared sample (3.0 g) is weighed into an empty pycnometer. The pycnometer is filled with a solvent (methanol) and then weighed. From the weight of the sample and the weight of the solvent displaced by the sample, the specific gravity is calculated using the following equation:

$$SG = \frac{Dry\ sample\ weight\ (g)}{Weight\ of\ solvent\ displaced\ (g)}\ x\ Specific\ Gravity\ of\ the\ Solvent$$

Specific gravity measurements on selected drill core pulps using this pycnometer method were completed in 2022 (1,899 samples) and 2004 (408 samples).

11.6 Data Management

Data are verified and double-checked by senior geologists on site for data entry verification, error analysis, and adherence to analytical quality-control protocols. All measured and observed data are collected digitally using MS Excel software.

11.7 Quality Assurance / Quality Control

Sampling QA/QC programs are set in place to ensure the reliability and trustworthiness of exploration data. They include written field procedures and independent verifications of drilling, surveying, sampling, assaying, data management, and database integrity. Appropriate documentation of quality control measures and regular analysis of quality-control data are essential for the Project data and form the basis for the quality-assurance program implemented during exploration.

Analytical quality control measures typically involve internal and external laboratory control measures implemented to monitor sampling, preparation, and assaying precision and accuracy. They are also essential to prevent sample mix-up and monitor the voluntary or inadvertent contamination of samples. Sampling QA/QC protocols typically involve regular duplicate and replicate assays as well as the insertion of blanks and standards (certified reference materials). Routine monitoring of quality control samples is undertaken to ensure that the analytical process remains in control and confirms the accuracy and precision of laboratory analyses. In addition to laboratory internal quality control protocols, sample batches should be evaluated for evidence of suspected cross-sample contamination, certified reference material performance evaluated relative to established warning and failure limits to ensure the analytical process remains in control while maintaining an acceptable level of accuracy and precision, duplicate and replicate assay performance evaluated, and any concerns communicated to the laboratory in a timely fashion. Check assaying is typically performed as an additional reliability test of assaying results. These checks involve re-assaying a set number of coarse rejects and pulps at a second umpire laboratory.

Arizona Metals' QA/QC program comprises the systematic insertion of standards or certified reference materials (CRMs) and blanks. Field duplicate samples were added to the program beginning in 2023. QC samples are inserted into the sample sequence at an insertion frequency of approximately one (1) sample per 20 samples for CRMs and blanks, and one (1) sample per 40 samples for field duplicates. A total of 10.6% of samples assayed have been QC samples in the drilling programs from 2020 to 2025. Combined routine QC sample statistics for this period are presented in Table 11.3. All QC samples listed were analyzed by the primary analytical lab (ALS).

Table 11.3: Routine QC Sample Statistics for Arizona Metals Core Sampling 2020 - 2025

Original Samples	Standards	Blanks	Field Duplicates	QC Sample Total	QC Sample %
11,533	618	614	139 pairs	1,317	10.6%

Sample batches with suspected cross-sample contamination or certified reference materials returning assay values outside of the mean ± 3SD control limits are considered analytical failures by the Company, and affected batches are re-analyzed to ensure data accuracy when deemed warranted.

ALS has its own internal QA/QC program, which is reported in the assay certificates, but no account is taken of this in the determination of batch acceptance or failure.

11.7.1 Certified Reference Material

A selection of six (6) CRMs has been used to-date by Arizona Metals in the course of the Kay Project drill program: multi-element standards from CDN Resource Laboratories in Langley, B.C. (CDN-ME-1404, CDN-ME-1410, CDN-ME-1707, CDN-ME-1902, CDN-ME-1903, and CDN-ME-2101). The means, standard deviations (SD), warning, and control limits for standards are utilized as per the QA/QC program described below.

CRM performance and analytical accuracy are evaluated using the assay concentration values relative to the certified mean concentration to define the Z-score relative to the sample sequence with warning and failure limits. Warning limits are indicated by a Z-score of between ±2 SD and ±3 SD, and control limits / failures are indicated by a Z-score of greater than ±3 SD from the certified mean. Sample batches with certified reference materials returning assay values outside of the mean ± 3SD control limits, or with suspected cross-sample contamination indicated by blank sample analysis, are considered analytical failures and selected affected batches are re-analyzed to ensure data accuracy.

For geochemical exploration analysis methods, laboratory benchmark standards are to achieve a precision and accuracy of plus or minus 10% (of the concentration) ±1 Detection Limit (DL) for duplicate analyses, in-house standards and client submitted standards, when conducting routine geochemical analyses for gold and base metals. These limits apply at, or greater than, 20 times the limit of detection. For samples containing coarse gold, native silver or copper, precision limits on duplicate analyses can exceed plus or minus 10% (of the concentration).

For mineralized material grade analysis methods, laboratory benchmark standards are to achieve a precision and accuracy of plus or minus 5% (of the concentration) ± 1 DL for duplicate analyses, in-house standards and client-submitted standards. These limits apply at 20 times the limit of detection. As in the case of routine geochemical analyses, samples containing coarse gold, native silver, or copper are less likely to meet the expected precision levels for mineralized material grade analysis.

CRM analytical results for the Arizona Metals drilling programs are summarized in Table 11.4 to Table 11.8 for Ag, Au, Cu, Pb and Zn to evaluate analytical accuracy (bias), precision (average coefficient of variation, CVAVR%), warning rates, and failure rates. Shewhart CRM control charts for Ag, Au, Cu, Pb and Zn for the Arizona Metals drilling programs are presented in Figure 11.1 to Figure 11.5.

The QA/QC program from 2020 - 2025 included the insertion of a total of 618 CRM samples (Table 11.3). The combined CRM failure rates during this period were 0.6% for Ag, 2.9% for Au, 1.3% for Cu, 0.3% for Pb, and 3.4% for Zn. CRM analytical results confirm acceptable analytical accuracy (bias less than ±5%) and acceptable analytical precision (CVAVR% within ±5%) for Ag, Au, Cu, Pb and Zn. The QP considers this CRM performance acceptable and within industry standards. Review of the Company's CRM QC program indicates that there are no significant issues with the drill core assay data.

Table 11.4: CRM Sample Ag Performance at ALS for the 2020-2025 Drill Programs

CRM Ag ppm	Certified Value		2020-2025							
	Mean	SD	Count	Mean	Bias %	CV_{AVR}%	Warning # > 2SD	Warning % > 2SD	Failure # > 3SD	Failure % > 3SD
CDN-ME-1404	59.1	1.35	14	59.9	1.4	1.7	1	7.1%	0	0.0%
CDN-ME-1410	69	1.9	108	70.3	1.8	2.2	11	10.2%	1	0.9%
CDN-ME-1707	27.9	1.45	185	27.8	-0.2	2.8	2	1.1%	0	0.0%
CDN-ME-1902	349	8.5	306	354.4	1.5	1.9	28	9.2%	3	1.0%
CDN-ME-1903	180	5.5	3	177.3	-1.5	1.2	0	0.0%	0	0.0%
CDN-ME-2101	48	2	2	49.0	2.1	3.2	0	0.0%	0	0.0%
Total	**-**	**-**	**618**	**-**	**-**	**-**	**42**	**6.8%**	**4**	**0.6%**

Table 11.5: CRM Sample Au Performance at ALS for the 2020-2025 Drill Programs

CRM Au ppm	Certified Value		2020-2025							
	Mean	SD	Count	Mean	Bias %	CV_{AVR}%	Warning # > 2SD	Warning % > 2SD	Failure # > 3SD	Failure % > 3SD
CDN-ME-1404	0.897	0.032	14	0.885	-1.3	3.4	1	7.1%	0	0.0%
CDN-ME-1410	0.542	0.024	108	0.546	0.7	3.8	7	6.5%	2	1.9%
CDN-ME-1707	2.02	0.107	185	2.067	2.3	5.9	21	11.4%	11	5.9%
CDN-ME-1902	5.38	0.21	305	5.350	-0.6	3.2	22	7.2%	5	1.6%
CDN-ME-1903	3.035	0.121	3	2.980	-1.8	5.1	1	33.3%	0	0.0%
CDN-ME-2101	0.765	0.0435	2	0.793	3.6	2.5	0	0.0%	0	0.0%
Total	**-**	**-**	**617**	**-**	**-**	**-**	**52**	**8.4%**	**18**	**2.9%**

Table 11.6: CRM Sample Cu Performance at ALS for the 2020-2025 Drill Programs

CRM Cu ppm	Certified Value		2020-2025							
	Mean	SD	Count	Mean	Bias %	CV_{AVR}%	Warning # > 2SD	Warning % > 2SD	Failure # > 3SD	Failure % > 3SD
CDN-ME-1404	4840	110	14	4790	-1.0	1.4	0	0.0%	0	0.0%
CDN-ME-1410	38000	850	108	37605	-1.0	1.7	8	7.4%	1	0.9%
CDN-ME-1707	27200	550	185	26944	-0.9	1.4	7	3.8%	1	0.5%
CDN-ME-1902	7810	135	306	7700	-1.4	1.7	41	13.4%	6	2.0%
CDN-ME-1903	12300	300	3	12333	0.3	0.6	0	0.0%	0	0.0%
CDN-ME-2101	13200	300	2	13200	0.0	0.3	0	0.0%	0	0.0%
Total	**-**	**-**	**618**	**-**	**-**	**-**	**56**	**9.1%**	**8**	**1.3%**

Table 11.7: CRM Sample Pb Performance at ALS for the 2020-2025 Drill Programs

CRM Pb ppm	Certified Value		2020-2025							
	Mean	SD	Count	Mean	Bias %	CV_{AVR}%	Warning # > 2SD	Warning % > 2SD	Failure # > 3SD	Failure % > 3SD
CDN-ME-1404	3810	90	14	3791	-0.5	1.3	1	7.1%	0	0.0%
CDN-ME-1410	2480	60	108	2472	-0.3	1.5	3	2.8%	0	0.0%
CDN-ME-1707	970	30	185	948	-2.2	2.1	2	1.1%	1	0.5%
CDN-ME-1902	22000	500	306	21726	-1.2	1.6	10	3.3%	1	0.3%
CDN-ME-1903	10600	200	3	10500	-0.9	1.2	0	0.0%	0	0.0%
CDN-ME-2101	8270	190	2	8455	2.2	1.6	0	0.0%	0	0.0%
Total	**-**	**-**	**618**	**-**	**-**	**-**	**16**	**2.6%**	**2**	**0.3%**

Table 11.8: CRM Sample Zn Performance at ALS for the 2020-2025 Drill Programs

CRM Zn ppm	Certified Value		2020-2025							
	Mean	SD	Count	Mean	Bias %	CV_{AVR}%	Warning # > 2SD	Warning % > 2SD	Failure # > 3SD	Failure % > 3SD
CDN-ME-1404	20800	350	14	20657	-0.7	1.0	1	7.1%	0	0.0%
CDN-ME-1410	36820	420	108	36531	-0.8	1.7	19	17.6%	14	13.0%
CDN-ME-1707	5390	80	185	5344	-0.9	1.6	31	16.8%	7	3.8%
CDN-ME-1902	36600	1150	306	36173	-1.2	1.5	2	0.7%	0	0.0%
CDN-ME-1903	17500	350	3	17017	-2.8	2.1	1	33.3%	0	0.0%
CDN-ME-2101	14880	285	2	14875	0.0	0.6	0	0.0%	0	0.0%
Total	**-**	**-**	**618**	**-**	**-**	**-**	**54**	**8.7%**	**21**	**3.4%**

Figure 11.1: CRM Control Chart of Ag for the 2020-2025 Drill Programs

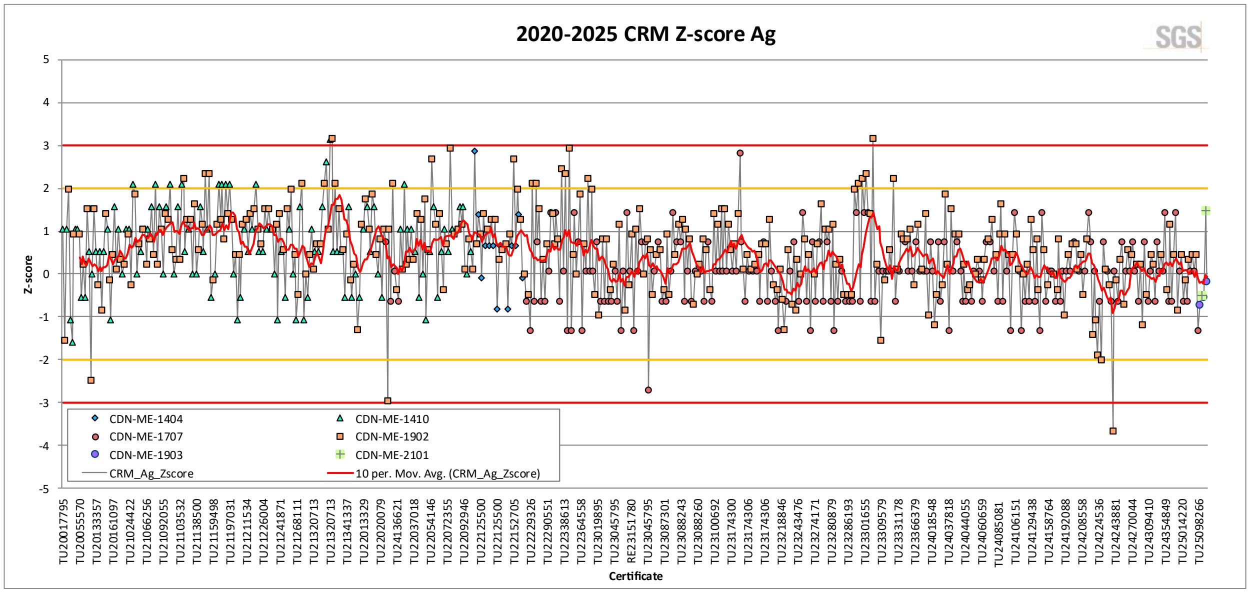


Figure 11.2: CRM Control Chart of Au for the 2020-2025 Drill Programs

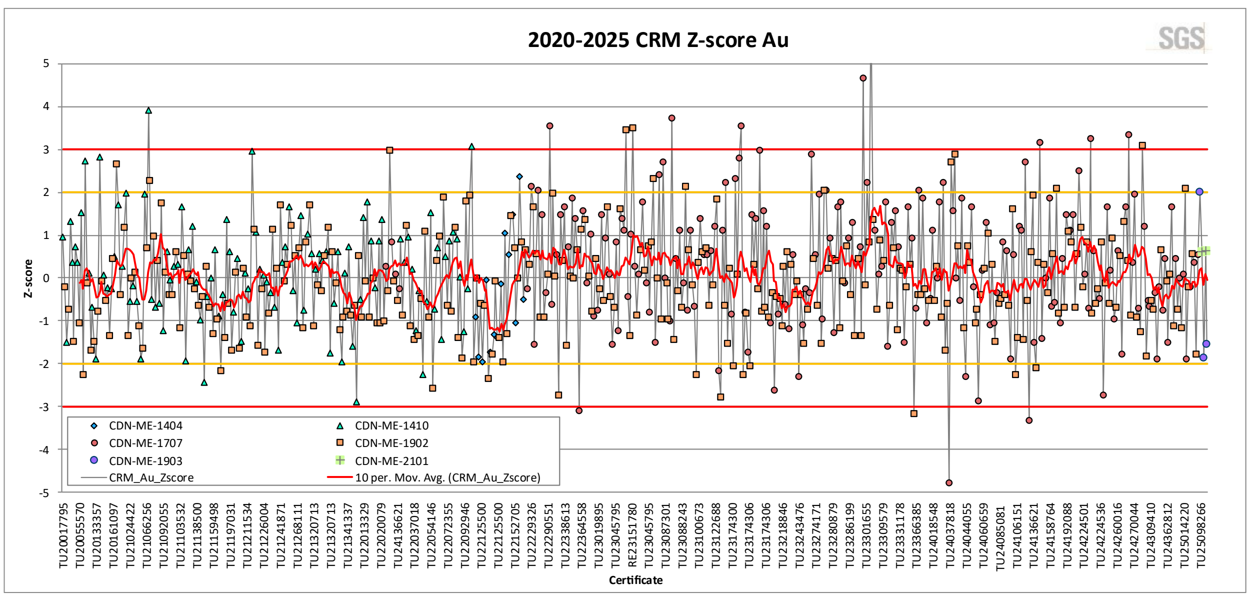


Figure 11.3: CRM Control Chart of Cu for the 2020-2025 Drill Programs

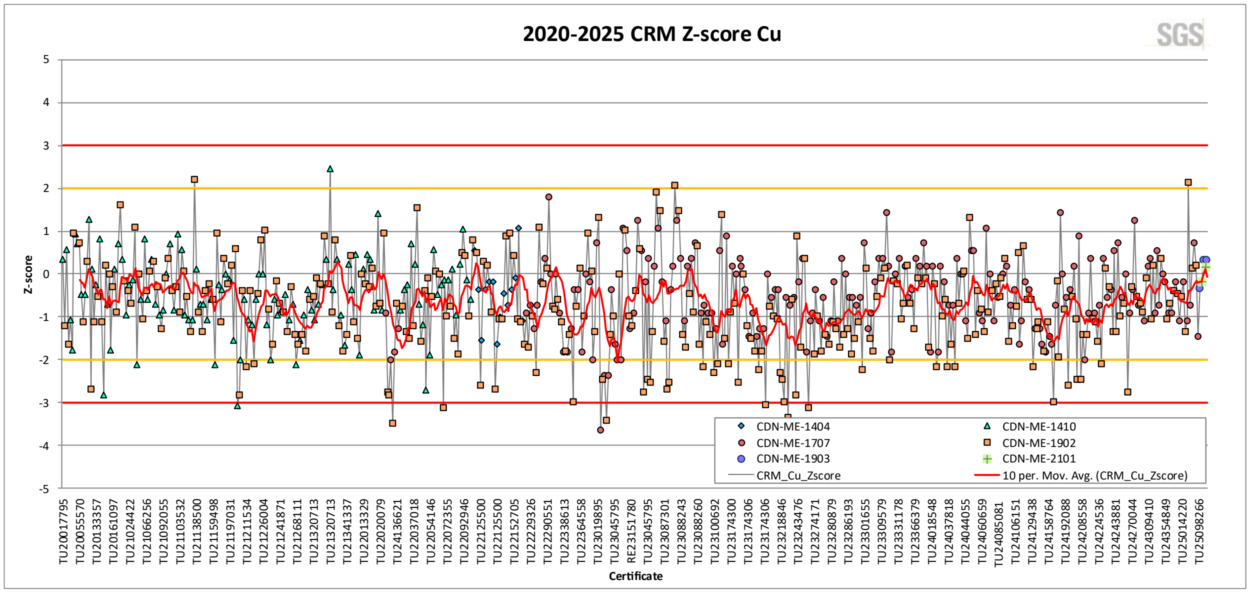


Figure 11.4: CRM Control Chart of Pb for the 2020-2025 Drill Programs

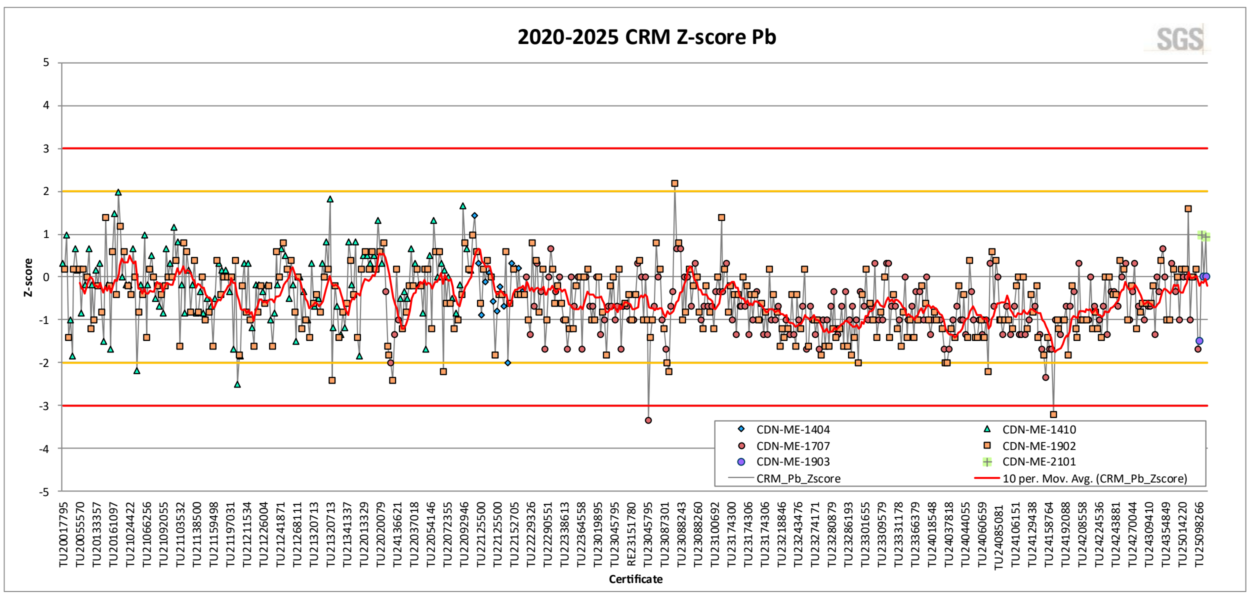
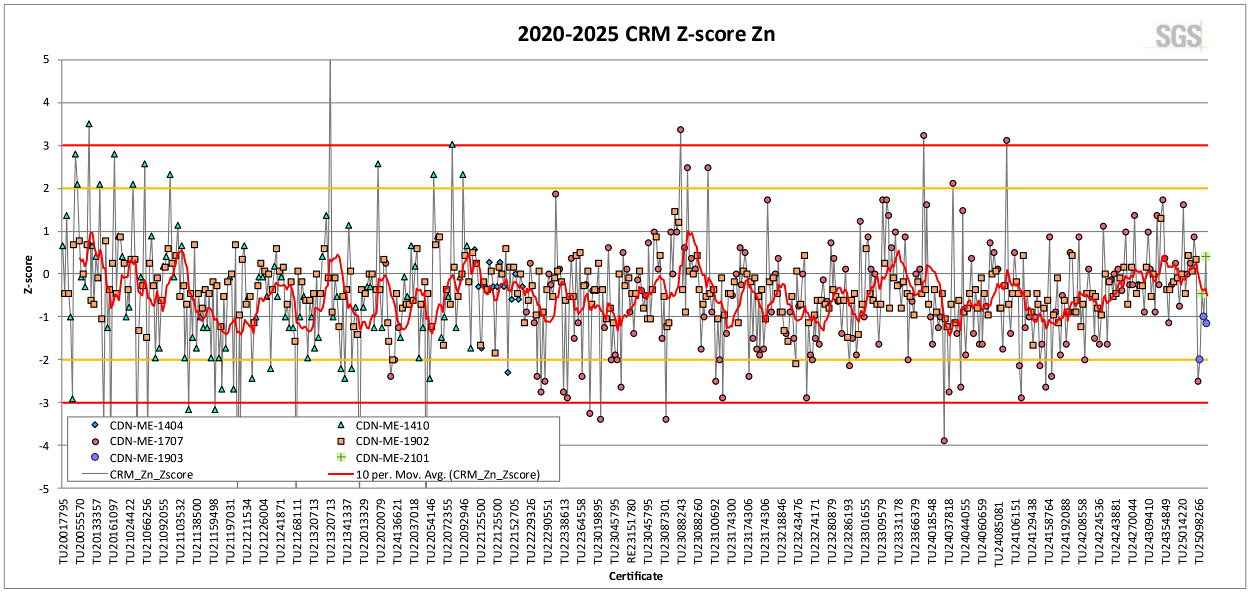


Figure 11.5: CRM Control Chart of Zn for the 2020-2025 Drill Programs



11.7.2 <u>Blank Material</u>

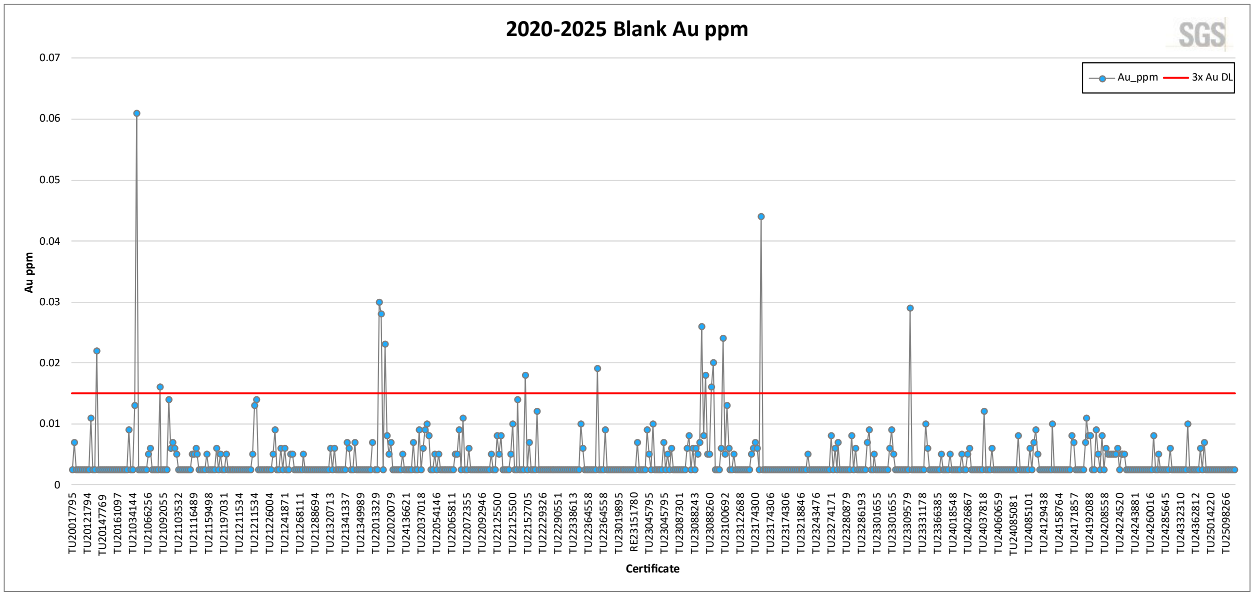
Certified blank reference samples sourced from CDN Resource Laboratories in Langley, B.C. (CDN-BL-9 and CDN-BL-10) were inserted into the sample stream in the field to determine the degree of sample carryover contamination after sample collection, particularly during the sample preparation process. This material has recommended values of less than 0.01 ppm Au established by a third party through round robin lab testing.

The QA/QC program from 2020 – 2025 included the insertion of a total of 614 blank samples (Table 11.3). For blank sample values, failure is more subjective. Some carryover within sample batches is to be expected in routine sample preparation. To minimize sample carryover within a batch, equipment is cleaned thoroughly with compressed air to remove any remaining loose material. For routine protocols, with samples of similar weights, sample carryover is usually considered acceptable if it is less than 1.0%. To ensure no batch-to-batch carryover occurs, standard quality control procedures include passing barren wash material through crushing and pulverizing equipment at the start of each new batch of samples.

Evaluation of blank samples using a failure ceiling for Au of 0.015 ppm (3x detection limit) indicates that the combined blank failure rate from 2020 – 2025 was 2.4%. The highest blank samples returned values of 0.06 ppm Au (Figure 11.6).

The blank failure rate is considered acceptable by industry standards. Based on the low risk of cross-sample carryover contamination and the low amounts of Au sample carryover that may have contaminated blank material, it is considered unlikely that there is a carryover contamination issue with the Project drilling data.

Figure 11.6: Blank Sample Chart of Au for the 2020-2025 Drill Programs



11.7.3 Duplicate Material

Field duplicate sampling was added to Arizona Metals' QA/QC program beginning in 2023. From 2023 – 2025, a total of 139 field duplicate (½ core) samples were assayed (Table 11.3). Duplicate samples were analyzed at the primary lab (ALS) to evaluate analytical precision and sampling error.

Figure 11.7 illustrates the comparative assay results and precision of duplicate sample analyses for Ag, Au, Cu, Pb and Zn.

To obtain a relatively accurate estimate of the sampling precision or average relative error, a large number of duplicate data pairs are required. Reliably determining the base metal data precision, which typically exhibits relatively small average relative errors (such as 5%), would require 500 – 1,000 duplicate data pairs, while reliable determination of gold data precision, which typically exhibits relatively large average relative errors (such as 25%), would require greater than 2,500 duplicate data pairs (Stanley and Lawie, 2007).

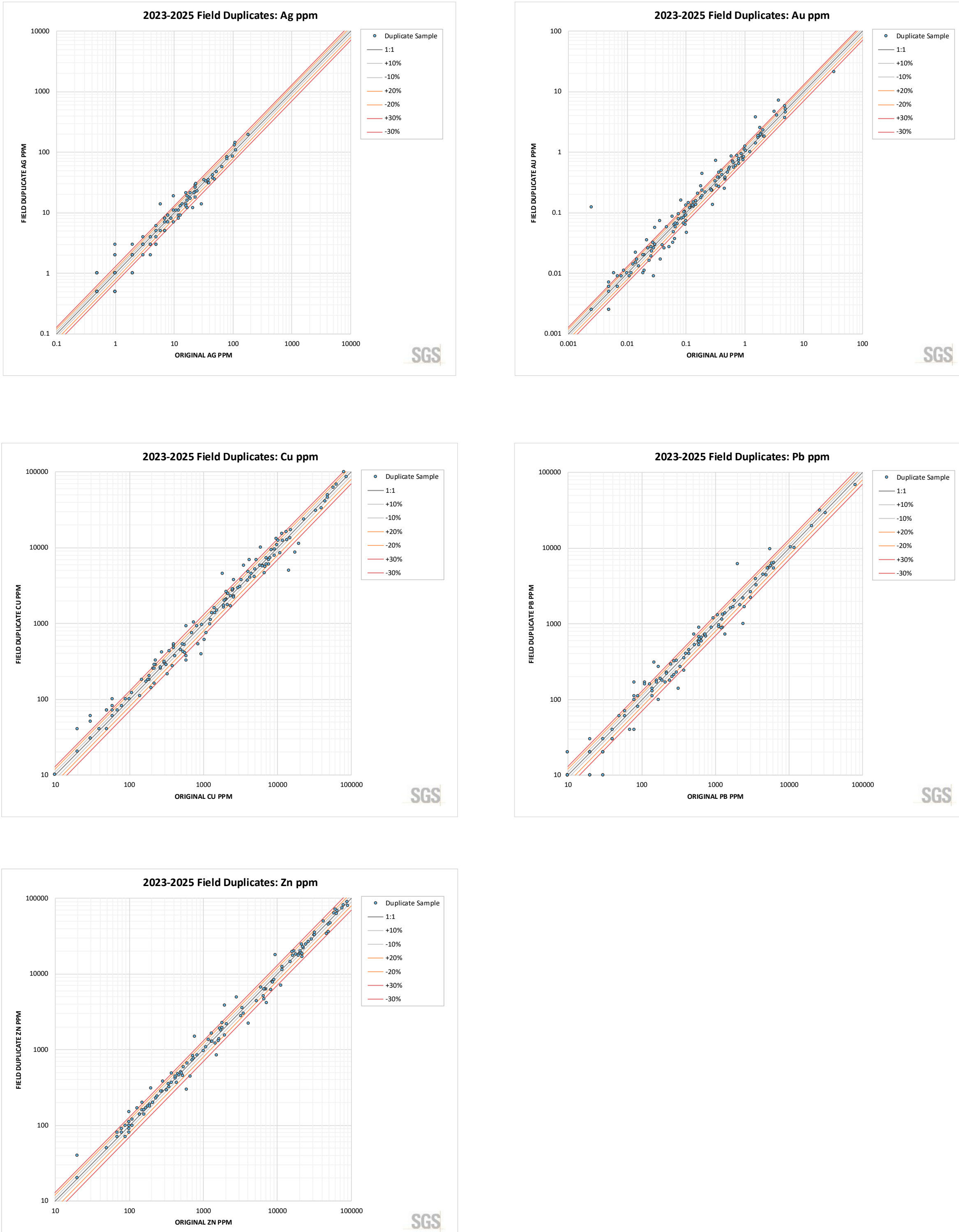
In the case of the Kay deposit, based on the current duplicate data set size for field duplicates, analysis of the precision should be considered approximate in nature only for all elements until a larger dataset is available. The average relative error, as quantified by the Average Coefficient of Variation (CVAVR%) for Ag, Au, Cu, Pb, and Zn, is shown in Table 11.9, calculated using the root mean square coefficient of variation calculated from the individual coefficients of variation.

The preliminary estimates of precision errors (CVAVR%) for Kay sampling indicate that the sampling precision is acceptable by industry standards for duplicates for this style of mineralization (Abzalov, 2008). The precision of duplicates should continue to be monitored as the drill program progresses and the size of the duplicate data set becomes more representative.

Table 11.9: Average Relative Error of Duplicate Samples for Ag, Au, Cu, Pb and Zn from the 2023-2025 Drill Programs

Drillhole Series	Duplicate Type	Count	Ag CV_{AVR}%	Au CV_{AVR}%	Cu CV_{AVR}%	Pb CV_{AVR}%	Zn CV_{AVR}%
2023-2025 Drilling	Field	139 duplicate pairs	22.9	25.3	19.3	21.6	14.8

Figure 11.7: Plots of Field Duplicate Samples for Ag, Au, Cu, Pb and Zn from the 2023-2025 Drill Programs



11.8 QP's Comments

It is the QP's opinion, based on a review of all possible information, that the sample preparation, analyses and security used on the Project by the Company meet acceptable industry standards (past and current). Review of the Company's QA/QC program indicates that there are no significant issues with the drill core assay data. The data verification programs undertaken on the data collected from the Project support the geological interpretations, and the analytical and database quality, and therefore, the data can support the estimation of Indicated and Inferred mineral resources.

12. DATA VERIFICATION

12.1 Introduction

The following section summarises the data verification procedures that were carried out, completed and documented by the Authors for this Technical Report, including verification of all drill data collected by Arizona Metals during their 2020 to 2025 drill programs, as of the effective date of the MRE.

12.2 Drill Sample Database

An independent verification of the assay data in the drill sample database used for the current MRE was conducted. Approximately 30% of the digital assay records were randomly selected and checked against the available laboratory assay certificate reports. Assay certificates were available for all diamond drilling completed by Arizona Metals. The assay database was reviewed for errors, including overlaps and gapping in intervals, and typographical errors in assay values. In general, the database was in good condition. A limited number of minor errors were noted and corrected during the validation.

Verifications were also carried out on drill hole locations, down hole surveys, lithology, SG and topography information. The database is considered of sufficient quality to be used for the current MRE.

The sample preparation, analyses, and security (see Section 11) completed by Arizona Metals for the Property were reviewed. Based on a review of all possible information, the sample preparation, analyses, and security used on the Project by Arizona Metals, including QA/QC procedures, are consistent with standard industry practices, and the drill data can be used for geological and resource modelling, and resource estimation of Indicated and Inferred mineral resources.

12.3 Site Visit – Allan Armitage

Armitage personally inspected the Property on October 25-26, 2023, and April 7-8, 2024, accompanied on both site inspections by Chris Steuer, Project Manager for Arizona Metals. During the site visit, Armitage inspected the core logging and core sampling facilities and core storage areas near Phoenix. Armitage examined a number of selected mineralized core intervals from recently completed diamond drillholes from the Property. Armitage examined accompanying drill logs and assay certificates, and the assays were examined against the drill core mineralized zones. Armitage inspected and reviewed current core sampling, QA/QC, and core security procedures.

- As drilling and core logging were in progress during the time of the site inspections, Armitage had the opportunity to review and discuss the entire path of the drill core, from the drill rig to the logging and sampling facility and finally to the laboratory. Armitage is of the opinion that the current protocols in place, as have been described and documented by Arizona Metals, are adequate.
- The Author participated in multiple field tours of the Property area, including visits to several outcrops to review the local geology, the drill, recent drill sites, and areas of historic shafts.
- As a result of the site inspections, Armitage was able to become familiar with conditions on the Property, was able to review and gain an understanding of the geology and various styles of mineralization, was able to verify the work done, and, on that basis, can review and recommend to Arizona Metals an appropriate exploration program.

12.4 Site Visit – Ben Eggers

Eggers conducted a site visit to the Project on May 30, 2025, accompanied by Chris Steuer, Project Manager and Ben Somps, Senior Exploration Geologist for Arizona Metals. The site visit consisted of a field tour of the Property and inspection of the core logging and sampling facilities and core storage areas at the Project.

The field tour of the Property area included visits to several outcrops to review the local geology and recent drill sites. All areas were easily accessible by road, and the bedrock geology is well exposed on the Property. Validation checks of drillhole collar locations were completed from a selection of five (5) drill pads used to target mineralization on the Property. Recent collars were observed on several drill pads; however, ongoing reclamation requirements and the repeated use of drill pads for successive drillholes mean that permanent retention of drillhole collar monuments is not possible. Collar locations were validated with the use of a handheld GPS.

During the site visit, selected mineralized core intervals were examined from seven diamond drillholes intersecting Kay mineralization in both the South and North (Kay2) lenses at a range of depths and spanning Arizona Metals' drilling programs completed in 2021, 2022, and 2024. The accompanying drill logs, long sections, and assays were examined against the drill core mineralized zones. Current core sampling, QA/QC and core security procedures were reviewed. Core boxes for drillholes reviewed are properly stored, easily accessible and well labelled. Sample tags are present in the boxes, and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones.

The site visit to the Kay core logging, sampling, and storage facilities included the inspection of the areas used for the geologists to log and photograph core, the areas for cutting and sampling core, the core storage areas, and the office area. Drilling was in progress during the time of the site visit, and an inspection of the active drill was completed. The entire path of the drill core, from the drill rig to the logging and sampling facility and finally to the laboratory, was reviewed and discussed. The QP is of the opinion that the current protocols in place, as have been described and documented by the Company, are adequate.

As a result of the site visit, the QP was able to become familiar with conditions on the Property, was able to observe and gain an understanding of the geology and various styles of mineralization, was able to verify the work done, and, on that basis, can review and recommend to the Company an appropriate exploration program.

The site visit completed in May 2025 is considered current, per Section 6.2 of NI 43-101CP. To the Authors' knowledge, there is no new material, scientific or technical information about the Property since that personal inspection. The Technical Report contains all material information about the Property.

12.5 Conclusion

All geological data has been reviewed and verified as being accurate to the extent possible, and to the extent possible, all geologic information was reviewed and confirmed. There were no significant or material errors or issues identified with the drill database. Based on a review of all possible information, Armitage is of the opinion that the database is of sufficient quality to be used for the current Indicated and Inferred MRE.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A metallurgical test work program was completed in July 2025 at SGS in Burnaby, British Columbia. The work was performed in continuation of the Early-Stage Metallurgical Evaluation (ESME) program completed on April 17, 2023. The objective of the test work program was to assess and optimize the flotation recovery and upgrading of the valuable constituents on a proportional (master) composite sample representative of the Kay Mine deposit composition. The scope of the test work included mineralogical studies, open batch and closed-circuit flotation, cyanidation, diagnostic leaching, and Albion Oxidative Treatment on samples from the Kay Mine Project. The relevant metallurgical data and findings from the 2023-2025 program are summarized in this section.

13.2 Previous Metallurgical Test Work

Metallurgical testwork, including Bond ball mill work index, Knelson gravity concentration and rougher flotation on gravity tails, was performed as part of an Early-Stage Metallurgical Evaluation of four (4) composites from the Kay Mine in Arizona, USA. The primary objective of the program was to provide a preliminary understanding of the deposit through grindability, gravity separation, and flotation. These early-stage results provide a snapshot of the metallurgical potential of the deposit.

Composites were blended using assay rejects from ALS Tucson and contained chalcopyrite, sphalerite, galena, pyrite, and arsenopyrite, with non-sulfide gangue comprised of quartz, chlorites, ankerite, iron oxides, and dolomite in varied proportions.

Grindability tests showed the material is moderately soft with respect to the SGS database, with bond work ranging from 9.5-12.6 kWh/t. Mineralogical characterization determined that all sulfides exhibited very good exposure and liberation properties. Gravity concentration recovered up to ~21% of the gold at ~1% mass pull, producing a concentrate grading ~327 g/t Au. Given the ultrafine gold particle size (~2 µm mean; 100% <6 µm), gravity is expected to make only a minor contribution to overall gold recovery. Any further gravity cleaning would likely trade recovery for higher concentrate grade and is not expected to materially improve overall recovery. Exploratory rougher kinetic flotation tests on the gravity tailings produced good separation between Cu/Pb and Zn, owing to the good liberation of the sulfide minerals. Overall recoveries, including both gravity and rougher flotation concentrates, yielded recoveries of 62-98% Cu, 66-92% Pb, 79-98% Zn, 64-98% Au, and 83-99% Ag.

13.2.1 Bond Work Index Tests

Bond Ball Mill Work Index (BWi) tests were conducted on four (4) representative composite samples to assess the comminution characteristics of the mineralized material. The results are summarized in Table 13.1. The measured Work Index values range from 9.5 to 12.6 kWh/t, classifying the tested material from “very soft” to “moderately soft” relative to global BWi datasets.

Table 13.1: Bond Ball Mill Grindability Test Results Summary

Sample ID	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kW h/t)	Hardness Percentile	Category
K-Met-01	100	2,114	118	1.98	12.6	31	Moderately Soft
K-Met-02	100	2,127	114	2.12	11.6	21	Soft
K-Met-03	100	2,185	109	2.33	10.4	13	Soft
K-Met-04	100	2,108	110	2.65	9.5	8	Very Soft

No additional grinding testwork was conducted. For design purposes, a Bond Ball Mill Work Index of 12.2 kWh/t, corresponding to the 85th percentile of the previously reported sample results, was adopted for sizing the primary and secondary grinding circuits.

No regrind testwork has been carried out for Cu, Zn, or pyrite concentrates. Instead, signature plot data from comparable projects were applied, together with the specified concentrate mass pull and target grind requirements (F_{80}, P_{80}), to estimate the specific energy demand for each regrind stage. Based on this approach, the predicted specific energy requirements are 43.5 kWh/t for the Cu/Pb regrind mill and 70.5 kWh/t for the Zn regrind mill.

13.2.2 Gravity

Gravity separation tests were conducted to evaluate the amenability of the mineralized material to concentration using a Knelson concentrator. The results are summarized in Table 13.2. Recoveries were below 17% for composites K-MET-01, K-MET-02, and K-MET-03, and approximately 22% for K-MET-04, the highest Au-grade sample. These recoveries are considered low and are attributed to the extremely fine grain size of gold, with a mean particle size of ~2 µm and 100% passing 6 µm. Knelson concentrators are generally effective for free gold above ~20 µm, and therefore, limited recovery is expected under the observed conditions.

Table 13.2: Gravity Test Results Summary

Composite-Test	Target K80	Actual K80	Product	Weight	Au Assays							Au Dist.						
				%	Au	Ag	Cu	Pb	Fe	Zn	S	Au	Ag	Cu	Pb	Fe	Zn	S
	µm	µm			g/t	g/t	%	%	%	%	%	%	%	%	%	%	%	%
K-MET-01-G1	75	60	Knelson Con	1.07	12.5	73.2	0.47	6.93	9.08	31.1	33.1	16.64	2.95	0.11	75.74	38.72	2.16	3.19
			Knelson Tails	98.9	0.68	26.1	4.79	0.02	0.16	15.3	10.9	83.36	97.05	99.89	24.26	61.28	97.84	96.8
			Direct Head		0.82	32.2	4.51	0.08	0.18	15.2	11.6							
			Calculated Head	100	0.81	26.6	4.74	0.10	0.25	15.5	11.1	100.0	100.0	100.0	100.0	100.0	100.0	100.0
			Acc. (%)		*98%*	*83%*	*105%*	*124%*	*140%*	*102%*	*96%*							
K-MET-02-G1	75	62	Knelson Con	1.03	28.8	84.6	0.27	2.75	6.14	33.9	33.2	8.38	1.80	1.05	3.85	0.87	1.78	1.86
			Knelson Tails	99.0	3.26	47.90	0.27	0.71	7.26	19.4	18.1	91.62	98.20	98.95	96.15	99.13	98.22	98.1
			Direct Head		3.64	53.8	0.25	0.73	7.3	19.3	18.5							
			Calculated Head	100	3.52	48.3	0.27	0.73	7.2	19.5	18.3	100.0	100.0	100.0	100.0	100.0	100.0	100.0
			Acc. (%)		*97%*	*90%*	*106%*	*101%*	*99%*	*101%*	*99%*							
K-MET-03-G1	75	56	Knelson Con	1.04	6.4	104.0	1.11	0.63	4.21	35.4	40.1	2.64	2.08	1.08	2.12	0.88	1.88	2.13
			Knelson Tails	99.0	2.46	51.10	1.06	0.31	4.96	19.3	19.3	97.36	97.92	98.92	97.88	99.12	98.12	97.9
			Direct Head		1.78	58.8	1.12	0.31	5.0	19.8	19.8							
			Calculated Head	100	2.50	51.6	1.06	0.31	5.0	19.5	19.5	100.0	100.0	100.0	100.0	100.0	100.0	100.0
			Acc. (%)		*140%*	*88%*	*95%*	*99%*	*100%*	*98%*	*99%*							
K-MET-04-G1	75	58	Knelson Con	1.07	327.0	294.0	5.28	0.06	0.17	33.3	31.4	21.47	3.04	8.12	0.04	0.01	1.57	1.35
			Knelson Tails	98.9	12.90	101.0	0.64	1.92	13.60	22.5	24.8	78.53	96.96	91.88	99.96	99.99	98.43	98.7
			Direct Head		14.70	101.0	0.63	1.89	13.2	21.7	24.9							
			Calculated Head	100	16.25	103.1	0.69	1.90	13.5	22.6	24.9	100.0	100.0	100.0	100.0	100.0	100.0	100.0
			Acc. (%)		*111%*	*102%*	*110%*	*101%*	*102%*	*104%*	*100%*							

13.3 Sample Selection and Preparation

A total of 5,431 assays from holes KM-23-99 were categorized into three (3) metal clusters (Cu, Zn-Pb, and Zn-Pb-Cu). From these assays, a total of 3,201 assays were selected for a specific metal cluster, and the resulting percentages (shown in Table 13.3) were used to create the master composite for the test program.

Table 13.3: Master Composite Blend Recipe

Composite	Metal Cluster	N(Assays)	Proportion
K-MET-01	Cu	863	27.0%
K-MET-02	Zn-Pb	1,162	36.3%
K-MET-03	Zn-Pb-Cu	1,176	36.7%
K-MET-04	High Au	-	-
Total		**3,201**	**100%**

An initial master composite (MC-1A) was produced using the three (3) composites K-MET-01 to 03 blend proportions shown in Table 13.3. The material used was 10 mesh and homogenized, divided into 2 kg bags, and then retrieved from cold storage to create the master composite blend. This master composite blend was homogenized and split into 2 kg test charges using a rotary splitter. The initial master composite 1A blend was used until the flotation test MC-F15, when stock was exhausted. A second master composite (MC-1B) blend was then created using the same blend proportions with the remaining K-MET-01 to 03 material from cold storage. The MC-1B blend was used for all subsequent testwork.

After blending and homogenization steps, each master composite was submitted for head analysis. The MC-1A was also submitted for mineralogy, targeting sulfide, visible gold deportment, and mercury analyses. The head assays of the MC-1A and MC-1B blends were comparable, suggesting reasonable continuity between the composite blends. The calculated silver head grades from flotation testing fall between the average of the direct silver assay measurements.

A summary of head assays, measured by direct chemical analysis and calculated from flotation testwork are summarized in Table 13.4.

Table 13.4: Summary of Master Composite Head Assays

	Method	Fusion with ICP or XRF								LECO		Fire Assay (Gravimetric)		3-Acid Digest / ICP
	Element unit	Cu %	Pb %	Zn %	Fe %	Al %	Ca %	Mg %	Si %	S(t) %	C(t) %	Au g/t	Ag g/t	As %
MC-1A	Assay	1.71	0.38	4.26	17.9	2.95	3.71	3.71	10.7	17.0	3.02	1.95	39.7	1.71
	Calculated	1.85	0.38	4.47	18.9	-	-	-	-	17.0	-	2.17	43.5	1.66
MC-1B	Assay	1.70	0.42	4.19	18.4	2.87	3.60	3.65	10.9	17.2	3.02	2.24	48.5	1.92
	Calculated	1.85	0.40	4.57	18.7	-	-	-	-	16.9	-	2.10	45.5	1.67

13.4 Mineralogy

Master composite 1A was stage crushed to a P_{80} of 106 µm and submitted for TIMA-X using the Particle Mineral Analysis (PMA) mode of measurement. For modal and sulfide mineralogical analyses, the sample was sized in three (3) size fractions: +75 µm, +25 µm, and -25 µm to mitigate stereological biases. For the visible gold deportment study, the sample was subjected to heavy liquid separation (HLS) at a specific gravity (SG) of 2.9, and the HLS sinks were further upgraded with a super-panner. All the products generated were then individually studied.

13.4.1 Modal Mineralogy

The modal mineralogy, as summarized in Table 13.5, identified the sulfide gangue minerals as pyrite (23.5%) and arsenopyrite (3.9%). Non-sulfide minerals consisted primarily of silicates such as quartz (18.3%) and plagioclase (1.1%), carbonates such as dolomite / ankerite (17.3%) and siderite (5%), and phyllosilicates such as chlorites (13%), micas (2.6%), and clays (1.1%).

Table 13.5: Master Composite 1A, TIMA Mineralogy Modals

Survey		CA20M-00000-211-18426-01 / MI7008-JUN23						
Project		SGS Met Ops (Kay Mine)						
Sample		K-Met - Master Comp						
Fraction		Combined	+75 µm		+25 µm		-25 µm	
Mass % of Size Fraction [%]		100.0	32.1		31.0		36.9	
Median Particle Size (µm)		13	66		20		5	
		Sample	Sample	Fraction	Sample	Fraction	Sample	Fraction
Mineral Mass (%)	Pyrite	23.5	10.1	31.4	7.67	24.8	5.73	15.5
	Sphalerite	6.78	1.62	5.04	2.43	7.85	2.73	7.40
	Chalcopyrite	4.90	1.15	3.58	2.05	6.61	1.70	4.62
	Arsenopyrite	3.93	1.66	5.17	1.19	3.86	1.07	2.91
	Galena	0.42	0.13	0.41	0.12	0.40	0.16	0.44
	Tetrahedrite-Tennantite	0.24	0.04	0.13	0.09	0.31	0.10	0.27
	Other Sulphides	0.05	0.02	0.05	0.01	0.03	0.02	0.05
	Quartz	18.3	7.57	23.6	4.68	15.1	6.06	16.4
	Plagioclase	1.09	0.15	0.46	0.22	0.71	0.73	1.97
	K-Feldspar	0.01	0.01	0.02	0.00	0.01	0.00	0.01
	Amphibole/Pyroxene	0.12	0.07	0.21	0.03	0.10	0.02	0.05
	Micas/Illite	2.61	0.54	1.69	0.64	2.06	1.43	3.86
	Chlorites	12.8	2.19	6.82	3.34	10.8	7.22	19.6
	Clays	1.14	0.19	0.59	0.26	0.85	0.69	1.86
	Other Silicates	0.31	0.08	0.25	0.10	0.33	0.13	0.34
	Fe-Oxide/Hydroxide	0.57	0.17	0.54	0.15	0.48	0.24	0.66
	Rutile	0.17	0.03	0.10	0.04	0.13	0.10	0.26
	Other Oxides	0.66	0.04	0.12	0.06	0.19	0.57	1.53
	Calcite	0.01	0.00	0.01	0.00	0.01	0.01	0.02
	Dolomite/Ankerite	17.3	5.09	15.8	6.05	19.6	6.12	16.6
	Siderite	5.00	1.24	3.87	1.76	5.69	1.99	5.40
	Other Carbonates	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Sulphates/phosphates	0.11	0.02	0.05	0.02	0.07	0.07	0.19
	Other	0.06	0.03	0.08	0.03	0.08	0.01	0.03
	Total	100.0	32.1	100.0	31.0	100.0	36.9	100.0

13.4.2 Sulfide Mineralogy

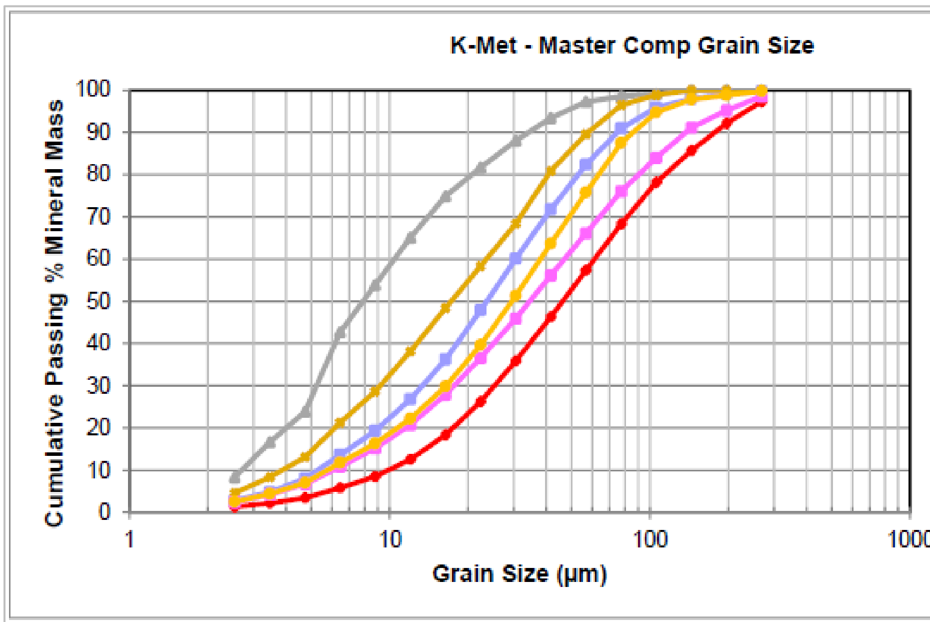
The key sulfide mineral deportment and grain size results, as presented in Table 13.6 and Figure 13.1, are summarized as follows:

- Pyrite accounted for 72.1% of the sulfur content, followed by 12.8% as sphalerite, 9.9% as chalcopyrite, and 4.5% in arsenopyrite.
- The primary copper-bearing mineral was chalcopyrite (94.2%), with 4.6% total copper deportment in tetrahedrite-tennantite minerals.
- Arsenopyrite accounted for most of the arsenic content at 98.7%.
- The presence of zinc and lead was entirely in the sphalerite and galena grains, respectively.
- The measured sulfide mineral P_{80} grain sizes ranged from 20 to 144 microns, with pyrite and galena at the coarsest and finest particle sizes, respectively.

Table 13.6: Master Composite 1A, Key Mineral Deportment (normalized)

	Mineral Name	Combined	+75 µm	+25 µm	-25 µm
Sulphur Deportment	Pyrite	72.1	80.6	69.5	63.4
	Sphalerite	12.8	7.99	13.6	18.5
	Chalcopyrite	9.86	6.02	12.2	12.4
	Arsenopyrite	4.47	4.91	4.01	4.42
	Galena	0.32	0.26	0.28	0.45
	Tetrahedrite-Tennantite	0.35	0.16	0.41	0.53
	Other Sulphides	0.09	0.07	0.06	0.14
	Other	0.05	0.02	0.02	0.14
	Total	100.0	100.0	100.0	100.0
Copper Deportment	Chalcopyrite	94.2	94.9	94.9	93.0
	Tetrahedrite-Tennantite	4.65	3.59	4.47	5.55
	Other Sulphides	0.89	1.29	0.54	1.02
	Other Oxides	0.24	0.20	0.11	0.41
	Total	100.0	100.0	100.0	100.0
Arsenic Deportment	Pyrite	0.73	0.99	0.67	0.41
	Arsenopyrite	98.7	98.5	98.8	98.9
	Tetrahedrite-Tennantite	0.14	0.06	0.21	0.22
	Other Sulphides	0.41	0.43	0.30	0.49
	Total	100.0	100.0	100.0	100.0
Pb & Zn Dep.	Sphalerite	100.0	100.0	100.0	100.0
	Galena	100.0	100.0	100.0	100.0

Figure 13.1: Sulfide Mineralogy Grain Sizes



Mineral	P_{80}, µm
Pyrite	114
Arsenopyrite	90
Chalcopyrite	63
Sphalerite	53
Tetrahedrite	40
Galena	20

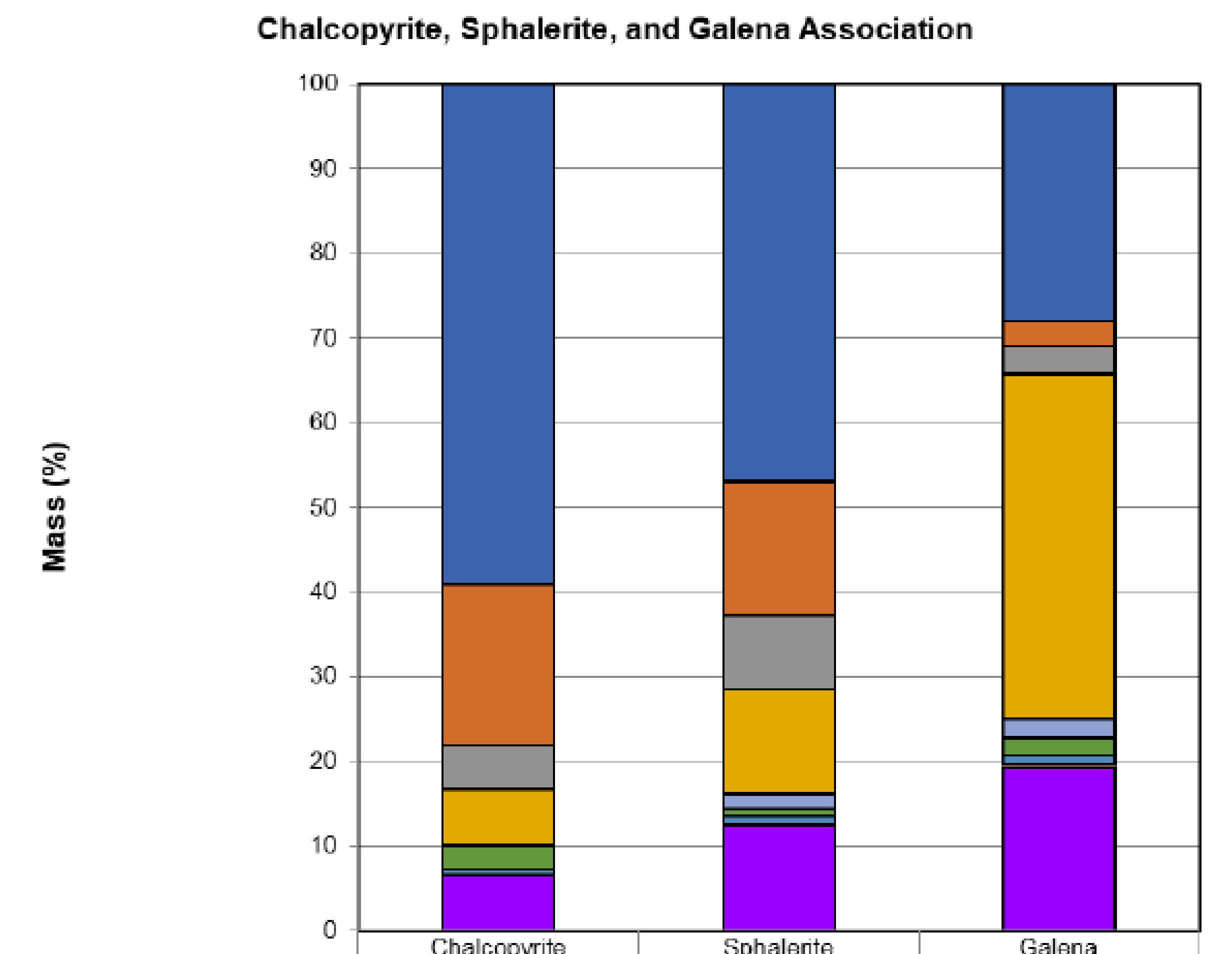
Table 13.7: Master Composite 1A, Key Mineral Liberation and Exposure

	Mineral Name	Combined	+75 µm	+25 µm	-25 µm
Chalcopyrite Exposure	Exposed	86.6	62.3	90.7	97.9
	50-80%	6.45	13.9	6.23	1.70
	30-50%	2.60	8.35	1.42	0.14
	10-30%	1.96	7.09	0.64	0.07
	0-10%	0.20	0.62	0.10	0.03
	Locked	2.23	7.75	0.88	0.14
	Total	100.0	100.0	100.0	100.0
Sphalerite Exposure	Exposed	74.1	31.0	77.5	96.6
	50-80%	12.9	25.5	15.7	2.93
	30-50%	5.54	16.8	3.98	0.25
	10-30%	3.60	12.63	1.54	0.09
	0-10%	0.73	2.83	0.16	0.00
	Locked	3.11	11.2	1.07	0.11
	Total	100.0	100.0	100.0	100.0
Galena Exposure	Exposed	50.3	15.0	40.2	86.9
	50-80%	15.0	14.0	24.7	8.39
	30-50%	6.21	7.98	10.8	1.24
	10-30%	4.70	9.43	4.48	1.00
	0-10%	0.77	1.37	1.06	0.06
	Locked	23.0	52.2	18.7	2.42
	Total	100.0	100.0	100.0	100.0
Arsenopyrite Exposure	Exposed	52.4	25.1	56.1	90.8
	50-80%	22.3	28.4	26.9	7.65
	30-50%	10.8	17.9	9.63	0.90
	10-30%	8.64	17.1	4.38	0.33
	0-10%	1.18	2.19	0.83	0.01
	Locked	4.70	9.31	2.23	0.28
	Total	100.0	100.0	100.0	100.0
Pyrite Exposure	Exposed	70.9	50.5	79.3	95.9
	50-80%	17.2	26.1	15.7	3.48
	30-50%	5.9	11.2	3.09	0.33
	10-30%	3.80	8.01	1.03	0.09
	0-10%	0.63	1.40	0.09	0.01
	Locked	1.51	2.80	0.79	0.18
	Total	100.0	100.0	100.0	100.0

Table 13.7 shows that (at a feed P_{80} of 106 µm), the chalcopyrite, galena, and sphalerite minerals displayed good liberation and exposure characteristics as the combined liberation ranged from 50 to 87% with greater than 50% exposure across the combined size fractions. This suggests potential for successful rougher flotation of the copper, lead and zinc minerals. As expected, a higher degree of liberation and exposure was observed in the sub-25 µm particles with a liberation range of 86.9 to 97.9% at above 50% exposure. Arsenopyrite and pyrite displayed similar characteristics to the key sulfide minerals, indicating the potential

to facilitate their rejection from the copper-lead and zinc concentrates through the deployment of appropriate regrinding and reagent schemes.

Figure 13.2: Key Sulfide Associations



	Chalcopyrite	Sphalerite	Galena
Pure	59.2	46.9	28.0
Free	19.0	15.8	2.99
Liberated	5.24	8.87	3.32
Binary : Other Sulphides	6.54	12.3	40.7
Binary : Carbonates	0.00	1.74	2.14
Binary : Mica/Chlor/Clays	2.78	0.81	2.14
Binary : Quartz/Feld	0.75	1.01	1.06
Binary : Oxides	0.00	0.11	0.29
Binary : Other Minerals	0.00	0.01	0.02
Complex	6.53	12.4	19.3

The combined percentage of pure, free, and liberated chalcopyrite, sphalerite, and galena was 83.4%, 71.6% and 34.3%, respectively, while complex associations were 6.5%, 12.4% and 19.3% for the same suite of minerals as shown in Figure 13.2. With 40.7% of the galena in binary form with other sulfides, regrinding of the copper lead concentrate would be necessary to facilitate selective upgrading of the lead content. Sphalerite, to a lesser degree than galena, will also benefit from regrinding prior to the respective cleaner circuits. In general, the liberation and association data show similar trends towards potential recovery and upgradeability of the copper, lead and zinc minerals.

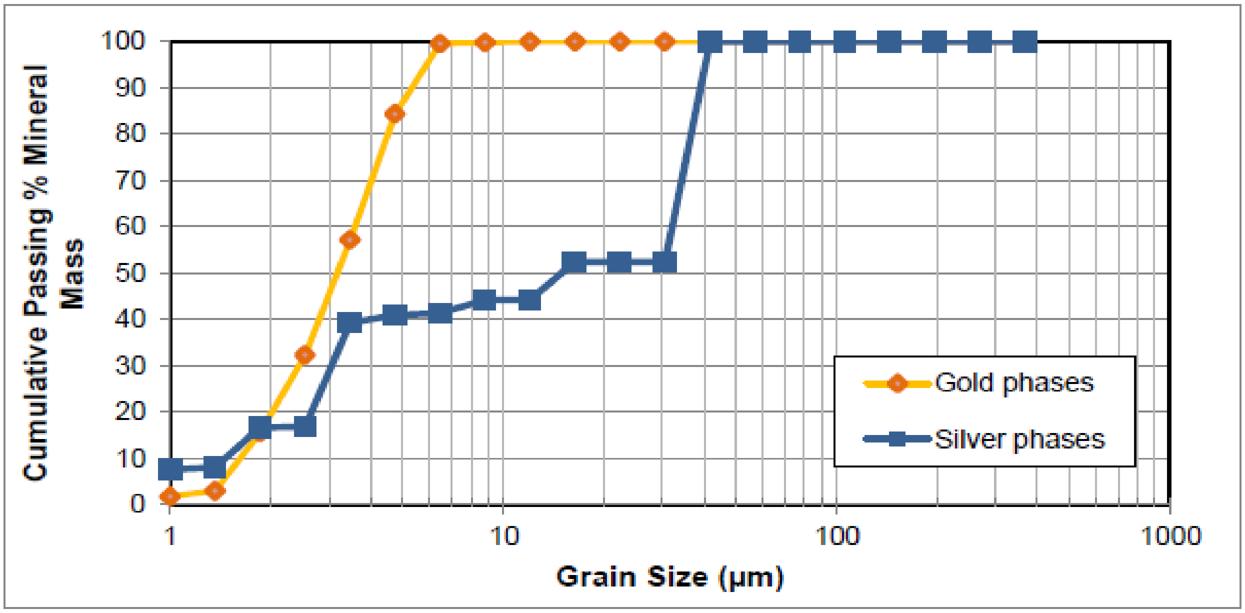
13.4.3 Gold Mineralogy

The visible gold deportment (grains > 0.5 µm) within the master composite, presented in Table 13.8, show 43.5% as native and 48.4% as electrum, with minor amounts of gold-tellurides in petzite (3.8%) and calaverite (3.3%). In total, 130 grains were observed during scanning. The gold grain size was ultrafine with a mean of two (2) microns and 100% passing six (6) microns (Figure 13.3).

Table 13.8: Master Composite Visible (> 0.5 µm) Gold Deportment

Mineral Name	Combined	+75 µm Sink	-75 µm Sink	HLS Float
Gold	43.5	21.5	46.6	0.00
Electrum	48.4	70.4	45.4	0.00
Petzite	3.77	3.40	3.83	0.00
Calaverite	3.32	4.10	3.22	0.00
Silver Phases	0.91	0.59	0.96	0.00
Total	**100.0**	**100.0**	**100.0**	**0.00**

Figure 13.3: Master Composite Gold and Silver Grain Size



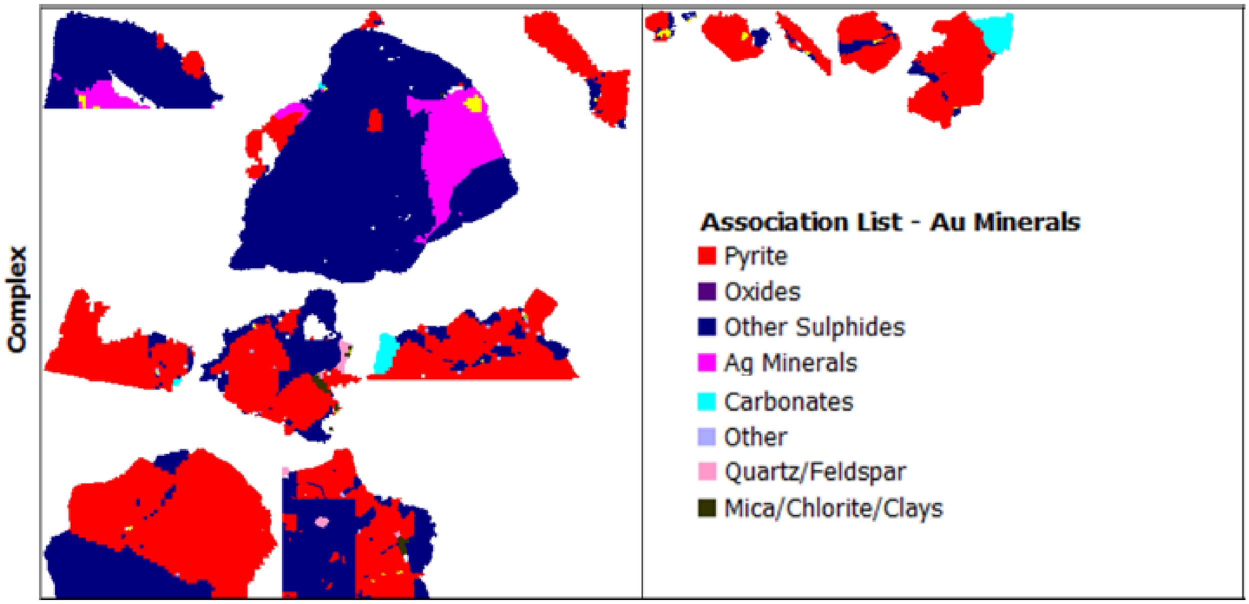
The normalized gold associations illustrated in Table 13.9 show that 20% of the observed grains were pure or liberated particles, while 7.6% were associated with carbonates. Sulfide associations constituted at least 46.4% of the grains, comprising pyrite and other sulfides, primarily arsenopyrite.

There is potential for an extra ~25% when accounting for associations labelled as "Complex," which, according to images in Figure 13.4, predominantly indicate pyrite / arsenopyrite associations. Within the sulfides, the gold was observed to occur as inclusions and along fractures, which indicates that ultrafine grinding and/or oxidation processes would be necessary to expose mineral surfaces for leach extraction.

Table 13.9: Master Composite Gold Associations (Normalized)

Mineral Name	Combined	>90 µm	>80 µm	>70 µm	>60 µm	>50 µm	>40 µm	>30 µm	>20 µm	>10 µm	>5 µm	<5 µm
Pure Au Minerals	5.38	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	83.0
Free Au Minerals	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Lib Au Minerals	14.6	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	65.7	0.00
Au Minerals : Ag Minerals	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Minerals : Pyrite	13.0	12.4	43.4	0.00	42.6	0.00	47.1	67.0	0.00	0.00	0.00	0.00
Au Minerals : Other Sulphides	33.4	26.5	29.5	0.00	57.4	4.91	0.00	22.6	77.8	0.18	0.22	0.00
Au Minerals : Quartz/Feld	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Minerals : Mica/Chlor/Clays	0.20	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.41	0.00	0.00
Au Minerals : Carbonates	7.59	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	34.1	0.00
Au Minerals : Oxides	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Minerals : Other Minerals	0.04	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.67
Complex	25.8	61.1	27.0	0.00	0.00	95.1	52.9	10.4	22.2	98.4	0.00	16.3
Total	100.0	100.0	100.0	0.00	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Figure 13.4: Complex-Gold Association Grain Image

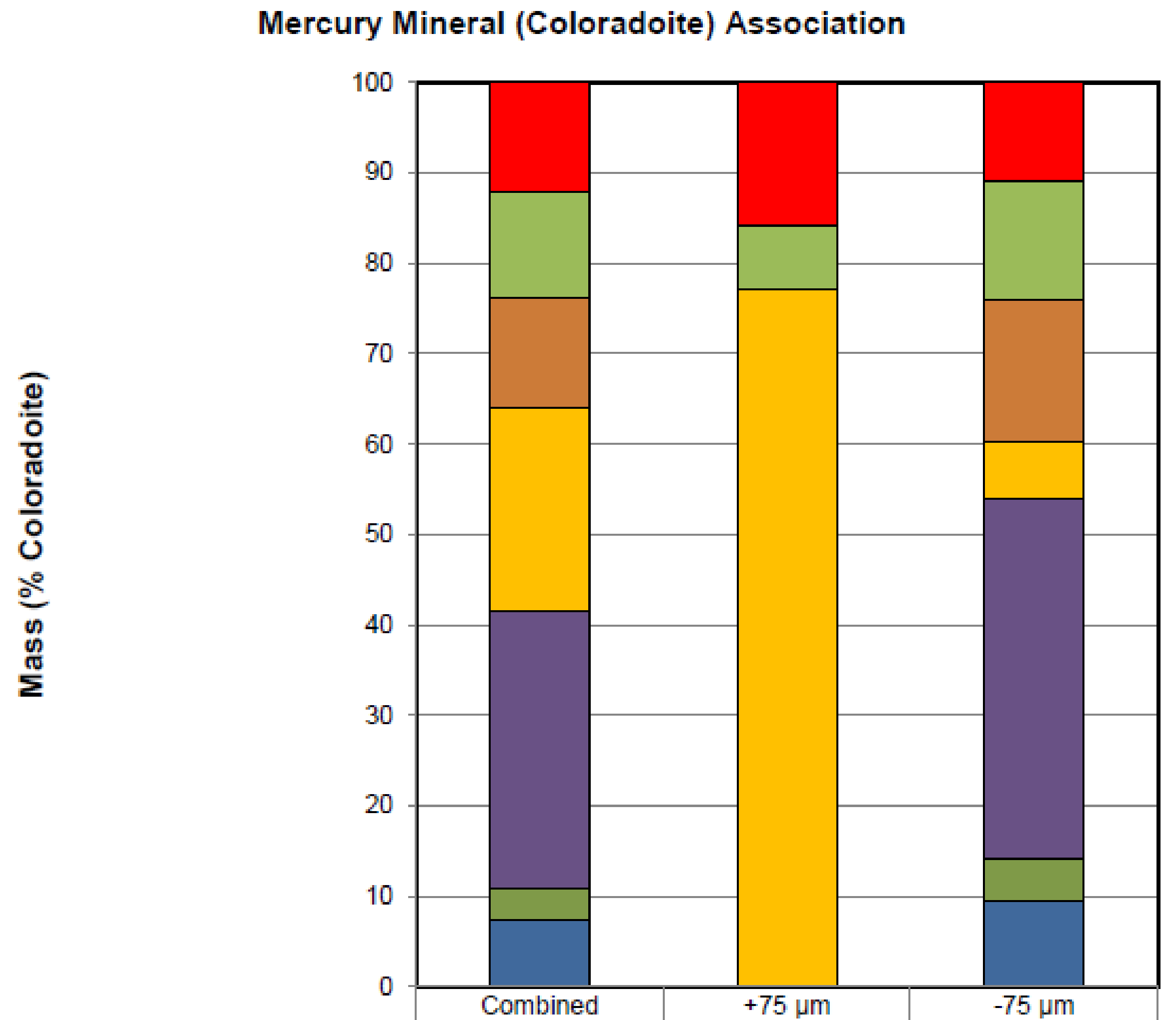


13.4.4 Mercury Mineralogy

A high concentration of mercury was assayed in both the copper-lead and the zinc cleaner concentrates produced from open batch (F26-27) and locked cycle flotation (LCT1) tests, grading 246-256 g/t Hg on the zinc concentrate. As a result, targeted re-examination of the gold deportment study data was performed to assess the mineralogical characteristics of the mercury content in the HLS sink products.

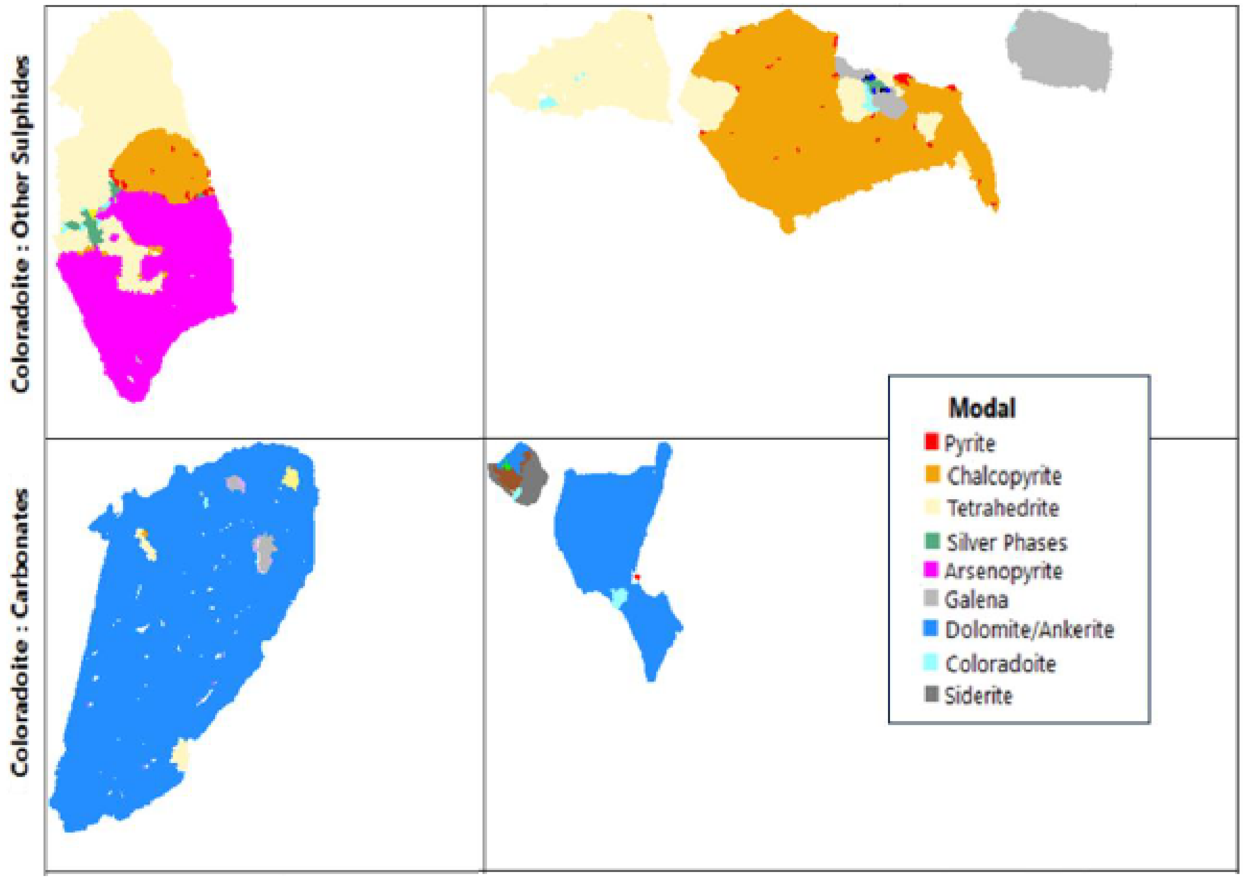
In total, 50 mercury grains were observed through scanning, and the mercury content was entirely identified within a HgTe mineral called coloradoite. The coloradoite had a measured P_{80} grain size of 14 µm, and the mineral association data, as summarized in Figure 13.5, show that 10.9% of the coloradoite was classified as pure, free, and liberated, while 30.6% was associated with tellurium phases and 22.6% with pyrite. The remaining coloradoite was present as binary inclusions (Figure 13.6), within other sulfide minerals such as chalcopyrite, galena, tetrahedrite, and arsenopyrite (12.1%), and carbonate minerals such as dolomite / ankerite and siderite (11.7%).

Figure 13.5: Coloradoite Association for Master Composite 1A



	Combined	+75 µm	-75 µm
Complex	12.1	15.8	11.0
Coloradoite : Other Minerals	0.00	0.00	0.00
Coloradoite : Oxides	0.00	0.00	0.00
Coloradoite : Carbonates	11.7	7.02	13.1
Coloradoite : Mica/Chlor/Clays	0.00	0.00	0.00
Coloradoite : Quartz/Feld	0.00	0.00	0.00
Coloradoite : Other Sulphides	12.1	0.00	15.7
Coloradoite : Pyrite	22.6	77.2	6.28
Coloradoite : Tellurium Phases	30.6	0.00	39.8
Lib Coloradoite	3.63	0.00	4.71
Free Coloradoite	0.00	0.00	0.00
Pure Coloradoite	7.26	0.00	9.42

Figure 13.6: Other Sulfides and Carbonate Mineral Grains with Coloradoite



Mineralogical analysis of LCT1 zinc cleaner concentrate reaffirmed the findings from the feed material, as the coloradoite had a mean of four (4) microns, and none were detected to be associated with the sphalerite.

13.5 Flotation

The main objective of flotation was to assess the separability of copper-lead and zinc into separate flotation concentrates, whilst minimizing arsenic within each concentrate. This was approached by testing various arsenic rejection techniques, such as pulp pH control and selective oxidation of arsenopyrite, while utilizing selective copper collectors such as Aero 5100 and Aero 3894. Specific targets were set for primary grinding and rougher concentrate regrinding to enhance liberation, as indicated by the mineralogical analysis. A summary of test objectives is presented in Table 13.10.

Table 13.10: Flotation Test Objective Summary

ID	Type	Objective
MC-F1	Rougher Kinetic	Baseline Rougher
MC-F2	Rougher Kinetic	F1 with reduced depressent
MC-F3	Rougher Kinetic	F2 with Aero 3894 collector in Cu/Pb circuit
MC-F4	Rougher Kinetic	F2 with coarser primary grind
MC-F5	Cleaner Kinetic	Baseline Cleaner
MC-F6	Cleaner Kinetic	Aero 3894 Collector with Sodium humate (NaHA) & higher pH in Zn cleaner
MC-F7	Cleaner Kinetic	Aero 3894 Collector with H2O2 and pH 11 in Zn cleaner
MC-F8	Sequential Cleaning	Cleaner flotation with F6 conditions
MC-F9	Sequential Cleaning	F8 with higher Aero 3894 dosage
MC-F10	Rougher Kinetic	F3 with higher Aero 3894 dosage and additional conditioning time
MC-F11	Rougher Kinetic	Aero 5100 Collector in Cu/Pb circuit
MC-F12	Rougher Kinetic	F11 but with addition of Aero 208
MC-F13	Rougher Kinetic	F2 with Aero 7261 Depressant
MC-F14	Sequential Cleaning	Cleaner flotation with F11 conditions and NaHA in Zn cleaners
MC-F15	Sequential Cleaning	F14 with changed NaHA addition points
MC-F16	Sequential Cleaning	Re-Baseline (New MC)
MC-F17	Rougher Kinetic	F11 but with shorter Cu/Pb circuit rougher times
MC-F18	Rougher Kinetic	F17 with higher Cu/Pb and Zn collectors dosage
MC-F19	Rougher Kinetic	F18 with higher pH and additional Zn kinetic sample
MC-F20	Rougher Kinetic	F18 with Aero 7261 in Cu/Pb circuit and add Pyrite circuit
MC-F21	Rougher Kinetic	F18 with higher Zn pH and shorten Zn kinetics
MC-F22	Rougher Kinetic	F21 with higher CN/ZnSO4, Aero 5100 in Zn circuit, and add Pyrite circuit
MC-F23	Rougher Kinetic	F22 with shorter times & reduced collector in Cu/Pb and Aero 7279 in Zn circuit
MC-F24	Rougher Kinetic	F23 with new frother and increased CuSO4 dosage
MC-F25	Rougher Kinetic	F22 with higher Aero 5100 & pH 9 in Cu/Pb circuit, and Aerofroth 76A frother
MC-F26	Sequential Cleaning	F18 Cu/Pb rougher and F22 Zn rougher conditions, cleaning with Aero 5100
MC-F27	Sequential Cleaning	F26 with scavengers and increase of pH & collectors in Zn cleaner

13.5.1 <u>Test Conditions</u>

Flotation was performed on 2 kg charges of the master composite blend. The first batch of master composite (MC-1A) was used for flotation tests MC-F1 through MC-F15, while the second batch of master composite (MC-1B) was used in test MC-F16 onwards. All flotation products were assayed for Au, Ag, Cu, Pb, Zn, As, Fe and S.

Testing was performed using a Denver (D-12) flotation machine with pressurized air flow rates typically between 8 and 12 L/min and controlled through rotameter adjustment. Primary grinding was performed in a laboratory rod mill with stainless steel grinding rods. Pulp densities for roughers and cleaners were typically around 30% and 10%, respectively, with pH maintained through lime (CaO) addition. A summary of the reagents and general conditions employed for each test is displayed in Table 13.11 and Table 13.12. Reagents are shown along with their respective dosage strengths in grams per tonne of primary feed.

Table 13.11: Flotation Reagent Summary

Reagent	Reagent Type	Description	Addition Format
Lime	pH Modifier	Calcium Oxide	Powder
$CuSO_4$	Activator	Copper Sulphate	Powder
NaCN	Depressant	Sodium Cyanide	Powder
$ZnSO_4$	Depressant	Zinc Sulphate	Powder
NaHA	Depressant	Sodium Humate	Powder
Aero 7261A	Depressant	Water soluble polymer	0.2% solution (water)
Aerophine 3418A	Collector	Dialkyl dithiophosphinate	0.2% solution (water)
Aero 3894	Collector	IPETC, Dialkyl thionocarbamate	neat - dropwise
Aero 5100	Collector	Modified IPETC	neat - dropwise
Aero 7279	Collector	Formulated IPETC	neat - dropwise
Aerofloat 208	Collector	Short chain dialkyl dithiophosphate	neat - dropwise
SIPX	Collector	Sodium Isopropyl Xanthate	0.2% solution (water)
H_2O_2	Oxidant	Hydrogen Peroxide	50% solution (water)
MIBC	Frother	Methyl Isobutyl Carbinol	0.2% solution (water)
Aerofroth 76A	Frother	Ethylhexanol-based Alcohol	neat - dropwise

Table 13.12: Summary of Flotation Conditions

Test	Primary Grind				Cu/Pb Circuit									Zn Circuit											Pyrite Circuit			
					Rougher				Regrind		Cleaner			Rougher				Regrind			Cleaner							
ID	P80	pH	NaCN	$ZnSO_4$	Collector-g/t	Frother-g/t	Reagent 2	pH	P80	$ZnSO_4$	Collector-g/t	Frother-g/t	pH	$CuSO_4$	Collector-g/t	Frother-g/t	pH	P80	CuSO4	Reagent	Collector-g/t	Frother-g/t	Reagent 2	pH	H_2SO_4	Collector-g/t	Frother-g/t	pH
MC-F1	57	7	300	900	3418A-100	MIBC-30	-	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F2	54	7	100	300	3418A-100	MIBC-30	-	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F3	55	7	100	300	3894-50	MIBC-30	-	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F4	83	7	100	300	3418A-100	MIBC-30	-	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F5	80	7	100	300	3418A-100	MIBC-30	-	9.5	16	50	3418A-15	MIBC-5	10.5	500	SIPX-60	MIBC-20	10	22	200	-	SIPX-10	MIBC-5	-	11	-	-	-	-
MC-F6	61	7	100	300	3894-50	MIBC-30	-	9.5	16	50	3894-10	MIBC-5	10.5	500	SIPX-60	MIBC-25	10	20	200	NaHA-625	SIPX-10	MIBC-12	-	11.5	-	-	-	-
MC-F7	60	7	100	300	3894-50	MIBC-30	-	9.5	15	50	3894-10	MIBC-5	10.5	500	SIPX-60	MIBC-25	10	18	200	-	SIPX-10	MIBC-12	H2O2-2000	11	-	-	-	-
MC-F8	56	7	100	300	3894-50	MIBC-30	-	9.5	14	50	3894-15	MIBC-5	10.5	500	SIPX-60	MIBC-25	10	18	200	NaHA-625	SIPX-10	MIBC-12	-	11.5	-	-	-	-
MC-F9	55	7	100	300	3894-65	MIBC-30	-	9.5	16	50	3894-25	MIBC-5	10.5	500	SIPX-60	MIBC-25	10	25	200	NaHA-625	SIPX-10	MIBC-12	-	11.5	-	-	-	-
MC-F10	54	7	100	300	3894-65	MIBC-30	-	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F11	51	7	100	300	5100-65	MIBC-30	-	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F12	57	7	100	300	5100-65	MIBC-30	208-25	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F13	52	7	100	300	3418A-100	MIBC-30	7261-25	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F14	63	7	100	300	5100-65	MIBC-30	-	9.5	16	50	5100-25	MIBC-5	10.5	500	SIPX-60	MIBC-35	10	16	200	NaHA-625	SIPX-25	MIBC-12	-	11.5	-	-	-	-
MC-F15	60	7	100	300	5100-65	MIBC-30	-	9.5	17	50	5100-25	MIBC-10	10.5	500	SIPX-60	MIBC-35	10	16	200	NaHA-415	SIPX-10	MIBC-12	NaHA-210	11.5	-	-	-	-
MC-F16	62	7	100	300	5100-65	MIBC-30	-	9.5	17	50	5100-25	MIBC-10	10.5	500	SIPX-60	MIBC-35	10	16	200	NaHA-415	SIPX-10	MIBC-22	NaHA-210	11.5	-	-	-	-
MC-F17	59	7	100	300	5100-65	MIBC-30	-	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F18	60	7	100	300	5100-82	MIBC-30	-	9.5	-	-	-	-	-	500	SIPX-76	MIBC-20	10	-	-	-	-	-	-	-	-	-	-	-
MC-F19	56	7	100	300	5100-82	MIBC-30	-	11	-	-	-	-	-	500	SIPX-75	MIBC-20	11.5	-	-	-	-	-	-	-	-	-	-	-
MC-F20	68	7	100	300	5100-82	MIBC-25	7261-50	9.5	-	-	-	-	-	500	SIPX-60	MIBC-20	10	-	-	-	-	-	-	-	850	SIPX-60	MIBC-10	7
MC-F21	60	7	100	300	5100-82	MIBC-30	-	9.5	-	-	-	-	-	500	SIPX-75	MIBC-30	11.5	-	-	-	-	-	-	-	-	-	-	-
MC-F22	60	7	150	450	5100-82	MIBC-30	-	9.5	-	-	-	-	-	500	5100-45	MIBC-30	11.5	-	-	-	-	-	-	-	2500	SIPX-30	-	7
MC-F23	60	7	150	450	5100-57	MIBC-30	-	9.5	-	-	-	-	-	500	7279-45	MIBC-30	11.5	-	-	-	-	-	-	-	-	-	-	-
MC-F24	60	7	150	450	5100-57	76A-30	-	9.5	-	-	-	-	-	750	7279-45	76A-30	11.5	-	-	-	-	-	-	-	-	-	-	-
MC-F25	52	7	150	450	5100-90	76A-30	-	9	-	-	-	-	-	500	5100-45	76A-30	11.5	-	-	-	-	-	-	-	3500	SIPX-30	-	7
MC-F26	56	7	100	300	5100-82	MIBC-25	-	9.5	16	50	5100-20	MIBC-10	9.5	500	5100-45	MIBC-30	11.5	24	200	NaHA-625	5100-9	MIBC-9	-	11.5	2350	SIPX-30	-	7
MC-F27	59	7	100	300	5100-107	MIBC-40	-	9.5	15	50	5100-25	MIBC-10	10	500	5100-45	MIBC-35	11.5	14	200	NaHA-625	5100-25	MIBC-37	-	11.8	2200	SIPX-30	-	7

13.5.2 Rougher Flotation

The initial rougher flotation tests focused on optimization of the copper-lead circuit. Collectors and depressants were evaluated to maximize copper recovery while reducing the misplacement of zinc and arsenic to the rougher copper-lead concentrate. Although the lead, gold, and silver are typically recovered in the copper-lead concentrate, copper recovery was the primary performance driver for the testwork. Zinc rougher circuit investigations were mostly performed after the copper-lead circuit was partially optimized.

The baseline test results, using Aerophine 3418A (copper-lead circuit) and SIPX (zinc circuit) collectors, indicate that reducing the NaCN/$ZnSO_4$ depressant dosage to 100/300 g/t in test MC-F2 (P_{80} 55 µm) increased the copper circuit rougher recovery to 96% Cu, while zinc misplacement increased from 15% to 24%. Coarsening the primary grind in test MC-F4 (P_{80} of 80 µm) with the same reduced zinc depressant dosage had a slight detrimental effect on selective recovery performance, by increasing zinc misplacement at lower copper recovery. The high depressant dosage combination (three (3) times the lower dosage) used in test MC-F1 generally provided the lowest zinc misplacement of 15% to the copper-lead concentrate, but at the expense of copper, gold and silver performance.

13.5.2.1 Collector Screening

Four (4) sulfide collectors were screened in the copper-lead circuit, with 3418A (MC-F2) providing baseline results. Copper-lead rougher flotation was performed at pH 9.5, with a target primary grind of 55 µm, with equal NaCN/$ZnSO_4$ and MIBC dosages for all tests. Aero 3894 and Aero 5100 are thionocarbamate collectors known for their copper chelating ability and good selectivity over iron sulfides. Aerofloat 208 is a dithiophosphate collector known for selectivity against arsenic and iron sulfides. Zinc rougher flotation was performed using SIPX at pH 10.

Table 13.13: Rougher Flotation Collector Screening

Test	Product	Wt.	Assay								Distribution							
		%	Au g/t	Ag g/t	Cu %	Pb %	Zn %	As %	Fe %	S %	Au %	Ag %	Cu %	Pb %	Zn %	As %	Fe %	S %
MC-F2 3418A (Baseline)	Cu/Pb Rougher Con	18.3	5.58	201	9.87	1.77	6.06	1.94	26.6	28.8	46.3	84.9	**95.9**	**81.5**	24.1	21.0	25.4	31.3
	Zn Rougher Con	26.7	4.02	18.3	0.16	0.18	12.8	4.59	33.8	39.8	48.7	11.3	2.2	12.4	**74.1**	72.7	47.2	63.1
MC-F3 Aero 3894	Cu/Pb Rougher Con	16.0	3.60	201	11.4	1.21	4.63	1.14	22.4	22.3	30.7	71.1	**92.7**	**50.7**	15.7	11.0	18.8	20.9
	Zn Rougher Con	32.9	3.65	33.2	0.32	0.50	12.0	4.25	33.0	38.8	63.8	24.0	5.4	42.4	**83.0**	84.4	56.8	74.2
MC-F11 Aero 5100	Cu/Pb Rougher Con	17.9	4.52	192	9.85	1.41	4.86	1.38	22.3	23.8	37.0	79.8	**94.8**	**65.3**	19.3	15.1	21.0	24.8
	Zn Rougher Con	34.0	3.86	21.6	0.20	0.32	10.52	3.92	31.2	37.1	60.1	17.1	3.6	28.5	**79.5**	81.4	56.1	73.4
MC-F12 Aero 5100 Aerofloat 208	Cu/Pb Rougher Con	20.3	4.23	167	8.59	1.32	4.78	1.47	22.7	23.8	38.8	79.2	**94.4**	**71.4**	21.5	18.1	24.2	28.0
	Zn Rougher Con	32.2	4.02	23.7	0.23	0.28	10.8	4.02	31.4	37.5	58.5	17.8	4.1	23.6	**77.1**	78.6	53.2	70.3

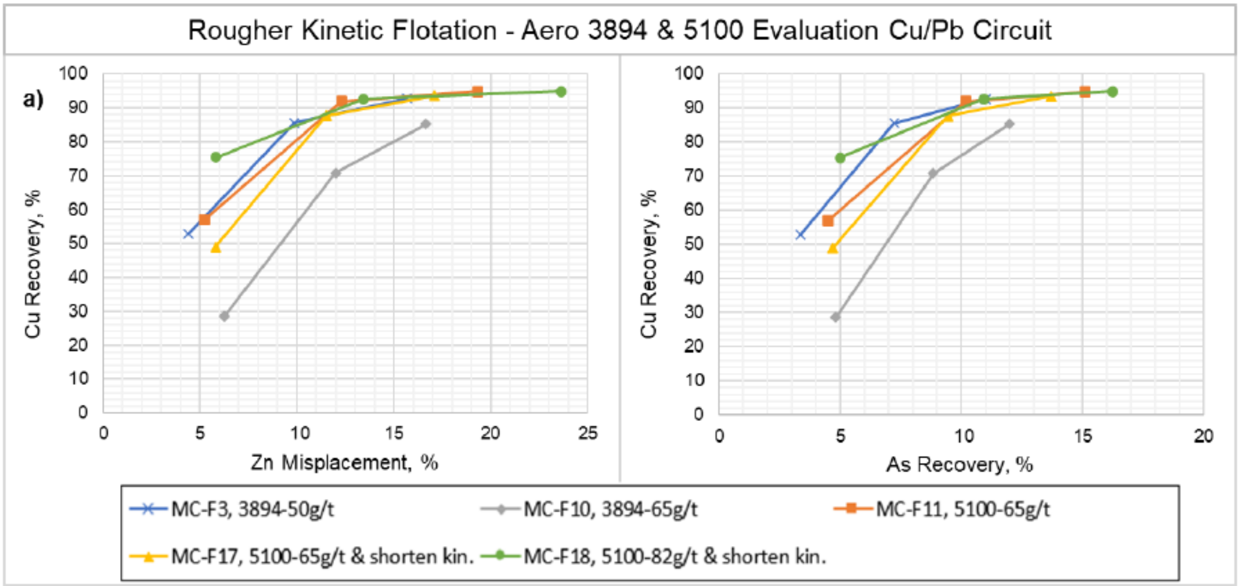
The three (3) collectors tested performed well when compared to the baseline test. Of the two (2) thionocarbamate collectors, Aero 3894 (Test MC-F3) was the most selective over arsenic, while Aero 5100 (Test MC-F11) had the highest copper recovery, which coincided with a proportionally higher mass yield. The combination of Aero 5100 and Aerofloat 208 (Test MC-F12) did not show any benefits under the conditions tested. Zinc misplacement followed a similar path to arsenic, with Aero 3894 the best performer in that respect. By employing selective collectors, zinc and arsenic recovery in the copper-lead circuit was reduced by up to 35 and 48%, respectively (Table 13.13), with minimal impact on copper recovery. Conversely, the rejection of arsenopyrite reduced the recovery of gold and silver to the copper-lead concentrate due to mineralogical associations.

13.5.2.2 Thiocarbonate Collectors Evaluation

Based on the encouraging results from the screening tests, some additional tests were performed to assess the impact of collector dosages and/or reduced froth collection times. The copper-lead rougher circuit kinetics and overall results, summarized in Figure 13.7 and Table 13.14, suggest that adjustments with the Aero 5100 collector were moderately successful. The recovery of arsenic and zinc to the copper-lead concentrate decreased by ~10% by shortening the rougher kinetics time to six (6) minutes (MC-F17) as compared to (MC-11) with a rougher kinetic time of nine (9) minutes. Increasing the Aero 5100 dosages to 82 g/t and shortening the kinetics time resulted in slightly improved copper-lead recoveries, with higher arsenic and zinc recoveries to the copper-lead concentrate.

Increasing the Aero 3894 collector dosage to 60 g/t (MC-F10) provided unexpectedly lower copper recoveries than at 50 g/t (MC-F3), with similar deportment of arsenic, gold, silver, and zinc to the copper-lead concentrate. Test results could not be repeated for Aero 3894, as observed from the poor correlation (R2 value of 0.04) between copper recovery and mass yield; contrarily, stronger correlations (R2 values of 0.93 to 0.97) for Fe and S, indicating copper recovery as an anomaly.

Figure 13.7: Thionocarbamate Evaluation



**Note: a) Zn Misplacement vs. Cu Recovery, b) As vs. Cu Recovery.*

Table 13.14: Rougher Flotation, Aero 3894 and 5100 Evaluation

Test	Conditions	Product	Wt.	Assay								Distribution							
				Au	Ag	Cu	Pb	Zn	As	Fe	S	Au	Ag	Cu	Pb	Zn	As	Fe	S
			%	g/t	g/t	%	%	%	%	%	%	%	%	%	%	%	%	%	%
MC-F3	Aero 3894 -50g/t	Cu/Pb Rougher Con	16.0	3.60	201	11.4	1.21	4.63	1.14	22.4	22.3	30.7	71.1	**92.7**	50.7	15.7	11.0	18.8	20.9
		Zn Rougher Con	32.9	3.65	33.2	0.32	0.50	12.0	4.25	33.0	38.8	63.8	24.0	5.4	42.4	**83.0**	84.4	56.8	74.2
MC-F10	Aero 3894 -60g/t	Cu/Pb Rougher Con	16.5	3.30	182	9.55	1.28	4.58	1.19	21.2	22.1	27.1	72.1	**85.3**	55.5	16.6	12.0	18.6	21.2
		Zn Rougher Con	36.2	3.88	28.5	0.66	0.42	10.3	3.85	30.3	36.5	69.8	24.7	12.9	39.5	**82.0**	84.9	58.3	76.7
MC-F11	Aero 5100 -65g/t	Cu/Pb Rougher Con	17.9	4.52	192	9.85	1.41	4.86	1.38	22.3	23.8	37.0	79.8	**94.8**	65.3	19.3	15.1	21.0	24.8
		Zn Rougher Con	34.0	3.86	21.6	0.20	0.32	10.5	3.92	31.2	37.1	60.1	17.1	3.6	28.5	**79.5**	81.4	56.1	73.4
MC-F17	Aero 5100 -65g/t	Cu/Pb Rougher Con	16.8	3.81	200	10.0	1.49	4.54	1.35	22.3	23.5	30.5	76.0	**93.5**	63.9	17.1	13.7	19.8	23.0
	Shorten kinetics	Zn Rougher Con	34.8	4.01	26.2	0.25	0.34	10.5	3.96	31.1	37.0	66.5	20.7	4.8	29.9	**81.4**	83.3	57.2	75.0
MC-F18	Aero 5100 -82g/t	Cu/Pb Rougher Con	17.9	3.72	201	9.48	1.46	5.99	1.52	22.2	24.0	32.7	79.8	**94.9**	67.8	23.6	16.2	21.1	25.2
	Shorten kinetics	Zn Rougher Con	34.0	3.88	22.8	0.19	0.31	10.0	3.97	31.1	36.6	64.7	17.3	3.5	27.2	**75.1**	80.5	56.1	73.1

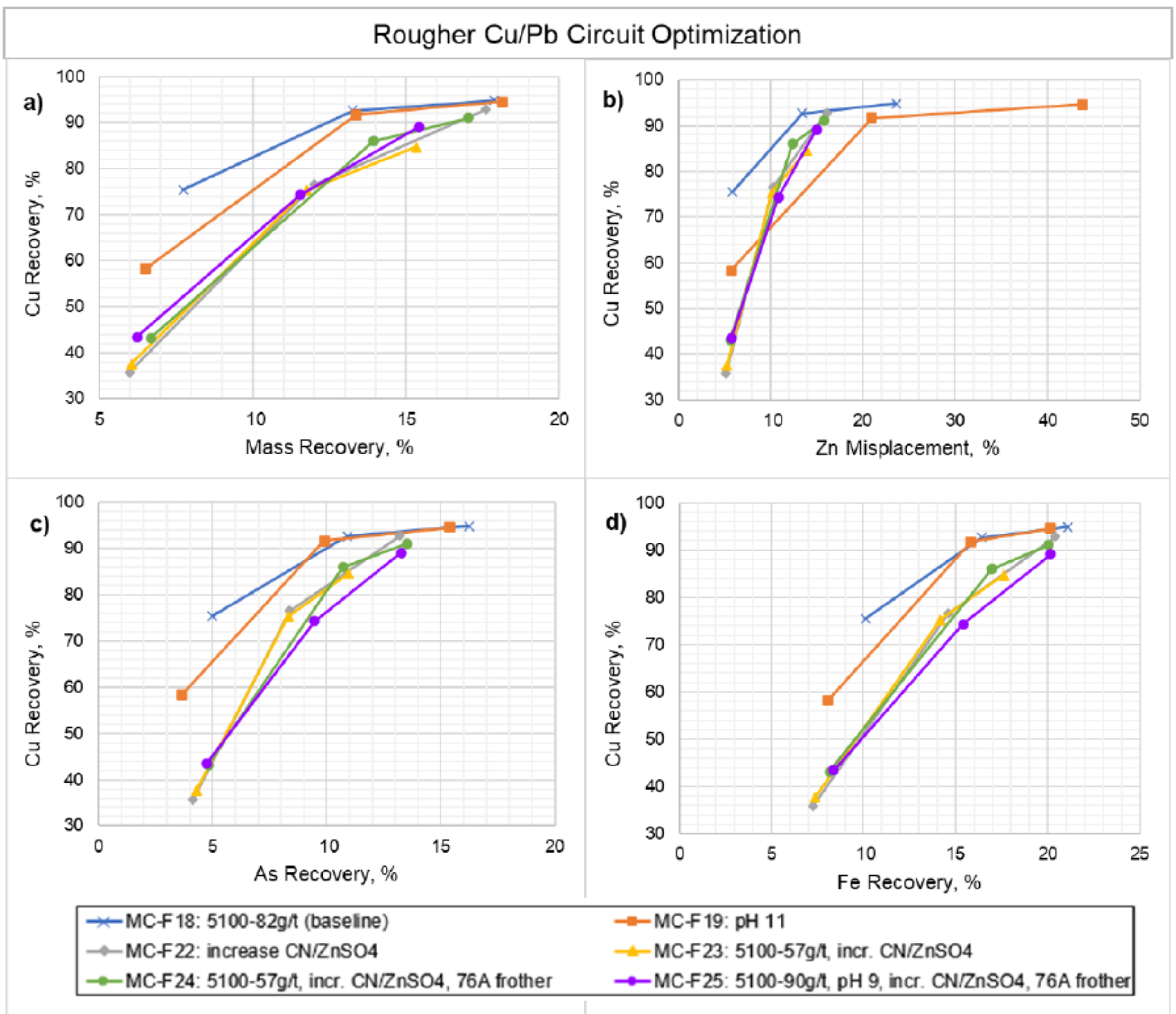
13.5.3 Iron Sulfide Depressant

Investigation of Aero 7261A as an iron sulfide depressant was conducted to determine if arsenopyrite could be rejected from the copper-lead concentrate. The first test (MC-F13) was performed using Aerophine 3418A as a collector and 20 g/t 7261A in the copper-lead rougher circuit, while the second test (MC-F20) utilized Aero 5100 with 50 g/t of 7261A. The copper kinetics with 7261A were slower for the first minute of flotation for both collector types, but recoveries converged by the next kinetics point; the same trend was observed with iron deportment. Zinc misplacement to the copper-lead concentrate was reduced by 8% and 23% for Aerophine 3418A and Aero 5100, respectively. The arsenic, gold, lead, and silver recoveries were relatively unaffected by Aero 7261A at the dosages tested. This depressant was not subject to any further testing.

13.5.4 Copper-Lead Rougher Optimization

A series of rougher flotation tests was performed to assess copper-lead circuit performance while adjusting various conditions such as the pulp pH, Aero 5100 collector dosage, NaCN/$ZnSO_4$ dosage, and replacing MIBC with Aerofroth 76A. The primary grind size remained at the target of 55 µm for these tests. The test results are summarized in Figure 13.8 and Table 13.15, with test MC-F18 serving as a baseline for comparative purposes.

Figure 13.8: Copper-Lead Rougher Optimization



**Note: a) Cu Recovery Kinetics, b) Cu vs. Zn Recovery, c) Cu vs. As Recovery, d) Cu vs. Fe Recovery.*

Table 13.15: Copper-Lead Rougher Optimization Results

Test	Conditions	Product	Wt.	Assay								Distribution							
			%	Au g/t	Ag g/t	Cu %	Pb %	Zn %	As %	Fe %	S %	Au %	Ag %	Cu %	Pb %	Zn %	As %	Fe %	S %
MC-F18	Aero 5100 - 82g/t (baseline)	Cu/Pb Rougher Con	17.9	3.72	201	9.48	1.46	5.99	1.52	22.2	24.0	32.7	79.8	**94.9**	**67.8**	23.6	16.2	21.1	25.2
		Zn Rougher Con	34.0	3.88	22.8	0.19	0.31	10.0	3.97	31.1	36.6	64.7	17.3	3.5	27.2	**75.1**	80.5	56.1	73.1
MC-F19	pH=11	Cu/Pb Rougher Con	18.1	4.33	189	9.40	1.44	10.7	1.41	21.1	25.4	36.0	77.6	**94.6**	**67.2**	43.9	15.4	20.2	26.6
		Zn Rougher Con	24.0	4.81	26.9	0.24	0.36	10.0	5.16	27.4	31.2	53.1	14.6	3.2	22.4	**54.6**	74.3	34.7	43.3
MC-F22	Increase NaCN/ZnSO$_4$	Cu/Pb Rougher Con	17.6	3.63	193	9.89	1.48	4.17	1.28	22.1	22.6	31.1	77.1	**92.8**	**65.0**	16.1	13.2	20.4	23.0
		Zn Rougher Con	15.4	3.24	34.5	0.45	0.46	23.9	2.98	21.0	30.7	24.3	12.1	3.7	17.9	**80.8**	26.9	17.1	27.3
MC-F23	Aero 5100 - 57g/t Increase NaCN/ZnSO$_4$	Cu/Pb Rougher Con	15.3	3.94	205	10.1	1.63	4.08	1.19	21.8	22.4	27.1	70.6	**84.7**	**62.5**	13.9	11.0	17.6	20.0
		Zn Rougher Con	16.7	3.88	39.4	1.31	0.45	22.1	2.77	20.1	29.1	29.2	14.8	12.0	18.8	**82.3**	27.8	17.8	28.3
MC-F24	Aero 5100 - 57g/t Increase NaCN/ZnSO$_4$ Aerofroth 76A	Cu/Pb Rougher Con	17.0	3.50	193	9.51	1.51	4.19	1.34	22.3	22.9	29.3	73.6	**91.1**	**66.5**	15.8	13.5	20.0	22.6
		Zn Rougher Con	16.9	3.22	36.2	0.59	0.41	21.7	3.02	20.8	30.2	26.8	13.7	5.6	18.1	**81.1**	30.2	18.6	29.6
MC-F25	Aero 5100 - 90g/t pH=9 Increase NaCN/ZnSO$_4$ Aerofroth 76A	Cu/Pb Rougher Con	15.4	4.24	216	10.9	1.73	4.44	1.43	24.6	25.9	32.5	75.3	**89.1**	**70.5**	15.0	13.3	20.2	23.4
		Zn Rougher Con	18.0	2.95	32.5	0.80	0.31	20.8	2.82	20.4	28.6	26.5	13.3	7.6	14.9	**82.2**	30.7	19.5	30.2

The copper-lead rougher optimization results, as compared with the baseline test (MC-F18), are summarized as follows:

- Increase pulp pH to 11:
  - Slower initial copper and arsenic kinetics, but final rougher recoveries were comparable to the baseline pH of 9.5.
  - Zinc misplacement to the copper-lead rougher concentrate increased to 44%, nearly double the baseline results.
- Increase NaCN/ZnSO$_4$ to 150/450 g/t:
  - By increasing the depressant dosage by 1.5x the baseline test, slower copper kinetics were observed with a relatively lower final copper recovery to the copper-lead concentrate.
  - Improved arsenic and zinc rejection by around 19% and 32%, respectively, when compared to the baseline results.
- Decrease Aero 5100 to 57 g/t, elevate NaCN/ZnSO$_4$, MIBC or Aerofroth 76A:
  - Decreasing the Aero 5100 dosage to 57 g/t with 150/450 g/t NaCN/ZnSO$_4$ for test MC-F23 (with MIBC) achieved the best rejection of arsenic and zinc in this set of tests, but copper recovery to the copper-lead concentrate decreased to 85%.
  - Replacing MIBC with Aerofroth 76A (test MC-F24) yielded a higher recovery of 91% Cu, but this was still lower than 95% Cu achieved in the baseline test. It is, however, noted that arsenic and

zinc rejection improved by roughly 17% and 33%, respectively, as compared to the baseline results.

- Increase Aero 5100 to 90 g/t, decrease pulp pH to 9:
  - Lastly, with the NaCN/$ZnSO_4$ at 150/450 g/t and continued use of the 76A frother, increasing the Aero 5100 collector dosage and lowering the pulp pH to 9 did not provide any measurable performance improvements when compared to the MC-F24 and the baseline tests.

The rougher copper-lead optimization tests show that the baseline conditions from test MC-F18 generally provided the best combination of copper rougher recovery along with arsenic and zinc rejection from the rougher concentrate. By elevating the NaCN/$ZnSO_4$ dosage to 150/450 g/t, benefits in arsenic and zinc rejections were seen, but at the expense of lower copper, lead, gold, and silver recoveries. Adjustments to the pulp pH and Aero 5100 dosages did not provide any meaningful improvements to the baseline results.

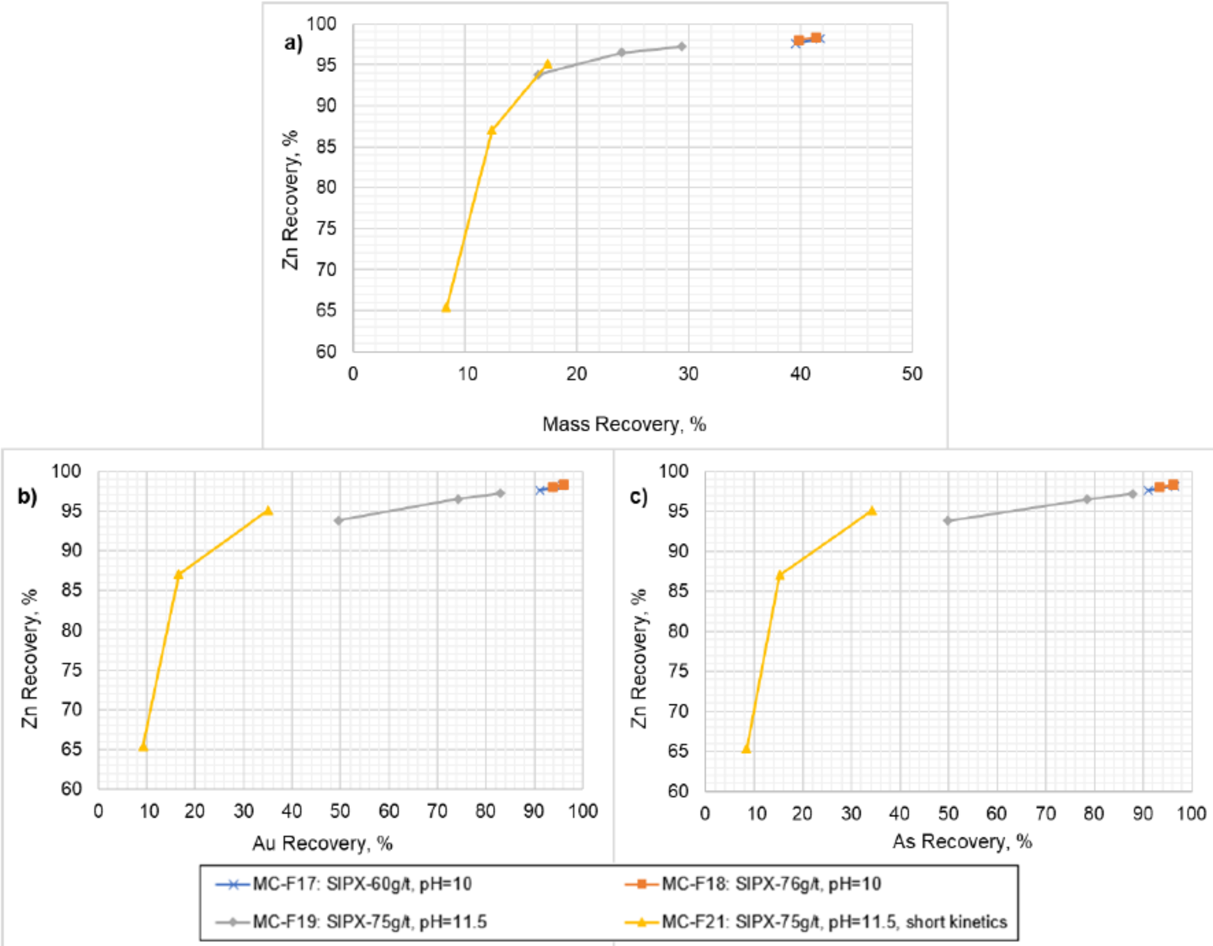
13.5.5 Zn Rougher Circuit Optimization

For the initial flotation tests (up to MC-F17), the reagent scheme for the zinc rougher circuit remained the same and consisted of 500 g/t copper sulfate ($CuSO_4$) to activate the sphalerite, SIPX (60 g/t) as the collector, MIBC as the frother, and a pulp pH of 10. Upon partial optimization of the copper-lead rougher circuit, various zinc flotation conditions were evaluated to maximize the recovery of zinc and reject the arsenic (penalty) and gold (non-payable) content in the zinc rougher concentrate. Parameters such as collector type / dosage, pulp pH (10 vs. 11.5), frother type (MIBC vs. Aerofroth 76A), and rougher kinetics were investigated.

The normalized SIPX collector test results are presented in Figure 13.9, with test MC-F17 serving as a baseline for the various parameter adjustments. The plotted results are normalized to reflect the zinc, gold and arsenic content that are available for recovery from the zinc circuit feed. The test results show:

- No change in zinc, gold and arsenic flotation performance was seen between the SIPX dosages of 60 g/t (baseline) and 76 g/t (MC-F18).
- By increasing the pulp pH to 11.5 with the elevated SIPX dosage (MC-F19), the zinc stage rougher mass recovery decreased by nearly 30% while the zinc stage recovery remained high at 97%. A slight improvement in arsenic and gold rejection was also observed.
- Shortening the zinc flotation residence time to 1.2 minutes (MC-F21) produced significant improvements, highlighted by a decrease in rougher mass and an improvement in zinc grade. Zinc recovery remained high at 95%. Arsenic and gold recoveries were still quite high at about 35%.

Figure 13.9: Normalized Zinc Rougher Kinetics with SIPX

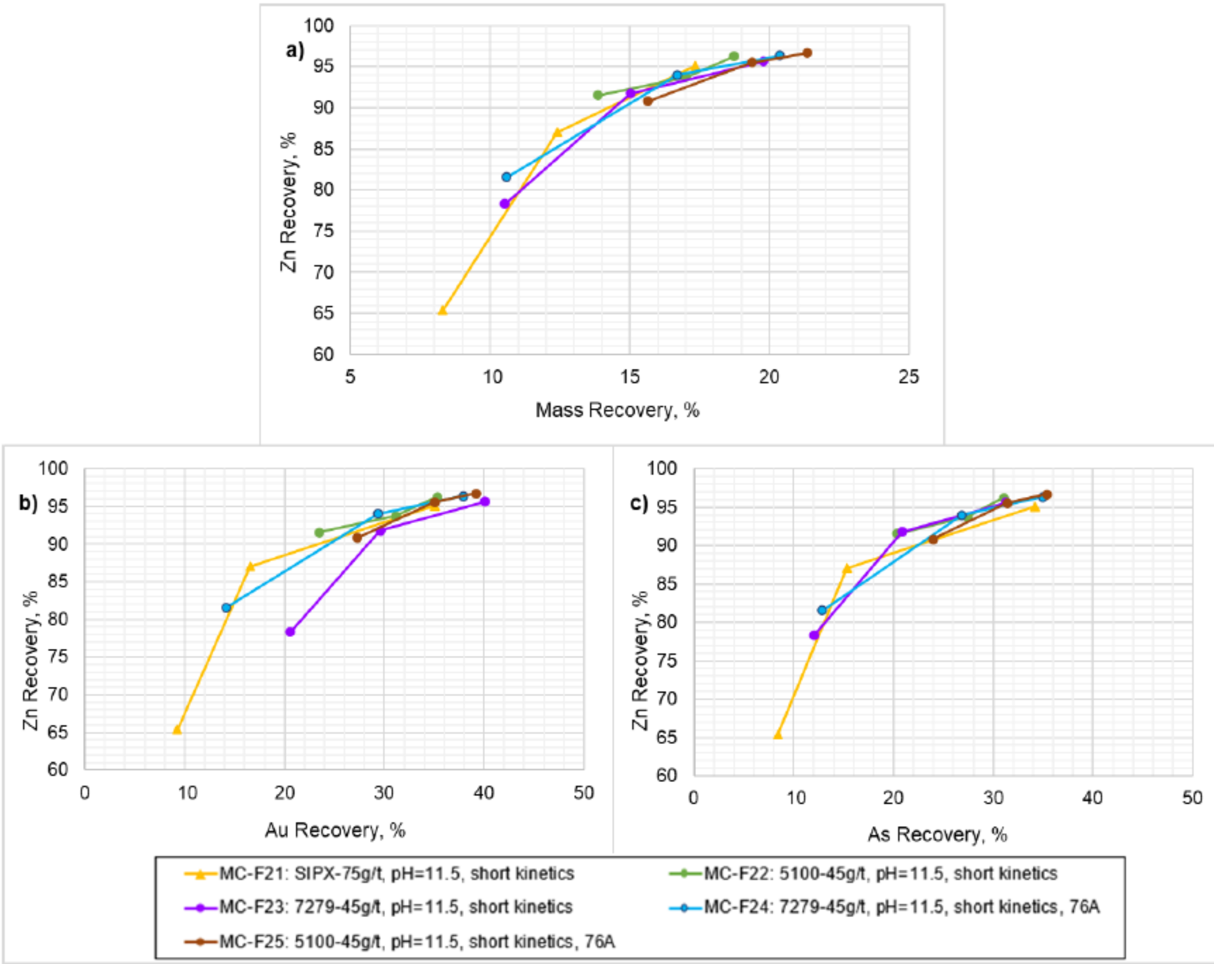


**Note: a) Zn Recovery Kinetics, b) Zn vs. Au Recovery, c) Zn vs. As Recovery.*

The shortened zinc rougher circuit residence time at a pulp pH of 11.5 was further evaluated with thionocarbamate collectors, Aero 5100 and 7279, which have good selectivity against iron sulfides. The results are presented in Figure 13.10 with test MC-F21 included as a comparative baseline.

Replacing the SIPX collector with Aero 5100 (test MC-F22) and Aero 7279 (test MC-F23) improved arsenic rejection by about 10%, while gold recovery to the zinc rougher concentrate increased by 14% with Aero 7279. The use of Aerofroth 76A as the frother simply increased the rougher concentrate mass recovery without providing any improvements in zinc circuit performance. In general, the thionocarbamate collectors performed similarly to SIPX with respect to zinc recovery, with improvements observed in arsenic rejection.

Figure 13.10: Normalized Zinc Rougher Kinetics with Thionocarbamate



**Note: a) Zn Recovery Kinetics, b) Zn vs. Au Recovery, c) Zn vs. As Recovery.*

13.5.6 Pyrite Rougher Flotation

Pyrite flotation was included in the later stages of testwork to recover the gold and silver content in the zinc rougher tails into an iron sulfide concentrate. With emphasis on rejection of the iron sulfide diluents (pyrite / arsenopyrite) from the copper-lead and zinc circuits, the associated gold and silver content were also lost to the rougher flotation tailings.

The pyrite flotation conditions were straightforward, with the pulp pH adjusted to 7 using H_2SO_4, followed by dosing with SIPX collector. No efforts were made to optimize these conditions. In general, up to 41% and 8% of the overall gold and silver were recovered in the concentrate in batch tests. Most of the Au and Ag losses were to the base metal rougher concentrates, and high stage recovery was achieved in pyrite / arsenopyrite flotation. The recovery of the precious metals content is anticipated to be higher in closed circuit, where the iron sulfides from the various cleaning tailings streams would be available for recovery in the pyrite / arsenopyrite flotation circuit.

13.6 Cleaner Flotation

13.6.1 Baseline Testing

An initial baseline cleaner kinetic test (MC-F5) was performed to assess general efficiencies of the copper, lead and zinc circuits. The rougher conditions were adopted from test MC-F4 (3418A in the Cu/Pb circuit, and SIPX in the Zn circuit), and the same set of collectors was used in the respective cleaning circuits.

The copper-lead and zinc circuit regrind sizes were 16 and 22 microns (P_{80}'s), respectively, and cleaner pulp pHs were maintained at 10.5 and 11.

The test results, as summarized in Table 13.16, displayed good cleaning efficiencies. Copper-lead circuit stage recoveries ranged between 91-94% for the copper, lead, and silver, with about 50% mass rejection to the cleaner tailings. Indications of effective arsenopyrite rejection in the zinc circuit were also observed, with 58% of the arsenic that entered the zinc cleaning circuit reported to the cleaner tailings. Despite that, the arsenic grades in the concentrates were still quite high, with 1.0-1.5% As in the copper concentrates, and 2.5-3.4% As in the zinc concentrates.

Table 13.16: Baseline Cleaner Kinetic Test

Test	Product	Wt.	Assay								Distribution							
		%	Au g/t	Ag g/t	Cu %	Pb %	Zn %	As %	Fe %	S %	Au %	Ag %	Cu %	Pb %	Zn %	As %	Fe %	S %
MC-F5	Cu/Pb Rougher Con	21.2	6.27	173	8.32	1.47	5.55	1.72	25.6	27.9	52.5	80.5	94.3	77.7	26.3	23.6	28.8	34.8
	Cu/Pb Cleaner Con 1	6.7	10.4	418	22.3	3.59	3.67	0.99	27.5	32.8	27.3	61.1	79.4	59.5	5.5	4.3	9.7	12.9
	Cu/Pb Cleaner Con 1-2	8.7	11.0	380	19.5	3.08	4.88	1.21	27.5	32.5	37.8	72.5	91.1	66.8	9.5	6.8	12.7	16.7
	Cu/Pb Cleaner Con 1-3	10.1	10.0	343	17.2	2.78	6.97	1.37	27.0	32.6	39.9	75.9	92.9	69.8	15.7	8.9	14.4	19.4
	Cu/Pb Cleaner Con 1-4	11.1	9.66	316	15.7	2.58	8.07	1.47	26.7	32.5	42.4	77.1	93.4	71.2	20.1	10.6	15.7	21.3
	Cu/Pb Cleaner Tail	10.1	2.54	15.4	0.16	0.26	2.77	2.00	24.4	22.7	10.1	3.4	0.9	6.5	6.3	13.0	13.0	13.5
	Zn Rougher Con	32.1	3.58	23.4	0.25	0.22	10.1	3.54	28.8	33.6	45.3	16.4	4.3	17.7	72.2	73.4	49.0	63.4
	Zn Cleaner Con 1-1	9.4	3.25	45.6	0.49	0.40	31.0	2.53	19.0	33.5	12.0	9.4	2.5	9.3	64.9	15.3	9.4	18.5
	Zn Cleaner Con 1-2	11.8	3.71	44.8	0.48	0.40	26.7	2.95	21.3	33.9	17.3	11.6	3.1	11.7	70.6	22.5	13.3	23.6
	Zn Cleaner Con 1-3	13.9	4.06	42.5	0.45	0.38	23.0	3.40	23.3	34.0	22.3	12.9	3.4	13.1	71.4	30.5	17.1	27.8
	Zn Cleaner Tail	18.2	3.21	8.80	0.09	0.10	0.20	3.65	33.1	33.2	23.1	3.5	0.9	4.5	0.8	42.9	31.9	35.6
	Rougher Tail	46.7	0.12	3.00	0.06	0.04	0.14	0.10	8.98	0.67	2.2	3.1	1.5	4.6	1.5	3.0	22.2	1.8
	Head (calc.)		2.53	45.6	1.87	0.40	4.47	1.55	18.9	17.0	100	100	100	100	100	100	100	100
	Direct Head		1.95	39.7	1.71	0.38	4.26	1.71	17.9	17.0								

13.6.2 Arsenopyrite Rejection in Zinc Cleaner Circuit

Based on the arsenic cleaning efficiency from the baseline cleaner test (MC-F5), two (2) arsenic rejection measures, using sodium humate (NaHA, MC-F6) and hydrogen peroxide (H_2O_2, MC-F7) were tested to reduce arsenic levels further to produce a Zn concentrate with acceptable grades for smelting. Both tests adopted the same target regrind sizes, cleaner kinetics scheme and SIPX collector in the zinc circuit; Aero 3894 replacing Aerophine 3418A in the copper-lead circuit.

The NaHA dosage and pulp pH conditions were extracted from literature, where optimum arsenopyrite rejection was observed with a NaHA dosage of > 300 mg/L at pH 11.5 using a xanthate collector. NaHA and a $CuSO_4$ activator were added to the regrind mill, which was adjusted to pH 11.5 with lime. For test MC-F7, hydrogen peroxide was added to the regrind mill discharge and allowed to condition for 15 minutes at natural pH (pH 7) prior to the addition of flotation reagents. Like test MC-F6, the $CuSO_4$ activator was added to the regrind mill. After conditioning, the pulp was adjusted to pH 11 prior to reagent addition and aeration.

The 1st cleaner kinetics test results, as summarized in Table 13.17, showed that the copper-lead cleaner conditions achieved high stage recoveries of copper, lead, gold, and silver with good mass rejection to the 1st cleaner tailings. Arsenic rejection responded positively with NaHA/H_2O_2 addition, and these tests achieved comparatively higher arsenic rejection than the baseline test.

As shown in Table 13.18, pre-treatment with NaHA performed slightly better than hydrogen peroxide, with 76% arsenic rejected in test MC-F6 as compared to 73% in test MC-F7. Based on these results, the NaHA depressant was selected for subsequent cleaner flotation testwork.

Table 13.17: 1st Cleaner Kinetics with Arsenic Depressant / Oxidant in Zinc Cleaner Circuit

Test	Product	Wt.	Assay								Distribution							
			Au	Ag	Cu	Pb	Zn	As	Fe	S	Au	Ag	Cu	Pb	Zn	As	Fe	S
		%	g/t	g/t	%	%	%	%	%	%	%	%	%	%	%	%	%	%
MC-F6 Sodium Humate	Cu/Pb Rougher Con	12.1	5.17	235	12.6	1.40	5.04	1.13	22.6	24.5	28.4	65.0	90.2	48.7	13.2	8.7	14.4	17.6
	Cu/Pb Cleaner Con 1	3.8	7.34	384	27.6	2.16	2.15	0.56	28.0	33.2	12.7	33.3	62.1	23.6	1.8	1.3	5.6	7.5
	Cu/Pb Cleaner Con 1-2	4.9	7.16	384	26.0	2.23	2.69	0.65	27.4	32.6	15.9	42.9	75.4	31.5	2.9	2.0	7.1	9.4
	Cu/Pb Cleaner Con 1-3	6.3	8.16	395	23.3	2.30	4.13	0.84	26.3	31.6	23.3	56.5	86.3	41.6	5.6	3.3	8.7	11.7
	Cu/Pb Cleaner Con 1-4	7.0	7.76	376	21.4	2.19	4.85	0.92	25.5	30.7	24.6	59.9	88.2	44.0	7.3	4.0	9.4	12.6
	Cu/Pb Cleaner Tail	5.1	1.64	44.2	0.68	0.32	5.29	1.43	18.6	16.2	3.8	5.2	2.1	4.7	5.9	4.6	5.0	4.9
	Zn Rougher Con	38.2	3.98	36.4	0.35	0.41	10.3	3.65	30.4	35.7	69.1	31.8	8.0	45.6	85.5	88.4	61.3	80.6
	Zn Cleaner Con 1-1	11.6	3.29	67.0	0.70	0.46	27.6	2.20	20.4	34.3	17.4	17.8	4.8	15.4	69.3	16.2	12.5	23.5
	Zn Cleaner Con 1-2	13.2	3.41	68.0	0.72	0.49	27.6	2.28	20.5	34.2	20.5	20.5	5.7	18.8	78.8	19.1	14.3	26.7
	Zn Cleaner Con 1-3	14.2	3.59	69.2	0.73	0.56	26.8	2.36	20.8	34.1	23.2	22.5	6.2	22.8	82.8	21.3	15.6	28.7
	Zn Cleaner Tail	23.9	4.21	16.9	0.13	0.33	0.53	4.42	36.1	36.6	45.9	9.3	1.8	22.8	2.7	67.1	45.7	51.9
	Rougher Tail	49.7	0.11	2.80	0.06	0.04	0.12	0.09	9.22	0.64	2.5	3.2	1.8	5.7	1.3	2.9	24.2	1.9
	Head (calc.)		2.20	43.7	1.69	0.35	4.62	1.58	18.9	16.9	100	100	100	100	100	100	100	100
	Direct Head		1.95	39.7	1.71	0.38	4.26	1.71	17.9	17.0								
MC-F7 Hydrogen Peroxide	Cu/Pb Rougher Con	13.9	4.33	213	11.1	1.55	5.00	1.16	22.6	23.8	23.9	64.0	80.2	52.0	18.3	9.9	16.2	19.8
	Cu/Pb Cleaner Con 1	4.0	7.07	376	27.0	2.47	2.18	0.62	28.0	32.7	11.1	32.3	55.9	23.7	2.3	1.5	5.7	7.8
	Cu/Pb Cleaner Con 1-2	5.6	7.18	391	24.6	2.78	3.07	0.75	27.1	31.8	16.0	47.6	72.2	37.7	4.5	2.6	7.9	10.7
	Cu/Pb Cleaner Con 1-3	6.9	7.06	379	21.5	2.72	4.28	0.92	26.2	30.7	19.4	56.8	77.6	45.4	7.8	3.9	9.4	12.7
	Cu/Pb Cleaner Con 1-4	7.8	6.60	354	19.3	2.52	5.28	1.01	25.5	29.9	20.5	60.0	79.0	47.7	10.9	4.8	10.3	14.0
	Cu/Pb Cleaner Tail	6.1	1.40	30.4	0.38	0.29	4.65	1.36	18.8	16.1	3.4	4.0	1.2	4.3	7.4	5.1	5.9	5.8
	Zn Rougher Con	36.8	5.04	41.4	0.96	0.47	8.27	3.87	31.7	35.6	73.6	33.0	18.5	42.1	80.0	87.2	60.3	78.4
	Zn Cleaner Con 1-1	7.8	3.63	98.0	3.20	0.80	30.2	2.16	20.1	33.9	11.2	16.5	13.0	15.0	61.6	10.3	8.1	15.7
	Zn Cleaner Con 1-2	11.7	4.40	94.3	2.67	0.95	24.8	2.62	22.8	34.1	20.4	23.8	16.2	26.8	76.0	18.7	13.8	23.8
	Zn Cleaner Con 1-3	13.4	4.66	91.7	2.45	0.95	22.3	2.88	24.1	34.1	24.8	26.6	17.1	30.8	78.4	23.6	16.7	27.3
	Zn Cleaner Tail	23.4	5.25	12.6	0.11	0.20	0.26	4.43	36.0	36.4	48.8	6.4	1.3	11.3	1.6	63.6	43.7	51.0
	Rougher Tail	49.3	0.13	2.80	0.05	0.05	0.13	0.10	9.18	0.61	2.5	3.0	1.3	6.0	1.7	2.9	23.4	1.8
	Head (calc.)		2.52	46.2	1.91	0.41	3.81	1.63	19.3	16.7	100	100	100	100	100	100	100	100
	Direct Head		1.95	39.7	1.71	0.38	4.26	1.71	17.9	17.0								

Table 13.18: Arsenic Rejection in Zinc 1st Cleaners

Test	Arsenic Recovery, %			As Rejected in Zn Cleaners
	Zn Rougher Con	Zn Cleaner Con	Zn Cleaner Tail	
MC-F5 (Baseline)	73.4	30.5	42.9	58%
MF-F6 (Sodium Humate)	88.4	21.3	67.1	76%
MC-F7 (Hydrogen Peroxide)	87.2	23.6	63.6	73%

13.6.3 Sequential Cleaner Flotation with Aero 3894

Sequential cleaner tests (MC-F8 and MC-F9) were performed using Aero 3894 in the copper-lead circuit, adopting the flotation conditions defined in test MC-F7. The copper cleaning shown in Figure 13.11 and Table 13.19 showed that increasing the collector dosage in the cleaner circuit (MC-F9) achieved better copper-grade recovery performance than the test (MC-F8) with 15 g/t of Aero 3894, but with a higher zinc misplacement. Copper recoveries were lower than expected based on the test results from MC-F3 (92% Cu, 16% wt.) and MC-F7 (80.2% Cu, 13.9% wt.), where the same rougher conditions were maintained.

Figure 13.11: Copper-Lead Cleaning Efficiency with Aero 3894

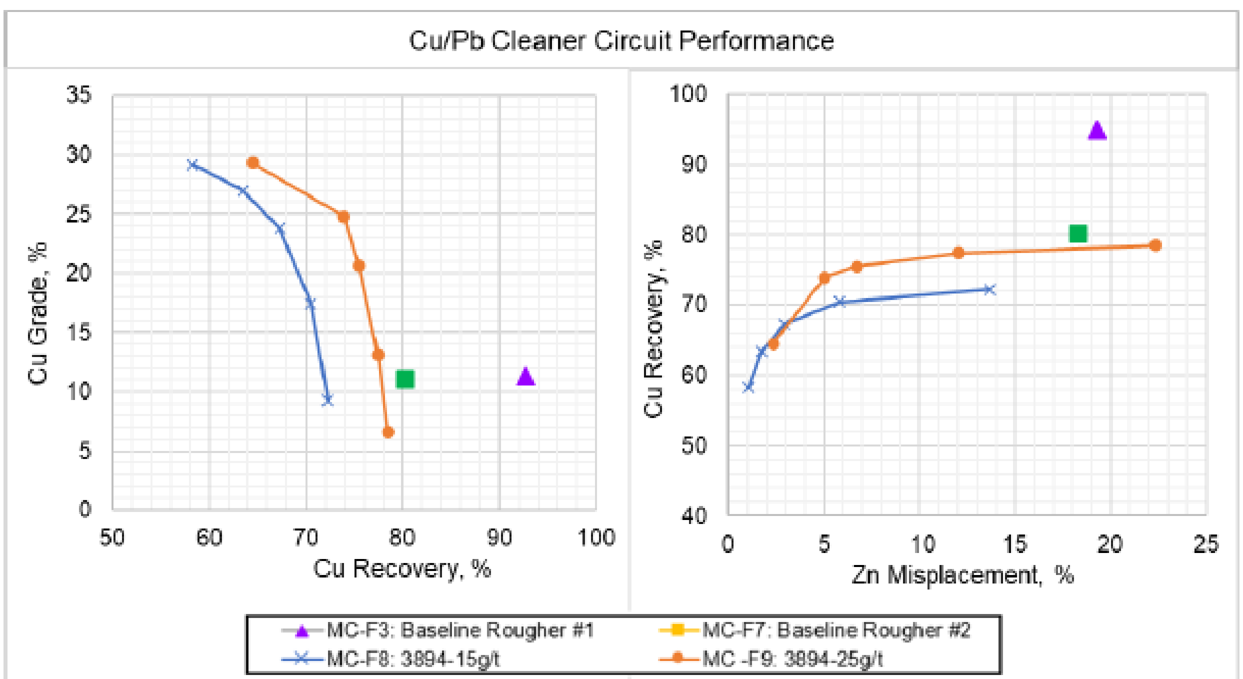
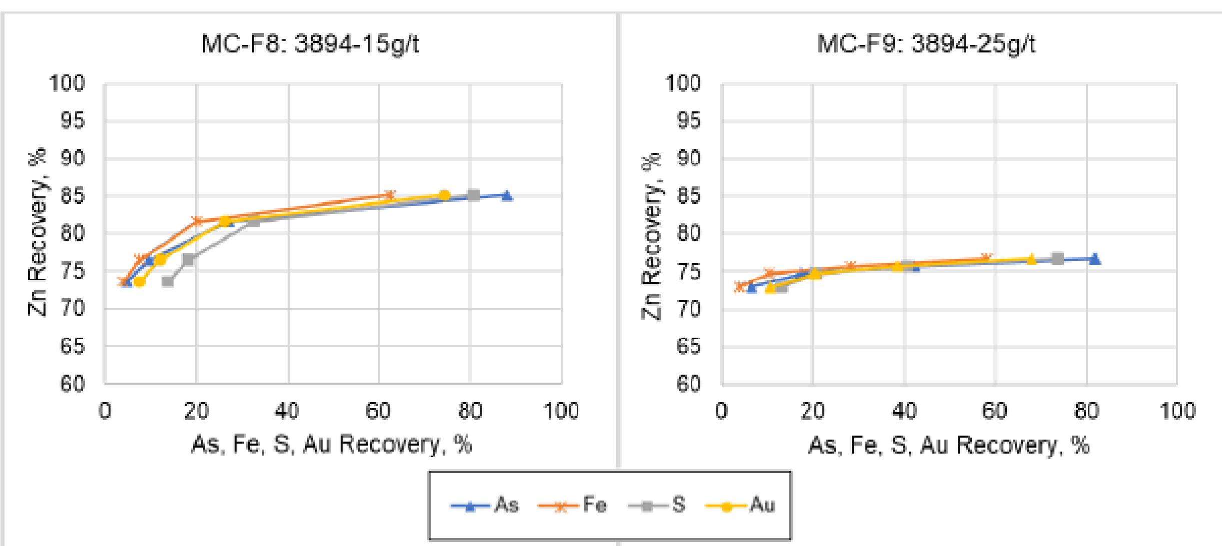


Table 13.19: Sequential Cleaner Flotation, Arsenopyrite Rejection Using NaHA

Test	Product	Wt.	Assay								Distribution							
			Au	Ag	Cu	Pb	Zn	As	Fe	S	Au	Ag	Cu	Pb	Zn	As	Fe	S
		%	g/t	g/t	%	%	%	%	%	%	%	%	%	%	%	%	%	%
MC-F8: 3894-15g/t	Cu/Pb Cleaner 4 Con	3.6	6.58	480	29.2	2.66	1.30	0.47	27.1	34.1	11.3	43.7	58.3	25.3	1.0	1.0	5.1	7.3
	Cu/Pb Cleaner 3 Con	4.3	6.72	462	27.0	3.02	1.87	0.63	26.9	33.2	13.6	49.5	63.4	33.8	1.8	1.6	6.0	8.4
	Cu/Pb Cleaner 2 Con	5.1	6.56	437	23.8	3.10	2.60	0.82	26.2	31.8	16.0	56.4	67.3	41.8	2.9	2.4	7.0	9.6
	Cu/Pb Cleaner 1 Con	7.3	5.45	346	17.5	2.55	3.63	1.03	24.1	27.7	19.0	63.7	70.5	49.1	5.8	4.3	9.2	12.0
	Cu/Pb Rougher Con	14.1	3.50	191	9.24	1.45	4.40	1.17	21.0	21.4	23.6	68.2	72.2	54.1	13.7	9.6	15.5	17.9
	Cu/Pb Combined Clnr Tail	10.5	2.45	92.1	2.40	1.04	5.46	1.41	18.9	17.0	12.3	24.5	14.0	28.9	12.7	8.6	10.4	10.6
	Zn Cleaner 3 Con	7.0	2.24	51.0	4.67	0.59	47.9	1.15	10.7	33.0	7.5	9.0	18.0	10.8	73.6	4.6	3.9	13.6
	Zn Cleaner 2 Con	9.2	2.71	49.8	3.90	0.60	37.6	1.79	15.7	33.1	12.0	11.6	19.9	14.6	76.5	9.6	7.6	18.1
	Zn Cleaner 1 Con	16.4	3.34	43.0	2.55	0.58	22.6	2.84	23.5	33.7	26.1	17.8	23.1	24.8	81.6	26.9	20.1	32.7
	Zn Rougher Con	39.1	3.98	28.8	1.21	0.40	9.89	3.89	30.6	34.8	74.3	28.4	26.2	40.9	85.2	87.9	62.4	80.7
	Zn Combined Clnr Tail	32.1	4.36	23.9	0.46	0.36	1.63	4.49	34.9	35.2	66.9	19.4	8.2	30.1	11.6	83.3	58.5	67.0
	Rougher Tail	46.8	0.09	2.90	0.06	0.04	0.11	0.09	9.08	0.52	2.0	3.4	1.6	4.9	1.1	2.5	22.2	1.4
	Head (calc.)		*2.09*	*39.6*	*1.81*	*0.38*	*4.54*	*1.73*	*19.2*	*16.9*								
MC-F9: 3894:25g/t	Cu/Pb Cleaner 4 Con	4.0	6.54	466	29.3	2.61	2.60	0.49	27.5	34.2	13.0	46.7	64.5	28.0	2.3	1.2	5.9	8.2
	Cu/Pb Cleaner 3 Con	5.4	6.69	440	24.8	3.11	4.14	0.78	26.6	32.6	18.0	59.7	73.9	45.1	5.0	2.6	7.7	10.5
	Cu/Pb Cleaner 2 Con	6.6	5.96	378	20.7	2.73	4.49	0.94	25.3	30.1	19.6	62.8	75.4	48.5	6.7	3.8	8.9	11.9
	Cu/Pb Cleaner 1 Con	10.7	4.37	253	13.1	1.89	4.96	1.15	22.5	24.9	23.3	68.2	77.4	54.5	12.0	7.5	12.9	15.9
	Cu/Pb Rougher Con	21.5	2.78	133	6.58	1.03	4.58	1.21	19.8	19.2	29.9	72.3	78.5	59.8	22.3	16.0	22.9	24.9
	Cu/Pb Combined Clnr Tail	17.6	1.93	57.8	1.44	0.67	5.03	1.38	18.1	15.9	16.9	25.6	14.0	31.8	20.0	14.8	17.0	16.7
	Zn Cleaner 3 Con	6.8	3.17	51.0	4.05	0.93	47.6	1.55	10.2	32.3	10.7	8.7	15.2	17.0	73.0	6.4	3.7	13.1
	Zn Cleaner 2 Con	10.8	3.80	48.6	2.91	0.78	30.7	2.63	18.0	32.1	20.4	13.2	17.3	22.7	74.8	17.3	10.4	20.7
	Zn Cleaner 1 Con	21.1	3.65	35.9	1.60	0.50	15.8	3.26	25.0	32.1	38.4	19.1	18.7	28.5	75.7	42.1	28.2	40.6
	Zn Rougher Con	38.2	3.56	26.2	0.95	0.35	8.87	3.50	28.4	32.1	67.9	25.2	20.2	35.9	76.8	81.8	58.2	73.7
	Zn Combined Clnr Tail	31.5	3.64	20.8	0.29	0.22	0.53	3.92	32.3	32.1	57.2	16.5	5.0	18.9	3.8	75.3	54.5	60.6
	Rougher Tail	40.2	0.11	2.50	0.06	0.04	0.10	0.09	8.79	0.60	2.2	2.5	1.3	4.3	0.9	2.3	18.9	1.4
	Head (calc.)		*2.00*	*39.7*	*1.81*	*0.37*	*4.42*	*1.64*	*18.7*	*16.7*								

The selectivity for zinc over arsenic minerals (Figure 13.12) was observed in the zinc cleaning circuit for both tests, confirming the rejection of arsenopyrite selectively and that the use of NaHA can be suitable for the master composite 1A and 1B material, owing to the good liberation of key sulfide minerals. The gold content was rejected, and about 90% of gold entering the zinc cleaner circuit reported to the combined cleaner tail, reaffirming the strong correlation between gold and sulfide minerals.

Figure 13.12: Sequential Cleaner Flotation, Zinc Circuits, Metals Rejection



13.6.4 Sequential Cleaner Flotation with Aero 5100

Five (5) cleaner flotation tests using Aero 5100 in the copper-lead circuit and SIPX or Aero 5100 in the zinc circuit were performed, with each test including 625 g/t NaHA depressant for arsenopyrite rejection along with 200 g/t $CuSO_4$ activator. While the target primary and regrind sizes were the same for the copper-lead circuit (60 µm and 15 µm, respectively), the Aero 5100 collector dosages and cleaner pulp pH were adjusted based on intermediate test results. For the zinc circuit, collector dosages and a coarser regrind were evaluated. The key conditions for the five cleaner tests were as follows:

- Test MC-F14: considered as the baseline test, conditions include:
  - Rougher conditions from test MC-F11 were used.
  - Copper-lead cleaners at pH 10.5 with 25 g/t Aero 5100.
  - Zinc cleaners regrind of 16 µm, NaHA addition in the regrind, pH 11.5, and 25 g/t SIPX.
- Test MC-F15: same conditions as MC-F14, except the NaHA addition points were split between the regrind and post-regrind conditioner. SIPX in the zinc cleaners was also reduced to 10 g/t.
- Test MC-F16: baseline test on master composite 1B, using the MC-F15 procedure.
- Test MC-F26: adjustments from the baseline MC-F16 conditions include:
  - Aero 5100 increased to 82 g/t in the copper-lead rougher circuit.
  - Copper-lead cleaner circuit pH was reduced to 9.5, and the Aero 5100 dosage was 20 g/t.
  - SIPX collector was replaced with Aero 5100 in the zinc rougher (45 g/t) and cleaner (9 g/t) circuits.
  - The zinc regrind size was increased to 24 µm.
- Test MC-F27: same conditions as MC-F26, but with scavenger stages, increased zinc cleaner pH to 11.8, and Aero 5100 to 25 g/t, and zinc regrind size back to 16 µm.

Table 13.20: Sequential Cleaner Flotation with Aero 5100 (Cu Circuit) and SIPX/Aero 5100 (Zn Circuit)

Test	Product	Wt.	Assay								Distribution							
		%	Au g/t	Ag g/t	**Cu %**	Pb %	**Zn %**	As %	Fe %	S %	Au %	Ag %	**Cu %**	Pb %	**Zn %**	As %	Fe %	S %
MC-F14	Cu Cleaner 4 Con	3.9	6.16	**494**	**30.5**	2.64	2.58	0.40	27.6	33.8	12.3	41.2	**65.7**	27.4	2.3	0.9	5.7	7.8
	Cu Rougher Con	18.9	3.32	181	8.86	1.30	4.57	1.32	21.5	22.2	31.9	73.1	92.1	65.1	19.6	14.7	21.5	24.8
	Zn Cleaner 3 Con	5.2	1.72	75.0	0.92	0.43	**58.4**	0.84	4.68	33.5	4.5	8.3	2.6	5.9	**68.6**	2.5	1.3	10.2
	Zn Rougher Con	35.8	3.59	31.9	0.32	0.32	9.73	3.91	29.9	34.8	65.7	24.4	6.4	30.0	79.3	82.7	56.7	73.5
	Zn Combined Cleaner Tail	30.6	3.91	24.6	0.22	0.30	1.53	4.43	34.1	35.0	61.2	16.1	3.8	24.1	10.7	80.2	55.5	63.3
MC-F15	Cu Cleaner 4 Con	3.6	6.67	485	**31.7**	2.29	1.69	0.39	28.8	34.2	11.0	37.7	**61.3**	21.8	1.3	0.8	5.5	7.2
	Cu Rougher Con	17.1	3.94	202	9.93	1.35	4.77	1.35	22.1	23.2	31.2	75.2	92.2	61.7	18.1	13.6	20.1	23.6
	Zn Cleaner 3 Con	2.6	1.84	60.0	0.95	0.32	**58.1**	0.78	6.36	33.1	2.2	3.4	1.4	2.2	**33.7**	1.2	0.9	5.2
	Zn Rougher Con	34.8	4.04	28.3	0.31	0.34	10.52	4.03	30.2	35.4	65.2	21.4	5.9	31.8	80.9	82.1	55.9	73.1
	Zn Combined Cleaner Tail	32.1	4.22	25.7	0.26	0.34	6.63	4.29	32.2	35.6	62.9	18.0	4.6	29.6	47.1	80.9	55.0	67.9
MC-F16	Cu Cleaner 4 Con	4.2	6.26	508	**26.2**	3.45	8.27	0.44	23.9	33.4	12.7	46.8	**62.4**	37.5	7.8	1.1	5.4	8.2
	Cu Rougher Con	15.9	3.71	202	9.64	1.53	4.47	1.29	21.7	23.3	28.8	71.0	87.7	63.4	16.2	12.0	18.5	21.8
	Zn Cleaner 3 Con	3.6	1.24	80.0	1.86	0.34	**58.2**	0.48	4.07	32.7	2.2	6.4	3.8	3.2	**47.7**	1.0	0.8	6.9
	Zn Rougher Con	36.5	3.80	31.7	0.51	0.32	9.96	3.95	29.4	35.2	67.7	25.6	10.7	30.3	82.6	84.2	57.6	75.6
	Zn Combined Cleaner Tail	32.8	4.08	26.4	0.36	0.32	4.67	4.34	32.2	35.5	65.5	19.2	6.8	27.1	34.9	83.2	56.8	68.7
MC-F26	Cu Cleaner 3 Con	4.0	7.21	601	**30.3**	3.45	3.13	0.51	26.4	34.1	12.1	54.2	**67.2**	34.7	2.8	1.2	5.5	7.7
	Cu Rougher Con	15.6	5.81	223	10.9	1.62	6.30	1.57	22.9	25.9	38.0	78.4	94.2	63.4	22.2	13.9	18.5	22.9
	Zn Cleaner 3 Con	4.4	1.46	58.3	0.41	0.64	**59.8**	0.49	3.70	33.6	2.7	5.8	1.0	7.1	**59.2**	1.2	0.8	8.3
	Zn Rougher Con	18.4	3.55	27.6	0.29	0.43	17.92	3.30	21.2	28.2	27.5	11.5	3.0	19.9	74.8	34.5	20.3	29.4
	Zn Combined Cleaner Tail	14.1	4.20	18.1	0.25	0.36	4.89	4.18	26.6	26.6	24.8	5.7	2.0	12.9	15.6	33.3	19.4	21.1
MC-F27	Cu Cleaner 3 Con	6.2	6.31	559	**29.3**	1.82	2.01	0.80	28.7	33.4	20.6	58.4	**78.6**	21.9	2.3	3.8	11.3	15.7
	Cu Rougher + Scav Con	20.5	4.68	244	10.8	1.88	7.11	1.75	24.2	26.9	50.1	83.6	94.7	73.9	26.5	27.1	31.3	41.4
	Cu Rougher Con	19.6	4.72	251	11.2	1.90	6.66	1.71	24.4	26.9	48.5	82.3	94.2	71.9	23.8	25.5	30.2	39.7
	Zn Cleaner 3 Con	4.7	1.89	38.0	0.54	0.43	**56.3**	1.41	6.26	32.8	4.7	3.0	1.1	3.9	**48.6**	5.1	1.9	11.7
	Zn Rougher + Scav Con	20.7	3.76	35.9	0.44	0.51	19.2	3.67	22.3	30.1	40.7	12.4	3.9	20.5	72.4	57.7	29.2	46.9
	Zn Rougher Con	18.5	3.71	36.3	0.45	0.53	21.3	3.58	21.6	30.6	35.9	11.2	3.6	18.7	71.8	50.3	25.2	42.6
	Zn Combined Cleaner Tail	10.7	4.06	33.1	0.40	0.49	7.26	4.18	26.6	28.4	22.8	6.0	1.9	10.1	14.2	34.1	18.0	23.0

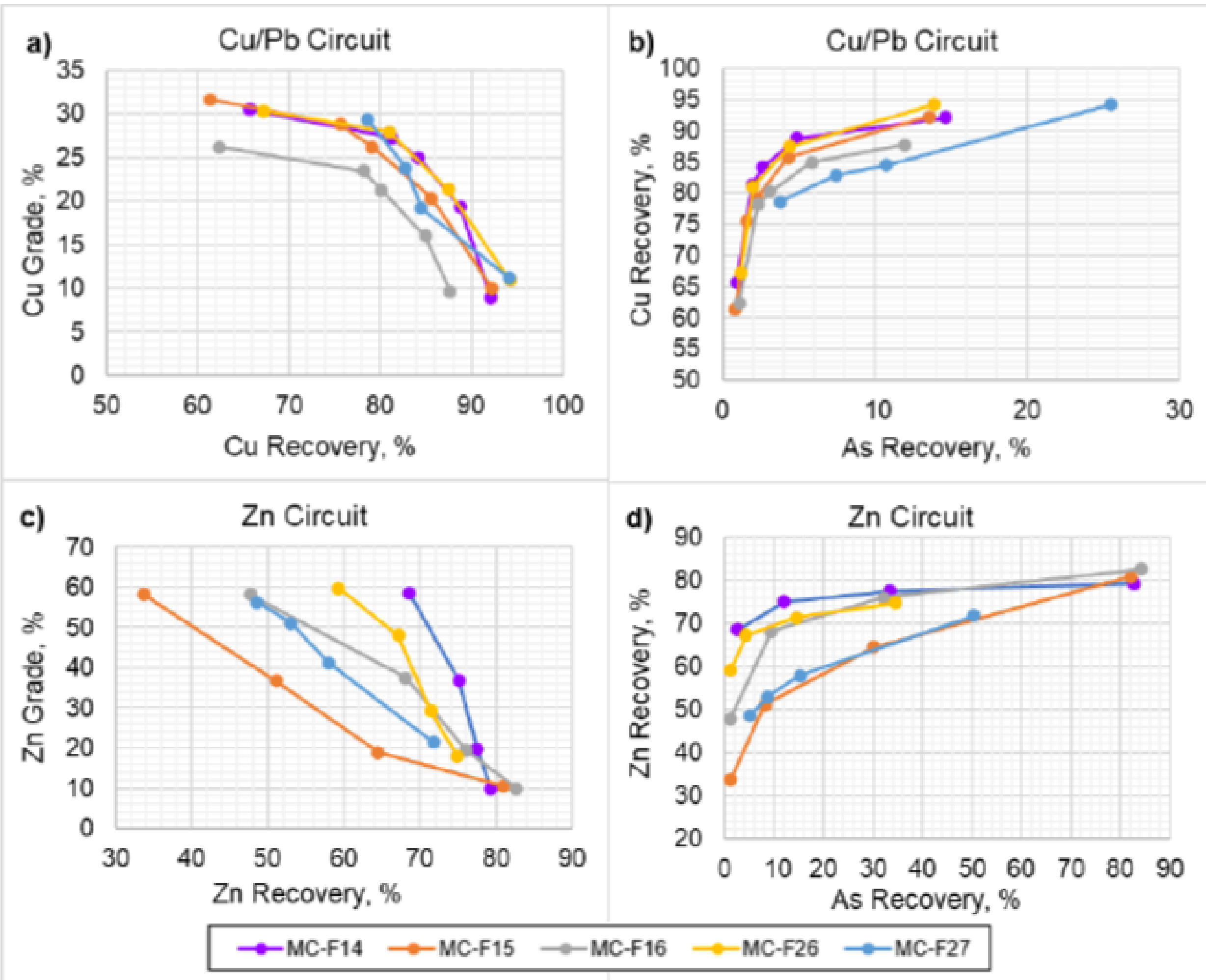
The sequential cleaner flotation test results are summarized in Table 13.20 and Figure 13.13. Flotation using copper-selective collector Aero 5100, at a cleaner pH of 10.5 (tests MC-F14 to F16), produced copper-lead concentrate grades ranging between 26 and 32% Cu with less than 0.44% As. Cleaner copper-lead concentrate mass yields were between 3.6 and 4.2%, and overall copper recovery from 61 to 66% Cu. Zinc misplacement was low for tests MC-F14 and MC-F15, but applying the procedure on master composite 1B (test MC-F16) resulted in 8% zinc misplacement.

Through Aero 5100 collector dosage adjustments in the copper-lead circuit, and with a decrease of pulp pH to 9.5 (test MC-F26), copper recovery to the cleaner concentrate increased to 67%, while zinc misplacement fell to 2.8%. Copper grade to the cleaner concentrate remained consistent at 30% Cu, but the arsenic grade increased slightly to 0.5% As.

At a cleaner pulp pH of 10, the cleaning stages achieved the best concentrate copper recovery of 79% with a grade of 29% Cu. However, arsenic recovery was the highest amongst the Aero 5100 tests, with 26% and 4% reporting to the rougher and cleaner concentrates (Table 13.17). This was unexpected, as this test utilized the same rougher conditions as test MC-F26. The addition of the copper-lead scavenger in the rougher circuit did not have much impact on recovery performance, while the 1st cleaner-scavenger recovered half of the available arsenic and zinc to the 1st cleaner-scavenger concentrate.

Overall, the test conditions for the copper-lead tests achieved concentrate grades between 26-32% Cu. However, zinc misplacement increased in the rougher copper-lead concentrate using the Aero 5100 collector, which was effectively decreased through the cleaning stages. Recoveries were from 11-21% for gold and 38-58% for silver, with test MC-F27 achieving the highest recoveries of the precious metal content.

Figure 13.13: Copper Cleaner Flotation, Copper-Lead Circuit



**Note: a) Cu Grade-Recovery and b) Cu vs. As Recovery; Zinc Circuit: c) Zn Grade-Recovery and d) Zn vs. As Recovery.*

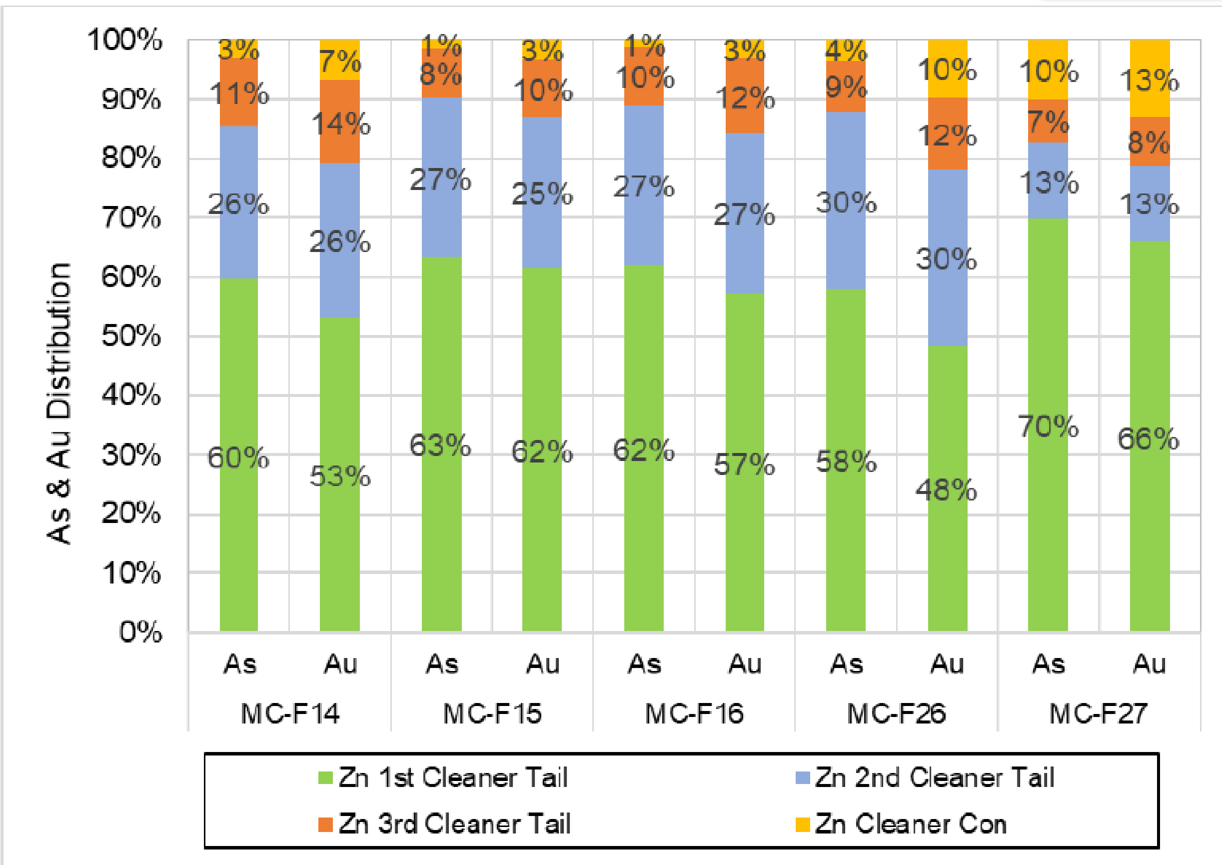
The low recoveries for gold were due to the association of gold with iron sulfides.

The zinc circuit produced cleaner concentrates with grades between 58 and 60% Zn at a wide recovery range of 33-69% recovery. Zinc cleaner concentrate recoveries in tests MC-F15 and MC-F16 were the lowest due to a reduced SIPX dosage, though concentrate grades remained at 58% Zn. The Aero 5100 collector test at pH 11.5 (MC-F26) outperformed the test at pH 11.8 (MC-F27) with respect to zinc cleaning efficiency. The best stage recovery of zinc to the cleaner concentrate was achieved at pH 11.5, with 25 g/t of SIPX (test MC-F14) or 9 g/t of Aero 5100 (test MC-F26).

Arsenic rejection from the zinc cleaner concentrate ranged from 96-99% at pH 11.5, and the performance decreased to 90% arsenic rejection at pH 11.8.

The normalized arsenic and gold rejection profile in the zinc cleaners shown in Figure 13.14 suggests that the gold deportment closely follows the arsenic deportment.

Figure 13.14: Normalized Arsenic Rejection in Zinc Cleaner Circuit

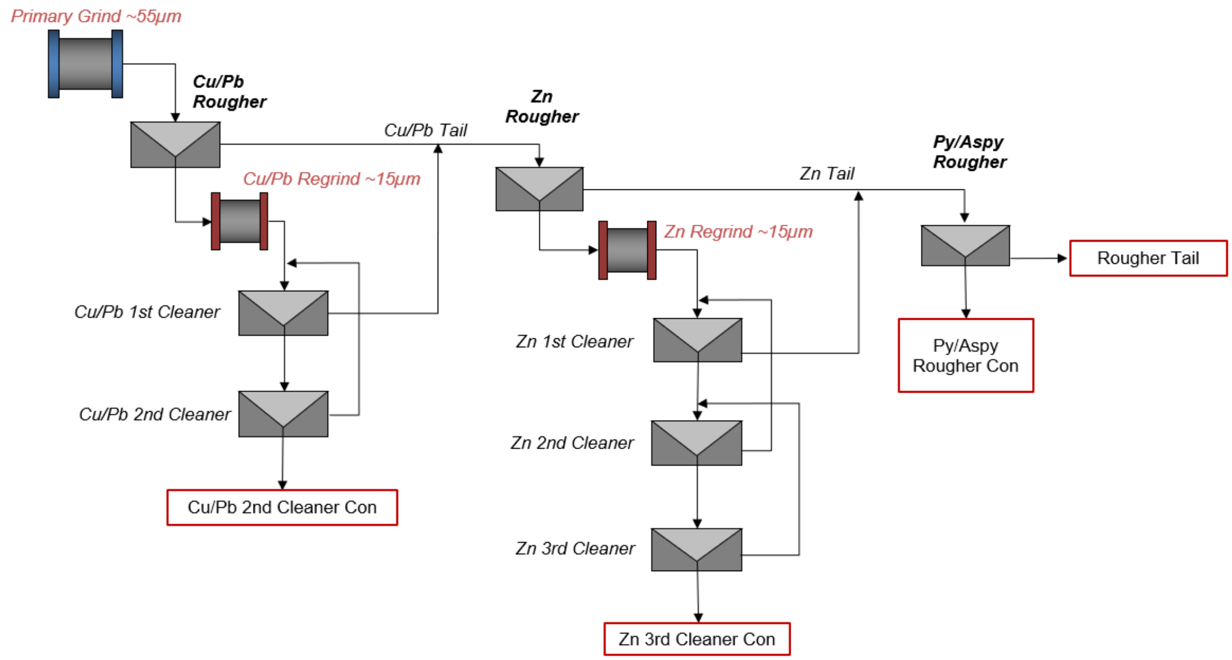


13.7 <u>Locked Cycle Tests</u>

A five-cycle closed circuit flotation test (test MC-LCT1) was performed with rougher and cleaner conditions adopted from test MC-F22 and MC-F26, respectively. The purpose of the test was to determine the effects of circulating streams on the final copper-lead and zinc concentrate grades and recoveries at steady state.

Copper-lead cleaning was carried out in two (2) stages (instead of three (3)), and the target regrind 80% passing sizes for both the copper-lead and zinc circuits were 15 µm. Aero 5100 collector was used in both circuits for its selectivity against iron sulfides, and SIPX was only used in the pyrite flotation circuit. The MC-LCT1 flowsheet is presented in Figure 13.15, and the detailed procedure can be found in the SGS Report 18426-01A.

Figure 13.15: Locked Cycle Test (MC-LCT1) Flowsheet



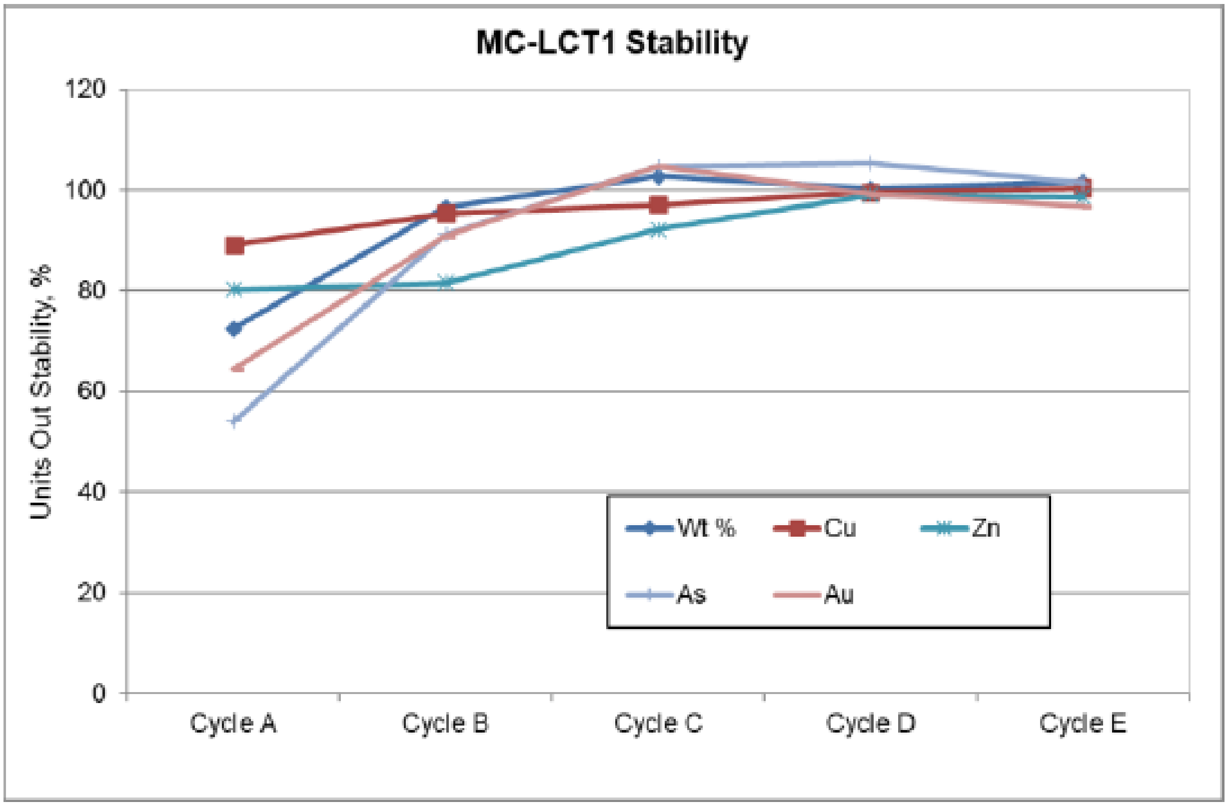
Source: SGS Testwork Report

Table 13.21 presents a summary of the LCT results. Results from Cycle C-E were used for developing the mass balance as the test attained stability for the last three (3) cycles, as shown in Figure 13.16. The LCT performed well, producing a copper concentrate with 88% copper recovery and 27.1% Cu grade. The zinc concentrate achieved 76% recovery with a zinc grade of 58.7%.

Table 13.21: Locked Cycle Flotation Test (MC-LCT1) Summary for Cycles C-E

Product	Weight	Assays %, g/t								% Distribution							
	%	Cu	Fe	Pb	Zn	S	As	Au	Ag	Cu	Fe	Pb	Zn	S	As	Au	Ag
Cu/Pb 2nd Cleaner Con	5.7	27.1	27.2	3.32	4.24	33.7	0.98	7.8	527.7	88.3	8.2	50.0	5.7	11.2	3.1	20.8	66.8
Zn 3rd Cleaner Con	5.5	0.43	5.95	0.60	58.7	34.0	1.31	1.71	47.7	1.3	1.7	8.6	75.9	10.8	4.0	4.4	5.8
Pyrite Rougher Con	31.4	0.38	35.1	0.37	2.10	37.1	4.29	4.23	29.3	6.8	57.8	30.4	15.6	67.9	74.4	62.0	20.3
Pyrite Rougher Tail	57.4	0.11	10.7	0.07	0.21	3.00	0.59	0.48	5.61	3.5	32.3	11.0	2.8	10.0	18.6	12.8	7.1
Head (calc)	*100*	*1.76*	*19.1*	*0.38*	*4.24*	*17.2*	*1.81*	*2.14*	*45.3*	*100*	*100*	*100*	*100*	*100*	*100*	*100*	*100*
Head (direct)		*1.70*	*18.4*	*0.42*	*4.19*	*17.2*	*1.92*	*2.24*	*48.5*								

Figure 13.16: Locked Cycle Test (MC-LCT1) Stability



Subsamples from each cycle of the copper-lead and zinc cleaner concentrates were combined and submitted for multi-element assays. The detailed assay results are appended, while the common penalty elements measured in the respective concentrates are summarized in Table 13.22. The elevated levels of arsenic and mercury in the cleaner concentrates are of concern from the smelter contract standpoint. SGS recommends evaluation by the hydrometallurgical processes route to reduce these grades to acceptable levels.

Table 13.22: Summary of Multi-Element Scan on Copper and Zinc Concentrates

Product	As g/t	Hg g/t	Bi g/t	Cd g/t	Sb g/t	F %
MC-LCT1 Cu 2nd Cleaner Con (A-E)	9,510	68	23.2	87.8	9,740	< 0.005
MC-LCT1 Zn 3rd Cleaner Con (A-E)	13,000	256	2	1,490	489	< 0.005

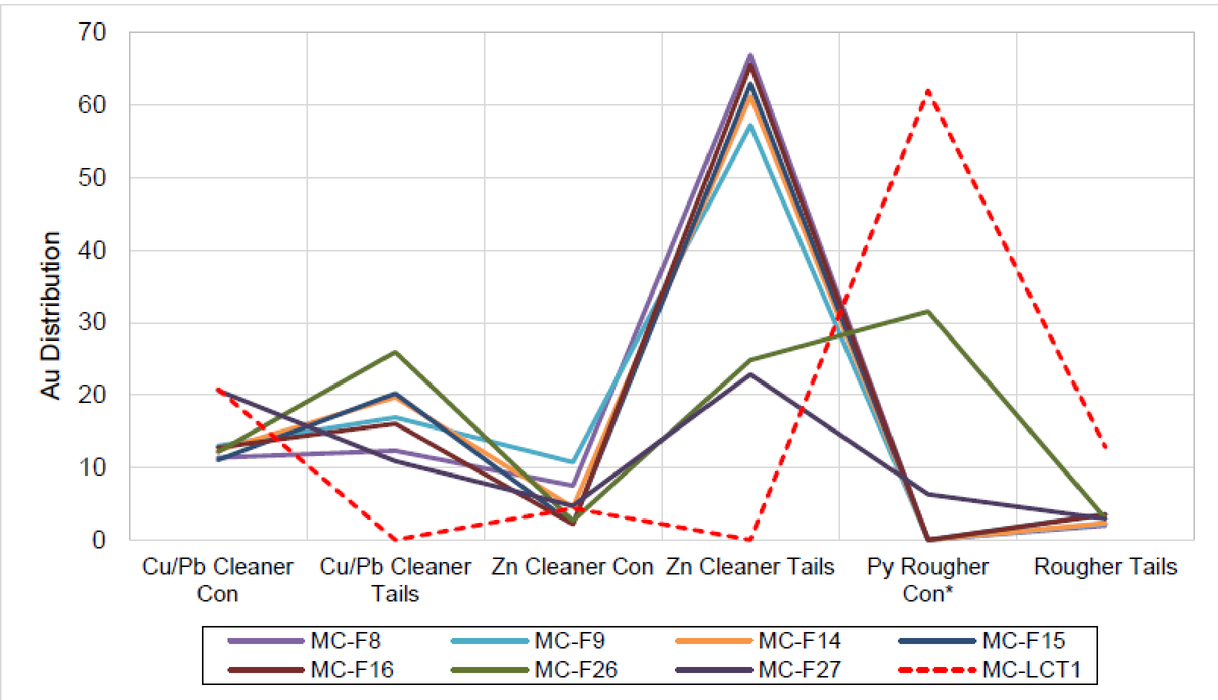
13.8 Solid-Liquid Separation

No filtration or thickening testwork has been undertaken to date. For the purposes of this study, industry-standard design parameters have been applied, benchmarked against comparable operations.

13.9 Gold Distribution in Testwork Products

Gold is primarily concentrated with 57-67% gold in the zinc cleaner tails in the open circuit batch tests, in the absence of the pyrite flotation circuit. Consequently, initial refractory gold testwork focused on the zinc cleaner tailings. A shift in the distribution of gold was observed in the locked cycle test, where the gold in the copper-lead and zinc cleaner tailings is recovered in the pyrite concentrate, as shown in Figure 13.17. Subsequent gold extraction testwork was performed on the pyrite concentrate.

Figure 13.17: Gold Distribution in Open Batch Cleaner and Locked Cycle Flotation



13.10 Investigation of Gold Recovery

An initial bottle roll test performed on the MC-F8 zinc cleaner tailings achieved only 15% gold dissolution over 72 hours using relatively intensive cyanidation conditions. Considering the ultrafine nature and iron sulfide association of the gold, dissolution through standard cyanidation was predictably unsuccessful.

Recovering the gold will require refractory gold processing options to oxidize the sulfide minerals and liberate the locked gold particles. After the initial testing, the Albion treatment process for oxidation of the sulfides was tested on the MC-LCT1 pyrite concentrate and achieved positive gold and silver dissolution results. Lastly, cyanidation of the MC-LCT1 final tailings was performed to assess the recovery of the remaining gold and silver, which were not recovered by flotation.

13.11 Zinc Cleaner Tailings Cyanidation

The zinc cleaner tails from the flotation test MC-F8 were combined to form a single composite. The weights of each component in the blend are detailed in Table 13.23, and the measured P_{80} particle size was 18 µm. Cyanidation (test MC-C1) was performed using 2 g/L NaCN and 250 g/t Pb $(NO_3)_2$, with a pulp density of 40% solids (w/w). Oxygen was sparged into the pulp, maintaining dissolved oxygen (DO) levels above 20 mg/L throughout the test. The pH was kept at 10.5-11 using lime (CaO). Solution kinetic samples were analyzed for gold after 8, 24 and 48 hours, with the final PLS and residue samples taken at the 72-hour test termination. Final extractions were 15% gold, along with 24% Cu and 10% Zn in the PLS. The respective cyanide and lime consumption was 7.9 kg/t and 2.1 kg/t over 72 hours.

Table 13.23: MC-F8 Combined Zinc Cleaner Tailings – Cyanidation Feed Sample

Sample	Mass (g)
Zn Cleaner 1 Tail	415.8
Zn Cleaner 2 Tail	117.3
Zn Cleaner 3 Tail	22.2
Total	**555.3**

The assays and extraction summary in Table 13.24 show the cyanide leachable gold extracted within the first 8 hours.

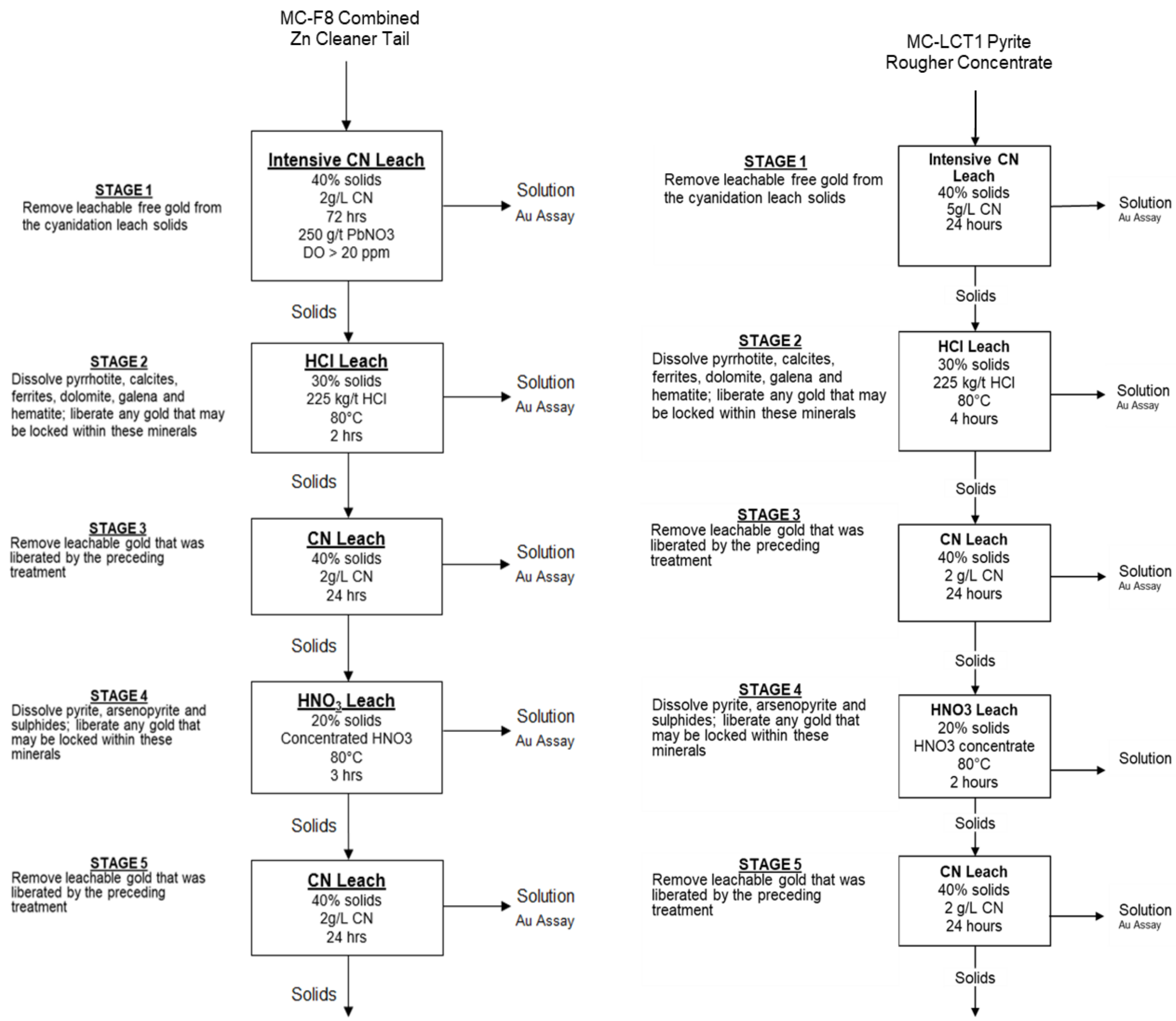
Table 13.24: Zinc Cleaner Tailings Cyanidation Assays and Extraction (Test MC-C1)

Product	Assays					Extraction				
	Au g/t	As %	Cu %	Zn %	Fe %	Au g/t	As %	Cu %	Zn %	Fe %
Head (direct)	4.36	4.49	0.46	1.63	34.9					
8hr PLS	0.46	-	0.05	0.04	0.0002	16.1	-	19.0	4.8	-
24hr PLS	0.44	-	0.05	0.06	0.0004	15.4	-	20.5	6.7	-
48hr PLS	0.42	-	0.06	0.07	0.0007	14.7	-	21.4	8.1	-
72hr PLS	0.44	-	0.06	0.09	0.0010	15.4	-	24.0	9.9	-
Residue	3.69	4.42	0.31	1.23	33.8					
Head (calc)	4.36		0.41	1.37	33.8					

13.12 Diagnostic Leach

Diagnostic leach tests were conducted on the MC-F8 zinc cleaner tailings cyanidation residue and the MC-LCT1 pyrite rougher concentrate to assess the deportment of their gold content and the extent to which it may be refractory. The 5-stage leach details and objectives from each stage are presented in Figure 13.18.

Figure 13.18: Diagnostic Leach Test Flowsheet



Test results on the zinc cleaner concentrate are summarized in Table 13.25. The overall gold extracted from the zinc cleaner tailings was 83.4%, indicating that 16.6% remained locked in silicates. Of the leached gold, 65.1% was locked in sulfides, 2.5% was locked in carbonates, and 15.4% was free milling.

Table 13.25: Diagnostic Leach Results on Zinc Cleaner Tailings

Test #	Sample ID	Test Stages	Head Au Grade		Leach Residue		PLS**
			Direct (g/t)	Calc. (g/t)	Au Assay (g/t)	Au Distribution* (%)	Overall Au Extraction (%)
MC-DL1	MC-F8 Combined Zn Cleaner Tail	Stage 1 - CN Leach (test C1)	4.36	4.36	3.69	84.6	15.4
		Stage 2 - HCl Leach	3.69	4.23	4.22	99.7	0.2
		Stage 3 - CN Leach	4.23	4.20	4.08	97.1	2.5
		Stage 4 - HNO_3 Leach	4.08	3.75	20.1	99.8	0.1
		Stage 5 - CN Leach	20.1	18.7	3.91	20.3	65.1
		Au Overall Extraction (i.e. during D.Leach only)					**83.4**
		Au Remaining in D.Leach Residue					**16.6**

**Note: *Au distribution in solid at each stage; ** PLS - Pregnant Leach Solution.*

Duplicate diagnostic leach tests were conducted on the pyrite concentrate, and the results presented in Table 13.26 show that the average gold extracted from the pyrite concentrate was 78.9%, of which 63.3% was locked in sulfides, 0.9% locked in carbonates, and 14.1% was free milling.

Table 13.26: Diagnostic Leach Results on Pyrite Concentrate

Test #	Sample ID	Test Stages	Head Au Grade		Leach Residue		PLS**
			Direct (g/t)	Calc. (g/t)	Au Assay (g/t)	Au Distribution* (%)	Overall Au Extraction (%)
MC-DL2	MC-LCT1 Combined Pyrite Concentrate (A-E)	Stage 1 - Intensive CN Leach	3.98	3.93	3.40	86.0	14.0
		Stage 2 - HCl Leach	3.40	3.42	3.89	99.4	0.5
		Stage 3 - CN Leach	3.89	3.93	3.93	99.0	0.8
		Stage 4 - HNO_3 Leach	3.93	3.70	11.8	99.7	0.3
		Stage 5 - CN Leach	11.8	11.8	3.16	26.2	62.3
		Au Overall Extraction (i.e. during D.Leach only)					**77.9**
		Au Remaining in D.Leach Residue					**22.1**
MC-DL3	MC-LCT1 Combined Pyrite Concentrate (A-E)	Stage 1 - Intensive CN Leach	3.98	3.99	3.44	85.8	14.2
		Stage 2 - HCl Leach	3.44	3.33	3.90	99.7	0.2
		Stage 3 - CN Leach	3.90	3.83	3.82	99.0	0.9
		Stage 4 - HNO_3 Leach	3.82	3.76	11.5	99.7	0.2
		Stage 5 - CN Leach	11.5	11.8	2.85	23.8	64.3
		Au Overall Extraction (i.e. during D.Leach only)					**79.9**
		Au Remaining in D.Leach Residue					**20.1**

**Note: *Au distribution in solid at each stage; ** PLS - Pregnant Leach Solution.*

An average of 21.1% of the gold remained in the diagnostic leach residue (typically locked in silicates); this was unexpectedly high, as the gold mineralogy study suggested gold was not associated with silicates. It was suspected that the standard diagnostic leach procedure possibly provided only partial decomposition of the relatively high sulfide content. The stage 5 residues were therefore submitted for sulfide-sulfur and tellurium assays, and the results are presented in Table 13.27. The assays show a considerable amount of undissolved sulfide content remains in the final diagnostic leach residue, with gold tellurides potentially contributing to incomplete gold dissolution. The actual amount of gold associated / locked in silicates is therefore likely to be lower than the ~21% indicated from the diagnostic leach tests. A more aggressive diagnostic leach procedure should be considered for future testwork.

Table 13.27: Sulfide-Sulfur and Tellurium Assays

Sample ID	S=%	Te g/t
MC-DL2 Residue	26.0	10.6
MC-DL3 Residue	27.9	12.2

13.13 Albion Oxidate Treatment and Cyanidation

Scoping Albion pretreatment testing was performed on the MC-LCT1 pyrite concentrate to assess the potential of oxidizing the sulfide minerals and releasing the refractory gold for cyanidation. Based on instructions from Glencore Technology, the feed sample was subject to a sodium assisted neutral Albion leach, where the sample was ultrafine ground to 10 µm (P_{80}), the pulp was adjusted to and maintained at pH 4.5 with $CaCO_3$, the temperature was kept at a target of 95ºC, and oxygen was continuously supplied to the leach reactor at 1 L/min, with agitation for 72 hours. The detailed Albion procedure and test data can be found in SGS Report 18426-01A.

The $CaCO_3$ and H_2SO_4 consumptions for the scoping Albion test were high, measuring 899 kg/t and 151 kg/t (pyrite concentrate feed), respectively. Glencore noted that no efforts were made towards reagent optimization for the initial test, as the objective was to determine the general amenability of the Albion process for the pyrite concentrate. The sulfide oxidation rate, calculated from the Albion feed and residue, was 91%. Due to the large quantity of $CaCO_3$ addition and its conversion to gypsum, the weight of the combined residues increased from a feed weight of 750 g to 1,838 g and diluted the gold grade to 1.45 g/t for the final residue. As shown in Table 13.28, assays on the Albion kinetic solution samples measured below the detection limit of 0.01 mg/L Au, and the gold content remained in the solids amongst the kinetic residues, scum residue, and final residue samples.

Table 13.28: Albion Oxidative Treatment - Gold Balance

Product	Weight / Volume g/mL	Gold	
		Assay g/t, mg/L	Distribution %
24 h Final Preg Solution	475.5	< 0.01	-
24 h Residue	82.9	2.10	6.2
48 h Final Preg Solution	446.7	< 0.01	-
48 h Residue	91	1.81	5.9
72 h Scum Preg Solution	1,826.2	< 0.01	-
72 h Scum Residue	445.5	1.58	25.1
72 h Final Preg Solution	4,897.6	< 0.01	-
72 h Final Residue	1,218.4	1.45	62.9
Head (calc.)	750	3.75	100.0
Head (dir.)		3.90	

The Albion test residue was subject to a two-stage cyanidation with the following conditions as defined by the client:

- Stage 1:
  - Adjust pulp to 30% solids (w/w), adjust and maintain pH to target 11.5 with CaO, add 500 g/t $Pb(NO_3)_2$, and pre-aerate for four (4) hours with oxygen addition.
  - Leach with 2 g/L NaCN (maintained) for 48 hours with oxygen addition.
  - Test ended at 48 hours, and the pulp was filtered with three (3) displacement washes. The 48-hour PLS and residue subsamples were submitted for gold and silver assays.
- Stage 2:
  - The Stage 1 residue was repulped to 30% solids (w/w), adjusted pH to target 12 with NaOH, added 500 g/t $Pb(NO_3)_2$, and pre-aerated for 16 hours.
  - Leach with 2 g/L NaCN (maintained) for 24 hours with oxygen addition.
  - Test ended at 24 hours, and the pulp was filtered with three (3) displacement washes. The 24-hour PLS and residue subsamples were submitted for gold and silver assays.

As shown in Table 13.29, cyanidation of the Albion-treated samples extracted 98% Au and 94% Ag from the pyrite rougher concentrate. The dissolution of gold was complete after the Stage 1 – 48-hour leach. The

Stage 2 leach was able to extract an additional 21% Ag from the Stage 1 cyanidation residue. Overall, the test results show that the pyrite rougher concentrate is amenable to the Albion oxidation process with respect to gold and silver extraction performance. This implies almost all the ~21% gold remaining after the diagnostic leach procedure was also liberated by the Albion oxidation process.

Table 13.29: Cyanidation of Albion Treated Sample

Product	Assays		Extraction		Reagent Addition		Reagent Consumption	
	Au g/t, mg/L	Ag g/t, mg/L	Au %	Ag %	NaCN kg/t	CaO/NaOH* kg/t	NaCN kg/t	CaO kg/t
Stage 1: 48 Hours PLS	0.63	3.45	98.0	73.0	7.99	6.69	3.68	6.63
Stage 2: 24 Hours PLS	<0.05	0.66	0.0	21.0	5.38	5.49*	9.08	-
Total: 72 Hours PLS	0.33	2.06	98.0	94.0	-	-	-	-
Final Residue	0.04	0.50	2.0	6.0	-	-	-	-

13.14 Rougher Tailings Cyanidation

The rougher tailings from test MC-LCT1 (cycles C-E) were combined in equal portions to form a single composite with a measured P_{80} particle size of 51 µm. Cyanidation (test MC-C2) was performed on the rougher tailings composite using the same conditions as test MC-C1 (zinc cleaner tailings leach). Solution kinetic samples were analyzed for gold and silver after 4, 24 and 48 hours, with the final PLS and residue samples taken at the 72-hour test termination. Final extractions were 28% gold and 68% silver in the PLS. The respective cyanide and lime consumption were 2.9 kg/t and 0.9 kg/t over 72 hours. The assays and extraction summary in Table 13.30 shows most of the cyanide leachable gold and silver were extracted within the first 4 and 8 hours, respectively.

Table 13.30: Rougher Tailings Cyanidation Assays and Extractions (Test MC-C2)

Product	Assays		Extraction	
	Au g/t	Ag g/t	Au %	Ag %
Head (direct)	0.48	5.60		
4hr PLS	0.09	1.80	25.6	47.6
24hr PLS	0.10	2.34	28.5	62.0
48hr PLS	0.10	2.42	28.5	64.1
72hr PLS	0.10	2.59	28.4	68.5
Residue	0.38	1.80	71.6	31.5
Head (calc)	0.53	5.71		

13.15 Summary of Metallurgical Recoveries

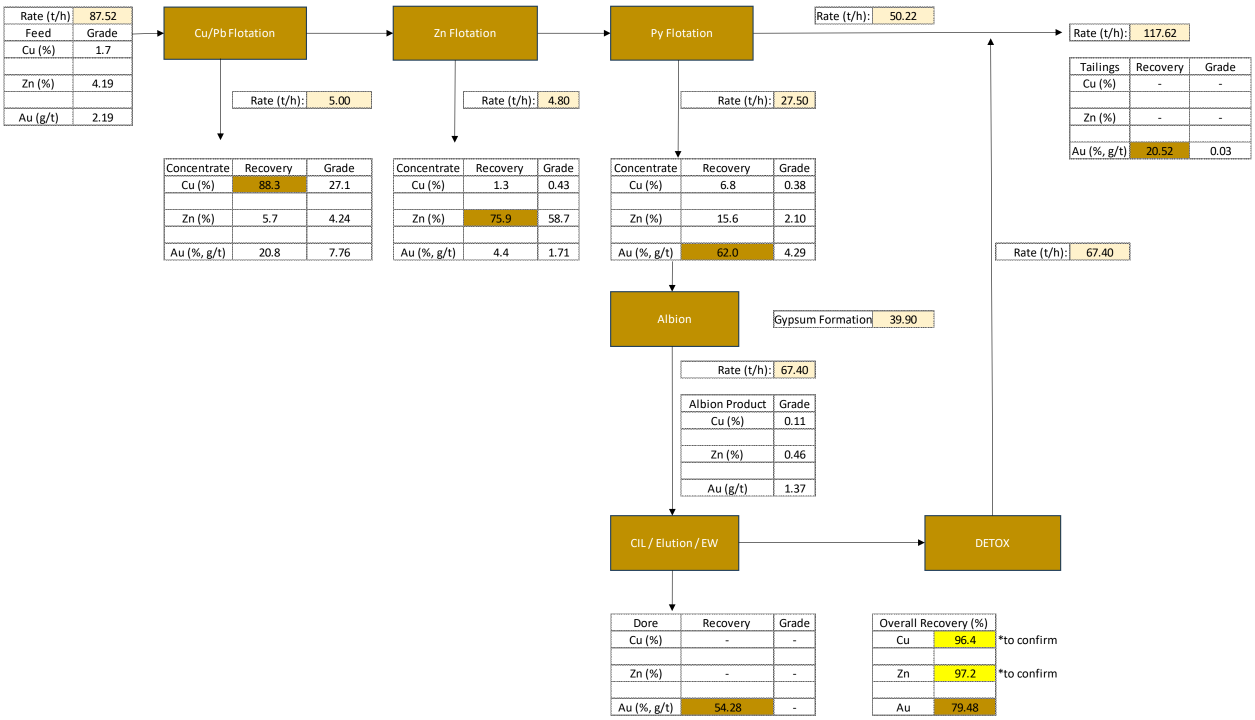
As shown in Figure 13.19, the metallurgical testwork has demonstrated that copper, lead, and zinc can be recovered into saleable concentrates, with flotation achieving up to 88% Cu recovery at concentrate grades of ~27% Cu, and zinc recoveries of up to 76% at ~59% Zn grade. Silver showed moderate recoveries to the Cu-Pb concentrate (38–58%), while gold recoveries to conventional concentrates were low, reflecting the refractory nature of the mineralization.

The primary challenge identified is the very fine gold grain size (mean ~2 µm; 100% passing 6 µm), with much of the gold locked within pyrite and arsenopyrite. Without an oxidation step, such as Albion, overall gold recovery remained low, and the pyrite concentrate assays below 5 g/t Au, which is not commercially attractive. Incorporation of Albion oxidative pretreatment followed by cyanidation achieved gold recoveries of ~98% and silver recoveries of ~94%, confirming that refractory treatment is essential for economic performance.

In addition to the refractory gold, deleterious elements present further metallurgical challenges. Arsenic levels of 0.4–0.5% in the Cu concentrate can represent a marketing issue. Mercury, though present at lower levels, also requires careful monitoring, as even trace amounts can result in penalties or rejections in concentrate sales. Both elements will require ongoing evaluation to determine whether selective flotation strategies, blending, or downstream treatment options can mitigate their impact.

Overall, the testwork indicates that base metal recoveries are within expected ranges, and that achieving acceptable gold recovery requires the inclusion of a refractory gold treatment stage.

Figure 13.19: Stage-wise and Overall Recoveries



14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

The following section describes the MRE for the Kay Deposit. Completion of the current MRE involved the assessment of a drill hole database, which included all data for surface drilling completed to June 17, 2025. Completion of the current MRE also included updated three-dimensional (3D) mineral resource models (resource domains), a 3D topographic surface model, 3D models of historical underground workings, and available written reports.

The Inverse Distance Squared ("ID2") calculation method restricted to mineralized domains was used to interpolate grades for Au (g/t), Ag (g/t), Cu (ppm), Pb (ppm) and Zn (ppm) into a block model for the Kay Deposit.

Indicated and Inferred mineral resources are reported in the summary tables in Section 14.10. The MRE presented below takes into consideration that the Kay Deposit may be mined by underground mining methods.

The reporting of the current MRE complies with all disclosure requirements for Mineral Resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects. The classification of the MRE is consistent with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definitions). In completing the updated MRE, the Author uses procedures and methodologies that are generally consistent with industry standard practices, including those documented in the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019 CIM Guidelines).

14.2 Drill Hole Database

To complete the current MRE for the Kay Deposit, a validated drill hole database comprising a series of comma delimited spreadsheets containing surface diamond drill hole information was provided by Arizona Metals. The database included hole location information, down-hole survey data, assay data for all metals of interest, lithology data and density data. The data in the geochemistry / assay tables included data for the elements of interest including Ag (g/t), Au (g/t), Pb (ppm), Zn (ppm) and Cu (ppm). After review of the database, the data was then imported into GEOVIA GEMS version 6.8.3 software ("GEMS") for statistical analysis, block modeling and resource estimation. No errors were identified when importing the data. The data was validated in GEMS, and no erroneous data, data overlaps or duplication of data was identified.

The updated database provided by Arizona Metals for the MRE included data for 233 surface diamond drill holes completed on the Property, totalling 133,912 m (Table 14.1) (Figure 14.1 and Figure 14.2). The database totals 11,533 assay intervals representing 14,066 m of drilling. The average assay sample length is 1.21 m.

The database was checked for typographical errors in drill hole locations, down-hole surveys, lithology, assay values and supporting information on source of assay values. Overlaps and gapping in survey, lithology and assay values in intervals were checked. All assays had analytical values for Ag (g/t), Au (g/t), Pb (ppm), Zn (ppm) and Cu (ppm).

Table 14.1: Project Drill Hole Totals

Deposit Area	Drill Holes	Drill Hole #	Total Length (m)	No. of Assays	Tot. Assay Length (m)	Avg. Assay Length (m)	SG Values
Kay Deposit	233	KM-20-01 – KM-25-181	133,912	11,533	14,006	1.21	2,307

Figure 14.1: Plan View - Distribution of Surface Drill Holes on Property (WGS 84), on Topography

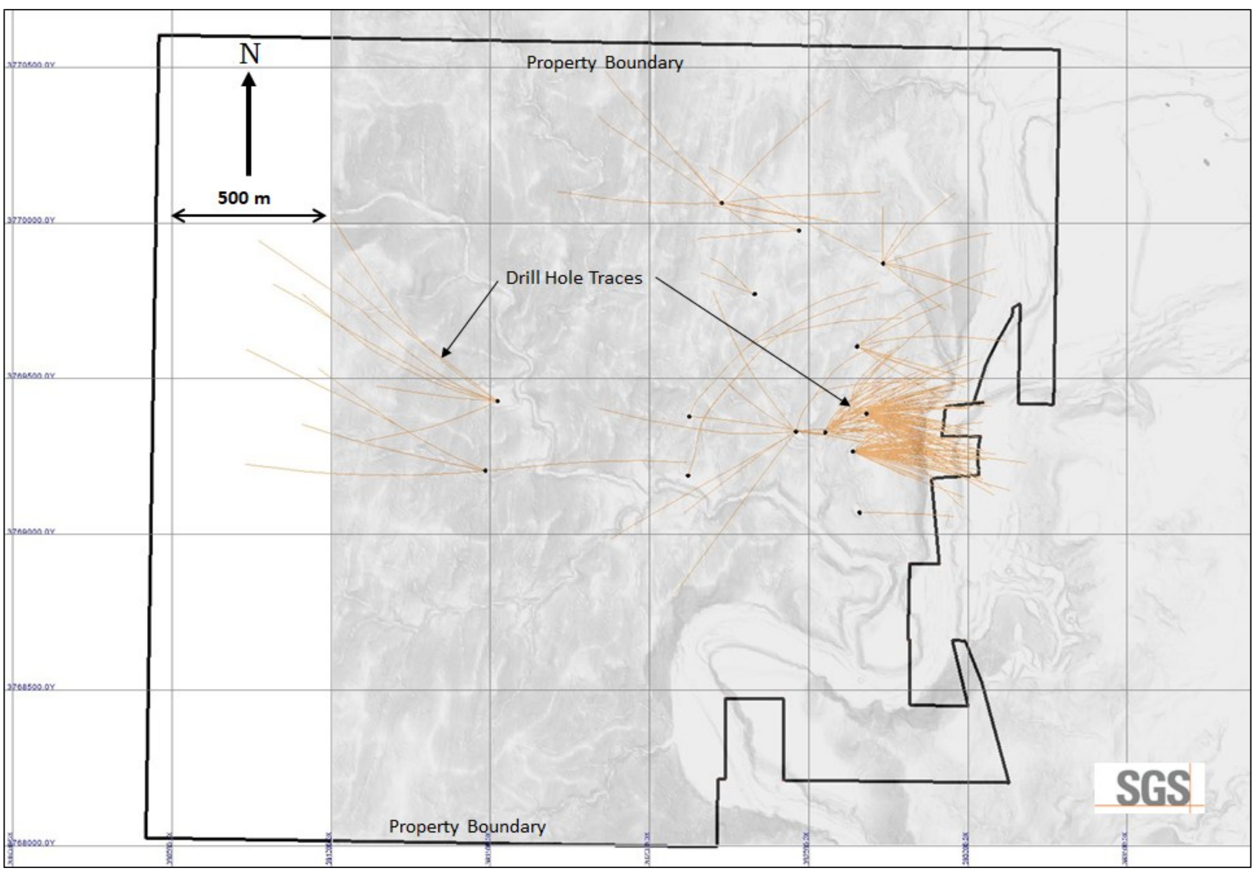
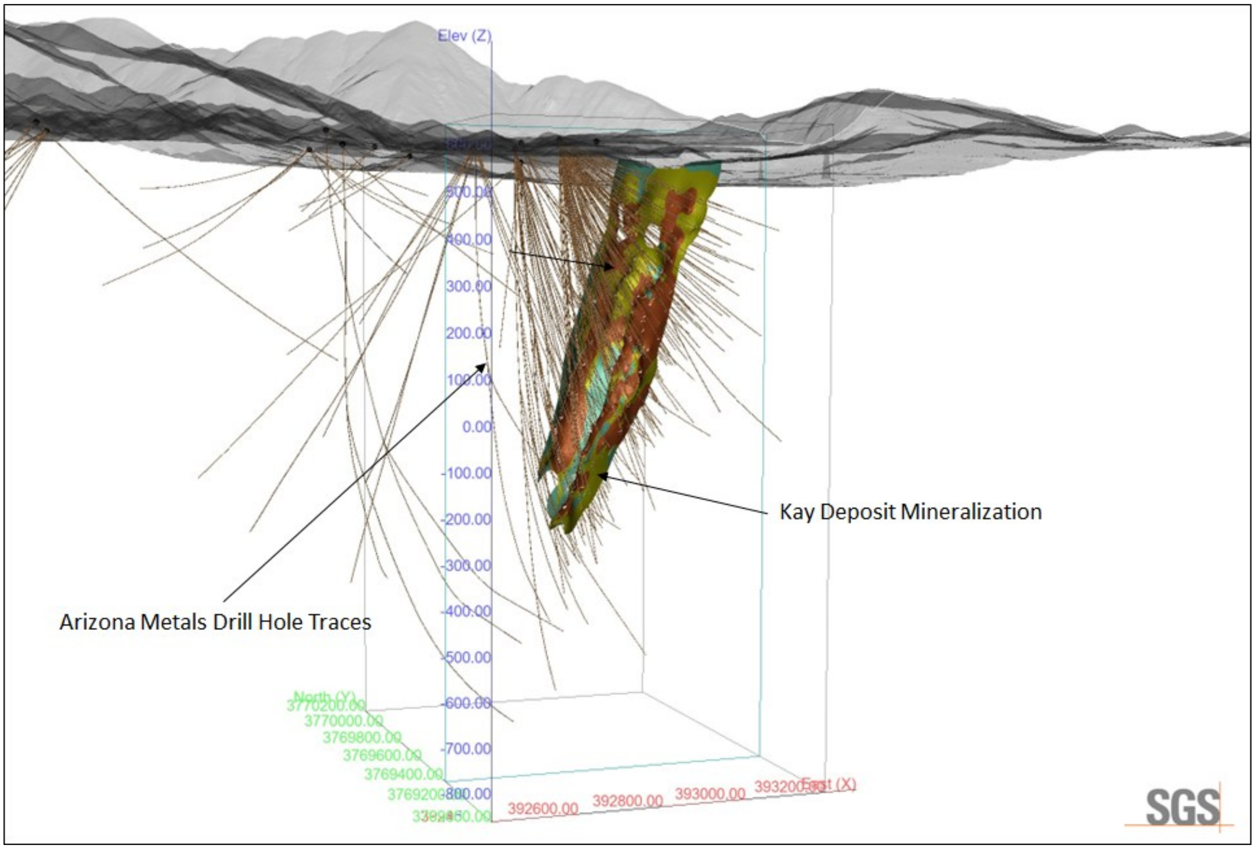


Figure 14.2: Isometric View Looking Northeast - Distribution of Surface Drill Holes in the Kay Deposit Area (WGS84)



14.3 <u>Mineral Resource Modelling</u>

For the current MRE, in collaboration with Arizona Metals, SGS constructed two (2) three-dimensional ("3D") resource models and four (4) lithology models for the Kay deposit (Table 14.2) (Figure 14.3 to Figure 14.7) in Leapfrog Geo version 2025.1.0.

Host rock lithology models were constructed incorporating drilling data, surface mapping, and structural interpretations in addition to SGS field and drill core observations. Lithology models comprise the Hangingwall Mafic Sequence (MVS), Felsic Volcanic Sequence (FVS), Graphite-rich Horizon (GH), and the Mineralization Horizon (MIN-Horizon). The MIN-Horizon model was constructed using the Leapfrog Geo Vein tool from assays greater than 0.5% CuEq and was used to establish the bounding limits of the subsequently constructed resource models. The MIN-Horizon model is consistent with the interpretation that within the property-scale isoclinal folding the sulphide lenses are affected by steeply plunging tight folds (parasitic S-folds).

The Kay drillhole database and drill core was reviewed to evaluate the geological continuity and internal variability with respect to mineralization styles, metal zonation patterns, and density. The deposit displays complex internal variability of mineralization style, density, and relative metal distributions. Mineralization within the MIN-Horizon model was sub-domained using CuEq grade as a proxy for mineralization style and density. Two (2) resource models were constructed: a semi-massive to massive sulphide, high-grade domain (MIN-HG) and a stringer sulphide, low-grade domain (MIN-LG), to domain appropriate density and capping values in the estimation process.

The MIN-HG and MIN-LG resource models were constructed using the Leapfrog Geo Indicator RBF numerical modelling tool with a structural trend based on the folded MIN-Horizon model. The MIN-HG resource model was established from assay intervals above 1.5% CuEq constrained by the MIN-Horizon model. The MIN-LG resource model was established from assay intervals above 0.5% CuEq, outside of the MIN-HG model, and constrained by the MIN-Horizon model.

A digital elevation surface model (LiDAR) was provided for the Property area. All 3D resource models were clipped to topography and limited to the Property boundary.

Mineralization in the Kay sulphide lens resource models extends for up to 400 m along strike and up to 850 m vertically (900 m down plunge). The mineralization horizon in general dips at 73º towards 260º (W) with local variations in strike and dip resulting from steeply plunging tight parasitic folds. The principal plunge direction of the sulphide lenses is 68º towards 300º (WNW) and appears to be influenced in part by steeply plunging tight parasitic folds.

The Author has reviewed the resource models on plan view and in section view and in the Author's' opinion the models are well constructed and appear to be representative of the mineralization identified on the Property and the distribution of the Cu-Au-Zn-Pb-Ag mineralization within these sulphide lenses. Models were reviewed by Arizona Metals during the modelling process and refined by SGS before final resource estimation. Models have been extended beyond the limits of the current drilling for the purpose of providing guidance for continued exploration. However, the extension of the mineral resource beyond the limits of drilling is limited by the search radius during the interpolation procedure (a maximum of 110 m in the plunge direction past drilling).

14.3.1 <u>Specific Gravity</u>

The author was provided with a database of 2,307 SG measurements for the current MRE, including samples from LG and HG mineralization and waste rocks.

Based on a review of the available SG data, it was decided that a fixed value be used for each resource model. The average density used by domain for the current MRE is presented in Table 14.2.

It is recommended that Arizona Metals continue to collect additional SG data as drilling continues. As the SG data collection is restricted to drilling prior to 2025, it is strongly recommended that Arizona Metals go back and collect data from the 2025 drill core.

Table 14.2: Property Domain Descriptions

Model	Rock Code	Block Rock Code	SG
	Gems		
LITH - MIN-HG_1.5	KMHG	1	3.40
LITH - MIN-LG_0.5_1.5	KMLG	2	2.95
LITH - MIN-Horizon	KMHORIZ	103	2.88
LITH - FVS	SCHIST	101	2.80
LITH - GH	GRSCHIST	102	2.85
LITH - MVS	METAVOLC	100	2.90

Figure 14.3: Plan View - Property Geology Models

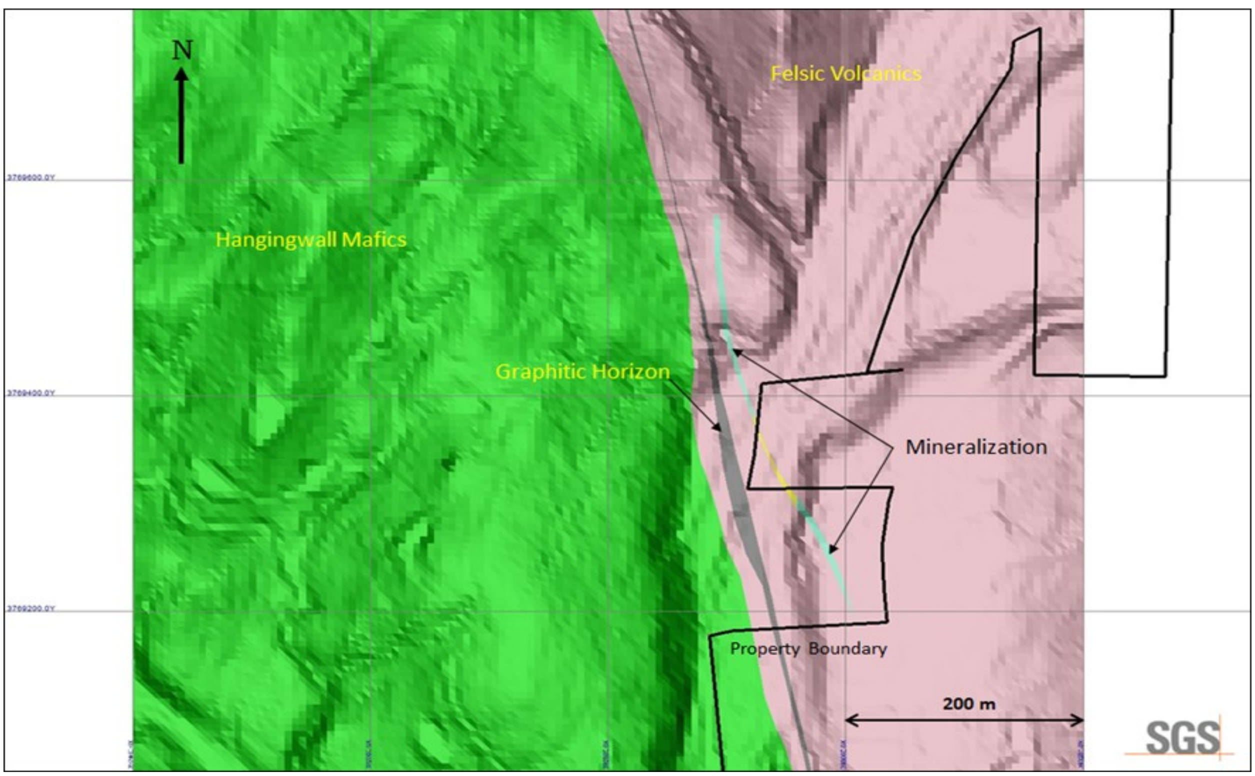
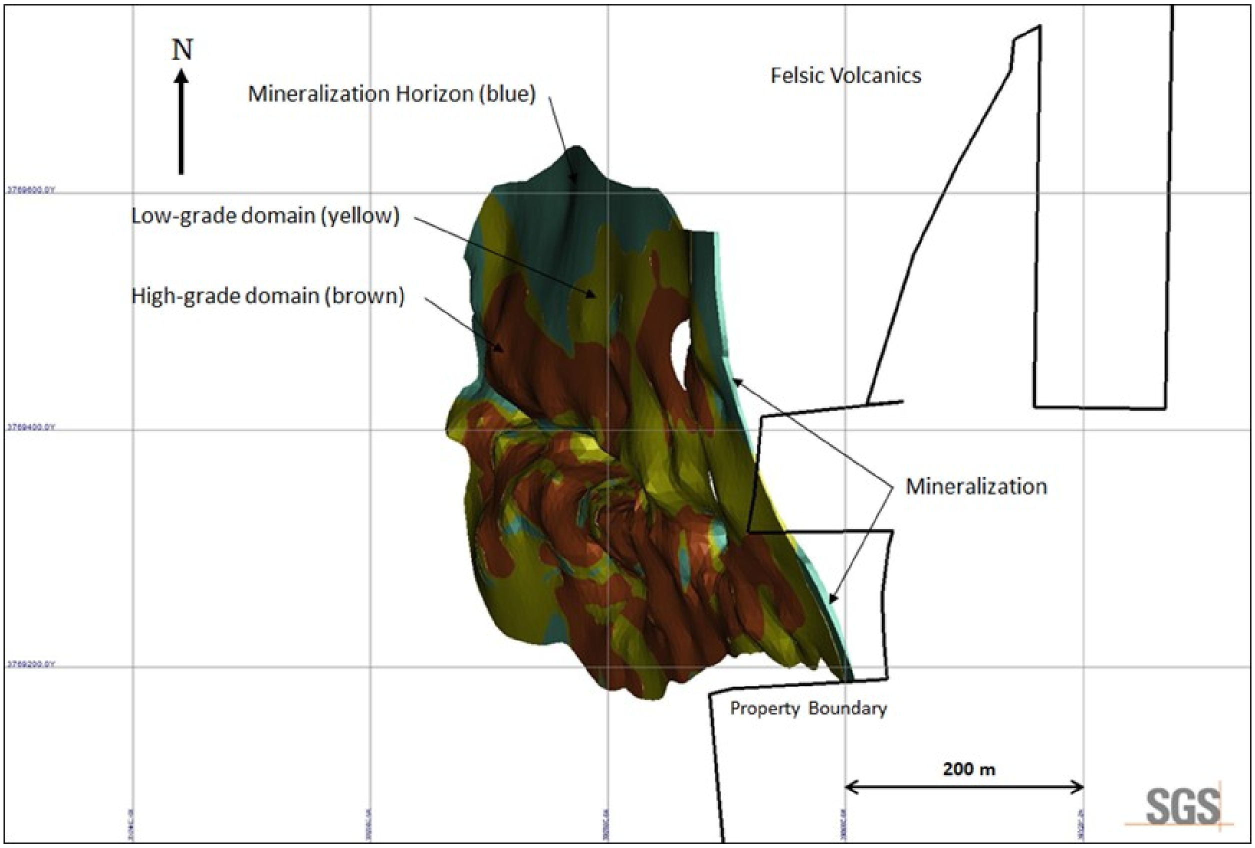


Figure 14.4: Plan View - Property Mineral Resource Models



**Note: Projected intersection of mineralization model with surface; mineralization does not crop out on adjacent properties.*

Figure 14.5: Isometric View Looking NNW: Property Geology Models

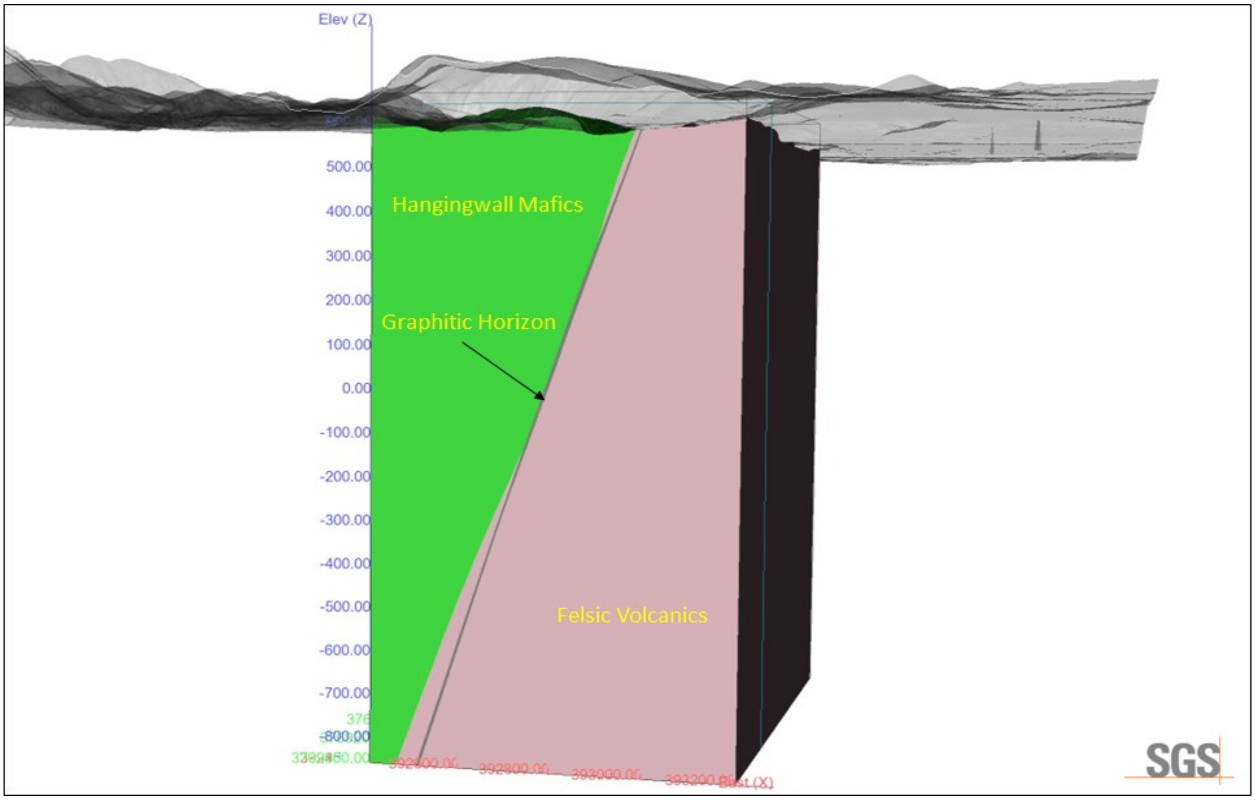


Figure 14.6: Isometric View Looking NNW - Property Mineral Resource Models

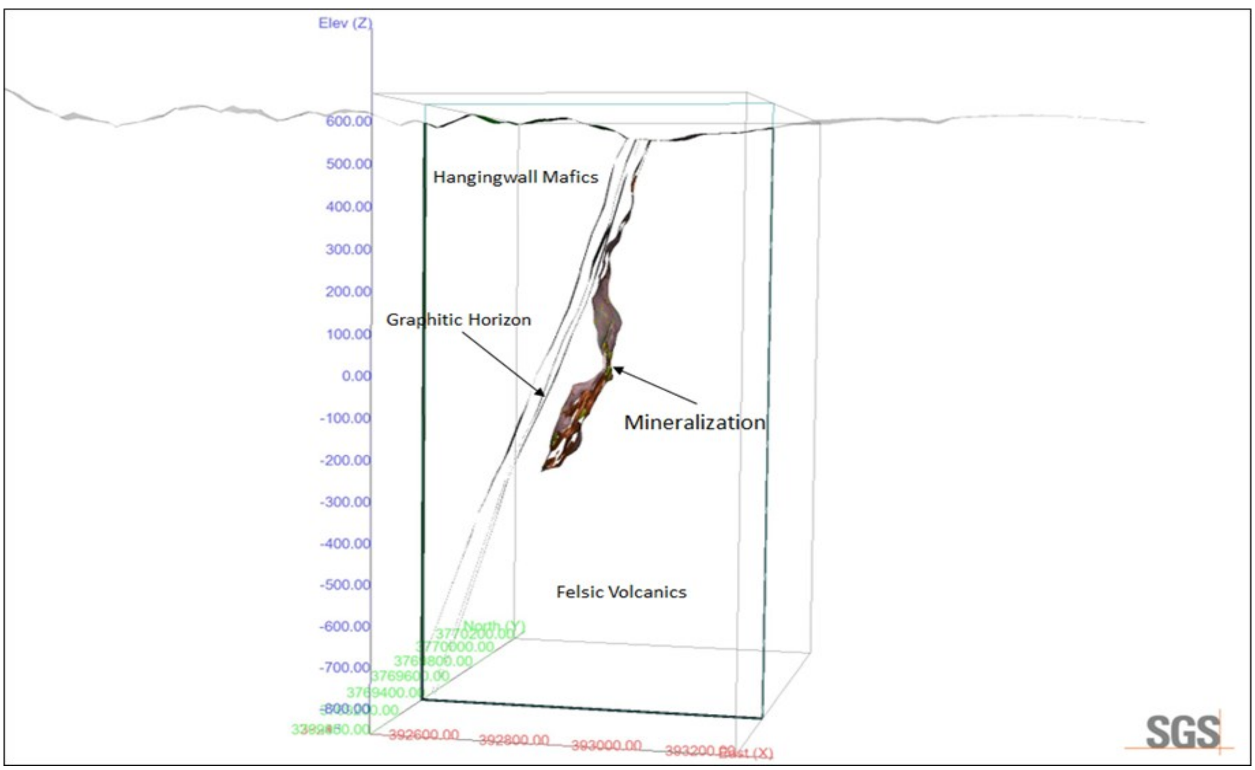
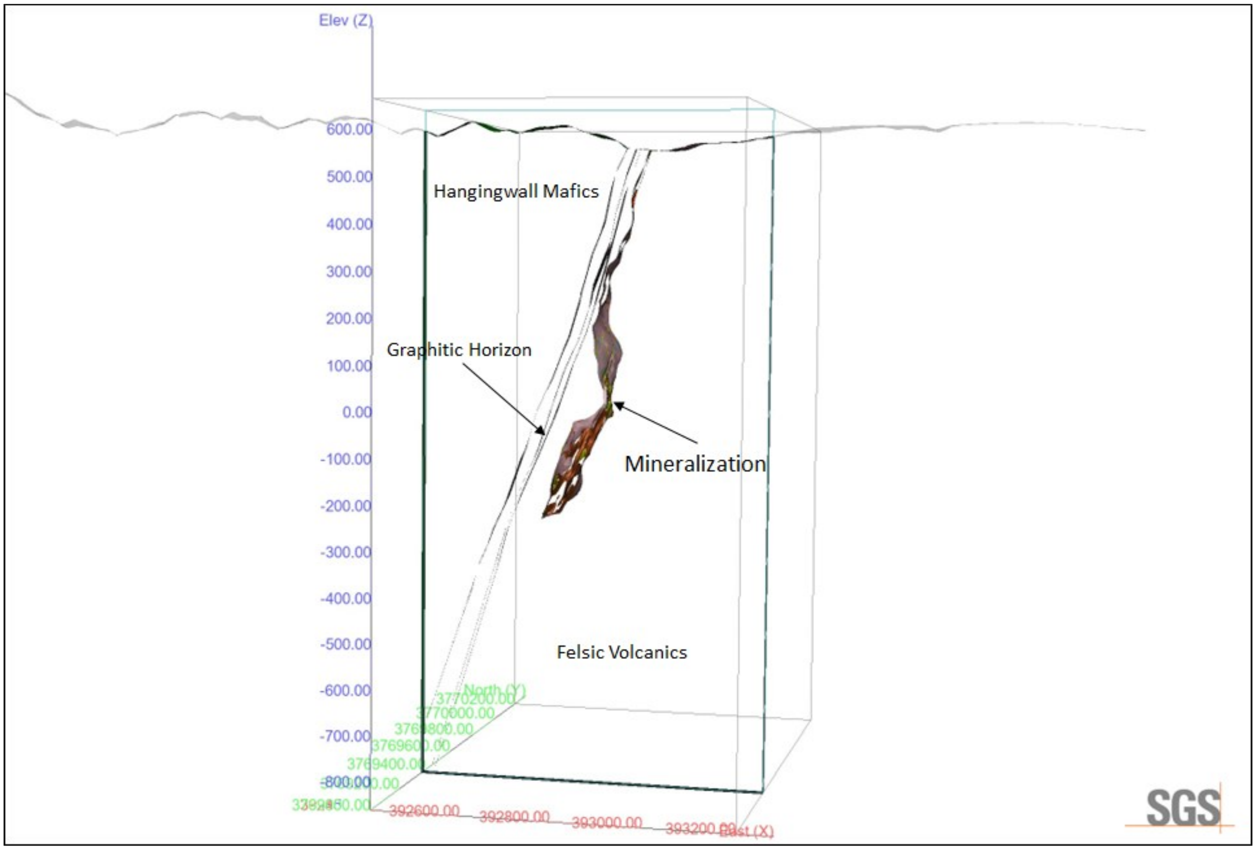


Figure 14.7: Isometric View Looking NNW – Property Mineral Resource Models and Geology Models – Section 3769375N



14.4 Compositing

The assay sample database available for the resource modelling totalled 11,533 samples representing 14,006 m of drilling (Table 14.1). A statistical analysis of the assay data from within the mineralized domains, is presented in Table 14.3 to Table 14.5. There is a total of 3,492 assays within the mineral resource domains.

Table 14.3: Statistical Analysis of Drill Assay Data from Within the Kay Deposit Resource Domains, High Grade Domain

Variable	Au g/t	Ag g/t	Cu ppm	Pb ppm	Zn ppm
Total # Assay Samples	2,159				
Average Sample Length	1.10 m				
Minimum Grade	0.00	0.00	10	10	90
Maximum Grade	273	1,250	207,000	102,000	279,000
Mean	2.19	40.8	14,148	4,781	33,987

Variable	Au g/t	Ag g/t	Cu ppm	Pb ppm	Zn ppm
Standard Deviation	6.62	70.2	24,171	8,489	40,703
Coefficient of variation	3.02	1.72	1.70	1.78	1.20
97.5 Percentile	10.7	195	88,000	28,700	141,250

Table 14.4: Statistical Analysis of Drill Assay Data from Within the Kay Deposit Resource Domains, Low Grade Domain

Variable	Au g/t	Ag g/t	Cu ppm	Pb ppm	Zn ppm
Total # Assay Samples	1,333				
Average Sample Length	1.20 m				
Minimum Grade	0.00	0.00	5.00	10	50
Maximum Grade	21.9	272	106,500	36,200	300,000
Mean	0.34	10.7	3,911	1,030	6,679
Standard Deviation	0.89	18.9	7,756	1,963	12,799
Coefficient of variation	2.60	1.77	1.98	1.91	1.92
97.5 Percentile	1.58	51.5	23,700	5,845	31,950

Table 14.5: Statistical Analysis of Drill Assay Data from Within the Kay Deposit Resource Domains, Low Grade + High Grade Domain

Variable	Au g/t	Ag g/t	Cu ppm	Pb ppm	Zn ppm
Total # Assay Samples	3,492				
Average Sample Length	1.14 m				
Minimum Grade	0.00	0.00	0.00	0.00	0.00
Maximum Grade	273	1,250	207,000	102,000	300,000
Mean	1.48	29.3	10,240	3,349	23,563
Standard Deviation	5.31	58.3	20,222	7,024	35,537
Coefficient of variation	3.58	1.99	1.97	12.10	1.51
97.5 Percentile	8.75	168	73,000	24,250	128,000

The average length of all assay sample intervals is 1.14 m and ranges from 0.06 m to 2.90 m. Of the 3,492 assays, approximately 39% of the assays are >1.25 m; 64% of the assays are >1.00 m. To minimize the dilution and over-smoothing due to compositing, a composite length of 1.50 m was chosen as an appropriate composite length for all areas, for the current MRE.

For the current MRE, composites were generated starting from the collar of each drill hole. Un-assayed intervals were given a value of 0.0001 for Au, Ag, Cu, Pb and Zn. Composites were then constrained to the individual mineral domains. The constrained composites were extracted to point files for statistical analysis and capping studies. The constrained composites were grouped based on the mineral domain (rock code) of the constraining resource model.

A total of 2,688 composite sample points occur within the resource models. A statistical analysis of the composite data from within the mineralized domains, by area, is presented in Table 14.6 to Table 14.8.

Table 14.6: Composite Data Statistical Analysis Within the Kay Deposit Resource Domains, High Grade Domain

Variable	Au g/t	Ag g/t	Cu ppm	Pb ppm	Zn ppm
Total # Assay Samples	1,615				
Average Sample Length	1.50 m				
Average SG	3.36				
Minimum Grade	0.01	0.08	18.3	7.49	13.3
Maximum Grade	185	671	181,469	53,943	217,781
Mean	2.14	38.9	13,334	4,712	33,543
Standard Deviation	5.38	54.9	20,726	7,140	35,283
Coefficient of variation	2.52	1.41	1.55	1.52	1.05
97.5 Percentile	9.35	183	74,027	24,048	128,030

Table 14.7: Composite Data Statistical Analysis Within the Kay Deposit Resource Domains, Low Grade Domain

Variable	Au g/t	Ag g/t	Cu ppm	Pb ppm	Zn ppm
Total # Assay Samples	1,073				
Average Sample Length	1.50 m				
Average SG	2.95				
Minimum Grade	0.00	0.00	0.00	0.00	0.00
Maximum Grade	0.33	156	87,608	19,267	172,797
Mean	0.51	10.3	3,565	1,004	6,254

Variable	Au g/t	Ag g/t	Cu ppm	Pb ppm	Zn ppm
Standard Deviation	1.57	13.6	5,651	1,620	8,941
Coefficient of variation	1.33	1.31	1.58	1.61	1.43
97.5 Percentile	1.33	43.3	16,201	5,163	26,789

Table 14.8: Composite Data Statistical Analysis from Within the Kay Deposit Resource Domains, Low Grade + High Grade Domain

Variable	Au g/t	Ag g/t	Cu ppm	Pb ppm	Zn ppm
Total # Assay Samples	2,688				
Average Sample Length	1.50 m				
Average SG	3.19				
Minimum Grade	0.00	0.00	0.00	0.00	0.00
Maximum Grade	185	671	181,469	63,406	217,781
Mean	1.41	27.5	9,434	3,231	22,650
Standard Deviation	4.28	45.6	17,138	5,915	30,959
Coefficient of variation	3.02	1.66	1.82	1.83	1.37
97.5 Percentile	7.73	146	63,402	20,388	113,410

14.5 Grade Capping

A statistical analysis of the composite database within the resource models (the "resource" population) was conducted to investigate the presence of high-grade outliers which can have a disproportionately large influence on the average grade of a mineral deposit. High-grade outliers in the composite data were investigated using statistical data (Table 14.6 to Table 14.8), histogram plots, and cumulative probability plots of the composite data. The statistical analysis was completed by deposit area and was completed using GEMS.

After review, it is the opinion that capping of high-grade composites to limit their influence during the grade estimation is necessary for Au, Ag, Cu, Pb and Zn for all domains. A summary of grade capping values within the mineralized domains, by area, is presented in Table 14.9. In the opinion of the author, the capping applied to the deposit composites has had the desired effect of limiting the influence of high-grade outliers on the global MRE. The capped composites are used for grade interpolation into the Kay Deposit block models.

Table 14.9: Composite Capping Summary – by Domain

	Total # of Composites	Attribute	Capping Value	# Capped	Mean of Raw Composites	Mean of Capped Composites	CoV of Raw Composites	CoV of Capped Composites
High Grade Domain	1,615	Au g/t	26.0	4	2.14	2.03	2.52	1.38
		Ag g/t	290	11	38.9	37.8	1.41	1.25
		Cu ppm	130,000	7	13,334	13,223	1.55	1.51
		Pb ppm	30,000	19	4,712	4,545	1.52	1.38
		Zn ppm	180,000	4	33,543	33,509	1.05	1.05
Low Grade Domain	1,073	Au g/t	2.00	12	0.51	0.31	1.33	1.17
		Ag g/t	75.0	7	10.3	10.1	1.31	1.18
		Cu ppm	60,000	2	3,565	3,533	1.58	1.49
		Pb ppm	--	0	1,004	1,004	1.61	1.61
		Zn ppm	100,000	1	6,254	6,186	1.43	1.28

14.6 Block Model Parameters

The Kay Deposit mineral resource domains are used to constrain composite values chosen for interpolation, and the mineral blocks reported in the estimate of the MRE. A block model, within UTM coordinate space, was created for the Kay Deposit (Table 14.10, Figure 14.8 and Figure 14.9). A block model, with dimensions in the x (east m), y (north m) and z (level m) directions, was placed over the resource models, with only that portion of each block inside the models (and within the Property boundary) recorded as part of the MRE (% block model). The block size for each block model was selected based on drillhole spacing, composite length, the geometry and shape of the mineralized domains, and the selected mining method (underground bulk mining). At the scale of the deposit models, the selected block size for each model provides a reasonable block size for discerning grade distribution, while still being large enough not to mislead when looking at higher cut-off grade distribution within the model. The models were intersected with surface topography to exclude blocks, or portions of blocks, that extend above the bedrock surface.

Table 14.10: Deposit Block Model Geometry

Block Model	Kay Deposit		
	X (East)	Y (North)	Z (Level)
Origin (WGS 84)	392610	3769125	670 m
Extent (blocks)	220	115	475
Block Size	2 m	5 m	2 m
Rotation (counterclockwise)	0°		

Figure 14.8: Plan View - Kay Deposit Mineral Resource Block Model and Mineralization Domains

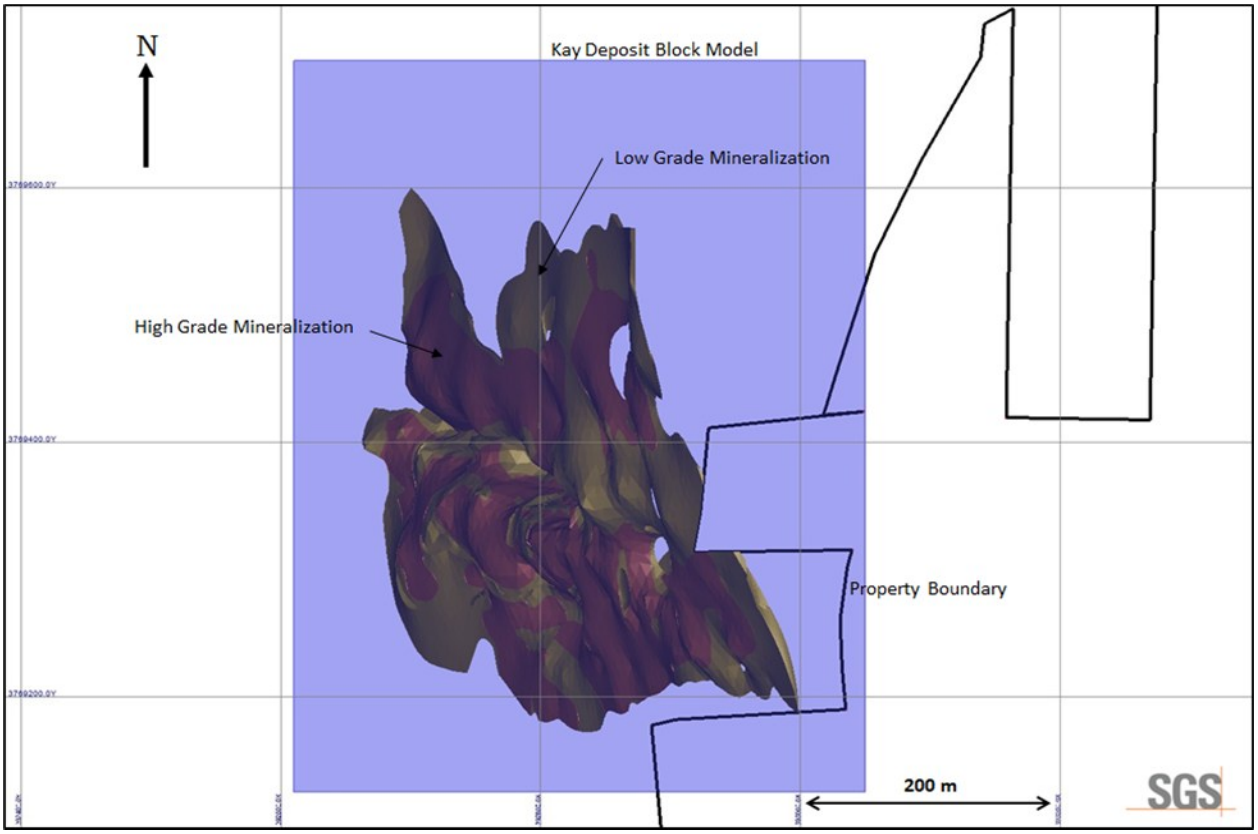
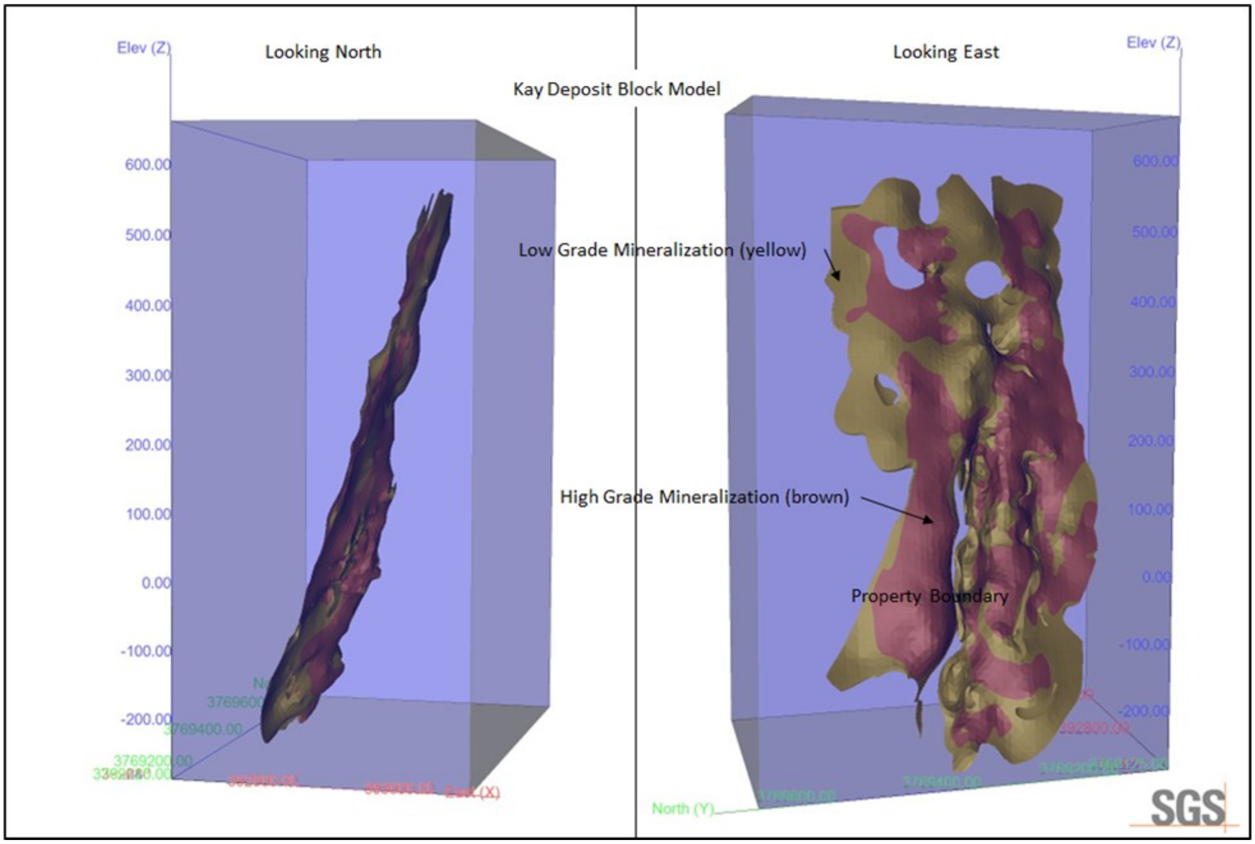


Figure 14.9: Isometric View looking North (left) and East (right) of the Kay Deposit Mineral Resource Block Model and Mineralization Domains



14.7 Grade Interpolation

Gold, silver, copper, lead, and zinc were estimated for each mineralization domain within the block model. Blocks within each mineralized domain were interpolated using composites assigned to that domain. However, it was decided to treat the boundary between the low grade and high-grade domain as a soft boundary, i.e., the interpolation procedure was allowed to see composites across the boundary. To generate grade within the blocks, the inverse distance squared (ID2) interpolation method was used for all domains.

For all domains, the search ellipse used to interpolate grade into the resource blocks was interpreted based on orientation and size of the mineralized domains, and the distribution of data within each domain. The search ellipse axes are generally oriented to reflect the observed preferential long axis (geological trend) of the domain and the observed trend of the mineralization down dip / down plunge (Table 14.11).

A three-pass search procedure was used to interpolate grade into all the blocks in the mineralization domains (Table 14.11): blocks were classified as Indicated if they were populated with grade during

Pass 1 and Pass 2 of the interpolation procedure, and Inferred if they were populated with grade during Pass 3 of the interpolation procedure.

For the high-grade domain, grades were interpolated into blocks using a minimum of seven (7) and maximum of 12 composites to generate block grades during Pass 1 (maximum of three (3) sample composites per drill hole) of a three-pass procedure (Table 14.11), minimum of five (5) and maximum of 12 composites to generate block grades during Pass 2 (maximum of three (3) sample composites per drill hole), and minimum of three (3) and maximum of 12 composites to generate block grades during Pass 3 (maximum of two (2) sample composites per drill hole). For the low-grade domain, grades were interpolated into blocks using a minimum of five (5) and maximum of eight (8) composites to generate block grades during Pass 1 and Pass 2 (maximum of three (3) sample composites per drill hole) of a three-pass procedure, and minimum of three (3) and maximum of eight (8) composites to generate block grades during Pass 3 (maximum of two (2) sample composites per drill hole).

Table 14.11: Grade Interpolation Parameters for the Kay Deposit

Parameter	Domain – Kay Deposit HG and LG		
	Pass 1 Indicated	Pass 2 Indicated	Pass 3 Inferred
Calculation Method	Inverse Distance squared		
Search Type	Ellipsoid		
Principle Azimuth	295°		
Principle Dip	-68°		
Intermediate Azimuth	5°		
Anisotropy X Range	35	65	110
Anisotropy Y Range	20	40	70
Anisotropy Z Range	7.5	15	30
Min. Samples	7 (5)	5 (5)	3 (3)
Max. Samples	12 (8)	12 (8)	12 (8)
Min. Drill Holes	3 (2)	2	2

**Note: Values in Brackets are adjusted for the Low-Grade Domain.*

14.8 Mineral Resource Classification Parameters

The MRE presented in this Technical Report is disclosed in compliance with all current disclosure requirements for mineral resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2016). The classification of the current MRE into Indicated and Inferred mineral resources is consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves, including the critical requirement that all mineral resources “have reasonable prospects for eventual economic extraction”.

The current MRE is sub-divided, in order of increasing geological confidence, into Indicated and Inferred categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource. There are no Measured Mineral Resources reported.

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

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

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

14.8.1 Measured Mineral Resource

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

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

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

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

14.8.2 Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit.

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

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

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

14.8.3 Inferred Mineral Resource

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

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

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

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

14.9 Reasonable Prospects of Eventual Economic Extraction

The general requirement that all Mineral Resources have “reasonable prospects for economic extraction” implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade considering extraction scenarios and processing recoveries. To meet this requirement, the Author considers that the mineralization on the Kay Property is amenable to underground extraction.

To determine the quantities of material offering “reasonable prospects for economic extraction” by underground mining methods, reasonable mining assumptions to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be “reasonably expected” to be mined from underground are used. Based on the location, depth from surface and depth extent, size, shape, general thickness, orientation and grade of the of the mineralized zones within the project area, it is envisioned that the deposits may be mined using underground bulk mining methods such as Longhole Stoping (LHS). The underground parameters used, based on this potential mining methods, are summarized in Table 14.12. Underground Mineral Resources are reported at a base case cut-off grade of 1.00% CuEq. A base case cut-off grade of 1.00% CuEq is applied to identify blocks that will have reasonable prospects of eventual economic extraction.

The reporting of the underground resource is presented undiluted and in situ, constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction. The underground mineral resource grade blocks were quantified above the base case cut-off grade, below topography and within the 3D constraining mineralized models (the constraining volumes).

Table 14.12: Parameters Used for Considering an Underground Cut-off Grade

Parameter SGS 2025	Value	Unit
Gold Price	$2,200.00	US$ per ounce
Silver Price	$26.00	US$ per ounce
Copper Price	$4.10	US$ per pound
Lead Price	$1.00	US$ per pound
Zinc Price	$1.35	US$ per pound
Processing Cost (incl. crushing) + Treatment and Refining	$24.00	US$ per tonne milled
Underground Mining Cost	$49.00	US$ per tonne mined
Underground General and Administrative	$5.00	US$ tonne of feed
Gold Recovery	76	Percent (%)
Silver Recovery	75	Percent (%)
Copper Recovery	92	Percent (%)
Lead Recovery	76	Percent (%)
Zinc Recovery	85	Percent (%)
Mining loss / Dilution (underground)	10/10	Percent (%) / Percent (%)
Cut-off Grade (CuEq)		
Kay Deposit Underground	1.00	Percent (%)

14.10 Mineral Resource Statement

The MRE for the Project is presented in Table 14.13 (Figure 14.10 and Figure 14.11).

Highlights of the Project Mineral Resource Estimate are as follows:

- The underground MRE includes 9.28 million tonnes grading 1.39 g/t Au, 27.6 g/t Ag, 0.97% Cu, 0.33% Pb, and 2.39% Zn in the Indicated category, and 0.86 million tonnes grading 1.06 g/t Au, 15.4 g/t Ag, 0.87% Cu, 0.20% Pb, and 1.68% Zn in the Inferred category, at a base-case cut-off grade of 1.00% CuEq.

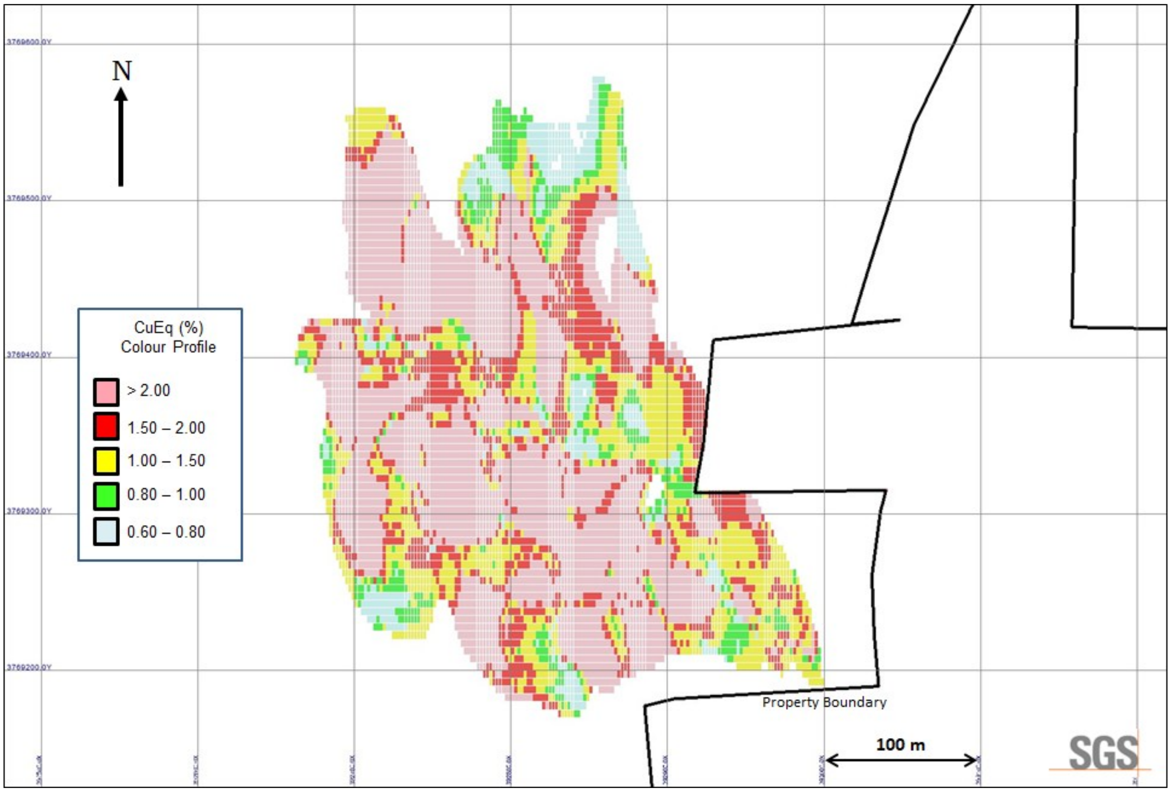
Table 14.13: Kay Property Mineral Resource Estimate, June 17, 2025

Tonnes (Mt)	Average Grade						Contained Metal					
	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	CuEq (%)	Au (koz)	Ag (koz)	Cu (Mlbs)	Pb (Mlbs)	Zn (Mlbs)	CuEq (Mlbs)
Indicated												
9.28	1.39	27.6	0.97	0.33	2.39	3.18	415	8,253	197.9	67.3	490.1	650.6
Inferred												
0.86	1.06	15.4	0.87	0.20	1.68	2.44	29	423	16.4	3.8	31.8	46.1

Kay Deposit Mineral Resource Estimate Notes:

1. *The effective date of the Kay Project Mineral Resource Estimate (MRE) is June 17, 2025. This is the close-out date for the final mineral resource drilling database.*
2. *The mineral resource was estimated by Allan Armitage, Ph.D., P. Geo. of SGS Geological Services, an independent Qualified Person as defined by NI 43-101. Armitage conducted site visits to the Kay Deposit on two (2) occasions, on October 25-26, 2023, and April 7-8, 2024. The mineral resource was peer reviewed by Ben Eggers, MAIG, P. Geo. of SGS Geological Services, an independent Qualified Person as defined by NI 43-101. Eggers conducted a site visit to the Kay Property on May 30, 2025.*
3. *The classification of the current MRE into Indicated and Inferred mineral resources is consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves.*
4. *All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.*
5. *All mineral resources are presented undiluted and in situ, constrained by continuous 3D wireframe models (considered mineable shapes), and are considered to have reasonable prospects for eventual economic extraction.*
6. *Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that most Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
7. *The Kay Project MRE is based on a validated drill hole database which includes data from 234 surface diamond drill holes completed between 2020 and May 2025. The drilling totals 133,912 m (including wedge holes). The resource database totals 11,533 assay intervals representing 14,006 m of data.*
8. *Grades for Au, Ag, Cu, Pb and Zn are estimated for each mineralization domain using 1.50 m capped composites assigned to that domain. To generate grade within the blocks, the inverse distance squared (ID2) interpolation method was used for all domains.*
9. *Average density values were assigned to each domain based on a database of 2,307 samples.*
10. *Based on the size, shape, and orientation of the deposit, it is envisioned that the deposits may be mined using underground bulk mining methods such as Longhole Stoping. The MRE is reported at a base case cut-off grade of 1.00% CuEq. The mineral resource grade blocks are quantified above the base case cut-off grade and within the constraining mineralized wireframes (considered mineable shapes).*
11. *The underground base case cut-off grade of 1.00% CuEq considers metal prices of $4.10/lb Cu, $1.00/lb Pb, $1.35/lb Zn, $2,200/oz Au and $26/oz Ag, assumed metal recoveries of 92% for Cu, 76% for Pb, 85% for Zn, 76% for Au and 75% for Ag, a mining cost of US$49.00/t rock and processing, treatment and refining, transportation and G&A cost of US$29/t mineralized material.*
12. *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*

Figure 14.10: Plan View – Mineral Resource Block Grades (upper) and Block Class (lower) for the Kay Deposit MRE



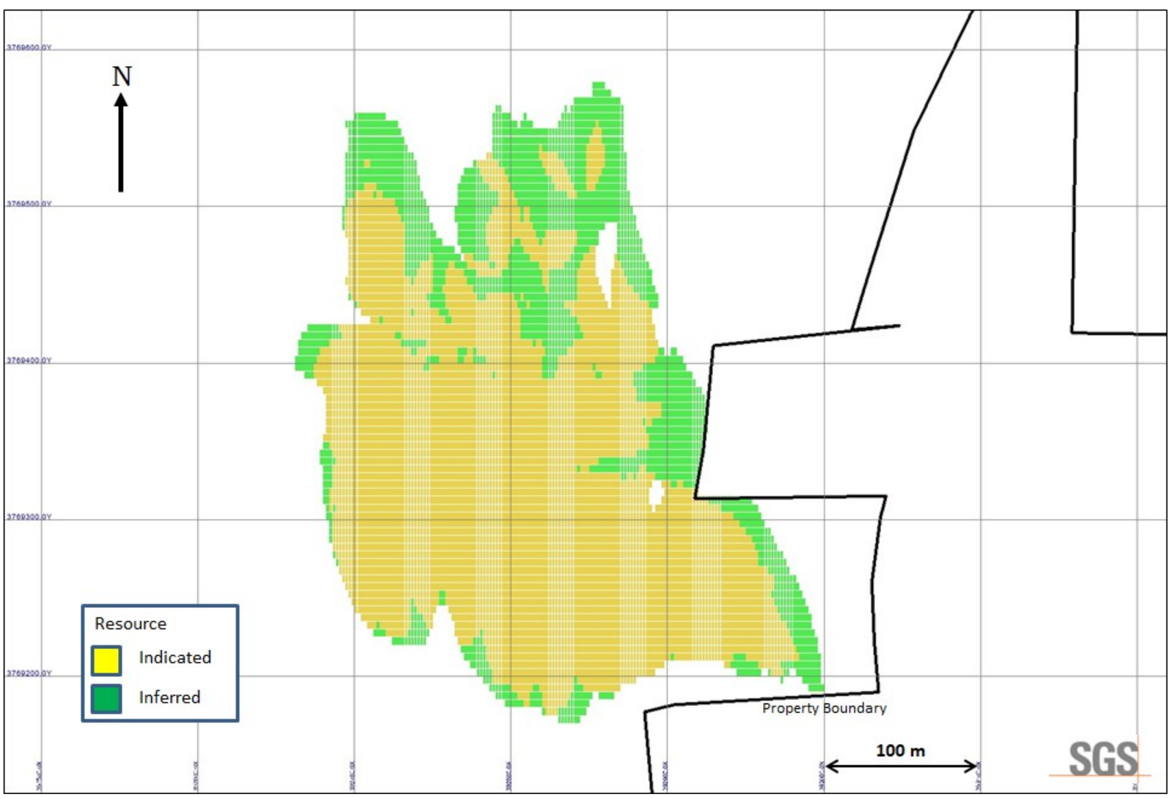
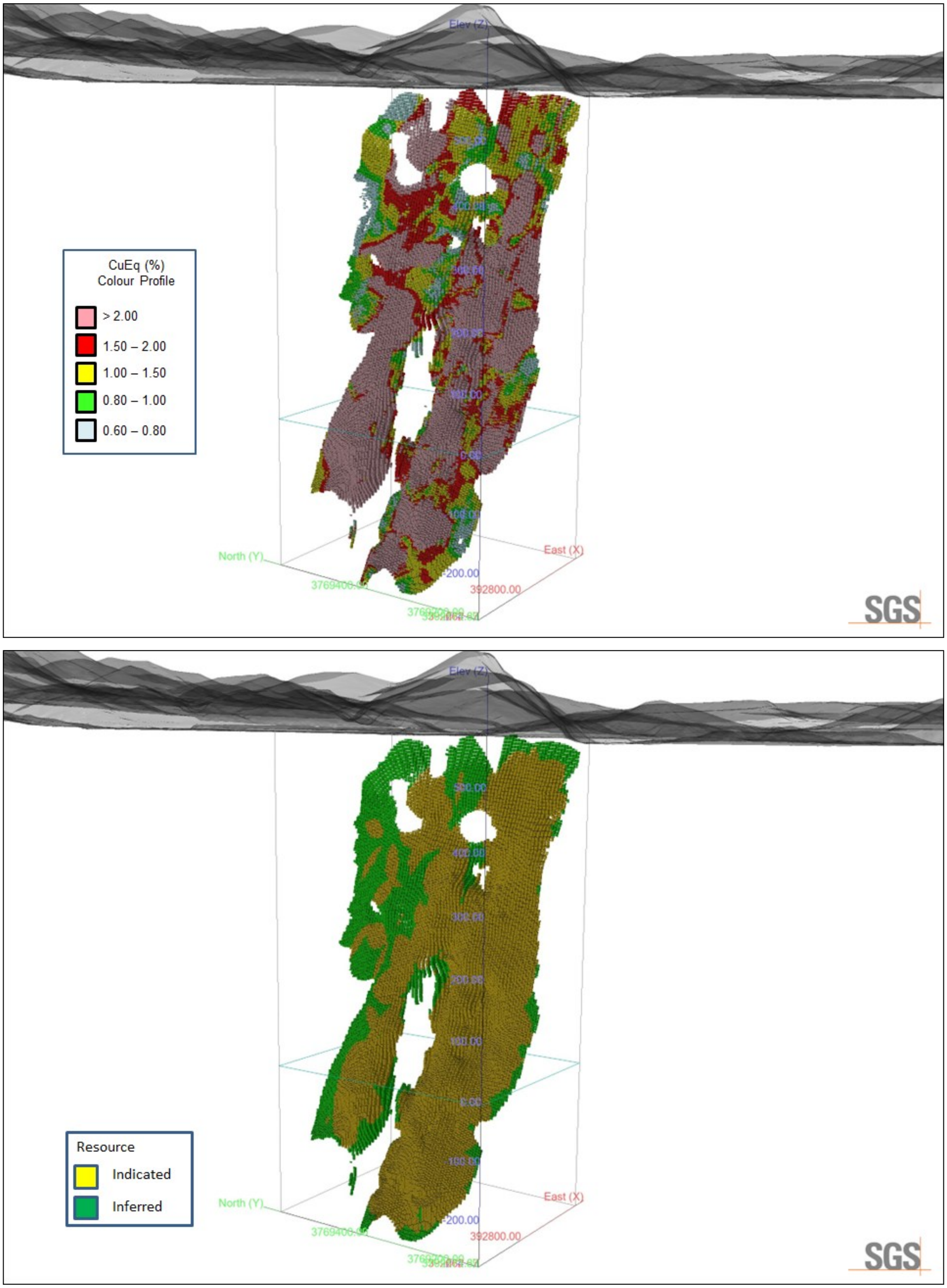


Figure 14.11 : Isometric View Looking NE – Mineral Resource Block Grades (upper) and Block Class (lower) for the Kay Deposit MRE



14.11 Model Validation and Sensitivity Analysis

Visual checks of block grades against the composite data and assay data on vertical section showed good correlation between block grades and drill intersections.

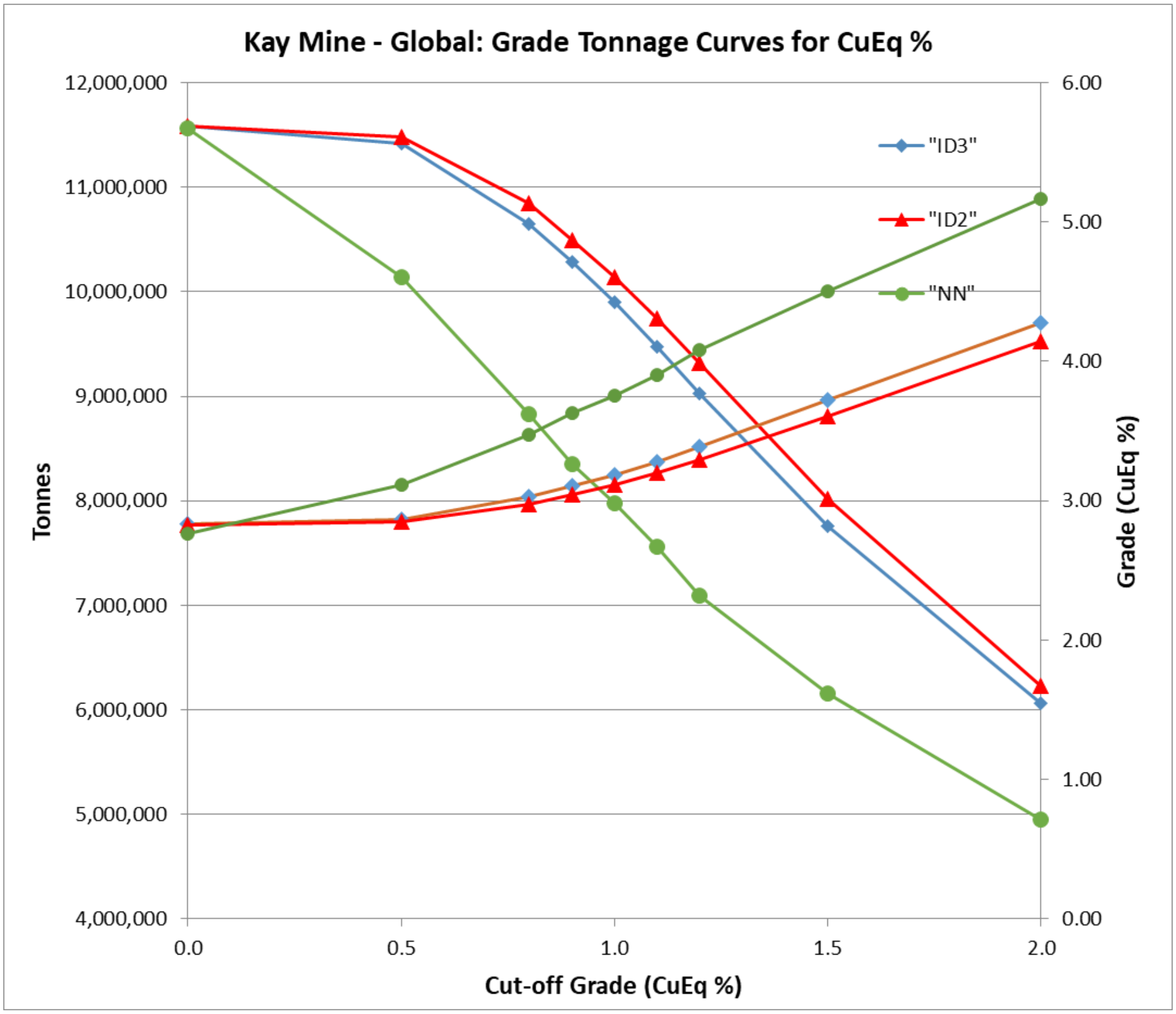
A comparison of the average capped composite grades, average assay grades and average block model grades, by model / domain is shown in Table 14.14. The block model average grades compared well with the capped composite average grades.

For comparison purposes, additional grade models were generated using a varied inverse distance weighting (ID^3) and nearest neighbour (NN) interpolation methods. The results of these models are compared to the chosen models (ID^2) at various cut-off grades in a grade / tonnage graph shown in Figure 14.12. In general, the ID^2 and ID^3 models show similar results, and both are much more conservative and smoother than the NN model. For models well-constrained by wireframes and well-sampled (close spacing of data), ID^2 should yield very similar results to other interpolation methods such as ID^3 or Ordinary Kriging.

Table 14.14 : Comparison of Average Assay Grades, Composite Grades with Block Model Grades

Domain	Variable	Number of	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
HG + LG	Assays	3,492	1.48	29.3	1.02	0.33	2.36
	Composites Capped	2,688	1.34	26.8	0.94	0.31	2.26
	Blocks	230,789	1.23	24.3	0.88	0.29	2.11

Figure 14.12 : Comparison of ID³, ID² & NN Models for the Kay Deposit



14.11.1 Sensitivity to Cut-off Grade

The Kay Project Mineral Resource has been estimated at a range of cut-off grades presented in Table 14.15 to demonstrate the sensitivity of the resources to cut-off grades. The current Mineral Resource is reported at a base-case cut-off grade of 1.00% CuEq (highlighted).

Note: Values in these tables reported above and below the base-case cut-off 1.00% CuEq for underground Mineral Resources should not be misconstrued with a Mineral Resource Statement. The values are only presented to show the sensitivity of the block model estimates to the selection of the base case cut-off grade. All values are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.

Table 14.15: Kay Property Mineral Resource Estimate at Various CuEq % Cut-off Grades, June 17, 2025

Cut-off Grade (CuEq %)	Tonnes (Mt)	Average Grade						Contained Metal					
		Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	CuEq (%)	Au (koz)	Ag (koz)	Cu (Mlbs)	Pb (Mlbs)	Zn (Mlbs)	CuEq (Mlbs)
Indicated													
0.80	9.90	1.32	26.5	0.93	0.31	2.28	3.04	421	8,444	202.4	68.6	498.6	662.9
0.90	9.59	1.36	27.1	0.95	0.32	2.34	3.11	418	8,353	200.3	68.0	494.6	657.1
1.00	9.28	1.39	27.6	0.97	0.33	2.39	3.18	415	8,253	197.9	67.3	490.1	650.6
1.10	8.94	1.43	28.3	0.99	0.34	2.46	3.26	411	8,134	194.9	66.4	484.5	642.7
1.20	8.60	1.47	28.9	1.01	0.35	2.52	3.35	406	8,001	191.7	65.5	478.4	633.9
1.50	7.47	1.62	31.3	1.09	0.38	2.75	3.65	389	7,506	179.7	61.7	453.3	600.4
Inferred													
0.80	0.94	1.00	14.6	0.82	0.19	1.57	2.30	30	443	17.1	3.9	32.6	47.8
0.90	0.90	1.02	14.9	0.85	0.19	1.62	2.37	30	433	16.8	3.9	32.2	47.1
1.00	0.86	1.06	15.4	0.87	0.20	1.68	2.44	29	423	16.4	3.8	31.8	46.1
1.10	0.80	1.11	16.0	0.89	0.21	1.78	2.54	28	410	15.7	3.7	31.2	44.7
1.20	0.72	1.17	16.8	0.93	0.22	1.92	2.69	27	390	14.9	3.5	30.4	42.8
1.50	0.55	1.37	18.8	1.05	0.25	2.28	3.11	24	333	12.8	3.0	27.7	37.8

14.12 Disclosure

All relevant data and information regarding the Project are included in other sections of this Technical Report. There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading.

The Authors are not aware of any known mining, processing, metallurgical, environmental, infrastructure, economic, permitting, legal, title, taxation, socio-political, or marketing issues, or any other relevant factors not reported in this technical report, that could materially affect the updated MRE.

15. MINERAL RESERVE ESTIMATES

No Mineral Reserves have been estimated for the Kay Mine Project. This Technical Report supports a Preliminary Economic Assessment (PEA) prepared in accordance with National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and Form 43-101F1 and is based on Mineral Resources as disclosed in Chapter 14 of this Report, including Inferred Mineral Resources.

Under the *CIM Definition Standards on Mineral Resources and Mineral Reserves* (adopted by CIM Council on May 10, 2014, as amended), Mineral Reserves can only be declared on the basis of a Pre-Feasibility Study (PFS) or Feasibility Study (FS) that demonstrates, at the time of reporting, that economic extraction can be reasonably justified. A PEA is a preliminary, scoping-level study and does not meet the minimum level of engineering, economic, and modifying-factor assessment required for the conversion of Mineral Resources to Mineral Reserves. Accordingly, no Proven or Probable Mineral Reserves are declared in this Report.

Readers are cautioned that the PEA is preliminary in nature. The mine plan and economic analysis presented in this Report include the use of Inferred Mineral Resources, which 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 the results of the PEA will be realized, nor that Inferred Mineral Resources will be converted to Measured or Indicated Mineral Resources. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. The estimated production schedule, capital and operating cost estimates, and financial results presented in Chapters 16 through 22 of this Report should not be considered to be, and are not, a Pre-Feasibility Study or a Feasibility Study.

16. MINING METHODS

16.1 Summary

The Kay Mine is planned as a mechanized long-hole open stoping underground mine. The milling rate is planned at 0.7 Mtpa with a ramp-up period of nine (9) months during the first operational period. The mill will operate for 10 years. A PEA is preliminary in nature and is intended to provide only an initial, high-level review of the Project's potential and design options. The PEA mine plan and economic model include numerous assumptions and the use of Inferred Mineral Resources. Inferred Mineral Resources are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and to be used in an economic analysis, except as allowed in PEA studies. There is no guarantee that Inferred Mineral Resources can be converted to Indicated or Measured Mineral Resources, and, as such, there is no guarantee the Project economics described herein will be achieved.

The underground operation consists of a single mine accessed via one surface portal located south of the surface infrastructure area. The selected mining method consists of long hole open stoping (LHOS), specifically sublevel transverse stoping and sublevel longitudinal stoping.

The life-of-mine (LOM) for the underground operation is estimated at 12.5 years, encompassing construction and development, pre-production, and full production phases. Of this total, the underground mine is expected to operate in production for approximately 10 years, including a nine (9) months ramp-up period. The pre-production phase is anticipated to last approximately two and a half (2½) years following portal construction, allowing sufficient underground development to be completed to support steady-state production.

The underground mine is planned to supply the mill feed at an average rate of 1,918 tonnes per day of mineralized material. The mine plan includes the excavation of approximately 39.3 km of lateral development and 2.3 km of vertical development.

A total of approximately 6.55 million tonnes (Mt) of mineralized material is expected to be mined at average diluted grades of 1.01% Cu, 2.67% Zn, 1.60 g/t Au, and 29.07 g/t Ag. The primary production fleet will consist of 15-t diesel-powered load–haul–dump (LHD) units in combination with 45-t underground haul trucks for the transport of all mined material.

16.2 Geotechnical Considerations

Alius Mine Consulting ("Alius") carried out a geotechnical assessment that included data from a geotechnical photo-logging program and a review of historical data, aimed at characterizing the geotechnical conditions and rock mass near the planned mining area. The various components of the assessment and their findings are outlined in the following sections.

16.2.1 Rock Mass Characterization

The rock mass characterization completed for this study includes estimates of intact rock strength, rock mass quality classification, and structural fabric interpretation. Given the absence of laboratory testing and detailed geotechnical core logging, several inputs rely on engineering judgment, supported by review of available drill core photographs, benchmarking against comparable volcanogenic massive sulfide (VMS) deposits, and published references.

Uniaxial compressive strength (UCS) values were derived from literature sources relevant to the identified lithologies. The mineralized zones are mainly hosted within felsic metavolcanic rocks, including meta-rhyolites and felsic schists. For these units, minimum and maximum UCS values of 60 MPa and 75 MPa, respectively, were used, consistent with published ranges reported for similar rock types (Brady and Brown, 2006; Deere, 1966; Hoek and Brown, 1997).

The wider host sequence surrounding the mineralized zones consists of mafic schists and metasedimentary units. For these lithologies, UCS values between 40 MPa and 50 MPa were chosen, based on comparable literature values for altered mafic and metasedimentary rocks in metamorphosed volcanic sequences (Brady and Brown, 2006; ISRM, 2007; Vutukuri et al., 1978).

The rock mass was characterized through photographic review of drill core from 27 drillholes distributed across the S2 mineralized zone. Only drillholes that fully intersected both the hanging wall and footwall were considered representative and included in the assessment. Rock mass quality was evaluated using the Norwegian Geotechnical Institute (NGI) Q and Q rock mass classification systems (Barton et al., 1974). The resulting Q values for the main lithologies are summarized below.

Table 16.1: Q Rock Mass Classification

Lithology	Q_{25}	Q_{50}
Felsic Rock	6.0	8.5
Mineralized Material	11.7	15.0

**Note: Q classifications are equal to those of Q as the Jw/SRF ratio is assumed to be one (1).*

Structural information was derived from manually logged joint orientation data collected during historical core logging. These structures are mostly interpreted as foliation-related features, consistent with observations made during the photo-logging review. The structural dataset indicates that the rock mass is characterized by a single dominant joint set, subparallel to foliation. Joint orientation measurements were plotted on a stereonet to evaluate the geometry of this main structural fabric. The resulting stereographic projection shows an average dip of approximately 73° with a dip direction of 275°.

For the stope stability assessment, the stress environment used in the empirical analyses was defined based on representative in-situ stress conditions evaluated at a depth of 600 m below the surface, approximately two-thirds of the maximum mineralized zone depth. The vertical stress (σv) was treated as the minor principal stress and calculated from the weight of the overlying rock column. The major horizontal stress (σH), assumed to be perpendicular to the hanging wall and footwall, is considered to be equal to 1.5 times the vertical stress. The minor horizontal stress (σh), acting perpendicular to the stope end walls, is assumed to be 1.3 times the vertical stress.

Table 16.2: In-situ Stress Properties

Rock Density (kg/m^3)	Depth Below Surface (m)	σ_v (MPa)	σ^H (MPa)	σ^h (MPa)
3,100	600	18.2	27.4	23.7

16.2.2 Stope Sizing

Stope stability was evaluated using established empirical open stope design methods, including the Mathews–Potvin stability chart method (Potvin, 1988), the Equivalent Linear Overbreak Sloughing (ELOS) approach (Clark and Pakalnis, 1998), and probabilistic extensions to the Mathews method (Mawdesley et al., 2001). These methods are commonly used in underground hard-rock mining and are suitable for the current stage of the Project.

The analyses assess stope stability by examining the relationship between the modified stability number (N′) and the hydraulic radius (HR) of each exposed stope surface. Stability evaluations were carried out separately for the stope back, hanging wall, footwall, and end walls of representative stopes.

The modified stability number (N′) includes rock mass quality, in-situ stress conditions, joint orientation relative to stope surfaces, and gravity effects. The hydraulic radius indicates the geometric exposure of each stope surface and is defined as the ratio of surface area to perimeter. For each stope surface, N′ and HR values were calculated and plotted on published stability and ELOS design charts to evaluate the expected stability performance and dilution.

16.2.3 Lithological Exposure Scenarios

Two (2) lithological exposure scenarios were evaluated to account for variability in wall rock conditions:

- Hanging wall and footwall developed within mineralized rock.
- Hanging wall and footwall exposed to felsic host rock.

For both scenarios, stope backs and end walls were assumed to be developed within the mineralized unit. Preliminary mine planning indicates that stopes are roughly evenly distributed between these two (2) exposure conditions. Stability and dilution analyses were completed for both cases.

A graphitic horizon interpreted as a fault zone was excluded from the empirical analyses because it lies more than 15 m away from the modelled mineralized zones and is assumed not to influence stope stability at this stage. The potential impact of this structure should be reassessed during future, more detailed design phases.

16.2.4 Stope Dimensions

Based on the empirical stability analyses and initial stope optimization, recommended stope dimensions were selected to minimize dilution while maintaining an acceptable probability of failure. These dimensions are presented in Table 16.3.

Table 16.3: Stope Dimensions

Dimension	Maximum
Strike Length (m)	13
Width (m)	15
Vertical Height (m)	25

The mineralized zone typically ranges from approximately 14 to 30 m in width. As a result, most stopes are expected to be mined using a transverse orientation.

16.2.5 Stability Assessment

The stability chart results indicate that most stope surfaces fall within stable or transitional stability zones. The following general conclusions can be drawn:

Table 16.4: Stope Surface Stability Classification

Stope Surface	Stability Classification	Interpretation
Back	Stable to transitional	Stable to stable with support
End Walls	Stable	Stable
Footwall	Stable	Stable
Hanging Wall	Stable to transitional	Stable to stable with support

At this stage, cable bolting is not expected to be required for hanging walls and footwalls. However, local support might be required for stopes with larger widths, especially where the hanging wall is exposed to the felsic unit.

Additionally, the possible presence of the graphitic fault within the hanging wall might require revising support assumptions in later design stages. Stopes with shallow dips (e.g., less than 60°) may also require specific support standards to ensure proper stability.

16.2.6 Dilution Estimates

Dilution was assessed using the ELOS method, employing the same stope geometries and lithological exposure scenarios as in the stability analyses. ELOS estimates were determined for stope widths ranging from 10 to 15 m.

Based on the recommended stope dimensions, the expected dilution outcomes are as follows:

Table 16.5: Predicted Equivalent Linear Overbreak / Slough (ELOS) by Stope Surface

Stope Surface	Expected ELOS (m)	Stability Interpretation
Back	< 1.0 m (assumed to be mineralized material)	Minor sloughing
End Walls	Negligible (< 0.0 m)	Stable
Footwall	< 0.5 m	Very minor sloughing
Hanging Wall	< 0.75 m	Minor sloughing

These results suggest that dilution is likely to be low and manageable for the proposed stope geometries.

16.2.7 Backfill

Backfill strength requirements were estimated using the analytical method of Belem et al. (2022) for cemented rockfill (CRF) in transverse and longitudinal stopes. A design factor of safety (FoS) of 2.0 is adopted for this stage of the Project.

Primary stopes require a minimum CRF strength of approximately 550 kPa, assumed to be achievable with about 5% binder content. Underhand mining plug pours require a higher strength backfill of approximately 1,200 kPa, corresponding to about 10% binder. The last secondary stope adjacent to an access may be filled with uncemented rockfill (URF). Where URF cannot be used, a minimum strength of 100 kPa should be achieved.

Where underhand mining occurs, overlying stopes must contain higher-strength CRF, representing at least 70% of the fill volume for stopes wider than 10 m and 60% for stopes narrower than 10 m.

These values are preliminary and should be validated through laboratory testing using site-specific materials.

Table 16.6: Cemented Backfill Strength Requirements

Stope Type	Strength Requirement (kPa)	Assumed Binder Content
Transverse and Longitudinal Stopes	550	5%
Transverse (secondary) – Last Stope	N/A (URF)	0%
Plug for Underhand Mining	1,200	10%

**Notes: URF = Uncemented Rockfill.*

16.2.8 Ground Support

Ground support recommendations are based on empirical design methods and common industry practices used in Canadian and Australian underground mines. They are suitable for PEA-level design. The systems outlined below assume typical rock mass behaviour and should be reviewed and optimized as additional mapping and geotechnical data become available during development.

Permanent development (ramps and infrastructure) is planned to be supported on a 1.2 × 1.2 m pattern using 2.4 m resin rebars in the back and 1.5 m friction bolts in the walls, with 4 × 4”, 6-gauge welded wire mesh extending to approximately 1.5 m above the floor. Intersections in permanent development with spans up to 7.5 m are expected to remain stable with primary support only, while larger intersections (7.5–10.0 m) will require secondary support consisting of 6 m grouted cable bolts installed on a 2.0 × 2.0 m pattern.

Temporary development, including footwall drifts, drawpoints, and mineralized material drift, is planned to be supported on a 1.2 × 1.2 m pattern using 1.8 m inflatable bolts in the back and 1.5 m friction bolts in the walls, with 4 × 4”, 6-gauge mesh and similar wall coverage. Intersections in temporary development will require 6 m cable bolts installed on a 2.0 × 2.0 m pattern for spans up to approximately 8.5 m.

Vertical development support depends on raise diameter. Raises with a 3.0 m diameter are planned to be supported using 1.8 m friction bolts arranged in a 1.2 × 1.2 m pattern with mesh, while 4.5 m diameter raises should use 2.4 m resin rebars on the same pattern with mesh coverage.

Back support for transverse stopes is planned by using cable-bolt rings, each comprising four (4) 6 m grouted cable bolts per ring at approximately 2.1 m spacing, providing a stable stope back under expected conditions. Support requirements may be adjusted locally based on rock mass variability, and additional reinforcement could be necessary near brows or other areas with increased stress concentration.

These recommendations are preliminary and could be refined once site-specific structural mapping and performance observations become available.

16.3 Hydrogeology

16.3.1 Review of Existing Hydrogeological Information

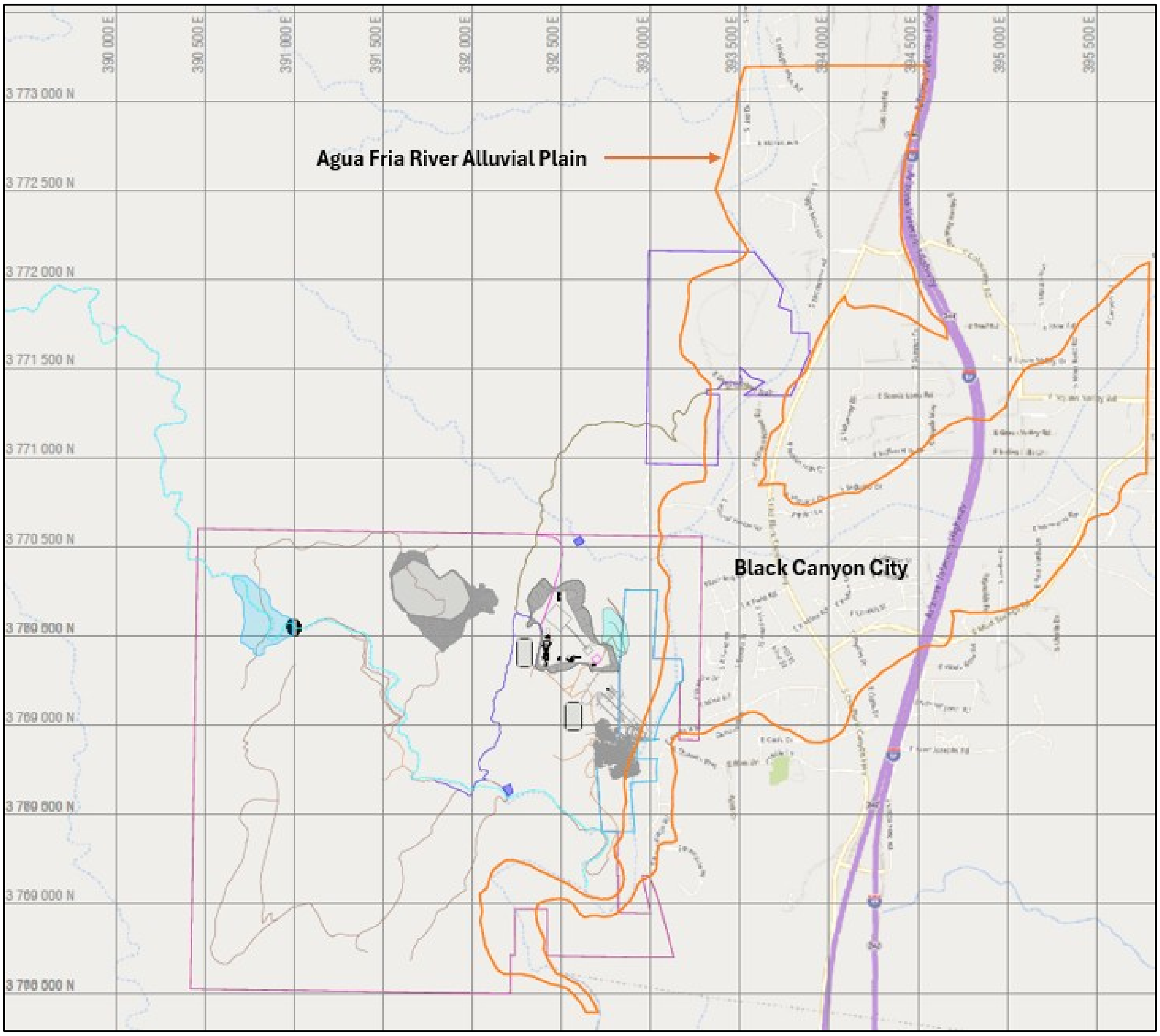
A review of historical reports provided by Arizona Metals Corp indicates that no site-specific hydrogeological data, such as the hydraulic conductivity of the bedrock, is available to support calculations of groundwater inflow into the existing mine workings. To assess the presence of major aquifers in the Project area, publicly available hydrogeological information was examined.

The Kay Mine is situated within the Agua Fria watershed, in the Black Canyon City area of Yavapai County, Arizona. According to the Arizona Water Atlas Volume 5 (2009), the two (2) principal aquifers in the Agua Fria Basin are the basin-fill deposits (the Agua Fria River alluvial plain) and sedimentary rock formations (conglomerates). While conglomerates are not present on the Kay Mine property, the site does intersect with the alluvial plain of the Agua Fria River (see Figure 16.1).

Based on the geological map provided, this alluvial plain, referred to as "river wash" in the illustration below, consists of unconsolidated sand and gravel. This formation may contribute to groundwater inflow into the planned underground mine, particularly if permeable geological structures exist that connect the mine workings to the Agua Fria River alluvium.

Currently, there is no hydrogeological data available to confirm such connectivity. A detailed hydrogeological investigation will be required to assess this potential interaction in the next phase of the Project.

Figure 16.1: Agua Fria River Alluvial Plain



16.3.2 <u>Review of Historic Dewatering Pumping Rate Information</u>

A review of historical reports provided by Arizona Metals Corp indicates that no records are available regarding the dewatering pumping rates of the former mine during its operational period. Additionally, a literature review revealed no publicly available data on dewatering rates for other historical mines located within a 75-kilometre radius of the Kay Mine. Dewatering rates reported for the Resolution Copper mine

(USDA Forest Service, 2020), located 150 km southeast of the Kay mine, are between 4,400 and 5,400 m^3/day. The depth of this mine is 2,000 m.

No information regarding the water quality of the flooded underground workings was identified during the review of historical reports.

16.4 Underground Mining

16.4.1 Underground Mining Method

The selected underground mining method is long hole open stoping (LHOS), employing either transverse or longitudinal stoping configurations. Backfill will be employed to fill the voids created by the stoping activities. The stoping sequence will be carried out in an ascending manner from an initial undercut. In most cases, stopes will be mined using an upper access for drilling and a lower access for mucking. Stopes located immediately below sill pillars will require redevelopment of the upper access through backfill to enable drilling access.

LHOS is a commonly employed underground mining method for competent hard-rock orebodies. The initial phase of LHOS is the mine development phase, during which the primary underground excavations are constructed, including the decline, level accesses, haulage drifts, drawpoints, and associated infrastructure (e.g., refuges, explosive storage, maintenance facilities, electrical substations, gear bays, ventilation infrastructure, and safety egress). These excavations provide access to the stoping areas and support the production activities. The development phase also includes the development of both overcut and undercut drifts.

The overcut drift is developed for production drilling of the stoping area, while the undercut provides access for mineralized material extraction from the stope. Once development of the drilling and extraction drifts is completed, a slot raise, typically raise bored or conventionally drilled and blasted, is developed to create the initial void for production drilling and blasting activities.

The production drilling phase involves drilling of long vertical or inclined holes at regular intervals along the length and width of the stoping area. Following completion of production drilling, the blasting phase commences with the loading of the production drill holes with bulk explosives, initiating products, and stemming.

During the blasting phase, the controlled use of explosives fractures the rock surrounding the slot raise and production drill holes. Once blasting is completed and blasting gases have been cleared from the mine, the fragmented material is mucked from the stope through the undercut drift using load–haul–dump (LHD) units. Depending on stope size and geometry, up to three (3) blasting and mucking cycles may be required to fully extract the material from the stope. The broken material is subsequently loaded into haul trucks and transported to surface.

LHOS is a non-entry mining method, as the stoping area is not accessible to personnel once production has commenced. The use of remote-controlled load-haul-dump (LHD) units is therefore required to fully recover the blasted material from the stope. The final phase of LHOS is the backfilling stage. Depending on the LHOS variant employed and the mining sequence, stopes may be backfilled with cemented material, uncemented material, or a combination of both. For this Project, the selected cemented backfill is cemented rockfill (CRF), while uncemented backfill will typically consist of rockfill (RF) composed of waste rock.

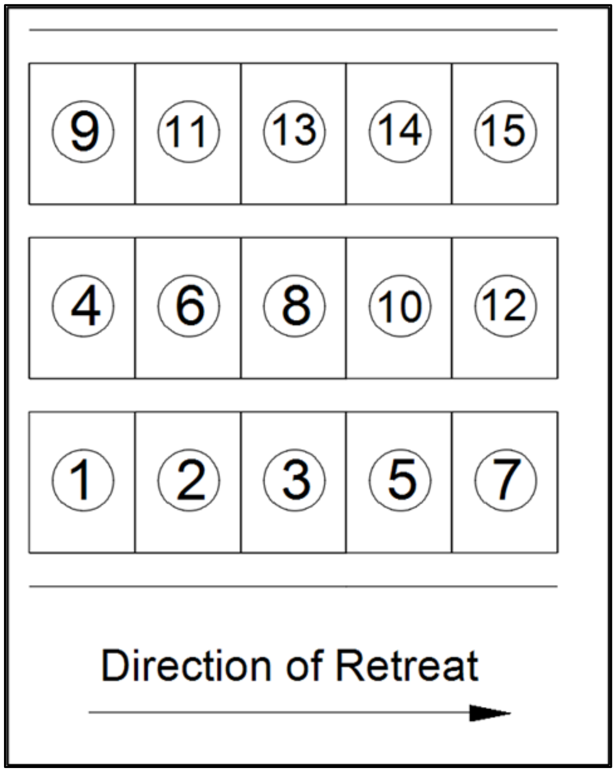
This non-entry mining method offers several advantages, including but not limited to high productivity, operational flexibility, low operating costs, efficient extraction of mineralized material, and improved worker safety.

The longitudinal variant of LHOS is used for the following situations:

- The stoping width is less than 8 m.
- Adjacent to longitudinal LHOS stopes where transverse LHOS is not practical.

Longitudinal mining areas are accessed by developing the overcut and undercut drifts inside the stoping area along the strike of the orebody. Once development reaches the extremity of the longitudinal mining area, the production cycle of the initial stope can commence. Subsequent stopes will be mined following the same production cycle while retreating toward the main access, which is located either at an extremity or near the center of the orebody. Figure 16.2 illustrates the typical longitudinal stoping sequence.

Figure 16.2: Underground Mine Sublevel Longitudinal Stoping Typical Stope Sequence



Source: GMS 2026 (not to scale).

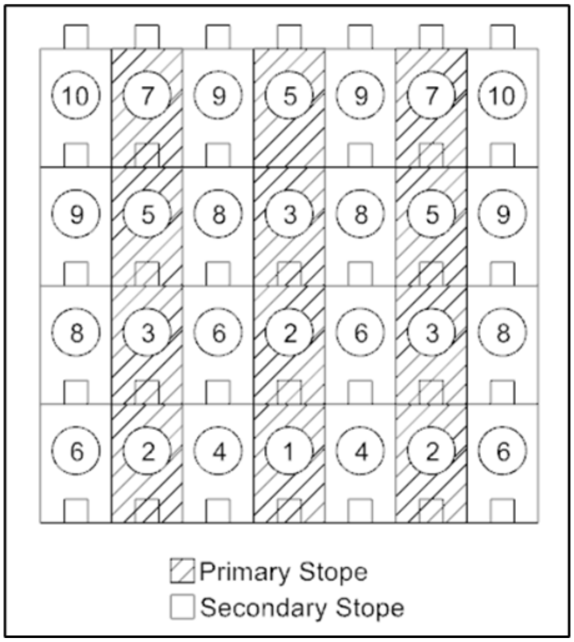
The transverse variant of LHOS is used for the following situations:

- The stoping width exceeds 8 m.
- Adjacent to transverse LHOS stopes where longitudinal LHOS is not practical.

The transverse mining area will be accessed by developing a haulage drift parallel to the strike of the orebody. This haulage drift will maintain a reasonable standoff distance from the stoping area to preserve ground stability during production. Perpendicular to the haulage drift, a series of evenly spaced, parallel drawpoints will provide access to the undercut and overcut drifts developed across the width of the orebody.

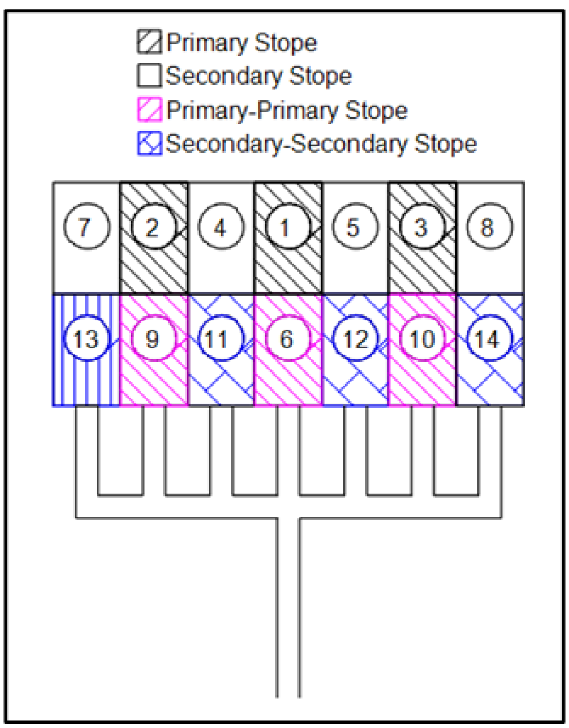
Once the haulage drift, drawpoints and mineralized material drifts have been developed, the production cycle of the initial primary stope can commence. Subsequent primary stopes will be mined following the same production cycle while retreating towards the extremities of the orebody. Stopes will be sequenced using an overhand approach, such that after two (2) lifts of primary stopes are mined, the secondary stope between these two (2) primary stopes can be mined at the first lift. This primary-secondary transverse mining approach provides advantages in terms of production efficiency and operational flexibility, as multiple stopes can be active simultaneously. A disadvantage of this method is the requirement to excavate a haulage drift along the full length of the mineralized zone. However, the production cycle may begin prior to completion of the haulage drift to the extremities of the orebody. Figure 16.3 illustrates the typical configuration of sublevel transverse stoping, while Figure 16.4 illustrates the typical transverse mining stope sequence.

Figure 16.3: Typical Transverse Mining Stope Sequence



Source: GMS 2026 (not to scale).

Figure 16.4: Typical Transverse Mining Stope Sequence - Plan View



Source: GMS 2026 (not to scale).

16.4.2 Cut-Off Value and Optimization Parameters

The break-even cut-off value (COV) represents the minimum economic value of mineralized material required for its extraction and processing to be economically viable. It corresponds to the point at which the costs of mining, processing, and refining are equal to the revenues generated from the sale of the payable commodities.

The processing plant is designed to produce three (3) concentrates (copper, zinc, and pyrite), enabling the recovery of copper, zinc, gold, and silver. Accordingly, the use of an NSR-based cut-off provides a consistent and integrated measure of the potential revenue per tonne, reflecting the combined contribution of all payable metals as well as associated processing and smelting terms.

Table 16.7 summarizes the economic parameters. Table 16.8 through Table 16.10Table 16.10 present the technical parameters used to calculate the NSR values for the copper, zinc and pyrite concentrates.

Table 16.7: Economic Parameters

Metal	Unit	Price	Refining Cost
Copper	USD/lb	4.10	0.08
Zinc	USD/lb	1.35	0.00
Gold	USD/oz	2,200.00	15.00
Silver	USD/oz	26.00	1.00
Royalty Rate	%	0	

Table 16.8: Copper Concentrate Parameters

Payable Metal	% Recovery	% Payable	Minimum Payable
Copper (Cu)	92.00%	95.00%	-
Gold (Au)	20.80%	95.00%	1 g/t
Silver (Ag)	66.00%	95.00%	30 g/t
Moisture Content	9%		
Treatment Charges	50 USD/dmt		
Transportation	130 USD/wmt		
Transport Loss	0.2%		

Payable Metal	% Recovery	% Payable	Minimum Payable
Concentrate Grade	27.1%		
Contaminant Metal*	**Unit Price (US$/dmt)**	**Penalty Limit**	**Increment**
Arsenic (As)	2.00	0.1%	0.1%
Antimony (Sb)	4.00	0.1%	0.1%
Mercury (Hg)	0.20	10 ppm	1 ppm
Zinc (Zn) + Lead (Pb)	2.00	3%	1%

**Note: Grades within concentrate are fixed for the following metals: As 0.98%; Hg 68 g/t; Zn 4.24%; Pb 3.32%.*

Table 16.9: Zinc Concentrate Parameters

Payable Metal	% Recovery	% Payable	Minimum Payable
Zinc (Zn)	80.00%	95.00%	-
Gold (Au)	4.40%	95.00%	1 g/t
Silver (Ag)	3.06%	95.00%	30 g/t
Moisture Content	9%		
Treatment Charges	100 USD/dmt		
Transportation	130 USD/wmt		
Transport Loss	0.2%		
Concentrate Grade	58.7%		
Contaminant Metal	**Unit Price (US$/dmt)**	**Penalty Limit**	**Increment**
Lead (Pb)	2.00	3.5%	1%
Mercury (Hg)	1.50	250 ppm	100 ppm

Table 16.10: Pyrite Concentrate Parameters

Payable Metal	% Recovery
Gold (Au)	60.80%
Silver (Ag)	15.94%
Moisture Content	9%
Concentrate Grade	4.23 g/t Au

Albion Process	
Payable Metal	**% Recovery**
Gold (Au)	91.74%
Silver (Ag)	92.63%

To evaluate the potentially economic portion of the Mineral Resource Estimate, a cut-off value was calculated for the selected underground mining method (Long hole open stoping, LHOS). Table 16.11 summarizes the parameters used to estimate the mine's break-even cut-off value and presents the detailed mining operating cost (OPEX) estimates for Kay Mine.

Table 16.11: Underground Mine COV

Item	Units	Value
Mine Operating Cost - LHOS	USD/t milled	61.76
Processing Cost (Power Incl.)	USD/t milled	47.41
General & Administration Cost	USD/t milled	7.00
Material Rehandling	USD/t milled	3.50
Break-even Cut-Off Value (No Sustaining Capital)	USD/t milled	119.97

A zone-by-zone approach was then used to validate the economic viability of the individual zones by incorporating zone-specific sustaining capital costs into the economic analysis.

16.4.3 Mineralized Material Included in Life-of-Mine

16.4.3.1 Resource Block Model

The mineral resources block models were provided by SGS as two (2) separate block models and imported into Deswik CAD™. The parent block dimensions of the models were 2.0 m × 5.0 m × 2.0 m. Both models were sub-blocked using a factor of 4 in the X and Y directions, with sub-blocking in the Z direction constrained by wireframes and subsequently combined into a single block model for evaluation purposes.

The evaluation of the Potentially Extractable Portion of the Mineral Resources Estimate, referred to as mineralized material mined in the Kay Mine PEA, includes the following categories of resources: Indicated and Inferred.

A PEA is preliminary in nature and is intended to provide only an initial, high-level review of the Project's potential and design options. The PEA mine plan and economic model include numerous assumptions and the use of Inferred resources. Inferred resources are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves and to be used in an economic analysis, except as allowed for in PEA studies. There is no guarantee that Inferred resources can be converted to Indicated or Measured resources, and as such, there is no guarantee the Project economics described herein will be achieved.

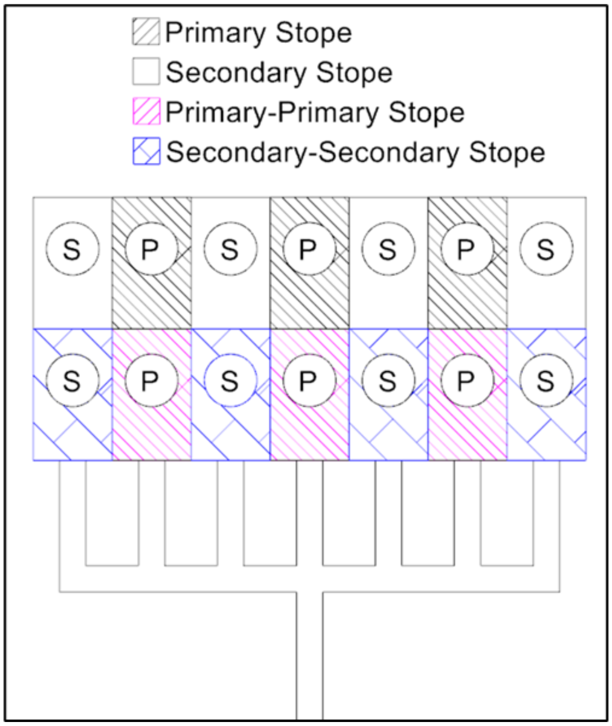
16.4.3.2 Dilution & Mining Losses

Dilution parameters were assigned to each stope to estimate the additional dilution experienced during mining operations. An equivalent linear overbreak slough (ELOS) of 0.75 m was applied to the stope hanging wall, and 0.5 m was applied to the footwall. In addition, mining losses of 5% and dilution associated with backfill interaction were incorporated following stope optimization. Mining dilution (also referred to as backfill dilution) is applied to the stopes based on the selected mining method and extraction sequence. This dilution is considered unavoidable and is assumed to have a null grade as it consists of backfill material. Backfill dilution was estimated using an equivalent linear dilution of 0.5 m on the stope floor and 0.5 m on the stope walls (per wall in contact with backfill). Table 16.12 shows the various backfill dilution factors used depending on the stope type, while Figure 16.5 illustrates the different stope types for sublevel transverse stoping to reflect the multiple possible situations.

Table 16.12: Underground Mine Backfill Dilution Parameters

Parameters	Units	Value
Sublevel Transverse Stoping		
Primary Stope	%	2.0
Secondary Stope	%	9.7
Primary - Primary Stope	%	6.2
Secondary - Secondary Stope	%	13.9
Sublevel Longitudinal Stoping		
Primary Stope	%	5.9

Figure 16.5: Underground Mine Transverse Stope Types – Plan View



Source: GMS 2026 (not to scale).

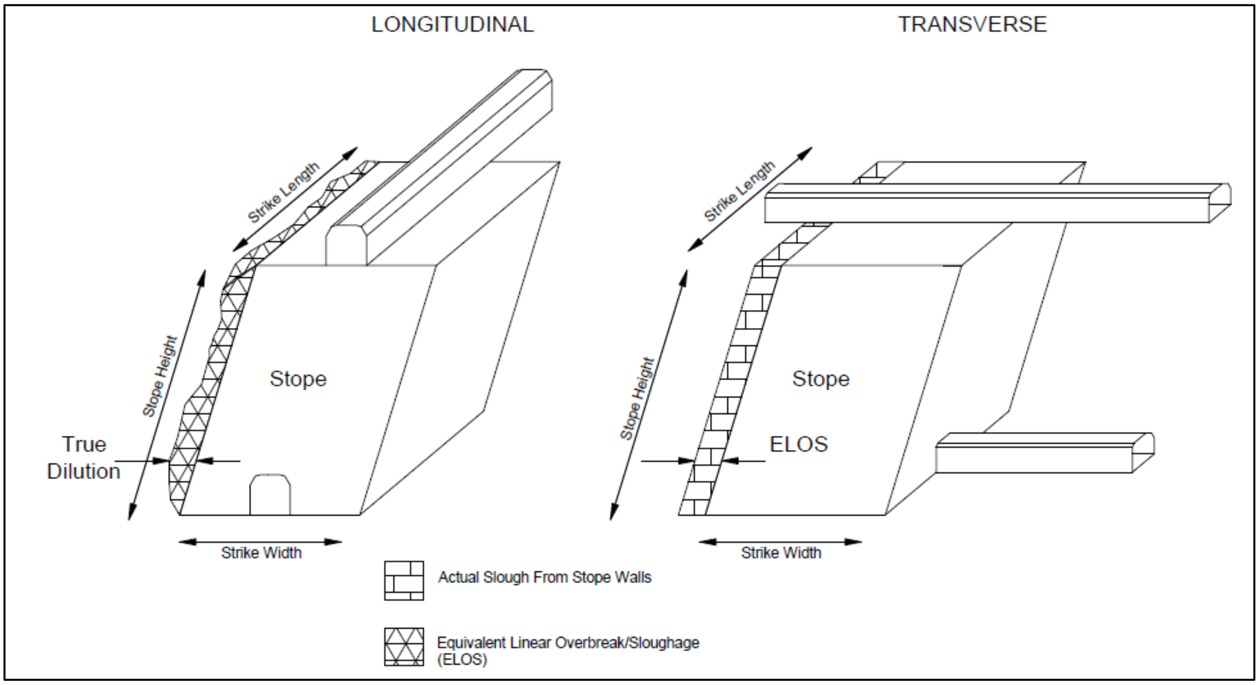
16.4.3.3 <u>Stope Optimization</u>

A series of iterations with the stope optimizer tool of Deswik™ software was performed to obtain the best possible stope shapes. The stope geometry parameters used in the stope optimizer tool are summarized in Table 16.13. Figure 16.6 presents a schematic representation of the various mining methods and shows that a consistent naming convention is applied to all stope types, regardless of the access location.

Table 16.13: Long Hole Open Stoping - Stope Optimizer Parameters

Item	Value
Deswik Stope Optimizer Parameters	
Stope Height	25 m
Strike Length	13 m
Maximum Mining Width (HW to FW)	15 m
HW Dilution	0.75 m
FW Dilution	0.5 m
Minimum Dip	55°
Crown Pillar Thickness	NA

Figure 16.6: Underground Mine Stope Types – Isometric View



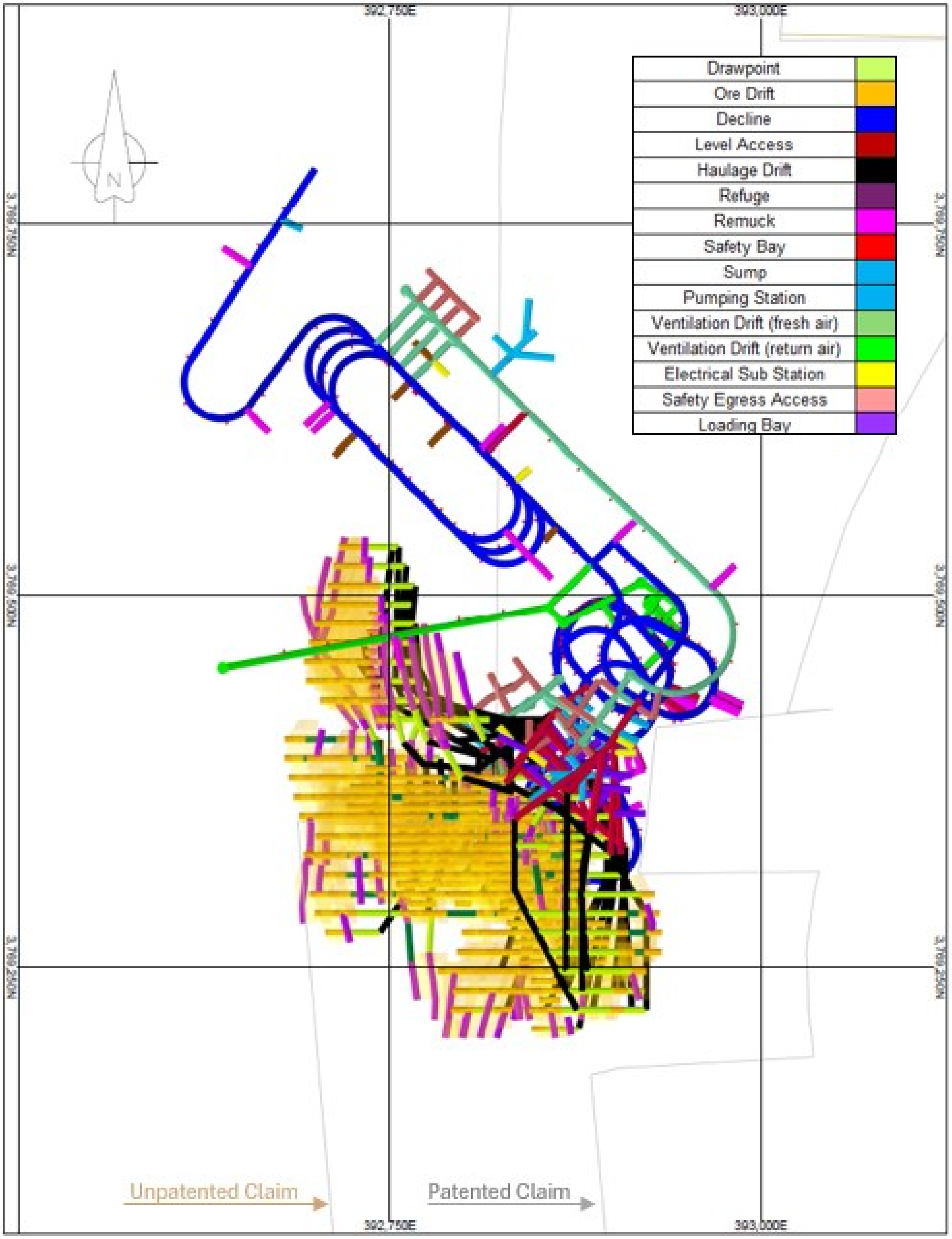
16.4.4 <u>Underground Mine Design</u>

16.4.4.1 <u>Development Design</u>

The underground operation consists of a single mine accessed via one (1) surface portal located south of the surface infrastructure area.

Figure 16.7 gives an overview of the position of the production zone relative to the mining claims.

Figure 16.7: Underground Mine Plan View

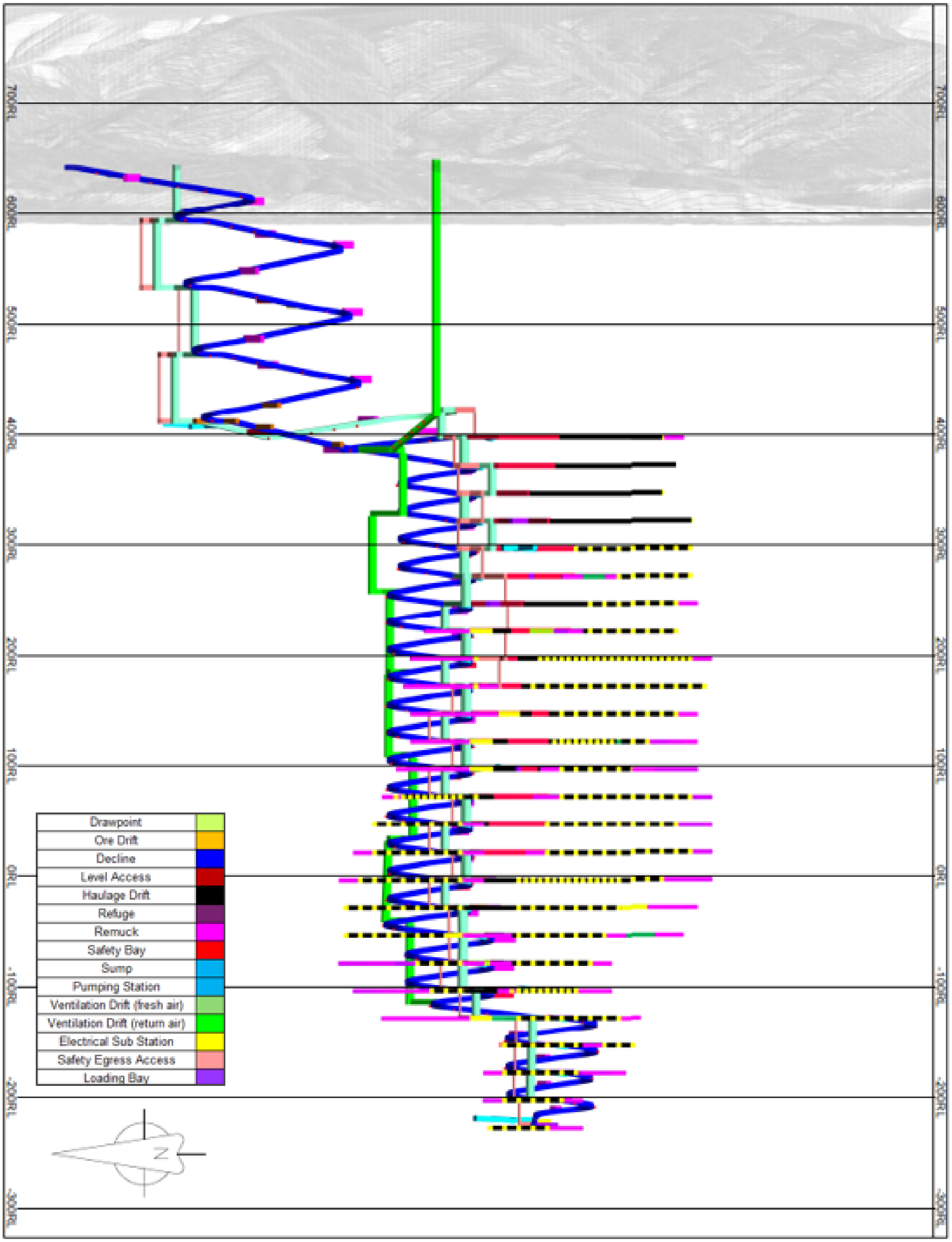


Source: GMS 2026 (not to scale).

From the portal, a ramp is developed to reach the production zone. Each production level is accessed through a level access and a haulage drift that leads to the crosscuts that are either driven perpendicular to the strike length of the stoping area for transverse stoping or along the strike of the mineralized zone for longitudinal stoping.

For a typical level access, the following infrastructure is included: sump, electrical bay, fresh air access and safety egress. For a typical haulage drift, a remuck is included. Figure 16.8 illustrates the overall development required.

Figure 16.8: Underground Mine Development Longitudinal View – Looking East



Source: GMS 2026 (not to scale)

Table 16.14 summarizes some of the most relevant parameters used for mine design, while Table 16.15 presents the various development factors applied to mine design. Table 16.16 identifies the different development types present in the mine and their respective dimensions.

Table 16.14: Underground Mine Lateral Development Parameters

Item	Measurements / Specifications
Ramp & Lateral Development	
Ramp Gradient	Nominal +/-15%
Ramp Gradient at Intersections	Reduced to 10%, then 5% & Flat at Intersections
Turn Radius (Ramp)	25 m
Ramp - Orebody Offset (minimum)	80 m
Footwall Drift - Orebody Offset (minimum)	20 m

Table 16.15: Underground Mine Development Factors

Parameters	Units	Value
Development Allowances		
Ramp (e.g., safety bays, TDB for fans, intersection slashes, cut-outs for pump boxes)	%	5
Lateral Development (CAPEX) (e.g., safety bays, TDB for fans, intersection slashes)	%	5
Lateral Development (OPEX) (e.g., TDB for fans, cutouts for remote mucking)	%	2.5
Infrastructure (e.g., slashes, TDB for fans and services)	%	2.5
Production Drift	%	0
Raises	%	0
Overbreak		
Development in Waste	%	10
Development in Mineralized Material	%	0
Raises	%	0

Table 16.16: Underground Mine Development Type and Dimensions

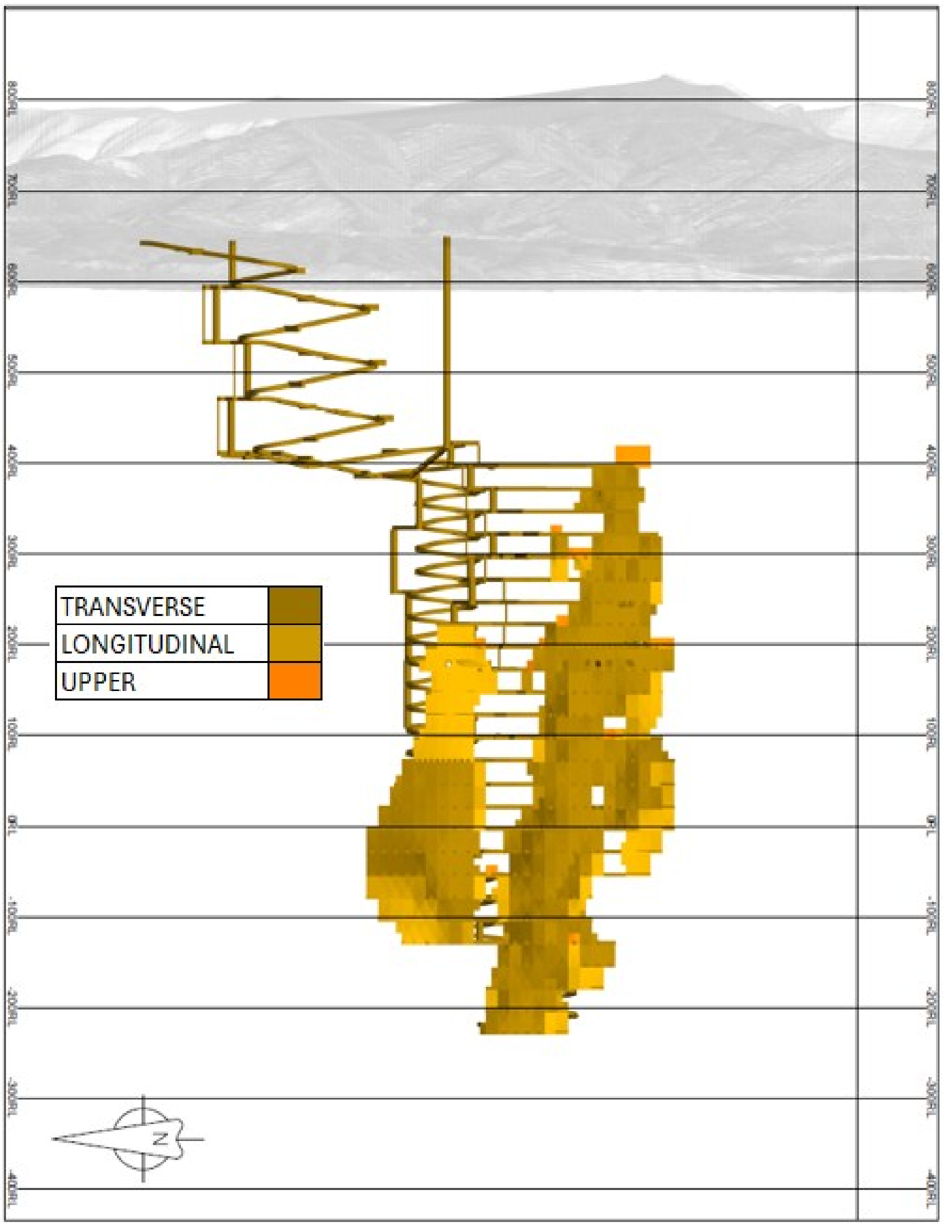
Development Type	Width (m)	Height (m)	Length (m)
CAPEX Development			
Decline	5.0	5.0	Variable
Level Access	5.0	5.0	Variable
Haulage Drift	4.5	4.5	Variable
Refuge	5.0	7.0	20.0
Remuck	5.0	7.0	20.0
Safety Bay	1.5	2.0	2.0
Sump	5.0	5.0	12.5
Pumping Station	5.0	5.0	Variable
Ventilation Drift	5.0	5.0	Variable
Explosive Magazine	5.0	5.0	20.0
Cap Magazine	5.0	5.0	20.0
Anfo Parking	5.0	5.0	20.0
Electrical Sub Station	5.0	5.0	12.5
Safety Egress Access	5.0	5.0	Variable
Loading Bay	5.0	5.0	20.0
Vertical CAPEX Development			
Fresh Air Raise (Drop Raise)	3.55	3.55	Variable
Return Air Raise (Drop Raise)	3.55	3.55	Variable
Return Air Raise (Raise Bore)	4.0 Dia.	-	Variable
Emergency Egress	1.2 Dia.	-	Variable
OPEX Development			
Drawpoint	4.5	4.5	20.0
Production Drift	4.5	4.0	Variable

Additionally, on the final levels where underlying stopes are present, once these stopes have been blasted and backfilled, sill development will be undertaken to provide overhead access for drilling. These developments must be executed on a just-in-time basis to avoid ground stability and ground support complications.

16.4.4.2 Stope Design

The Kay Mine mineralized zone is predominantly sub-vertical and exhibits significant variability in stope thickness. Where the thickness exceeds 15 m, the stope is subdivided to maintain a maximum stope width of 15 m. Approximately 11% of the stope tonnage will be mined using the sublevel longitudinal stoping variant of the LHOS mining method, while 88% will be mined using the sublevel transverse stoping variant. The remaining 1% of the total stope tonnage will be mined using the upper stoping variant of the LHOS mining method. Figure 16.9 presents the distribution of stopes mined, classified by mining method.

Figure 16.9: Underground Mining Method - Longitudinal View - Looking East

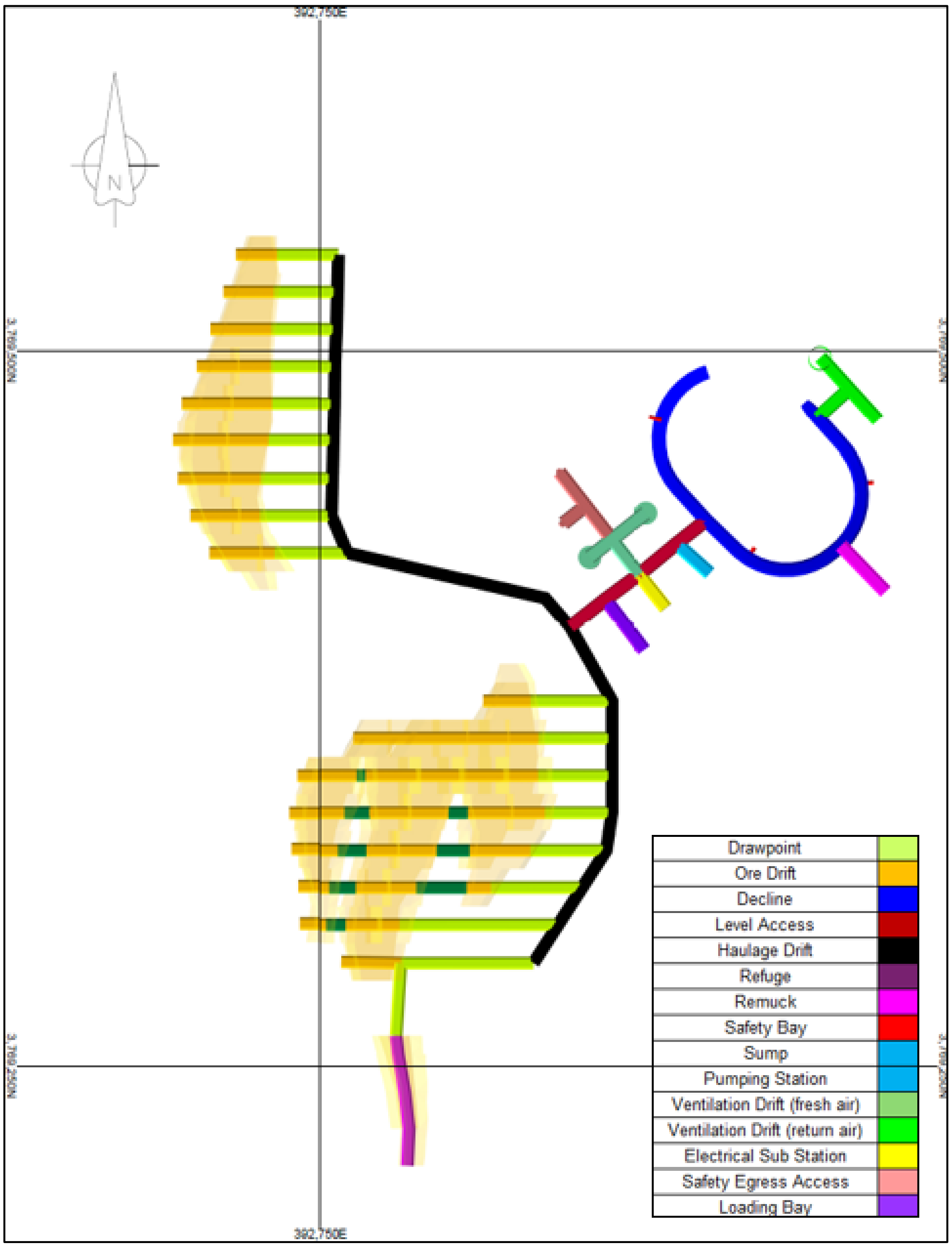


Development and production activities will be carried out concurrently in multiple mining blocks to maintain the availability of sufficient development headings and stopes.

The longitudinal stopes have an average thickness of 8.0 m and a stope tonnage averaging 8,000 t. The transverses stopes have an average thickness of 11.7 m and a stope tonnage averaging 11,600 t.

Figure 16.10 shows a plan view of a typical underground mine production level.

Figure 16.10: Underground Mine Typical Production Level - Plan View



Source: GMS 2026 (not to scale).

16.4.4.3 Physicals Summary

Table 16.17 summarizes the total development metres. It is important to note that all development quantities are expressed in linear metres rather than equivalent metres. Stoping and development tonnages and grades, including both mineralized material and waste development quantities, are summarized in Table 16.18.

Table 16.17: Underground Mine Design Summary

Development Type	Unit	Total
Lateral Development		
Main Decline	m	7,745
Level Access	m	2,311
Haulage Drift	m	4,729
Infrastructures	m	6,378
Sub-total CAPEX Development	m	21,163
OPEX Development	m	18,108
Total Lateral Development	m	39,271
Vertical Development		
Raise Bore	m	232
Drop Raise	m	1,295
Emergency Egress	m	757
Total Vertical Development	m	2,284

Table 16.18: Underground Mine Physicals Summary

Item	Unit	Total
Development Physicals		
Development Mineralized Material	t	586,115
	% Cu	0.84
	% Zn	2.11
	g/t Au	1.23
	g/t Ag	23.42
Development Waste	t	2,167,782

Item	Unit	Total
Production Physicals		
Stoping Mineralized Material	t	5,961,436
	% Cu	1.03
	% Zn	2.73
	g/t Au	1.64
	g/t Ag	29.62
Total Mine Physicals		
Total Underground Mineralized Material	t	6,547,550
	% Cu	1.01
	% Zn	2.67
	g/t Au	1.60
	g/t Ag	29.07

16.4.5 Underground Mine Operations

16.4.5.1 Development

Mine operations commence with the development phase. At Kay Mine, mechanized pneumatic mining equipment will be used to carry out the required lateral and vertical development to access the deposit. The development cycle begins with drilling of the heading, performed using a two-boom electric / hydraulic development drill (jumbo). The drilling pattern incorporates a reamed cut, allowing a drilled length of 4.66 m to achieve an effective break of approximately 95%. The planned drillhole diameter is 51 mm; however, this parameter may be adjusted during operations based on blasting performance. The jumbo penetration rate is estimated at 1.20 m/min for regular holes and 0.50 m/min for reamed holes. The average drilling time per round, including delays, is estimated at 4.0 h per round, although this value is strongly dependent on excavation dimensions. The advance rate per jumbo is assumed to be 10 m/d when multiple faces are available. Of this total, 5.0 m/d is allocated to the priority heading, with the remaining 5.0 m/d distributed equally between the next two (2) development headings (2.5 m/d each).

Once drilling is completed, the blasting crew will load the drillholes with explosives. Bulk emulsion explosive will be used for development blasting. Perimeter control drilling will be implemented to minimize dilution. Loaded rounds will be blasted at the end of each shift. A one-hour re-entry period is planned following blasting to allow for gas clearance. Both the main access and ventilation exhaust raises will be monitored using gas detectors.

The third step of the development cycle involves mucking the blasted material from the development face using a load-haul-dump (LHD) unit. Material is removed from the face as quickly as possible to allow the subsequent stages of the development cycle to proceed. LHD performance is influenced by the ramp gradient and the haul distance between the development face and the remuck location and therefore can vary significantly. To minimize haulage distances, the unloading point (remuck) is planned to be located within a maximum distance of 150 m from the working face.

The final step of the development cycle is the installation of ground support at the development face. Ground support will be installed using a platform bolter. The ground support pattern is described in the corresponding section of this report.

Major vertical development from surface, such as the ventilation exhaust raise, will be executed by a contractor specializing in raise boring. It is assumed that a raise-boring crew can advance the raise at a rate of approximately 3.0 m/d. This rate includes the pilot drilling, reaming and the raise-boring process. In terms of in-mine raise excavation, most of the vertical development is planned as drop-raises or raise-bored safety egresses that will be done by the owner’s equipment.

16.4.5.2 Stoping

The second stage of mine operations is the production phase, also referred to as the stoping phase. The stoping phase begins with stope preparation, the first step of which is slot raise drilling. Slot raises are assumed to be drilled using a mobile raise-boring machine equipped with a 30-inch boring head. The next step in the stoping phase is the drilling of production holes, which will be carried out using top-hammer production drills. Once a stope has been fully drilled, the production holes are loaded with bulk emulsion explosives. At Kay Mine, a mechanized explosive loader will be used to load the production holes in preparation for blasting. Depending on stope size and geometry, up to three (3) blasting and mucking cycles may be required to extract all material from a stope. Blasting of loaded stopes will be conducted at the end of a shift. A one-hour re-entry period is planned following blasting to allow for gas clearance.

Broken material will be mucked using 14-t diesel-powered load–haul–dump (LHD) units to a remuck location or dumped directly into 45-t diesel-powered underground haul trucks. Mineralized material from the stopes will then be hauled to surface via the decline and discharged either directly into the crusher or onto the ROM pad, depending on crusher availability.

Once all material has been extracted from a stope, the backfilling process, the final step of the stoping cycle, can be initiated. For transverse mining, primary stopes and constrained secondary stopes (i.e., those adjacent to another planned secondary stope) will be backfilled with cemented rockfill, while unconstrained secondary stopes will be backfilled with uncemented rockfill to achieve cost savings. For longitudinal mining, all stopes will be backfilled using a combination of cemented rockfill (CRF) and uncemented rockfill. The final stope in the sequence, which will not be in contact with a subsequent stope, will be backfilled with uncemented rockfill.

This backfilling process is expected to take between one (1) and two (2) weeks, depending on stope size. A short cure period of seven (7) days and a long cure period of twenty-eight (28) days are planned for each stope backfilled with CRF to allow the backfill to achieve the required strength and to minimize backfill dilution. Waste rock for backfilling activities will be primarily sourced from development waste, with any additional material supplied from surface. Cement slurry for CRF will be transported from surface using mobile transmixers. Further details regarding the CRF batch plant and backfilling process are provided in Section 0.

16.4.5.3 Box-cut

The Project will require the construction of a single mine portal. The portal, located to the south of the infrastructure area, will provide access to the orebody. Excavation will be limited to the minimum required to establish a stable face in competent rock from which tunnel development can commence. Detailed portal design and ground support guidelines will be developed during subsequent phases of the Project.

16.4.5.4 Raise Collar

It is currently planned to have two (2) raise collars. Detailed designs for these raise collars will be developed during a subsequent phase of the Project.

16.4.6 Underground Mine Development and Production Rates

The targeted underground mine production rate for Kay Mine is 1,910 tpd, equivalent to approximately 0.7 Mt of mineralized material per year. This total includes stope production of 1,730 tpd and lateral development within mineralized material of 180 tpd. The achieved production rate varies slightly over the life-of-mine (LOM), as the quantity of development within mineralized material is not constant. Multiple mining blocks are mined simultaneously to maintain the targeted underground mine production rate. The production rate for the Kay Mine underground operation is calculated using the Deswik™ mining sequence, based on the task-specific production rates presented in Table 16.19.

Table 16.19: Underground Mine Scheduler Task Rates

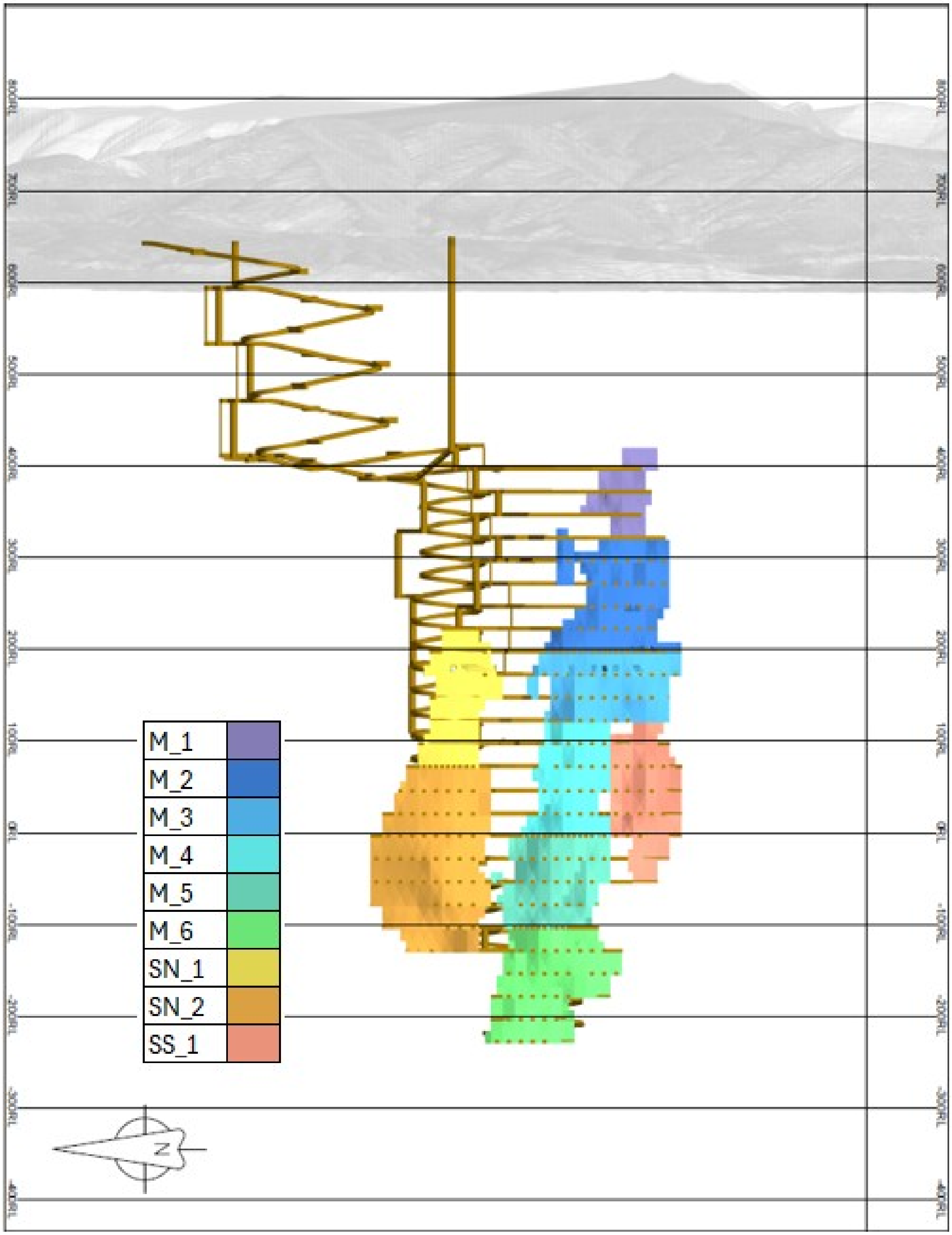
Parameter	Units	Rate
Single-Face Development Rate	m/d/unit	5.0
Multi-Face Development Rate	m/d/unit	10
Stope Preparation	d	5
Stope Cables	m/d	120
Slot Raise Drilling Rate	m/d	5
Production Drilling Rate - Transverse	m/d	200
Production Drilling Rate - Longitudinal	m/d	200
Production Drilling Factor - Transverse	t/m drilled	12.0
Production Drilling Factor - Longitudinal	t/m drilled	8.0
Blasting Delay	d	3
Mucking Rate	t/d	875
Rockfill Rate	t/d	800
Cemented Rockfill Rate	t/d	640
Short Cure Time	d	7
Long Cure Time	d	28
Maximum Stoping	t/d	1,750

16.4.7 Underground Mine Development and Production Sequencing

Once excavation and construction of the portal are completed, the development phase will commence with the advancement of the main decline toward the production zone.

While the objective is to access the first stoping areas as quickly as possible, the development of the primary ventilation and safety egress networks is also a priority, as these are essential to allow stoping to begin. The initial focus is to achieve rapid entry into production at the top of the orebody. Ramp development will then continue to provide access to additional mining blocks, enabling the simultaneous extraction of multiple blocks and supporting a consistent feed to the mill throughout the life-of-mine (LOM). Figure 16.11 presents the distribution of stopes mined, classified by mining blocks.

Figure 16.11: Underground Mining Blocks – Longitudinal View - Looking East



The mineralized material production profile (mined material) of the underground mine is summarized by zone in Figure 16.12, and the underground mine production plan (mined material) is presented in Table 16.20.

Figure 16.12: Underground Mine Mineralized Material Production by Sub-Zone

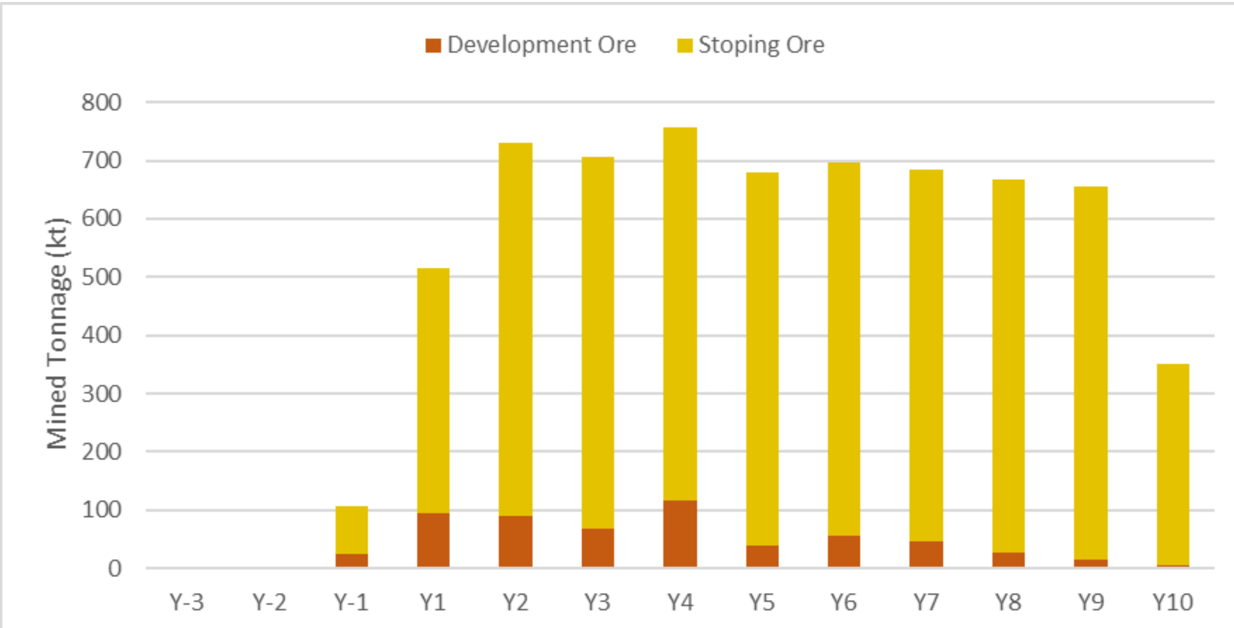


Table 16.20: Underground Mine Production Plan

Kay Mine	Unit	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Development Physicals															
Development Mineralized Material	kt	586	-	0.3	25	94	91	68	116	40	57	46	26	16	6
	% Cu	0.84	-	0.61	0.62	0.68	1.00	0.79	0.91	0.98	0.84	0.88	0.71	0.66	0.68
	% Zn	2.11	-	0.77	1.69	2.22	2.07	1.89	1.90	2.05	2.66	2.19	2.23	2.44	2.72
	g/t Au	1.23	-	0.12	0.76	0.97	1.19	1.16	1.08	1.47	1.88	1.49	1.32	1.26	1.44
	g/t Ag	23.42	-	7.10	19.39	30.29	24.64	21.08	20.16	21.62	24.11	22.77	23.94	22.55	15.14
Development Waste	kt	2,168	111	274	473	430	418	256	64	8	33	55	20	19	6
Production Physicals															
Stoping Mineralized Material	kt	5,961	-	-	82	421	639	639	640	639	639	639	640	639	344
	% Cu	1.03	-	-	0.79	0.73	0.85	0.98	1.18	1.10	1.22	1.24	1.09	0.94	0.79
	% Zn	2.73	-	-	1.71	2.59	2.54	2.48	2.64	2.70	2.63	2.48	2.95	3.34	3.25
	g/t Au	1.64	-	-	0.97	1.05	1.14	1.19	1.48	1.78	1.92	1.68	2.08	2.16	1.88
	g/t Ag	29.62	-	-	20.52	31.59	34.71	30.73	28.25	26.66	29.52	25.13	32.60	31.63	25.24
Total UG Mine Physicals															
UG Mined Mineralized Material	kt	6,548	-	0	108	516	729	707	757	679	696	685	667	655	350
	% Cu	1.01	-	0.61	0.75	0.72	0.87	0.96	1.14	1.09	1.19	1.21	1.07	0.94	0.79
	% Zn	2.67	-	0.77	1.71	2.53	2.49	2.43	2.53	2.66	2.63	2.46	2.92	3.32	3.24
	g/t Au	1.60	-	0.12	0.92	1.03	1.15	1.19	1.42	1.76	1.92	1.67	2.05	2.14	1.87
	g/t Ag	29.07	-	7.10	20.25	31.35	33.45	29.80	27.01	26.36	29.08	24.97	32.26	31.41	25.06
UG Mined Waste	kt	2,168	111	274	473	430	418	256	64	8	33	55	20	19	6

16.4.8 Underground Mine Equipment

Underground equipment requirements were determined based on the operating hours required to achieve the projected production and development rates defined in the mine plan.

During the production years, haulage cycles consider the distances from the loading point in the footwall drifts to the level access, then up the ramp to the surface destination (crusher or pads and stockpiles). Mucking and hauling cycles are defined based on fixed distances between the stopes and the truck loading points, or between the stopes and the truck loading bays.

Quantities of non-critical auxiliary equipment were estimated based on the scale of the operation or were derived from the requirements of other equipment. Table 16.21 summarizes the equipment requirements for two (2) distinct stages of the life of mine (LOM): the mine pre-production stage and the full-production stage.

Table 16.21: Underground Mine Mobile Equipment Fleet

Equipment Type	Avg. Qty Pre-Production	Avg. Qty Production*	Max Qty LOM
Jumbo – 2 Boom	2	2	2
Bolter	3	3	4
Production Drill – Top Hammer	1	1	1
Production Drill – Easer L	1	1	1
Cable Bolter	1	1	1
LHD - Type I	2	4	4
Truck – Type I	3	7	8
Explosive Truck - Development	1	1	1
Explosive Truck - Production	1	1	1
Scissor Lift - Development	2	2	2
Scissor Lift - Construction	1	1	1
Boom Truck	1	1	1
Fuel & Lube Truck	1	1	1
Water Truck	1	1	1
Block Holer	1	1	1
Scaler	1	1	1

Equipment Type	Avg. Qty Pre-Production	Avg. Qty Production*	Max Qty LOM
Cassette Truck	1	2	2
Shotcrete Sprayer	1	1	1
Shotcrete Mixer	1	1	1
Transmixer	1	1	1
Grader	1	1	1
Backhoe Loader	1	1	1
Personnel Carrier	1	1	1
Tractor	2	3	3
Light Vehicle	4	6	6
Total UG Mobile Equipment	36	46	48

**Note: Reference Year 3.*

16.4.9 Underground Mine Labour

The Kay Mine property is located immediately adjacent to the town of Black Canyon City, approximately 69 km (43 miles) north of the city of Phoenix, in central Arizona, USA. It is anticipated that the majority of the workforce will be sourced from the region. Operational positions are planned to be staffed on a rotational work schedule. Management and support service personnel will be employed on a standard 5 days on / 2 days off (5/2) schedule. A total workforce of 238 workers is expected to be employed for the Kay Mine underground operation. The summary of the underground mine labour at the peak of the mine is described in Table 16.22.

Table 16.22: Underground Mine Workforce Summary

Position	Schedule	Max Qty
UG Engineering		
Chief Mining Engineer	5/2	1
Assistant Chief Mining Engineer	5/2	1
Mine Engineer	5/2	4
Mine Technician	5/2	4
Surveyor	5/2	4

Position	Schedule	Max Qty
UG Geology		
Chief Geologist	5/2	1
Senior Geologist	5/2	2
Geologist	5/2	2
Geology Technician	5/2	4
UG Mine Operations		
Mine Manager	5/2	1
Mine Superintendent	5/2	1
Mine Ops. General Foreman	14-14	2
Clerk	5/2	1
Supervisor	14-14	8
Trainer	14-14	2
Long-Hole Driller	14-14	6
Blasters	14-14	8
Scoop Operators	14-14	16
Truck Operators	14-14	32
Jumbo Operator	14-14	8
Rock Bolter	14-14	14
Cable Bolter	14-14	2
Scissor Lift Operator	14-14	12
Level Services	14-14	8
Grader Operator	14-14	2
U/G Constructions	14-14	4
Sumps and Services Labour	14-14	2
Boom Truck Operator	14-14	2
Fuel Truck Operator	14-14	2
Lamps-Dry	14-14	2
Labour Spare	14-14	8

Position	Schedule	Max Qty
UG Mine Maintenance		
Superintendent	5/2	1
General Foreman	14-14	4
Mechanical Engineer	14-14	2
Trainer	14-14	2
Maintenance Planner	14-14	2
Mechanics - Mobile Equipment	14-14	27
Mechanics - Fixed Equipment	14-14	4
UG Mine Electrical		
Supervisor	14-14	4
Planner	14-14	2
Trainer	14-14	2
Electricians - Mobile Equipment	14-14	18
Electricians - Instr. Technicians	14-14	2
Electricians - Fixed Equipment	14-14	2

16.4.10 Underground Mine Ventilation and Cooling

16.4.10.1 Ventilation Fresh Air Requirements

Ventilation requirements for the underground mine are primarily governed by diesel emissions from the mobile equipment fleet. Light vehicle airflow requirements are defined in accordance with CANMET guidelines, while other mobile equipment requirements are established in compliance with MSHA standards. These criteria are used to determine the airflow demand for each equipment type, with applicable attenuation factors applied based on the estimated equipment utilization. Table 16.23 summarizes the typical ventilation fresh air requirements for equipment used underground. Preliminary Ventsim™ ventilation models have been developed, and simulations have been completed for the maximum productivity scenario.

Table 16.23: Underground Mine Fresh Air Requirements per Equipment

Equipment	Engine	HP	CFM / EQUIP	Utilization Factor
Jumbo – 2 Boom	Cummins QSB4.5	170	8,500	50%
Bolter	Cummins QSB4.5	170	8,500	50%
Production Drill – Type I	Mercedes OM904 series	147	6,500	50%
Production Drill – Type II	Mercedes OM904 series	147	6,500	50%
Cable Bolter	Mercedes OM904 series	147	6,500	50%
LHD – Type I	Volvo TAD1340VE	343	15,500	80%
Truck – Type I	Volvo TAD1341VE (Tier 2)	603	30,000	80%
Explosive Truck - Development	Cummins QSB4.5	170	8,500	65%
Explosive Truck - Production	Cummins QSB4.5	170	8,500	65%
Scissor Lift - Development	Cummins QSB4.5	170	8,500	65%
Scissor Lift - Construction	Cummins QSB4.5	170	8,500	65%
Boom Truck	Cummins QSB6.7	193	11,000	65%
Fuel & Lube Truck	Cummins QSB6.7	193	11,000	65%
Water Truck	Volvo TAD580	141	13,600	65%
Block Holer	Cummins QSB4.5	170	8,500	65%
Scaler	Volvo TAD580	141	13,600	65%
Cassette Truck	Volvo TAD580	141	13,600	65%
Shotcrete Sprayer	Cummins QSB4.5	170	8,500	65%
Shotcrete Mixer	Cummins QSB4.5	170	8,500	65%
Shotcrete and Concrete Transport	Cummins QSB 6.7	193	11,000	65%
Grader	CAT C7 ACERT™ (Tier 2)	145	20,864	65%
Backhoe Loader	CAT 3054	86	10,000	65%
Personnel Carrier	Volvo TAD582	215	13,600	65%
Tractor	Mercedes OM904 series	147	6,500	65%
Light Vehicle	1 Hz PCNA	127	7,300	65%

Table 16.24Table 16.24 illustrates the CFM requirement per equipment type, while Table 16.25 demonstrates the maximum ventilation fresh air requirements quantities used for the Kay Mine underground operation.

Table 16.24: Underground Mine Fresh Air Requirements per Equipment Type

Equipment	Qty Total	CFM Requirement
Jumbo – 2 Boom	2	8,500
Bolter	4	17,000
Production Drill – Type I	1	3,250
Production Drill – Type II	1	3,250
Cable Bolter	1	3,250
LHD – Type I	4	49,600
Truck – Type I	8	192,000
Explosive Truck - Development	1	5,525
Explosive Truck - Production	1	5,525
Scissor Lift - Development	2	8,500
Scissor Lift - Construction	1	5,525
Boom Truck	1	7,150
Fuel & Lube Truck	1	7,150
Water Truck	1	5,525
Block Holer	1	5,525
Scaler	1	8,840
Cassette Truck	2	17,680
Shotcrete Sprayer	1	5,525
Shotcrete Mixer	1	5,525
Shotcrete and Concrete Transport	1	7,150
Grader	1	13,562
Backhoe Loader	1	6,500
Personnel Carrier	1	8,840
Tractor	3	12,675
Light Vehicle	6	28,470

Table 16.25: Maximum Ventilation Requirement

Item	Max CFM
CFM Requirement	442,000
Contingency	28,000
Total Ventilation	470,000

16.4.10.2 Ventilation Design

A phased ventilation approach will be implemented for the Kay Mine. Temporary systems will support the main decline development until the permanent ventilation network is constructed and commissioned. The permanent system is designed to meet fresh air requirements from pre-production through full production.

The temporary ventilation system has been designed to satisfy the fresh air requirements for a single underground development crew advancing the main ramp. The required airflow will be generated by a 250 HP fan and conveyed to the active heading via 48-inch diameter rigid ventilation ducting.

The mine will be ventilated using a mechanized push ventilation system consisting of one (1) fresh air raise (FAR), one (1) return air raise (RAR), with the main decline acting as an exhaust. A total of two (2) main fans will be installed in parallel underground at the base of the main fresh air raise. These fans will force air through the fresh air raise system, which connects to each level access to supply fresh air to the mine headings. Stale air will be exhausted via the decline, and the associated exhaust raise system. The emergency egress network has been planned adjacent to the fresh air network to ensure the provision of fresh air within the egress raises. Table 16.26 summarizes the design parameters for the two (2) main fans, including installed power (kW), operating pressure, and airflow.

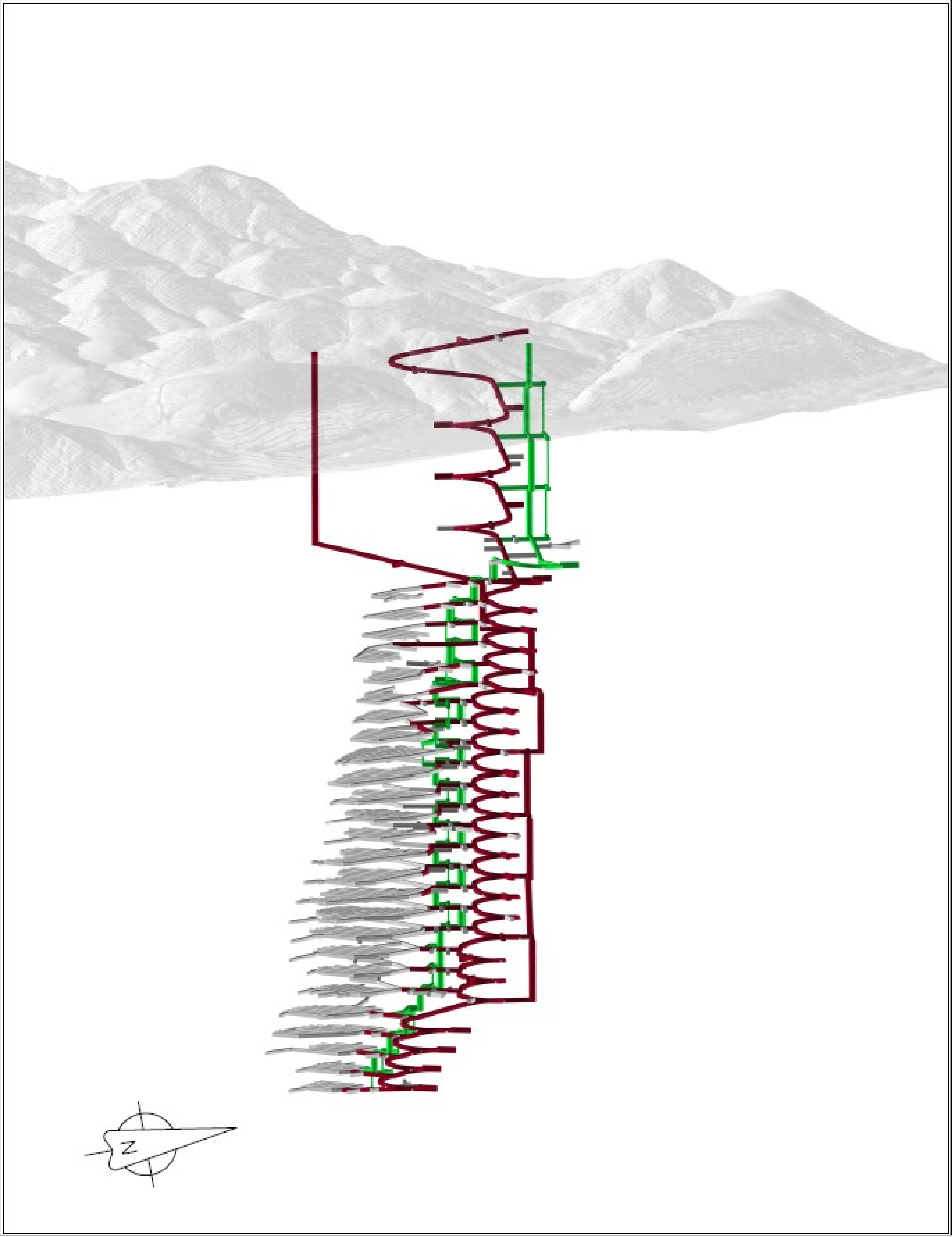
Table 16.26: Underground Mine Ventilation Main Fans Design Values

Item	Unit	Design Value
Main Fans		
Fans	kW	2 x 600
Pressure	In. wg.	12.3
Airflow	kCFM	470

Major vertical development from surface, such as the main ventilation exhaust raise, will be executed by a contractor specializing in raise boring. The remaining raise infrastructure will consist of a series of drop raises.

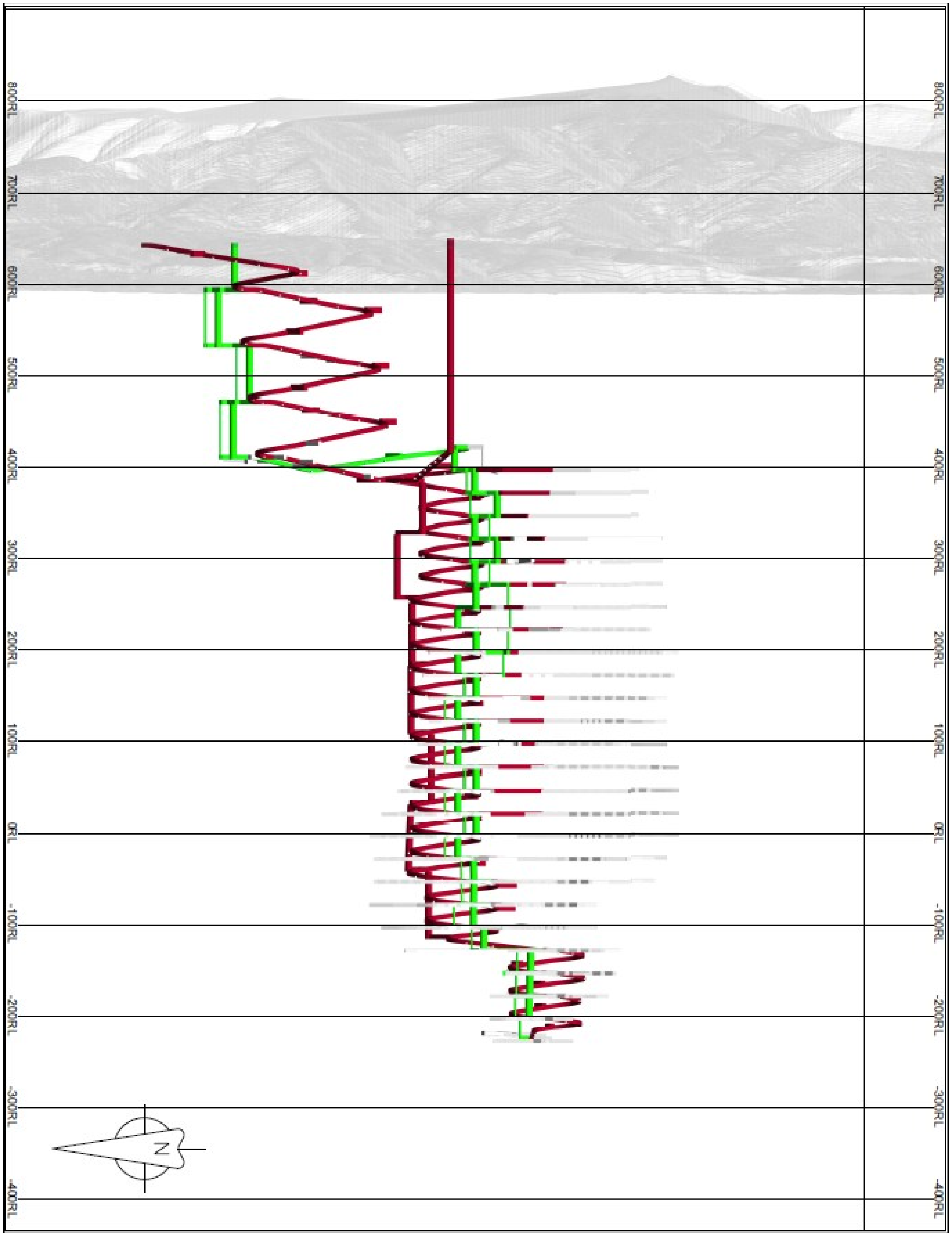
Figure 16.13 and Figure 16.14 illustrate the permanent underground mine ventilation network. Fresh airflow is indicated in green, the exhaust system in red, and areas supplied by auxiliary ventilation fans in grey.

Figure 16.13: Underground Mine Ventilation Network – Isometric View



Source: GMS 2026 (not to scale).

Figure 16.14: Underground Mine Ventilation Network – Longitudinal View - Looking East



Source: GMS 2026 (not to scale).

The permanent ventilation systems will operate under a range of pressures and airflow rates to accommodate varying operating conditions across the mine. Ventilation control walls installed at the fresh air raise (FAR) access on each level will ensure that adequate quantities of fresh air are distributed to the appropriate workplaces. On each level, access to the FAR network at the level access will promote flow-through ventilation along the extended portions of the production levels

As the mine site is located in a region where ambient temperatures can exceed 40°C during the summer months, the installation of a water-cooled refrigeration system at the intake raise is planned. The cooling system is intended to provide discharge air at approximately 12°C to support suitable underground working conditions. Cooling requirements and system design criteria will be further evaluated and confirmed during subsequent phases of the Project.

16.4.11 Underground Mine Services

16.4.11.1 Electrical Distribution

The power generation infrastructure will be located near the processing plant and will power the underground mine through a 4.16 kV, 60 Hertz (Hz) distribution network stepping down to a voltage of 1,000 V. The permanent system will deliver power via cables installed along the decline to supply the main production areas.

During the decline development, one (1) mobile sub-station will be installed every 600 m. As development progresses, the substation will be relocated to the subsequent electrical bay. Near infrastructures with high power requirements, such as pumping stations, mobile substations will initially be used to power the development and then left in place to supply the infrastructures.

For the main production area, an electrical bay will be developed at each of the three (3) production levels. Each substation will supply its level as well as the immediate upper and lower levels. To supply the upper and lower levels, the electrical cables will pass through service holes.

16.4.11.2 Dewatering

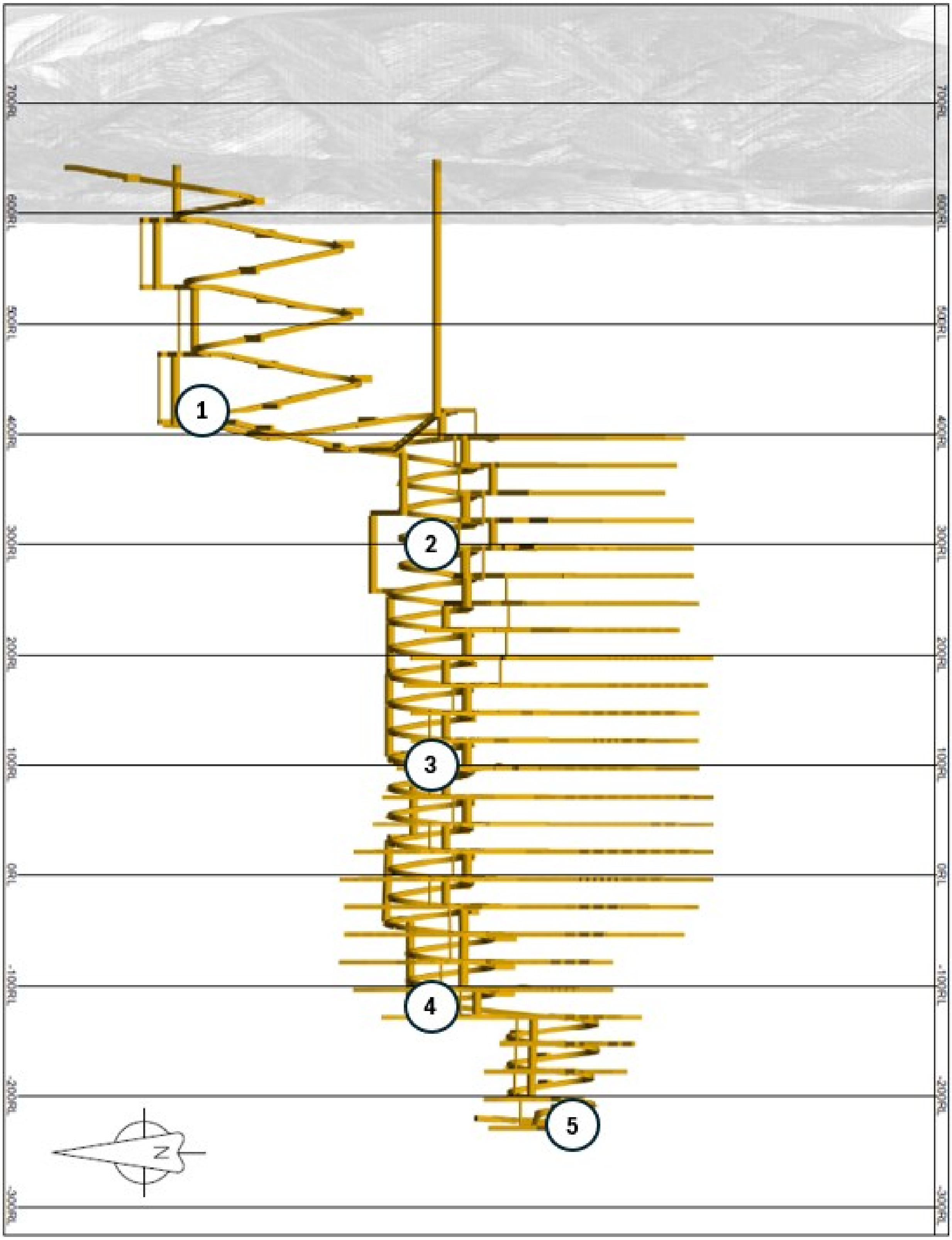
In the absence of site-specific hydrogeological data and a completed mine water balance, dewatering requirements for the Kay Mine cannot be reliably quantified at this stage. A review of historical technical reports for the Kay Mine indicates that no records are available regarding underground dewatering or pumping rates from previous operations.

A review of historical data from mines in the region has established an estimated dewatering requirement of 5,000 m³/day (3,472 L/min). This estimate is subject to significant uncertainty and will be refined during a subsequent phase of the Project.

Water collected from underground workings will be pumped to surface through a series of staged pumping stations and discharged into a surface storage pond. This water will be recycled to support ongoing mining and milling operations. Five (5) pumping stations have been strategically planned at various elevations to progressively convey underground water to surface. Infiltration water and operational water will gravitate through the workings and be collected in designated sumps, from which it will be pumped to surface. Each pumping station will be equipped with settling bays to facilitate solids management and sludge handling.

Figure 16.15 illustrates the dewatering network, while Table 16.27 summarizes the mine pumping horsepower (HP) requirements.

Figure 16.15: Underground Mine Dewatering Network Longitudinal View - Looking East



Source: GMS 2026 (not to scale).

Table 16.27: Underground Mine Pumping HP Requirements

	HP Required
Pumping Station 1	882
Pumping Station 2	390
Pumping Station 3	245
Pumping Station 4	107
Pumping Station 5	50

16.4.11.3 Cemented Rockfill Plant

The use of the sublevel transverse stoping and sublevel longitudinal stoping variants of the LHOS mining method requires the use of cement backfill. Cemented rockfill (CRF) has been selected as the cemented fill material for the underground mine. CRF is a mix of waste rock and cement slurry. In this case, the cement slurry will be produced on surface at the same concrete batch plant used for construction purposes. Cement slurry will be transported underground to the desired location by mobile transmixers.

The waste rock for rockfill and cemented rockfill will primarily be sourced from development waste, while any additional waste required will be obtained from surface. Waste material sourced from surface will be crushed and backhauled underground in the same trucks that transported the mineralized material to the surface. On production levels, the truck will dump waste into a drawpoint that is subsequently mucked out. Transmixers will discharge cement slurry into a cement slurry reservoir located at another identified draw point.

Cement will be pumped to the mixing pit, and an LHD will add the waste into the mixing pit according to the specified recipe and cement content required for the stope. Once mixed to specification, the cemented rockfill will be transported by LHD and dumped from the upper level to fill the open stope. The planned cement content is 5% for regular stopes and 10% for sill level stopes, where stopes will be mined under a backfilled stope. Figure 16.16 and Figure 16.17 illustrate the CRF process.

Figure 16.16: General CRF Process Map

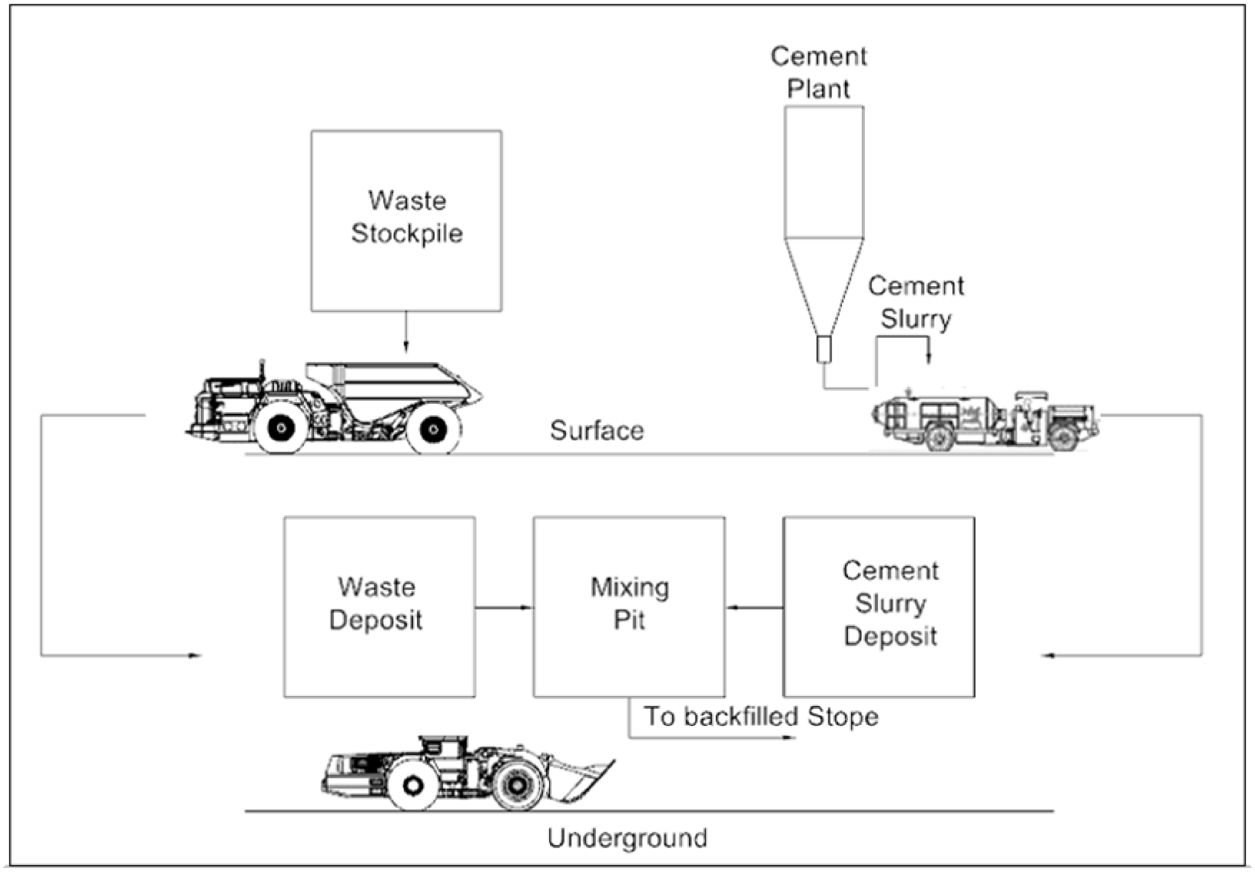
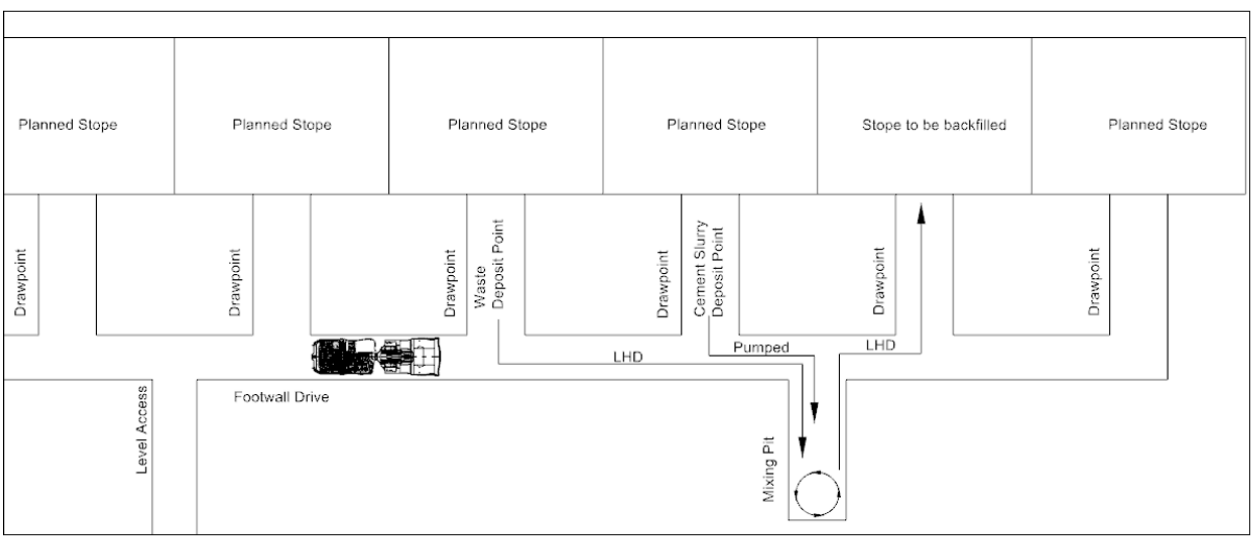


Figure 16.17: Production Level CRF Process Map



16.4.11.4 Compressed Air

The compressed air supply will be provided by a series of electric compressors installed on surface. The compressed air piping network will be installed along the decline, in the main drifts and in the emergency escapeways throughout the mine. Compressed air will provide power to the dewatering pumps servicing the development headings, to handheld drills, to some air-powered actuators, as well as to other air-powered equipment. The compressors will also provide an emergency air supply to the refuge stations.

During the development phase, prior to the installation of the permanent compressed air network, a temporary compressor will be utilized to supply compressed air for mining operations. This temporary unit will be positioned near the portal to ensure effective air delivery to the work areas.

16.4.11.5 Communications

The underground communication network consists of an LTE system that will be installed on site and will be expanded over the LOM. Mobile equipment operators, light vehicles, and supervisors will be equipped with LTE phones to communicate with surface personnel. LTE will also enable the use of remote-control features for selected equipment.

16.4.11.6 Fuel Storage and Distribution

Fuel will be stored on surface. A fuel truck will form part of the mobile equipment fleet to distribute fuel to underground equipment that cannot readily travel to the underground fuel station for refuelling.

16.4.11.7 Explosives Storage and Handling

One (1) underground explosive magazine and one (1) detonator magazine will be installed at designated locations. Explosives will be delivered to the portal by the selected explosive supplier and subsequently transported to the underground magazines by flatbed service truck for later use.

16.4.11.8 Personnel and Underground Material Transportation

Supplies and personnel will access the underground mine via the main ramp. Several personnel carriers, such as land cruisers, will be used to transport workers from surface to the underground mine. Supervisors and technical services will also use some light vehicles for transportation underground. The maintenance team, as well as mechanical and electrical personnel, will utilize maintenance tractors equipped with attachments such as forks, buckets, booms, or other tools to assist them in performing their duties.

Additionally, a cassette truck will be used to move supplies from the surface to the underground active headings, stopes and material storage.

16.4.11.9 Equipment Maintenance

The maintenance for all equipment is done following the supplier's recommendations. To maximize equipment lifespan, rebuilds are planned following the supplier rebuild cycle recommendations. Major mechanical maintenance activities will be carried out at the surface workshop. Some minor maintenance and small emergency interventions will be performed underground in a dedicated maintenance bay excavated and equipped for this purpose.

16.4.12 Underground Mine Safety Measures

16.4.12.1 Emergency Exits

The main decline will provide primary egress from the underground workings. For secondary egress, some independent egress raises will be excavated between production levels for most of the mine. It is important to note that some ventilation transfer drifts between the different zones will also serve as secondary egress. The independent safety egress raises will be equipped with prefabricated modular Laddertube systems.

16.4.12.2 Refuge Stations

Refuge stations will be positioned so that all employees can access a refuge in less than 15 minutes from the moment they leave their workplace, with refuge stations located at intervals of not more than 1,000 metres along main travel ways.

Each refuge station will be equipped with the following:

- Telephone or radio for communication with surface, independent of mine power supply.
- Compressed air, water lines, and water supply.
- Emergency lighting.
- Hand tools and sealing material.
- Plan of the underground work showing all exits and the ventilation plans.
- All other necessary items according to the applicable regulation.
- Fire protection.

16.4.12.3 Fire Protection

Underground mobile vehicles will be equipped with automatic fire suppression systems in accordance with best practices. Fire extinguishers will be provided and maintained in accordance with regulations and best practices at electrical installations, pump stations, fuel stations, service garages and wherever a fire hazard exists. Every vehicle will carry at least one (1) fire extinguisher of adequate size and proper type.

16.4.12.4 Mine Rescue

Fully trained and equipped mine rescue teams will be established in accordance with applicable regulations. Mine rescue equipment, including a dedicated underground emergency vehicle and a foam generator, will be available on site.

Rescue teams will be trained for surface and underground emergencies. An emergency response plan will be developed and continuously updated as the mine and regulations evolve.

16.4.12.5 Emergency Stench System

A mine stench gas warning system will be installed at the main surface ventilation system. This system is designed to inject a specific dosage of stench gas, calculated based on airflow quantity. This gas with a particular smell would alert the workers to an emergency as soon as they smell the gas. Another mine stench gas warning system will be installed at the mine compressed air system as a second means to alert underground workers in the event of an emergency.

16.5 Production Schedule

16.5.1 Processing Schedule

The mill schedule includes a six (6)-month pre-production period followed by a nine (9)-month ramp-up during the production period. Ramp-up commences at approximately 55% of nameplate throughput and increases monthly to reach full throughput after nine (9) months. The commercial throughput of 0.7 Mtpa is then maintained for approximately seven (7) years, followed by a gradual ramp-down over the final two (2) years to approximately 50% throughput in the last production year. Mill feed will be maximized through a combination of direct feed from the underground mine and rehandled stockpiled material.

Figure 16.18 and Table 16.28 depict the milled tonnage and feed grade per period going into the mill.

Figure 16.19 shows the quantity of waste or mineralized material over the LOM.

Figure 16.18: Mill Feed and Feed Grade

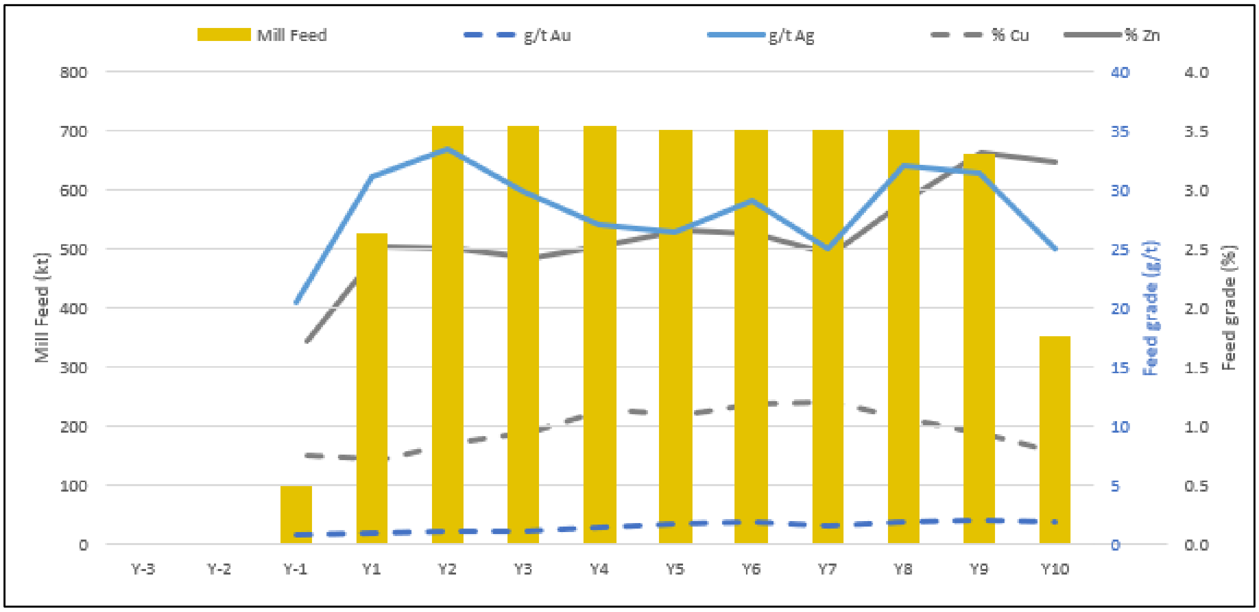


Figure 16.19: Mining Tonnage and Mining Grade

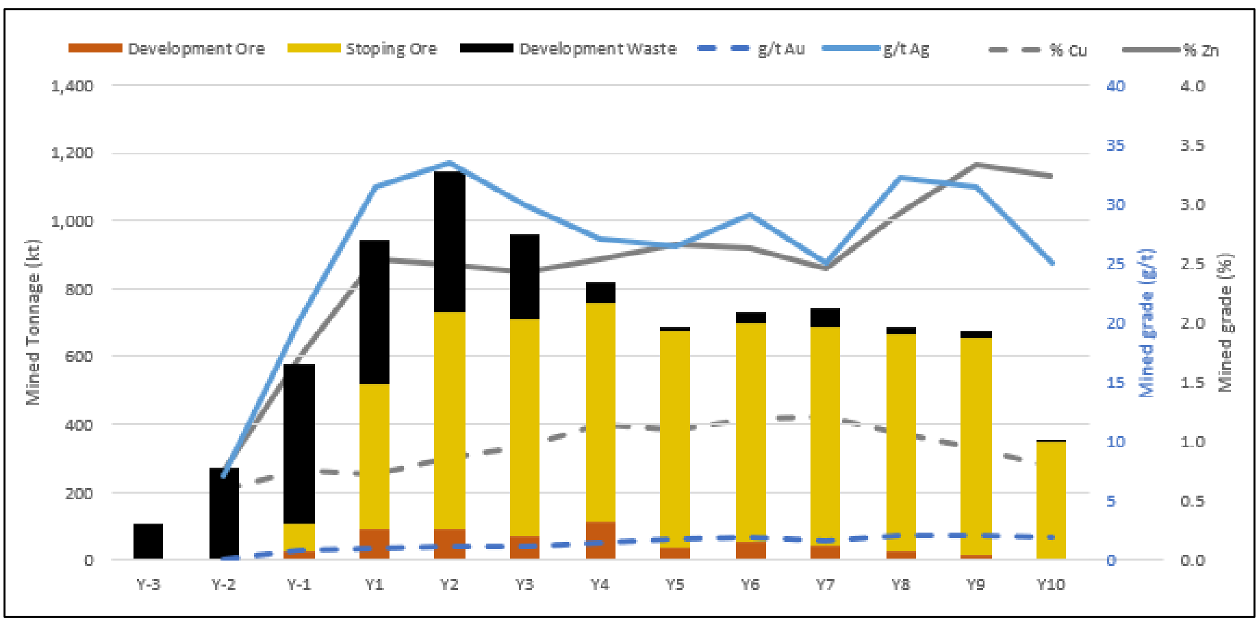


Table 16.28: Global Underground Mining and Mill Feed

	Unit	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Mine Physicals															
Mined Mineralized Material	kt	6,548	-	0.3	108	516	729	707	757	679	696	685	667	655	350
	% Cu	1.01	-	0.61	0.75	0.72	0.87	0.96	1.14	1.09	1.19	1.21	1.07	0.94	0.79
	% Zn	2.67	-	0.77	1.71	2.53	2.49	2.43	2.53	2.66	2.63	2.46	2.92	3.32	3.24
	g/t Au	1.60	-	0.12	0.92	1.03	1.15	1.19	1.42	1.76	1.92	1.67	2.05	2.14	1.87
	g/t Ag	29.07	-	7.10	20.25	31.35	33.45	29.80	27.01	26.36	29.08	24.97	32.26	31.41	25.06
Mined Waste	kt	2,168	111	274	473	430	418	256	64	8	33	55	20	19	6
Mill Feed															
Milled Mineralized Material	kt	6,548	-	-	99	525	705	705	705	700	700	700	700	658	350
	% Cu	1.01	-	-	0.76	0.72	0.87	0.96	1.14	1.09	1.18	1.21	1.07	0.94	0.79
	% Zn	2.67	-	-	1.72	2.51	2.50	2.42	2.53	2.65	2.63	2.46	2.90	3.32	3.24
	g/t Au	1.60	-	-	0.93	1.03	1.14	1.19	1.42	1.75	1.91	1.66	2.02	2.14	1.87
	g/t Ag	29.07	-	-	20.45	31.12	33.50	29.81	27.01	26.43	29.07	25.05	32.08	31.39	25.06

17. RECOVERY METHODS

17.1 Introduction

The proposed process plant design for the Kay Mine Project is based on a standard metallurgical flowsheet to treat copper-zinc-lead volcanogenic massive sulphide material to produce copper-lead concentrate, zinc concentrate and gold from pyrite concentrate. The flowsheet is based on metallurgical test work described in Section 13, industry standards and conventional unit operations.

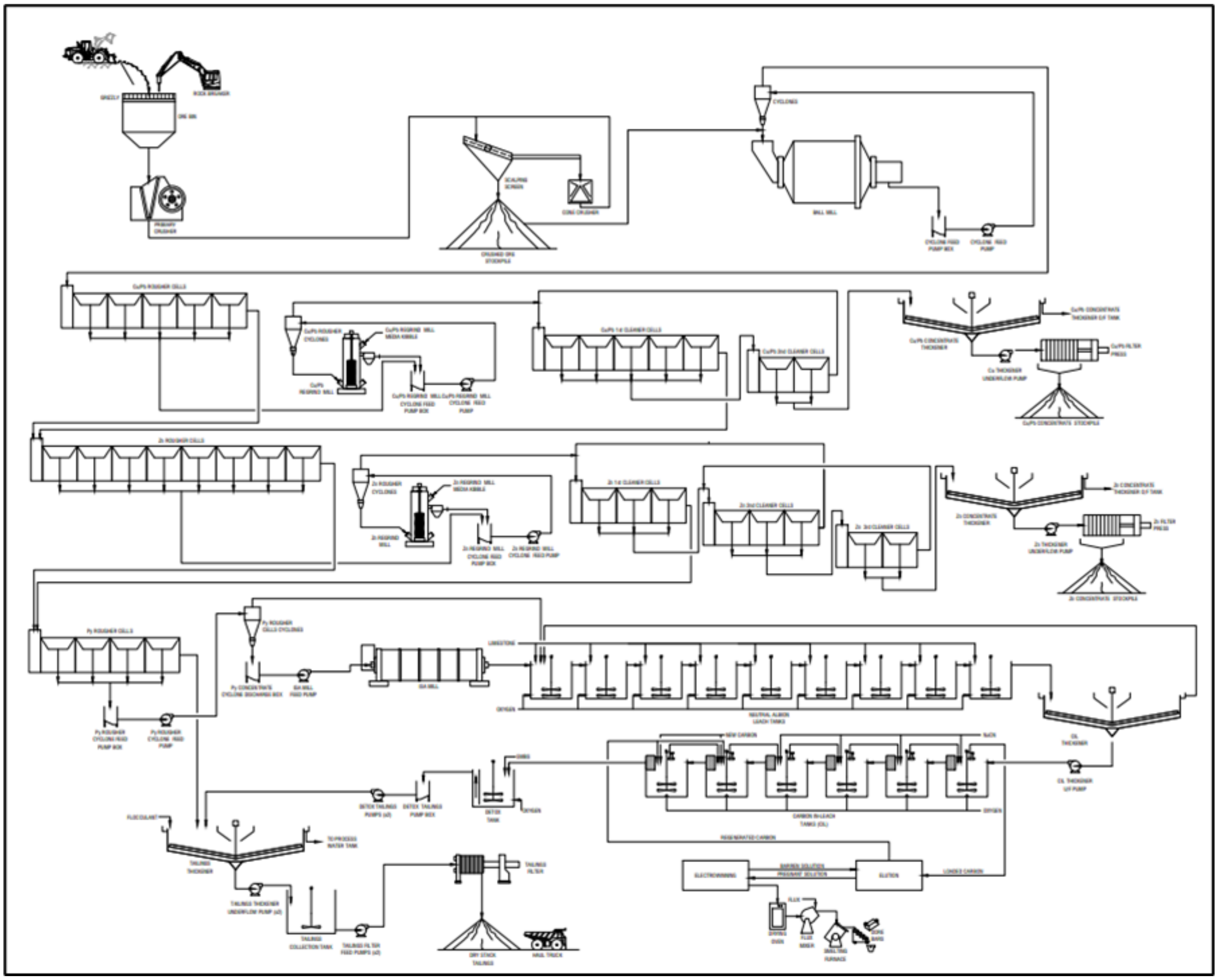
The process plant is designed to nominally treat 0.7 Mtpy of fresh rock and will consist of comminution, Cu-lead flotation, zinc flotation and pyrite flotation circuits followed by dewatering circuit for the copper-lead and zinc concentrate products. Pyrite concentrate will be treated in an Albion Process followed by cyanidation, adsorption, desorption and recovery (ADR) circuit for recovering gold. Pyrite flotation tailings and cyanidation tailings post cyanide destruction will be filtered, and the filtered tailings / dry stack tailings will be trucked to the tailings' storage facility. Figure 17.1 presents the overall flowsheet for the Kay Mine Project.

The key project design criteria for the process plant are listed below:

- Nominal throughput of 0.7 Mtpy of fresh rock.
- Crushing plant availability of 70%.
- Grinding, flotation, concentrate dewatering circuit availability of 91% using standby equipment in critical areas, and reliable power supply.
- Comminution circuit to produce a primary grind size of (P_{80}) 80% passing 60 µm.
- Flotation circuit consisting of roughers, regrind circuit followed by cleaner flotation for Cu-Pb and zinc recovery. For pyrite recovery, rougher flotation is considered.
- Concentrates dewatering including thickening and filtration circuits for Cu-Pb and Zn products.
- Albion treatment for the pyrite concentrate containing refractory gold followed by cyanidation for extracting the gold.
- Detoxification of the cyanidation tailings.
- Tailings Filtration for producing dry stack tailings.
- Sufficient process plant control to minimize the need for continuous operator interface and to allow for manual override and control when required.
- Equipment selection based on suitability for the required duty, reliability, and ease of maintenance.

- Plant layout that provides ease of access to all equipment for operating and maintainability, while facilitating concurrent construction activities in multiple areas of the plant.

Figure 17.1: Overall Flowsheet



17.2 Process Design Criteria

The proposed process plant will consist of the following unit operations:

- Crushing of the ROM by two-stage crushing.
- Coarse material stockpile and reclaim.
- Grinding consisting of a ball mill with hydrocyclones producing a ground product P_{80} of 60 µm.

Flotation circuit consisting of roughers, regrind and cleaners for producing separate copper and zinc concentrates, as well as single stage rougher flotation to produce pyrite concentrate.

- Concentrate thickening and filtration for the Cu-Pb and Zn concentrates.
- Albion processing including concentrate regrind and oxidative leaching of the refractory pyrite flotation concentrate.
- Cyanidation, elution and electrowinning of the Albion-treated product for producing Doré.
- Cyanide detoxification of the tailings prior to disposal to the dry stack tailings facility.
- Dewatered pyrite flotation tailings to the dry stack tailings facility.
- Water systems (potable water, treated water, gland seal water and process water).

Key process design criteria are summarized in Table 17.1.

Table 17.1: Key Process Design Criteria

Area	Criteria	Unit	Nominal Value
General	Nominal Annual Throughput	t/y	700,000
	Nominal Daily Throughput	t/d	1,920
	Crusher Plant Availability	%	70
	Process Plant Availability	%	91
	Design Copper Head Grade	%	2.05
	Design Zinc Head Grade	%	5.25
	Design Lead Grade	%	0.48
	Design Gold Grade	g/t	2.51
	Design Silver Grade	g/t	52.92
	Copper Recovery	%	88.30
	Design Cu Concentrate Grade	%	32.52
	Zinc Recovery	%	75.90
	Design Zinc Grade	%	64.57
	Pyrite concentrate (Gold Recovery)	%	62.00
	Pyrite concentrate (Silver Recovery)	%	20.30
	Gold Overall Recovery (including Albion)	%	80.0

Area	Criteria	Unit	Nominal Value
Crushing & Storage	Crusher Work Index	kWh/t	10.7
	Run of Mine (ROM), Maximum Size	mm	500
	Crusher Circuit Product Size (P_{80})	mm	10.7
	Crushed mineralized material Stockpile Capacity (live)	h	12
Grinding	SMC A x b	-	53.3
	Bond Ball Mill Work Index (85th percentile)	kWh/t	12.2
	Primary Mill Dimensions	m	Ø 3,9 x 5,2
	Grinding Circuit Product Size (P_{80})	µm	60
Copper Flotation Circuit	Rougher Flotation	-	5 x OK16
	Regrind mill	-	HIG 500
	Regrind Circuit Product Size (P_{80})	µm	15
	1st Cleaner Flotation	-	5 x OK3
	2nd Cleaner Flotation		2 x OK3
Zinc Flotation Circuit	Rougher Flotation	-	8 x OK3
	Regrind mill	-	HIG 500
	Regrind Circuit Product Size (P_{80})	µm	15
	1st Cleaner Flotation	-	3 x OK3
	2nd Cleaner Flotation	-	3 x OK3
	3rd Cleaner Flotation	-	2 x OK3
Pyrite Flotation Circuit	Rougher Flotation	-	4 x OK16
Copper-lead Concentrate Dewatering	Cu-Pb Concentrate Thickener	-	9.7 m dia. conventional
	Filter Feed Tank (Conc. holding capacity)	h	1
	Cu-Pb Concentrate Filter Press (6 t/h)	-	PF 16/19 M12 1 45
Zinc Concentrate Dewatering	Zn Concentrate Thickener	-	9.7 m dia. conventional
	Filter feed Tank (Conc. holding capacity)	h	1
	Zn Concentrate Filter Press (6 t/h)	-	PF 16/19 M12 1 45
Pyrite Concentrate Treatment by Albion Process	Regrind Mill (Isa Mill)	-	M7500
	Regrind Circuit Product Size (P_{80})	µm	10 -12
	Concentrate Storage Tank	m^3	799
	OxiLeach Reactors	-	9 x OLR1700

Area	Criteria	Unit	Nominal Value
	Slurry Cooling Tower	-	1
	Oxygen plant	t/d	329
Carbon in Leach	CIL Thickener	-	22 m dia.
	CIL Tanks	-	6 x 870 m^3
Adsorption, Desorption, Recovery	2-tonne ADR plant	-	-
Cyanide Detox	Cyanide Detox Tank	-	135 m^3
Tailings	Tailings Thickener	-	22 m dia.
	Tailings Filter Press	-	GHT2500F18

17.3 Process Plant Description

17.3.1 Crushing Circuit

Material from the open pit will be transported to the plant by rear dump trucks. The trucks will tip directly to the primary crusher dump pocket. Material from the primary crusher dump pocket will feed a Jaw crusher. A rock breaker will be installed to assist in breaking down oversize material retained on the stationary grizzly.

Crushed material from the jaw crusher will feed the double deck inclined scalping screen. The scalping screen operates in closed circuit with the cone crusher. The screen oversize will feed the cone crusher while the undersize reports to the crushed rock stockpile. An apron feeder and sacrificial conveyor will withdraw the crushed material. A belt magnet at the sacrificial conveyor discharge will recover any trash metal. The sacrificial conveyor will discharge crushed material on to the primary ball mill feed conveyor. The primary ball mill feed conveyor will be fitted with a weightometer to monitor the mill feed throughput and to control the apron feeder variable speed drive (VSD).

The crushing circuit will be serviced by a single dust collection system consisting of multiple extraction hoods, ducting, and a baghouse. Dust collected from this system will be discharged onto the stockpile feed conveyor.

17.3.2 Material Stockpile

The crushed rock stockpile will have a live capacity of approximately 1,050 tonnes (equivalent to 12 hours of mill feed). Two (2) reclaim apron feeders located underneath the stockpile will be installed with variable speed drives (VSDs) to control the reclaim rate feeding the grinding circuit. The apron feeders installed within the surface tunnel are equipped with a water spray system at the chute and an integrated dust collection system designed to mitigate airborne particulate matter generated during the material handling. The surface tunnel is designed with an engineering ventilation system to maintain adequate airflow, ensuring occupational health and safety compliance while minimizing dust accumulation. This ventilation system is designed to control air quality, humidity, and temperature, which are critical factors for the safe and efficient operation of mechanical equipment within the surface tunnel.

17.3.3 Grinding

Reclaimed material from the stockpile will feed a 3.9 m diameter by 5.2 m effective grinding length (EGL) ball mill via the ball mill feed conveyor. The ball mill will be installed with 1,350 kW fixed speed motor. A belt-scale on the ball mill feed conveyor will monitor the feed rate. Process water will be added to the ball mill to maintain a 75% slurry discharge density. A trommel on the ball mill discharge will remove grinding media scats and a small number of pebbles. Slurry from the grind cyclone feed pump box will be pumped to a cyclone cluster of nine (9) (seven (7) operating / two (2) standby) 250 mm hydrocyclones for size classification. The hydrocyclones have been designed for a 350% circulating load.

17.3.4 Flotation

The cyclone overflow from the grinding circuit will flow via gravity to the copper-lead flotation circuit. The copper-lead flotation circuit consists of five (5)16 m^3 Metso OK16 or similar cells for rougher flotation, a HIG 500 regrind mill for rougher concentrate grinding followed by five (5) 3 m^3 Metso OK3 or similar cells for 1st cleaner flotation and two (2) 3 m^3 Metso OK3 or similar cells for 2nd cleaner flotation for producing a final copper-lead concentrate. The 2nd cleaner tailings will be pumped back to the 1st cleaner flotation.

The copper-lead roughers and 1st cleaner tailings will be pumped to the zinc flotation circuit. The zinc flotation circuit consists of eight (8) 3 m^3 Metso OK3 or similar cells for rougher flotation, a HIG 500 regrind mill for rougher concentrate grinding followed by three (3) 3 m^3 Metso OK3 or similar cells for 1st cleaner flotation, 2nd cleaner flotation (three (3) 3 m^3 Metso OK3 or similar cells) and 3rd cleaners (two (2) 3m^3 Metso OK3 or similar cells) for producing a final zinc concentrate. The tailings from the 3rd cleaners and 2nd cleaners will be pumped countercurrently to the 2nd cleaners and to the 1st cleaners. The zinc roughers tailings and the zinc 1st cleaner tailings will be pumped to the pyrite flotation circuit.

The pyrite rougher flotation circuit consists of four (4) 16 m^3 Metso OK16 or similar mechanical cells for rougher flotation, for producing a pyrite concentrate. The rougher flotation tailings will be pumped to the tailings thickener prior to sending to the tailings' filtration.

17.3.5 Albion Process

The pyrite concentrate containing refractory gold will be processed in an Albion Process plant. The Albion plant consists of an IsaMill (M7500) with 1,719 kW installed power motor for producing a ground product with a P_{80} 10-12 µm. The finely ground concentrate will be stored in 799 m^3 concentrate storage tank with an agitator having a 64-kW motor. The concentrate pumped from storage tank will be leached in 9 x OLR1700 OxiLeach reactors with 316 kW motor on each of the nine reactors. The leached slurry will be pumped to a cyclone cluster. The cyclone underflow will be stored in a slurry cooling tower to reduce slurry temperature. The cooled slurry will be pumped to a 22 m Ø CIL thickener by a CIL transfer pump.

17.3.6 Cyanidation of Albion Product

The CIL thickener will increase the slurry density for the downstream cyanidation process. Flocculant will be added to the thickener feed to promote the settling of solids. The thickener underflow at 45% solids will be pumped to the Carbon in Leach (CIL) circuit consisting of six (6) x 870 m^3 OxiLeach tanks for leaching of the gold from the pyrite circuit. Sodium cyanide and activated carbon will be added to the leach tanks for leaching gold and adsorption on to the activated carbon. The leach tanks will provide a total residence time of 48 hours, and the tanks will be sparged with oxygen.

Tailings from the CIL circuit will be pumped to a carbon safety screen to capture any residual carbon particles. Captured carbon will be collected in bags and processed to recover any residual gold and silver. The carbon safety screen undersize will flow by gravity to the cyanide destruction circuit.

17.3.7 Cyanide Detoxification

The cyanide destruction circuit will consist of 135 m^3 mechanically agitated tank, providing a retention time of 2 hours. The conventional SO_2 / O_2 air process will be used for cyanide destruction. Treated slurry will flow by gravity to the cyanide destruction tailings pump box for pumping to the tailings' thickener.

The cyanide destruction circuit will treat CIL tailings, process spills from various contained areas and process bleed streams: cold cyanide barren solution effluent, acid wash effluent and area sump pump discharge.

Oxygen will be sparged into the cyanide destruction tank. Lime slurry will be added to maintain the pH of 8.5 and copper sulphate will be added as a catalyst. Sodium metabisulphite (SMBS) will be dosed into the system as a source of SO_2. The process will reduce WAD cyanide in solution to 10.5 mg/L. The total cyanide and WAD cyanide level in solutions will eventually drop to lower levels via natural degradation and dilution from rainwater in the tailings' storage facility.

17.3.8 Acid Wash and Elution

Loaded carbon from the CIL circuit will be pumped to a two (2)-tonne Adsorption, Desorption, Recovery (ADR) circuit and screened to the acid wash column where it will be treated with hydrochloric acid to remove inorganic foulants such as calcium, magnesium, sodium salts, and silica. The carbon will first be rinsed with fresh water. Acid will then be pumped from the acid wash circulation tank to the acid wash column and then pumped upward through the acid wash vessel and overflow back to the acid wash circulation tank. The carbon will then be rinsed with fresh water to remove the acid and any mineral impurities. Fresh acid will be pumped from drums into the acid wash tank when required. A recessed impeller pump will transfer acid washed carbon from the acid wash vessel into one of the elution vessels using recycled carbon transfer water. Carbon slurry will discharge directly into the top of one of the elution vessels.

The carbon stripping (elution) cycle will utilize barren solution to strip gold rich carbon to create a pregnant solution. The strip circuit will be equipped with two (2) strip columns that can hold 10 t of carbon each to allow more than one (1) strip per day, depending on the feed to the plant. During the strip cycle, solution containing approximately 2.0% hydroxide and 0.2% sodium cyanide, at a temperature of 150°C and 500 kPa will be circulated through the strip vessel. Solution exiting the top of the elution vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold solution. The heated barren solution will then be heated again through the primary heat exchanger using heated water to bring the solution to its final temperature.

The hot barren solution will then be pumped into the elution column through the carbon bed and recirculated multiple times creating a pregnant solution. A barren solution tank will store barren solution, and a pregnant solution tank will store pregnant solution. The elution column can also be used as a cold strip circuit to remove copper from carbon if copper levels are too high.

17.3.9 Carbon Regeneration

Once stripped of gold, transport water will transfer the carbon from the elution vessel to the carbon dewatering screen. The screen acts as both a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity to the carbon regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank. A diesel fired kiln will be utilized to treat 10 t of carbon per day, equivalent to 100% regeneration of carbon. The regeneration kiln discharge carbon will be transferred to the carbon quench tank by gravity, cooled by process water and stored in the regenerated sized carbon tank prior to being pumped back into the CIL circuit.

To compensate for carbon losses by attrition, fresh carbon is added to the carbon pre-attrition tank along with fresh water to mix and activate the carbon. The fresh carbon will then drain into the regenerated carbon tank.

17.3.10 Electrowinning and Gold Room

The pregnant solution generated from the elution column will be pumped to electrowinning cells from the pregnant solution tank. These cells will operate on a single-pass basis to produce a gold sludge. The barren solution will be collected in the barren solution pump box where it will be pumped to the barren solution tank. The primary flow from the barren solution tank pump box returns the solution to the elution circuit where it will be reused as barren stripping solution for the elution column.

The electrowinning cathodes will be manually transferred from the electrowinning cells to the cathode washing tank where a high-pressure washer will be used to dislodge gold sludge from the cathode surface. The sludge will be filtered by a filter press. The resulting filter cake will be dried in a drying oven, and the resulting filtrate will be pumped back to the barren solution pump box tank within the refinery.

The dried filter cake will then be transferred manually into the electric smelting furnace with flux materials where it will be batch smelted into gold Doré bars and stored in a secure vault.

17.3.11 Concentrate Dewatering

The final copper-lead concentrate from the 2nd cleaner flotation will be pumped to a 9.7 m diameter conventional thickener followed by pressure filtration to decrease the concentrate moisture content. Flocculant will be added to the thickener feed to accelerate the settling of solids. The thickener overflow will report to a thickener overflow tank which is then pumped to the process water tank. The filtered concentrate will be discharged on to a copper-lead concentrate stockpile.

The final zinc concentrate from the 3rd cleaner flotation will be pumped to a 9.7 m diameter thickener followed by pressure filtration to decrease the concentrate moisture content. Flocculant will be added to the thickener feed to accelerate the settling of solids. The thickener overflow will report to a thickener overflow tank which is then pumped to the process water tank. The filtered concentrate will be discharged on to a zinc concentrate stockpile.

17.3.12 Tailings Storage

Tailings from the cyanide destruction circuit and tailings from pyrite rougher flotation will feed a 22 m diameter tailings thickener. The thickener overflow will report to the process water tank whereas the underflow will feed a tailings collection tank prior to feeding a 3,050 x 2,640 mm, 81 plates filter press to produce a dry stack tailing. Dried/filtered tailings will be trucked to the dedicated Tailings Storage Facility (TSF).

17.4 Reagents

Reagents consumed within the process plant will be prepared on site and distributed via various reagent handling and makeup systems. These reagents include sodium cyanide, quicklime, zinc sulphate, copper sulphate, Aero 5100, sodium isopropyl xanthate, sodium humic acid, and flocculant.

For the management of unexpected reagent spills, the reagent preparation and storage facilities will be located within containment areas designed to accommodate more than the content of the largest tank. Where required, each reagent system will be located within its own containment area to facilitate its return to its respective storage vessel and to avoid the mixing of incompatible reagents. Storage tanks will be equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eye wash stations and showers, and Material Safety Data Sheet (MSDS) stations will be located throughout the facilities. Sumps and sump pumps will be provided for spillage control.

The reagents will be mixed, stored, and then delivered to the primary grinding mill, copper-lead flotation, zinc flotation, pyrite flotation, thickener circuits. Dosages will be controlled by flow meters and control valves. The capacity of the storage tanks will be sized to typically handle one (1) day of production. The reagents will be delivered in dry form.

17.4.1 Sodium Cyanide

Sodium cyanide will be added as a zinc depressant in the grinding mill and in cyanidation circuit as a gold lixiviant. The cyanide will be shipped in briquette form by road to site in 18-t ISO containers and stored in the cyanide mixing facility; separate from the reagent storage and mixing facility. The sodium cyanide will be mixed with fresh water to form a cyanide solution for use in the carbon in leach (CIL) circuit.

17.4.2 Quicklime

Quicklime will be used as a pH modifier in flotation, in CIL circuit and will be supplied in dry form in bulk bags. Quicklime will be slaked, sent to a grit separator, and then added into a mix tank to prepare a milk of lime slurry, and transferred to a storage tank before addition into the process.

17.4.3 Zinc Sulphate

Zinc sulphate ($ZnSO_4$) will be used as a zinc depressant in the grinding circuit and will also be used as a catalyst for cyanide destruction. The copper sulphate will be supplied as a dry flake in one (1)-tonne bulk bags and stored in the reagent storage area adjacent to the reagents mixing facility. The zinc sulphate will be mixed with fresh water to form a zinc sulphate solution ready for use in the processing facility.

17.4.4 Copper Sulphate

Copper sulphate ($CuSO_4$) will be used as an activator in zinc flotation circuit and cyanide destruction. The copper sulphate will be supplied as a dry flake in one (1)-tonne bulk bags and stored in the reagent storage area adjacent to the reagents mixing facility. The copper sulphate will be mixed with fresh water to form a copper sulphate solution ready for use in the processing facility.

17.4.5 Sodium Humate

Sodium Humate (NaHA), will be used for Arsenic depression and will be supplied in one (1.0)-tonne bulk bags as a dry reagent. NaHA will be stored in the reagent storage area where it will be transferred to the mixing facility to produce a NaHA solution prior to use in the zinc regrind circuit.

17.4.6 Aero 5100 Dithiophosphate

Aero 5100 dithiophosphate will be used as a sulphide collector in the flotation circuit. The reagent will be supplied in liquid form in standard bulk containers. Aero 5100 will be diluted with fresh water prior to being dosed into the flotation circuit.

17.4.7 Methyl Isobutyl Carbinol

Methyl Isobutyl Carbinol (MIBC) will be used as a frother in flotation. It will be supplied in totes and stored in the reagent storage area adjacent to the reagent mixing facility.

17.4.8 Sodium Hydroxide

Sodium Hydroxide (NaOH) also known as caustic soda, will be used in Albion oxidative leach process, in elution and cyanide mixing. NaOH will be supplied in 1.2 tonne bulk bags and will be mixed with fresh water prior to being used in gold elution circuit, and cyanide mixing tank.

17.4.9 Hydrochloric Acid

Hydrochloric acid (HCl) will be supplied in liquid form in totes and will be stored in the reagent storage area adjacent to the reagent mixing facility. HCl will be used to remove inorganic foulants (Carbonates) from carbon in the acid wash process within the elution plant.

17.4.10 Activated Carbon

Activated carbon will be delivered in bulk bags. Carbon will be added to the carbon attrition tank as required for carbon make-up to the CIP inventory. The addition point will allow attritioning of any friable carbon particles with subsequent fines removal on the sizing screen prior to entering the CIP tanks.

17.4.11 Sodium Meta Bisulphate

Sodium metabisulphite ($Na_2S_2O_5$), also known as SMBS, will be the source of SO_2 for the cyanide destruction process and will be supplied in 1.2-tonne bulk bags as a dry reagent. SMBS will be stored in the reagent storage area where it will be transferred to the mixing facility to produce a SMBS solution prior to use in the cyanide destruction process.

17.4.12 Flocculant

Flocculant is a liquid polymer that will be used in the thickeners to settle solids. It will be supplied in 700 kg bulk bags as a dry reagent. Flocculant will be shipped by road to site, offloaded by forklift, and stored in the reagent storage area adjacent to the reagents mixing facility. Flocculant will be diluted using fresh water and further diluted using an inline mixer with process water prior to being added into the processing facility.

17.5 Plant Services

17.5.1 Plant & Instrumentation Air

Three (3) air compressors will provide plant and instrument air for the process plant. Plant air receivers will act as a buffer storing air to account for variations in demand prior to being distributed throughout the process plant including the oxygen generation plant. Instrument air will be dried before being stored in the instrument air receivers and distributed throughout the plant.

17.5.2 Oxygen Generation

An oxygen generation plant will be used to provide industrial grade oxygen for the Albion, CIL circuit and cyanide destruction circuit. The plant air compressors will supply air to the oxygen generation circuit. The oxygen generation plant will include an oxygen plant air drier, a Pressure Swing Adsorption (PSA) oxygen generator, and an oxygen plant receiver.

17.5.3 Treated Water

Raw water will be pumped to the plant treated water tank by vertical turbine pumps from the catchment pond. The plant treated water tank will serve as storage for water supply.

Treated water will be used to supply the following services:

- Crushing circuit dust suppression water.
- Mining underground equipment.
- Reagent preparation water.
- Slurry pumps gland seal water.
- Cooling water systems, i.e., mill motors cooling.
- Make-up water for the process water system.

Raw water from the treated water tank will be pumped through multimedia filters to remove particulates.

17.5.4 Raw Water Buffer Tank

Feed to the raw water tank will be supplied by borewells. The raw water buffer tank will feed the truck shop, fuel bay area, sewage treatment plant, fire water tank and emergency domestic water tank.

17.5.5 Fire Water

Fire water will receive water from the raw water buffer tank and will have a minimum capacity to feed the plant and permanent campfire suppression systems, fire hydrants and hose reels.

17.5.6 Process Water

Process water will comprise of copper concentrate thickener overflow, zinc concentrate thickener overflow, CIL leach thickener overflow, tailings thickener overflow and tailings reclaim water (filtrate) from Cu-Pb filtration, Zn filtration and tailings filtration. Process water will be stored in the process water storage tank and distributed by the process water pumps, in a duty – standby configuration.

17.5.7 Potable Water

Feed to the potable water system is supplied by treated water tank. The water will be treated in a vendor-supplied potable water plant to produce potable water for the process plant facilities distribution. The potable water will be used in the process plant for safety showers and washrooms.

17.5.8 Gland Seal Water

Water for the gland seal water system will be supplied by treated water from the treated water tank. The gland seal water tank will store and distribute gland water to the plant with gland seal water pumps in a duty-standby configuration.

To prevent particulates from causing damaged gland seals throughout the plant, the water feeding the gland water tank will pass through 25-micron particulate filters.

17.6 Metallurgical Accounting

Several samplers will be provided throughout the plant to generate composite shift samples from key process streams. Two (2) types of sampling will be performed, metallurgical and process control sampling.

Metallurgical samplers will be used to generate shift composite samples that will be assayed for plant metallurgical accounting. The following process streams will be equipped with metallurgical samplers:

- Secondary grind cyclone overflow.
- Final copper-lead concentrate.

- Final zinc concentrate.
- Rougher pyrite concentrate.
- Rougher pyrite tailings.
- Cyanidation tailings.

The metallurgical samplers will sample feed, concentrate and tailings products which will allow an accurate metal balance of the plant to be completed.

Process control sampler will generate samples used to monitor unit processes in the plant. The process control samplers will be used to generate shift composite samples on process streams that will provide plant operation performance data.

The following process streams will be equipped with process control samplers:

- Cu/Pb rougher concentrate.
- Cu/Pb rougher tailings.
- Copper-Lead rougher regrind cyclone overflow.
- Copper-Lead 2nd cleaner tailings.
- Zn rougher concentrate.
- Zn rougher tailings.
- Zn rougher regrind cyclone overflow.
- Zn 2nd cleaner tailings.
- Zn 3rd cleaner tailings.
- Pyrite rougher tailings.
- Leach feed.
- Leach tailings.
- Final tailings.
- Pregnant solution to electrowinning.
- Barren solution after electrowinning.

All samplers will produce 5-10 L of slurry that can be transported to the assay laboratory for further analysis.

A weightometer on the stockpile feed conveyor will measure primary crushed mineralized material tonnage, and a weightometer on the primary ball mill feed conveyor will determine mill feed tonnage.

A manual belt cut sampling point on the primary ball mill feed conveyor will allow for the collection of a mill feed head grade sample for cross-checking with the calculated head grade. This sample will also be utilized to establish the moisture content of the mill feed.

Regular surveys of the gold and silver in circuit will allow a reconciliation of precious metals in the feed and the concentrates.

Water supplied and used in the various areas will be continuously monitored.

Reconciliation of the reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates to unit operations will be measured and accumulated using flowmeters.

17.7 Plant Control System

The following provides a broad overview of the control strategy that will be employed for the process plant.

The general control philosophy for the process plant will be one with a moderate level of automation and remote-control facilities to allow critical process functions to be carried out with minimal operator intervention. Instrumentation will be provided within the plant to measure and control key process parameters.

The main control room, located in the process plant office, will house PC-based operator interface terminals (OIT) and a single server. These workstations will act as the control system supervisory control and data acquisition (SCADA) terminals. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS).

17.8 Plant Consumption

17.8.1 Energy

The power demand for the process plant, along with the rest of the project, will be discussed in Section 18.6.

17.8.2 Reagents & Consumables

Reagent storage, mixing and pumping facilities will be provided for all reagents for the process plant. Reagents and consumables usage are summarized in Table 17.2 and Table 17.3, assuming 100% fresh rock.

Table 17.2: Reagents Consumption

Description – Flotation Reagents	Delivered Form	Average Usage (kg/t)
Sodium Cyanide	Briquette ISO tank	0.10
Lime (@90% CaO)	1.5 t bag	3.45
Sulphuric Acid (H_2SO_4) (10%)	1,100 L IBC	0.05
Zinc Sulphate	Powder	0.55
Aero 5100 (Modified IPTEC)	Liquid	0.18
Copper Sulphate	1.2 t bags (dry)	0.50
Methyl Isobutyl Carbinol (MIBC)	Liquid	0.06
Sodium Humate (NaHA)	Powder	0.83
Flocculant	700 kg bags (dry)	0.005
Albion Process Reagents		
Limestone – Calcium Carbonate	1.5 t bag	373.1
Caustic Soda- NaOH	1.2 t bags	6.6
Flocculant	700 kg bags (dry)	0.03
Leaching – Acid Wash, Elution Reagents		
Activated Carbon	500 kg bags (dry)	0.03
Sodium Cyanide NaCN	Briquette ISO tank	0.31
Lime (@90% CaO)	1.5 t bag	1.14
Sodium Meta Bisulphate (SMBS)	1.2 t bags (dry)	1.12
Copper Sulphate ($CuSO_4$)	1.2 t bags (dry)	0.09
Sodium Hydroxide	1.2 t bags (dry)	0.09
Hydrochloric Acid (32% strength)	1,100 L IBC	0.30
Flocculant	700 kg bags (dry)	0.02

Source: GMS, 2025

Table 17.3: Consumables Consumption

Description	Delivered Form	Usage
Jaw Crusher – Fixed Jaw Consumption	lot	8.1 sets / year
Jaw Crusher – Moving Jaw Consumption	lot	5.9 sets / year
Cone Crusher – Liner Consumption	lot	10.9 set / year
Primary Ball Mill Liners	lot	1.0 sets / year
Copper Regrind Mill Liners	lot	0.4 sets / year
Zinc Regrind Mill Liners	lot	0.4 sets / year
Primary Ball Mill Grinding Media (100 mm)	bulk	0.52 kg/t
Secondary Ball Mill Grinding Media (38 mm)	bulk	0.37 kg/t
Copper Regrind Mill Grinding Media (4 mm)	bulk	0.35 kg/t
Zinc Regrind Mill Grinding Media (4 mm)	bulk	0.54 kg/t
Albion Regrind mill (IsaMill) Liner consumption	bulk	0.7 sets / year
Albion Regrind mill (IsaMill) Ball consumption	bulk	0.20 kg/t
Trash Screen panels	lot	1.0 sets / year
Loaded carbon Screen panels	lot	1.1 sets / year
Carbon Quench Screen Panels	lot	1.1 sets / y
Carbon Safety Screen Panels	lot	1.1 sets / y
Interstage Screen Panels	lot	0.6 sets / y

Source: GMS, 2025

17.9 Process Plant Personnel

The personnel for the process plant will consist of management, operations, maintenance, and laboratory. Operating staff will work 8-hour x 3 shifts on a rotation cycle and management will work 8-hour days.

Annual process plant personnel requirements are provided in Table 17.4.

Table 17.4: Process Plant Personnel

Department	Position	# Employees
Process Management	Process Manager	1
Technical Support	Senior Metallurgist	1
	Metallurgical Technician	1

Department	Position	# Employees
Laboratory	Chief Assayer	1
	Senior Assayer	2
	Technicians	4
Process Operations	Operations Superintendent	1
	Supervisor	4
	Control Room Operator	4
	Crusher Operator	4
	Grinding Operator	4
	Regrind Operator	4
	Flotation Operator	4
	Concentrate Dewatering Operator	4
	Reagents Operator	1
	Albion & Leaching	4
	Leaching, Electrowinning & Goldroom	12
Maintenance	Maintenance Superintendent	1
	Reliability Engineer	1
	Maintenance Planner	2
	Materials Planning Clerk	1
	Mechanical Supervisor	2
	Plant Fitter (millwright)	2
	Electrical Supervisor	1
	Electrician	2
	Instrumentation Technician	1
	System Integrator	1
Total		**72**

Source: GMS, 2025

18. PROJECT INFRASTRUCTURE

The Project infrastructure is designed to support the operation of an underground mine feeding a process plant with a nominal 0.7 Mtpa, operating on a 24-hour per day, 7-day per week basis. It is designed in consideration of local conditions and topography.

18.1 Site Layout

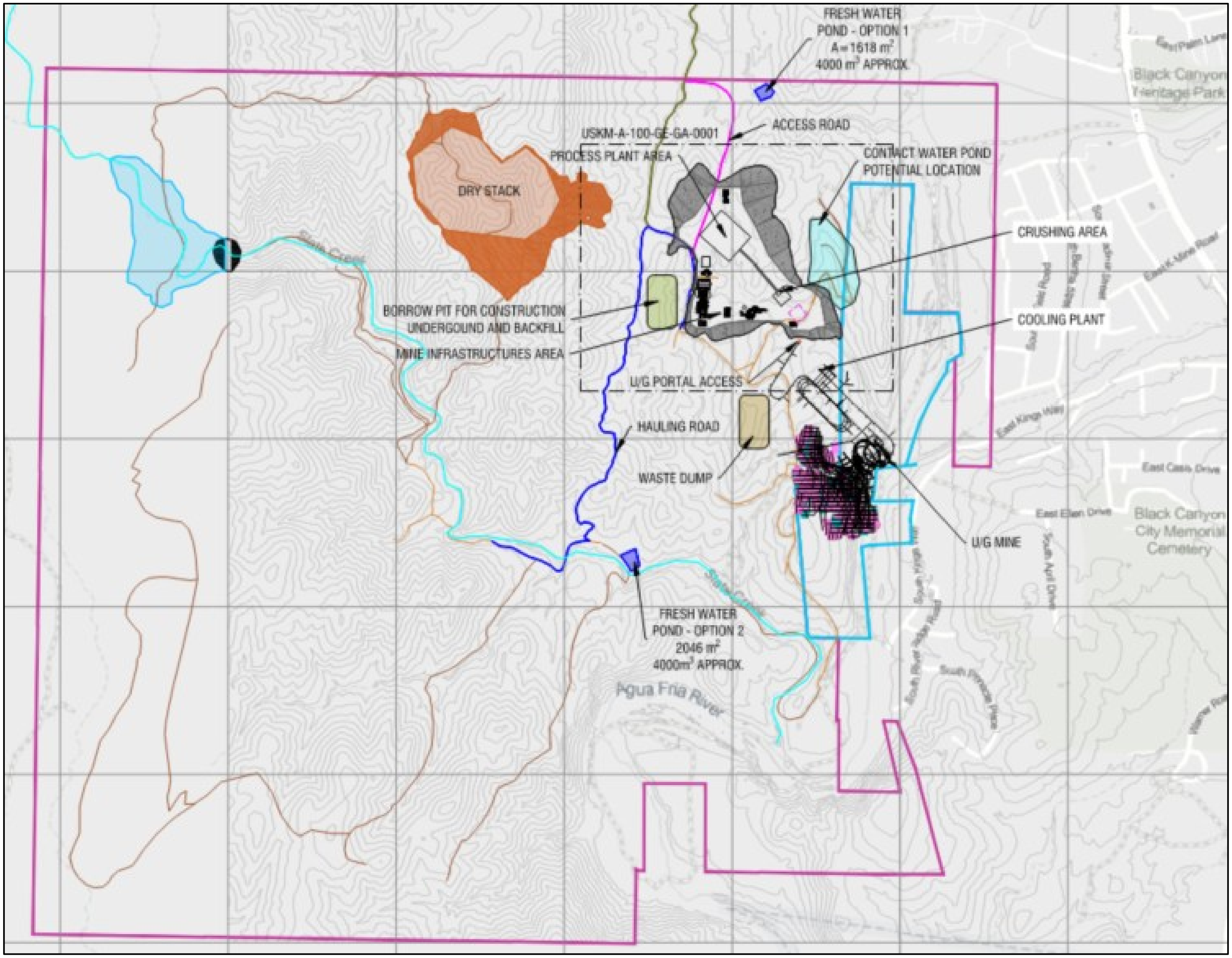
Figure 18.1 illustrates the Project site plan, presenting the on-site infrastructure. This plan is designed to minimize environmental impacts, provide secure site access, minimize construction costs, and enhance operational efficiency. It shows the proposed mine, process plant, the various surface infrastructures and the various access roads.

Site facilities will include mine infrastructure, process plant infrastructure, and supporting facilities, including:

- Water Management Infrastructure:
  - Process Water and Fire Water Systems.
  - Potable Water System.
  - Contact Water Collection Pond.
  - Sewage Treatment System.
  - Effluent Treatment Plant.
- Mine Infrastructure:
  - Mine Maintenance Facility and Warehouse.
  - Mine Administration Building.
  - Mine Dry.
  - Explosives Storage Facilities.
  - Dry Stack Tailings Storage Facility (DSTSF).
- Process Infrastructure:
  - Mineral Processing Facility.
  - Assay and Metallurgical Laboratory.
  - Reagent Storage Facilities.
  - Compressor Room.

  - Crushed Mineralized Material Stockpile and Run-of-Mine (ROM) Pad.
- Support Infrastructure:
  - Electrical Substation and Power Distribution Infrastructure.
  - Communications Infrastructure.
  - Fuel Storage and Distribution Facilities.
  - Kitchen and Lunchroom Facilities.
  - Security Building and Controlled Site Access.

Figure 18.1: Project Site Plan



18.2 Roads

The Project road network will comprise access roads, service roads, haul roads, and roads servicing the Dry Stack Tailings Storage Facility (DSTSF). These roads will provide connectivity between the process plant, underground mine infrastructure, ROM pad, stockpile areas, DSTSF, and supporting site facilities. The road network has been developed to support safe and efficient movement of personnel, materials, and mining equipment throughout the life of mine.

Roads will be designed in accordance with applicable industry standards and will incorporate appropriate drainage and erosion control measures, including roadside ditches, culverts, cross-drains, and localized erosion protection where required. Expected road design parameters are summarized in Table 18.1.

Table 18.1: Type of Roads, Length, and Design Parameters

Description	Access Road	DSTSF Road	Service Road	Haul Road
Design Vehicle	WB-20	Komatsu HM300	HL-93 / Single Unit Truck	SANDVIK TH545i / Komatsu HM300
Minimum Lane Width (m)	3.5	1.75 x Width of Truck (if 2 lanes)	3.5	1.75 x Width of Truck (if 2 lanes)
Number of Lanes	2	2	2	2
Shoulder Width (m)	1.5	2	1.5	2
Design Speed (km/h)	40	30	40	30
Minimum Horizontal Curve Radius (m)	45	150	45	150
Total Length (km)	**3**	**1.5**	**1.9**	**1.5**

The primary site access road will extend approximately 3 km from Highway 17 to the main mine infrastructure area and will be designed to accommodate WB-20 design vehicles. Service roads throughout the site will generally follow a similar gravel-surfaced configuration and will provide access between operational and support facilities.

Haul roads and DSTSF access roads will be designed to accommodate the largest planned mining equipment, including the Komatsu HM300 haul truck and the Sandvik TH545i underground haul truck. These roads will connect the infrastructure pad, underground mine access, process plant, ROM pad, stockpile areas, and the DSTSF. Road geometries, operating widths, gradients, and turning radii will be developed to support safe and reliable operation under anticipated site operating conditions.

18.3 Water Management Infrastructure

Surface water management infrastructure will be implemented across the Project site to minimize contact between clean runoff and mine-affected areas, while protecting operational infrastructure from flooding and water accumulation. The water management system will include perimeter drainage ditches, culverts, contact water collection ponds, sedimentation controls, and runoff diversion structures.

Runoff from industrial areas, including the truck shop, fueling facilities, process plant, and mineralized material handling areas, will be collected separately from non-contact runoff and directed toward designated contact water management facilities. Oil-water separators will be installed at hydrocarbon handling areas to remove residual hydrocarbons prior to reuse or discharge.

The dry-stack tailings storage facility will incorporate runoff collection systems and stormwater diversion channels designed to minimize infiltration and maintain operational stability. Detailed hydraulic and hydrological design will be completed during future engineering phases.

18.3.1 Process Water and Fire Water

Process water for the Project will be supplied primarily from on-site boreholes and integrated into the site-wide water management system to support mining and process plant operations. Water distribution infrastructure will include storage tanks, pumping systems, and piping networks designed to provide a reliable water supply throughout the operational areas.

A dedicated fire water storage tank will be installed to supply the site fire protection system, including fire hydrants, sprinkler systems, and emergency fire suppression infrastructure serving the process plant and associated surface facilities. All major site buildings will be connected to the centralized fire protection network supplied by the dedicated fire water system. Specialized fire suppression systems will be installed in high-risk areas, including laboratory facilities and selected process plant areas, where required.

18.3.2 Potable Water

A potable water treatment system will be installed to supply domestic water to the mine dry, office buildings, and other support facilities. Treated water will be used for showers, washrooms, sinks, and general domestic services throughout the site infrastructure.

At the current stage of the Project, Potable drinking water is expected to be supplied through bottled water systems during operations. Evaluation of long-term potable water supply alternatives, including additional groundwater sources and treatment requirements, will be completed during future engineering phases.

18.3.3 Contact Water Collection Pond

Contact water generated from mining, mineralized material handling, and process plant activities will be collected and conveyed to an on-site contact water collection pond. Sources of contact water are expected to include runoff from operational areas, process infrastructure, truck wash facilities, and the dry-stack tailings storage facility.

The contact water collection pond will provide temporary storage and sediment settling prior to reuse within site operations or further treatment, as required. The contact water management system remains conceptual at the PEA stage, and detailed design criteria, treatment requirements, and discharge strategies will be refined during subsequent engineering and environmental studies.

18.3.4 Sewage Treatment and Oil-Water Separation

Domestic wastewater generated from the mine dry, administration buildings, and process plant facilities will be collected and treated through a centralized sewage treatment system designed in accordance with applicable environmental regulations. Treated effluent will be discharged in compliance with regulatory requirements, while residual sludge will be transported to an approved disposal facility.

Oil-water separation systems will be installed in areas where hydrocarbon contamination may occur, including the truck shop, fueling facilities, and equipment wash bays. Water collected from these areas will be treated to remove residual hydrocarbons and suspended solids prior to reuse or discharge within the site water management system.

18.3.5 Effluent Treatment Plant

An allowance for an effluent treatment plant (ETP) has been included in the current Project capital cost estimate to address the anticipated treatment requirements for contact water prior to environmental discharge. Given the preliminary stage of the Project, the exact treatment processes and design criteria have not yet been fully defined.

As additional site water quality and flow data become available through future engineering and environmental studies, the treatment requirements and associated infrastructure may be refined, which could result in adjustments to the current cost estimates and facility configuration.

18.4 Mine Infrastructure

18.4.1 Mine Maintenance Facility and Warehouse

The Mine Maintenance Facility and Warehouse will be located within the Balance of Plant (BOP) area and positioned to provide direct access to the underground mining fleet and surface support equipment. The facility will be designed to support maintenance activities for underground haul trucks, mobile mining equipment, and light vehicles.

The truck shop will include vehicle maintenance bays, overhead cranes, personnel access doors, overhead vehicle doors, tool storage areas, offices, training rooms, and a segregated welding bay. The final number and configuration of maintenance bays will be refined during subsequent engineering phases based on the selected mining fleet and operational requirements. Separate maintenance areas for heavy-duty and light vehicles will be incorporated to improve operational safety and efficiency.

The facility will also include air compressors, air filtration systems, and distribution systems for gear oil, engine oil, transmission oil, grease, coolant, and windshield washer fluid. Floor drainage systems will be connected to an oil-water separator prior to discharge to the site contact water management system.

A dedicated truck wash bay will accommodate underground mining equipment and surface support vehicles, as well as light vehicles. Wash water will be treated to remove sediments and hydrocarbons prior to reuse or discharge in accordance with the site water management strategy.

The warehouse will include storage areas for spare parts, consumables, personal protective equipment (PPE), tools, and operational materials required to support mining activities. The building will include personnel access areas, overhead doors, and loading areas designed to facilitate cargo handling and equipment deliveries. Dedicated storage areas will also be provided for lubricants and greases, including a bulk tank room, indoor oil storage areas, and sheltered tote storage. The warehouse structure is currently planned as a fabric-covered building designed for operation throughout the life of mine.

18.4.2 Mine Administration Building

The Mine Administration Building will accommodate site management, operations personnel, technical services, dispatch, mine rescue, medical response services, meeting rooms, and training facilities. The building is expected to consist primarily of prefabricated modular units installed on a concrete foundation and located adjacent to the primary operational infrastructure.

The facility will also include first response equipment and emergency coordination areas to support rapid deployment during emergency situations.

18.4.3 Mine Dry

The Mine Dry will support underground mining operations and will include changing rooms, showers, washrooms, locker facilities, storage areas, and associated utilities for mine and maintenance personnel. The facility is expected to accommodate approximately 70 male and 30 female personnel.

The building will include:

- Male and female changing rooms with showers and washrooms.
- Locker facilities.
- Storage rooms.
- Electrical room.
- Janitorial room.
- Water heating systems.

Ventilation and dehumidification systems will be designed to manage moisture generated from underground mining activities and maintain suitable drying conditions for underground work clothing.

18.4.4 Explosive Storage Facilities

Explosives storage facilities will be constructed on site to support underground mining operations and will be designed in accordance with applicable regulatory and safety requirements. During early Project phases, prefabricated surface explosive storage facilities are expected to be utilized. Long-term storage strategies may include underground explosive storage depending on operational requirements and regulatory approvals.

18.4.5 Dry Stack Tailings Storage Facility (DSTSF)

A dry-stack tailings management system is proposed for the Project to support the storage of filtered tailings generated from mineral processing operations. Tailings produced by the process plant will be dewatered through dedicated filtration systems prior to transport to the Dry Stack Tailings Storage Facility (DSTSF). The filtered tailings are expected to achieve a moisture content suitable for truck transport, placement, and compaction.

Filtered tailings will be transported from the process plant to the DSTSF using haul trucks operating on dedicated haul roads. Tailings placement will be completed in controlled lifts and compacted during placement to improve geotechnical stability, reduce permeability, and minimize infiltration from precipitation events. The stacking sequence and deposition strategy will be developed to support stable operations throughout the life of mine while maximizing storage efficiency within the facility footprint.

The DSTSF layout has been developed conceptually at the PEA stage based on preliminary topographic and site planning considerations. The facility is expected to incorporate perimeter containment embankments, internal drainage features, stormwater diversion channels, and contact water collection systems designed to manage runoff and maintain operational stability. Runoff generated within the facility footprint will be directed toward designated contact water management infrastructure for collection and management as part of the overall site water management strategy.

Surface water diversion channels and associated drainage infrastructure will be constructed around the DSTSF to minimize contact between non-contact runoff and tailings storage areas. Erosion protection measures, sediment control systems, and water conveyance infrastructure will be incorporated into the final design to support long-term operational performance and environmental management objectives.

The dry-stack tailings management approach is expected to reduce overall water consumption by maximizing water recovery within the process plant and minimizing the volume of water reporting to the tailings' storage facility. Recovered water from the filtration process and contact water management systems may be reused within the process plant, subject to operational requirements and water quality considerations.

The final location, storage capacity, embankment configuration, drainage systems, operational sequencing, and closure strategy for the DSTSF will be refined during future geotechnical, hydrological, hydrogeological, and environmental investigations. Detailed design activities during subsequent engineering phases will include geotechnical characterization, seepage assessments, water balance evaluations, stability analyses, and closure planning in accordance with applicable regulatory guidelines and industry standards.

18.5 Process Infrastructure

18.5.1 Mineral Processing Facility

The mineral processing facility will house the principal mineralized material processing equipment and associated material handling systems required to support the proposed plant throughput. The facility will include crushing, grinding, flotation, concentrate filtration, gold recovery, and tailings filtration systems designed to support the production of copper and zinc concentrates, as well as gold. These process systems are further described in Chapter 17.

Separate filter press systems are planned for zinc and copper concentrate dewatering, as well as for tailings dewatering to support dry-stack tailings disposal. The process plant complex will also include control room facilities, process supervision offices, training areas, and maintenance coordination spaces required to support plant operations. The final layout and detailed design of these facilities will be refined during subsequent engineering phases.

18.5.2 Assay and Metallurgical Laboratory

The assay, metallurgical, and environmental laboratory facilities will support grade control, process monitoring, metallurgical testing, and environmental monitoring activities within a centralized laboratory building.

The laboratory will include dedicated areas for sample preparation, analytical chemistry, instrumentation, leach testing, and environmental testing. Laboratory safety systems will include fume hoods, scrubbers, gas detection systems, make-up air systems, and associated ventilation infrastructure designed in accordance with industrial laboratory standards.

The facility is expected to include analytical equipment for XRF analysis, carbon-sulfur determination, and wet chemistry testing. Final equipment lists, laboratory layouts, and staffing requirements will be refined during detailed engineering.

18.5.3 Reagent Storage Facilities

Process reagents will be stored within a dedicated steel-framed building equipped with insulated walls and roof panels. Reagents will be segregated using containment walls and curbing systems designed to prevent cross-contamination and contain potential spills.

Dedicated storage areas will be provided for individual reagent types in accordance with applicable safety and handling requirements. Detailed reagent handling systems and storage arrangements will be further developed during subsequent engineering phases.

18.5.4 Compressor Room

Compressed air systems required for underground mining operations and process plant services will be housed within a dedicated compressor room. The facility will supply compressed air to underground mining infrastructure, maintenance facilities, and selected process plant systems. Equipment selection and detailed layout will be finalized during future engineering phases.

18.5.5 Crushed Mineralized Material Stockpile and Run-of-Mine Pad

A run-of-mine (ROM) mineralized material stockpile will be located near the primary crushing facility to provide operational flexibility between mining and processing activities. The stockpile is intended to accommodate short-term fluctuations in mine production and process plant throughput while supporting continuous plant operation.

The stockpile design, storage capacity, and overall footprint have been developed at a conceptual level appropriate for the PEA stage and will be refined during subsequent engineering phases.

18.6 Support Infrastructure

18.6.1 Fuel Storage and Distribution Facilities

Fuel storage and distribution facilities will be installed to support underground mining operations and surface support operations throughout the Project site. The fuel distribution system will include dedicated dispensing infrastructure for underground mobile mining equipment, service vehicles, and light-duty vehicles.

Diesel fuel will be stored in an above-ground storage tank equipped with secondary containment systems designed in accordance with applicable environmental and safety requirements. The conceptual design will include diesel fuel storage capacity sufficient to provide approximately four (4) days of operational autonomy under average operating conditions. The relatively limited on-site storage capacity has been selected considering the close proximity of existing public fuel supply infrastructure.

Fuel handling and dispensing infrastructure will be installed on concrete containment pads and will incorporate spill prevention and containment measures. The detailed design and configuration of the fuel distribution system will be refined during subsequent engineering phases.

18.6.2 Kitchen and Lunchroom Facilities

Kitchen and lunchroom facilities will be provided to support site personnel during operations. The facilities will include designated food preparation areas, hygiene stations, freezer and dry storage areas, and associated support spaces. The lunchroom is expected to accommodate approximately 48 operational and technical personnel at maximum capacity.

The facility will also include boot wash stations and personal protective equipment storage areas. Catering services are currently anticipated for food preparation and delivery during operations. Commercial kitchen equipment and dedicated HVAC systems will be installed as required.

18.6.3 Security Building and Controlled Site Access

Site security infrastructure will include a controlled access gate and security building to manage personnel and vehicle access to the Project site. CCTV systems will monitor critical operational areas, including the process plant, truck shop, parking areas, and other key infrastructure.

Additional security measures will be implemented for high-value process areas and operational infrastructure as required.

18.6.4 Power Supply and Distribution

Electrical power for the Project will be supplied by the local utility through a new feeder connection from the regional electrical grid to the main site substation. The main substation will include two (2) 20/26 MVA power transformers and associated 13.8 kV switchgear infrastructure designed to support underground mining operations, mineral processing activities, and support surface infrastructure.

Electrical distribution to underground mining infrastructure will be supplied through dedicated transformers equipped with 4.16 kV secondary windings, while process plant equipment and ancillary facilities will be supplied at 4.16 kV, 480 V, and 208/120 V, depending on operational requirements and equipment load characteristics. The average electrical load for the Project is currently estimated at approximately 6.4 MW, with peak demand anticipated to reach approximately 9 MW.

Emergency power generation systems will be installed to maintain critical operational and safety systems during utility power interruptions. The current conceptual design includes two (2) diesel generators rated at approximately 2 MW each operating at 13.8 kV. Emergency power systems will provide backup electrical supply for critical underground and surface infrastructure, including mine ventilation systems, communications systems, water management infrastructure, emergency lighting, control systems, and other safety-related equipment required to maintain safe operating conditions during outage events.

The Project electrical distribution system will be configured at 13.8 kV, 60 Hz, and will consist of overhead distribution lines, transformers, switchgear assemblies, motor control centers, cableways, and distribution panels. Electrical loads throughout the Project will be supplied at operating voltages including 4.16 kV, 480 V, and 208/120 V.

The process plant will represent the primary electrical load centre and will be supplied directly from the main substation switchgear. Additional Project infrastructure, including the Dry Stack Tailings Storage Facility (DSTSF), effluent treatment plant, administration buildings, mine maintenance facility, warehouse, fuel storage facilities, explosives storage facilities, water management infrastructure, water treatment plant, sewage treatment plant, and communications infrastructure, will be supplied through overhead distribution lines connected to the Project electrical network.

Power distribution infrastructure will also support underground electrical systems associated with mine development and underground operations. The detailed design, redundancy philosophy, and configuration of the electrical distribution network and emergency power systems will be refined during subsequent engineering phases.

18.6.5 Communications Infrastructure

The Project communications infrastructure will include redundant fibre optic and microwave communication systems to support operational communications, process control systems, internet connectivity, and underground safety systems throughout the construction and operational phases of the Project. Off-site communications infrastructure will provide high-speed network connectivity to the Project site, while on-site fibre optic infrastructure will connect the various operational and support facilities through an integrated communications backbone.

A centralized data centre will be established to support the Project's core network and computer systems. The facility will include continuous power supply and cooling systems and is expected to accommodate a minimum of three (3) standard equipment racks.

Radio communications systems will be installed to support construction activities, underground mining operations, operational coordination, and personnel safety. The communications system is currently expected to include a trunking-based radio network with multiple communication channels, supported by handheld and mobile radio units distributed throughout the site.

Underground communications infrastructure will utilize leaky feeder technology or equivalent systems to maintain continuous communication with underground personnel and mobile mining equipment. These systems will support operational coordination, personnel tracking, and emergency response activities within underground workings.

A fibre-based Gigabit Passive Optical Network (GPON) will also be implemented to provide communications services to operational buildings and personnel support facilities throughout the Project site. Mobile communications infrastructure, including 4G/LTE coverage, will be evaluated and implemented as required to support reliable site-wide communications during operations.

Internet service provider arrangements and mobile communications systems will be further evaluated during subsequent engineering phases to ensure reliable communications coverage, operational redundancy, and cost-effective long-term operation.

19. MARKET STUDIES AND CONTRACTS

The Kay Mine Project is expected to produce gold and silver in doré bars, as well as copper concentrate containing payable copper, gold and silver, and zinc concentrate containing payable copper, zinc, gold and silver.

19.1 Metal Market

19.1.1 Copper Market

Copper (Cu) is a base metal commonly used in electrical infrastructure, construction, and manufacturing applications. Copper is traded in established global markets, with prices determined through transparent and widely reported pricing mechanisms. Prices are typically quoted in United States dollars (USD) per pound.

19.1.2 Zinc Market

Zinc (Zn) is a base metal primarily used in the galvanization of steel to prevent corrosion, as well as in the production of alloys and other industrial applications. Zinc is traded in established global markets, with prices determined through commonly referenced international pricing mechanisms. Prices are typically quoted in United States dollars (USD) per pound.

19.1.3 Gold Market

Gold (Au) is a precious metal that is traded globally and used primarily as a store of value and for investment purposes, with additional limited industrial and commercial applications. Gold is transacted daily through banks and trading platforms at a spot price for immediate delivery. Prices are typically quoted in United States dollars (USD) per troy ounce.

19.1.4 Silver Market

Silver (Ag) is a precious metal with both industrial and investment applications, including use in electronics, solar energy, and manufacturing. Silver is traded in established international markets through spot and futures transactions. Pricing reflects both investment demand and industrial consumption. Prices are typically quoted in United States dollars (USD) per troy ounce.

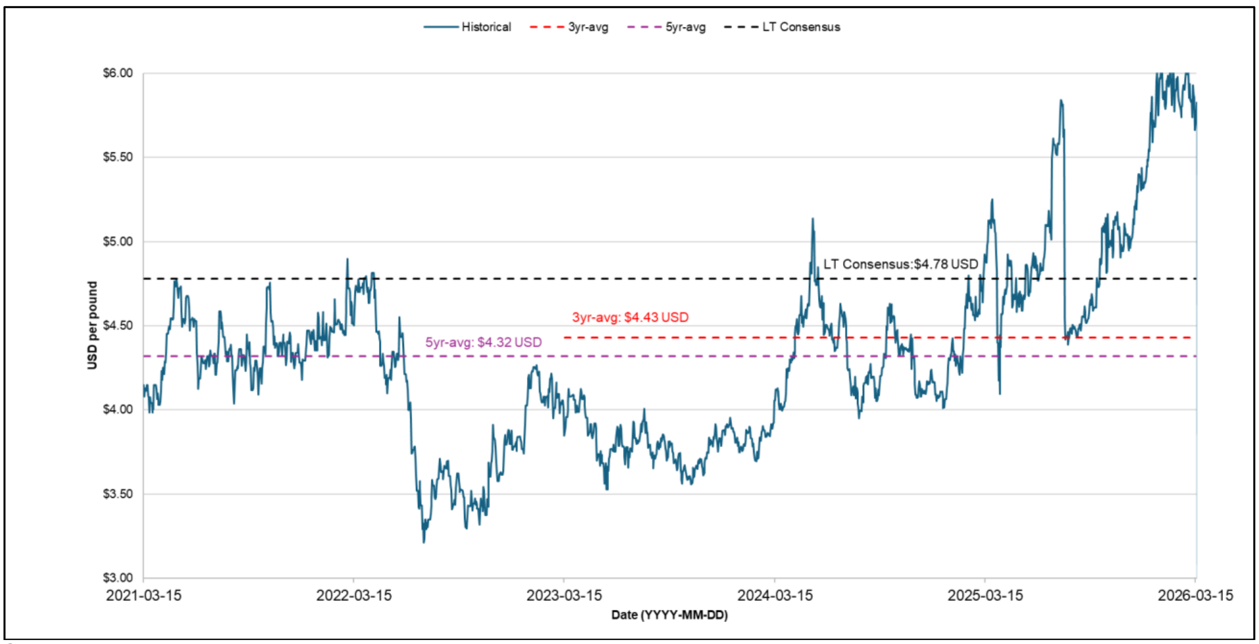
19.2 Metal Price

The metal prices are typically established based on review of historical prices, long-term broker consensus forecast and pricing assumptions used by industry peers. The long-term consensus prices as of March 3, 2026, are USD 4.78 per pound for copper, USD 1.23 per pound for zinc, USD 3,515 per troy ounce for gold, and USD 45.26 per troy ounce for silver (source: Broker Consensus Estimates from CIBC Capital Markets).

The financial analysis for the Kay Mine Project considered a copper price of $4.70/lb, a zinc price of $1.27/lb, a gold price of $3,100/oz and a silver price of $38.00/oz. An exchange rate of 1.34 Canadian dollars per US dollar (1.34 CAD/USD) was used for this PEA.

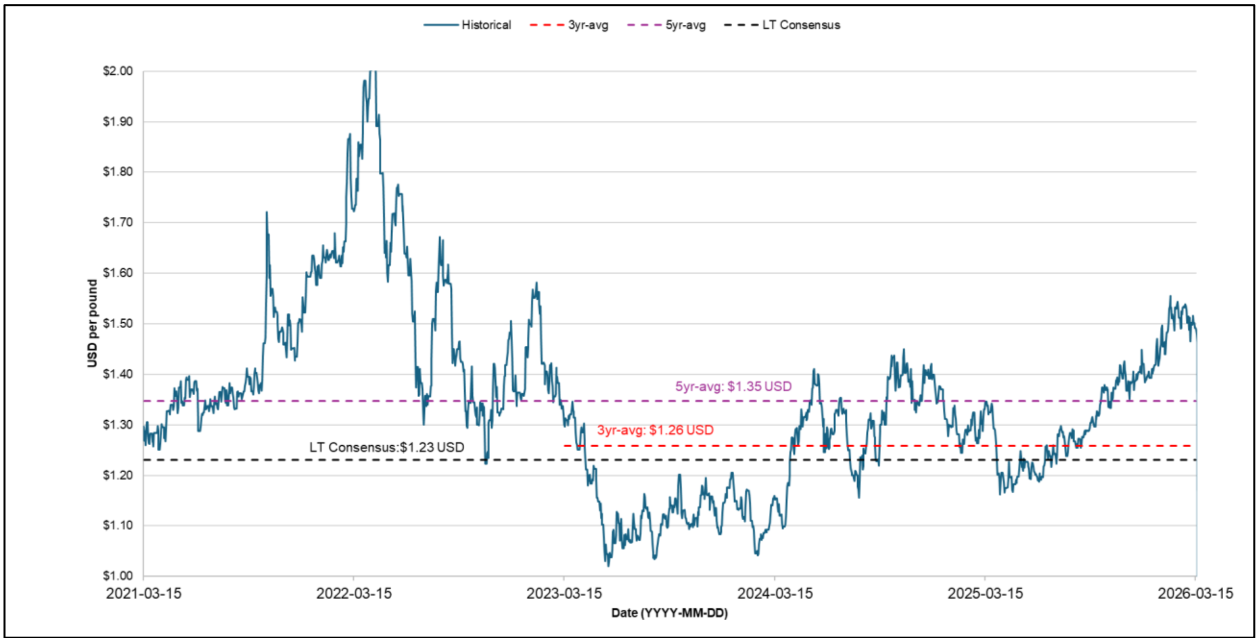
Figure 19.1 to Figure 19.4 show the historical daily average value of copper, zinc, gold and silver for the past five (5) years.

Figure 19.1: Daily Copper Price



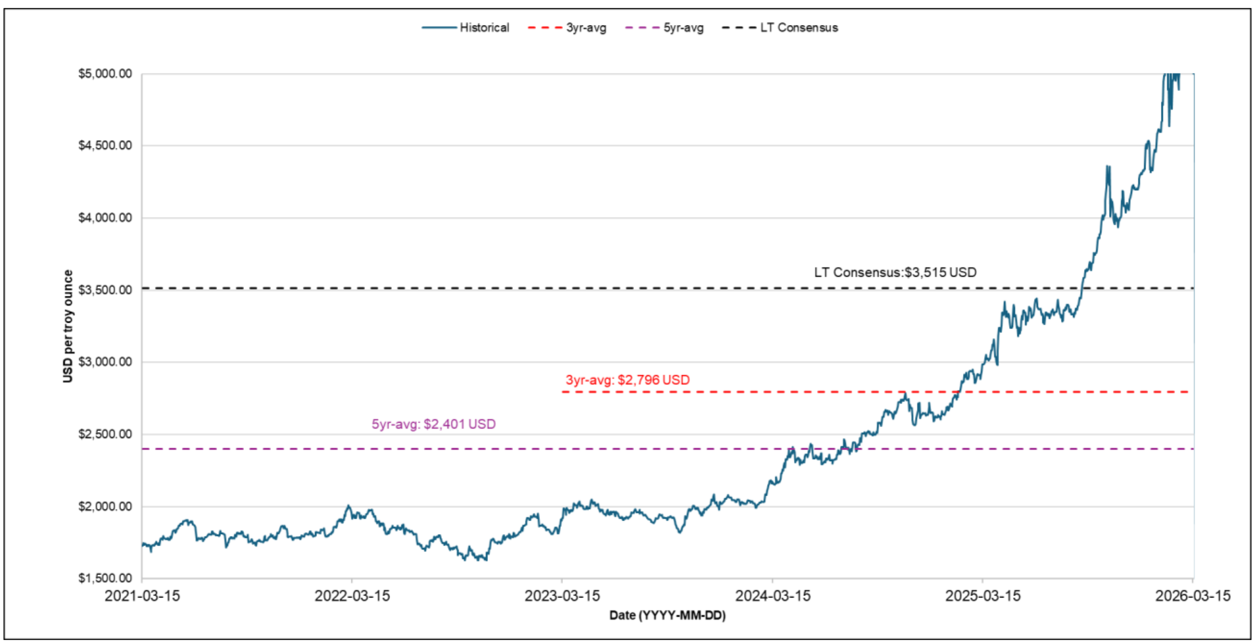
Source: Metal Price API

Figure 19.2: Daily Zinc Price



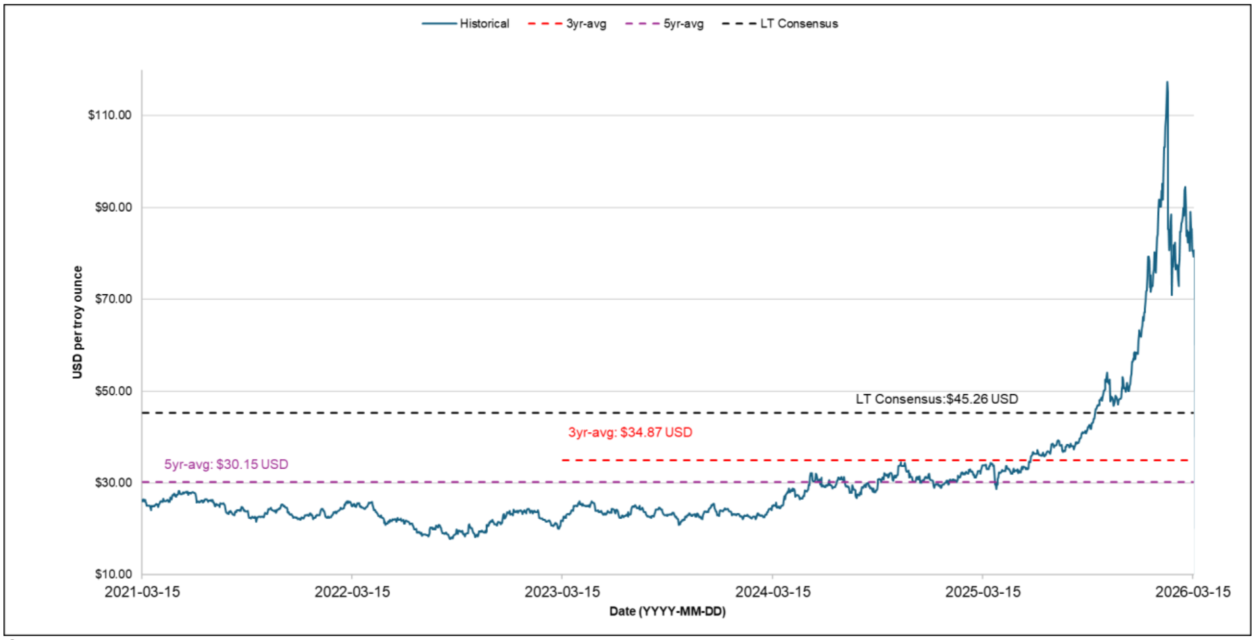
Source: Metal Price API

Figure 19.3: Daily Gold Price



Source: Metal Price API

Figure 19.4: Daily Silver Price



Source: Metal Price API

19.3 Contracts

There are currently no refining agreements or sales contracts in place for the Project that are relevant to this Technical Report. Any future sales or processing arrangements are expected to be negotiated on terms consistent with prevailing industry practices. There are no mining, concentrating, smelting, refining, transportation, handling, sales and hedging, forward sales contracts or arrangements for the Project. This situation is typical for a development stage project, still several years away from production

19.4 Concentrates

The PEA considers the production of two (2) concentrates: copper and zinc. A pyrite concentrate is also assumed to be processed through an Albion Process™ circuit, followed by cyanide leaching, to recover gold and silver, which are recovered as doré bars.

Table 19.1 summarizes the refining cost for each metal. Table 16.8 and Table 19.3 present the metal payability factors, concentrate grades, treatment and refining charges, transportation costs, and applicable smelting terms based on industry-standard assumptions for copper and zinc concentrates, respectively. Table 16.10 presents the concentrate grade for the pyrite concentrate, as well as the percentage recovery for the Albion Process used to process the recovered gold and silver into doré bars.

Table 19.1: Refining Cost

Metal	Unit	Refining Cost
Copper	USD/lb	0.05
Zinc	USD/lb	0.00
Gold	USD/oz	5.00
Silver	USD/oz	0.50

Table 19.2: Copper Concentrate Parameters

Copper Concentrate			
Payable Metal	**% Payable**	**Minimum Payable**	
Copper (Cu)	95.00%	-	
Gold (Au)	95.00%	1 g/t	
Silver (Ag)	95.00%	30 g/t	
Treatment Charges	50 USD/dmt		
Transportation	130 USD/wmt		
Transport Loss & Insurance	0.2%		
Concentrate Grade	27.1%		
Contaminant Metal*	**Unit Price (USD/dmt)**	**Penalty Limit**	**Increment**
Arsenic (As)	2.00	0.1%	0.1%
Antimony (Sb)	4.00	0.1%	0.1%
Mercury (Hg)	0.20	10 ppm	1 ppm
Zinc (Zn) + Lead (Pb)	2.00	3%	1%

**Grades within concentrate are fixed for the following metals: As 0.98%; Hg 68 g/t; Zn 4.24%; Pb 3.32%.*

Table 19.3: Zinc Concentrate Parameters

Zinc Concentrate			
Payable Metal	**% Payable**	**Minimum Payable**	
Zinc (Zn)	95.00%	-	
Gold (Au)	95.00%	1 g/t	
Silver (Ag)	95.00%	30 g/t	
Treatment Charges	100 USD/dmt		
Transportation	130 USD/wmt		

Zinc Concentrate			
Payable Metal	**% Payable**	**Minimum Payable**	
Transport Loss & Insurance	0.2%		
Concentrate Grade	58.7%		
Contaminant Metal	**Unit Price (USD/dmt)**	**Penalty Limit**	**Increment**
Lead (Pb)	2.00	3.5%	1%
Mercury (Hg)	1.50	250 ppm	100 ppm

Table 19.4: Pyrite Concentrate Parameters

Pyrite Concentrate	
Concentrate Grade	4.23 g/t
Albion Process	
Payable Metal	**% Recovery**
Gold (Au)	91.74%
Silver (Ag)	92.63%

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

AMC is currently acknowledged to conduct up to five (5) acres of mineral exploration activities under a Notice with the Bureau of Land Management, Hassayampa Field Office (BLM). In addition, AMC filed an Exploration Plan of Operations (EPO) with the BLM in January 2026 to allow for expanded exploration operations. The BLM is currently processing the EPO, including the completion of an Environmental Assessment (EA), in order to comply with the National Environmental Policy Act (NEPA). As part of the Notice and EPO, AMC has completed a number of environmental baseline studies within and adjacent to the Kay Mine Project. These studies looked at cultural resources, biological resources, threatened and endangered (T&E) species, BLM sensitive species, native plants, and Waters of the United States (WOTUS). It is anticipated that the BLM will approve the EPO in the first quarter of 2027. The EPO Project Area is the same as the Kay Mine Project Area, except it does not include any private lands.

20.1 Existing Environmental Conditions

The Kay Mine Project is located immediately adjacent to the town of Black Canyon City, approximately 69 km (43 miles) north of the city of Phoenix, in central Arizona. The Kay Mine Project is located in Sections 4, 5, 8, and 9, Township 8 North, Range 2 East (Gila and Salt River meridian), in the Tip Top mining district in Yavapai County, Arizona. The surface disturbing facilities associated with the Kay Mine are located entirely on BLM-managed public land, while the underground portion of the mine is located below BLM-managed public land and private land.

Small historical mine dumps exist onsite at the No. 1, No. 2, and No. 3 shafts and are likely to contain sulfide minerals, particularly pyrite, which have the potential for producing acidic surface waters as they oxidize. The mineralization at the Kay Mine Project contains significant arsenic, above 10%, as discovered in previous AMC drill samples. Given the proximity of these mine dumps to the Aqua Fria River, AMC will consult with a local environmental consultant to evaluate whether any environmental risk exists from these historic mine features.

20.2 Permitting and Baseline Studies

This section of the NI 43-101 Technical Report summarizes the permits that will likely be required to conduct mining and mineral processing activities at the Kay Mine Project. The details of the Kay Mine Project and activities are not fully designed at this time; however, some general design criteria are known. The Kay Mine Project will be an underground mining operation and associated surface facilities with an estimated ten-year mining and processing life.

The following baseline studies occurred in support of the EPO and EA: a Class III cultural resources inventory; a Biological Evaluation (BE); and a Waters of the United States (WOTUS) technical memorandum. The Class III cultural resources inventory, conducted between June and August 2025, identified one (1) previously recorded site, 27 newly identified sites, and 148 isolated occurrences. Out of the 28 sites documented, six (6) are recommended eligible for listing on the National Register of Historic Places (NRHP), and 22 are not recommended eligible for listing. All 148 isolated occurrences are not recommended as eligible for the NRHP due to a lack of significance under the NRHP eligibility criteria.

The BE evaluated the potential occurrence within the EPO project area of special status species, defined as those species designated by the U.S. Fish and Wildlife Service (USFWS) as Endangered, Threatened, Proposed for listing, or Candidate for listing under the Endangered Species Act (ESA); species protected under the Bald and Golden Eagle Protection Act (BGEPA); and species designated as sensitive by the BLM for the Phoenix District. The BE identified the Threatened, yellow-billed cuckoo, and the BLM sensitive lowland leopard frog and Sonoran Desert tortoise as present in the EPO project area.

A technical memorandum was prepared for the Kay Exploration Project, in the same area as the mine, that evaluates the surface water features for potential as WOTUS. Three (3) primary surface water features intersect the Review Area: Agua Fria River, Black Canyon Creek, and Slate Creek, the latter two (2) of which are tributaries to the former. Tributaries to these features also occur in the Kay Mine Project Area, as well as an unnamed ephemeral drainage in the southwestern corner of the Kay Mine Project Area. While most of the surface water features were not identified as potential WOTUS, both Black Canyon Creek and the Agua Fria River support ephemeral and intermittent flows and may be considered WOTUS. There are also potential wetland features along the Agua Fria River, which may be considered WOTUS.

20.3 Project Design

In general, the proposed underground mine will consist of one (1) mine portal and two (2) ventilation raises. AMC plans the construction, operation, reclamation, and closing of this mining operation. The major components include:

- Underground mine.
- Mine portal.
- Ventilation raises.
- Waste rock dump.
- Borrow pit.
- Crushing and conveying system.

- Water pond and water management.
- Fire Protection, Potable Water, Sewage Water, and Effluent Water Treatment.
- Warehouse, Canteen and Lunchroom, and First Response Services.
- Workshop and truck shop.
- Laboratory, reagents storage and mill office.
- Fuel storage facility.
- Compressor room.
- Electrical substation and power distribution.
- Dry-stack tailings storage facility.
- Explosives storage.
- Crushed mineralized material stockpile.
- Mineral processing facility.
- Water supply and delivery system.
- Run-of-mine (ROM) pad.

The mine is planned for 10 years of production at a processing rate of 700,000 tons of mineralized material per year. A 30-month construction phase is required prior to the start of operations. South of the portal, a temporary waste rock dump will be constructed to store approximately one million tons of waste rock. From Year 3 to Year 5 of the production phase, this waste rock will be returned underground and used as backfill to provide ground support for the stopes. An additional 1.7 million tons of rock sourced from a borrow pit located west of the infrastructure will be required to backfill stopes through Year 10.

The mineralized material and waste rock would be extracted from the underground mine using conventional longhole open stoping mining methods of drilling, blasting, mucking, and hauling. AMC would use load-dump-haul units to load the blasted mineralized material and waste into the haul trucks, which would transport the material to the surface. Mineralized material would be transported directly to the crusher or ROM pad and mineralized material stockpile, depending on crusher availability. A processing facility will deliver mineral concentrates to two (2) separate filter press systems for the production of Cu/Pb and Zn concentrates, as well as a separate filter press system for the dewatering of the processing facility tailings to support a dry-stack tailings method. The processing facility is conceptual in nature at this PEA stage and is designed to support the proposed plant throughput, with layout and design details to be developed in subsequent engineering phases.

Water management for the Project includes the following:

- Process water will be supplied from on-site wells, with a water management system designed to support plant operations.
- Fire water will be stored in a dedicated tank.
- The potable water treatment plant will be installed to treat water. Treated water will be used for showers, toilets and sinks in the dry and office complexes. Water from the treatment plant is not intended for consumption, and bottled drinking water will be supplied throughout the site for drinking.
- Contact water generated from mining and processing activities will be collected and conveyed to an on-site containment pond. This will include contact water from surface facilities, tailings management, and mine-related activities. The containment pond will provide temporary storage and basic settling prior to reuse in site operations or further management as required. The contact water management system is conceptual at this PEA stage, and detailed design, treatment requirements, and discharge criteria will be defined in subsequent engineering and environmental studies.

Domestic wastewater from the mine dry, process plant, and office facilities will be collected and treated in a centralized sewage treatment system. Treated water will be discharged, while sludge will be disposed of at an approved location consistent with applicable regulatory standards.

Oil-water separators will be placed near contamination-prone areas, such as the truck shop and fuel bay. Wash bay water will be skimmed for oil and reused for equipment cleaning.

20.4 Environmental Baseline Data Needs

AMC’s current baseline characterization activities have been focused on metallurgical testing for mineral resource estimates, as well as the studies for the Notice and EPO. Environmental baseline data collection, in addition to the Notice and EPO studies, that will be necessary to complete permit applications, includes groundwater characterization, geochemical characterization of the tailings and waste rock, an air quality impact assessment, and socioeconomic evaluation. The scheduling of this work needs to be coordinated with the overall Kay Mine Project development schedule.

The hydrologic characterization testing would include the installation of groundwater wells and piezometers. One or more groundwater pump tests would be conducted to better understand the groundwater system and the overall dewatering and water supply requirements. Collection of water quality data would begin once the groundwater wells are installed. The data collected from this characterization will be used in a groundwater flow model to understand the potential effects of the Kay Mine Project water supply and mine dewatering activities on the groundwater system and the water supply for Black Canyon City.

Geochemical testing of tailings and waste rock is required to assess the non-potentially acid-generating (Non-PAG) and potentially acid-generating (PAG) characteristics of the materials. The main waste management consideration for the Project is the prevention and control of potential metal leaching and acidic solutions from the tailings and waste rock that would be produced during mine development or operations.

The air quality impact assessment will use the calculated emissions of criteria pollutants from the various Project activities in an air quality model to evaluate the air quality impacts relative to the applicable ambient air quality standards.

The socioeconomic assessment will use Kay Mine Project employment and fiscal information to evaluate the Project's potential effects on the community of Black Canyon City, the surrounding area, and the County of Yavapai.

20.5 Project Permits

In order to conduct mining and processing activities, the Project will need specific permits from the United States Department of the Interior Bureau of Land Management (BLM) and several State of Arizona agencies, including the Arizona State Mine Inspector and several units within the Arizona Department of Environmental Quality (ADEQ). The following is a list of the major permits that will be required, followed by a brief discussion of each. None of the permits is currently in the application stage.

- BLM Plan of Operations.
- Arizona Mined Land Reclamation Permit.
- Aquifer Protection Permit.
- Air Quality Permit.
- AZPDES Multi-Sector General Permit (MSGP) for mining.
- AZPDES Individual Permit.

20.5.1 BLM Plan of Operations

A Plan of Operations (Plan) is required for submittal to and approval by the BLM in compliance with 43 Code of Federal Regulations 3809 (43 CFR 3809). A Plan outlines the processes that a mining project would undergo to prevent unnecessary or undue degradation of public lands by operations authorized by the mining laws. Submittal of the Plan will trigger review under NEPA, though it is not currently clear whether the review would be completed as an Environmental Assessment or an Environmental Impact Statement.

20.5.2 Mined Land Reclamation Plan

A Mined Land Reclamation Plan, described under the Arizona Mined Land Reclamation Act (A.R.S. §§ 27-901 et seq.) and administered by the Arizona State Mine Inspector (ASMI), is required for any operator conducting surface mining operations that result in surface disturbance of five (5) or more acres on private lands. Prior to commencing operations, the operator must submit a Reclamation Plan to ASMI for review and approval. The applicant must demonstrate the following: that disturbed mined lands will be reclaimed to a safe, stable, and non-hazardous condition capable of supporting an approved post-mining land use; that the Reclamation Plan adequately addresses grading, revegetation, and long-term stabilization of all disturbed areas, including waste rock dumps, tailings, haul roads, and processing facilities; and that sufficient financial assurance — in the form of a performance bond, letter of credit, or other ASMI-approved instrument — has been established to cover the full estimated cost of reclamation in the event of operator default.. Early integration of reclamation planning into project design will be beneficial to long-term planning and will inform the financial assurance calculation.

20.5.3 Aquifer Protection Permit

An Aquifer Protection Permit (APP), issued by the Arizona Department of Environmental Quality (ADEQ) Groundwater Protection, Reclamation and Reuse Section, is needed if one of the Project facilities discharges a pollutant that has a reasonable probability of reaching an aquifer. The applicant for an APP must demonstrate the following: that Best Available Demonstrated Control Technology (BADCT) will be utilized to prevent or eliminate the discharge of pollutants, that aquifer water quality standards will not be violated in groundwater at the point of compliance, that the applicant has financial and technical capability to comply with the permit, and that the property has been properly zoned for the activity. An APP also needs to be obtained if the mine site contains a non-municipal solid waste landfill. Early collection of groundwater quality data and development of the groundwater model will inform the APP application.

20.5.4 Air Quality Permit

An Application for a Class II Air Permit for those portions of the stationary sources that have the potential to emit pollutants must be prepared using forms provided by the ADEQ Air Quality Permits Section. The application includes the following main components: a detailed description of each process at the facility; a flow diagram for all processes; a detailed emission inventory and calculations; and a comprehensive list of all equipment. Local meteorological data informs the development of these application materials.

20.5.5 Arizona Pollutant Discharge Elimination System Multi-Sector General Permit for Mining

For discharges of industrial stormwater to protected receiving waters, coverage under the AZPDES program is required. Stormwater management will be addressed through the preparation of a stormwater pollution prevention plan (SWPPP) and submittal of a Notice of Intent to discharge through the ADEQ Surface Water Section, Stormwater and General Permits Unit. The SWPPP will be developed in compliance with the AZPDES Multi-Sector General Permit (MSGP) for Mining.

20.5.6 Arizona Pollutant Discharge Elimination System / Individual Permit

For compliance with the CWA, facilities that plan the discharge of municipal, domestic, and non-domestic (industrial) discharges of pollutants to protected surface waters shall obtain an AZPDES Individual Permit. A permit application (Forms 1 and 2D) is required for submittal to and approval by ADEQ.

20.5.7 Minor Permits and Applications

In addition to the above-noted permits, Table 20.1 lists potential other notifications or ministerial permits that will likely be necessary to conduct the mining operations.

Table 20.1: Required Minor Permits and Notifications

Notification / Permit	Agency
Federal	
Notification of Commencement of Operation	U.S. Department of Labor Mine Safety and Health Administration (MSHA)
MSHA Identification Number and MSHA Coordination	MSHA
Hazardous Waste Identification Number	U.S. Environmental Protection Agency (EPA)
Clean Water Act Section 404 Nationwide Permit	Army Corps of Engineers
Explosives User Permit	Bureau of Alcohol, Tobacco, Firearms and Explosives
Radio License	Federal Communications Commission
State	
Notice of Start-up of Mine Operations	Arizona State Mine Inspector
Hazardous Waste, Treatment, Storage and Disposal Permit	ADEQ Hazardous Permits Unit
Special Waste Identification Number	ADEQ Solid Waste Unit

Notification / Permit	Agency
Notice of Intent to Drill, Deepen, or Modify a Monitor / Piezometer / Environmental Well	Arizona Department of Water Resources, Groundwater Permitting and Wells Section
Fire Safety Inspection and Permit	Arizona Office of the State Fire Marshal
Local	
Yavapai County Mining / Metallurgical Use Exemption	Yavapai County Chief Zoning Inspector

20.6 Social and Community

The Kay Mine is located next to the town of Black Canyon City, in Yavapai County, Arizona, approximately 69 kilometres (43 miles) north of Phoenix, Arizona. Access to the Kay Mine Project is from Interstate 17 and Old Black Canyon Highway. AMC has a community outreach page on their website where individuals can submit questions, join AMC's mailing list, and learn more about the current mineral exploration activities and what actions are being taken to minimize community impacts. AMC is also a sponsor of the Black Canyon City Heritage Park, mining museum, local food bank, and nearby school sports teams. In June 2025, an External Relations Plan was developed for AMC's proposed exploration project. The same strategies could continue to be used for the Kay Mine Project. The plan outlines suggestions for how to achieve consistent company and project messaging, methods of community engagement, such as community partnerships and participation in community events, social media communications, and lists the main key stakeholder groups that AMC should consider engaging. AMC has been engaging with the BLM, county, and state regulators on their current and planned exploration activities.

20.7 Waste Characterization

Currently, AMC has not commenced their material characterization program for the tailings and waste rock that will be managed within the Kay Mine Project Area. AMC plans to dispose of all hazardous and non-hazardous wastes, excluding waste rock and processed mineralized material from the mining operation, at off-site, state-approved waste disposal sites.

20.8 Closure and Reclamation Strategy

20.8.1 Regulatory Framework

Permits and entitlements required for construction and operations are described in detail in Section 20.5. The following section addresses specific closure and reclamation obligations in the context of those permits and entitlements.

Closure and reclamation of the proposed Kay Mine underground mining operation, which is planned to be located on both public and private lands, will be regulated under an integrated federal and state permitting framework. Agencies involved in the permitting process include the Bureau of Land Management (BLM), the Arizona State Mine Inspector (ASMI) and the Arizona Department of Environmental Quality (ADEQ).

Given that the Project is located on public lands administered by the BLM, the Project is subject to federal surface management requirements under the Federal Land Policy and Management Act (FLPMA), 43 United States Code (U.S.C.) § 1701 et seq., and implementing regulations at Title 43 Code of Federal Regulations (43 CFR) Subpart 3809.

Given the potential for facilities planned as part of the Project to create discharges (or potential discharges) to an aquifer or vadose zone in the State, A.R.S. Title 49 Chapter 2 authorizes the ADEQ to require those who intend to construct and operate a facility that creates these discharges (or potential discharges) to obtain an Aquifer Protection Permit (APP). While typically considered an operational permit, the APP program also considers the eventual cessation of operations and the restoration of vadose and aquifer conditions associated with those permitted facilities.

Although these regulatory programs operate independently; closure planning, reclamation design, financial assurance, and post-closure monitoring obligations should be developed in a coordinated manner to ensure compliance with all applicable statutory and regulatory requirements.

20.8.1.1 Federal Requirements – Plan of Operations and NEPA

Mining operations on public lands administered by the BLM that exceed casual use require approval of a Plan of Operations pursuant to 43 CFR § 3809.11. The Plan of Operations governs development, operation, closure, reclamation, and post-closure monitoring of the Project. Under the FLPMA of 1976, 43 U.S.C. § 1732, and 43 CFR § 3809.5 and § 3809.420, the BLM must ensure that operations prevent “unnecessary or undue degradation” of public lands.

The Plan of Operations must include detailed descriptions of the planned underground development, surface facilities, waste rock storage areas, tailings facilities, water management infrastructure, and include a comprehensive reclamation and closure plan. A third-party reclamation cost estimate must be provided to support the calculation of an appropriate reclamation financial guarantee required under 43 CFR § 3809.552. The financial assurance must be posted prior to surface disturbance pursuant to 43 CFR § 3809.500.

Approval of the Plan of Operations constitutes a major federal action subject to the National Environmental Policy Act (NEPA), 42 U.S.C. § 4321 et seq. For a project of this scale and complexity, preparation of an Environmental Impact Statement pursuant to 42 U.S.C. § 4332(2)(C) and 40 CFR Parts 1,500 through 1,508 is anticipated. The NEPA process evaluates potential direct, indirect, and cumulative impacts of the Project, including impacts to groundwater and surface water, geochemical behaviour of tailings and waste rock, underground mine flooding, geotechnical and seismic stability, air quality, biological resources, cultural resources, and socioeconomics. The NEPA Record of Decision establishes binding mitigation measures and closure performance standards incorporated into the approved Plan of Operations.

20.8.1.2 Arizona State Mine Inspector –Reclamation Plan

As previously discussed, mining and exploration operations on private lands in Arizona, involving more than five (5) acres of disturbance, require approval of a Mined Land Reclamation Plan from the ASMI pursuant to A.R.S. § 27-921 and § 27-922. ASMI authority is limited to surface reclamation and physical stability, and groundwater protection is regulated separately by the ADEQ under Title 49.

The Reclamation Plan must describe grading and recontouring, erosion control, revegetation, demolition and removal of surface structures, and closure of mine portals and shafts to ensure long-term public safety. The plan must be accompanied by a third-party reclamation cost estimate and an approved financial assurance mechanism pursuant to A.R.S. § 27-922. Arizona law prohibits duplicative bonding for the closure of specific facilities secured under the APP and covered by an approved Reclamation Plan; however, all surface reclamation components must be fully financially assured.

Similarly, the ASMI-approved Reclamation Plan will defer to the BLM for the closure and reclamation of facilities and structures on BLM-administered lands. Therefore, it is likely that the property will have two (2) integrated closure plans, one with ASMI and the other with the BLM.

Bond release under ASMI jurisdiction is contingent upon the demonstration of successful reclamation, including revegetation success typically evaluated over multiple growing seasons, and confirmation that reclaimed landforms are stable and safe.

20.8.1.3 Arizona Department of Environmental Quality – Aquifer Protection Permit

Mining operations in Arizona that include facilities that discharge or have the potential to discharge pollutants to the vadose zone or aquifer require an individual APP pursuant to A.R.S. § 49-241 and § 49-243. For the Project, facilities anticipated to require APP authorization include the dry-stack tailings facility, waste rock storage areas, process ponds, certain mill and support facilities, and the underground workings subject to mine flooding.

The APP program requires that facilities be designed, constructed, and operated using Best Available Demonstrated Control Technology (BADCT) to achieve the greatest degree of discharge reduction practicable pursuant to A.R.S. § 49-243(B)(1). The permit application must include detailed engineering design, hydrogeologic and geochemical investigations defining the point of compliance pursuant to Arizona Administrative Code (A.A.C.) R18-9-A206, baseline groundwater quality characterization, predictive groundwater modelling, and a closure strategy with associated cost estimate pursuant to A.A.C. R18-9-A201 and R18-9-A209.

Closure under the APP program generally requires demonstration of clean closure of discharging facilities unless otherwise authorized. Clean closure requires removal of process liquids, sludges, and contaminated materials; proper characterization and disposal of residual materials; and elimination of future discharge potential. Financial assurance must be provided pursuant to A.R.S. § 49-243(N) to ensure third-party implementation of closure and post-closure monitoring obligations. Groundwater monitoring at established points of compliance must continue until aquifer conditions stabilize and no constituent exceeds established alert levels pursuant to A.A.C. R18-9-A210.

20.8.2 Status of Permit Applications

20.8.2.1 Bureau of Land Management - Plan of Operations

Although prior exploration activities have been authorized under a notice-level exploration filing or other temporary authorization, full mine development and operations cannot commence until the BLM has approved a comprehensive Plan of Operations, and the operator has posted an adequate reclamation financial guarantee in accordance with 43 CFR § 3809.500 and § 3809.552.

The Plan of Operations must describe all proposed underground and surface disturbances, including mine development, waste rock storage areas, the dry-stack tailings facility, process facilities, water management infrastructure, access roads, utilities, and ancillary structures. The Plan of Operations must also include a detailed reclamation and closure strategy, geochemical characterization data, water balance analysis, and a third-party reclamation cost estimate sufficient for the BLM to establish the appropriate financial assurance amount.

Approval of the Plan of Operations constitutes a major federal action subject to NEPA, 42 U.S.C. § 4321 et seq. For a project of this scale, preparation of an Environmental Impact Statement pursuant to 42 U.S.C.

§ 4332(2)(C) would be anticipated. The NEPA review process includes public scoping, preparation of draft and final environmental documents, interagency consultation, and issuance of a Record of Decision. Federal permitting timelines are therefore driven primarily by the scope and complexity of the NEPA analysis rather than by fixed statutory deadlines.

At the present conceptual design stage, no Plan of Operations has been developed or submitted to the BLM for consideration. A submittal of this nature is typically undertaken once engineering design has advanced sufficiently to define the footprint and technical specifications of all major facilities, generally at or beyond a preliminary engineering level adequate to support detailed environmental analysis and third-party reclamation cost estimation. Consequently, the current level of project design does not yet support initiation of the formal BLM permitting process or detailed reclamation cost estimation.

20.8.2.2 Arizona State Mine Inspector - Reclamation Plan

Although exploration activities previously conducted by Kay Mine are subject to an exploration-level Reclamation Plan, Kay Mine must submit and obtain approval for a Mined Land Reclamation Plan for privately held lands prior to initiating mining operations involving more than five (5) acres of disturbance, pursuant to A.R.S. § 27-921 and § 27-922. Unreclaimed disturbances from prior or ongoing exploration activities may be incorporated into the disturbance footprint of the operational Reclamation Plan or reclaimed under the existing exploration-level plan.

Future mining operations (on private lands) described in this document will require approval of a Mined Land Reclamation Plan as established under A.R.S. § 27-901 et seq. The plan should be developed once Kay Mine has largely completed design drawings for all surface disturbances and structures subject to ASMI jurisdiction.

The closure of discharging facilities as defined under APP rules, including tailings impoundments, process ponds, and waste rock stockpiles, must be included within the approved Reclamation Plan for purposes of grading, stabilization, and revegetation, even though detailed groundwater protection and clean closure requirements are regulated separately under the APP and approved by ADEQ. The current stage of project design is insufficient to support completion of a Reclamation Plan at this time.

20.8.2.3 Arizona Department of Environmental Quality - Aquifer Protection Permit

Future mining operations described in this document will eventually require approval of an individual APP pursuant to A.R.S. § 49-241 and § 49-243. The APP will regulate facilities that discharge or have the potential to discharge pollutants to the vadose zone or aquifer, including the dry-stack tailings facility, waste rock storage areas, process ponds, mill facilities, and underground workings subject to mine flooding.

Although ADEQ allows pre-application meetings and certain preliminary technical discussions at approximately 30 percent design, the APP application must ultimately include sufficiently advanced engineering designs, hydrogeologic characterization, geochemical data, and predictive groundwater modelling to demonstrate compliance with BADCT requirements under A.R.S. § 49-243(B)(1). In practice, formal APP submittal generally occurs once project design has advanced to approximately 75 percent completion for all regulated facilities. Accordingly, the current conceptual design stage does not yet support submittal of a complete APP application.

Closure of discharging facilities must be addressed within the APP application package, including a closure strategy and third-party cost estimate pursuant to A.R.S. § 49-243 and A.A.C. R18-9-A201. Although closure costs for discharging facilities must be identified in the APP, Arizona statutes prohibit duplicative bonding for the same closure elements under separate regulatory programs.

20.8.2.4 Known Requirements to Post Performance or Reclamation Bonds

Aside from the pending reclamation plan and associated financial assurance for exploration activities at the Site, Kay Mine currently has no obligations to post performance or reclamation bonds for full mine development and operations. However, prior to commencing mine construction or operations, the Kay Mine will be required to obtain approval of a Plan of Operations from the BLM pursuant to 43 CFR § 3809.11 and to post an adequate reclamation financial guarantee in accordance with 43 CFR § 3809.500 and § 3809.552. The BLM financial guarantee must be sufficient to cover the estimated cost for a third party to implement all reclamation, closure, and post-closure monitoring requirements described in the approved Plan of Operations, including earthwork, demolition, tailings and waste rock stabilization, water management infrastructure removal, and long-term site stabilization measures. The financial guarantee must be in place prior to surface disturbances associated with mine development.

Kay Mine must also obtain approval of a Mined Land Reclamation Plan from the ASMI pursuant to A.R.S. § 27-921 and § 27-922 and post a reclamation bond sufficient to secure surface grading, stabilization, revegetation, structural demolition, and portal closure activities on private lands. Arizona law prohibits duplicative bonding for closure components already secured under the APP; however, the operator must demonstrate that all reclamation and closure obligations are fully financially assured across all of the applicable regulatory programs.

In addition to federal financial assurance requirements, Kay Mine will be required to obtain and maintain an approved APP from the ADEQ pursuant to A.R.S. § 49-243 and to provide financial assurance sufficient to cover third-party implementation of closure and post-closure monitoring obligations for regulated discharging facilities pursuant to A.R.S. § 49-243(N). Accordingly, once the Project reaches a level of engineering design sufficient to support mine development and operations, Kay Mine will be required to submit and obtain approval of a BLM Plan of Operations, an ASMI-approved Reclamation Plan, and an ADEQ-approved APP, and to post the corresponding (and largely non-overlapping) financial assurance mechanisms prior to initiating construction or operational disturbance. Closure strategies and associated third-party cost estimates must be finalized, approved, and secured through appropriate financial assurance mechanisms before facility construction and operation may proceed.

20.8.3 Mine Closure Plans, Including Remediation and Reclamation Plans, and Associated Costs

G Mining provided Haley & Aldrich, Inc. (Haley & Aldrich) with a conceptual site layout and approximate dimensions of planned improvements on the property. Detailed design drawings or operational details are not available at this point in design, and the closure approach and estimated costs discussed in the following sections are therefore preliminary and based on the estimated closure costs outlined in PEAs for similar operations in the south-west United States.

The proposed location of site facilities is denoted in Kay Mine-Preliminary Economic Assessment- Regional Layout General Arrangement Plan View provided to Haley and Aldrich by G Mining.

Cost estimates for the closure of individual facilities were developed based on similar closure plans for mines in the southwestern United States and simply scaled based on the size of the respective facility. These similar closure plans are referenced in Table 21.12, where applicable.

20.8.3.1 Closure Components

20.8.3.1.1 Dry-Stack Tailings Facility Closure and Reclamation Approach

The approximate 10.1 million tonnes dry-stack tailings facility is planned to be located on BLM administered lands and will be subject to concurrent oversight by the BLM and ADEQ. Under 43 CFR § 3809.420 and § 3809.401, the BLM will require that the facility be designed, operated, and closed to prevent unnecessary or undue degradation of public lands and to achieve long-term static and seismic stability, erosion resistance, and protection of surface water and groundwater resources. Closure requirements will be incorporated into the approved Plan of Operations, and the Engineer of Record will be required to certify the design, construction, operation, and closure of the facility.

Under the APP program, ADEQ will evaluate geochemical characterization of the tailings to determine whether engineered infiltration controls are required to satisfy BADCT and groundwater protection standards pursuant to A.R.S. § 49-243. If tailings exhibit acid generation potential or metal leaching characteristics, an engineered cover system or other infiltration controls may be required. If geochemical testing demonstrates inert characteristics, closure may consist of regrading, stormwater control installation, placement of growth media, and revegetation, subject to regulatory approval. These geochemical studies have yet to be performed.

Closure costs for the dry-stack tailings facility are expected to include earthwork and regrading, potential buttress construction, cover system installation (if required), stormwater controls, growth media placement, revegetation, and long-term inspection and monitoring as required under the BLM approved Plan of Operations and the APP.

20.8.3.1.2 Underground Waste Rock and Development Rock Closure and Reclamation Approach

Underground waste rock and underground development rock will be characterized to determine acid generation and metal leaching potential. Under 43 CFR § 3809.420, the BLM will require that waste rock storage areas be reclaimed to achieve long-term physical and geochemical stability and prevent unnecessary or undue degradation. This includes demonstration of stability under static and seismic loading and implementation of drainage controls to minimize erosion and infiltration.

Closure cost estimates will incorporate earthwork, potential engineered controls, revegetation, monitoring, and long-term inspection obligations under all three (3) regulatory programs.

ADEQ will determine whether engineered controls are required to prevent exceedance of aquifer water quality standards at the point of compliance pursuant to A.R.S. § 49-243 and applicable APP rules. If reactive materials are identified, encapsulation, cover systems, or other BADCT measures may be required.

20.8.3.1.3 Mineral Processing Facility Closure and Reclamation Approach

The Mineral Processing facility will include crushing, grinding, flotation, and associated process support infrastructure. Closure will be implemented in accordance with the BLM-approved Plan of Operations, and the ADEQ-approved APP, as applicable.

Under 43 CFR § 3809.420, the BLM will require that the Mineral Processing facility be decommissioned and reclaimed in a manner that prevents unnecessary or undue degradation and achieves long-term site stability. Closure activities will include the removal of residual chemical or processing wastes, dismantling and removal of process equipment, piping, tanks, and ancillary infrastructure, unless retention is specifically approved for post-mining land use. Foundations can be removed or stabilized in place, as appropriate, to ensure long-term stability.

Under the APP program, ADEQ will require closure of process-related facilities in a manner that achieves clean closure unless otherwise approved. This will include removal of process solutions, reagents, concentrates, and residual materials, along with characterization and appropriate management of impacted materials. Confirmatory sampling will be required to demonstrate that no continued discharge to the vadose zone or aquifer will occur. Additional remedial measures may be required if closure performance standards are not achieved.

Closure cost components are expected to include demolition, equipment removal, material handling and disposal, foundation treatment, grading, growth media placement, revegetation, and confirmatory sampling and documentation. The extent of material impacts and remediation requirements remains uncertain at this stage and will be refined through future design and investigation.

20.8.3.1.4 Underground Workings Closure and Reclamation Approach

Closure of underground workings must address removal of fuels, chemicals, and hazardous materials; sealing of portals and shafts; and management of mine flooding. Under the BLM-approved Plan of Operations and 43 CFR § 3809.420, closure must ensure long-term public safety and prevent unnecessary or undue degradation, including the evaluation of subsidence risk and surface stability.

The BLM will require physical closure and stabilization of portals and shafts to ensure long-term public safety. Closure cost estimates will include portal sealing or backfilling, bulkheads if required, monitoring wells, and long-term groundwater monitoring as required under the APP and BLM authorization.

Predictive hydrologic and geochemical modelling will be required under the APP to evaluate rock-water interactions and potential groundwater impacts. If modelling indicates potential exceedance of alert levels at the point of compliance, corrective action requirements under A.A.C. R18-9-A210 would apply.

20.8.3.1.5 Mill, Process Areas, and Ponds Closure and Reclamation Approach

Under the BLM-approved Plan of Operations, surface facilities must be removed unless specifically approved for retention to support an identified post-mining land use. The BLM will require demolition, removal of hazardous materials, grading of foundations, and stabilization of disturbed areas to prevent unnecessary or undue degradation pursuant to 43 CFR § 3809.420.

The ASMI Reclamation Plan must address demolition and removal of structures not approved for post-mining use, regrading of disturbed areas, and revegetation consistent with post-mining land use objectives. Closure cost estimates must reflect demolition, waste disposal, grading, and revegetation in accordance with BLM, ADEQ, and ASMI requirements.

Process facilities and ponds must be decommissioned in accordance with APP closure requirements. Process liquids and sludges must be removed and properly characterized. Residual materials, including soils and liner systems, requiring off-site disposal must be managed in accordance with applicable waste regulations. Clean closure must be demonstrated prior to bond release under the APP.

20.8.3.1.6 Process and Chemical Ponds Closure and Reclamation Approach

Under the BLM Plan of Operations, closure of ponds must ensure elimination of discharge potential and long-term site stability, including regrading and stabilization of pond footprints to prevent erosion or ponding.

The ASMI-approved Reclamation Plan will address grading and stabilization of remaining surface depressions to achieve safe and stable post-mining conditions. These efforts are typically incorporated into the overall grading and Reclamation Plan.

The approved ADEQ APP will require that process ponds be drained and cleaned to remove remaining sludges and sediments. Liquids may be allowed to evaporate where permitted, but remaining sludges and sediments must be characterized and profiled for off-site transportation and disposal in accordance with applicable APP rules and regulations.

20.8.3.1.7 Structural Decommissioning Approach

The BLM will require removal of unnecessary surface structures, equipment, and materials unless specifically authorized for retention under the approved post-mining land use. Foundations must be removed or buried in a manner that ensures long-term stability and prevents environmental degradation pursuant to 43 CFR § 3809.420.

The ASMI-approved Reclamation Plan will address demolition and removal of surface facilities not specifically excluded from the plan. Certain structures, such as water wells, utility infrastructure, or buildings, may be retained if approved and compatible with post-mining land use. Any remaining surface depressions must be regraded to achieve safe and stable conditions. Inert materials generated from facility decommissioning may be buried on site if categorically inert or demonstrated to be inert through approved testing protocols.

The ADEQ-approved APP closure plan does not generally regulate structural demolition beyond ensuring that process liquids or residues are not discharged in an uncontrolled manner.

20.8.3.1.8 General Grading and Revegetation Approach

Under the BLM Plan of Operations, disturbed areas must be recontoured, stabilized, and revegetated to meet approved performance standards and prevent unnecessary or undue degradation of public lands pursuant to 43 CFR § 3809.420. Revegetation success criteria are typically performance-based and evaluated over multiple growing seasons.

The ASMI-approved Reclamation Plan will similarly address grading, site recontouring, and revegetation requirements. Roads and compacted areas must be ripped and scarified to promote revegetation. Stockpiles must be contoured to reduce erosion and enhance long-term stability. Inert materials generated during decommissioning may be buried on site if appropriately characterized.

There are typically no specific grading or revegetation standards included within an approved APP beyond stabilization efforts necessary to prevent long-term discharges to surface and groundwater.

20.8.3.1.9 Aquifer Restoration and Post-Closure Monitoring Approach

Under the BLM Plan of Operations, post-closure monitoring may include inspections of reclaimed landforms, stormwater controls, and tailings facilities to ensure continued stability and compliance with approved reclamation performance standards.

The ASMI-approved Reclamation Plan will require monitoring of revegetation success, erosion repair, fencing maintenance, and site condition verification prior to bond release.

Post-closure monitoring under the APP may include confirmation sampling related to clean closure of process areas and long-term groundwater monitoring at established points of compliance for a period of 25 years or longer. The Project will be required to maintain and sample monitoring wells until aquifer conditions stabilize and no constituent exceeds applicable alert levels. Monitoring may extend for 10 years or longer, depending on aquifer recovery and groundwater conditions following dewatering.

20.8.3.1.10 Post-Closure Monitoring and Bond Release

Final bond release under 43 CFR § 3809.590 will occur only after the BLM determines that all reclamation requirements have been completed, and long-term stability has been demonstrated in accordance with the approved Plan of Operations.

The ASMI will require monitoring of revegetation success and maintenance of reclaimed landforms prior to bond release pursuant to A.R.S. § 27-922.

Post-closure monitoring will be governed by the specific requirements of the APP, the ASMI Reclamation Plan, and the BLM approved Plan of Operations. ADEQ may require groundwater monitoring for an extended period beyond the completion of surface reclamation activities, depending on aquifer stabilization.

20.8.3.1.11 Current Project Status and Closure Cost Considerations

The Project is currently at a conceptual design stage. Detailed closure plans, engineering-level cost estimates, and final financial assurance amounts cannot be finalized until completion of advanced engineering design, geochemical characterization, hydrogeologic investigations, groundwater modelling, and regulatory consultation with the BLM, ADEQ and ASMI.

Closure cost estimates will be developed on a third-party implementation basis in accordance with 43 CFR § 3809.552, A.R.S. § 27-922, and § 49-243. Final bonding requirements will reflect the approved configuration of the underground workings, dry-stack tailings facility, waste rock storage areas, process facilities, and post-closure monitoring obligations.

21. CAPITAL AND OPERATING COSTS

The capital and operating cost estimates presented in this report are based on the preliminary design, engineering assumptions, and economic parameters established for the Kay Mine Project, located in Arizona, United States of America. These estimates have been developed to support the economic analysis of the Project and are consistent with the level of accuracy expected at the current stage of study, typically within the -30% +50% range for a Preliminary Economic Assessment (PEA).

Capital Costs include estimates for initial development, construction, infrastructure, equipment procurement, mine pre-production activities, indirect costs, contingency allowances, and owner's costs. These estimates are benchmarked against current market conditions, supplier quotations, and recent cost data from similar mining operations.

The base date of the capital expenditure (CAPEX) estimate is 2026 Q1, and the initial CAPEX duration is planned over a period of 30 months.

The initial capital cost estimate is presented in U.S. dollars (USD). Cost estimates were developed in the original (native) currency and, where required, converted to US dollars using an exchange rate of 1.34 CAD/USD.

The capital cost estimate is a built-up effort by the major facility and discipline. Each discipline performed estimation from the preliminary engineering and concepts that were developed for the Project. Unit costs from recently executed or estimated projects were used as reference benchmarks for this study. In addition, a budgetary price Request for Proposal (RFP) process was undertaken for certain cost elements.

Operating Costs have been developed on the basis of the proposed mine plan, anticipated throughput and process flow sheet. These costs encompass mining, processing, tailings management, site services, general and administrative (G&A) expenses, and sustaining capital. Labour rates, reagent consumption, fuel prices, and power tariffs reflect prevailing regional conditions and regulatory frameworks in Arizona.

The mining capital and operating cost estimates were developed by GMS to include the mine mobile equipment, i.e. primary, auxiliary, support and ancillary equipment, as well as pre-production mine development.

Mining infrastructures, namely haul roads, mine facilities, and processing plants, were developed by GMS.

The capital and operating cost estimates for the process plant were developed by GMS based on preliminary Process Design Criteria.

The dry stack tailings and overall site water management capital and operating cost estimates were developed by GMS.

The CAPEX estimate reflects an owner-managed project delivery model.

All the mining equipment purchase costs are captured in WBS (Work Breakdown Structure) Area 500. The equipment pricing includes the base machine with several required options, tires, fire suppression systems in most cases and assembly and commissioning when required.

Indirect costs consist of the labour costs for mine supervision, management, and technical support, as well as operating costs such as fuel, electricity, maintenance parts and consumables. Direct and indirect costs during pre-production were both captured in the CAPEX.

21.1 Initial Capital Expenditures

A summary of the capital expenditure is presented in Table 21.1.

Table 21.1: Capital Expenditures Summary

Capital Expenditures		k USD
100	Infrastructure	41,953
200	Power and Electrical	23,192
300	Water Management	20,619
400	Surface Operations	21,691
500	Mining	84,023
600	Process Plant	185,059
700	Construction Indirect	87,754
800	General Services	15,131
900	Pre-production and Commissioning	45,130
990	Contingency	84,126
Total		**608,678**

21.1.1 Infrastructure

A capital expenditures summary for infrastructure is presented in Table 21.2.

Table 21.2: Infrastructure Capital Expenditures

WBS	Infrastructure	k USD
110	Roads, Bridges and Fencing	7,617
120	Mine Infrastructure	11,827
130	Support Infrastructure	8,314
160	Process Plant Infrastructure	11,172
170	Fuel Systems Storage	1,356
180	Stockpile Pads	1,667
190	Offsite Facilities	
Total		**41,953**

21.1.2 Power Supply and Communications

Power and communications capital costs for the entire Project were combined within one general category, power and electrical, based on the average load installed in each area. The distribution per type of expenditure will be as presented in Table 21.3.

Table 21.3: Power Supply and Communications Capital Expenditures

WBS	Power & Electrical	k USD
210	Main Power Generation	9,800
220	Secondary Power Generation	1,100
230	Water Management Electrical Room	344
240	Service Electrical Room	177
250	Mine Electrical Room	2,477
260	Process Plant Electrical Rooms	4,242
270	Medium Voltage (MV) Distribution O/H Line	600
280	Automation Network	3,700
290	IT Network & Fire Detection	752
Total		**23,192**

21.1.3 Water Management

The water management capital cost distribution is presented in Table 21.4 and includes the following components: process plant water supply, mine water treatment plant, potable water treatment, sewage water management, fire protection, water ponds and overall water management, and the Tailings Storage Facility (TSF).

The TSF capital expenditure covers initial costs associated with pad preparation and water management pond construction, water return pipelines, as well as seepage collection ditches and sumps.

Table 21.4: Water Capital Expenditures

300	Water Management	k USD
310	Process Plant Water Supply	229
320	Mine Water Treatment Plant	11,974
330	Potable Water Treatment	951
340	Sewage Water	1,355
350	Fire Protection	966
360	Water Ponds and Water Management	1,701
370	Tailings Storage Facility	3,444
Total		**20,619**

21.1.4 Surface Operations

The capital costs estimate for surface equipment and batch plant were benchmarked from other recent projects as summarized in Table 21.5.

Table 21.5: Initial Surface Operations Capital Expenditures

400	Surface Operations	k USD
410	Surface Operations Equipment	20,896
430	Concrete Batch Plant	795
460	Borrow Pit	
470	Aggregate Quarry	
480	Aggregate Plant	
Total		**21,691**

21.1.5 Mining

The capital costs estimate for the mining areas include mining equipment, UG mine excavations and constructions. Costs are based on an owner-operated mine fleet.

A summary of the capital expenditures for mining is presented in Table 21.6.

Table 21.6: Initial Mining Capital Expenditures

500	Mining	k USD
510	Surface Mine Infrastructure	-
520	Surface Services infrastructure	-
530	UG Mine Excavation	35,771
540	UG Mine Infrastructure	4,261
550	UG Mobile Mine Equipment	37,276
560	UG Fixed Equipment & Mine Services	6,714
570	Backfill	-
580	Mine Shaft	-
590	OP Mine Equipment	-
Total		**84,023**

21.1.6 Process Plant and Related Infrastructure

Process plant capital costs were developed based on major equipment identified in the process design criteria. Costs for associated disciplines, such as civil, structural, piping, electrical, and instrumentation, were incorporated using factored estimates. The approach reflects the current study stage.

The capital cost estimates for the processing areas are presented in Table 21.7.

Table 21.7: Processing Capital Expenditures

600	Process Plant	k USD
601	Site Preparation / Road / Berms	9,310
610	Crushing	2,705
620	Grinding	4,758
630	Flotation and Regrind	25,614

600	Process Plant	k USD
640	Concentrate Handling	9,525
650	Albion & Lixiviation	65,251
660	Refinery	6,401
670	Reagents	4,580
680	Tailings Handling	6,951
690	Process Plant Services	49,964
Total		**185,059**

21.1.7 Construction Indirects

Indirect costs for the Project are estimated at $87.8M, inclusive of site management personnel, engineering, temporary facilities, construction tools and consumables, as well as fuel and energy consumption during the construction period. The estimate is based on factored allowances from direct costs and benchmark data from comparable projects. These costs reflect the current Project definition and exclude the owner's costs.

Construction Indirect Costs are presented in Table 21.8.

Table 21.8: Construction Indirect Capital

700	Construction Indirect	k USD
Total		**87,754**

21.1.8 General Services

General services and owners' costs were provided by Arizona Metals, except for the logistics cost (part of operating expenses). Logistics costs were calculated as a percentage of the material and equipment to be purchased.

Cost estimates are presented in Table 21.9.

Table 21.9: General Services Expenditures

800	General Services – Owner's Costs	k USD
810	G&A Departments	5,582
830	Operating Expenses	6,780

800	General Services – Owner's Costs	k USD
840	Environmental	218
850	Health and Safety	482
860	Site Insurance	2,069
Total		15,131

21.1.9 Pre-Production and Commissioning Expenditures

The development of the underground main decline and associated infrastructure to access the production zone accounts for the majority of pre-production expenditures, including processing costs incurred during mill preparation, personnel training, and commissioning.

Pre-production and commissioning expenditures are presented in Table 21.10.

Table 21.10: Pre-Production, Commissioning and Contingency Expenditures

900	General Services – Owner's Costs	k USD
910	Mining Pre-Prod.	25,400
950	Process Plant Pre-Prod.	6,471
960	First Fill, Spares & Consumables	13,258
990	Contingency	84,126
Total		129,256

Pre-Production mining costs will be incurred during the thirty (30) months prior to the start of commercial production; a tonnage of 108 kt of mineralized material and 859 kt of waste will be mined during this 30-month period.

Pre-production processing costs include an eight (8)-month period during which the processing workforce will be mobilized for training, commissioning, and ramp-up activities. The ramp-up period is expected to extend through the first nine (9) months of commercial production, during which process plant throughput is expected to increase progressively from approximately 55% of nameplate capacity to 100% design capacity.

The contingency accounts for uncertainties in scope, pricing fluctuations, design development, and potential construction risks. It was applied as a global factor, rather than itemized per discipline, to maintain consistency with industry-standard estimating practices. This allowance supports a realistic projection of total installed costs at the current stage of engineering. It was estimated at 25% of the direct costs.

21.2 Sustaining Capital

Sustaining capital for the mine includes additional equipment purchases and replacement for the underground operations, underground development and construction for an overall mining requirement of $87M. Equipment repairs that require replacing major components, such as engines, transmissions, and similar parts, are classified as sustaining capital.

Sustaining capital is presented in Table 21.11.

Table 21.11: Sustaining Capital Costs (k USD)

	TOTAL	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Total Sustaining Capital Costs	87,205	37,812	18,107	12,123	6,048	5,428	4,716	1,480	735	756	-

21.3 Closure Costs & Salvage Value

21.3.1 Closure Costs

The closure and long-term monitoring costs are estimated at $24,000,000 (USD) and $3,750,000 (USD), respectively, for a total closure and long-term monitoring cost of $27,750,000 (USD). The closure and long-term monitoring costs were based on engineering experience of closure and long-term monitoring costs for similar sites in the western United States. The low and high range estimates for the closure and long-term monitoring are summarized in Table 21.12. For the purpose of estimating closure costs, the salvage value of major plant equipment has been assumed to be zero. The estimated closure costs do not include any adjustment for net present value.

Closure costs would cover the closure of the following facilities:

- Tailings Storage Facility (TSF).
- Underground Mine.
- Mineral Processing Facility.
- Water Pond and Water Management.
- Fire Protection, Potable Water, Sewage Water, and Effluent Water Treatment.
- Warehouse, Canteen, and First Response Services.
- Workshop and Truck Shop.
- Laboratory, Reagents Storage and Mill Office.
- Fuel System Storage.
- Compressor Room.
- Electrical Substation.
- Explosives Storage.
- Crushed Mineralized Material Stockpile.
- Tailing Management.
- ROM Pad and Buried Services.

The closure site management and monitoring costs are estimated to be between $2.0M and $5.0M, carried out over 25 years of post-closure, assuming monitoring only and no perennial water treatment.

The total closure cost and monitoring post-closure are estimated to be between $11.7M and $36.7M.

Table 21.12: Closure Cost Summary

Facility	Basis / Typical Closure Tasks	Low Estimate k USDs	High Estimate k USD
Tailings Storage Facility	Regrading, cover / topsoil, seepage control, monitoring	3,000	10,000
Underground Mine	Portal sealing / bulkheads, underground waste rock, basic infrastructure removal	2,000	6,000
Mineral Processing Facility	Structural decommissioning and removal, residual treatment	1,100	1,300
Water Pond and Water Management	Ditch closure, liner removal, reshaping, short-term treatment	500	3,000
Fire Protection, Potable, Sewage and Effluent Water Treatment	System decommissioning, residual treatment	300	1,200
Warehouse, Canteen and First Response	Demolition, debris disposal, site restoration	150	800
Workshop and Truck Shop	Demolition, hazardous waste removal	300	1,000
Laboratory, Reagents Storage and Mill Office	Demo + hazmat handling	300	1,000
Fuel System Storage	Tank removal, soil remediation	250	1,000
Compressor Room	Demolition	100	400
Electrical Substation	Transformer removal, electrical grading	200	800
Explosives Storage	Magazine removal, safety compliance	150	500
Crushed Mineralized Material Stockpile	Regrading, erosion control	300	1,200
Tailing Management Systems	Pipelines, pumps removal	500	2,000
ROM Pad and Buried Services	Utility removal, grading	500	1,500
Subtotal Closure		**9,650**	**31,700**
Monitoring and Long-Term Water Treatment (25 yrs)	Sampling, lab analysis, periodic treatment	2,000	5,000
Total Closure and Monitoring Post-Closure		**11,650**	**36,700**

Kay Mine’s closure costs are primarily driven by a set of keys, interrelated factors. At the current conceptual stage, closure assumptions have generally been developed to reflect reasonable but conservative third-party implementation conditions, consistent with financial assurance expectations.

The most significant uncertainty is the duration of water treatment. Long-term management of Tailings Storage Facility (TSF) seepage, runoff, and underground water has the potential to extend for decades, depending on site-specific water quality and hydrogeologic conditions. At this stage, closure cost assumptions reflect a conservative treatment duration scenario, recognizing that ongoing site characterization, geochemical testing, and groundwater modelling may ultimately support a reduced treatment timeframe or allow for alternative closure strategies. As such, water management remains the dominant potential driver of post-closure financial liability.

The TSF closure method also has a major impact on closure costs. The Project is currently conceptualized as a dry-stack tailings facility, which represents a lower long-term risk profile relative to conventional slurry impoundments, particularly with respect to seepage, stability, and monitoring requirements. Closure assumptions are therefore aligned with a dry-stack configuration; however, they do not assume highly optimized performance beyond what can reasonably be supported without detailed geochemical and operational data. As a result, some conservatism remains in assumed earthwork quantities, stormwater controls, and monitoring requirements.

Potential acid rock drainage (ARD) and metal leaching represent another key uncertainty. In the absence of detailed geochemical characterization, closure assumptions incorporate a conservative allowance for potentially reactive materials, including the possibility of ARD generation. This approach is consistent with regulatory expectations under the Aquifer Protection Permit and the Bureau of Land Management (BLM) frameworks and avoids underestimating closure liabilities. Future testing may demonstrate more favourable geochemical conditions, which could reduce the need for engineered controls and long-term treatment.

Underground flooding management also influences closure cost and risk. Current assumptions reflect a controlled rebound scenario with monitoring and contingency provisions, rather than assuming either fully passive recovery or worst-case active treatment in perpetuity. This represents a balanced approach at the conceptual stage, with the understanding that predictive hydrogeologic modelling will be required to refine water level recovery rates and water quality outcomes.

Finally, regulatory financial assurance requirements imposed by the Arizona Department of Environmental Quality and the BLM require that closure costs reflect third-party implementation under reasonably conservative assumptions, including contingencies for uncertainty. The current closure cost framework is consistent with this requirement and does not rely on operator efficiencies or speculative performance improvements to reduce cost estimates.

Collectively, water treatment duration, TSF closure approach, ARD / metal leaching potential, underground flooding behaviour, and regulatory requirements represent the primary drivers and uncertainties shaping the Kay Mine's closure cost profile. While several aspects of the Project, such as the selection of dry-stack tailings, reflect inherently lower risk design choices, the overall cost estimate remains conservatively positioned pending additional engineering, geochemical, and hydrogeologic data that could support refinement and potential cost optimization.

21.3.2 Salvage Value

For the purpose of estimating closure costs, no credit has been taken for the salvage value of the major plant equipment; closure costs are therefore presented on a gross basis. The salvage value of the major plant equipment, estimated at USD 5 million, is recognized separately within the economic model and is not applied to reduce closure costs.

21.4 Operating Costs

The operating costs include mining, processing, general services and administration (G&A), royalties, concentrates transportation and refining, and power cost, which is included within each area. The average LOM direct operating cost is $138.47/t milled.

The power cost calculated with the rate sheet from the local utility is $0.0749/kWh.

Operating Costs are summarized in Table 21.13.

Table 21.13: Operating Costs Summary

Item	Total LOM Cost (M$)	Unit Cost ($/t milled)
Underground Mining	388	60.24
Processing	312	48.36
General Services & Administration	74	11.49
Total Direct Cost	**774**	**120.09**

Item	Total LOM Cost (M$)	Unit Cost ($/t milled)
Royalty	**-**	-
Transport and Refining	119	18.38
Total OPEX Cost	**893**	**138.47**

A summary of the total operating costs, including mining, milling, power, G&A and refining & transportation, as well as total cost per tonne milled, is presented in Table 21.14.

Table 21.14: Total Operating Costs Summary

Operating Cost Summary	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Underground Mining	M USD	388	38	45	41	46	39	43	43	38	35	22
Processing	M USD	312	26	33	33	33	33	33	33	33	32	20
General Services & Administration	M USD	74	8	8	8	8	8	8	8	8	7	6
Total Direct Cost	**M USD**	**774**	**72**	**86**	**82**	**87**	**80**	**84**	**84**	**79**	**74**	**48**
Royalty	M USD	-	-	-	-	-	-	-	-	-	-	-
Transport and Refining	M USD	119	8	12	12	13	13	14	13	14	13	6
Total OPEX Cost	**M USD**	**893**	**80**	**97**	**94**	**100**	**93**	**98**	**97**	**93**	**87**	**54**
Total OPEX Cost	t/milled	138.47	152.07	138.12	133.08	142.41	132.88	139.51	138.67	132.58	131.60	155.04

21.4.1 Mining Costs

A detailed mine cost build-up was developed from basic cost elements such as consumable prices, fuel prices, remuneration costs, and equipment productivities.

Equipment operating costs were determined from various sources, including primarily information from the major suppliers and benchmarked costs from operations in similar environments. Equipment operating costs were estimated for each equipment model, which includes operation and maintenance labour, parts (maintenance and repairs), fuel consumption, lubricant consumption, ground-engaging tools or tires if applicable.

A fuel price of $0.7331/L was considered for the purpose of this assessment.

The underground mining unit cost is $60.32/t MM mined. Table 21.15 presents the breakdown of mining costs by department.

Table 21.15: Underground Mining Cost Summary Total

Mining Costs (M USD)	**Total**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**
Mineralized Material Mined (Mt)	6.4	0.5	0.7	0.7	0.8	0.7	0.7	0.7	0.7	0.7	0.4
Total Diamond Drilling & Geology	10	1	1	1	1	1	1	1	1	1	1
Total Stope Preparation	56	9	9	7	9	4	5	5	3	3	2
Total Drilling & Blasting	27	2	3	3	3	3	3	3	3	3	2
Total Mucking & Hauling	58	3	5	6	6	7	7	7	7	7	3
Total Backfilling	55	4	6	5	7	5	7	7	6	5	3
Total Supervision	37	4	4	4	4	4	4	4	4	3	2
Total Mine Services	51	5	5	6	6	5	5	5	5	4	4
Total Surface Rehandling	8	1	1	1	1	1	1	1	1	1	0
Total Maintenance Services	17	2	2	2	2	2	2	2	2	2	1
Total Electrical Services	45	5	5	5	5	5	5	5	5	4	2
Total Technical Services	24.0	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.3	2.3	1.7
Total Mining Cost	**388**	**38**	**45**	**41**	**46**	**39**	**43**	**43**	**38**	**35**	**22**
Unit Cost ($/t MM Mined)	60.32	73.01	61.16	57.62	60.71	57.20	61.68	62.27	57.09	53.59	62.84

21.4.2 Processing Costs

The process flowsheet described in Section 17 is a conventional flotation and cyanidation circuit to treat refractory gold mineralized material. Processing cost will vary with the type of materials fed to the plant and their respective proportions. During operation, the average processing cost, including power cost, is $48.36/t milled.

The consumption estimates for grinding media and reagents (Table 21.16) are based on metallurgical test work results, modelling and historical data from operations in Eastern Canada. Pricing was compiled from 2025 supplier quotations and adjusted to reflect transportation costs to the Project site (Table 21.17). For certain minor reagents, historical prices were adjusted for inflation.

Labour costs were determined using wage rates and total compensation packages (including salary, benefits, and incentives), provided by the client.

Table 21.16: Grinding Media and Reagent Consumption

Consumables	Unit	Consumable Rate
Grinding Media & Liners		
Ball Mill Grinding Media	kg/t	0.37
Cu Regrind Mill Media	kg/t	0.35
Zn Regrind Mill Media	kg/t	0.54
Albion Regrind Mill Media	kg/t	0.20
Ball Mill Liners	Sets/y	0.4
Regrind Mill Liners	Sets/y	0.4
Albion regrind Mill Liners	Sets/y	0.7
Reagents - Flotation		
Collector Aero 5100	kg/t	0.18
Frother MIBC	kg/t	0.06
Calcium Oxide	kg/t	3.45
Sulfuric acid	kg/t	0.05
Copper Sulfate	kg/t	0.50
Zinc Sulfate	kg/t	0.55
Sodium Humate	kg/t	0.82
Sodium Cyanide	kg/t	0.10

Consumables	Unit	Consumable Rate
Flocculant	kg/t	0.05
Reagents - Albion		
Calcium Carbonate (Limestone)	kg/t	373.1
Caustic	kg/t	6.6
Flocculant	kg/t	0.03
Reagents - Leaching, Cyanide Detox		
Sodium Cyanide	Kg/t	0.4
Activated Carbon	kg/t	0.03
Lime	kg/t	1.02
Sodium Metabisulfite	kg/t	1.12
Copper Sulfate	kg/t	0.10
Lime	kg/t	0.11
Hydrochloric Acid	Kg/t	1.15
Flocculant	kg/t	0.02

Table 21.17: Consumables Operating Cost

Consumable Operating Cost $/t Milled		
Consumables	**Price ($/unit or set)**	**$/t**
Grinding Media & Liners		
Ball Mill Grinding Media	1,741	0.64
Cu Regrind Mill Media	3,693	1.28
Zn Regrind Mill Media	3,693	2.01
Albion Regrind Mill Media	3,693	0.74
Ball Mill Liners	232,100	0.13
Regrind Mill Liners	344,588	0.20
Albion Regrind Mill Liners	344,588	0.33
Reagents - Flotation		
Collector Aero 5100	4,431	0.80
Frother MIBC	2,743	0.18
Calcium Oxide	217	0.75
Sulfuric acid	74	0.01

Consumable Operating Cost $/t Milled		
Consumables	**Price ($/unit or set)**	**$/t**
Copper Sulfate	2,969	1.48
Zinc Sulfate	633	0.35
Sodium Humate	1,583	1.31
Sodium Cyanide	3,007	0.30
Flocculant	5,085	0.03
Reagents - Albion		
Calcium Carbonate (Limestone)	37	0.01
Caustic	791	0.01
Flocculant	4,220	0.01
Reagents - Leaching, Cyanide Detox		
Sodium Cyanide	2,565	0.78
Activated Carbon	2,957	0.09
Lime	185	0.19
Sodium Metabisulfite	405	0.45
Copper Sulfate	2,969	0.29
Lime	185	0.02
Hydrochloric Acid	315	0.13
Flocculant	4,338	0.07

The estimated electricity cost of $0.0749/kWh is based on benchmark data from similar mining projects in Ontario. It is used to evaluate operating expenses for site infrastructure and processing facilities.

Total yearly processing costs, excluding power, are presented in Table 21.18.

Table 21.18: Total Yearly Processing Costs

Processing Costs (M USD)	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Tonnes Milled (Mt)	6.4	0.5	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.4
Mill Labour	70	7	7	7	7	7	7	7	7	7	7
Metallurgical and Labs	6	0	1	1	1	1	1	1	1	1	0
Maintenance & Supplies	23	2	3	3	3	2	2	2	2	2	1
Reagent & Consumable	91	7	10	10	10	10	10	10	10	9	5
Power	111	9	12	12	12	12	12	12	12	11	6
Tailing & Water Management	10	1	1	1	1	1	1	1	1	1	1
Total Process Costs	312	26	33	33	33	33	33	33	33	32	20
Processing Cost ($/t milled)	48.36	50.38	47.46	47.46	47.46	47.53	47.53	47.53	47.54	48.23	57.64

21.4.3 General and Administration

General Services include general management, accounting and finance, IT, environmental and social management, human resources, supply chain, surface support, security, health and safety. In most cases, these services represent fixed costs for the site as a whole. The General Services costs exclude certain costs, such as refining costs and transportation, and environmental rehabilitation costs.

The average G&A operating costs over the life-of-mine are $11.49/t milled or an average of $8M per year.

21.4.4 Total Operating Costs

Total operating costs are presented on a yearly basis as shown in Table 21.19.

Table 21.19: Operating Cost Summary

Operating Cost Summary	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Production Highlights												
Tonnage Milled	kt	6,449	525	705	705	705	700	700	700	700	658	350
Tonnage Mined MM UG	kt	6,440	516	729	707	757	679	696	685	667	655	350
Tonnage Mined Waste UG	kt	1,309	430	418	256	64	8	33	55	20	19	6
Operating Costs												
Underground Mining	M US$	388	38	45	41	46	39	43	43	38	35	22
Processing	M US$	312	26	33	33	33	33	33	33	33	32	20
General & Administration	M US$	74	8	8	8	8	8	8	8	8	7	6
Total Direct Cost	M US$	774	72	86	82	87	80	84	84	79	74	48
Royalty Cost	M US$	-	-	-	-	-	-	-	-	-	-	-
Transport & Refining	M US$	119	8	12	12	13	13	14	13	14	13	6
Total OPEX Cost	M US$	893	80	97	94	100	93	98	97	93	87	54
Unit Operating Costs												
Underground Mining Cost	t/MM mined	60.32	73.01	61.16	57.62	60.71	57.20	61.68	62.27	57.09	53.59	62.84
Underground Mining Cost	t/milled	60.24	71.73	63.28	57.76	65.15	55.48	61.29	60.90	54.37	53.31	62.84
Processing Cost	t/milled	48.36	50.38	47.46	47.46	47.46	47.53	47.53	47.53	47.54	48.23	57.64
General & Administration Cost	t/milled	11.49	14.81	10.96	10.90	10.93	10.97	10.97	10.97	11.00	10.19	16.38
Total Direct Cost	t/milled	120.09	136.92	121.70	116.11	123.53	113.99	119.80	119.41	112.91	111.74	136.86
Royalty Cost	t/milled	-	-	-	-	-	-	-	-	-	-	-
Transport & Refining	t/milled	18.38	15.15	16.43	16.97	18.88	18.89	19.71	19.26	19.66	19.86	18.19
Total OPEX Cost	t/milled	138.47	152.07	138.12	133.08	142.41	132.88	139.51	138.67	132.58	131.60	155.04

22. ECONOMIC ANALYSES

The economic and financial analysis presented in this Preliminary Economic Assessment has been prepared using a discounted cash flow (DCF) methodology, evaluated on both a pre-tax and after-tax basis. The results of the economic evaluation are expressed in terms of Net Present Value (NPV), Internal Rate of Return (IRR), and payback period for the Kay Mine Project. Economic results are based on a 100% ownership basis. All economic figures are presented in real terms (i.e., excluding the effects of inflation) and are denominated in 2026 Q1 United States dollars (USD), unless otherwise indicated. The economic model excludes any Project debt or equipment financing.

The economic model projects annual cash flows over the life of the Kay Mine Project, aligned with the level of engineering and design appropriate for a Preliminary Economic Assessment (PEA). These projections are based on estimates for sales revenue, operating costs (OPEX), capital expenditure (CAPEX) and other costs. CAPEX is categorized into three (3) components: initial capital, sustaining capital, closure and reclamation costs. OPEX includes expenses related to labour, reagents, maintenance, supplies, services, fuel and power. Additional costs, such as depreciation and taxes, are calculated in accordance with the current mine and processing plans.

The Preliminary Economic Assessment (PEA) is preliminary in nature and includes Inferred Mineral Resources. Mineral Resources are considered to be too geologically speculative to have economic considerations applied to them in a manner which would allow for classification as Mineral Reserves. As such, there is no assurance that the outcomes projected in the PEA will be realized.

22.1 Cautionary Statements

The results of the economic analyses discussed in this section represent forward-looking information as defined under the Canadian securities law. These results are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

The forward-looking information includes, but is not limited to, the following:

- Currency exchange rate fluctuation.
- Assumed prices for valued metals.
- Cost inflation.
- Unexpected variations in the amount of mineralized material and material grade.
- Geotechnical or hydrogeological considerations during mining that differ from the assumptions.

- The proposed mine production plan.
- Assumptions regarding mining dilution and mining recovery.
- The recovery rates of valued metals in the processing plant.
- Proposed sustaining and operating costs.
- Failure of plant, equipment, and processes to operate as anticipated.
- Assumptions regarding closure costs.
- Assumptions regarding environmental, social and licensing risks.
- Labour and materials availability.
- Labour and materials costs being approximately consistent with the assumptions in the report.
- Ability to maintain social licence to operate.
- Unrecognized environmental risks.
- Unforeseen reclamation expenses.
- Changes to tax rates.

22.2 Assumptions

22.2.1 Metals Price

The determination of valued metal prices is described in Section 19. The long-term gold price assumption used in the base case scenario for this assessment is copper priced at $4.70/lb Cu, zinc priced at $1.27/lb Zn, gold priced at $3,100/oz Au, and silver priced at $38.00/oz Ag.

22.2.2 Exchange Rate

Cost estimates were developed in the original (native) currency and, where required, converted to US dollars using an exchange rate of 1.34 CAD/USD.

22.2.3 Other Assumptions

The other key assumptions used in economic analysis are as follows:

- Discount rate 5%.
- Cashflow discounted at the start of construction.

- All cost estimates are in constant Q1 2026 United States dollars with no inflation or escalation factors considered.

22.3 Metal Production and Revenues

Over the life of the Project, three (3) concentrates are expected to be produced, containing payable copper, zinc, gold, and silver. Based on assumed metallurgical recoveries and payability factors, total payable metal production over the life of mine is estimated at 127 Mlbs of copper, 293 Mlbs of zinc, 258 koz of gold, and 4,712 koz of silver. During the pre-production phase, payable quantities of copper, zinc, gold, and silver are expected to generate an estimated net revenue of $18 million (net of penalties, transportation, refining, and royalty costs). Over the operating period, additional payable copper, zinc, gold, and silver are anticipated, generating an estimated net revenue of $1,812 million. (net of penalties, transportation, refining, and royalty costs).

Table 22.1 presents the LOM Mill Production Schedule Summary. Figure 22.1 and Figure 22.2 present mill feed grade and Payable metal production.

Table 22.1: Milling Production Schedule Summary

Milling Schedule	Unit	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Tonnage Milled	kt	6,548	-	-	99	525	705	705	705	700	700	700	700	658	350
Copper Head Grade	%	1.01	-	-	0.76	0.72	0.87	0.96	1.14	1.09	1.18	1.21	1.07	0.94	0.79
Zinc Head Grade	%	2.67	-	-	1.72	2.51	2.50	2.42	2.53	2.65	2.63	2.46	2.90	3.32	3.24
Gold Head Grade	g/t	1.60	-	-	0.93	1.03	1.14	1.19	1.42	1.75	1.91	1.66	2.02	2.14	1.87
Silver Head Grade	g/t	29.07	-	-	20.45	31.12	33.50	29.81	27.01	26.43	29.07	25.05	32.08	31.39	25.06
Contained Copper	Mlbs	146	-	-	2	8	13	15	18	17	18	19	17	14	6
Contained Zinc	Mlbs	386	-	-	4	29	39	38	39	41	41	38	45	48	25
Contained Gold	koz	337	-	-	3	17	26	27	32	39	43	37	45	45	21
Contained Silver	koz	6,119	-	-	65	525	759	676	612	595	654	564	722	664	282
Average Copper Recovery	%	92%	-	-	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%
Average Zinc Recovery	%	80%	-	-	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Average Gold Recovery	%	86%	-	-	86%	86%	86%	86%	86%	86%	86%	86%	86%	86%	86%
Average Silver Recovery	%	85%	-	-	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Recovered Copper	Mlbs	134	-	-	2	8	12	14	16	15	17	17	15	13	6
Recovered Zinc	Mlbs	309	-	-	3	23	31	30	31	33	32	30	36	39	20
Recovered Gold	koz	290	-	-	3	15	22	23	28	34	37	32	39	39	18
Recovered Silver	koz	5,201	-	-	55	446	645	574	520	506	556	479	614	565	240
Payable Copper	Mlbs	127	-	-	1	7	12	13	15	15	16	16	14	12	5
Payable Zinc	Mlbs	293	-	-	3	22	29	29	30	31	31	29	34	37	19
Payable Gold	koz	258	-	-	2	13	20	20	24	30	33	28	35	35	16
Payable Silver	koz	4,712	-	-	50	410	591	522	467	455	501	427	558	514	218

Figure 22.1: Mill Production Schedule – Feed Grade

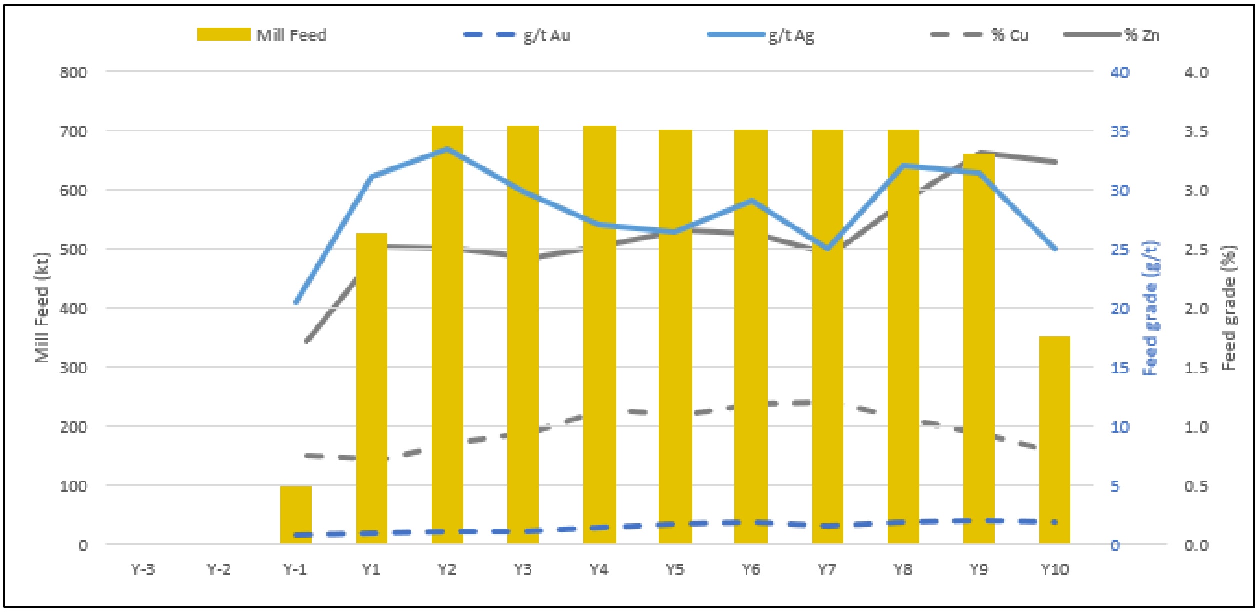
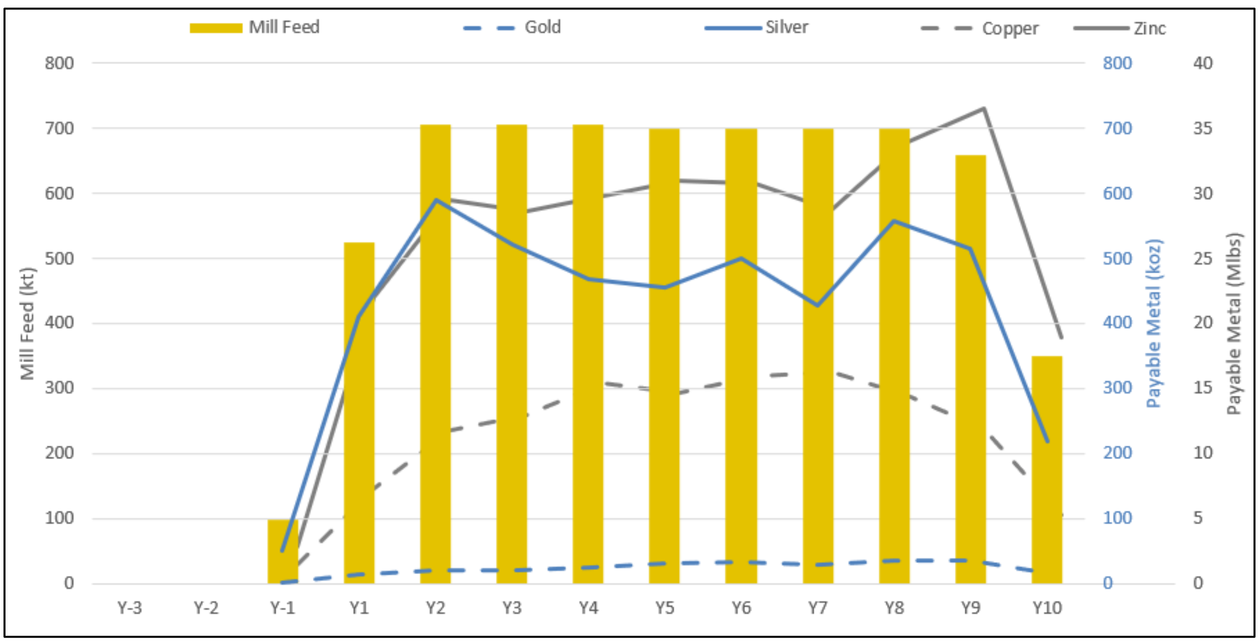


Figure 22.2: Mill Production Schedule – Payable Metal



22.4 <u>Capital Expenditures</u>

The capital expenditures include initial capital expenditures as well as the sustaining capital expenditures to be spent after commencement of commercial operations.

22.5 Initial Capital

The initial CAPEX for Project construction, including processing facilities, mine equipment purchases, pre-production activities, infrastructures and other direct and indirect costs, is estimated to be $609 million before by-product metal credits from pre-production concentrate sales and doré production. The total initial CAPEX includes a contingency of $84 million (excl. working capital adjustment).

22.6 Sustaining Capital

Sustaining capital is required during operations. For the underground mining, the sustaining included underground capitalized development, equipment purchases and replacement, and major repairs. The sustaining capital is estimated at $87 million.

22.7 Working Capital

Working capital requirements were estimated based on 30-day accounts receivable, 30-day accounts payable and other current liabilities. Working Capital during construction is estimated at USD 19 million, and in operation is expected to fluctuate over the life-of-mine, where the remaining balance is credited at the end of the life-of-mine.

22.8 Closure Cost and Salvage Value

Reclamation and closure costs include infrastructure decommissioning, site preparation and revegetation, as well as maintenance and post-closure monitoring for twenty-five (25) years after operations. Closure costs are estimated to be a total of USD 28 million. In addition, an annual surety bond has been included as financial assurance, estimated at USD 7 million, bringing the total closure cost to USD 35 million.

A salvage value is estimated for the major process plant equipment. The salvage value is estimated at $5.34 million.

22.9 Operating Cost Summary

The operating costs include mining, processing, general services and administration (G&A), concentrates transportation and refining, and power cost, which is included within each area. Operating cost summaries are presented by year in Table 22.2 and Table 22.3. The average LOM operating cost is $138.47/t. milled.

Table 22.2: Operating Cost Summary

Operating Cost Summary	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Production Highlights (excl. Pre-Prod.)												
Tonnage Milled	kt	6,449	525	705	705	705	700	700	700	700	658	350
Tonnage Mineralized Material Mined UG	kt	6,440	516	729	707	757	679	696	685	667	655	350
Tonnage Mined Waste UG	kt	1,309	430	418	256	64	8	33	55	20	19	6
Payable Copper	Mlbs	126	7	12	13	15	15	16	16	14	12	5
Payable Zinc	Mlbs	290	22	29	59	30	31	31	29	34	37	19
Payable Gold	koz	256	13	20	20	24	30	33	28	35	35	17
Payable Silver	koz	4,662	410	591	522	467	455	501	427	558	514	218
Operating Costs (excl. Pre-Prod.)												
Underground Mining	M USD	388	38	45	41	46	39	43	43	38	35	22
Processing	M USD	312	26	33	33	33	33	33	33	33	32	20
General & Administration	M USD	74	8	8	8	8	8	8	8	8	7	6
Total Direct Cost	**M USD**	**774**	**72**	**86**	**82**	**87**	**80**	**84**	**84**	**79**	**74**	**48**
Royalty Cost	M USD	-	-	-	-	-	-	-	-	-	-	-
Transport & Refining	M USD	119	8	12	12	13	13	14	13	14	13	6
Total OPEX Cost	**M USD**	**893**	**80**	**97**	**94**	**100**	**93**	**98**	**97**	**93**	**87**	**54**

Table 22.3: Operating Cost Summary per Tonne

Operating Costs per Tonnes Mined		Average	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Underground Mining Cost	t/MM	60.32	73.01	61.16	57.62	60.71	57.20	61.68	62.27	57.09	53.59	62.84
Operating Costs per Tonnes Milled												
Underground Mining Cost	t/milled	60.24	71.73	63.28	57.76	65.15	55.48	61.29	60.90	54.37	53.31	62.84
Processing Cost	t/milled	48.36	50.38	47.46	47.46	47.46	47.53	47.53	47.53	47.54	48.23	57.64
General & Administration Cost	t/milled	11.49	14.81	10.96	10.90	10.93	10.97	10.97	10.97	11.00	10.19	16.38
Total Direct Cost	**t/milled**	120.09	136.92	121.70	116.11	123.53	113.99	119.80	119.41	112.91	111.74	136.86
Royalty Cost	t/milled	-	-	-	-	-	-	-	-	-	-	-
Transport & Refining	t/milled	18.38	15.15	16.43	16.97	18.88	18.89	19.71	19.26	19.66	19.86	18.19
Total OPEX Cost	**t/milled**	138.47	152.07	138.12	133.08	142.41	132.88	139.51	138.67	132.58	131.60	155.04

22.10 Royalties

The Kay Mine is not subject to any royalties payable to third parties.

22.11 Taxation

McGovern Hurley prepared the tax model used for the economic analysis, as described in Section 3. The Project is subject to three (3) levels of taxation: State mining tax, State income tax and Federal income tax. The Kay Mine Project will pay approximately USD 73 million in tax payments over the life-of-mine.

22.12 Economics

The main economic metrics used to evaluate the Project consist of net undiscounted after-tax cash flow, net discounted after-tax cash flow or NPV, IRR and payback period. A 5% discount rate was applied to the cash flow to derive the NPV for the Project on a pre-tax and after-tax basis.

A summary of the Project's economic results is presented in Table 22.4. The total after-tax cash flow over the Project life is $259M, and the NPV 5% is $40M pre-tax and $-6M after-tax. The after-tax Project cash flow results in a 7.5-year payback period from the commencement of commercial operations with an IRR of 6.0% pre-tax and 4.9% after-tax.

Table 22.4: Project Economic Results Summary

Assumptions	Unit	Base Case
Copper Price	$/lb	4.70
Zinc Price	$/lb	1.27
Gold Price	$/oz	3,100
Silver Price	$/oz	38.00
Exchange Rate	USD : CAD	1.34
Fuel Price	$/L	0.73
Mine Life	yrs	10
Production Summary (Life-of-Mine)		
Underground		
Mineralized Material Mined	Mt	6.55
Copper Grade	%	1.01
Zinc Grade	%	2.67

Assumptions	Unit	Base Case
Gold Grade	g/t	1.60
Silver Grade	g/t	29.07
Mill Feed		
Average Milling Throughput	Mtpa	0.7
Average Daily Throughput	tpd	1,918
Total Mill Feed Tonnes	Mt	6.55
Copper Head Grade	%	1.01
Zinc Head Grade	%	2.67
Gold Head Grade	g/t	1.60
Silver Head Grade	g/t	29.07
Contained Copper	Mlbs	146.1
Contained Zinc	Mlbs	385.9
Contained Gold	koz	336.8
Contained Silver	koz	6,119.2
Average Copper Recovery (%)	%	92
Average Zinc Recovery (%)	%	80
Average Gold Recovery (%)	%	86
Average Silver Recovery (%)	%	85
Recovered Copper	Mlbs	134.4
Recovered Zinc	Mlbs	308.7
Recovered Gold	koz	289.7
Recovered Silver	koz	5,201.3
Payable Copper	Mlbs	127.4
Payable Zinc	Mlbs	292.7
Payable Gold	koz	258.1
Payable Silver	koz	4,712.2
Operating Costs (LOM average)		
Underground Mining Cost	$/t milled	60.24
Processing Cost	$/t milled	48.36
General & Administration Cost	$/t milled	11.49
Total Direct Cost	**$/t milled**	**120.09**

Assumptions	Unit	Base Case
Royalty Cost	$/t milled	-
Transport & Refining	$/t milled	18.38
Total OPEX Cost	**$/t milled**	**138.47**
Capital Costs		
Initial Capital	M$	609
Sustaining Capital	M$	**87**
Closure Costs	M$	35
Total Capital Cost	M$	**731**
Construction Working Capital	M$	19
Salvage Value	M$	5
Financial Evaluation Pre-Tax		
Free Cash Flow	M$	332
Pre-Tax NPV 5%	M$	40
Pre-Tax IRR	%	6.0
Payback	yrs	7.3
Financial Evaluation After-Tax		
Free Cash Flow	M$	259
After-Tax NPV 5%	M$	-6
After-Tax IRR	%	4.9
Payback	yrs	7.5

Table 22.5 presents the Base Case economic results, the annual Project cash flows and production.

Table 22.5: Base Case Economic Results, Annual Project Cash Flows and Production

Cash Flow	Unit	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Copper Price	$/lbs	4.70	4.70	4.70	4.70	4.70	4.70	4.70	4.70	4.70	4.70	4.70	4.70	4.70	4.70	4.70
Zinc Price	$/lbs	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27
Gold Price	$/oz	3,100	3,100	3,100	3,100	3,100	3,100	3,100	3,100	3,100	3,100	3,100	3,100	3,100	3,100	3,100
Silver Price	$/oz	38.00	38.00	38.00	38.00	38.00	38.00	38.00	38.00	38.00	38.00	38.00	38.00	38.00	38.00	38.00
Exchange Rate	USD-CAD	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34
Payable Copper	Mlbs	127	-	-	1	7	12	13	15	15	16	16	14	12	5	-
Payable Zinc	Mlbs	293	-	-	3	22	29	29	30	31	31	29	34	37	19	-
Payable Gold	koz	258	-	-	2	13	20	20	24	30	33	28	35	35	16	-
Payable Silver	koz	4,712	-	-	50	410	591	522	467	455	501	427	558	514	218	-
Gross Revenue	M$	1,950	-	-	19	119	176	180	203	219	235	217	241	231	108	-
Underground Mining	M$	414	3	6	16	38	45	41	46	39	43	43	38	35	22	-
Processing	M$	318	0	0	6	26	33	33	33	33	33	33	33	32	20	-
General & Administration	M$	82	2	3	4	8	8	8	8	8	8	8	8	7	6	-
Royalty	M$	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Transport & Refining	M$	120	-	-	1	8	12	12	13	13	14	13	14	13	6	-
Transfer to CAPEX	M$	(41)	(4)	(9)	(28)	-	-	-	-	-	-	-	-	-	-	-
Total OPEX Cost	M$	893	-	-	-	80	97	94	100	93	98	97	93	87	54	-
EBITDA	M$	1,057	-	-	19	39	79	86	103	126	138	120	148	144	54	-
Initial CAPEX	M$	(609)	(59)	(255)	(294)	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	M$	(87)	-	-	-	(38)	(18)	(12)	(6)	(5)	(5)	(1)	(1)	(1)	-	-
Closure Cost & Monitoring	M$	(35)	(1)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	
Salvage Value	M$	5	-	-	-	-	-	-	-	-	-	-	-	-	5	-
Working Capital	M$	-	(4)	(4)	(11)	(7)	0	(1)	(2)	(2)	(1)	1	(2)	1	9	23
Pre-Tax Cash Flow	M$	332	(64)	(262)	(289)	(9)	58	71	92	116	129	118	143	141	65	23
Taxes	M$	(73)	-	-	(0)	(1)	(3)	(4)	(4)	(5)	(5)	(5)	(11)	(26)	(9)	-
After-Tax Cash Flow	M$	259	(64)	(262)	(290)	(10)	55	67	88	112	124	113	131	116	57	23

22.13 Sensitivity Analysis

A sensitivity analysis was carried out with the base case as described above. An interval of ± 30% versus base case values was considered with increments of 10%. The sensitivity analysis was carried out on the metal price, the OPEX and the Initial CAPEX. The Project's financial performance is most sensitive to the metal price and much less to the operating costs and initial capital expenditures. The results of the sensitivity analysis of the Project in terms of NPV, IRR and Payback are summarized in Table 22.6, Table 22.7 and Table 22.8Table 22.8. Figure 22.3 to Figure 22.6**Erreur! Source du renvoi introuvable.** show the sensitivity of the After-Tax Total Cashflow, NPV 5%, IRR and Payback.

Table 22.6: Gold Price Sensitivity

Metal Price	Pre-Tax				After-Tax			
	NPV 0% (M USD)	NPV 5% (M USD)	IRR	Payback Period (yrs)	NPV 0% (M USD)	NPV 5% (M USD)	IRR (%)	Payback Period (yr)
70%	-253	-239	-6.0%	-	-269	-369	-6.5%	-
80%	-58	-226	-1.3%	-	-81	-241	-1.8%	-
90%	137	-93	2.6%	8.5	105	-114	2.0%	8.7
Base Case	332	40	6.0%	7.3	259	-6	4.9%	7.5
110%	527	173	9.0%	6.3	405	97	7.4%	6.6
120%	722	306	11.7%	5.6	551	198	9.7%	5.9
130%	917	439	14.3%	5.1	696	299	11.9%	5.3

Table 22.7: OPEX Sensitivity

OPEX	Pre-Tax				After-Tax			
	NPV 0% (M USD)	NPV 5% (M USD)	IRR	Payback Period (yr)	NPV 0% (M USD)	NPV 5% (M USD)	IRR (%)	Payback Period (yr)
70%	556	193	9.4%	6.1	427	113	7.8%	6.4
80%	480	142	8.3%	6.5	371	73	6.8%	6.8
90%	406	90	7.1%	6.9	315	34	5.8%	7.1
Base Case	332	40	6.0%	7.3	259	-6	4.9%	7.5
110%	258	-10	4.8%	7.7	203	-45	3.9%	7.9

OPEX	Pre-Tax				After-Tax			
	NPV 0% (M USD)	NPV 5% (M USD)	IRR	Payback Period (yr)	NPV 0% (M USD)	NPV 5% (M USD)	IRR (%)	Payback Period (yr)
120%	187	-59	3.5%	8.1	148	-85	2.8%	8.4
130%	115	-108	2.2%	8.7	84	-128	1.6%	8.9

Table 22.8: Initial CAPEX Sensitivity

Initial CAPEX	Pre-Tax				After-Tax			
	NPV 0% (M USD)	NPV 5% (M USD)	IRR	Payback Period (yr)	NPV 0% (M USD)	NPV 5% (M USD)	IRR	Payback Period (yr)
70%	514	211	11.3%	5.8	442	165	10.1%	6.0
80%	454	154	9.2%	6.3	381	108	8.1%	6.5
90%	393	97	7.5%	6.8	320	51	6.4%	7.0
Base Case	332	40	6.0%	7.3	259	-6	4.9%	7.5
110%	271	-17	4.6%	7.7	198	-63	3.5%	8.0
120%	210	-74	3.4%	8.1	137	-120	2.3%	8.5
130%	149	-131	2.3%	8.6	76	-177	1.2%	9.1

Figure 22.3: After-Tax Total Cash Flow Sensitivity (M$)

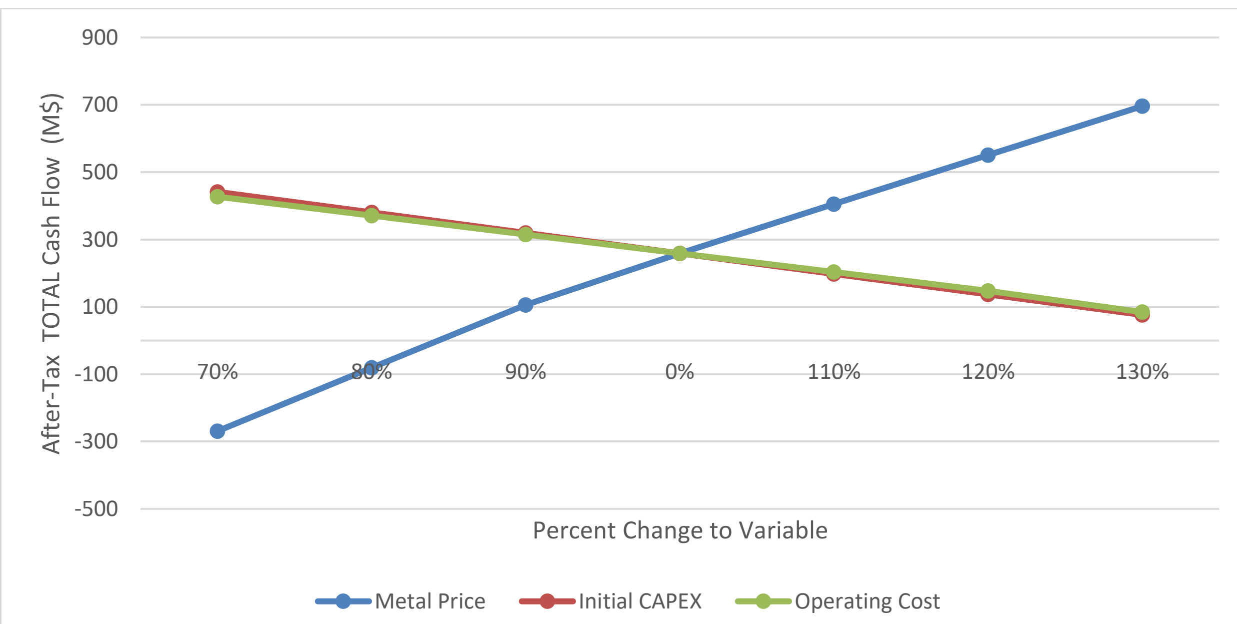


Figure 22.4: After-Tax NPV (5%) Sensitivity (M$)

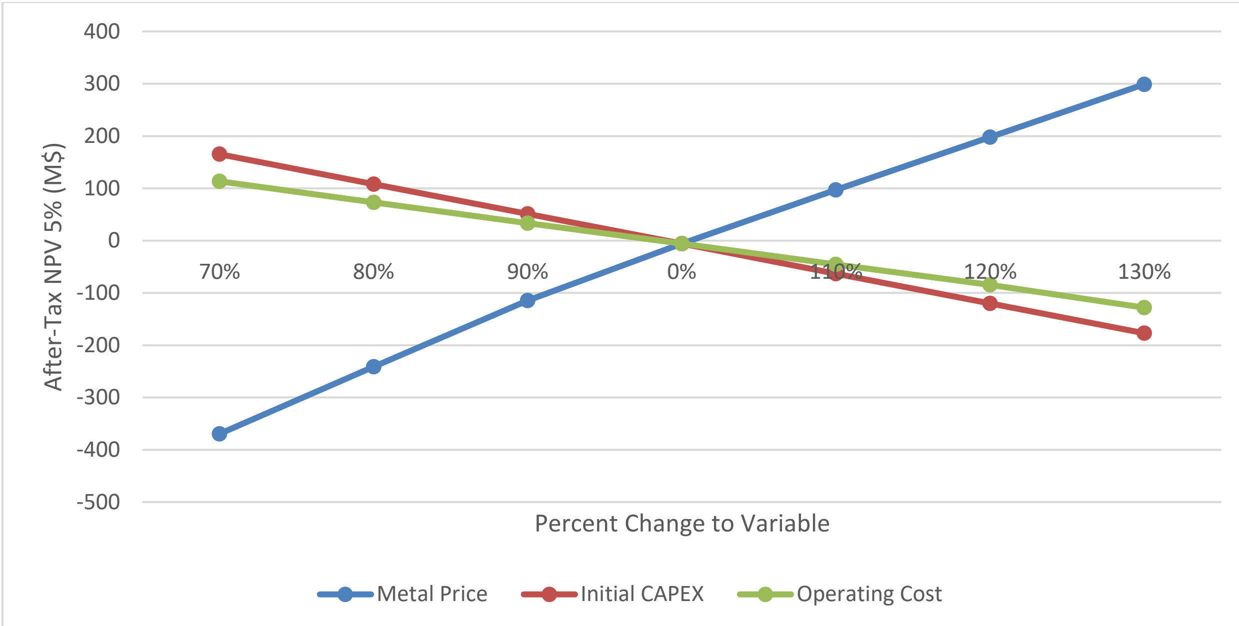


Figure 22.5: After-Tax Internal Rate of Return Sensitivity

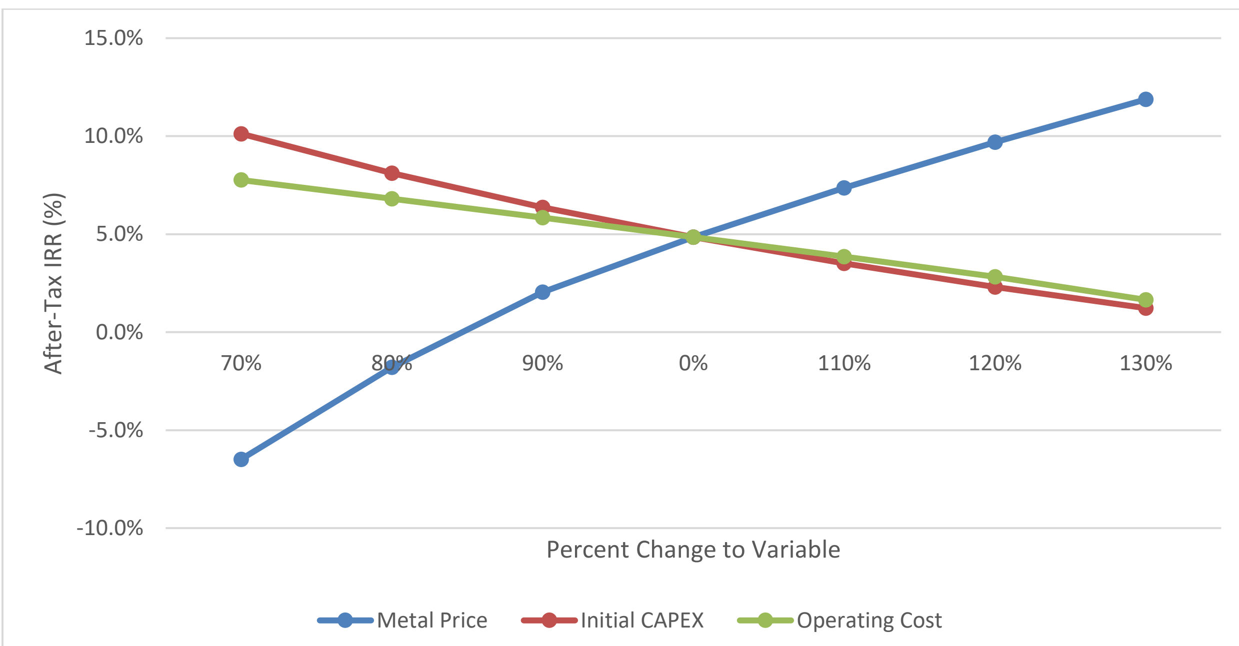
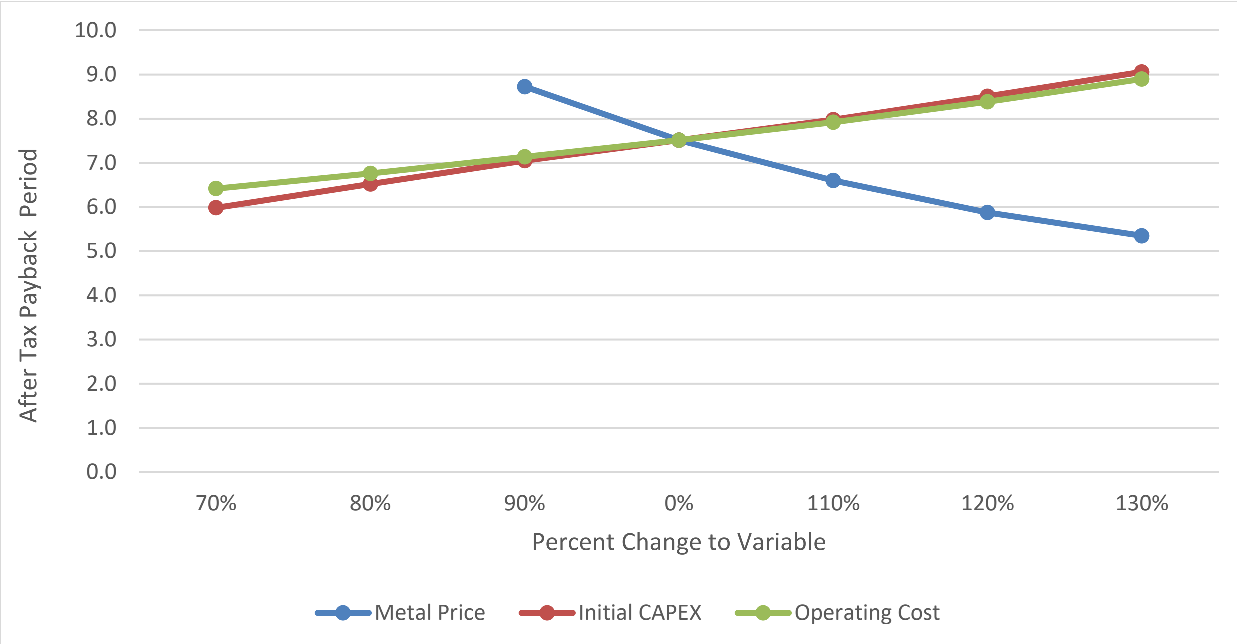


Figure 22.6: After-Tax Payback Period Sensitivity



23. ADJACENT PROPERTIES

There is no information on properties adjacent to the Property necessary to make the Technical Report understandable and not misleading.

24. OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information available that is necessary to make the Technical Report understandable and not misleading. To the Authors' knowledge, there are no significant risks and uncertainties that could reasonably be expected to affect the reliability or confidence in the exploration information or MRE.

25. INTERPRETATION AND CONCLUSIONS

25.1 Summary

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this PEA Report is the evaluation of the potential technical feasibility and potential economic viability of the Project, notably the development of an underground mine, including processing facilities and related infrastructures.

This NI 43-101 Technical Report indicates that the Project demonstrates potential technical feasibility; however, it does not yet support robust economic viability under current assumptions. Additional work will be required to strengthen the resource base and improve Project economics before advancing toward a future Pre-Feasibility Study, including expanded drilling to increase the overall mineralized inventory, as well as step-out and deeper exploration to identify extensions of higher-grade zones. Continued and expanded metallurgical test work, including variability testing, process optimization, and evaluation of alternative recovery methods, is also required to enhance recoveries, improve concentrate quality, and reduce processing costs.

Table 25.1 shows the Preliminary Economic Assessment.

Table 25.1: Preliminary Economic Assessment

Assumptions	Unit	Base Case
Copper Price	$/lb	4.70
Zinc Price	$/lb	1.27
Gold Price	$/oz	3,100
Silver Price	$/oz	38.00
Exchange Rate	USD : CAD	1.34
Fuel Price	$/L	0.73
Mine Life	yrs	10
Production Summary (Life-of-Mine)		
Underground	**Unit**	**Base Case**
Mineralized Material Mined	Mt	6.55
Copper Grade	%	1.01
Zinc Grade	%	2.67

Assumptions	Unit	Base Case
Gold Grade	g/t	1.60
Silver Grade	g/t	29.07
Mill Feed	**Unit**	**Base Case**
Average Milling Throughput	Mtpa	0.7
Average Daily Throughput	tpd	1,918
Total Mill Feed Tonnes	Mt	6.55
Copper Head Grade	%	1.01
Zinc Head Grade	%	2.67
Gold Head Grade	g/t	1.60
Silver Head Grade	g/t	29.07
Contained Copper	Mlbs	146.1
Contained Zinc	Mlbs	385.9
Contained Gold	koz	336.8
Contained Silver	koz	6,119.2
Average Copper Recovery (%)	%	92
Average Zinc Recovery (%)	%	80
Average Gold Recovery (%)	%	86
Average Silver Recovery (%)	%	85
Payable Copper	Mlbs	127.4
Payable Zinc	Mlbs	292.7
Payable Gold	koz	258.1
Payable Silver	koz	4,712.2
Operating Costs (LOM average)		
Underground Mining Cost	$/t milled	60.24
Processing Cost	$/t milled	48.36
General & Administration Cost	$/t milled	11.49
Total Direct Cost	***$/t milled***	***120.09***
Royalty Cost	$/t milled	-
Transport & Refining	$/t milled	18.38
Total OPEX Cost	***$/t milled***	***138.47***

Assumptions	Unit	Base Case
Capital Costs		
Initial Capital Costs	M$	609
Sustaining Capital	M$	**87**
Closure Costs	M$	35
Total Capital Cost	***M$***	***731***
Construction Working Capital	M$	19
Salvage Value	M$	5
Financial Evaluation Pre-Tax		
Free Cash Flow	M$	332
Pre-Tax NPV 5%	M$	40
Pre-Tax IRR	%	6.0
Payback	yrs	7.3
Financial Evaluation After-Tax		
Free Cash Flow	M$	259
After-Tax NPV 5%	M$	-6
After-Tax IRR	%	4.9
Payback	yrs	7.5

25.2 Mineral Resource Estimate

Completion of the current MRE involved the assessment of a drill hole database, which included all data for surface drilling completed through the end of May 2025. Completion of the current MRE also included updated three-dimensional mineral resource models (resource domains), a 3D topographic surface model, 3D models of historical underground workings, and available written reports. The Inverse Distance Squared calculation method restricted to mineralized domains was used to interpolate grades for Au (g/t), Ag (g/t), Cu (ppm), Pb (ppm) and Zn (ppm) into a block model for the Kay Deposit. The MRE for the Kay Deposit takes into consideration that the Kay Deposit may be mined by underground mining methods.

The reporting of the current MRE complies with all disclosure requirements for Mineral Resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects. The classification of the MRE is consistent with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definitions). In completing the updated MRE, the Author uses procedures and methodologies that are generally consistent with industry standard practices, including those documented in the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019 CIM Guidelines).

To complete the current MRE for the Kay Deposit, a validated drill hole database comprising a series of comma-delimited spreadsheets containing surface diamond drill hole information was provided by Arizona Metals. The database included hole location information, down-hole survey data, assay data for all metals of interest, lithology data and density data. The data in the geochemistry / assay tables included data for the elements of interest, including Ag (g/t), Au (g/t), Pb (ppm), Zn (ppm) and Cu (ppm). After review of the database, the data were then imported into the GEOVIA GEMS version 6.8.3 software for statistical analysis, block modelling, and resource estimation. No errors were identified when importing the data. The data was validated in GEMS, and no erroneous data, data overlaps or duplication of data was identified.

The updated database provided by Arizona Metals for the MRE included data for 234 surface diamond drill holes, completed on the Property, totalling 133,912 m. The database totals 11,533 assay intervals representing 14,066 m of drilling. The average assay sample length is 1.21 m.

For the current MRE, in collaboration with Arizona Metals, the authors constructed two (2) three-dimensional resource models and four (4) lithology models for the Kay deposit in Leapfrog Geo version 2025.1.0.

Host rock lithology models were constructed incorporating drilling data, surface mapping, and structural interpretations in addition to SGS field and drill core observations. Lithology models comprise the Hangingwall Mafic Sequence (MVS), Felsic Volcanic Sequence (FVS), Graphite-rich Horizon (GH), and the Mineralization Horizon (MIN-Horizon). The MIN-Horizon model was constructed using the Leapfrog Geo Vein tool from assays greater than 0.5% CuEq and was used to establish the bounding limits of the subsequently constructed resource models. The MIN-Horizon model is consistent with the interpretation that within the property-scale isoclinal folding, the sulfide lenses are affected by steeply plunging tight folds (parasitic S-folds).

The Kay drillhole database and drill core were reviewed to evaluate the geological continuity and internal variability with respect to mineralization styles, metal zonation patterns, and density. The deposit displays complex internal variability of mineralization style, density, and relative metal distributions. Mineralization within the MIN-Horizon model was sub-domained using CuEq grade as a proxy for mineralization style and density. Two (2) resource models were constructed: a semi-massive to massive sulfide, high-grade domain (MIN-HG) and a stringer sulfide, low-grade domain (MIN-LG), to domain-appropriate density and capping values in the estimation process.

The MIN-HG and MIN-LG resource models were constructed using the Leapfrog Geo Indicator RBF numerical modelling tool with a structural trend based on the folded MIN-Horizon model. The MIN-HG resource model was established from assay intervals above 1.5% CuEq constrained by the MIN-Horizon model. The MIN-LG resource model was established from assay intervals above 0.5% CuEq, outside of the MIN-HG model, and constrained by the MIN-Horizon model.

A digital elevation surface model (LiDAR) was provided for the Property area. All 3D resource models were clipped to topography and limited to the Property boundary.

Mineralization in the Kay sulfide lens resource models extends for up to 400 m along strike and up to 850 m vertically (900 m down plunge). The mineralization horizon in general dips at 73º towards 260º (W) with local variations in strike and dip resulting from steeply plunging tight parasitic folds. The principal plunge direction of the sulfide lenses is 68º towards 300º (WNW) and appears to be influenced in part by steeply plunging tight parasitic folds.

The Author has reviewed the resource models on plan view and in section view, and in the Author's opinion, the models are well constructed and appear to be representative of the mineralization identified on the Property and the distribution of the Cu-Au-Zn-Pb-Ag mineralization within these sulfide lenses. Models were reviewed by Arizona Metals during the modelling process and refined by SGS before final resource estimation. Models have been extended beyond the limits of the current drilling for the purpose of providing guidance for continued exploration. However, the extension of the mineral resource beyond the limits of drilling is limited by the search radius during the interpolation procedure (a maximum of 110 m in the plunge direction past drilling).

25.2.1 Mineral Resource Statement

Highlights of the Project Mineral Resource Estimate are as follows:

- The underground MRE includes 9.28 million tonnes grading 1.39 g/t Au, 27.6 g/t Ag, 0.97% Cu, 0.33% Pb, and 2.39% Zn in the Indicated category, and 0.86 million tonnes grading 1.06 g/t Au, 15.4 g/t Ag, 0.87% Cu, 0.20% Pb, and 1.68% Zn in the Inferred category, at a base-case cut-off grade of 1.00% CuEq.

Table 25.2: Kay Property Mineral Resource Estimate, June 17, 2025

nes (Mt)	Average Grade						Contained Metal					
	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	CuEq (%)	Au (koz)	Ag (koz)	Cu (Mlbs)	Pb (Mlbs)	Zn (Mlbs)	CuEq (Mlbs)
Indicated												
9.28	1.39	27.6	0.97	0.33	2.39	3.18	415	8,253	197.9	67.3	490.1	650.6
Inferred												
0.86	1.06	15.4	0.87	0.20	1.68	2.44	29	423	16.4	3.8	31.8	46.1

Kay Deposit Mineral Resource Estimate Notes:

1) *The effective date of the Kay Project Mineral Resource Estimate (MRE) is June 17, 2025. This is the close-out date for the final mineral resource drilling database.*
2) *The mineral resource was estimated by Allan Armitage, Ph.D., P.Geo. of SGS Geological Services, an independent Qualified Person as defined by NI 43-101. Armitage conducted site visits to the Kay Deposit on two (2) occasions, on October 25-26, 2023, and April 7-8, 2024. The mineral resource was peer reviewed by Ben Eggers, MAIG, P.Geo. of SGS Geological Services, an independent Qualified Person as defined by NI 43-101. Eggers conducted a site visit to the Kay Property on May 30, 2025.*
3) *The classification of the current MRE into Indicated and Inferred mineral resources is consistent with the current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves.*
4) *All figures are rounded to reflect the relative accuracy of the estimate, and numbers may not add due to rounding.*
5) *All mineral resources are presented undiluted and in situ, constrained by continuous 3D wireframe models (considered mineable shapes), and are considered to have reasonable prospects for eventual economic extraction.*
6) *Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that most Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
7) *The Kay Project MRE is based on a validated drill hole database, which includes data from 234 surface diamond drill holes completed between 2020 and May 2025. The drilling totals 133,912 m (including wedge holes). The resource database totals 11,533 assay intervals representing 14,006 m of data.*
8) *Grades for Au, Ag, Cu, Pb and Zn are estimated for each mineralization domain using 1.50 m capped composites assigned to that domain. To generate grades within the blocks, the inverse distance squared (ID^2) interpolation method was used for all domains.*
9) *Average density values were assigned to each domain based on a database of 2,307 samples.*
10) *Based on the size, shape, and orientation of the deposit, it is envisioned that the deposits may be mined using underground bulk mining methods such as Longhole Stoping. The MRE is reported at a base case cut-off grade of 1.00% CuEq. The mineral resource grade blocks are quantified above the base case cut-off grade and within the constraining mineralized wireframes (considered mineable shapes).*
11) *The underground base case cut-off grade of 1.00% CuEq considers metal prices of $4.10/lb Cu, $1.00/lb Pb, $1.35/lb Zn, $2,200/oz Au and $26/oz Ag, assumed metal recoveries of 92% for Cu, 76% for Pb, 85% for Zn, 76% for Au and 75% for Ag, a mining cost of USD 49.00/t rock and processing, treatment and refining, transportation and G&A cost of USD 29/t mineralized material.*
12) *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*

25.3 Mineral Reserve Estimate

This Preliminary Economic Assessment (PEA) of the Kay Mine Project is based on indicated and Inferred Mineral Resources. Because of the inclusion of Inferred Resources, it is not applicable to determine Mineral Reserves at this stage of the Project. Economic zones will be classified as mineralized material only.

25.4 Mining

Kay Mine is planned as a mechanized long-hole open stoping underground mine. The milling rate is planned at 0.7 Mtpa with a ramp-up period of 9 months during the first operational period. The mill will run for 10 years

The underground operation consists of a single mine accessed via one (1) surface portal located south of the surface infrastructure area. The selected mining method consists of long-hole open stoping (LHOS), specifically sublevel transverse stoping and sublevel longitudinal stoping.

The life-of-mine (LOM) for the underground operation is estimated at 12.5 years, encompassing construction and development, pre-production, and full production phases. Of this total, the underground mine is expected to operate in production for approximately 10 years, including a nine (9)-month ramp-up period. The pre-production phase is anticipated to last approximately two and a half (2½) years following portal construction, allowing sufficient underground development to be completed to support sustained full production.

The underground mine is planned to operate at an average production rate of 1,910 tonnes per day (tpd) of mineralized material, of which approximately 1,730 tpd will be sourced from stope production and 180 tpd from lateral development. The mine plan includes the excavation of approximately 39.3 km of lateral development and 4.0 km of vertical development.

A total of approximately 6.55 million tonnes (Mt) of mineralized material is expected to be mined at average diluted grades of 1.01% Cu, 2.67% Zn, 1.60 g/t Au, and 29.07 g/t Ag. The primary production fleet will consist of 15-t diesel-powered load–haul–dump (LHD) units in combination with 45-t underground haul trucks for the transport of all mined material.

25.5 Processing

The development of the Kay Mine Project process flowsheet is based on the metallurgical test work and comminution simulations conducted during the study phase.

The crushing and grinding circuits have been evaluated through simulation using industry-recognized modelling software. Evaluation of flotation circuits for producing Cu-Pb, Zn and pyrite concentrates has been based on open rougher-cleaner flotation and a single locked cycle test.

Scoping level Albion oxidative leaching testwork indicates that the refractory pyrite concentrate is amenable to Albion, followed by cyanidation for gold recovery. Further optimization testwork is needed to validate and confirm the selection.

Collectively, the test work and engineering assessments provide a sound technical basis for the proposed process design, with potential for further optimization during the next study phases.

25.6 Infrastructure

The proposed infrastructure for the Kay Mine Project has been developed to support a 0.7 Mtpa underground mining and mineral processing operation over the planned life of mine. The infrastructure layout has been established considering local topography, operational requirements, environmental management objectives, construction efficiency, and integration with the surrounding site conditions.

The Project infrastructure includes water management infrastructure, mine infrastructure, process infrastructure, and supporting surface facilities required to sustain underground mining and mineral processing operations.

Water management infrastructure will include process water and fire water systems, potable water treatment facilities, contact water collection ponds, sewage treatment systems, and an effluent treatment plant. Surface water management systems, including diversion ditches, culverts, runoff conveyance infrastructure, sediment control measures, and contact water collection systems, will be implemented throughout the Project site to manage contact and non-contact water.

Mine infrastructure will include the underground mine access area, mine maintenance facility and warehouse, mine administration building, mine dry, explosives storage facilities, and the Dry Stack Tailings Storage Facility (DSTSF). The DSTSF will store filtered tailings generated from the process plant and will incorporate drainage and runoff management systems designed to support long-term operational stability and water management objectives.

Process infrastructure will include the mineral processing facility, assay and metallurgical laboratory, reagent storage facilities, compressor room, crushed mineralized material stockpile, and run-of-mine

(ROM) pad. The process plant is designed to support the production of copper and zinc concentrates, as well as gold.

Supporting infrastructure will include electrical substations and power distribution systems, emergency backup power generation systems, communications infrastructure, fuel storage and distribution facilities, kitchen and lunchroom facilities, and site security infrastructure with controlled site access.

The Project infrastructure will be connected through a network of access roads, service roads, haul roads, and DSTSF access roads designed to support the safe and efficient movement of personnel, materials, and mining equipment throughout the site.

25.7 Environmental, Social and Permitting Considerations

The Project would require a range of federal, state, and local permits to authorize mining, protect environmental resources, and ensure regulatory compliance. At the federal level, a BLM Plan of Operations is required under 43 CFR 3809 and administered by the BLM. Approval of this plan would trigger environmental review under the NEPA, likely as an Environmental Assessment.

At the state level, several key permits are required. A Mined Land Reclamation Plan, administered by the Arizona State Mine Inspector, ensures disturbed lands are reclaimed to a safe and stable condition and requires financial assurance. An Aquifer Protection Permit (APP) and a Class II Air Quality Permit must be obtained from the ADEQ to address potential groundwater contamination and air emissions. Additionally, the Project must comply with the CWA through coverage under AZPDES, including both a Multi-Sector General Permit for stormwater and, if applicable, an Individual Permit for direct discharges to surface waters.

The Project would also require various supporting permits and notifications. These include federal requirements such as mine registration and safety compliance with the MSHA, hazardous waste identification with the EPA, and potential CWA Section 404 authorization from the USACE. Additional permits may be needed for explosives use, communications, water wells, hazardous and solid waste handling, and fire safety. At the local level, approvals such as a mining / metallurgical use exemption from Yavapai County may also be required.

Overall, the permitting framework is comprehensive and involves coordination across multiple agencies, with early planning (e.g., for reclamation, groundwater protection, and stormwater management) critical to successful Project approval.

25.8 Capital and Operating Costs

Life-of-mine Project capital costs are estimated to total USD 731 million, consisting of the following three (3) distinct phases:

- Initial Capital Expenditure – This phase includes all costs to develop the property with a process plant designed to nominally treat 0.7 Mtpa of fresh rock. Initial capital costs total $609 million (including $84 million of contingency). The initial capital excludes pre-production net revenue of $18 million. The construction phase extends over a 30-month design, construction, pre-production and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations and the underground mining development. Sustaining capital costs are estimated to be $87 million.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine for twenty-five (25) years after operations. Closure costs are estimated to be a total of USD 28 million. In addition, an annual surety bond has been included as financial assurance, estimated at US$7 million, bringing the total closure cost to US$35 million.

The Capital and sustaining expenditures are summarized in Table 25.3 according to the level 1 work breakdown structure (WBS). Expenditures are presented in US dollars.

Table 25.3: Capital Expenditure Summary

Capital Expenditures (k USD)	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
100 – Infrastructure	41,953	-	41,953
200 – Power and Electrical	23,192	-	23,192
300 – Water Management	20,619	-	20,619
400 – Surface Operations	21,691	-	21,691
500 – Mining	84,023	87,205	171,228
600 – Process Plant	185,059	-	185,059
700 – Construction Indirect	87,754	-	87,754
800 – General Services / Owner's Cost	15,131	-	15,131
900 – Pre-production, Start-up, Comm.	45,130	-	45,130
990 – Contingency	84,126	-	84,126
Total	**608,678**	**87,205**	695,883

The operating costs include mining, processing, general services and administration ("G&A"), royalties, concentrates transportation and refining, and power cost, which is included within each area. The average LOM operating cost is $138.47/t milled.

25.9 Economic Analysis

The economic analysis is presented in real terms (i.e., excluding the effects of inflation) and is denominated in 2026 Q1 United States dollars (USD), unless otherwise indicated.

The economic analysis was conducted using a discount rate of 5% and long-term metal price assumptions of copper at $4.70/lb Cu, zinc at $1.27/lb Zn, gold at $3,100/oz Au, and silver at $38.00/oz Ag. Cash flows were discounted from the start of construction, and all costs before this period were considered as sunk costs.

The total after-tax cash flow over the Project life is $259M, and the NPV 5% is $40M pre-tax and $-6M after-tax. The after-tax Project cash flow results in a 7.5-year payback period from the commencement of commercial operations with an IRR of 6.0% pre-tax and 4.9% after-tax.

The Project's financial performance is most sensitive to the metal price and much less to the operating costs and initial capital expenditures.

25.10 Project Risks and Opportunities

25.10.1 Risks

25.10.1.1 Mineral Resources

A portion of the contained metal of the Kay Deposit, at the reported cut-off grades for the MRE, is in the Inferred Mineral Resource classification. It is reasonably expected that the majority of Inferred Mineral resources could be upgraded to Indicated Minerals Resources with continued exploration.

The mineralized structures (mineralized domains) are relatively well understood. However, due to the limited drilling in some areas, all mineralization zones might be of slightly variable shapes from what has been modelled. A different interpretation from the current mineralization models may adversely affect the current MRE. Continued drilling may help define with more precision the shapes of the zones and confirm the geological and grade continuities of the mineralized zones.

25.10.1.2 Mining

The mine plan and economic analysis presented in this Report include the use of Inferred Mineral Resources, which 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 the results of the PEA will be realized, nor that Inferred Mineral Resources will be converted to Measured or Indicated Mineral Resources. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Risks in the mining discipline include changes to the following factors and assumptions:

- Continued inflationary pressure.
- Interpretations of mineralization geometry and continuity in mineralized zones.
- Ability of the mining operation to meet the production targets and development rates.
- Operating cost assumptions.
- Mine operation and process plant recoveries.
- Metal prices.

25.10.1.3 Geomechanical

The current underground design is based on a preliminary geomechanical assessment using empirical methods, literature-derived parameters, and interpretation of drill-core photographs. Although the rock mass is generally classified as moderate to good quality and suitable for transverse longhole open stoping, several uncertainties remain that may affect excavation stability, dilution, and support requirements during mining.

A primary risk comes from limited direct geotechnical data. Intact rock strength, joint conditions, and rock mass quality were estimated based on comparable rock types rather than through lab testing or detailed logging. RQD values were obtained from photographs, where it was difficult to distinguish natural from mechanical breaks, and groundwater and stress-reduction factors were assumed. As a result, the actual rock mass quality may differ from current assumptions, potentially leading to greater overbreak, increased support requirements, or smaller stope spans.

Rock mass behaviour is expected to be structurally controlled. The presence of a dominant foliation-parallel joint set dipping steeply introduces anisotropy that may govern hanging wall performance. Local wedge or planar failures may occur, especially in shallow-dipping stopes or where stopes are unfavourably oriented relative to foliation. Although empirical stability charts indicate stable to transitional conditions, structurally controlled failures can still develop locally and increase dilution.

An additional uncertainty is the presence of a nearby graphitic horizon interpreted as a fault zone that was not included in the stability analysis. If mining approaches or interacts with this structure, localized loss of confinement and larger-scale hanging wall instability could occur, requiring design modifications and increased ground support.

Stress conditions were estimated rather than directly measured, assuming a moderately high horizontal stress regime at approximately 600 m depth. If actual stresses exceed the assumed values, stress concentrations around stopes and brows could lead to instability or require rehabilitation. Similarly, groundwater conditions were not assessed and might decrease joint strength and increase dilution beyond predicted levels.

Finally, backfill strength and support systems were established using empirical design approaches without site-specific testing. If cemented rockfill strength or rock mass response deviates from expectations, backfill failures may occur during mining of adjacent stopes.

25.10.1.4 Processing

The process plant design chosen for the study relies on conventional flotation technology, which is not efficient in recovering fine particles. This may result in under-optimized metallurgical performance. Alternatively, Glencore Jameson Cell and Metso Concorde Cell technologies should be evaluated. Further, while Albion oxidative pretreatment of pyrite concentrate followed by cyanidation was the only method tested, other refractory gold recovery processes were not evaluated. Testing on these processes should be considered in the next study phases.

25.10.1.5 Infrastructure

The current infrastructure design for the Kay Mine Project has been developed at a conceptual level appropriate for a Preliminary Economic Assessment and remains subject to refinement during subsequent engineering phases. Although the proposed infrastructure configuration is considered technically feasible, several risks and uncertainties remain associated with detailed design development, construction execution, permitting, and long-term operational performance.

Water management represents a key infrastructure risk due to the proximity of operational infrastructure to local drainage features and the requirement to effectively separate contact and non-contact water throughout the site. Additional hydrological and hydrogeological investigations will be required to refine water management criteria, stormwater diversion requirements, contact water storage capacity, and long-term treatment requirements.

The Dry Stack Tailings Storage Facility (DSTSF) configuration remains conceptual at the PEA stage. Additional geotechnical, hydrological, and environmental investigations will be required to confirm foundation conditions, drainage requirements, facility staging, and long-term closure performance. Changes to the DSTSF configuration or water management requirements may impact the overall site layout and capital cost estimates.

Electrical infrastructure and power supply requirements are based on preliminary load estimates and conceptual distribution layouts. Future engineering studies will be required to confirm utility connection requirements, final substation configuration, emergency power generation requirements, and detailed electrical distribution design.

The Project infrastructure capital cost estimate has been prepared to a PEA level of accuracy and remains subject to uncertainties associated with market conditions, labour availability, equipment pricing, contractor availability, and ongoing inflationary pressures. Future engineering, geotechnical investigations, and vendor engagement may result in modifications to the current infrastructure scope and associated capital cost estimates.

25.10.2 Opportunities

25.10.2.1 Mineral Resources

There is an opportunity in the Kay Deposit area to extend known mineralization at depth, on strike and elsewhere on the Property and to potentially convert Inferred Mineral Resources to Indicated Mineral Resources. Arizona Metal’s intentions are to direct their exploration efforts towards resource growth with a focus on extending the limits of known mineralization and testing other targets on the greater Kay Property.

25.10.2.2 Mining

Opportunities in the mining discipline include changes to the following factors and assumptions:

- Increase the size of underground stopes.
- Review optimal mining throughput.
- Reduction of ventilation and cooling requirements.

- Reduction of the dewatering requirements.

25.10.2.3 Geomechanical

As additional geotechnical data become available during advanced studies and early development, there is an opportunity to optimize the geomechanical recommendations.

A collection of laboratory strength data, detailed core logging, and in situ stress measurements would enable improved rock mass classification and stability analyses. This could reduce uncertainty in stability and may justify a larger hydraulic radius than currently assumed. Stope dimensions might be increased beyond the preliminary 13 m strike length, 5–15 m width, and 25 m height while still maintaining acceptable stability performance.

Dilution estimates are currently conservative due to uncertainty in rock mass strength and structural behaviour. Improved geomechanical characterization and monitoring of stope performance may reduce anticipated dilution.

Backfill assumptions offer room for optimization. Cemented rockfill strength requirements and binder contents were chosen using general analytical methods and have not yet been verified with site materials. Laboratory testing and mix design improvements could lower binder content.

Ground support recommendations are currently based on standard empirical patterns suitable for a wide range of conditions. Implementing an observational ground control approach during development, such as mapping and performance monitoring, would help refine support requirements.

25.10.2.4 Processing

There is an opportunity to improve the grade and recovery of base and precious metals by evaluating advanced technologies such as the Glencore-Jameson Cell and Metso-Concorde Cell. These technologies offer improved flotation selectivity, particularly for the recovery of fine and ultrafine particles.

Testing on these technologies during the next study phases would help achieve optimal metallurgical performance.

25.10.2.5 Infrastructure

Opportunities exist to further optimize the Project infrastructure layout, construction strategy, and water management systems during subsequent engineering phases. Additional engineering studies and site investigations may identify opportunities to reduce earthworks quantities, improve operational efficiencies, and optimize the integration of surface infrastructure.

Further optimization of the water management system, including refinement of contact water management strategies, runoff diversion systems, and process water recycling, may reduce long-term water treatment requirements and improve overall water reuse throughout the operation.

The Dry Stack Tailings Storage Facility (DSTSF) configuration may also benefit from additional optimization studies focused on facility staging, tailings placement strategies, drainage design, and material handling logistics. Opportunities may exist to reduce haul distances, minimize containment requirements, and optimize long-term closure performance.

Additional opportunities may also be identified through detailed power distribution studies, equipment selection optimization, and refinement of communications and operational support infrastructure. Given the relatively short Project life, opportunities may also exist to optimize certain support buildings through the use of lighter or modular construction approaches where operationally appropriate. Evaluation of alternative building configurations and construction standards will be completed during subsequent engineering phases. Further engineering development may improve constructability, operational reliability, and maintenance efficiency while reducing overall capital and operating costs.

As the Project advances toward future study phases, additional geotechnical, hydrological, hydrogeological, and civil engineering investigations will support refinement of the infrastructure design basis and may further improve the overall Project configuration and execution strategy.

26. RECOMMENDATIONS

The results of the financial analysis presented in this Preliminary Economic Assessment (PEA) demonstrate potential technical feasibility; however, it does not yet support robust economic viability under current assumptions. It is recommended to carry out additional work before advancing toward a future Pre-feasibility Study (“PFS”) for the Project. The proposed budget total discussed in this section is $11.1M and is summarized in Table 26.1.

Table 26.1: Cost Estimate Associated with Recommendations

Description	k USD
Exploration and Drilling	9,132
Land and Property Fees	42
Metallurgical Testing Program	500
Climatic Study for Colling Assessment	55
Geomechanical Drilling, Testing and Engineering	808
Hydrogeology Field Investigation	600
Total	**11,137**

26.1 Geology / Exploration

The Kay Project deposits contain underground Indicated and Inferred Mineral Resources that are associated with well-defined mineralized trends and models. All deposits are open along strike and at depth.

The Project has potential for delineation of additional Mineral Resources. Given the prospective nature of the Kay Property, it is the opinion of the QP that the Property merits further exploration and that a proposed plan for further work by Arizona Metals is justified.

It is recommended that Arizona Metals conduct further exploration on the Project, subject to funding and any other matters which may cause the proposed exploration program to be altered in the normal course of its business activities or alterations which may affect the program as a result of exploration activities themselves.

The following work is recommended:

- A minimum of 5,000 m additional drilling within the Kay Deposit: infill drilling to convert Inferred material to Indicated, expansion drilling to north, south, and at depth, as recommended by MRE report authors. This drill program should be contingent on results and could be more than recommended.

- A minimum of 7,500 m exploration drilling outside the Kay Deposit once the EXPO permit is approved, including road construction, EXPO permit support, and reclamation bonding. This drill program should be contingent on results of the AI studies described below and on initial drill results and could be more than recommended.
- Property-wide artificial intelligence studies to evaluate exploration drill targets outside the Kay Deposit. Data evaluated in the AI studies includes soil and rock geochemistry; ground EM geophysics; airborne magnetic and VTEM geophysics; gravity geophysics; an airborne hyperspectral survey; multi-element drill assays; and all lithologic, structure, and geologic data gathered during drill logging. The intent of the AI studies is to comprehensively target drilling for expansion of the Kay deposit and generation of additional exploration targets outside the Kay deposit.

26.2 Mining

The following work is recommended:

- Carry out a comprehensive climatic study to obtain site-specific data on ambient (surface) temperatures, geothermal gradient, and virgin rock temperature in order to refine the assessment of underground cooling requirements and associated infrastructure.
- Conduct a trade-off analysis on equipment selection to minimize ventilation demand.
- Perform a trade-off study to determine the optimal mining throughput.

26.2.1 Geomechanical

The current confidence in the geotechnical database and rock mass characterization is deemed adequate at this stage. Additional geotechnical investigation is necessary to support upcoming engineering studies and to minimize uncertainties related to excavation stability, dilution, and ground support requirements.

A geotechnical logging program should be carried out on drillholes near planned underground infrastructure and stoping zones. Laboratory testing should be conducted on core samples from the main lithologies. The results will enable the refinement of intact strength parameters and empirical stability inputs currently estimated from published literature.

Further characterization of the graphitic fault horizon and other potential shear or fracture zones is necessary. The hanging wall graphitic structure should be included in future design models, as its presence could adversely affect stope stability and increase dilution.

Once site-specific rock mass parameters and structural data are available, numerical modelling should be used to complement empirical design methods. The modelling should evaluate stress redistribution around stopes following the planned mining sequence.

26.2.2 Hydrogeology

The following presents high-level recommendations for the next phase of engineering studies. The primary objective is to characterize baseline groundwater flow conditions and assess water quality within both the bedrock and the flooded underground mine workings. This includes evaluating the permeability of the bedrock surrounding the proposed new mine and the historic mine targeted for dewatering, as well as investigating the potential hydraulic connection between the mine workings and the Agua Fria River alluvial plain.

The data collected through these investigations will support the development of a conceptual and numerical hydrogeological model of the site. This model will be used to estimate groundwater inflow into the mine workings and assess potential impacts on the surrounding groundwater resources.

To support this effort, a hydrogeological field investigation should be undertaken, including:

- Packer tests in boreholes to define bedrock hydraulic conductivity vs depth profiles, with a focus on geological structures that could connect the mine workings with the alluvial plain of the Agua Fria River.
- Perform a pumping test if the packer test results show high permeability zones in bedrock.
- Install monitoring wells to define baseline groundwater flow direction and groundwater quality.
- Obtain water samples from various depths within the flooded mine shafts scheduled for dewatering. The results will support the planning and design of appropriate water treatment measures for the extracted water.

26.3 Processing

26.3.1 Metallurgical Testing and Mineral Processing

While the initial metallurgical test programs have demonstrated that the selected flowsheet is appropriate for the Kay Mine ore type, more testwork is needed to optimize the process selection. The following testwork programs are proposed:

- Comminution testwork for sizing comminution circuit and evaluating the crushing and grinding characteristics for the Kay Mine ore types (Variability testing).
- Open circuit and Locked cycle Flotation test work on variability samples for assessing the grade-recovery variability for the Kay mineralization.
- Signature plot tests will be required to evaluate the energy requirements for reducing the particle size in the copper, zinc, and Albion circuits.
- Static and dynamic settling tests for sizing thickeners (concentrate, pre-leach and tailings thickeners)
- Cu-Pb and Zn Concentrate filtration tests for the selection and sizing of filtration units.
- Viscosity tests on concentrate products and final tailings.
- Testing using Albion, Pressure oxidation, and Biological oxidation technology (BIOX) for increasing gold recoveries.
- Cyanide detoxification tests on tailings produced from the cyanidation testwork for sizing cyanide detoxification equipment.
- Tailings filtration testing for producing dry-stack tailings.
- Acid Base Accounting (ABA) and humidity cell testing for environmental compliance.
- It will be necessary to define all water sources and to close the site water balance, explicitly accounting for seasonal variability.
- Evaluate the application of column cells, Jameson cell and Concorde cell in flotation for grade and recovery improvement in Cu/Pb, Zn and Pyrite circuits.
- It should be evaluated whether to operate the copper and zinc regrind circuits in open circuit, by directing the mill discharge to combine with the cyclone overflow and feed the cleaner flotation stages, without returning the product to the cyclone.

26.3.2 Recovery Methods

Current PEA-level metallurgical test work has provided a basis for the preliminary design of the recovery circuits. However, opportunities remain to further optimize gold recovery, improve reagent consumption, and enhance operational efficiency. The following recommendations are proposed to validate and refine the recovery methods during the next study phases:

- **Grinding Circuit Simulation:** Grinding circuit evaluation to be carried out utilizing a fine grinding technology for an energy-efficient grinding unit for producing a finer grind product.
- **Flotation Optimization Tests:** Tests for optimizing Cu-Pb, Zn and pyrite flotation performance using different flotation configurations (reagents, conditions, and flotation sequence) followed by locked cycle tests (LCT) on the selected processing option.
- **Jameson Cell Technology**: Jameson Cell tests to evaluate the potential of Jameson cell technology for replacing the mechanical cells for recovery improvements.
- **Column Flotation, Metso Concorde Cell Technology** to be evaluated for improving the recovery.
- **Signature Plot Testwork:** Signature plot testwork needs to be performed on Cu-Pb and Zn rougher concentrates for the selection and sizing of regrind mills.
- **Gold Recovery from Pyrite Tailings:** Different processing options like Biological Treatment (BOIX), Pressure Oxidation (POX) need to be explored in addition to the Albion Process to select the optimal processing route for recovering the gold from pyrite flotation tailings.
- **Gold Recovery from Refractory Pyrite Concentrate:** Optimization testwork should be carried out on the Albion process, including other processing options (Pressure Oxidation, BIOX) for treating the refractory gold in the pyrite concentrate to select the optimal processing route.
- **Cyanidation Testwork for CIL, ADR Circuit Design:** Depending on the treatment route selected for processing the refractory gold, a cyanidation step is needed to recover the gold. Tests to be performed to select and size the equipment for the cyanidation circuit, adsorption, desorption and refining (ADR) circuit.
- **Detoxification Tests:** Cyanide detox tests to be performed for selection and sizing of the detox equipment.
- Evaluate heap leaching for processing the low-grade material to explore the potential of processing the high-grade material in the process plant and the low-grade material by heap leaching.

26.4 Infrastructure

A number of future studies and site investigations have been identified to further evaluate and refine the Project infrastructure design and to support consideration of advancement toward future study phases, including a potential Pre-Feasibility Study (PFS). These activities are intended to reduce technical uncertainty, improve infrastructure integration, refine capital and operating cost estimates, and support future permitting and Project evaluation activities.

Key opportunities for the infrastructure development at the Kay Mine Project are as follows:

- Reduction of capital cost uncertainty through additional engineering studies and collection of site-specific data, including geotechnical, geochemical, hydrological, and hydrogeological investigations to support refinement of foundation systems, civil designs, and water management and treatment infrastructure.
- Evaluation of opportunities to share selected infrastructure with other potential projects in the region.
- Completion of detailed hydrological and hydrogeological analyses, including refinement of the site water balance, determining water intake and discharge requirements and support effective recycling of water recovered from the DSTSF and contact water management systems.
- Completion of additional geotechnical investigations for the process plant, DSTSF, buildings, and major infrastructure corridors to support optimization of foundation systems, earthworks quantities, and construction methodologies, which may reduce capital costs.

Additional engineering is recommended to further refine the overall site layout, including process infrastructure, underground and surface support infrastructure, water management infrastructure, and utility corridors. Updated quantity take-offs and vendor quotations would improve estimate accuracy and support future Project evaluation and decision-making activities.

26.4.1 Water Management

The preliminary design of the water management infrastructure for the Kay Mine site was prepared based on currently available data. To further refine the water management infrastructure design, additional hydraulic, hydrological, and hydrogeological studies should be undertaken.

These investigations should include field monitoring programs, detailed surveying, and data collection activities required to establish baseline hydraulic and hydrological conditions and support detailed site-wide hydraulic and hydrological modelling. The resulting information will support refinement of the site water balance, surface water diversion systems, contact water management infrastructure, and long-term water treatment requirements.

26.4.2 Other Infrastructure

The following work is recommended:

- Advance engineering design of all infrastructure components, including roads, buildings, power systems, communications systems, fuel storage facilities, water systems, and support facilities, to support future study phases. This work should include development of detailed design criteria, layout optimization, and coordination with geotechnical, environmental, and operational requirements.
- Complete additional geotechnical investigations in the areas designated for the process plant, stockpiles, roads, buildings, and major infrastructure corridors. The collected data should support foundation design, slope stability assessments, drainage planning, and refinement of earthworks quantities.
- Further optimize the site layout through value engineering exercises evaluating haul road alignments, cut-and-fill balances, utility corridors, and the proximity between operational areas. Reduction of haul distances and earthworks quantities may provide opportunities to reduce capital and operating costs.
- Advance utility systems and capacity planning, including refinement of electrical load distribution, water demand forecasting, process water requirements, and effluent generation estimates. The sizing and configuration of tanks, pumps, pipelines, treatment systems, and emergency backup systems should be refined based on updated operational requirements and redundancy criteria.
- Evaluate opportunities for modular or lighter construction approaches for selected support buildings considering the relatively short Project life. Alternative construction methodologies may improve constructability and reduce capital costs where operationally appropriate.
- Continue permitting activities associated with key infrastructure components, including road construction, water abstraction and discharge, sewage treatment systems, fuel storage facilities, and electrical infrastructure. Continued engagement with regulatory agencies and stakeholders should be maintained throughout future study phases to support Project evaluation and potential development activities.

26.5 Environmental Fieldwork and Studies

Key studies and future coordination efforts needed to support permitting and Project development include:

- **Environmental and technical studies:** groundwater characterization, geochemical testing of tailings and waste rock, air quality impact assessment, and a socioeconomic evaluation.
- **Regulatory and agency coordination:** consultation with SHPO, Tribal governments, and the USFWS for compliance with the ESA.
- **Additional analyses:** potential noise modelling and species-specific surveys (including ESA- and Migratory Bird Treaty Act-related nest clearance surveys), depending on construction timing.
- **Updates to existing studies:** revisions to the BE and cultural resources reports if the Project footprint changes or new lands are acquired.

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