NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA

June 11, 2021

Effective Date: March 31, 2021

Prepared for: Capstone Mining Corp.

Authors:
Clay Craig, P.Eng., Capstone Mining Corp.
Tony J. Freiman, PE, Wood, Environment & Infrastructure Solutions, Inc.
J. Todd Harvey, SME-RM, Global Resource Engineering, Ltd.
Garth Kirkham, P.Geo., FGC, Kirkham Geosystems Ltd.
Colleen Roche, P.Eng., Capstone Mining Corp.
Klaus Triebel, CPG, Capstone Mining Corp.
Edward C. Wellman, PE, PG, CEG, Independent Geomechanics LLC

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Capstone Mining Corp. (“Capstone”) by Global Resource Engineering, Ltd., Independent Geomechanics LLC, Wood, Environment & Infrastructure Solutions, Inc. and Capstone Mining Corp., collectively the “Report Authors”. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors’ services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Capstone Mining Corp. subject to terms and conditions of the individual contracts with the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of, or reliance on, this report by any third party is at that party’s sole risk.

Suite 2100 – 510 West Georgia Street,
Vancouver, BC, V6B 0M3, Canada
Tel: 604-684-8894 Fax: 604-688-2180
www.capstonemining.com

CERTIFICATE OF QUALIFIED PERSON

I, Clay Craig, do hereby certify that:

1) I am currently employed as the Manager, Mining and Evaluations by:
Capstone Mining Corp.
Suite 2100 – 510 West Georgia Street,
Vancouver, BC, V6B 0M3, Canada

2) This certificate applies to the report titled "NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA", with an effective date of March 31, 2021.

3) I am a graduate of University of British Columbia, with a Bachelor of Applied Science in Geological Engineering in 1994. I have been practicing my profession since 1994 and my relevant experience for the purpose of this technical report includes work in Mine Planning and Mine Engineering management at a number of open pit mines and development projects, as well as Mineral Reserves estimation at numerous projects. I have worked on and been involved with NI43-101 studies on a number of projects including the Oyu Tolgoi and Fort Knox mines.

4) I am a Registered Member of Engineers and Geoscientists British Columbia, P.Eng., EGBC# 25189.

5) I visited the Pinto Valley Mine property most recently April 26 to 29, 2021.

6) I am responsible for Sections 1.6, 1.7, 2.5, 3, 12.2, 15, 16.1, 16.3-16.11, 21, 22, 24, 25.2.2, 25.2.3 and 26.4, and my contributions to Sections 2.3, 2.4, 25.1, 25.3 and 27 in the report titled "NI 43-101 Technical Report on the Pinto Valley Mine, Miami, Arizona" with an effective date of March 31, 2021.

7) I have had prior involvement with the property as employee of Capstone Mining Corp., from October 2017 to October 2020 as Technical Services Superintendent and from June 2017 to September 2017 as Chief Geologist.

8) I am not independent of Capstone Mining Corp. as defined in Section 1.5 of National Instrument 43-101.

9) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of a Qualified Person as defined in National Instrument 43-101.

10) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that, at the effective date of the Technical Report, to the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

11) I have read the sections for which I am responsible for and the National Instrument 43-101, Standards for Disclosure of Mineral Projects and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Dated this June 11th, 2021 in Miami, Arizona, USA.

"Signed and Sealed"

Clay Craig, P.Eng.
Capstone Mining Corp.

Tony J Freiman, PE
Wood Environment & Infrastructure Solutions, Inc.
4600 East Washington Street, Suite 600
Phoenix, Arizona 85034

CERTIFICATE OF QUALIFIED PERSON

I, Tony J. Freiman, do hereby certify that:

1) I am currently employed as the Principal Geotechnical Engineer by:

Wood Environment & Infrastructure Solutions, Inc.
4600 East Washington Street, Suite 600
Phoenix, Arizona 85034

2) This certificate applies to the report titled “NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA”, with an effective date of March 31, 2021 (the Technical Report).

3) I am a graduate of the University of Arizona with a Bachelor of Science in Civil Engineering in 1985. I have been practicing my profession since 1985 and have worked on or been involved in previous mining studies, where I have been responsible for the design of tailings and mine infrastructure facilities. Previous studies include, among others, the Asarco LLC Ray Mine Elder Gulch Tailings Storage Facility. I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of a Qualified Person as defined in National Instrument 43-101.

4) I am a Registered Professional Engineer in the State of Arizona (License No. 23982).

5) I visited the Pinto Valley Mine property on May 6, 2021 to conduct the Engineer of Record quarter annual inspection of the active tailing storage facilities. I also visited the Pinto Valley Mine property on February 9 and 10, 2021 to conduct the Engineer of Record annual inspection of the active and inactive tailing storage facilities.

6) I am responsible for the following sections in the Technical Report:
18.2, 18.3, 18.4.2, 18.5.2, 18.6.1, 25.2.7 and 26.5 and my contributions to tailings storage in Sections: 1.9, 2.3, 2.4, and 25.1 and 25.3.

7) I have had prior involvement with the property since 1997 and since 2013 as Engineer of Record for the Pinto Valley Mine tailings storage facility and as independent qualified person co-author of the reports entitled “NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA”, with effective dates of January 1, 2014 and of January 1, 2016.

8) I am independent of Capstone Mining Corp. as independence is defined in Section 1.5 of National Instrument 43-101.

9) I have read NI 43-101 and I certify that the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument and Form.

10) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and I certify that, at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of this Technical Report that I am responsible for contains all the scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated this June 11th, 2021 in Phoenix, Arizona, USA.

“Signed and Sealed”

Tony J. Freiman, PE
Wood Environment & Infrastructure Solutions, Inc.

CERTIFICATE OF QUALIFIED PERSON

I, J. Todd Harvey, do hereby certify that:

1) I am currently employed as the President and Director of Process Engineering by:

Global Resource Engineering Ltd.
600 Grant Street, Suite 975
Denver, CO 80203 USA

2) This certificate applies to the report titled "NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA", with an effective date of March 31, 2021 (the Technical Report).

3) I am a graduate of Queen's University with a Ph.D. in Mining Engineering (major in process engineering) in 1995. I have been practicing my profession since 1987 and have worked on or been involved in previous mining studies disclosed under NI 43-101 standards, where I have been responsible for process plant designs, capital and operating costs and mine infrastructure facilities. Previous studies include, among others, Bear Creek Mining, Corani Project, NI 43-101 Report, Independent Technical Report and Preliminary Economic Assessment Kilgore Project, Preliminary Economic Assessment – Technical Report, Imperial Gold Project, California, USA. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of a Qualified Person as defined in National Instrument 43-101.

4) I am a Registered Member of the Society for Mining, Metallurgy & Exploration (SME).

5) I visited the Pinto Valley Mine property on several occasions and most recently May 4 and 5, 2021.

6) I am responsible for the following sections in the Technical Report: 12.4, 13, 17, 25.2.4, 25.2.5, 26.2, 26.3 and my contributions to Sections: 2.3, 2.4, 25.1, 25.3 and 27.

7) I have had prior involvement with the property since 2019 as process engineering consultant.
8) I am independent of Capstone Mining Corp. as independence is defined in Section 1.5 of National Instrument 43-101.

9) I have read NI 43-101 and I certify that the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument and Form.

10) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that, at the effective date of the Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this June 11th, 2021 in Denver, Colorado.

"Signed and Sealed"

J. Todd Harvey, SME-RM
Global Resource Engineering Ltd.

Kirkham Geosystems Ltd.

6331 Palace Place
Burnaby, B.C.
V5E 1Z6

Phone: (604) 529-1070
gdkirkham@shaw.ca

CERTIFICATE OF QUALIFIED PERSON

I, Garth D. Kirkham, P.Geo, FGC, do hereby certify that:

1) I am currently employed as the President by:
Kirkham Geosystems Ltd.
6331 Palace Place Burnaby,
BC, V5E 1Z6

2) This certificate applies to the report titled "NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA", with an effective date of March 31, 2021 (the Technical Report).

3) I am a graduate of the University of Alberta in 1983 with a BSc. I have continuously practiced my profession since 1988. I have worked on and been involved with NI43-101 studies on the Mineral Park, Halilağa, Ajax and Tres Chorreras porphyry deposits.

4) I am a Professional Geoscientist, Registered Member of the Engineers and Geoscientists of British Columbia.

5) I visited the Pinto Valley Mine property on May 14th, 2013 and April 16th -17th, 2015.

6) I am responsible for Sections 1.5, 12.1.2, 14 and 25.2.1 and my contributions to Sections 2.3, 2.4, 25.1, 25.3 and 27 of this Technical Report.

7) I have had prior involvement with the property as independent qualified person for the reports entitled "Pinto Valley Mine Life Extension – Phase 3 (PV3) Pre-Feasibility Study" with effective date of 1st of January, 2016, "Pinto Valley Mine Prefeasibility Study NI 43-101 Technical Report" with effective date of 1st of January, 2014 and "Resource Estimate for the Pinto Valley Deposit NI 43-101 Technical Report" with effective date of 8th of May, 2013 and amended date of 11th of December, 2013.

8) I am independent of Capstone Mining Corp. as defined in Section 1.5 of National Instrument 43-101.

9) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.

10) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that, at the effective date of the Technical Report, to the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

11) I have read the sections for which I am responsible for and the National Instrument 43-101, Standards for Disclosure of Mineral Projects and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Dated this June 11th, 2021 in the Burnaby, BC, Canada

"Signed and Sealed"

Garth Kirkham, P.Geo.
Kirkham Geosystems Ltd.

Suite 2100 – 510 West Georgia Street,
Vancouver, BC, V6B 0M3, Canada
Tel: 604-684-8894 Fax: 604-688-2180
www.capstonemining.com

CERTIFICATE OF QUALIFIED PERSON

I, Colleen Roche, do hereby certify that:

1) I am currently employed as the Operations Support Manager by:

Pinto Valley Mine
Capstone Mining Corp.
P.O Box 100, 2911 N. Forest Service Rd. 287,
Miami, AZ, 85539, USA

2) This certificate applies to the report titled "NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA", with an effective date of March 31, 2021.

3) I am a graduate of University of British Columbia with a Master of Engineering in Mining Engineering in 2008 and of McGill University with a Bachelor of Engineering in Mining Engineering in 2000. I have been practicing my profession since 2000 and my relevant experience for the purpose of this Technical Report includes operational duties at Minto Mine and Pinto Valley Mine related to pit design, preparation of permit documents, environmental department oversight, oversight of tailings, heap leach operations, facilities and budget preparation.

4) I am a Registered Member of Engineers and Geoscientists British Columbia, P.Eng., EGBC# 36041.

5) I visited the Pinto Valley Mine property most recently April 26 to 29, 2021.

6) I am responsible for Sections 1.1, 1.2, 1.8, 2.1, 2.2, 2.6, 3 to 6, 12.5, 18.1, 18.4.1, 18.5.1, 19, 20, 23, 25.2.6, 25.2.8 and 26.6, and my contributions to Sections 1.9, 2.3, 2.4, 25.1, 25.3 and 27 in the report titled "NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA", with an effective date of March 31, 2021.

7) I have had prior involvement with the property as employee of Capstone Mining Corp. in operations management roles since November 2015 at Pinto Valley Mine.

8) I am not independent of Capstone Mining Corp. as defined in Section 1.5 of National Instrument 43-101.

9) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of a Qualified Person as defined in National Instrument 43-101.

10) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that, at the effective date of the Technical Report, to the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

11) I have read the sections for which I am responsible for and the National Instrument 43-101, Standards for Disclosure of Mineral Projects and Form 43-101F1. This Technical Report has been prepared in compliance with that instrument and form.

Dated this June 11th, 2021 in Miami, Arizona, USA.

"Signed and Sealed"

Colleen Roche, P.Eng.
Capstone Mining Corp.

Suite 2100 – 510 West Georgia Street,
Vancouver, BC, V6B 0M3, Canada
Tel: 604-684-8894 Fax: 604-688-2180
www.capstonemining.com

CERTIFICATE OF QUALIFIED PERSON

I, Klaus Triebel, do hereby certify that:

1) I am currently employed as the Chief Resource Modeler by:
Pinto Valley Mine
Capstone Mining Corp.
P.O Box 100, 2911 N. Forest Service Rd. 287,
Miami, AZ, 85539, USA

2) This certificate applies to the report titled "NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA", with an effective date of March 31, 2021.

3) I am a graduate of University of Alaska Fairbanks, with a M.Sc. Geological Engineering in 1990 and of University of Applied Science – Mining, Energy and Environmental Science, Bochum, Germany with a B.Eng. Mining Engineering in 1981. I have been practicing my profession since 1981 and my relevant experience for the purpose of this technical report includes work as a geologist on projects at the exploration, development/start-up and operating stages. I have worked on and been involved with NI43-101 studies of similar scope on Kinross' open pit Fort Knox TR "Technical Report for the Fort Knox Mine Prepared for the Kinross Gold Corporation and Fairbanks Gold Mining Incorporated".

4) I am a Registered Member of the American Institute of Professional Geologists, AIPG# 10657.

5) I visited the Pinto Valley Mine property most recently April 26 to 29, 2021.

6) I am responsible for Sections 1.3, 1.4, 7, 8, 9, 10, 11, 12.1.1, 26.1, and my contributions to Sections 2.3, 2.4, 25.1, 25.3 and 27 in the report titled "NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA", with an effective date of March 31, 2021.

7) I have had prior involvement with the property as employee of Capstone Mining Corp., since November 2020 as Chief Resource Modeler and from May 2018 to October 2020 as Chief Geologist.

8) I am not independent of Capstone Mining Corp. as defined in Section 1.5 of National Instrument 43-101.

9) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of a Qualified Person as defined in National Instrument 43-101.

10) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that, at the effective date of the Technical Report, to the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

11) I have read the sections for which I am responsible for and the National Instrument 43-101, Standards for Disclosure of Mineral Projects and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Dated this June 11th, 2021 in Gold Canyon, Arizona, USA.

"Signed and Sealed"

Klaus Triebel, CPG
Capstone Mining Corp.

Independent Geomechanics LLC
2332 Decatur St Apt 4
Denver, CO 80211
Phone: (520) 232-4467

CERTIFICATE OF QUALIFIED PERSON

I, Edward C. Wellman, do hereby certify that:

1) I am currently employed as the Principal Geological Engineer by:

Independent Geomechanics LLC
2332 Decatur St Apt 4
Denver, CO 80211 USA

2) This certificate applies to the report titled "NI 43-101 Technical Report on the Pinto Valley Mine, Arizona, USA", with an effective date of March 31, 2021 (the Technical Report).

3) I am a graduate of the University of Nevada with a Master of Science in Geological Engineering in 1997. I graduated with a Bachelor of Science degree in Geosciences from the University of Arizona in 1994. I have been practicing my profession since 1996 and have worked on or been involved in previous mining studies disclosed under NI 43-101 standards, where I have been responsible for the geotechnical inputs to mine design in for both open-pit and underground mining. Previous studies include, among others, the Almaden Minerals Ixtaca Project, and the Royal Gold Peak Gold Project. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence, and affiliation with a professional association, I meet the requirements of a Qualified Person as defined in National Instrument 43-101.

4) I am a Registered Member of the Society for Mining, Metallurgy & Exploration (SME).

5) I visited the Pinto Valley Mine property on March 19, 2021.

6) I am responsible for the following sections in the Technical Report:
12.3, 16.2 and my contributions Sections: 2.3, 2.4 and 27.

7) I have had prior involvement with the property since 2006 as the principal consultant on slope stability for the project.

8) I am independent of Capstone Mining Corp. as independence is defined in Section 1.5 of National Instrument 43-101.

9) I have read NI 43-101 and I certify that the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument and Form.

10) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that, at the effective date of the Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this June 11th, 2021 in Denver, Colorado.

(Original Signed and Sealed)

Edward C. Wellman, PE, PG, CEG
Independent Geomechanics LLC

Table of Contents

Table of Figures

Table of Tables

1 Summary

The purpose of this report is to disclose updated Mineral Resource and Mineral Reserve estimates for the Capstone Mining Corp. (Capstone) affiliate owned Pinto Valley Mine (PVM) for the planned life of mine (LOM) through 2039.

Capstone and Pinto Valley Mining Corp. (PVMC) personnel compiled this report and contributed to multiple report sections. Consultants that contributed to this report include Global Resource Engineering (GRE), Independent Geomechanics LLC, Kirkham Geosystems Ltd. and Wood PLC (Wood). Personnel from each of these companies have signed off as QPs, as defined in National Instrument 43-101 (NI 43-101), for their specific sections of the Technical Report that they are responsible for preparing.

This Technical Report has an effective date of March 31, 2021. All information and assumptions discussed in this report were determined as of the effective date. In Section 1, tables and production statistics are reported in imperial and metric units. Both imperial and metric units are used throughout this report and are identified for each table and figure as appropriate. Cost estimates are based on 2021 US Dollars (US$).

Capstone is a Canadian-based metals mining company committed to the responsible development of the company's assets. Capstone's focus is on the production of copper at two producing mines, including Pinto Valley Mine in the United States and Cozamin Mine in Mexico. Capstone also owns 100% of the Santo Domingo Project in Chile, and a portfolio of exploration properties. Capstone's headquarters are in Vancouver, Canada, and the company is listed on the Toronto Stock Exchange. Further information is available at www.capstonemining.com.

1.1 Project Description

PVM is located in Gila County, Arizona, at the west end of the Globe-Miami mining district, approximately 11 miles west of Globe and 80 miles east of Phoenix via U.S. Highway 60, at 33°23′32″ N and 110°58′15″ W.

PVM's moderate, semi-arid climate allows for year-round operation. Average annual precipitation is 18.9 inches, although the area experiences periods of extended drought, including 2019 through to the publication of this Technical Report. Pinto Valley Mine has several water sources including a private wellfield with multiple wells, a system of water catchments with pumpback capabilities, and reclaim systems on operating tailings impoundments. Nevertheless, extended periods of drought have the potential to impact the operation of PVM. Average temperatures range from a high of 97°F in July to a low of 34°F in January. The terrain surrounding the mine property is generally mountainous with elevations ranging from 3,500 ft to 5,000 ft above mean sea level (amsl).

The historic mining towns of Miami and Globe are the closest communities to PVM, with a total population of approximately 10,000 residents. Some services are available locally in these communities, with the remainder of services coming from the greater Phoenix and Tucson areas.

PVM is an open pit mine that produces copper and molybdenum concentrates and copper cathode. Administration, ore processing and tailings and waste rock storage facilities and related infrastructure are located within 3 miles of the pit on PVM property or on adjacent National Forest System land administered by the Tonto Nation Forest. The processing facility consists of three crushing stages, grinding in six ball mills, copper flotation stages, a molybdenum flotation circuit, and associated thickeners for concentrates and tailings. The two existing tailings storage facilities (TSFs) will provide adequate tailings storage for the planned life of mine through 2039.

1.2 History and Ownership

The Globe-Miami mining district is one of the oldest and most productive mining districts in the United States, with more than 15 billion pounds (lb) of copper produced since its first recorded production in 1878.

The PVM property was originally owned by Miami Copper Company in 1909, and was held by numerous owners until the early 1970s when the mine, mill and other infrastructure were constructed by Cities Service Company. The mine was owned by Newmont, Magma Copper and subsequently BHP Billiton (BHP) until October 2013, when BHP sold PVM to PVMC.

PVM has been operating since 1974, with a brief shutdown in 1983, and subsequent shutdowns from 1998 to 2007, and from 2008 to 2012. Since restarting in 2012, PVM has operated continuously.

1.3 Geology and Mineralization

Several mines and numerous prospects have been developed in the Globe-Miami mining district. Larger mines in the district are porphyry copper deposits associated with Paleocene (63 megaanni [Ma]–59 Ma) granodiorite to granite porphyry stocks. The porphyry copper deposits have been dismembered by faults and affected by later erosion and oxidation. Vein deposits and possible exotic copper deposits are also found within the district.

The primary minerals of the porphyry copper deposits are chiefly pyrite and chalcopyrite, with minor amounts of molybdenite; gold and silver are recovered as by-products. Sphalerite and galena occur locally in non-economic occurrences. Hydrothermal alteration associated with the deposits include potassic, argillic, sericitic, and propylitic mineral assemblages.

The PVM deposit is a hypogene ore body with chalcopyrite, pyrite, and minor molybdenite as the only significant primary sulfide minerals. The primary host rock for the PVM porphyry copper deposit is the Precambrian-age Lost Gulch Quartz monzonite, which is equivalent to the Ruin Granite. Formation of the deposit was associated with the intrusion of small bodies and dikes of granite porphyry and granodiorite.

1.4 Exploration, Sampling and Drilling

Surface mapping and drilling are the main sources of data for the resource estimation.

A total of 951 drill holes have informed the Mineral Resource in this Technical Report. 778 of these holes were drilled prior to Capstone's ownership, and consisted of core, rotary, and churn drillholes. Churn holes defined much of the early mineralization, which has been mostly mined out. Since Capstone's ownership in 2013 an additional 173 core and RC holes have been drilled, generally targeting to infill upcoming production volumes to Measured drill spacing.

The data was confirmed as acceptable for use in resource estimation through database audits and QA/QC programs. For data prior to 2006, data quality was confirmed using a re-assaying program that included certified reference materials. All drilling campaigns after 2006 conform to industry standards for QA/QC.

1.5 Mineral Resource

To update the Mineral Resource presented in this Technical Report, surfaces and solids were generated for lithology and structural domains, and an indicator-based grade shell was generated at a 0.08% Cu threshold. A 45 ft composite length was used in order to minimize the smoothing of the grades, reduce the influence of very high-grade samples, and to match the 45 ft pit benches.

Updated density values were applied based upon lithology and alteration using information from 305 samples.

The block model grades for copper were estimated using Ordinary Kriging, with molybdenum being estimated by Inverse Distance. During grade estimation, search orientations were designed to follow the general trend of the mineralization in each of the zone domains. The great majority of blocks (greater than 98%) were estimated in a single pass for each domain, with the remaining areas of non-typical drillhole geometry receiving a “finishing” estimation from two further passes. The primary estimation pass required a minimum of five composites and a maximum of eight, with a maximum of three from any one drillhole. The Mineral Resource listed in Table 1-1 are for % copper (Cu) and % molybdenum (Mo) at a base-case cut-off grade of 0.14% Cu.

The Measured and Indicated Mineral Resource at PVM are inclusive of the Mineral Resource converted to a Mineral Reserve using modifying factors, including, but not limited to mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors. The Inferred Mineral Resource was not considered for conversion to a Mineral Reserve. Inferred Mineral Resources are estimated using limited geological evidence compared to Measured and Indicated Resources; this evidence is adequate to imply but not verify sufficient continuity of grade or geology. However, it is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to Indicated Mineral Resource with continued exploration and are consistent with the definition of Mineral Resources and their confidence categories in CIM (2014).

Table 1-1: Mineral Resource at 0.14% Copper Cut-off, as of March 31, 2021

Classification	Tonnes (M Tonnes)	Cu %	Mo %	Contained Copper (M lb)	Contained Molybdenum (M lb)
Measured (M)	619.9	0.33	0.006	4,442.7	83.4
Indicated (I)	782.5	0.26	0.005	4,493.6	88.0
Total M & I	**1,402.3**	**0.29**	**0.006**	**8,934.6**	**170.0**
Inferred	170.6	0.26	0.006	967.6	20.7

The Mineral Resource is classified according to CIM (2014) definitions, estimated following CIM (2019) guidelines and has an effective date of March 31, 2021. The Independent Qualified Person for the estimate is Mr. Garth D. Kirkham, P.Geo., FGC., of Kirkham Geosystems Ltd. The economic assumptions include the following: $3.50/lb Cu, $10.00/lb Mo, 84.6% average Cu recovery, 8.9% average Mo recovery, $1.74/tonne average mining costs, $1.13/tonne G&A costs, $0.88/tonne operational support costs, $4.67/ton milling costs, and pit slopes by rock type. The Mineral Resource is reported inclusive of the Mineral Reserve. The last date for drilling data and mining activities is March 31, 2021. Rounding may result in apparent summation differences between tonnes, grade and contained metal.

Garth Kirkham, P.Geo., FGC is the independent qualified person responsible for the PVM Mineral Resource.

1.6 Mineral Reserve

The PVM Mineral Reserve presented in Table 1-2 was developed in line with industry guidelines by tabulating the contained Measured and Indicated (Proven and Probable) material inside of the designed pit at the mill cut-off grades. The schedule utilizes a variable cut-off grade to the mill that fluctuates between 0.17 to 0.21% Cu. The final pit design and the Mineral Reserve do not include low grade leach dump material in the economic analysis or Mineral Reserve.

Table 1-2: Mineral Reserve at a variable cut-off of 0.17-0.21%, as of March 31, 2021

Classification	Tonnes (M Tonnes)	Cu %	Mo %	Contained Copper (M lb)	Contained Molybdenum (M lb)
Proven	241.6	0.34	0.007	1,833	35.6
Probable	139.4	0.28	0.006	877	17.4
Proven and Probable	**381.0**	**0.32**	**0.006**	**2,710**	**53.0**

The Mineral Reserve has an effective date of March 31, 2021 and was prepared by Clay Craig, P.Eng., Manager, Mining and Evaluations at Capstone Mining Corp. The economic assumptions include the following: $3.00/lb Cu, $10.00/lb Mo, 86.0% average Cu recovery, 8.5% average Mo recovery, $1.68/tonne average mining costs, $1.13/tonne G&A costs, $0.88/tonne Ops Support costs, $4.67/ton milling costs, and pit slopes by rock type. The Mineral Reserve is reported at a variable cut-off ranging from 0.17% to 0.21% copper. Tonnage measurements are in metric units. Copper and molybdenum grades are reported as percentages. Contained metal is reported as million pounds. Rounding may result in apparent summation differences between tonnes, grade and contained metal.

The Qualified Person for the Mineral Reserve is Clay Craig, P.Eng., Manager, Mining and Evaluations at Capstone Mining Corp. Scientific and technical information about the Mineral Reserve is based on forward-looking information. Metal price assumptions, resource modelling assumptions, modifying factors applied and risks are explained in the relevant sections of this report. Changes in these could impact the Mineral Reserve in a positive or negative way.

1.7 Mining Methods and Life of Mine Plan

PVM utilizes conventional open-pit hard rock mining methods, employing drilling, blasting, loading and hauling to move copper bearing sulfide ore to the primary crusher. The life of mine plan (LOMP) in this Technical Report continues mine life to 2039. Total mining rates will average 52.6 M tonnes per year from 2021 through 2031, then decrease from 2032 to 2039, as shown in Figure 1-1. No significant changes are made to mining equipment fleets relative to current capacity. The areas mined in this study are the south east, east and north walls of the Pinto Valley pit, along with deepening the pit with every pushback. Waste rock is to be placed on the Main Dump and a new dump named the West Dump, situated in a valley immediately west of the Main Dump.

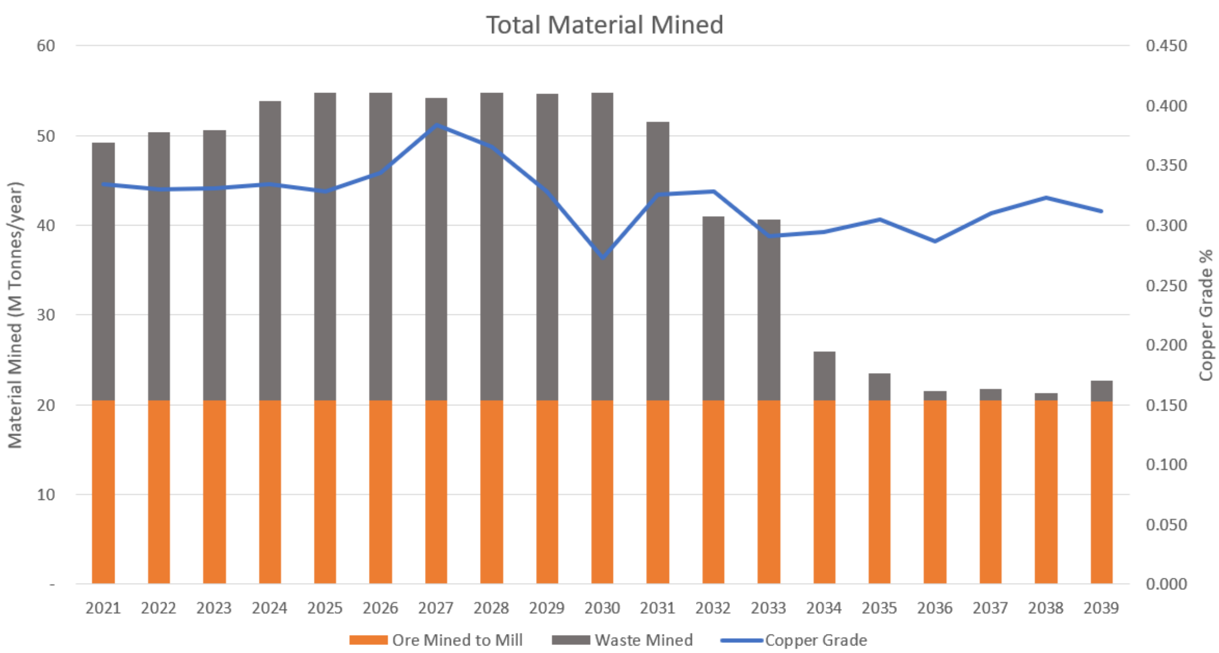


Figure 1-1: Life of Mine Plan – Mill Rate of 56,000 tpd (2021+)

1.8 Permitting

Pinto Valley Mine possesses the requisite permits for continued operation through 2026 at the current mill throughput rate. Additional permits are required to extend the operation to 2039 as discussed in Section 20.

1.9 Conclusions and Recommendations

Pinto Valley Mine is a viable mining operation that began operations in 1974 and has been operated continuously by Capstone since October 2013. Based on the findings summarized in this Technical Report, the QPs believe that PVM is capable of sustaining production through the depletion of its current Mineral Reserve, and make the recommendations listed in Table 1-3. The Mineral Resource and Mineral Reserve estimates were performed to industry best practices as described in CIM (2019) and conform to the definitions in CIM (2014).

Capstone holds all required mining concessions, surface rights, and rights of way to support mining operations through the depletion of the March 31, 2021 Mineral Reserve estimate. Permits held by Capstone and expected to be issued to Capstone in 2021 are sufficient to ensure that mining activities within PVM are carried out within the regulatory framework required by the various levels of government.

Understanding of the regional geology, lithological, structural, and alteration controls of the mineralization at PVM are sufficient to support estimation of the Mineral Resource and Mineral Reserve. The Mineral Resource and Mineral Reserve estimates, copper grade cut-off strategy, and operating and capital cost estimates were generated using industry-accepted methodologies and actual PVM performance standards and operating costs. Metallurgical expectations are reasonable, based on stable metallurgical results generated from actual production data and recently completed studies. Reviews of the environmental, permitting, legal, title, taxation, socio-economic, marketing and political factors for PVM support the declaration of Mineral Reserves.

PVM has several water sources including a private wellfield with multiple wells, a system of water catchments with pumpback capabilities, and reclaim systems on operating tailings impoundments. Nevertheless, extended periods of drought have the potential to impact the operation of PVM.

The design of existing tailings storage facilities provides adequate capacity to store tailings generated through depletion of the current Mineral Reserve. This assumes that proper tailings management continues, including the development of competent tailings beaches and the management of embankment pore water to allow the continued safe construction of the tailings storage facilities.

Based on current regulations and laws, Capstone has addressed PVM's environmental impact. Closure provisions are appropriately considered in the LOMP. There are no known significant environmental, social or permitting issues that are expected to prevent continued mining at PVM.

Table 1-3: Summary of Recommendations

Recommendations – overall implementation cost estimated at US$13 million
Improve QAQC on drillhole analysis through increased insertion of certified reference materials created from PVM material at an approximate cost of US$ 10,000 to be completed over three months.[1]
Further evaluate the geotechnical impact of key structures such as the Bummer fault to the Mineral Reserve and adjust the final pit design. Estimated cost is $100,000, at two to three months to complete.[2]

To improve the performance of the PVM plant[3]:

- Optimize the grinding circuit to reduce the product P80 size. This has a major impact on copper rougher recovery. Estimated cost for modeling and consulting $50,000, six months duration.
- Investigate the potential benefits of improving the cyclone system to reduce bypass and circulating loads. This should improve grinding efficiency and reduce the P80 product size. Estimated cost for modeling and consulting $30,000, three months duration.
- Investigate the fundamental issue with the past poor metallurgical recovery in diabase. Estimated cost for mineralogy and flotation test work $75,000, duration three months.
- Reduced rougher flotation pH (less than pH 10) should be investigated as it will improve the recovery of locked pyrite/chalcopyrite particles. Estimated cost for test work, plant trials and analysis $25,000, duration three months.
- Continue to optimize the molybdenum circuit with the new reagent scheme. Estimated cost for test work, plant trials and analysis $100,000, duration three to four months.
- Investigate the impact of recycling water from the molybdenum circuit using the new reagent suite. Costs and duration are included in the above recommendation.

To improve coppery recovery[3]:

- Upgrade the level control system for the rougher flotation circuit to maximize mass pull and mineral recovery, at an estimated cost of $200,000 and timeline of six months.
- Upgrade the reagent dosing system to improve the reagent addition control, at an estimated cost of $150,000 and timeline of three months.
- Upgrade the rougher concentrate pumps to reduce overflow situations and allow for maximized mass pull at coarser grinds, at an estimated cost of $250,000 with a completion estimate of six to nine months.
- Upgrade the tailings thickeners to more efficient units that yield higher density outputs and greater water recovery. The upgrade is in progress, with completion expected in July 2021 at an estimated cost of $5 to $7 million.

Infrastructure recommendations are focused on water supply. PVM must continue its efforts to maximize water efficiency, increase recycling of water and minimize water usage.[4]

Monitoring and control of the phreatic levels in the TSF embankments is critical to the performance of the facilities. Additional geotechnical field investigations, including cone penetration testing, exploratory drilling, laboratory testing and engineering analyses will be required if phreatic levels in the TSF embankments rise above predicted levels. Methods to control or mitigate the phreatic rise would be developed. A contingency of $6.6 M has been included in the LOMP tailings storage capital cost estimate for these efforts, expended in 2021 through 2034.[5]

To obtain and maintain permits to operate, PVM's personnel should, as part of their regular duties[4]:

- Plan to evaluate the impacts of climate change on PVM's ability to operate within permit terms and conditions by the end of 2022;
- Update the MLRP following USFS issuance of a MPO and authorization to proceed;
- Stay abreast of continuously evolving mining regulatory regime and best practices.

Table 1-5 Notes:

1. QP Garth Kirkham, P.Geo., FGC
2. QP Clay Craig, P.Eng.
3. QP J. Todd Harvey, SME-RM
4. QP Colleen Roche, P.Eng.
5. QP Tony J. Freiman, PE

Opportunities identified for PVM by the QPs are presented in Table 1-4.

Table 1-4: Summary of Opportunities

Opportunities
Upgrade classification of the current Inferred Resource to Indicated class by decreasing the drill hole spacing with future drill programs.[1]
Continue regional exploration and property evaluations within reasonable trucking distance of the plant.[1]
Evaluate steps required to include gold and silver in the Mineral Resource estimate[1] and the Mineral Reserve estimate.[2]
Optimize plant throughput with improvements to the crushing circuit, finer blast fragmentation and the rougher flotation circuit performance.[3]
Complete planned studies into technologies that could potentially enhance dump leach performance and increase copper cathode production to 300 to 350 million pounds from mineralized waste over the LOM, such as the patented catalytic technology from Jetti Resources (expected 2022), coarse particle flotation technology by Eriez Flotation Division (expected the second half of 2021), and pyrite agglomeration (expected 2022). [3]
Improve current site infrastructure through continuous maintenance and possible use of large scale solar power.[4]
Continue to review the mining fleet, investigate innovative technologies to maximize fuel usage, explore debottlenecking scenarios and complete the blast fragmentation study to optimize operational performance.[2]
Perform progressive reclamation of tailings storage facility embankments and waste rock dumps.[4]
Consider alternative tailings storage approaches to allow greater throughput and potentially expand the value of the PVM Mineral Reserve[2].
Advance tailings management practices by working towards conformity with the Global Industry Standard on Tailings Management.[5]
Continue to foster collaborative relationships with stakeholders and peers to maximize benefits of the project[4]

Table 1-4 Notes:

1. QP Garth Kirkham, P.Geo., FGC
2. QP Clay Craig, P.Eng.
3. QP J. Todd Harvey, SME-RM
4. QP Colleen Roche, P.Eng.
5. QP Tony J. Freiman, PE

Risks to PVM identified by the QPs are summarized in Table 1-5. While there are significant risks with all mining projects, many of those risks are mitigated as PVM is in full production with the required infrastructure already in place.

Table 1-5: Summary of Risks

Risks
Revisions to the Mineral Resource[1] and Mineral Reserve[2] estimates caused by changes in assumptions, including changes to geological interpretations after new drilling, and possible impacts of Inferred Mineral Resource material encountered while mining the Mineral Reserve.[1]
Impacts of financial market conditions, including supply, demand and prices of base metals, goods and services to the Mineral Reserve.[2]

Impacts of regulatory and permitting environment complexities, changes and delays, especially concerning mining activities, tailings storage and the closure plan.[2,4,5]
Water supply shortages and/or regulatory changes impacting water supply required for processing under the current LOMP[3] could also impact the Mineral Reserve[2]
Design pit slope angles could be unsuitable for actual mining conditions and other unforeseen geotechnical conditions may occur.[2]
Use of new reagents for the molybdenum circuit are not guaranteed to perform as expected when applied at the PVM plant as it is currently configured.[3]
Given the age of PVM infrastructure such as buildings and facilities including the mill, unforeseen failures may occur[3] and investment is needed to ensure long-term reliability[4].
Feasibility of continued upstream embankment raises and future tailings storage capacity could be jeopardized if consistent tailings management procedures are not strictly followed to ensure the development of competent tailings beaches and to control embankment pore water pressures.[5]
Delay in receiving USFS authorization for mining activities described in the MPO (expected October 2021) and delays or inability to obtain approval for closure plan prior to planned closure activities.[4]

Table 1-5 Notes:
1. QP Garth Kirkham, P.Geo., FGC
2. QP Clay Craig, P.Eng.
3. QP J. Todd Harvey, SME-RM
4. QP Colleen Roche, P.Eng.
5. QP Tony J. Freiman, PE

2 Introduction

2.1 Description of the Issuer

Capstone Mining Corp. is a Canadian-based metals mining company committed to the responsible development of the company's assets. Capstone's focus is on the production of copper at two producing mines, including Pinto Valley Mine in the United States and Cozamin Mine in Mexico. Capstone also owns 100% of the Santo Domingo Project in Chile, and a portfolio of exploration properties. Capstone's headquarters are in Vancouver, Canada, and the company is listed on the Toronto Stock Exchange. Further information is available at www.capstonemining.com.

This Technical Report was prepared by Capstone to disclose an updated Mineral Resource and Mineral Reserve for Pinto Valley Mine in Gila County, Arizona, USA. Pinto Valley Mine is an operating open-pit copper-molybdenum mine with a nominal throughput of 56,000 tonnes per day, and was acquired by Capstone in October 2013.

2.2 Terms of Reference

This Technical Report discloses an updated Mineral Resource and Mineral Reserve for Pinto Valley Mine. The Mineral Reserve presented herein was updated to reflect changes since the previous Technical Report filed on February 23, 2016 with an effective date of January 1, 2016 (PV3-2016-PFS), including mining and drilling activities.

Preparation of this Technical Report followed National Instrument 43-101, Standards of Disclosure for Mineral Projects ("NI 43-101") and the report was written in accordance with Form 43-101F1. Estimates of the Mineral Resource and Mineral Reserve follow industry best practices as defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM, 2019). Classification of the Mineral Resource and Mineral Reserve conform to CIM Definition Standards (CIM, 2014). The effective date of this Technical Report is March 31, 2021.

2.3 Qualified Persons

QPs for this Technical Report are listed in Table 2-1.

Table 2-1: Qualified Persons for this Technical Report

Qualified Persons
Clay Craig, P.Eng., Manager, Mining and Evaluations, Capstone Mining Corp., not Independent within the meaning of NI 43-101.
Tony Freiman, PE, Principal Geotechnical Engineer, Wood, Environment & Infrastructure Solutions, Inc. Independent within the meaning of NI 43-101.
J. Todd Harvey, SME-RM, Global Research Engineering, Ltd. Independent within the meaning of NI 43-101.
Garth Kirkham, P.Geo., FGC, President, Kirkham Geosystems Ltd. Independent within the meaning of NI 43-101.
Colleen Roche, P.Eng., Operations Support Manager, Capstone Mining Corp., not Independent within the meaning of NI 43-101.

Qualified Persons
Klaus Triebel, CPG, Chief Resource Modeler, Capstone Mining Corp., not Independent within the meaning of NI 43-101
Edward C. Wellman, PE, PG, CEG, Principal Geomechanic, Independent Geomechanics LLC. Independent within the meaning of NI 43-101.

2.4 Qualified Person Site Visits

Site inspections were undertaken by each of the QPs listed in Table 2-1. Dates listed do not include travel time associated with the visit.

Clay Craig, P.Eng., Manager, Mining and Evaluations, Capstone Mining Corp., visits PVM at least weekly since October 2020. For the purposes of this Technical Report, the scope of recent site visits included review of inputs required for Mineral Reserve estimation including permitting and mineral tenure, long range planning, confirmation of costs and general site overview with respect to execution of the mine plan. Clay Craig's prior involvement with PVM was as Superintendent, Technical Services from October 2017 to October 2020. Clay Craig is not Independent within the meaning of NI 43-101.

Tony Freiman, PE, Principal Geotechnical Engineer, Wood, Environment & Infrastructure Solutions, Inc. has had involvement with PVM since 1997 and continues to visit quarterly. His most recent visit was on May 6, 2021 to conduct the Engineer of Record quarter annual inspection of the active tailing storage facilities and on February 9 and 10, 2021 to conduct the Engineer of Record annual inspection of the active and inactive tailings storage facilities. Tony Freiman is Independent within the meaning of NI 43-101.

J. Todd Harvey, SME-RM, Global Research Engineering, Ltd. has visited PVM on multiple occasions over the last several years including as recently as May 4 and 5, 2021. The scope of the visits included a review of mill operating data, processing circuit and equipment, MetSim Model development, and an evaluation of in-progress metallurgy studies. J. Todd Harvey is Independent within the meaning of NI 43-101.

Garth Kirkham, P.Geo., FGC, President, Kirkham Geosystems Ltd. has had involvement with PVM since 2013, most recently visiting the property April 15 and 16, 2015 to view mineralization in the pit, inspect the mine site infrastructure, core logging and processing facilities and core storage facilities. Garth Kirkham is Independent within the meaning of NI 43-101.

Colleen Roche, P.Eng., Operations Support Manager, Capstone Mining Corp., has worked at PVM since 2015 and is on-site weekly. For the purposes of this Technical Report, the scope of the most recent visit April 26 to 29, 2021 included review of data associated with land ownership, mineral rights, permitting, environmental monitoring and site infrastructure inspection. Colleen Roche is not Independent within the meaning of NI 43-101.

Klaus Triebel, CPG, Chief Resource Modeler, Capstone Mining Corp., has worked at PVM since 2018 and is on-site weekly. For the purposes of this Technical Report, the scope of the most recent visit April 26 to 29, 2021 included review of geological setting and mineral estimation data including drilling operations, exploration, sample analysis and density measurements. Klaus Triebel is not Independent within the meaning of NI 43-101.

Edward C. Wellman, PE, PG, CEG, Principal Geomechanic, Independent Geomechanics LLC, has visited PVM periodically since 2006 associated with work on slope stability analysis. His most recent site visit was March 19, 2021 where he reviewed pit geotechnical conditions. Edward C. Wellman is Independent within the meaning of NI 43-101.

2.5 Information Sources, Effective Dates and References

The effective date of this report is based on the Mineral Resource and Mineral Reserve estimates dated March 31, 2021.

Sources of data for the report and the corresponding effective dates are as follows:

- Drilling information including collar surveys, downhole surveys, geological and geotechnical logging and assays up to December 16, 2019
- Production and processing information, from historical operators pre-2013 and collected by Capstone from 2013 up until March 31, 2021, including the month-end production survey dated March 31, 2021 used in reporting the Mineral Resource and Mineral Reserve estimates
- Mineral Resource and Mineral Reserve estimates: March 31, 2021
- Environmental, regulatory and social or community aspects to April 30, 2021
- Maintenance of mining concessions to June 4, 2021
- Metallurgical test work to April 30, 2021
- Geotechnical inputs including pit slope performance data, geotechnical core logging data, laboratory strength testing, geotechnical standard operating procedures and ground control design procedures, collected from historical operators pre-2013 and by Capstone from 2013 to April 30, 2021

In addition, other reports, opinions and statements of lawyers and other experts are discussed in Section 3.

Units in this report include SI (metric) system (Système International d'Unités - International System of Units) except where otherwise noted due to customary use at PVM and for some units which are deemed industry standards, such as troy ounces (oz) for precious metals and pounds (lb) for base metals. All currency values are in US dollars.

Table 2-2 defines acronyms that are used in this Technical Report.

Table 2-2: Acronyms

Acronym	Name
ADEQ	Arizona Department of Environmental Quality
ADWR	Arizona Department of Water Resources

Acronym	Name
Ag	silver
AMEC	AMEC or Amec Foster Wheeler, now Wood
APP	Aquifer Protection Permit
ASCu	acid-soluble copper
ASMIO	Arizona State Mine Inspector’s Office
Au	gold
AZPDES	Arizona Pollutant Discharge Elimination System
BHP	BHP Billiton Ltd.
BLM	Bureau of Land Management
BWi	Bond ball work index
CAA	Clean Air Act
Capstone	Capstone Mining Corp.
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CPP	cumulative probability plot
Cu	copper
Cu-Mo	copper-molybdenum
CWA	Clean Water Act
dmt	dry metric tonne
ESA	Endangered Species Act
F	Fahrenheit
FEIS	Final Environmental Impact Statement
FLEET	Flotation Economic Evaluation Tool
FMI	Freeport-McMoRan Copper & Gold Inc.
FOB	fine ore bin
FOS	factor of safety
FR	Forest Road
ft	foot
G&A	general and administrative
gpm	gallons per minute
GSI	geological strength index
hp	horsepower
HSEC	Health, Safety, Environment, and Community
IFRS	International Financial Reporting Standards
IMC	Independent Mining Consultants, Inc.
IT	information technology
kt	kilotonne (1,000 metric tonnes)
kW	kilowatt
kWh	kilowatt hour
lb	pound
LOM	life of mine

Acronym	Name
LOMP	life of mine plan
m	meter
M	million
Ma	megaannus (million years)
MBWi	modified Bond work index
mi^2	square miles
MLRP	Mined Land Reclamation Plan
mm	millimeter
Mo	molybdenum
MSG	Multi-Sector General
MSHA	Mine Safety and Health Administration
M tonnes	million tonnes (metric)
NEPA	National Environmental Policy Act
NFS	National Forest System
NI 43-101	National Instrument 43-101 – Standards of Disclosure for Mineral Projects
NN	Nearest Neighbor
NPV	net present value
NSR	net smelter return
OSA	maximum overall slope angle
PLS	pregnant leach solution
POC	point-of-compliance
PV2	Pinto Valley Phase 2 Study – mine life to 2026
PV3	Pinto Valley Phase 3 Study – mine life to 2039
PVM	Pinto Valley Mine
PVMC	Pinto Valley Mining Corp.
PVO	Pinto Valley Operation (Pinto Valley Mine under BHP ownership)
QA/QC	quality assurance / quality control
QP	Qualified Person
Quadra	Quadra FNX Mining Ltd.
RC	reverse circulation
RHD	relative half difference
RMR	rock mass rating
ROD	Record of Decision
ROM	run-of-mine
RQD	rock quality designation
Stpd	short tons per day
Skyline	Skyline Assayers and Laboratories
SMARRCO	San Manuel Arizona Railroad Company
SMU	selective mining unit

Acronym	Name
SR 77	State Route 77
SRK	SRK Consulting (U.S.), Inc.
SRP	Salt River Project
SWPPP	Stormwater Pollution Prevention Plan
SX-EW	solvent extraction and electrowinning
t	metric tonne
TC/RC	treatment charge / refining charge
TNF	Tonto National Forest
ton	short ton (2000 lb)
tonne	metric tonne (1000 kg)
tpd	tonnes per day
tph	tonnes per hour
TSF	tailings storage facility
UCS	uniaxial compressive strength
US$	US Dollars
US 60	US Highway 60
USACE	US Army Corps of Engineers
USFS	US Forest Service
USFWS	US Fish and Wildlife Service
VFD	variable frequency drive
WRI	WestLand Resources Inc.
WWTP	wastewater treatment plant
yd	yard
yr	year
~	approximately
°	degrees
µm	microns
σ	standard deviation

2.6 Previous Technical Reports

Capstone has previously filed the following Technical Reports on PVM:

- Bush, G., Freiman, T., Hoag, C., Kirkham, G., Major, K., Marek, J., 2016: Pinto Valley Mine Life Extension – Phase 3 (PV3) Pre-Feasibility Study NI 43-101 Technical Report on the Pinto Valley Mine, Miami, Arizona: technical report prepared by Amec Foster Wheeler Environment & Infrastructure Inc., Kirkham Geosystems Ltd., KWM Consulting Inc., Independent Mining Consultants, Inc. and SRK Consulting (U.S.), Inc. for Capstone Mining Corp., effective Date: January 1, 2016
- Freiman, T., Hoag, C., Kirkham, G., Lawson, M., Major, K., Marjorkiewicz, A., Marek, J., 2014: Pinto Valley Mine 2014 Prefeasibility Study NI 43-101 Technical Report on the

Pinto Valley Mine, Miami, Arizona: technical report prepared by Stantec – Mining, Amec Environment & Infrastructure, Inc., SRK Consulting (U.S.), Inc., Kirkham Geosystems Ltd., KWM Consulting Inc., Adam M Consulting Inc. and Independent Mining Consultants, Inc. for Capstone Mining Corp., effective Date: January 1, 2014

- Kirkham, G., 2013: Pinto Valley Property Mineral Resource Estimate NI 43-101 Technical Report (Amended and Restated): technical report prepared by Kirkham Geosystems Ltd. for Capstone Mining Corp., effective Date: February 28, 2013

3 Reliance on Other Experts

In preparing this Technical Report, the QPs have fully relied upon certain work, opinions and statements of lawyers and other experts. The authors consider the reliance on other experts, as described in this section, as being reasonable based on their knowledge, experience and qualifications. Expert content was incorporated in the following sections:

- Marc A. Marra, Counsel, Fennemore, for a legal opinion pertaining to the ownership of mining concessions by PVMC in Section 4.2 (June 7, 2021).
- Ashley Woodhouse, Director, Marketing of Capstone Mining Corp., for specialized commodity market knowledge summarized in Section 19.
- Timothy Ralston, CHMM, REM, Manager Land, Permitting & Regulatory Affairs of PVMC for environmental and regulatory considerations detailed in Section 20.

4 Property Description and Location

PVM owns approximately 10 square miles (mi[2]) of patented land, 467 unpatented mining claims around the perimeter of the patented land and a 27 acre ranch including 33,000 acre grazing allotment within the Tonto National Forest administered by the U.S. Department of Agriculture - Forest Service (USFS).

4.1 Location

PVM is located at the west end of the Globe-Miami mining district, approximately 11 miles west of Globe and 80 miles east of Phoenix, Arizona via U.S. Highway 60, at 33°23′32″ N and 110°58′15″ W (Figure 4-1).

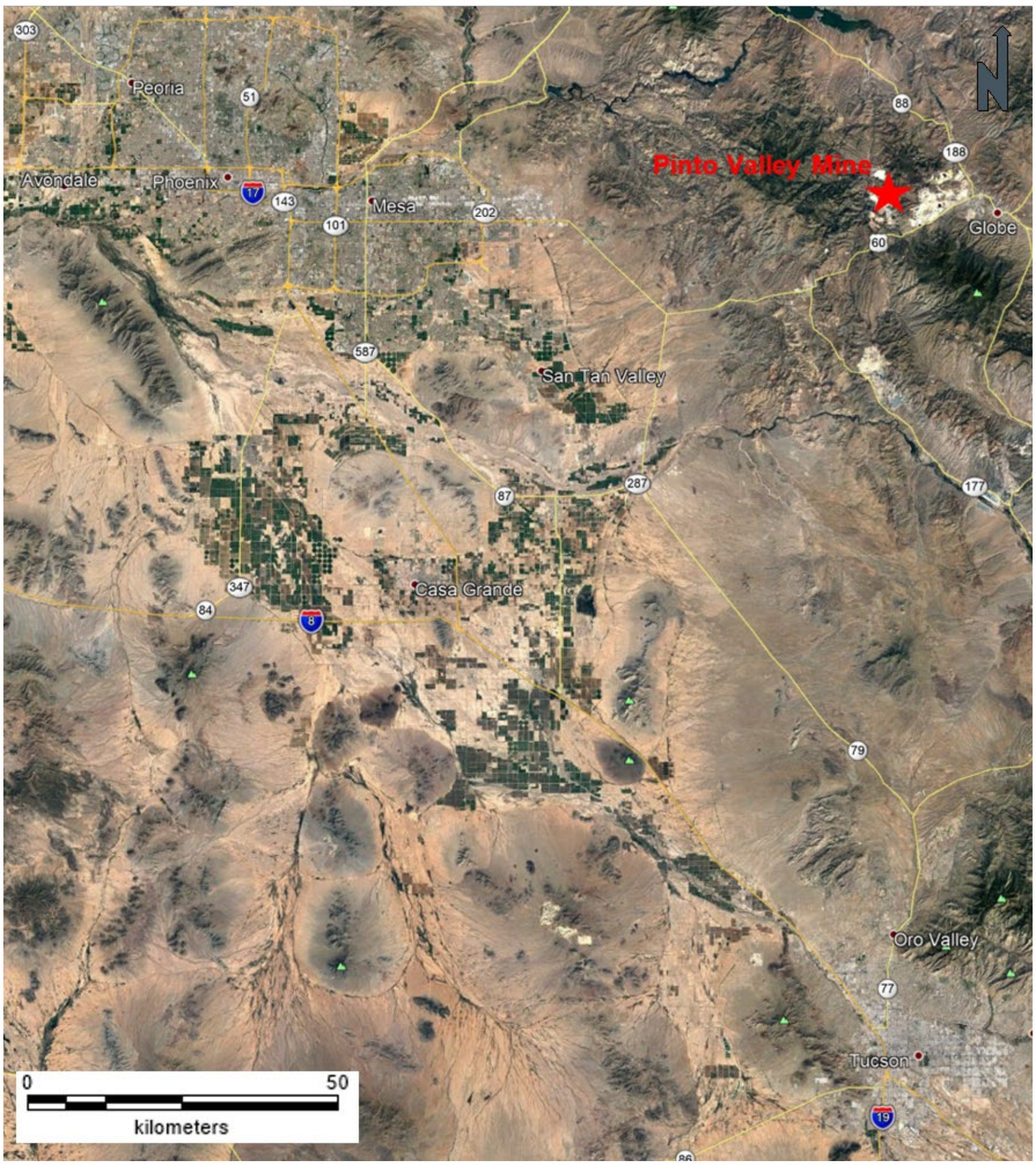


Figure 4-1: Pinto Valley Mine Location Map (Google Earth, 2021)

4.2 Tenure, Ownership and Encumbrances

PVM is a combination of fee land, patented mining and mill site claims, and unpatented mining (lode) and mill site claims, as shown in Figure 4-2. As a whole, the land supports open-pit mining, ore processing, tailings storage, waste rock disposal, and the operation of an SX-EW plant. Operations on the unpatented lode claims and mill sites are pursuant to the provisions of the US General Mining Law of 1872, subject to plan of operations and other use authorization approvals from the USFS. The document is supported by a proposed Plan of Operations. Use of the unpatented mining claims and mill sites will be governed by the final Plan of Operations.

PVMC owns the patented lands in fee parcels, which provides for ownership of both surface and mineral rights (Table 4-1). The fee lands are located by legal description and recorded at the Gila County Recorder's Office. These lands are subject to annual property tax payments. The unpatented mining claims and mill sites are subject to annual claims maintenance fee payments and notice of intent to hold mining claims.

Adjacent to and nearby the patented land are 467 unpatented lode mining claims (Table 4-2) and mill sites (Table 4-3). The unpatented mining claims and mill sites were staked on federal land administered by the USFS.

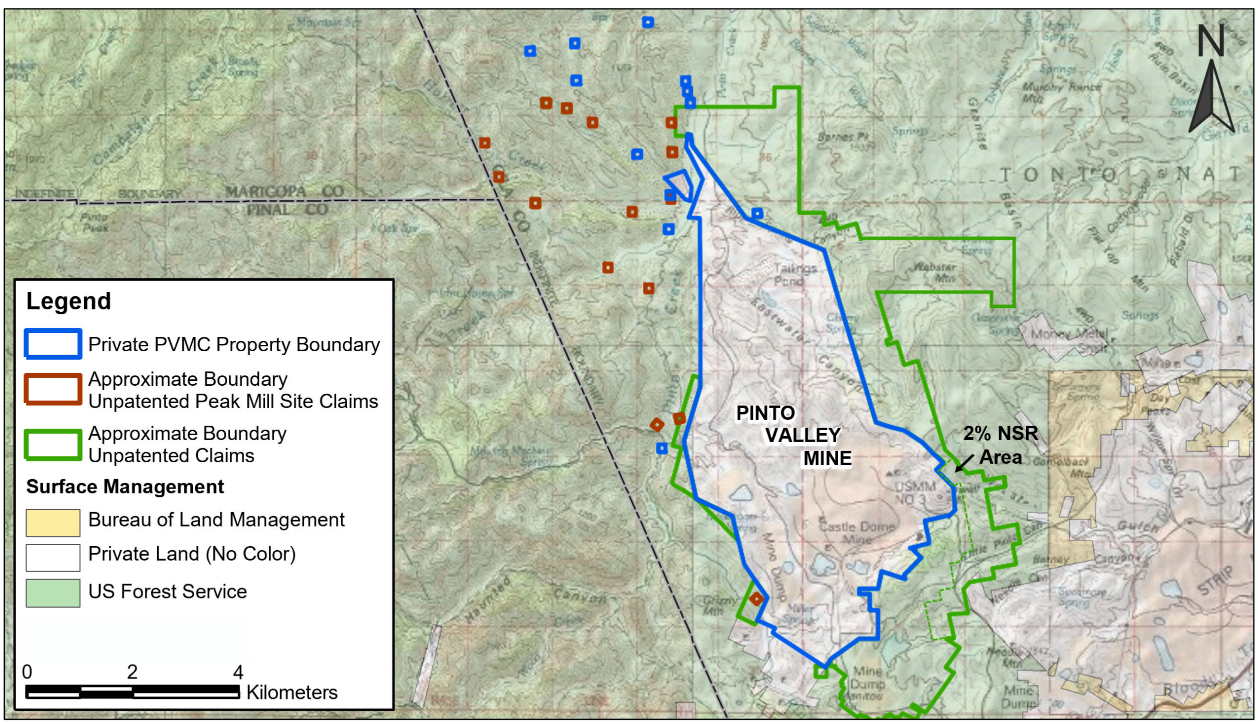


Figure 4-2: Pinto Valley Tenure

Table 4-1: PVM Patented Land (includes patented lode and mill site claims)

Tax Parcel No.	Description/Location
204-10-007E	Tract 40, according to Map No. 474, Gila County, Arizona except that portion lying south and west of a straight line between the northwest corner of the Lizard #27 unpatented claim (A.M.C. 28018 recorded at Book 47, Records of Mines, page 348 of the Official Records of Gila County, Arizona) and the northeast corner of the Alto patented claim (Mineral Survey No. 2676, Patent No. 167985)
204-10-007A	Part of Tract 40, according to Map No. 474, Gila County, Arizona
204-10-007B	A parcel of land within Tract 40 according to Map No. 474, Gila County, Arizona
204-10-004	Tract 41, according to Map No. 474, Gila County, Arizona
204-10-006	A portion of Homestead Entry Survey No. 71, as shown on plat on file in the B.L.M. as granted by Patent recorded in Book 21, page 465, Gila County, Arizona
204-10-005	Layton Ranch consisting of 27.570 acres, being part of Homestead Entry Survey No. 71, as shown on plat on file in the B.L.M. as granted by Patent recorded in Dkt 57, page 314, Gila County, Arizona
204-10-002	Homestead Entry Survey No. 441, as shown on plat on file in the B.L.M. as granted by Patent recorded in Dkt 57, page 314, Gila County, Arizona

Table 4-2: PVM Unpatented Claims – Lode Claims (270)

Lode Claim	BLM Serial No.	Lode Claim	BLM Serial No.
BOB 1	AMC384552	PVMC 3	AMC449741
BOB 2	AMC384553	PVMC 4	AMC449742
BOB 3	AMC384554	PVMC 8	AMC449743
BOB 4	AMC384555	PVMC 9	AMC449744
BOB 5	AMC384556	PVMC 22	AMC449757
BOB 6	AMC384557	PVMC 23	AMC449758
COPPER BOTTOM NO 1	AMC27751	PVMC 24	AMC449759
COPPER BOTTOM #2 A[1]	AMC35830	PVMC 25	AMC449760
COPPER BOTTOM NO 5 A[1]	AMC35831	PVMC 26	AMC449761
COPPER BOTTOM NO 6	AMC27752	PVMC 27	AMC449762
COPPER BOTTOM #7 A[1]	AMC35832	PVMC 28	AMC449763
COPPER BOTTOM NO 8	AMC27753	PVMC 29	AMC449764
COPPER BOTTOM NO 9	AMC27754	PVMC 30	AMC449765
COPPER BOTTOM NO 10	AMC27755	PVMC 31	AMC449766
COPPER BOTTOM #11 A[1]	AMC35833	PVMC 32	AMC449767
COPPER BOTTOM #12 A[1]	AMC35834	PVMC 33	AMC449768
COPPER BOTTOM #13 A[1]	AMC35835	PVMC 34	AMC449769
COPPER SITE AMENDED	AMC27742	PVMC 35	AMC449770
COWBOY NO 1 AMENDED	AMC27738	PVMC 36	AMC449771
COWBOY NO 2 AMENDED	AMC27739	PVMC 37	AMC449772
COWBOY NO 3 AMENDED	AMC27740	PVMC 38	AMC449773
CWL 1	AMC422569	PVMC 39	AMC449774
CWL 2	AMC422570	PVMC 40	AMC449775
CWL 3	AMC422571	PVMC 41	AMC449776

Lode Claim	BLM Serial No.
CWL 4	AMC422572
CWL 5	AMC422573
CWL 6	AMC422574
CWL 7	AMC422575
CWL 8	AMC422576
CWL 9	AMC422577
CWL 10	AMC422578
CWL 11	AMC422579
CWL 12	AMC422580
CWL 13	AMC422581
CWL 14	AMC422582
CWL 15	AMC422583
CWL 16	AMC422584
CWL 17	AMC422585
CWL 18	AMC422586
CWL 19	AMC422587
CWL 20	AMC422588
CWL 21	AMC422589
D 1	AMC215329
D 2	AMC215330
D 3	AMC215331
D 4	AMC215332
D 5	AMC215333
D 6	AMC215334
D 7	AMC215335
D 8	AMC215336
D 9	AMC215337
D 10	AMC215338
D 11	AMC215339
D 12	AMC215340
D 13	AMC215341
D 16	AMC215344
D 17	AMC215345
E-1	AMC364145
E-2	AMC364146
E-3	AMC364147
HILL NO 2	AMC27757
HILL NO 3	AMC27758
HILL NO 4	AMC27759
HILL NO 5	AMC27760
HILL NO 6	AMC27761
HILL NO 7	AMC27762
HILL NO 8	AMC27763
HILL NO 9	AMC27764
HAMMER	AMC27747
HILL NO 10	AMC27765
HILL NO 11	AMC319065
HILL NO 12	AMC319066
HILL NO 13	AMC319067
E-4	AMC364148

Lode Claim	BLM Serial No.
PVMC 42	AMC449777
PVMC 43	AMC449778
PVMC 44	AMC449779
PVMC 45	AMC449780
PVMC 46	AMC449781
PVMC 47	AMC449782
PVMC 48	AMC449783
PVMC 49	AMC449784
PVMC 50	AMC449785
PVMC 51	AMC449786
PVMC 52	AMC449787
PVMC 53	AMC449788
PVMC 54	AMC449789
PVMC 55	AMC449790
PVMC 56	AMC449791
PVMC 57	AMC449792
PVMC 58	AMC449793
QUARTZ SITE (A)[1]	AMC35792
QUARTZ SITE NO 2(A)[1]	AMC35793
QUARTZ SITE NO 3(A)[1]	AMC35794
QUARTZ SITE NO 9(A)[1]	AMC35800
QUARTZ SITE NO 10(A)[1]	AMC35801
QUARTZ SITE NO 15(A)[1]	AMC35806
QUARTZ SITE NO 16(A)[1]	AMC35807
QUARTZ SITE NO 17(A)[1]	AMC35808
QUARTZ SITE NO 18(A)[1]	AMC35809
QUARTZ SITE NO 19(A)[1]	AMC35810
QUARTZ SITE NO 20(A)[1]	AMC35811
QUARTZ SITE NO 21(A)[1]	AMC35812
QUARTZ SITE NO 22(A)[1]	AMC35813
QUARTZ SITE NO 23(A)[1]	AMC35814
QUARTZ SITE NO 24(A)[1]	AMC35815
QUARTZ SITE NO 25(A)[1]	AMC35816
QUARTZ SITE #28(A)[1]	AMC35820
RITA	AMC27741
RJ #1	AMC370110
RJ #2	AMC370111
RJ #3	AMC370112
RJ #4	AMC370113
RJ #5	AMC370114
RJ #6	AMC370115
RJ #7	AMC370116
RJ #8	AMC370117
RJ #9	AMC370118
RJ #10	AMC370119
RJ #11	AMC370120
RJ #12	AMC370121
RJ #13	AMC370122
RJ #14	AMC370123
RJ #15	AMC370124

Lode Claim	BLM Serial No.	Lode Claim	BLM Serial No.
EAST WATER	AMC40744	RJ #16	AMC370125
JANIE NO 1 A[1]	AMC35824	RJ #17	AMC370126
JANIE NO 2 A[1]	AMC35825	RJ #18	AMC370127
JOAN	AMC27748	RJ #19	AMC370128
K 1	AMC138512	RJ #20	AMC370129
K 10	AMC138521	RJ #21	AMC370130
K 11	AMC138522	RJ #22	AMC370131
K 12	AMC138523	RJ #23	AMC370132
K 13	AMC138524	RJ #24	AMC370133
K 14	AMC138525	RJ #25	AMC370134
K 15	AMC138526	RJ #26	AMC370135
K 16	AMC138527	RJ #27	AMC370136
K 17	AMC138528	RJ #28	AMC370137
K 18	AMC138529	RJ #29	AMC370138
K 19	AMC138530	RJ #30	AMC370139
K 2	AMC138513	RJ #31	AMC370140
K 20	AMC138531	RJ #32	AMC370141
K 21	AMC138532	RJ #33	AMC370142
K 22	AMC138533	RJ #34	AMC370143
K 23	AMC138534	RJ #35	AMC370144
K 24	AMC138535	RJ #36	AMC370145
K 25	AMC138536	RJ #37	AMC370146
K 26	AMC138537	RJ #38	AMC370147
K 27	AMC138538	RJ #39	AMC370148
K 28	AMC138539	RJ #40	AMC370149
K 29	AMC138540	RJ #41	AMC370150
K 3	AMC138514	RJ #42	AMC370151
K 30	AMC138541	RJ #43	AMC370152
K 31	AMC138542	RJ #44	AMC370153
K 32	AMC138543	RJ #45	AMC370154
K 33	AMC138544	RJ #46	AMC370155
K 34	AMC138545	RJ #47	AMC370156
K 35	AMC138546	RJ #48	AMC370157
K 4	AMC138515	RJ #49	AMC370158
K 5	AMC138516	RJ #50	AMC370159
K 6	AMC138517	RJ #51	AMC370160
K 7	AMC138518	RJ #52	AMC370161
K 8	AMC138519	RJ #53	AMC370162
K 9	AMC138520	RJ #54	AMC370163
MARY 1	AMC7735	RJ #55	AMC370164
MARY 2	AMC7736	RJ #56	AMC370165
MARY 3	AMC7737	RJ #57	AMC370166
MARY 4	AMC7738	RJ #58	AMC370167
MARY 5	AMC7739	RJ #59	AMC370168
MARY 6	AMC7740	RJ #60	AMC370169
MIDNIGHT TEST	AMC27750	RJ #61	AMC370170
PINE TREE AMENDED	AMC27743	RJ #62	AMC370171
PVMC 1	AMC449739	RJ #63	AMC370172
PVMC 2	AMC449740	RJ #64	AMC370173
PVMC 10	AMC449745	RJ 65	AMC384697

Lode Claim	BLM Serial No.	Lode Claim	BLM Serial No.
PVMC 11	AMC449746	RJ 66	AMC384698
PVMC 12	AMC449747	RJ 67	AMC384699
PVMC 13	AMC449748	RJ 68	AMC384700
PVMC 14	AMC449749	RJ 69	AMC384701
PVMC 15	AMC449750	RJ 70	AMC384702
PVMC 16	AMC449751	RJ 71	AMC384703
PVMC 17	AMC449752	RJ 72	AMC384704
PVMC 18	AMC449753	SILVER BELL (A)[1]	AMC35821
PVMC 19	AMC449754	SOUTHERN CROSS	AMC27749
PVMC 20	AMC449755	TUNNEL	AMC27994
PVMC 21	AMC449756	ZEE NO 2	AMC28059

1. Lode claim subject to a 2% NSR royalty payable to William E. Bohme and Eula Belle Bohme (half interest) and Patricia M. Green.

Table 4-3: PVM Unpatented Claims – Mill Site Claims (197)

Mill Site Claim	BLM Serial No.	Mill Site Claim	BLM Serial No.
CWM 1	AMC422590	EWC28	AMC460547
CWM 10	AMC422599	EWC29	AMC460548
CWM 11	AMC422600	EWC3	AMC460522
CWM 12	AMC422601	EWC30	AMC460549
CWM 13	AMC422602	EWC31	AMC460550
CWM 14	AMC422603	EWC32	AMC460551
CWM 15	AMC422604	EWC33	AMC460552
CWM 16	AMC422605	EWC34	AMC460553
CWM 17	AMC422606	EWC35	AMC460554
CWM 18	AMC422607	EWC36	AMC460555
CWM 19	AMC422608	EWC37	AMC460556
CWM 2	AMC422591	EWC38	AMC460557
CWM 20	AMC422609	EWC39	AMC460558
CWM 21	AMC422610	EWC4	AMC460523
CWM 22	AMC422611	EWC40	AMC460559
CWM 23	AMC422612	EWC41	AMC460560
CWM 24	AMC422613	EWC42	AMC460561
CWM 25	AMC422614	EWC43	AMC460562
CWM 26	AMC422615	EWC44	AMC460563
CWM 27	AMC422616	EWC45	AMC460564
CWM 28	AMC422617	EWC46	AMC460565
CWM 29	AMC422618	EWC47	AMC460566
CWM 3	AMC422592	EWC48	AMC460567
CWM 30	AMC422619	EWC49	AMC460568
CWM 31	AMC422620	EWC5	AMC460524
CWM 32	AMC422621	EWC50	AMC460569
CWM 33	AMC422622	EWC51	AMC460570
CWM 34	AMC422623	EWC52	AMC460571
CWM 35	AMC422624	EWC53	AMC460572
CWM 36	AMC422625	EWC54	AMC460573
CWM 37	AMC422626	EWC55	AMC460574
CWM 38	AMC422627	EWC56	AMC460575
CWM 39	AMC422628	EWC57	AMC460576

Mill Site Claim	BLM Serial No.	Mill Site Claim	BLM Serial No.
CWM 4	AMC422593	EWC58	AMC460577
CWM 40	AMC422629	EWC59	AMC460578
CWM 41	AMC422630	EWC6	AMC460525
CWM 42	AMC422631	EWC60	AMC460579
CWM 43	AMC422632	EWC61	AMC460580
CWM 44	AMC422633	EWC62	AMC460581
CWM 45	AMC422634	EWC63	AMC460582
CWM 46	AMC422635	EWC64	AMC460583
CWM 47	AMC422636	EWC65	AMC460584
CWM 48	AMC422637	EWC66	AMC460585
CWM 49	AMC422638	EWC67	AMC460586
CWM 5	AMC422594	EWC68	AMC460587
CWM 50	AMC422639	EWC69	AMC460588
CWM 51	AMC422640	EWC7	AMC460526
CWM 52	AMC422641	EWC70	AMC460589
CWM 53	AMC422642	EWC71	AMC460590
CWM 54	AMC422643	EWC72	AMC460591
CWM 55	AMC422644	EWC73	AMC460592
CWM 56	AMC422645	EWC74	AMC460593
CWM 57	AMC422646	EWC75	AMC460594
CWM 58	AMC422647	EWC76	AMC460595
CWM 59	AMC422648	EWC77	AMC460596
CWM 6	AMC422595	EWC78	AMC460597
CWM 60	AMC422649	EWC79	AMC460598
CWM 61	AMC422650	EWC8	AMC460527
CWM 62	AMC422651	EWC80	AMC460599
CWM 63	AMC422652	EWC81	AMC460600
CWM 64	AMC422653	EWC82	AMC460601
CWM 65	AMC422654	EWC9	AMC460528
CWM 66	AMC422655	M1	AMC142565
CWM 67	AMC422656	P 1	AMC327204
CWM 68	AMC422657	P 10	AMC327213
CWM 69	AMC422658	P 11	AMC327214
CWM 70	AMC422659	P 12	AMC327215
CWM 71	AMC422660	P 13	AMC327216
CWM 72	AMC422661	P 14	AMC327217
CWM 73	AMC422662	P 15	AMC327218
CWM 74	AMC422663	P 16	AMC327219
CWM 75	AMC422664	P 17	AMC327220
CWM 76	AMC422665	P 18	AMC327221
CWM 77	AMC422666	P 2	AMC327205
CWM 78	AMC422667	P 3	AMC327206
CWM 79	AMC422668	P 4	AMC327207
CWM 8	AMC422597	P 5	AMC327208
CWM 80	AMC422669	P 6	AMC327209
CWM 9	AMC422598	P 7	AMC327210
EWC1	AMC460520	P 8	AMC327211
EWC10	AMC460529	P 9	AMC327212
EWC11	AMC460530	PEAK 34	AMC97644
EWC12	AMC460531	PEAK 34-B	AMC460483

Mill Site Claim	BLM Serial No.	Mill Site Claim	BLM Serial No.
EWC13	AMC460532	PEAK 8-B	AMC460482
EWC14	AMC460533	PEAK 94	AMC129535
EWC15	AMC460534	PEAK NO 10	AMC28028
EWC16	AMC460535	PEAK NO 14	AMC28031
EWC17	AMC460536	PEAK NO 15	AMC28032
EWC18	AMC460537	PEAK NO 17	AMC28033
EWC19	AMC460538	PEAK NO 22	AMC28036
EWC2	AMC460521	PEAK NO 24	AMC28037
EWC20	AMC460539	PEAK NO 25	AMC28038
EWC21	AMC460540	PEAK NO 28	AMC28040
EWC22	AMC460541	PEAK NO 3	AMC28023
EWC23	AMC460542	PEAK NO 7	AMC28025
EWC24	AMC460543	PEAK NO 8	AMC28026
EWC25	AMC460544	PEAK NO 9	AMC28027
EWC26	AMC460545	TOE 20	AMC393547
EWC27	AMC460546		

1. Mill site claim subject to a 2% NSR royalty payable to William E. Bohme and Eula Belle Bohme (half interest) and Patricia M. Green.

4.3 Environmental Liabilities

PVM is an existing mine with existing environmental liabilities, as described in Section 20.

4.4 Permitting

All required permits for current operation to 2026 have been obtained . New permits are required to extend the operation to 2039. Refer to Section 20 for detailed permit information, including a list of required permits.

4.5 Royalties

There are 26 unpatented lode claims located outside of the PVM patented land boundary that have a 2% net smelter return (NSR) royalty payable to William E. Bohme and Eula Belle Bohme (half interest) and Patricia M. Green. The LOMP presented in this Technical Report does not impact those claims, and as such, no royalty payments are expected.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

PVM is located in a mountainous region of east-central Arizona, 11 miles west of Globe and 13 miles northeast of the town of Superior, as shown in Figure 5-1.

Primary road access to the mine is along US 60, which runs east and west 3 miles south of the mine site. The highway's maximum elevation of 4,600 ft occurs just west of PVM.

Forest Road 287 (FR 287) provides a paved connection from US 60 to the mine security gate. This route also serves as the main access for the neighboring Carlota Mine, owned by KGHM International Ltd.

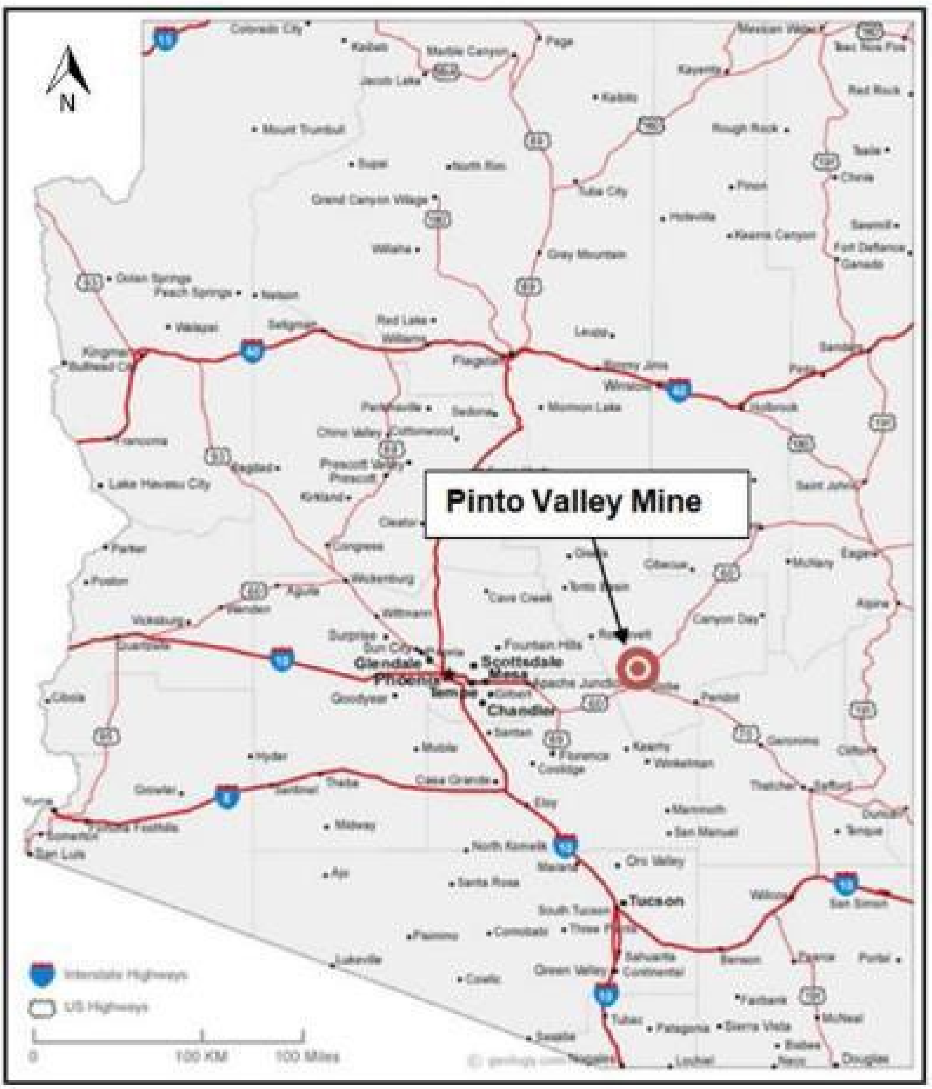


Figure 5-1: Pinto Valley Mine Accessibility

5.2 Climate and Topography

PVM's moderate, semi-arid regional climate allows for year-round operation. Average annual precipitation is 18.9 inches, occurring in a bimodal pattern. Most rainfall occurs during the winter and summer months, with dry periods in the spring and fall. May and June are typically the driest months of the year.

The maximum average annual temperature for the period of record is 77°F. July is the warmest month, with an average maximum temperature of 97°F, and January is the coldest month with an average minimum temperature of 34°F.

The terrain surrounding PVM is generally mountainous, and elevations at the mine range from approximately 3500 ft (1067 m) to 5000 ft (1524 m) above mean sea level.

5.3 Local Resources and Infrastructure

Miami and Globe are the main communities in the local area. Mining-related jobs are a significant source of employment in the area. These communities are supported by medical, fire, police, public works, transportation and recreational facilities. Medical facilities are available at the Cobre Valley Regional Medical Center located in Miami. The region has an adequate supply of permanent and temporary housing to accommodate PVM's workforce, though much of the workforce lives in the Phoenix Metropolitan Area.

5.4 Physiography

PVM is located in east-central Arizona, in the structural transition zone between the Sonoran section of the Basin and Range physiographic province to the south-southwest and the Colorado Plateau to the north. The terrain surrounding the mine property is generally mountainous, dominated by sharp landforms and prolific exposures of a variety of bedrock formations present in the region.

PVM lies entirely along the eastern flank of Pinto Creek, with numerous southwest-trending to northwest-trending ephemeral Pinto Creek tributaries crossing the property. Most of the headwaters of these tributaries originate along a regional surface water divide that runs north to south near the eastern PVM property line. All surface water runoff from the site ultimately flows into Pinto Creek, just west of the boundary of the property. Pinto Creek flows from the south to the north into Roosevelt Lake, an artificial impoundment constructed along the Salt River.

PVM is near the boundary of areas mapped as the Interior Chaparral biotic community and the Arizona Upland subdivision of Sonoran desert scrub biotic community.

6 History

The Globe-Miami mining district is one of the oldest and most productive mining districts in the United States, with its first recorded production occurring in 1878. Since that time, more than 15 billion pounds of copper have been produced in the district.

Ownership of PVM has changed numerous times since its inception. PVM originated as Miami Copper Company in 1909. In 1960, the Tennessee Corporation took over Miami Copper Company, and, in 1969, Cities Service Company merged with the Tennessee Corporation. At the time of construction and commissioning, PVM was owned by Cities Service Company, who had recently merged with Tennessee Corporation. Occidental Petroleum Corporation acquired Cities Service Company in late 1982 and sold the Miami operations to Newmont Mining Corporation in 1983. At this time, the company's name was changed to Pinto Valley Copper Corporation. In 1986, Newmont merged the Pinto Valley Copper assets into Magma Copper Company holdings, and Pinto Valley Copper became the Pinto Valley Mining Division of Magma Copper Company. In 1995, Broken Hill Proprietary Company Limited purchased Magma Copper Company. With the merger of Broken Hill Proprietary Company Limited and Billiton in 2001, the Pinto Valley Mining Division became Pinto Valley Operations of BHP. In 2013, Capstone affiliate-PVMC purchased the Pinto Valley Operations, now referred to as Pinto Valley Mine.

Prior to the construction of PVM, a chalcocite-enriched zone of the deposit was mined from 1943 until 1953 as the Castle Dome Mine.

Development of the PVM open pit began in 1972 and the concentrator went into production in 1974. The SX-EW plant began processing pregnant leach solution (PLS) from the leach dumps in 1981. A short shutdown occurred in 1983. In February 1998, mining and milling operations were suspended and environmental permits were maintained during the suspension of operations, as were the water and electrical systems. SX-EW facilities and cathode copper production continued during the suspension of mining and milling operations.

The mine has had two restarts since the 1998 shutdown. The mine resumed sulfide operations in mid-2007 for 18 months to January 2009 and then went into care and maintenance with only leaching operations continuing. The second restart began in December 2012 and included extensive rehabilitation of the site and purchase of a new mining fleet. During financial year ending June 30, 2013, sulphide mining resumed at Pinto Valley with production for financial year ending June 30, 2013 of 16.6 kt of copper concentrate and 4.9 kt of copper cathode.

Under Capstone ownership, PVM has produced 948 M lb of copper since 2014. PVM production since 2014 is summarized in Table 6-1.

Table 6-1: PVM Production Summary since 2014

Operating Statistics[1]	Q1-2021	2020	2019	2018	2017	2016	2015	2014
Production (contained metal in concentrate and cathode)								
Copper (000's lb)	36,410	118,968	117,629	119,067	126,393	151,788	133,921	143,585
Mining								
Ore (kt)	5,569	19,883	18,888	19,290	20,605	21,586	23,139	20,931
Waste (kt)	7,169	27,292	30,101	27,687	26,164	19,507	11,464	932
Milling								
Milled (kt)	5,229	19,674	18,665	19,246	19,655	20,565	17,730	17,231

Milled (average tpd)	58,095	53,755	51,137	52,728	53,849	56,189	48,576	47,209
Copper grade (%)	0.357	0.31	0.33	0.32	0.32	0.37	0.38	0.41
Recovery								
Copper (%)	85.6	85.0	85.1	84.6	89.2	87.6	87.4	88.9
Concentrate Production								
Copper Concentrate (dmt)	63,587	211,432	196,560	201,747	196,583	234,702	205,233	211,709
Copper Concentrate grade (%)	25.1	24.5	26.3	26.0	28.2	28.5	28.6	29.6
Property costs ($/t milled)	$11.15	$11.29	$11.17	$11.63	$11.00	$10.01	$11.55	$12.46
Payable copper produced (000s lb)	**35,177**	**114,978**	**113,645**	**115,018**	**122,118**	**146,667**	**129,450**	**138,860**
C1 cash costs ($/lb payable copper)	$1.94	$2.21	$2.05	$2.16	$1.95	$1.61	$1.97	$2.03

Notes:
1. Source of the operating statistics is Capstone's Form 51-102F1 Management Discussion & Analysis from December 2014 to March 2021. The abbreviation dmt refers to dry metric tonnes.

7 Geological Setting and Mineralization

PVM is located within the Globe-Miami mining district of central Arizona. Several mines and numerous prospects have been developed in the area. Larger mines in the district are porphyry copper deposits associated with Paleocene granodiorite to Granite Porphyry stocks. The porphyry copper deposits have been dismembered by faults and affected by later erosion and minor oxidation. Vein deposits and possible exotic copper deposits are also found within the district.

The Globe-Miami district contains igneous, metamorphic, and sedimentary rocks of Precambrian, Paleozoic, Paleogene, and Quaternary age. Precambrian basement rocks largely consist of Early Proterozoic Pinal Schist (~1700 Ma) intruded by granites correlative with peraluminous two-mica granite batholiths that comprise the Proterozoic basement rocks throughout southern Arizona and New Mexico. The Late Proterozoic Apache Group consists of (from oldest to youngest): the Pioneer Formation, including the basal Scanlan Conglomerate; the Dripping Spring Quartzite, including the Barnes Conglomerate; the Mescal Limestone; and, minor basalt closely associated with the Mescal. These units are intruded by Apache Diabase sills of various thicknesses.

Paleozoic rocks in the district are the Cambrian Troy Quartzite, Devonian Martin Limestone, Mississippian Escabrosa Limestone, and Pennsylvanian to Permian Naco Formation.

A large pluton of Schultze Granite was intruded into the Precambrian and Paleozoic wall rocks. Near the northern-most exposures at the Inspiration mineral deposit, it has various textures and compositions that have been called Granodiorite, Quartz Monzonite, and Porphyritic Quartz Monzonite. A separate Granite Porphyry has been mapped at Pinto Valley, Copper Cities, Diamond H, and Miami East, and is seen near the vein-controlled mineralization at Old Dominion.

Paleogene sedimentary and volcanic rocks cover the mineralized units. The Whitetail Conglomerate was formed as a result of regional uplift which contains weathered clasts of older rocks in a red iron oxide- rich, very fine-grained matrix. A Miocene ash-flow tuff, known as the Apache Leap Tuff, covered the area following the Whitetail Conglomerate, and further Basin and Range faulting and subsequent erosion produced the Paleogene to Quaternary Gila Conglomerate from all older rocks. On the west side of the Pinto Valley Open Pit, the Gila Conglomerate contains a basalt sill.

The hydrothermal ore deposits in the district comprise vein deposits and typical porphyry copper deposits. On the basis of predominant metals, the vein deposits can be further divided into copper veins, zinc-lead veins, zinc-lead-vanadium-molybdenum veins, manganese-zinc-lead-silver veins, gold- silver veins, and molybdenum veins. The primary minerals of the porphyry copper deposits are chiefly pyrite and chalcopyrite with minor amounts of molybdenite; gold and silver are recovered as by- products. Sphalerite and galena occur locally in very small amounts. Silicate alteration associated with the deposits includes potassic, argillic, sericitic, and propylitic alterations.

The deposit is a hypogene ore body with chalcopyrite, pyrite, and minor molybdenite as the only significant primary sulfide minerals. It is the underlying protore of the chalcocite-enriched Castle Dome deposit that was exhausted in 1953 (Peterson, 1962).

The primary host rock for the porphyry copper deposit is the Precambrian-age Lost Gulch Quartz Monzonite, which is equivalent to the Oracle Granite or Ruin Granite (Breitrick and Lenzi, 1987).

Formation of the deposit was associated with the intrusion of small bodies and dikes of granite porphyry and granodiorite that are of similar composition and age as the Schultze Granite (~61.2 Ma). Copper mineralization has been dated at 59.1 Ma (Creasey, 1980).

Primary sulfide ore minerals consist of pyrite, chalcopyrite, and minor molybdenite that occur in veins and microfractures, and less abundantly as disseminated grains, predominantly in biotite sites. The ore zone grades outward into a pyritic zone with higher total sulfide content, and the ore zone grades inward toward the low-grade core, which has lower total sulfides. Molybdenum distribution generally reflects copper distribution, with higher molybdenum values usually found in the higher-grade copper zones.

Sulfide deposition is controlled to some extent by the host rock. For the most part, the host is Lost Gulch Quartz Monzonite and porphyritic quartz monzonite, which are similarly altered and mineralized. The sulfide content decreases in Precambrian aplite intrusions. Aplite usually contains less than 0.25% copper, whereas adjacent quartz monzonite may have as much as 0.6% copper. The deficiency of copper in aplite is probably due to the absence of biotite, which makes up approximately 7% of quartz monzonite. Disseminated chalcopyrite shows an affinity for biotite, where it is seen to be either disseminated through the biotite or partially replacing it. Additional chalcopyrite is present in veins that cut both rock types.

Small intrusions of granite porphyry extend beyond the main mapped unit. Quartz monzonite and granite porphyry typically contain economic mineralization, while the granodiorite is typically weakly mineralized and is a minor source of ore.

The PVM ore trend has the appearance of a hook in plan view (Figure 7-2) and mimics the pit outline. Rock located south of the ore has decreasing sulfide content and numerous barren quartz veins. Rock located north of the ore has progressively more abundant, late-stage quartz-pyrite-sericite veins.

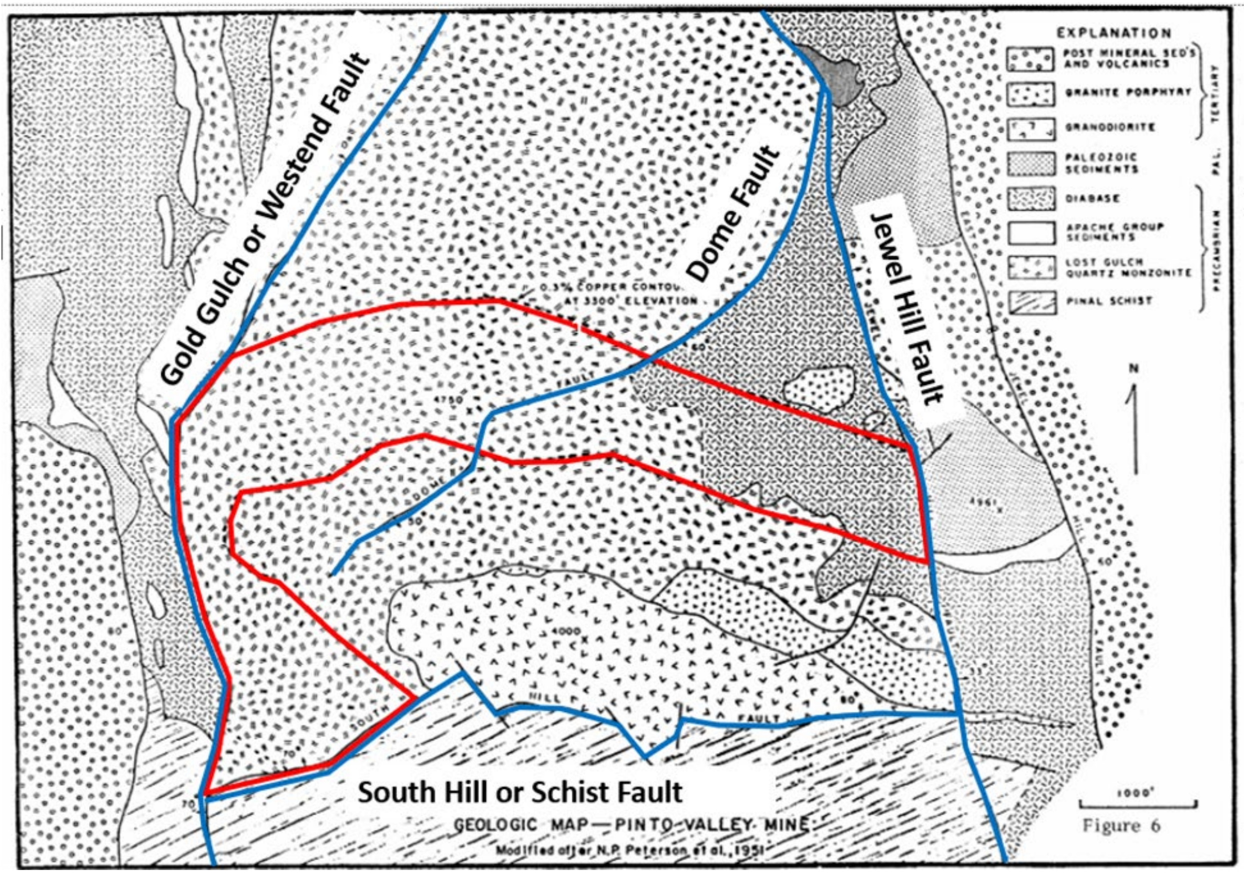


Figure 7-1: Plan view of hook shaped Pinto Valley ore trend (modified after Preece, R and Biard, B, 2012)

The South Hill Fault cuts the ore shell and associated alteration to the south and positions it against the Pinal Schist. It appears that the original configuration of the copper zone was that of a distorted, inverted bowl, with its long axis striking approximately N80E.

The deposit is bound by post-mineral faults (Figure 7-1). The South Hill Fault is on the south side of the deposit, the Jewel Hill Fault is on the east side, and the Gold Gulch Fault system is on the west side. Minor post-mineral normal displacement has taken place on the Dome Fault, a pre-mineral structure that strikes northeasterly across the north limb of the deposit.

Diabase forms thin dikes in pit exposures. These dikes commonly contain higher copper content than the surrounding Ruin Granite. In the eastern part of the deposit, a diabase sill lies at the top of the ore; west of the Gold Gulch Fault, diabase is mineralized by pyrite and chalcopyrite veins with abundant magnetite near mineralized granite porphyry.

A geological mapping exercise of PVM was conducted in early 2012 using the Anaconda method, producing three geographic information system-registered layers showing geology, alteration style, and mineralization.

7.1 Local Geology and Alteration

The following sections describe the main rock, alteration, and mineralization types on site (after Farjado, J. et all, 2012). Figure 7-2 shows PVM geology in plan view. Figure 7-3 illustrates the generalized columnar sections of sedimentary and volcanic rocks for the Castle Dome (i.e. PVM) area while Figure 7-4 presents a visual hypothetical and stylistic distribution of the major rock types of the Pinto Valley mine (modified after Peterson 1962).

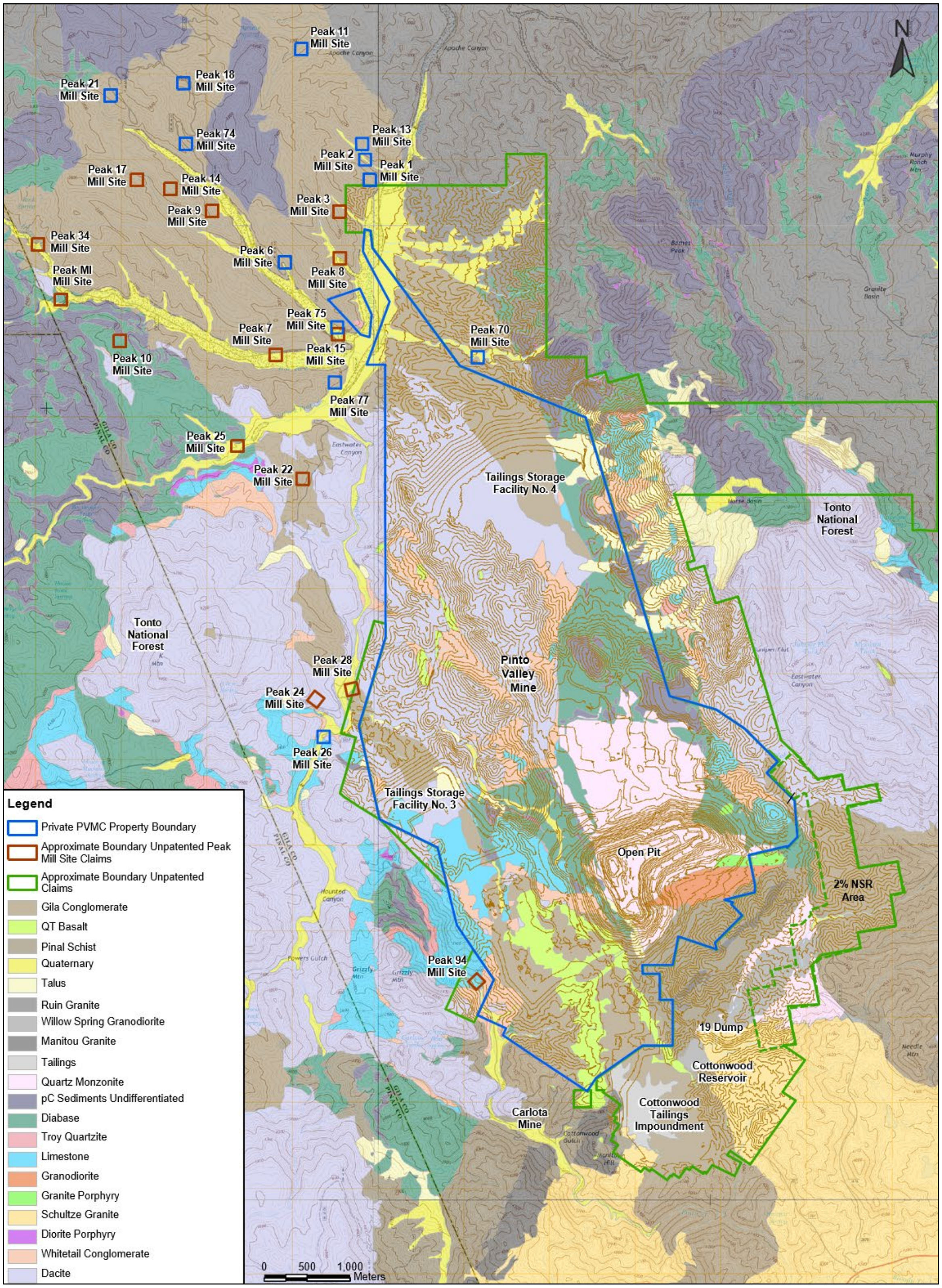


Figure 7-2: Pinto Valley Mine Geology Plan (Source: PVM property 2021, geology compilation by SRK, 2009)

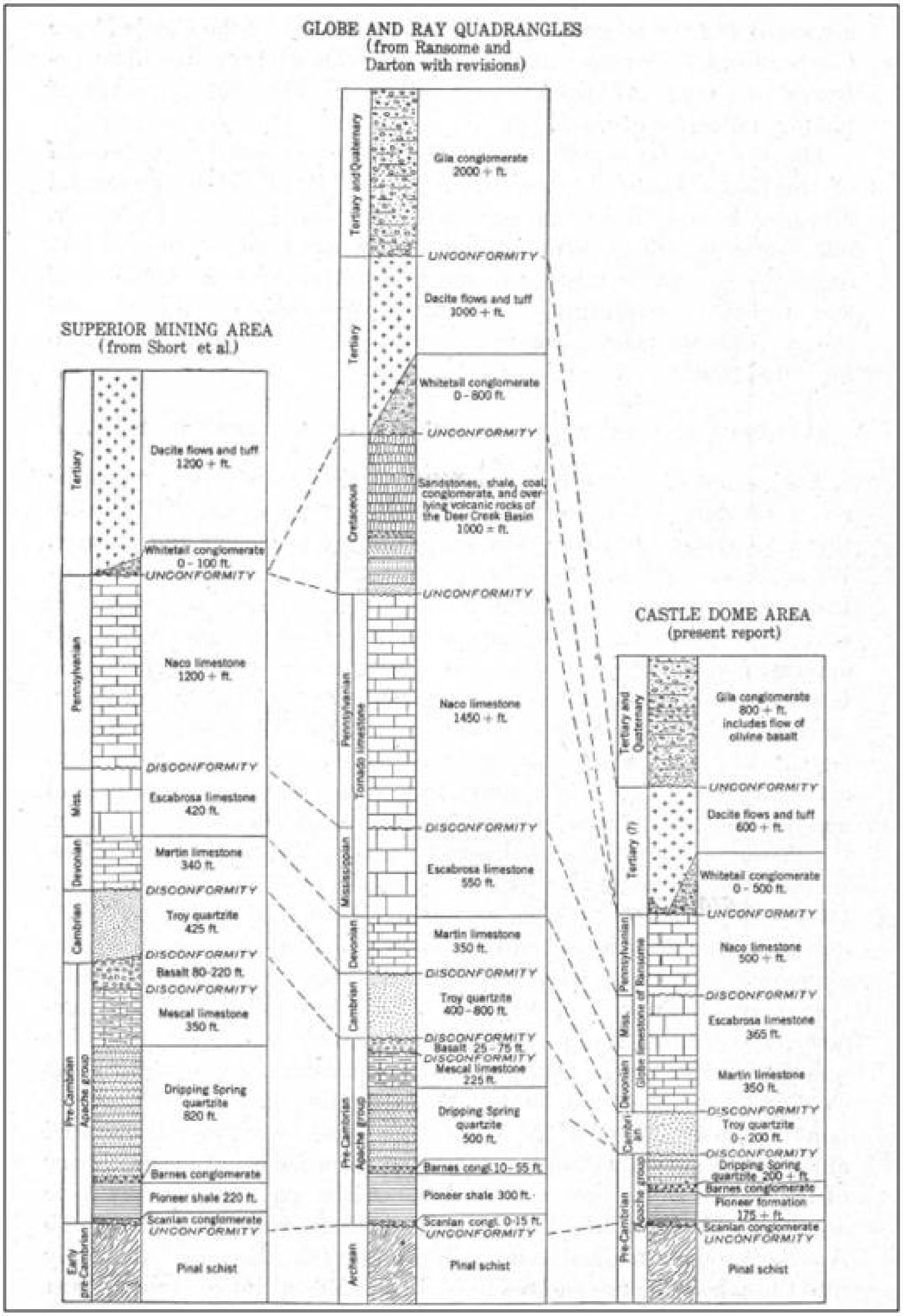


Figure 7-3: Generalized Columnar Lithology Sections for the Castle Dome Area (after Peterson et al., 1951)

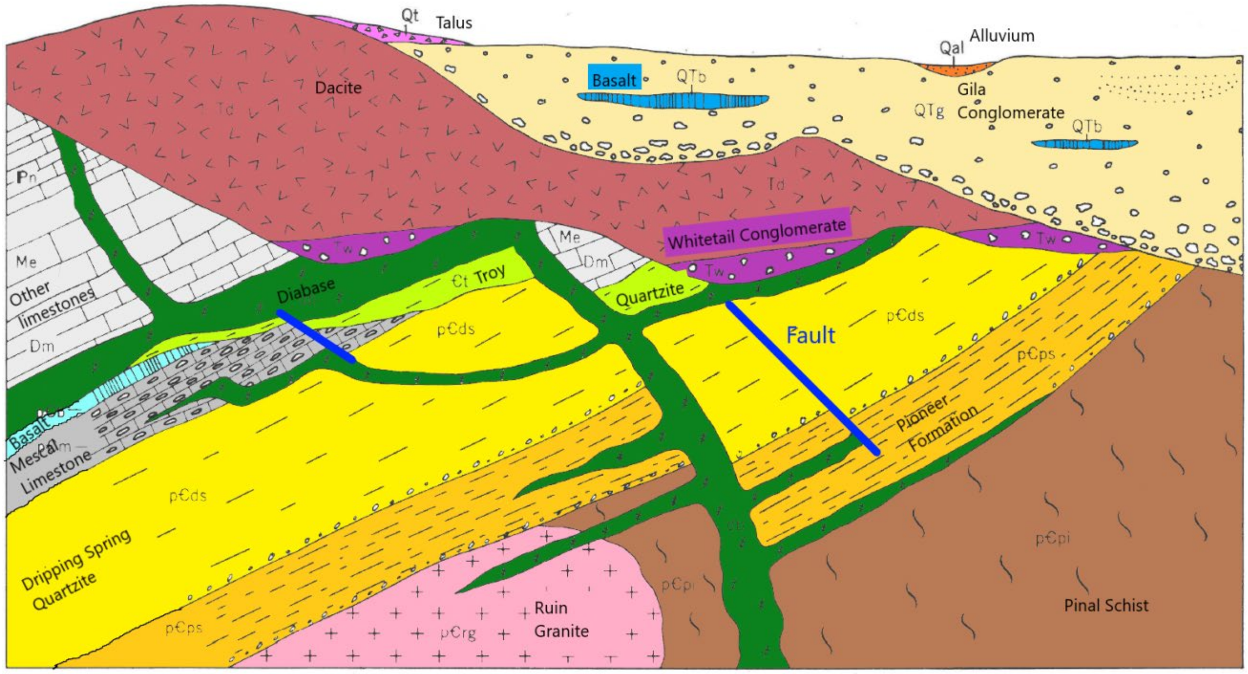


Figure 7-4: Hypothetical distribution of major rock types of the PVM area (modified after Peterson, 1962)

7.1.1 Pinal Schist

Lower Precambrian Pinal Schist is a fine-grained, well-bedded sediment dominated by biotite, lesser muscovite, and quartz, and in some areas, such as south of the South Hill Fault, bears garnet and chlorite. Grain sizes range from coarse quartz sericite schist to fine-grained quartz, sericite, and chlorite schist, which at times display magmatic segregation of biotite and quartz-rich seams up to 0.6 inches wide. The rock is extensively deformed, bearing tight to isoclinal folding and faulted extensively by various intrusive events.

7.1.2 Dripping Spring Quartzite

Precambrian Dripping Spring Quartzite contains a range of internal variation, from upper coarse- to medium-grained quartzite with cross bedding to lower thinly laminated fine-grained, well-sorted sediments at the base. This unit is typified by variably colored beds of fine sediment that display the well-sorted nature of the rock, which preserves current direction and energy regimes. Beds range from red brown to red-purple to purple-black, alternating with thin beds of arenaceous shale.

7.1.3 Mescal Limestone

Mescal Limestone, a sedimentary unit, was observed mainly in the northwestern part of the study area. It is composed of limestones, dolomites, and large amounts of chert. This Precambrian unit overlies the Pinal Schist and is overlaid by the Precambrian basalt.

7.1.4 Precambrian Basalt

Precambrian basalt, a basic volcanic unit, was recognized in the northern limit of the PVM tenements. This rock is black, with vesicles and some calcite calcedonic amygdales. This unit overlies the Mescal Limestone and is covered by the Troy Quartzite.

7.1.5 Troy Quartzite

Troy Quartzite, a Cambrian unit, is a distinct marker unit underlying the Martin Limestone, with unconformable boundaries separating upper and lower limestone units. Welded by cherts and siliceous cements, this fine-grained sediment is very resistant to weathering; therefore, it forms ridges and escarpments adjacent to limestone units. Where outcropped, the quartzite is a well bedded, well- sorted unit that forms gullies and gorges when exposed, sculptured by surface waterways. A quartzite conglomerate bed exists at the base of this unit that comprises well rounded quartz pebbles in a sandy silicified matrix; iron oxide staining gives this rock its characteristic red-brown color.

7.1.6 Martin Limestone

Martin Limestone is a massive sequence of layered brown- to gray-colored carbonaceous rocks with only a minor presence of fossil fragments. It is interbedded with fine red sandstones and shales. This unit overlies the Troy Quartzite and it underlies the Escabrosa Limestone.

7.1.7 Escabrosa Limestone

Escabrosa Limestone is a more massive, poorly bedded limestone; this unit outcrops as bold cliff faces, appearing medium to light gray, underlying the Naco Group (limestone). Mississippian in age, these lower beds appear oolitic with nodular calcareous formations; some beds contain crinoid fragments.

7.1.8 Naco Group

Naco Group (hereafter referred to by the informal site name of Naco Limestone) is dominantly thinly bedded limestone units and has a mid-gray color with thin laminations of calcareous sediments and marls separating limestone beds displaying crinoids, bivalves, and other marine fossil fragments. Lower horizons and the basal unit consist primarily of cherts, marls, and well-bedded calcareous sediments.

7.1.9 Whitetail Conglomerate

Paleogene in age, the Whitetail Conglomerate is distinguished from other sedimentary units by the exclusion of dacite and tuff. Mostly well bedded and often hematite-rich in both matrix and coating of clasts, this unit outcrops only where it is revealed by the erosion of the dacite cover. The unit is matrix- supported, displaying gradational fining-up sequences. Clasts are subrounded to angular in a poorly sorted matrix, with some quartzite horizons comprising well rounded quartz-rich and lithic fragments cemented by coarse quartz sands. This unit overlies and postdates mineralization, and therefore has little potential for economic value.

7.1.10 Gila Group (Conglomerate)

The Gila Group (hereafter referred to by the local site name Gila Conglomerate) overlies and is the youngest of all sedimentary units of Paleogene and Quaternary age. The unit is distinguished by the inclusion of all local rock types: the Apache Group, Paleozoic limestones, diabase, and dacite tuff, with some Pinal Schist fragments. Poorly sorted, but in parts moderately well stratified, it is compositionally matrix-supported. The unit is composed of dominantly cobble- to pebble-sized subrounded clasts. The composition of the rock is highly variable, often representing the dominant local lithology. Clast sizes decrease to the east of the project area, where the unit becomes more of a distal fan conglomerate with bedding stratification. The Gila Conglomerate overlies and postdates mineralization, and therefore has little potential for economic value.

7.1.11 Alluvium

Paleogene and Quaternary alluvium is a poly-lithologic detritus of some boulder-sized, but mostly cobble- and more finely sized, poorly sorted and poorly cemented sediments. Detritus lines the low-lying areas, commonly occurring at the base of steep slopes undergoing active erosion. Components often show evidence of reworking, resedimentation, and welding by modern calcrete and silcrete cements.

7.2 Intrusive Phases

In the PVM area, a series of intrusive bodies has been mapped with litho-chemistries ranging from intermediate to acid digenetic composition. Units have been classified according to mineralogy, crosscutting, and inclusion. The intrusive history of porphyry copper (molybdenum) emplacement in the Pinto Valley district is classified into pre-, intra-, and post mineralization stages. Descriptions of copper bearing intrusive events are detailed below.

7.2.1 Pre-Mineralization Intrusions

7.2.1.1 Manitou Granite

Manitou Granite is prevalent in the southeast portion of the study area, approximately 2,300 ft from the PVM pit. Occupying an area of approximately 0.08 mi2 and outcropping as elongated bodies trending in a northeasterly direction, this unit intrudes the Precambrian Pinal Schist basement. The Manitou granite itself has been intruded by Precambrian Ruin Granite and a series of fine- and coarse-grained aplitic intrusive phases related to this magmatic event. The Schultze Granite was the last unit to intrude the Manitou Granite in a much later Paleogene period.

Macroscopically, this rock is dark brown with a phaneritic texture; it is equigranular and medium-grained, with anhedral crystals of quartz (20%), subhedral undifferentiated mafic minerals (7%), anhedral muscovite (5%), orthoclase (25%), and subhedral-euhedral plagioclase (38%).

This unit has a prevalent slight to moderate foliation that has deformed the original equigranular texture. Minerals are generally elongated, with the long axis of grains ordered in a preferred orientation or, in some cases, partially destroyed; this has been observed in some locations with respect to mafic minerals. Manitou Granite is the youngest Precambrian intrusive.

7.2.1.2 Willow Spring Granodiorite

Willow Spring Granodiorite is an intrusive unit that outcrops in the southeastern sector of the study area, occupying approximately 0.16 mi2. This unit outcrops as elongated bodies trending north- northeast, intruded by Precambrian Ruin Granite and Paleogene Schultze Granite. It is also in fault contact with the Gila Conglomerate unit.

Macroscopically, this granite is mottled by dark brown minerals and has a slightly porphyritic, phaneritic, and inequigranular texture with medium-sized grains. The rock is composed of quartz, anhedral- subhedral (15%); biotite-amphibole (12%), which is partially replaced by chlorite; orthoclase (15%), subhedral-euhedral, ranging from 4 to 10 mm; and subhedral-euhedral plagioclase (38%).

Crosscutting relationships and Creasey (1980) dated this intrusive unit as Precambrian.

7.2.1.3 Ruin Granite

Ruin Granite is an intrusive unit that outcrops over an area of approximately 0.81 mi2, which has been exposed primarily by the excavation of the PVM pit. This rock is the primary host rock of copper mineralization in economic concentrations and has been dated to Precambrian age (Creasey 1980). The granite has also experienced a series of magmatic-hydrothermal events, resulting in the emplacement of porphyry copper systems. The Ruin Granite is in fault contact with the Pinal Schist unit to the south and a stacked series of faults to the west, with repetitious sedimentary units. Granites of the southeastern sector of the study area have been intruded by the Willow Spring Granodiorite and Paleogene Schultze Granite, and in one area it is in fault contact with the Gila Conglomerate. A zone to the north of the PVM pit puts the Ruin Granite in contact with Precambrian Dripping Spring Quartzite sediments and diabase dike intrusions.

Macroscopically, this rock is pinkish-brown, with phaneritic, inequigranular coarse texture and anhedral quartz crystals (25%), anhedral-subhedral biotite (7%), anhedral muscovite (3%), subhedral-euhedral orthoclase (35%) with some phenocrysts up to 60 mm, and subhedral euhedral plagioclase (38%). In much of the Pinto Valley pit, orthoclase is much more dominant than plagioclase, which may result from alteration.

There has been a series of aplitic phases related to Ruin Granite emplacement; the highest concentration of these is in the southeastern sector of the outcrop. Numerous small dikes also occur within the PVM pit. The aplitic intrusive units are pinkish-brown and dominated by equigranular quartz. They have a fine grained, sugary texture and are dominated by potassic feldspar. The intrusive complex related to the Ruin Granite is Precambrian in age (Creasey, 1980).

7.2.1.4 Diabase

Diabase is a sub-volcanic Cretaceous or later unit that is most prevalent in the northern area of the project; but it also occurs as sills and minor dikes throughout most of the project area. This unit occupies approximately 0.58 mi^2 of the project area. The diabase most commonly intrudes Precambrian units, such as the Apache Group sediments and Ruin Granite. The unit is generally covered by post sedimentary units, including the Martin, Escabrosa, and Naco Limestones, and is partially covered by Gila Conglomerate and the Apache Leap Tuff.

This unit is of fine- to medium-grained mafic composition, bearing pyroxene and hornblende mafic minerals, and lesser plagioclase. This unit has different phases, with early medium to coarse textures that range to later, fine-grained textured intrusions.

The diabase commonly contains 1%–2% disseminated pyrite and trace chalcopyrite, but will bear stronger sulfide content, especially chalcopyrite, when proximal to a porphyritic source.

7.2.1.5 Schultze Granite

Schultze Granite is Paleogene in age; this plutonic body has been dated at 61 Ma from similar outcrops sampled in the Miami-Inspiration area (Creasey, 1980). This unit represents the main pre-mineral stage of the Laramide intrusions and the magmatic source of the metal-bearing porphyritic intrusions in the district. This unit outcrops generally in the southern part of the project area with batholitic dimensions of 0.58 mi2 outcrops. This unit has also been observed intruding the Ruin Granite and Pinal Schist. In some places, the unit is covered by Quaternary basalt and is in fault contact with the Gila Conglomerate.

Macroscopically, this rock has phaneritic texture and inequigranular texture of medium- to coarse-sized grains, with books of biotite (8%), subhedral 1 mm–3 mm sizes; quartz (20%), subhedral 2 mm–8 mm sizes; K-feldspar of orthoclase variety (25%), subhedral-euhedral 3 mm–15 mm sizes; and plagioclase (47%) with 2 mm–4 mm sizes.

7.2.2 Intramineral Granite Porphyry

In the Pinto Valley district, a suite of porphyritic intrusive units have been identified that have age and genetic relationships with a number of igneous events. Intrusives were found to have a composition varying from quartz monzonite to granite. The following sections describe these units.

7.2.2.1 Early Granite Porphyry

A family of porphyritic intrusives appears in the form of dikes and stocks in the central sector of the PVM pit. A number of small finger-like projections stemming from granitic porphyry stocks and dikes also exist in the western section of the pit, with a predominant northeast trend. This early granite porphyry unit has been observed intruding the country rock Ruin Granite, and as having been crosscut by the intramineral late granodiorite phases.

Macroscopically, the rock is pinkish-brown to gray, phaneritic, and of porphyritic texture with an inequigranular grain shape. Mineral composition comprises 40% phenocrysts with approximately 60% groundmass, characterized by aggregates of quartz and feldspar: quartz eye phenocrysts (3%–7%) are euhedral-subhedral and range between 2 and 4 mm in size; books of biotite (5%–8%) are subhedral and range between 1 mm and 3 mm; orthoclase feldspar (20%–25%) is euhedral subhedral and ranges between 3 mm and 5 mm; and plagioclase (60%–65%) is subhedral-euhedral, ranging between 2 mm and 5 mm.

A number of additional observations of this unit’s magmatic-hydrothermal activity suggests this intrusive phase is responsible for introducing mineralization into the PVM system. A clear relationship exists between the development of strong late magmatic and early hydrothermal potassic alteration (K- feldspar, biotite, and silica). Early hydrothermal activity has also produced extensive quartz “A” vein development, along with sulfide mineralization where chalcopyrite content is greater than pyrite. The presence of quartz “B” veinlets with minor molybdenite content also occurs in close proximity to the “A” vein sets.

7.2.2.2 Intramineral Granite Porphyry

An intramineral phase of porphyry has been identified in the northeastern sector of the PVM pit. Intramineral granite porphyry outcrops as a stock elongate in an east west direction hosted in Ruin Granite, though crosscutting relationships were not observed between this and the earlier granite porphyry. This intrusive unit mainly crosscuts the Ruin Granite, Pinal Schist, and diabase.

In a hand specimen this rock is brown-gray with a phaneritic texture, and inequigranular with a strong porphyritic texture, including 40%–45% phenocrysts and 55%–60% as groundmass with aggregates of quartz and feldspar. Mineralogically, eye quartz forms 10%–15% of the rock; grains are euhedral subhedral and range between 2 mm and 4 mm. Books of biotite form 3%–5%; grains are subhedral and range between 1 mm and 3 mm. Orthoclase feldspar forms 30%–35%; grains are euhedral-subhedral and range between 4 mm and 10 mm. Plagioclase forms 50%–55%; grains are subhedral euhedral and range between 2 mm and 5 mm.

This porphyritic unit exhibits minor hydrothermal alteration, and only displays minor potassic alteration and “A” quartz vein sets. Minor disseminated mineralization has been observed. The unit was found with a zone of strong phyllic alteration in the PVM deposit associated with extensive “D” veining. The observed mineralogy and alteration styles suggest that this intrusive was emplaced later in the magmatic hydrothermal history of the PVM porphyry copper deposit.

7.2.2.3 Intramineral-Late Granodiorite

The Intramineral-Late granodiorite unit outcrops as a large body in the southeastern area of PVM, with a second zone in the west mapped as northeast trending minor bodies. Crosscutting relationships suggest that this unit intruded into both porphyritic units in the mine.

In a hand specimen, this rock is gray-brown with a phaneritic texture. It is equigranular, of fine- to medium-sized grain with the following mineral composition: hornblende (5%, subhedral-euhedral, 1 mm–2 mm); books of biotite (5%, subhedral, 1 mm–2 mm); K-feldspar (10%, subhedral, 2 mm–3 mm); quartz (12%); and crystals of plagioclase (68%, subhedral–euhedral, 2 mm–3 mm).

This unit exhibits only minor mineralization as 1%–2% disseminated pyrite chalcopyrite; thin quartz veins exist but are generally unmineralized. Only weak hydrothermal alteration was observed and described as a weak potassic alteration; this suggests that this intrusive unit was injected late in the Laramide intrusive history. Crosscutting relationships indicate that this unit truncates the late magmatic potassic event.

7.2.2.4 Porphyritic Granodiorite

The porphyritic granodiorite intrusive unit was observed in the southwestern boundary of PVM as a small body intruding into the Pinal Schist and the Schultze Granite.

In a hand specimen, this unit has medium-sized grains, inequigranular with some porphyritic textures, with the following mineral composition: quartz (10%, anhedral, 1 mm–3 mm); books of biotite (8%, subhedral-euhedral, 1 mm–3 mm); hornblende (2%, subhedral, averaging 1 mm–2 mm); K feldspar (15%, subhedral, 2 mm–4 mm); and plagioclase (65%, subhedral-euhedral, 2 mm–6 mm).

This intrusive was found at a site that had been disturbed by a small shaft and old workings. Copper oxide was evident, coating rocks close to the mouth of the small mine opening. Minor hydrothermal alteration was observed as chlorite replacing mafic minerals. This intrusive is most probably related to the granodioritic intrusive event in the PVM area.

7.2.2.5 Breccia Porphyry

Near the southeastern boundary of the PVM area, two sub-outcrops of a unit with intrusive brecciaed features were found. This unit is called the breccia porphyry and intrudes the Ruin Granite as a small dike swarm.

In a hand specimen, this unit displays a brecciated texture, composed predominantly of a groundmass material (~70%), with surrounding fragments of rock and broken eye quartz (10%–15%) that range in size from 2 mm to 4 mm.

This unit was tested using a portable infrared mineral analyzer spectrometer for hydrothermal alteration minerals, revealing an upper crustal association of dickite-kaolinite-pyrite. Some leaching of minerals, mainly jarosite and minor goethite, was also confirmed by TerraSpec analysis. This is an extremely important finding because it can be concluded that the mineral is associated with the advanced argillic alteration zone in the upper crust.

Microscopic study of thin sections revealed the presence of a brecciated texture. Intrusive fragments of granite monzogranite were observed with clearly defined borders, but some had moderately rounded margins, indicating a lack of any reaction with the matrix. The matrix is composed of rock flour, various clay species, disseminated dickite, and traces of muscovite and brown biotite.

Features described in this rock suggest a stage of phreatic brecciation, possibly related to the activity of a nearby hydrothermal system.

7.3 Regional Structural Framework

A number of structural events were identified during the mapping exercise, showing a high level of complexity in both the extent of deformation and the timing of the various events. Considerable deformation of the units has persisted from the Precambrian era to Paleogene Basin and Range events, involving the reactivation of many earlier structures.

The main structures identified in the project are related directly to a set of lineaments, faults, and fractures with a north-south orientation (Figure 7-1).

The oldest fault observed is the northeast-southwest striking South Hill Fault. Field observations suggest that this fault controlled the emplacement of all the Precambrian intrusive phases along a northeast trend. The last reactivation along this fault has reverse movement, with a northwestern dip that has truncated mineralization of the PVM deposit; this fault has placed the Ruin Granite over the Pinal Schist.

Most north-south structures are a product of extensional deformation from the Basin and Range event; the best example is the Gold Gulch Fault that separates, via horst and graben blocks, the Apache Group sediments and the Ruin Granite, respectively. Other large faults are the Dome Fault and the Jewel Hill Fault, with normal movement and more restricted deformational features.

Locally, the fault systems at surface present a north-northwest pattern with normal movements. Some minor reverse and transcurrent faults were observed and are closely related to extremely large structures, such as the Riedel-type faults, which all show subvertical dips.

8 Deposit Types

PVM is classified as a copper-molybdenum porphyry system. Extensive literature exists on porphyry deposits due to their large size and economic importance. The following description of a porphyry deposit is from a summary by Sillitoe (2010).

Porphyry deposits are typically centered on polyphase stocks and porphyry dyke swarms, with skarn deposits formed adjacent to and epithermal deposits above the porphyry mineralization. The metal endowment of a porphyry system is related to the geochemistry of the oxidized magmas that contribute to the formation of the stocks and dykes, with gold and/or molybdenum commonly found in association with copper. Porphyry deposits typically occur in association with Mesozoic and Paleogene intrusions, probably as a result of poor preservation of older rocks. A generalized porphyry copper cross section is depicted in Figure 8-1.

Porphyry systems are typically zoned from a potassic-altered (biotite-potassium feldspar) core overlying barren, calcic-sodic-altered rock, upward through phyllic altered (sericite or chlorite-sericite) margins to propylitic-altered (chlorite-epidote) rocks (Figure 8-2). Porphyry systems also grade upward into advanced argillic and silicic alteration related to epithermal mineralization. Alteration zoning may be complex and overlapping due to successive injections of magma into country rocks. The vertical distance between porphyry mineralization and overlying epithermal mineralization may range from 1,000 yd to several thousand yards.

Hypogene copper mineralization is disseminated and veinlet-hosted in addition to being zoned from bornite-rich in the core through chalcopyrite to pyrite in distal areas. Magnetite (in copper-gold porphyries) and molybdenite (in copper-molybdenum porphyries) are common accessory minerals.

Quartz veins and veinlets as stockworks and sheeted arrays are present throughout these systems, and typically occur in a sequence from early quartz-feldspar “A” veins, through quartz-sulfide (mainly chalcopyrite-molybdenite) “B” veins with potassic altered margins, to late sulfide-dominant (primarily pyrite) “D” veins with phyllic altered margins (Gustafson and Hunt 1975).

Veining in copper-gold deposits may differ slightly, with quartz-magnetite-chalcopyrite and magnetite- dominant “M” veins present or dominant (Arancibia and Clark 1996).

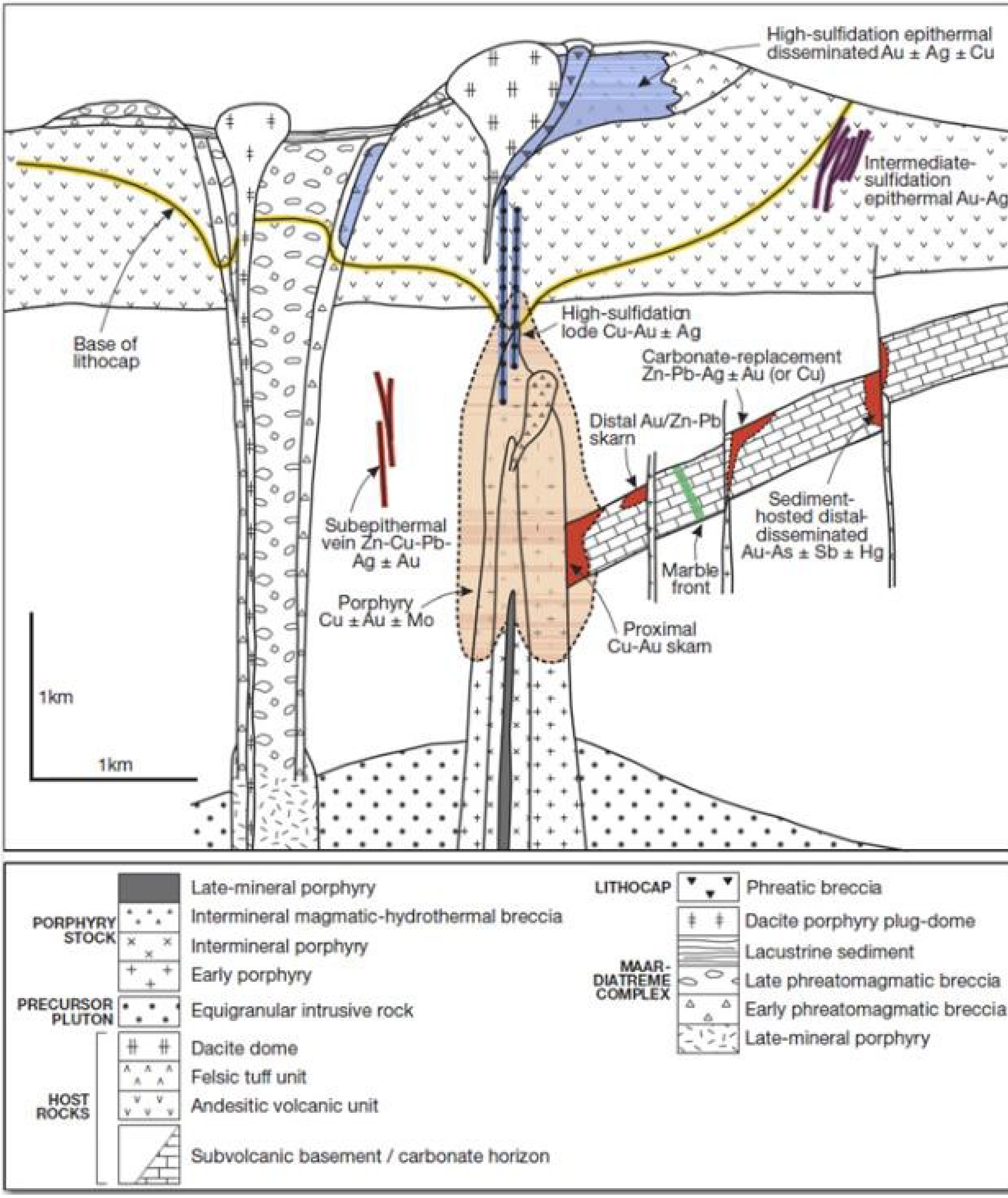


Figure 8-1: Anatomy of a Telescoped Porphyry System (Sillitoe, 2010)

Due to the large amount of disseminated pyrite in most porphyry systems, these systems are susceptible to supergene weathering and leaching. Copper is oxidized and leached from areas above the water table and deposited as chalcocite and other supergene copper minerals at or near the water table, leading to enrichment in copper grades. Supergene chalcocite enrichment can increase grades locally by 200% to 300% or more, with a significant impact on the overall economics of these deposits.

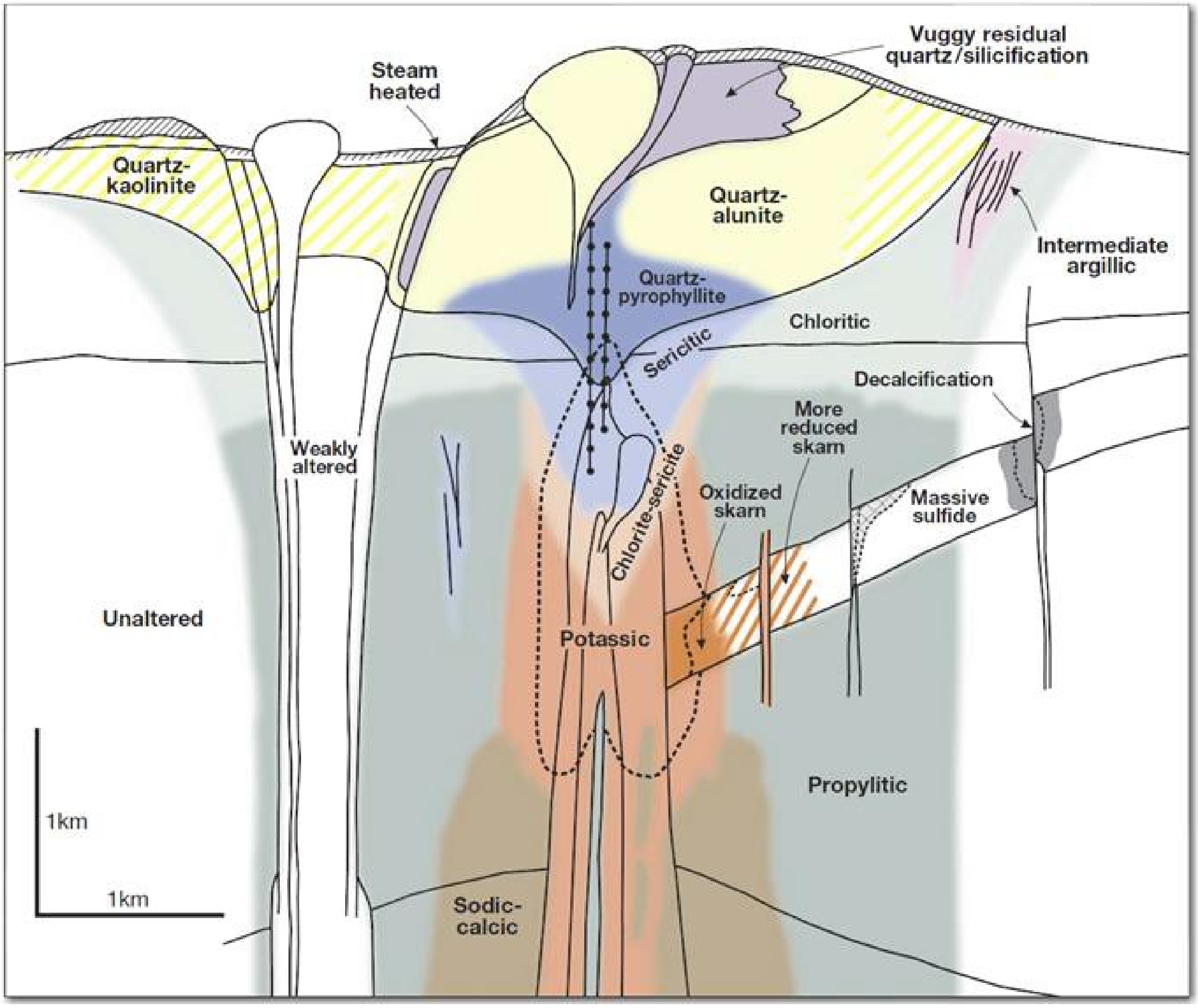


Figure 8-2: Generalized Alteration-Mineralization Zoning Pattern for Telescoped Porphyry Copper Deposits (Sillitoe, 2010)

Proximal skarn deposits are typically located laterally from porphyry deposits where the igneous body intrudes calcareous host rocks (Meinert, 2000). They consist of replacement bodies within (endoskarn) or marginal to (exoskarn) the causative intrusion. Skarn may be particularly well developed in limestones and other calcium or carbonate-rich rocks. Skarn alteration assemblages include garnet, pyroxene, wollastonite, magnetite, actinolite, pyrite, magnetite, and chalcopyrite.

Copper-molybdenum porphyry and skarn mineralization are all found in close proximity within the PVM area. Skarn is a relatively minor unit in comparison the scale of the overall porphyry deposit.

Mineralization is associated with an overlap of phyllic and potassic alteration (Figure 8-3), a supergene chalcocite blanket, and adjacent areas of hornfelsing and skarn alteration.

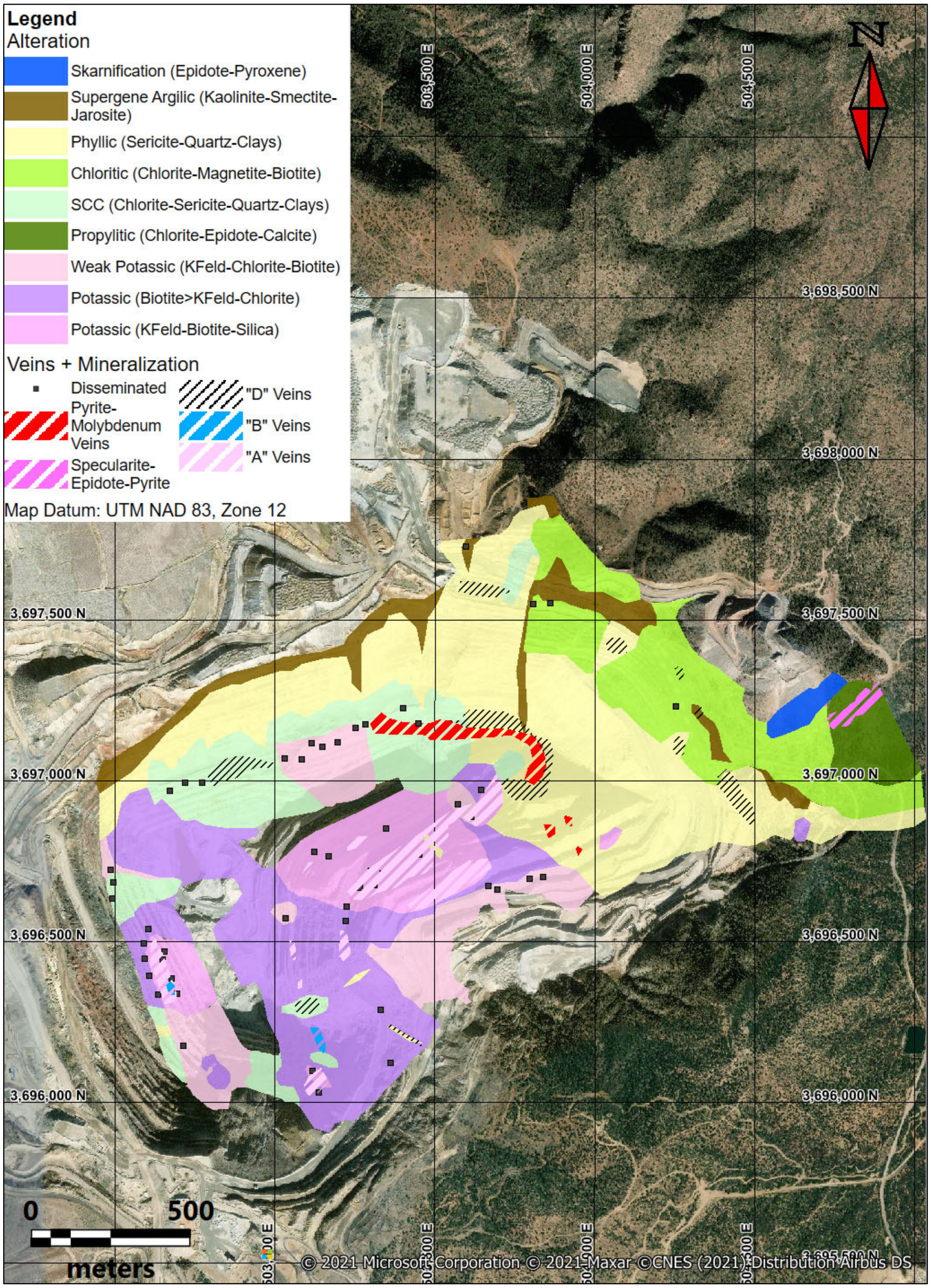


Figure 8-3: Pinto Valley Mine Alteration and Mineralization Plan Map (after BHP, 2012)

9 Exploration

No exploration has been completed since the mine was purchased in 2013.

10 Drilling

Historical drilling documentation by previous owners was limited to internal reports, and there were no other listings for historical data, methods used, or pre-2010 drilling procedures.

The pre-2006 PVM drilling programs comprised a combination of core, rotary, and churn drillholes. Churn holes defined much of the early Castle Dome mineralization, which has been mined out. Post-Castle Dome holes were drilled on an original spacing of 400 ft east-west and 200 ft north-south. Later, drilling was done to infill the original grid to 200 ft spacing in some areas. Drilling that has occurred since the 1986 construction of the block model includes 10 core holes (E52 through E61) and three RC rotary holes (RC62 through RC64) drilled in 1992. From January 1996 to April 1997, 67 RC exploration and infill holes were drilled including 48 RC holes (AD- and NR-Series totaling 29,665 ft) drilled in 1996 and 19 RC holes (WW- and 97-Series totaling 8,520 ft) drilled in 1997. The WW- and 97 Series were drilled in the interior pit and through the Gold Gulch and Continental Faults. Seven of the exploration holes were drilled east of the existing pit; these laid the groundwork for future plans of an east pit expansion, known as the Satellite Pit.

The current PVM drill hole database contains a significant amount of drilling that defined the grades in the block model that have been mined out, especially as they relate to the historic Castle Dome mining activity.

All drillhole collar locations were surveyed for historical and current drilling campaigns. The majority of the drillholes are vertical and, therefore, do not have downhole surveys. However, a majority of the inclined holes do have downhole surveys.

Drilling campaigns from 2006 to 2008 had various purposes, including delineation, exploration, geotechnical, and resource classification upgrade drilling. The campaigns included 18 G-Series geotechnical holes and 11 HW-Series holes drilled in 2007, and 17 PZ-Series holes, 17 S-Series holes, 24 B-Series holes, and 4 DH-Series holes drilled in 2008.

A BHP drilling campaign in 2010 focused on near-pit exploration, while the 2011 and 2012 campaigns focused on infill drilling for resource classification upgrade in support of restarting operations. Ten holes were drilled in 2010, 40 holes were drilled in 2011, and 64 holes were drilled in 2012.

In 2014, Capstone drilled 10 geotechnical holes. An infill RC program consisting of 43 holes was carried out in 2015 aimed at 2016 and 2017 production areas. Additionally, a three-hole geotechnical program was completed. Further infill-drilling with RC, diamond drillholes (DDH) and DDH with an RC pre-collarTable 10-1: Drilling Summary, 2015 to 2020, was carried out in 2016, 2017, 2018 and 2019 (Table 10-1).

Table 10-1: Drilling Summary, 2015 to 2020

Year	2015	2016	2017	2018	2019	2020
# of drillholes available for model update	897	901	919	942	951	951
Drilling Campaign Summary	Infill: 43 RC (9,010 feet) 19 PRC Geotech: 3 RC (1,100 feet)	Infill: 4 RC (3,370 feet)	Infill: 17 RC holes (15,820 feet) 1 DDH[1] with RC pre-collar (1,950 feet)	Infill: 22 RC holes (14,280 feet) 1 DDH[1] with RC pre-collar (1,090 feet)	Infill: 8 DDH[1] with RC pre-collar (12,460 feet) 1 DDH[1] (600 feet)	None

1. DDH core is HQ size, with a diameter of 63.5mm.

Data from these drill programs were incorporated into the 2021 block model.

A total of 951 drillholes were used for the PVM resource estimate model developed in 2021 to support this Technical Report.

11 Sample Preparation, Analyses and Security

11.1 Sampling and Analytical Procedures

RC drill cuttings are blown into a cyclone and collected at 10-foot intervals. The recovered material was split to 12.5% of the original volume, using a rotary splitter at the drill, since 2015.

Diamond drill core is placed in wax-covered core boxes with depth markers for every drill run of up to 10 ft then transported to the core handling facility by PVM employees or the drilling contractor. QuickLogs are done at core reception which includes initial lithology and a visual estimation of mineralization and alteration, particularly biotite content. The mine is set up on a bar code system for ease of handling and to track the core and samples. There is a triple bar code tag: the first tag is for the half core that remains in the box, the second tag is for the split that is sent to the lab for analysis, and the third tag is for the coarse duplicate and is used to tag the pulps and rejects. The core is logged for geology and split by saw at one of two stations.

The detailed geological logs are entered into an acQuire® relational database system which also records the collar, survey, assay, lithology, alteration, mineralization, and geotechnical (RQD) data. These data are tagged and tracked using the bar codes, and all subsequent assay information provided by the laboratory, including the QA/QC data, is linked to the database. A dispatch report is created which is then sent to the laboratory and subsequently matched against the shipments. Deviations and discrepancies are reported and investigated. Any updated assay data from the laboratory is linked to the bar code system and relayed to the company electronically via Excel® CSV files and imported into acQuire® automatically. The data are imported into MineSight™ for the purpose of resource estimation.

A number of different companies and laboratories have provided assay services to PVM over the years. Details of sampling and assaying procedures used during the earlier stages of operation are not readily available. Procedures used by outside labs that ran assays for some of the later drilling campaigns, such as those performed by Mountain States for the RC holes and Chemex for the AD holes, are also not readily available. The analytical procedures requested by PVM for assays procedures of contract laboratories since 2013 are in line with industry standards for total copper and molybdenum (3 or 4-acid digestion with ICP finish) but procedures were BHP-specific with respect to acid soluble copper (i.e., digestion with 10% sulfuric acid, placed in a hot bath at 40°C, and read after 40 minutes).

PVM contracted Skyline Assayers and Laboratories (Skyline) since 2015 to assay samples informing the resource model. Skyline picks up the bagged samples directly from PVM. For sample preparation, Skyline enters all data into their laboratory information system. If necessary, samples are dried for eight to 24 hours at 225° to 250°F. Before processing, washed-river-rock is fed through the crusher to prevent contamination from the previous batch. The sample is then crushed to produce a nominal 70 to 80% minus 10 mesh product, which is transferred to a Jones or Gilson Splitter. After blending three times, a parent and reject pan are established, and the parent poured back into the splitter, repeating the procedure until 250 to 300 grams of material remains and is poured into a labeled envelope. Between samples, the crusher and splitter are cleaned out using compressed air to minimize cross contamination. During pulverization, each envelope is poured into a pulverizing bowl, where between 90 seconds and two minutes of pulverization results in a pulp to a nominal 95% minus 150 mesh. Between batches, the bowl is cleaned out with silica sand.

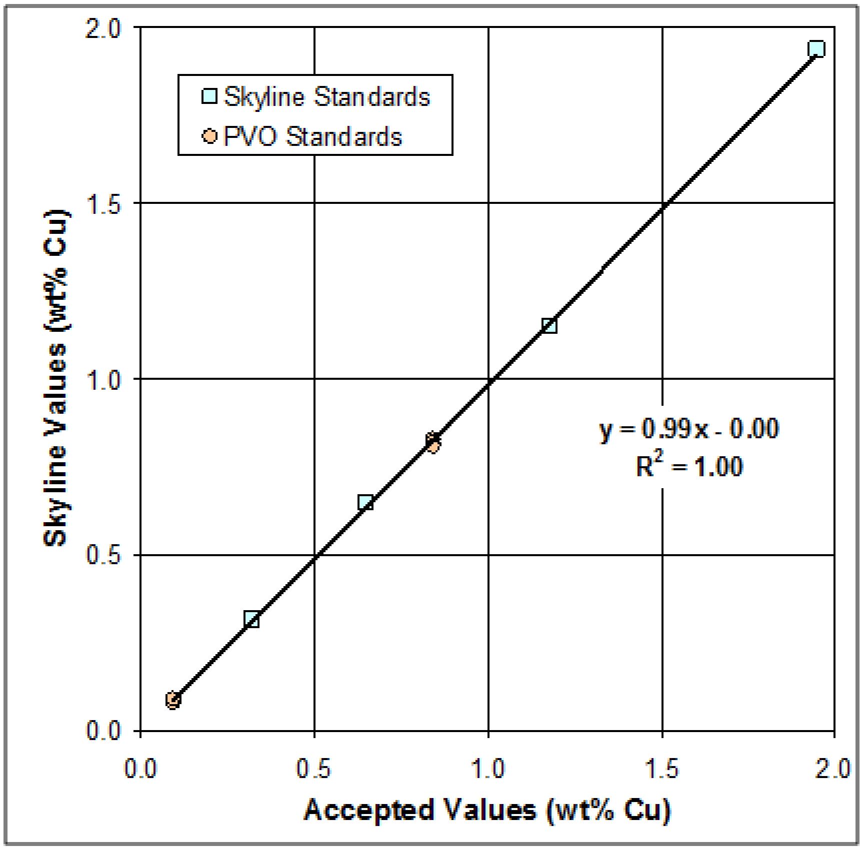
11.2 Verification of Pre-2006 Data

Independent audits of the Pinto Valley assays were conducted in 1992 and 2000. Results were as follows:

- assay values in the Pinto Valley database were reliably entered;
- total copper assays in the Pinto Valley database were reproducible and could be considered representative within normally-accepted limits of error;
- total copper assays in holes below the current pit base could also be considered representative within normally-accepted limits of error, except in the deeper parts of some RC holes where they may be low-biased. However, using these assays to estimate grades in the model is acceptable because they will tend to provide a conservative rather than an overly optimistic estimation of grades;
- acid soluble assays in the Pinto Valley database vary considerably depending on the drilling campaign and;
- reserves, resources, and production at Pinto Valley were reported as sulfide copper, which was calculated by subtracting acid soluble copper from total copper. Because biases exist in the acid soluble copper assays, this procedure generates sulfide copper values that are biased relative to each other as a function of the drilling campaign. However, sulfide copper values are only slightly lower than overall total copper values, so it can be reasonably assumed that the sulfide copper values were also globally correct within normally-accepted limits of error.

As part of the start-up Feasibility Study done in 2006, a QA/QC program was conducted on 101 randomly selected drillhole assay interval pulp samples and 15 randomly selected core assay intervals. Samples were sent to Skyline Assayers and Laboratories in Tucson, Arizona to be analyzed for total copper and acid soluble copper. Skyline was instructed to analyze the samples for acid soluble copper using BHP lab procedures. Before the lab processed these samples, BHP provided instructions for the pulp sample analytical procedures and also provided a sequential pulp sample list. Included in this QA/QC program for the Feasibility Study were seven sets of a known National Institute of Standards and Technology (NIST) standard pulps: Copper Ore Mill Heads standard at 0.84% total copper, and a Copper Mill Tails standard at 0.091% total copper. These known standard sets were inserted in sequential order for analysis preceding the 15th pulp sample in the analytical run. All relative precisions are discussed at a 95% confidence level (estimated using the Student's T-distribution).

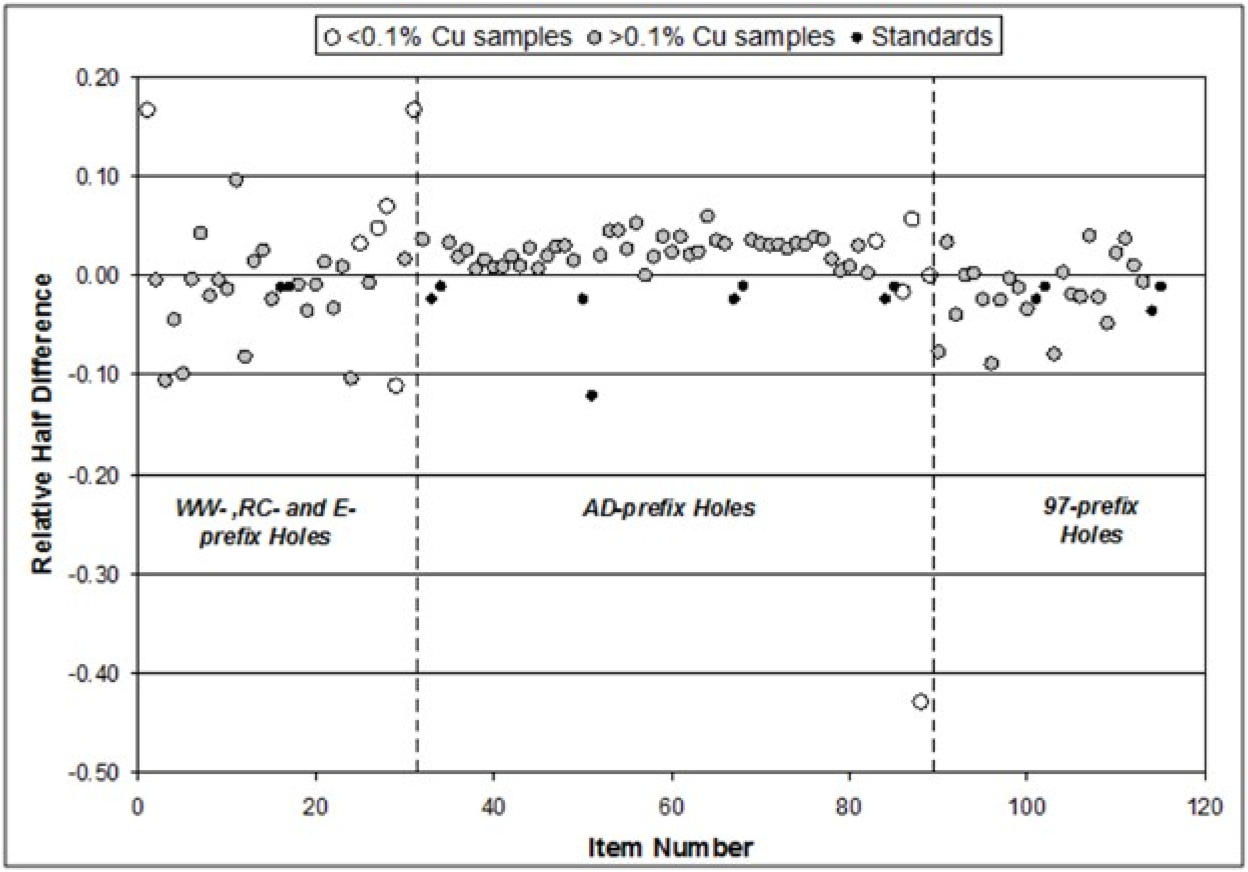
The analytical results from the standard samples are shown in Figure 11-1; both include standards supplied by the Pinto Valley Operations (PVO) project team and those used by Skyline Labs for internal QA/QC. A relative bias of -2% (Skyline is lower than the mean values) is determined from these samples, with a relative precision of 4% for the standards greater than 0.1% Cu and 10% for the reference sample containing 0.09% Cu. These results provide an estimated precision for pulp and instrumentation sampling.



Note: PVO refers to PVM under BHP ownership; wt% Cu = %Cu

Figure 11-1: Analytical Results from Standard Reference Materials, 2006

The re-assay program for stored pulp samples shows that historical quality control measures used in the PVM analytical laboratory were within acceptable limits, although variables: at times they were extremely good, but at other times they were less so. The relative half differences (RHDs) of the samples are presented in sequential order in Figure 11-2, which shows that the drillhole series is well correlated with the variability and bias of repeat assays. Because of the consistent results from the reference standards included in the samples submitted to Skyline, it can be assumed that the variability in the drilling programs originate with the analytical precision at PVM and not at Skyline.



Note: Samples are shown in sequential order of analyses, but are grouped by drillhole identification; percentages refer to %Cu.

Figure 11-2: Relative Half Differences in Replicate Pulp Analyses (compares original PVM copper assays with Skyline repeats), 2006

Table 11-1 shows the statistical summaries of the 2006 QA/QC program on replicate pulp assays, broken down by drilling campaign. Although similarities exist between the WW-, RC-, and E-Series holes, there are only limited samples from the latter two series, and these tend to be low-grade. Because the WW- and 97-Series holes were both drilled at approximately the same time and were drilled at a much different time than the remaining holes, these holes should be categorized as having similar laboratory quality practices. The AD-Series holes seem to have been assayed under different protocols and are grouped with the E-Series because of their similar drilling dates. Additional information presented below further suggests this grouping for the purpose of estimating analytical uncertainty. Based on the replicate pulp program, the AD- and E-Series holes have a relative bias of +2.5% (original assays higher than Skyline) and precision of 6%, compared to the remaining holes that have a bias and precision of approximately −1.5% and 9%, respectively.

Table 11-1: Analytical Results for Replicate Pulp Assays 2006 Pinto Valley Mine QA/QC Program

Drillhole Program	Data Subset	No.	Copper Average (%)		Linear Fit Slope	Average		Relative Precision ‡
			Skyline	PVM		RHD*	ARHD†	
WW-, RC- & E-Series	All Data	29	0.254	0.247	0.95	0	0.049	0.138
	>0.1% Cu Only	23	0.313	0.303		−0.017	0.035	0.103
AD-Series	All Data	50	0.277	0.291	1.05	0.016	0.034	0.135
	>0.1% Cu Only	45	0.302	0.318		0.025	0.025	0.058
97-Series	All Data	22	0.300	0.290	0.91	−0.016	0.029	0.08
All Samples	All Data	101	0.275	0.278	1.00	0.004	0.037	0.123
	>0.1% Cu Only	90	0.304	0.307		0.005	0.029	0.074

* RHD defined as (PVM − Skyline)/(PVM + Skyline).
† Absolute relative half difference.
‡ Relative precision calculated as the square root of the average squared relative half difference at the 95% confidence level, as estimated through Student's t-distribution.

Fifteen field duplicates of split core from drillholes lying in sequence between E-21 and E-60 are summarized in Figure 11-3. The relative bias between the two core halves is nearly identical to that seen in lab assays for the AD-Series holes, with PVM core assays approximately 3% higher grade than the replicate values. The relative precision of the two core halves at copper grades above 0.1% Cu is slightly more than double the analytical precision of AD-Series pulp replicates. The AD-Series replicate pulp assays plot on a near-perfect least squares linear fit from the E-Series duplicate core assays (Figure 11-3), which suggests an excellent correlation between field duplicates.

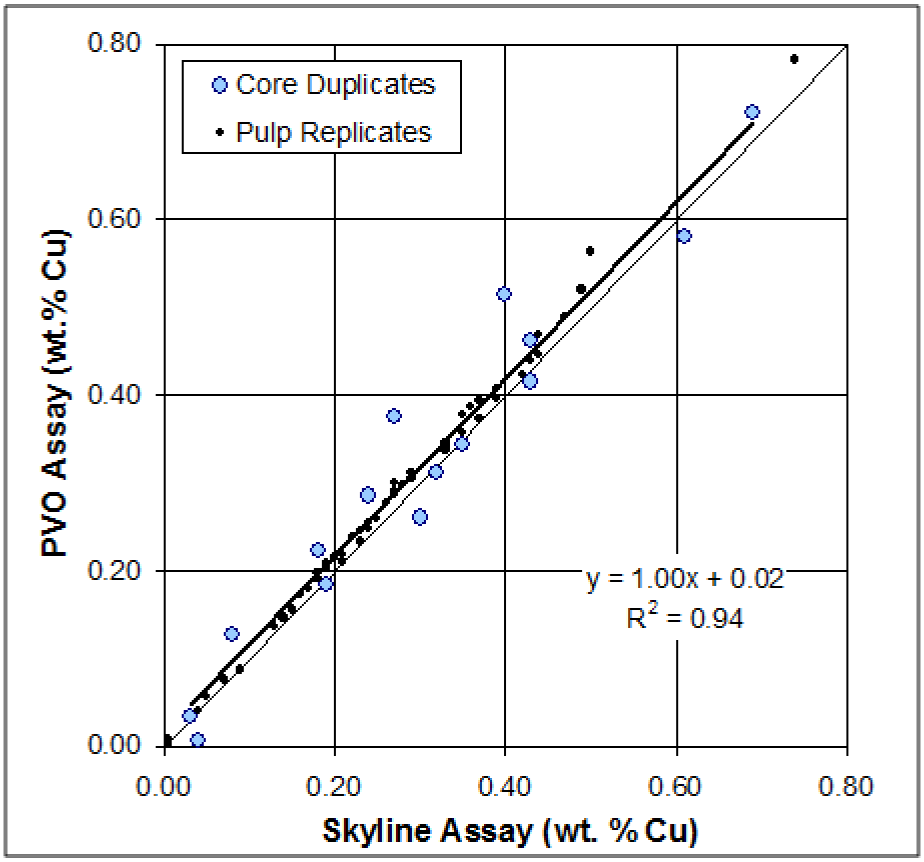


Figure 11-3: Comparison of 15 Field Duplicate Samples from 2006 PVM QA/QC Program

Based on the E- and AD-Series results, the total relative sampling standard deviation for the split core samples above 0.1% Cu is estimated to be approximately 8%; 86% of the sampling variance is due to core splitting and sample preparation errors, and 14% is due to analytical variance within the PVM lab. Instrumentation errors associated with the QA/QC analytical process are responsible for approximately 0.5% of the total variance. The relative bias of approximately 2.5% between PVM and Skyline is the result of an absolute bias of −2.7% between Skyline and the international standard; these results are summarized in Table 11-2.

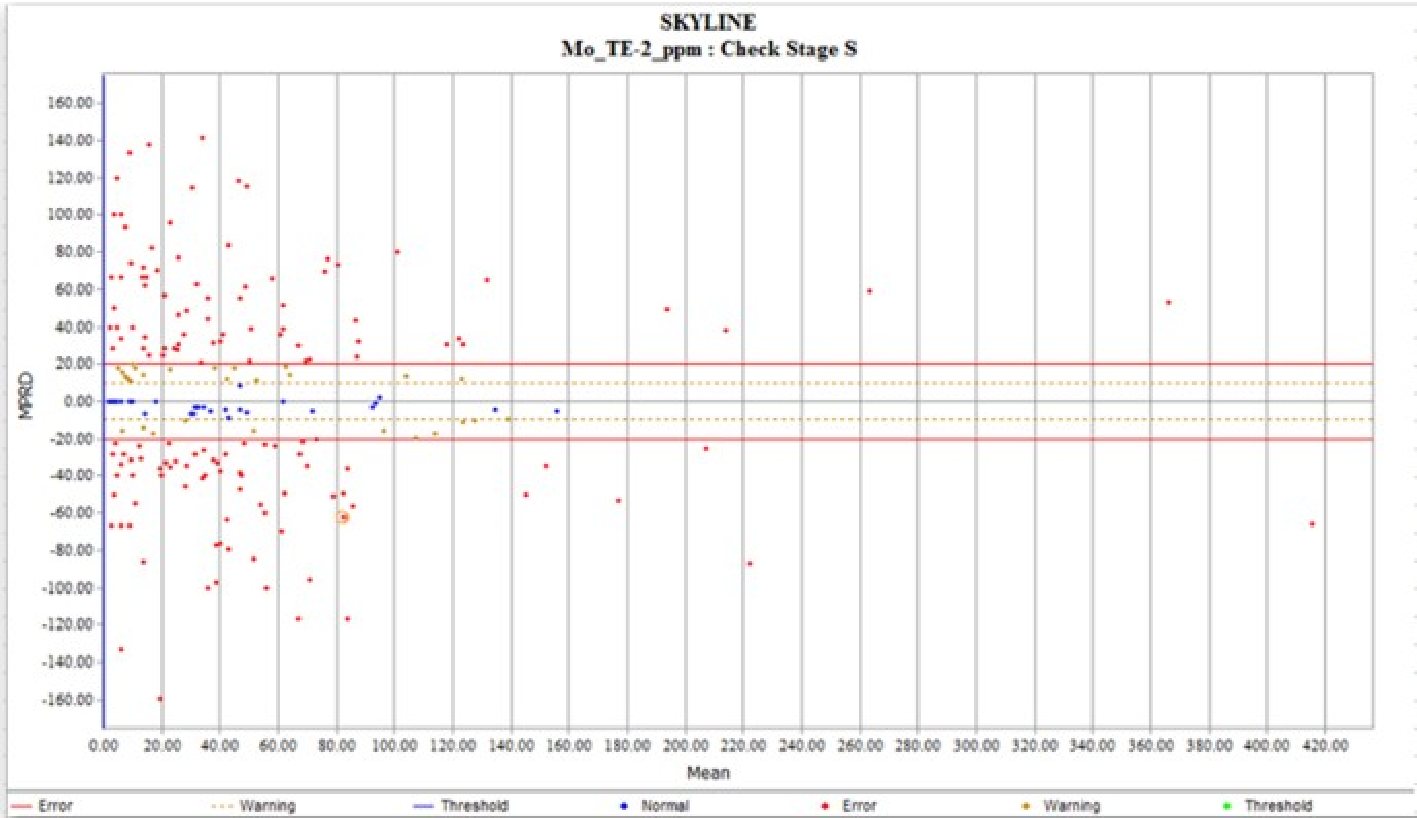
The sampling and preparation errors of the RC samples could not be fully determined due to a lack of field duplicates. Field sampling of RC cuttings are generally associated with lower variances than sampling of drill core, which can offset the higher laboratory variances measured for the 1996 and 1997 programs. The analytical bias seen in these samples, corrected for the Skyline bias, is estimated to be 4% lower than the international standards.

Table 11-2: Total and Stepwise Sampling Estimates and Analytical Variances

Drillhole Samples	No.	Total Relative Errors			Stepwise Relative Error		
		Bias	Standard Deviation	Variance	Bias	Standard Deviation	Variance
Core Sampling Variance (E-Series core duplicates)	12	0.032	0.0760	0.00577	0.006	0.070	0.00495
PVM Analytical Variance (AD-Series pulp replicates)	45	0.025	0.0287	0.00082	−0.001	0.028	0.00079
Skyline Analytical Variance (reference material)	7	−0.027	0.0058	0.00003	−0.027	0.006	0.00003
RC Variance (WW- and 97-Series)	Unknown						
PVM Analytical Variance (WW- and 97-Series pulp replicates)	43	−0.017	0.0454	0.00206	−0.044	0.045	0.00203
Skyline Analytical Variance (reference material)	7	−0.027	0.0058	0.00003	−0.027	0.006	0.00003

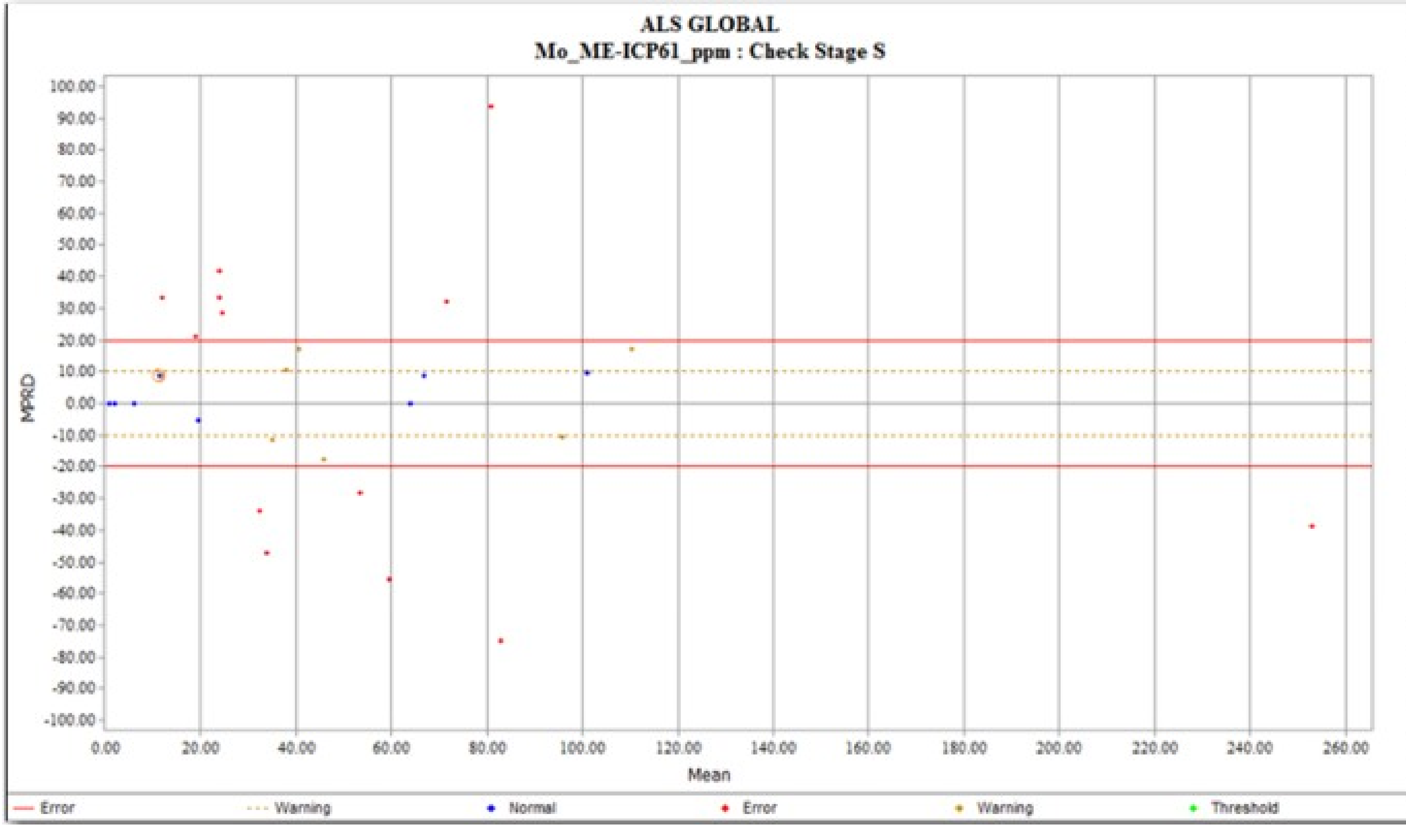
The PVM QA/QC procedures have been based on leading practices as defined by BHP and used throughout BHP's group of assets. These have been developed in conjunction with other BHP base metal mines. These processes continue to be utilized on-site, to the best of the authors' knowledge.

Prior to the 2010 through 2013 drilling campaigns, there is limited information with respect to the molybdenum analyses and QA/QC. Charts shows the respective laboratory; Skyline and ALS Global, results of the analyses for the field (Figure 11-4 and Figure 11-5), coarse (Figure 11-6 and Figure 11-7) and pulp (Figure 11-8) duplicates. The molybdenum QA/QC illustrate that quality control measures used at both laboratories are variable and that there is a relatively high failure rate for all analyses methods. As there are no reference sample analyses (ie. Standards), it is difficult to ascertain whether the cause of the issues and lack of analytical precision originate at PVM or at Skyline and ALS Global however with both laboratories experiencing similar failure rates.



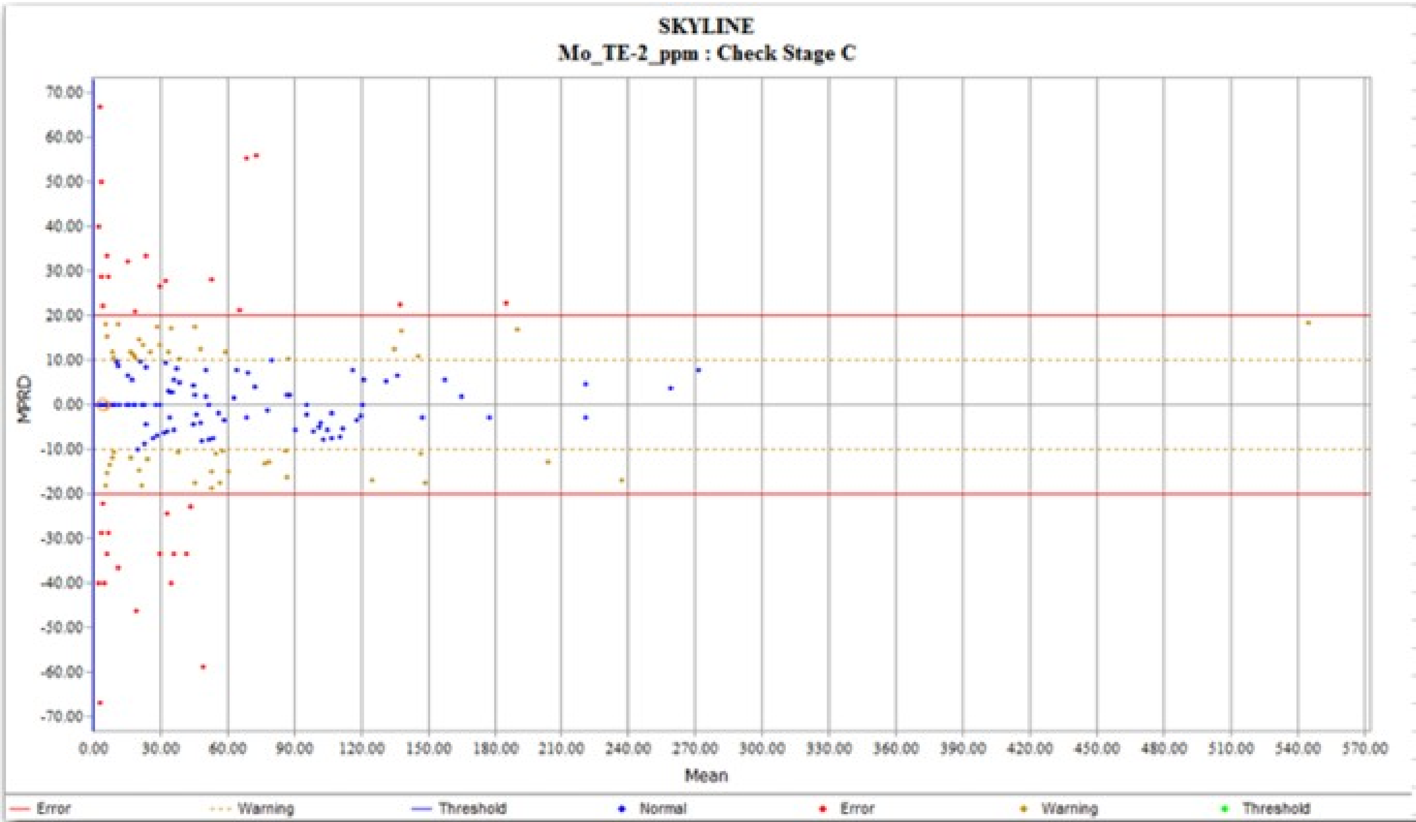
Statistics		
Mo_TE-2_	Original	Check
# of Analy	217	
# of Analy	149	
# Outside	179	
# Outside	151	
Mean	47.53	48.2074
Median	30	32
Min	2	2
Max	463	552
Range	461	550
Variance	3305.26	3680.63
Coeff. of V	1.2096	1.2585
Pop. Std.	57.4914	60.6682
q1 (25%)		
q3 (75%)		
Bias	0.0143	
Corr. Coef	0.7977	
RMA Error	2.8525	
RMA Error	0.0401	
RMA Error	52.6604	
Mean AM	40.2641	
% of Popu	87.0968	
% of Popu	82.4885	
% of Popu	76.0369	
% of Popu	69.5853	

Figure 11-4: Comparison of Field Duplicate Samples – Skyline



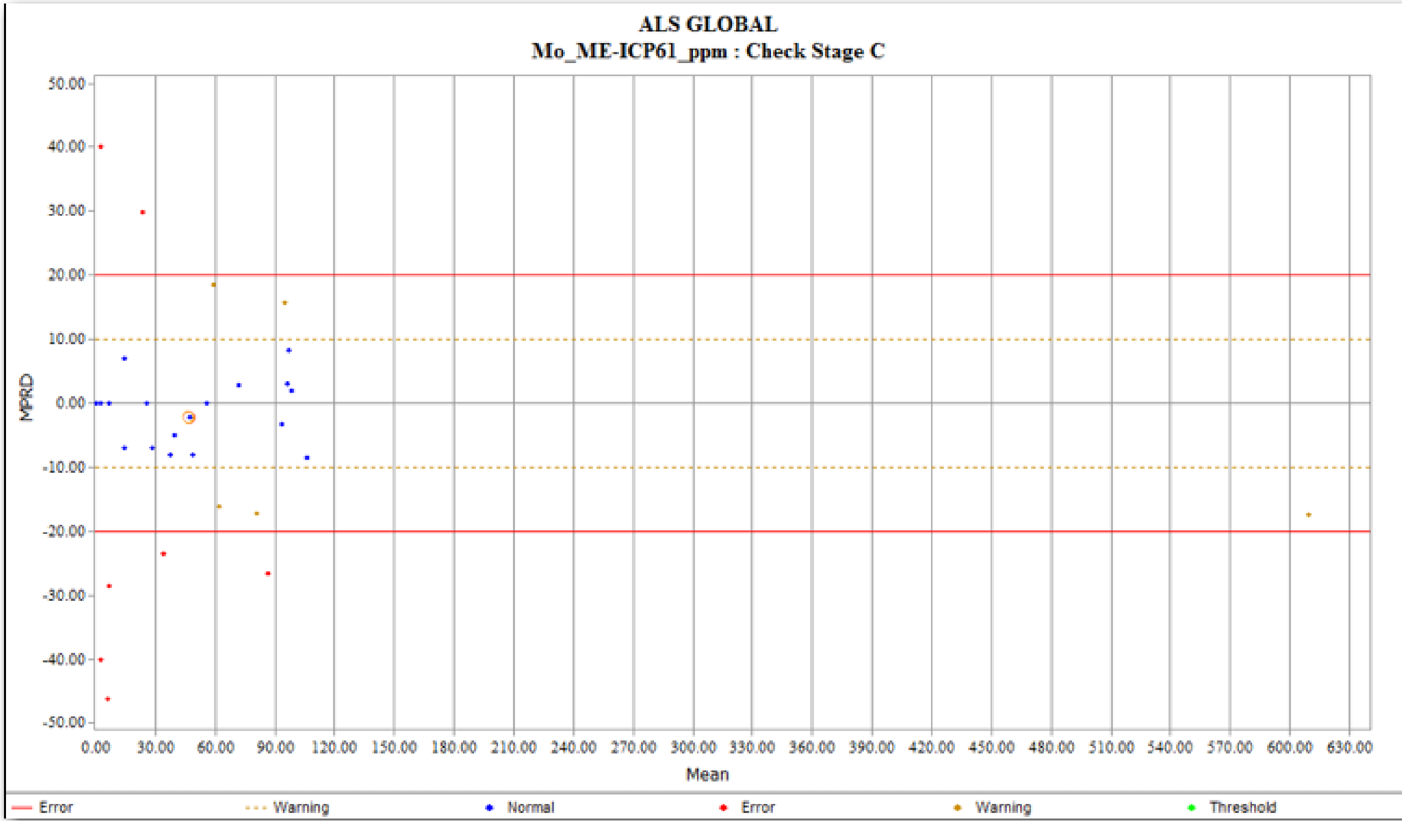
Statistics		
Mo_ME-IC	Original	Check
# of Analy	37	
# of Analy	8	
# Outside	28	
# Outside	19	
Mean	53.5405	54.7838
Median	40	38
Min	1	1
Max	204	302
Range	203	301
Variance	1934.9	2862.44
Coeff. of V	0.8216	0.9766
Pop. Std.	43.9875	53.5018
q1 (25%)		
q3 (75%)		
Bias	0.0232	
Corr. Coef	0.831	
RMA Error	7.4244	
RMA Error	0.1005	
RMA Error	40.4608	
Mean AM	27.3779	
% of Popu	89.1892	
% of Popu	75.6757	
% of Popu	62.1622	
% of Popu	51.3514	

Figure 11-5: Comparison of Field Duplicate Samples – ALS Global



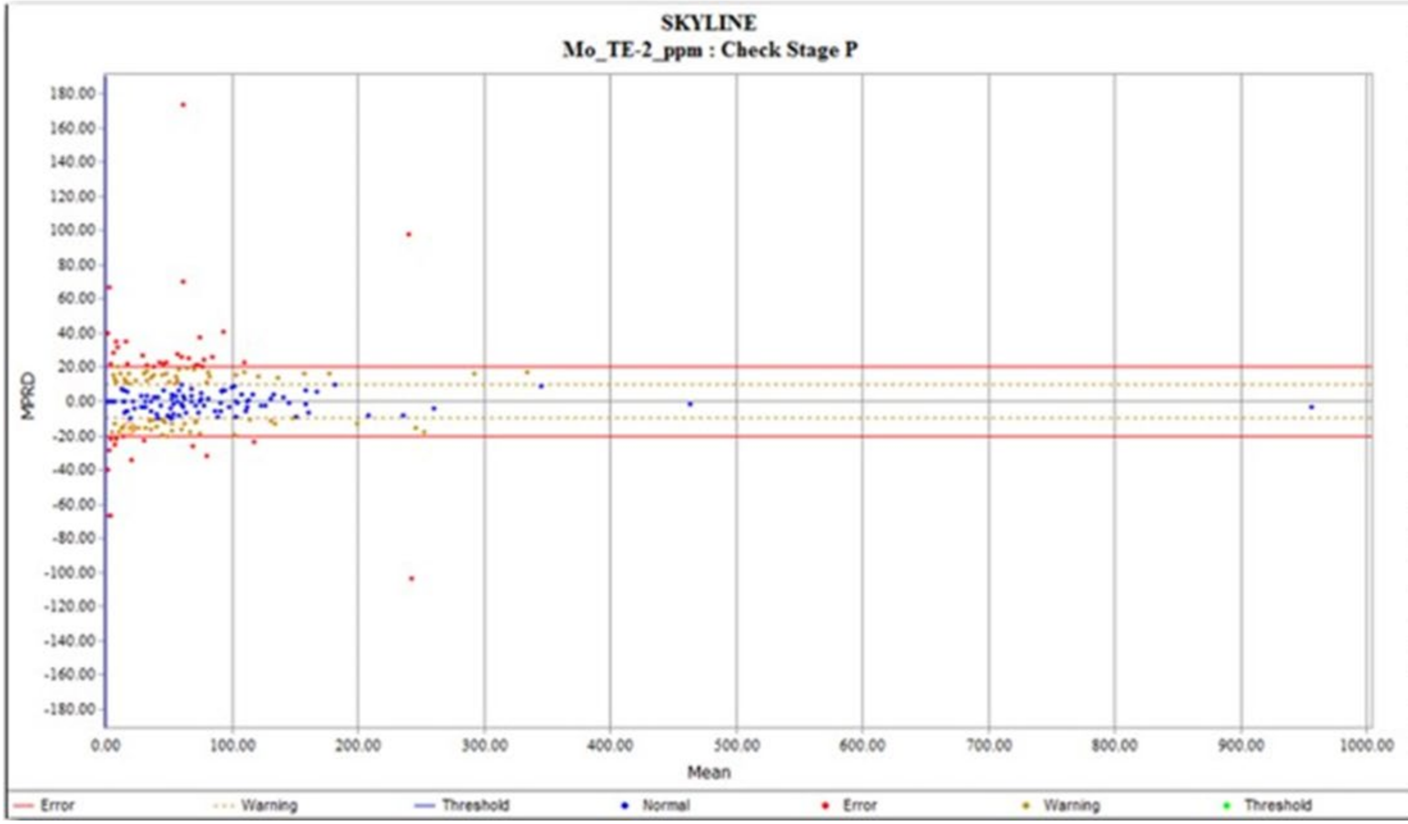
Statistics		
Mo_TE-2_	Original	Check
# of Analy	222	
# of Analy	147	
# Outside	102	
# Outside	42	
Mean	48.9009	48.3153
Median	27.5	26
Min	2	2
Max	596	495
Range	594	493
Variance	4245.16	3717.61
Coeff. of V	1.3324	1.262
Pop. Std. I	65.1549	60.9722
q1 (25%)		
q3 (75%)		
Bias	-0.012	
Corr. Coef	0.9754	
RMA Error	0.9743	
RMA Error	0.0136	
RMA Error	19.6584	
Mean AM	12.2335	
% of Popu	63.5135	
% of Popu	45.9459	
% of Popu	30.6306	
% of Popu	18.9189	

Figure 11-6: Comparison of Coarse Duplicate Samples - Skyline



Statistics		
Mo_ME-IC	Original	Check
# of Analy	43	
# of Analy	7	
# Outside	19	
# Outside	10	
Mean	68.0233	73.4884
Median	39	41
Min	1	1
Max	557	663
Range	556	662
Variance	12805.8	18090.6
Coeff. of V	1.6636	1.8302
Pop. Std. I	113.163	134.501
q1 (25%)		
q3 (75%)		
Bias	0.0803	
Corr. Coef	0.9972	
RMA Error	1.6942	
RMA Error	0.0126	
RMA Error	13.6129	
Mean AM	13.3641	
% of Popu	62.7907	
% of Popu	44.186	
% of Popu	44.186	
% of Popu	23.2558	

Figure 11-7: Comparison of Coarse Duplicate Samples – ALS Global



Statistics		
Mo_TE-2_	Original	Check
# of Analy	225	
# of Analy	117	
# Outside	112	
# Outside	44	
Mean	71.4222	69.9511
Median	53	52
Min	2	2
Max	940	973
Range	938	971
Variance	8082.77	8287.67
Coeff. of V	1.2588	1.3014
Pop. Std. I	89.9042	91.0367
q1 (25%)		
q3 (75%)		
Bias	-0.0206	
Corr. Coef	0.9574	
RMA Error	1.8439	
RMA Error	0.018	
RMA Error	36.5204	
Mean AM	13.6739	
% of Popu	68	
% of Popu	49.7778	
% of Popu	35.5556	
% of Popu	19.5556	

Figure 11-8: Comparison of Pulp Duplicate Samples – Skyline

11.3 QA/QC Post-2006

Assays are imported to the PVM server for approval. This is done for each batch according to the criteria above. The following procedures are used to approve the QA/QC for each batch:

- Enter the PVM data portal, and select the area, project, and batch list.
- Review QA/QC results, particularly “Company Standards,” blanks, and “Lab Standards,” for approval.
- Review the field duplicates, coarse duplicates, pulp duplicates, and lab assay repeats as well.

This information is then compiled to generate a QA/QC report detailing any errors associated with the splitting and crushing procedures for that particular batch.

All drilling since the May 2015 Mineral Resource estimate presented in the PV3-2016-PFS was performed with a comprehensive QAQC program in place that included the regular submission of blanks, certified reference materials (CRM) and duplicate samples along with the in-house QAQC program conducted by the analyzing lab, Skyline, and reported on all received assay certificates. Assay results were routinely inspected when received to ensure that all QAQC criteria were satisfied before results were deemed acceptable and entered into acQuire™. Assays incorporated into the PV3-2016-PFS estimate included samples from the GTH-14, GTH-15, and PVRC-15 series for a total of 2,426 samples. (Capstone, 2016) The Mineral Resource estimate presented in this report includes an additional 54 drillholes from the RC-16, RC-17, RC-18, RCD-17 and RCD-19 series, along with two diamond drillholes for a total of 4,842 additional samples.

Blanks consisted of high purity silica chips in 2015 and felsic intrusive chips from 2016 to 2019. Blanks were inserted in the sample stream in order to detect any contamination. A total of 206 blanks were inserted over the course of the drilling programs, with the vast majority of these reporting below detection limit values for Cu and Mo. All analyses fall below the 5x detection limit failure criteria (detection limits are 0.01% for Cu and 0.0005% for Mo) that have been specified, meaning all blank analyses are considered acceptable.

CRM purchased from CDN Resource Laboratories Ltd. of Langley, British Columbia, Canada were included in every sample submittal to Skyline to confirm the accuracy of analysis across various grades. CRM failures were set as values beyond three standard deviations from the certified mean, or more than two consecutive values between two and three standard deviations, either above or below the certified mean. Of the 73 CRM submitted from 2015 through the end of resource drilling in 2019, there were four failures for copper and one failure for molybdenum, including one CRM that failed for both copper and molybdenum. The CM-37 failure for both copper and molybdenum was attributed to a sample switch caused by incorrectly inserting sample tags at the core yard and was not reanalyzed. CRM typically returned values distributed about the mean for copper with the exception of CM-22 (biased 3% low overall with two values just below acceptable limits that were not reanalyzed), however molybdenum was biased low for every CRM (ranging from 2 to 12% low, within acceptable limits for the CRM). CRM performance is summarized in Table 11-3. The two CM-22 failures were not reanalyzed because all other values in the drillhole were much lower than the certified mean of the CRM.

Table 11-3: CRM Performance Summary, 2015 to 2019

CRM Name	Acceptable Range	# Inserted	# Failures	Performance Comments
CM-22[1]	0.995 ± 0.039% Cu 0.020 ± 0.003% Mo	7	2 Cu	Cu – Low failures (0.940% and 0.955% vs. 0.956%) Cu - Biased low by 3% overall Mo – Biased low by 12% overall
CM-33[2]	0.346 ± 0.030% Cu 0.025 ± 0.003% Mo	23	0	Mo – Biased low by 9% overall
CM-35[3]	0.248 to 0.018% Cu 0.029 to 0.003% Mo	12	0	Cu – acceptable, values do not appear biased Mo – Biased low by 2% overall
CM-37[3]	0.212 to 0.018% Cu 0.027 to 0.003% Mo	15	1 – Cu+Mo	1 failed sample for both Cu and Mo, attributed to a sample tag insertion switch at the core yard Cu – Biased high overall by 2% Mo – Biased low overall by 9%
CM-31[2]	0.082 to 0.006% Cu 0.009 to 0.003% Mo	16	1 Cu	Cu – Minor fail at upper limit (0.09 vs. 0.088% Cu) Mo – Biased low overall by 12%

CRM acceptable ranges are ±3 standard deviations. CRM were purchased from CDN Resource Laboratories Ltd., Langley, Canada.
1. CM-22 was prepared using 767 kg of a granitic rock blended with 33 kg of a Cu-Au-Mo concentrate.
2. CM-33 and CM-31 were prepared using ore from a North American calc-alkalic copper-molybdenum altered quartz monzonite porphyry intrusion. Mineralization is principally pyrite, chalcopyrite and molybdenite that occurs in veins, stockworks and disseminations.
3. CM-35 and CM-37 were prepared using ore from a south-central Far East K-silicate, silica and sericite altered intermediate volcanic and related intrusive rocks with Cu-Au porphyry-style mineralization.

Duplicate samples include field, coarse, and pulp duplicates, which are inserted in the sample stream to test the variation between measurements at the various stages in the sample collection, preparation, and analysis processes, respectively. Overall analytical variance for duplicate pairs inserted over the course of the drilling programs can be seen in Table 11-3 and Table 11-4 for Cu and Mo, and display expected trends of decreasing variance moving from field to pulp duplicates.

Table 11-4: Duplicate Pair Analytical Variance for %Cu

Duplicate Type	Bias (%Cu)	Correlation Coefficient	Average Mean Pair Relative Difference (%)
Field	-0.008	0.967	10.57
Coarse	-0.014	0.980	5.47
Pulp	-0.012	0.998	2.26

Table 11-5: Duplicate Pair Analytical Variance for %Mo

Duplicate Type	Bias (%Mo)	Correlation Coefficient	Average Mean Pair Relative Difference (%)
Field	0.040	0.901	15.90
Coarse	0.049	0.957	11.99
Pulp	0.062	0.992	10.87

A total of 245 field duplicates were submitted for analysis with a scatterplot of the results shown in Figure 11-9 and Figure 11-10. Warning and error lines are set at respective 15% and 20% limits with regard to the relative difference between duplicate pair assays. Returned Cu analyses show that 17 duplicate pairs lie between warning and error limits and 31 lie outside of error limits, while Mo analyses show that 8 duplicate pairs lie between warning and error limits and 70 lie outside of error limits.

A total of 104 coarse duplicates were submitted for analysis with a scatterplot of the results shown in Figure 11-11 and Figure 11-12. Warning and error lines are set at respective 10% and 15% limits with regard to the relative difference between duplicate pair assays. Returned Cu analyses show that six duplicate pairs lie between warning and error limits, and nine lie outside of error limits; Mo analyses show that 5 duplicate pairs lie between warning and error limits and 27 lie outside of error limits. This scatter results in correlation coefficients of 0.980 and 0.957 for Cu and Mo, respectively, with average mean pair relative differences (AMPRD) of 5.47 for Cu and 11.99 for Mo.

A total of 92 pulp duplicates were submitted for analysis with a scatterplot of the results shown in Figure11-13 and Figure 11-14. Warning and error lines are set at respective 5% and 10% limits with regard to the relative difference between duplicate pair assays. Returned Cu analyses show that nine duplicate pairs lie between warning and error limits, and five lie outside of error limits; Mo analyses show that five duplicate pairs lie between warning and error limits and 26 lie outside of error limits. This scatter results in correlation coefficients of 0.998 and 0.992 for Cu and Mo, respectively, with average mean pair relative differences (AMPRD) of 2.26 for Cu and 10.87 for Mo.

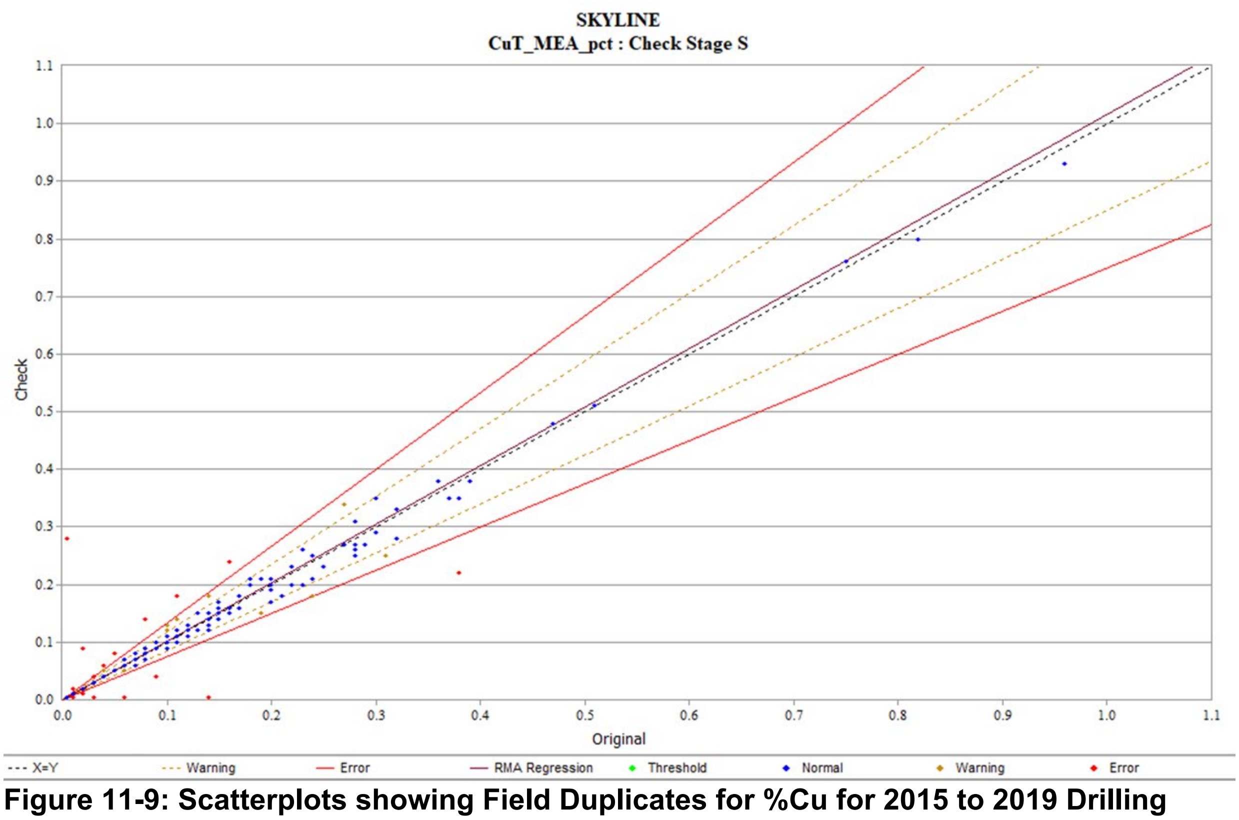


Figure 11-9: Scatterplots showing Field Duplicates for %Cu for 2015 to 2019 Drilling

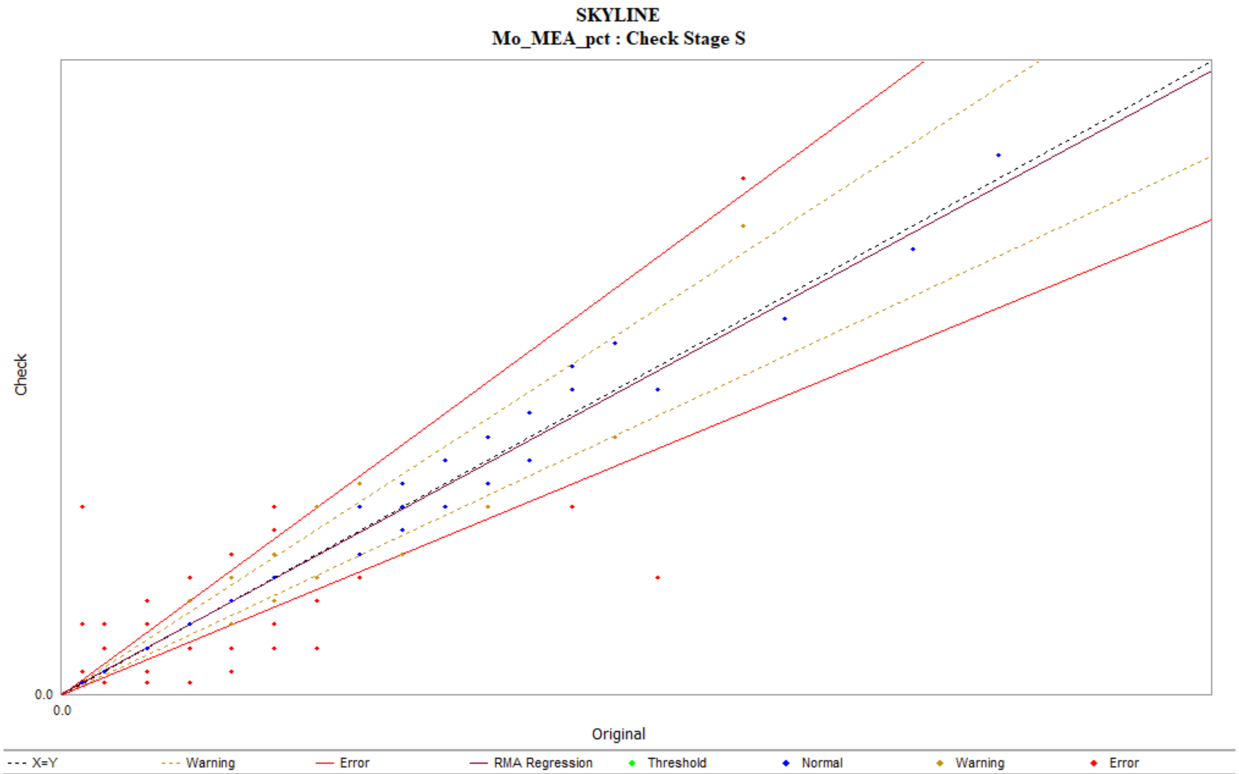


Figure 11-10: Scatterplots showing Field Duplicates for %Mo for 2015 to 2019 Drilling

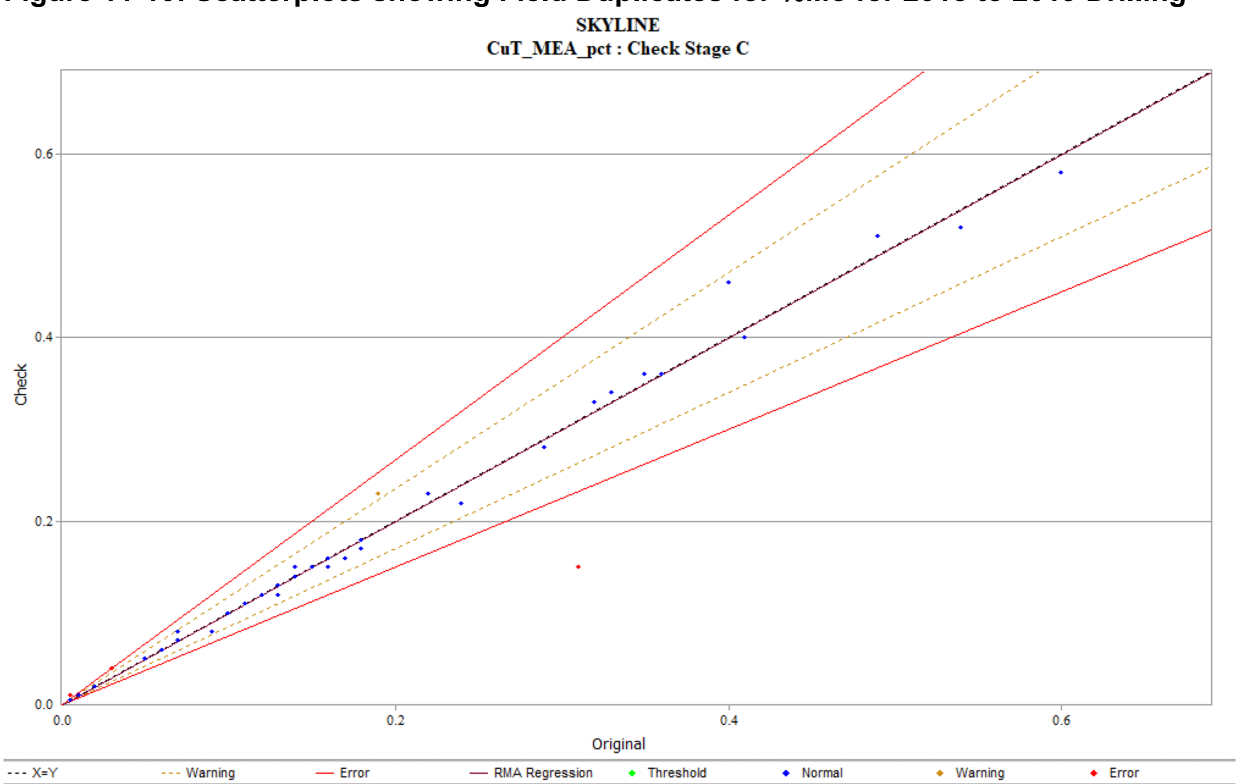


Figure 11-11: Scatterplots Coarse Duplicates for %Cu for 2015 to 2019 Drilling

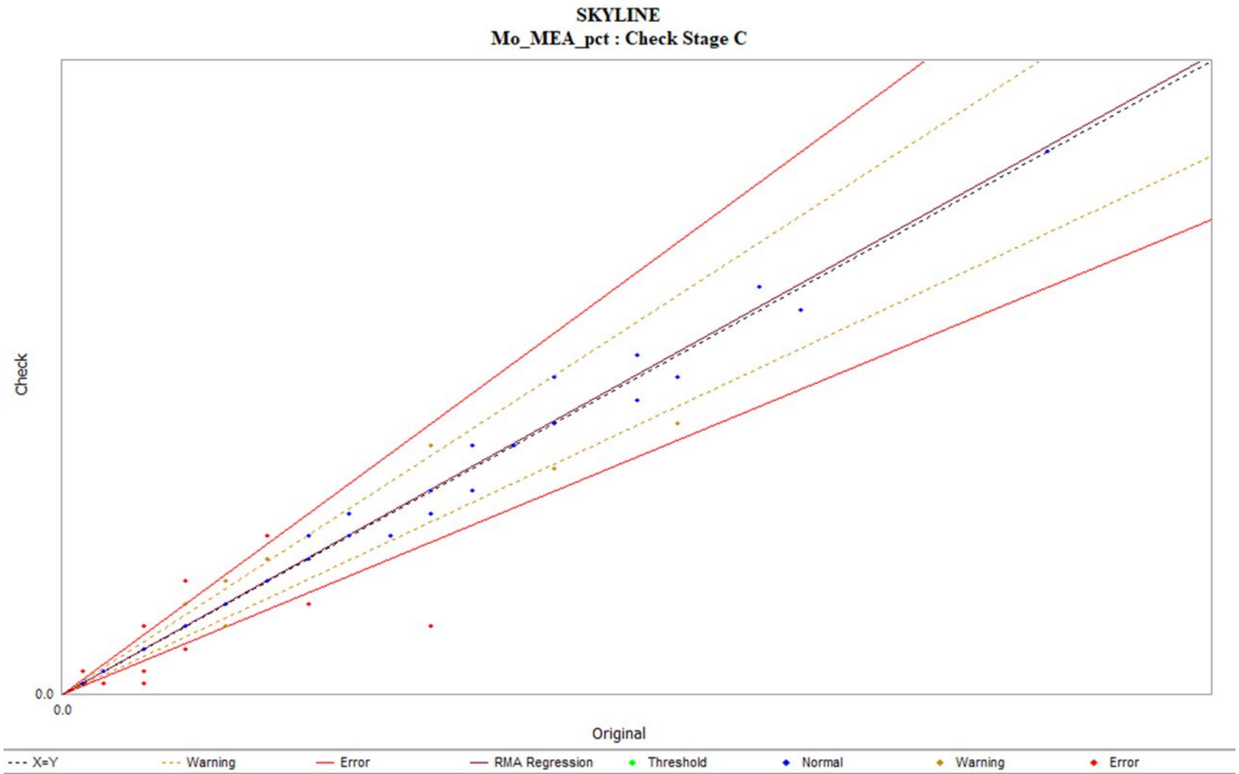


Figure 11-12: Scatterplots Coarse Duplicates for %Mo for 2015 to 2019 Drilling

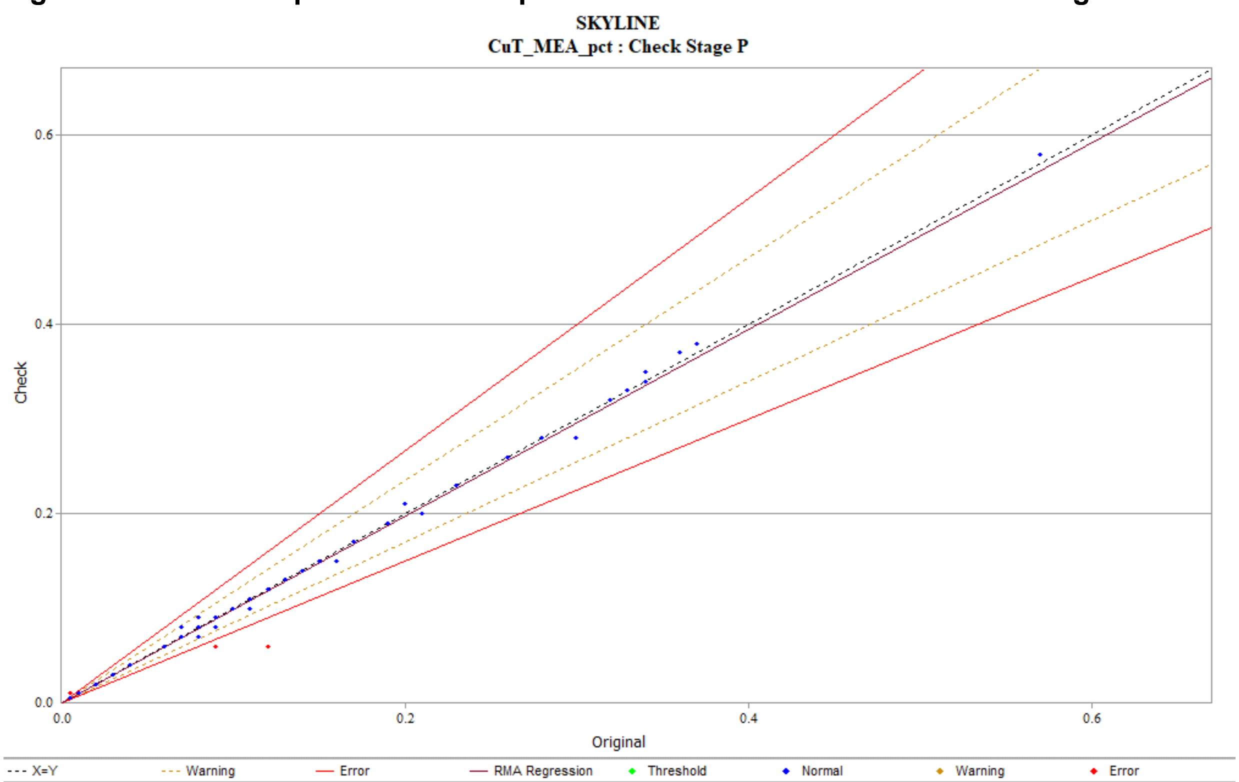


Figure 11-13: Scatterplots Pulp Duplicates for %Cu for 2015 to 2019 Drilling

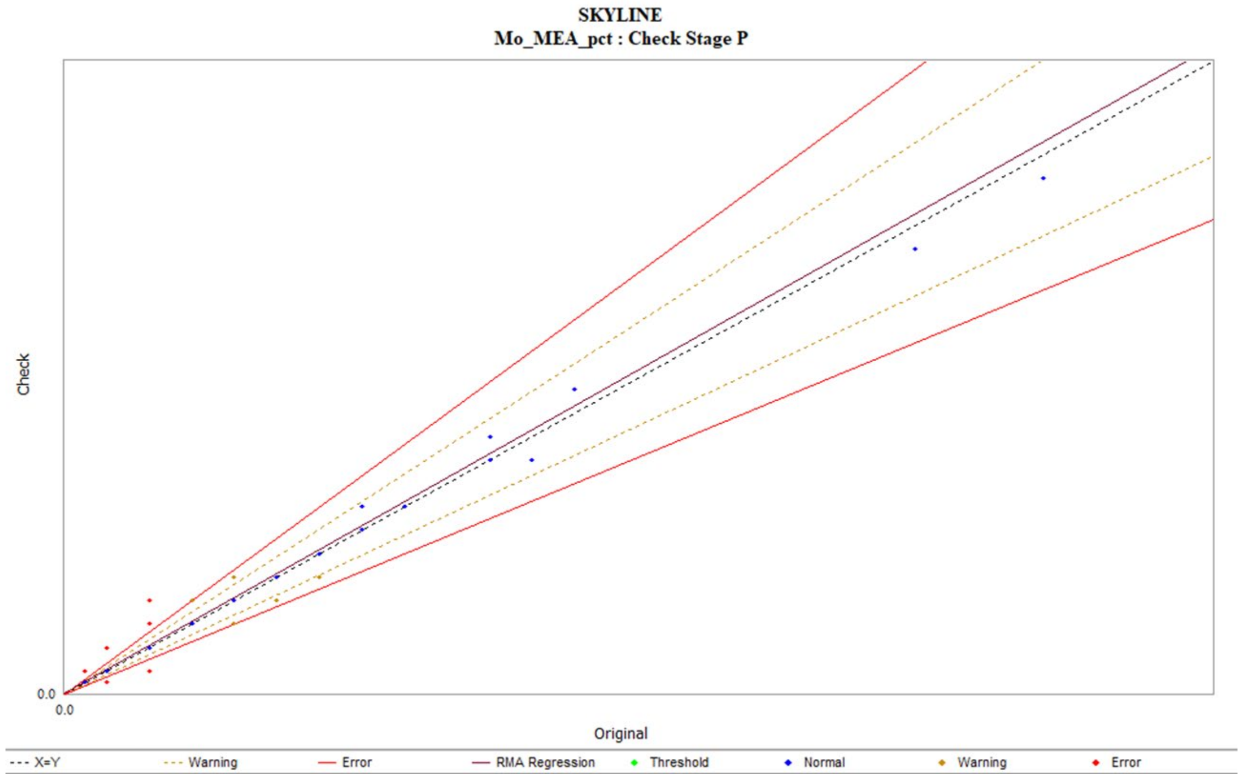


Figure 11-14: Scatterplots Pulp Duplicates for %Mo for 2015 to 2019 Drilling

11.4 Bulk Density

From August to October 2018, PVM measured density on drillcore from five major lithologies with 17 types of alteration. A total of 305 values were used to create average bulk densities for the five lithologies. (PVMC, 2019) The Mineral Resource and Mineral Reserve estimates in this report use these bulk density values.

Drillcore was weighed three times, first in air (unsealed), then again in air after application of a permanent sealant, and finally in water (sealed).

The density was then determined using the following formula:

$$Density = \frac{Ma}{(Msa - Msw) - ((Msa - Ma)/Psealer)}$$

Where:

Ma is the mass of the dry rock in air (first measurement)

Msa is the mass of the sealed rock in air (second measurement)

Msw is the mass of the sealed rock in water (third measurement)

Psealer is the density of the liquid sealant

Data collected was checked for reasonableness and completeness; two values where weight in air was recorded as less than weight in water were rejected, as were four values where alteration was not recorded. These rejected measurements are not included in the total of 305.

QA/QC on the data included internal checks to confirm values were sufficiently reproducible between geologists on adjacent sample intervals (33 Ruin Granite samples, 16 Diabase samples) and a comparison of the entire 10-foot sample interval versus shorter lengths (four samples), as well as submission of confirmation samples on adjacent intervals by different methods at ALS (35 samples, paraffin coating used) and Skyline (14 samples, no coating used). Variability was within expected parameters because identical samples could not be used for the internal or external checks as a permanent sealant was applied at PVM.

Table 11-6: Updated Density Measurements, 2018

Lithology	# Measurements	Bulk Density (2018)
11 – Ruin Granite	171	2.61 g/cm^3
30 - Diabase	49	2.75 g/cm^3
45 - Limestone	10	2.71 g/cm^3
50 – Granodiorite	50	2.58 g/cm^3
60 - Granite Porphyry	25	2.62 g/cm^3

In the PV3-2016-PFS, a density value of 2.51 g/cm^3 was applied to all rock types. Overall, the densities used in this Mineral Resource and Mineral Reserve are 3% higher.

11.5 Recommendations

To increase certainty of copper and molybdenum analysis, CRM used should be created from PVM material and should be inserted more frequently in analytical batches with at least one per batch of 20 samples (5% insertion rate).

An approximate cost to create CRM from PVM materials is US$ 10,000 to cover preparation including homogenization, round-robin testing and certification.

Timely communication with the external resource laboratory is required to resolve issues, such as the low bias observed in some CRM.

11.6 Conclusions

In the QP’s opinion, the sample preparation, analysis QA/QC and security protocols follow accepted industry standards. Based on the data and results, it is the authors opinion that the compiled database is valid and of sufficient quality to use in mineral resource estimation.

12 Data Verification

12.1 Verification of Geology, Drilling, Sampling Preparation, Analyses and Security

12.1.1 Verification of Geology, Drilling, Sample Preparation, Analyses and Security

Klaus Triebel is present at the mine on a weekly basis. He conducts frequent pit mapping and remote data analyses (i.e. propeller drone flight interpretations) to verify lithology and structural interpretations. He is familiar with most of the literature references cited where geology is presented. He was in charge of the 2019 drilling campaign including the bidding process and drilling activities. He frequently visited the core shed to assure proper logging and sampling of RC cuttings and core and monitored the shipping process to insure security of the samples. Under his supervision assays were entered into the database.

No issues were identified and the QP is confident that the statements regarding geology, drilling, sample preparation, analyses and security in this Technical Report are valid.

12.1.2 Verification of Geology, Drilling, Sampling, Analyses and Security for use in Mineral Resource Estimates

Garth Kirkham, P.Geo., FGC, visited the property on May 14, 2013 and April 16-17, 2015. The site visits included an inspection of the core logging facilities, offices, pit tour, outcrops, drill collars, core storage facilities, core receiving area, and core sawing stations, and a tour of the major centers and surrounding towns that are affected by the mining operation.

The tour of the offices and core logging and storage facilities showed a clean, well-organized, professional environment. On-site staff led the author through its chain of custody and methods used at each stage of the logging and sampling process.

The QP randomly selected four complete drillholes from the database and laid the core out at the core storage area. Site staff supplied the logs and assay sheets so the author could verify the core and logged intervals. The data correlated with the physical core, and no issues were identified. In addition, the author toured the complete core storage facility, pulling and reviewing core throughout the tour. No issues were identified and recoveries appeared to be very good to excellent.

The QP is confident that the data and results are valid, based on the site visit and inspection of all aspects of the project; this confidence extends to the methods and procedures used. It is the opinion of the independent author that all work, procedures, and results have adhered to best practices and industry standards required by NI 43-101. No duplicate or verification samples were taken to verify assay results in historical work, but the author believes that the work was conducted by a well-respected, large, multi- national company that employs competent professionals who adhere to industry best practices and standards. Current practices include additional QAQC to verify assay results.

The QP also visited Skyline on 15 May 2013. The laboratory tour was performed by Jim Martin, Senior Chemist and Arizona Registered Assayer (No. 11122), who provided a complete review of the laboratory facilities, laboratory preparation procedures, instrumentation, assay methods, QA/QC protocols, and reporting procedures. The laboratory appeared to be operated in a very professional manner, as is expected from a widely used North American laboratory facility. Skyline, because of its long standing service to many large copper mines, appears to specialize in and have extensive experience with the assay processes and procedures for copper. Skyline has been ISO 17025 certified since 2008.

The reconciliation of production grades as compared to those defined by drill data (both legacy and current) and predicted by the block model which resulted in excellent correlations particularly within the core mine block. Reconciliation of the production data further away from the mine block, particularly within the Castle Dome area were less favorable, however an extensive remodeling of the deposit was completed to rectify these discrepancies and are now within reasonable tolerances.

12.2 Verification of Inputs into Mineral Reserve Estimate

QP Clay Craig compared the Mineral Resource models supporting the Mineral Reserve to drilling, grade control sampling, geological modeling and an evaluation of monthly and annual reconciliations.

The QP has reviewed annual historical values for all costs, as well as reasonability of future projections. These have been used to support parameters used during Lerchs-Grossmann optimization and mine planning.

The QP considers that reconciliations of actual equipment capacities and productivities from recent months and years have been appropriately considered while establishing reliable projections of mining fleet requirements in the mine plan.

The Mineral Resource models and other data provided were confirmed as adequate for use in Mineral Reserve estimation for this Technical Report.

12.3 Verification of Considerations for Geotechnical Factors

QP Edward C. Wellman supervised collection of geotechnical data used in this Technical Report, validated the data using several geotechnical indices combined with laboratory strength testing and observed outcomes of operational activities at the recommended angles. The QP's opinion is the geotechnical data is adequate to support the resulting open pit mine slope angle recommendations included in this Technical Report.

12.4 Verification of Mineral Processing, Metallurgical Testing Data and Recovery

QP J. Todd Harvey obtained data in this analysis directly from the distributed control system, the data historian and the accounting systems. The data was validated by direct observations of the plant performance and review/comparison of historic publicly available audited production figures.

12.5 Environmental, Regulatory and Social or Community Data Verification

Several verification mechanisms are in place for the Pinto Valley Mine to confirm the validity and accuracy of these data for inclusion in this Technical Report.

The QP, Colleen Roche P.Eng. has access to available data as part of her duties directly managing the environmental function at Pinto Valley Mine over the last five years and supporting the function for three additional years in a corporate role including due diligence during the original acquisition. In this capacity she has led hiring efforts for key environmental personnel and engaged consultants to review regulatory submissions, supported the construction of comprehensive environmental databases and overseen the collection of the data ensuring use of industry standard methods and analysis of samples using accredited laboratories. She has also reviewed in detail the site closure and reclamation plan and its costing since 2013. Field work has been conducted routinely by nature of the QP's on-site role to verify the conditions and assumptions that underscore the environmental data used in this report.

Capstone has established internal policies and controls to manage the environmental, regulatory and social or community aspects for Pinto Valley mining operations. These are periodically reviewed by operational and corporate management for their effectiveness in a culture which follows the principle of continuous improvement. The QP is of the opinion that a reasonable level of verification has been completed and that no material issues have been left unidentified in the course of collecting and analysing the data described in this report. In reaching this opinion, the QP has also relied upon the work of other consultants in the specific project areas of this Technical Report (listed in Section 3). Data review and verification undertaken with respect to the environmental and regulatory aspects of the Pinto Valley Mine operation and closure adequately support the summary, conclusions and recommendations presented in the Technical Report in these areas.

13 Mineral Processing and Metallurgical Testing

This report provides an update to the Technical Report published by Capstone on PVM in 2016 (Capstone, 2016). J. Todd Harvey of Global Resource Engineering Ltd (GRE) has been retained to provide updates based on current PVM operations and test work.

PVM has been in continuous operation for approximately 47 years with two copper price-related shutdowns occurring from 1998 to 2007 and from 2008 to 2012, and a short shutdown in 1983. The process plant is a conventional porphyry copper concentrator that produces a primary copper sulfide flotation concentrate and a by-product molybdenum flotation concentrate. The plant flowsheet is typical of its era with primary through tertiary crushing, ball milling and conventional flotation. The mill has undergone a number of process optimizations during its operating life. The most recent upgrades have been undertaken to replace aging equipment and optimize throughput and recovery.

In 2014 Capstone commenced the PV3 project (Capstone, 2014) to define the extension of the mine life. The 2016 Technical Report provided additional technical support for the mine life extension with the addition of new metallurgical test work on future ores and the identification of plant optimization opportunities. This latest Technical Report provides further support for the LOMP presented herein.

13.1 Metallurgical Recovery Overview

The metallurgical recoveries at PVM have been reasonably consistent since the restart in 2014. Copper recovery has averaged 85% with salable concentrate grades ranging from 24.5% Cu to 29.6% Cu with by-product credits for gold, and silver. The LOMP assumption for future concentrate production is 25% Cu. The molybdenum circuit has operated intermittently since the restart and currently a new reagent scheme is being evaluated. Molybdenum recovery has averaged approximately 8% over the last 7 years.

Based on the projected copper feed grades over the life of mine, copper recovery should be consistent with operational levels ranging from 85% to 88% averaging 86% based on an average anticipated feed grade of 0.32% Cu. Similarly, for the molybdenum circuit, recovery is anticipated to range from 9% to 11% averaging 10% based on an average projected feed grade of 0.006% Mo.

These recovery figures do not include any adjustment for potential process improvements.

13.2 Ore Lithologies

The mineralized material at PVM has been classified into a series of lithologies. The main lithologies with relevant rock codes and the life of mine anticipated tonnage distributions are outlined in Table 13-1.

Table 13-1: Lithological Distribution – Life of Mine

Lithology	Tonnage (%)
30 - Diabase	0.03%
50 - Granodiorite	0.24%

60 - Granite Porphyry	2.91%
72 - Aplite	0.40%
11 – Ruin Granite	96.42%
Grand Total	100%

The distribution of mineralized materials at PVM is dominated by the Ruin Granite with a small proportion of Granite Porphyry. As such, the focus of the majority of test work and analysis has been placed on the Ruin Granite. The minor lithologies have been highlighted in the report in terms of their potential impact on plant performance.

13.3 Metallurgical Test Work

13.3.1 Grinding

The 2016 Technical Report provides a detailed analysis of the Bond ball mill work index (BWi) testing for the main lithologies. A wide variety of grinding tests have been undertaken over the mine's life with test data available dating back to 1993. A major test program was undertaken by SGS Minerals Services (SGS Minerals Services, 2013) in 2013 on drill core samples in support of the original PV2 Project. Further grinding analysis was conducted by Base Metallurgical Laboratories (BML) in 2015 (Base Metallurgical Laboratories, 2015). The focus of the BML work was primarily on the Ruin Granite with several samples of mixed lithologies, including Granodiorite and Diabase lithologies (Table 13-2).

Table 13-2: Ore Lithological Distribution for the Life of Mine

Sample	Lithology	Bond Ball Mill Work Index (kWh/t)
ROM Composite	11/30/50 – Ruin Granite/ Diabase/Granodiorite	13.6
Eastern Pushback Low-level Length	11– Ruin Granite	14.1
Eastern Pushback Mid-level Length	11– Ruin Granite	13.2
Eastern Pushback Upper-level Length	11/30 - Ruin Granite/Diabase	17.1
Northern Pushback Low-level Length	11/50 – Ruin Granite/Granodiorite	14.5
Northern Pushback Mid-level Length	11– Ruin Granite	13.2
Aplite Composite	11– Ruin Granite	13.1
Average		**14.1**

In the 2016 Technical Report (Capstone, 2016) the BWi was tabulated for each lithology as shown in Table 13-3.

Table 13-3: 2016 PV3 Technical Report Lithological Distribution and Bond Work Index – Life of Mine

Lithology	Tonnage (%)	Bond Ball Mill Work Index (kWh/t)
30 - Diabase	2.9%	17.1
50 - Granodiorite	0.3%	13.1
60 - Granite Porphyry	2.4%	15.0

72 - Aplite	0.4%	13.8
11 – Ruin Granite	94.1%	14.1-14.9
Total/Average	**100%**	**14.5**

The most recent test work indicates that the Ruin Granite work index is distributed over a narrow range from 13.1 kWh/t to 14.1 kWh/t. Previous analysis completed by BHP indicated that the Ruin Granite may have a bimodal work index distribution being split into "soft" and "hard" classifications depending on the location within the pit with the work index ranging from approximately 13 to 15 kWh/t.

Diabase samples have consistently shown a high hardness ranging from 17.0 kWh/t to 17.5 kWh/t. The proportion of Diabase over the life of mine has been reduced and now makes up only 0.03% of the mineable mineralized material (from the originally reported value of 2.9% in the 2016 Technical Report) as a result of its reduced copper recovery. The inclusion of harder lithologies in the ore blend will tend to reduce mill throughput.

A recent comminution study conducted by SRK Consulting (Canada) Inc. (SRK) (SRK, 2021) updated the steady-state models of the Fine Crushing Plant (FCP) and grinding circuits to assist the ongoing operational optimization. SRK had previously provided models of both circuits based on survey data collected in 2019. Recently, PVM has made significant changes in blast fragmentation, FCP, and grinding circuit operating conditions which have been updated in the JKSimMet models. Opportunities to improve circuit performance were investigated using the models and simulation results.

The comparison of survey results showed that the finer blasting increased the "fines" component (minus 1/2 inch) in the FCP feed from 35% in 2019 up to 47% in 2020. The yearly average was approximately 32%. The FCP feed P80 was 86.8 mm in 2019 and 61.9 mm in 2020. The effect of the finer measured FCP feed was noticeable in the survey results, with a lower circulating load around the tertiary screens and a lower amp draw for both secondary and tertiary crushers. The ball mill feed size distributions were similar for both surveys with a P80 of 10.8 mm and 10.9mm for 2019 and 2020, respectively.

The 2020 survey measured a significant improvement in FCP feed size with 40% to 50% minus 12 mm due to changes in blasting practices (see Figure 13-1). The solids vs. dotted lines in Figure 13-1 show the shift in FCP feed size since October 2019. Screens 1, 2, and 3 are parallel, each with its own dedicated secondary cone crusher. In addition, the secondary crusher feed measured only 10% +4 inch (100mm) compared with up to 20% in 2019.

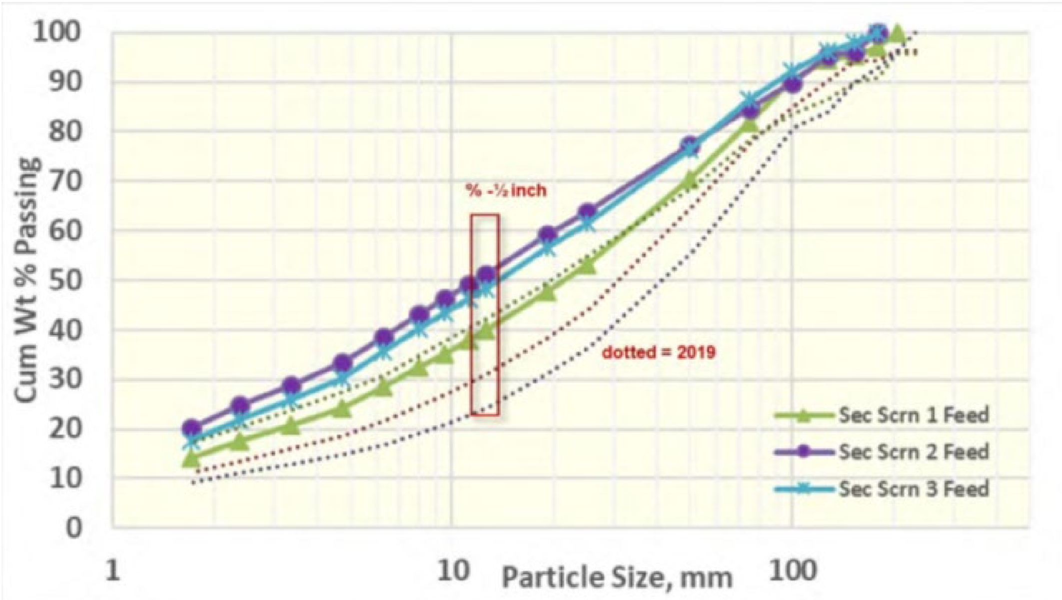


Figure 13-1: FCP Feed Size – 2020 and 2019 Surveys

SRK reports that in both FCP plant surveys, the secondary screen undersize was considerably finer than the tertiary screen undersize with the FCP product a combination of the two (see Figure 13-2). A finer FCP feed produces a finer secondary screen undersize, but the tertiary crushers generate all of the tertiary screen undersize.

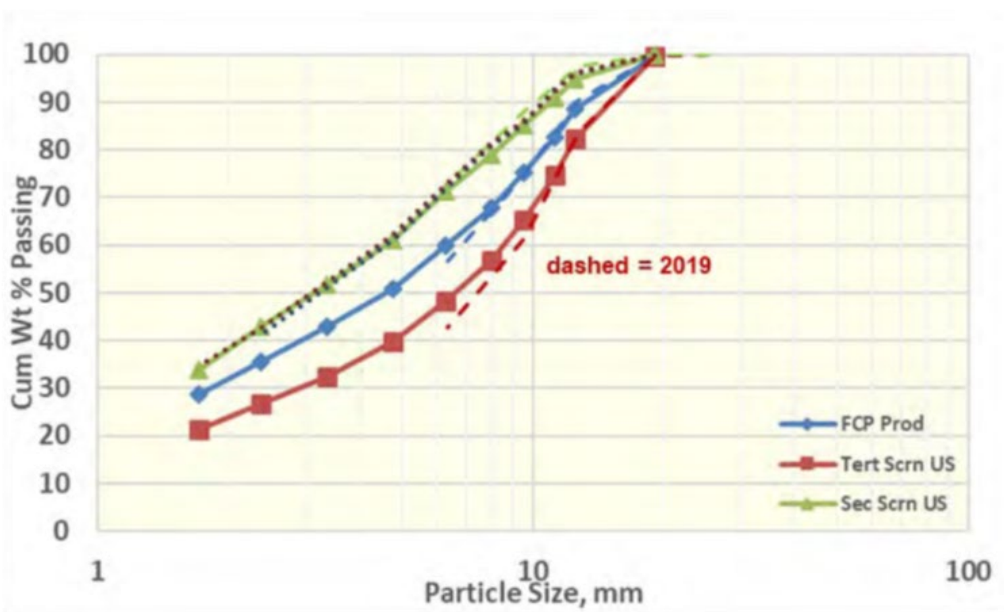


Figure 13-2: FCP Feed Size – 2020 and 2019 Surveys

To produce a finer overall FCP product (ball mill feed), the tertiary screen undersize needs to be shifted towards the secondary undersize distribution. For comparison, Figure 13-3 shows the two screen undersize distributions with a range of ball mill feed distributions at various feed tonnages. The ball mill feed size range lies within the two screen undersize distributions. With the current FCP screen size, the finest ball mill feed is the secondary undersize and the coarsest feed is the tertiary undersize.

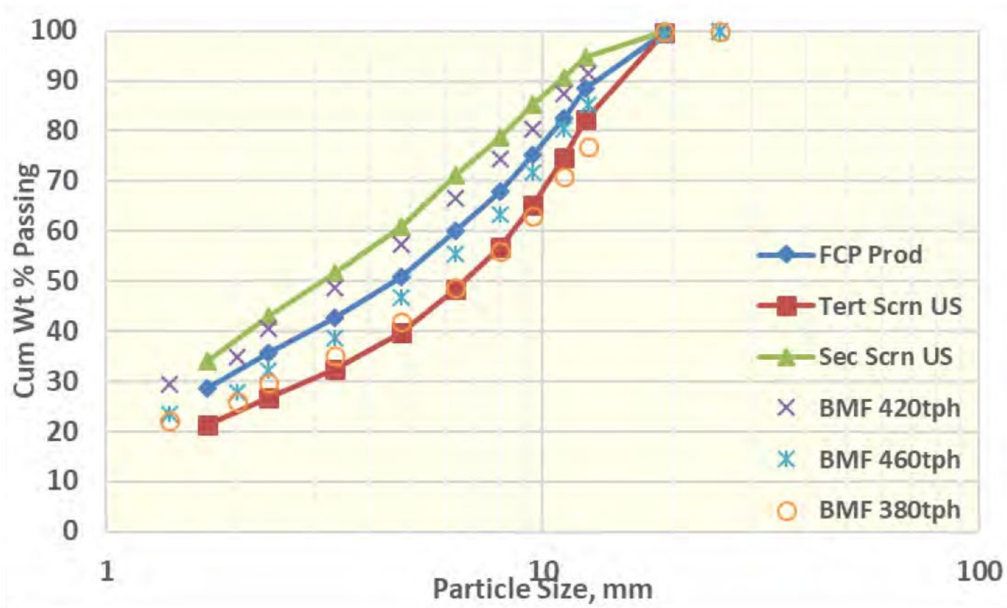


Figure 13-3: Comparison of Ball Mill Feed Size Vs. FCP Range Of Products

SRK indicates that a change in screen panel opening, for both the secondary and tertiary crushers would shift both screen undersize and result in a finer FCP product. Whether this option has potential would depend on the resulting circulating load to the tertiary screens.

PVM have achieved a significant improvement in blast fragmentation fines based on the two FCP surveys conducted by SRK. The current KPI target is 30% minus 12 mm, in the blasted product. SRK has recommended that the KPI target be raised to 35% to provide a greater fines proportion in FCP feed which will allow more fines to bypass the FCP and reduce the circulating load and crusher power draw.

SRK developed a simple model of ball mill throughput as a function of ore hardness (Bond Work Index, BWi) and feed size. Figure 13-4 shows the estimated change in grinding line capacity for a range of percentage 6.35 mm (+¼ inch) and two BWi values.

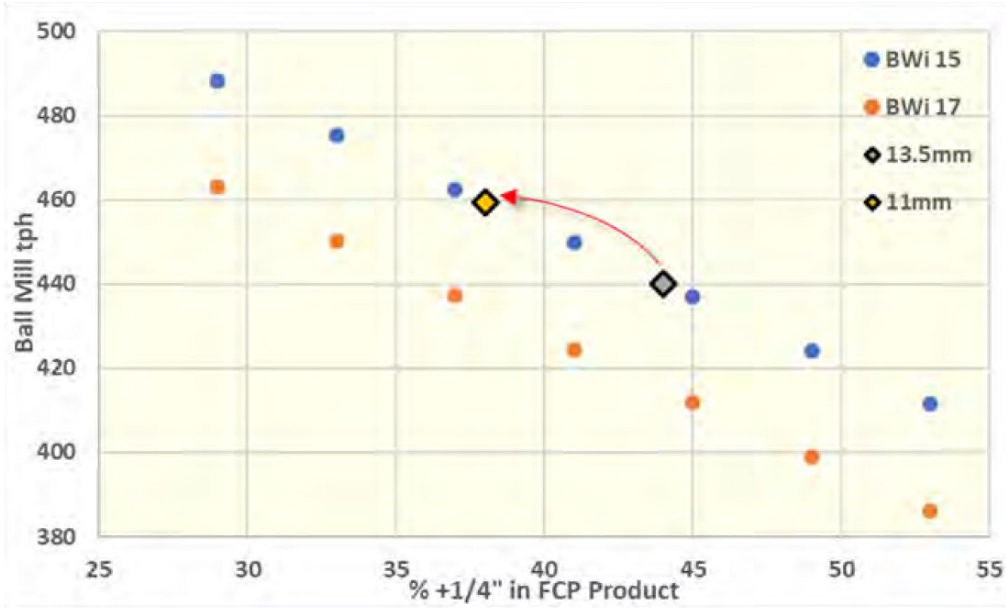


Figure 13-4: Estimated Effect of Fine Screen Panels On Ball Mill #2 Throughput

Changing the screen panels from 13.5 mm to 11 mm results in a 6% reduction in +¼ inch (6.35 mm) material which is approximately equivalent to a 20 tonnes per hour (tph) increase in grinding line throughput for a BWi of 15kWh/t. SRK has stated that changes in feed size have a much greater impact on ball mill circuit capacity than ore hardness, due to the moderate BWi values that are prevalent in PVM material.

SRK's review of the grinding circuit indicated that the final cyclone overflow size distribution (flotation feed) is very sensitive to the grinding throughput. In the grinding survey, tonnage was stepped from 420 tph to 480 tph in 20 tph increments. The impact is consistent with a flotation feed P80 coarsening from 350 µm to 413 µm over the tonnage range (see Figure 13-5).

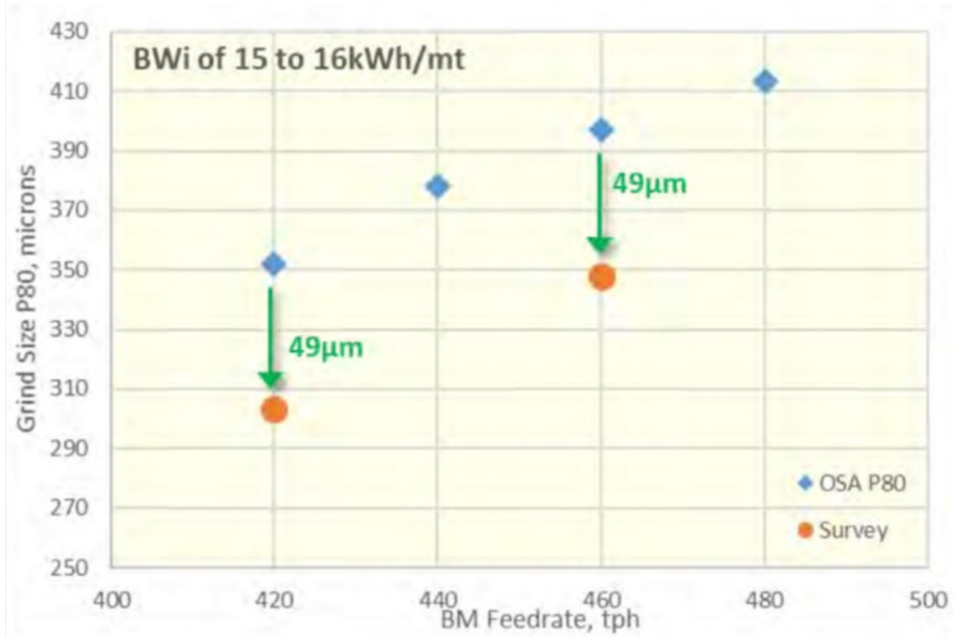


Figure 13-5: Comparison of On-stream Analyzer Sample P80 Vs. Cyclone Overflow P80 Values

There is some indication that the on-stream analyzer samples tend to be consistently coarser than the grab samples taken by SRK during the survey (Figure 13-5). During the various surveys the operating work index (OWi) was compared to the laboratory Bond Work Index (BWi). The OWi matched the BWi closely with an average value of around 15 kWh/t.

SRK has also identified opportunities to improve the grinding mill performance with modifications to the existing cyclone packs. To improve grinding circuit limitations due to circulating load, water addition and/or sump level limitations, SRK investigated utilizing smaller vortex finder sizes in the current cyclones. The simulation result with 10 inch vortex finders reduced the circulating load but slightly coarsened the P80 size (311 µm vs 303 µm). For this simulation, the cyclone feed was maintained at 67% solids with a pressure drop of 59 kPa (8.5 psi). The smaller vortex finders decreased the total water addition by 20%. This produced an overflow solids concentration of 42% which matched the results during Survey 2 at a 460 tph feed rate.

SRK also investigated the regrind circuit performance. This is a reverse grinding configuration with the feed material being cycloned before introduction to the ball mill. The review indicated that the regrind mill performance is adequate but there is room for improvement around the cyclone system. Previous studies conducted by Weir Minerals (2019) have indicated that the current primary cyclones suffer from a significant bypass proportion (43%). This high bypass results in a high circulating load and a coarser grind size (P80). A 43% bypass indicates that 43% of the material is already product size and is reporting back to the ball mill. A 10% decrease in bypass could potentially improve the throughput by 5%.

The goals of these investigations were to identify opportunities to increase the mill throughput while still maintaining adequate flotation recovery. However, for this current Technical Report the target maximum tonnage rate is 56,000 tpd and the focus has shifted from maximizing tonnage to maximizing the copper recovery possible at this throughput. The impact of grind size on flotation recovery is discussed in subsequent sections.

GRE has reviewed the plant operating data from 2018 through to March 2021 to determine the operating work index and also the grinding circuit power draw.

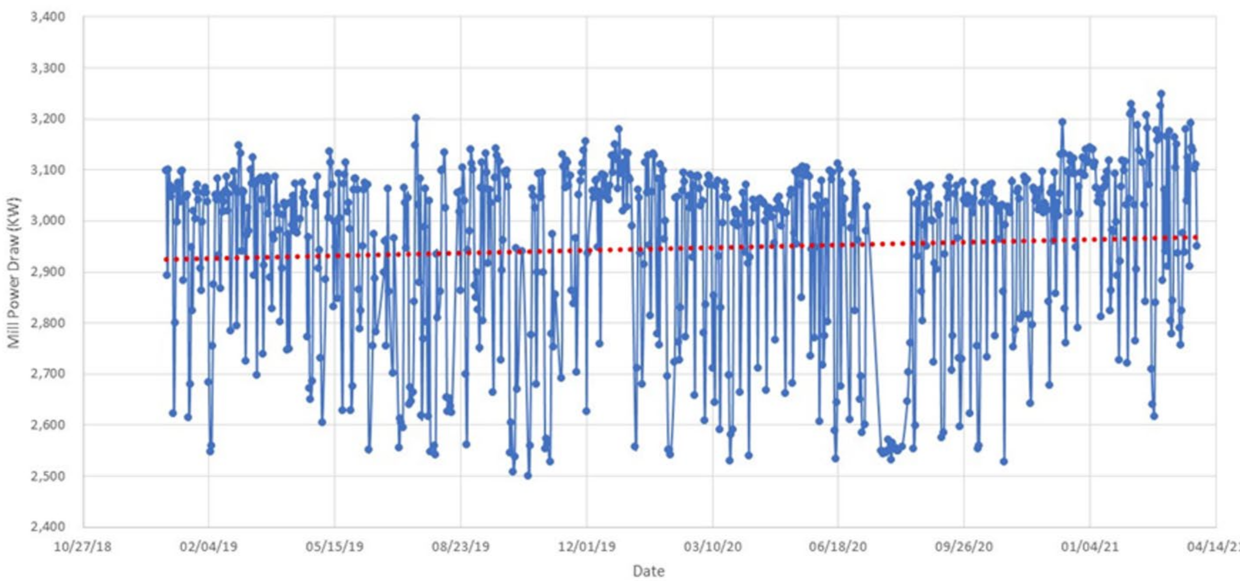


Figure 13-6: Mill Power Draw 2018 – 2021

The primary grinding mills are equipped with 2,983 kW (4,000 hp) nameplate synchronous motors. During this period, the average power draw was 2,950 kW with a peak power draw of 3,250 kW. Similar power draws were seen for the data for 2020 and 2021 (Figure 13-6). Over the last three years, PVM has successfully improved the power draw from the mills, as shown by the red dotted trend line in Figure 13-6. From the data (Figure 13-7) it appears that the mills should be able to operate consistently in the range of 3,100 kW (note that the power draw is consistently beyond the nameplate mill motor power). The majority of the mill power draw fluctuations are due to external forces such as the availability of fine ore feed, not mill operations.

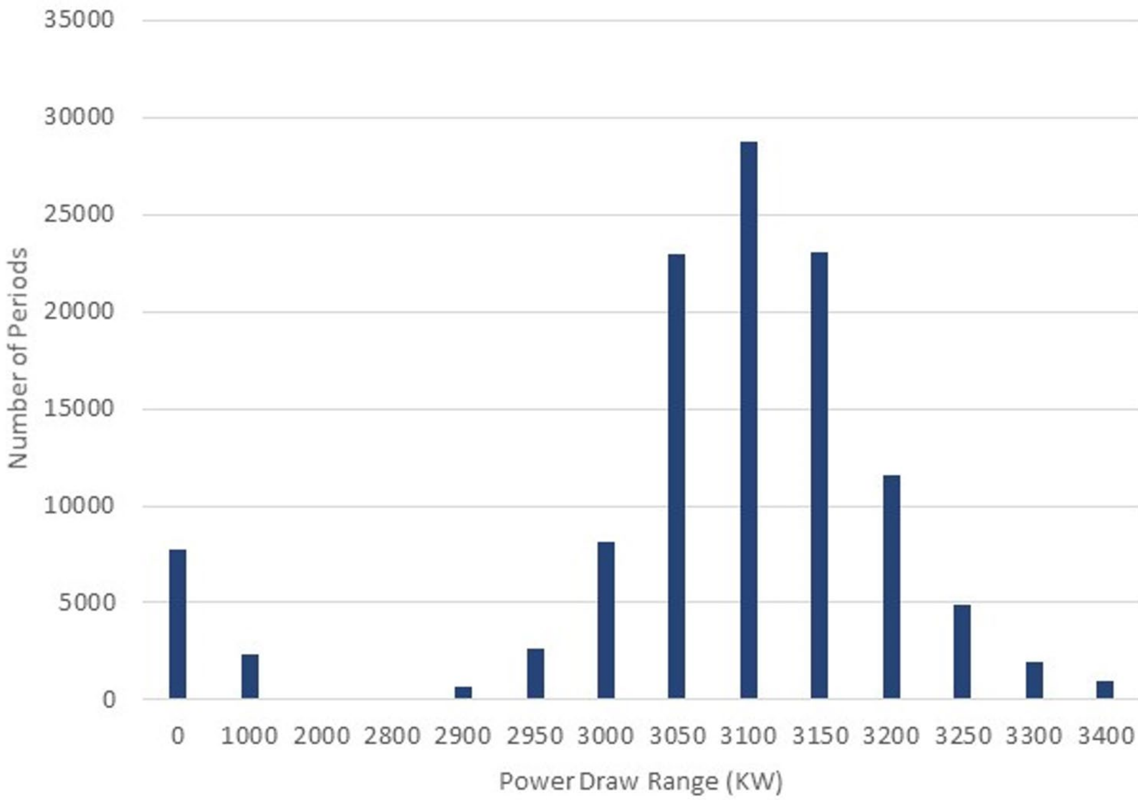


Figure 13-7: Mill Power Draw Ranges 2018 – 2021

The Bond work index specific energy was tabulated for the same period and is shown in Figure 13-8.

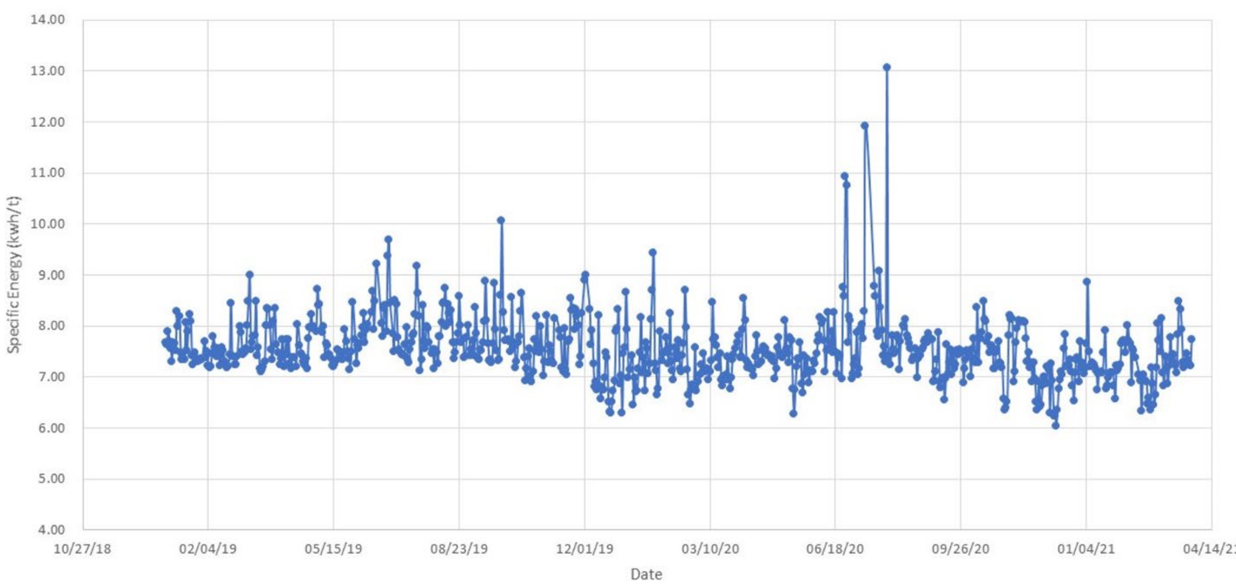


Figure 13-8: Specific Energy Mill Power Draw 2018 - 2021

The average specific energy for the period was 7.50 kWh/t and depending on the assumed product size produced the OWi equates to approximately 15 kWh/t (F80 11 mm, P80 320 µm).

Previous studies have examined the mill capacity in terms of grind size based on the installed power. Utilizing the peak power draw of 3,100 kW at 95% efficiency allows for the grinding circuit capacity to be estimated for a range of P80 product sizes, as shown in Figure 13-9.

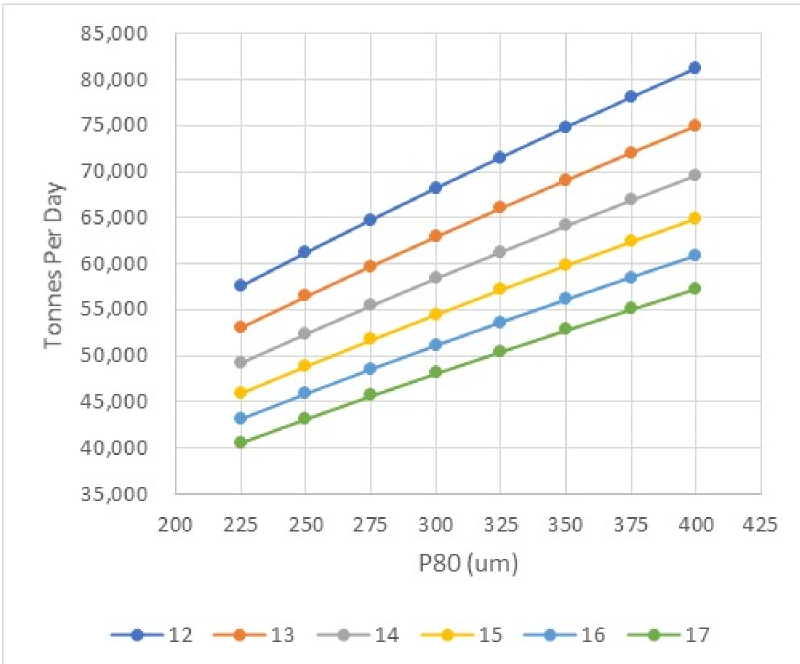


Figure 13-9: Mill Throughput At Various Work Index and Product Sizes

Assuming the desired mill capacity of 56,000 tpd with a work index of 15 kWh/t, with a feed product size of 11 mm, a P80 product size of 313 µm should be achievable.

13.3.2 Copper Flotation

Significant flotation test work has been conducted over the life of PVM; details of much of this past work are available in the 2016 Technical Report (Capstone, 2016). In general, the results suggest that the copper recovery in flotation is a function of the feed grade (total copper and oxide copper), mass pull, grind size (P80) and throughput (retention time). GRE has examined the plant's production statistics for the period of January 1, 2014 through to March 30, 2021 along with associated test work to develop flotation recovery predictions. A significant portion of the most recent test results has been summarized in an investigation conducted in 2020 by 911Metallurgy Corp. (2020).

The metallurgy of the PVM deposit is well understood and relatively straightforward. There have been several metallurgical reports produced for this site from ALS Metallurgy Kamloops (ALS) (2014), Blue Coast Metallurgy, Ltd (Blue Coast) (2019), FL Smidth (FLS) (2017), BaseMetLabs (2015) and Amelunxen Mineral Processing Ltd (Aminpro) (2017). These reports are available in the PVM library for review.

The rougher circuit mass pull becomes increasingly more important as the grind of the flotation feed coarsens. As the grind coarsens, there are less liberated particles, and in order to maintain the recovery, more binary particles need to be floated, requiring a higher mass pull. Figure 13-10 shows the data for the plant operation in the 1990s and in 2019 as produced by Blue Coast. The 2019 data is centering close to the 4% mass pull, and the 1990 data was typically above the 6% mass pull line. Mass pull has historically been closer to 6 to 8%.

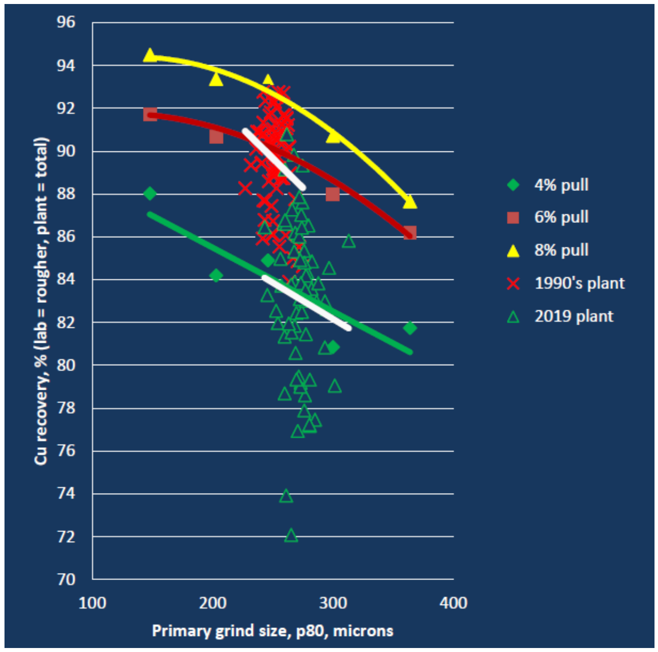


Figure 13-10: 2019 Q1 Plant Data Overlaid on Previous Data

The historical plant data show a fundamental relationship between mass pull and final copper recovery, but there is significant scatter and a resulting low R-squared value (0.016) as shown in Figure 13-11.

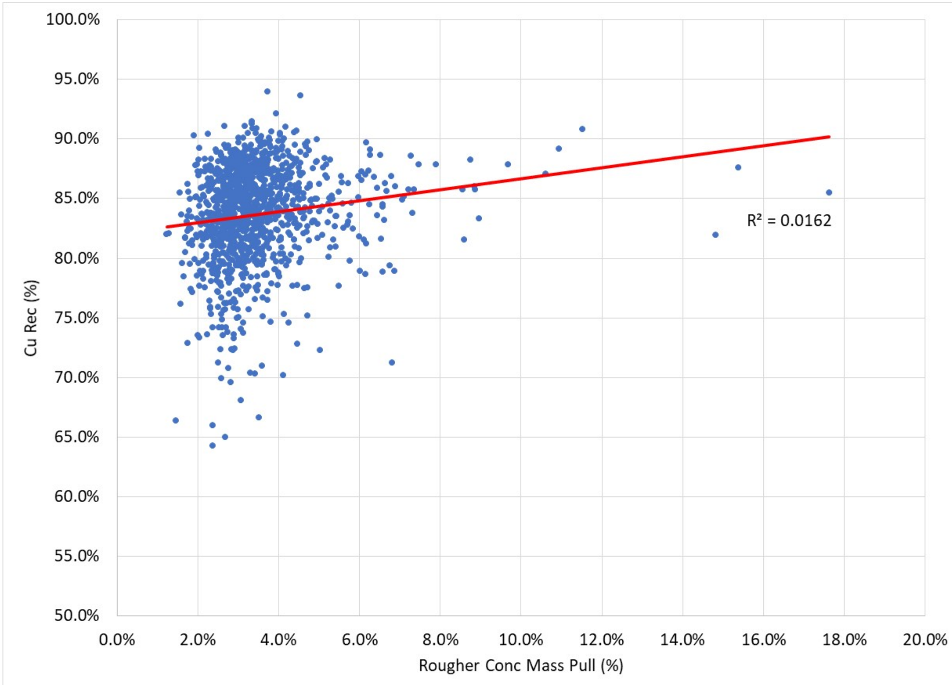


Figure 13-11: Final Copper Recovery and Rougher Mass Pull

Figure 13-12 shows the relationship between final copper recovery and the plant feed grade (total copper). A better relationship is exhibited with an R-squared value of 0.15.

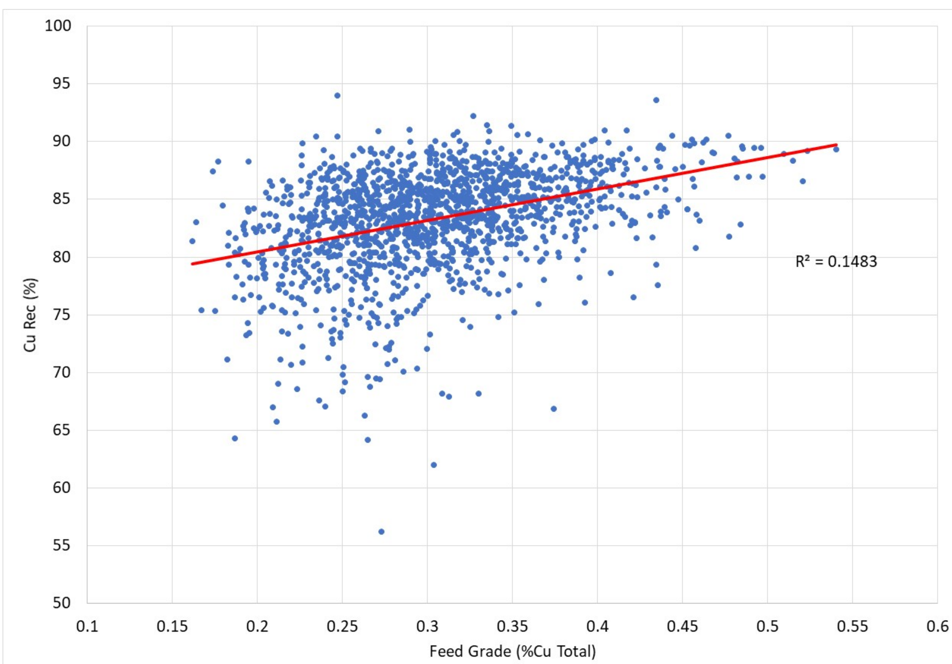


Figure 13-12: Final Copper Recovery and Feed Grade (Cu Total)

As feed grade increases, the copper recovery tends to increase in part likely due to the existence of a constant tail due to fine-grained minerals not being fully liberated. If a portion of the tailings copper is unliberated and constant, a higher feed grade will naturally produce a higher recovery.

Similarly, as the proportion of oxide material in the feed increases, copper recovery will naturally decrease. The copper oxide minerals do not respond well to sulfide flotation resulting in losses to the tailings (Figure 13-13).

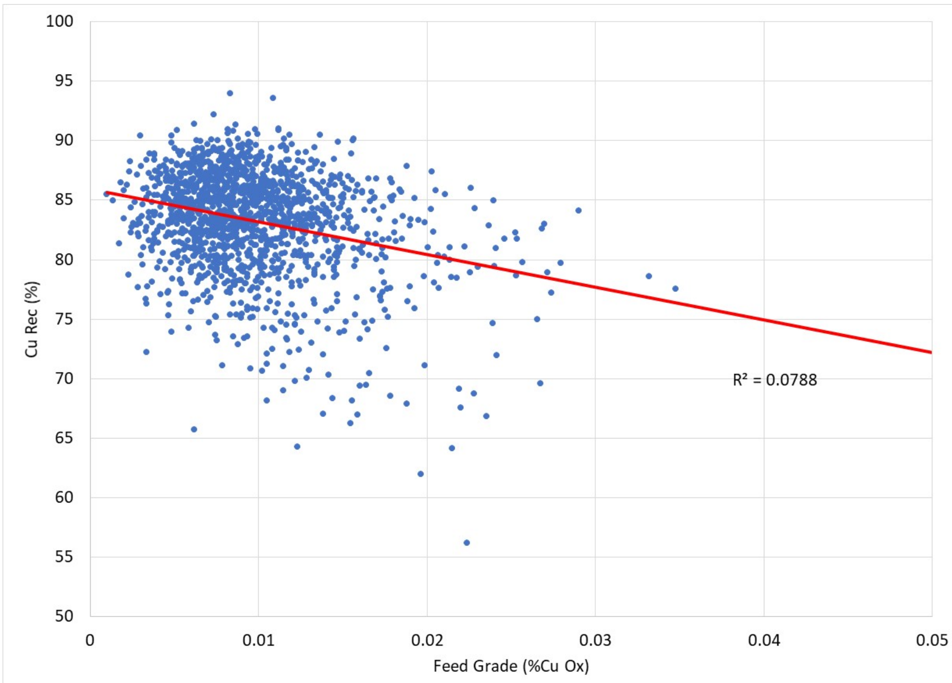


Figure 13-13: Final Copper Recovery and Feed Grade (Cu Oxide)

The majority of these recovery drivers only show moderate correlations when examined as a simple regression. In order to fully explore the impact of these variables, a multiple regression analysis was completed (see Section 13.3).

The current primary grind is relatively coarse, but the test work has shown that a K80 of 250 to 300 μm is acceptable with the appropriate mass pull. Figure 13-14 shows a composite of the flotation feed particle size over the last two years. The most recent data show that the feed is averaging around a P80 of 350 μm.

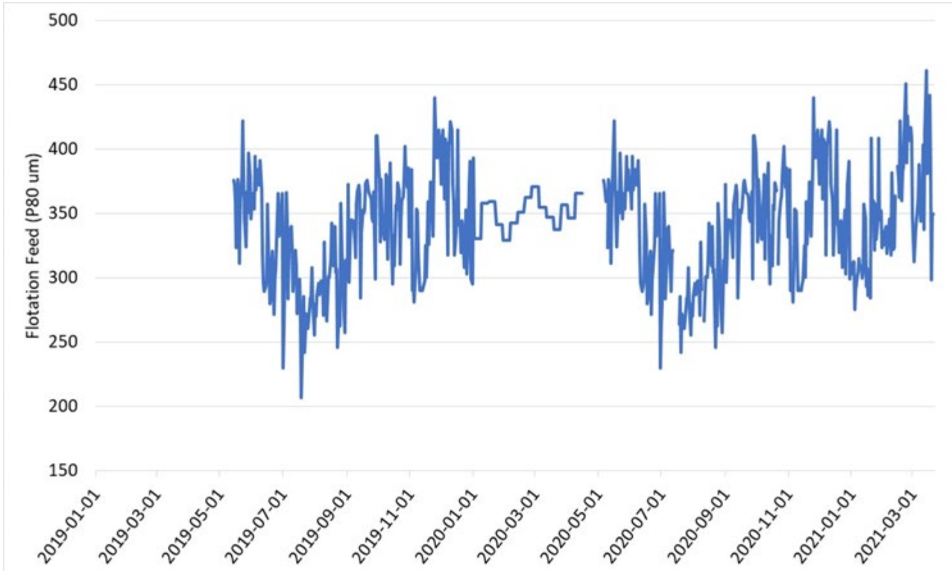


Figure 13-14: Flotation Feed P80

The grinding circuit appears to be producing bimodal distributions in the flotation feed. There is an excess of both coarse and fine material. It has been reported that approximately two-thirds of the lost copper is in the minus 75 and plus 355 µm size fractions, according to Blue Coast (Figure 13-15). This bimodal distribution relates back to the cyclone performance bias that was discussed earlier.

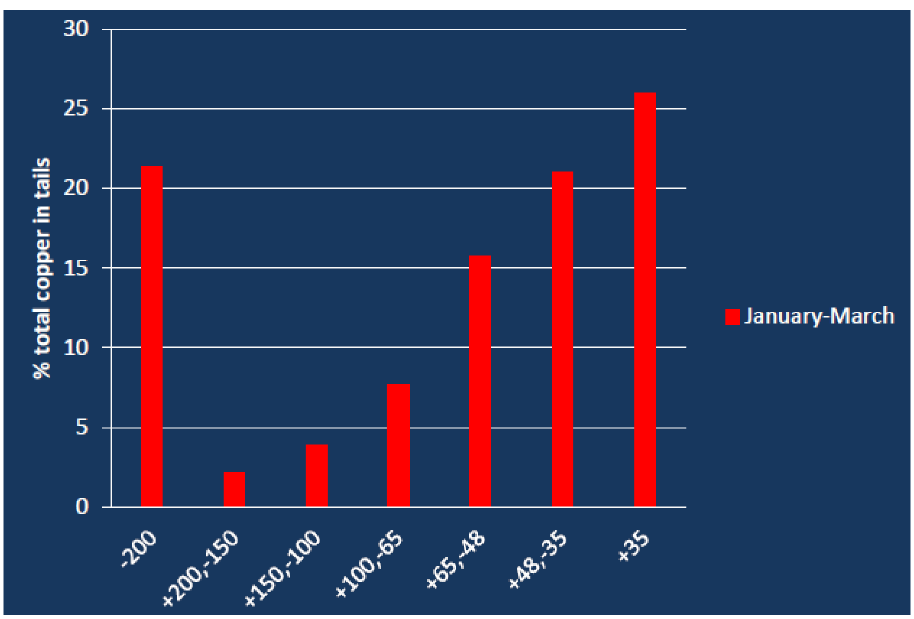


Figure 13-15: 2019 Q1 Tailings Copper Losses

Figure 13-16 shows that at a primary grind below a K80 of 300 µm the liberation of chalcopyrite is greater than 60%. At this liberation, a good rougher concentrate should be obtainable. This conclusion has also been supported by test work conducted by BaseMetLabs and ALS.

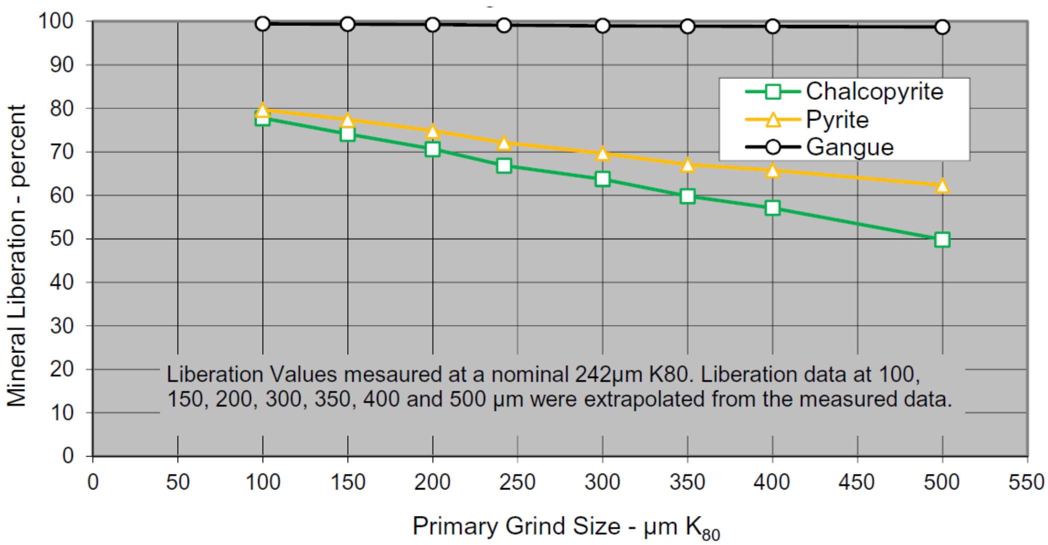


Figure 13-16: The Effect of Primary Grind Sizing on Mineral Liberation - Rougher Feed

Figure 13-17 shows a recovery-mass pull curve for both copper and molybdenum at various primary grinds. It is evident from this work that a primary grind of approximately 300 µm or less will provide the desired recovery providing the mass pull is suitable. As liberation decreases, it is necessary to increase the mass pull to maintain recovery.

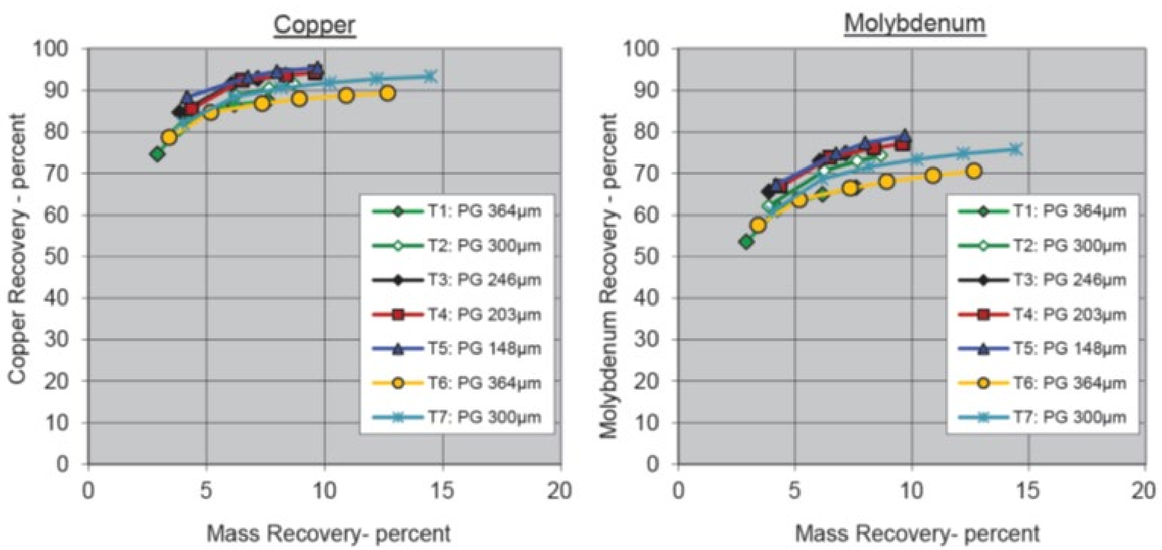


Figure 13-17: The Effect of Primary Grind Sizing on Copper and Molybdenum Recovery

The laboratory data had indicated that moving from a K80 primary grind size of 300 µm to 250 µm a 2% increase in copper recovery was achieved (for the same mass pull). However, given that the grinding energy is fixed, reducing the K80 has a large impact on the throughput (Figure 13-9). A 50 µm change in the primary grind results in a decrease in plant throughput of approximately 220 tonnes per hour. A 2% increase in recovery is not usually economically justified when it requires an 8% decrease in throughput.

The performance of the regrind circuit has a major impact on recovery for both copper and molybdenum (see Figure 13-18). Moving from a regrind K80 of 44 µm (current) to 18 µm could improve copper recovery up to 4% with a 2 to 3% improvement in grade and molybdenum recovery could increase 8 to13%. The regrind performance has not been assessed to determine what fineness of grind is obtainable in the current regrind circuit.

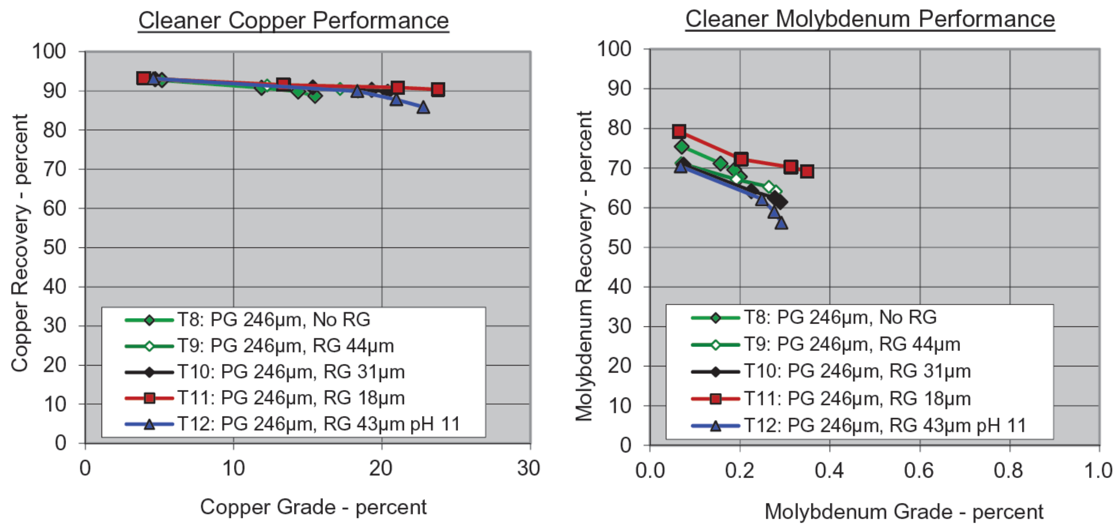


Figure 13-18: The Effect of Regrind Sizing on Copper and Molybdenum Recovery

13.3.3 Molybdenum Flotation

Over the course of the PVM life, the molybdenum circuit has been idled at various periods due to the molybdenum market price. Molybdenum is a by-product of the copper plant and the feed grade is low averaging 0.0056% over the last four years. The other issue with the molybdenum circuit is that the primary depressant reagent employed is sodium hydrosulfide (NaHS). NaHS will evolve toxic H_2S gas when exposed to pH regimes just slightly below the operating pH of the molybdenum circuit. As a result of the potential danger to plant operators, PVM has undertaken a program to examine a new reagent suite. It should be noted that PVM has taken all the necessary precautions to utilize NaHS safely in the plant but, as a risk mitigation process, has elected to explore alternative reagents.

The molybdenum recovery traditionally has been fairly low, maxing out at around 20%, with more typical recoveries for similar plants being around 40 to 50%. Aminpro (Amelunxen Mineral Processing Ltd., 2020a) has shown that molybdenum recoveries in the laboratory can reach 50% with grades approaching 50% Mo. On inspection of the molybdenum plant, a survey by Aminpro indicated that current performance was hindered by a lack of flotation time in the 1st stage cleaners. The molybdenum circuit was shut down in late 2019 and was recently refurbished.

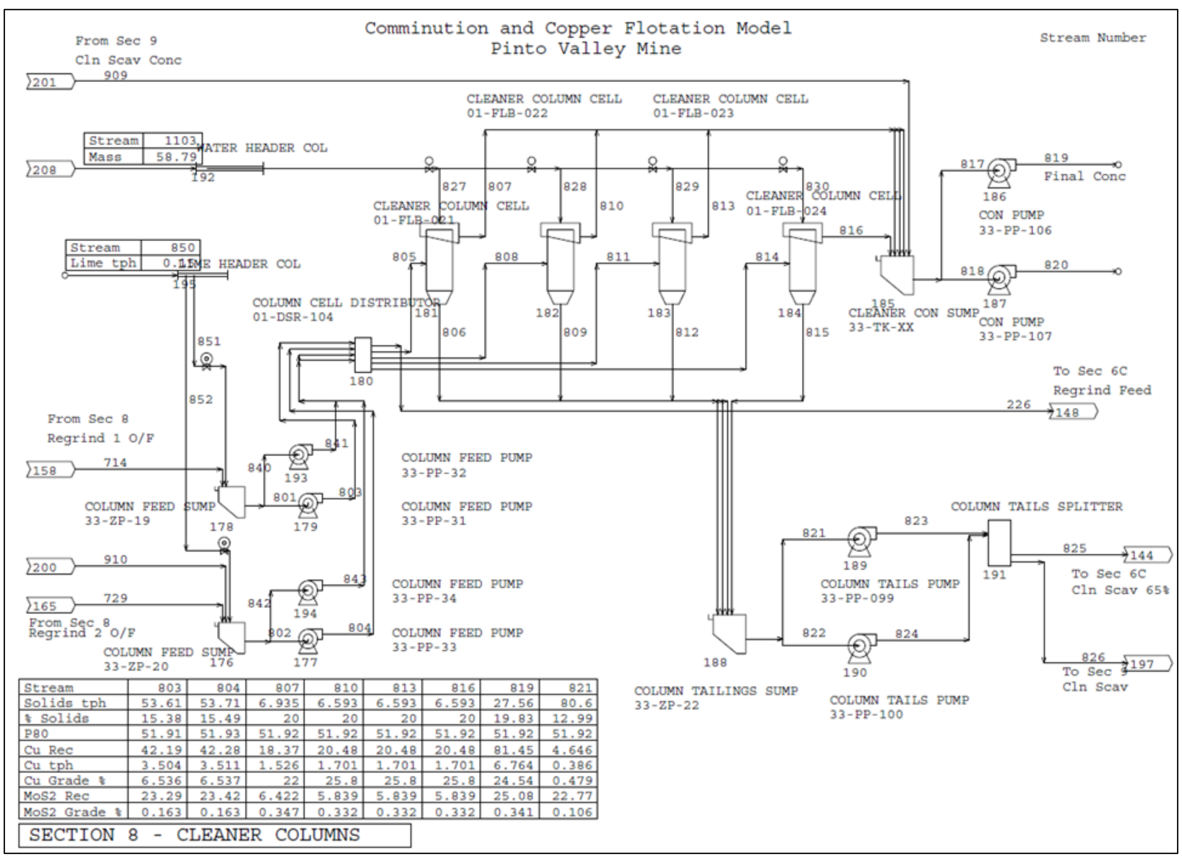
Orfom D8 (Marketed by Chevron Phillips) is a reagent that is designed to replace NaHS. The reagent has not seen significant industrial application because it produces a tenacious froth that hinders pumping and general concentrate flow. The use of D8 is often accompanied by the addition of defoamer chemicals. D8 can be used at any pH, operates with ambient air in the cells (NaHS operates with nitrogen), and presents no operator danger. Aminpro recently undertook a study to evaluate the transition to D8 at the PVM plant (Amelunxen Mineral Processing Ltd., 2020b)

The results from the Aminpro test indicate that good copper-molybdenum separation can be achieved with the new D8 reagent and that rougher recovery of molybdenum close to 90% can be achieved. Overall, molybdenum recovery after cleaning was close to 50%. Plant trials are currently underway and no production details are currently available, as a result of this, the estimated molybdenum performance cited later in this report is based on the conventional molybdenum reagent suite.

13.4 Process Modeling

A significant amount of process modeling has been conducted for the PVM process plant. The majority has been focused on the crushing and grinding circuit using Bruno™ or JKSimMet™. GRE has also developed a model for the complete process plant using MetSim™. The model was developed to allow the analysis of various production scenarios and to provide an estimation of various stream flows that can't or are not physically be measured in the plant.

A complete metallurgical balance has been produced from the MetSim™ model using plant data to validate the output. The cleaner column flotation circuit flowsheet has been provided as an example (Figure 13-19).



Stream	803	804	807	810	813	816	819	821
Solids tph	53.61	53.71	6.935	6.593	6.593	6.593	27.56	80.6
% Solids	15.38	15.49	20	20	20	20	19.83	12.99
P80	51.91	51.93	51.92	51.92	51.92	51.92	51.92	51.92
Cu Rec	42.19	42.28	18.37	20.48	20.48	20.48	81.45	4.646
Cu tph	3.504	3.511	1.526	1.701	1.701	1.701	6.764	0.386
Cu Grade %	6.536	6.537	22	25.8	25.8	25.8	24.54	0.479
MoS2 Rec	23.29	23.42	6.422	5.839	5.839	5.839	25.08	22.77
MoS2 Grade %	0.163	0.163	0.347	0.332	0.332	0.332	0.341	0.106

Figure 13-19: MetSim™ Cleaner Flotation Model

13.5 Recovery Models

This section outlines the development of a baseline recovery model for the PVM concentrator. The objective was to develop a copper recovery model that could be utilized in the analysis of the open pit mine resource. The model allows copper recovery to be estimated based on the material characteristics and applied to each block within the resource block model.

Copper and molybdenum recovery models were previously presented in historical NI-43-101 Technical Reports (Capstone, 2014), (Capstone, 2016). The previous copper recovery model was a single variable formula based solely on total copper head grade although a review of other variables was conducted including mill throughput. The previous models also relied on laboratory estimation from metallurgical test work.

The revised model examined the most recent laboratory test work, detailed concentrator data from January 2017 through to February 2021 as well as the reconciled copper production records from the third-party smelter for that same period.

The data employed in the development of the recovery models were sourced from the plant data supplied by PVM and from metallurgical test work reports.

13.6 Historic Copper Model

The historical models were presented in the report titled “Pinto Valley Mine Life Extension - Phase 3 (PV3) Pre-Feasibility Study NI 43-101 Technical Report Pinto Valley Mine, Miami, Arizona” published January 1, 2016 (Capstone, 2016).

There are three historic recovery models available, one was produced in 2007 and two were produced in 2013 (Table 13-4). These models were developed by BHP, the property owner at the time.

Table 13-4: Historic Recovery Models for Copper, 2007 to 2013

Year	Model	Equation Number
2007	%Cu Rec = 8.2381 ln(%Cu) + 94.19	1
2013	%Cu Rec = [9.8864(%Cu) + 87.041] x 0.973 (Cleaner)	2
2013	%Cu Rec = -18.465$(\%Cu)^2$ + 7.6287(%Cu) + 96.67 (Cleaner)	3

A 5% intercept recovery deduction was selected for diabase ore types based on suggested plant performance as follows (Table 13-5):

Table 13-5: Historic Recovery Models for Copper in Diabase, 2013

Year	Model	Equation Number
2013	%Cu Rec (Diabase) = [9.8864(%Cu) + 82.041] x 0.973 (Cleaner)	4

The previous Technical Report, PV3-2016-PFS, indicated that the molybdenum recovery is dependent on the ore feed grade and the performance of the bulk concentrate separation circuit. Molybdenum is a by-product stream with a very low feed grade.

The estimated molybdenum circuit recovery to the molybdenum concentrate when the circuit is operating is 47% (Table 13-6). Currently, a plant trial is underway testing new reagents with the goal of excluding NaHS from the circuit. No plant results have been provided at this stage.

Table 13-6: Historic Recovery Models for Molybdenum, 2013

Year	Model	Equation Number
2013	%Mo Rec = 0.47 (Cleaner)	5

The previous Technical Report, PV3-2016-PFS, identified several variables beyond total copper feed grade that impacted copper recovery including oxide copper grade, flotation feed size (P80), mill throughput, ore types (Diabase). Only the Diabase influence was quantified, the balance of the variables was not considered directly in the copper recovery model. The models assume that a target grind size will be maintained and throughput adjusted accordingly.

13.7 Current Copper Model

The current recovery model developed is based on the review of four years of concentrator production data and a myriad of laboratory metallurgical test reports. The laboratory data was analyzed first to determine what dependent variables related to the copper recovery were evident (statistically significant). These variables, among others, were then evaluated for the available concentrator production data.

A review of the laboratory reports indicated that the following variables appeared to influenced copper recovery:

- Copper grade – oxide and total
- Ore type, mainly Diabase
- Flotation feed particle size P80
- Concentrate mass pull

A review of the plant operating statistics indicated that these additional variables may be important:

- Mill throughput
- Flotation retention time
- Cyclone overflow density

All of the plant data was evaluated on a daily average basis. A simple regression was utilized to identify variables that may impact the copper recovery. This was followed by a multiple regression analysis approach to determine the set of dependent variables that were statistically significant. For this exercise statistical significance was defined as a p-value less than 5% at a 95% confidence interval.

A total of 1,541 data points were available for the daily concentrator production representing a time period from January 1, 2017 to March 30, 2021. This data was directly downloaded from the control system historian. This data was augmented with additional data from external sources including the daily screen analysis for the flotation feed (P80), the cyclone overflow solids percentage, retention time calculations for the flotation circuit and the cleaner circuit assays.

In general, the regression data sets ranged from 590 data points to the full set of 1,541 points. The data sets were reduced in size for a number of reasons:

- Incomplete data for the variable being examined such as P80
- Filtering of the data based on mill throughput to remove outliers
- Filtering the data to target certain data trends

After conducting multiple regression investigations, a good relationship was established between final copper recovery and the following dependent variables:

- Copper total feed grade
- Copper oxide feed grade
- Flotation feed particle size (P80)
- Cyclone overflow solids percentage
- Flotation retention time

A total of 612 data points were used in the analysis and the data was filtered to remove low tonnage data from the set (tonnage less than 35,000 tonnes per day). Periods with tonnage below this setpoint represent operational outliers.

Since the recovery model is typically employed for open pit optimization the concentrator variables in the equation (P80 and cyclone overflow solids) were fixed at the average for the data set to produce a final equation for copper recovery.

The regression produced an R-squared value of 0.43 which is considered excellent for this type of raw plant data. The output from the regression analysis is shown in Table 13-7.

Table 13-7: Regression Analysis Results – Copper

Summary Output	
Regression Statistics	
Multiple R	0.651
R Square	0.426
Adjusted R Square	0.420
Standard Error	0.023
Observations	612 final conc recovery, tpd >35k tpd

Analysis of Variance					
	Degrees of Freedom	***Sum-of-Squares***	***Mean Squares***	***F-Test***	***Significance F***
Regression	5	0.235	0.0471	89.48	1.95E-70

Residual	606	0.319	0.0005			
Total	611	0.554				

	Coefficients	***Standard Error***	***t Stat***	***P-value***	***Lower 95%***	***Upper 95%***
Intercept	0.9790	0.01536	63.4041	~0	0.9438	1.004123
Feed Grade Cu %	26.0629	1.6694	15.6123	~0	22.7844	29.34135
Feed Grade %CuOx	-344.6257	24.5278	-14.0504	~0	-392.7960	-296.456
Ret Time (min)	0.0014	0.0006	2.5403	0.0113	0.0003	0.0025
RO Feed P80 (um)	~0	~0	-3.5235	0.0005	-0.0001	~0
Cyc OF solids %	-0.4748	0.0383	-12.4078	~0	-0.5499	-0.3996

Substituting the concentrator operating variables with the average for the data set provides a final copper recovery equation as follows (Table 13-8):

Table 13-8: Copper Recovery with Average Variables, 2021

Year	Model	Equation Number
2021	Final Cu Recovery = 0.804368 + 26.063*(Cu Grade %) - 344.626*(Cu Grade OX%)	6

Where:
Cu Grade % - is the concentrator total copper feed grade in percent
Cu Grade Ox% - is the concentrator oxide copper feed grade in percent

The complete concentrator recovery model is provided in Table 13-9.

Table 13-9: PVM Complete Concentrator Recovery, 2021

Year	Model	Equation Number
2021	Final Cu Recovery = 0.97896 + [26.063*(Cu Grade %)] + [-344.626 * (Cu Grade OX%)] + [-0.0000881 * P80] + [-0.475*Cyclone Overflow Solids%] + [0.0014*Retention Time]	7

Where:
Cu Grade % - is the concentrator total copper feed grade in percent
Cu Grade Ox% - is the concentrator oxide copper feed grade in percent
P80 – flotation feed particle size P80 in microns – data average 341 um
Cyclone Overflow Solids% - flotation feed solids density in percent – data average = 35.0%
Retention Time – flotation retention time in minutes – data average = 15.5

As indicated by the model equation, total copper recovery improves with increasing copper feed grade, decreases with increasing oxide concentrations, decreases with increasing particle size and solids content and increases with increasing retention time.

The model was compared to the last 3.5 years of production on a monthly reconciled basis (Figure 13-20). On the advice of PVM staff, 2017 was excluded because there were major issues with the final concentrate reconciliation.

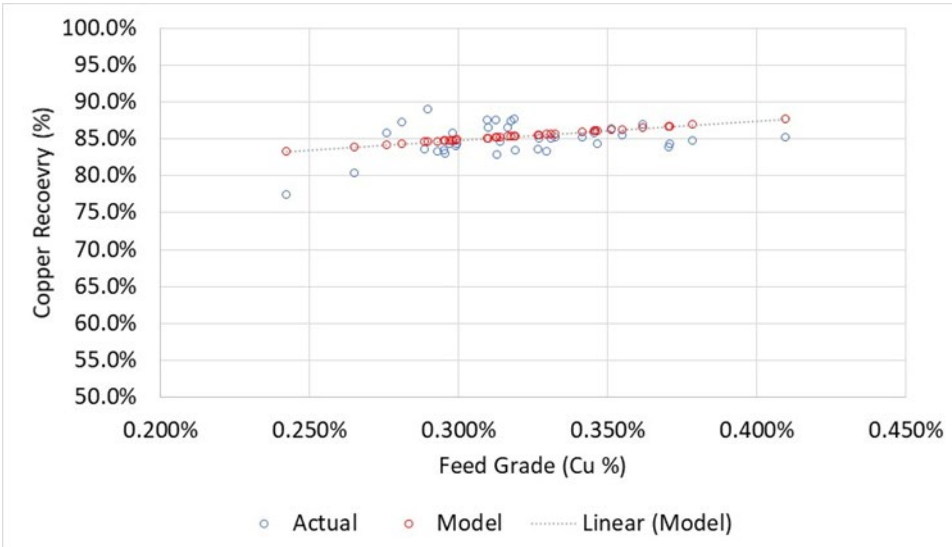


Figure 13-20: Actual and Model Copper Recovery

The model fits the data reasonably well with approximately 50% of the actual recovery being both above and below the model line.

In the previous Technical Report PV3-2016-PFS, a second model was developed to include an allowance for the Diabase lithology due to its poor copper recovery performance. The Diabase domain represents less than 1% of the current Mineral Resource and as such it is a small contributor to the overall economics. However, because its metallurgical response is very different from the main domain of Ruin Granite it has been included. The Ruin Granite domain contains approximately 96% of the Mineral Resource (Table 13-2).

The data for the performance of Diabase ore types is limited. PVM concentrator staff report that the ore is very hard and reduces grinding throughput, and they also report lower copper recoveries with Diabase blends. Unfortunately, quantifying this with concentrator data is impossible because the lithologies being processed are not quantified in a manner suitable for statistical analysis.

The previous Technical Report PV3-2016-PFS provided a 5% baseline copper recovery deduction for Diabase lithologies. Several specific flotation tests were conducted to gauge the response of pure Diabase ores. The results of the testing showed a significant decrease in baseline copper recovery for these ore types. The laboratory data was utilized to develop a model adaptation for the Diabase lithology. The copper recovery model equation 6 was adjusted to provide for a Diabase component as shown in Table 13-10.

Table 13-10: PVM Copper Recovery in Diabase, 2021

Year	Model	Equation Number
2021	Final Cu Recovery = 0.804368 + 26.063*(Cu Grade %) - 344.626*(Cu Grade OX%) – 0.18*(Diabase Wt%)	8

Where:
Cu Grade % - is the concentrator total copper feed grade in percent
Cu Grade Ox% - is the concentrator oxide copper feed grade in percent
Diabase Wt% - is the amount of Diabase ore present in the sample in weight percent

The complete concentrator recovery model is provided in Table 13-11.

Table 13-11: PVM Complete Concentrator Recovery, 2021

Year	Model	Equation Number
2021	Final Cu Recovery = 0.97896 + [26.063*(Cu Grade %)] + [-344.626 * (Cu Grade OX%)] + [-0.0000881 * P80] + [-0.475*Cyclone Overflow Solids%] + [0.0014*Retention Time] + [– 0.18*Diabase Wt%]	9

Where:
Cu Grade % - is the concentrator total copper feed grade in percent
Cu Grade Ox% - is the concentrator oxide copper feed grade in percent
P80 – flotation feed particle size P80 in microns – data average 341 um
Cyclone Overflow Solids% - flotation feed solids density in percent – data average = 35.0%
Retention Time – flotation retention time in minutes – data average = 15.5
Diabase Wt% - is the amount of Diabase ore present in the sample in percent

Comparing the test work results to the model revealed that the model prediction was moderately higher than the laboratory test work copper recovery. The model was adjusted for the Diabase content recognizing that the recovery model overpredicts the laboratory results. Figure 13-21 shows a series of laboratory flotation tests that included three Diabase ore samples and several non-Diabase samples.

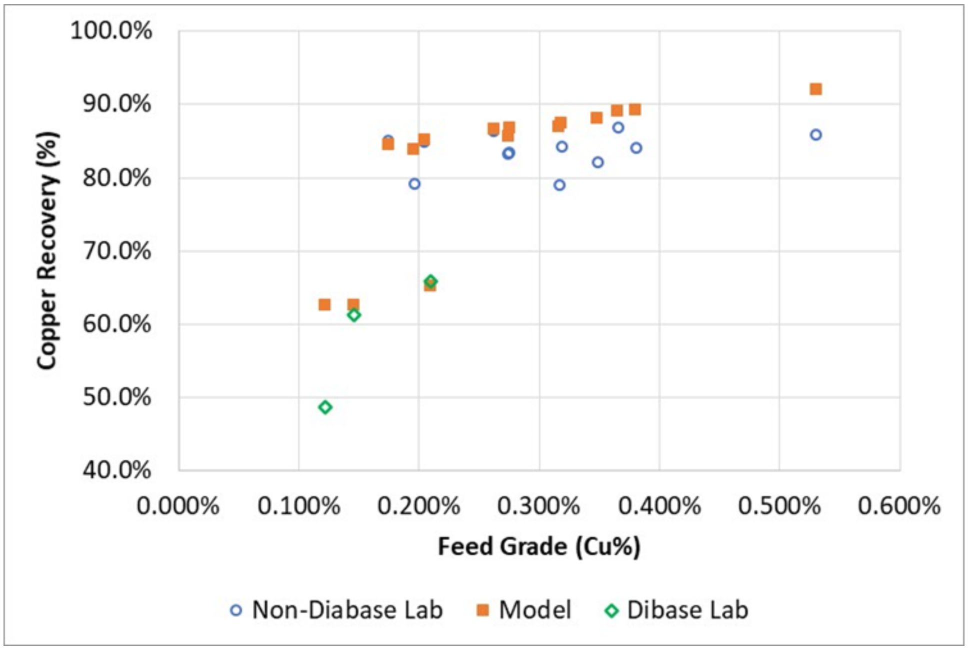


Figure 13-21: Actual and Model Copper Recovery with Diabase

As shown, the presence of Diabase reduces the expected copper recovery by 0.18% per percent Diabase present (18% maximum deduction). Figure 13-22 shows the copper recovery model for an example ore of 0.30% Cu with varying Diabase content (equation 9).

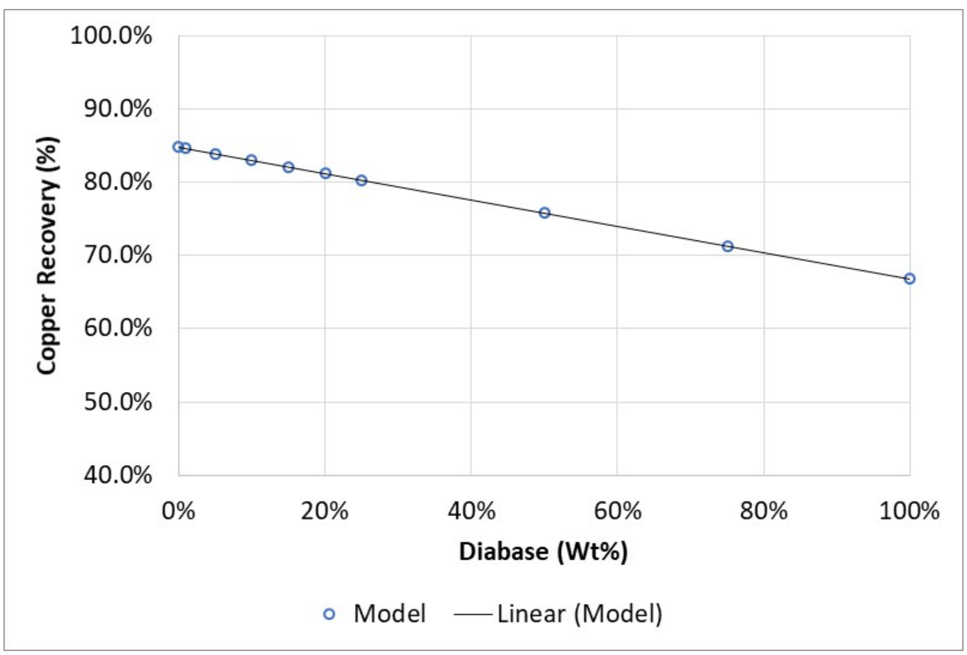


Figure 13-22: Copper Recovery with Various Diabase Contents

13.8 Current Molybdenum Model

The previous Technical Report PV3-2016-PFS established the molybdenum recovery at a constant value set at 47%. This report did indicate that the molybdenum recovery was both a function of the copper recovery and the molybdenum feed grade but did not consider these in the recovery equation. A regression analysis was conducted on the plant data with the results shown in Table 13-12.

Table 13-12: Regression Analysis Results - Molybdenum

Summary Output	
Regression Statistics	
Multiple R	0.511
R Square	0.261
Adjusted R Square	0.260
Standard Error	0.125
Observations	1,285 final conc recovery, tpd >35k tpd

Analysis of Variance					
	Degrees of Freedom	***Sum-of-Squares***	***Mean Squares***	***F-Test***	***Significance F***
Regression	2	7.057	3.528	226.526	~0
Residual	1282	19.968	0.0156		
Total	1284	27.025			

	Coefficients	***Standard Error***	***t Stat***	***P-value***	***Lower 95%***	***Upper 95%***
Intercept	-1.1490	0.07646	-15.0286	~0	-1.29901	-0.9991
Feed Grade Mo %	1116.2776	143.6831	7.7690	~0	834.3977	1398.1574
Cu Recovery %	1.8088	0.0903	20.03268	~0	1.6317	1.98597

The current model for molybdenum recovery uses both the copper recovery and the molybdenum feed grade plus a cleaner flotation recovery factor, as shown in Table 13-13.

Table 13-13: PVM Molybdenum Recovery, 2021

Year	Model	Equation Number
2021	Final Molybdenum Recovery = [-1.149 + 1116.28 *Moly Grade% + 1.8088 * (Cu Recovery%)]*0.2	10

Where:
Moly Grade % - is the concentrator total molybdenum feed grade in percent
Cu Recovery% - is the concentrator final copper recovery in percent
0.2 = moly recovery in the molybdenum cleaner circuit – historic performance

A total of 1,285 concentrator data points were employed representing data from January 2014 to March 2021. The R squared of the regression analysis was approximately 0.3 which is a reasonable fit for operating plant data. Figure 13-23 shows the reconciled production and the model predictions for the same period.

As shown, the model tends to predict a slightly higher molybdenum recovery than observed in the plant but this can be adjusted through the cleaner recovery factor. The data shown was also developed based on the previous reagent system (using NaHS) and a new reagent system is currently being tested in the plant (using D8 and no NaHS). The predicted recovery model is only valid for the previous reagent scheme.

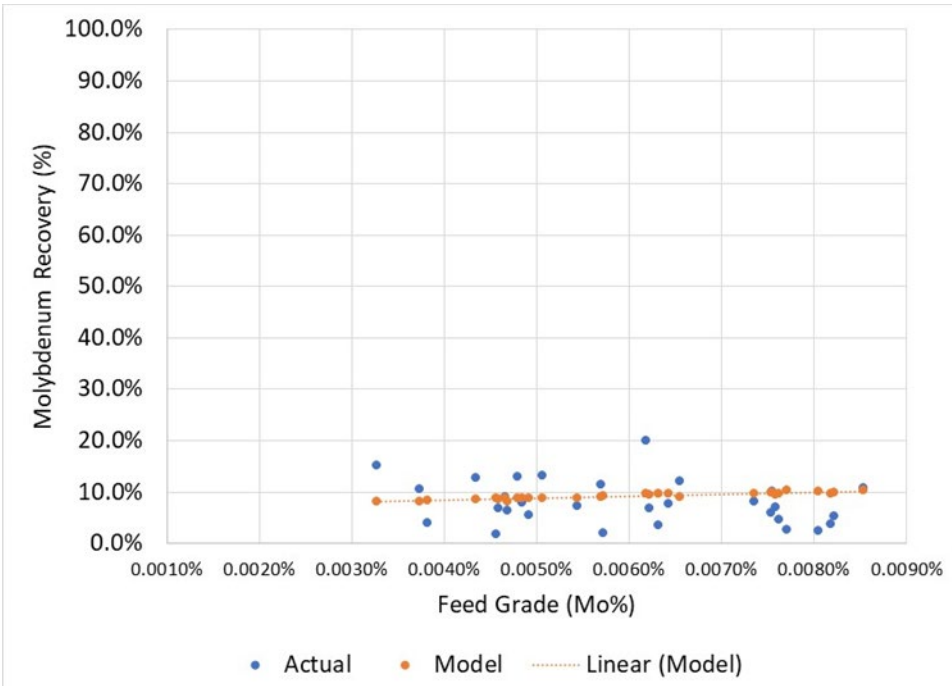


Figure 13-23: Actual and Model Molybdenum Recovery

13.9 Metallurgical Variability

A discussed previously, the majority of the PVM Mineral Resource is in the Ruin Granite. This is typical of the ore that has been more recently processed in the PVM circuit. The range of work index measurements for the various test samples was very narrow with Ruin Granite ranging from 13.5 to 15.5 kWh/t. The flotation test work has also been consistent for the Ruin Granite. The Diabase ores have a higher work index and poorer metallurgical performance but the expected impact on mill performance is minimal with the Diabase blended with the Ruin Granite.

13.10 Processing Factors or Deleterious Elements

No processing factors or deleterious elements that may have a significant effect on potential economic extraction have been identified.

13.11 Risks

There are minimal metallurgical risks associated with the copper production from the PVM plant as the plant has a long production history and the feed material is well characterized. The main risk is the incorporation of a new reagent scheme for the molybdenum circuit. The new reagents have been commercially adopted by other similar plants but there is no guarantee that they will perform as expected when applied at the PVM plant as it is currently configured.

Water supply is assumed to be sufficient to complete processing as tested. Uncertainty inherent to assumptions in hydrological aspects, especially in in the case of more frequent or lengthy drought periods and climate change, could impact PVM's ability to process ore and recover copper and molybdenum as described.

13.12 Opportunities

The following opportunities exist to enhance the performance of the PVM plant:

- Optimize the crushing circuit to remove properly sized product as soon as possible from the circuit. This has the potential to improve plant throughput and reduce the primary grind product particle size.
- Define the potential additional improvement that can be derived from finer blast fragmentation. This has the potential to improve FCP throughput and reduce circulating loads.
- Optimize the flotation circuit performance in terms of mass pull to maximize the rougher flotation recovery.

13.13 Recommendations

The following recommendations are intended to improve the performance of the PVM plant:

- Optimize the grinding circuit to reduce the product P80 size. This has a major impact on copper rougher recovery. Estimated cost for modeling and consulting $50,000, six months duration.
- Investigate the potential benefits of improving the cyclone system to reduce bypass and circulating loads. This should improve grinding efficiency and reduce the P80 product size. Estimated cost for modeling and consulting $30,000, three months duration.
- Investigate the fundamental issue with the flotation of diabase materials. This material has now been almost fully excluded from the Reserve because of its poor metallurgical performance. Estimated cost for mineralogy and flotation test work $75,000, duration three months.
- Reduced rougher flotation pH (less than pH 10) should be investigated as it will improve the recovery of locked pyrite/chalcopyrite particles. Estimated cost for test work, plant trials and analysis $25,000, duration three months.
- Continue to optimize the molybdenum circuit with the new reagent scheme. Estimated cost for test work, plant trials and analysis $100,000, duration three to four months.
- Investigate the impact of recycling water from the molybdenum circuit using the new reagent suite. Costs and duration are included in the above recommendation.

14 Mineral Resource Estimate

14.1 Introduction

This section of the Technical Report describes the mineral resource estimation methodology and summarizes the key assumptions considered by the QP to update the mineral resource estimate for the PVM deposit.

The mineral resource estimate was prepared under the supervision of Mr. Garth D. Kirkham, P.Geo., FGC, of Kirkham Geosystems Ltd. and is effective March 31, 2021. Kirkham is an "independent qualified person" within the meaning of NI 43-101 for the purposes of mineral resource estimates contained in Section 14.10 of this Technical Report.

The mineral resource estimate for this deposit was originally presented in a technical report, PV3-2016-PFS, dated February 23, 2016 with an effective date of January 1, 2016. The effective date of the updated block model used to generate the estimate of mineral resources in this technical report is March 31, 2021.

The mineral resources presented in this report are derived from the same resource block model that was created for the 2016 PFS, but there have been adjustments to the estimate in response to drill programs conducted through 2019, a revised geological model, a revised grade shell model, correction to the molybdenum database, a more accurate estimation of rock density based on new data, and adjustments to the estimation strategy based on five years of production reconciliation and geological and geochemical data analysis.

In the opinion of the QP, the mineral resource estimate reported herein is a reasonable representation of the mineralization found at the PVM Project at the current level of sampling. The mineral resources have been estimated in conformity with generally accepted CIM *Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines* (November 29, 2019) and are reported in accordance with NI 43-101.

The mineral resources were classified according to their proximity to the sample data locations and are reported, as required by NI 43-101, according to the CIM *Definition Standards for Mineral Resources and Mineral Reserves* (May 2014).

Mineral resources are not mineral reserves and they do not have demonstrated economic viability There is no certainty that all or any part of the mineral resource will be converted into a mineral reserve upon application of modifying factors.

Acid soluble copper (ASCu) was estimated to identify and provide guidance regarding the oxide/sulfide interface. However, ASCu is not reported within this Technical Report; the ASCu data within the database is incomplete and therefore not reliable to accurately estimate resources attributable to the oxides. Much of the oxides appear to have been extracted during historical operations. In addition, the majority of remaining ASCu represents localized post-mining oxidation of fresh exposures. Therefore, drillholes that pre-date those mining exposures are not suitable for estimating the post-mining oxidation.

14.2 Data Evaluation

The data used to update the geological model and subsequently generate the Mineral Resource estimate consisted of collar locaionts, downhole survey readings, lithology, mineralization type and assay results (Cu%, ASCu%, Mo%) in .csv format.

Table 14-1 lists the drillholes used in PV3-2016-PFS and in this Technical Report. No holes were drilled in 2020, resulting in a total of 951 drillholes representing 706,177.8 ft of drilling with Cu% assays. Most of these reside within the surrounding pit area.

There are a total of 260,279 individual samples in the PVM Project database, the majority of which have been analyzed for Cu%, ASCu% and Mo%. Sample data for copper and molybdenum have been extracted from the main database and imported into MineSight® to develop the resource model.

The dataset included: 1) assay values, 2) zeroes which represent actual zeroes on assay certificates/documents and are likely to represent an older methodology before ½ detection limit values were used in place of zero and; 3) negative values that are typically -1, and represent intervals for which there is a conclusive absence of data. Table 14-2 shows the complete database for statistics Cu%, ASCu% and Mo%, respectively. Figure 14-1 shows a plan view for the drillholes used in the analysis.

Table 14-1: Drillholes used in 2015 and 2021 Resource Estimates

Drillhole Source	Number
Original BHP resource drillholes	778
PVRC-13 series	59
GTH-14 series	10
PZ-14 series	4
PVRC-15 series	43
GTH-15 series	3
Sub-total used in 2015 Resource Estimate	**897**
2016 drilling (4 RC)	4
2017 drilling (17 RC , 1 DDH with RC pre-collar)	18
2018 drilling (22 RC , 1 DDH with RC pre-collar)	23
2019 drilling (8 DDH with RC pre-collar, 1 DDH)	9
Total used in 2021 Resource Estimate	**951**

Table 14-2: Assay Statistics

Element	#Valid Samples	Max	Mean	SD	CV
Cu%	94092	8.16	0.264	0.239	0.904
ASCu%	81402	0.62	0.005	0.006	1.147
Mo%	84785	5.96	0.011	0.073	6.933

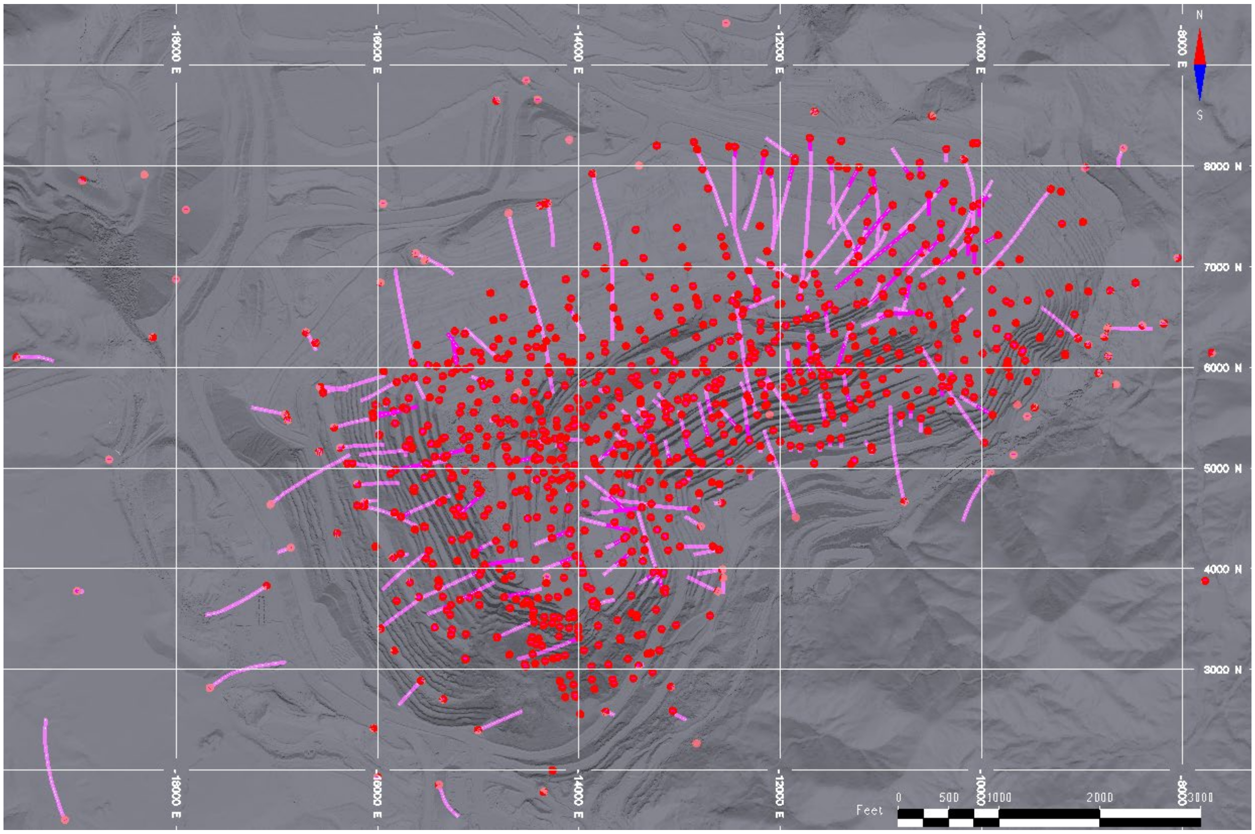


Figure 14-1: Plan View Showing Drillholes Used with December 31, 2020 Topography (source: Kirkham, 2021)

14.3 Geologic Model and Domains

As described in Section 9 (Deposit Types), the PVM deposit is classified as a copper-molybdenum porphyry system. Surfaces and solids were generated for lithology wireframes and structural domains.

The geologic information derived from observations during core logging provide lithology code designations for the various rock units present on the property. The lithology domains were revised and remodeled for this Technical Report. The numeric coding for the lithology domains is provided in Table 14-3.

Table 14-3: PVM 2021 Model Update Lithology Codes

Lithology	Code
Pinal Schist*	10
Ruin Granite	11
Granite of Manitou Hill	13
Pioneer Formation	20

Dripping Spring Quartzite	21
Quartzite (generic)	25
Diabase	30
Troy Quartzite	40
Limestone	41
Granite Porphyry	60
Granodiorite	50
Whitetail Conglomerate	90
Dacite*	95
Gila Conglomerate*	100
Basalt*	101

*Not a significant lithology in the Mineral Resource

Lithologies have been interpreted by:

- Snapping wireframe nodes to lithology changes of drillholes.
- Digitizing lithology boundaries of the pre-mining topography (Peterson et.al, 1951) and correlating with sub-terranean information.
- Digitizing lithology boundaries over various older pit lithology maps.
- Using trace element blasthole assay information as proxies for lithologies.
- Interpreting high- and low-density pit LIDAR scans.
- Completing frequent pit mapping.
- Incorporating drone data to delineate lithologies and structural elements where physical access was prohibitive.

As shown in Table 14-3 the PVM lithological model consists of 15 lithological groups. The primary lithological units representing potential ore-hosts are the Ruin Granite, Granodiorite, Diabase and Granite Porphyry.

A comparison of surface geology from pre-mine conditions to the current mining topography is shown in Figure 14-2.

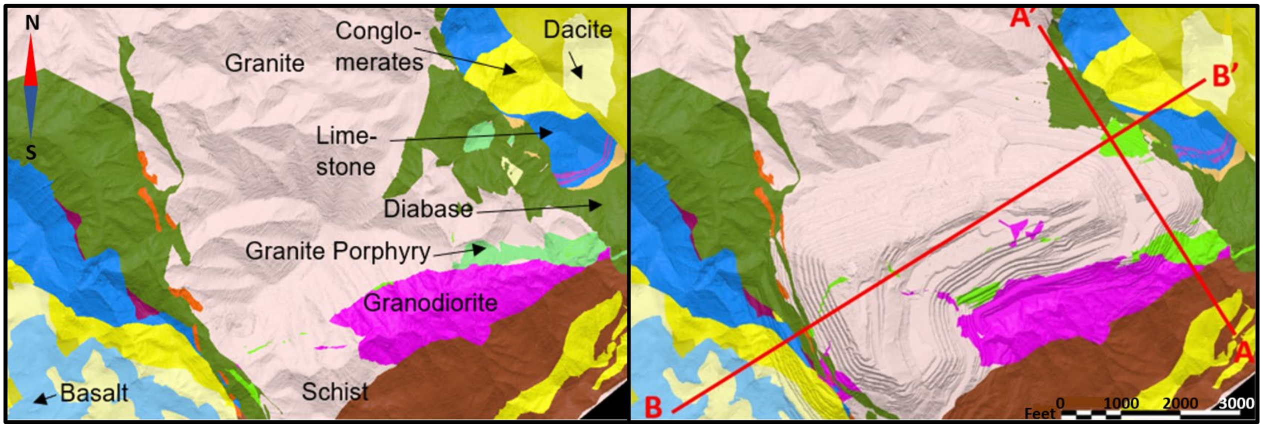


Figure 14-2: Comparison of Lithologies before Mining (left) and December 31, 2020 Topography (right) with Cross Section Reference lines (source: Kirkham, 2021)

Faults were not explicitly acknowledged during field mapping, with the exception of the Schist, Jewel Hill, Gemini and West End faults. Cross sections are shown in Figure 14-3 and Figure 14-4, with section reference lines shown in red in Figure 14-2.

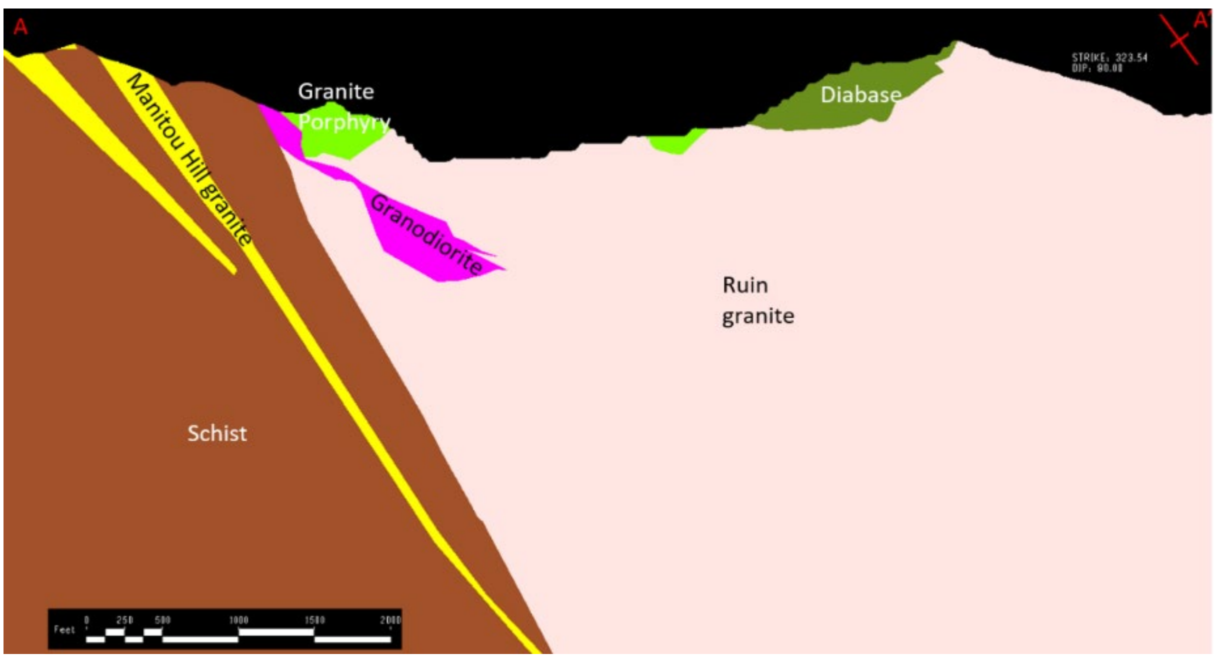


Figure 14-3: Cross section A-A' of Lithological Model Looking West (source: Kirkham, 2021)

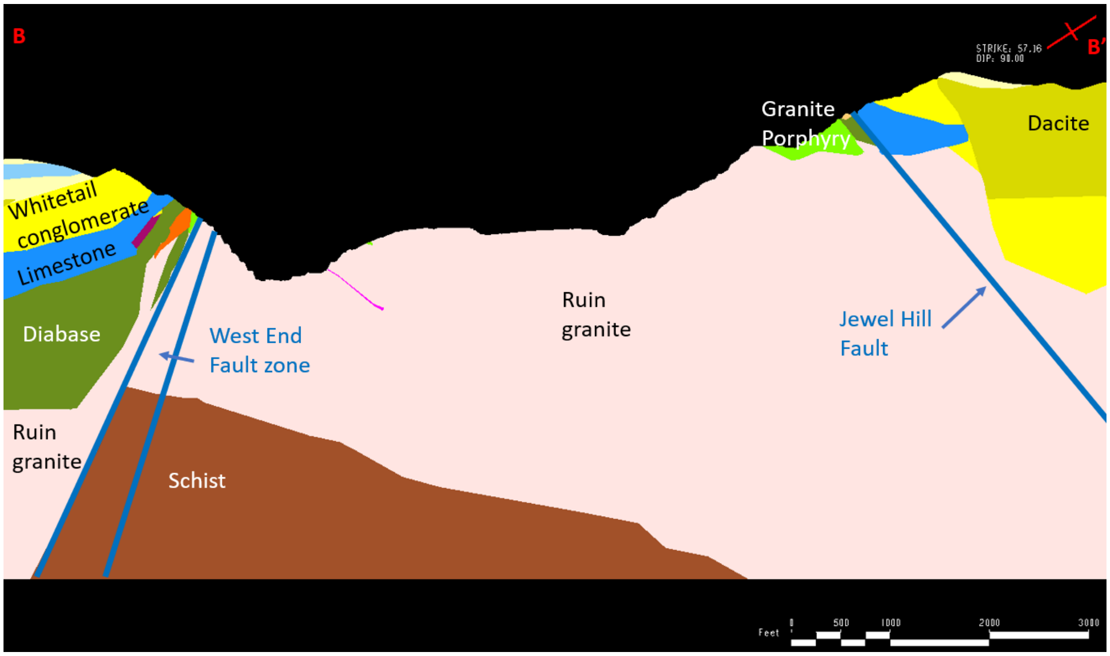


Figure 14-4: Cross section B-B' of Lithological Model with Stylized Fault Traces Looking North (source: Kirkham, 2021)

The primary host rock for the PVM deposit is the Precambrian Ruin Granite. Mineralization occurred with the intrusion of two Paleogene units: the Granite Porphyry and the Granodiorite. The Granite Porphyry is considered to have been the main mineralizing unit and as such it also contains generally economic mineralization but the Granodiorite formed very late in the mineralizing event and contains very little in the way of economic copper grades.

A lesser, amount of mineralization is found within the West End fault zone (Figure 14-5 and Figure 14-6). This fault zone consists of a shear zone that is bounded by at least two relatively close major faults, making it the most structurally complex fault block in the model. The primary host rock for mineralization within this zone is again the Precambrian Ruin Granite, which is cut by several slightly younger aplite sills (APLITE unit) and a single, branching Precambrian diabase dyke.

This West End fault zone is further complicated by the stratigraphically overlying basalt, Gila and Whitetail conglomerates, Paleozoic Limestone as well as the later intrusion of the Paleogene Granodiorite. Pinal Schist is the basement unit within the Gold Gulch fault block and is intruded by the Ruin Granite.

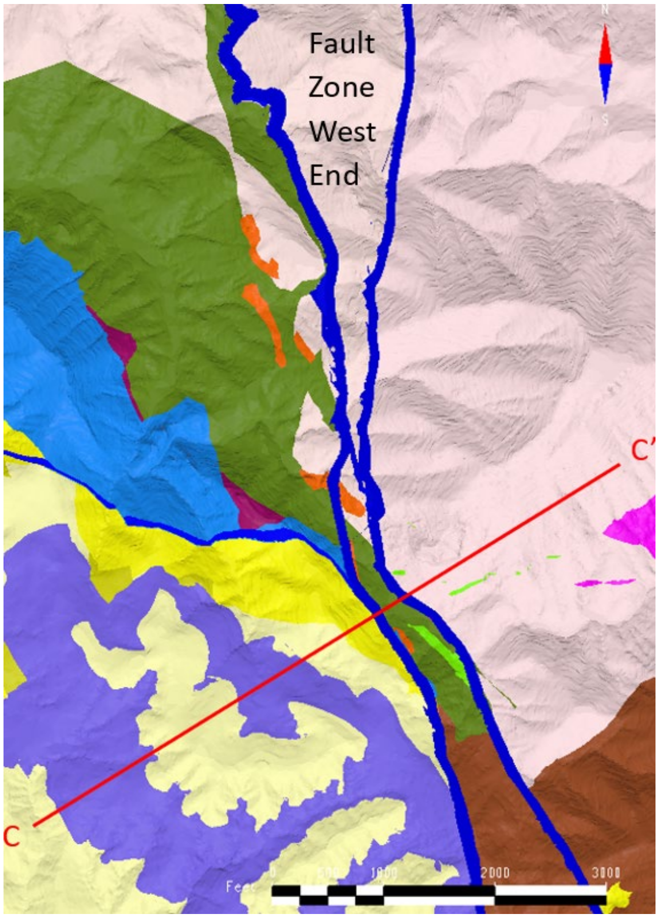


Figure 14-5: Plan View of Lithological Model for West End Fault Zone with Location of Cross Sections (C-C') (source: Kirkham, 2021)

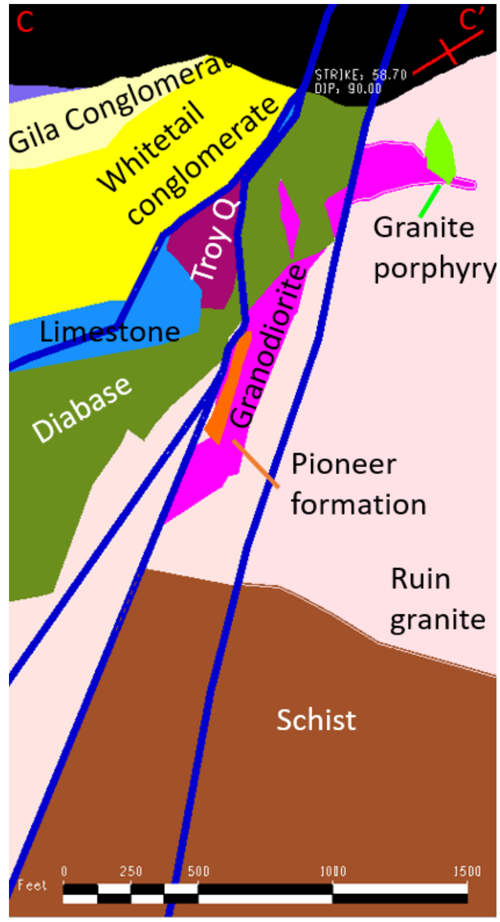


Figure 14-6: Southwest-Northeast Cross Sections (C-C') (source: Kirkham, 2021)

14.3.1 Structural Domains

In addition to the lithology wireframes, domains were interpreted that either separated different mineralization trends/truncations or consisted of established faults. The model in this Technical Report used domain boundaries from important faults that still impact current mining. These are the West End/Gold Gulch fault zone, South Hill or Schist fault, Jewel Hill, Gemini and Dome faults. In addition to these notable faults, an abrupt change in Cu and other geochemical values was modeled as a structural boundary. This feature lies just on the north side of the main trend, in the northeast side of the deposit.

Figure 14-7 shows a plan view of stylized fault traces of the pre-mine topography and Figure 14-8 represents faults in relation to the current pit surface.

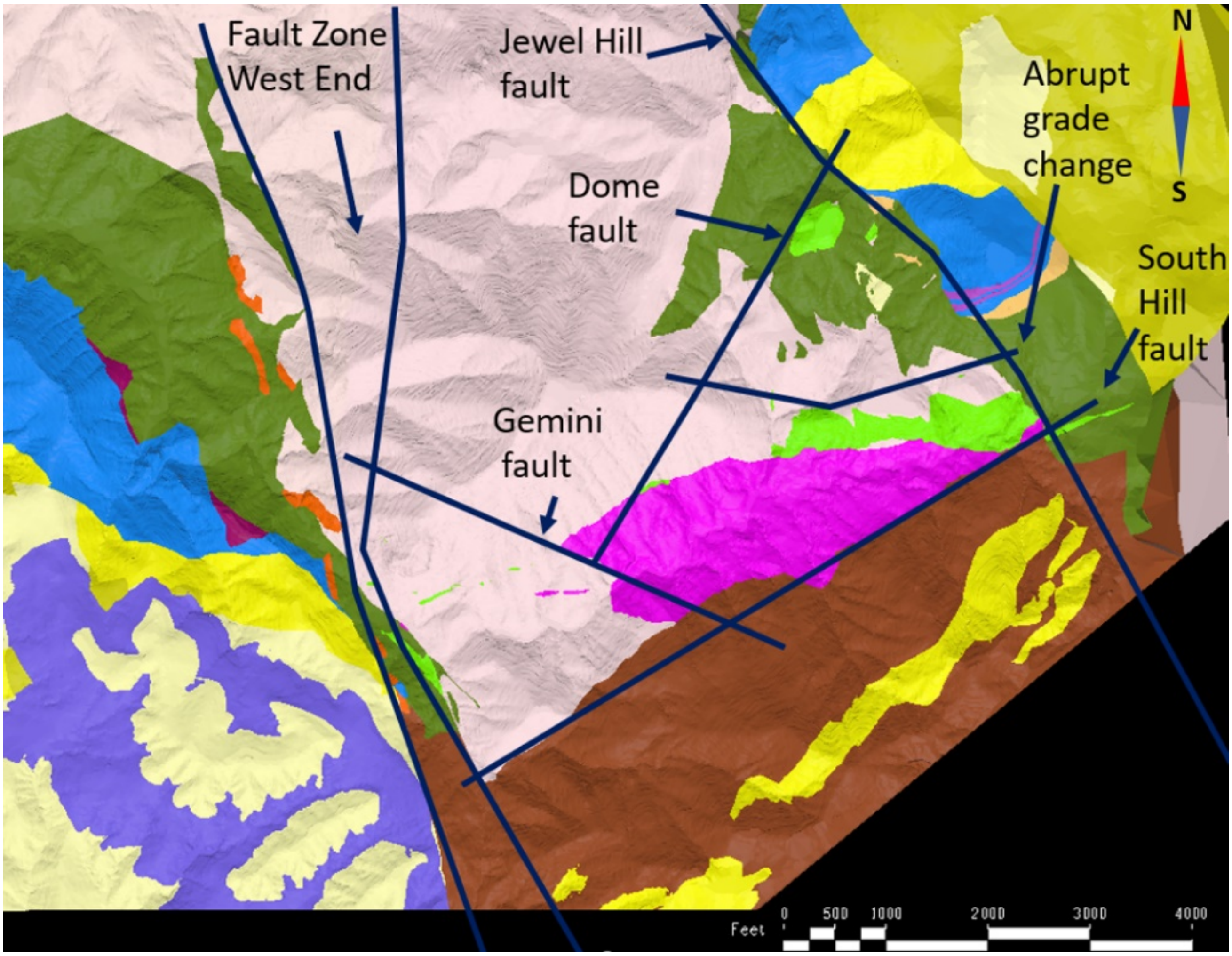


Figure 14-7: Plan View Showing Major Faults (source: Kirkham, 2021)

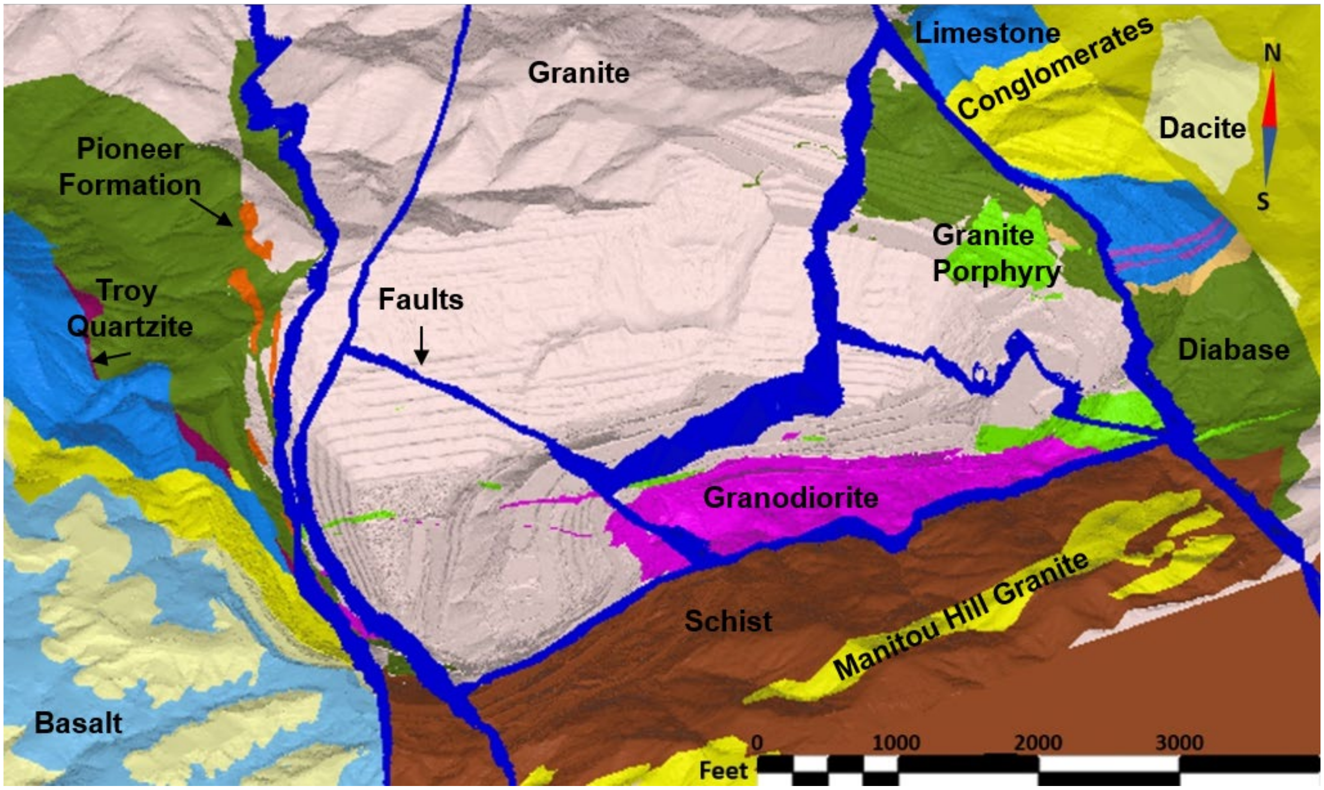


Figure 14-8: Lithology and Fault Surfaces of the Current Pit Topography to March 31, 2021 (oblique view looking north) (source: Kirkham, 2021)

Based on the structures shown in Figures 14-7 and Figure 14-8, the final structural domains are represented in Figure 14-9.

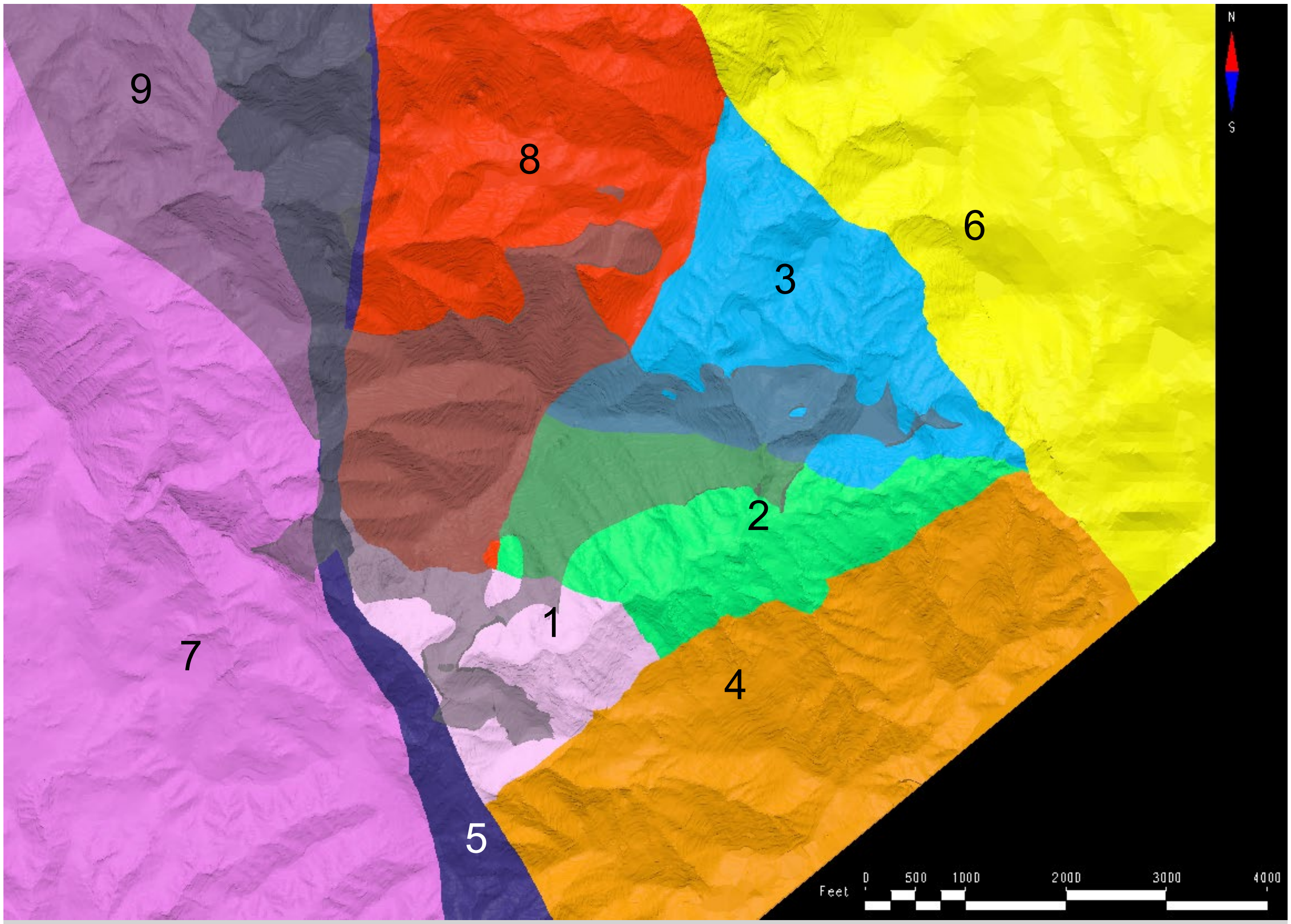


Figure 14-9: Structural Domain Representation Below the Original Pre-mining Topography (source: Kirkham, 2021)

Domain 9 (gray shading), shown in Figure 14-9, is a near-surface domain containing very high grade copper intercepts interpreted to represent secondary supergene enrichment. This portion of the deposit is largely mined out, and the majority of the remaining material is located on the western side of the deposit. This domain is given precedence over other domains during coding of composites and blocks.

14.3.2 Indicator-Based Grade Shell

To better control smoothing between waste and potentially mineralized material, an indicator-based grade shell was generated at a 0.08% Cu threshold. Composites were coded with an Indicator value of "1" if their Cu grade was below 0.08% and with "0" if above. The indicator value was then interpolated into blocks, and if the resulting value was greater than 0.45, then the block was considered "outside" the 0.08% Cu grade shell. Note that these indicator values may be considered, in this context, a quantification of probability where a value of 0.45 is essentially a 45% probability.An inverse distance squared (ID2) procedure was used for a weighted interpolation of the indicator. Although ID2 is a rapid technique, the impact of a 0.08%

grade shell on mill ore greater than 0.17% is expected to be very low. A single interpolation pass was performed for each structural domain using ellipsoidal search parameters that were established for the large estimation pass of total copper (Section 14.6). A minimum of one and a maximum of and six composites per block were used, with a maximum of three per hole.

The 0.45 probability threshold was established by comparing ID2 results to a Nearest Neighbor (NN) estimation of the indicator. Table 14-4 lists various thresholds and validation volumes that were used to compare the ID2 volumes against the corresponding NN volume.

Table 14-4: Calibration of Indicator Grade Shell Threshold Against NN Indicator Volume

Method - Threshold	Volume	
	December 31, 2020 Reserve solid (ft^3)	December 31, 2020 Resource solid (ft^3)
NN	1,974,467,966	11,525,623,687
ID2 - 0.45	2,070,663,554	11,697,912,487
ID2 - 0.50	1,963,924,750	11,195,355,106
ID2 - 0.55	1,844,763,931	10,656,753,028

14.4 Composites

Assessment of composite lengths showed that a 45 ft fixed-length composites would minimize the smoothing of the grades, and reduce the influence of typically narrow, higher-grade samples. This would also match the 45 ft pit bench height used for mine planning.

A change in lithology triggered the beginning of a new composite, and remnants at the end of a hole were added to the previous composite if it was shorter than half the 45 ft composite length. The impact of any variation on composite length was mitigated by the use of length-weighting the composite during interpolation. Each composite was coded by lithology, structural domain and grade shell.

Univariate lithology statistics for Cu and Mo assays/composites are listed in Table 14-5 and Table 14-6, respectively, and univariate domain statistics for Cu and Mo assays/composites are listed in Table 14-7 and Table 14-8, respectively.

Table 14-5: Univariate Statistics for Lithology Coded Cu Assays and Composites

Univariate Statistics for Length-Weighted Cu%						
Lithology	Assays Cu%			Composites Cu%		
	Num Samples	Weighted Mean	Weighted CV	Num Samples	Weighted Mean	Weighted CV
Schist	1,358	0.007	2.00	313	0.007	1.40
Ruin Granite	80,185	0.286	0.78	13,692	0.285	0.69
Pioneer Formation	204	0.071	1.68	53	0.071	1.27
Dripping Spring Quartzite	49	0.007	0.89	12	0.007	0.50
Generic Quartzite	0	-	-	0	-	-
Diabase	5,831	0.104	1.77	1,115	0.104	1.56
Troy Quartzite	178	0.461	1.12	43	0.461	0.86

Limestone	705	0.062	3.35	149	0.061	2.83
Granodiorite	2,647	0.104	0.90	590	0.103	0.79
Granite Porphyry	1,788	0.206	0.70	477	0.205	0.63
Whitetail Conglomerate	458	0.021	2.89	102	0.022	2.61
Dacite	44	0.002	1.13	10	0.002	0.87
Gila Conglomerate	182	0.009	0.88	44	0.009	0.79
Basalt	46	0.009	s1.43	11	0.009	0.61

Table 14-6: Univariate Statistics for Lithology Coded Mo Assays and Composites

Univariate Statistics for Length-Weighted Mo%						
Lithology	Assays Mo%			Composites Mo%		
	Num Samples	Weighted Mean	Weighted CV	Num Samples	Weighted Mean	Weighted CV
Schist	1,248	0.0006	1.99	280	0.0006	1.25
Ruin Granite	69,167	0.0056	1.13	11,800	0.0055	0.87
Pioneer Formation	179	0.0011	1.52	48	0.0011	1.40
Dripping Spring Quartzite	46	0.0005	0.81	11	0.0005	0.72
Generic Quartzite	0	-	-	0	-	-
Diabase	5,020	0.0009	1.75	998	0.0009	1.50
Troy Quartzite	178	0.0036	1.38	43	0.0036	1.10
Limestone	690	0.0011	2.63	144	0.0011	2.41
Granodiorite	2,338	0.0024	1.38	536	0.0023	1.08
Granite Porphyry	1,447	0.0041	0.89	366	0.0041	0.75
Whitetail Conglomerate	435	0.0006	1.34	93	0.0006	1.88
Dacite	44	0.0001	-	10	0.0001	-
Gila Conglomerate	182	0.0005	0.49	44	0.0005	0.42
Basalt	46	0.0005	1.01	11	0.0005	0.69

Table 14-7: Univariate Statistics for Cu Assays and Composites by Structural Domain

Univariate Statistics for Length-Weighted Cu%						
Domain	Assays Cu%			Composites Cu%		
	Num Samples	Weighted Mean	Weighted CV	Num Samples	Weighted Mean	Weighted CV
1	29,323	0.284	0.67	5,655	0.284	0.59
2	13,189	0.272	0.72	2,393	0.267	0.64
3	8,634	0.123	0.90	1,363	0.125	0.76
4	1,116	0.006	1.66	258	0.006	1.11
5	3,442	0.227	1.02	688	0.227	0.91
6	1,038	0.039	1.75	189	0.039	1.33
7	4,169	0.081	2.50	954	0.088	2.09
8	17,391	0.272	0.64	2,745	0.272	0.56
9	13,884	0.372	0.91	2,188	0.368	0.78

Table 14-8: Univariate Statistics for Mo Assays and Composites by Structural Domain

Univariate Statistics for Length-Weighted Mo%						
Domain	Assays Mo%			Composites Mo%		
	Num Samples	Weighted Mean	Weighted CV	Num Samples	Weighted Mean	Weighted CV
1	26,870	0.0051	1.00	5,249	0.0051	0.75
2	12,358	0.0064	0.94	2,185	0.0063	0.82
3	8,420	0.0035	1.63	1,322	0.0035	1.31
4	1,017	0.0006	2.15	229	0.0006	1.36
5	3,005	0.0051	1.38	627	0.0050	1.12
6	599	0.0007	0.85	131	0.0007	0.82
7	4,037	0.0009	2.15	930	0.0013	2.25
8	16,081	0.0057	0.88	2,530	0.0057	0.72
9	7,176	0.0055	1.97	990	0.0055	1.33

The basic statistics shown in these tables indicates that the copper and molybdenum data are reasonably well distributed. The coefficient of variation (CV) in copper and molybdenum composites and assays are typically lower in structural domains than in lithology wireframes. The same lithologies can be located in different parts of the deposit which can expose them to different grade regimes, which produces a higher CV.

The lowest CVs for Cu% are in the predominantly higher-grade structural domains 1, 2, and 8. High CV's for Cu% in structural domains 6 and 7 reflect negligible amounts of erratic mineralization at the far ends of the model. The CV's are considered relatively high for such a homogeneous, consistent deposit, as is commonly the case for copper prophyries, therefore some level of grade limiting or cutting is warranted.

14.5 Outliers

Cumulative frequency plots were analysed to determine the threshold levels at which to implement a grade-limiting strategy. One is by physically cutting the grades of the composites, and the other is by limiting the influence that a high-grade composite has by limiting the distance to which it contributes to the grade of a block estimate. Therefore, within a given radius of an estimated block, composites were uncapped, and beyond that radius the composite was reduced to the capping threshold. As stated, capping thresholds for each domain were determined by analyzing the cumulative probability plots (CPP) and histograms to determine a reasonable capping threshold.

Kinks or gaps at the high end of the CPP graph correspond to a capping grade that was then verified by grade distributions of the histogram. The outlier radius was generally set to 10% of the shortest search-ellipsoid distance.

Figure 14-10 shows an example of CPP for domain 1, which applied outlier restriction for composites above 0.7% Cu.

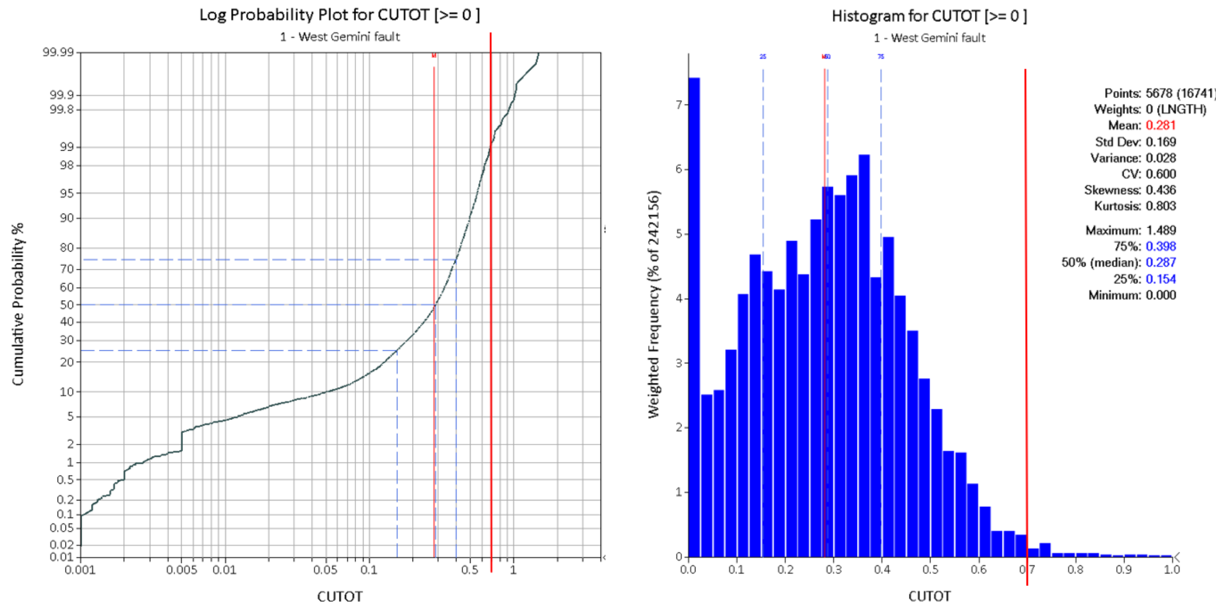


Figure 14-10: Example of a Cumulative Probability Plot and Histogram Showing Capping Strategy for Cu%

Table 14-9 shows the capping grades for all domains in the four primary lithologies that potentially host ore (rock types 11, 30, 50 and 60).

Table 14-9: Capping Parameters for Estimation of Copper in Structural Domains for Rock Types 11, 30, 50 and 60

Domain	1	2	3	4	5	6	7	8	9
Capping Grade	0.70	0.62	0.40	0.02	0.30	0.20	0.70	0.70	1.10
Capping Distance	235	90	66	225	150	130	100	65	80

All remaining rock types applied a 500 ft outlier search restriction to any composite greater than 0.2% Cu. This was true for all structural domains for the remaining rock types. For Mo estimation, a 150 ft outlier search restriction was applied to any composite grading greater than 0.04% Mo.

14.6 Contact Profile Analysis

Contact profiles evaluate the nature of grade trends between two domains. A contact profile analysis was performed to determine whether domains are suited to serve as a hard or soft boundary. Hard contacts do not allow composites from neighbouring domains to be used for block interpolation while a soft boundary may include composites from neighbouring domains.

Most domains were hard contacts with only domains 1-2, 1-8 and 2-8 treated as soft boundaries.

14.7 Variography

The semi-variogram is a common function used to measure the spatial variability within a deposit. Spatial variability and continuity in a mineral deposit depend on both the distance and direction between points of comparison. If the variability is related to the direction of comparison, then the deposit is said to exhibit anisotropic tendencies which can be summarized by an ellipse fitted to the ranges in the different directions.

Components of a variogram include the nugget, the sill, and the range. Often samples compared over very short distances (including samples from the same location) show some degree of variability. As a result, the curve of the variogram often begins at a point on the y-axis above the origin; this point is called the nugget. The nugget is a measure of both the natural variability of the samples over very short distances, and the variability by errors during sample collection, preparation, and assaying.

Typically, variability between samples increases as the distance between the samples increase, while samples that are closer together are more similar to each other. Eventually, the degree of variability between samples reaches a constant or maximum value; this is called the sill, and the distance between samples at which this occurs is called the range.

Using the 2021 structural domains, copper variography was performed for each domain using Snowden's Supervisor. The 2021 model used correlograms with two exponential structures. The correlogram is normalized to the variance of the data and is less sensitive to outlier values. This generally gives cleaner results. Correlograms were generated for the distribution of copper within each of the structural domains. Lithology was not considered because more than 97% of the Mineral Resource being hosted by Ruin Granite. Table 14-10 lists the variographic model parameters for copper.

Table 14-10: Cu Variogram Model Parameters by Structural Domain

Domain	Structure	C0	C1 or C2	MineSight ranges in ft			MineSight rotations in degrees		
				Major	Minor	Vert	Rot1	Rot2	Rot3
1	1st	0.17	0.66	3,216	1,839	933	96	22.5	20.4
	2nd		0.17	3,797	2,344	2,635	96	22.5	20.4
2	1st	0.13	0.03	119	1,674	617	87.8	6.4	39.6
	2nd		0.84	5,502	3,762	870	87.8	6.4	39.6
3	1st	0.16	0.26	1,188	708	659	100	0	30
	2nd		0.58	3,895	1,878	660	100	0	30
4	1st	0.16	0.31	3,178	2,074	1,146	9.3	-37.8	-153.4
	2nd		0.53	3,179	5,759	2,227	9.3	-37.8	-153.4
5	1st	0.1	0.51	698	2,090	898	-25	8.6	59.6
	2nd		0.39	5,060	4,952	1,458	-25	8.6	59.6
6	1st	0.24	0.25	2,317	99	499	120	0	-90
	2nd		0.51	2,501	1,309	2,007	120	0	-90
7	1st	0.26	0.59	1,343	513	318	24.6	75.9	44.6
	2nd		0.15	1,599	1,931	982	24.6	75.9	44.6
8	1st	0.02	0.91	2,702	1,470	632	77.7	6.4	39.6

	2nd		0.07	2,703	1,471	633	77.7	6.4	39.6
9	1st	0.09	0.64	731	732	411	98.9	6.7	18.9
	2nd		0.27	2,084	2,965	755	98.9	6.7	18.9

Correlograms were not generated for the distribution of molybdenum due to low economic contribution, and because an Inverse Distance interpolator was used for Mo (see Sec 14-8).

14.8 Densities

For the 2021 model, updated variable density values were applied based on lithology and alteration. These values were established using the results from approximately 300 density tests conducted in 2018 on five lithologies and 17 alteration types.

Table 14-11 shows the test results in SI density units.

Table 14-11: Density Statistics by Lithology and Alteration Type

	Ruin Granite		Diabase		Limestone		Grano-diorite		Granite Porphyry		Weighted Averages	
Alteration	**SG**	**#**	**SG**	**#**	**SG**	**#**	**SG**	**#**	**SG**	**#**	**SG**	**#**
Sericite (60)	2.57	74					2.59	27	2.59	16	2.58	117
Biotite (40)	2.62	41	2.63	5							2.62	46
Quartz-Sericite (55)	2.64	22	2.89	6			2.65	3	2.68	8	.269	39
Weak magnetite in biot sites (45)	2.64	10	2.77	14							2.72	24
Deep Quartz-Sericite vns (57)	2.62	8									2.62	8
Green Sericite overprint (59)	2.63	5									2.63	5
Strong magnetite in biot sites (48)	2.63	4									2.63	4
Weak magnetite in biot sites + qtz vns (46)	2.64	4									2.64	4
Biotite cut by qtz vns (44)	2.70	3									2.70	3
Epidote (15)			*2.64*[1]	*2*							2.64	2
Biotite-Chlorite (41)			*2.57*[1]	*1*							2.57	1
Clay (90)			2.73	3							2.73	3
Chlorite (70)			2.73	18			*2.58*[1]	*1*	*2.59*[1]	*1*	2.72	20
K-Feld (20)							2.57	19			2.57	19
No alteration (105)					*2.72*[2]	*3*					0.72	3
Calc-silicate (86)					*2.64*[2]	*1*					2.64	1
Garnet (82)					*2.71*[2]	*6*					2.71	6
Weighted averages	2.61	171	2.75	49	2.71	10	2.58	50	2.62	25	2.63	305

1. A total of 5 samples were not used due to insufficient sample coverage.
2. The ten limestone samples were averaged to a single value for density assignment.

Blocks coded with lithology-alterations listed in Table 14-11 were assigned the corresponding density. Blocks without corresponding alterations were assigned a lithology average. For schist and conglomerates, densities were taken from the 2015 PV3 Geomechanical Report by SRK (2015). Basalt was given an average value taken from the literature. All remaining rock was assigned a default value of 2.63 g/cm^3.

Block model assignment of alteration codes were assigned by NN, and were used exclusively in the density assignment of blocks.

Figure 14-12 shows the density distribution for all blocks on 3320 Bench, as an example.

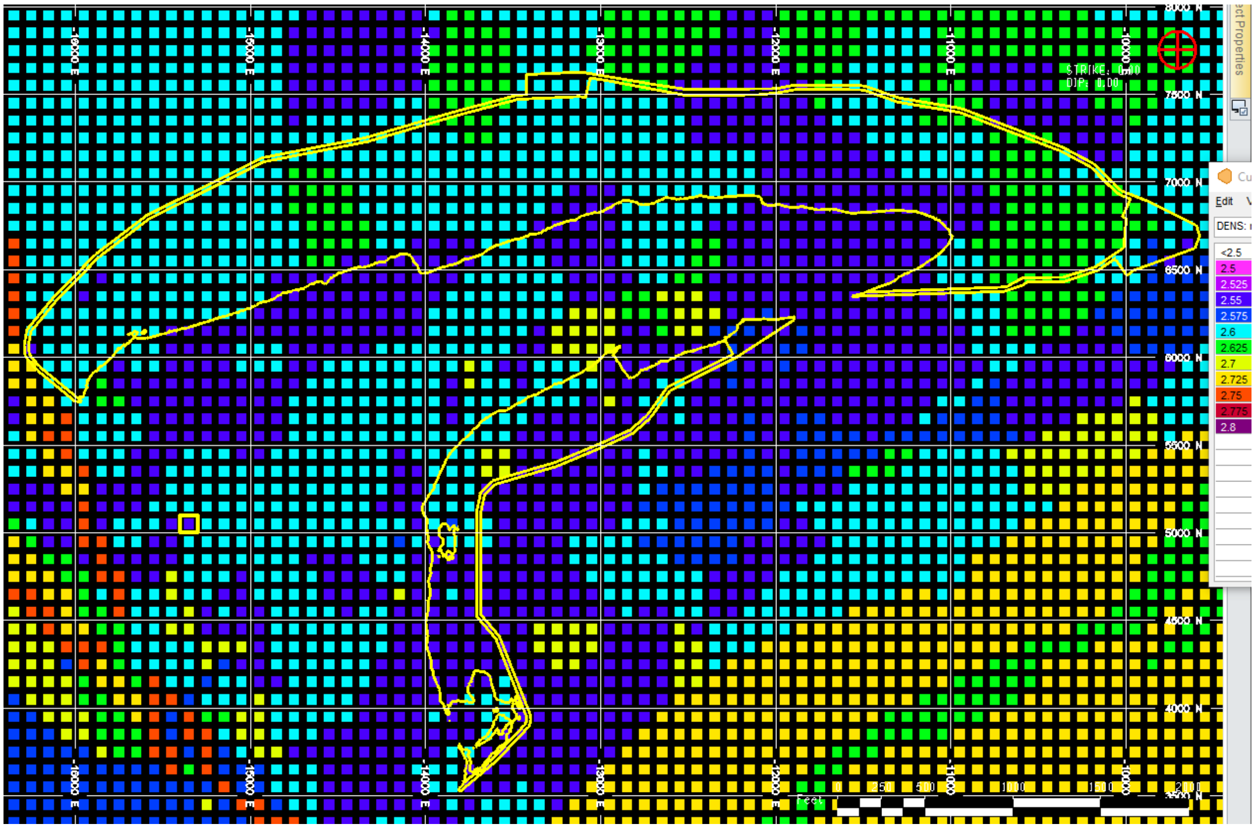


Figure 14-11: Densities with Remaining Reserve Outline at 3320 Bench (Source: Kirkham, 2021)

14.9 Block Model Definition

The 2021 block model used the same dimensions as the March 2016 model, which are shown in Table 14-12. The block model is orthogonal and non-rotated, reflecting the orientation of the deposit. The chosen block size was 100 ft × 100 ft × 45 ft to roughly reflect the available drillhole spacing and bench height and to adequately represent the deposit with a representative number of discrete points in three dimensions. The block size corresponds to the same selective mining unit (SMU) used for past operations.

Table 14-12: Block Model Bounds

Direction	Min	Max	Block Size (ft)	Block Count
X (columns/i)	-25,000	-7,000	100	180
Y (rows/j)	-2,000	10,000	100	120
Z (levels/k)	-10	5,030	45	112

The block model grades for Cu were estimated using Ordinary Kriging. Three estimation passes were performed for each estimation domain. The majority of blocks were estimated in the short estimation pass (approximately 99% of reserve blocks and 98% of resource blocks). The medium and long estimation passes interpolated grades into areas of unusual data geometry that were not assigned in the short estimation pass. Search distances for these passes were based on multiples of the longest variogram range, or D80 (the distance at which 80% of the sill is reached). Table 14-13 lists the ellipsoidal search parameters for each structural domain.

The long estimation pass used a minimum of one and a maximum of twelve composites, with a maximum of three composites per hole. Similarly, the medium estimation pass used was a minimum of four and maximum of eight composites, with a maximum of three composites per hole. The short estimation pass used a minimum of five and maximum of eight composites, with a maximum of three composites per hole.

Estimation domains were defined by each combination of lithology group, structural domain, and grade shell. In almost all cases, these estimation domains applied "hard boundaries" and therefore blocks were only interpolated using composites containing the same combination of lithology, structural domain, and grade shell. The few boundaries considered to be "soft" were structural domains 1-2, 2-8 and 1-8, which enabled the sharing of composites during estimation.

Table 14-13: Ellipsoidal Search Parameters for Cu Grade Estimation

	Domain	1	2	3	4	5	6	7	8	9
Long pass (Run 1) in feet	Major	7,600	11,000	7,800	6,400	10,100	5,000	3,200	5,400	4,200
	Minor	4,700	7,600	3,800	11,600	10,000	2,600	4,000	2,600	6,000
	Vert	5,300	1,800	1,320	4,500	3,000	4,000	2,000	1,300	1,600
Medium pass (Run 2) in feet	Major	3,800	5,500	3,900	3,200	5,050	2,500	1,600	2,700	2,100
	Minor	2,350	3,800	1,900	5,800	5,000	1,300	2,000	1,300	3,000
	Vert	2,650	900	660	2,250	1,500	2,000	1,000	650	800
Short pass (Run 3) in feet	Major	1,600	2,600	1,400	1,400	1,150	1,100	600	1,400	650
	Minor	850	1,800	700	2,000	1,600	400	300	750	550
	Vert	550	450	300	800	600	700	180	300	250
All passes	RotZ	96	88	100	9	-25	120	25	78	99
	RotX	23	6	0	-38	9	0	75	6	7
	RotY	20	40	30	-153	60	-90	45	40	19

Please refer Figure 14-9 to view domain locations. Rot is MineSight Rotation in degrees.

Mo was estimated by an Inverse distance squared procedure. The amount of data available for estimating Ag, Au and Fe is limited, and therefore estimating these elements in the same manner would not result in meaningful results.

14.10 Mineral Resource Classification

The mineral resources for the PVM Project were classified in accordance with the CIM *Definition Standards for Mineral Resources and Mineral Reserves* (May 2014). The classification parameters are defined relative to the distance between copper sample data and are intended to encompass zones of reasonably continuous mineralization that exhibit the desired degree of confidence. These parameters are based on visual observations and statistical studies. Classification parameters are based primarily on the nature of the distribution of copper data because copper is the main contributor to the relative value of this polymetallic deposit.

The spatial variation pattern incorporated in the semi-variogram and the drillhole spacing can be used to help predict the reliability of estimation for copper metal. In this case, there are two potentially economic metals, however copper is the greatest contributor to NSR. Therefore, copper variation will dominate estimation uncertainty and ultimately determine drill spacing. The measure of estimation reliability or uncertainty is expressed by the width of a confidence interval or the confidence limits.

It is possible to calculate the drillhole spacing that is required to achieve the target level of reliability. For instance, Indicated resources may be adequate for planning purposes in most prefeasibility work. For feasibility studies, it is not uncommon to require Measured resources to define the production within the payback period, and then Indicated resources for would be required scheduling beyond payback time.

In the case of the current deposit, there is some information available from several domains, and the spacing between holes varies with much of the data at a 200 ft spacing. Results from this study will be validated against current and future drilling.

Confidence intervals are intended to estimate the reliability of estimation for different volumes and levels of drillhole spacing. A narrower interval implies a more reliable estimate, and attempts should be made to have enough closely spaced holes in the drilling to accurately determine the spatial correlation structure of copper samples less than 200 ft apart.

The following details the grid spacing for each resource category to classify Measured, Indicated and Inferred resources:

Measured Mineral Resources

A spacing of 250 ft may be required to classify Measured Resources. This spacing was based upon five years of production reconciliation. When this level of reconciliation data is available in sufficient quantity and quality, this can help improve methods based on kriging variance. From 2016 through 2020, the results indicate that the tonnes and grade of volumes equivalent to annual production (approximately x Mtonnes) can be estimated with ±15% uncertainty 90% of the time when drill holes are spaced on a nominal 250 ft grid pattern.

Indicated Mineral Resources

Mineral Resources in the Indicated category are estimated from multiple drillholes located on a nominal 500 ft square grid pattern.

Inferred Mineral Resources

Mineral Resources in the Inferred category include model blocks that do not meet the criteria for Indicated category material but are within a maximum distance of 750 ft from a drillhole.

The spacing distances are intended to define contiguous volumes, and they should allow for some irregularities due to actual drillhole placement. The final classification volume results typically must be smoothed manually to come to a coherent classification scheme.

To further ensure confidence and continuity, the blocks were displayed at the chosen thresholds of approximately 250 ft and 500 ft to the nearest composite, and boundaries were digitized to create a smooth surface and to reduce the “spotted dog” effect. Solids were then created and coded back into the model by majority code, and using greater than 50% partials to be classified as measured or indicated. The remainder that is greater than 500 ft, but not more than 750 ft from nearest composite, was classified as Inferred. A snapshot of the classification distribution on 3095 Bench, as an example, is presented in Figure 14-12 showning Measured (red), Indicated (green) and Inferred (blue) resources. For reference, Figure 14-12 also outlines the the PV3-2016-PFS Mineral Resource (blue) and Mineral Reserve (yellow) extents on 3095 Bench.

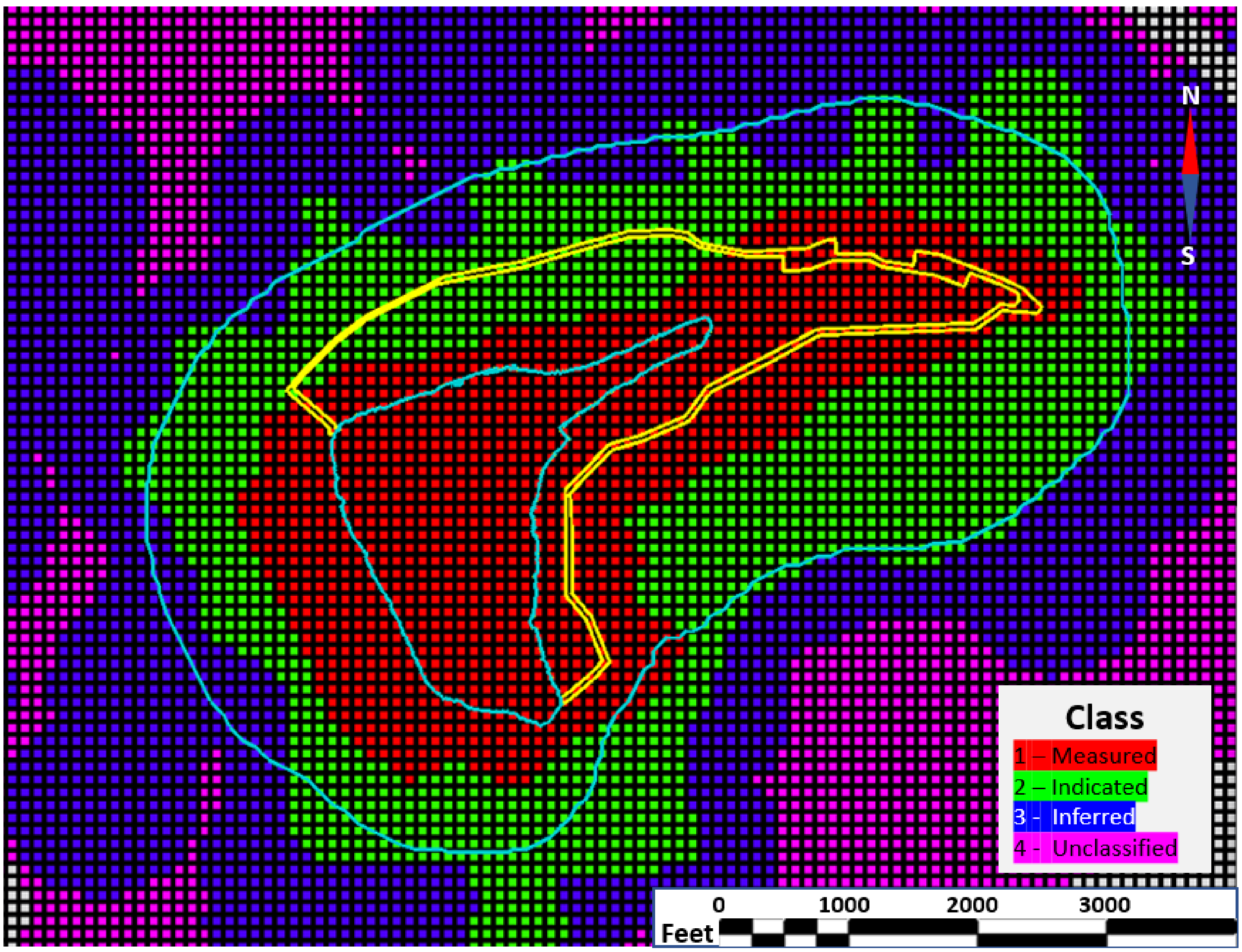


Figure 14-12: Classification Distribution on the 3095 Bench (Source: Kirkham, 2021)

14.11 Model Validation

A graphical validation was done on the block model. The purposes of the graphical validation are as follows:

- Visually check the reasonableness of the estimated grades based on the estimation plan and the nearby composites.

- Compare the general drift and the local grade trends of the block model to the drift and local grade trends of the composites.
- Ensure that all required blocks are estimated.
- Check that, within the model blocks, the topography has been properly accounted for.
- Check the manual order-of-magnitude estimates for tonnage to determine reasonableness.
- Inspect and clarify, when necessary, the high-grade blocks created as a result of outliers.

A full set of cross sections and plans were used to visually check the block model, showing the block grades and the composite. There was no evidence that any blocks were wrongly estimated. It appears that every block grade can be explained as a function of the following:

- Surrounding composites
- Correlogram models used
- Estimation plan applied

These validation techniques include the following:

- A visual inspection done on a section-by-section and plan-by-plan basis.
- Inspection of grade tonnage curves.
- Histograms of varying cut-off grades that demonstrate a relatively uniform, normal distribution.
- Swath plots (drift analysis) that compare the ordinary kriged blocks with the inverse distance and nearest neighbor estimates.
- Inspection of histograms to determine the distance of the first composite to the nearest block and the average distance to blocks for all composites used.
- Model checks for change of support.
- Detailed reconciliations with blasthole and production data.

14.12 Mineral Resource

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) define a Mineral Resource as follows:

> *[A] concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics, and continuity of a mineral*

resource are known, estimated or interpreted from specific geological evidence and knowledge.

The Measured and Indicated Mineral Resources at PVM are inclusive of the Mineral Resource converted to a Mineral Reserve using modifying factors, including, but not limited to mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

The Inferred Mineral Resource was not considered for conversion to a Mineral Reserve. Inferred Mineral Resources are estimated using limited geological evidence compared to Measured and Indicated Resources; this evidence is adequate to imply but not verify sufficient continuity of grade or geology.

The “reasonable prospects for eventual economic extraction” requirement generally implies that quantity and grade estimates meet certain economic thresholds and that Mineral Resources are reported at an appropriate cut-off grade, taking into account extraction scenarios and processing recovery.

The “reasonable prospects for eventual economic extraction” were tested using floating cone pit shells based on reasonable economic assumptions. The floating cone pit shells are an optimization for theoretical economic breakeven pit wall locations. The economic assumptions are detailed in Table 14-14.

Table 14-14: ‘Reasonable Prospects of Eventual Economic Extraction’ Parameters

Parameter	Value
Metal Price	
Copper	$3.50/lb
Molybdenum	$10.00/lb
Recovery to concentrate	
Copper	84.6%
Molybdenum	8.9%
Site Costs	
Mining	Ore: $1.52/tonne Waste: $1.48/tonne
Haulage Cost Increment per Bench	+ $0.011/tonne ore per bench below 3995 el. +$0.015/tonne waste per bench below 4400 el.
Operations Support	$0.88/tonne
Processing	$4.67/tonne
G&A	$1.13/tonne
Pit Wall Slopes	
Granite OSA	45°
Granite Porphyry OSA	45°
All other rock types	Variable (refer to Table 15-1)

The pit optimization results are used solely for the purpose of testing the “reasonable prospects for eventual economic extraction” and do not represent an attempt to estimate Mineral

Reserves. The optimization results are used to assist with the preparation of a Mineral Resource statement and to select and appropriate reporting assumptions.

It is important to note that the mineral resources are reported below the most current topography at the effective date which is the March 31, 2021 topographic surface. The volume is further reduced by areas of known backfill within the pit. A solid was created between the March 31, 2021 topography and the “reasonable prospects” pit. The volumes were reported as a percentage of the partial blocks as opposed to the less accurate method of whole block reporting. Figure 14-14 shows a plan view of the Cu% grades for blocks located within the 2021 “reasonable prospects” pit.

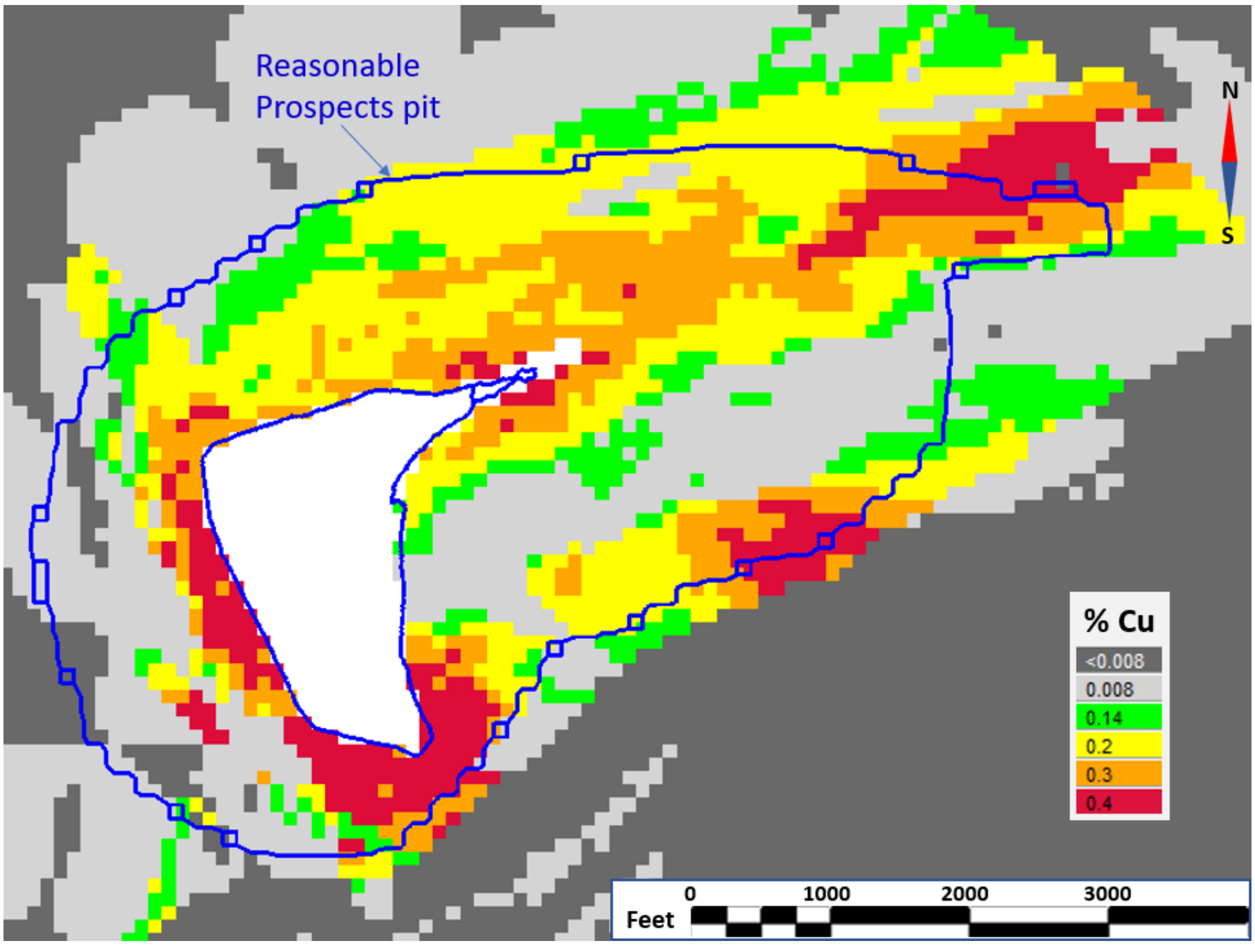


Figure 14-13: Plan View of Block Model at Elevation 3050’ with Cu % Grades (source: Kirkham, 2021)

The Mineral Resources listed in Table 14-14 are for Cu% and Mo% at a base-case cut-off grade of 0.14% Cu.

The Mineral Resources were estimated under the supervision of Kirkham Geosystems Ltd. and are effective March 31, 2021. The estimate includes results from drill programs conducted through 2019 and a revised geological model, revised grade shell model and adjustments to the estimation strategy based on five years of production reconciliation and geological and geochemical data analysis.

Table 14-15: Mineral Resource at 0.14% Cu Cut-off Grade, as of March 31, 2021

Classification	Tonnes (M Tonnes)	Cu%	Mo%	Contained Copper (M lb)	Contained Molybdenum (M lb)
Measured (M)	619.9	0.33	0.006	4,442.7	83.4
Indicated (I)	782.5	0.26	0.005	4,493.6	88.0
Total M&I	**1,402.3**	**0.29**	**0.006**	**8,934.6**	**170.0**
Inferred	170.6	0.26	0.006	967.6	20.7

The Mineral Resource is classified according to CIM (2014) definitions, estimated following CIM (2019) guidelines and has an effective date of March 31, 2021. The Independent Qualified Person for the estimates is Mr. Garth D. Kirkham, P.Geo., FGC., of Kirkham Geosystems Ltd. The economic assumptions include the following: $3.50/lb Cu, $10.00/lb Mo, 84.6% average Cu recovery, 8.9% average Mo recovery, $1.74/tonne average mining costs, $1.13/tonne G&A costs, $0.88/tonne operational support costs, $4.67/ton milling costs, and pit slopes by rock type. The Mineral Resource is reported inclusive of the Mineral Reserve. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The last date for drilling data and mining activities is March 31, 2021. Rounding may result in apparent summation differences between tonnes, grade and contained metal.

Mineral Resources are not Mineral Reserves until they have demonstrated economic viability. Mineral Resource estimates do not account for a resource's mineability, selectivity, mining loss, or dilution.

These estimates include Inferred Mineral Resources that are normally considered too geologically speculative for the application of economic considerations; therefore, they are unable to be classified as Mineral Reserves. 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.

14.13 Comparison of January 1, 2016 and March 31, 2021 Resource Estimates

A comparison of the January 1, 2016 Mineral Resource estimate (Capstone, 2016) and the March 31, 2021 Mineral Resource shows a 1% decrease in the Measured and Indicated Resource tonnage (3% decrease in contained copper, 24% decrease in contained molybdenum) and 35% increase in the Inferred resource tonnage. Depletion for mining activities resulted in the most significant decrease, offset by the addition of tonnes in the Measured and Indicated categories with infill drilling and revised density values based on measurements from drillcore (increased tonnage by 4%). The Inferred resource increased primarily because cut-off grade was lowered to 0.14% copper in the Mineral Resource presented in this Technical Report compared to the 0.17% copper cut-off grade in PV3-2016-PFS. Long-term economic and operating assumptions for this Mineral Resource include the projection of higher copper commodity prices (possibly 6% higher than those used in the 2016 model: $3.50/lb vs. $3.30/lb), with copper recovery estimated at 5% lower (85.2% vs. 88%) and operating cost assumptions estimated to be 5% higher ($8.42/tonne vs. $8.00/tonne).

Table 14-16: Comparison of Mineral Resources at 0.14% Cu Cut-off Grade as at March 31, 2021 and at 0.17% Cu Cut-off Grade as at January 1, 2016

Classification	Tonnage (M Tonnes)	Cu%	Mo%	Contained Copper (M lb)	Contained Molybdenum (M lb)
March 31, 2021 Mineral Resource[1]					
Measured (M)	619.9	0.33	0.006	4,442.7	83.4
Indicated (I)	782.5	0.26	0.005	4,493.6	88.0
Total M&I	**1,402.30**	**0.29**	**0.006**	**8,934.6**	**170.0**
Inferred	170.6	0.26	0.006	967.6	20.7
January 1, 2016 Mineral Resource[2]					
Measured (M)	647.9	0.34	0.008	4,843.4	118.6
Indicated (I)	772.3	0.26	0.006	4,387.8	105.6
Total M&I	1,420.2	0.30	0.007	9,231.5	224.1
Inferred	126.0	0.25	0.005	686.7	13.9
Difference					
Total M&I	-1%	-3%	-14%	-3%	-24%
Inferred	35%	4%	20%	41%	49%

1. The Mineral Resource is classified according to CIM (2014) definitions, estimated following CIM (2019) guidelines and has an effective date of March 31, 2021. The Independent Qualified Person for the estimates is Mr. Garth D. Kirkham, P.Geo., FGC., of Kirkham Geosystems Ltd. The economic assumptions include the following: $3.50/lb Cu, $10.00/lb Mo, 84.6% average Cu recovery, 8.9% average Mo recovery, $1.74/tonne average mining costs, $1.13/tonne G&A costs, $0.88/tonne operational support costs, $4.67/ton milling costs, and pit slopes by rock type. The Mineral Resource is reported inclusive of the Mineral Reserve. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The last date for drilling data and mining activities is March 31, 2021. Rounding may result in apparent summation differences between tonnes, grade and contained metal.
2. The Mineral Resource is classified according to CIM (2014) definitions, estimated following CIM (2003) guidelines and has an effective date of January 1, 2016. The Independent Qualified Person for the estimates is Mr. Garth D. Kirkham, P.Geo., FGC., of Kirkham Geosystems Ltd. The economic assumptions include the following: $3.30/lb Cu, $10.00/lb Mo, 88% average Cu recovery, 50% average Mo recovery, $1.50/tonne average mining costs + operational support costs, $1.50/tonne G&A costs, $5.00/ton milling costs, and a pit slopes of 45°. The Mineral Resource is reported inclusive of the Mineral Reserve. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The last date for drilling data and mining activities is December 31, 2015. Rounding may result in apparent summation differences between tonnes, grade and contained metal.The 2016 Mineral Resource is described in full in the Capstone’s Technical Report on the Pinto Valley Mine, Miami, Arizona, published February 23, 2016.
3. These estimates include Inferred Mineral Resources that are normally considered too geologically speculative for the application of economic considerations; therefore, they are unable to be classified as Mineral Reserves. 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.

14.14 Model Risks

Mineral resource models represent a best estimate because there are inherent risks related to the type of input data used, the interpretation of data and geology, subjective confidence levels and the lack of continuity for Acid Soluble Copper (ASCu) data. Every effort is taken to mitigate these potential risks however, the following should be noted in the case of the mineral resource estimate reported herein.

There remains some uncertainly related to the interface between oxides and sulfides as the data, specifically the historical data, has limited ASCu sampling results. Modelling has been performed with the existing dataset and is used as a guide to locate the interface. Estimation risks are limited within the core of the deposit where most of the oxide has been mined out however, there is some uncertainty within the pit walls when pushing back.

By definition, the Inferred resources for any deposit are relatively uncertain. Additional drilling will be required to more accurately characterize the grades when Inferred resources are encountered during mining.

14.15 Opportunities

The following opportunities are available with respect to the Mineral Resource estimate:

- Upgrade the classification of a substantial portion of the current Mineral Resource, from Inferred to Indicated class by decreasing the drill hole spacing in future drill programs. (Garth Kirkham, P.Geo., FGC)
- Continue regional exploration and property evaluations within reasonable trucking distance to the plant. (Garth Kirkham, P.Geo., FGC)
- Evaluate steps required to include gold and silver in the Mineral Resource estimate, (Garth Kirkham, P.Geo., FGC)

14.16 Recommendations

There are no recommendations related to Mineral Resources.

15 Mineral Reserve Estimate

The Mineral Reserve was developed by Capstone and is the total of all Proven and Probable category material planned for processing in the LOMP. Development of the final open pit design and Mineral Reserve was based on the best economic limit, subject to the design tailings storage capacity of TSF4. There were no limits or constraints incorporated from the current operating permit at Pinto Valley. The qualified person for the Mineral Reserve is Clay Craig of Capstone Mining Corp.

The Mineral Reserve at PVM is based on industry standard mine planning practices, as are applied at similar open pit mines. The Lerchs-Grossmann algorithm combined with practical phase designs and trials of alternative mine production schedules were used to set the LOMP and production schedule. The Mineral Reserve is the total of all material planned for processing within the mine production schedule.

Early in the development of this LOMP, a series of preliminary phases or push backs were designed that initially targeted a floating cone that was based on a copper price of $3.00/lb. Additional expansions beyond that price were also developed using metal prices as high as $3.50/lb copper.

Mine schedules were developed that progressively added pushbacks to the mine life. The impact of each additional pushback was evaluated economically.

When complete, the schedule analysis did not support the production of all material that was within the $3.00/lb floating cone. Two pushbacks (east and north walls) were proven to be economic. The west and southwest pushbacks did not contribute to the overall project economics on a time value of money basis because of their high strip ratios and the time required to release the ore at depth. The east and north walls of the floating $3.00 cone were used as the guide to develop the final PV3 pushback sequence for this study.

15.1 Lerchs-Grossmann Cone Optimization

The Lerchs-Grossmann (LG) computer algorithm is a tool that provides guidance to mine design. The algorithm applies approximate costs and recoveries, and approximate pit slope angles, to establish theoretical economic breakeven pit wall locations. Economic benefit was applied to measured and indicated (proven and probable) ore only for this study. Inferred category mineralization is considered to be waste in the project evaluation.

The economic input applied to the cone algorithm is necessarily preliminary, as it is one of the first steps in the development of the mine plan. The cone geometries should be considered as approximate, as they do not assure access or working room. The important result of the cones is the relative change in geometry between cones of increasing metal prices. Lower metal prices result in smaller pits, which provide guidance to the design of the initial pushbacks. The change in pit geometry as metal prices are increased indicates the best directions for the succeeding phase expansions to the ultimate pit.

Table 15-1 summarizes the input data to the LG optimization. Process recoveries and estimated process costs were developed from actual production data from 2016 through 2020. Slope angles were provided by Edward C. Wellman of Independent Geomechanics LLC, PVM's geotechnical consultant since 2006. Mine operating costs were based on actual costs from 2016 through 2020. LG optimization required overall slope angles for input. The overall slope angle (OSA) that was used in the optimization to account for the impact of haul roads are shown in Table 15-1.

Multiple LG optimizations were completed at a range of metal prices. The base case metal prices for design were set at $3.00/lb copper price and $10.00/lb molybdenum. Molybdenum prices for the multiple cones were set at the same price due to their negligible impact on optimum pit location.

The costs and recoveries result in the following simplified cut-off grades if the contribution of molybdenum is not included in the cut-off calculation.

- Internal or Marginal Cut-off = 0.16% Total Copper
- Breakeven Cut-off = 0.205% Total Copper

Cones were optimized by assigning ore above a cut-off of 0.17% Cu. This figure, falling between the internal and breakeven cut-offs, reflected the intent to elevate the cut-off to maximize project value given limits on annual mill throughput and total tailings capacity.

Figure 15-1 illustrates the LG optimization that was produced at $3.00/lb copper. Figure 15-2 illustrates the final pit design that was used to develop the Mineral Reserve. Comparing the two indicates that the west and southwest areas that were economic in the floating cone but not economic in the phase by phase analysis were not included in the final pit design.

Table 15-1: Base Case Floating Cone Input – Pinto Valley Project

Cone Input Item	Cost or Recovery
Mining Cost Inputs to Cones	
Direct Mine Operating Cost	$1.52/tonne Ore, $1.48/tonne Waste
Mine Sustaining CAPEX	$0.25/tonne
Haulage Cost Increment per Bench	$0.011/tonne /bench depth above/below 3995 bench for Ore, $0.015/tonne /bench below 4400 bench for Waste
Average Total Mining Cost With Haul Increment & Sustaining CAPEX	$1.93/tonne
Bench Discounting	1.3%/bench of depth below the 4400 bench
Process Costs for Mill Operation	
General and Administrative Cost	$1.13/tonne
Milling Cost	$4.67/tonne
Operations Support Cost	$0.88/tonne
Flotation Process Recovery - Copper[1]	
All Except Diabase	Recovery (%) = 0.7768 + 26.063 * (Cu%) / 100
Diabase	Recovery (%) = 0.5968 + 26.063 * (Cu%) / 100
Molybdenum Recovery	Recovery = [-1.149 + 1116.28 * (Mo%)/100 + 1.8088*(Cu Recovery%)/100] * 0.2
Concentrate Transport Costs	$140.40/wet tonne Cu con based upon 80% int'l & 20% domestic
Treatment and Refining Costs for Copper	

Moisture Content of Copper Concentrate	9%
Copper Concentrate Grade	25.0%
Copper Smelting Recovery	96.5% subject to a minimum deduction of 1 unit
Copper Smelting Cost	$80/dry tonne
Copper Refining Cost	$0.080 (refining cost/lb recovered Cu)
Total Cost /lb recovered Cu	$0.5228 (SMRF cost/lb recovered Cu – Au & Ag credits ignored during cone optimization)
Treatment and Refining Costs for Molybdenum	
Molybdenum Concentrate Grade	Minimum 47.00%
Net Transport and Roasting Cost for Molybdenum	$1.00 (cost/lb recovered Mo)
Metal Prices for Base Case	
Copper Price	$3.00/lb Cu
Molybdenum Price	$10.00/lb Mo

1. From May 10, 2021 Report by J. Todd Harvey

Slope Angles, Independent Geomechanics[2] Recommendations	Floating Cone Slope (IRA Angles/OSA Angles)
Pinal Schist	27°/27° OSA Used for Floating Cone
Conglomerates and Paleogene Volcanics	35°
West wall Diabase/Limestone	45°
East wall Diabase/Limestone	46°
Ruin Granite & Granite Porphyry All walls	48°/45°
Diabase in Gold Gulch/West End Shear Zone	32°
Granodiorite in Gold Gulch/West End Shear Zone	28°
Granodiorite outside of Shear Zone	40°
Tailings	27°
Waste Rock, All	32°

2. Slope Angle Modifications have been reviewed at least annually by QP Edward C. Wellman, Independent Geomechanics LLC.

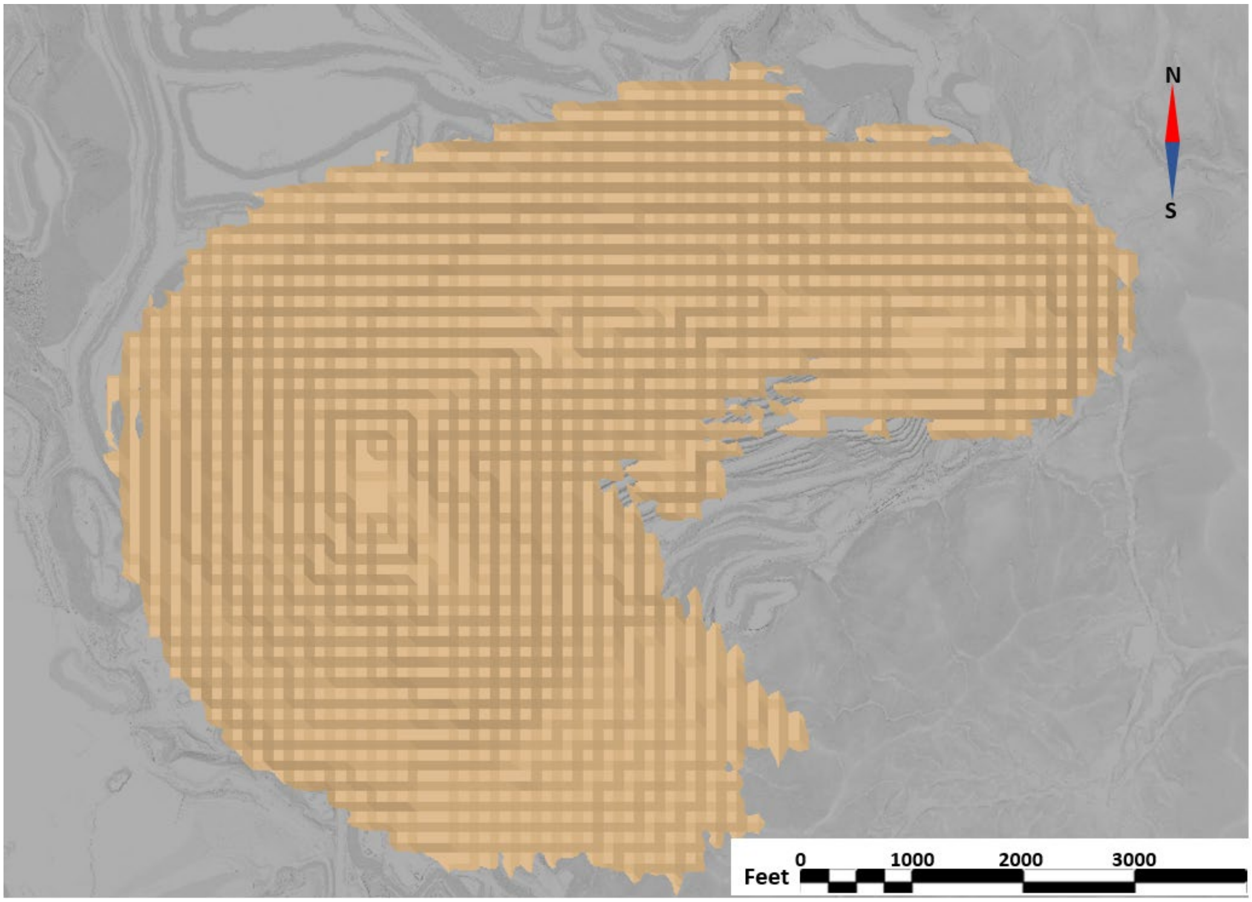


Figure 15-1: Plan view of Lerchs-Grossmann Optimization at $3.00/lb Copper (tan), used as Guidance for Phase Design.

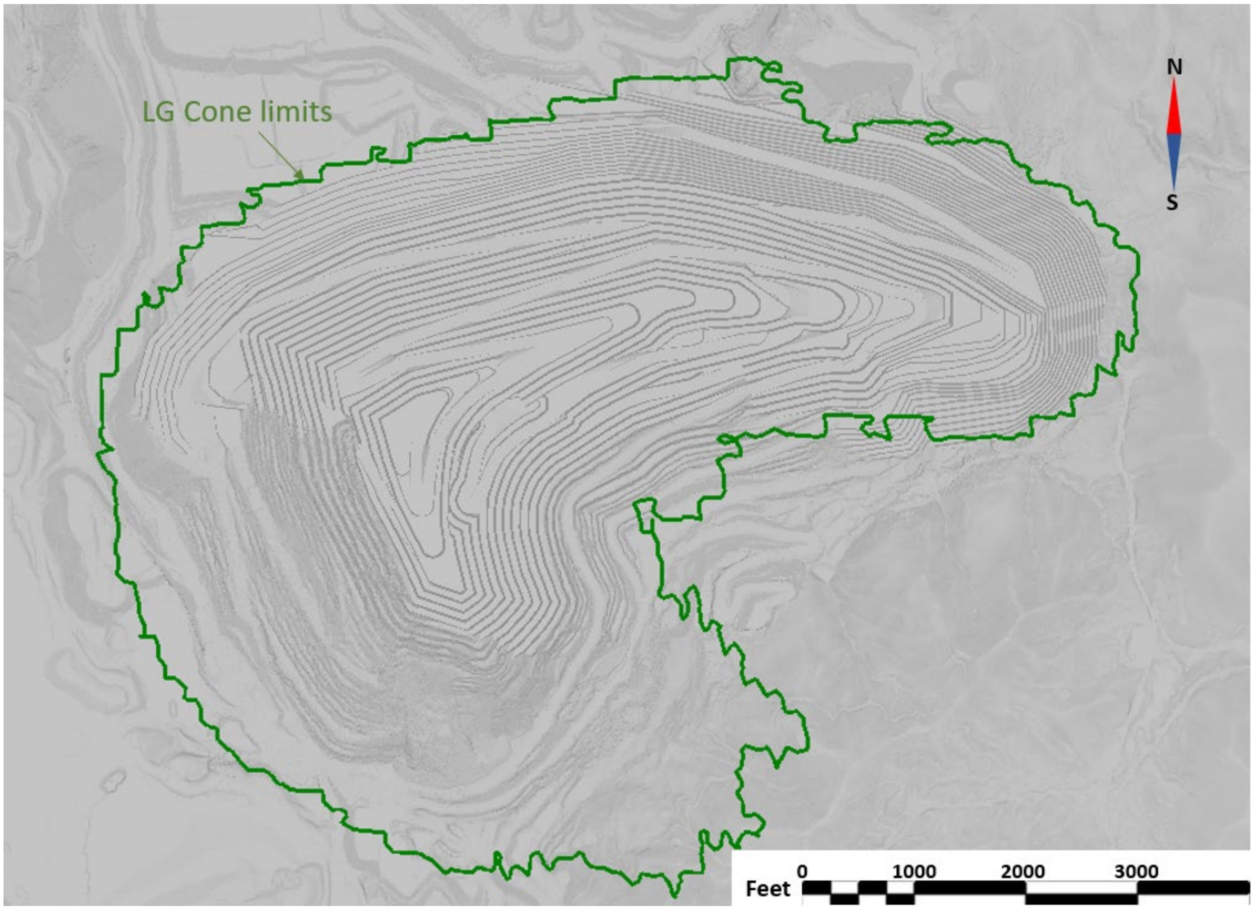


Figure 15-2: Plan view of Mineral Reserves Pit Design compared to limit of Lerchs-Grossmann at $3.00/lb Copper

Criteria influencing the Mineral Reserves pit design are described further in Section 16. Given the tailings limitations envisioned in this LOMP, the phase sequence was designed generally to the Northern and Northeastern limits of the LG optimization. Volumes to the Southwest were excluded from the Reserves pit design due to the presence of a landfill on the west side of the pit. Volumes on the South were excluded from the Reserves pit design due to the increased study requirements of excavating large volumes of Pinal Schist.

To optimize the impacts of challenging access ramps, phase designs often incorporate the mining of the lowest 2-4 segments of redundant access ramps, thus improving ore release in underlying benches.

15.2 Statement of Mineral Reserve

The Mineral Reserve was developed by tabulating the contained measured and indicated (proven and probable) material inside of the designed pit at the mill cut-off grades. The Mineral Reserve is based on the PVM schedule presented in Table 16-3. The schedule adjusts cut-off grade over time to improve the mine's net present value (NPV). Initial cut-off grades are lower to overcome the stripping hurdle in phase 3A, then increase to maximize copper production within the TSF4 tailings storage capacity.

PVM has operated a run-of-mine (ROM) dump leach on low-grade sulfide material for many years. This study assumes that leaching will cease after 2021 due to waste rock storage requirements. ROM dump leach is not included in the floating cone analysis or tabulated in the Statement of Mineral Reserve. Material that is incurred at ROM leach grade is treated as waste.

Table 15-2 summarizes the Mineral Reserve at PVM remaining after March 31, 2021.

The Mineral Reserve and the LOMP are based on a variable density informed by 305 measurements (see Section 14).

The qualified person for the estimation of the Mineral Reserve is Clay Craig of Capstone Mining. Mr. Craig has reviewed in detail the Pinto Valley block model that was constructed under the direct supervision of Mr. Kirkham.

Table 15-2: Mineral Reserves at a variable cut-off of 0.17-0.21%, as of March 31, 2021

Classification	Tonnes (M Tonnes)	Cu %	Mo %	Contained Copper (M lb)	Contained Molybdenum (M lb)
Proven	241.6	0.34	0.007	1,833	35.6
Probable	139.4	0.28	0.006	877	17.4
Proven and Probable	**381.0**	**0.32**	**0.006**	**2,710**	**53.0**

The Mineral Reserve has an effective date of March 31, 2021 and was prepared by Clay Craig, P.Eng., Manager, Mining and Evaluations at Capstone Mining Corp. The economic assumptions include the following: $3.00/lb Cu, $10.00/lb Mo, 86.0% average Cu recovery, 8.5% average Mo recovery, $1.68/tonne average mining costs, $1.13/tonne G&A costs, $0.88/tonne Ops Support costs, $4.67/ton milling costs, and pit slopes by rock type. The Mineral Reserve is reported at a variable cut-off ranging from 0.17% to 0.21% copper. Tonnage measurements are in metric units. Copper and molybdenum grades are reported as percentages. Contained metal is reported as million pounds. Rounding may result in apparent summation differences between tonnes, grade and contained metal.

15.3 Risks

As reserve models are an estimate based on certain assumptions and interpretations, they have certain inherent risks. Risks to the PVM Mineral Reserve as outlined in this report include, but may not be limited to:

- Changes to the resource model, potentially resulting from revised interpretation and/or the results of additional drilling and sampling.
- Changes to financial assumptions, including metal pricing.
- Significant changes to land tenure or the permitting requirements, including anticipated timelines for renewals of permits currently in place.
- Technical challenges such as water supply shortages or geotechnical stability of the open pit or tailings storage facilities.

15.4 Opportunities

Opportunities to potentially expand and increase the value of the PVM Mineral Reserve include:

- Evaluation of options for increased tailings storage capacity. This would be a key step towards converting a large tonnage of potential ore on the Southwest side of the deposit into Reserves.
- Assaying of gold and silver values in historical and future exploration drillholes, with the intention of assessing whether confidence in these values can be brought to levels sufficient for inclusion in Mineral Reserves.
- Cost-benefit analysis of transporting the western landfill to assess mining of high grade material in that area.

15.5 Recommendations

To support the PVM Mineral Reserve, a geotechnical evaluation of key structures such as the Bummer fault during mining of ‘internal’ pit phases 3B and western 3A, and application of resultant observations to adjust final 3C pit design. Estimated cost is $100,000, with duration of one month for each of 2-3 key structures.

16 Mining Methods

PVM is an open-pit hard-rock mine, producing copper bearing sulfide ore to a conventional milling and flotation concentrator. Conventional open-pit mining utilizes the cycle of drilling, blasting, loading, and hauling of material to the respective destinations. Ore is hauled to the primary crusher for processing and waste rock material is hauled to waste storage facilities. Mining is accomplished on 45 ft benches and the LOMP is reported in metric tonnes. The qualified person for this section is Clay Craig of Capstone Mining Corp.

This Technical Report incorporates a mill throughput of 56,000 tonnes per day (20,440 ktonnes/yr) from 2021 through 2039. The mine production schedule was developed with the goal of maintaining mill feed and maximizing the mine's net present value (NPV).

The LOMP schedules movement of an average of 144,121 tpd (52,505 ktonnes/yr) of total material from 2021 to 2031. Beginning in 2032 the waste mined begins to fall, and the total material movement reduces to slightly more than the mill ore feed rate.

The LOMP presented in this section was developed by PVM under the supervision of Mr. Craig, and is based upon the 2021 block model of the deposit that was developed under the supervision of Mr. Kirkham. The model blocks are 100 ft by 100 ft on plan with 45 ft bench heights.

16.1 Phase Design

PVM has been operated intermittently since the 1970s. There are areas of the existing PVM pit that have had slope stability issues over time. Specific areas are the southwest corner of the pit in the Pinal Schist and the northeast side of the pit near the Bummer Fault. Designof phase expansions in these areas must consider the geotechnical impacts and practical constraints they impose. Phase designs developed for this Technical Report do not mine any of the Pinal Schist. Updated stability analysis by Edward C. Wellman of Independent Geomechanics, LLC. (IG) indicates that the Bummer Fault is unlikely to have a significant adverse impact to the LOMP. Previous instability in this area was caused by a lack of adequate pit slope dewatering and poor blasting techniques.

Areas to the east and north of the current PVM pit will be expanded. The phase that pushes back the northern wall will mine through some historic waste and leach dump material before encountering solid rock. That material is mined at a shallower angle (32 degrees) than the solid rock below.

The phase designs were partially guided by the results of the floating cones that were summarized in Section 15. However, far more important in the design of the mine phases are the challenges of working with the existing access pattern, minimum/practical mining widths, and the geotechnical constraints on the design.

The current PVM operation mines two phases that incorporate the current pit bottom as well as east and north expansions. PVM has also designed three pushbacks that expand the pit beyond 2027 that further develop the pit to the east and north.

In total, there are five phase designs that were used as input to the development of the PVM schedule. The mining phases are a combination of work completed by PVM staff in the long range, short range, geotechnical, and operation groups. The phase designs in order of extraction in the current schedule are: PV2C (Castle Dome), PV2B (Jewel Hill), PV3A, PV3B and PV3C.

Castle Dome phase is the current primary ore pushback on the south side of the PVM pit and will continue through 2025. Jewel Hill is an eastern pushback that was designed to continue the operation through 2027. The Jewel Hill pushback design has been continually optimized to account for geotechnical issues as well as accommodating operational challenges.

The location of the phase designs Jewel Hill, Castle Dome, and PV3 are illustrated in Figure 16-1.

Inter-ramp slope angles for the phase design are summarized in Section 16.2. The overall and inter-ramp slopes were reviewed and recommended by slope stability consultant Edward C. Wellman of Independent Geomechanics, LLC. In addition to slope angles, the following road and pushback geometries complete the mine design parameters:

- Haul Road Width: 125 ft and 115 ft
- Haul Road Grade: 10% maximum
- Minimum Widths Between Pushbacks: 300 ft nominal

The tonnage and grade at multiple cut-off grades were tabulated from the designed phases on a bench- by-bench basis. Those tabulations were used as input to the development of the mine production schedule.

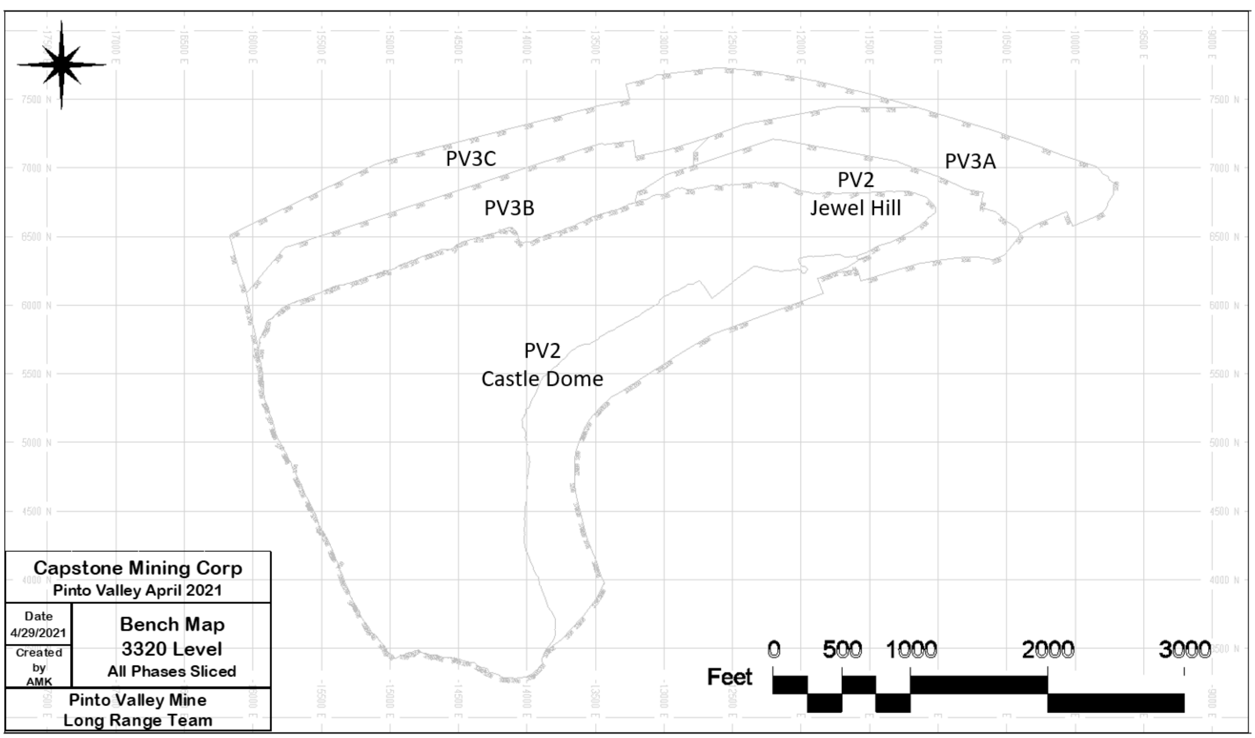


Figure 16-1: Location of Phase Designs on the 3860 Bench

16.2 Slope Stability

This section describes the geotechnical inputs to the mine design. These consist of the rock mass characterization used to develop the slope angle recommendations for the mine design.

16.2.1 Rock Mass Characterization

The rock mass for the PVM pit has been domained primarily as a function of lithology. Open-pit mapping and several geotechnical drilling programs have been completed to inform the current study. These studies were completed by PVM and its geotechnical consultants, including Call & Nicholas (2013; 2014) and SRK (2015) for the PV3 study.

Rock mass characterization has been completed by mapping and drill core using several geotechnical indices, including Rock Quality Designation (RQD), Rock Mass Rating (RMR), and the Geological Strength Index (GSI). In addition to laboratory strength testing of core samples, these classifications have been used to develop rock mass strengths used in stability analysis. Following is a summary of the rock mass characteristics of the major lithologies and geotechnical units used to develop the recommended slope angles for the PVM pit. Numerical values presented in ranges indicate the 30th and 70th percentile values for the parameters described.

Ruin Granite – The granitic intrusion is classified as a quartz-monzonite-porphyry based on mineral composition and texture. The mineralogy is mainly consistent. It is locally intruded by thin bodies of diabase and aplite / alaskite. This rock forms the primary ore host and includes a granite porphyry alteration type along its southern margin. The unit is slightly to moderately weathered along fractures. The rock is strong with typical hardness values ranging from R3 to R4. The Uniaxial Compressive Strength (UCS) ranges from 1,000 to 12,000 psi as a function of weathering and microfractures in the core samples. The UCS averages 8,200 psi. RQDs range from 55% to 85%. The rock mass rating system indicates fair to good rock ranging from RMR of 50 to 60.

Diabase – The Diabase is a massive equigranular fine-grained intrusive unit present in both the west wall and the PV3 east wall. The unit is typically R3 to R4 in hardness. UCS strength ranges primarily as a function of alteration and weathering. UCS values average around 9,500 psi but can typically range from 3,000 to 15,000 psi. RQDs range from 60% to 90%, and the rock mass rating indicates fair rock (RMR 50 to 55).

Limestone – The Devonian Martin Limestone, Mississippian Escabrosa Limestone, and Pennsylvanian Naco Limestone are present to the west of the pit and the east. These are massive and variably shaly and cherty limestone units. The limestone is a weak to average strength with typical hardness values ranging from R2 to R3. The UCS ranges from 3,000 to 9,000 psi. RQDs range from 10% to 90%, and the rock mass rating indicates poor to fair rock (RMR 45-50).

Weak rock mass - Rock units to the west of the Gold Gulch Fault are generally weak and behave as soil-like materials. These include the Basalt, Gila, and Whitetail conglomerate units and are assigned as one domain for slope design purposes. These units primarily consist of conglomerates with several different facies. These facies range from sandstone to siltstone beds, which were observed in the eastern section of the pit. A matrix-supported conglomerate and breccia are more typical on the west wall, with localized ash and weak layers within the units. Where intact samples could be collected, the intact strength of the weak rock mass is typically 1,000 to 2,000 psi. Rock hardness is typically R2 or less, with localized R3 zones. RQDs range from 0% to 40%, and RMR ranges from 20 to 30

Granodiorite - The unit is moderately to highly weathered and altered. Locally it may be weathered to a residual granular quartz material. The strength of the granodiorite is that of weak rock, with a typical hardness of R2 to R3. UCS values range from 1,000 to 5,000 psi with an average value of 2,500. RQDs range from 30% to 40%, and the rock mass rating indicates poor rock.

Pinal Schist - The schist is strongly foliated. Changes in foliation are observed and typically correspond to displaced units within the Schist Hill gravity slide and in-situ Pinal Schist. The Pinal Schist is completely weathered and decomposed into a residual soil between the Gold Gulch Fault and West End Fault. Locally this level of alteration also occurs in localized beds within the schist. The intact strength ranges from 500 to 3,000 psi. Locally it can be harder, but more typically is weaker than rock hardness R2, or less than 3,500 psi. RQDs range from 0% to 40%, and the rock mass rating indicates very poor to poor rock (RMR 0 to 30).

16.2.2 Open Pit Mine Slope Angle Recommendations

The stability of the pit slope design meets slope acceptance criteria with a minimum factor of safety (FOS) of 1.3. The slope angle recommendations assume a 45 ft bench height. In competent rock mass areas, including the Ruin Granite, a 90 ft high double bench height may be used based on the selected mining equipment. This method includes two 45 ft single benches stacked with a catch bench. Using a double bench configuration permits a steeper interramp slope angle (ISA) in competent ground.

The minimum catch bench width is developed using the modified Ritchie Criteria (Ryan and Pryor, 2001). Minimum bench widths are as follows: 24 ft widths for 45 ft bench height and 33 ft widths for 90 ft bench heights. A design catch bench width of 48 ft for 90 ft double benches was used in the current Technical Report study as listed in Table 16-1. Other parameters include a bench face angle of 70 degrees used to determine the maximum intrerramp angle of 48 degrees. Maximum angle values are for the Ruin Granite. Other geotechnical domains use the slope angles listed in Table 16-2.

Six stability cross-sections were analyzed for the project, and the minimum FOS exceeds 1.3, meeting industry-accepted standard guidelines for open-pit slope stability (Read and Stacey, 2009). The stability analysis was completed by PVM mine personnel and reviewed by the slope stability consultant Edward C. Wellman of Independent Geomechanics.

Table 16-1: Open-pit Slope Angle Criteria

Criteria	Value	Units
Bench Increment	45	ft
Bench Height	45-90	ft
Bench Face Angle (BFA)	70	degrees
Design Bench Width	33-48	ft
Minimum Bench Width	24-33	ft

Maximum Interramp Slope Angle (ISA)	48	degrees
Maximum Overall Slope Angle (OSA)	48	degrees
Minimum Factor of Safety (FOS)	1.3	

Source: Independent Geomechanics, 2021

The recommended slope angle in each domain of the pit is the flattest angle produced by the catch bench, inter-ramp, or overall slope analyses. Catch bench reliability for the double-bench area in the Ruin Granite will exceed 80% based on prior analysis. The recommended ISAs are summarized in Table 16-2.

Table 16-2: Interramp Slope Angles

Rock Type / Design Sector	Maximum Interramp Slope Angle	Notes and Comments
Ruin Granite Granite Porphyry Aplite	48°	Applies to all sectors and wall orientations, including single bench and double bench. Use single bench for Ruin Granite RQD <50 or weak rock mass. Granite Porphyry may use Ruin Granite angle, 48°, for slope heights less than 61m (200-ft). CB width minimum is 7.3m (24-ft), BFA adjusted to meet criteria.
Paleozoic Limestone and younger units	45° / 46°	Northwest Wall (45°) / Northeast Jewel Hill (46°) recommendations for Single Bench. CB width minimum is 7.3m (24-ft), BFA adjusted to meet criteria.
Apache Group Gila Conglomerate	35°	West Wall – Single Bench
Post-Mineralization Unit - Basalt, Gila, and Whitetail Conglomerate	35°	Single Bench
Post-Mineralization Unit - Whitetail Conglomerate on Jewel Hill Northeast	45°	Single Bench and slope height less than 121.90m (400-ft)
Granodiorite	40°	Overall Wall Stability Governs Angle
Diabase	45° / 46°	Northwest (45°) and Northeast (46°) wall recommendations for single bench. CB minimum is 7.3m (24-ft), BFA adjusted to meet criteria.
Pinal Schist	27°	Limit equilibrium slope angle. FOS~1.0. Expect deformation at angles of 23° from toe of slope. Single bench.
Waste Rock and Tailings	32° / 27°	Tailings material recommended angle of 27°
Diabase between Faults (Gold Gulch and West End)	32°	Overall Wall Stability Governs Angle
Granodiorite between Faults (Gold Gulch and West End)	28°	Overall Wall Stability Governs Angle

Source: Wellman, 2021

Mining of slopes in the Pinal Schist is not planned in the current design. For reference, slopes in the Pinal Schist have a history of displacement and have been assessed using limit equilibrium models to have a FOS of 1.0. PVM has safely maintained operational activities with displacing slopes using a range of mitigating controls based on observation, a trigger action response plan, and a slope monitoring program.

In addition, a Granite Porphyry unit was intersected in the southeast section of the pit that was originally interpreted as part of the Granodiorite but Granite Porphyry has recently been re-assessed for strength and the determination has been made that it behaves like Ruin Granite. Granite Porphyry has been observed to be high strength and has not presented operational challenges to date.

Slope angle recommendations listed in Table 16-2 are for depressurized slopes. PVM installs horizontal drains and employs other slope depressurization measures as part of normal mining operations.

16.3 Mine Production Schedule

PVM production is currently planned at 56,000 tpd for 2021. The mine production schedule was developed to release and deliver this quantity of ore to the mill while maximizing the mine's NPV.

The total material production rates in the mine were selected after the development of several alternative schedules that compared alternatives of mining equipment loading capacity.

The alternative schedules and equipment trade-off evaluations were completed with input from PVM engineering and operations departments to confirm the production capacity of the equipment being considered. The best economic schedule resulted using the current equipment fleet without adding any new or larger units.

Currently, PVM operates the following primary loading equipment:

- 2, Cat 994K front end loader equipped with 27.5 m^3 (36 yd^3) buckets.
- 2, Cat 994H front end loader equipped with 17.2 m^3 (22.5 yd^3) buckets.
- 1, Hitachi EX5600 hydraulic front shovel with 29.1 m^3 (38 yd^3) bucket

The high mobility and low diesel consumption of the two 994K Loaders will be critical to achieving the LOMP. PVM has established the following total material movement schedule based on the production capacities of the two 994K Front Loaders and assisted by one EX5600 hydraulic shovel as well as the 994H loaders.

An average of approximately 52,500 k tonnes/year is maintained until the release of ore from the final pushback in the year 2031 is secured and there is a minimal waste to be mined.

The LOMP and schedule presented in this Technical Report are a continuation and extension of the current operation. Road pioneering and bench development is scheduled to begin in 2021 for the development of the PV3A pushback.

Cut-off grades were established to maximize the mine's NPV for the selected equipment capacity and total material rate. The cut-off grade for the mine schedules are based on total copper cut-off grades. The mill feed cut-off grade changes over time. The cut-off grade is kept close to 0.17% Cu through 2028 due to high stripping requirements in these years. After 2028, the cut-off is increased as the mine schedule allows, in order to improve head grade and increase copper produced within the current tailings design limitations. PVM Engineers recognize that further cut-off grade optimization and increases in mined capacity could be realized if additional tailings capacity were to be secured.

The mine extraction and mill feed schedule is illustrated in Table 16-3. Mining to supply mill feed continues through 2039. Material below mill cut-off in any given period is sent as waste. Limited 'surge' stockpiling occurs to ensure the mill is continually fed. Long term stockpiling of the mill material has negative economic impacts due to weathering that reduces recovery. The mine extraction and mill feed schedule are based on the Proven and Probable Mineral Reserve only.

Figure 16-2: Life of Mine Plan – Mill Rate of 56,000 TPD (2021+)Figure 16-2 illustrates the mine schedule in graphic form, including the full year for 2021 that has not been depleted for Q1 production.

Table 16-3: Mine Extraction Plan (Mill Rate 56,000 TPD) + Mill Feed Schedule 2021 to 2039

Year	Cut-off Grade	Ore Mined to Mill			Waste	Total Mined	Contained Metal in Concentrate	
	% Cu	M Tonnes	% Cu	% Mo	M Tonnes	M Tonnes	M lb Copper	M lb Moly
2021[1]	0.17	20.4	0.33	0.006	28.8	49.2	129.8	0.25
2022	0.17	20.4	0.33	0.007	30.0	50.4	128.2	0.31
2023	0.18	20.4	0.33	0.006	30.1	50.6	128.6	0.27
2024	0.17	20.5	0.33	0.007	33.3	53.8	130.2	0.32
2025	0.20	20.4	0.33	0.007	34.4	54.8	127.4	0.30
2026	0.17	20.4	0.34	0.007	34.4	54.8	133.8	0.32
2027	0.17	20.4	0.38	0.009	33.8	54.2	150.7	0.42
2028	0.17	20.5	0.37	0.010	34.3	54.8	143.4	0.45
2029	0.19	20.4	0.33	0.008	34.3	54.7	127.3	0.33
2030	0.21	20.4	0.27	0.006	34.4	54.8	104.7	0.26
2031	0.21	20.4	0.33	0.007	31.1	51.6	126.4	0.30
2032	0.20	20.5	0.33	0.006	20.5	41.0	127.8	0.24
2033	0.21	20.4	0.29	0.006	20.2	40.6	112.1	0.24
2034	0.21	20.4	0.29	0.007	5.4	25.9	113.6	0.29
2035	0.21	20.4	0.31	0.007	3.0	23.5	117.9	0.29
2036	0.19	20.5	0.29	0.005	1.1	21.6	110.6	0.20
2037	0.21	20.4	0.31	0.004	1.4	21.8	119.8	0.18
2038	0.21	20.4	0.32	0.004	0.9	21.3	125.2	0.15
2039	0.21	20.4	0.31	0.004	2.4	22.7	120.4	0.15
Total	**0.19**	**388.5**	**0.32**	**0.006**	**413.5**	**802.1**	**2,378.0**	**5.3**

1. Twelve months of 2021 mine extraction is shown. Under this schedule, mine extraction for the first quarter of 2021 comprises 6,524 M Tonnes of ore at 0.31% Cu and 0.006% Mo with 7,201 M Tonnes waste totaling 13,725 M Tonnes mined with production of 38.0 M lb copper metal and 0.08 M lb molybdenum metal.

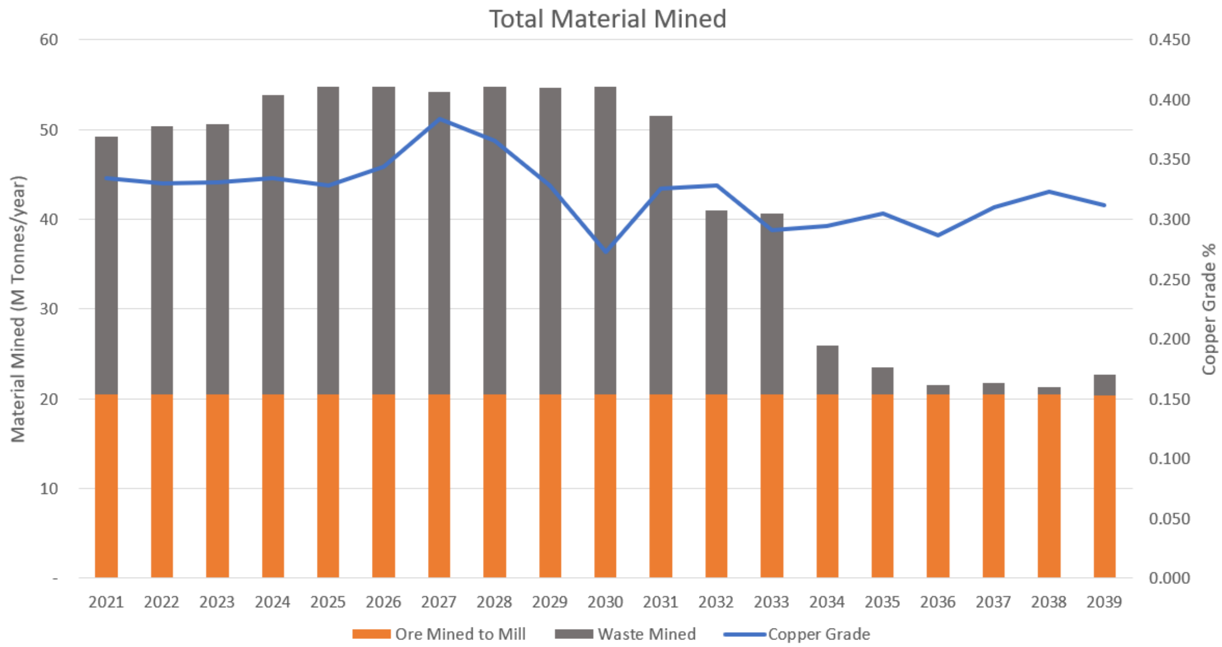


Figure 16-2: Life of Mine Plan – Mill Rate of 56,000 TPD (2021+)

16.4 Mine Material Storage

PVM has historically operated a dump leach operation in the Gold Gulch drainage that collects pregnant leach solution and produces copper cathode by SX/EW. PVM is evaluating options for the continued use of select leach areas, but the LOMP presented in this Technical Report assumes no significant future production of copper cathode from the leach operation.

During each year of the LOMP, waste material is generally allocated to the waste storage area based on the shortest haul time. Surge stockpiles for short term ore surpluses have been created around the crusher and western pit areas but are limited to 500 to 800 kt.

There are two dumps and one strategic limestone stockpile considered in the LOMP. The Main Dump encompasses the historic leach pads and generally also expands eastward. The strategic limestone stockpile lies to the east of the Main Dump. The West Dump is planned in the Gold Gulch area, scheduled to fill in the valley to the northwest of the pit and west of the Main Dump. That area is the current location of the pregnant solution collection pond from dump leaching, a part of the SX-EW process. Prior to 2023 the area will be available for waste rock storage. Starting in 2023, the LOMP begins mining from the north wall with the shortest haul being the West Dump in Gold Gulch. Construction for the West Dump has been assumed to utilize high dump heights (up to 700 ft). When the dump is at capacity, it will be dozed to 28.6 degrees.

The Main Dump is built in 45 ft vertical lifts. Dumps utilized slope angles of 36.0 degrees with 16 ft setbacks between lifts, for an average of 30.0 degrees in between access ramps. Overall slope angles accounting for access ramps typically range from 25.0 - 27.5 degrees.

A consistent density of 0.0591 tonne/ft^3 (swell factor of 1.25) has been used to calculate all waste rock and stockpile storage capacities. Figure 16-11 illustrates the ultimate (2039) configuration of PVM waste rock storage facilities.

16.5 End of Period Mine Drawings

End of period mine and dump drawings for the LOMP are presented in Figure 16-3 through Figure 16-10. These figures include the Mine Plan of Operations Boundary (MPO Boundary) that extends beyond the Private PVMC Property Boundary in places.

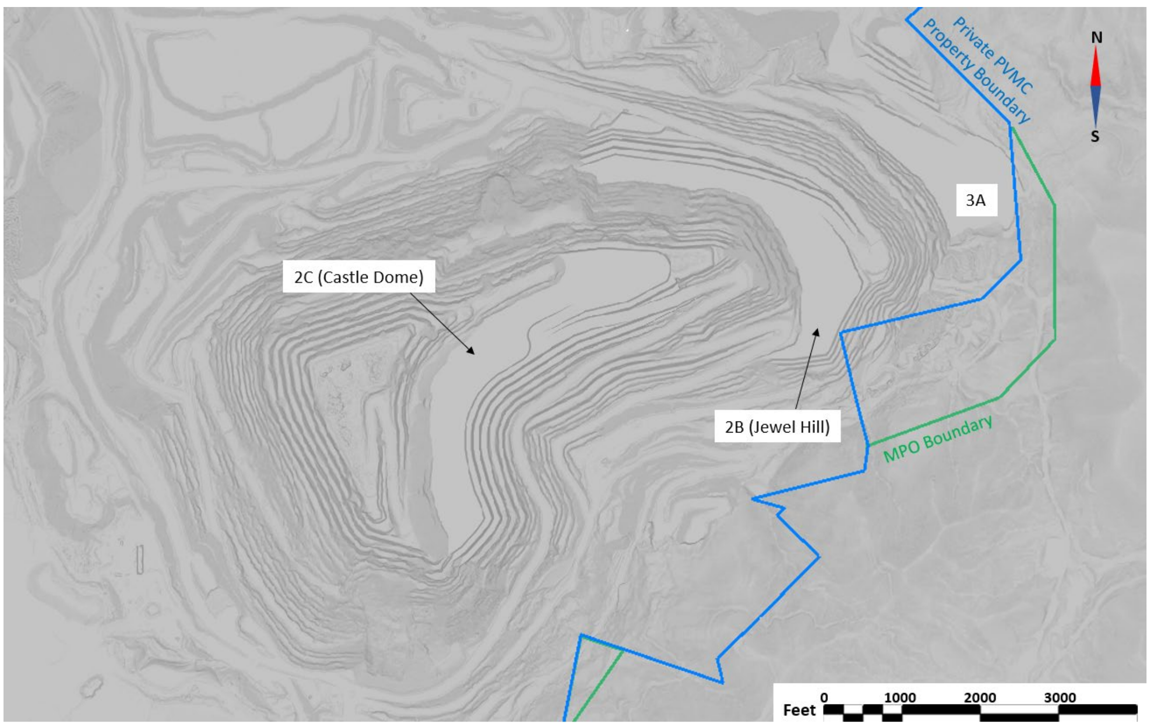


Figure 16-3: Planned Open Pit Position at end of 2021

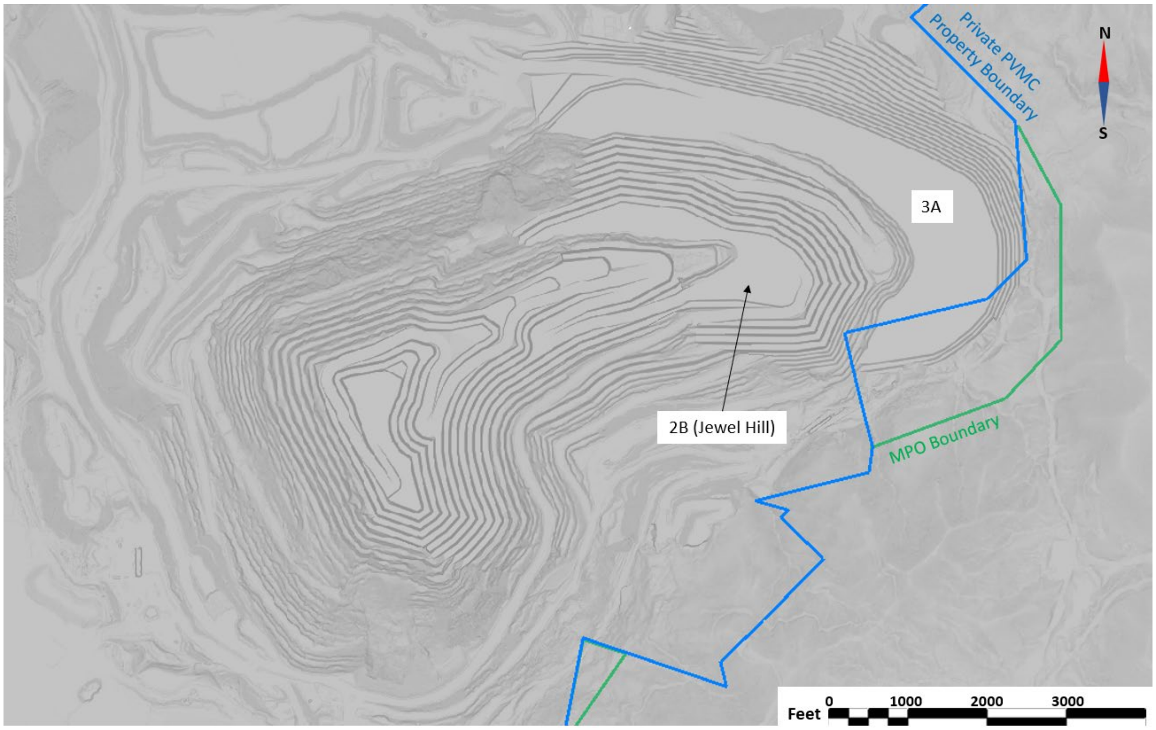


Figure 16-4: Planned Open Pit Position at end of 2025

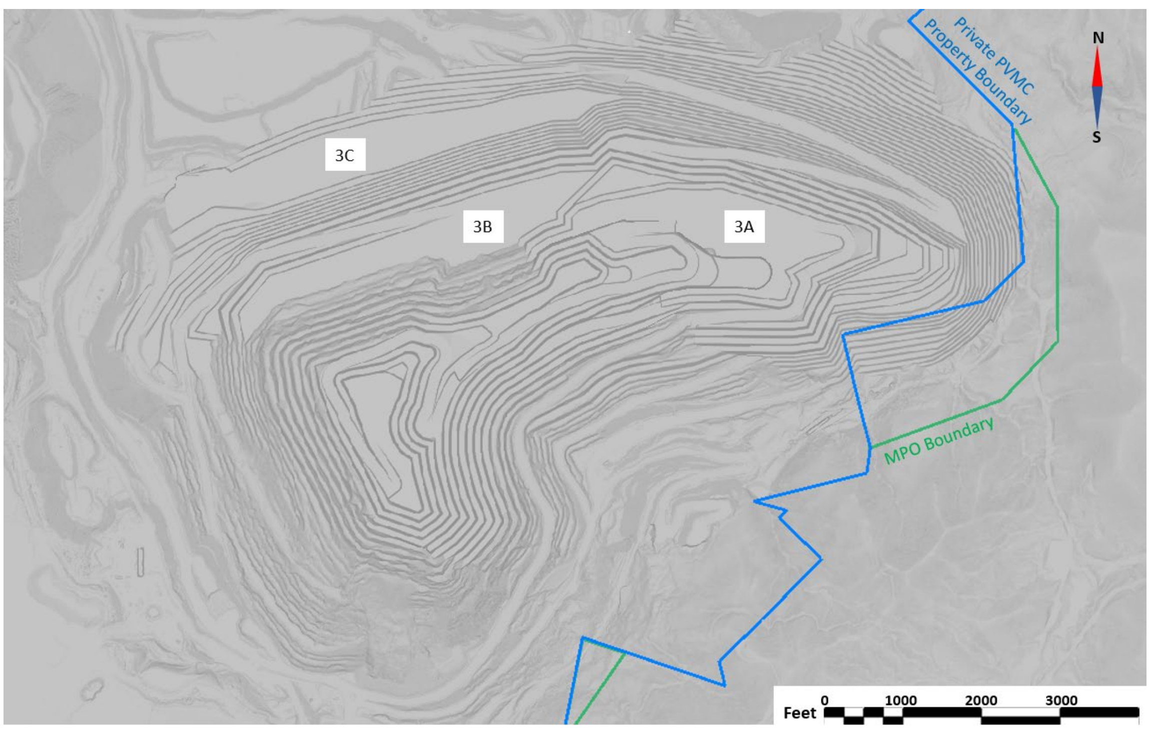


Figure 16-5: Planned Open Pit Position at end of 2030

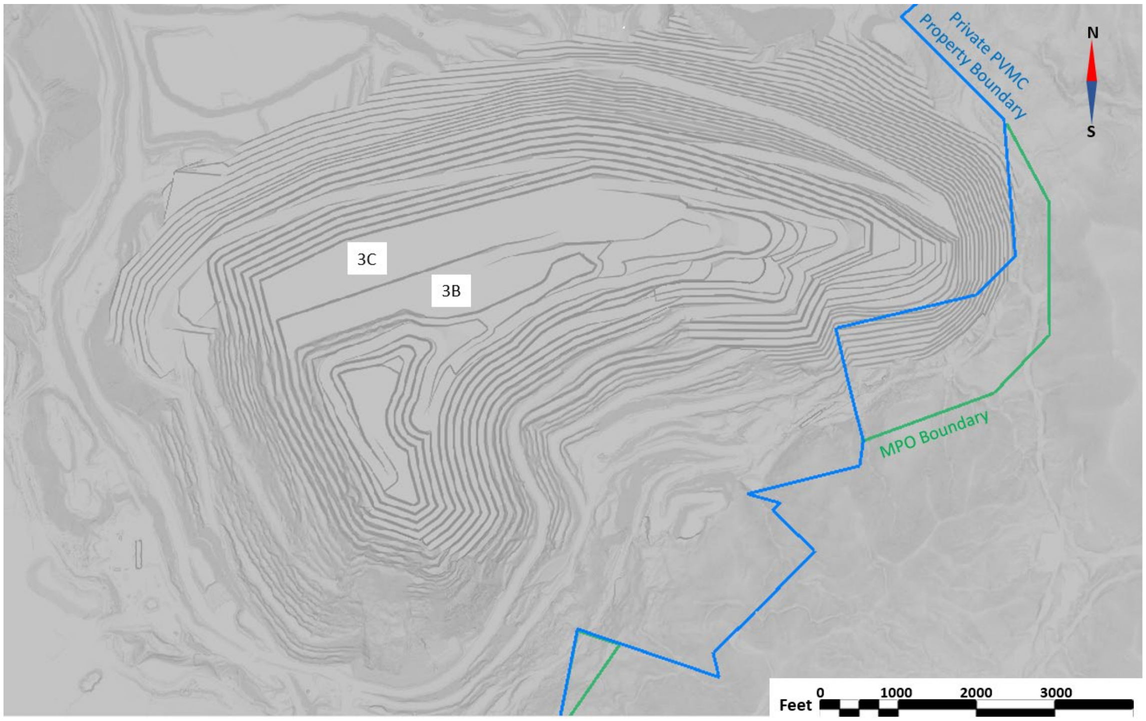


Figure 16-6: Planned Open Pit Position at end of 2035

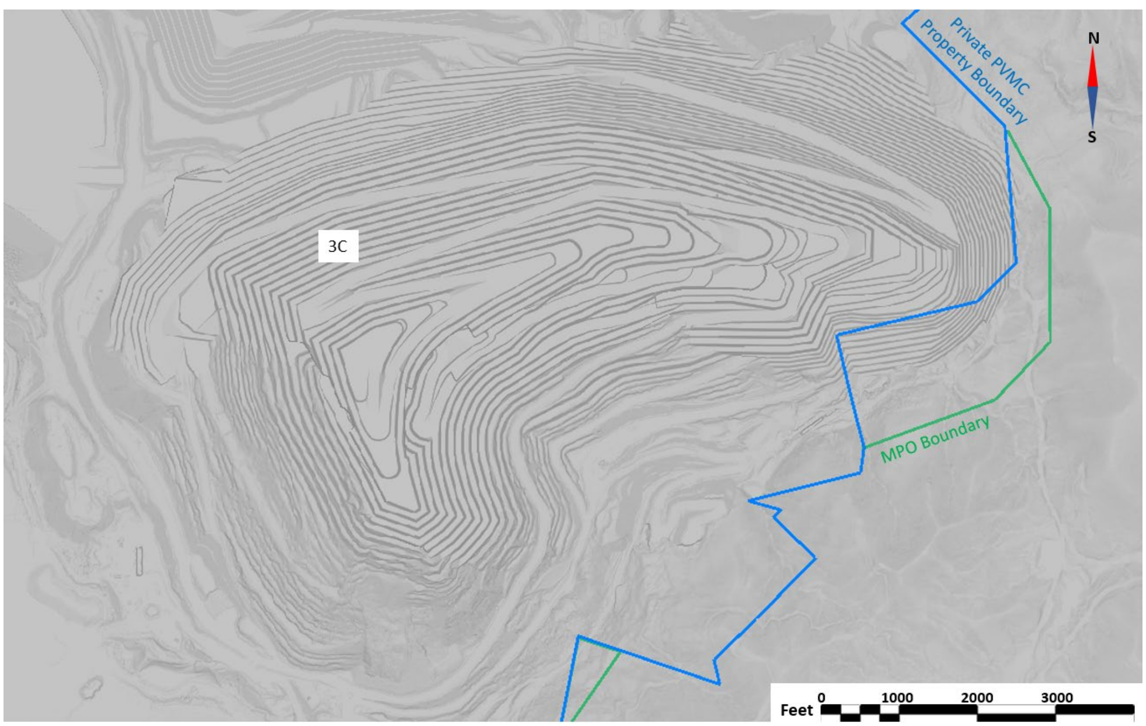


Figure 16-7: Planned Open Pit Position at end of 2039

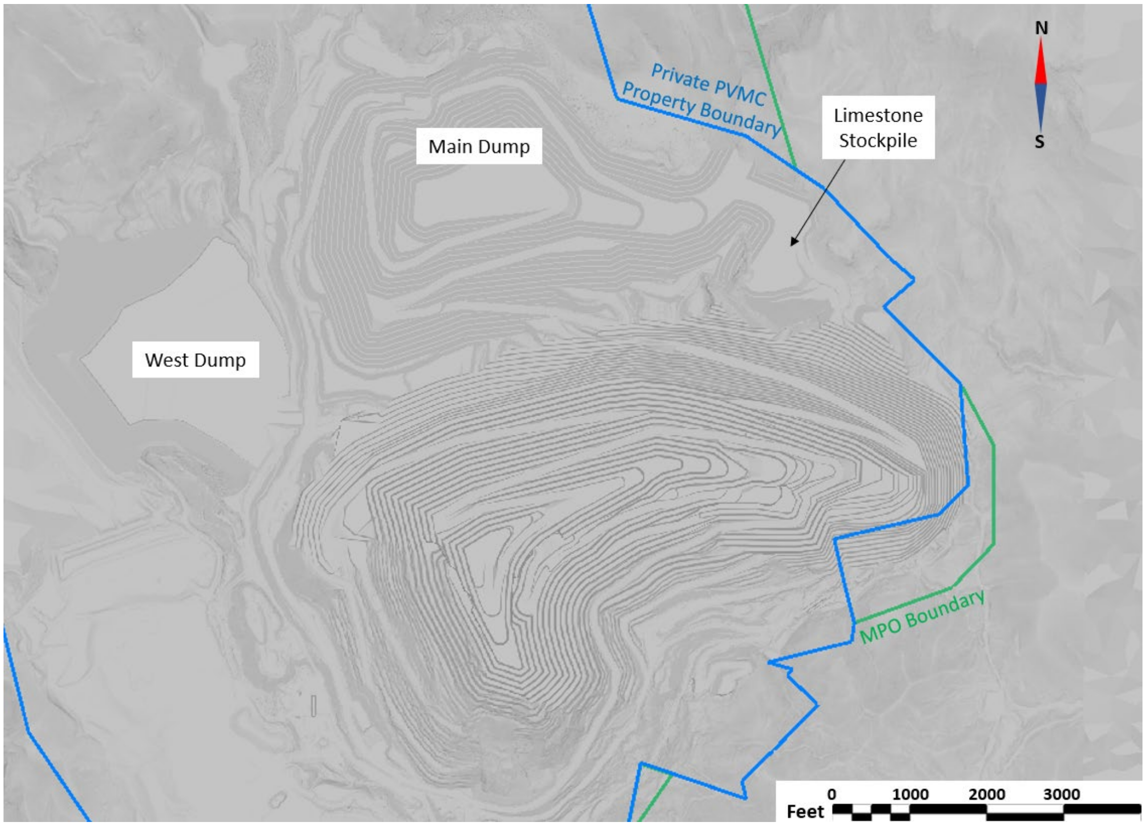


Figure 16-8: Ultimate PV3-2021 Configuration for Open Pit and Waste Dumps

16.6 Mine Operations and Equipment

Mine equipment requirements for the LOMP were calculated based on the annual mine production schedule, the mine work shift schedule, and equipment productivity estimates. The fleet is structured to ensure sufficient critical equipment is always available to meet minimum operational requirements; not all equipment is in use at the same time. Total material handled by the mining equipment peaks at an average of 52.5 million tonnes per year from 2021 through 2031.

The decision to continue with the current equipment fleet without additional units was described earlier in this section in the trade-off between capital investment and mine head grade benefits.

Table 16-4 presents a summary of the total mine fleet that is required for the LOMP. This table takes the fleet replacements and rebuilds into account to show available units on site.

Mine equipment requirements were not estimated for the following activities:

- Construction of major surface water diversion channels and settlement ponds and dams, other than the ditching and sedimentation ponds for the waste storage areas.
- Road construction outside of the immediate mine area.
- Clearing brush and stripping topsoil-growth media in advance of mining or dumping.

- Contouring or reclamation of dumps at the end of the mine life.

Table 16-4: Mine Major Equipment Fleet Requirement

Equipment Type	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Blasthole Drill	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2
Pre-split Drill	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1
CAT 789D Haul Truck	19	19	19	19	19	19	19	19	19	19	19	19	12	11	10	10	6	6	6
Liebherr T264 Haul Truck	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
CAT 994K	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
CAT 994H	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	0	0	0
Hitachi EX5600 Shovel	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0	0
CAT 992G	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
CAT D10T Track Dozer	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1	1	1	1	1
Liebherr 776 Dozer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
CAT D9T Track Dozer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
CAT 834H Wheel Dozer	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1
CAT 16M Motor Grader	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2
CAT 777F Water Truck	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2
CAT 777 Fuel/Lube Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
CAT 980 Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0
Total	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**50**	**37**	**36**	**32**	**32**	**27**	**27**	**27**

16.6.1 Mine Equipment Requirements Assumptions

The LOMP was largely scheduled utilizing the existing major mine equipment at PVM. This information has been combined with input from the mine site to establish the overall mine equipment list. The mine is scheduled to work two 12 hour shifts per day, seven days a week, 52 weeks per year.

Major equipment unit types addressed are:

- Blasthole drills
- Loading equipment (loaders and shovels)
- Haul trucks
- Major auxiliary equipment

Table 16-5 presents a summary of the maximum availabilities and use of availabilities that have been applied for each equipment type. Equipment is assumed to accumulate 11.0 metered hours per shift.

Table 16-5: Maximum Equipment Availabilities and Utilization

Equipment Type	Availability	Utilization
Blasthole Drill	80%	85%
Pre-split Drill	80%	85%
CAT 789D Haul Truck	89%	84%
Liebherr T264 Haul Truck	83%	84%
CAT 994K	84%	83%
CAT 994H	84%	83%
Hitachi EX5600 Shovel	80%	86%
CAT 992G	85%	85%
CAT D10T Track Dozer	85%	88%
Liebherr 776 Dozer	85%	88%
CAT D9T Track Dozer	85%	88%
CAT 834H Wheel Dozer	85%	85%
CAT 16M Motor Grader	85%	85%
CAT 777F Water Truck	85%	78%
CAT 777 Fuel/Lube Truck	85%	75%
Excavator	85%	75%
CAT 980 Loader	85%	85%

PVM employs the CAT Minestar Dispatch system. As part of this system, certain operational delays such as ‘queue to dump’ and ‘queue at crusher’ are considered as part of a truck’s cycle time and thus are excluded from the Utilization figure. These delays are nonetheless accounted for during Long Range Mine Planning with MineSight™ MPSO. Further, ‘standby’ delays are included within the Utilization metric at Pinto Valley. Regular reconciliation is performed to align predicted and actual metrics.

16.7 Operating Equipment Requirements

16.7.1 Drilling

Drill productivity was based upon recent historical performance. The current practice utilizes different drill patterns for ore and waste with a higher powder factor in the ore to induce more breakage and improve crushing and grinding throughput. Compared to 2016, a significant improvement in the proportion of blasted material under 0.5 inches was realized through use of these updated drill and blast protocols.

All mined material requires drilling and blasting, with the exception of the old waste and leach dumps and the low-grade stockpiles. Table 16-6 summarizes the production drill requirements per year. This table does not include the pre-split drilling requirements that are instrumental to high-quality wall slopes.

Table 16-6: Drill Requirements – Blast Hole Drill

Year	ORE		WASTE		TOTAL		Shifts per Year	Drill Fleet	# of Crews	# Operators
	kt	Shifts	kt[1]	Shifts	kt	Shifts				
2021	20,440	896	29,060	792	49,500	1,688	730	3	4	12
2022	20,440	896	30,560	833	51,000	1,729	730	3	4	12
2023	20,090	880	30,110	821	50,200	1,701	730	3	4	12
2024	20,496	898	33,304	908	53,800	1,806	730	3	4	12
2025	20,440	896	34,360	937	54,800	1,832	730	3	4	12
2026	20,440	896	34,360	937	54,800	1,832	730	3	4	12
2027	20,440	896	33,756	920	54,196	1,816	730	3	4	12
2028	20,496	898	34,304	935	54,800	1,833	730	3	4	12
2029	20,440	896	34,360	937	54,800	1,832	730	3	4	12
2030	20,440	896	34,360	937	54,800	1,832	730	3	4	12
2031	20,440	896	31,360	855	51,800	1,751	730	3	4	12
2032	20,496	898	20,504	559	41,000	1,457	730	2	4	8
2033	20,090	880	20,495	559	40,585	1,439	730	2	4	8
2034	20,440	896	5,439	148	25,879	1,044	730	2	4	8
2035	20,440	896	3,027	83	23,467	978	730	2	4	8
2036	20,496	898	1,082	29	21,578	928	730	2	4	8
2037	20,440	896	1,353	37	21,793	933	730	2	4	8
2038	20,440	896	906	25	21,346	920	730	2	4	8
2039	20,084	880	2,352	64	22,436	944	730	2	4	8
Total	**387,529**	**16,982**	**415,052**	**11,313**	**802,580**	**28,296**				

1. Waste tonnage does not include fill material where no drilling is required.

16.7.2 Loading

The loading fleet consists of two Cat 994K front end loader equipped with 27.5 m^3 (36 yd^3) buckets, two Cat 994H front end loaders equipped with 17.2 m^3 (22.5 yd^3) buckets, and one Hitachi EX5600 hydraulic front shovel equipped with 29.1 m^3 (38 yd^3) buckets. These units are matched with Cat 789D and Liebherr T264 haul trucks.

The wheel loaders are the primary loading units at the mine due to their low operating costs and high mobility. On average, one operating 994K loader will contribute about ~25% of the loading capacity. On average, one operating 994H loader will contribute about ~20% of the loading capacity. The remaining hydraulic shovel serves as a supporting loading unit to the mobile loader fleet. The second EX5600 shovel has been decommissioned and in the process of being sold in 2021. The shift away from shovels to wheel loaders was made due to the greatly reduced diesel consumption and maintenance cost of the CAT 994K units.

16.7.3 Hauling

The current truck fleet at PVM consists of 19 Caterpillar 789D haul trucks and four Liebherr T264 haul trucks. Truck productivity was estimated using haul time simulation based upon regular reconciliation against actual values recorded within the Minestar dispatch system. PVM measured haul profiles for each time period, material type, pushback and destination for input to simulation. In total 124 profiles were measured. Truck haulage times of each profile were calculated and the resulting tonnage per truck shift was used to calculate the required truck operating shifts for each year.

Each of the trucks will be rebuilt once at about 50,000 hours and replaced at about 100,000 hours. The CAT 789D haul truck load target is 185 dry tonnes and for the Liebherr T264 haul trucks the load target is 215 dry tonnes.

Table 16-7 summarizes the loading and hauling fleet requirements for the LOMP. From 2021 to 2031 an average of 21 trucks are required. The number of required trucks falls from 2031 to 2039 due to the reduced waste tonnes required to expose ore. The rebuild and replacement schedule maintains 23 trucks in the fleet through 2031.

Table 16-7: Loading Unit and Haul Truck Requirements

Year	Total kt	Loading Units				Haul Trucks		
		Loader kt	Shovel kt	Units	# of Operators	Truck Hours	Units	# of Operators
2021	49,500	47,028	2,472	5	18	147,333	19	76
2022	51,000	47,028	3,972	5	18	175,292	23	92
2023	50,200	47,028	3,172	5	18	179,085	23	92
2024	53,800	47,028	6,772	5	20	151,429	19	76
2025	54,800	47,028	7,772	5	20	158,490	20	80
2026	54,800	47,028	7,772	5	20	154,381	20	80
2027	54,196	47,028	7,169	5	20	159,454	20	80
2028	54,800	47,028	7,772	5	20	162,300	21	84
2029	54,800	47,028	7,772	5	20	151,030	19	76
2030	54,800	47,028	7,772	5	20	144,607	19	76
2031	51,800	47,028	4,772	5	20	177,684	23	92
2032	41,000	41,000	-	4	16	161,093	21	84
2033	40,629	40,629	-	4	16	173,559	22	88
2034	25,529	25,529	-	3	12	98,076	13	52
2035	23,467	23,467	-	3	12	96,594	12	48
2036	21,578	21,578	-	3	12	89,991	12	48
2037	21,793	21,793	-	3	12	98,855	13	52
2038	21,346	21,346	-	3	12	103,669	13	52
2039	22,739	22,739	-	3	12	126,242	16	64
Total	**802,577**	**735,384**	**67,193**			**2,709,163**		

16.7.4 Major Auxiliary Equipment

Major auxiliary equipment refers to the mine major equipment which is not directly responsible for production, but which is scheduled on a regular basis. Equipment operating requirements, operating cost per shift, and personnel requirements were estimated for this equipment.

The primary function of the auxiliary equipment is to support the major production units, and provide safe and clean working areas. Equipment types included in the auxiliary mine fleet are:

- 3, Caterpillar D10T Track Dozers
- 1, Caterpillar D9T Track Dozers

- 1, Liebherr 776 Track Dozer
- 2, Caterpillar 834H Wheel Dozers
- 3, Caterpillar 16M Graders
- 3, Caterpillar 777F Water Truck
- 1, Caterpillar 980H Auxiliary Loader
- 2, Caterpillar Excavators
- 1, Caterpillar 777F Fuel and Lube Trucks

The operating hours and shift requirements for the auxiliary equipment are based on current and planned scheduling practice at PVM.

16.8 Mine Personnel Requirements

Mine operations and maintenance labor manpower are provided to operate and maintain the equipment listed previously. Table 16-8 summarizes the required hourly personnel and the mine salaried and supervisory staff for the mine life.

Table 16-8: Hourly and Mine Supervisory Personnel

Hourly - Mine Operations	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Operations	165	190	190	168	174	174	174	181	168	168	193	168	174	112	106	106	112	112	131
Maintenance	61	70	70	62	64	64	64	67	62	62	71	62	64	41	39	39	41	41	48
Hourly Total	**226**	**260**	**260**	**230**	**238**	**238**	**238**	**248**	**230**	**230**	**264**	**230**	**238**	**153**	**145**	**145**	**153**	**153**	**179**

Salaried - Mine Operations	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Management	6	6	6	6	6	6	6	6	6	6	6	6	6	4	4	4	4	4	4
Operations	12	12	12	12	12	12	12	12	12	12	12	12	12	9	9	9	9	9	9
Maintenance	10	10	10	10	10	10	10	10	10	10	10	10	10	8	8	8	8	8	8
Engineering	9	9	9	9	9	9	9	9	9	9	9	9	9	6	6	6	6	6	6
Geology	4	4	4	4	4	4	4	4	4	4	4	4	4	3	3	3	3	3	3
Salaried Total	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**30**	**30**	**30**	**30**	**30**	**30**

16.9 Risks

Risks to the mining methods and LOMP presented in this section include:

- Slope stability issues in specific areas of the PVM pit, including the southwest corner of the pit in the Pinal Schist and the northeast side of the pit near the Bummer Fault, in particular where blasting techniques are not tailored to these areas.
- Availability of skilled labor to achieve mine plan.

- Availability of replacements for aging mine fleet, particularly during 2026-2028 when 789D fleet is reaching optimum lifespan.

16.10 Opportunities

Opportunities to optimize mining methods at PVM include:

- Continue to review the mining fleet to optimize type and size of equipment used, including consideration of electric trucking and trolley assist
- Investigate innovative technologies that may improve the ratio of material moved to fuel usage, such as the use of specialized fuel additives for heavy equipment
- Complete the blast fragmentation study in progress, expected by the end of 2021
- Study current workflows to determine possible opportunities to debottleneck mining

16.11 Recommendations

There are no recommendations related to mining methods.

17 Recovery Methods

17.1 Introduction

The PVM concentrator was built in 1973, and the basic processing flowsheet has largely been unchanged since that time. The original nameplate capacity was 36,287 tonnes per day (tpd) (40,000 short tons per day (stpd)). Significant improvements have been incorporated over the years, resulting in the current 56,000 tpd base case capacity. The PVM plant is designed to treat porphyry copper ore with minor molybdenum. The main minerals of interest are chalcopyrite and molybdenite, with by-product quantities of precious metals including gold and silver.

The PVM concentrator flowsheet includes:

- Crushing - primary through to tertiary crushing
- Grinding – conventional ball mills
- Flotation – copper and molybdenum - conventional cells for roughing and scavenging and columns for primary cleaning
- Copper concentrate thickening and filtration
- Molybdenum concentrate thickening, filtration and drying
- Tailings thickening
- Tailings impoundments

Figure 17-1 shows a simplified process flowsheet of the PVM concentrator.

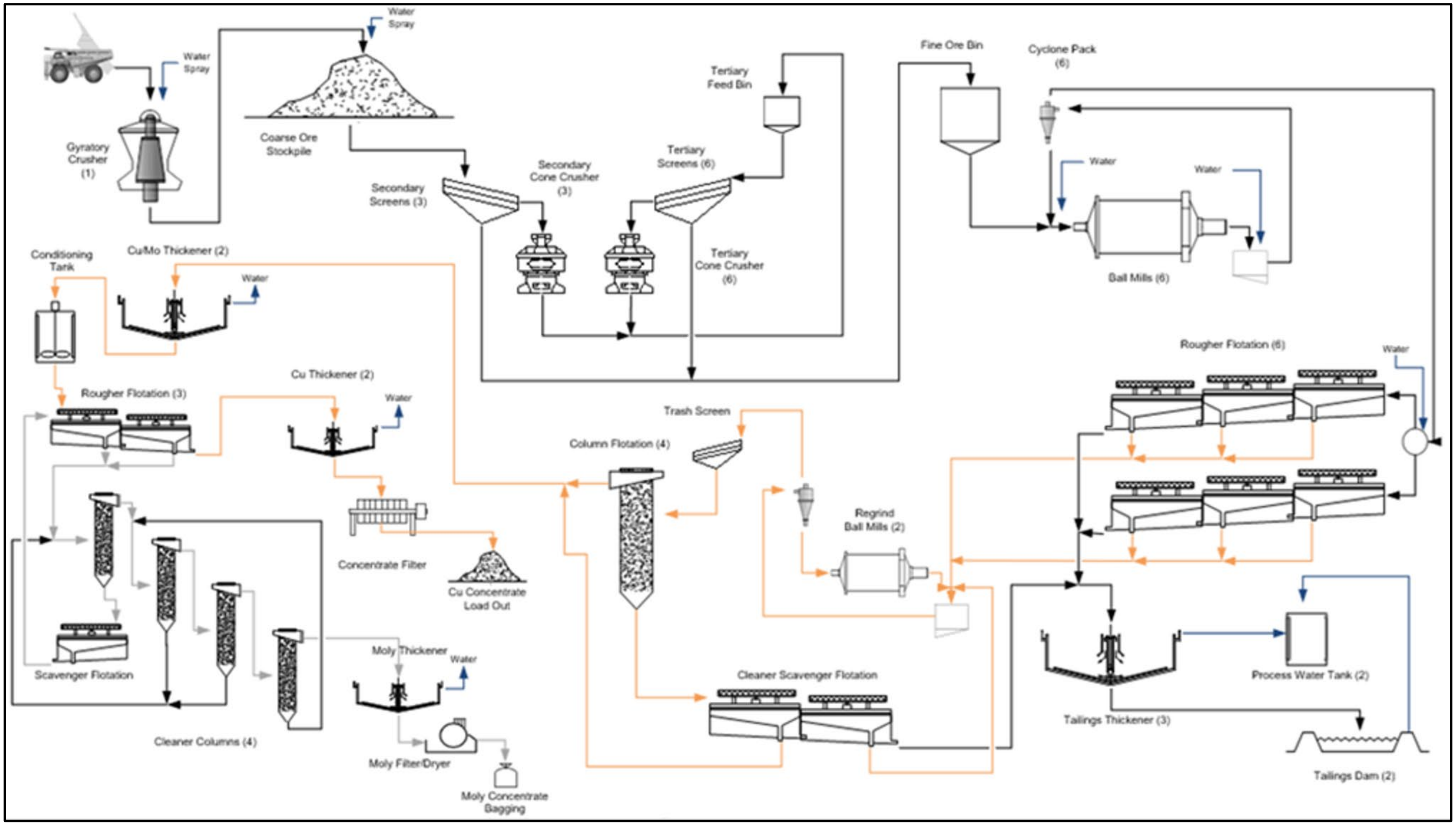


Figure 17-1: Simplified PVM Process Flowsheet

17.2 Plant Optimization

Improvements in the plant since Capstone acquired PVM have resulted in the ability to increase throughput significantly from nameplate capacity. Continuous operating and maintenance improvements have increased the availability and utilization, as well as capacity, in all the circuits. This has allowed the plant to meet its new production target with the potential for further expansion.

These improvements have included minor expansions/changes to both the copper and molybdenum flotation circuits, and a new filter plant installed in 2006. The most recent major upgrades (2020) have focused on the fine crushing plant (FCP) and include:

- Replacement of the existing secondary 7 ft diameter Symons with a Raptor 9000 (in progress with completion expected in 2021). This represents a change in installed power from 373 kW to 671 kW (500 to 900 hp) and an upgrade to more modern crushing technology.
- Replacement of the 7 x 16 ft Simplicity double deck screens to 8 x 20 ft Ludowici Banana Double Deck Screens for secondary crushers.
- Replacement of the 8 x 20 ft Simplicity double deck screens to 8 x 20 ft Ludowici Banana Double Deck Screens for tertiary crushers.
- Mill shell replacements. The ball mill shells at PVM are original equipment and are being systematically replaced due to wear with one shell replacement completed in 2020 and a second shell scheduled for 2021.
- Other improvements include improved blast fragmentation, flotation level control upgrades, operator training, flotation circuit adjustments including flowsheet and reagent optimization. Currently, the molybdenum circuit is being optimized with a new reagent suite to potentially eliminate the use of NaHS.

17.3 Plant Process Design Criteria

The various unit operations of PVM have been evaluated to develop the basic process design criteria for the plant. The target concentrator throughput is 56,000 tpd. Where possible, the operating data has been compared to test work and/or simulation models to determine suitability for the design criteria. The key elements of the process design criteria include the following (operating data has been employed unless stated otherwise):

Primary Crusher

- Availability Average for 2020: 86%
- Feed Rate: up to 4,300 tph
- Product Size Distribution (typical)
- P80: 61.9 mm

- 32% passing 12 mm

Secondary/Tertiary Crushers

- Availability Average for 2020: 83%
- Feed Rate: up to 2,900 tph
- Product Size Distribution (typical)
  - P80: 10.9 mm
  - 87.5% passing 12 mm

Grinding Circuit

- Availability Average for 2020: 93%
- Work Index: 14.5 kWh/t (nominal)
- Daily Throughput: 56,000 tpd (nominal)
- Cyclone overflow P80: 330 µm (typical)
- Overflow Density: 37% solids (target)

Copper Flotation

- Roughers
  - Rougher Feed Density: 35% solids (target)
  - Flotation Residence Time: 17 minutes (nominal)
  - Mass Pull: 5-8% (target)
  - Concentrate Grade: 4-6% Cu (typical)
  - Recovery: 88-90% (typical)
- Cleaners
  - Concentrate Grade: 2020 average 25% Cu
  - Final Recovery: 2020 average 85%

Copper Concentrate Filtration

- Availability Average for 2020: 93%
- Throughput: 2020 average 580 tpd as dry concentrate
- Moisture: less than 10% (typical)

Final Tailings Thickening

- Throughput: 2020 average 2,400 tpd

- Underflow Solids: 2020 average 55% solids
- Overall Water Recovery: 60%

17.4 Process Description

The following are the descriptions of the primary unit processes in the PVM mill. Based on the evaluation of the existing plant equipment, current operating results, process modeling, and maintaining the design criteria and operating parameters identified in Sections 17.1 through 17.3, it has been determined that the PVM mill has an estimated capacity of 56,000 tpd. With further circuit optimization, and improved availability, it is believed that the crushing and grinding circuit throughput can be increased further.

17.4.1 Primary Crushing

ROM ore is delivered by haul truck to a Fuller Traylor™ 60 inch × 89 inch gyratory primary crusher. The trucks discharge directly into the crusher, which is set in a dump pocket. The crushed ore is withdrawn from a surge pocket under the crusher by an apron feeder, which discharges onto the primary conveyor.

The primary conveyor transports the primary crushed ore to the coarse ore stockpile, which has a nominal live capacity of 27,215 tonnes (30,000 tons). Based on digital size analysis of the primary crusher discharge, the P80 of the primary crushed ore is approximately 88.9 mm (3.5 inches). The fine product distribution is attributed to ore fragmentation and mine blasting practices. The discharge product averages 32% passing 12 mm (half inch). The quantity of fines in the feed has a significant influence on the production capacity of the FCP and this has been the focus of blast fragmentation optimization.

17.4.2 Secondary and Tertiary Crushing (Fine Crushing)

The primary crush ore is reclaimed from the coarse ore stockpile by six apron feeders, which feed three coarse ore reclaim belts. Each coarse ore reclaim belt discharges onto an 8 X 20 ft Ludowici double-deck vibrating screen. Two of the screen's oversize reports to two secondary 7 ft Nordberg™ standard cone crushers and one screen feeds the new Raptor 900 standard cone crusher. Screen undersize from the secondary screens is sent to the fine ore bin (FOB), with a nominal live capacity of 40,000 tonnes (44,000 tons).

The secondary crushers operate in an open circuit. Crusher product from all three secondary crushers is forwarded via a common conveyor system to the tertiary crusher feed bin. Ore is withdrawn from the tertiary crusher feed bin by six feeders and delivered directly to six 8 X 20 ft Ludowici double-deck vibrating screens. The screen undersize from the tertiary screens is sent via a common conveyor system to the FOB. The screen oversize is crushed by six 7 ft Nordberg™ tertiary shorthead cone crushers. The product from the six shorthead crushers is added to the secondary crusher product on the common conveyor system to feed the tertiary feed bin, and tertiary crushing is operated in a closed circuit. At the current plant throughput, the P80 of the fine-crushing plant is approximately 11 mm (0.43 inches).

17.4.3 Grinding

Fine ore is reclaimed from the FOB and fed directly to six 18 ft × 21 ft, 2,983 kW (4,000 hp) Allis-Chalmers™ overflow ball mills. Each ball mill is an independent circuit consisting of a discharge sump, pump, and cyclone cluster. Water is added to the ball mill feed to achieve the desired percent solids content for grinding.

Additional water is required at the ball mill discharge sump to maintain the optimal operation of the cyclones. Each circuit is equipped with three 838 mm (33-inch) inclined cyclones. Cyclone overflow slurry gravity feeds the rougher flotation banks, while the underflow discharges back to the ball mill feed sump. The ball mills operate in a closed circuit with the cyclones, with a circulating load estimated at 400%.

Xanthate (SIPX), dithiophosphate (DAP), diesel, and lime are added to the grinding circuit to prepare the ore in the slurry for flotation. A pH target of 10 is utilized for flotation.

17.4.4 Flotation

The flotation circuit operates as a bulk copper and molybdenum flotation process. Subsequent differential flotation is designed to produce the final individual copper and molybdenum concentrates. The rougher flotation circuit is operated to maximize recovery of the primary sulfide minerals from the gangue. Subsequent cleaner flotation of the bulk concentrate is operated to maximize the copper concentrate grade while minimizing copper recovery losses.

The flotation reagents used include SIBX (Xanthate), C-2420 (dithiophosphate), diesel (for molybdenum recovery) and Flottec F-171 (frother). The majority of the flotation reagents are added to the ball mill feed with “kickers” of reagents added down the flotation bank where required.

Regrinding of the rougher concentrate is required to provide increased mineral liberation to allow cleaner flotation to produce high concentrate grades. The molybdenum is co-recovered to the bulk cleaner concentrate with the copper. The molybdenum flotation circuit provides the separation of the copper and molybdenum into respective salable concentrates.

17.4.4.1 Rougher Flotation

The rougher flotation circuit consists of 65 28.3 m^3 (1,000 ft^3) Wemco™ cells configured into three banks each with two trains (Sections 1,2,3), with cyclone overflow from two ball mills combined to feed each of the banks. The frother is added to the head of the rougher flotation cells, with supplemental reagents added as required down the bank. The rougher section is operated in open circuit, with the rougher tailings reporting directly to the final tailings.

17.4.4.2 Regrind

The rougher concentrate is delivered to the regrind ball mill circuit. The regrind circuit consists of two 11 x 15-ft regrind ball mills driven by a 373 kW (500 hp) synchronous motor. The regrind circuit is a reverse configuration common for regrinding. The rougher concentrate is combined with the regrind ball mill discharge and pumped to the closed-circuit cyclones (a bank of four 500 mm (20 inches) diameter cyclones for each mill).

The cyclone overflow is fed to the cleaner flotation circuit, while the underflow is sent to two regrind mills. The regrind mills operate in a closed circuit with the cyclones. The target product for regrind cyclone overflow is P80 of 50 µm.

17.4.4.3 Cleaner Flotation

The cleaner circuit consists of four 2.4 m diameter by 12.2 m tall (8 ft diameter by 40 ft tall) column flotation cells operated in parallel. The column cell concentrate, the final copper-molybdenum bulk concentrate, contains 24% to 29% Cu and 0.35% to 0.7% Mo. The column cell tails are sent to the cleaner scavenger flotation bank. The cleaner scavenger bank comprises 15 8.5 m^3 (300 ft^3) Wemco™ flotation cells. The first bank of cells produces the final concentrate, and concentrate from the remaining cells is recirculated to the column cells via the regrind circuit. The tails of the cleaner scavenger bank report to final tailings.

17.4.4.4 Molybdenum Plant

The bulk copper-molybdenum concentrate from the cleaner circuit is thickened before being sent to the molybdenum plant. The plant comprises four banks of Agitair™ rougher cells of six 1.4 m^3 (50 ft^3) cells each and four stages of cleaning using column cells. Sodium hydrosulfide (NaHS) is added to the slurry to provide depression of copper and iron sulfides. Diesel is added as a molybdenum promoter.

17.4.5 Concentrate Dewatering

The molybdenum rougher tailing becomes the final copper concentrate reporting to one of two 27.4 m (90 ft) copper thickeners. The final molybdenum product is thickened in a 7.9 m (26 ft) molybdenum thickener, filtered on a disk filter, dried in a rotary dryer, and bagged for shipment.

The final copper concentrate is thickened to 60% solids and flows by gravity from the copper thickeners to either of two 900 m^3 (238,000 gallons) copper concentrate slurry storage tanks. The slurry is pumped from the storage tanks to the filter plant. The concentrate is filtered in an Eimco™ plate and frame pressure filter and conveyed to the copper concentrate storage shed for loadout.

17.4.6 Tails Thickening

Tailings from the three rougher banks and the cleaner scavenger bank are combined and feed three 106 m diameter (350 ft) tailings thickeners where overflow water is reclaimed, and the tails are thickened and sent on to the TSFs.

TSF4 is the primary location for the disposal of tailings from the Pinto Valley mill. TSF3 is used only for initial start-up and in an emergency should a problem arises with the TSF4.

In early 2016, a new tailings pump station and pipeline were installed. Details of tailings pumping and storage are described in Section 18.

17.4.7 Process Water

Water supply for processing is delivered to the facilities through a system of above-ground and buried pipelines that generally follow road alignments. Sources include the Cottonwood Reservoir (formerly the decant pond of the now inactive Cottonwood TSF), the Mine Reservoir (a concrete-lined 2.32 acre pond) and the Peak Well system.

Supplemental process water is pumped from a neighbouring property, BHP Copper's Copper Cities Unit Diamond H pit. Capstone signed a Water Supply Agreement with BHP Copper effective until October 2025; the agreement is subject to water availability and BHP Copper's own requirements. Operational water use is closely monitored at PVM, and water conservation improvements are on-going, resulting in the reuse of 93% of the water used by PVM.

PVM maintains a site water balance that indicates that existing water sources will be sufficient to execute the LOMP, though periods of severe drought have the potential to limit PVM's ability to process ore at planned throughput rates.

17.5 SX-EW

The PVM SX-EW plant was built and commissioned in 1981 to process solutions from the leach grade material placed on the run-of-mine (ROM) leach dumps north of the pit. Through 1998, approximately 450 M tonnes of 0.13% Cu material had been placed on the leach dumps, resulting in peak production of 10 to 15 M lb of cathode copper per year in the early 2000s. Over the last few years, the SX-EW has produced in the range of 3 to 5 M lb of cathode per year due to the declining residual copper inventory in the leach piles. A moderate quantity of fresh material was placed on the leach pad in 2020.

In the PV3-2016-PFS, the leach area and pregnant solution pond area were slated for future decommissioning and conversion to waste rock storage after suitable rinsing and drainage. PVM is evaluating options for continued use of select leach areas (see Section 17.7 Opportunities). Effluent from the dump leach will continue to be processed in the SX-EW plant for the foreseeable future.

17.6 Risks

Key risks to the operation of the PVM plant as planned include:

- The plant was originally built in 1973 and has been well maintained, but much of the original equipment remains in service. The PVM technical team has a good understanding of the replacements and upgrades required to maintain high utilization and availability. The PVM team has undertaken a systematic program to address these issues, however, given the age of the equipment, unforeseen failures may be a factor.
- Insufficient water supply for prolonged periods of time could affect PVM's ability to process material economically, or at the rate presented in the current LOMP. Future water sources may not be available or available at a suitable cost. PVM's on-going water conservation efforts may not be sufficient if water availability decreases due to climate change or other factors.

17.7 Opportunities

A number of opportunities have been identified to increase copper recovery from the ore and waste rock at PVM:

- PVM is reviewing potential opportunities to enhance dump leach performance and increase copper cathode production from mineralized waste over the life of mine, with the study expected to conclude in 2022. Proof of concept was demonstrated in a test application of patented catalytic technology from Jetti Resources to an existing dump leach.

- An investigation into recovering additional minerals from the copper circuit cleaner tailings (Pyrite Agglomeration) has been initiated, with results expected in 2022. The material would be leached in conjunction with dump leach material. The goal is to enhance copper recovery and improve dump leach acid availability.
- Pilot plant testing of coarse particle flotation technology by Eriez Flotation Division suggests a 6 to 8% increase to overall copper recovery is achievable (Capstone, 2021). Investigation continues into other possible benefits such as lower grinding costs by increasing grind size, and reductions to specific water and specific energy use of the plant. Results of the study are anticipated the second half of 2021.

17.8 Recommendations

PVM is evaluating a number of potential upgrades to the plant. The following recommendations are related to improving plant copper recovery:

- Upgrade the level control system for the rougher flotation circuit to maximize mass pull and mineral recovery. The estimated cost of this upgrade is $200,000 with completion estimate of six months.
- Upgrade the reagent dosing system to improve the reagent addition control. The estimated cost of this upgrade is $150,000 with a complete estimate of three months.
- Upgrade the rougher concentrate pumps to reduce overflow situations and allow for maximized mass pull at coarser grinds. The estimated cost of this upgrade is $250,000 with completion estimate of six to nine months.
- Upgrade the tailings thickeners to more efficient units that yield higher density outputs and greater water recovery. The upgrade is in progress, with completion expected in July 2021. The upgrade cost is estimated at $5 to $7 million. Center wells are currently being fabricated and set for delivery to site starting in mid-June. Piping and other ancillary equipment is ordered and set for delivery in late June.

18 Project Infrastructure

18.1 Site Infrastructure

Existing PVM infrastructure includes:

- Mine Equipment Maintenance Facilities (North Barn, Main Shop, wash bays, tire change area)
- Offices complexes (admin, mine, mill)
- Heavy and light vehicle fuel storage and distribution
- Explosives Plant
- Pit dewatering pumps and pipelines
- Concentrate dewatering, storage and loadout
- Warehousing and Change Rooms
- Stormwater ponds and pumping systems
- Internal roads and access road FR 287
- Water wells and water pumping systems
- First aid facility
- Assay lab
- Power lines and transformers
- Tailings storage and distribution facilities
- Waste dumps (see Section 16.4 for details)

All infrastructure is currently adequate to support the life of mine through 2039, with the exception of tailings storage and distribution facilities which will continue to be expanded as needed within their established design.

18.1.1 Water Supply and Storage Details

PVM uses water for processing ore and for personnel use. Water sources include the Peak Well system and water pumped from neighbouring properties, BHP Copper's Copper Cities Unit and and Old Dominion Mine. Capstone signed a water supply agreement with BHP Copper effective until October 2025; the agreement is subject to water availability and BHP Copper's own requirements. Buried and above ground pipelines for water supply generally follow road alignments. Additionally, two water storage tanks are situated immediately north of the Cottonwood Reservoir: an 110,000-gallon steel-walled potable water tank and a 650,000-gallon steel-walled fire/service water tank. Both tanks are above-ground installations resting on concrete foundations.

18.1.2 Electrical Supply Details

PVM purchases electricity from grid sources that provide a stable, continuous supply for that meets operational requirements. In Arizona, grid electricity is generated from a combination of nuclear, coal, natural gas and renewables like hydro, biomass, solar and wind. Electrical consumption is monitored primarily as a cost driver through monthly billing and metering. PVM adjusts electricity use during peak demand periods. The mill grinding circuit is the largest electricity use at PVM.

18.2 Tailings Storage

PVM currently operates two tailings storage facilities (TSFs), TSF3 and TSF4. TSF4 is the primary storage facility, and TSF3 is typically used when maintenance is required on the TSF4 tailings distribution system or during plant upset conditions. Inactive TSFs at PVM include Cottonwood TSF and TSF1/2, which have been closed and largely reclaimed.

TSF4 has been evaluated and was determined to be capable of storing an additional 393 M tons (375 M tonnes) dry weight of tailings at a settled density of 92 lb/ft^3 (1.47 t/m^3). TSF4 would rise from the current crest elevation of approximately 3,950 ft to the planned ultimate crest elevation of 4,250 ft at an average 17 ft-per-year rate of rise.

TSF3 is capable of storing an additional 21 M tons (19 M tonnes) dry weight of tailings at a settled density of 90 lb/ft^3 (1.44 t/m^3). TSF3 is designated to be used primarily as a backup to TSF4, whenever maintenance or repairs are needed in the tailings transport system to TSF4.

18.2.1 Tailings Characteristics

Average annual mill throughput is expected to be approximately 62,000 tons (56,000 tonnes) per day through 2039. The total tailings storage requirement is 420 M tons (381 M tonnes) over the remaining mine life. An average mill availability of 95% is anticipated. The tailings distribution and water reclamation systems have been evaluated for a tailings production rate of 65,000 tons (58,900 tonnes) per day. The design tailings pipeline flow rate is calculated to be 12,930 gpm with 55% solids content in the tailings slurry.

Tailings gradation test results indicate that approximately 10% of tailings particles are finer than 4.8 µm, approximately 50% of particles are finer than 112 µm, and approximately 80% of particles are finer than 360 µm.

18.2.2 Tailings Storage Facility Design

18.2.2.1 Design of TSF4

Figure 18-1 presents the ultimate proposed layout of TSF4. TSF4 was placed in service in 1977. The crest of TSF4 was at an elevation of approximately 3,950 ft in the first quarter of 2021. The tailings impoundment surface immediately below the dam crest is at an elevation of approximately 3,930 feet. The LOMP mill production will result in TSF4 being raised to elevation 4,250 ft by 2039. The dam is constructed in an upstream manner, with a cycloned sand shell.

At present, the TSF4 footprint is constrained by the patented land boundary. Boundary dams are required along the eastern edge of the facility to prevent the migration of tailings to the adjacent USFS land. The boundary dams are currently being built in a staged manner using cyclone sands and earthen material. As described in Section 20.3, USFS approval of the consolidated Mine Plan of Operations (MPO) is required to allow placement of tailings beyond the current property boundary. The tailings deposition plan assumes that required authorization will be secured in 2021, to allow TSF4 tailings deposition beyond the current property boundary.

18.2.2.2 Design of TSF3

TSF3 was placed in service in 1973 and was operated until early 2009 to a crest elevation of approximately 3,740 ft, with a maximum height of 451 ft. In 2011 and 2012, the configuration of the facility was altered by shifting the active deposition inward from the dam crest. The inset TSF3 embankment is constructed to an elevation of approximately 3,782 ft. The TSF3 footprint is also currently constrained by the patented land boundary although tailings deposition is authorized on a portion of National Forest System land.

The TSF3 main embankment will be raised in a sequence of centerline and upstream lifts. These raises will provide 6.6 M tons (6 M tonnes) of storage capacity to an elevation of 3,780 ft. Once the consolidated MPO is approved, TSF3 can be raised to an ultimate elevation of 3,857 ft, which will provide 21 M tons (19 M tonnes) of tailings storage.

18.2.3 Stability and Seepage Analysis

Past performance and stability analyses of TSF3 and TSF4 indicate that these facilities meet ADEQ stability safety requirements for static and pseudo-static (associated with a seismic event of a 975-year recurrence interval) conditions. Dynamic analyses of TSF3 and TSF4 were completed in 2015. The results indicate that for design-level ground motions caused by earthquakes with magnitudes of 5.0 to 6.5 and for Peak Ground Acceleration (PGA) values about 0.085g, the damage to the embankment is “none or slight.”

Wood has performed annual reviews of the operation of TSF3 and TSF4 from 1998 to the present. Studies and monitoring of TSF4 reveal that the facility benefits from drainage provided by the underlying permeable foundation materials. These fluids are recovered in downgradient wells.

18.2.4 Hydrologic Analysis

Hydrologic analyses have been performed for TSF3 and TSF4 for their current and ultimate configurations. Both facilities were determined to be capable of containing the stormwater inflows associated with the Probable Maximum Flood (PMF).

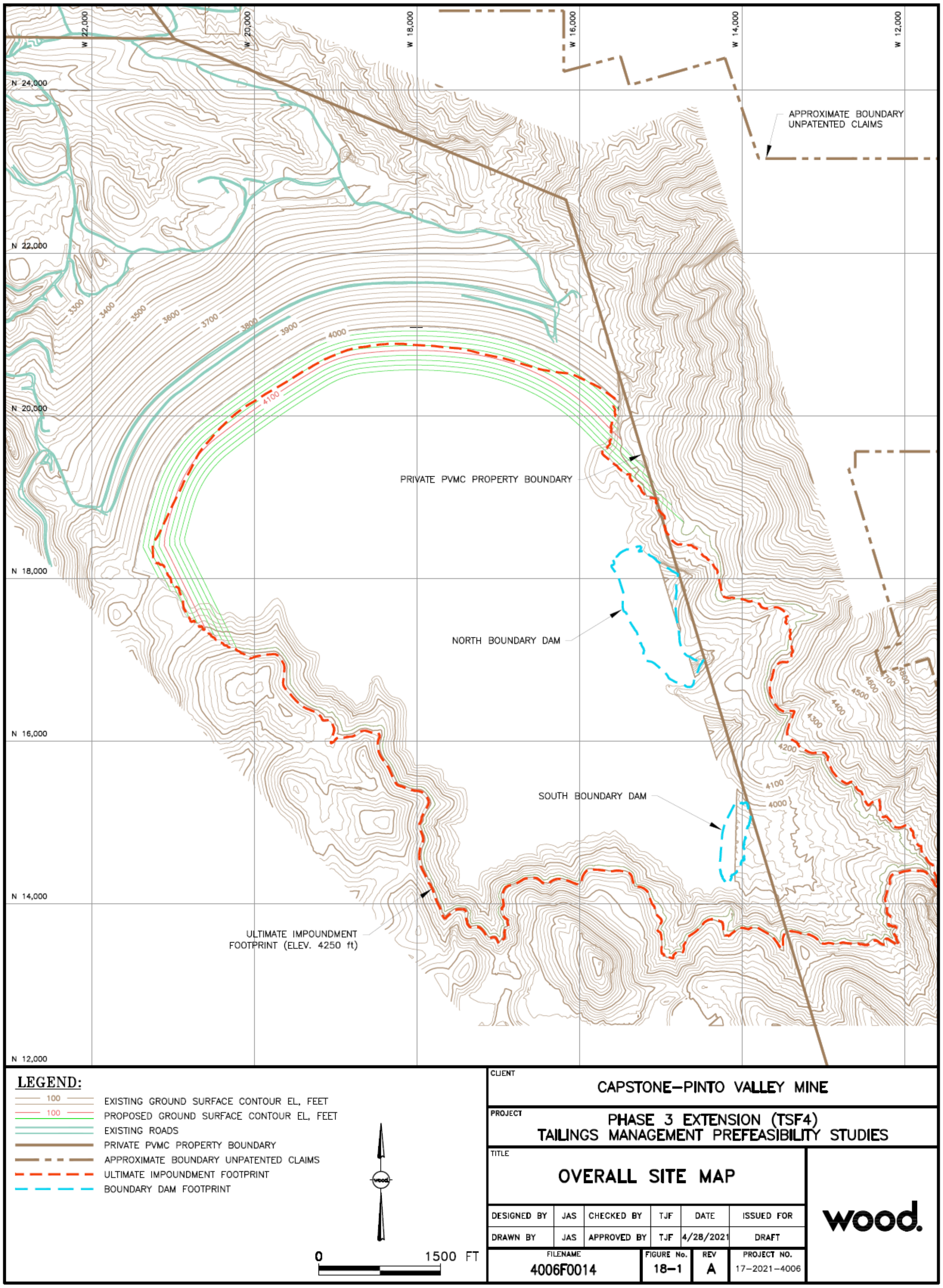


Figure 18-1: Ultimate TSF4 Footprint

18.3 Tailings Storage Facilities Operation

The mill tailings are thickened to approximately 50% to 57% solids by weight, and are conveyed from the thickener collection box to the booster pump station for approximately 7,500 ft through a 34-inch diameter DR 15.5 HDPE pipeline operating under gravity open flow conditions.

The tailings' settled density was measured, in earlier studies, between 85 lb/ft^3 and 114 lb/ft^3 at an in-situ moisture content of 4% to 33%. Average dry densities of 90 lb/ft^3 for TSF3 and 92 lb/ft^3 for TSF4 were used for the volumetric calculations. These estimated averages were developed based on the measured values and a review of literature data and have accounted for additional capacity that may be gained from future consolidation of existing tailings.

18.3.1 TSF4 Operation

A tailings booster pump station, currently configured with three stages of pumping, delivers tailings to TSF4 through a 30-inch lined steel or 30-inch HDPE pipeline. Upgrades to the pump station will be made in 2021 to increase pump availability and reduce the frequency of upset conditions, allowing for more controlled deposition to TSF3. Additional upgrades to the pump station and associated infrastructure are anticipated as the construction of the facility proceeds.

Crane mounted clustered cyclones are being used for tailings distribution at TSF4. Each cluster consists of 12 cyclones fed from a single feed line. The cyclone sand underflow is used to construct the dam embankment. The slimes are collected and discharged to the beach. Single point discharge deposition outfalls are used when the cranes are being moved or otherwise out of service.

The existing reclaim water system consists of barge-mounted pumps and two booster pump stations conveying water from the southern end of the TSF4 decant pool to the Mill Water Supply Tank. The nominal design flow rate is 6,500 gpm.

The existing dual barge pumps are Hazleton Model 12DA Type VNCC with 21.5-inch impellers. The drivers are Westinghouse 186 kW, 3-phase, 4,160 V, 1,174 rpm motors. Both booster pump stations have two parallel Allis-Chalmers Model 3415 12 × 14 horizontal split case pumps with full-size 45.72 mm impellers rated for 1,780 rpm. The drivers are GE 450 kW, 3-phase, 4,160 V, 1,800 rpm motors. All of the reclaim pumps are controlled (kept within their rated pump curve) by back pressure valves that are partially closed (increasing the back pressure) as the pond level rises.

As the TSF4 decant pond rises, the booster pump station will likely need to be moved upgradient. Various pump types and configurations are being considered to optimize the system, taking into consideration storm volumes that must be managed and operational needs.

18.3.2 TSF3 Operation

One stage of pumping is used to deliver tailings from the pump station to TSF3. A header pipe along the east and north and northwest crests of TSF3 is being constructed to allow for spigot discharge along the entire TSF crest. Spigots will be located at 52 ft intervals. Block valves are located at 500 ft internals.

The existing reclaim water system at TSF3 consists of a single trailer-mounted 8 ft by 6 ft self-priming centrifugal pump with an engine drive.

18.3.3 TSF Monitoring

A series of piezometers have been installed to monitor pore pressures at the active and inactive TSFs. The piezometers include open standpipes and directly embedded vibrating wire piezometers. Piezometers measurements are performed through an automated data acquisition system. Additional piezometers will be installed as the TSFs are raised to ensure that pore pressures are within acceptable ranges.

Monitoring reports of the piezometer readings are prepared and reviewed by the designated PVM engineer to verify that the TSFs are operated, and are performing, as designed. An annual monitoring report, which includes a summary of the facility performance and the piezometer measurements, is submitted to ADEQ.

The active TSFs are inspected quarterly, and the inactive TSFs are inspected annually by PVM’s Engineer of Record. Feedback is provided on the operation of the TSFs and recommendations, if necessary, are provided to the PVM management and engineering groups. Capstone Mining Corp. retains an independent tailings expert on a bi-annual basis to perform an independent review of all active and inactive TSFs at PVM.

18.4 Risks

18.4.1 Risks to PVM with Respect to Infrastructure

PVM infrastructure such as buildings and facilities are generally in good condition and are suitable for continued usage through the life of mine described in this report, but some facilities are relatively old, and continued investment will be needed to ensure their long-term reliability.

18.4.2 Risks to PVM with Respect to Tailings Management

The upstream construction method used to raise PVM’s TSF3 and TSF4 requires consistent tailings management procedures to ensure the development of competent tailings beaches and to control embankment pore water pressures. If these procedures are not followed, it can jeopardize the feasibility of continued upstream embankment raises, and limit future tailings storage capacity.

18.5 Opportunities

18.5.1 Opportunities at PVM with respect to Infrastructure

For long-term increased energy efficiency, PVM may consider increased use of solar power where appropriate, such as lighting in administrative buildings, light stands in remote areas of the site and for electronic signage. PVM is also assessing the feasibility of a larger scale solar installation to significantly decrease the amount of grid power used.

18.5.2 Opportunities at PVM with respect to Tailings Management

Tailings management at PVM is currently governed by a tailings management system developed following the guidance of the Mining Association of Canada’s *Guide to the Management of Tailings Facilities* (MAC, 2021). To ensure ongoing conformance with international best practices, PVM may consider also working towards conformity with the *Global Industry Standard on Tailings Management* (GTR, 2020).

18.6 Recommendations

18.6.1 Tailings Management Recommendations

Monitoring and control of pore pressures in the TSF embankments is critical to the performance of the facilities. Additional geotechnical field investigations, including cone penetration testing, exploratory drilling, laboratory testing and engineering analyses are likely to be needed. Methods to control or mitigate pore pressures would be developed if these data indicate that they are necessary. A total of $6.6 M has been included in the life of mine tailings management budget for these efforts, and is reflected in the costs presented in Section 21.

19 Market Studies and Contracts

19.1 Copper

Copper and copper-based alloys are used in a variety of applications to increase standards of living. Its continued production and use are essential for society's economic development. Historically, urbanisation has been the key driver of copper consumption growth consisting of new housing, power infrastructure, and consumer goods as newly urbanised residents populated their homes with household appliances. Currently, the most significant driver of copper consumption growth is being derived from a shift to renewable energy generation and low carbon technologies to address climate change. Major economies look to emerge from the pandemic on the back of a green recovery fueled by government stimulus which consists of investment towards sustainability, renewable energy power generation and low-carbon modes of transportation, such as electric vehicles (EVs). The consumption of refined copper in its variety of applications is expected to increase at a compound average growth rate (CAGR) of 2.3% between 2020 and 2040, requiring significant primary mine development over the period. This energy transition signifies a material demand shift that requires copper to contribute to society's development well into the future.

19.2 Global Production

Since 1900, demand for refined copper increased from less than 0.5 Mt to an expectation of over 24 Mt in 2021. Demand for primary copper is expected to grow at an average rate of 2% per annum over the next 20 years representing an incremental rise of 9 Mt over this period. Figure 19-1 describes the distribution of copper consumption by region of the world and end uses.

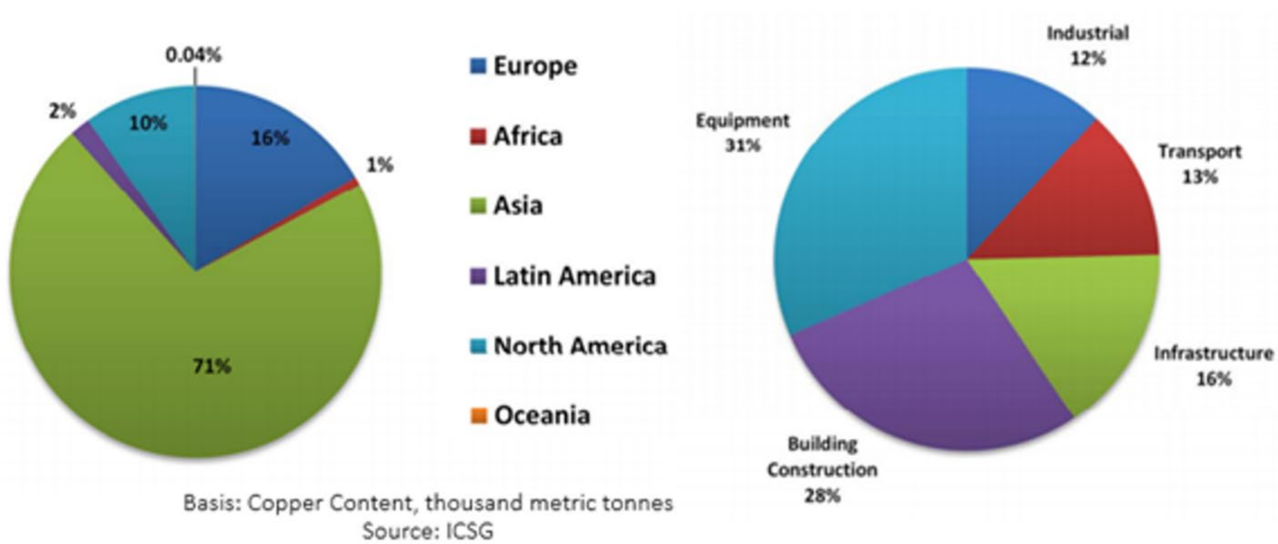


Figure 19-1: Major Uses of Copper: Usage by Region and End-Use Sector – 2019

19.3 Copper Concentrate Market

Smelter production capability (concentrate demand) forecast for 2021 is 21.96 Mt (Cu contained), representing a potential 8.8% increase from 2020. Smelter output capability growth is expected to continue at an overall CAGR of 4.0% over the next five years. However, with the timing of some projects remaining uncertain, mines subject to disruptions, and a lack of concentrates and secondary feed material available, actual smelter production is expected to be restricted to only 20.63 Mt. Global smelter production capability is forecast to grow by an average of 0.9% per annum through to 2040 totalling 4.1 Mt of additional smelter production capability, with over 60% of that coming from China.

19.4 Supply

Global mine production is expected to continue to grow during 2021, reaching 22 Mt, 5.7% higher than the total achieved during 2020. Additions to mine capacity from 2020 through 2025 represent a CAGR of 2.7%. Beyond 2025, a significant supply shortfall of 16 Mt by 2040 is expected to arise as mine supply growth slows relative to demand, and base case mine supply growth faltering due to grade attrition and reserve depletion. The attrition at existing mines is expected to result in a decline of 13.8 Mt by 2040, a CAGR of 2.0%. The mining industry will require new investment to meet these expanding requirements, especially given the seven to 10-year time horizon required to bring a new mine into production. As a result, beyond 2025, the primary market will return to a deficit unless mine projects are developed.

19.5 Copper Prices

Copper has come under a spotlight in the commodities market as the post-pandemic recovery is expected to be extremely copper intensive. Government stimulus aimed at low carbon technologies to move away from traditional energy sources are boosting copper's growth trajectory as it is a vital component needed to implement the shift into renewable energy technologies. Demand for copper is expected to rise as governments gear investments towards sustainability, while supply constraints will drive prices higher. As of now, over the forecast period to 2040, it is expected that the market will not have enough copper supply to meet increasing demand leading to a sustained rise in prices.

19.6 Treatment Charge / Refining Charge

China is the leading importer of copper concentrate and has the greatest capacity for smelting and refining. As a result, Chinese smelters will negotiate and set the marginal cost in the form of an annual Benchmark Treatment and Refining Charge. The annual treatment charge / refining charge (TC/RC) in 2020 was $62/6.2. For 2021, the annual TC/RC decreased to $59.5/5.95 because of reduced supply expectations.TC/RC's are expected to be similar in 2022 with an incremental increase into 2023 and 2024 as new mine supply enters the market.

19.7 Pinto Valley Mine

PVM's current and long term copper concentrate production forecast is 190 to 273 dry kilotonnes per year. The copper concentrate has numerous potential destinations; smelters in Arizona (domestic) and smelters internationally, mostly in Asia due to the geographical location, but material has also been shipped to further destinations such as Europe in the past. The domestic bound concentrate is trucked directly to the smelters from PVM. The international bound concentrate is trucked in rotainers directly to the Port of Guaymas in western Mexico, and then loaded onto ships destined for the receiving smelter.

19.8 Markets and Contracts

The quality of the copper concentrate produced at PVM is highly desired by traders and smelters due to an above-average copper content and lack of deleterious elements. The proliferation of high arsenic bearing concentrates in the market has only strengthened the demand for concentrates of PVM quality.

Contracting strategy will be based on the existing contracts of varying tenure with an emphasis on increasing domestic sales providing a higher NSR. Copper concentrate will be sold internationally and to smelters in North America (domestic).

In all cases, the concentrate price will be based on the published London Metals Exchange prices averaged over the applicable quotational period for the payable element (copper), less applicable TC/RC.

For the molybdenum sulfide concentrate, sales are being made on an ad hoc basis to interested parties. The molybdenum concentrate price is based on the published molybdenum oxide price, less a discount to cover the costs of converting the sulfide material to oxide through roasting as well as logistics costs.

The full annual production of copper cathode has been sold through a competitive tendering process to a single buyer. Cathode pricing is based on the COMEX or London Metals Exchange pricing for copper cathode at the applicable quotational period for the monthly deliveries.

20 Environmental Studies, Permitting and Social or Community Impact

This section summarizes the environmental issues identified on site, existing permit status, environmental monitoring, and permitting efforts that may be required throughout the life of mine presented in this study. PVM's health and safety, and social and community programs are also discussed.

20.1 Current Environmental and Regulatory Context

20.1.1 Key Environmental Regulations and Permits

The principal permitting agencies providing oversight of the operation and closure of PVM are the Arizona Department of Environmental Quality (ADEQ), the Arizona State Mine Inspector's Office (ASMIO), the Arizona Department of Water Resources (ADWR), and the USFS.

The majority of the environmental permits for the operation are in effect for the life of mine (or until substantive changes are needed). Exceptions are the Arizona Pollutant Discharge Elimination System (AZPDES) Individual permit, and Multi-Sector General Permit (MSGP) for stormwater discharge and the Air Pollution Control Permit, which are included in permit programs that require renewal by ADEQ every five years.

Other minor permits, registrations, and licenses required for operation include a federal MSHA registration number, Arizona state mining ID number, hazardous materials and hazardous waste ID numbers, solid waste management inventory number, waste water, leach field, septic permits, radio and other communications licenses, blasting operator registrations, exploration/well drilling permits, and laboratory/nuclear instrumentation licenses and registrations.

As described in detail in Section 20.3, the USFS is currently completing a National Environmental Policy Act (NEPA) review of PVM's Mine Plan of Operations (MPO), which describes existing and future use of National Forest System (NFS) lands surrounding the private PVM property on which the majority of mining activities occur. The Final Environmental Impact Statement (FEIS) and Draft Record of Decision (ROD) were published in April 2021; the final ROD is scheduled to be published in September 2021, and the USFS authorization to proceed with the mining activities described in the final MPO is expected in October 2021.

Status of the environmental permitting for the operation are discussed in Section 20.3, below.

20.1.2 Site Management Systems and Training

Environmental systems are in place at PVM to ensure that all permit compliance monitoring and reporting obligations are properly managed, and that the staff have the relevant training to ensure the programs and compliance requirements are implemented. Workflow management of routine inspections, sampling events, and reporting are coordinated by the environmental superintendent. All employees receive environmental orientation training related to air quality, groundwater, stormwater, hazardous materials, and waste management; environmental department staff receive specialized environmental and regulatory training from internal and external sources.

20.1.3 Site Health, Safety, and Industrial Hygiene Program

Safety is at the heart of PVM's business philosophy and "work responsibly" is one of Capstone's four core values. Safety is built into all levels of the business, at the highest level it is part of Capstone's "Code of Conduct" and governance through Capstone Environment, Health, Safety and Sustainability (EHSS) Policy overseen by an EHSS Committee. Each new employee is required to make a commitment to safety acknowledged by signing the Code of Conduct.

PVM is subject to health and safety regulations under the supervision of the Mine Safety and Health Administration (MSHA), ASMIO, Arizona Department of Transportation (ADOT), and other federal and state agencies. New miner training, annual refresher training and training certification to operate specialized equipment is handled internally at site. Specific health and safety plans and traffic management plans will be developed for projects that are outside of routine operations throughout the mine life.

Safety incidents and accidents are reported via an integrated internal notification system. Incidents, property damage, and injuries are investigated by the area supervisors, assisted by PVM's Health, Safety and Environment (HSE) department and other relevant internal personnel, to review the causes, and develop preventative measures and remedial plans.

The safety procedures and personal protective equipment requirements for routine work, emergency reporting and response protocols, and training such as risk review procedures and environmental training are well documented and reinforced by the HSE department as part of employee training. The HSE department also oversees industrial hygiene-related programs such as hearing conservation programs that test employee hearing, monitor noise levels, and work with operations, as required, to mitigate excessive noise. The department activities include a respiratory protection program, fatigue management, annual medical examinations, the "back-to-work" assessment following a workplace illness or injury, the drug and alcohol program, fit for duty program, and new employee medical screenings.

20.1.4 Site Characterization Studies to Support Environmental Permitting

Site studies have been completed to support prior and current state and federal environmental permit approvals and authorizations. The characterization studies include: climate, groundwater, stormwater, surface water, ore and gangue mineral types and metal concentrations, geochemical behavior of relevant rock and wastes (i.e. non-mineralized formations, waste rock, leach ore, tailings, and pit walls), process solutions, biological and cultural resources, waters of the United States (WOTUS) delineations, and plant species suitable for revegetation. In addition to these studies, the 2021 Final EIS prepared by the USFS includes a detailed discussion on recent studies related to: air quality, biological resources, greenhouse gases and climate change, cultural resources, resources of tribal interest, environmental justice, fire and fuels management, geology minerals and geotechnical stability, paleontology, hazardous and nonhazardous materials, land ownership, livestock grazing, noise, public health and safety, recreation and wilderness, socioeconomic conditions, soils, traffic and transportation, visual resources, water resources and hydrogeochemistry (USFS, 2021).

20.2 Environmental Issues, Monitoring, and Management

The environmental monitoring and management is driven by federal and state regulatory requirements and Capstone's commitment to environmental stewardship and reducing impacts of the operation on the environment. Systems are in place to ensure that all permit compliance monitoring and reporting obligations are properly managed. A summary of the monitoring and management program is presented below.

20.2.1 Groundwater

Water quality analyses of various constituents are required on a quarterly, annual, and biennial basis from designated seeps/springs, and Point-of-Compliance (POC) wells. Routine self-monitoring report forms are submitted or uploaded to ADEQ with the results of water quality monitoring and site inspections.

Aquifer Protection Permit (APP) compliance reporting requires an annual demonstration of the adequacy of pit containment (i.e. the open pit is a sink with evaporation exceeding the groundwater flowing inwards from surrounding areas into the open pit). A comprehensive groundwater report is required by ADEQ every five years to assess adequacy of POC wells and the passive hydraulic containment in the open pit.

The USFS will require groundwater monitoring as a mitigation measure in conjunction with approval of the MPO. The Comprehensive Water Resources Monitoring and Mitigation Plan (appended to the MPO) describes the groundwater quality and water level monitoring that will be implemented following the USFS authorization to proceed with the mining activities described in the final MPO. The monitoring program will utilize components of the existing ADEQ-required monitoring described above, as well as new monitoring wells. Results of this monitoring program would be reported to the USFS annually, along with the surface water monitoring described below.

20.2.2 Surface Water, Stormwater, Process Water, and Wastewater

Monitoring and sampling of mixed stormwater and seepage water discharges at specific outfall locations and seeps that could enter Pinto Creek is required by AZPDES Permit No. AZ0020401; discharge limits are set for a number of constituents. Site facility maps are updated regularly to show seepage zones, constructed and natural drainage structures, pipelines, pump stations and pump capacities, and all monitoring points associated with Clean Water Act compliance. Best management practices (BMPs) have been established for all non-discharge containment structures. Routine inspections and maintenance are conducted to ensure BMPs are maintained.

The USFS will also require surface water monitoring as a mitigation measure in conjunction with the MPO approval. The Comprehensive Water Resources Monitoring and Mitigation Plan (appended to the MPO) describes the surface water quality and flow monitoring that will be implemented following the USFS authorization to proceed with the mining activities described in the final MPO. The monitoring program will utilize components of the existing ADEQ-required monitoring described above, as well as new monitoring points and whole effluent toxicity (WET) testing. Results of this monitoring program would be reported to the USFS annually, along with the groundwater monitoring described above.

The wastewater treatment plant (WWTP) is authorized to operate with a maximum daily flow of 25,000 gallons per day. The WWTP system has compliance monitoring requirements and discharge limits set for fecal coliform and total nitrogen as measured downgradient of the chlorination tank on the effluent line.

20.2.3 Air Quality and Abatement

As part of the mine's synthetic minor air quality permit, specific controls, including dust collectors, electro-static precipitators and water sprays are used at key areas of the facility to reduce the generation and distribution of air pollutants. The permit requires regular monitoring of the air quality control equipment and air quality at the facility.

20.2.4 Noise Monitoring and Abatement

Maintaining a healthy environment with respect to proper lighting, and acceptable noise and vibration levels is part of the safety program. The facilities and equipment operating in high noise environments have been designed to reduce noise to the best extent possible through use of structural barriers or noise reduction materials. Employees are required to wear hearing-protective equipment appropriate to the noise level and duration of exposure. The HSE department maintains a Hearing Conservation program and works with operations, as appropriate, to mitigate excessive noise.

20.2.5 Tailings Storage

Per current APP requirements, a maximum disposal rate of 29 million tonnes (32 million short tons) is permitted to be disposed on TSF3 and TSF4 on an annual basis, and design heights are in effect for both facilities. Limited capacity remains on TSF3 so discharge to this facility is capped at a specified number of days per year. The APP has been modified to establish a new design height of 4,250-ft elevation for TSF4. TSF3 and TSF4 are inspected quarterly and after significant rainfall events for evidence of crest failure, toe slippage, substantial cracks, and erosion features. TSF1 has been reclaimed and is inspected for seepage and erosion features. Wood performs an annual inspection of all impoundments and prepares a report that is written and stamped by an Arizona-registered engineer and submitted to ADEQ. Associated seepage toe drains and caissons are to be kept free of debris, sediments, vegetation, and other obstructions.

Environmental monitoring for the expanded TSF4 does not require additional downgradient POC wells or surface water discharge point monitoring from the seepage and stormwater collection ponds during the operational period. The USFS will require seepage monitoring during the post-closure period as a mitigation measure for Capstone's use of NFS lands. The Post-Closure Tailings Seepage Management and Mitigation Plan describes the seepage water quality monitoring that will be implemented following mine closure, pursuant to the USFS authorization to proceed with the mining activities described in the final MPO.

Any new monitoring related to the expanded facilities, including compliance with tailings disposal and design height limits, will be conducted as required by the relevant governmental agencies.

20.2.6 Waste Rock Storage

Waste rock is currently placed in the Main Dump on top of the leach piles. An additional dump, planned to be built starting in approximately 2023, was permitted through an amendment to the

APP. There are a number of historic waste rock storage facilities at PVM, some of which have been consolidated through time into single facilities. One facility (Southside Dump 14) is in post-reclamation status. Each facility has a specific design storage capacity limit and is to be managed to reduce acid rock drainage.

20.2.7 Hazardous, Regulated, and Solid Wastes

Hazardous and regulated wastes are collected and stored on site prior to shipment to a recycling facility or permitted waste disposal facility as appropriate. Chemicals brought on site must be approved in advance by PVM.

A solid waste landfill facility to dispose of construction debris is located within the footprint of the Northside Dump 9.3, west of the open pit and north of the North Barn. Capstone uses standard industry methods to cover and compact active dump areas to eliminate blowing debris, and disease vectors. PVM has a scrap program in place to minimize waste placed in the solid waste landfill. Office and non-hazardous shop refuse is collected by a local hauler.

20.2.8 Process and Stormwater Ponds / Catchment Berms

The PLS and Raffinate ponds, and stormwater ponds, basins and catchments are inspected on a routine frequency and after significant storm events. Inspection results and required repairs are documented.

20.2.9 Remaining Evaluations

Selected data have been collected and studies updated to reflect changes expected for the remaining life of mine and to support PV3 permit approvals. These include data or studies related to climate, biological and cultural resources, WOTUS, groundwater, and geochemical behavior of representative PV3 tailings and waste rock. If Capstone elects to further optimize the resource or revises the LOMP designs considered in this report, additional environmental evaluations and permitting may be required.

20.3 Environmental Permit Review Related to Ongoing Operations

The following analysis is a brief review of current federal and state permits and requirements with respect to new, expanding and ongoing operations and closure planning at PVM. Any potential actions or permit amendments that will be required are noted.

20.3.1 Federal Permitting

20.3.1.1 Mining Plan of Operations (USFS)

PVM's existing operations on NFS land are conducted pursuant to a series of prior issued plans of operation, rights of way and special use authorizations. Continuation and expansion of those operations are the subject of a pending MPO approval. The pending MPO will consolidate all prior approved plans of operations and certain expired rights-of-way and special use authorizations for water and power lines in addition to addressing legacy third-party encroachments. In addition, the MPO will authorize the planned expansion of the mine pit, TSF3 and TSF4 onto NFS lands.

A proposed MPO was submitted to the USFS in May 2016, and has been the subject of an ongoing NEPA evaluation. USFS approval of a final MPO is expected in October 2021. The final MPO also includes a Reclamation Plan addressing operations on NFS lands disturbed by PVM activities which also incorporates relevant components of state-required reclamation plans summarized in Section 20.3.2.

The NEPA analysis included public involvement (through scoping and comment on a published draft EIS) and addressed a number of environmental resource areas (e.g., endangered species, historic properties, air quality, water quantity and quality, etc.). The FEIS and draft ROD were published in April 2021; the final ROD is scheduled to be published in September 2021. The NEPA process also requires that the lead agency (in this case, the USFS) develop alternatives to the proposed action. The two alternatives considered by the USFS were the “no-action” alternative, which would consist of no use of NFS lands, and Alternative 1, which would consist of continuation of existing authorized uses, but no new expansion of the Open Pit, TSF3 and TSF4.

MPO approval also included USFS consultations with the relevant agencies and entities under the Endangered Species Act and the National Historic Preservation Act (discussed in Sections 20.3.1.4 and 20.3.1.5, respectively). Those consultations have been completed.

20.3.1.2 Forest Road Use

Currently, FR 287, also commonly referred to as the Pinto Valley Road, provides access to the mine site and administrative facilities from US Highway 60. FR 287 is a public road that traverses NFS land and the PVM property. It provides public access to ranches and for recreation uses of adjacent Tonto National Forest (TNF) land. Public use of the segment through PVM’s private land is authorized by a reserved easement and is maintained by PVM. This segment of the road may be relocated from its current alignment to accommodate mining activities, if needed.

The pending MPO obligates the USFS to issue PVMC a road use permit containing the relevant terms and conditions for continued use and maintenance of the paved portion of FR 287 on NFS land. On an annual basis, PVMC and the USFS will meet to discuss and if needed, update the road use permit for all permitted maintenance and commercial uses of the paved portion of FR 287 including road conditions, safety, signage and will work together to identify and address any issues.

20.3.1.3 Clean Water Act Compliance

Placement of dredged or fill material into “waters of the United States” (WOTUS) would require authorization by the USACE pursuant to the agency’s authority under Section 404 of the Clean Water Act (CWA). Surface water features within the footprints of waste rock dumps, TSF3 or TSF4 expansion were evaluated for the presence of jurisdictional WOTUS under the CWA. An Approved Jurisdictional Delineation (AJD) was issued by USACE. The USACE concurred thatno currently planned facilities would impact any WOTUS. If it is determined that any future operation will impact WOTUS, a Section 404 permit application will be submitted.

20.3.1.4 Endangered Species Act Compliance

Impacts to threatened or endangered species listed by the U.S. Fish and Wildlife Service (USFWS) under the Endangered Species Act (ESA) must be evaluated as part of USFS’s MPO approval process. The USFS completed a Biological Assessment of the project in 2020, and found that five species listed under the ESA had a potential to occur at or near the PVM:

- Ocelot
- Arizona hedgehog cactus
- Southwestern willow flycatcher
- Western yellow-billed cuckoo
- Gila topminnow

USFS concluded that the project "may affect, but is not likely to adversely affect" each of these species except the Gila topminnow, which would not be affected by the project.

USFS also identified designated critical habitat (finalized in April 2021) for the western distinct population segment of the yellow-billed cuckoo along certain segments of Pinto Creek. The agency determined that the project would "not [be] likely to result in destruction or adverse modification" of the critical habitat.

The USFWS concurred with the USFS determinations in a Biological and Conference Opinion conducted in accordance with the ESA Section 7 consultation process.

The USFS will require biological resources monitoring as a mitigation measure . The Biological Resources Monitoring and Mitigation Plan describes the program that will be implemented following the USFS authorization to proceed with the mining activities described in the final MPO. The monitoring program includes survey requirements for western yellow-billed cuckoo and Arizona hedgehog cactus in locations where these species may occur.

20.3.1.5 National Historic Preservation Act Compliance

Impacts to cultural resources that are eligible for listing on the National Register of Historic Places (NRHP) must be evaluated as part of the USFS's MPO approval process. Cultural resource surveys have been completed to support ongoing planning on both private and public lands at PVM and in the vicinity. There are numerous cultural resources that are considered eligible for the NRHP and would be impacted by the project. PVMC prepared a Historic Properties Treatment Plan (HPTP) that describes the data recovery effort for those sites. The HPTP was developed in accordance with the requirements of Section 106 of the National Historic Preservation Act, in consultation with the USFS, Advisory Council on Historic Preservation (ACHP), Arizona State Historic Preservation Office (SHPO), and Native American Tribes. Per a Memorandum of Agreement (MOA) executed by the PVMC, USFS, and SHPO, the HPTP will be implemented following the USFS authorization to proceed with the mining activities described in the final MPO.

20.3.2 State Permitting

20.3.2.1 Aquifer Protection Permit (APP)

Several significant amendments to the APP were completed to address facility expansion and modifications within the private and NFS lands. A perspective view showing the proposed facilities is shown in Figure 20-1.

The changes included design modifications to existing facilities, addition of new facilities, closure or removal of permitted facilities that are no longer needed, updates to BADCT demonstrations, updates to the closure and post-closure strategy and the site-wide closure and post-closure cost estimates. The following permit modifications were completed:

- Design and height limit for TSF4 and update the BADCT demonstration;
- Design and BADCT demonstration for changes to TSF4 boundary dams to consolidate the number of boundary dams from four to two and for their construction by placement of earthen material or cyclone sand;
- Design for Main Dump and update the BADCT demonstration;
- Design and BADCT demonstration for new waste rock dump (West Dump) to be placed in Gold Gulch;
- Design and BADCT demonstration for a PLS draindown capture facility to replace the decommissioned Gold Gulch PLS Pond 1A; and
- Update the site-wide closure and post-closure strategy and costs.

20.3.2.2 Arizona Pollutant Discharge Elimination System (AZPDES)

PVM currently holds an individual AZPDES permit for process water discharges to downstream receiving waters. No additional process water discharges are proposed a and the AZPDES permit modification is anticipated to be secured.

20.3.2.3 Arizona Multi-Sector General Stormwater Permit (AZMSG)

The existing Storm Water Pollution Prevention Plan (SWPPP) will be modified to include the LOM facilities and BMPs. Stormwater runoff from the Main Dump remain mainly within the non-discharging Open Pit Basin as designated on the SWPPP map; stormwater run-on flows to this new dump will be minimized through use of berms and/or ditches. PVM’s SWPPP is reviewed at least annually to update the location of seepage zones, pump locations, stormwater berms/diversion ditches, and any new or expanded facilities, as needed.

20.3.2.4 CWA Section 401 Water Quality Certification

It was determined that PV3 expansion would not result in discharges to navigable waters of the U.S., none of the expanded activities being authorized in the MPO will result in new point sources discharges to navigable waters requiring a permit under Section 402 of the CWA, that no CWA Section 404 permit is required, and therefore no Section 401 water quality certification is required.

20.3.2.5 State Antiquity Regulations

State regulations prescribe specific obligations with regard to the discovery of human remains and/or funerary objects during ground-disturbing activities. If human remains and/or funerary objects are encountered as part of construction activities, work in these areas must be stopped until the remains can be properly recovered and repatriated. Any such discoveries must be treated in compliance with Arizona Revised Statute 41-865 under the jurisdictional authority of the Director of the Arizona State Museum (ASM). For cultural resources work on private land, a burial agreement for the treatment and disposition of human remains, funerary objects, sacred objects, objects of cultural patrimony, Pinto Valley Mine MOA Page 8 objects of tribal patrimony, or formal non-human burials has been developed in coordination with ASM.

20.3.2.6 Air Quality Control Permit

PVM currently operates under a Clean Air Act (CAA) Class II "Synthetic Minor" air quality control permit. To operate under a Class II Synthetic Minor permit, PVM agreed to limit throughput and operate control devices to effectively manage emissions and ensure that no Criteria Pollutants generated exceed 100 tons per year. The current permit allows for PVM to operate up to 65,300 tonnes tons per day (72,000 tonnes) with the existing equipment. The throughput of 56,000 tonnes per day requires no major equipment changes to the plant, resulting in the current permit being adequate to maintain operations for the life of the mine.

20.3.2.7 Dam Safety Permits

Dam Safety permits and water withdrawal permits are administered by the ADWR. Tailings dams are explicitly excluded from jurisdiction and no new dam safety permits are anticipated. Two existing ADWR permitted dam structures within the Gold Gulch that impound PLS and surface water will be modified or withdrawn, as appropriate.

Any wells and borings requiring removal for execution of the LOMP will be abandoned per ADWR abandonment requirements.

20.3.2 Mined Land Reclamation Plan

PVM is subject to closure and post-closure reclamation requirements by ASMIO. The original Mined Land Reclamation Plan (MLRP) for PVM was approved in 1998 and has been updated, as needed, to reflect substantive changes to the LOMP and closure designs. The MLRP was modified in 2016 to update the LOM disturbance acreage and the reclamation costs. Reports tabulating new disturbance and new reclamation completed during the year are filed annually with ASMIO.

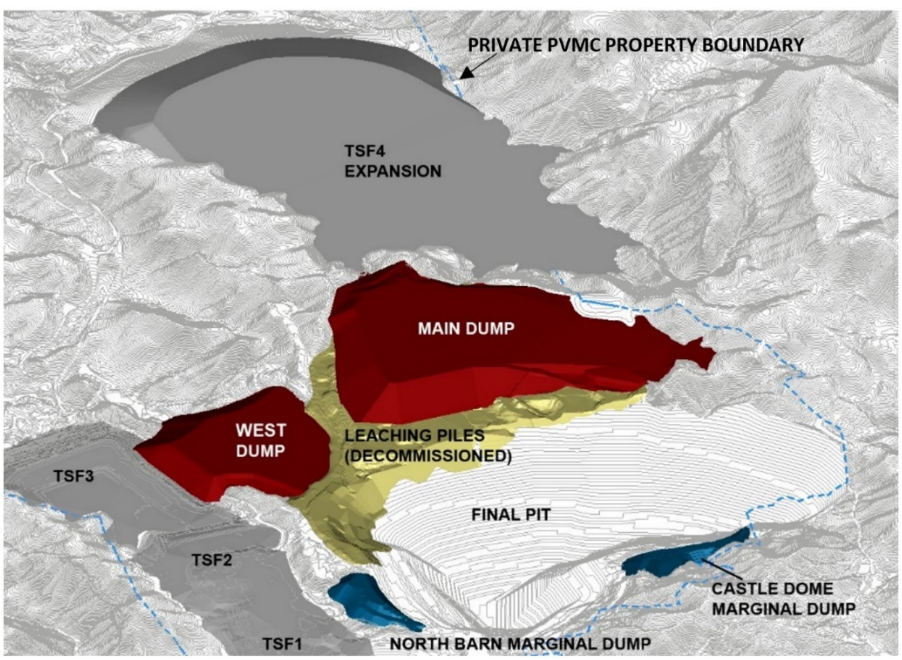


Figure 20-1: Perspective view looking north, -30 degrees at planned LOM waste rock, leaching, and tailings facilities. Future decommissioned LP (gold), existing and expanded tailings (gray), marginal grade dumps (blue) and planned LOM waste rock (red)

20.3.2.1 Certificate of Environmental Compatibility

It is anticipated that the mine life extension will not require a new substation or modification of the existing high-voltage transmission line.

20.3.3 Summary of Key Permitting Considerations

A summary of key permitting considerations is provided in Table 20-1. As noted in the table, USFS MPO authorization is the critical path schedule driver, and involves agency approval and actions that are beyond Capstone control.

Table 20-1: Summary of Key Permitting Considerations.

Permit Effort	Agency	Assumptions/ Key Considerations	Estimated Timeframe	Schedule Start Point
Federal				
Mine Plan of Operations (including NEPA review)	USFS	NEPA evaluation has been completed and the final ROD is expected in October 2021.	6 months [from April 2021]	Under way.
Forest Road Use Permit	USFS	The USFS will issue a Road Use Permit for future cost share of maintenance related to commercial use of the paved portion of FR 287.	6 months	Under way.
Clean Water Act Compliance	USACE	Current planned activities do not require a CWA Section 404 permit.	NA	NA
Endangered Species Act Compliance	Lead federal agency and USFWS	Formal consultation for potential impacts to ESA listed species completed, Biological and Conference Opinion issued by USWFS.	NA	NA
National Historic Preservation Act Compliance	Lead federal agency and SHPO	Formal consultation for potential impacts to NRHP-eligible sites completed. HPTP developed and approved. Data recovery will need to be completed before sites are disturbed.	1-2 years for final data recovery report; however, construction would be permitted following data recovery and an end-of-field work report.	Data recovery may begin upon receipt of USFS MPO authorization to proceed.
State				

Aquifer Protection Permit	ADEQ	APP amendment for waste rock and tailings facilities completed.	NA	NA
Individual AZPDES Permit	ADEQ	Existing permits are in place, no amendments required.	NA	NA
Arizona Multi-Sector General Stormwater Permit (AZMSGP)	ADEQ	The current AZMSG-2019-002 expires on December 31, 2024.	NA	NA
Air Quality Control Permit	ADEQ	No amendments required.	NA	NA
Dam Safety Permit	ADWR	PV3 will not utilize jurisdictional impoundments (greater than 25 ft embankment height or greater than 50 ac-ft storage capacity). Two existing ADWR permitted dam structures within the Gold Gulch that impound PLS and surface water will be modified or withdrawn, as appropriate.	NA 6 months	NA Underway
Mined Land Reclamation Plan	ASMIO	MLRP was updated in 2016 and will be updated again following issuance of the USFS MPO but is not required for authorization to proceed.	NA	NA
Certificate of Environmental Compatibility	Arizona Corporation Commission	Not required.	NA	NA

20.4 Mine Closure and Reclamation

Conceptual closure designs and cost estimates are based on the current LOMP for mining and waste disposal facilities.

20.4.1 Closure and Reclamation Plans

General plans for end-of-life closure and post-closure activities is summarized in the *Closure and Post-Closure Strategy*, the MLRP, and the MPO *Reclamation Plan*. These documents address the general methods and goals for site closure and reclamation within guidance and requirements provided by state and federal agencies (ADEQ, ASMIO, USFS). Each agency focuses on its specific regulatory responsibilities. Detailed closure plans are required to be submitted to ADEQ prior to any closure construction activities. As part of the MLRP reporting requirements, PVM provides ASMIO with an annual tally of new disturbance and reclamation acreages. The MLRP primarily focuses on ensuring post-mining public safety and the reclamation and revegetation of disturbed lands. The MLRP is updated as needed to reflect substantive changes in facility configurations or cost assumptions. Capstone and the USFS are currently collaborating on a plan for reclamation of disturbance to NFS lands. This *Reclamation Plan* must be completed and approved by the USFS prior to authorization to proceed with the mining activities described in the final MPO, which is expected in October 2021.

20.4.2 Closure and Post-Closure Costs

Closure and post-closure costs are tabulated by permit or approval program. Financial assurance demonstrations for estimated closure and post-closure costs are updated with the relevant agencies (ADEQ, ASMIO, USFS), as needed, to reflect changes in the configuration of mining and waste disposal facilities. The APP-related closure costs address the closure and post-closure monitoring activities related to discharging facilities (tailings, waste rock dumps, leaching pile, process ponds). Post-closure monitoring costs are included in this estimate. As part of the closure activities, 47 wells that will not be needed for post-closure monitoring would be abandoned. The MLRP-related closure costs address closure activities related to ensuring public safety, facility regrading, and revegetation.

APP-related closure costs were last updated and approved by ADEQ in April 2021 to reflect mine expansion on private land and disturbance through 2039.

Site-wide closure and post-closure costs have been estimated based on the updated LOMP to 2039 discussed in this Technical Report. The costs were calculated based on assumptions documented in the *Closure and Post-Closure Strategy*, regrade designs and quantities prepared by Wood and SRK.

Based on the LOM configuration discussed in this Technical Report, the total closure and post-closure costs for the APP and MLRP programs are estimated to be:

- $102.2 M – ADEQ APP program,
- $15.7 M – ASMI MLRP program,

The cost for PVM closure and reclamation of private land under the APP and MLRP programs is estimated to be $113.5 M including $97.8 M in closure costs and $4.4 M in post-closure costs, and $39.8 M in Owner's Costs. Post-closure costs include site inspections, maintenance, monitoring. Owner's costs include internal G&A and labor to support the 30-year post-closure period as well as closure designs, environmental studies and permit amendments and routine environmental reporting, as needed.

As noted above, Capstone and the USFS are currently collaborating on a plan for reclamation of disturbance to NFS lands. This Reclamation Plan and reclamation cost estimate must be completed and approved by the USFS prior to issuance of the notice to proceed with the mining activities described in the final MPO. PVMC will also be required to provide a satisfactory financial assurance mechanism to the USFS for the estimated reclamation cost.

Capstone has reviewed the estimated closure and post-closure costs and believe the cost estimates are reasonable.

20.5 Social and Community

Capstone is committed to its employees and to the communities in which it works to operate under high standards of corporate environmental and social responsibility (Capstone, 2014; 2015). PVM operates in accordance with recognized industry standards while complying with local and applicable regulations and laws.

PVM has established relationships with its communities of interest and stakeholders and assigns dedicated personnel to this aspect of its business. Communication channels are in place, and forums for direct interaction with stakeholders are held as required. Arizona's political climate is stable and the state has a long history of copper resource development.

All levels of management and staff participate in community involvement initiatives, community affairs personnel manage and track communication with stakeholders, ensuring timely responses to community needs. Engagement with community stakeholders is proscribed according to PVM's *Community Engagement Procedure*. The procedure outlines stakeholder identification, documentation processes for stakeholder engagement, communication strategies for information requests and distributing information, donations, sponsorships and employee support, employee involvement, memberships, documentation policies for grievances/complaints, and key roles within the organization with respect to community engagement. Additional to the procedure are a stakeholder register and a stakeholder analysis log containing a record of communications with stakeholders.

PVM has policies and procedures in place to address security and emergency management. Capstone follows the Capstone Code of Conduct for compliance with local regulations and to ensure business ethics in its relationships with its employees, suppliers, vendors, contractor firms, regulators, and local communities. Specific policies include:

- A Whistleblower Policy (Fraud reporting and Investigation);
- A Code of Conduct Policy that outlines the official complaint procedure; and
- An Anti-bribery Policy complements the Code of Conduct with additional guidance on compliance with applicable anti-bribery and corruption laws and regulations.

20.6 Risks

20.6.1 Risks to PVM with respect to Permitting, Environmental Studies and Social and Community Impact

- Delay in receiving USFS authorization for mining activities described in the MPO (expected October 2021).

20.6.2 Risks to PVM with respect to Closure Planning

- Delay or inability to obtain approval for closure plan prior to planned closure activities.

20.7 Opportunities

Opportunities related to permitting, environmental studies and social and community impact include:

- Continue to foster collaborative relationships with stakeholders and peers to maximize benefits of the project;
- Look for opportunities to perform progressive reclamation of tailings storage facility embankments and waste rock dumps.

20.8 Recommendations

The following activites are recommended in order for PVM to be successful in obtaining and keeping its permits to operate, as part of PVM personnel regular duties:

- Prepare a plan to evaluate the impacts of climate change on PVM's ability to operate within permit terms and conditions before the end of 2022;
- Update the MLRP following USFS issuance of a MPO and authorization to proceed; and
- Stay abreast of continuously evolving mining regulatory regime and best practices.

21 Capital and Operating Costs

21.1 Operating Costs

The LOM operating cost for PVM is projected to average $9.94/tonne milled. These costs do not include TC/RC and concentrate transportation costs. Operating costs are detailed in Table 21-1.

Table 21-1: Unit Operating Cost Summary

Item	Units	Life of Mine Average Cost
Mining Cost	$/t moved	1.68
Mining Cost	$/t milled	3.26
Milling Cost	$/t milled	4.67
Operations Support	$/t milled	0.88
G&A Cost	$/t milled	1.13
Total	**$/t milled**	**9.94**

21.1.1 Mine Operating Costs

Mine operating costs were estimated for this study based on the mine plan and equipment list. The following assumptions were made in calculating the mine operating costs:

- Costs are in 2021 US$.
- Diesel fuel at $2.26 per gallon.
- Explosives at $0.22/t blasted.
- Labor and equipment costs are based on recent historical values and are adjusted for projected trends in major consumable pricing.

Operating costs do not include:

- Planned component replacement program costs, which are capitalized.
- Post mining reclamation costs.
- Process costs from the primary crusher.
- Assay laboratory and assay costs for blast holes.
- Exploration programs

The mine costs average to $1.68/t moved. The breakdown by category is shown in Table 21-2. Life of mine average cost per tonne milled is $3.26.

Table 21-2: Mine Unit Cost Summary

Cost Type	Life of Mine Average Cost
	US$/tonne mined
Drilling – Operations and Maintenance	0.09

Blasting – Operations	0.24
Loading – Operations and Maintenance	0.28
Hauling – Operations and Maintenance	0.39
Mine Ancillary – Operations and Maintenance	0.10
Mine – Administration	0.56
Mine – LME Heavy Equipment Maintenance	0.02
Total	**1.68**

21.1.2 Plant Operating Costs

The mill operating cost estimates include all costs related to the process facilities, including the primary/secondary/tertiary crushing, mill, and concentrate. The budgets are based on current operating conditions, with details for power consumption and costs, consumables (including wear materials and reagents) and direct and indirect labor costs. The unit cost at 56,000 tpd is $4.67/tonne milled. The cost breakdown of the LOM cost is shown in Table 21-3.

- Costs are in 2021 US$
- Power cost is $0.064/kWh
- Labor and non-capitalized maintenance costs are based on recent operational values

Table 21-3: Process Operating Cost Summary

Cost Type	Life of Mine Average Cost (US$/tonne ore)
Workforce	1.06
Mechanical and Electrical Parts	0.85
Liners and Grinding Media	0.84
Power	0.79
Contractors and Consultants	0.71
Reagents and Water	0.33
Diesel, Gas and Lubricants	0.07
Other G&A Expenses	0.01
Tires & Accessories	0.01
Drill parts & Explosives	0.00
Total	**4.67**

21.1.3 Operations Support Costs

The Operations Support OPEX costs include tailings distribution and pumping costs, outlying areas, assay lab, and light vehicle maintenance. Hydrometallurgy costs are also included in this category but may not continue for the full LOM presented in this report. A value of $0.88/tonne milled is applied for the LOM, and is based upon recent historical values.

21.1.4 General and Administrative Costs

General and administrative costs are estimated to average $1.13/tonne milled. This amount is based upon recent historical expenditures.

21.2 Capital Costs

Life of mine capital costs have been estimated for the continued operation of PVM through 2039, as shown in Table 21-4.

Table 21-4: Capex Cost Summary

Cost Type	Units	Life of Mine Total
Site Sustaining	US$M	100.2
Mine Sustaining	US$M	379.6
Expansionary	US$M	76.0
Total	**US$M**	**555.8**

The $555.8M capital costs equate to $0.25/lb of payable copper over the LOM.

21.2.1 Site Sustaining Capital

The plant and site sustaining capital costs have been estimated to total $100.2 M. These costs cover capital to maintain the mill, tailings, site infrastructure, light vehicles and water systems as well as permitting and engineering costs related to executing the longer mine plan.

21.2.2 Mine Sustaining Capital

Sustaining capital costs for the mine have been estimated to total $379.6 M, largely comprising additions and replacement of mining fleet, planned component replacements and associated support services. The PVM mining fleet was summarized in Section 16.

21.2.3 Expansionary Capital

Expansionary capital costs for PVM have been estimated to total $76.0 M, including upgrades to the mill to ensure it can consistently achieve the planned throughput throughout the life of mine, and relocation of the PLS pond to allow for the construction of the West Dump in Gold Gulch.

22 Economic Analysis

As Pinto Valley Mine is a producing mine and no material expansion of current production is proposed, an economic analysis is not required for this Technical Report. A positive economic outcome of the Mineral Reserve was confirmed by QP Clay Craig, P.Eng.

Mineral Resources are not used during economic analysis, although the Mineral Resources are reported under the constraints of Reasonable Prospects of Economic Extraction as described in Section 14.

23 Adjacent Properties

23.1 Adjacent Properties

The PVM site is in proximity to KGHM's Carlota Mine adjacent to PVM, BHP's inactive operations in the Globe-Miami area, and Freeport-McMoRan's Miami operation. The sources of the information included in this section are historic records, published reports, and public websites as well as publicly disclosed information by KGHM International Ltd. and Freeport-McMoRan Inc. (FMI). The QP has been unable to verify the information and that the information herein is not necessarily indicative of the mineralization on the property that is the subject of the Technical Report.

23.2 Carlota Mine

The Carlota Mine is adjacent to PVM and is under the sole ownership of KGHM Carlota Copper Company, a subsidiary of KGHM International Ltd., which acquired Quadra FNX Mining Ltd. (Quadra) in 2012. Discovered in 1900, mining activity and development of the predominately oxide copper ores at the Carlota Mine progressed through several owners until Quadra purchased the property from Cambior Inc. in 2005. Quadra commissioned the open-pit mine in 2008, and produces copper cathode using ROM dump leach and solvent extraction methods. The Carlota Mine became one of the first copper mines designated and permitted under modern environmental legislation (KGHM, 2014).

Nearing closure and reclamation, the Carlota Mine is operating consistent with the objectives described in the Carlota permits. The mine's timeline for closure is in accordance with the current permits and Arizona environmental regulations. In March 2021, KGHM International Ltd. announced plans to sell Carlota Mine (Kitco, 2021).

23.3 BHP Globe-Miami Area Operations

BHP maintains four closed mining and processing units in the Globe-Miami area east of PVM. The Copper Cities, Miami, and Solitude units are approximately 5 miles east of PVM, north of the Town of Miami. The Old Dominion Unit is 10 miles east of PVM, adjacent to the City of Globe and is a source of water pumped to PVM via the Copper Cities Unit. The Copper Cities Unit consists of two open pit porphyry copper mines (Copper Cities Deep Pit, and the Diamond H Pit) that operated from 1951 to 1975 with associated processing facilities. Current usage for the Diamond H pit is for stormwater and sludge management from treated water; the pit is a reservoir for supplemental process water pumped to PVM under agreement with BHP.

23.4 FMI Miami Operations

The Freeport-McMoRan Inc. (FMI) Miami Operation, located approximately 5 miles east of PVM adjacent to the Town of Miami, includes an open pit copper mine, SX-EW plant, a smelter and a rod mill. Total recorded production (1915-2015) from FMI's Miami operation was 4,217,263 short tons of copper and 2,873 short tons of molybdenum. On August 27, 2015, FMI announced that mining operations at Miami would be discontinued owing to low metal prices. FMI currently produces copper through leaching material already placed on stockpiles, which is expected to continue until 2023. (FMI, 2021)

24 Other Relevant Data and Information

No additional information or explanation is necessary to make the Technical Report understandable and not misleading.

25 Interpretation and Conclusions

Pinto Valley Mine has been successfully operated by Capstone Mining Corp. since 2014. Based on the findings of this Technical report, the QPs believe that PVM is capable of sustaining production through the depletion of the Mineral Reserve. Relevant geological, geotechnical, mining, metallurgical and environmental data from PVM have been reviewed by the QPs to obtain an acceptable level of understanding in assessing the current state of the operations.

25.1 Conclusions

Capstone holds all required mining concessions, surface rights, and rights of way to support mining operations through the depletion of the March 31, 2021 Mineral Reserve estimate. Permits currently held by Capstone and expected to be issued to Capstone in 2021 are sufficient to ensure that mining activities within PVM are carried out within the regulatory framework required by the various levels of government.

Understanding of the regional geology, lithological, structural, and alteration controls of the mineralization at PVM are sufficient to support estimation of the Mineral Resource and Mineral Reserve. The Mineral Resource and Mineral Reserve estimates, copper grade cut-off strategy, and operating and capital cost estimates were generated using industry-accepted methodologies and actual PVM performance standards and operating costs. Metallurgical expectations are reasonable, based on stable metallurgical results generated from actual production data and recently completed studies. Reviews of the environmental, permitting, legal, title, taxation, socio-economic, marketing and political factors for PVM support the declaration of Mineral Reserves.

The Mineral Resource estimate was completed using reasonable and appropriate parameters and is acceptable for use in Mineral Reserve estimation. The confirmation of ‘Reasonable Prospects of Eventual Economic Extraction’ is based on the following assumptions: $3.50/lb Cu, $10.00/lb Mo, 84.6% average Cu recovery, 8.9% average Mo recovery, $1.74/tonne average mining costs, $1.13/tonne G&A costs, $0.88/tonne operational support costs, $4.67/tonne milling costs, and pit slopes by rock type. The Mineral Resource is classified according to CIM (2014) definitions and estimated following CIM (2019) guidelines.

The Mineral Resource, as of March 31, 2021, at a 0.14% Cu cut-off grade, is:

- Measured – 619.9 M Tonnes at 0.33% Cu and 0.006% Mo
- Indicated – 782.5 M Tonnes at 0.26% Cu and 0.005 % Mo
- Inferred – 170.6M Tonnes at 0.26% Cu and 0.006% Mo

The Independent Qualified Person for the Mineral Resource estimate is Mr. Garth D. Kirkham, P.Geo., FGC., of Kirkham Geosystems Ltd.

The Mineral Reserve is based on the following economic assumptions: $3.00/lb Cu, $10.00/lb Mo, 86.0% average Cu recovery, 8.5% average Mo recovery, $1.68/tonne average mining costs, $1.13/tonne G&A costs, $0.88/tonne Ops Support costs, $4.67/tonne milling costs, and pit slopes by rock type. The Mineral Reserve is reported at a variable cut-off ranging from 0.17% to 0.21% copper.

The Mineral Reserve, as of March 31, 2021, at a variable cut-off ranging from 0.17% to 0.21% total copper:

- Proven – 241.6 M Tonnes at 0.34% Cu and 0.007% Mo
- Probable – 139.4 M Tonnes at 0.28% Cu and 0.006% Mo

Clay Craig, P.Eng., Manager, Mining and Evaluations at Capstone Mining Corp. is the Qualified Person for the Mineral Reserve estimate. Mr. Craig is not Independent of Capstone within the meaning of NI 43-101.

PVM has several water sources including a private wellfield with multiple wells, a system of water catchments with pumpback capabilities, and reclaim systems on operating tailings impoundments. Nevertheless, extended periods of drought have the potential to impact the operation of PVM.

The design of existing tailings storage facilities provides adequate capacity to store tailings generated through depletion of the current Mineral Reserve. This assumes that proper tailings management continues, including the development of competent tailings beaches and the management of embankment pore water to allow the continued safe construction of the tailings storage facilities.

Based on current regulations and laws, Capstone has addressed PVM's environmental impact. Closure provisions are appropriately considered in LOMP. There are no known significant environmental, social or permitting issues that are expected to prevent continued mining at PVM.

25.2 Risks

25.2.1 Risks related to the Mineral Resource

The Mineral Resource model is an estimate based on input data with inherent risk, including:

- Interpretation of data and geology, subjective confidence, continuity of grade, lack of continuous ASCu analysis and impact of oxidation.
- Inferred Resources, while not in the Mineral Reserve or LOMP, may be encountered during mining. Inferred Resources for any deposit are relatively uncertain by definition. Additional drilling will be required to more accurately characterize the grades when Inferred Resource material is encountered during mining.

25.2.2 Risks related to the Mineral Reserve

As reserve models are an estimate based on certain assumptions and interpretations, they have certain inherent risks. Risks to the PVM Mineral Reserve as outlined in this report include, but may not be limited to:

- Changes to the Mineral Resource estimate, potentially resulting from revised interpretation and/or the results of additional drilling and sampling.
- Changes to financial assumptions, including metal pricing.

- Significant changes to land tenure or permitting requirements, including anticipated timelines for renewals of permits currently in place.
- Technical challenges such as water supply shortages or geotechnical stability of the open pit or tailings storage facilities.

25.2.3 Risks related to Mining Methods and the Life of Mine Plan

Risks to the mining methods and LOMP presented in this Technical Report include:

- Slope stability issues in specific areas of the PVM pit, including the southwest corner of the pit in the Pinal Schist and the northeast side of the pit near the Bummer Fault, in particular where blasting techniques are not tailored to these areas.
- Availability of skilled labor to achieve mine plan.
- Availability of replacements for aging mine fleet, particularly during 2026-2028 when 789D fleet is reaching optimum lifespan.

25.2.4 Risks related to Mineral Processing and Metallurgical Testing

- The new reagents for the molybdenum circuit have been commercially adopted by other similar plants but there is no guarantee that they will perform as expected when applied at the PVM plant as it is currently configured.

25.2.5 Risks related to Metal Recovery

Key risks to the operation of the PVM plant as planned include:

- The plant was originally built in 1973, given the age of the equipment, unforeseen failures may be a factor.
- Insufficient water supply for prolonged periods of time could affect PVM's ability to process ore economically, or at the rate presented in the LOMP. Future water sources may not be available or available at a suitable cost. PVM's on-going water conservation efforts may not be sufficient if water availability decreases due to climate change or other factors.

25.2.6 Risks relating to Site Infrastructure

- PVM infrastructure such as buildings and facilities are generally in good condition and are suitable for continued usage through the LOMP described in this report, but some facilities are relatively old, and continued investment will be needed to ensure their long-term reliability.

25.2.7 Risks relating to Tailings Management

- The upstream construction method used to raise PVM's TSF3 and TSF4 requires consistent tailings management procedures to ensure the development of competent tailings beaches and to control embankment pore water pressures. If these procedures are not followed, it can jeopardize the feasibility of continued upstream embankment raises, and limit future tailings storage capacity.

25.2.8 Risks relating to Permitting, Environmental Studies and Social and Community Impact and Closure Planning

- Delay in receiving USFS authorization for mining activities described in the MPO (expected October 2021).
- Delay or inability to obtain approval for closure plan prior to planned closure activities.

25.3 Opportunities

The authors of this Technical Report have noted the following opportunities:

- Upgrade classification of the current Inferred Resource to Indicated class by decreasing the drill hole spacing with future drill programs (Garth Kirkham, P.Geo., FGC)
- Continue regional exploration and property evaluations within reasonable trucking distance of the plant. (Garth Kirkham, P.Geo., FGC)
- Evaluate steps required to include gold and silver in the Mineral Resource estimate (Garth Kirkham, P.Geo., FGC) and the Mineral Reserve estimate (Clay Craig, P.Eng.)
- Optimize plant throughput with improvements to the crushing circuit, finer blast fragmentation and the rougher flotation circuit performance (J. Todd Harvey, SME-RM)
- Complete planned studies into technologies that could potentially enhance dump leach performance and increase copper cathode production to 300-350 million pounds from mineralized waste over the LOM, such as the patented catalytic technology from Jetti Resources (expected 2022), coarse particle flotation technology by Eriez Flotation Division (expected the second half of 2021), and pyrite agglomeration (expected 2022). (J. Todd Harvey, SME-RM)
- Improve current site infrastructure through continuous maintenance and possible use of large scale solar power (Colleen Roche, P.Eng.)
- Continue to review the mining fleet, investigate innovative technologies to maximize fuel usage, explore debottlenecking scenarios and complete the blast fragmentation study to optimize operational performance (Clay Craig, P.Eng.)
- Consider alternative tailings storage approaches to allow greater and potentially expand the value of the PVM Mineral Reserve (Clay Craig, P.Eng.)
- Advance tailings management practices by working towards conformity with the *Global Industry Standard on Tailings Management* (GTR, 2020). (Tony J. Freiman, PE)
- Maximize maximize benefits of the project by continuing to foster collaborative relationships with stakeholders and peers (Colleen Roche, P.Eng.)
- Perform progressive reclamation of tailings storage facility embankments and waste rock dumps. (Colleen Roche, P.Eng.)
- Continue to foster collaborative relationships with stakeholders and peers to maximize benefits of the project. (Colleen Roche, P.Eng.)

26 Recommendations

The following recommendations have been identified by the authors of the Technical Report.

26.1 Recommendation related to Sampling (Section 11)

To increase certainty of copper and molybdenum analysis, CRM used should be created from PVM material and should be inserted more frequently in analytical batches with at least 1 per batch of 20 samples (5% insertion rate).

An approximate cost to create CRM from PVM materials is US$ 10,000 to cover preparation including homogenization, round-robin testing and certification. The project can be completed in about three months.

26.2 Recommendations related to Mineral Processing and Metallurgical Testing

The following recommendations are intended to improve the performance of the PVM plant:

- Optimize the grinding circuit to reduce the product P80 size. This has a major impact on copper rougher recovery. Estimated cost for modeling and consulting $50,000, six months duration.
- Investigate the potential benefits of improving the cyclone system to reduce bypass and circulating loads. This should improve grinding efficiency and reduce the P80 product size. Estimated cost for modeling and consulting $30,000, three months duration.
- Investigate the fundamental issue with the flotation of diabase materials. This material has now been almost fully excluded from the Mineral Reserve because of its poor metallurgical performance. Estimated cost for mineralogy and flotation test work $75,000, duration three months.
- Reduced rougher flotation pH (less than pH 10) should be investigated as it will improve the recovery of locked pyrite/chalcopyrite particles. Estimated cost for test work, plant trials and analysis $25,000, duration three months.
- Continue to optimize the molybdenum circuit with the new reagent scheme. Estimated cost for test work, plant trials and analysis $100,000, duration three to four months.
- Investigate the impact of recycling water from the molybdenum circuit using the new reagent suite. Costs and duration are included in the above recommendation.

26.3 Recommendations related to Metal Recovery

PVM is evaluating a number of potential upgrades to the plant. The following recommendations are related to improving plant copper recovery:

- Upgrade the level control system for the rougher flotation circuit to maximize mass pull and mineral recovery. The estimated cost of this upgrade is $200,000 with completion estimate of six months.

- Upgrade the reagent dosing system to improve the reagent addition control. The estimated cost of this upgrade is $150,000 with a complete estimate of three months.
- Upgrade the rougher concentrate pumps to reduce overflow situations and allow for maximized mass pull at coarser grinds. The estimated cost of this upgrade is $250,000 with completion estimate of six to nine months.
- Upgrade the tailings thickeners to more efficient units that yield higher density outputs and greater water recovery. The upgrade is in progress, with completion expected in July 2021. The upgrade cost is estimated at $5 to $7 million. Center wells are currently being fabricated and set for delivery to site starting in mid-June. Piping and other ancillary equipment is ordered and set for delivery in late June.

26.4 Recommendations related to the Mineral Reserve

To improve understanding of the impact of key structures and the Mineral Reserve, the following is recommended:

- Geotechnical evaluation of key structures such as the Bummer fault during mining of ‘internal’ pit phases 3B and western 3A, and application of resultant observations to adjust final 3C pit design. Estimated cost is $100,000, with duration of one month for each of 2-3 key structures.

26.5 Recommendations related to Tailings Management

Monitoring and control of the phreatic levels in the TSF embankments is critical to the performance of the facilities. Additional geotechnical field investigations, including cone penetration testing, exploratory drilling, laboratory testing and engineering analyses will be required if phreatic levels in the TSF embankments rise above predicted levels. Methods to control or mitigate the phreatic rise would be developed. A contingency of $6.6 M has been included in the LOM tailings storage capital cost estimate for these efforts, expended in 2021 through 2034.

26.6 Recommendations related to Permitting Management

The following activites are recommended in order for PVM to be successful in obtaining and keeping its permits to operate, as part of PVM personnel regular duties:

- Prepare a plan to evaluate the impacts of climate change on PVM’s ability to operate within permit terms and conditions by the end of 2022;
- Update the MLRP following USFS issuance of a MPO and authorization to proceed; and
- Stay abreast of continuously evolving mining regulatory regime and best practices.

27 References

911Metallurgy Corp. 2020. Site Visit Summary and Plant Analysis. Denver: 911Metallurgy Corp.

ALS Metallurgy Kamloops. 2014. Metallurgical Assessment of Pinto Valley Operations. Kamloops: ALS Metallurgy Kamloops.

Amelunxen Mineral Processing Ltd. 2017. Review of the Pinto Valley Bulk Processing Plant. Vancouver: Amelunxen Mineral Processing Ltd.

Amelunxen Mineral Processing Ltd. 2020a. Historical Testwork on Pinto Valley Moly Plant Feed during 2020. Santiago: Amelunxen Mineral Processing Ltd.

Amelunxen Mineral Processing Ltd. 2020b. Trialing Cu-Mo Separation Tests. Santiago: Amelunxen Mineral Processing Ltd.

Base Metallurgical Laboratories. 2015. PV3 Metallurgical Testing of Future Ore from Pinto Valley. Kamloops: Base Metallurgical Laboratories.

Blue Coast Metallurgy, Ltd. 2019. A Review of Metallurgy at the Pinto Valley Mill. Vancouver: Blue Coast Metallurgy, Ltd.

Arancibia, O. N., and A. H. Clark. 1996. “Early Magnetite Amphibolite Plagioclase Alteration Mineralization in the Island Copper Porphyry Copper Gold Molybdenum Deposit, British Columbia.” *Economic Geology* 91: 402–38.

Barteczko, A. and C. Denina. 2021. ”Polish miner KGHM looking to sell two U.S. copper assets -sources”. *Kitco News*. March 4, 2021. Web. https://www.kitco.com/news/2021-03-04/UPDATE-1-Polish-miner-KGHM-looking-to-sell-two-U-S-copper-assets-sources.html

BHP Copper Inc. 2012. *Pinto Valley Geology Summary*. Unpublished internal memorandum.

Breitrick, R. A., and G. W. Lenzi. 1987. *Pinto Valley Copper Deposit: Arizona Geological Survey Special Paper 5*.

Burgin, Lorraine B. 1976. *Bureau of Mines Information Circular 8702: Time Required in Developing Selected Arizona Copper Mines*. Washington: U.S. Department of the Interior.

Call & Nicholas, Inc. 2013. *Slope Stability Analysis – Pinto Valley Phase 2*. Unpublished consultant’s report prepared by D. Childs, S. Young, D. Lowe, and E. Wellman for BHP Billiton Pinto Valley, July 2013, 156 p., 4 appendices.

Call & Nicholas, Inc. 2014. *Slope Stability Analysis Update – Pinto* Valley. Prepared by D. Childs, S. De Bruin, L. Standridge, B. Weston, T. Ryan, for Capstone Mining Corporation, 87 p.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM). 2014. *CIM DEFINITION STANDARDS - For Mineral Resources and Mineral Reserves.* CIM Council, May 10, 2014. Web.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM). 2019. *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines*. CIM Council, November 29, 2019. Web.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM). 2020. *2020 CIM Guidance on Commodity Pricing and Other Issues related to Mineral Resource and Mineral Reserve Estimation and Reporting*. *Canadian Institute of Mining, Metallurgy and Petroleum*. CIM Council, August 28, 2020. Web.

Capstone Mining Corp. 2014. *Pinto Valley Mine Life Extension – 2014 Prefeasibility Study, NI 43-101 Technical Report, Pinto Valley Mine, Miami, Arizona*. Miami: Capstone. Dated January 1, 2014.

Capstone Mining Corp. 2016. *Pinto Valley Mine Life Extension - Phase 3 (PV3) Pre-Feasibility Study, NI 43-101 Technical Report, Pinto Valley Mine, Miami, Arizona*. Miami: Capstone. Dated June 12, 2013.

Capstone Mining Corp. 2020. *Capstone to Expand Leaching at Pinto Valley; 300 to 350 Mlbs of Low-Cost, Copper Cathode Production Potential*. July 27, 2020. Web. https://s25.q4cdn.com/701614211/files/doc_news/archive/f0dfe538-cf6d-4dbd-a614-40426be7207b.pdf

Capstone Mining Corp. 2021. *Capstone Beats 2020 Production and Cost Guidance*. February 24, 2021. Web. https://s25.q4cdn.com/701614211/files/doc_financials/2020/q4/v2/NR-2020-Q4-Results.pdf

Creasey, S. C. 1980. “Chronology of Intrusion and Deposition of Porphyry Copper Ores, Globe-Miami District, Arizona.” *Economic Geology* 75: 830–44.

Davis, B. M. 1997. *Some Methods of Producing Interval Estimates for Global and Local Resources*. SME Preprint 97-5.

E&MJ News. 2012. “BHP Billiton Restarting Pinto Valley.” *Engineering and Mining Journal*. Accessed April 15, 2021. https://www.e-mj.com/news/us-and-canada/bhp-billiton-restarting-pinto-valley/

Freeport-McMoRan Copper & Gold Inc. 2014. “Miami Mine.” Accessed April 15, 2021. https://www.fcx.com/operations/northamerica.

Fajardo, Juan; Pizarro, Alejandro and Mckie, Sarah. 2012. Head of Resources Planning & Development - Brownfield Exploration - BHP Copper USA Inc. *Pinto Valley Surface Mapping Report (Geology_Chapter)*

FL Smidth. 2017. Flotation Test Work on Two Clay and One Regular Ore (Revision 1). Salt Lake: FL Smidth.

Global Tailings Review (GTR). 2020. *Global Industry Standard on Tailings Management*. GTR, August 5, 2020. Web.

Gustafson, L. B., and J. P. Hunt. 1975. “The Porphyry Copper Deposit at El Salvador, Chile.” Economic Geology 70: 857−912.

Hedenquist, J. W.; A. Arribas, Jr.; and E. Gonzalez-Urien. 2000. “Exploration for Epithermal Gold Deposits: Reviews.” Economic Geology 13: 245–77.

Independent Geomechanics LLC, 2021. *PV3 Technical Report Update – Draft Geotechncial Section 16.2*. Memo prepared by E. Wellman for Pinto Valley Mine.

Journel, A., and C. J. Huijbregts. 1978. Mining Geostatistics. London: Academic Press.

KGHM International. 2014. “Carlota Mine.” Accessed 18 March 2014. www.kghminternational.com/our-operations/open-pit/carlota-mine/default.aspx.

Fennemore. 2021. *Report on Status of Title; Pinto Valley Mine*. Prepared by Marc A. Marra for Pinto Valley Mining Corp.

Mining Association of Canada (MAC). 2021. *A Guide to the Management of Tailings Facilities* Version 3.2. MAC, April 7, 2021. Web.

Meinert, L. D. 2000. “Gold in Skarns Related to Epizonal Intrusions: Reviews.” *Economic Geology* 13: 347–75.

Peterson, N. P., C. M. Gilbert, and G. L. Quick. 1951. *Geology and Ore Deposits of the Castle Dome Area, Gila County, Arizona: U.S. Geological Survey Bulletin 971*. Washington: U.S. Government Printing Office.

Peterson, Nels Paul. 1962. *Geology and Ore Deposits of the Globe-Miami District, Arizona: U.S. Geological Survey Professional Paper 342*. Washington: U.S. Government Printing Office.

Pinto Valley Mining Corp. 2019. *Memo – Density Testing*. Unpublished internal report prepared for Pinto Valley Mining Corp.

Preece, R. and Baird, B. 2012. *June 2012 Mineral Resource & Ore Reserve Competent Persons Report: Pinto Valley.* Prepared for BHP Copper Inc.

Read, J. and Stacey, P. 2009. *Guidelines for Open Pit Slope Design (LOP).* CSIRO, Victoria, Australia.

Ryan, T, & Pryor, P. 2001. *Designing catch benches and interramp slopes*. In: Hustrulid WA, McCarter MK, Van Zyl DJA, editors. Slope stability in surface mining. Littleton, CO: Soc Min Eng; 2001.

Sillitoe, R. H. 2010. “Porphyry Copper Systems.” *Economic Geology* 105: 3–41.

SGS Minerals Services. 2013. Standard Bond Ball Mill Grindability Test. SGS.

SRK, 2015. *PV3 Geomechanical Data Report, Pinto Valley Mine, Miami, AZ*. Report prepared by E. Wellman & M. Levy for Capstone Mining Corporation, 78 p., 5 appendices

SRK Consulting (Canada) Inc. 2021. Pinto Valley PV3 Optimisation Comminution Model Development. Vancouver: SRK Consulting (Canada) Inc.

Tri-City Fire District & Tri-City Ambulance Service. 2014. “Our Services.” Accessed 5 March 2021. www.tricityfiredistrict.com/our-servies.aspx

U.S. Department of Agriculture – Forest Service (USFS). 2021. “Pinto Valley Mine Final Environmental Impact Statement”. 4 volumes.

Wood Environment and Infrastructure, Inc. (Wood). 2021. “Tailings Storage Facilities, 2020 Monitoring Program, Pinto Valley Mine.

Weir Minerals. 2019. Pinto Valley Cyclone Trial & Grind Circuit Overview. Weir Minerals.