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# PINTO VALLEY MINE
# 2014 PREFEASIBILITY STUDY
## NI 43-101 Technical Report
## Pinto Valley Mine, Miami, Arizona

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# 1.0 SUMMARY

## 1.1 Introduction

The purpose of this report is to disclose the current mineral resource and mineral reserve estimates for the Capstone Mining Corp. (Capstone) Pinto Valley Mine (PVM) through a planned extension of milling operations into 2026 and solvent extraction and electrowinning (SX-EW) operations through 2030. The information contained herein has been compiled by Qualified Persons (QPs) and is aligned with National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

The PVM property was purchased by Capstone from BHP Billiton Ltd. (BHP) in October 2013 and consists of an open-pit mining operation, mill, and an SX-EW facility. The mill produces copper and molybdenum concentrates. Copper cathodes are produced through an SX-EW plant by leaching run-of-mine (ROM) material above 0.10% copper (Cu) and below 0.18% Cu grade. The majority of the copper concentrate produced is trucked to the San Manuel Arizona Railroad Company (SMARRCO) in San Manuel, Arizona, for transport by rail to the port of Guaymas, Mexico, for distribution to international markets. SMARRCO was included as part of the PVM purchase package.

PVM has been in operation since 1974 with two shutdown periods for economic and strategic reasons. The mine and plant were restarted most recently in December 2012, following a large capital improvement and maintenance project, and are expected to remain fully operational for the life of the mine.

The focus of Capstone's work at PVM in the near future (commonly referred to as Pinto Valley Phase 2 [PV2]) will be to improve reliability and cost efficiencies. PV2 is a planned extension of PVM operations beyond the Pinto Valley Phase 1 (PV1) restart of mining and plant operations. PV2 is based on sustaining ore mining and processing capacities at approximately 55,100 tons per day (tpd) (50,000 tonnes per day [t/d]) for the first 2 years, increasing to 57,300 tpd (52,000 t/d) into 2026.

PVM has existing infrastructure to support the operational needs for the PV1 and PV2 operations. Minor upgrades to mine maintenance facilities have been identified to increase the effectiveness and efficiency of the operation. As mining activities in the pit progress, some existing infrastructure will need to be relocated or modified.

Capstone is a Canadian-based metals mining company committed to the responsible development of the mine's assets and surrounding environment. Capstone's focus is on the production of copper at three producing mines: the Pinto Valley copper-molybdenum (Cu-Mo) mine located in Arizona, United States; the Cozamin copper-silver-zinc-lead mine (Cu-Ag-Zn-Pb) in Zacatecas, Mexico; and the Minto copper-gold-silver mine in Yukon, Canada. 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.

Stantec – Mining (Stantec) contributed to multiple sections of this report and compiled the report for Capstone. Other consultants contributing to this report include Kirkham Geosystems Ltd.; Independent Mining Consultants, Inc. (IMC); KWM Consulting Inc.; AMEC Environment & Infrastructure, Inc. (AMEC); SRK Consulting (U.S.), Inc. (SRK); and Adam M Consulting Inc. Personnel from each of these companies have signed off as QPs, as defined in NI 43-101, for their specific responsibilities.

This technical report has an effective date of 01 January 2014. 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. In the remainder of the report, imperial and metric units are used interchangeably and are identified for each table and figure. Cost estimates are based on December 2013 US Dollars (US$).

## 1.2 Project Description

PVM is an established, fully operational Cu-Mo producing mine with silver byproducts. The mine is an open-pit operation, with an L-shaped pit that is approximately 1,100 feet (ft) deep, 5,000 ft wide, and 7,000 ft long. The ore processing, tailings storage, waste rock, and maintenance facilities are located on the property in close proximity to the pit. The ore processing facility consists of three crushing stages, six ball mills, three copper flotation stages, a molybdenum flotation circuit, and associated thickeners for increasing the density of concentrates and tailings. There are two tailings storage facilities (TSFs) that are currently operational. An SX-EW facility is also located on the property and produces copper cathode (Figure 1-1).

**Figure 1-1. Current Pinto Valley Site Facilities**

## 1.3 Accessibility, Climate, Local Resources, and Physiography

PVM is located in Gila County approximately 80 miles east of Phoenix, Arizona, and 3 miles north of US Highway 60 (US 60) on Forest Road (FR) 287. The site can also be accessed from Tucson, Arizona, by traveling north on State Route 77 (SR 77), then west on US 60.

PVM's moderate, semi-arid regional climate allows for year-round operation. The average annual precipitation is 18.92 inches. May and June are typically the driest months of the year and may result in local drought conditions. Thunderstorms occur during the summer. The average annual maximum temperature is 77 degrees Fahrenheit (°F), and the average annual minimum temperature for January, the coolest month, is 34°F.

The historic mining towns of Miami and Globe are the closest to PVM, with a total population of approximately 10,000 residents in 2011. Local services are used if available, with the remainder of services coming from the greater Phoenix area.

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 is entirely within the Pinto Creek watershed, where elevations range from 3,500 ft to 5,000 ft above mean sea level.

## 1.4 History and Ownership

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

The PVM property was originally owned by Miami Copper Company in 1909 and transitioned, through acquisitions and mergers by the Tennessee Corporation and Cities Service Company, to Occidental Petroleum Corporation. Occidental sold the property to Newmont Mining Corporation in 1983, who changed the name to Pinto Valley Copper Corporation (Pinto Valley Copper).

In 1986, Pinto Valley Copper became the Pinto Valley Mining Division of Magma Copper Company. In 1995, Broken Hill Proprietary Company Limited purchased Magma Copper Company, and after merging with Billiton in 2001, the Pinto Valley Mining Division became Pinto Valley Operations of BHP. In 2013, Capstone purchased Pinto Valley Operations, now referred to as Pinto Valley Mine.

## 1.5 Geology and Mineralization

Several mines and numerous prospects have been developed in the Globe-Miami district. Larger mines in the district are porphyry copper deposits associated with Paleocene (59 megaanni [Ma]–63 Ma) 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, Tertiary, and Quaternary age.

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 byproducts. Sphalerite and galena occur locally in very small amounts. Silicate alteration associated with the deposits include potassic, argillic, sericitic, and propylitic alteration 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 Oracle or Ruin granite. Formation of the deposit was associated with the intrusion of small bodies and dikes of granite porphyry and granodiorite.

## 1.6 Exploration, Sampling, and Drilling

Surface mapping and drilling have been the main sources of additional data for the technical report. The surface mapping for geotechnical information focused primarily on the bedding planes, major structures, and overall geological strength index.

In addition, a number of new copper mineralization occurrences were identified during the brownfield surface mapping campaign in the Pinto Valley district. Three principal target zones were identified and are named the Kosi, Bondi, and Mati Prospects.

The pre-2006 PVM drilling programs comprised a combination of core, rotary, and churn drill holes. Churn holes defined much of the early mineralization, which has been mined out. 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 3 reverse circulation (RC) rotary holes (RC62 through RC64) drilled in 1992. From January 1996 to April 1997, 67 RC exploration and infill holes were drilled: 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.

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.

The drilling campaign in 2010 focused on 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.

A total of 1,031 drill holes were supplied for PVM, which formed the original database as delivered by BHP; however, after rigorous review, 794 drill holes had assay data and could be validated and verified.

## 1.7 Mineral Resource

Solids were supplied for the lithology domains, along with the grade shells used for the estimation of resources. In addition, surfaces were supplied for the major fault planes.

It was determined that a 45 ft composite length minimizes the smoothing of the grades, but also reduces the influence of typically narrow, very high-grade samples; this appears to be an optimal interval length from the standpoint of regularization. The main justification to adopt the 45 ft composite length is because the mine has been using this length, and consistency with past practices is desirable. In addition, the selective mining unit (SMU) being used for mine planning is a 45 ft bench.

A radius of influence of 150 ft has been applied to values greater than 1.6% total copper (TCu) and 0.05% Mo. The radius of influence is 150 ft.

The average bulk dry density for ore-grade mineralized rock, primarily Lost Gulch Quartz monzonite, is 12.75 $ft^3$/ton. Although the in situ bulk dry densities for all PVM rock types range from 12.1 $ft^3$/ton for Pinal schist to 13.0 $ft^3$/ton for Whitetail conglomerate, 12.75 $ft^3$/ton has been used.

The block model grades for copper and molybdenum were estimated using ordinary kriging. During grade estimation, search orientations were designed to follow the general trend of the mineralization in each of the zone domains. The estimation plan includes the following.

- Store the mineralized zone code and percentage of mineralization.
- Estimate the grades for each of the metals using ordinary kriging in a single pass.
- Include a minimum of 6 composites and a maximum of 15, with a maximum of 5 from any 1 drill hole.

The mineral resources are listed in Table 1-1. and Table 1-2 for %TCu and %Mo. These mineral resources are listed at a base-case cutoff grade of 0.18% TCu.

**Table 1-1. Mineral Resources at 0.18% TCu Cutoff (imperial units)**

| Classification | M Tons* | %TCu | %Mo | Contained Cu (M lb†) | Contained Mo (M lb) |
|---|---|---|---|---|---|
| Measured (M) | 728.0 | 0.34 | 0.008 | 5,014.3 | 119.4 |
| Indicated (I) | 994.9 | 0.28 | 0.006 | 5,491.8 | 123.4 |
| **Total M & I** | 1,722.8 | 0.30 | 0.007 | **10,506.0** | **242.8** |
| Inferred | 64.1 | 0.23 | 0.005 | 298.5 | 6.8 |

Note: Summation errors due to rounding.
*M tons = millions of tons
†M lb = millions of pounds

**Table 1-2. Mineral Resources at 0.18% TCu Cutoff (metric units)**

| Classification | Mt* | %TCu | %Mo | Contained Cu (Mt) | Contained Mo (Mt) |
|---|---|---|---|---|---|
| Measured (M) | 660.4 | 0.34 | 0.008 | 2,274.4 | 54.2 |
| Indicated (I) | 902.5 | 0.28 | 0.006 | 2,491.0 | 56.0 |
| **Total M & I** | **1,563.0** | **0.30** | **0.007** | **4,765.4** | **110.1** |
| Inferred | 58.2 | 0.23 | 0.005 | 135.4 | 3.1 |

Note: Summation errors due to rounding.
*Mt = millions of tonnes

## 1.8 Mineral Reserve and Mine Plan

### 1.8.1 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 cutoff grades. The reserve statement was constrained by property boundaries and the capacity of permitted tailings facilities. The mine plan and schedule that is presented in Section 16 utilizes a declining cutoff grade to the mill that starts at breakeven cutoff in 2014 and reduces to internal cutoff grade in 2023.

The cutoff grade for the calculation of mineral reserves is 0.18% TCu for production years 2014 through 2022, which is the low-grade stockpile cutoff. The remaining years of the mine life, 2023 through 2025, utilize a 0.17% TCu cutoff.

PVM has operated a ROM dump leach on low-grade sulfide ores for many years. The final pit design and the mineral reserve have not included the ROM dump leach in the floating cone analysis or in the mineral reserve. Material that is incurred at ROM leach grade is reported in the mining plan for allocation to storage areas, but it is not incorporated into the statement of mineral reserves in Table 1-3 and Table 1-4.

Table 1-3 and Table 1-4 summarize the mineral reserves at PVM after 01 January 2014. Table 1-3 states the reserves in the imperial units that are incorporated throughout this report. For clarity and convenience when reporting in metric jurisdictions, Table 1-4 is a direct conversion of imperial units (millions of tons [M tons]) to metric units (millions of metric tonnes [Mt]).

All tonnage tabulations for mineral reserves and the mining plan are based on an in-place rock density of 12.75 $ft^3$/ton (2.513 $t/m^3$).

**Table 1-3. Mineral Reserves – Pinto Valley Mine – 01 January 2014 (imperial units)**

| Classification* | Cutoff %TCu | Ore (M tons) | %TCu | %Mo |
|---|---|---|---|---|
| Proven | 0.18–0.17 | 241 | 0.33 | 0.008 |
| Probable | 0.18–0.17 | 15 | 0.33 | 0.008 |
| Proven + Probable | 0.18–0.17 | 256 | 0.33 | 0.008 |

Note: Summation errors due to rounding.

*The mineral reserves stated above are limited to match the tailings storage capacity.

**Table 1-4. Mineral Reserves – Pinto Valley Mine – 01 January 2014 (metric units)**

| Classification | Cutoff %TCu | Ore (Mt) | %TCu | %Mo |
|---|---|---|---|---|
| Proven | 0.18–0.17 | 219 | 0.33 | 0.008 |
| Probable | 0.18–0.17 | 13 | 0.33 | 0.008 |
| Proven + Probable | 0.18–0.17 | 232 | 0.33 | 0.008 |

Note: Summation errors due to rounding.

### 1.8.2 Mining

PVM utilizes open-pit hard-rock mining methods. Copper-bearing sulfide ore from the mine is unloaded in the primary crusher, then conveyed to a mill. The current mine plan extends the PVM operational life to 2026 and respects the property boundaries and current permitted tailings storage capacity of just over 250 M tons (TSF3 and TSF4). There is additional material that could be mined if those constraints were removed.

Ore production to the mill is planned to continue at 55,100 tpd (20,117 kilotons/year [ktons/yr]) for the first 2 years, followed by an increase to 57,300 tpd (20,922 ktons/yr) for the remaining life of the mine plan. The mine production schedule was developed with the goal of maintaining mill feed and maximizing the project return on investment.

Table 1-5, Table 1-6, and Figure 1-2 summarize the mine production schedule.

The total material rate is tied to current mine equipment productivity, with an intent to expand the mine fleet to meet the strip ratio of 0.65:1. Material movement increases from 25,000 ktons/yr in the first year to 35,600 ktons/yr in the second year, and finally to 46,600 ktons/yr for 4 years until the ore is released from the final pushback and the required material movement drops off. The mine is scheduled to operate 365 days per year with two 12-hour shifts per day. The current bench height is being maintained at 45 ft.

The current mining fleet consists of the following.

- 2 Cat 6420 Blasthole Drills
- 3 Cat 994 Front-End Loaders
- 15 Cat 789 Haul Trucks

The fleet will expand to a maximum of the following.

- 3 Blasthole Drills
- 1 Cat 994 Front-End Loader
- 2 Hydraulic Shovels, 35 $yd^3$
- 19 Cat 789 Haul Trucks by 2017

**Table 1-5. Pinto Valley Mine Production Schedule (imperial units)**

| Year | | Mill Ore | | | | Low-Grade Stockpile, +0.18% TCu | | | Dump Leach, +0.10% | | Waste | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Cutoff %TCu | Ore ktons | %TCu | %Mo | LG Stkp* ktons | %TCu | %Mo | Leach ktons | %TCu | Material ktons | Material ktons |
| 1 | 2014 | 0.22 | 20,117 | 0.39 | 0.008 | 1,202 | 0.20 | 0.007 | 2,408 | 0.15 | 1,155 | 24,882 |
| 2 | 2015 | 0.22 | 20,117 | 0.34 | 0.008 | 4,735 | 0.20 | 0.007 | 5,991 | 0.14 | 4,775 | 35,618 |
| 3 | 2016 | 0.20 | 20,922 | 0.38 | 0.008 | 1,994 | 0.19 | 0.004 | 10,038 | 0.15 | 13,646 | 46,600 |
| 4 | 2017 | 0.20 | 20,922 | 0.33 | 0.008 | 934 | 0.19 | 0.008 | 10,076 | 0.13 | 14,668 | 46,600 |
| 5 | 2018 | 0.20 | 20,922 | 0.33 | 0.009 | 4,168 | 0.19 | 0.007 | 9,423 | 0.15 | 12,087 | 46,600 |
| 6 | 2019 | 0.19 | 20,922 | 0.32 | 0.010 | 2,948 | 0.18 | 0.006 | 16,789 | 0.15 | 4,807 | 45,466 |
| 7 | 2020 | 0.20 | 20,922 | 0.33 | 0.011 | 2,590 | 0.19 | 0.005 | 8,108 | 0.14 | 869 | 32,489 |
| 8 | 2021 | 0.20 | 20,922 | 0.33 | 0.008 | 1,964 | 0.19 | 0.005 | 4,335 | 0.15 | 672 | 27,893 |
| 9 | 2022 | 0.18 | 20,922 | 0.32 | 0.006 | | | | 4,129 | 0.14 | 341 | 25,392 |
| 10 | 2023 | 0.17 | 20,922 | 0.31 | 0.006 | | | | 5,307 | 0.13 | 1,077 | 27,306 |
| 11 | 2024 | 0.17 | 20,922 | 0.35 | 0.007 | | | | 1,245 | 0.14 | | 22,167 |
| 12 | 2025 | 0.17 | 6,911 | 0.34 | 0.007 | | | | | | | 6,911 |
| **Total** | | | **235,443** | **0.34** | **0.008** | **20,535** | **0.19** | **0.006** | **77,849** | **0.14** | **54,097** | **387,924** |

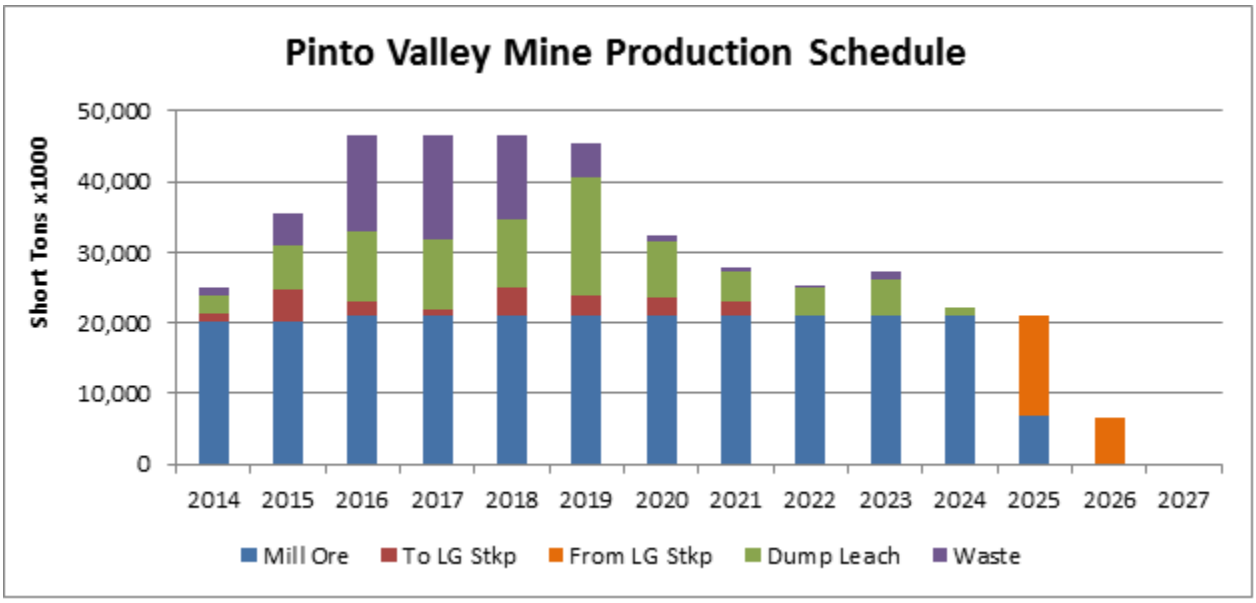
*Low-Grade Stockpile

**Table 1-6. Pinto Valley Mine Production Schedule (metric units)**

| Year | | Mill Ore | | | | LG Stockpile, +0.18% TCu | | | Dump Leach +0.10% | | Waste | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Cutoff %TCu | Ore ktonnes | %TCu | %Mo | LG Stkp* ktonnes | %TCu | %Mo | Leach ktonnes | %TCu | Material ktonnes | Material ktonnes |
| 1 | 2014 | 0.22 | 18,250 | 0.39 | 0.008 | 1,090 | 0.20 | 0.007 | 2,185 | 0.15 | 1,048 | 22,573 |
| 2 | 2015 | 0.22 | 18,250 | 0.34 | 0.008 | 4,296 | 0.20 | 0.007 | 5,435 | 0.14 | 4,332 | 32,312 |
| 3 | 2016 | 0.20 | 18,980 | 0.38 | 0.008 | 1,809 | 0.19 | 0.004 | 9,106 | 0.15 | 12,379 | 42,275 |
| 4 | 2017 | 0.20 | 18,980 | 0.33 | 0.008 | 847 | 0.19 | 0.008 | 9,141 | 0.13 | 13,307 | 42,275 |
| 5 | 2018 | 0.20 | 18,980 | 0.33 | 0.009 | 3,781 | 0.19 | 0.007 | 8,548 | 0.15 | 10,965 | 42,275 |
| 6 | 2019 | 0.19 | 18,980 | 0.32 | 0.010 | 2,674 | 0.18 | 0.006 | 15,231 | 0.15 | 4,361 | 41,246 |
| 7 | 2020 | 0.20 | 18,980 | 0.33 | 0.011 | 2,350 | 0.19 | 0.005 | 7,355 | 0.14 | 788 | 29,474 |
| 8 | 2021 | 0.20 | 18,980 | 0.33 | 0.008 | 1,782 | 0.19 | 0.005 | 3,933 | 0.15 | 610 | 25,304 |
| 9 | 2022 | 0.18 | 18,980 | 0.32 | 0.006 | | | | 3,746 | 0.14 | 309 | 23,035 |
| 10 | 2023 | 0.17 | 18,980 | 0.31 | 0.006 | | | | 4,814 | 0.13 | 977 | 24,772 |
| 11 | 2024 | 0.17 | 18,980 | 0.35 | 0.007 | | | | 1,129 | 0.14 | | 20,110 |
| 12 | 2025 | 0.17 | 6,270 | 0.34 | 0.007 | | | | | | | 6,270 |
| **Total** | | | **213,590** | **0.34** | **0.008** | **18,629** | **0.19** | **0.006** | **70,623** | **0.14** | **49,076** | **351,919** |

*Low-Grade Stockpile

**Figure 1-2. Pinto Valley Mine Production Schedule (imperial units)**




Note: LG Stkp = Low-Grade Stockpile

Figure 1-3 illustrates the final pit and storage facilities in year 2025. Low-grade stockpiles are stored to the southeast and to the west of the pit. Dump leach material is delivered to the northwest of the pit, and waste is stored to the northeast of the pit.

**Figure 1-3. Final Pit and Dump Configuration – End of 2025**

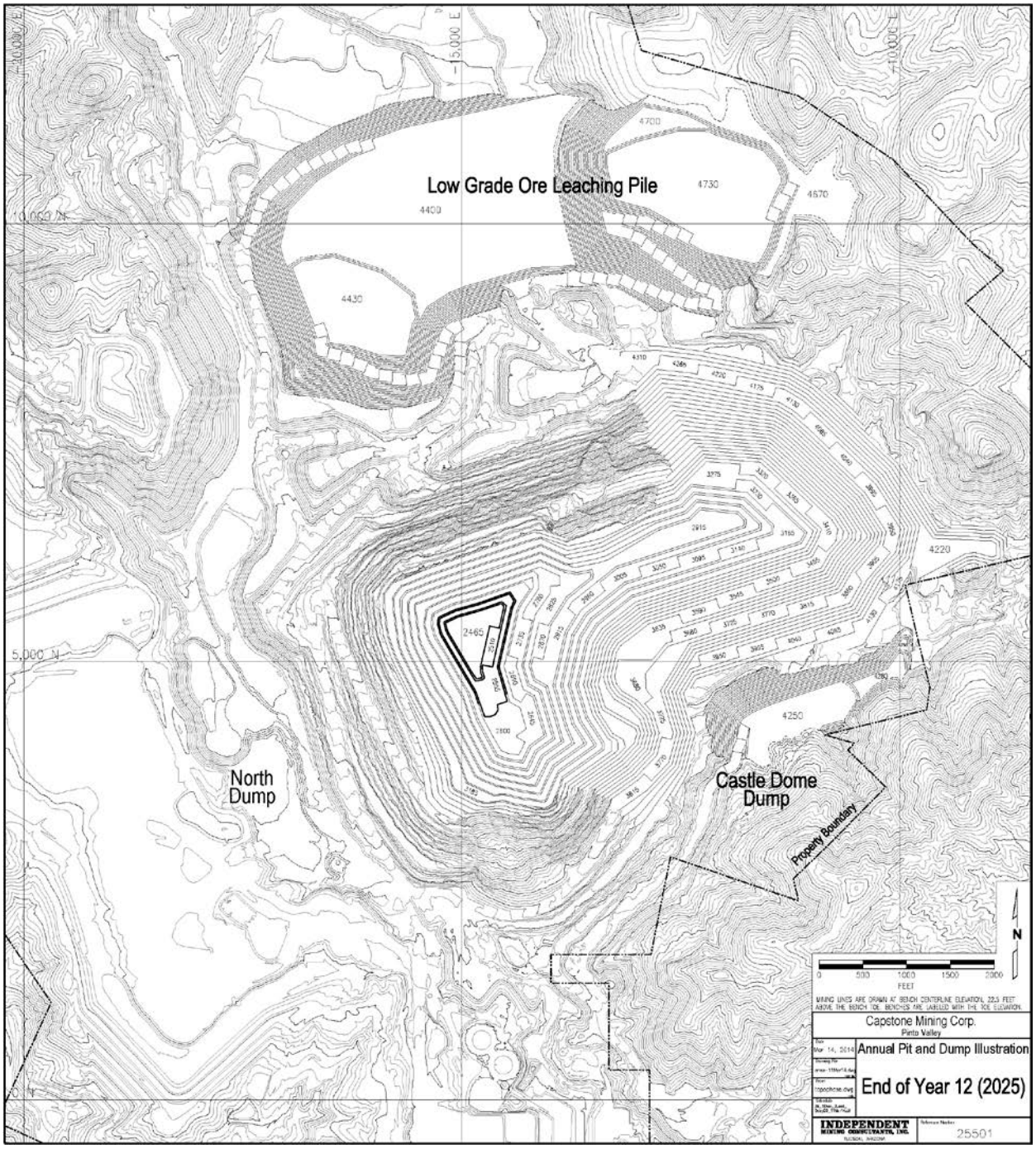


Table 1-7 and Table 1-8 show the mill feed schedule through 2026.

**Table 1-7. Pinto Valley Mill Feed Schedule (imperial units)**

| Year | | Mill Ore | | | |
|---|---|---|---|---|---|
| | | Cutoff %TCu | Ore ktons | %TCu | %Mo |
| 1 | 2014 | 0.22 | 20,117 | 0.39 | 0.008 |
| 2 | 2015 | 0.22 | 20,117 | 0.34 | 0.008 |
| 3 | 2016 | 0.20 | 20,922 | 0.38 | 0.008 |
| 4 | 2017 | 0.20 | 20,922 | 0.33 | 0.008 |
| 5 | 2018 | 0.20 | 20,922 | 0.33 | 0.009 |
| 6 | 2019 | 0.19 | 20,922 | 0.32 | 0.010 |
| 7 | 2020 | 0.20 | 20,922 | 0.33 | 0.011 |
| 8 | 2021 | 0.20 | 20,922 | 0.33 | 0.008 |
| 9 | 2022 | 0.18 | 20,922 | 0.32 | 0.006 |
| 10 | 2023 | 0.17 | 20,922 | 0.31 | 0.006 |
| 11 | 2024 | 0.17 | 20,922 | 0.35 | 0.007 |
| 12 | 2025 | 0.17–0.18* | 20,922 | 0.24 | 0.006 |
| 13 | 2026 | 0.18* | 6,524 | 0.19 | 0.006 |
| **Total** | | | **255,978** | **0.33** | **0.008** |

*Stockpile

**Table 1-8. Pinto Valley Mill Feed Schedule (metric units)**

| Year | | Mill Ore | | | |
|---|---|---|---|---|---|
| | | Cutoff %TCu | Ore ktonnes* | %TCu | %Mo |
| 1 | 2014 | 0.22 | 18,250 | 0.39 | 0.008 |
| 2 | 2015 | 0.22 | 18,250 | 0.34 | 0.008 |
| 3 | 2016 | 0.20 | 18,980 | 0.38 | 0.008 |
| 4 | 2017 | 0.20 | 18,980 | 0.33 | 0.008 |
| 5 | 2018 | 0.20 | 18,980 | 0.33 | 0.009 |
| 6 | 2019 | 0.19 | 18,980 | 0.32 | 0.010 |
| 7 | 2020 | 0.20 | 18,980 | 0.33 | 0.011 |
| 8 | 2021 | 0.20 | 18,980 | 0.33 | 0.008 |
| 9 | 2022 | 0.18 | 18,980 | 0.32 | 0.006 |
| 10 | 2023 | 0.17 | 18,980 | 0.31 | 0.006 |
| 11 | 2024 | 0.17 | 18,980 | 0.35 | 0.007 |
| 12 | 2025 | 0.17–0.18† | 18,980 | 0.24 | 0.006 |
| 13 | 2026 | 0.18† | 5,918 | 0.19 | 0.006 |
| **Total** | | | **232,219** | **0.33** | **0.008** |

*kilotonnes (kilotonne = 1,000 metric tonnes)

†Stockpile

## 1.9 Mineral Processing and Metallurgical Testing

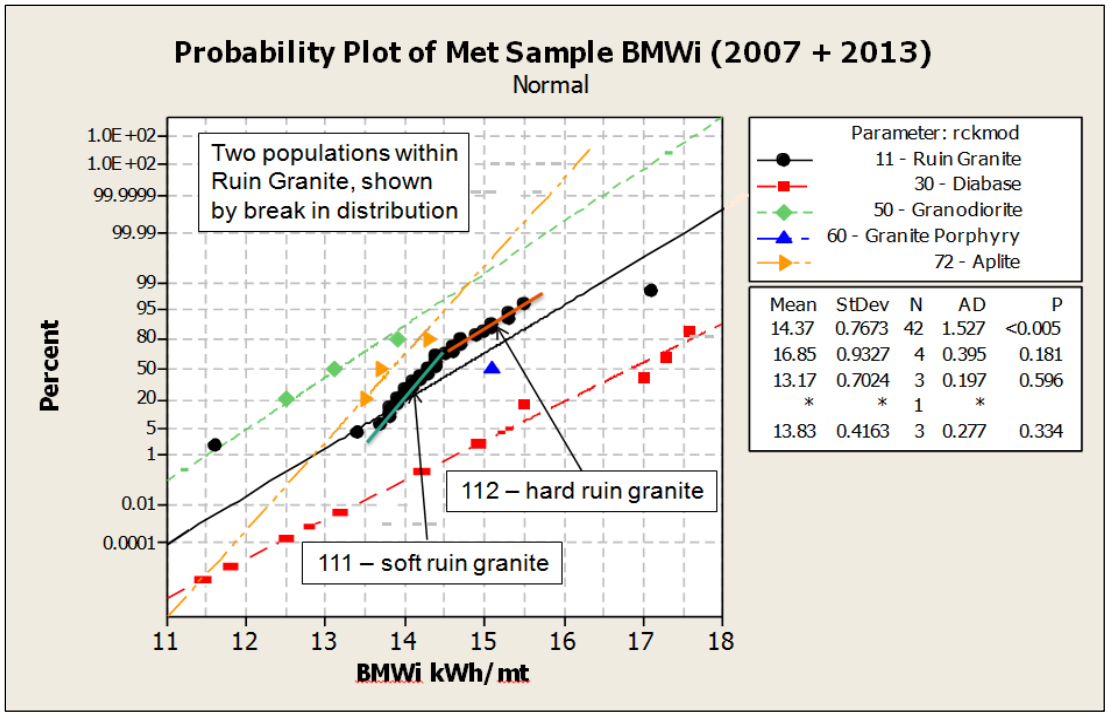
Ore processing through the mill at PVM started in 1974 with a few extended periods of downtime, and was most recently restarted in 2012. The mill has undergone some process optimizations during its operating life with equipment upgrades and/or replacements. As part of the PV1 restart and PV2 projects, a number of metallurgical test programs have been completed. Most of this testwork focused on operating plant performance monitoring to develop predictive models for the various unit operations. The development of the design criteria and metallurgical models for PV2 has focused on current and historical process operating performance.

Metso developed a Bruno model of the existing crushing circuit that could then be used to determine the circuit constraints. Using primary crusher discharge digital analysis to determine the feed characteristics to the fine-crushing (secondary and tertiary) circuit, the model indicated that the existing circuit capacity would be approximately 55,100 tpd with the crushing circuit operating at 75% availability.

Fine crushed ore product is conveyed to the fine ore bin (FOB) for distribution to the six single-stage ball mills. Shift operating data for the crushing plant and grinding circuit since September 2013, have shown that the ball mill feed (crushing plant product) $F_{80}$ has averaged 0.43 inches (11 millimeters [mm]). A review of the cyclone overflow distributions for each of the grinding circuits indicates that the grinding product $P_{80}$ has averaged 280 microns (µm). These process operating conditions were used to develop a work index model using the Bond ball work index (BWi) equations to determine the plant throughput.

BWi tests and/or modified Bond work index (MBWi) tests were competed on drill core samples. Figure 1-4 provides an analysis of the resulting BWi index data. The predominant ore type has been identified as Ruin granite, and the data indicates that the work index is distributed over a narrow range from 13.5 kilowatt hours/tonne (kWh/t) to 15.5 kWh/t.

**Figure 1-4. Bond Ball Mill Work Index**




Note: BMWi kWh/mt = Ball mill work index kWh/metric tonne

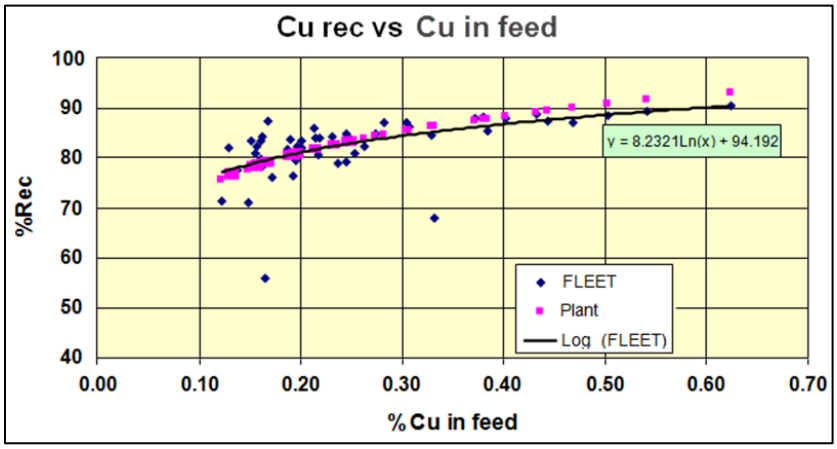
The PVM staff also provided an analysis of the monthly ore mined tonnage and related that to the drill hole work index information for the areas mined. For the months of August, September, and October 2013, the average work index from the mine block model was approximately 14.7 kWh/t. Applying this work index, the model would have limited mill throughput to 49,200 tpd (44,700 t/d), on a 95% operating availability basis. Based on the instantaneous readings from the weightometers on the mill feed conveyors, the ball mill feed rate has recently averaged about 400 tons per hour (tph). Assuming a mill availability of 95%, the estimated plant throughput with six mills is approximately 55,100 tpd (50,000 t/d).

Evaluation of the ore processing facility has been based primarily on recent operating practices and performance. From the review of the crushing and grinding circuit operation, it was determined that the mill should operate at an average daily throughput of 55,100 tpd (50,000 t/d) once the target availabilities of 75% crushing and 95% grinding have been achieved. This conclusion has been based on maintaining the fine particle size distribution of ore delivered to the fine-crushing plant and on a nominal ore work index of 14.7 kWh/t. From the analysis of the plant data, indications are that the mill is operating at 10% higher throughput than the work index model indicates.

A review of historical operating data indicated that the crusher and mill had operated for extended periods at throughput rates greater than 55,100 tpd. The planned increase to 57,300 tpd (52,000 t/d) in 2016 will be achieved through a continuous improvement program and increased operator experience with the mill equipment.

There has been a significant amount of work completed on flotation recoveries for the mill. The primary findings of the data reviewed in 2012/2013 have been included in Figure 1-5 and Table 1-9.

**Figure 1-5. 2007 FLEET Model**




Note: FLEET = Flotation Economic Evaluation Tool

The model that was developed predicted a copper cleaner recovery of 93.34%, which reduced the overall copper recovery to 85%. The historical cleaner recovery at PVM was indicated in one of the reviewed reports to be about 97.3%. Typical porphyry copper cleaners operate above the 97.3% level. It was concluded that the model was predicting lower recoveries than are being achieved in the plant.

In January 2013, composite samples for a froth flotation study and comminution testing were delivered to SGS in Tucson, Arizona. The rougher flotation testing was conducted for 25 minutes at a grind size of approximately 80% passing 270 µm. The testing of the composite samples indicated a total copper rougher recovery ranging from 95.06% to 91.98%.

Table 1-9 compares rougher / cleaner combined recovery from the Flotation Economic Evaluation Tool (FLEET) model developed in 2007 to the 2013 rougher recovery model and the overall recovery interpretations, using a fixed cleaner flotation recovery (97.3%) and the FLEET cleaner recovery model.

**Table 1-9. Copper Flotation Recovery Model**

| 2007 FLEET* | | 2013 (Rougher) | | 2013 (Cleaner) | |
|---|---|---|---|---|---|
| %TCu | %Recovery | %TCu | %Recovery | CR† = 97.3 | CR (FLEET) |
| 0.10 | 75.2 | 0.10 | 88.03 | 85.65 | 85.61 |
| 0.20 | 80.9 | 0.20 | 89.02 | 86.61 | 86.75 |
| 0.30 | 84.3 | 0.30 | 90.01 | 87.58 | 87.57 |
| 0.40 | 86.6 | 0.40 | 91.00 | 88.54 | 88.05 |
| 0.50 | 88.5 | 0.50 | 91.98 | 89.50 | 88.18 |
| 0.60 | 90.0 | 0.60 | 92.97 | 90.46 | 87.95 |
| 0.70 | 91.3 | 0.70 | 93.96 | 91.42 | 87.35 |
| 0.80 | 92.4 | 0.80 | 94.95 | 92.39 | 86.36 |
| 0.90 | 93.3 | 0.90 | 95.94 | 93.35 | 84.98 |

*As predicted by the FLEET models: 2007 (Total): %Recovery = 8.2381ln(%TCu) + 94.19; 2013 (Rougher): %Recovery = 9.8864(%TCu) + 87.041; 2013 (Cleaner): %Recovery = $-18.465(\%TCu)^2 + 7.6287(\%TCu) + 96.67$

†Cleaner recovery

Based on the information that has been presented, it is recommended that the copper flotation recovery model adopt a simple linear model to represent the rougher flotation recovery followed by a fixed cleaner flotation recovery.

- Copper Rougher Recovery
  - $[R_{RCu} = (9.8864) \times (\%Cu) + 87.041]$
- Copper Cleaner Recovery
  - $[R_{CCu} = 97.3\%]$
- Overall Copper Recovery
  - $[R_{Cu} = ((9.8864) \times (\%Cu) + 87.041) \times 0.973)]$

This recovery model applies to 99% of the ore processed. Diabase ores forming the remaining 1% of ore processed will have a 5-6% lower recovery.

The SGS flotation testing was also performed on the composite samples for Molybdenum recovery. The rougher bulk concentrate recoveries ranged from 81.76% to 69.69%. Molybdenum is a byproduct stream that has a low mill feed grade. Because the molybdenum recovery is dependent on the ore feed grade and the performance of the bulk concentrate separation circuit, the plant recovery of molybdenum disulfide ($MoS_2$) from the copper (bulk) cleaner concentrate will be another step decrease in molybdenum recovery from the test results. The overall estimated molybdenum circuit recovery to molybdenum concentrate when the circuit is operating is 47.9%, fairly typical of byproduct molybdenum operations.

## 1.10 Project Infrastructure

PVM has sufficient infrastructure to support current operations, but minor upgrades to existing facilities will be required to support the work plan. This report investigates the mining, processing, support, and concentrate handling infrastructure required for the mine plan that extends the operational life to 2026.

Access and property roads generally have graded or paved well-maintained, all-weather surfaces. Local access to the property is via US 60 and FR 287.

PV2 mine development pushbacks are planned for the south and east sides of the open pit. The expanded ultimate pit perimeter will require relocations of the east-side power feeder lines, explosives magazines, ammonium nitrate storage silos, and pit dewatering pipelines. A new power line isolator (circuit breaker) will allow complete isolation of the east power loop for construction and future maintenance. The construction of a new south perimeter stormwater diversion channel will divert stormwater away from the pit and minimize the need for catchments within the mine development areas. The in-pit dewatering pumps, piping, and electrical equipment will be upgraded and relocated as required to enable continued necessary mining activities on the lower benches of the pit.

The mining fleet is currently serviced at the Main Shop and the North Barn. All work on the mine equipment will be moved to the Main Shop to ensure that maintenance work is performed more efficiently. At the Main Shop, upgrades will include a new air compressor system, new offices, relocation of the light-vehicle fueling station, and a lubrication facility.

Also in support of the mining fleet move to the Main Shop, the light-vehicle shop will be moved to the North Barn, the existing truck wash near the Main Shop will be retrofitted and expanded, and a new tire maintenance facility will be built.

As part of the PV2 stage of operations, there will be marginal-grade material that will be mined during the remaining mine life; this material may be feasible to process in the mill or on the Low-Grade Ore Leaching Pile, depending on economics of the time period in which it is excavated. This marginal-grade material is planned to be stockpiled in two areas for easy access. A marginal-grade dump will be built on the south side of the pit in an area within the open pit drainage sub-basin and partially overlying the plant area for the historic Castle Dome Mine. A second marginal-grade dump will be placed west of the pit near the North Barn. Gold Gulch Dump, a new waste rock storage facility, will be constructed on the eastern portion of the Low-Grade Ore Leaching Pile.

PVM currently operates two TSFs. TSF4 is the primary storage facility, and TSF3 is used when maintenance is required on the TSF4 tailings distribution system or during plant upset conditions.

The total tailings storage requirement is 253.6 M tons through 2026. The tailings distribution and water reclamation systems have been evaluated for an average production rate of 57,300 tpd. Upgrades to the pumping infrastructure are required to handle the additional tailings material and higher raise elevations on the TSFs.

TSF4, placed in service in 1977, has been evaluated and was determined to be capable of storing 246.0 M tons (dry weight) of tailings at a settled density of 92 lb/ft$^3$. TSF4 would rise from the current approximately (~) 3,800 ft crest elevation to the permitted 4,005 ft crest elevation at an average 17 ft/yr rate of rise.

TSF3 was placed into service in 1973. TSF3 is designated to be used intermittently for a maximum of 32 days per year, or 10 ft of raise per year. Within the current patented land boundary, TSF3 is capable of storing 7.1 M tons of tailings at a settled density of 90 lb/ft$^3$, providing about 4 years of intermittent usage. Once the US Forest Service (USFS) Plan of Operations is amended, TSF3 will be capable of storing 23.0 M tons of tailings.

SMARRCO, a wholly owned subsidiary of Capstone, owns and operates a public carrier railroad. Copper concentrate is trucked to SMARRCO's train load-out facility in San Manuel, Arizona. Temporary in-car concentrate storage is available at SMARRCO's rail switching yard in Hayden, Arizona, with 80,000 tons of combined covered storage available in Guaymas, San Manuel, and PVM. The train load-out facility and railroad were upgraded in 2012 for the restart of PVM.

The nearby tri-city communities of Globe, Miami, and Claypool, Arizona, have a mature support services infrastructure. Local services include a regional hospital, emergency response and fire protection teams, and Gila County government offices. Normal communications channels using land-based facilities, radio, cell phone, microwave, and satellite are available. Additional services are available within 100 miles in the Phoenix and Tucson metropolitan areas. Gasoline and diesel fuels are trucked to the property. Power is supplied by Salt River Project (SRP), a major utility serving customers in central Arizona.

## 1.11 Environment

PVM has well-established environmental, health, and safety procedures and protocols that adhere to federal and state regulatory requirements and to internal corporate guidance to reduce impacts to the environment and to provide a safe work environment for employees and contractors. Numerous site characterization studies have been completed at PVM to support operations, environmental permitting efforts, closure planning, and other investigations. PVM is subject to health and safety regulations under the supervision of the Mine Safety and Health Administration (MSHA), Arizona State Mine Inspector's Office (ASMIO), Arizona Department of Transportation, and other federal and state agencies. 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. PVM has established relationships with its communities of interest and stakeholders; communication channels are in place for direct interaction with stakeholders as required.

## 1.12 Closure Plan

Planning for site closure and the post-closure monitoring period, and preparing a demonstration of financial assurance for the company to address closure and post-closure costs is managed under the Mined Land Reclamation Plan (MLRP) and Aquifer Protection Permit (APP) programs administered by ASMIO and the Arizona Department of Environmental Quality (ADEQ), respectively. The MLRP identifies post-mining land uses and addresses post-closure public safety, reclamation, and revegetation of disturbed lands. The APP closure plan focuses on reducing post-closure groundwater impacts through engineering designs and controls, site inspections, and groundwater monitoring.

The capital costs associated with closure and reclamation of PVM are estimated at $129.1 million (M) and are spread out from 2026 to 2055. This is based on the $87.1 M currently indicated by the APP and MLRP for the current mine, and an additional $42 M to cover an additional reclamation bond related to the approved Plan of Operations currently under review by the USFS, owner's costs including internal general and administrative (G&A) and labor to support the 30-year post-closure period, and the studies to support permit amendments, increased land disturbances and other impacts of executing the 12-year mine plan.

## 1.13 Permitting

PVM has all of the necessary permits to conduct mining activities through 2026, with the exception of an amendment to the Plan of Operations that is currently under review by the USFS. The Plan of Operations is a compilation of several special use permits and other authorizations for Capstone to use USFS lands adjacent to the private lands where mining activities occur. This plan is subject to a National Environmental Policy Act (NEPA) review that was initiated under BHP and is continuing with Pinto Valley Mining Corp. as the proponent. Once approved, it is expected that an additional financial assurance will be required by the USFS, likely in 2015. As of 31 December 2013, PVM was approximately 50% through the process, with no material risks identified. Capstone's assessment is that any modifications to the Plan of Operations following this review would not likely result in any material change to permitted operating parameters or financial obligations under any applicable permits. The major instruments or authorizations permitting and governing operations for the project include an APP and an Air Quality Permit from the ADEQ, an MLRP approved by ASMIO, and a Plan of Operations approved by the USFS.

## 1.14 Sustaining Capital Cost and Operating Cost

### 1.14.1 Sustaining Capital Cost

Sustaining capital for PVM over the 12-year mine life totals $187.9 M and is in addition to ongoing operating costs. This capital has been allocated into the following categories.

- Mine Equipment
  - Mine equipment necessary to carry out the mine plan, including hydraulic excavators, trucks, and support equipment.
- Mine Infrastructure and Mine Maintenance
  - Pit dewatering pumps and piping, pit water diversions, maintenance shop upgrades, electrical power loop relocation, and mine offices.
- Mill and SX-EW
  - Maintenance and upgrades of current equipment.
- Tailings and Water
  - Construction of tailings dams, tailings line replacement, and tailings and reclaim water pumping booster stations upgrades.
- Other Infrastructure
  - On-site and off-site infrastructure and an estimate for future sustaining capital.
- Miscellaneous Upgrades and Engineering Studies
  - Accounting system upgrades, information technology (IT) infrastructure, light vehicles, and engineering studies.

Sustaining capital expenditure by category is detailed in Table 1-10.

**Table 1-10. Sustaining Capital Requirements**

| Category | LOM* Requirement (US$ M) |
|---|---|
| Mine Equipment | 47.6 |
| Mine Infrastructure and Mine Maintenance | 13.0 |
| Mill and SX-EW | 23.5 |
| Tailings and Water | 41.3 |
| Other Infrastructure | 51.2 |
| Miscellaneous Upgrades and Engineering Studies | 11.3 |
| **Total** | **187.9** |

*Life-of-Mine

### 1.14.2 Operating Cost

The life-of-mine (LOM) operating cost for PVM is projected to be $9.40/ton milled (excluding SX-EW, freight, and treatment charge / refining charge [TC/RC] costs). Costs for SX-EW production of copper are projected to be $1.80/lb Cu (Table 1-11).

**Table 1-11. Operating Cost Summary**

| Operating Cost | Imperial | Metric |
|---|---|---|
| Mining Cost | $1.98/ton moved | $2.18/tonne moved |
| Mining Cost | $3.16/ton milled | $3.48/tonne milled |
| Milling Cost | $4.85/ton milled | $5.35/tonne milled |
| G&A Cost | $1.39/ton milled | $1.53/tonne milled |
| **Total** | **$9.40/ton milled** | **$10.37/tonne milled** |
| SX-EW Cost | $1.80/lb Cu cathode | - |

Note: Summation errors due to rounding.

## 1.15 Economic Analysis

### 1.15.1 Cash Flow Basis

A financial evaluation of PVM was performed to assess the economic viability of continuing mill operations until 2026 and SX-EW operations until 2030. The determination of economic viability was carried out by calculating a pre-tax and after-tax net present value (NPV). The NPV is calculated using annual cash flow projections over the life of the mine, based on estimates of capital expenditures, production costs, sales revenue, and taxes. The sales revenue is based on the sale of copper concentrate (including silver credits), molybdenum concentrate, and copper cathode.

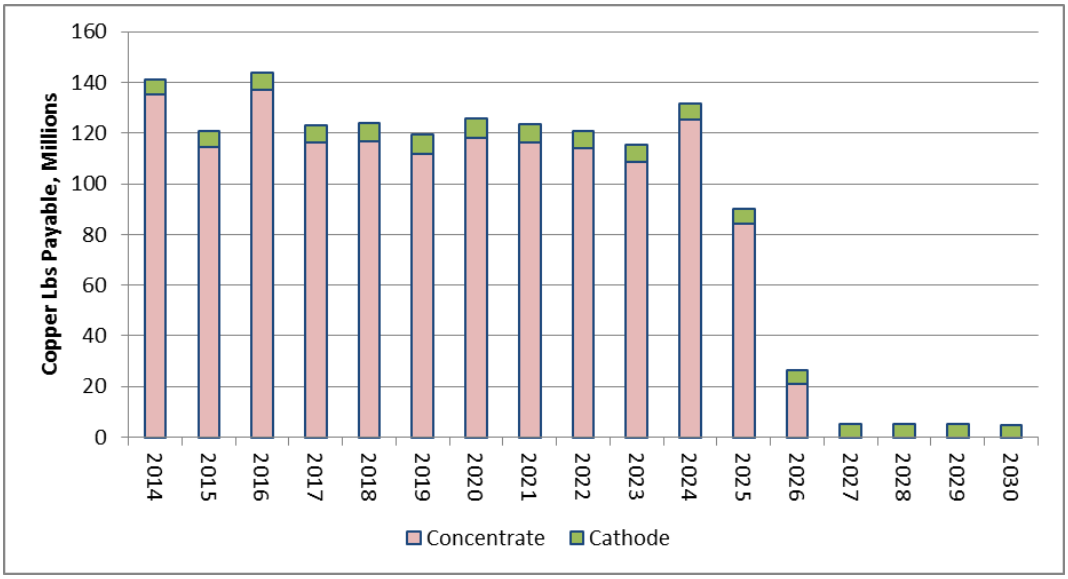
Key assumptions included in the economic evaluation are detailed in Table 1-12.

**Table 1-12. Key Assumptions in Economic Analysis**

| Item | Description |
|---|---|
| Copper Price – Average 2014–2022 | $3.15/lb |
| Copper Price – Long-Term (2023+) | $2.75/lb |
| Molybdenum Price | $12.5/lb |
| Silver Price | $20.0/oz |
| Copper Concentrate Grade – Cu | 27.5% |
| Copper Payable | 96.5% |
| Copper Treatment Charge | $85/dry tonne |
| Copper Refining Charge | $0.085/lb Cu |
| Average Concentrate Shipping Cost | $137.1/wet tonne |

Payable copper produced will range between 115 M lb/yr and 140 M lb/yr, until processing of low-grade stockpiles begins in 2025. Payable copper production by year is shown on Figure 1-6.

**Figure 1-6. Payable Copper by Year**




## 1.15.2 Economic Evaluation

The economic evaluation period begins 01 January 2014. The base-case financial indicators have been determined with 100% equity financing of the sustaining capital. Any acquisition cost or expenditures prior to 01 January 2014 have been treated as "sunk" cost and have not been included in the analysis. Costs are fourth-quarter 2013 dollars, and no escalation is applied.

The cash flow analysis shows an operating cash flow of $1,602 M, which results in a pre-tax NPV (8%) of $931 M and a post-tax NPV (8%) of $738 M, as shown in Table 1-13.

**Table 1-13. Revenue and Total Production Costs**

| Financial Analysis | US$ M |
|---|---|
| Copper Revenue | 4,656 |
| Molybdenum Revenue | 210 |
| Silver Revenue | 52 |
| **Total Revenue** | **4,918** |
| Site Costs | 2,600 |
| Concentrate Transport, TC/RCs | 716 |
| Operating Cash Flow | 1,602 |
| Sustaining Capital Costs | 188 |
| Closure Costs | 129 |
| Net Cash Flow – Pre-Tax, Undiscounted | 1,285 |
| **NPV (8%) Pre-Tax** | **931** |
| Taxes Payable | 274 |
| Net Cash Flow – Post-Tax, Undiscounted | 1,011 |
| **NPV (8%) Post-Tax** | **738** |

## 1.16 Interpretation and Conclusions

### 1.16.1 Key Outcomes

Key outcomes of this report include the following.

- Measured and indicated resources are 1.722 billion tons at 0.30% Cu and 0.07% Mo using a 0.18% cutoff grade.
- Proven and probable reserves are 256 M tons at 0.33% Cu and 0.08% Mo, using a variable cutoff between 0.17% and 0.18%.
- The LOM strip ratio is 0.65:1.
- Ore mining and processing capacities will be sustained at 55,100 tpd for the first 2 years, increasing to 57,300 tpd into 2026.
- PVM has all of the necessary permits to conduct mining activities through 2026, with the exception of an amendment to the approved Plan of Operations, which is currently under review by the USFS.
- LOM annual production is 119.5 M lb Cu contained in concentrate plus 6.3 M lb of cathode copper, and 1.4 M lb Mo and 235,000 troy ounces (oz) of silver credited to concentrate annually.

- LOM C1 (net direct cash cost) is estimated to be $2.00/lb of payable copper (net of copper cathodes and molybdenum and silver byproduct credits).
- Total sustaining capital costs are estimated to be $187.9 M.
- After-tax NPV, discounted at 8%, is $738 M.

PVM has been in operation since 1974, developing a porphyry copper ore body using open-pit mining techniques and processing technology to produce a copper concentrate that is well accepted by the market. The PV2 study extends the mine life by 8 years into 2026, while maintaining current fundamental mine and mill operational status. The conclusions of the QPs are detailed in the following sections of the report. Based on the site technical and financial data evaluated by the QPs for the technical report, the planned PV2 extension of the mine life into 2026 is justified.

## 1.17 Recommendations

PVM has significant mineral resource that has not been evaluated in the PV2 study due to permitted tailings storage area and property boundaries. The evaluation of the options to develop this resource may offer significant opportunity to Capstone.

# 2.0 INTRODUCTION

Capstone is a leading intermediate exploration and mining company with a focus on copper production. Capstone is based in Vancouver, British Columbia, and owns three operating mines located throughout North America. These include PVM in Miami, Arizona; Cozamin Mine in Zacatecas, Mexico; and Minto Mine in Yukon, Canada. In addition, Capstone has advanced projects in Region III, Chile, and British Columbia, Canada.

Capstone purchased PVM from BHP in October 2013, after BHP had completed a significant capital improvement project and resumed mining operations in December 2012. Sulfide ore from the open-pit mine is processed by an on-site mill, which produces copper concentrate and molybdenum concentrate. A hydrometallurgical processing circuit includes heap leach pads and an SX-EW plant for treatment of low-grade ores and production of cathode copper.

The ore deposit on the PVM property was initially mined for 10 years starting in 1943 and was known at the time as the Castle Dome Mine. It was developed as Pinto Valley Mine in 1972 by Cities Service Company. The mill began production of copper concentrate in 1974. Mining operations were curtailed and then restarted on two occasions for strategic reasons. Maintenance and capital improvements have been performed over the years to improve plant reliability and production.

Currently, the sulfide ore and SX-EW processes are both fully operational. Capstone, based on the results of this study, has scheduled mine development activities that extend milling production into 2026 and SX-EW activities to 2030.

## 2.1 Overview

This technical report was prepared for Capstone to disclose copper and molybdenum mineral resources and reserves, along with operational plans for the mine, and conforms to NI 43-101 standards with an effective date of 01 January 2014.

### 2.1.1 Terms of Reference

This technical report has been prepared by Stantec on behalf of Capstone. Other companies engaged in the preparation of the technical report include the following.

- AMEC Environment & Infrastructure, Inc.
- SRK Consulting (U.S.), Inc.
- Independent Mining Consultants, Inc.
- Kirkham Geosystems Ltd.
- KWM Consulting Inc.
- WestLand Resources Inc. (WRI)
- Adam M Consulting Inc.

Measurements are stated primarily in imperial units; weights are in pounds, short tons, and troy ounces, except in reference to copper concentrate production, certain geologic measurements, and other testwork; these are reported in metric units. Tables and graphs are clearly labeled with the units of measurement used.

## 2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

The tables in this section provide standard references used in this report. Frequently used abbreviations and acronyms are listed in Table 2-1.

**Table 2-1. Acronyms and Abbreviations**

| Acronym / Abbreviation / Symbol | Name |
|---|---|
| ADEQ | Arizona Department of Environmental Quality |
| ADWR | Arizona Department of Water Resources |
| Ag | silver |
| AMEC | AMEC Environment & Infrastructure, Inc. |
| APP | Aquifer Protection Permit |
| ASCu | acid-soluble copper |
| ASMIO | Arizona State Mine Inspector's Office |
| 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 |
| CNI | Call & Nicholas, Inc. |

**Table 2-1. Acronyms and Abbreviations**

| Acronym / Abbreviation / Symbol | Name |
|---|---|
| Cu | copper |
| Cu-Mo | copper-molybdenum |
| CWA | Clean Water Act |
| ESA | Endangered Species Act |
| F | Fahrenheit |
| 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 |
| hp | horsepower |
| HSEC | Health, Safety, Environment, and Community |
| IMC | Independent Mining Consultants, Inc. |
| IT | information technology |
| JORC | Joint Ore Reserves Committee |
| ktons | kiloton (1,000 short tons) |
| ktonnes | kilotonne (1,000 metric tonnes) |
| kW | kilowatt |
| kWh | kilowatt hour |
| lb | pound |
| LOM | life-of-mine |
| LRMC | long-run marginal cost |
| M | million |
| Ma | megaannus (million years) |
| MBWi | modified Bond work index |
| $mi^2$ | square miles |
| MLRP | Mined Land Reclamation Plan |
| mm | millimeter |
| Mo | molybdenum |
| $MoS_2$ | molybdenum disulfide |
| MSG | Multi-Sector General |
| MSHA | Mine Safety and Health Administration |
| Mt | million tonnes |
| M tons | million short tons |
| NEPA | National Environmental Policy Act |
| NI 43-101 | National Instrument 43-101 – Standards of Disclosure for Mineral Projects |
| NPV | net present value |
| NSR | net smelter return |

**Table 2-1. Acronyms and Abbreviations**

| Acronym / Abbreviation / Symbol | Name |
|---|---|
| oz | troy ounce (12 oz per lb) |
| PLS | pregnant leach solution |
| POC | point-of-compliance |
| PV1 | Pinto Valley – Phase 1 (2014–2018) |
| PV2 | Pinto Valley – Phase 2 (2014–2026) |
| PVM | Pinto Valley Mine |
| QA/QC | quality assurance / quality control |
| QP | Qualified Person |
| Quadra | Quadra FNX Mining Ltd. |
| RC | reverse circulation |
| RHD | relative half difference |
| ROM | run-of-mine |
| RQD | rock quality designation |
| Skyline | Skyline Assayers and Laboratories |
| SMARRCO | San Manuel Arizona Railroad Company |
| SMU | selective mining unit |
| SR 77 | State Route 77 |
| SRK | SRK Consulting (U.S.), Inc. |
| SRP | Salt River Project |
| Stantec | Stantec – Mining |
| SWPPP | Stormwater Pollution Prevention Plan |
| SX-EW | solvent extraction and electrowinning |
| t | metric tonne |
| TC/RC | treatment charge / refining charge |
| TCu | total copper |
| t/d | metric tonnes per day |
| TNF | Tonto National Forest |
| ton | short ton |
| tpd | tons per day |
| tph | tons per hour |
| tpoh | tons per operating hour |
| TSF | tailings storage facility |
| US$ | US Dollars |
| US 60 | US Highway 60 |
| USACE | US Army Corps of Engineers |
| USFS | US Forest Service |
| USFWS | US Fish and Wildlife Service |
| V | volt |
| VFD | variable frequency drive |
| WRI | WestLand Resources Inc. |
| WWTP | wastewater treatment plant |
| yd | yard |

**Table 2-1. Acronyms and Abbreviations**

| Acronym / Abbreviation / Symbol | Name |
|---|---|
| yr | year |
| Zn | zinc |
| ~ | approximately |
| ° | degrees |
| µm | microns |
| σ | standard deviation |

A glossary of geological and mining terms is provided in Table 2-2.

**Table 2-2. Glossary**

| Term | Explanation |
|---|---|
| Alluvium | Clay, silt, sand, gravel, or similar material deposited by running water. |
| Argillic | Hydrothermal alteration that introduces clay minerals, including kaolinite, smectite, and illite. |
| Assay | The chemical analysis of mineral samples to determine the metal content. |
| Biotite | Common phyllosilicate mineral within the mica group. |
| Bornite | A copper iron sulfide mineral with a chemical composition of $Cu_5FeS_4$. |
| Breccia | A rock composed of broken fragments of minerals or rock cemented together by a fine-grained matrix. |
| Capital expenditure | All expenditures that are not classified as operating costs. |
| Chalcopyrite | A copper iron sulfide mineral with a chemical composition of $CuFeS_2$. |
| Coefficient of variation | A unit-less statistical measure of the dispersion of the data set normalized to the mean. |
| Collar | The mouth or opening of a borehole. |
| Composite | A combination of more than one sample result to provide an average result over a larger distance. |
| Concentrate | A metal-rich product resulting from a mineral enrichment process such as gravity concentration or floatation, in which most of the desired mineral has been separated from waste material in the ore. |
| Cretaceous | A geological period from 145 Ma to 66 Ma. |
| Crushing | The initial process of reducing ore particle size in order to render it more amenable for further processing. |
| Cutoff grade | The level of mineral in an ore below which it is not economically feasible to mine it. |
| Diabase | A fine-grained mafic igneous rock with fine plagioclase crystals. |
| Dilution | Unwanted waste that is mined with ore. |
| Dip | The angle of drilling (or of a structure) relative to horizontal. |
| Disulfide | A sulfide containing two atoms of sulfur in its molecule or empirical formula. |

**Table 2-2. Glossary**

| Term | Explanation |
|---|---|
| Ecotone | A transitional area of vegetation between two different plant communities, such as forest and grassland, with some characteristics of each. |
| Eocene | Geologic epoch (56 Ma–33.9 Ma). A subpart of the Paleogene period (23.03 Ma–66 Ma). |
| Fault | The surface of a fracture along which movement has occurred. |
| Galena | A natural mineral form of lead sulfide, with a chemical composition of PbS. |
| Gangue | A nonmetallic or worthless metallic mineral associated with the ore minerals. |
| General Mining Law of 1872 | US law that authorizes and governs prospecting and mining on federal public lands. |
| Grade | The measure of concentration of a mineral within mineralized rock. |
| Histogram | Diagrammatic representation of data distribution by calculating the frequency of occurrence. |
| Hypogene | Applied to mineral deposits or ore deposits formed by ascending hot waters. |
| Igneous | Primary crystalline rock formed by the solidification of magma. |
| JORC | Joint Ore Reserves Committee. A professional code of practice that sets minimum standards for the public reporting of mineral exploration results, mineral resources, and ore reserves. |
| Kriging | Statistical concept in which data at certain locations help to estimate a value for a point that has not been sampled. Kriging weighs known grades such that variation of the estimation is minimized and the standard deviation is equal to zero. |
| Lithological | Geological description pertaining to different rock types. |
| Mean | Arithmetic average. |
| Milling | The process by which ore is crushed, ground, and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product. |
| Molybdenum | A silvery-white, very hard, metallic element in the chromium group. Used in nickel-based alloys that are heat- and corrosion-resistant. |
| Nugget | The typical difference (for an individual domain) in grade between samples taken, for those samples that are immediately adjacent to one another. |
| Pluton | A body of igneous rock that cools below the surface of the earth. |
| Population | In geostatistics, a population formed from grades having identical or similar characteristics. Ideally, one given population is characterized by a linear distribution. |
| Porphyry | An igneous rock consisting of large-grained crystals in a fine-grained matrix. |
| Protore | The rock below the sulfide zone of a supergene enrichment deposit. |
| Pyrite | An iron sulfide with the chemical composition of $FeS_2$. |
| Reserves | Mineable geological resources. |

**Table 2-2. Glossary**

| Term | Explanation |
|---|---|
| Resources | Geological resources that are both mineable and un-mineable. |
| Sample | Specimen with analytically determined grade values for the components being studied. |
| Sedimentary | Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks. |
| Sill | The variation value at which a variogram reaches a plateau. |
| Skarn | Lime-bearing siliceous rock produced by the metamorphic alteration of limestone or dolomite. |
| Smelting | A high-temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from gangue components that accumulate in a less dense molten slag phase. |
| Sphalerite | The chief zinc sulfide mineral with a chemical composition of ZnS. |
| Stockwork | An interconnected network of veins. |
| Strike | The direction of line that is formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction. |
| Sulfide | A sulfur-bearing mineral. |
| Supergene | Relating to the enrichment of mineral deposits by solutions moving downward through the rocks. |
| Tailings | Finely ground waste rock from which valuable minerals or metals have been extracted. |
| Thickening | The process of concentrating solid particles in suspension. |
| Variation | In statistics, the measure of dispersion around the mean value of a data set. |
| Variogram | A graph that displays the variability of an element by increasing the spacing between samples. |
| Vein | A sheet-like body of crystallized minerals intruded into a host rock. |
| Vug | A small cavity in rock, usually lined with a crystalline mineral incrustation. |

## 2.3 Sources of Information

The information included in this technical report is based on data collected at PVM during recent site visits, information provided by Capstone (as provided by drilling programs conducted from 2011 to 2013), public information, previous reports, and industry standards. Key documents that have been relied upon are listed in Section 27 – References.

Capstone issued two NI 43-101 technical reports for PVM in 2013. The first report, *Pinto Valley Property Mineral Resource Estimate – NI 43-101 Technical Report*, was prepared by Garth Kirkham, P.Geo., and released on 12 June 2013. The report was based on drilling programs and information provided by both BHP and the Joint Ore Reserves Committee (JORC)-compliant report, *June 2012 Mineral Resource & Ore Reserve Competent Persons Report: Pinto Valley*.

The second report, *Pinto Valley Property Mineral Resource Estimate – NI 43-101 Technical Report (Amended and Restated)*, also prepared by Garth Kirkham, was issued by Capstone on 11 December 2013. This revision includes information related to Capstone's purchase of PVM and general plans for the mine.

## 2.4 Qualified Persons

The following QPs, as defined in NI 43-101, have undertaken work in the preparation of the technical report.

Tony J. Freiman, PE, is responsible for Section 18.4.

Ms. Corolla Hoag, CPG, SME-RM, is responsible for Sections 1.11, 1.12, 1.13, 20, and 21.3.

Mr. Garth Kirkham, P.Geo., FGC, is responsible for Sections 1.3, 1.4, 1.5, 1.6, 1.7, 4, 5, 6, 7, 8, 9, 10, 11, 12, and 14.

Mr. Mel K. Lawson, SME-RM, is responsible for Sections 1.1, 1.2, 1.10, 1.16, 1.17, 2, 3, 18.1, 18.2, 18.3, 18.5, 18.6, 18.7, 18.8, 21.2.2, 23, 24, 25, 26, and 27.

Mr. Kenneth W. Major, P.Eng., is responsible for Sections 1.9, 13, 17, 21.4.2, and 21.4.3.

Mr. Adam Majorkiewicz, P.Eng., is responsible for Sections 1.14, 1.15, 19, 21.1, 21.4.4, 21.4.5, 22.1.2, 22.1.3, 22.1.4, 22.1.5, 22.1.6, 22.2, 22.3, and 22.4.

Mr. John Marek, PE, SME-RM, is responsible for Sections 1.8, 15, 16, 21.2.1, 21.4.1 and 22.1.1.

## 2.5 Site Visits and Scope of Inspections

Site visits were performed as follows.

Mr. Tony J. Freiman worked at the PVM site at various times in 1997 and 1998. Mr. Freiman also made numerous trips to the mine site between May and November 2013 for work on TSF3 and TSF4. Most recently, Mr. Freiman visited the project on 19 December 2013, 06 and 09 January 2014, and 12 February 2014. During the site visits, Mr. Freiman examined the TSFs and surrounding areas, supervised ongoing geotechnical activities pertaining to tailings embankment design, and held conversations with the mine engineering group pertaining to the design and operation of the TSFs.

Ms. Corolla Hoag visited the Pinto Valley property numerous times from 2006 to 2013 for environmental permitting, environmental reviews, and site characterization work. Her site visit was on 13 January 2014. The purpose of the visit was to discuss the current status of the Pinto Valley Health, Safety, Environment, and Community (HSEC) training programs; environmental permits; site monitoring and management plans; and operational expenditure budgets with PVM HSEC management personnel, and to tour the mining, processing, and waste disposal facilities.

Mr. Garth Kirkham most recently visited the property on 14 May 2013 and the analytical laboratory facilities on 15 May 2013. The site visit included an inspection of the mine site infrastructure, core logging facilities, offices, pit, outcrops, core storage facilities, core receiving area, and core sawing stations.

Mr. Mel K. Lawson made numerous site visits from April 2012 to present, the most recent of which was on 14 March 2014. During the site visits, Mr. Lawson was involved in the engineering, procurement, and construction management activities associated with the PV1 restart of PVM operations by BHP, and in ongoing engineering studies related to BHP's plans for PV2 operations. The purpose of the most recent visit was to inspect overall site conditions and infrastructure.

Mr. Kenneth W. Major's most recent site visit was 06–08 January 2014. The purpose of this visit was to review the mill and heap leach process facilities.

Mr. Adam Majorkiewicz visited the property 12–13 March 2014. The purpose of this visit was to review the project scope and collect relevant information.

Mr. John Marek's site visit was on 17 December 2013. The purpose of this visit was to conduct a mine tour, talk with mine engineering staff, and ensure that he was aware of all constraints that may affect future planning studies.

# 3.0 RELIANCE ON OTHER EXPERTS

This technical report has been prepared by QPs on behalf of Capstone. The QPs have relied on the legal and permitting status provided by Capstone in formulating their conclusions, opinions, and estimates, and on information provided by other technical experts. The relevant QPs believe that it is reasonable to rely on these experts, based on the fact that the experts have the necessary education, professional designations, and experience on matters relevant to the technical report.

John Marek, QP, relied upon Edward Wellman, PE, geotechnical consultant for Sections 15 and 16.

Adam Majorkiewicz, QP, relied upon Joe Fernandez, mining engineer, AJF Mining Consultants LLC, for the review and compilation of operating costs used in this report.

See Section 27 for a complete list of references used in this report.

## 3.1 Mineral Tenure and Surface Rights

During the acquisition of PVM and associated assets, Capstone completed a comprehensive due-diligence review of all legal, tenure, environmental, and permit conditions associated with the transfer of ownership from BHP to Capstone. All legal obligations, titles, and permits have been transferred to Capstone as a condition of the October 2013 sale closure. As a consequence of this process, the QPs have relied on the information provided by Capstone concerning legal opinions, mineral title, surface rights, and environmental permits to be current with no outstanding issues that would affect the opinions expressed in this technical report.

For the purpose of this report, the QPs have relied on mineral tenure and surface rights information provided by Capstone and are not qualified to express any legal opinion with respect to the property titles and current ownership. The QPs have not researched mineral rights, mining claims, or underlying property agreements for PVM and express no opinion as to their status.

## 3.2 Environmental and Work Program Permitting

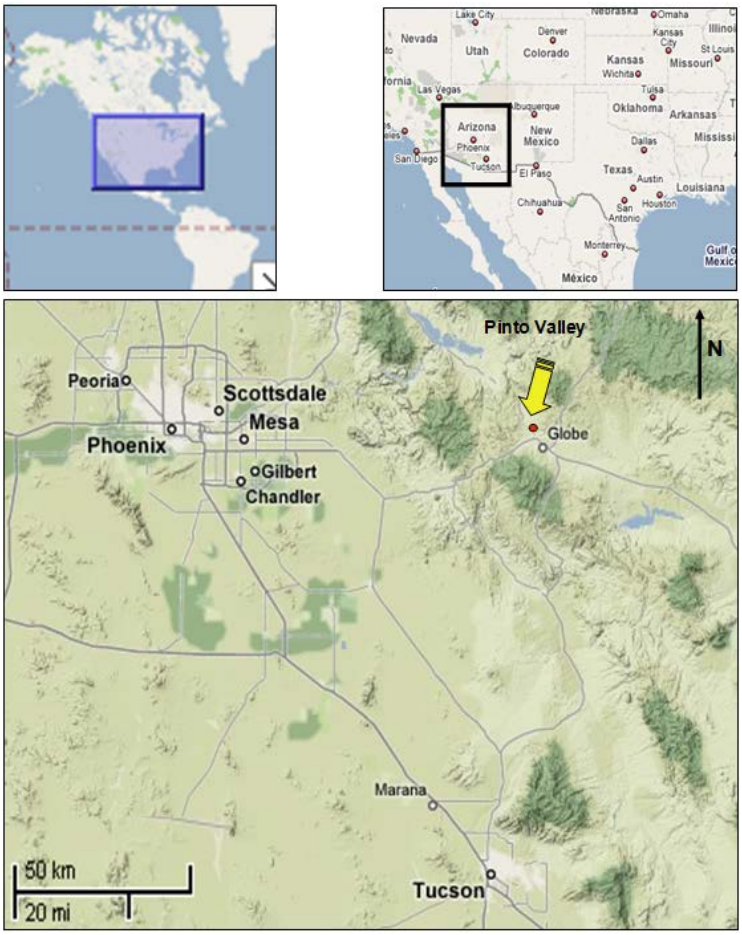
The QP has obtained information regarding historic PVM site characterization studies and the current status of Capstone's health, safety, and community outreach programs from Capstone's HSEC staff. The QP has obtained information regarding the current environmental permitting status of PVM through opinions and data supplied by experts retained by Capstone and from information supplied to the QP by the Capstone environmental staff. This information was used in Section 20, and the numerous site characterization documents are referred to, as relevant, in Section 20 of this report. The QP has fully relied upon, and disclaims responsibility for, information derived from such experts.

# 4.0 PROPERTY DESCRIPTION AND LOCATION

## 4.1 Location

The PVM property includes an open-pit mine and mill complex, heap leach pads, pregnant leach solution (PLS) collection areas, and SX-EW facilities. This property is located at the west end of the Globe-Miami mining district, approximately 6 miles west of the town of Miami and 11 miles west of Globe in Gila County, Arizona, at 33°23′32″ N and 110°58′15″ W (Figure 4-1). The property consists of approximately 9.5 square miles ($mi^{2)}$ of contiguous patented land claims and several small nearby claims, at an elevation ranging between 3,000 ft and 4,900 ft above mean sea level.

**Figure 4-1. Pinto Valley Mine Location Map**




BHP 2013

## 4.2 Property Description

PVM is an open-pit operation that is approximately 1,100 ft deep, 5,000 ft wide, and 7,000 ft long. The pit is L-shaped and is near the on-site infrastructure. There are suitable maintenance facilities for large earth-moving equipment and for the mill and general personnel. Two previous tailings dams have been rehabilitated and two tailings dams are currently operational. An SX-EW facility is located on the western edge of the property.

## 4.3 Tenure, Ownership, and Encumbrances

On 28 April 2013, Capstone entered into a purchase agreement with BHP, pursuant to which Capstone proposed to purchase, through a wholly owned US subsidiary, 100% interest in PVM and SMARRCO for $650 M. The purchase was finalized in October 2013.

PVM is a combination of fee land, patented mining and mill site claims, and unpatented mining and mill site claims. As a whole, the land supports open-pit mining, ore processing, tailings storage, waste rock disposal, and the operation of an SX-EW plant. The unpatented mining claims and mill sites are accessible under the provisions of the US General Mining Law of 1872, subject to approval from the USFS after the completion of an environmental analysis under NEPA. The document is supported by a proposed Plan of Operations governing portions of the property. The NEPA review process includes interagency consultation on project alternatives and the mitigation of environmental impacts.

Use of the fee lands and patented mining claims and mill sites are governed by an MLRP and an APP, both issued by the ADEQ. The applicable permits, rights, and Plan of Operations have all been transferred from BHP to Capstone.

The core of PVM consists of 69 patented lode mining claims. Also included in the property are 53 patented mill sites. Adjacent to and nearby the patented claims are 329 unpatented lode mining claims and mill sites. Most of the unpatented mining claims and mill sites were staked on federal land administered by the USFS, but a limited number of the unpatented mining claims and mill sites are on federal land administered by the Bureau of Land Management (BLM). Seven parcels of fee (private) land are associated with the property.

Capstone owns the patented mining claims and fee land parcels, which are private lands that provide the owner with both surface and mineral rights. Capstone also owns the patented mill sites. The patented mining claim block, located in the core of the property, is indicated in the field by surveyed brass caps on short pipes cemented into the ground. The fee lands are located by legal description and recorded at the Gila County Recorder's Office. The patented mining claims, mill sites, and fee lands are subject to annual property taxes. As long as the property taxes are paid annually on these claims, there is no expiration date.

Capstone also owns the unpatented lode mining claims and mill sites that are adjacent to and nearby the patented claims. Wooden posts and stone cairns mark the unpatented mining claim corners, end lines, and discovery monuments; all of these have been surveyed. The unpatented mining claims and mill sites have no expiration date and are maintained by filing the required documents with the BLM, providing the required records to Gila County, and paying an annual maintenance fee to the BLM of $140 per claim. A 2% net smelter return (NSR) royalty applies to 26 of the unpatented mining claims located outside of the property boundary.

## 4.4 Permits

All required permits for commercial operations have been obtained and are current, except for the USFS Plan of Operations, which is in the process of being amended. Refer to Section 20 – Environmental Studies, Permitting, and Social or Community Impact for detailed permit information, including a list of required permits. The current key environmental permits are listed in Table 4-1.

**Table 4-1. Pinto Valley Permits Status**

| Permit | Permit Number | Expiration | Comments |
|---|---|---|---|
| USFS Plan of Operations | N/A | | Renewal in progress. |
| Air Quality Class II Synthetic Minor | 54118 | 05 Aug 2017 | Transferred 11 Oct 2013. |
| AZPDES[1] Process Water | AZ0020401 | 27 Nov 2013 | Administratively continued, renewal in progress. |
| AZPDES Stormwater Multi-Sector General Permit | AZMSG 2010-003 / AZMSG-78423 | N/A | Approval date 11 Oct 2013. |
| Area Wide APP | P-100329 | Life of Facility | Signed 11 Oct 2013. |
| General APP – Concentrate Truck Wheel Wash | P-100639 | 20 Sep 2016 | Type 3.03 Vehicle & Equipment Washes. |
| Public Water System | AZ0404321 | Life of Facility | Non-Transient Non-Community Water System. |
| Hazardous Waste | EPA[2] ID: AZT000624353 | N/A | Hazardous Waste Site ID. |
| Special Waste | Generator Number 320276 | N/A | Special Condition PCS Only. |
| Non-Municipal Landfill | ICU95-821, Account No. 164 | Life of Facility | Place ID 145013, became part of the APP in 2011. |
| Off-Road Tire Burial | Burial Cell A | Life of Facility | Per AAC[3] R-18-8-704. |
| MLRP | N/A | Life of Facility | June 1998 approval, updated in 2013. |
| CERCLA[4] Preliminary Assessment Low Hazard 2003 | N/A | N/A | Hazardous Waste Generator I.D.: AZT000624353. Assigned December 1997 – CERCLIS[5]. |
| ADWR[6] Dam Safety Permit | 4.13 | Life of Facility | Gold Gulch 2 Dam. |
| ADWR Dam Safety Permit | 4.16 | Life of Facility | Slack / Conklin Dam. |
| ADWR Dam Safety Permit | 4.17 | Life of Facility | Gold Gulch 1A Dam. |
| Radioactive ARRA[7] | 04-008 | 31 Jul 2015 | Radioactive Material. |
| Radioactive ARRA | 04-I-2050 | 31 May 2016 | X-Ray Registration. |
| ADWR Well Registration Permit | Various | N/A | Permits for water production and monitoring wells. |

Compiled by SRK from data provided by Capstone 2014.

[1] Arizona Pollutant Discharge Elimination System

[2] Environmental Protection Agency

[3] Arizona Administrative Code

[4] Comprehensive Environmental Response, Compensation, and Liability Act

[5] Comprehensive Environmental Response, Compensation, and Liability Information System

[6] Arizona Department of Water Resources

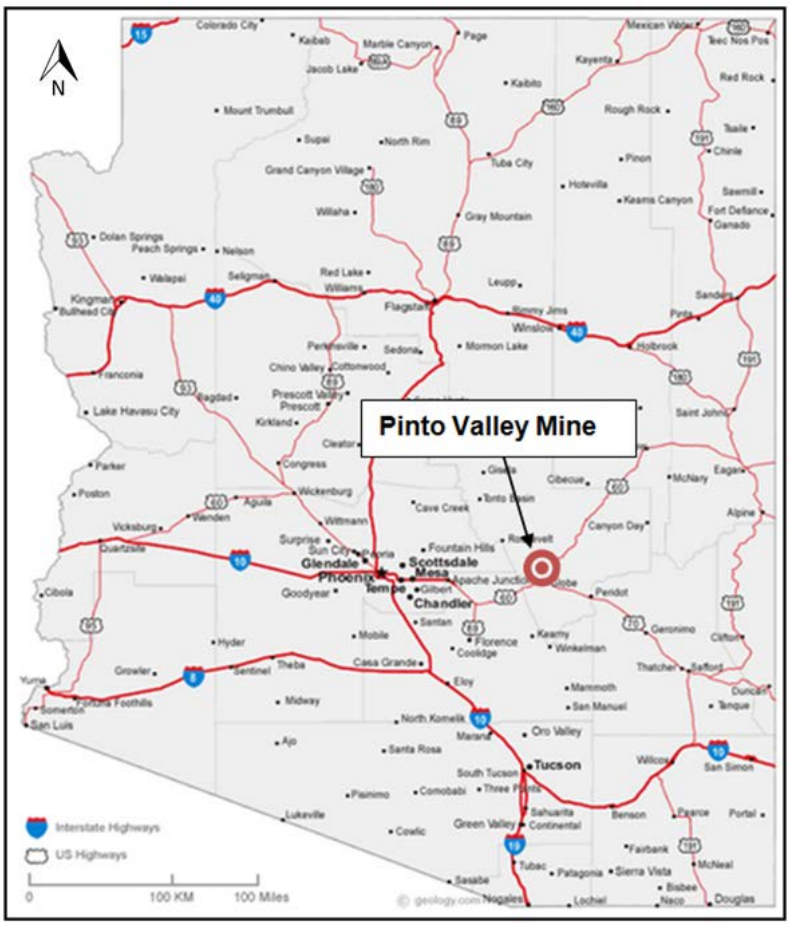
[7] American Recovery and Reinvestment Act

# 5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

## 5.1 Accessibility

PVM is located in Gila County approximately 6 miles west of Miami, Arizona. Access to the mine is via US 60, approximately 80 miles east of Phoenix, then 3 miles on FR 287 (Figure 5-1). The site can also be accessed from Tucson, Arizona (100 miles to the south), by traveling north on SR 77, then west on US 60 to FR 287.

**Figure 5-1. Pinto Valley Mine Location**




geology.com 2014

## 5.2 Climate

PVM's moderate, semi-arid regional climate allows for year-round operation. The average annual precipitation is 18.92 inches and falls in a bimodal pattern. Most rainfall occurs during the winter and summer months, with dry periods in the spring and fall. Heavy rains, windstorms, and lightning may occur during the summer period of July to early September. These are generally of short duration and occur in the afternoon. Precipitation during the winter months (December–March) is less intense and usually occurs as long, steady storms. Although snow may occur, it does not typically accumulate. May and June are typically the driest months of the year and may result in local drought conditions.

The National Oceanic and Atmospheric Administration's *Climate Atlas of the United States* and the Western Regional Climate Center records include data from a station in Miami, Arizona, approximately 6 miles east of the mine site. The average annual maximum temperature for the period of record at this station is 77°F.[1] July is the warmest month, with an average maximum temperature of 97°F. The average annual minimum temperature for January, the coolest month, is 34°F.

In 2011, the town of Miami, located 6 miles east of the mine, had approximately 1,800 residents, and the town of Globe (the Gila County seat), located 11 miles east of the mine, had approximately 7,500 residents. Copper mining provides the largest number of jobs in the area. Due to the long-standing mining tradition in the area, local services are already in place, and the current level of services is deemed adequate for the needs of the mine. Details of the infrastructure for the PVM site are included in Section 18 herein.

## 5.3 Physiography

PVM is located in east-central Arizona on federal land, 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 is entirely within the Pinto Creek watershed, where local elevations range from 3,000 ft to 5,000 ft above mean sea level.

[1] The period of record for the Miami, Arizona, station is from 1914 to 2005.

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 western boundary of the property. Pinto Creek flows from the south to the north into Roosevelt Lake, an artificial impoundment constructed along the Salt River.

Two types of hydrogeologic units are present at the site. The first and uppermost is the alluvial system, which is a near-surface groundwater system consisting of shallow-circulating water moving in the alluvium and the upper weathered portions of the underlying bedrock. The second is the bedrock system, which consists of deeply circulating groundwater moving through fractures and joints in the consolidated bedrock underlying the area. Some units / sections of the bedrock system act more like the alluvial system, including deeper weathered portions of the fractured bedrock and the Gila conglomerate.

PVM is near the boundary of areas mapped as the Interior Chaparral biotic community and the Arizona Upland subdivision of Sonoran desertscrub biotic community. Plant species on the property that are characteristic of the Arizona Upland community include saguaro, blue palo verde, velvet mesquite, catclaw, four-wing saltbush, ocotillo, and Engelmann prickly-pear. Plant species more characteristic of the Interior Chaparral community include Arizona white oak, shrub live oak, one-seed juniper, point-leaf manzanita, sugar sumac, skunkbush, and canotia.

A variety of mammals, birds, reptiles, and amphibians make up the wildlife community at PVM. Because the property is located on the ecotone between two major plant communities, wildlife diversity on the site represents species adapted to both communities. Common wildlife species observed on site include rock squirrel, coyote, mule deer, Gambel's quail, Cooper's hawk, mourning dove, Bell's vireo, western scrub-jay, phainopepla, and canyon towhee. Most of the species observed have wide environmental tolerances and are present in both plant communities on the property.

The southwestern parts of the mine are near the perennial reach of Pinto Creek. The Pinto Creek riparian zone is dominated by Fremont cottonwood, Goodding willow, Arizona sycamore, Arizona cypress, and seep willow.

# 6.0 HISTORY

The Globe-Miami 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 Globe-Miami mining district.

PVM originated as Miami Copper Company in 1909. In 1960, the Tennessee Corporation took over Miami Copper Company. In 1969, Cities Service Company merged with the Tennessee Corporation. Occidental Petroleum Corporation acquired Cities Service Company in late 1982 and sold the Miami operations to Newmont Mining Corporation in February 1983. At this time, the company's name was changed to Pinto Valley Copper Corporation. In November 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 December 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 purchased Pinto Valley Operations, now referred to as Pinto Valley Mine.

A chalcocite-enriched zone of the deposit was mined from 1943 until 1953 as the Castle Dome underground mine. Subsequently, PVM went into production in 1974. Ore from the operation was processed, producing a copper concentrate. The SX-EW plant began processing PLS from the leach dumps in 1981. In February 1998, mining and milling were suspended. The mill was placed under care and maintenance, and the mining equipment fleet was sold. Operating and environmental permits were maintained during the suspension of operations, as were the water and electrical systems. Cathode copper production continued during the suspension of sulfide operations at the SX-EW facilities.

After an extensive rehabilitation project, the mine and mill were restarted in late 2007. PVM operated for 18 months and was then placed under care and maintenance in January 2009.

In 2011, BHP commissioned a study to determine the extent of rehabilitation required to restart mining operations. BHP approved the restart plan in November 2011, and mill operations resumed in December 2012. The declared resource and reserve statements (JORC-compliant) for PVM by BHP are published in *BHP Billiton Annual Report 2013*.

Table 6-1 and Table 6-2 are for historical reference only. Capstone has not completed the work necessary to verify the classification of the resource and reserve statement and is not treating the resource and reserve statement as NI 43-101 defined mineral resources and mineral reserves verified by a QP. The historical estimates should not be relied upon. PVM requires considerable further evaluation, which Capstone's management and consultants intend to carry out in due course. Capstone does not have a copy of the report that includes the resource and reserve statement signed by a professional geologist; therefore it cannot verify the resource or reserves or comment on whether the estimate was made in compliance with the current standards. Capstone is not relying on these historical estimates.

As of 30 June 2012, the PVM copper resources and reserves were reported as follows.

**Table 6-1. PVM 2012 BHP JORC-Compliant Resources (metric units) As of 30 June 2012**

| Ore Type | Measured Resources | | Indicated Resources | | Inferred Resources | | Total Resources* | |
|---|---|---|---|---|---|---|---|---|
| | Mt† | %TCu | Mt | %TCu | Mt | %TCu | Mt | %TCu |
| Sulfide | 48 | 0.37 | 142 | 0.41 | 1 | 0.37 | 191 | 0.40 |
| Low-Grade Leach | 15 | 0.20 | 58 | 0.19 | 5 | 0.18 | 78 | 0.19 |
| **Total** | **63** | **0.33** | **200** | **0.35** | **6** | **0.22** | **269** | **0.34** |

*Resources are inclusive of reserves. Sulfide resources and reserves are reported at a 0.25% TCu cutoff grade, while the low-grade leach resources and reserves are reported at a 0.10% TCu cutoff grade. BHP did not publish or report molybdenum resources or reserves.

†Million tonnes

**Table 6-2. PVM 2012 BHP JORC-Compliant Proven and Probable Reserves (metric units) As of 30 June 2012**

| Ore Type | Total Reserves | |
|---|---|---|
| | Mt | %TCu |
| Sulfide | 75 | 0.40 |
| Low-Grade Leach | 13 | 0.21 |
| **Total** | **88** | **0.37** |

PVM resumed mining in November 2012. Resources were recalculated and published in *BHP Billiton Annual Report 2013*. As of 30 June 2013, the PVM copper resources and reserves were reported as follows in Table 6-3 and Table 6-4.

**Table 6-3. PVM 2013 BHP JORC-Compliant Resources (metric units) As of 30 June 2013**

| Ore Type | Measured Resources | | Indicated Resources | | Inferred Resources | | Total Resources* | |
|---|---|---|---|---|---|---|---|---|
| | Mt | %TCu | Mt | %TCu | Mt | %TCu | Mt | %TCu |
| Sulfide | 127 | 0.39 | 353 | 0.39 | 85 | 0.37 | 565 | 0.39 |
| Low-Grade Leach | 49 | 0.20 | 224 | 0.18 | 106 | 0.17 | 379 | 0.18 |
| **Total** | **176** | **0.34** | **577** | **0.31** | **191** | **0.26** | **944** | **0.31** |

*Resources are inclusive of reserves. Sulfide resources and reserves are reported at a 0.25% TCu cutoff grade, while the low-grade leach resources and reserves are reported at a 0.10% TCu cutoff grade.

**Table 6-4. PVM 2013 BHP JORC-Compliant Proven and Probable Reserves (metric units) As of 30 June 2013**

| Ore Type | Proven Reserves | | Probable Reserves | | Total Reserves | |
|---|---|---|---|---|---|---|
| | Mt | %TCu | Mt | %TCu | Mt | %TCu |
| Sulfide | 25 | 0.37 | 42 | 0.41 | 67 | 0.39 |
| Low-Grade Leach | 6 | 0.21 | 7 | 0.20 | 13 | 0.21 |
| **Total** | **31** | **0.34** | **49** | **0.38** | **80** | **0.36** |

Sulfide mining resumed at PVM, with the production of 16.6 ktons Cu concentrate and 4.9 ktons of Cu cathode for the financial year ending 30 June 2013.

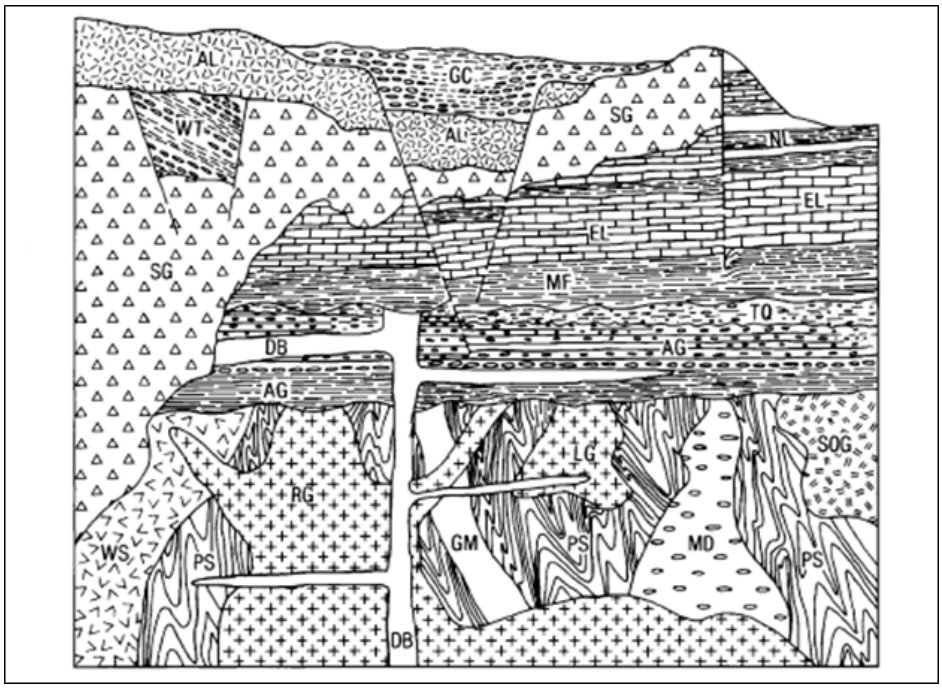
# 7.0 GEOLOGICAL SETTING AND MINERALIZATION

## 7.1 Geological Setting

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 (Creasey 1980) associated with Paleocene (59 Ma–63 Ma) 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, Tertiary, and Quaternary age. Figure 7-1 shows a diagrammatic sketch that indicates the age and spatial relationships of the major rock units. Figure 7-2 shows a simplified geological map of the western half of the district.

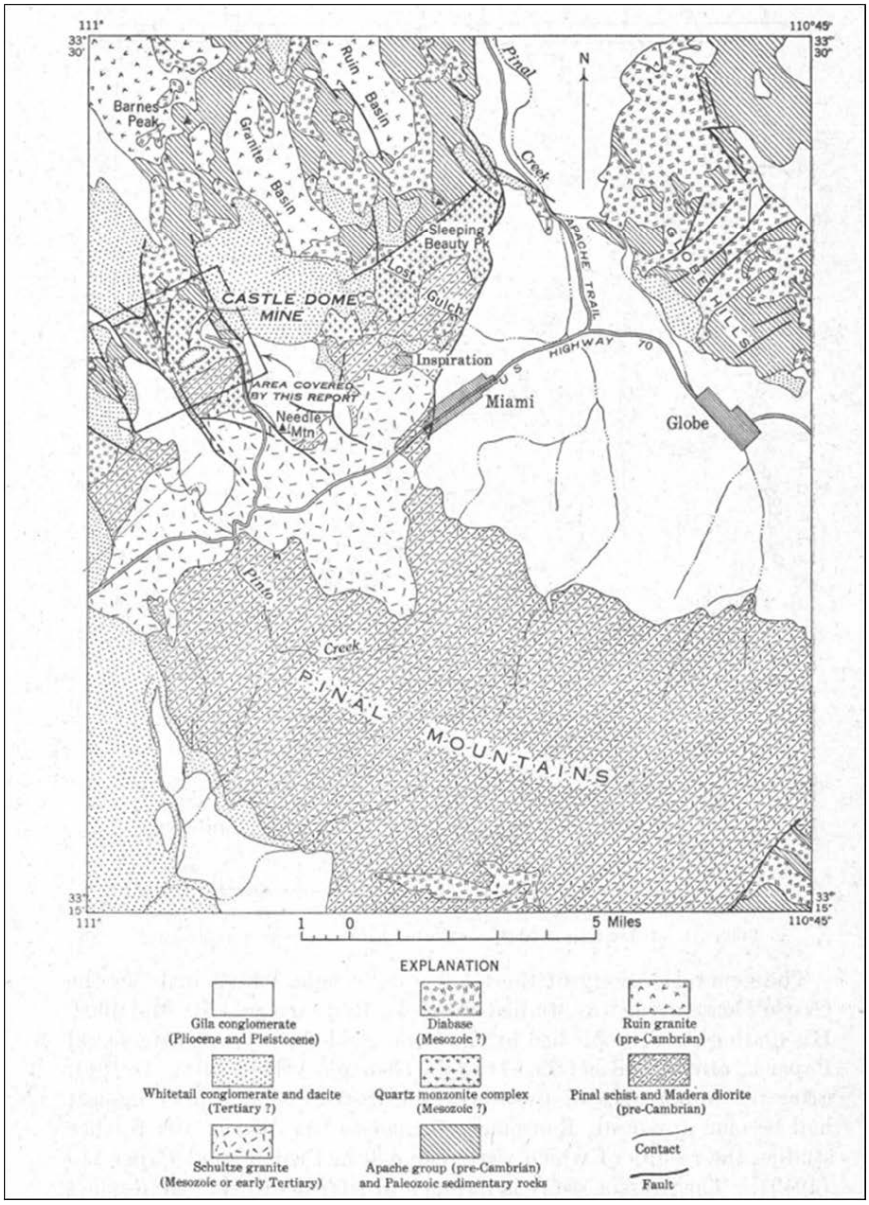
**Figure 7-1. Diagrammatic Sketch of the Geologic Relations of the Rock Units in the Globe-Miami District**




Creasey 1980

Abbreviations used for Figure 7-1 are as follows: AG, Apache group; AL, Apache Leap tuff; DB, diabase; EL, Escabrosa limestone; GC, Gila conglomerate; GM, granite of Manitou Hill; LG, Lost Gulch monzonite; MD, Madera diorite; MF, Martin limestone; NL, Naco limestone; PS, Pinal schist; RG, Ruin granite; SG, Schultze granite; SOG, Solitude granite; TQ, Troy quartzite; WS, Willow Spring granodiorite; WT, Whitetail conglomerate.

**Figure 7-2. Geological Map of the Western Half of the Globe-Miami District**




Creasey 1980

Precambrian basement rocks largely consist of early Proterozoic Pinal schist (~1,700 Ma) intruded by granites correlative with 1,450 Ma peraluminous two-mica granite batholiths that make up 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 1,100 Ma 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.

During the Eocene epoch (60 Ma–62 Ma), 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 (Olmstead and Johnson 1966, 143–50). Creasey (1980) refers to this as the porphyry phase of the Schultze granite. A separate granite porphyry has been mapped at PVM, Copper Cities, Diamond-H, and Miami East, and is seen near the vein-controlled mineralization at Old Dominion. Rocks identical to this granite porphyry are seen in the Miami-Inspiration mineral deposit, but they have not been systematically mapped as a separate unit.

Tertiary sedimentary and volcanic rocks cover the mineralized units. The Whitetail conglomerate was formed as a result of regional uplift (~32 Ma). Rocks of the Whitetail conglomerate contain weathered clasts of older rocks in a red iron oxide-rich, very fine-grained matrix, and detrital to exotic copper mineralization. A Miocene ash-flow tuff, known as the Apache Leap tuff, covered the area following the Whitetail conglomerate (21 Ma). Further Basin and Range faulting and subsequent erosion produced the Tertiary to Quaternary Gila conglomerate from all older rocks. On the west side of the PVM open pit, the Gila conglomerate contains a basalt sill.

## 7.2 Mineralization

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 (Peterson 1962, 151). The primary minerals of the porphyry copper deposits are chiefly pyrite and chalcopyrite with minor amounts of molybdenite; gold and silver are recovered as byproducts. Sphalerite and galena occur locally in very small amounts. Silicate alteration associated with the deposits includes potassic, argillic, sericitic, and propylitic alterations.

The PVM 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 et al. 1951).

The primary host rock for the PVM 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).

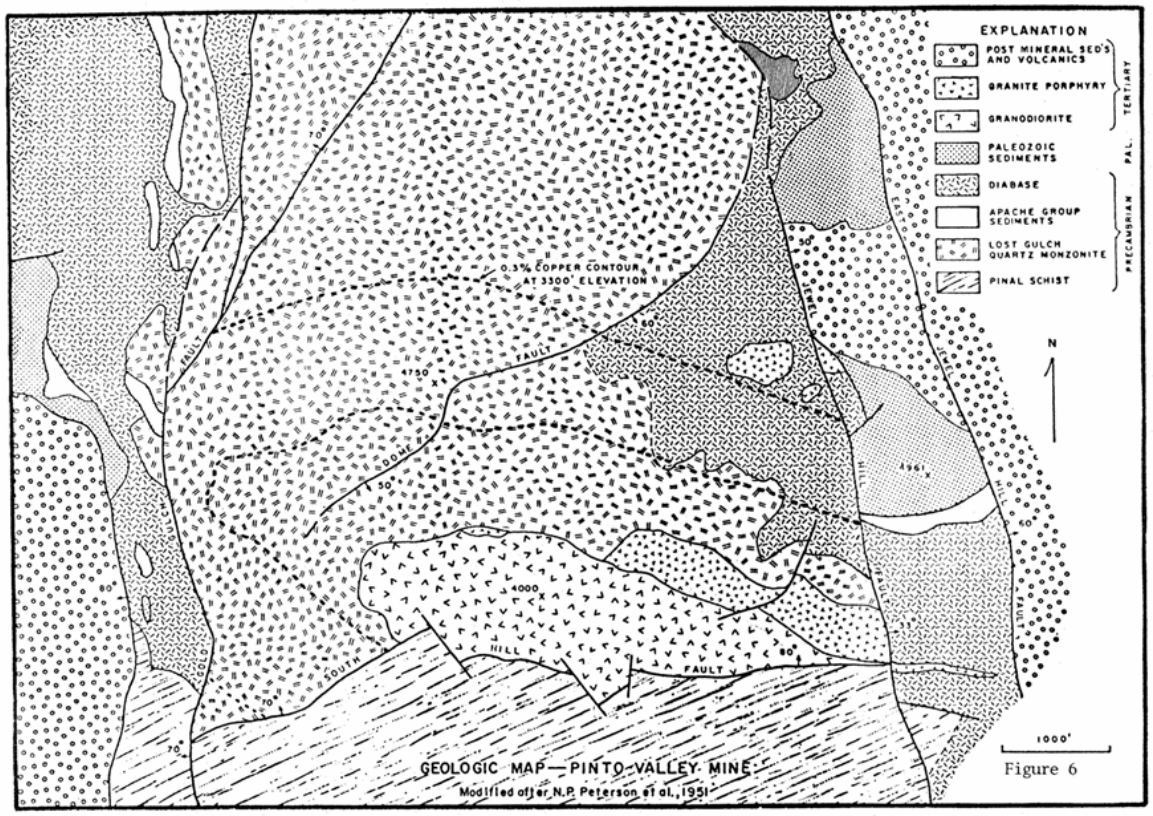
Primary sulfide ore minerals consist of pyrite, chalcopyrite, and minor molybdenite that occur in veins and microfractures, and less abundantly as disseminated grains, predominantly in biotite sites. The ore zone grades outward into a pyritic zone with higher total sulfide content, and the ore zone grades 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 at PVM 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, shown in Figure 7-3 as mimicking the pit outline. While quartz monzonite constitutes ore (more than 0.3% copper), the granite porphyry does not usually contain ore grades (~0.15%–0.2% copper). Granite porphyry contains sulfide veins, but generally lacks disseminated sulfides in biotite sites.

The shell has the appearance of a hook in plan view (see Figure 7-3) and mimics the pit outline. Rock located south of the ore has decreasing sulfide content and numerous barren quartz veins. This area has been interpreted as a low-grade core, and this low-grade zone corresponds spatially with the granite porphyry, which is seen as a poor lithologic host for ore-grade mineralization elsewhere in the deposit. Rock located north of the ore has progressively more abundant, late-stage quartz-pyrite-sericite veins.

**Figure 7-3. Surface Geology Map of Pinto Valley Mine**



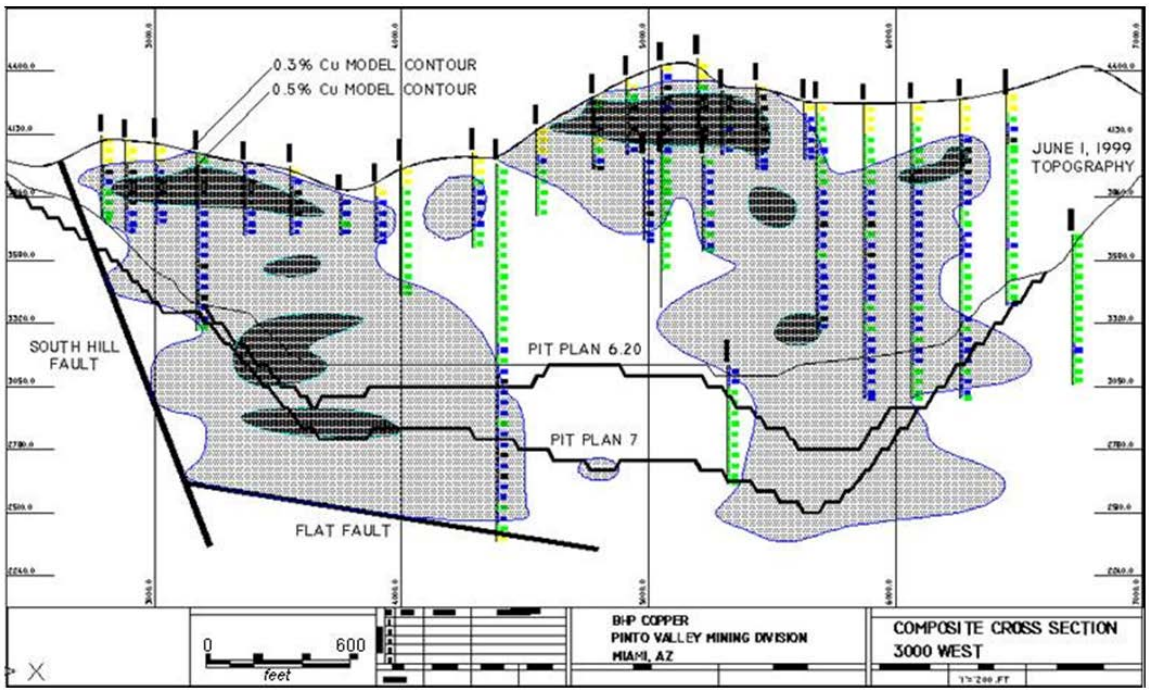

Peterson et al. 1951

Cross section 3000 West (Figure 7-4) shows drill holes and sulfide copper block model contours based on BHP's JORC-compliant 2007 block model. The section is drawn through the "hook" in quartz monzonite west of the large granodiorite and granite porphyry exposures. It further illustrates a central low-grade zone surrounded by an ore shell. The core of the shell dips steeply to the north.

The South Hill Fault cuts the ore shell and associated alteration to the south. The shallow, dipping Flat Fault cuts off the ore beneath the southern limb of the grade shell. The sections suggest 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. 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 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.

**Figure 7-4. Ore Body Cross Section 3000 W (looking west)**




BHP 2007

Diabase forms thin dikes in pit exposures. These dikes commonly contain higher copper content than the surrounding quartz monzonite. 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.

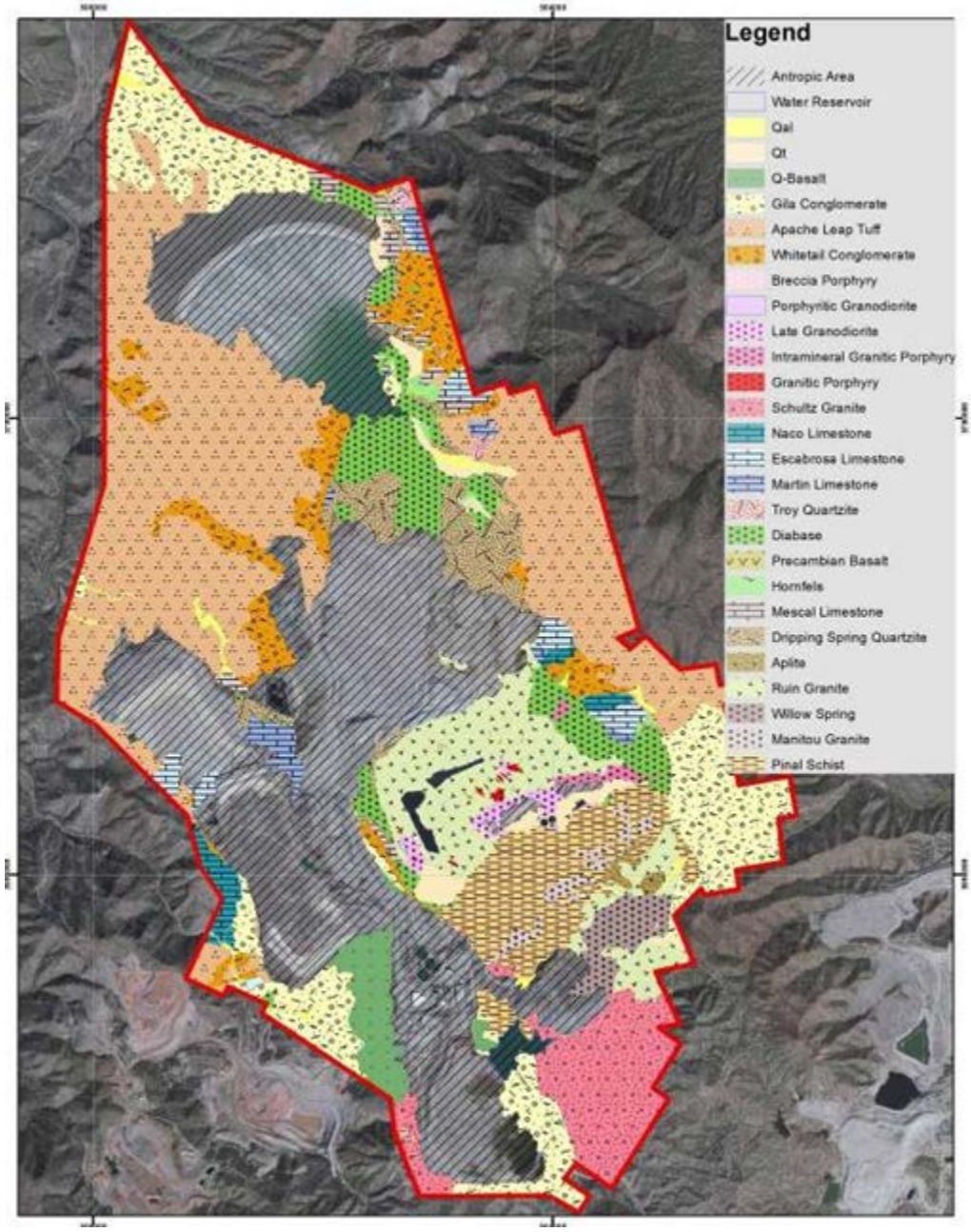
A total of 45 rock samples were submitted for analysis using ioGAS geostatistics. Both transmitted- and reflected-light thin sections were prepared for petrographic analysis of select samples. Spectral analysis of clays and micas from select sites was performed to determine if clay species were of hydrothermal origin.

Mapping of the regional PVM tenement has identified a number of new mineral occurrences. Copper mineralization was observed at a number of contacts between two genetically different granitic bodies. Surface exposures of porphyry breccia systems were also found bearing pyrite and chalcopyrite in a jarosite-dominated oxide precipitate. These sites were analyzed with field-portable TerraSpec, which detected dickite, indicating hydrothermal alteration. A number of massive magnetite / hematite seams bearing manganese, pyrite, and copper were mapped in skarn contacts around the fringe of limestone bodies. A skarn occurrence was found in contact with an intrusive diabase unit, bearing a stockwork of sulfide-rich "D" veins. A number of old workings were also found throughout the area, testing a range of copper-bearing geological settings such as porphyry stock, pegmatitic intrusive, mineralized skarn, intrusive contact, and oxide occurrence under tertiary cover.

### 7.2.1 Local Geology and Alteration

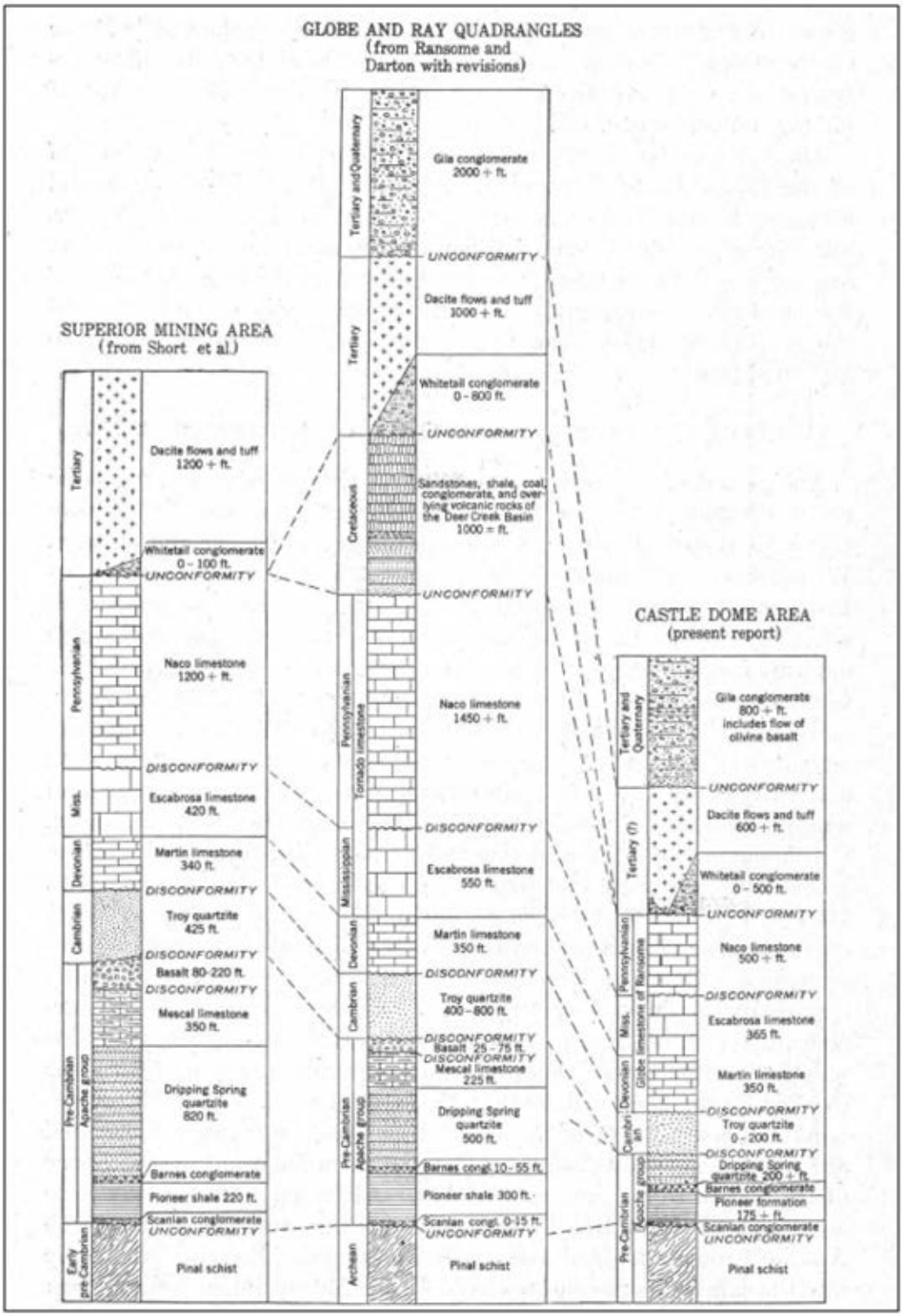
The following sections describe the main rock, alteration, and mineralization types on site. Figure 7-5 shows PVM geology in plan view, and Figure 7-6 illustrates the generalized columnar sections of sedimentary and volcanic rocks for the Castle Dome area. Figure 7-7 shows the PVM alteration types in plan view.

**Figure 7-5. Pinto Valley Mine Geology Plan**



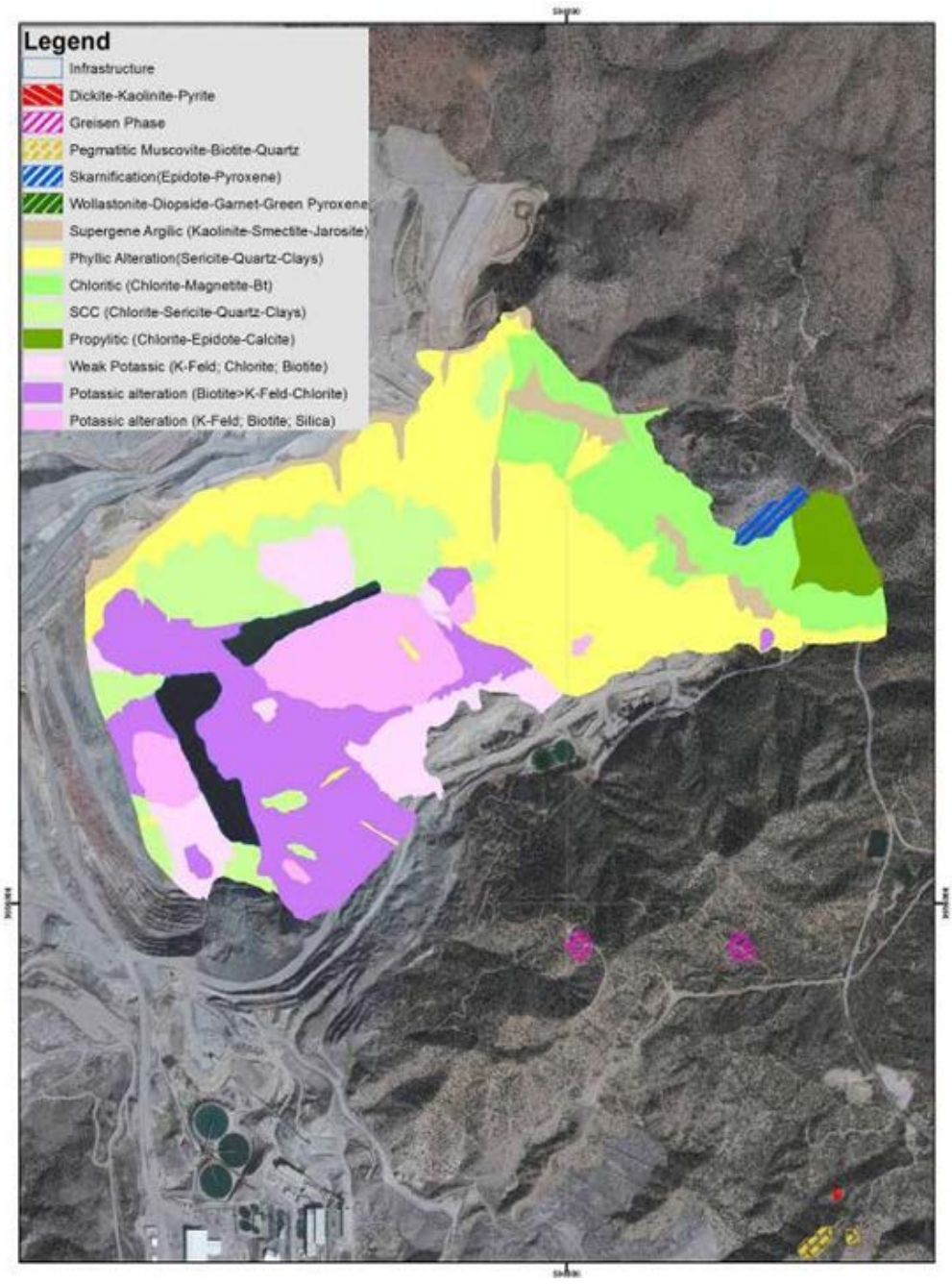

BHP 2012

**Figure 7-6. Generalized Columnar Sections of Sedimentary and Volcanic Rocks for the Castle Dome Area**




Peterson et al. 1951

**Figure 7-7. Pinto Valley Mine Alteration Plan**




BHP 2012

### 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 inch wide. The rock is extensively deformed, bearing tight to isoclinal folding and faulted extensively by various intrusive events.

### 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.

### 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.

### 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.

### 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.

### 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.

### 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.

### 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.

### Whitetail Conglomerate

Tertiary in age, the Whitetail conglomerate is distinguished from other sedimentary units by the exclusion of dacite and tuff lithologies. 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.

### 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 Tertiary and Quaternary age. The unit is distinguished by the inclusion of all local lithologies: 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.

#### Alluvium

Tertiary 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.

## 7.3 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.

### 7.3.1 Pre-Mineralization Intrusives

#### 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 $mi^2$ 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 Tertiary 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 mafics (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.

### Willow Spring Granodiorite

Willow Spring granodiorite is an intrusive unit that outcrops in the southeastern sector of the study area, occupying approximately 0.16 $mi^2$. This unit outcrops as elongated bodies trending north-northeast, intruded by Precambrian Ruin granite and Tertiary 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 mm to 10 mm; and subhedral-euhedral plagioclase (38%).

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

### Ruin Granite

Ruin granite is an intrusive unit that outcrops over an area of approximately 0.81 $mi^2$, 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 Tertiary 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%).

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 intrusives 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).

### 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 mafics minerals and lesser plagioclase. This unit has different phases, with early medium to coarse textures that range to later, fine-grained textured intrusions.

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

### Schultze Granite

Schultze granite is Tertiary 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 $mi^2$ 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.

## 7.3.2 Intra-Mineralization Intrusive Phases

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.

### 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 mm 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.

### 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 country rock, Pinal schist, and diabase lithologies.

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.

### 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.

### 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 (see Figure 7-3 and Figure 7-5).

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.

### Breccia Porphyry

Near the southeastern boundary of the PVM area, two sub-outcrops of a unit with intrusive brecciated features were found. This unit is called the breccia porphyry and intrudes the Ruin granite as a small dike swarm (see Figure 7-5).

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.

## 7.4 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 Tertiary 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 7-8).

The oldest fault observed is the 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 southeastern dip that has truncated mineralization of the PVM deposit; this fault has placed the Pinal schist over the Ruin granite.

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.

**Figure 7-8. Location and Distribution of the Main Structures of the Pinto Valley District**

Note: South Hill Fault, Gold Gulch Fault; Dome Fault; and Jewel Hill Fault are secondary structures. Other secondary structures are indicated in blue.

# 8.0 DEPOSIT TYPES

PVM is classified as a copper-molybdenum porphyry system. A large volume of 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.

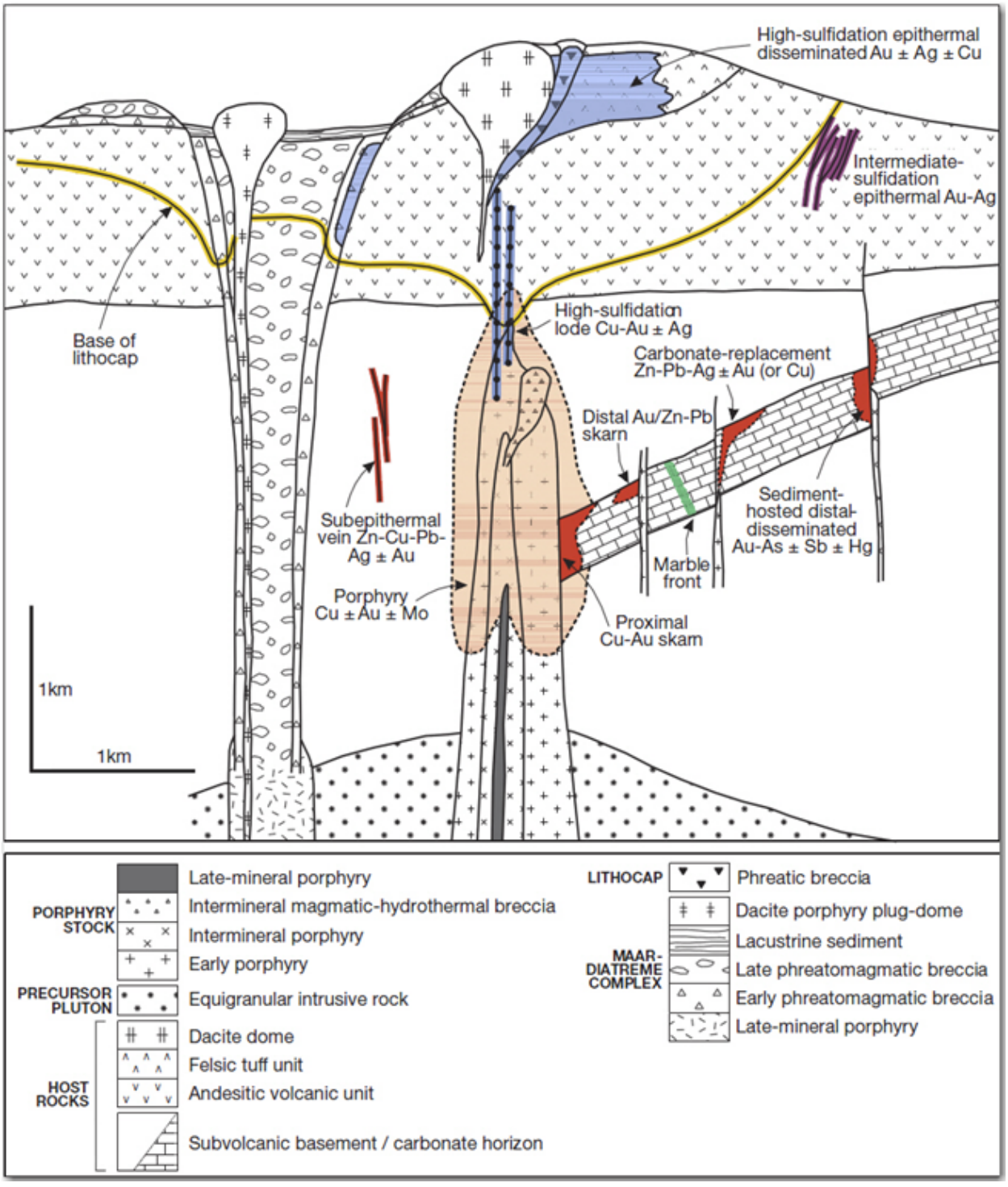
> Porphyry deposits are typically centred on polyphase stocks and porphyry dyke swarms, with skarn deposits formed adjacent to and epithermal deposits above the porphyry mineralization [Figure 8-1]. The metal endowment of a porphyry system is related to the geochemistry of the oxidized magmas that contribute 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 Tertiary intrusions, probably as a result of poor preservation of older rocks. (2010)

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 yards (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.

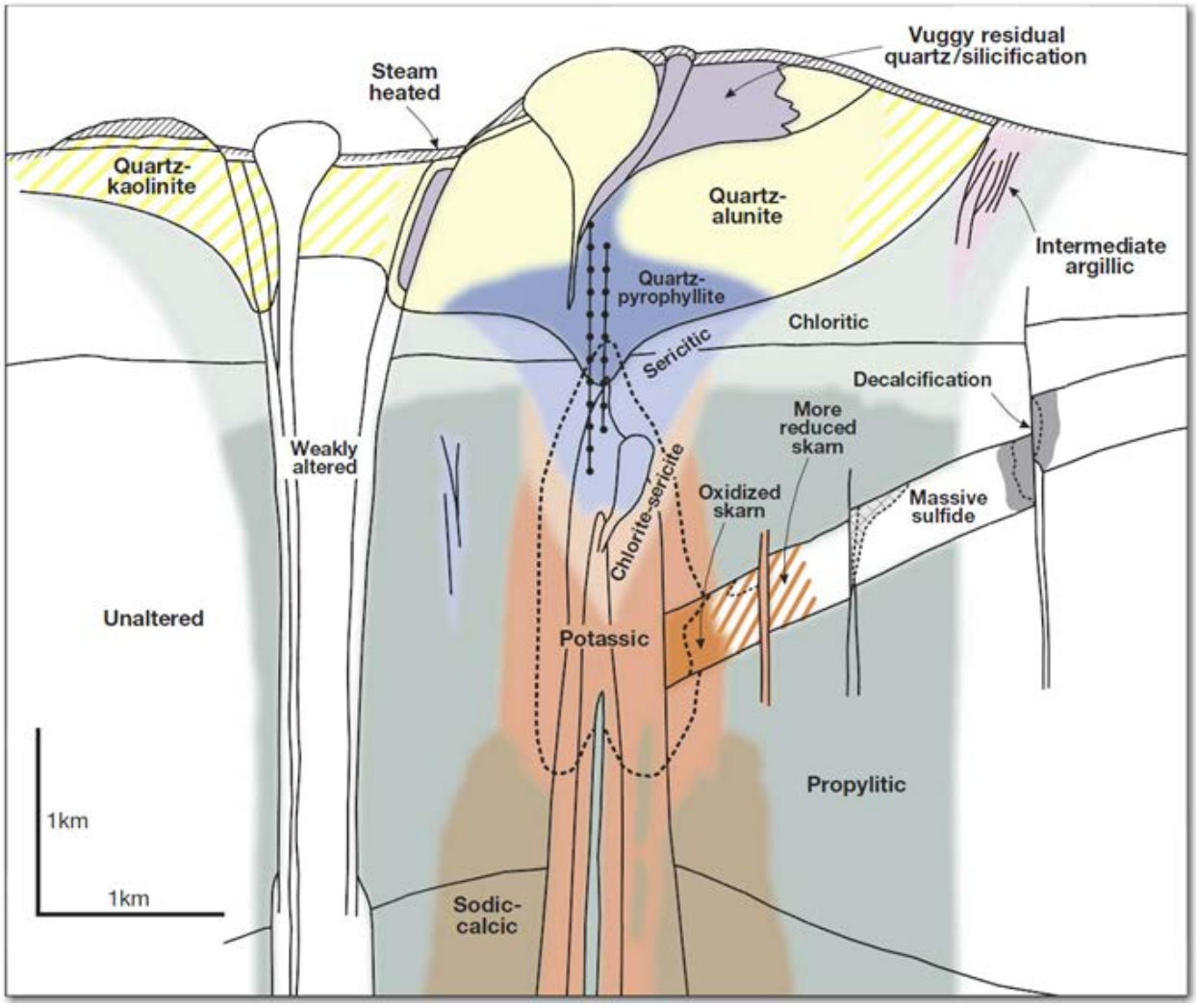
Quartz veins and veinlets as stockworks and sheeted arrays are present throughout these systems and typically occur in a sequence from early quartz-feldspar “A” veins, through quartz-sulfide (mainly chalcopyrite-molybdenite) “B” veins with potassic-altered margins, to late sulfide-dominant (primarily pyrite) “D” veins with phyllic-altered margins (Gustafson and Hunt 1975), as shown in Figure 8-3. 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**



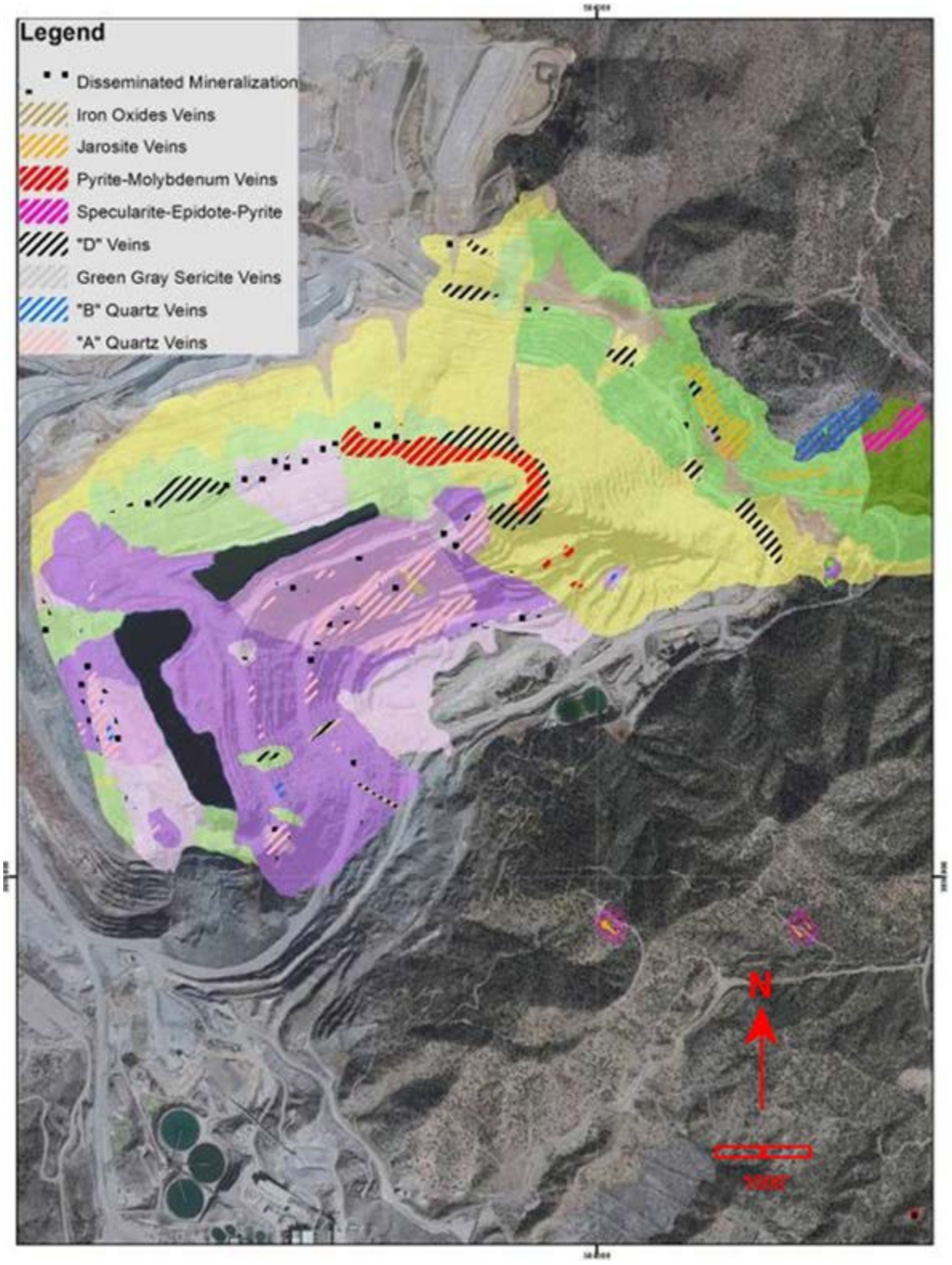

Sillitoe 2010

**Figure 8-2. Generalized Alteration-Mineralization Zoning Pattern for Telescoped Porphyry Copper Deposits**




Sillitoe 2010

**Figure 8-3. Pinto Valley Mine Alteration and Mineralization Plan Map**




BHP 2012

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%–300% or more, with a significant impact on the overall economics of these deposits.

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, a supergene chalcocite blanket, and adjacent areas of hornfelsing and skarn alteration.

# 9.0 EXPLORATION

Surface mapping has been the main source of additional data throughout the identification and selection phases of work for PV2. In preparation for PV2, two campaigns were conducted on separate occasions to improve both the geotechnical and geometallurgical knowledge of the deposit. All work stated in this section was performed by or on behalf of BHP.

The surface mapping for geotechnical information focused primarily on the bedding planes, major structures, and overall geological strength index. As a result, a more targeted geotechnical testing program was developed for the selection phase.

Various ore types were confirmed by using surface mapping and by reviewing core logs. Alteration zones and ore types were identified in the pit wall and correlated against core samples taken in previous drilling campaigns. The visual ore classification will be confirmed (and refined if necessary), using the laboratory petrographic facilities, LabSpec materials analysis equipment, and whole-rock chemical analysis.

Descriptions from the core logs were used to plot the correlation between rock type and alteration zone using ioGlobal software. Upon completion of this analysis, the primary ore types were classified to determine the necessary sampling program for the selection phase. Table 9-1 shows the proportion of ore types in the overall deposit for PV2. The most important ore types were narrowed down to Ruin granite, quartz monzonite, and diabase (Ore Types 1, 2, and 4, respectively, shown in Table 9-1). These ore types are based on relative abundance, gangue mineralogy, copper grade, alteration, and the potential impact on overall production (recovery, throughput, and consumption of reagents and energy).

**Table 9-1. Ore Type Summary for the Pinto Valley Deposit**

| Ore ID | Ore Type | %TCu Range | Average %TCu | Observed Proportion | Alteration |
|---|---|---|---|---|---|
| 1 | Ruin Granite / Quartz Monzonite | 0.2%–0.40% | 0.37% | 80%–85% | Potassic |
| 2 | Quartz Monzonite (with quartz veins) | 0.2%–0.40% | 0.36% | 6%–10% | Potassic |
| 3 | Granite Porphyry | 0.2%–0.25% | 0.22% | 5%–8% | Quartz Sericite |
| 4 | Diabase | 0.2%–1.0% | 0.45% | 2%–3% | Potassic |
| 5 | Aplite | 0.2%–0.25% | 0.20% | 1%–2% | Potassic |
| 6 | Quartzite / Granodiorite | 0.2%–0.25% | 0.22% | 0%–1% | No Alteration |

A number of new copper mineralization occurrences were identified during the brownfield surface mapping campaign in the Pinto Valley district. Three principal target zones are presented below.

## 9.1 Kozi Prospect

Mapping over the Ruin granite, southeast of the PVM pit, has shown a zone of small bodies with a brecciated texture (breccia porphyry) bearing evidence of hydrothermal alteration, relict sulfide boxwork, and pyrite grains. Outcrop is generally poor due to the steep angle of hillsides and narrow, but rounded, ridge tops. It is difficult to find moderately fresh rock in this area, as most surface material is extensively weathered and/or loose surface rubble.

Particular attention was paid to the bleached-looking alteration of feldspar; spectral analysis suggests the presence of dickite-kaolinite. Small, 1 mm diameter muscovite flakes appear to be a later generation of alteration; biotites are also altered and act as a nucleus around which sulfides have precipitated. Sulfide boxwork and primary pyrite grains were recognized. This zone of clay alteration appears to cover an area of approximately 215 $ft^2$ and is strictly related to the breccia porphyry.

Further evidence of alteration and porphyry emplacement was observed; this was confirmed to be a thin section of breccia with a rock-flour matrix, broken quartz fragments, and the lithics of granitic compositions, along with the presence of kaolinitic clays and dickite in the matrix.

The reflected-light thin section of this rock shows the presence of sulfides, mainly pyrite and minor chalcopyrite in the matrix of breccia, which has been partially replaced by goethite (these rocks are partially leached).

This reflected-light thin section clearly indicates that this rock is a breccia, suggesting a hydrothermal origin related with phreatic stages. This is because it has a majority of subrounded fragments and lower metal content along with the presence of rock-flour matrix and development of advanced argillic clays.

Mapping the boundary of Ruin granite 400 ft to the southeast of the breccia porphyry revealed a zone bearing copper oxides along the Schultze granite contact. Oxides coat the Schultze granite outcrop and subcrop, especially in an area surrounding a milky quartz vein, which is generally very vuggy and bears boxwork after sulfide dissolution. At one site, copper oxides dominantly comprise malachite and azurite, precipitating with siliceous cement over an area of approximately 250 $ft^2$ of subcrop. Chemical assay reports for samples taken from this area indicate >1% Cu (Table 9-2).

The copper oxide species indicates that it originates within the quartz veins. Veins contain abundant (up to 4%) coarse-grained chalcopyrite, with some grains up to 5 mm; some are moderately fresh, and goethite boxwork runs extensively through the center of the vuggy coarse vein. The Ruin granite-hosted disseminated chalcopyrite also contains coarse muscovite, calcite, and quartz. The rock displays some in situ oxidation of chalcopyrite, with copper oxide precipitation on the muscovites and micas. Contact-style mineralization between the two genetically different granites is most prevalent in the Ruin granite.

Microscopic description indicates coarse-grained texture granite with minor deformation of quartz, some subhedral orthoclase, and replacement of a later phase of coarse muscovite and calcite.

The chemical assay report for samples with chalcopyrite in the Ruin granite shows anomalous content of copper, molybdenum, silver, and zinc (Table 9-2).

**Table 9-2. Chemical Assay Results for Ruin and Schultze Granite**

| Sample | Cu ppm* | Mo ppm | Ag ppm | Zn ppm |
|---|---|---|---|---|
| 75163 Ruin Granite | 128 | 34.8 | <0.1 | 177 |
| 75164 Ruin Granite | 3,780 | 37.2 | 0.9 | 204 |
| 75165 Schultze Granite | >10,000 | 21.8 | <0.1 | 978 |
| 75181 Ruin Granite | 7,930 | 286.0 | 3.0 | 578 |

*parts per million

Float-bearing magnetite was encountered at Site 363, approximately 500 ft northeast of Site 361, in a creek line to the south. The source was not found; however, the float displayed a vuggy rock re-welded with hematite (martite).

Rock chip samples bearing magnetite returned anomalous copper values from multi-element assay reports. Surrounding this location, goethite and chalcopyrite grains up to 3 mm in diameter in Ruin granite were observed in a rock with moderate to strong coarse muscovite. This altered zone appears to be locally related to the intrusion of Schultze granite, but the anomalous mineralization of Cu-Mo-Ag-Zn suggests that this zone must be surrounding a hydrothermal system. Evidence of hydrothermal alteration was not observed in relation to the porphyritic system, but an intense hydrolytic event is related to the coarse muscovite in Ruin granite.

This prospect has two types of geological indicators of hydrothermal activity. The first indicator is the phreatic breccia, with occurrences of dickite-kaolinite and pyrite; this suggests that this zone has been exposed to advanced argillic alteration, as dickite-kaolinite exists in the upper crustal conditions following hydrothermal alteration associated with porphyry copper emplacement. The specimens tested may belong to the roots of an advanced argillic-altered zone, as evidenced by the presence of remnants that usually follow extensive weathering and erosion. The second indicator is related to the Cu-Mo anomalies in the Ruin granite; all the features suggest a possible connection to the pegmatite zones related to the intrusion of granitic magma—in this case, the Schultze granite.

## 9.2 Bondi Prospect

The Bondi Prospect is related to the Dripping Spring quartzite. Some zones with hornfels of biotite-magnetite outcrop near the tails facility as a 250 ft high cliff face and narrow gully incised by an active creek system. The quartzite is very fine- to fine-grained, well-bedded, and well-sorted quartz-dominated sediment with minor pebble conglomerate beds. On a number of cliff faces in this gully, copper oxides line exposed surfaces and natural cave formations. The oxides, including malachite and azurite, appear to have been introduced via seep of underground aquifer movement.

Sediments in some bands near the occurrence of copper oxides are studded with up to 4% fine-grained disseminated mineralization and narrow veins of pyrite grains, and possibly chalcopyrite. These beds are discrete, greater than 30 ft from oxide occurrences; sediments exhibit a much lower 1%–2% disseminated mineralization. The area warrants further review to identify the source of copper oxide precipitate.

A unit contact between the Mescal limestone and the Dripping Spring quartzite exhibited hematite and iron oxide seams in replacement silica-rich beds. Narrow seams of limestone have been replaced by a calc-silicate event, forming wollastonite crystals up to 3 mm in diameter, diopside, and disseminated pyrite and minor chalcopyrite.

Some bands have been metamorphosed and marbleized into light green, very hard, chert-rich beds, which are strongly outcropped and more resistant to weathering. In this area, limestone beds predominantly displayed disseminated magnetite over an area of 500 ft.

A series of massive magnetite veins is found in an outcrop measuring 130 ft long and 10 ft wide. As massive magnetite veins, these replacement beds are very hard and resistant to weathering, and contain abundant hematite oxide and some disseminated pyrite, with traces of chalcopyrite and bornite. These wide veins and narrower outlying magnetite veins are parallel-bedded, produced by dissolution and replacement of calcareous horizons.

This site displays a high development of prograde skarn alteration, indicated by the development of brown garnet, pyroxene, diopside, and wollastonite. Wollastonite grains are particularly well developed, up to 4 mm in diameter with well-formed crystal habits. Diopside and garnets are fewer and rarely occur in close proximity to massive metalliferous veins.

A retrograde mineral assemblage has also been observed comprising chlorite, magnetite, hematite, pyrolusite, and silica. Sulfide assemblage includes pyrite and traces of chalcopyrite and bornite in limestone beds. A diabase sill injected into the Mescal limestone has explored a bedding parallel zone, mapped in road cutting outcrops within the Mescal unit. The diabase unit has numerous iron oxide veins as a stockwork of near-vertical vein sets at regular ~1 ft intervals, which are between 0.25 inch and 0.5 inch in diameter. The vein wall rock interface alteration is also strongly iron-stained; very little quartz was observed in the iron oxide vein sets. The vein set postdates the diabase emplacement, as veins crosscut the body and penetrate a short distance into the surrounding limestone beds.

It is difficult to state the copper source at this prospect due to the oxidation observed in the quartzites, and it is important to note that this state is structurally complex and partially covered by post-mineral cover (Whitetail conglomerate). The skarn evidence may be related to the diabase intrusions, but the presence of strongly altered rock and a source of sulfur and metal can precede a close granitic source. It is important to note that a stockwork of leached veins was found in the diabase, which is possibly related to another fluid source.

## 9.3 Mati Prospect

Mapping in the limestone sequences 800 ft to the northeast of the pit revealed a skarnification zone, identified by prograde-stage development of pyroxene and wollastonite and retrograde-stage epidote. Both are associated with iron replacement stages in the limestone with injection of sulfide to the system, represented by pyrite-chalcopyrite and bornite. The supergene process was observed with the presence of copper oxide mineralization, mainly malachite and azurite.

This zone is related to old mine workings. The magnetite-sulfide bed had a 150 ft × 150 ft zone that varied from 13 ft to 40 ft orientated bedding parallel to the limestones.

This prospect is particularly interesting because the presence of copper sulfides related to the magnetite replacement suggests an origin associated with a Tertiary intrusion. This is because the host rocks are Paleozoic limestones that formed post-diabase intrusion; in this case, the only probable source must be related to a later intrusive.

## 9.4 Other Copper Oxide Exploration

Two small zones with copper oxide mineralization were identified. One is within the southwest boundary of the PVM property, near the Carlota Mine crossroad. Mineralization, primarily comprising malachite, was discovered at an old copper oxide mine and is related to a dike of porphyritic granodiorite. This rock intrudes the Schultze granite and is a restricted body with anomalous concentrations of greisens veins bearing copper sulfides. No disseminated mineralization was observed.

Another occurrence of copper oxides was found in an outcrop of Apache Leap tuff related to a small paleochannel in an active creek. The mineralization observed consists mainly of chrysocolla, black copper oxides, and minor malachite. This zone extends only 15 ft–18 ft, and no primary source was detected.

# 10.0 DRILLING

Drilling documentation was limited to internal reports, and there were no other listings for vintage data, methods used, or pre-2010 drilling procedures.

The pre-2006 PVM drilling programs comprised a combination of core, rotary, and churn drill holes. 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 3 RC rotary holes (RC62 through RC64) drilled in 1992. From January 1996 to April 1997, 67 RC exploration and infill holes were drilled: 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.

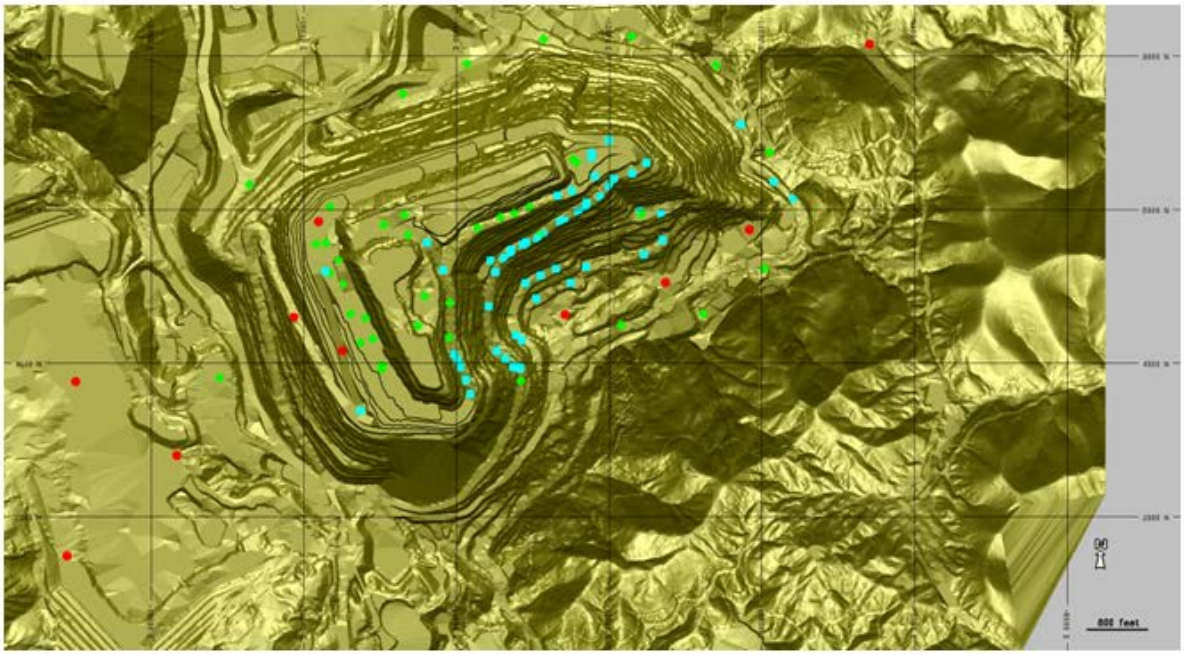
All drill hole collar locations were surveyed. The majority of the drill holes are vertical and therefore do not have downhole surveys. However, a majority of the inclined holes do have downhole surveys.

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.

The drilling campaign in 2010 focused on 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. Figure 10-1 illustrates a plan view with topography and the 2010, 2011, and 2012 drilling campaigns.

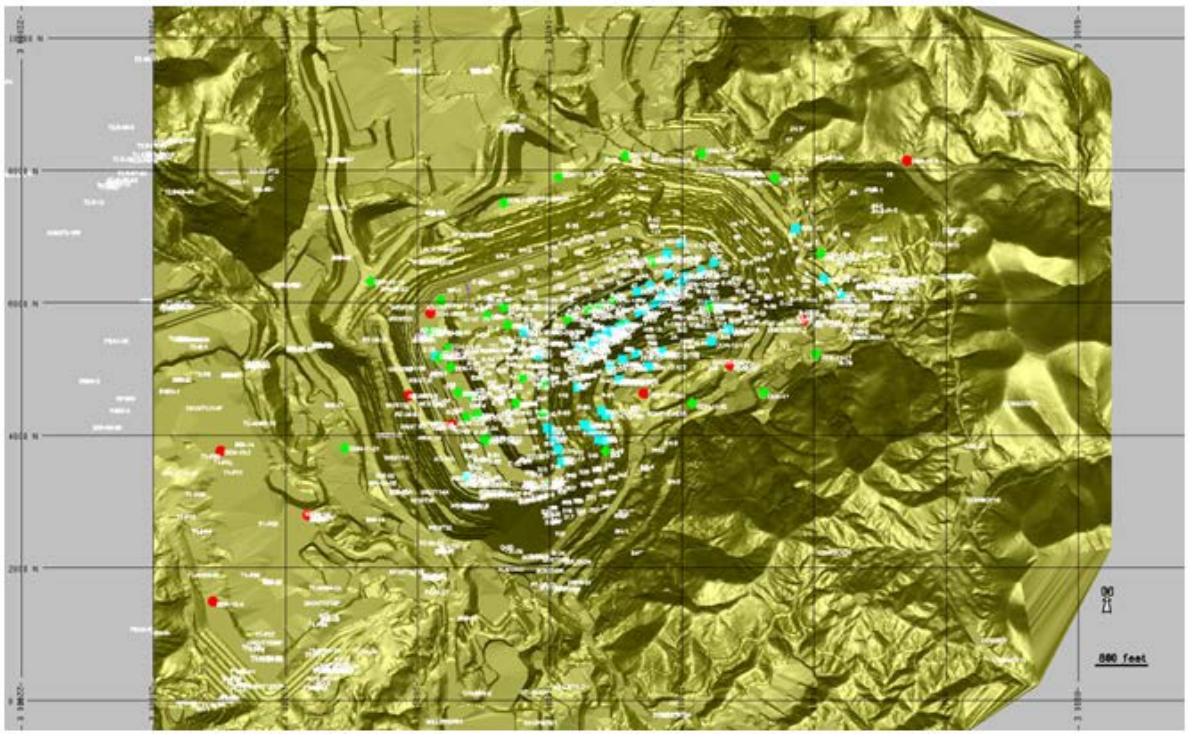
The combined database comprises 1,031 drill holes, as shown in Figure 10-2.

**Figure 10-1. Drill Hole Plan**



Note: 2010, 2011, and 2012 drilling is shown in red, green, and blue, respectively.

**Figure 10-2. Plan of All Drill Hole Collars**



Note: Assay results were pending for seven 2011 holes and thirty-four 2012 holes at the effective date of this report.

# 11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

This section describes the current protocols for handling the geological materials and preparing samples for analysis. Once drilling is completed, the core is transported to the core handling facility. Here, it is placed in wax-covered core boxes with depth markers for every drill run up to 10 ft. QuickLogs are completed 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 work stations.

The QuickLog data and the detailed logs are entered into an acQuire® relational database system that also records the collar, survey, assay, lithology, alteration, mineralization, and geotechnical (rock quality designation [RQD]) data. These data are tagged and tracked using the bar codes; all subsequent assay information provided by the laboratory, including the quality assurance / quality control (QA/QC) data, is linked to the database. The system is secured using protocols and procedures that appear to be extremely stringent. A dispatch report is created and sent to the laboratory, where it is 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 files and imported into acQuire® automatically. The data are imported into MineSight for the purpose of geological modeling and 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 currently in place at PVM are in line with industry standards for total copper, but procedures are Capstone-specific with respect to acid-soluble copper (i.e., digestion with 10% sulfuric acid placed in a hot bath at 100°F and read after 40 minutes). This procedure was used consistently at PVM and other mines in the Globe-Miami area operated by BHP and its predecessor companies.

Samples were assayed for total copper and acid-soluble copper. Composites representing 30 ft–50 ft of the sample rejects were made and assayed for total copper, oxide copper, molybdenum, and sulfur, as well as trace metals of gold and silver. Comparisons were made between the total copper and acid-soluble copper assays from the original assay intervals and the composite intervals.

Independent audits of the PVM assays were conducted in 1992 and 2000. Results were as follows.

- Assay values in the PVM database have been reliably entered.
- Total copper assays in the PVM database are reproducible and can be considered representative within normally accepted limits of error.
- Total copper assays in holes below the current pit base can 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 copper assays in the PVM database vary considerably depending on the drilling campaign.

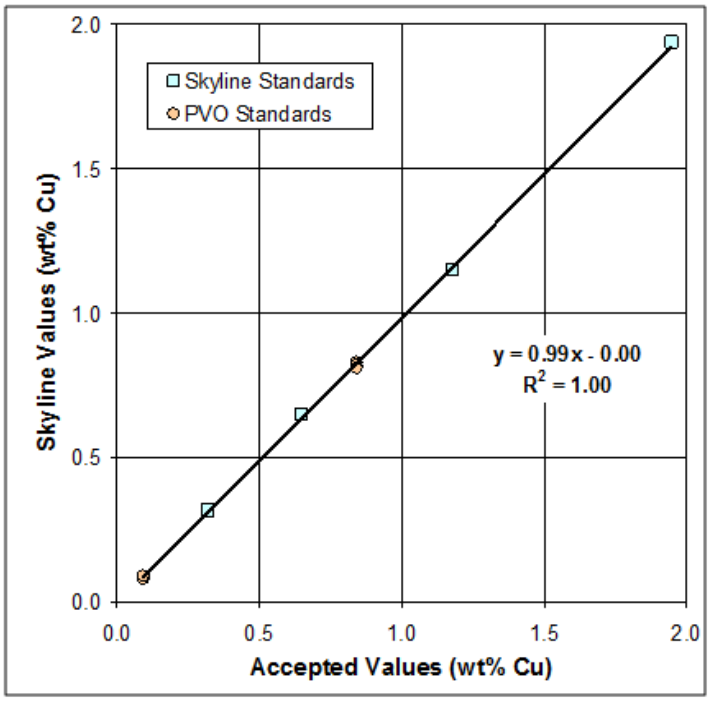
As part of the start-up feasibility study performed by BHP in 2006, a QA/QC program was conducted on 101 randomly selected drill hole assay interval pulp samples and 15 randomly selected core assay intervals. Samples were sent to Skyline Assayers and Laboratories (Skyline) in Tucson, Arizona, to be analyzed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

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

**Table 11-1. Analytical Results from Standard Reference Materials**

| | Standard | No. | Accepted Value | Average (Skyline) | Standard Deviation (Skyline) | Relative Difference | Relative Standard Deviation | Relative Precision |
|---|---|---|---|---|---|---|---|---|
| Internal Skyline | CGS-2 | 1 | 1.177 | 1.153 | N/A | –0.020 | N/A | N/A |
| | CGS-3 | 1 | 0.646 | 0.650 | N/A | 0.006 | N/A | N/A |
| | CGS-4 | 1 | 1.947 | 1.939 | N/A | –0.004 | N/A | N/A |
| | CGS-6 | 1 | 0.318 | 0.317 | N/A | –0.003 | N/A | N/A |
| PVM | High-Grade | 7 | 0.840 | 0.820 | 0.006 | –0.024 | 0.007 | 0.017 |
| | Low-Grade | 7 | 0.091 | 0.089 | 0.004 | –0.027 | 0.043 | 0.104 |
| **Total (all samples)** | | 18 | 0.589 | 0.579 | N/A | –0.018 | 0.033 | 0.070 |
| **Total (>0.1% TCu)** | | 11 | 0.906 | 0.891 | N/A | –0.017 | 0.021 | 0.048 |

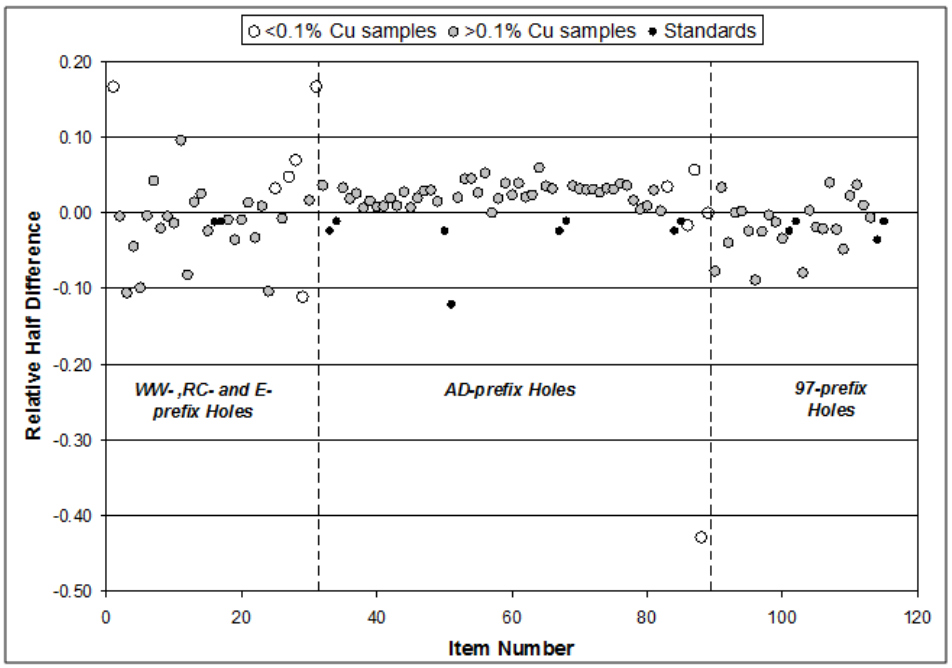
**Figure 11-1. Analytical Results from Standard Reference Materials**




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

The re-assay program for stored pulp samples shows that historical quality control measures used in the PVM analytical laboratory were variable: at times they were extremely good, but at other times they were less so, although still acceptable. The relative half differences (RHDs) of the samples are presented in sequential order in Figure 11-2, which shows that the drill hole 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 originates with the analytical precision at PVM and not at Skyline.

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




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

Table 11-2 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 11-2).

**Table 11-2. Analytical Results for Replicate Pulp Assays**
**2006 Pinto Valley Mine QA/QC Program**

| Drill Hole Program | Data Subset | No. | Copper Average (%) | | Linear Fit Slope | Average | | Relative Precision‡ |
|---|---|---|---|---|---|---|---|---|
| | | | Skyline | PVM | | RHD* | ARHD† | |
| WW-, RC-, and E-Series | All Data | 29 | 0.254 | 0.247 | 0.95 | 0.000 | 0.049 | 0.138 |
| | >0.1% TCu 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% TCu 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 | 0.080 |
| All Samples | All Data | 101 | 0.275 | 0.278 | 1.00 | 0.004 | 0.037 | 0.123 |
| | >0.1% TCu 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 RHD at the 95% confidence level, as estimated through Student's t-distribution.

Fifteen field duplicates of split core from drill holes lying in sequence between E21 and E60 are summarized in Table 11-3 and 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% TCu is slightly more than double the analytical precision of AD-Series pulp replicates (Table 11-3). 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.

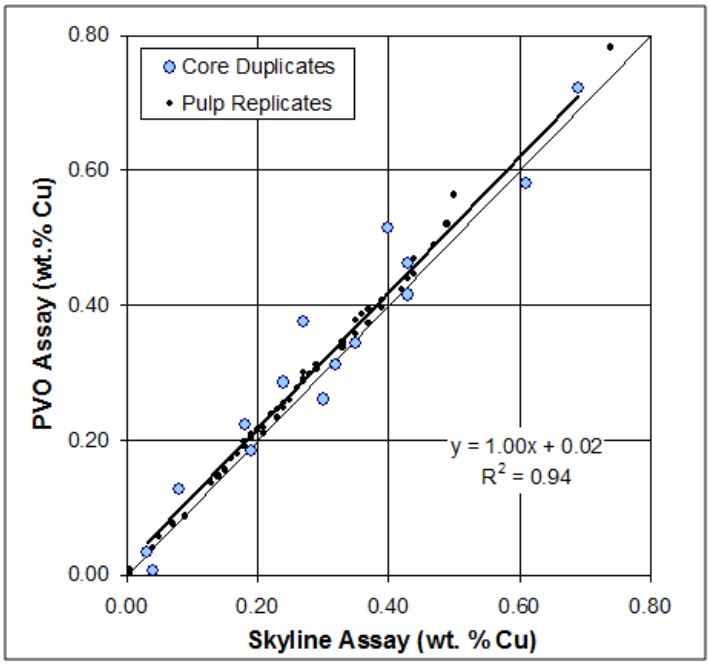
**Table 11-3. Analytical Results for Duplicate Core Preparation and Assays**

| Sample Set | No. | Skyline %TCu | PVM %TCu | RHD | ARHD* | Relative Precision |
|---|---|---|---|---|---|---|
| All Samples | 15 | 0.304 | 0.322 | –0.006 | 0.114 | 0.453 |
| Samples >0.1% TCu | 12 | 0.368 | 0.389 | 0.032 | 0.058 | 0.167 |

2006 PVM QA/QC program

*Absolute relative half difference

**Figure 11-3. Comparison of 15 Field Duplicate Samples**




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% TCu 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 11-4.

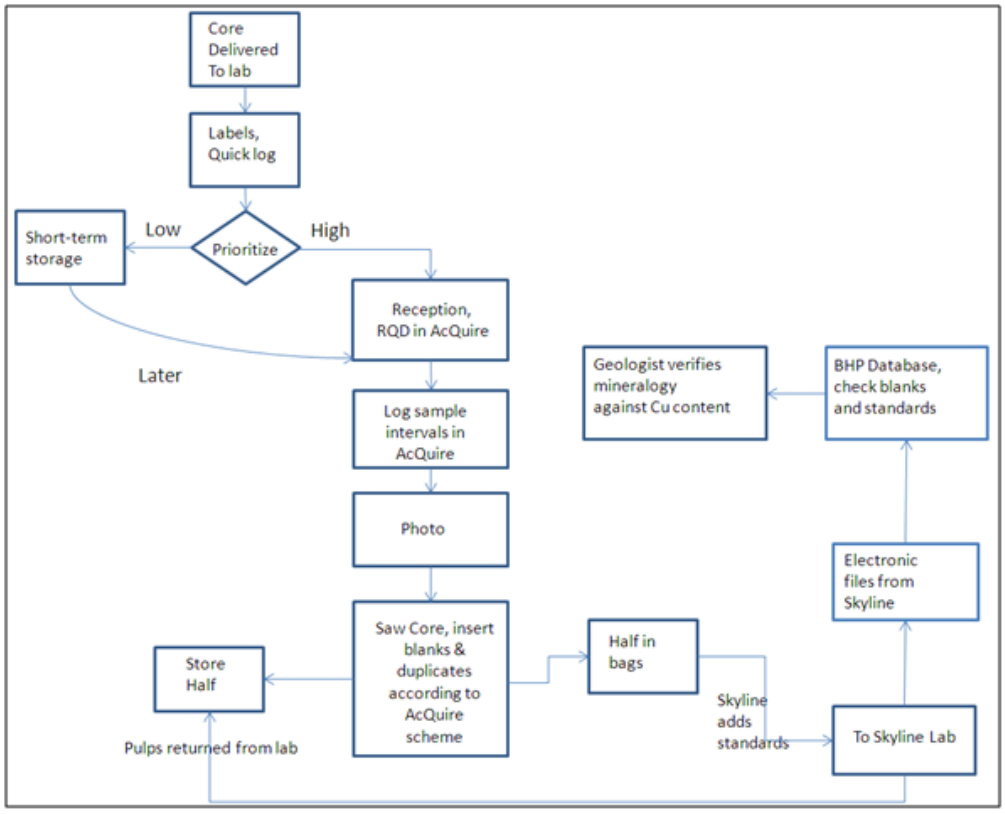
The sampling and preparation errors of the RC samples could not be fully determined due to a lack of field duplicates. Field sampling of RC cuttings are generally associated with lower variances than sampling of drill core, which can offset the higher laboratory variances measured for the 1996 and 1997 programs (Table 11-4). The analytical bias seen in these samples, corrected for the Skyline bias, is estimated to be 4% lower than the international standards.

**Table 11-4. Total and Stepwise Sampling Estimates and Analytical Variances**

| Drill Hole 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- / 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. The process, as shown in Figure 11-4, ensures that suitable checks are in place for each step of the sampling and data-gathering activities.

**Figure 11-4. Condensed Sample Handling and Chain-of-Custody Stream**




The following QA/QC criteria were used to validate the results and samples (ASCu represents acid-soluble copper).

1. **%TCu > %ASCu**
   a. Exceptions include the following.
      - %TCu < 0.1 and %ASCu < 0.1
      - %TCu/(%TCu – %ASCu) > 1.05
      - If not (%ASCu > %TCu), reject and report loss of precision to the laboratory and Capstone geologists, and send the following for reanalysis: 10 samples before and 10 samples after the rejected sample. Include results in the monthly QA/QC report.

2. **Blanks (Cu)**
   a. < 6 times %TCu, threshold limit = OK
   b. < 6 times %Mo, threshold limit = OK
   c. If not, reject and report loss of accuracy to the laboratory and Capstone geologists, and send the following for reanalysis: 10 samples before and 10 samples after the rejected sample. Include results in the monthly QA/QC report.
3. **Standards (Cu)**
   a. < 2σ = OK
   b. If > 2σ, reject and report loss of accuracy to the laboratory and Capstone geologists, and send the following for reanalysis: 10 samples before and 10 samples after the rejected sample. Include results in the monthly QA/QC report.
   c. If > 3σ, reject and report loss of accuracy to the laboratory and Capstone geologists, and send the following for reanalysis: 10 samples before and 10 samples after the rejected sample. Include results in the monthly QA/QC report.
4. **Field Duplicates, Crushing, and Pulp Duplicate (for Cu)**
   a. Protocols and procedures are in place to define sampling and laboratory errors within a large group of samples and batches; however, this is not used to reject a batch.

Assays are imported to the Capstone 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.

1. Enter the Capstone data portal, and select the area, project, and batch list.
2. Review QA/QC results, particularly "Company Standards," blanks, and "Lab Standards," for approval.
3. 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.

# 12.0 DATA VERIFICATION

Garth Kirkham, P.Geo., FGC, QP, visited the property on 14 May 2013. The site visit involved an inspection of the core logging facilities, offices, outcrops, historic 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 drill holes 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 QP 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, but the QP believes that the work is being conducted by a well-respected, large, multi-national company that employs competent professionals who adhere to industry best practices and standards.

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.

# 13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

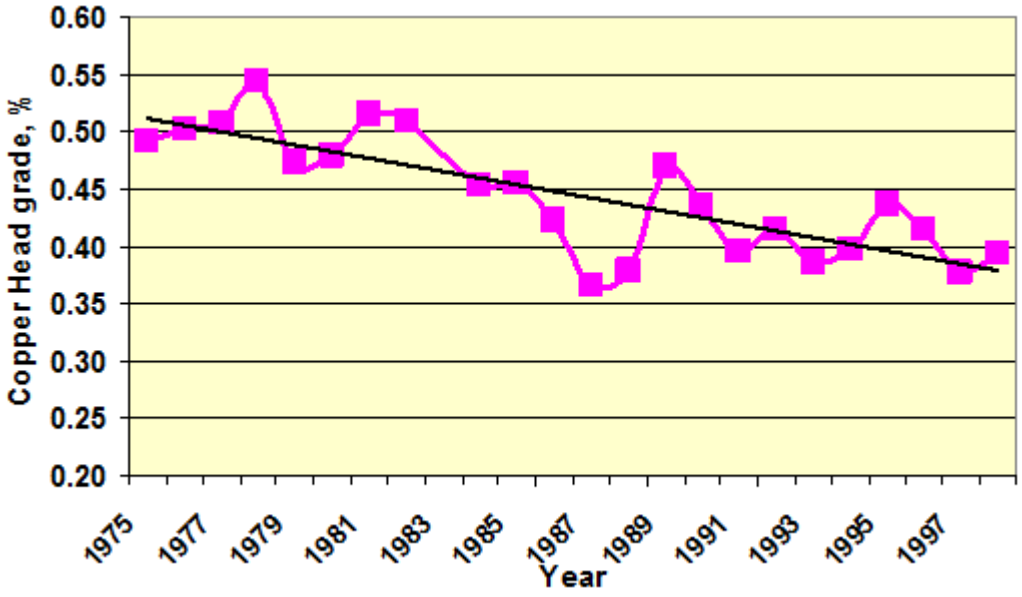
## 13.1 Introduction

PVM started operations in 1974. The mine and mill have been placed on care and maintenance due to economics a number of times, with the most recent restart of operations in December 2012. The mill has undergone a number of process optimizations during its operating life with equipment upgrades and/or replacements.

The plant feed grade of the mill between 1974 and 1998 was fairly consistent. The average copper head grade over the life of previous operating periods was 0.44% Cu. (Note: %Cu represents total %Cu, sulfide + soluble, in the mill feed). From Figure 13-1, it is evident that the head grade had been trending downward. The head grade is expected to average 0.33% Cu as operations continue.

When operations resumed in 1984, the plant was modified to process at a design tonnage of an average 63,000 tpd. The current processing rate in both the crushing plant and the mill has been lower than the design rate. This has been attributed to harder ore as the open-pit mine has become deeper. Equipment availability has also been a factor in the mill processing rate. Most recent projections have been estimated at a nominal 55,100 tpd.

**Figure 13-1. Pinto Valley Copper Head Grade 1974–1998**



As part of the 2012 restart and PV2 projects, a number of metallurgical testing programs have been completed and initiated. A portion of the testwork effort focused on attempting to develop predictive models for circuit performance.

This technical report for the PV2 project has primarily focused on the interpretations of the most recent operating results and comparing the mill results to the testwork where possible.

## 13.2 Metallurgical Testwork

### 13.2.1 Crushing

There has not been any testwork completed to evaluate the crushing plant equipment and configuration. Field measurements and operating data have been used to develop a crushing model (refer to Section 13.3.1) and the process design criteria (refer to Section 17.1).

Metso was contacted to develop a Bruno model of the crushing circuit to determine the circuit constraints. There was no specific process performance information provided to Metso for this preliminary modeling exercise. The preliminary results indicated the existing plant was limited to 1,925 tph at 0.39-inch (10 mm) product size. Table 13-1 represents the results of the Bruno simulations of the PVM crusher flow sheet using Metso handbook referenced data to represent the primary crusher product distribution and compares the predicted throughput for the existing 7 ft Symons cone crushers to newer crusher models available from Metso.

**Table 13-1. Metso Bruno Simulation Using Metso Standard Data (applying Metso handbook primary crusher discharge distribution)**

**BRUNO MODEL STANDARD CONDITIONS**

| | Secondary Capacity (tph) | | |
|---|---|---|---|
| P80 (mm) | 10 | 8.5 | 7 |
| P80 (inch) | 0.39 | 0.33 | 0.28 |
| | | | |
| MP800 | 3520 | 3300 | 2860 |
| HP800 | 2805 | 2475 | 2200 |
| HP500 | 2090 | 1815 | 1375 |
| Symons 7' | 1925 | 1760 | 1375 |
| | | | |
| | Daily (tpd) | | |
| 68% | Jul 1, 2013 to Jan 7, 2014 | | |
| MP800 | 57446 | 53856 | 46675 |
| HP800 | 45778 | 40392 | 35904 |
| HP500 | 34109 | 29621 | 22440 |
| Symons 7' | 31416 | 28723 | 22440 |
| | | | |
| | Daily (tpd) | | |
| 71% | Oct 1, 2013 to Jan 7, 2014 | | |
| MP800 | 59981 | 56232 | 48734 |
| HP800 | 47797 | 42174 | 37488 |
| HP500 | 35614 | 30928 | 23430 |
| Symons 7' | 32802 | 29990 | 23430 |
| | | | |
| | Daily (tpd) | | |
| 75% | Target Availability | | |
| MP800 | 63360 | 59400 | 51480 |
| HP800 | 50490 | 44550 | 39600 |
| HP500 | 37620 | 32670 | 24750 |
| Symons 7' | 34650 | 31680 | 24750 |

The PVM operations team provided SPLIT system data for the primary crusher discharge. The data indicated that there was a significant amount of fines in the primary crusher discharge. This information was provided to Metso. The revised Bruno model showed significant improvements to 2,365 tph at 0.39 inch (10 mm). Table 13-2 projects the increased throughputs for the various crusher models when considering the primary crusher discharge product distribution measured by the SPLIT system.

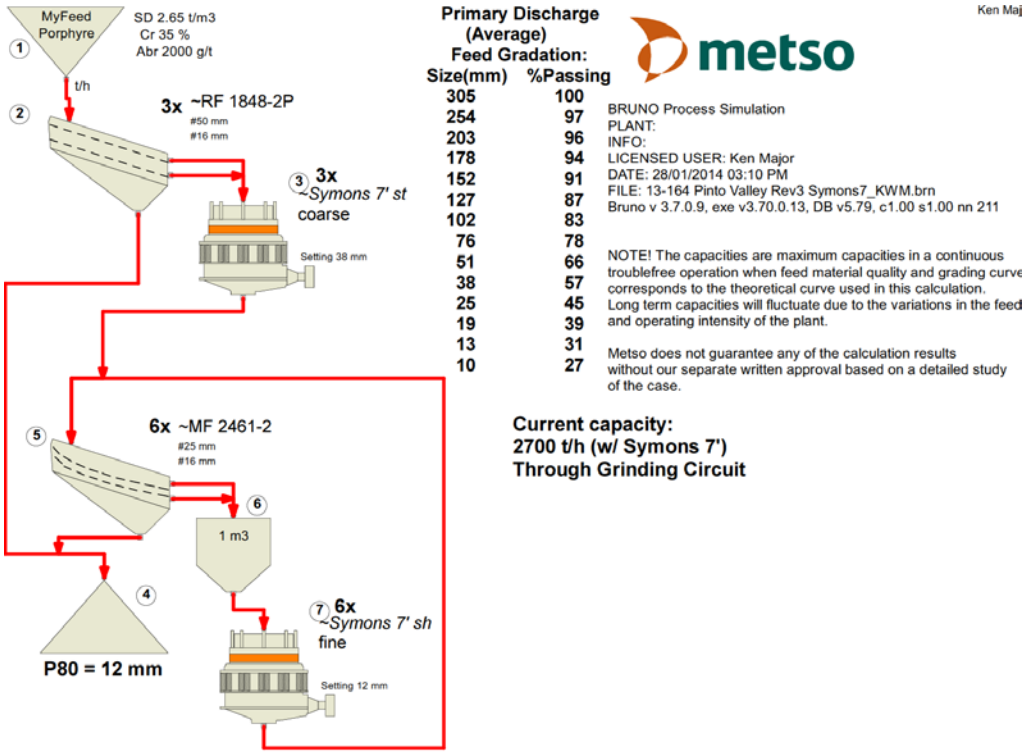
**Table 13-2. Metso Bruno Simulation Using Pinto Valley Mine System Data (applying Pinto Valley primary crusher discharge distribution)**

**BRUNO MODEL Pinto Valley Primary Crusher Discharge**

| | Secondary Capacity (tph) | | |
|---|---|---|---|
| P80 (mm) | 10 | 8.5 | 7 |
| P80 (inch) | 0.39 | 0.33 | 0.28 |
| | | | |
| MP800 | 4290 | 3300 | 2860 |
| HP800 | 3410 | 2475 | 2200 |
| HP500 | 2530 | 1815 | 1375 |
| Symons 7' | 2365 | 1760 | 1375 |
| | | | |
| | Daily (tpd) | | |
| 68% | Jul 1, 2013 to Jan 7, 2014 | | |
| MP800 | 70013 | 53856 | 46675 |
| HP800 | 55651 | 40392 | 35904 |
| HP500 | 41290 | 29621 | 22440 |
| Symons 7' | 38597 | 28723 | 22440 |
| | | | |
| | Daily (tpd) | | |
| 71% | Oct 1, 2013 to Jan 7, 2014 | | |
| MP800 | 73102 | 56232 | 48734 |
| HP800 | 58106 | 42174 | 37488 |
| HP500 | 43111 | 30928 | 23430 |
| Symons 7' | 40300 | 29990 | 23430 |
| | | | |
| | Daily (tpd) | | |
| 75% | Target Availability | | |
| MP800 | 77220 | 59400 | 51480 |
| HP800 | 61380 | 44550 | 39600 |
| HP500 | 45540 | 32670 | 24750 |
| Symons 7' | 42570 | 31680 | 24750 |

At 0.47-inch (12 mm) product size, the crushing plant instantaneous throughput was estimated using Bruno at 2,970 tph. These Bruno results are provided in Figure 13-2. The influence of the fine product distribution from the primary crusher (45% –1 inch) distribution on the fine-crushing plant (secondary and tertiary crushers) capacity was estimated to be 23%. The primary crusher does not generate an abundance of fines, and the observed distribution is likely a product of rock characteristics and design of the blasting procedures.

**Figure 13-2. Crusher Throughput at Crushing Plant ($P_{80}$ = 12 mm)**




Based on the various availabilities at the 0.47 inch (12 mm) product size, the Bruno model indicates that existing crusher facility is limited to the following.

- 68% availability = 48,470 tpd
- 71% availability = 50,610 tpd
- 75% availability = 53,460 tpd

It should be noted that the Bruno model has used screens with different profiles than the screens installed in order to attain throughputs similar to plant levels. The screens are a primary bottleneck in increasing plant throughput.

The July 2013–January 2014 crusher operating data collected by PVM was evaluated to determine site measurement of the crusher plant capacity. Nominal throughput for each secondary crusher circuit was estimated at about 1,000 tons per operating hour (tpoh) (2,930 tpoh combined). This compares well to the Bruno model (2,970 tph) that has been constructed for the PVM circuit.

### 13.2.2 Grinding

BWi tests and/or MBWi tests have been completed on drill core samples for the PV2 project. All BWi and MBWi data and analysis have been provided in metric units as defined by the units reported from testwork.

Figure 13-3 provides an analysis of the BWi data. Table 13-3 is an indication of the lithological ore distributions observed in the drill core for PV2.

**Figure 13-3. Bond Ball Mill Work Index**

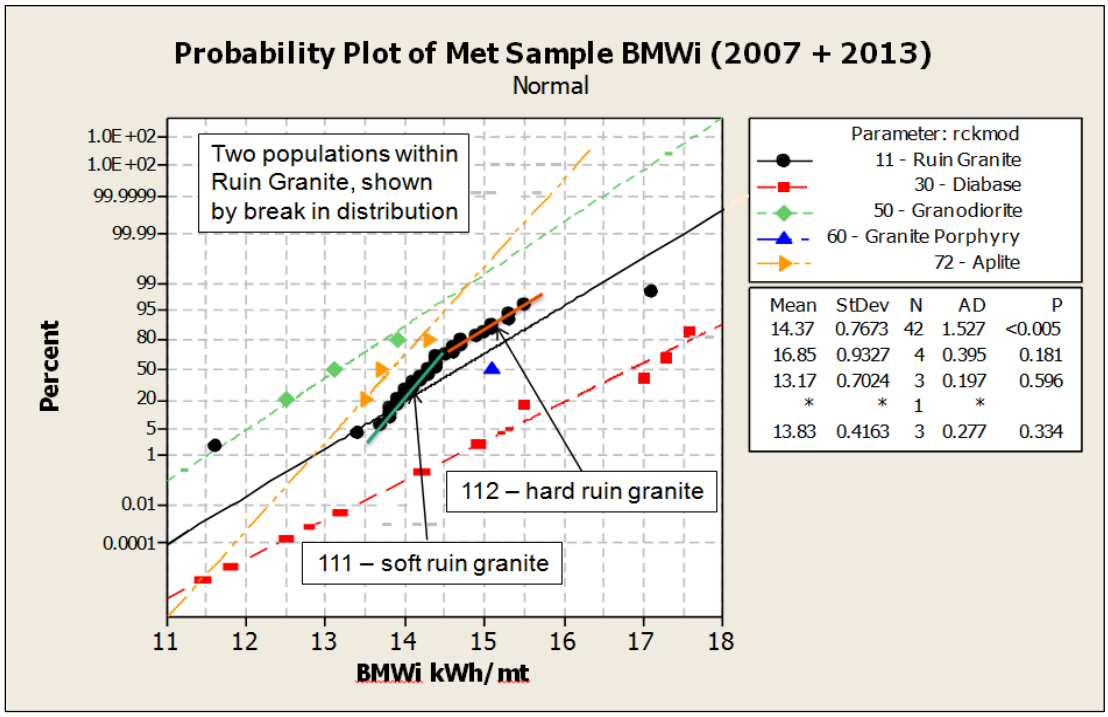



**Table 13-3. Ore Lithological Distribution for PV2 (imperial units)**

| Lithology Type | Mill Feed (ktons) | Mill Feed (%) | Average BWi (kWh/t) |
|---|---|---|---|
| Aplite Dikes | 11,279 | 4.4 | 13.8 |
| Diabase Dikes | 1,891 | 0.7 | 17.1 |
| Granite Porphyry | 3,008 | 1.2 | 15.0 |
| Granodiorite | 3,552 | 1.4 | 13.1 |
| Ruin Granite | 236,249 | 92.3 | 14.5 |
| **Total** | **255,979** | **100.0** | **14.5** |

The predominant ore type has been identified as Ruin granite, and the data indicate that the work index is distributed over a narrow range from 13.5 kWh/t to 15.5 kWh/t. Further analysis completed by PVM indicated based on the location within the pit that there may be an opportunity to split the Ruin granite into "soft" and "hard" classifications.

Diabase was found to have the highest MBWi, ranging from 17.0 kWh/t to 17.5 kWh/t. Diabase makes up 2.3% of the ore lithologies.

To determine the accuracy of the MBWi data, a comparison was generated versus BWi tests for identical samples. The maximum error between the BWi and MBWi results for the 2013 samples was 4.1%. The comparisons in 2007 and 2013 were both quite similar, with slopes close to unity indicating that there is no bias between the two tests.

Based on simplified grinding calculations and using the Ruin granite "soft" ore for the baseline, there could be a 12% decrease in mill throughput for the Ruin granite "hard" ore and a 38% decrease in mill throughput for the diabase ore. There is some limited control to try to maintain throughput, but this would require coarser grinds that could have a negative impact on recoveries.

Crushed ore is distributed to the FOB for distribution to the six single-stage ball mills. The ball mills are 18 ft diameter × 21 ft long with 4,000 horsepower (hp) motors. A review of the ball mills and the ball mill operating practices has shown that the ball mills have the capability to operate using the full mill power draw. Ball charging practices provide the ability to operate near or above the installed power.

Shift operating data for the crushing plant and grinding circuit have shown that the ball mill feed (crushing plant product), $F_{80}$, since September 2013 has been averaging approximately 0.43 inches. These readings are taken by the SPLIT system that uses digital analysis of the conveyor load to determine the size distributions.

A review of the cyclone overflow distributions for each of the grinding circuits indicates that the grinding product, $P_{80}$, has averaged approximately 280 µm.

A work index model based on the BWi equations was used to develop a throughput model that estimated the plant throughput based on ore work index for varying $F_{80}$ and $P_{80}$ conditions (Table 13-4).

**Table 13-4. Bond Ball Mill Work Index Modeling (metric units)**

| | $F_{80}$ = 6.5 mm $P_{80}$ = 315 µm t/d | | $F_{80}$ = 8 mm $P_{80}$ = 315 µm t/d | | $F_{80}$ = 9.5 mm $P_{80}$ = 315 µm t/d | | $F_{80}$ = 8 mm $P_{80}$ = 270 µm t/d | | $F_{80}$ = 11 mm $P_{80}$ = 280 µm* t/d | |
|---|---|---|---|---|---|---|---|---|---|---|
| Wi† | 95% | 24 hr | 95% | 24 hr | 95% | 24 hr | 95% | 24 hr | 95% | 24 hr |
| 12 | 69,466 | 73,122 | 64,998 | 68,419 | 60,821 | 64,022 | 59,134 | 62,246 | 56,042 | 58,991 |
| 13 | 64,359 | 67,746 | 60,092 | 63,254 | 56,147 | 59,102 | 54,812 | 57,696 | 51,773 | 54,498 |
| 14 | 59,252 | 62,371 | 55,186 | 58,090 | 51,473 | 54,182 | 50,489 | 53,146 | 47,505 | 50,005 |
| 14.3 | 57,720 | 60,758 | 53,714 | 56,541 | 50,071 | 52,706 | 49,192 | 51,781 | 46,224 | 48,657 |
| 15 | 54,145 | 56,995 | 50,280 | 52,926 | 46,799 | 49,262 | 46,167 | 48,596 | 43,236 | 45,512 |
| 15.1 | 53,634 | 56,457 | 49,789 | 52,409 | 46,332 | 48,770 | 45,734 | 48,141 | 42,809 | 45,062 |
| 16 | 49,038 | 51,619 | 45,373 | 47,761 | 42,125 | 44,342 | 41,844 | 44,046 | 38,967 | 41,018 |
| 17 | 43,931 | 46,243 | 40,467 | 42,597 | 37,451 | 39,422 | 37,522 | 39,496 | 34,699 | 36,525 |
| 17.3 | 42,399 | 44,630 | 38,995 | 41,048 | 36,049 | 37,946 | 36,225 | 38,131 | 33,418 | 35,177 |
| 18 | 38,824 | 40,867 | 35,561 | 37,433 | 32,777 | 34,502 | 33,199 | 34,946 | 30,430 | 32,032 |

*$F_{80}$ = 11 mm, $P_{80}$ = 280 µm: current PVM operating conditions.
†Wi = Work Index. 14.3 = soft Ruin granite ore, 15.1 = hard Ruin granite ore, and 17.3 = diabase ore.

The PVM operations team also provided an analysis of the monthly ore mined tonnage and related that to the drill hole work index information for the zones mined. For August, September, and October 2013, the average work indexes from the mine block model were 14.8 kWh/t, 14.6 kWh/t, and 14.7 kWh/t, respectively. Based on these work indexes, the model would have limited mill throughput to 49,200 tpd (44,700 t/d) on a 95% operating availability basis.

Various operating periods have been analyzed to evaluate the plant throughput. There has been a steady improvement of plant operating time through 2013, and the grinding circuit availability has been improving. Based on the instantaneous readings from the weightometers on the mill feed conveyors, it was estimated that the primary ball mill feed rate averaged approximately 400 tph. Assuming the mill availability will be 95%, the estimated throughput is approximately 55,100 tpd (50,000 t/d). From this data, the indications are that the mill is operating at 10% higher throughput than the work index model indicates.

A sample of mill feed was collected from a ball mill feed conveyor in January 2014 to confirm previous observations. The sample was sent to ALS in Kamloops for testing. The sample was screened to determine the feed size distribution and then was subjected to BWi tests at screen closing sizes of 106 µm and 300 µm. Based on the mine plan and geological evaluation of the mill feed, it was believed that the feed sample could have had as much as 8% diabase mixed with Ruin granite.

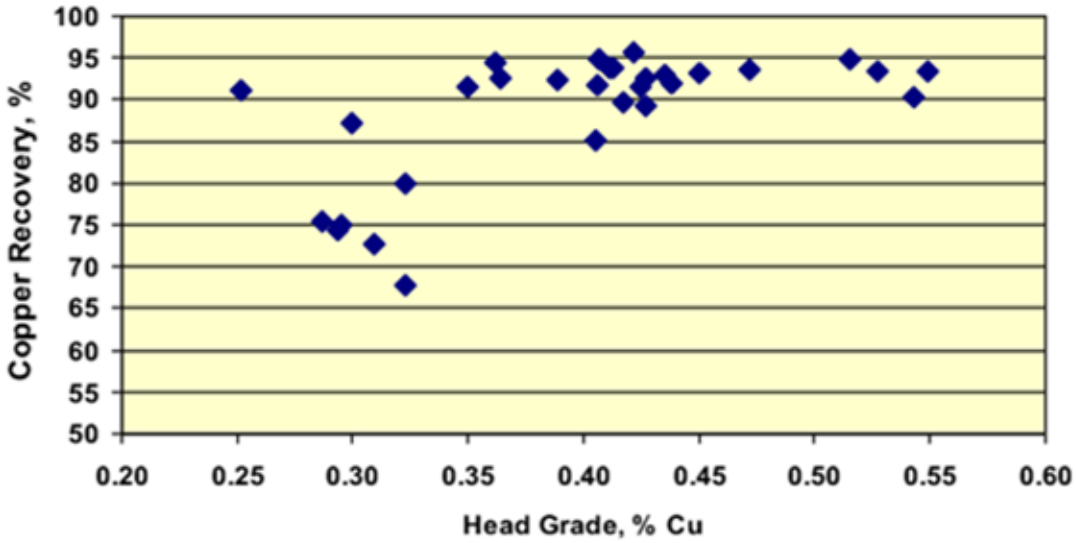
The test results indicated the following.

- Feed size distribution, $F_{80}$ = 12,434 µm
- BWi, 106 µm = 15.0 kWh/t
- BWi, 300 µm = 14.7 kWh/t

### 13.2.3 Flotation

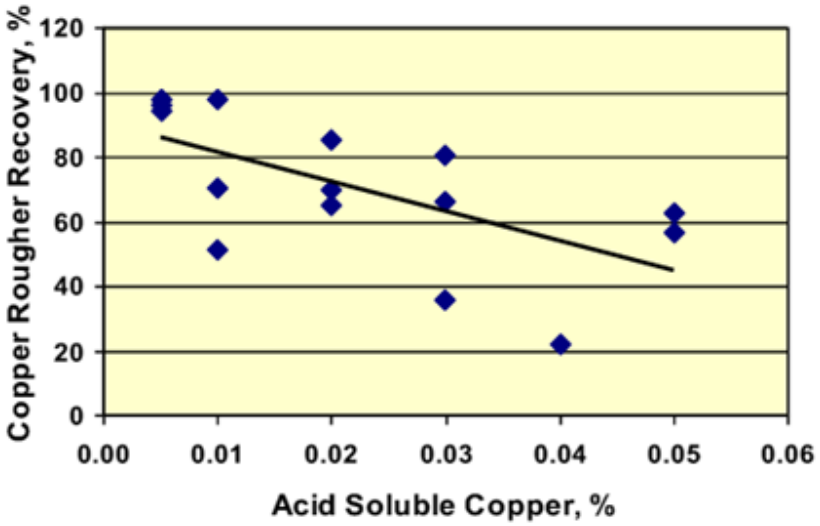
In October 2006, PVM completed a review of past metallurgical testwork. The report indicated that a compilation was made of all testwork that was conducted in the plant laboratory between 1992 and 1995. Figure 13-4 summarizes the grade-recovery relationship for rougher flotation.

**Figure 13-4. Copper Feed Grade vs. Recovery**



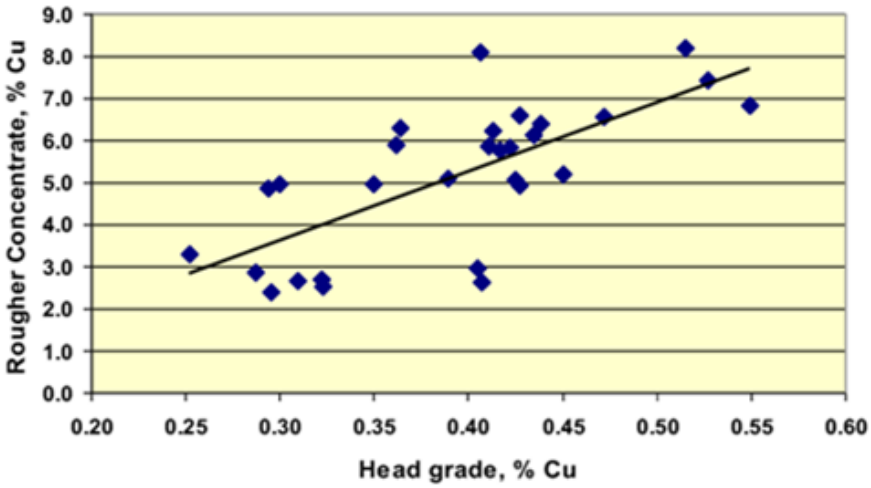
Low recoveries were attributed to the presence of soluble copper. Figure 13-5 provides an indication of the influence of acid-soluble copper on the rougher flotation recovery.

**Figure 13-5. Copper Recovery vs. Soluble Copper Content**



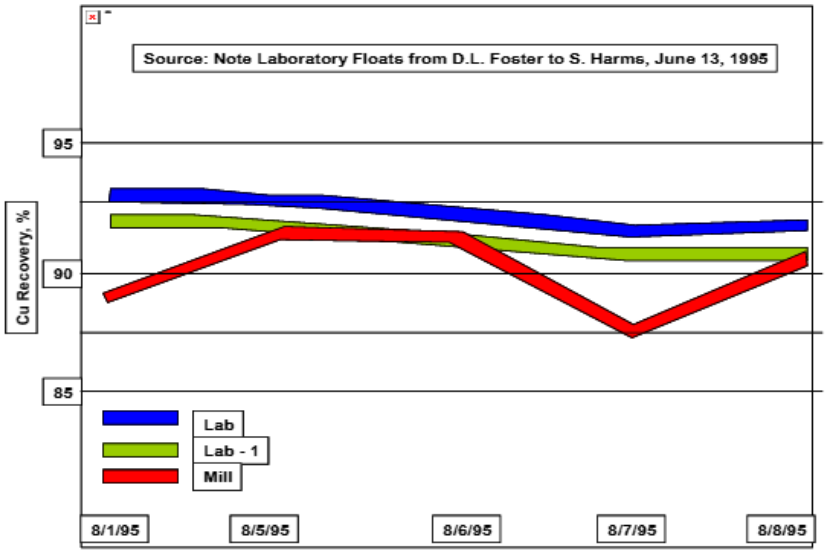

The review of the previous test data also focused on the influence of the copper feed grade to the copper rougher concentrate grade. The rougher concentrate feed grade is directly tied to the rougher flotation mass pull and the cleaner flotation cell requirements. Figure 13-6 illustrates the rougher concentrate grade versus mill feed grade relationship.

**Figure 13-6. Rougher Concentrate Grade vs. Mill Feed Grade (Cu)**



A comparison of plant recovery versus lab recovery is illustrated in Figure 13-7. This confirmed a rougher recovery of approximately 92%. It further indicates that on the average, the plant rougher recovery could be up to 2% lower than that obtained in the laboratory.

**Figure 13-7. Comparison of Plant to Laboratory Results**




For the PV2 project, PVM completed additional testwork with SGS. The bulk of this testwork focused on development of metallurgical simulations to represent the plant operation. The rougher kinetic flotation tests on the three ore type composites indicated that good recoveries can be achieved for the Ruin granite (potassic alteration) and granite porphyry samples, with approximately 90% recovery. For the Ruin granite (sericite alteration) sample, the recoveries were approximately 80%. Additional rougher flotation kinetics tests were conducted to assess the influence of reagent scheme, grind size, and %solids. No significant effect on final recovery was observed by changing the addition point of lime or collector. The current reagent suite tested with recovery >93% Cu. A limited interaction study of grind size and %solids indicates that there is significant interaction between these variables at laboratory scale. The highest rougher recovery is achieved at low %solids and finer grind size; however, at higher %solids, rougher recovery becomes insensitive to grind size. It was recommended to do additional testing and plant trials to validate these findings.

Open-circuit flotation tests with regrinding of rougher concentrate and three cleaning stages were performed on the metallurgy composite samples to evaluate the cleaning stage. Results showed that the copper grade was in the range of 23.0%–27.3% Cu, and recovery was between 79.3% and 84.8% Cu.

In January 2013, 11 composite samples for a froth flotation study and comminution testing were delivered to the SGS facility in Tucson, Arizona. The metallurgical study that was developed on 3 composite samples indicated that the copper and molybdenum were amenable to recovery by flotation. The metallurgical data developed are summarized in the following paragraphs.

The 11 composite samples were reconstituted at the lab per instructions from BHP and were subjected to sample preparation, sample characterization, and froth rougher flotation testing.

- Approximately 10 kg of sample from each composite sample were split for BWi and submitted for testing at SGS Lakefield.
- Approximately 20 kg of sample from Composite 1 and Composite 6 were split and submitted for SMC Test at SGS Vancouver.
- Approximately 10 kg of sample from each composite sample were stage crushed to 100% minus 10 mesh, homogenized using a Jones Riffle Splitter. Test charges of 1.6 kg for head assays, grind calibration, and rougher flotation testing were split using a rotary splitter.
- A test charge from the sericite sample was selected and submitted for mineral liberation analysis.
- The potassic and chalcocite composites were submitted for sequential copper analysis.

The rougher flotation testing was conducted for 25 minutes at a grind size of approximately 80%, passing 270 µm on 11 composite samples, using a scheme of reagents provided by BHP.

- Total copper rougher recovery ranged from 95.06% to 91.98%. The highest copper recovery of 95.06% was obtained on Composite 5 after a retention time of 25 minutes. From the results, a minimum test residence time of 15 minutes should be used for the mill flow sheet. A scale-up factor of 2 should be applied to the test residence time to determine the mill equipment requirements.
- Molybdenum recovery ranged from 81.76% to 69.69%. The highest molybdenum recovery of 81.76% was obtained on Composite 5 after a retention time of 25 minutes.

The metallurgical data developed on the rougher flotation kinetic tests conducted on the potassic, phyllic, sericite, chalcocite, and aplite composites have been summarized in Table 13-5.

**Table 13-5. Rougher Flotation Kinetics on the Potassic, Phyllic, Sericite, Chalcocite, and Aplite Composites Summary of Results**

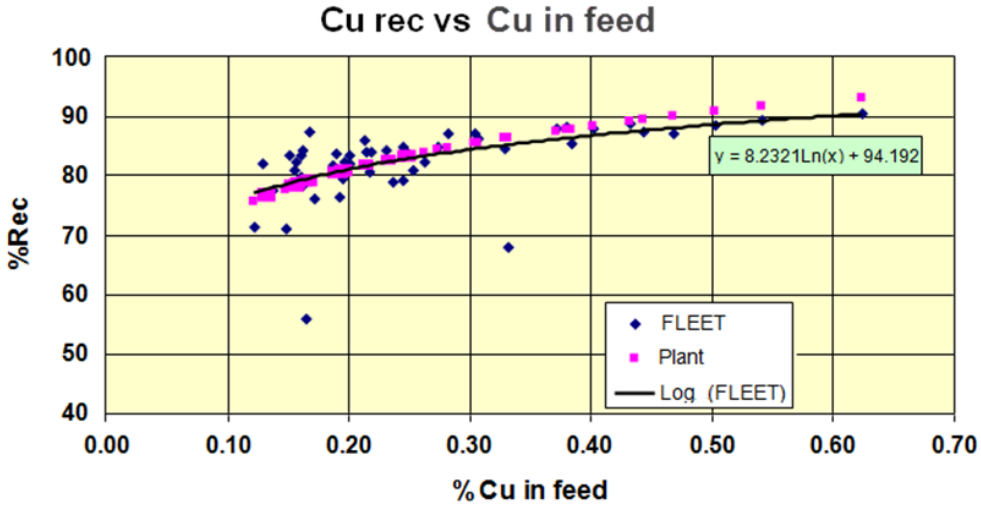
| Sample | Cumulative Time (Minutes) | Mass Recovery (%) | Cumulative Grade (%) | | | | | Cumulative Recovery (%) | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Cu | Mo (ppm*) | Fe | $S_T$ | Insol. | Cu | Mo | Fe | $S_T$ | Insol. |
| **Potassic** | **15** | **8.74** | **4.37** | **254** | **6.87** | **6.39** | **73.08** | **94.08** | **68.38** | **48.70** | **92.54** | **7.00** |
| | 25 | 13.54 | 2.85 | 170 | 4.89 | 4.18 | 78.60 | 95.06 | 70.74 | 53.71 | 93.80 | 11.66 |
| Calculated Head | | | 0.41 | 33 | 1.23 | 0.60 | 91.28 | | | | | |
| **Phyllic** | **15** | **7.53** | **4.60** | **3,007** | **11.82** | **12.83** | **61.45** | **91.86** | **85.68** | **54.41** | **81.36** | **5.18** |
| | 25 | 9.67 | 3.62 | 2,386 | 9.56 | 10.15 | 66.77 | 92.86 | 87.34 | 56.54 | 82.67 | 7.23 |
| Calculated Head | | | 0.38 | 264 | 1.64 | 1.19 | 89.34 | | | | | |
| **Sericite** | **15** | **6.74** | **2.37** | **485** | **3.73** | **3.12** | **80.54** | **89.80** | **73.65** | **29.92** | **83.37** | **5.92** |
| | 25 | 9.55 | 1.69 | 351 | 3.01 | 2.24 | 82.48 | 90.79 | 75.55 | 34.23 | 84.62 | 8.60 |
| Calculated Head | | | 0.18 | 44 | 0.84 | 0.25 | 91.63 | | | | | |
| **Chalcocite** | **15** | **7.94** | **4.83** | **373** | **13.07** | **13.92** | **59.53** | **90.41** | **78.63** | **58.09** | **87.43** | **5.36** |
| | 25 | 10.41 | 3.72 | 293 | 10.41 | 10.73 | 66.22 | 91.25 | 80.98 | 60.61 | 88.29 | 7.81 |
| Calculated Head | | | 0.42 | 38 | 1.79 | 1.26 | 88.20 | | | | | |
| **Aplite** | **15** | **5.84** | **3.54** | **646** | **14.00** | **15.24** | **57.95** | **89.39** | **77.59** | **54.88** | **84.42** | **3.81** |
| | 25 | 7.63 | 2.74 | 516 | 11.11 | 11.85 | 64.67 | 90.50 | 81.02 | 56.88 | 85.75 | 5.56 |
| Calculated Head | | | 0.23 | 49 | 1.49 | 1.05 | 88.85 | | | | | |

*parts per million

In summary, there has been a significant amount of work on the analysis of flotation recoveries for the Pinto Valley mill. The review completed in 2012/2013 by PVM for the PV2 project presentations generated a wide range of observations and conclusions. The primary findings have been included in the various charts and tables that follow.

Figure 13-8 illustrates the predictions for total copper recovery that were developed using flotation modeling techniques developed by SGS (FLEET). The FLEET models are typically developed by laboratory flotation tests. The results shown in Figure 13-8 indicate a good match to plant data.

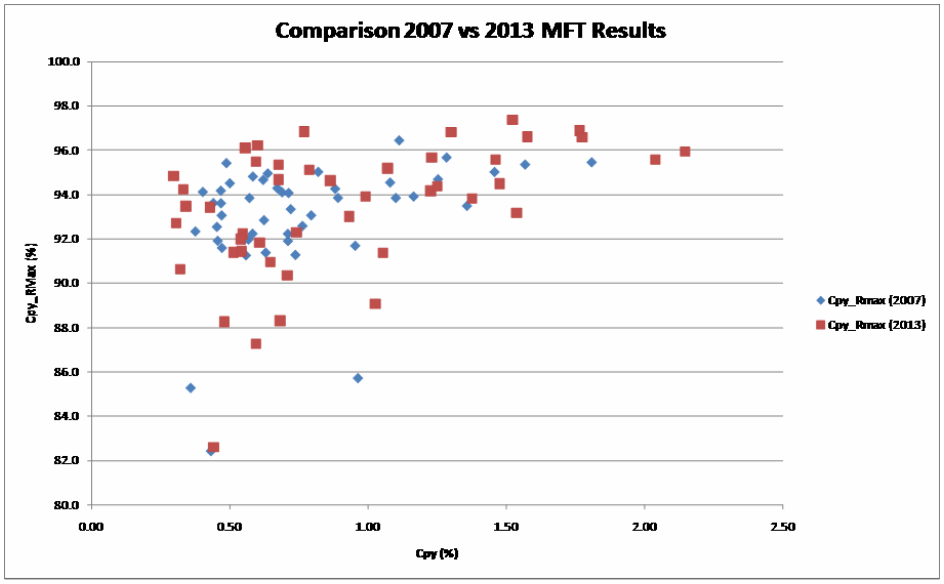
**Figure 13-8. 2007 FLEET Model**




In comparison to earlier recovery models based on a fixed final tail copper assay, the FLEET model that was generated appeared to provide a more accurate representation.

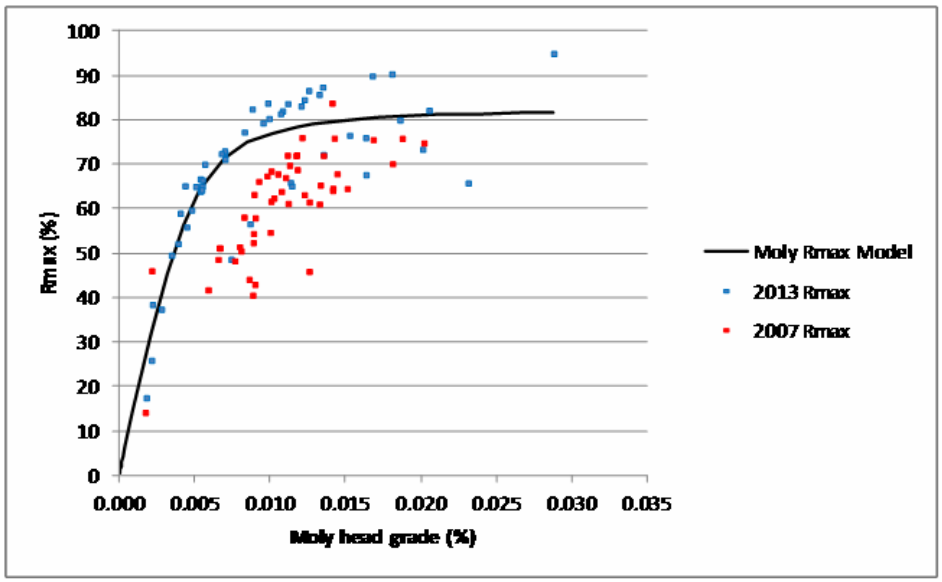
In Figure 13-9, efforts were made to compare 2013 data to the 2007 model and data. The results indicate that the rougher recoveries for copper sulfide mineralization were comparable for the two sets of data.

**Figure 13-9. Rougher Flotation Recovery of Copper Sulfides**




A comparison of the molybdenum recovery to the copper rougher concentrate has been shown in Figure 13-10. The results indicated that there was an increase in molybdenum recovery for the 2013 tests compared to 2007 steps. There was no indication in the review of documentation to explain any reasons for the differences.

**Figure 13-10. Molybdenum Recovery to the Copper Rougher Concentrate**



In Figure 13-11, the PVM report presented the predicted recovery developed from the models for the various stages of flotation. The overall molybdenum recovery (47.9%) was fairly typical of byproduct molybdenum operations. The recovery of $MoS_2$ from the copper (bulk) cleaner concentrate to a molybdenum concentrate will result in another step decrease in molybdenum recovery.

**Figure 13-11. Staged Recovery Predictions**

**Overall**

| Name | Con Grade (%) | | Recovery (%) | |
|---|---|---|---|---|
| | Mean | σ | Mean | σ |
| Cu | 27.14 | 4.62 | 85.18 | 6.44 |
| Fe | 31.63 | 2.63 | 12.53 | 4.39 |
| S | 36.41 | 2.97 | 32.97 | 23.04 |
| Mo | 0.39 | 0.38 | 47.88 | 23.71 |

**Rougher**

| Name | Con Grade (%) | | Recovery (%) | |
|---|---|---|---|---|
| | Mean | σ | Mean | σ |
| Cu | 5.84 | 2.95 | 91.26 | 4.43 |
| Fe | 18.27 | 4.94 | 36.01 | 13.67 |
| S | 20.09 | 5.80 | 90.52 | 5.92 |
| Mo | 0.13 | 0.13 | 77.79 | 35.57 |

**Cleaner**

| Name | Con Grade (%) | | Recovery (%) | |
|---|---|---|---|---|
| | Mean | σ | Mean | σ |
| Cu | 27.14 | 4.62 | 93.34 | 6.67 |
| Fe | 31.63 | 2.63 | 34.78 | 22.93 |
| S | 36.41 | 2.97 | 36.42 | 26.03 |
| Mo | 0.39 | 0.38 | 61.54 | 27.98 |

Note: Con Grade = Concentrate Grade.

The model that was developed predicted a copper cleaner recovery of 93.34%, which reduced the overall copper recovery to 85%. The historical cleaner recovery at PVM has indicated recoveries as high as 97.3%. Typical porphyry copper cleaners operate above the 97.3% level, and the model provided in Figure 13-11 could be predicting lower recoveries than are achieved in the plant.

Based on the variances observed in the interpretations of previous model results, the 2013 FLEET program completed by SGS has looked at separating the flotation model into rougher and cleaner components. The rougher flotation FLEET simulation shown in Figure 13-12 appeared to provide a reasonable analysis of the rougher flotation data.

**Figure 13-12. FLEET Simulated Rougher Flotation Recovery**

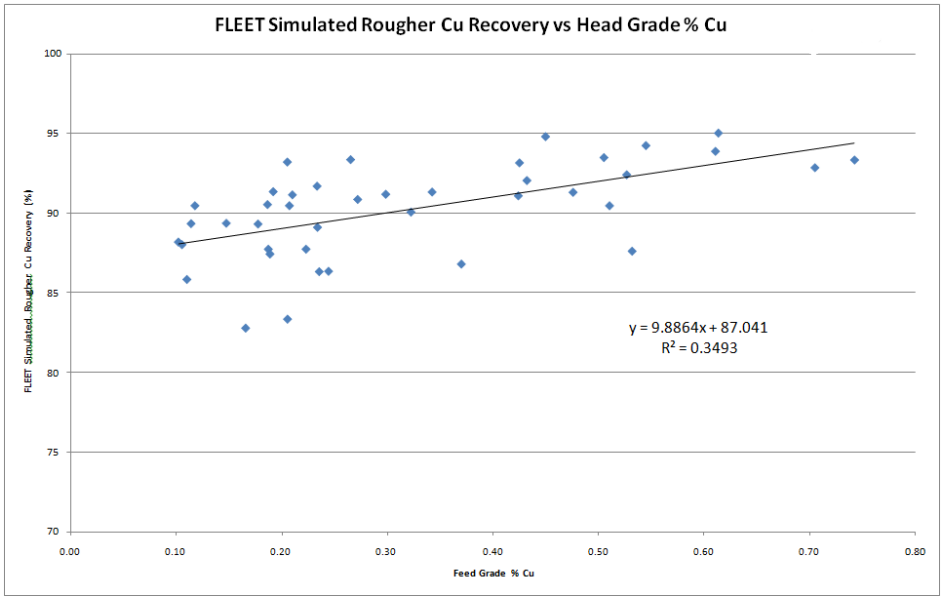
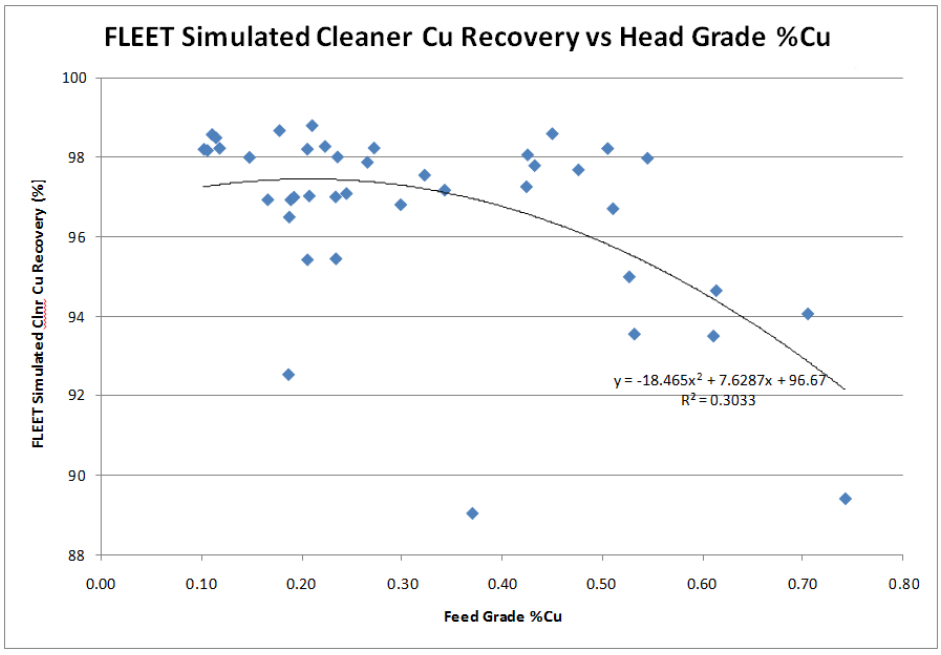



Figure 13-13 represents the FLEET interpretations for the copper recovery in the cleaner flotation circuit based on mill feed grade (%Cu).

**Figure 13-13. Cleaner Copper Recovery vs. Head Grade (%Cu)**



In Figure 13-14, the PVM copper-only model is compared to the FLEET model. The PVM model appears to cap the recoveries at higher copper feed grades.

**Figure 13-14. PVM Cu Flotation Model – FLEET Recovery vs. Fitted Models**

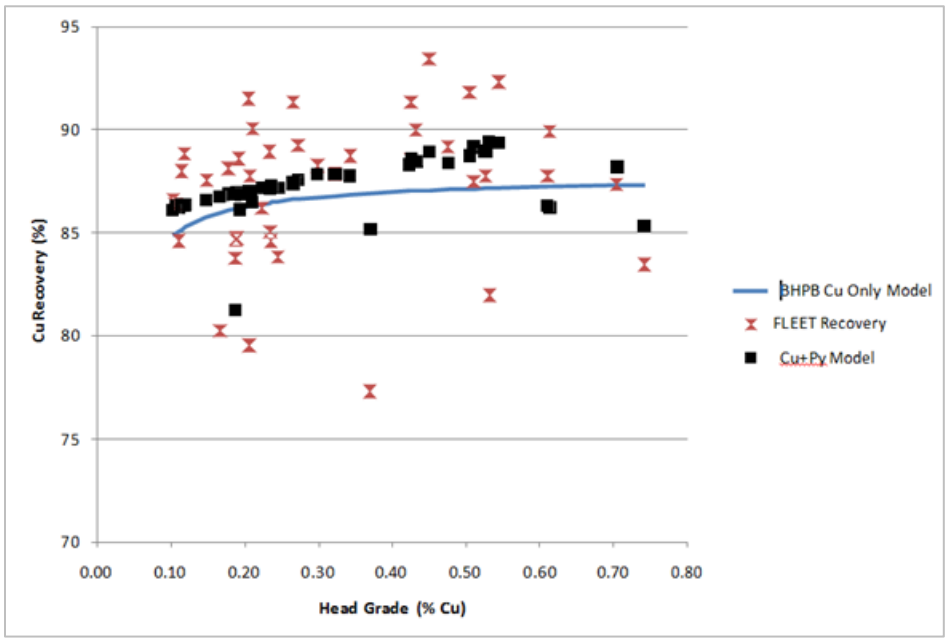



Table 13-6 looks at the combined recovery FLEET model developed in 2007 and compares it to the 2013 rougher recovery model and the overall recovery interpretations using a fixed cleaner flotation recovery (97.3%) and the FLEET cleaner recovery model.

**Table 13-6. Copper Flotation Recovery Model**

| 2007 FLEET* | | 2013 (Rougher) | | 2013 (Cleaner) | |
|---|---|---|---|---|---|
| %TCu | %Recovery | %TCu | %Recovery | CR† = 97.3 | CR (FLEET) |
| 0.10 | 75.2 | 0.10 | 88.03 | 85.65 | 85.61 |
| 0.20 | 80.9 | 0.20 | 89.02 | 86.61 | 86.75 |
| 0.30 | 84.3 | 0.30 | 90.01 | 87.58 | 87.57 |
| 0.40 | 86.6 | 0.40 | 91.00 | 88.54 | 88.05 |
| 0.50 | 88.5 | 0.50 | 91.98 | 89.50 | 88.18 |
| 0.60 | 90.0 | 0.60 | 92.97 | 90.46 | 87.95 |
| 0.70 | 91.3 | 0.70 | 93.96 | 91.42 | 87.35 |
| 0.80 | 92.4 | 0.80 | 94.95 | 92.39 | 86.36 |
| 0.90 | 93.3 | 0.90 | 95.94 | 93.35 | 84.98 |

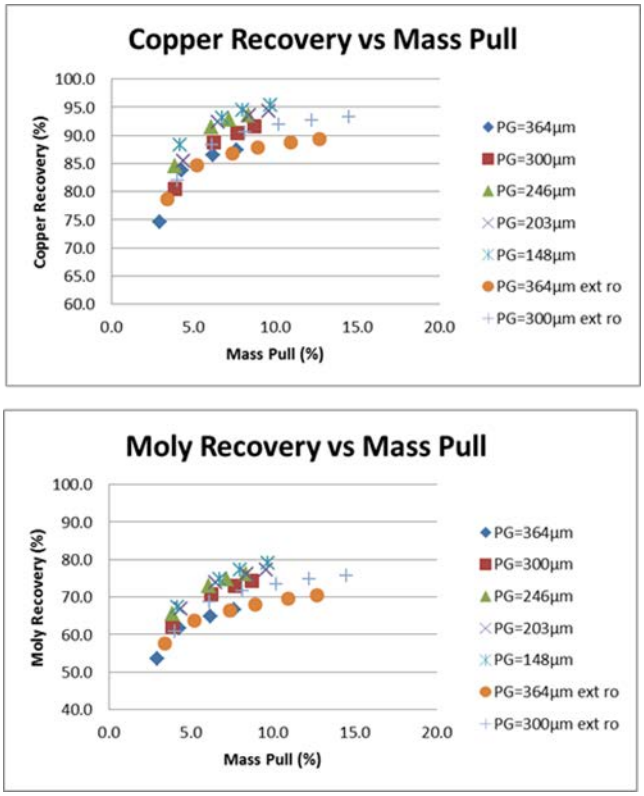
*As predicted by the FLEET models: 2007 (Total): %Recovery = 8.2381ln(%TCu) + 94.19; 2013 (Rougher): %Recovery = 9.8864(%TCu) + 87.041; 2013 (Cleaner): %Recovery = –18.465(%TCu)$^2$ + 7.6287(%TCu) + 96.67

†Cleaner recovery

The analysis of the FLEET cleaner flotation model indicates that the copper recovery peaks at a mill feed grade of about 0.50% Cu. Based on this inversion and declining cleaner flotation recovery with increasing grade, the application of the fixed cleaner recovery to the model appears to provide a more realistic expectation for the cleaner flotation circuit.

The sample that was collected from the ball mill feed in January 2014 and forwarded to ALS in Kamloops was prepared for use in a preliminary metallurgical test program. The focus of the program was to evaluate the rougher flotation recovery based on flotation feed size and flotation time. Figure 13-15 presents the results from the testwork for the copper and the molybdenum recovery to the rougher flotation concentrate.

**Figure 13-15. ALS Testwork Recovery vs. Mass Pull**



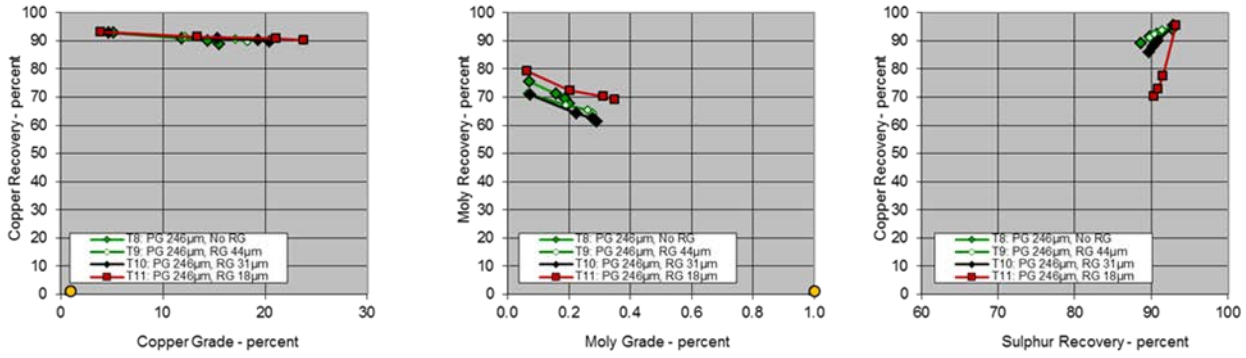
The testwork was completed at different grinds. The initial tests were based on eight minutes rougher flotation time. The results of the initial tests indicated that the copper and molybdenum recoveries and the mass pull to the rougher concentrate decreased as the flotation feed became coarser. The decrease appeared to become more significant as the flotation feed $K_{80}$ approached 300 µm. Based on the results of the preliminary tests, the $K_{80}$ 300 µm and $K_{80}$ 364 µm flotation tests were repeated with the laboratory flotation times extended to 20 minutes. The extended flotation times showed improved recoveries, but time did not achieve the same recoveries.

The grind / flotation time / recovery relationship has an impact on the throughput optimization of the PVM process facilities.

Rougher flotation testwork has been the primary focus of most of the metallurgical test programs. The focus on understanding the rougher flotation process is that the flotation feed grind size and ore work index have the most influence on plant throughput and metal recovery.

Very little recent testwork has been reported for cleaner flotation. Many of the reports and programs that touch on the cleaners target a regrind mill product size of 50 µm. The regrind kinetic testwork that has been initiated at ALS (shown in Figure 13-16) provides an indication of the influence of regrind size on metallurgical performance. The testwork results supported the plant bulk rougher concentrate target regrind $P_{80}$ of 50 µm for the recovery of copper. The testwork results also indicated that the molybdenum recovery in the bulk cleaner circuit could be increased by regrinding the bulk rougher concentrate to a finer product size, but additional work would be required to determine whether this influenced the molybdenum recovery to the molybdenum concentrate and/or resulted in a positive economic benefit.

**Figure 13-16. Metallurgical Influence of Regrind Product Size**



## 13.3 Metallurgical Models

### 13.3.1 Crushing

Based on the various availabilities at the 0.47-inch (12 mm) product size, the Bruno model indicates that the existing crusher facility is limited to the following.

- 68% availability = 48,470 tpd
- 71% availability = 50,610 tpd
- 75% availability = 53,460 tpd

It should be noted that the Bruno model has used screens with different profiles than the screens installed in order to attain throughputs similar to plant levels. The screens are a bottleneck in increasing plant throughput.

The July 2013–January 2014 crusher operating data collected by the mine was evaluated to determine site measurement of the crusher plant capacity. Nominal throughput for each secondary crusher circuit was estimated at approximately 1,000 tpoh (2,930 tpoh combined).

### 13.3.2 Grinding

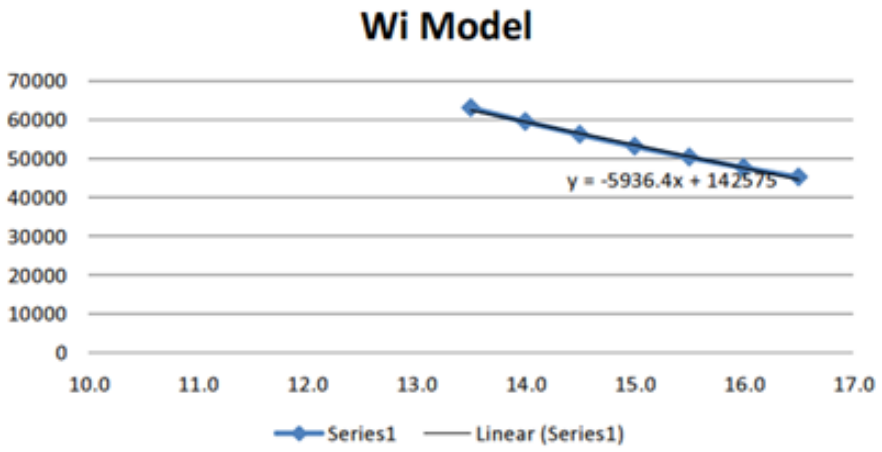
The PVM grinding circuit utilizes single-stage ball mill grinding to prepare the product from the fine-crushing facility for flotation. To model the grinding circuit, the focus has been to develop the model using the BWi calculations and incorporating the correction factors. The variables inputted to the Bond equation include the BWi (metric); ball mill feed size, $F_{80}$ (fine-crushing plant product); and the grinding circuit product size, $P_{80}$. When compared to the plant data for an ore work index of 14.6, there was an indication that the grinding circuit throughput was approximately 10% higher than projected by the work index model. The plant data was analyzed from three months of recent operating activity. Table 13-7 projects the plant throughput in tons per hour for a range of work indexes from 13.5 to 16.5. The 10% model adjustment for measured plant data has been included.

**Table 13-7. Production Model – Plant Throughput vs. Work Index**

| Plant Adjustment (tpd) | **62,980.7** | **59,445.1** | **56,161** | **53,106.5** | **50,261.8** | **47,609.6** | **45,134.2** |
|---|---|---|---|---|---|---|---|
| Model Throughput (tpd) | 57,255.2 | 54,041 | 51,055.5 | 48,278.6 | 45,692.6 | 43,281.5 | 41,031.1 |
| Model Throughput (t/d) | 52,050.2 | 49,128.2 | 46,414.1 | 43,889.6 | 41,538.7 | 39,346.8 | 37,301 |
| Grinding Availability | 95.0% | 95.0% | 95.0% | 95.0% | 95.0% | 95.0% | 95.0% |
| Plant Feed Rate (tons/hr) | 2,282.9 | 2,154.75 | 2,035.71 | 1,924.98 | 1,821.87 | 1,725.74 | 1,636.01 |
| Feed Size, $F_{80}$ (µm) | 11,000 | 11,000 | 11,000 | 11,000 | 11,000 | 11,000 | 11,000 |
| Product Size, $P_{80}$ (µm) | 280 | 280 | 280 | 280 | 280 | 280 | 280 |
| BWi (kWh/t) | 13.5 | 14.0 | 14.5 | 15.0 | 15.5 | 16.0 | 16.5 |
| Drive Losses | 3% | 3% | 3% | 3% | 3% | 3% | 3% |
| Unit Power (kWh/ton) | 7.84 | 8.31 | 8.79 | 9.30 | 9.83 | 10.37 | 10.94 |
| Motor – kW available | 17,904 | 17,904 | 17,904 | 17,904 | 17,904 | 17,904 | 17,904 |
| Motor – hp available | 24,000 | 24,000 | 24,000 | 24,000 | 24,000 | 24,000 | 24,000 |
| Motor – hp, installed | 24,000 | 24,000 | 24,000 | 24,000 | 24,000 | 24,000 | 24,000 |

Figure 13-17 provides a graph and linear equation for the interpretation of the model results for the specific conditions of $F_{80}$ = 11,000 µm and $P_{80}$ = 280 µm. The model and representative equation will change for varying feed and product conditions.

**Figure 13-17. Graphical Interpretation of Work Index Model ($F_{80}$ = 11,000 µm, $P_{80}$ = 280 µm)**



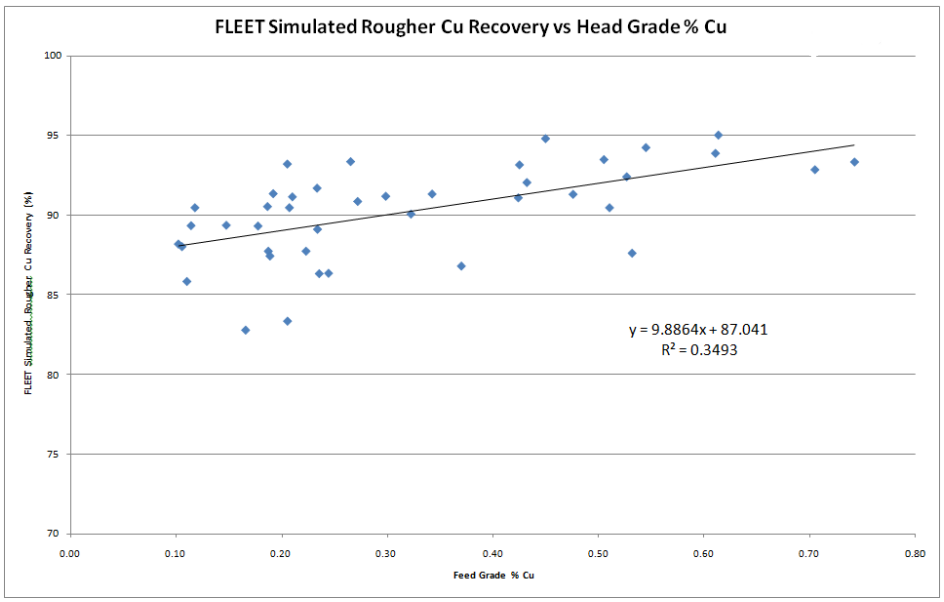

Based on the equipment installed and operating in the crushing circuit and the grinding circuit plant, and considering past processing history, it has been determined that the PVM process facility should operate at an average daily throughput of 55,100 tpd once the target availabilities of 75% crushing and 95% grinding have been achieved. These represent typical accepted industry design and operation availabilities for the respective circuits, and PVM has operated at or better than these targets during the operating mine life. Average ore work indexes and continuation or improvement of current blasting and crushing practices have also been considered when evaluating plant capacity.

Operations at the Pinto Valley mill have been improving steadily over the past 12 months. With optimization of the circuit operating parameters, it is believed that the plant throughput can be increased to 57,300 tpd. For the purposes of the PV2 evaluation, it has been assumed that a period of 2 years will be required for the optimization program. PVM has achieved these higher throughputs over extended periods in the past.

### 13.3.3 Flotation

The metallurgical review indicated that modeling of the flotation recovery has not produced consistent models. The PV2 project requires the pushback of the open pit in multiple directions, and the expectations should be that the mill performance will be very similar to the existing mill. Figure 13-18 is a representation of the FLEET simulated rougher flotation recovery for copper.

**Figure 13-18. Rougher Recovery**




From the review of the various models, it was apparent that none of the models that had been developed were very representative. Flotation of porphyry-type copper deposits typically requires a simple flow sheet approach.

Based on the information that has been presented, it is recommended that the copper flotation recovery model adopt a simple linear model to represent the rougher flotation recovery, followed by a fixed cleaner flotation recovery.

- Copper Rougher Recovery
  - [$R_{RCu}$ = (9.8864) × (%Cu) + 87.041]
- Copper Cleaner Recovery
  - [$R_{CCu}$ = 97.3% ]
- Overall Copper Recovery
  - [$R_{Cu}$ = ((9.8864) × (%Cu) + 87.041) × 0.973)]

Through the discussions, it was implied that the flotation of diabase ores resulted in recoveries 5%–6% lower than for the other lithologies. On review of the reports and presentation, there was no reference to flotation of diabase ores found. Based on the discussions, the following metallurgical model is recommended for determining the copper recovery for diabase ores.

- [$R_{Cu\ (diabase)}$ = ((9.8864) × (%Cu) + 82.041) × 0.973)]

The molybdenum recovery is dependent on the ore feed grade and the performance of the bulk concentrate separation circuit. Molybdenum is a byproduct stream that has a very low mill feed grade. The estimated molybdenum circuit recovery to molybdenum concentrate when the circuit is operating is 47%.

## 13.4 Metallurgical Variability

As indicated previously, most of the ore (96%) in the PV2 resource is 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. The flotation testwork 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.

# 14.0 MINERAL RESOURCE ESTIMATES

## 14.1 Introduction

This section details the methods, processes, and strategies used to calculate the mineral resource estimate for the PVM deposit.

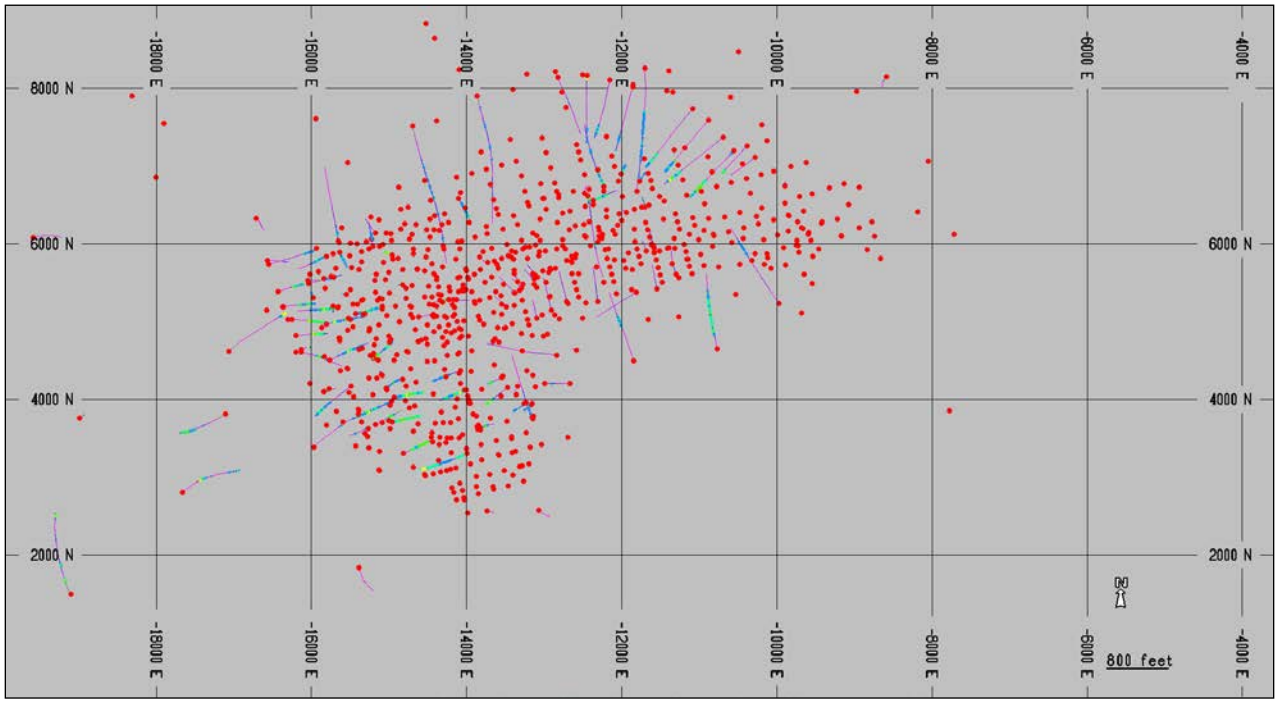
## 14.2 Data Evaluation

A total of 1,031 drill holes were supplied for PVM, which formed the original database as delivered by BHP; however, after rigorous review, 794 drill holes had assay data and could be validated and verified.

The drill hole database was supplied by Capstone in an electronic format. These data included drill hole collars, downhole surveys, lithology data, and assay data with downhole-from and downhole-to intervals in imperial units. The assay data included %TCu and %Mo.

Figure 14-1 shows a plan view for the drill holes used in the mineral resource estimate.

**Figure 14-1. Plan View Showing Drill Holes Used in Resource Estimate**



## 14.3 Computerized Geologic and Domain Modeling

Solids were supplied for the lithology domains along with the grade shells used for the estimation of resources. In addition, surfaces were supplied for the major fault planes. All surfaces were supplied in LeapFrog format.

The numeric coding for the lithology domains are as follows.

- 1, 2, 3, 4, 5, 6, 7, 8: Aplite Dikes
- 31, 32, 33: Diabase Dikes
- 50: Granodiorite
- 60: Granite Porphyry
- 11: Ruin Granite

The solids were adjusted to honor drill hole intercepts and to exclude outlying "orphan" volumes. The drill holes were then coded with the lithology solids. Numeric codes that denote the zones within the drill hole database were manually adjusted to ensure the accuracy of zonal intercepts. No assay values were edited or altered.

Figure 14-2 shows a plan view of the major fault planes that bound the PVM deposit. These are the West End, Gold Gulch, South Hill, Jewel Hill, Dome, and Bummer Faults. Figure 14-3 shows a plan view of the lithology solids within the main block of the PVM deposit, which have been interpreted and modeled in LeapFrog. The primary units are the Ruin granite, granodiorite, and granite porphyry, in addition to a series of flat-lying aplite dikes and near-vertical diabase dikes.

**Figure 14-2. Plan View Showing Major Faults**

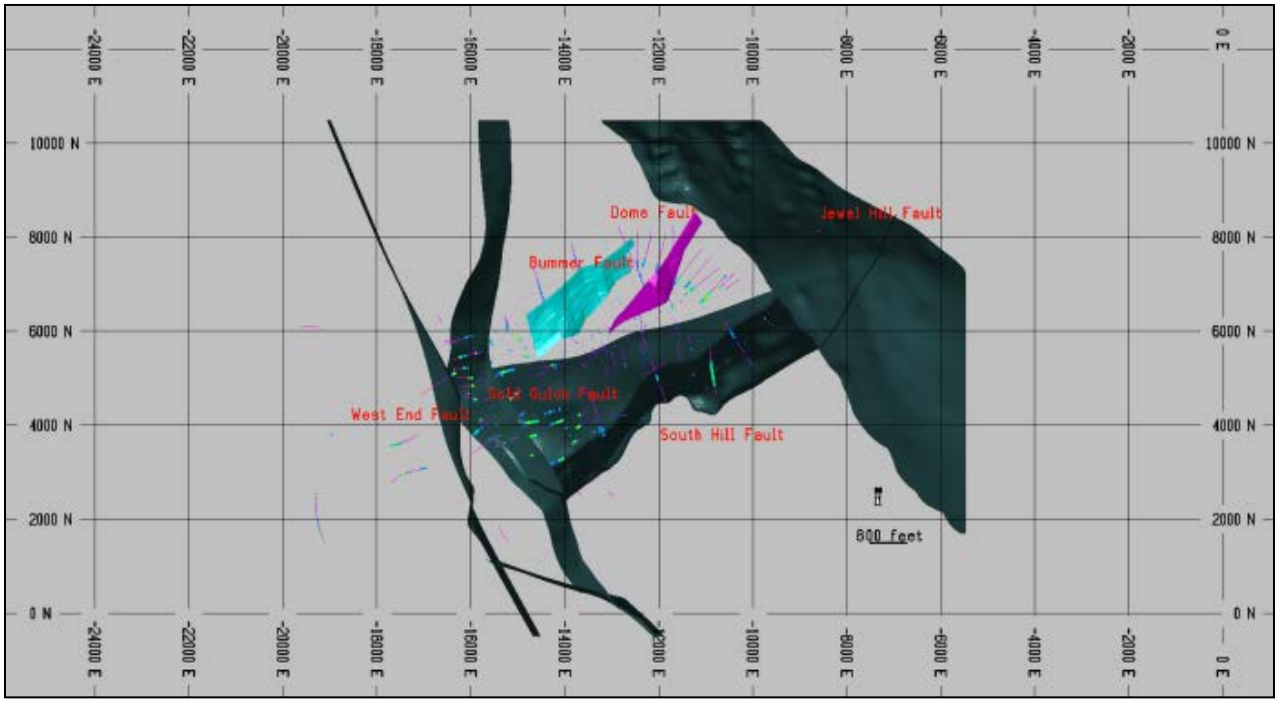
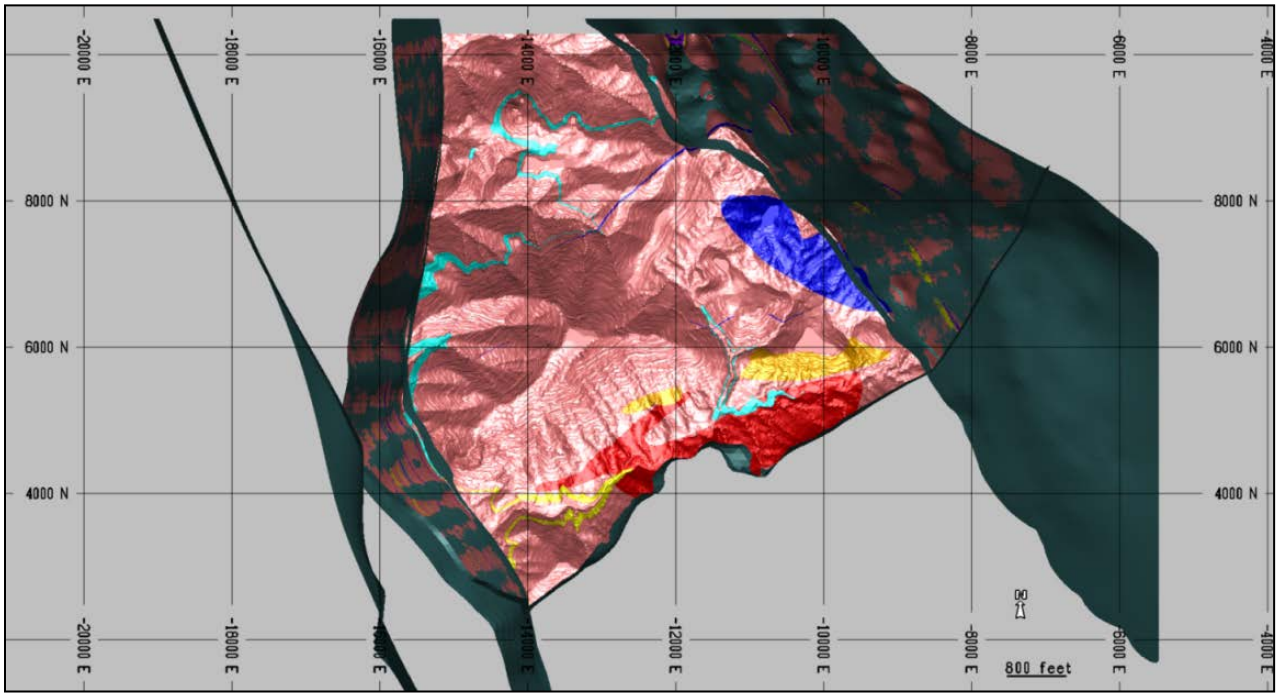


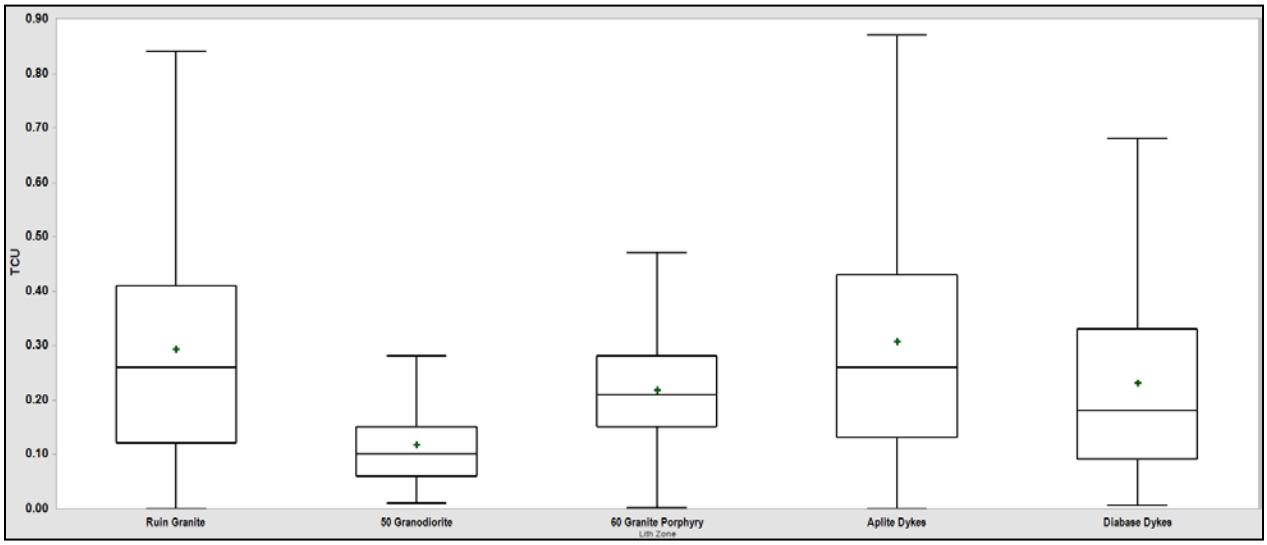

**Figure 14-3. Plan View of Lithology Domain Solids**




Note: Fault planes are shown in dark green, diabase dikes in dark blue, aplite dikes in light blue, granodiorite in red, granite porphyry in yellow, and Ruin granite in pink.

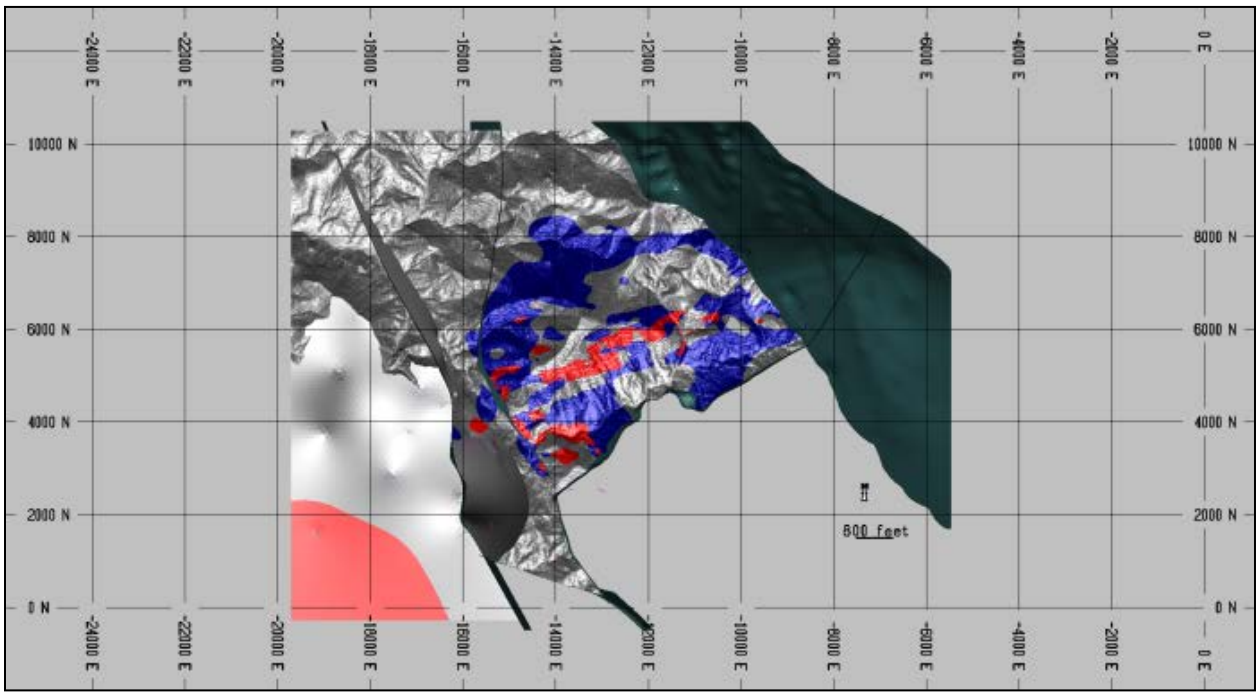
Of potential concern was whether the diabase and aplite dikes would require segregation as a separate set of domain classes during the estimation process. Box plots were run for the various lithology units, as shown in Figure 14-4, which illustrates that the aplite dikes are identical in nature to the Ruin granite, which they crosscut horizontally. Therefore, the Ruin granite and the aplite dikes were grouped as a common domain for the estimation. Furthermore, the diabase dikes are also very similar, as is the granite porphyry, so it was decided that creating subsets of the lithologic units within the mineralized envelopes was not necessary. The granodiorite exhibits lower grade than the other units; however, the granodiorite lies exclusively within the low-grade envelope.

**Figure 14-4. Box Plot with Domain Solids**



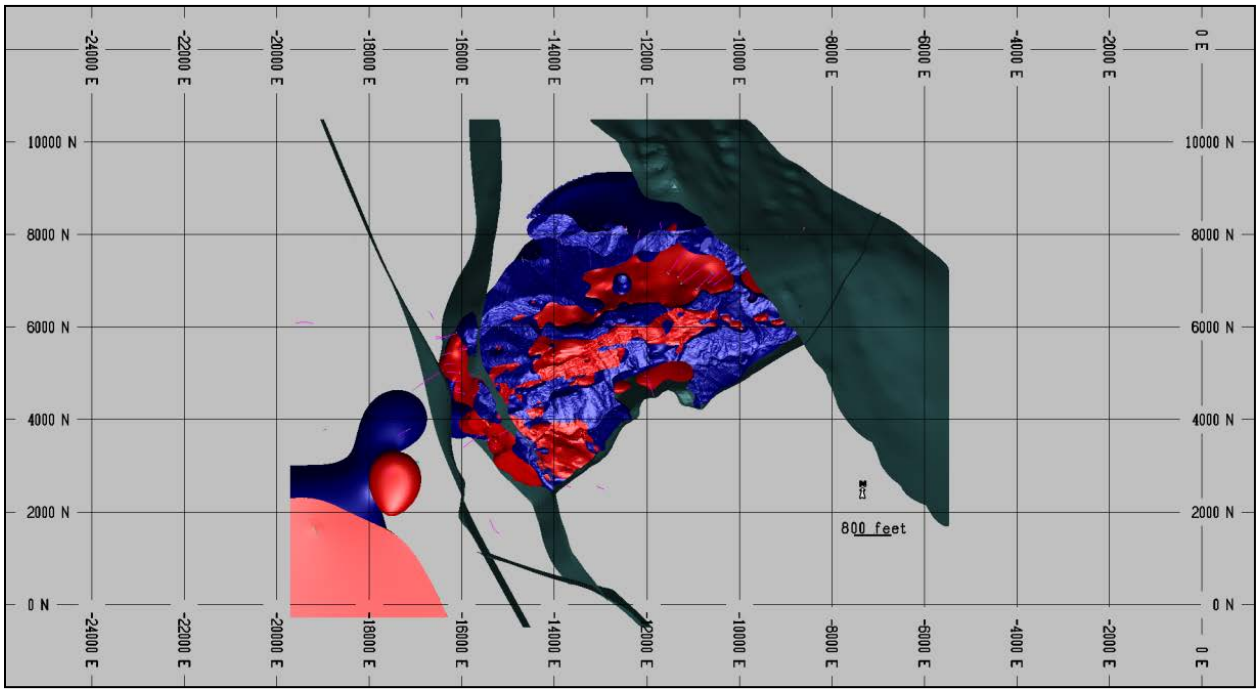

Mineralized envelopes were created using the copper composites at thresholds of 0.3% TCu and 0.1% TCu. In addition, the waste defined as <0.1% TCu was modeled as a solid. Figure 14-5, Figure 14-6, and Figure 14-7 show the three solids, with waste being gray, 0.1%–0.3% TCu being blue, and >0.3% TCu being red.

**Figure 14-5. Plan View of Mineralized Domain Solids – Waste, Low-Grade, and High-Grade**



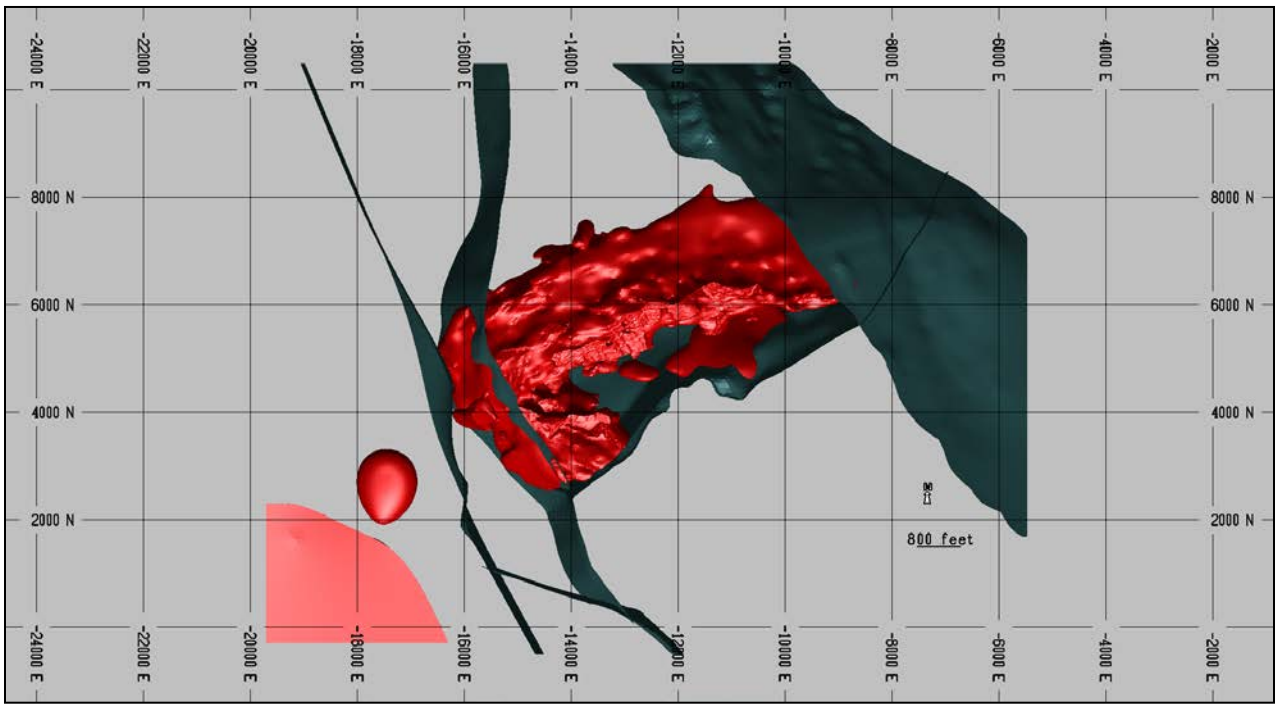

Note: Faults are shown in dark green, high-grade (>0.3% TCu) in red, low-grade (>0.1% TCu and <0.3% TCu) in blue, and waste solid (<0.1% TCu) in gray.

**Figure 14-6. Plan View of Mineralized Domain Solids – Low-Grade and High-Grade**




Note: Faults are shown in dark green, high-grade (>0.3% TCu) in red, and low-grade (>0.1% TCu and <0.3% TCu) in blue.

**Figure 14-7. Plan View Showing Mineralized Solids – High-Grade**

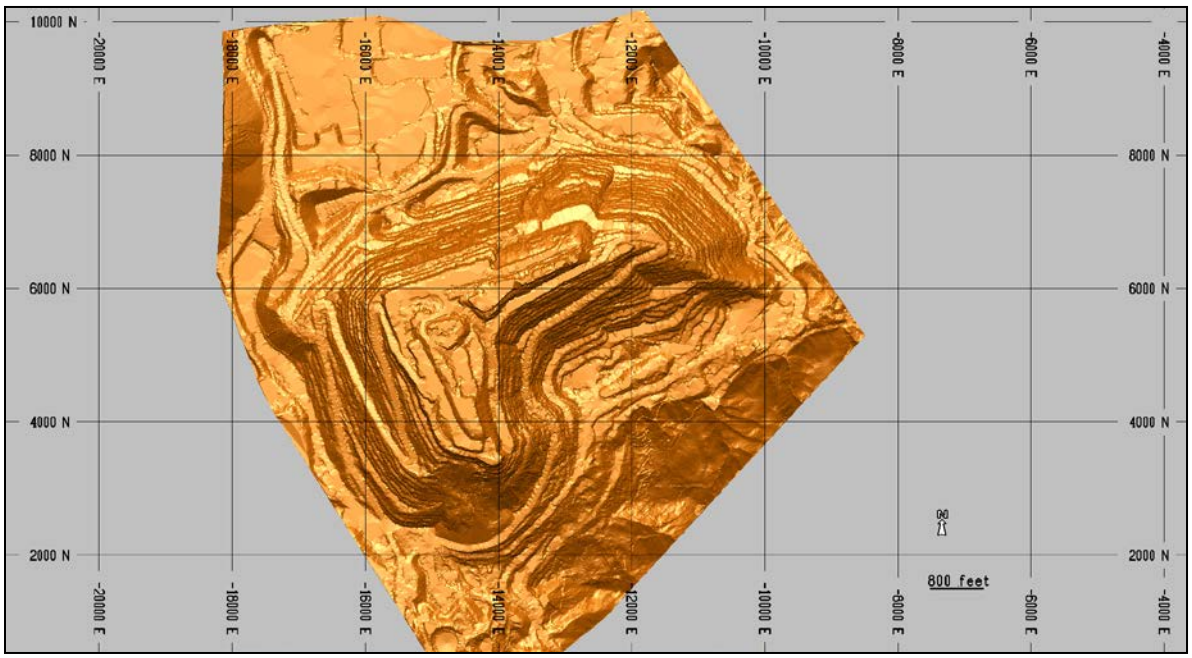



Note: Faults are shown in dark green, and high-grade (>0.3% TCu) in red.

## 14.4 Topography

The topography was obtained from a contour map, and a digital solid surface was created. The solids and contours are the result of merged surface, which incorporated recent 2014 survey data along with data derived from the 2011 topographic surface for areas away from the pit. This merged topography is used as the upper boundary surface of the deposit. Figure 14-8 shows the contours in plan view.

**Figure 14-8. Plan View of Topographic Solid**




## 14.5 Composites

It was determined that a 45 ft composite length minimizes the smoothing of the grades, but also reduces the influence of typically narrow, very high-grade samples; this appears to be an optimal interval length from the standpoint of regularization. The main justification to adopt the 45 ft composite length is because the mine has been using this length, and consistency with past practices is desirable. In addition, the SMU being used for mine planning is a 45 ft bench.

Figure 14-9 and Figure 14-10 show the box plots for copper and molybdenum within the mineralized envelopes. The basic statistics shown in the figures indicate that the copper and molybdenum data are reasonably distributed. The mean grade is 0.199% and 0.399% for copper, and 0.006% and 0.010% for molybdenum in the 0.1%–0.3% TCu and 0.3% TCu grade envelopes, respectively.

Copper and molybdenum composites have a relatively low coefficient of variation, ranging from 0.515 to 0.724 for copper and from 0.835 to 1.003 for molybdenum within the 0.3% TCu and 0.1%–0.3% TCu grade envelopes, respectively.

**Figure 14-9. Box Plot for Copper Composites by Zone**

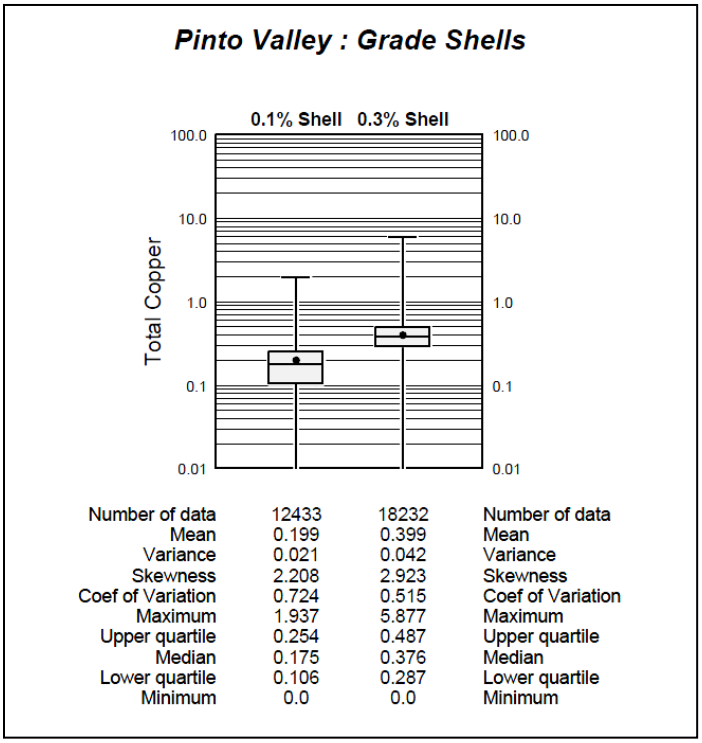



**Figure 14-10. Box Plot for Molybdenum Composites by Zone**

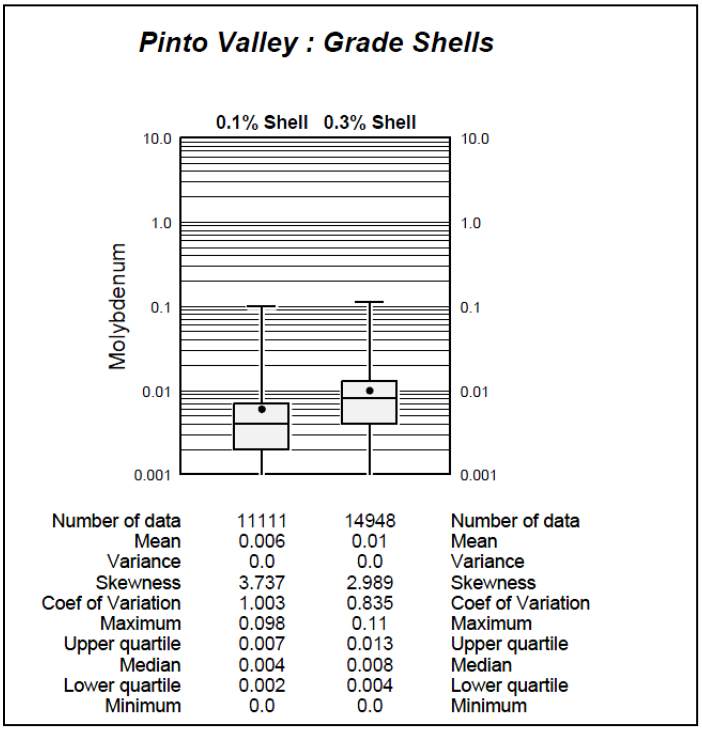


Figure 14-11 and Figure 14-12 show contact plots copper and molybdenum within the 0.1% and 0.3% grade envelopes for copper. As expected, the copper values show a clear, abrupt transition at the contact between the 0.1%–0.3% TCu and the 0.3% TCu grade envelopes. However, the contact plot for molybdenum shows little, if any, transition between the two grade envelopes. This shows that the molybdenum may be evaluated and interpolated as one continuous mineralized domain.

**Figure 14-11. Contact Plots for Copper**

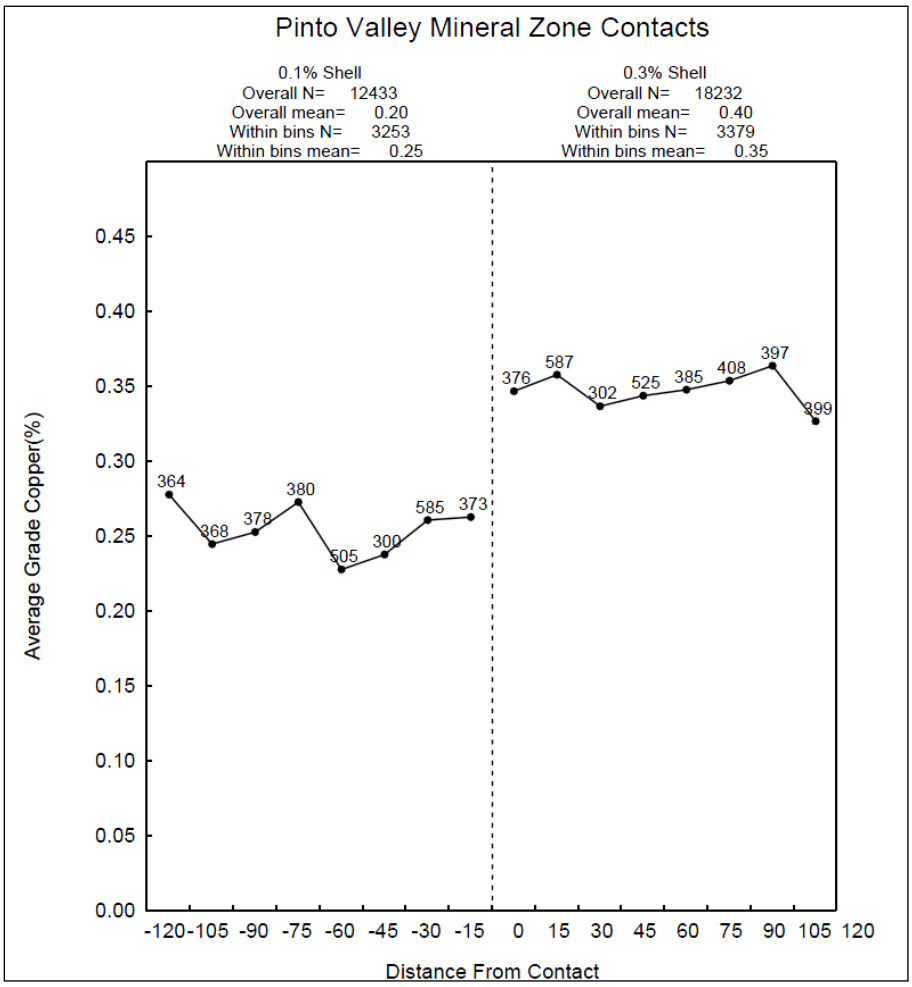
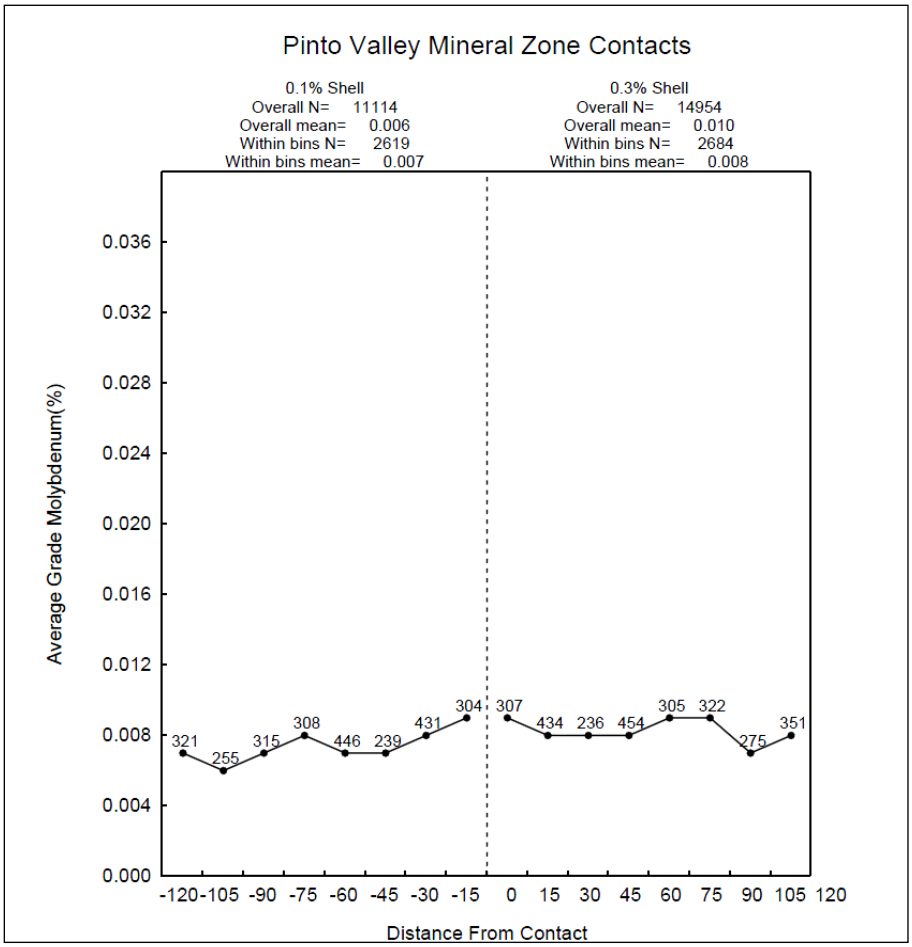


**Figure 14-12. Contact Plots for Molybdenum**




## 14.6 Outliers

Although there appear to be isolated high-grade outliers, there does appear to be a separate and distinct population that requires segregation or grade limiting. Cumulative frequency plots shown in Figure 14-13 and Figure 14-14 for %TCu and %Mo illustrate that at 1.6% TCu and 0.05% Mo, there is a break in the log normal plot. This represents 0.1% of the copper and 4% of the molybdenum composites, which require implementation of a grade-limiting strategy. Two methods are common in order to address high-grade outliers. One is by physically cutting the grades of the assays or composites, and the other is by limiting the influence that a high-grade sample has by limiting the distance to which it contributes to the grade of a block estimate. In the case of the PVM deposit, the distance (radius) threshold chosen was 150 ft, which equates to the adjacent, adjoining blocks and no farther.

**Figure 14-13. Cumulative Frequency Plot for Copper**

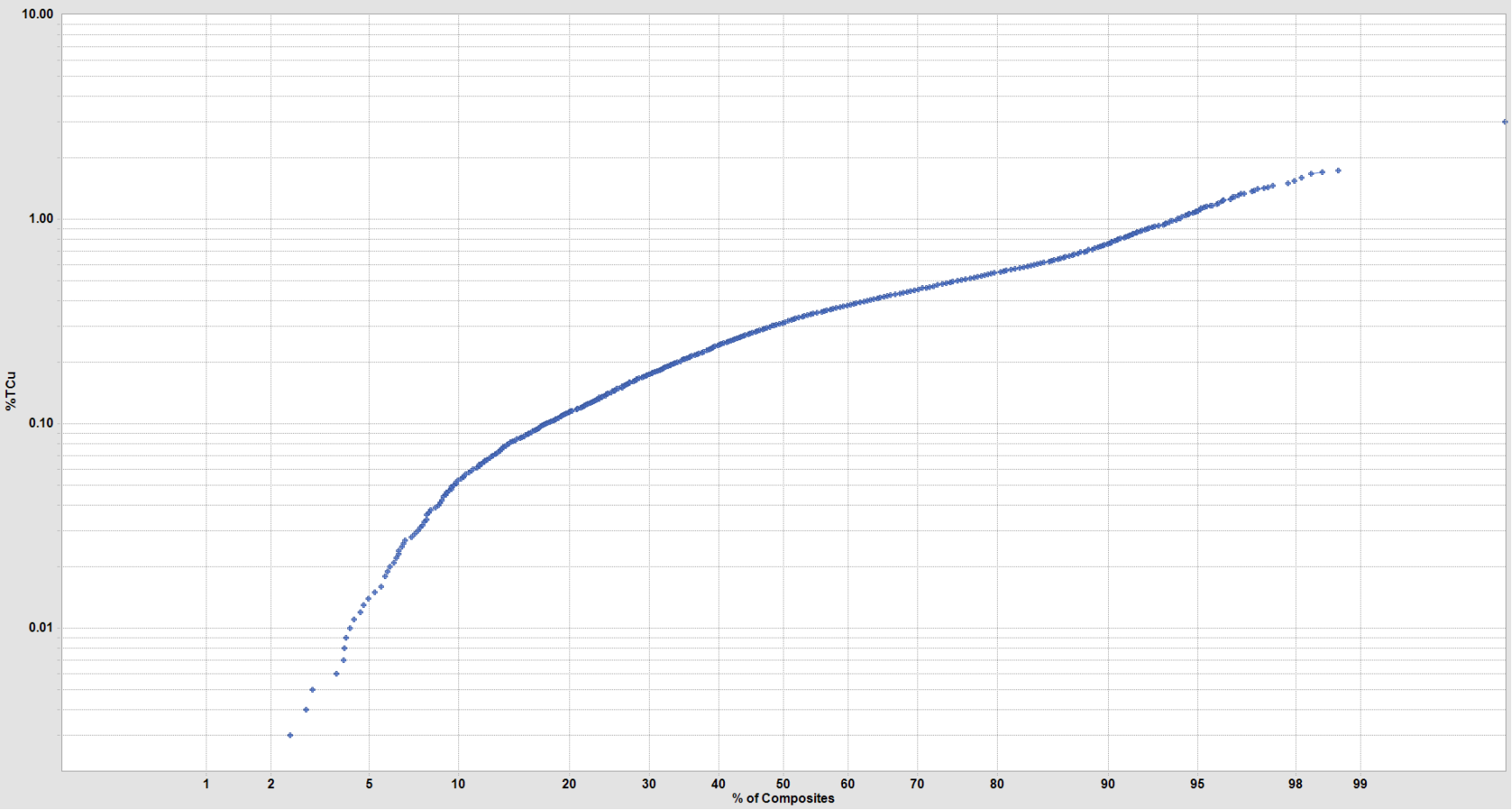
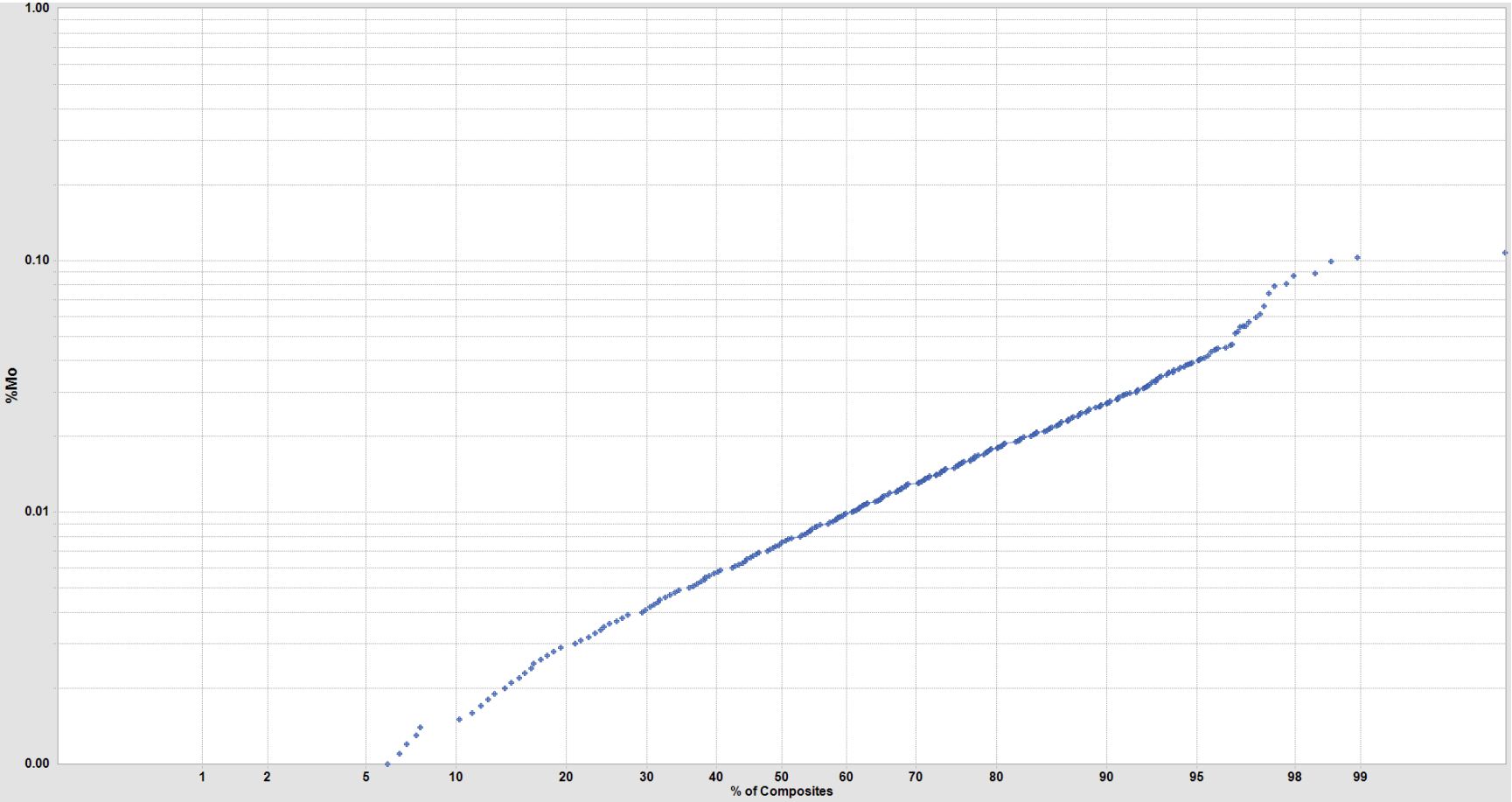


**Figure 14-14. Cumulative Frequency Plot for Molybdenum**



## 14.7 Tonnage Factor

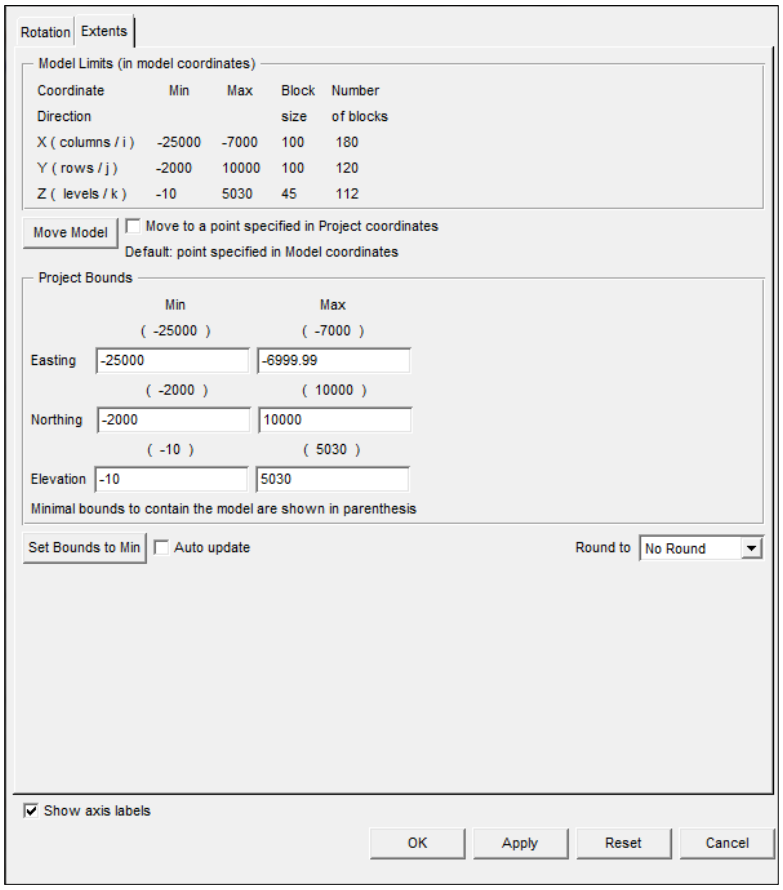
The average bulk dry density for ore-grade mineralized rock, primarily Lost Gulch quartz monzonite, is 12.75 $ft^3$/ton. Although the in situ bulk dry densities for all PVM rock types range from 12.1 $ft^3$/ton for Pinal schist to 13.0 $ft^3$/ton for Whitetail conglomerate, 12.75 $ft^3$/ton has been used for all reserve calculations. Although efforts to locate supporting documentation for this density data were unsuccessful, personal communications with former employees indicate that detailed density studies had been done. Further, production reconciliations tend to support the 12.75 $ft^3$/ton factor; it had been reported in production comparisons that even though the block model under-predicted tonnage, the 12.75 $ft^3$/ton tonnage factor that was used still provided a reasonable estimate. Since the resources remaining in the ground are geologically the same as the resources already mined from the primary zone, it is reasonable to assume that this density will also provide globally reasonable estimates of remaining resource tonnages.

## 14.8 Block Model Definition

The block model used to calculate the mineral resources was defined according to the limits shown in Figure 14-15.

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 drill hole spacing and to adequately represent the deposit with a representative amount of discrete points in three dimensions. In addition, this was the selected SMU for past operations and for going forward.

**Figure 14-15. Block Model Bounds**




## 14.9 Variography

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

The components of the 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 begins at a point on the y-axis above the origin; this point is called the *nugget*. The nugget is a measure of not only the natural variability of the data over very short distances, but also a measure of the variability that can be introduced due to errors during sample collection, preparation, and assaying.

Typically, the amount of variability between samples increases as the distance between the samples increases. 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*.

The spatial evaluation of the data was conducted using a correlogram instead of the traditional variogram. 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 and molybdenum in the various areas using the commercial software package Sage 2001, developed by Isaacs & Co. Correlogram model data are shown in Table 14-1.

**Table 14-1. Correlogram Model Data by Metal**

| Metal | C0 | C1 | C2 | Range Y | Range X | Range Z | Rotation Z | Rotation X | Rotation Y |
|---|---|---|---|---|---|---|---|---|---|
| %TCu | 0.120 | 0.447 | 0.433 | 63.7 | 186.9 | 117.2 | −52 | 46 | −6 |
| | | | | 787.3 | 561.9 | 1,204.4 | −14 | −8 | −89 |
| %Mo | 0.155 | 0.428 | 0.417 | 17.4 | 120.3 | 228.6 | −15 | 0 | 17 |
| | | | | 1,153.8 | 878.9 | 1,838.6 | −19 | −47 | −96 |

The block model grades for copper and molybdenum were estimated using ordinary kriging. Estimates were validated using the Hermitian Polynomial Change of Support model (Journel and Huijbregts 1978), also known as the Discrete Gaussian Correction. The ordinary kriging models were generated with a relatively limited number of composites to match the change of support or Herco (*Her*mitian *co*rrection) grade distribution. This approach reduces the amount of smoothing (also known as averaging) in the model, and while there may be some uncertainty on a localized scale, this approach produces reliable estimates of the potentially recoverable grade and tonnage for the overall deposit. The interpolation parameters are summarized by domain in Table 14-2.

**Table 14-2. Interpolation Parameters**

| Range X (ft) | Range Y (ft) | Range Z (ft) | Rotation Z | Rotation X | Rotation Y | Minimum Number Composites | Maximum Number Composites | Maximum Number Composites per DDH* |
|---|---|---|---|---|---|---|---|---|
| 750 | 750 | 300 | 0 | 0 | 0 | 6 | 15 | 5 |

*Diamond drill hole

During grade estimation, search orientations were designed to follow the general trend of the mineralization in each of the zone domains. The estimation plan includes the following.

- Store the mineralized zone code and percentage of mineralization.
- Estimate the grades for each of the metals using ordinary kriging in a single pass.
- Include a minimum of 6 composites and a maximum of 15, with a maximum of 5 from any one drill hole.

Figure 14-16 and Figure 14-17 show the resulting block model in plan view for copper and molybdenum grades, respectively.

**Figure 14-16. Plan View of Block Model Showing Copper Grade Model at 3230 Elevation**

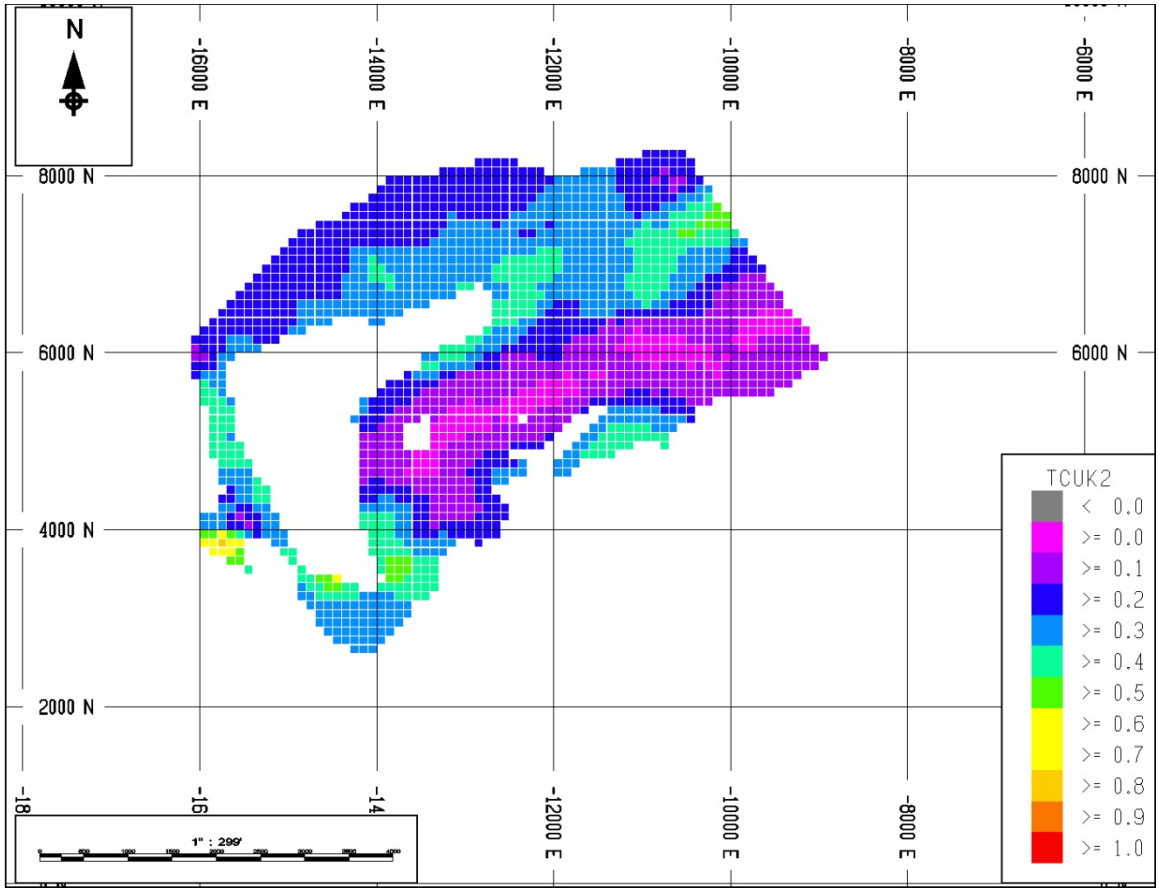
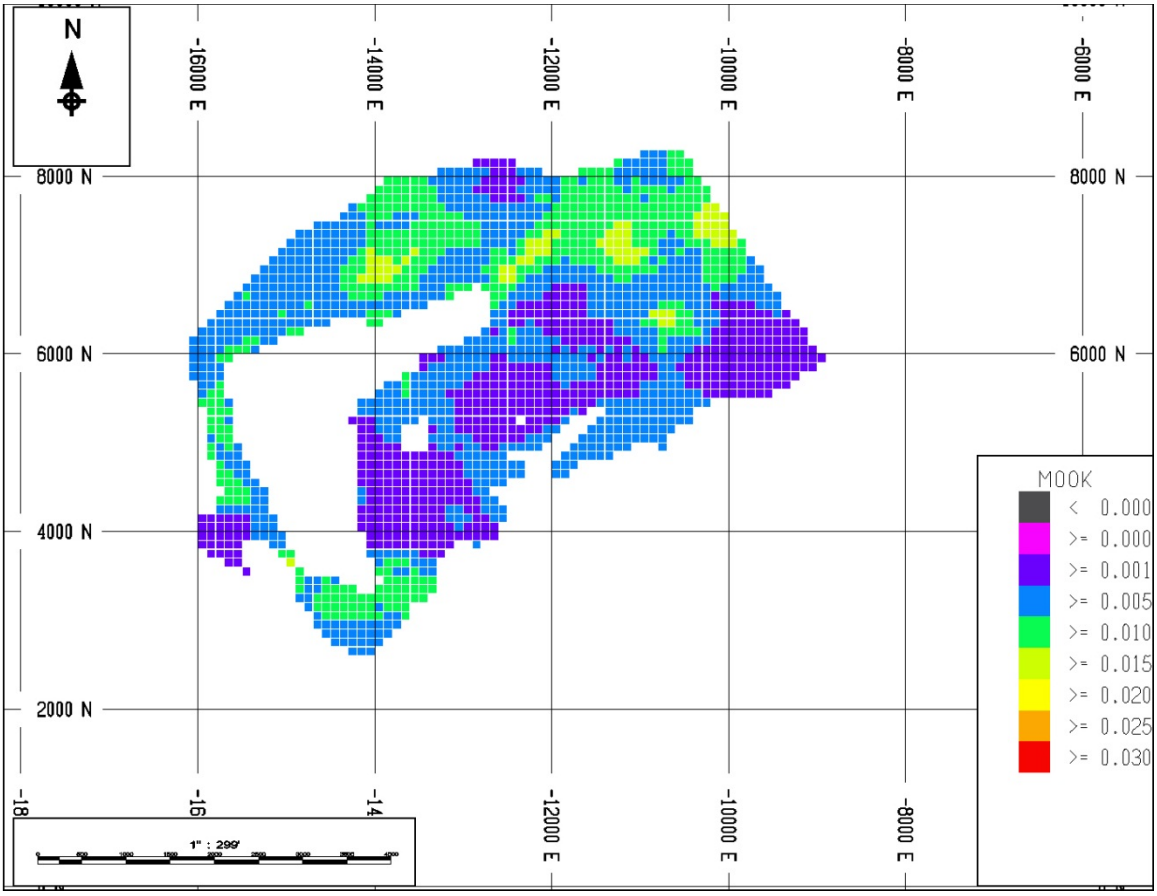


**Figure 14-17. Plan View of Block Model Showing Molybdenum Grade Model at 3230 Elevation**




## 14.10 Mineral Resource Classification

The spatial variation pattern incorporated in the variogram and the drill hole spacing can be used to help predict the reliability of estimation for copper metal. (In this case, there are at least two potentially economic metals, but copper is likely 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. Then, by knowing how reliably metal content must be estimated to adequately plan, it is possible to calculate the drill hole spacing necessary to achieve the target level of reliability. For instance, indicated resources may be adequate for planning 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 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 spacing of about 200 ft. Results from this study will be validated against current and future production.

Confidence intervals are intended to estimate the reliability of estimation for different volumes and levels of drill hole 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 method of estimating confidence intervals is an approximate method that has been shown to perform well when the volume being predicted from samples is sufficiently large (Davis 1997). In this case, the smallest volume where the method would most likely be appropriate is the production from 1 year. Using these guidelines, an idealized block configured to approximate the volume produced in one month is estimated by ordinary kriging, using the idealized grids of samples.

The kriging variances from the ideal blocks and grids are divided by 12 (assuming approximate independence in the production from month to month) to get a variance for yearly ore output. The square root of this kriging variance is then used to construct confidence limits under the assumption of normally distributed errors of estimation. For example, if the kriging variance for a block is $\sigma^2_m$ then the kriging variance for a year is $\sigma^2_y = \sigma^2_m/12$. The 90% confidence limits are then *confidence limit* $= \pm 1.645 \times \sigma_y$.

The confidence limits for a given production rate are a function of the spatial variation of the data and the sample or drill hole spacing. For this exercise, the drill hole grids tested were 900 ft × 900 ft, 600 ft × 600 ft, 300 ft × 300 ft, and 150 ft × 150 ft.

Further assumptions made for the confidence interval calculations are as follows.

- The variograms are appropriate representations of the spatial variability for presence of mineralization and metal grade.
- The tonnage factor for the domains is 12.75 $ft^3$/ton.
- Most of the uncertainty in metal production within zones is due to the fluctuation of metal grades and not to variation in the presence or absence of the unit.

The following details the grid spacing for each resource category to classify resources, based on the assumptions that were discussed above.

- Measured
  - Note that based on the Canadian Institute of Mining (CIM) definitions, continuity must be demonstrated in the designation of measured (and indicated) resources; therefore, no measured resources can be declared based on one hole. The uncertainty based on current information suggests a spacing of 150 ft may be required to classify measured resources.
- Indicated
  - Resources in this category could be delineated from multiple drill holes located on a nominal 500 $ft^2$ grid pattern.
- Inferred
  - Resources in this category include any material not falling in the categories above, and within a maximum 1,000 ft of one hole.

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

The study described above indicates a drill spacing of around 500 ft may be sufficient in delineating indicated resources.

To further ensure confidence and continuity, the blocks were displayed at the chosen thresholds of 150 ft and 500 ft to the nearest composite, and a boundary was digitized to create a smooth surface and to reduce the presence of isolated islands of blocks. A solid was then created and coded back into the model by majority code and using >50% partials to be classified as measured or indicated. The remainder that is greater than 500 ft, but not more than 100 ft from nearest composite, was classified as inferred.

## 14.11 Mineral Resources

The resources show reasonable prospects of economic extraction.

*CIM Definition Standards for Mineral Resources and Mineral Reserves* (CIM 2010) 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 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 "reasonable prospects for economic extraction" requirement generally implies that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cutoff grade, taking into account extraction scenarios and processing recovery.

The "reasonable prospects for economic extraction" were tested using floating cone pit shells based on reasonable economic assumptions. The economic assumptions include the following: $3.30/lb Cu, $10.00/lb Mo, 88% Cu recovery, 50% Mo recovery, $1.50/ton mining costs, $1.50/ton G&A costs, $5.00/ton milling costs, and a pit slope of 45° (Kirkham 2013b). The pit optimization results are used solely for the purpose of testing the "reasonable prospects for economic extraction" and were not used to estimate the mineral reserves in this technical report. The optimization results were used to assist with the preparation of a mineral resource statement and to select and appropriate reporting assumptions.

The mineral resources are listed in Table 14-3 for %TCu and %Mo. These mineral resources are listed at a base-case cutoff grade of 0.18% TCu.

**Table 14-3. Mineral Resources at 0.18% TCu Cutoff (imperial units)**

| Classification | M Tons | %TCu | %Mo | Contained Cu (M lb) | Contained Mo (M lb) |
|---|---|---|---|---|---|
| Measured (M) | 728.0 | 0.34 | 0.008 | 5,014.3 | 119.4 |
| Indicated (I) | 994.9 | 0.28 | 0.006 | 5,491.8 | 123.4 |
| **Total M & I** | **1,722.8** | **0.30** | **0.007** | **10,506.0** | **242.8** |
| Inferred | 64.1 | 0.23 | 0.005 | 298.5 | 6.8 |

Note: Summation errors due to rounding.

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. Also, there is no certainty that these inferred mineral resources will someday be converted into measured or indicated resources as a result of future drilling or after applying economic considerations.

## 14.12 Model Validation

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

- 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 filled in.
- 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 explain, when necessary, the high-grade blocks created as a result of outliers.

A full set of cross sections, long sections, and plans were used to check the block model visually, 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 cutoff 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.

# 15.0 MINERAL RESERVE ESTIMATES

The mineral reserve was developed from the block model that was developed by Garth Kirkham and summarized in Section 14 – Mineral Resource Estimates. The mine plan and mineral reserve were developed by IMC, and the QP for the mineral reserve is John Marek, PE, President of IMC. IMC and John Marek have not audited or verified the block model and have relied on the work of Mr. Kirkham.

The mineral reserve is the total of all proven and probable category ore that is planned for processing at the existing sulfide mill. Section 16 – Mining Methods details the development of a mine plan that delivers sulfide ore to the mill at the planned mill throughput and moves sufficient waste to assure continued release of that sulfide mill ore. The mineral reserve was established by tabulating the contained tonnage of measured and indicated material (proven and probable) within the mining plan for sulfide mill processing. The final pit design and the internal phase (pushback) designs were guided by the results of the floating cone algorithm.

The Pinto Valley mine plan is currently constrained by the capacity of the TSF. The TSF will hold approximately 250 M tons of tailings without requiring additional area or higher elevations above the current permitted level. The goal of this mine plan is to produce 250 M tons of ore that will generate the highest overall project return on investment. Open-pit guidance algorithms, such as cones or Lerch-Grossman routines, were not intended for that purpose, so their application must be modified for application to the development of this mine plan.

## 15.1 Floating Cones

The floating cone computer algorithm is a tool that provides guidance to mine design. The algorithm applies approximate costs and recoveries, along with approximate pit slope angles, to establish theoretical economic breakeven pit wall locations. All of the floating cone and mine plan discussions in this section and the subsequent sections address measured and indicated (proven and probable) ore only. Inferred-category mineralization is considered to be waste from this point forward in the project evaluation.

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. However, 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 floating cone. Process recoveries and estimated process costs were developed from budget and historical information provided by Capstone. Slope angles were provided by the geotechnical engineering consulting firm Call & Nicholas, Inc. (CNI). Mine operating costs were developed for input to the cones by IMC based on other recent projects and on budget estimates for the current year by Capstone.

Multiple floating cones were completed at a range of metal prices. Copper prices ranging from $2.00/lb to $2.75/lb were applied within the cone runs. The base case is defined at the $2.75/lb Cu price and $12.75/lb Mo. Molybdenum prices for the multiple cones were set at the same ratio to base case as copper. For example, a floating cone at $2.20/lb Cu utilized a molybdenum price of (2.20/2.75 × 12.50 = $10.00/lb).

PVM has historically received a precious metal credit from smelters that is primarily attributed to silver contained in concentrate. A simplified calculation of silver credit per pound of copper smelted is applied, as noted in Table 15-1.

The Pinto Valley pit is currently limited by property ownership. The property boundary limits the floating cone and mine plan designs in the southeast corner of the pit. The floating cone algorithm was limited to respect the property boundary.

Once all were completed, the multiple floating cones were analyzed in a unique way in order to accommodate the 250 M tons constraint on total ore. All of the 10 cone geometries that were generated could produce 250 M tons of ore. The larger cones could apply a high cutoff grade to produce 250 M tons, and smaller cones necessarily would require a low cutoff to produce that tonnage. The smaller cone with a lower cutoff would have the lowest overall cost per ton of ore. Alternatively, the large pit with the high cutoff would have the highest grade and highest gross income per ton of ore processed. The following steps were followed to establish the cone for design guidance.

**Analysis of Cone Results**

1. For each of the 10 cone geometries, find the total copper cutoff grade that produces 250 M tons of ore.
2. Apply that cutoff grade to tabulate the ore tonnage and grades, as well as the waste requirement for the cone.
3. Apply the base-case cone economics to each of the 10 cases, and determine which of the 250 M ton cone cases has the best non-discounted benefit to the project.
4. Apply the cone bench discounting routine to each of the 10 cases, and determine which of the 250 M ton cone cases has the best discounted (approximate) benefit for the project.

The results indicated that the $2.20/lb floating cone was the best overall economic target for long-term mine planning. Figure 15-1 illustrates the $2.20 floating cone, and Figure 15-2 illustrates the final pit design that was used for the reserve. Both maps illustrate the property boundary that was respected in the floating cones and mine plan.

## 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 | $1.40/ton |
| Haulage Cost Increment per Bench | $0.022/ton /bench depth below 3995 bench |
| Average Total Mining Cost With Haul Increment | $1.84/ton |
| Bench Discounting | **1%** / bench of depth below the 3995 bench |
| **Process Costs for Mill Operation** | |
| General and Administrative Cost | $1.50/ton |
| Milling Cost | $5.00/ton |
| **Flotation Process Recovery** | |
| Copper Recovery | |
| • All Except Diabase | Recovery (%) = ((Cu Grade × 9.8864) + 87.041) × 0.973 |
| • Diabase | Recovery (%) = ((Cu Grade × 9.8864) + 82.041) × 0.973 |
| Molybdenum Recovery | 50% |
| **Concentrate Transport Cost** | |
| Total Transport | $154.57/dry ton |
| **Treatment and Refining Costs for Copper** | |
| Moisture Content of Copper Concentrate | 9% |
| Copper Concentrate Grade | 27.50% |
| Copper Smelting Recovery | 96% |
| Copper Smelting Cost | $84.74/dry ton |
| Copper Refining Cost | $0.085 (refining cost/lb recovered Cu) |
| **Total Cost per Pound Recovered Copper** | **$0.5238 (SMRF* cost/lb recovered Cu)** |
| **Silver Credit per Pound of Copper** | |
| Silver Grade in Concentrate | 1.279 oz/ton |
| Payable Silver in Concentrate | 90% |
| Silver Price | $20.00/oz Ag |
| Silver Value per Ton of Concentrate | $23.02/ton |
| Silver Value per Pound of Copper Metal in Concentrate | $0.0436/lb Cu as credit against SMRF |
| **Net Copper Smelting Cost per Pound for Cones** | **$0.4802 (SMRF cost/lb recovered Cu)** |
| **Treatment and Refining Costs for Molybdenum** | |
| Molybdenum Payable from Roaster | 80% |
| Molybdenum Roasting Costs | $1.13/lb recovered Mo |
| Molybdenum Concentrate Transport Costs | $47.50/dry ton |
| Molybdenum Concentrate Moisture Content | 4.5% |
| Molybdenum Concentrate Grade | 55.0% |
| **Net Transport and Roasting Cost for Molybdenum** | **$0.0540 (cost/lb recovered Mo)** |
| **Metal Prices for Base Case** | |
| Copper Price | $2.75/lb Cu |
| Molybdenum Price | $12.50/lb Mo |

| Slope Angles (based on CNI recommendations) | Floating Cone Slope Angles |
|---|---|
| Pinal Schist Shear | 24° |
| Pinal Schist | 27° |
| Post Mineral Group West / Shear, West Wall Shear | 32° |
| Post Mineral Group East | 35° |
| Bummer Fault Zone | 36° |
| West Wall Diabase / Limestone | 40° |
| Ruin Granite West of Bummer Fault | 42° |
| Ruin Granite East of Gold Gulch Fault | 46° |
| Ruin Granite East of Bummer Fault | 48° |

*Smelting and Refining

The $2.20 floating cone did not meet a number of practical constraints required for execution of the mine plan.

- The $2.20 cone indicated that a very narrow (50 ft–100 ft) slice should be taken from the entire north wall of the pit. Mining on that narrow width is not possible, and that segment of the cone was necessarily ignored.
- The $2.20 cone indicated that a very narrow (50 ft–100 ft) slice should be taken from the south end of the pit in the area of the Pinal schist. Mining on that narrow width is not possible, and that segment of the cone was also ignored.
- Developing access to the pit floor for ore and providing an improved haul route for waste required mining more width than the cone indicates in the Castle Dome area (southeast) in order to mine on a practical width and develop mine access roads.

The costs and recoveries in Table 15-1 result in the following simplified cutoff grades if the contribution of molybdenum is not included in the cutoff calculation.

- Internal or Marginal Cutoff = 0.17% TCu
- Breakeven Cutoff = 0.22% TCu

IMC completed an additional analysis to determine the impact of molybdenum on the floating cone results and the cutoff grade calculations. In summary, molybdenum provides the equivalent value of about 0.02% TCu. Comparisons of grade tonnage curves between an NSR cutoff approach applying the molybdenum credit and a simple total copper cutoff grade were sufficiently close that the reserve and mine plan will be based on the total copper cutoff.

Figure 15-2 illustrates the final pit design that was used to develop the mineral reserve. Comparisons with the floating cone in Figure 15-1 indicate the areas where diversion from the cone was necessary to meet geotechnical constraints and to assure that the pit design could be operated efficiently.

The best way to understand the final pit design in Figure 15-2 is to understand that it is the final pushback in a series of four phases that were used to develop the mine schedule. Haul roads are 115 ft wide with a maximum gradient of 10%.

Pushbacks maintain a minimum of 300 ft of operating room for the life of this schedule. Inter-ramp slope angles that were used for the final pit and phase designs are summarized in Section 16.

**Figure 15-1. Floating Cone for Final Pit Design Based on $2.20/lb Copper (magenta line is the property boundary)**

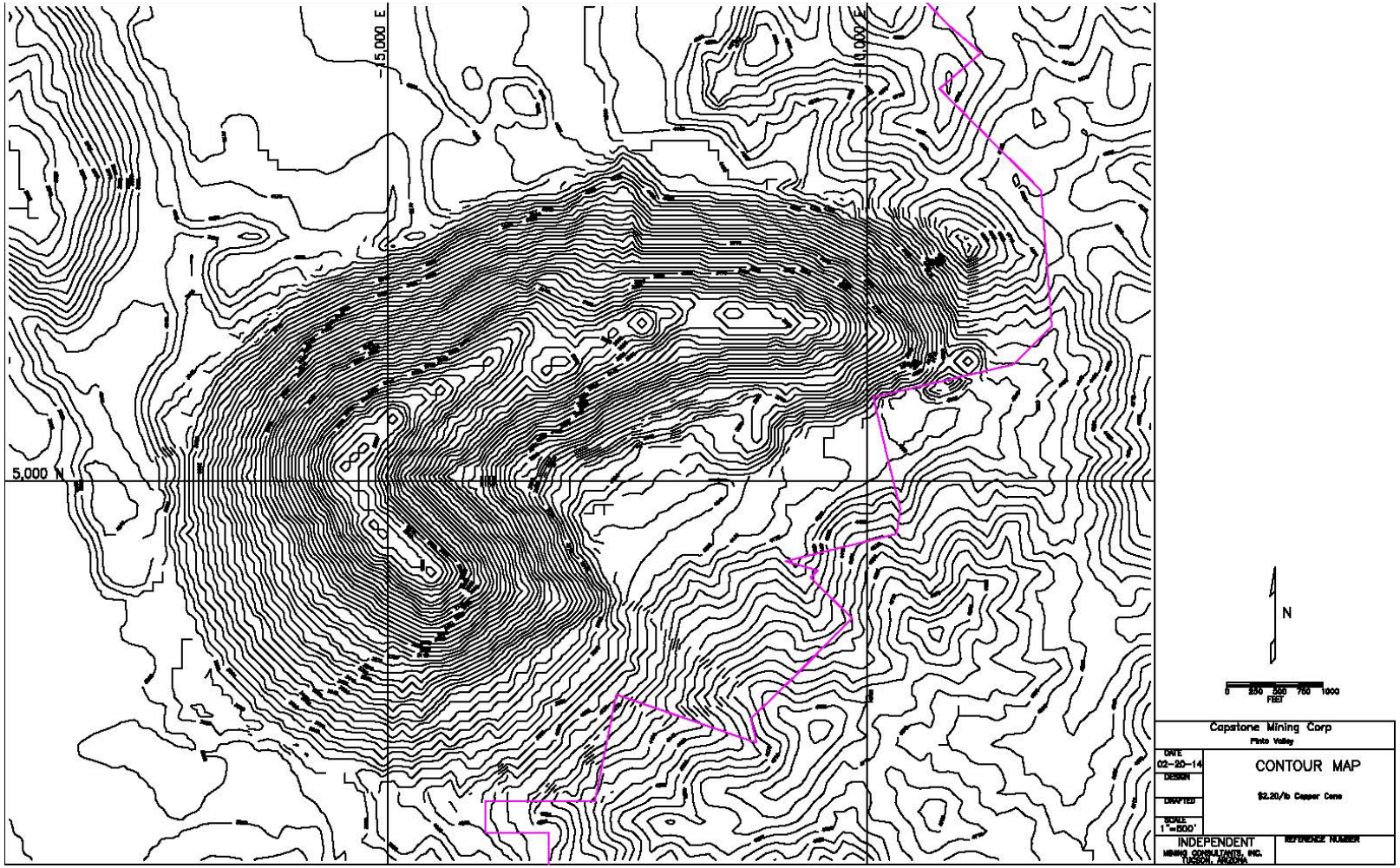
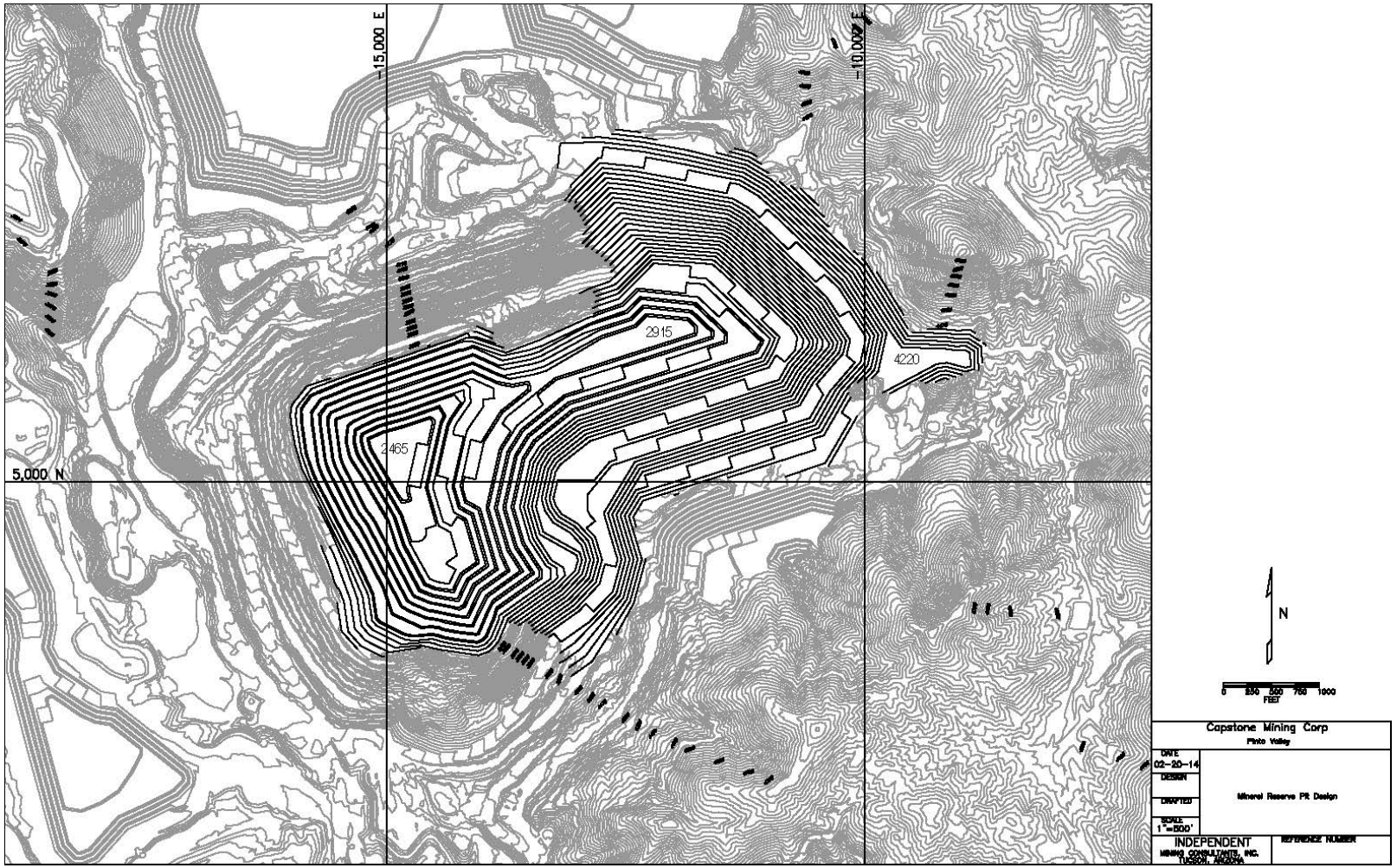


**Figure 15-2. Final Pit Design for Mineral Reserve Determination**



## 15.2 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 cutoff grades. The mine plan and schedule that is presented in Section 16 utilizes a declining cutoff grade to the mill that starts at breakeven cutoff in 2014 and reduces to internal cutoff grade in 2023.

During the years that the cutoff grade to the mill is above 0.18% TCu, material with grade between mill cutoff and 0.18% is stockpiled for future processing. The stockpile cutoff of 0.18% TCu is higher than internal cutoff because it includes the cost of rehandling from the stockpile to the mill. In summary, the cutoff grade for the calculation of mineral reserves is 0.18% TCu for production years 2014 through 2022. The remaining years of the mine life, 2023 through 2025, utilize a 0.17% TCu cutoff.

PVM has operated a ROM dump leach on low-grade sulfide ores for many years. The final pit design and the mineral reserve have not included the ROM dump leach in the floating cone analysis or in the mineral reserve. Material that is incurred at ROM leach grade is reported in the mining plan, but it is not incorporated into the statement of mineral reserves in Table 15-2 and Table 15-3.

Table 15-2 and Table 15-3 summarize the mineral reserves at PVM remaining after 01 January 2014. Table 15-2 states the reserves in the imperial units that are incorporated throughout this report. For clarity and convenience when reporting in metric jurisdictions, Table 15-3 is the result of a direct conversion of imperial units to metric units.

All tonnage tabulations for mineral reserves and the mining plan are based on an in-place tonnage factor of 12.75 $ft^3$/ton (2.513 $Mt/m^3$).

The QP for the estimation of the mineral reserve was John Marek of IMC. Mr. Marek did not audit or verify the block model or the statement of mineral resources. IMC and Mr. Marek have relied on the expertise of Garth Kirkham, the QP for the model and mineral resource.

Metal price changes could materially change the estimated mineral reserves in either a positive or negative way. Pit wall instability could impact the remaining reserves in a negative way, depending on the time and area of occurrence. Capstone and their geotechnical contractors have addressed pit wall stability.

PVM is currently an operating mine with all permits in place to produce the mineral reserve as stated in this report. Therefore, the environmental risk at PVM is limited when compared with green field projects that are currently under development elsewhere in the United States.

**Table 15-2. Mineral Reserves – Pinto Valley Mine (imperial units) 01 January 2014**

| Classification | Mineral Reserves[1] | | | |
|---|---|---|---|---|
| | Cutoff %TCu[2] | Ore M tons | %TCu | %Mo[3] |
| Proven | 0.18–0.17 | 241 | 0.33 | 0.008 |
| Probable | 0.18–0.17 | 15 | 0.33 | 0.008 |
| Proven + Probable | 0.18–0.17 | 256 | 0.33 | 0.008 |

[1]The mineral reserves stated above are limited to match the tailings storage capacity.
[2]Total copper in percent by weight.
[3]Molybdenum metal in percent by weight.

**Table 15-3. Mineral Reserves – Pinto Valley Mine (metric units) 01 January 2014**

| Classification | Mineral Reserves | | | |
|---|---|---|---|---|
| | Cutoff %TCu | Ore Mt | %TCu | %Mo |
| Proven | 0.18–0.17 | 219 | 0.33 | 0.008 |
| Probable | 0.18–0.17 | 13 | 0.33 | 0.008 |
| Proven + Probable | 0.18–0.17 | 232 | 0.33 | 0.008 |

# 16.0 MINING METHODS

PVM is planned to be continued as an open-pit mine. This mine plan is based on the block model developed by Garth Kirkham, as described in Section 14. The mine plan that is presented in this section was developed by IMC, with John Marek acting as the QP.

PVM is currently processing copper-bearing sulfide ore through a mill and flotation circuit to produce copper and molybdenum concentrates. This mine plan extends the PVM operational life to 2026. This mine plan respects the property boundaries and the current permitted tailings storage capacity of just over 250 M tons. There is additional material that could be mined if those constraints were removed.

Ore production to the mill is planned to continue at 55,100 tpd (20,117 ktons/yr) for the first 2 years, followed by an increase to 57,300 tpd (20,922 ktons/yr) for the remaining life of the mine plan. The mine production schedule was developed with the goal of maintaining mill feed and maximizing the project return on investment.

The total material rate is tied to current mine equipment productivity, with an intent to expand the mine fleet for increased material movement. The planned total material movement is 68,170 tpd (24,882 ktons/yr) the first year, 97,584 tpd (35,618 ktons/yr) the second year, and finally to 127,671 tpd (46,600 ktons/yr) for 4 years until the ore is released from the final pushback and the required material movement drops off. The mine is scheduled to operate 365 days/yr with two 12-hour shifts per day. The current bench height is being maintained at 45 ft.

The current mining fleet consists of the following equipment.

- 2 Blasthole Drills
- 3 Cat 994 Front-End Loaders
- 15 Cat 789 Haul Trucks

Appropriate auxiliary support equipment is included with this fleet.

A hydraulic shovel of 35 $yd^3$ capacity is planned for acquisition during mid-2015 so that it is active during the last quarter of 2015 through 2025.

The mine plan was developed with a phase approach. The phase designs, mine schedule, and mine equipment requirements are summarized in this section.

Table 16-1 and Figure 16-1 summarize the mine production schedule.

**Table 16-1. Pinto Valley Mine Production Schedule (imperial units)**

| Year | | Mill Ore | | | | Low-Grade Stockpile, +0.18% TCu | | | Dump Leach, +0.10% | | Waste | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Cutoff %TCu | Ore ktons | %TCu | %Mo | LG Stkp* ktons | %TCu | %Mo | Leach ktons | %TCu | Material ktons | Material ktons |
| 1 | 2014 | 0.22 | 20,117 | 0.39 | 0.008 | 1,202 | 0.20 | 0.007 | 2,408 | 0.15 | 1,155 | 24,882 |
| 2 | 2015 | 0.22 | 20,117 | 0.34 | 0.008 | 4,735 | 0.20 | 0.007 | 5,991 | 0.14 | 4,775 | 35,618 |
| 3 | 2016 | 0.20 | 20,922 | 0.38 | 0.008 | 1,994 | 0.19 | 0.004 | 10,038 | 0.15 | 13,646 | 46,600 |
| 4 | 2017 | 0.20 | 20,922 | 0.33 | 0.008 | 934 | 0.19 | 0.008 | 10,076 | 0.13 | 14,668 | 46,600 |
| 5 | 2018 | 0.20 | 20,922 | 0.33 | 0.009 | 4,168 | 0.19 | 0.007 | 9,423 | 0.15 | 12,087 | 46,600 |
| 6 | 2019 | 0.19 | 20,922 | 0.32 | 0.010 | 2,948 | 0.18 | 0.006 | 16,789 | 0.15 | 4,807 | 45,466 |
| 7 | 2020 | 0.20 | 20,922 | 0.33 | 0.011 | 2,590 | 0.19 | 0.005 | 8,108 | 0.14 | 869 | 32,489 |
| 8 | 2021 | 0.20 | 20,922 | 0.33 | 0.008 | 1,964 | 0.19 | 0.005 | 4,335 | 0.15 | 672 | 27,893 |
| 9 | 2022 | 0.18 | 20,922 | 0.32 | 0.006 | | | | 4,129 | 0.14 | 341 | 25,392 |
| 10 | 2023 | 0.17 | 20,922 | 0.31 | 0.006 | | | | 5,307 | 0.13 | 1,077 | 27,306 |
| 11 | 2024 | 0.17 | 20,922 | 0.35 | 0.007 | | | | 1,245 | 0.14 | | 22,167 |
| 12 | 2025 | 0.17 | 6,911 | 0.34 | 0.007 | | | | | | | 6,911 |
| **Total** | | | **235,443** | **0.34** | **0.008** | **20,535** | **0.19** | **0.006** | **77,849** | **0.14** | **54,097** | **387,924** |

*Low-Grade Stockpile

**Figure 16-1. Pinto Valley Mine Production Schedule**

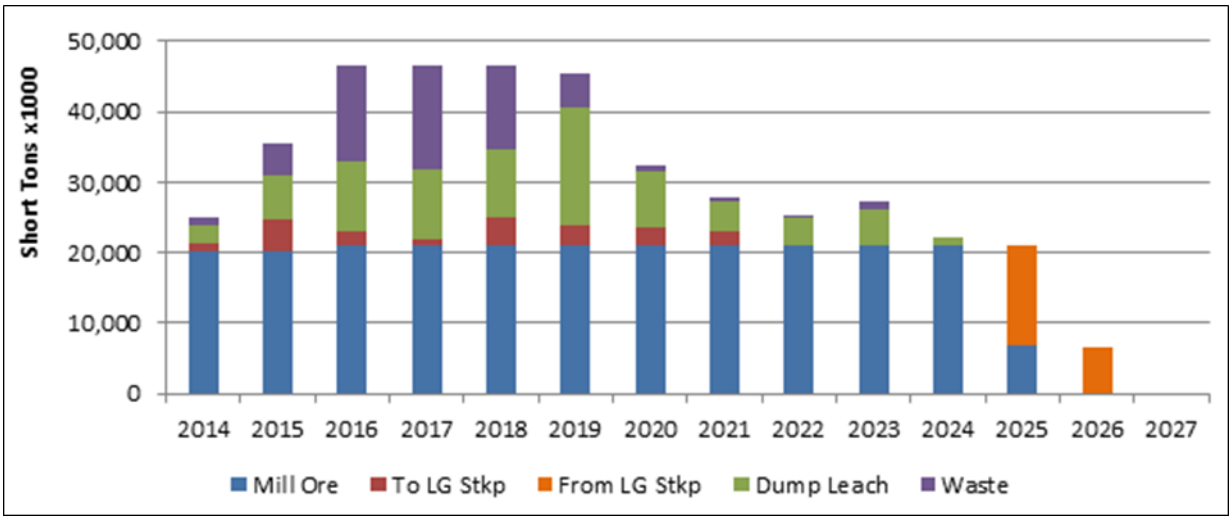


Table 16-2 summarizes the mill feed schedule.

**Table 16-2. Pinto Valley Mill Feed Schedule (imperial units)**

| Year | | Mill Ore | | | |
|---|---|---|---|---|---|
| | | Cutoff %TCu | Ore ktons | %TCu | %Mo |
| 1 | 2014 | 0.22 | 20,117 | 0.39 | 0.008 |
| 2 | 2015 | 0.22 | 20,117 | 0.34 | 0.008 |
| 3 | 2016 | 0.20 | 20,922 | 0.38 | 0.008 |
| 4 | 2017 | 0.20 | 20,922 | 0.33 | 0.008 |
| 5 | 2018 | 0.20 | 20,922 | 0.33 | 0.009 |
| 6 | 2019 | 0.19 | 20,922 | 0.32 | 0.010 |
| 7 | 2020 | 0.20 | 20,922 | 0.33 | 0.011 |
| 8 | 2021 | 0.20 | 20,922 | 0.33 | 0.008 |
| 9 | 2022 | 0.18 | 20,922 | 0.32 | 0.006 |
| 10 | 2023 | 0.17 | 20,922 | 0.31 | 0.006 |
| 11 | 2024 | 0.17 | 20,922 | 0.35 | 0.007 |
| 12 | 2025 | 0.17–0.18* | 20,922 | 0.24 | 0.006 |
| 13 | 2026 | 0.18* | 6,524 | 0.19 | 0.006 |
| **Total** | | | **255,978** | **0.33** | **0.008** |

## 16.1 Phase Design

PVM has been operated on and off since the 1970s. As a result, there is an extensive open pit at the site with developed access roads. Those access roads were reasonable and appropriate at the time they were designed and implemented, but they are not necessarily in the most optimal location for the continued production at PVM.

There are areas of the existing PVM pit that have had slope stability issues over time. Specific areas are the south end of the pit in the Pinal schist rock type and the northeast side of the pit near the Bummer Fault. Designing phase expansions in these areas must consider the geotechnical impacts and practical constraints they impose.

The phase designs were guided by the results of the floating cones (summarized in Section 15) and a declining cutoff grade. However, far more important in the design of the mine phases are the challenges of working with the existing access pattern and the geotechnical constraints on the design.

PVM planners have developed two major phase designs in the bottom of the existing pit. In addition to the pit bottom phases, IMC has designed two pushbacks that expand the pit to the 12-year life that is shown in Table 16-1. Those two pushbacks are subdivided with two additional sub-phases. The west-side pit floor was segregated into two parts in order to more correctly reflect the planned material movement of 2014 as a sub-phase.

In total, there are seven phases or sub-phase designs that were used as input to the development of the PVM schedule. The following items summarize the mine design parameters.

- Haul Road Width: 115 ft
- Haul Road Grade: 10% Maximum
- Minimum Width Between Pushbacks: 300 ft Nominal

Inter-ramp slope angles are summarized in the next subsection that was prepared by geotechnical contractor CNI. It should be noted that all of the phase designs avoid developing slopes in the Pinal schist. That unit has been problematic regarding slope stability in the past.

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

Figure 16-2 through Figure 16-4 illustrate the relative position of the phase designs on the 3905, 3770, and 3095 elevations. Phases PB2 and PB4 were designed by PVM staff and deepen the current pit floor. These are illustrated in Figure 16-4 on the 3095 bench.

**Figure 16-2. Relative Location of Phase Designs on the 3905 Bench**

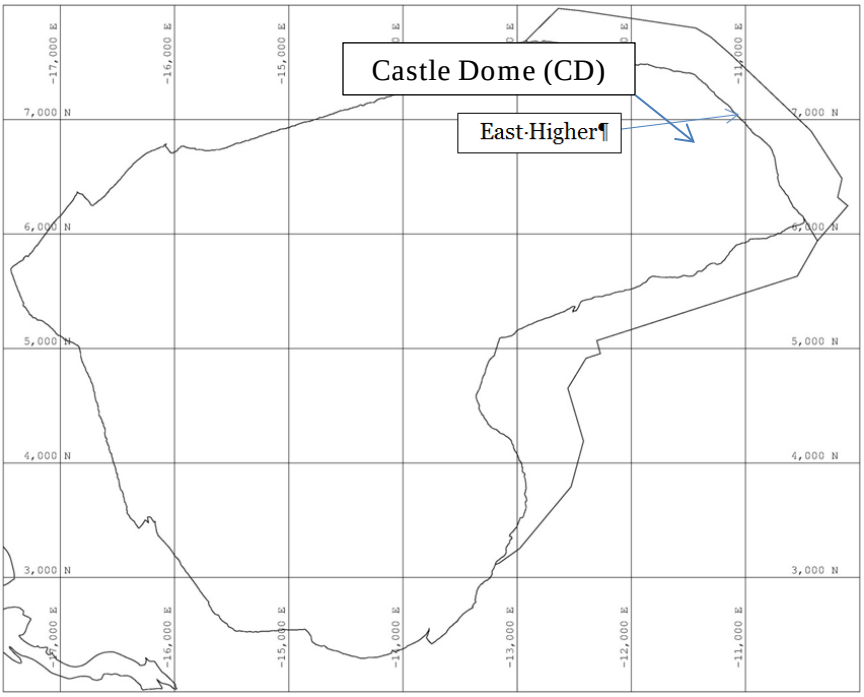



**Figure 16-3. Relative Location of Phase Designs on the 3770 Bench**

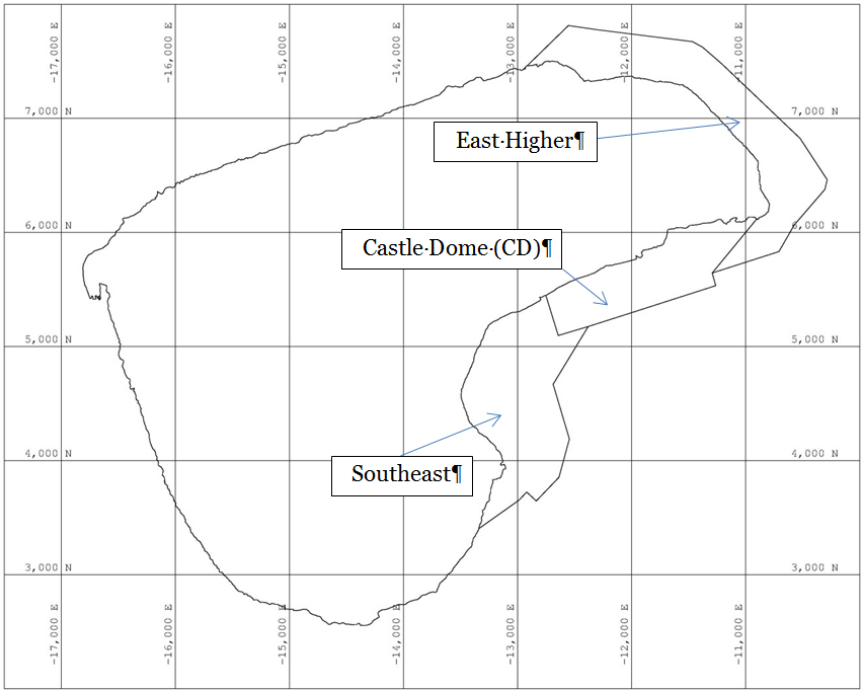
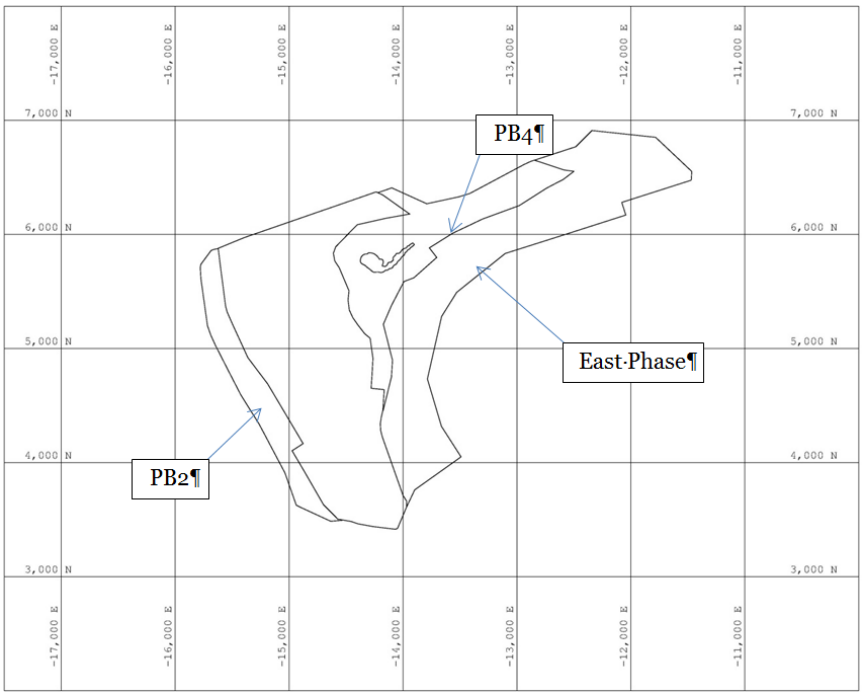


**Figure 16-4. Relative Location of Phase Designs on the 3095 Bench**




The Castle Dome Phase and the combined sub-phase called Southeast were designed by IMC with the following goals.

- Provide early access to ore in order to provide more time to remove waste from the East Phase that follows.
- Develop access from the southeast corner of the pit that will connect through the East Phase to deliver waste and leach to the north storage areas.
- Connect access to the existing south wall road to the crusher so that future deep East Phase ore can be delivered to the crusher.
- Develop the access in Items 2 and 3 without crossing the property boundary.

The property boundary in the southeast corner was respected by pushing in a temporary fill road. Currently, there is a road on the east side of the pit. Improvements to that road for mine haul trucks require that it be straightened and that the gradient be made consistent by lowering the south end of the road.

The Southeast sub-phase was developed as part of the Castle Dome Phase in order to provide access to more ore and allow some flexibility in the schedule process to develop ore when required.

The Castle Dome and Southeast Phases are followed closely in time by the East Phase. The East Phase develops ore near the surface, but requires substantial waste movement before sustained ore release is achieved. East-side access to the dumps is maintained through the East Phase.

The north corner of the East Phase is developed to effectively deal with the Bummer Fault zone with a "notch" design.

The East Phase has been sub-phased into an upper and lower area to facilitate mine scheduling. The relative timing between the Castle Dome Phase and the East Phase must be considered to maintain efficient access through the southeast corner at all times.

## 16.2 Slope Stability

CNI-recommended slope angles for the pit have been based on stability analyses and historical information. The pit slope design process involves determining the appropriate bench geometry and inter-ramp slope angles, as well as checking wall stability.

- To define safe catch bench widths to protect against rockfall, a catch bench design analysis based on rock fabric was performed with CNI's Backbreak program. Based on evaluation of the existing bench performance, a catch bench reliability of 70% was selected. To achieve this reliability in some sectors, controlled wall blasting techniques will be used. The bench scale design limits slope angles to 48° in the Ruin granite.
- To evaluate the potential of overall slope failures involving major faults or weak rock mass, the available geologic, hydrologic, and geotechnical data from PVM were assessed. Based on previous experience, a target factor of safety (FOS) of greater than 1.2 is achieved for overall stability.
- Mining of slopes in the Pinal schist is not planned in the design. For reference, slopes in the Pinal schist have a history of displacement and are at limit equilibrium FOS = 1.0. The PVM operation has safely maintained operational activities with displacing slopes using a range of mitigating controls based on observation and a slope monitoring program.
- Slope depressurization is recommended for operational considerations and is required for stability of the Whitetail conglomerate and the granodiorite in the west wall.
- 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.

CNI has reviewed the pit design for this study, and it meets the design criteria. Table 16-3 lists the inter-ramp angles. Figure 16-5 illustrates the geotechnical domains and slope design.

**Table 16-3. Inter-Ramp Slope Angles**

| Sector | 45 ft Bench | 90 ft Bench |
|---|---|---|
| Bummer Fault Zone | 36° | - |
| | 42° | 42° |
| Ruin Granite | - | 48° |
| Whitetail Southwest | 35° | - |
| Whitetail Northeast | 40° | - |
| Shear Zone | 32° | - |
| Granodiorite Shear Zone | 32° | - |
| Pinal Schist | 27° | - |
| Pinal Schist Shear Zone | 24° | - |
| Granodiorite | 46° | 46° |
| Limestone West | 40° | - |
| Diabase | 40° | - |

### 16.2.1 Geologic Units / Geotechnical Domains

Geotechnical domains are based on lithologies from the geologic model. The following data sets were used to develop rock mass properties based on the lithologies and to analyze pit slope stability.

#### Geomechanical Data

Eighteen oriented core holes have been drilled. Ten were drilled in 2007 and eight additional holes were completed in 2012. Joint orientations from these programs are included along with various cell mapping programs from 2006. Cell mapping provides structure orientation, length, and spacing data for analyzing the bench scale design. A total of 72 cells have been mapped since 2006. Geomechanical data, including recovery and RQD, have been collected on 175 core holes completed since 2007. Geomechanical drill data include recovery, RQD, percent broken core, percent whole core, percent rubble, and longest piece. These data were used to develop an RQD model which is used for rock-mass classifications and rock-mass strength estimates.

#### Rock Strength Testing

Rock strength testing has been carried out on all units in the mine. Intact uniaxial compression and triaxial compression tests have been completed on the Ruin granite, diabase, limestone, Gila and Whitetail conglomerates, granodiorite, and Pinal schist. Small-scale, direct-shear testing of fractures has been completed on all of these units. In addition, direct-shear testing of highly weathered Pinal schist, Whitetail conglomerate, diabase, and granodiorite has been completed to determine residual strengths; these were then compared with shear strengths obtained from analysis of existing slopes.

#### Hydrogeology

Slope stability was analyzed with a groundwater model developed from well data and horizontal drain data. Pore pressures are input into the overall stability analysis.

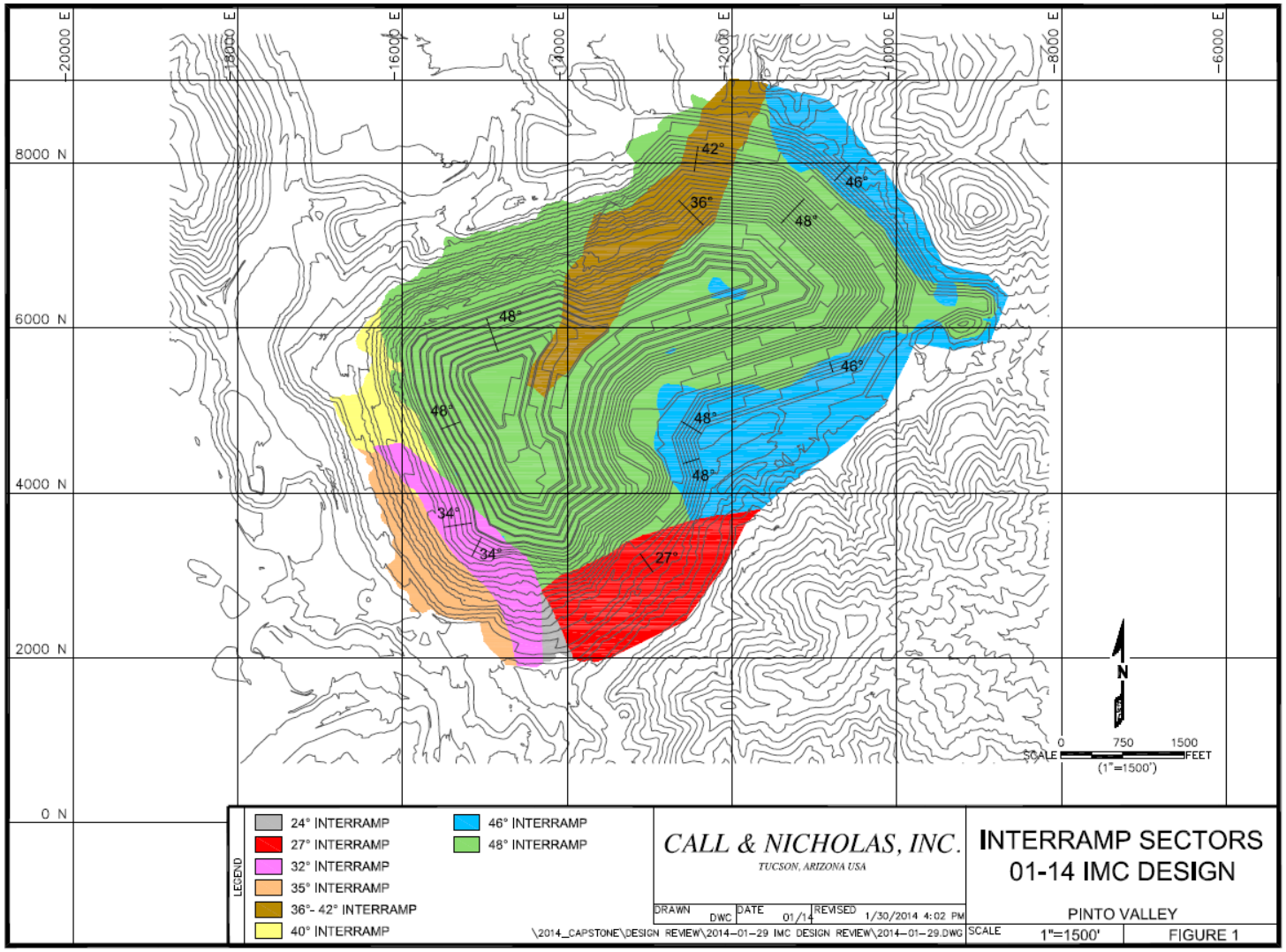
### 16.2.2 Analysis of Pit Slope Design, Bench, and Inter-Ramp Angle

CNI uses a probabilistic approach to determine the bench face angle distribution for a given reliability. This distribution is directly related to the amount of backbreak calculated from the structure length, spacing, and orientation, as well as joint shear strength. Backbreak is defined as the distance behind the crest that a vertical bench sloughs into the pit along daylighted geologic structures; therefore, bench face angles calculated in the analysis are defined as the angle from the bench toe to the bench crest after backbreak has occurred.

#### Overall Wall Stability – Inter-Ramp and Overall

SLOPE/W, a two-dimensional, limit-equilibrium slope stability analysis program, was used for both inter-ramp and overall slope stability analyses. Spencer's Method of Slices was used to calculate FOSs. FOS is defined as resisting force divided by the driving force.

**Figure 16-5. Inter-Ramp Slope Angle Sectors**



## 16.3 Mine Production Schedule

The mine production schedule was developed to deliver the required ore to the process plant and maximize the project return on investment. Multiple mine production schedules were developed that analyzed alternative cutoff grade strategies versus mine total material movement. Total material rates were tied to the size and number of loading units so that the final selected schedule would provide efficient use of the capital equipment employed.

The multiple schedules were evaluated on an NPV basis to establish the mine plan and the corresponding mineral reserve (Section 15). The best overall production schedule on an economic and practical basis was selected and is summarized in Table 16-1.

Several alternatives were evaluated before finalizing the schedule shown in Table 16-1.

- An alternative to the PB2 pit bottom design was considered that maintained dual access to the pit bottom at all times during the mine life. That alternative was rejected because the west-side road in PB2 contains substantial high-grade ore. Saving that ore and providing dual access is a conservative approach with minimal risk. However, producing that ore early in the mine plan provides substantial economic benefit to the project. On that basis, a single-access approach to the pit bottom was adopted.
- Alternative equipment units were considered with corresponding different total material ramp-up rates. Adding one loader for a total of four units in 2015 was compared with the final case that has two hydraulic shovels and one operating loader by 2016. The shovel-loader case was chosen due to the added loading efficiency and productivity of the hydraulic shovel.

PVM is currently operating, so there is no preproduction stripping. Minor road pioneering will be required to begin the extraction of the Castle Dome and East Phases on the east end of the pit.

The cutoff grade for the mine schedule is based on total copper cutoff grades. The mill ore feed cutoff grade changes over time, starting at 0.22% TCu and reducing to 0.17% TCu.

Low-grade material above a cutoff of 0.18% TCu and less than the mill feed cutoff will be stored in two stockpile storage facilities that are located to the southeast of the pit and west of the pit. The low-grade material is currently planned to be processed after the mine extraction of ore is completed.

Historically, PVM has placed low-grade sulfide mineralization above 0.10% TCu on a dump area that is northwest of the pit for leaching with sulfuric acid. That process will continue. IMC has reported the material in that grade range and has delivered it to the dump leach area. Material with grade less than 0.10% TCu is considered as waste and is delivered to the northeast of the current pit.

## 16.4 Waste, Waste Leach, and Low-Grade Storage

There are three types of storage material resulting from the PVM plan.

- Low-Grade Ore Stockpile, +0.18% TCu to Mill Cutoff
  - This material is planned to be re-handled and processed after the pit ore is exhausted.
- Dump Leach, +0.10% TCu to Stockpile Cutoff
  - This material is delivered to the Low-Grade Ore Leaching Pile northwest of the pit for dump leaching.
- Waste Storage Less than 0.10% TCu
  - This material is delivered to the Gold Gulch Dump, a waste storage area northeast of the pit.

Waste rock below the 0.18% stockpile cutoff that is high in clay content or calcareous will be delivered to the Gold Gulch Dump, as the material is not conducive to leaching of copper.

The annual mine schedule drawings at the end of the section illustrate the location and size development of the storage facilities. Two separate low-grade stockpiles have been designed: (1) Castle Dome Marginal Dump and (2) North Barn Marginal Dump. Material will be allocated to the two low-grade dumps based on haul distance.

The waste storage area, dump leach, and Castle Dome Marginal Dump are planned at overall angles of 29°. The lifts' heights are 30 ft high, which are dumped at angle of repose (37°) with 15 ft setbacks left every lift. The North Barn Marginal Dump is planned at angle of repose since it is intended to be re-mined and processed through the mill. The Castle Dome Marginal Dump is also planned for re-mining and processing, but geotechnical considerations have limited it to the 29° overall angle.

A consistent tonnage factor of 16.58 loose $ft^3$/ton (swell factor of 1.3) has been used to calculate all storage capacities.

## 16.5 Mine Equipment Requirements

IMC calculated the requirements for major mine equipment at PVM. This information has been combined with input from the mine site and other contractors to establish the overall mine equipment list.

IMC has addressed the requirements for the following major equipment units.

- Blasthole Drills
- Loading Equipment
- Haul Trucks

The mine work schedule for the equipment requirement calculations was as follows.

- 2 Shifts/Day
- 12 Hours/Shift
- 365 Days/Year
- 730 Scheduled Shifts/Year

IMC calculates availability and use of availability as follows.

- Availability = $\frac{\text{[number of shifts the machine is available to work]}}{730}$

- Use of Availability = $\frac{\text{[number of shifts the machine actually works]}}{\text{[available shifts]}}$

The following description summarizes the calculations completed by IMC to set the required numbers of each of the three types of equipment. The following procedures estimate the unit production capacity in one shift of 12 hours. That productivity assumes that the machine works the entire shift with normal production delays and efficiency.

### 16.5.1 Blasthole Drills

Blasthole drill productivity was calculated from historic penetration rates and powder factors that have been applied at PVM and compared against IMC experience with other projects. PVM uses relatively small-diameter drill holes for bench heights of this size and a relatively high powder factor.

The purpose of the small hole diameter and the high powder factor is to achieve more breakage in the pit to improve the loading productivity of the front-end loaders and to improve the throughput rate of ore through the primary crusher.

The top part of Figure 16-6 summarizes the estimated drill productivity per 12-hour shift, assuming that the machine is available and manned for the entire shift. The lower portion of the figure summarizes the drill fleet requirements and the number of required operating shifts to achieve the planned production rate at PVM.

## Figure 16-6. Blasthole Drill Productivity and Requirements, 9⅞ Bit (imperial units)

**Productivity**

| | Units | 9 7/8 Inch ANFO | 9 7/8 Inch Slurry |
|---|---|---|---|
| Hole Diameter | inches | 9.875 | 9.875 |
| Bench Height | ft | 45.0 | 45.0 |
| Subgrade | ft | 5.0 | 5.0 |
| Powder Loaded | SG* | 0.85 | 1.20 |
| Powder Factor | lb/ton | 0.64 | 0.64 |
| Bank Density | $ft^3$/ton | 12.75 | 12.75 |
| Powder Load | lb/ft | 28.21 | 39.83 |
| Powder Height | ft | 38.68 | 27.40 |
| Powder per Hole | lb/ton | 1,091.2 | 1,091.2 |
| Stemming Height | ft | 11.32 | 22.60 |
| Rock Mass per Hole | tons | 1,705 | 1,705 |
| Spacing and Burden | ft | 22.00 | 22.00 |
| Drilling Rate, No Move Time | ft/hr | 96.0 | 96.0 |
| Shift Drill Time, Bit on Ground | hr | 9.19 | 9.19 |
| Shift Total Drilling | ft | 881.89 | 881.89 |
| Shift Production | tons | 30,072 | 30,072 |

**Requirements**

| | Year | Ore ktons | Ore Shifts | Low-Grade to Stockpile ktons | Low-Grade to Stockpile Shifts | Leach + Waste ktons | Leach + Waste Shifts | Redrill @5% ktons | Redrill @5% Shifts | Total with Redrill ktons | Total with Redrill Shifts | Shifts/ Year | Fractional Fleet | Rounded Fleet |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 2014 | 20,117 | 669 | 1,202 | 40 | 3,563 | 118 | 1,244 | 41 | 26,126 | 868 | 730 | 1.56 | 2 |
| 2 | 2015 | 20,117 | 669 | 4,735 | 157 | 10,766 | 358 | 1,781 | 59 | 37,399 | 1,243 | 730 | 2.24 | 3 |
| 3 | 2016 | 20,922 | 696 | 1,994 | 66 | 23,684 | 788 | 2,330 | 77 | 48,930 | 1,627 | 730 | 2.93 | 3 |
| 4 | 2017 | 20,922 | 696 | 934 | 31 | 24,744 | 823 | 2,330 | 77 | 48,930 | 1,627 | 730 | 2.93 | 3 |
| 5 | 2018 | 20,922 | 696 | 4,168 | 139 | 21,510 | 715 | 2,330 | 77 | 48,930 | 1,627 | 730 | 2.93 | 3 |
| 6 | 2019 | 20,922 | 696 | 2,948 | 98 | 21,596 | 718 | 2,273 | 76 | 47,739 | 1,588 | 730 | 2.86 | 3 |
| 7 | 2020 | 20,922 | 696 | 2,590 | 86 | 8,977 | 299 | 1,624 | 54 | 34,113 | 1,135 | 730 | 2.05 | 3 |
| 8 | 2021 | 20,922 | 696 | 1,964 | 65 | 5,007 | 167 | 1,395 | 46 | 29,288 | 974 | 730 | 1.76 | 2 |
| 9 | 2022 | 20,922 | 696 | | 0 | 4,470 | 149 | 1,270 | 42 | 26,662 | 887 | 730 | 1.60 | 2 |
| 10 | 2023 | 20,922 | 696 | | 0 | 6,384 | 212 | 1,365 | 45 | 28,671 | 953 | 730 | 1.72 | 2 |
| 11 | 2024 | 20,922 | 696 | | 0 | 1,245 | 41 | 1,108 | 37 | 23,275 | 774 | 730 | 1.40 | 2 |
| 12 | 2025 | 6,911 | 230 | | 0 | | 0 | 346 | 12 | 7,257 | 242 | 300 | 1.06 | 2 |
| 13 | 2026 | | | | | | | | | | | | | |
| Total | | 235,443 | 7,832 | 20,535 | 682 | 131,946 | 4,388 | 19,396 | 643 | 407,320 | 13,545 | 8,330 | | |

| | |
|---|---|
| Shift Production | 30,072 tons |
| Mechanical Availability | 80% |
| Utilization of Availability | 95% |
| Maximum Utilization | 76% |

*Specific gravity

### 16.5.2 Loading Equipment

PVM currently has three Cat 994 front-end loaders as their loading fleet. Alternative production schedules were developed to evaluate alternative combinations of loading equipment. The alternatives were as follows.

1. Add a fourth Cat 994 in 2015, and maintain a total material movement of 43,400 ktons of total material per year (Four-Loader Case).
2. Add a 35 $yd^3$ hydraulic shovel in mid-2015 for planned production during the last quarter of 2015 through 2025. Schedule only two of the current three Cat 994 loaders (One-Shovel, Two-Loader Case). The total material rate increases to 46,600 ktons/yr in 2016.
3. Add two 35 $yd^3$ hydraulic shovels, and operate one of the Cat 994 loaders. One shovel is planned for operation in the last quarter of 2015, and the second is planned for operation at the beginning of 2016 (Two-Shovel, One-Loader Case).

A quick comparison between the three cases indicates that all three cases are similar economically. The case with two shovels and one loader (Case 3 above) was selected due to the productivity of the hydraulic shovels and the improved loading time for haul trucks when loaded by the shovel.

There is potential for PVM to continue beyond the life of this current mine plan. Given that potential, the hydraulic shovel is a better selection for the long term.

Figure 16-7 summarizes the productivity of both the 35 $yd^3$ shovel and the 22.5 $yd^3$ loaders on a 12-hour shift basis. The figure also summarizes the fleet requirements and the number of operating shifts that are required per year.

## Figure 16-7. Loading Equipment Productivity and Requirements (imperial units)

**Productivity**

| | Units | Hydraulic Shovel | Cat 994 |
|---|---|---|---|
| Bucket Capacity | $yd^3$ | 35.0 | 22.5 |
| Dry Bank Density | $ft^3$/ton | 12.75 | 12.75 |
| Swell | % | 40.0% | 40.0% |
| Moisture Content | % | 3.00% | 3.00% |
| Bucket Fill Factor | none | 0.95 | 0.90 |
| Tons/Pass (Dry) | tons | 50.29 | 30.63 |
| Tons/Pass (Wet) | tons | 51.80 | 31.55 |
| Shovel Cycle Time/Pass | min | 0.50 | 0.75 |
| Truck Spot Time | min | 0.50 | 0.50 |
| Passes/Truck | passes | 4.0 | 7.0 |
| Tons Per Truck (Dry) | tons | 192.5 | 192.5 |
| Tons Per Truck (Wet) | tons | 198.3 | 198.3 |
| Payload Fill Factor | none | 1.00 | 1.00 |
| Last Bucket | none | partial | partial |
| Total Time/Truck | min | 2.50 | 5.75 |
| Shift Loading Time | min | 550 | 550 |
| Truck Loads/Shift | loads | 220.00 | 95.65 |
| Shift Production (Dry) | tons | 42,347 | 18,412 |

**Requirements**

| Year | | Material Loaded by 994 Loader, 22.5 $yd^3$ | | | | | | Material Loaded by Hydraulic Shovel, 35 $yd^3$ | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 994 Loading | | Shifts/ | Fractional | Actual | Fleet | Hydraulic Shovel | | Shifts/ | Fractional | Actual | Fleet |
| | | ktons | Shifts | Year | Fleet | Fleet | Utilization % | ktons | Shifts | Year | Fleet | Fleet | Utilization % |
| 1 | 2014 | 24,882 | 1,351 | 730 | 2.29 | 3 | 62% | | | | | | |
| 2 | 2015 | 29,563 | 1,606 | 730 | 2.72 | 3 | 73% | 6,055 | 143 | 183 | 0.97 | 1 | 78% |
| 3 | 2016 | 4,660 | 253 | 730 | 0.43 | 1 | 35% | 41,940 | 990 | 730 | 1.68 | 2 | 68% |
| 4 | 2017 | 4,660 | 253 | 730 | 0.43 | 1 | 35% | 41,940 | 990 | 730 | 1.68 | 2 | 68% |
| 5 | 2018 | 4,660 | 253 | 730 | 0.43 | 1 | 35% | 41,940 | 990 | 730 | 1.68 | 2 | 68% |
| 6 | 2019 | 4,547 | 247 | 730 | 0.42 | 1 | 34% | 40,919 | 966 | 730 | 1.64 | 2 | 66% |
| 7 | 2020 | 3,249 | 176 | 730 | 0.30 | 1 | 24% | 29,240 | 690 | 730 | 1.17 | 2 | 47% |
| 8 | 2021 | 2,789 | 151 | 730 | 0.26 | 1 | 21% | 25,104 | 593 | 730 | 1.01 | 1 | 81% |
| 9 | 2022 | 2,539 | 138 | 730 | 0.23 | 1 | 19% | 22,853 | 540 | 730 | 0.92 | 1 | 74% |
| 10 | 2023 | 2,731 | 148 | 730 | 0.25 | 1 | 20% | 24,575 | 580 | 730 | 0.98 | 1 | 79% |
| 11 | 2024 | 2,217 | 120 | 730 | 0.20 | 1 | 16% | 19,950 | 471 | 730 | 0.80 | 1 | 65% |
| 12 | 2025 | 2,092 | 114 | 730 | 0.19 | 1 | 16% | 18,830 | 445 | 730 | 0.75 | 1 | 61% |
| 13 | 2026 | 652 | 35 | 300 | 0.15 | 1 | 12% | 5,872 | 139 | 300 | 0.57 | 1 | 46% |
| Total | | 89,241 | 4,847 | 9,060 | | | | 319,219 | 7,539 | 7,783 | | | |
| Shift Production | | | | 18,412 tons/shift | | | | | | 42,347 tons/shift | | | |
| Mechanical Availability | | | | 85.0% | | | | | | 85.0% | | | |
| Utilization of Availability | | | | 95.0% | | | | | | 95.0% | | | |
| Maximum Utilization | | | | 80.8% | | | | | | 80.8% | | | |

### 16.5.3 Haul Trucks

PVM currently operates 15 Cat 789 haul trucks. The nominal dry payload for the 789 is 192.5 tons. Estimated moisture is 3%, so the actual truck payload is 198.3 tons.

Haul profiles were measured for each material type to each destination for each year of the mine plan. Those haul profiles reflect the grade of each road segment and notes for the occurrences of switchbacks.

Truck productivities were calculated using haul time simulation. The haul times were calculated over each of the measured haul profiles incorporating acceleration and braking. Fixed times were applied differently if the truck was loaded by the 994 (5.75 + 2.0 minutes) or the hydraulic shovel (2.5 + 2.0 minutes).

The productivities were applied based on the tonnage allocation to both shovels and loaders to determine the overall truck fleet requirements.

Table 16-4 summarizes the truck fleet and operating shift requirements for the Cat 789 haul trucks.

**Table 16-4. Truck Fleet Requirements and Operating Shift Requirements from Simulation (imperial units)**

| Year | Loaded with Hydraulic Shovel | | | | | | Loaded with Cat 994 | | | | | Truck Fleet Requirements | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Total ktons | Mill Ore Shifts | Low-Grade Shifts | Stockpile Mining Shifts | Leach Shifts | Waste Shifts | Mill Ore Shifts | Low-Grade Shifts | Stockpile Mining Shifts | Leach Shifts | Waste Shifts | Total Shifts | Shifts/ Year | Fractional Fleet | Rounded Fleet | Fleet Utilization |
| 1 2014 | 24,882 | | | | | | 6,150 | 371 | | 825 | 576 | 7,923 | 730 | 13.44 | 14 | 78% |
| 2 2015 | 35,618 | 747 | 103 | | 180 | 170 | 4,158 | 621 | | 1,029 | 954 | 7,962 | 730 | 13.51 | 14 | 78% |
| 3 2016 | 46,600 | 4,722 | 348 | | 1,173 | 1,775 | 589 | 45 | | 161 | 239 | 9,052 | 730 | 15.36 | 16 | 77% |
| 4 2017 | 46,600 | 5,180 | 219 | | 1,808 | 2,716 | 640 | 27 | | 232 | 347 | 11,169 | 730 | 18.95 | 19 | 81% |
| 5 2018 | 46,600 | 4,038 | 684 | | 1,776 | 2,904 | 513 | 89 | | 226 | 360 | 10,590 | 730 | 17.97 | 18 | 81% |
| 6 2019 | 45,466 | 3,491 | 518 | | 3,646 | 1,348 | 452 | 67 | | 457 | 165 | 10,143 | 730 | 17.21 | 18 | 77% |
| 7 2020 | 32,489 | 3,936 | 653 | | 2,071 | 294 | 502 | 81 | | 255 | 35 | 7,827 | 730 | 13.28 | 14 | 77% |
| 8 2021 | 27,893 | 4,504 | 593 | | 1,278 | 241 | 565 | 72 | | 155 | 29 | 7,437 | 730 | 12.62 | 13 | 78% |
| 9 2022 | 25,392 | 5,078 | | | 1,324 | 135 | 628 | | | 160 | 16 | 7,341 | 730 | 12.45 | 13 | 77% |
| 10 2023 | 27,306 | 5,768 | | | 1,894 | 448 | 705 | | | 227 | 53 | 9,095 | 730 | 15.43 | 16 | 78% |
| 11 2024 | 22,167 | 6,680 | | | 469 | | 806 | | | 56 | | 8,011 | 730 | 13.59 | 14 | 78% |
| 12 2025 | 20,922 | 2,480 | | 102 | | | 297 | | 1,215 | | | 4,094 | 730 | 6.94 | 7 | 80% |
| 13 2026 | 6,524 | | | 106 | | | | | 1,223 | | | 1,329 | 300 | 5.49 | 6 | 74% |
| Total | 408,459 | 46,621 | 3,119 | 0 | 15,618 | 10,034 | 16,005 | 1,372 | 2,438 | 3,783 | 2,774 | 101,971 | 9,060 | | | |

| | |
|---|---|
| 85.0% | Mechanical Availability |
| 95.0% | Utilization of Availability |
| 80.8% | Maximum Utilization |

## 16.6 Mine Plan Drawings

Figure 16-8 through Figure 16-15 illustrate selected years of the mine plan. The pit progress reflects the tonnages reported in Table 16-1. The low-grade, waste, and leach storage area progress is also shown in the drawings.

**Figure 16-8. Pit and Dump Configuration – End of 2014**

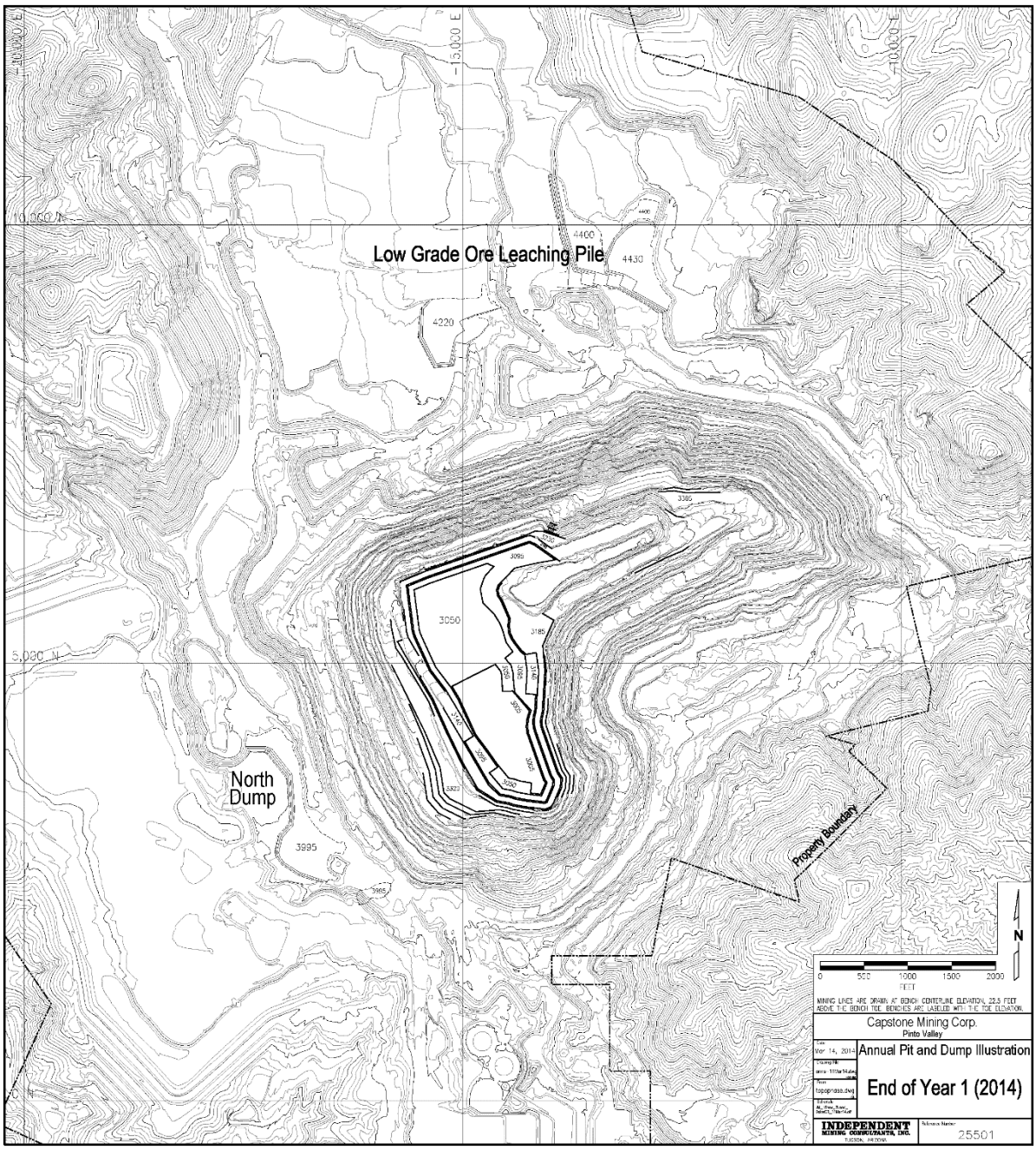


**Figure 16-9. Pit and Dump Configuration – End of 2015**

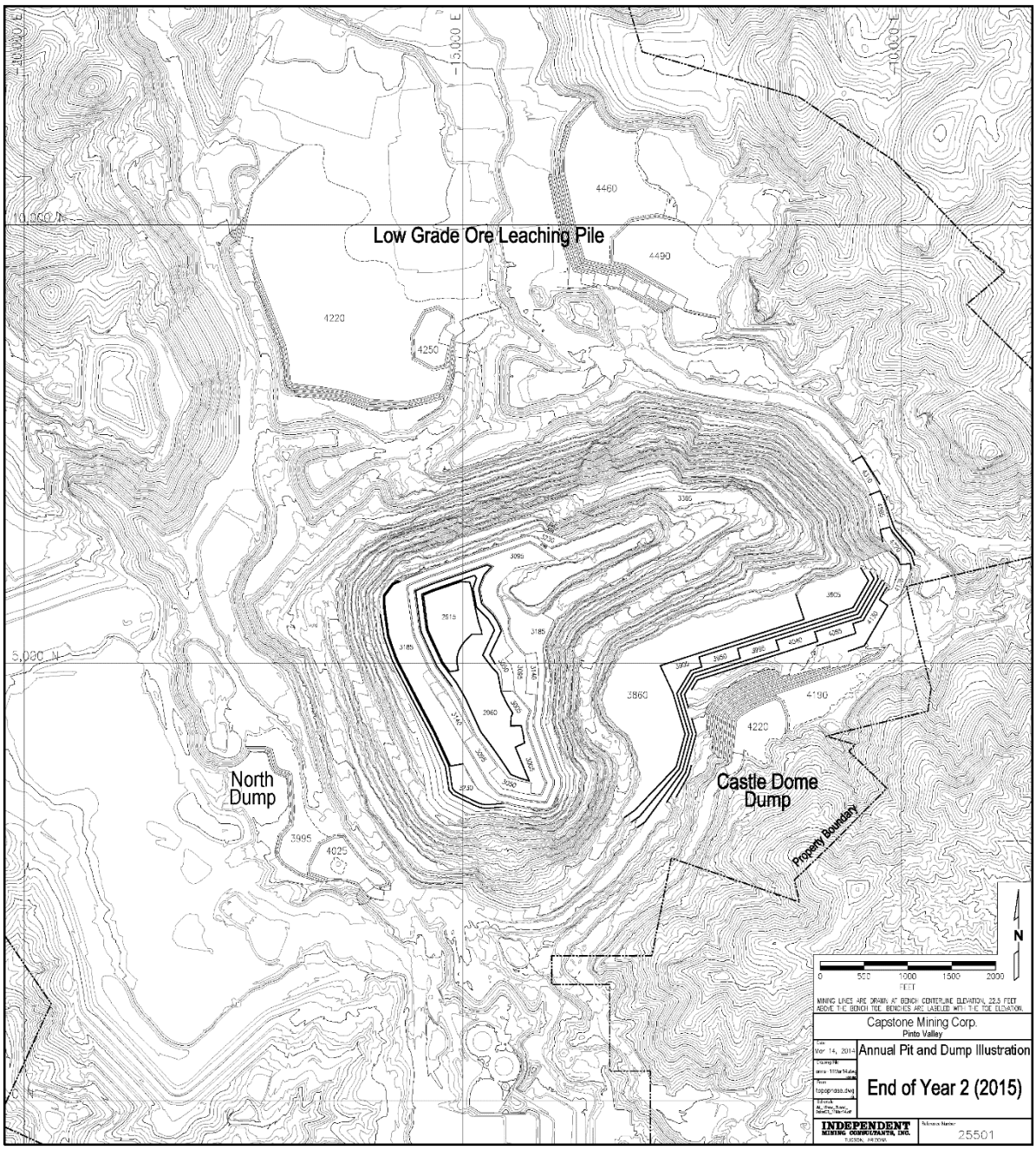


**Figure 16-10. Pit and Dump Configuration – End of 2016**

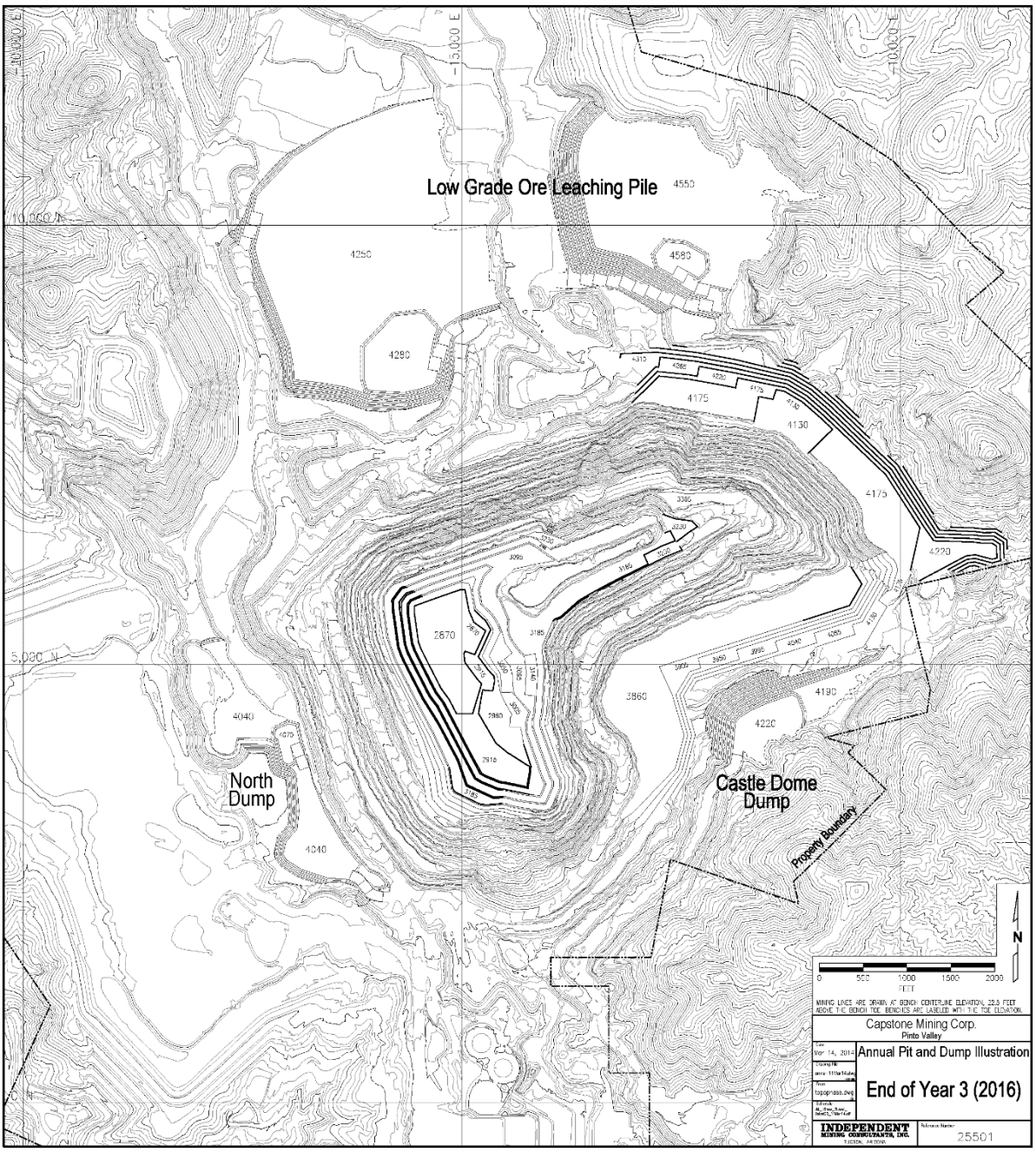


**Figure 16-11. Pit and Dump Configuration – End of 2017**

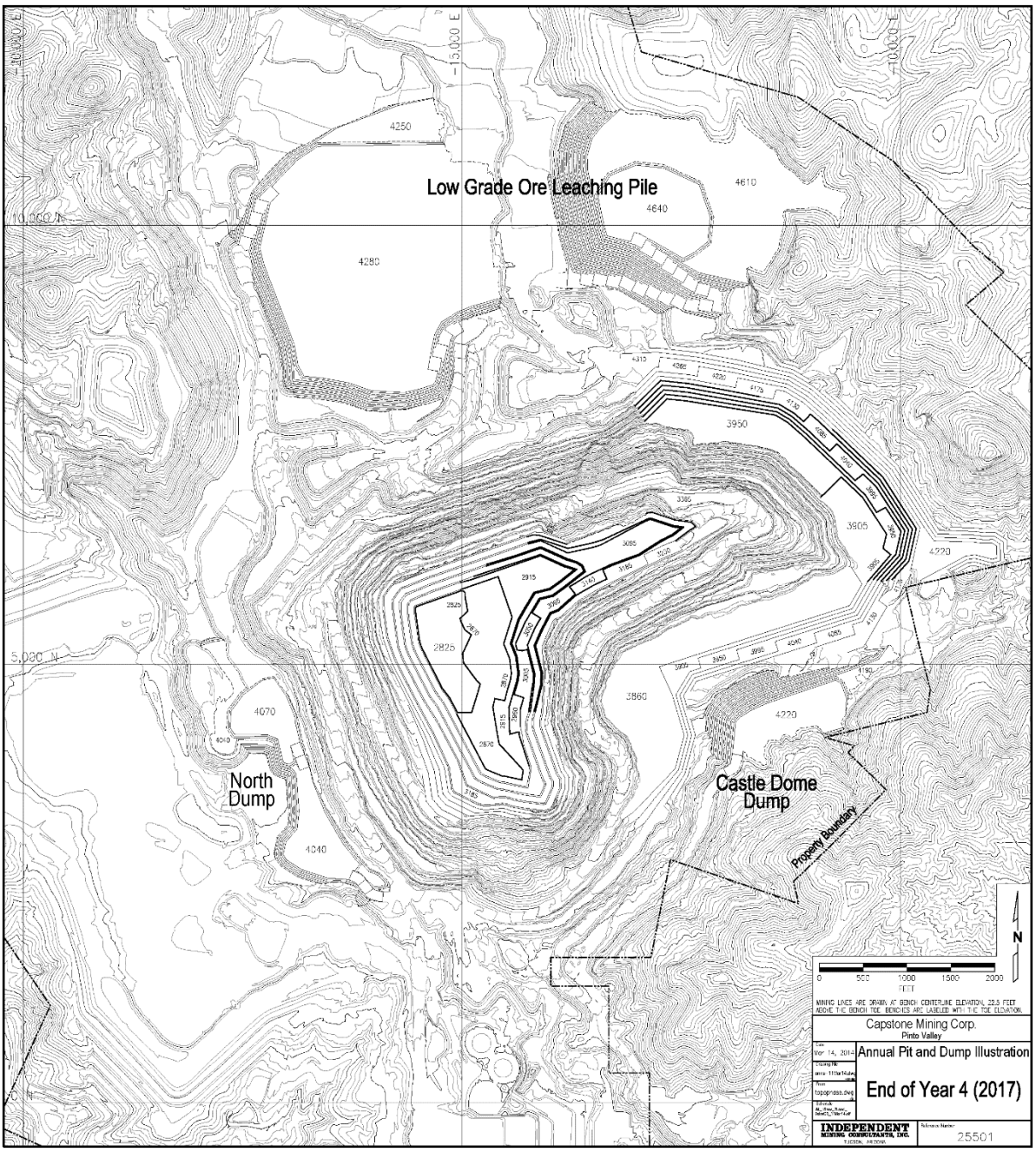


**Figure 16-12. Pit and Dump Configuration – End of 2018**

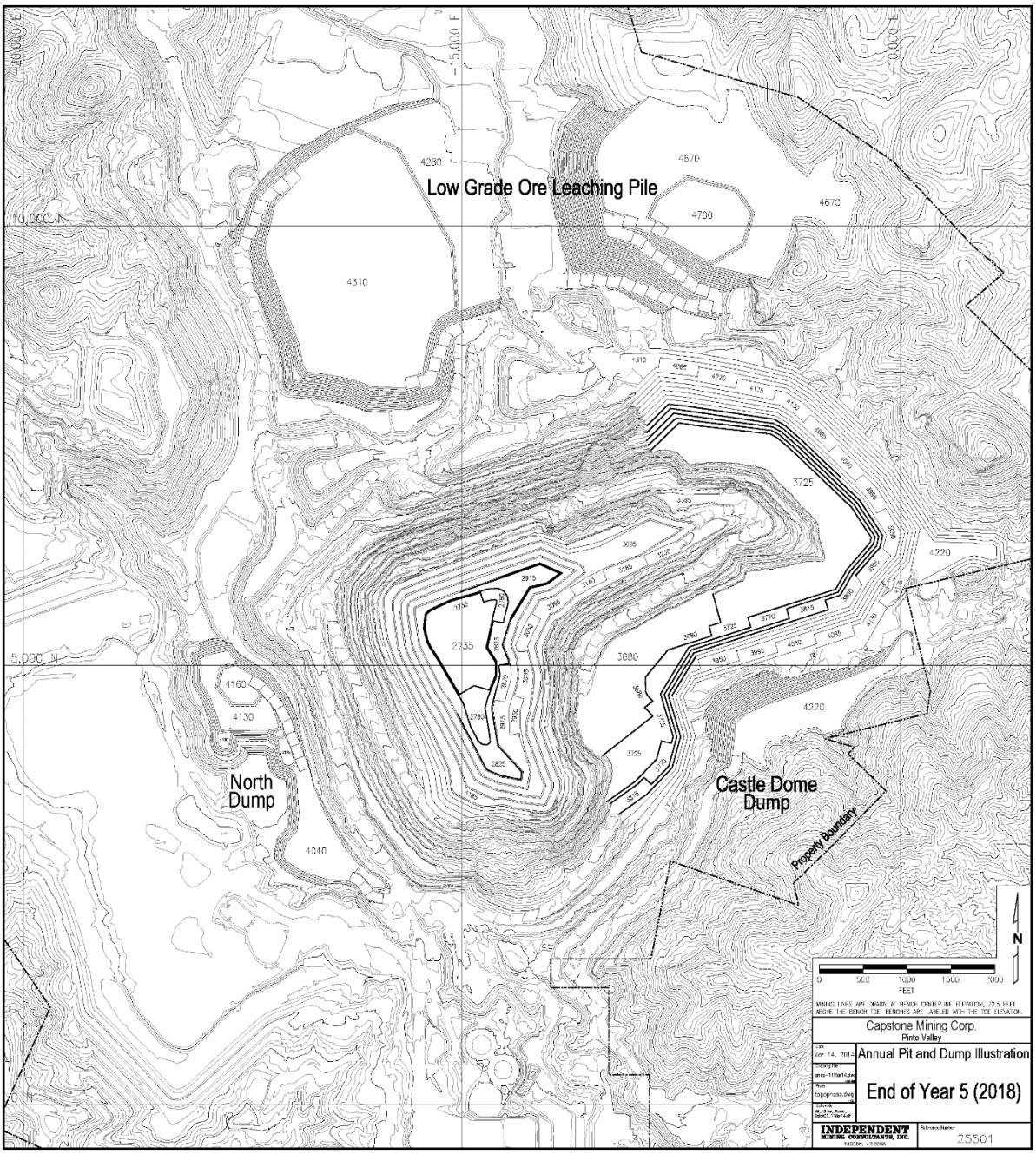


**Figure 16-13. Pit and Dump Configuration – End of 2020**

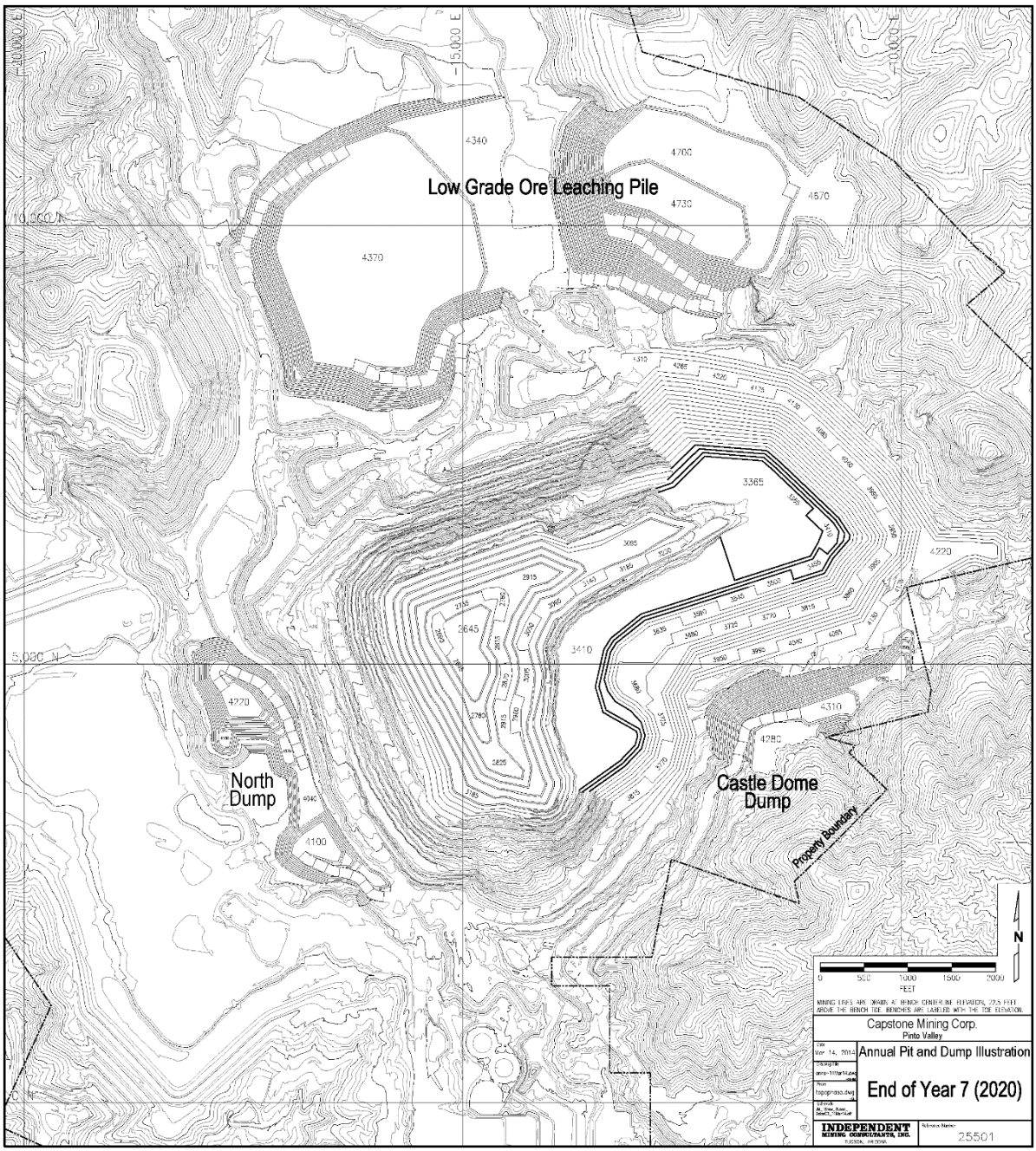


**Figure 16-14. Pit and Dump Configuration – End of 2023**

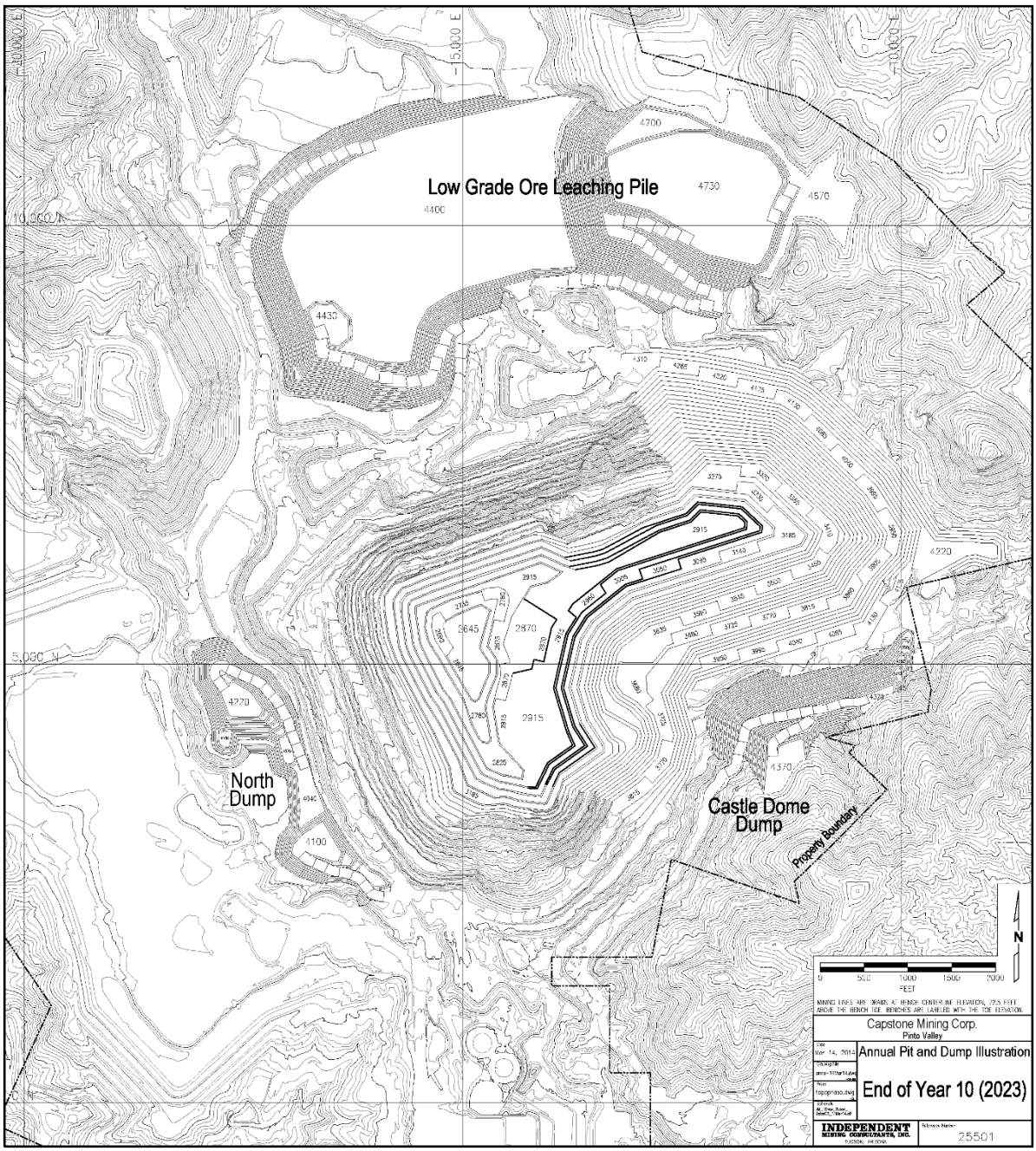
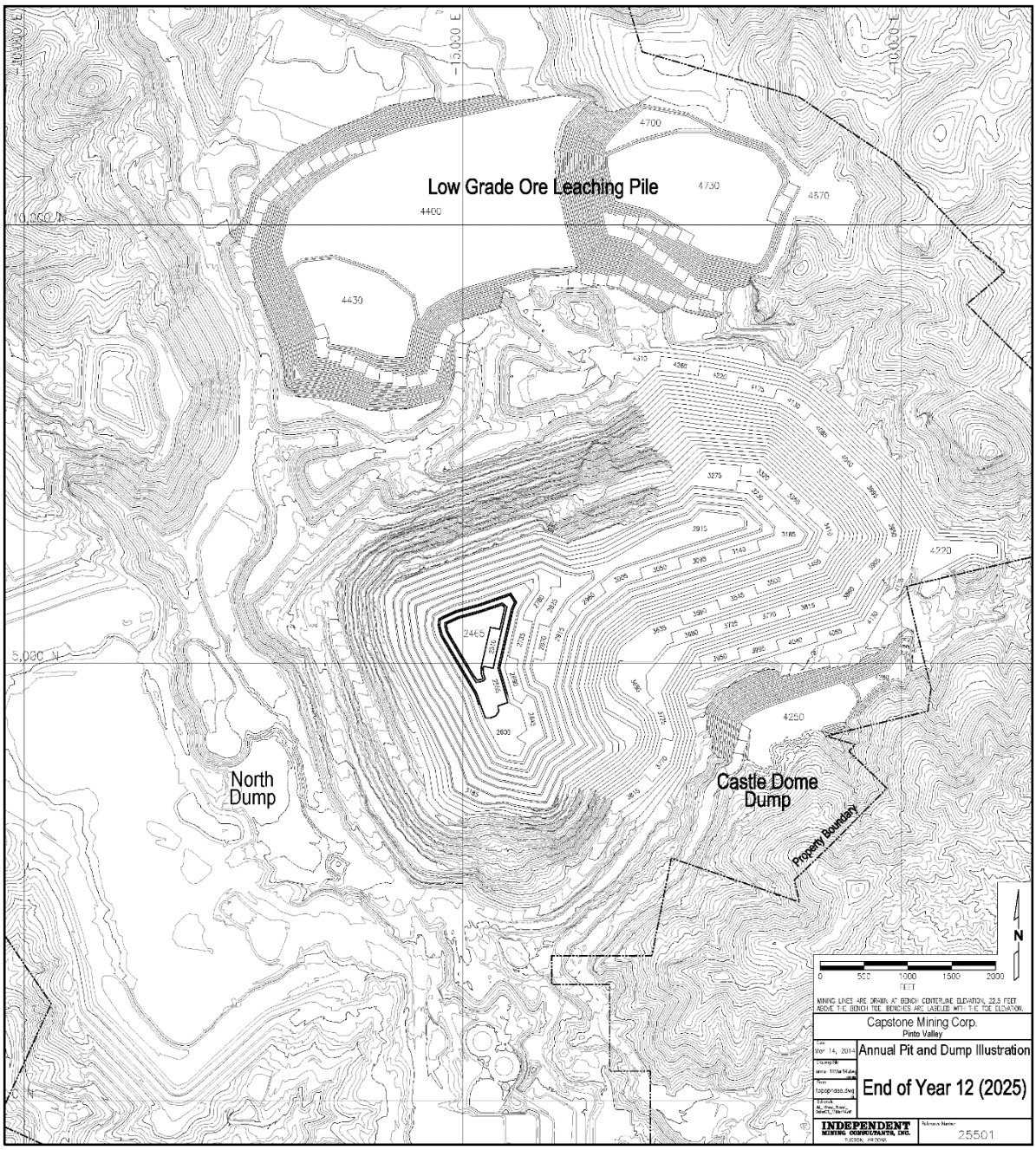


**Figure 16-15. Pit and Dump Configuration – End of 2025**



# 17.0 RECOVERY METHODS

## 17.1 Mill Process Design Criteria and Flow Sheet

The various unit operations of PVM have been evaluated to develop the basic process design criteria for the mill. Where possible, the operating data has been compared to testwork and/or simulation models to determine suitability for the design criteria. The key elements of the process design criteria include the following.

- Primary Crusher
  - Product Size Distribution (operating data)
    - 80%: 3.5 inches
    - 45%: 1 inch
    - 30%: 0.5 inch

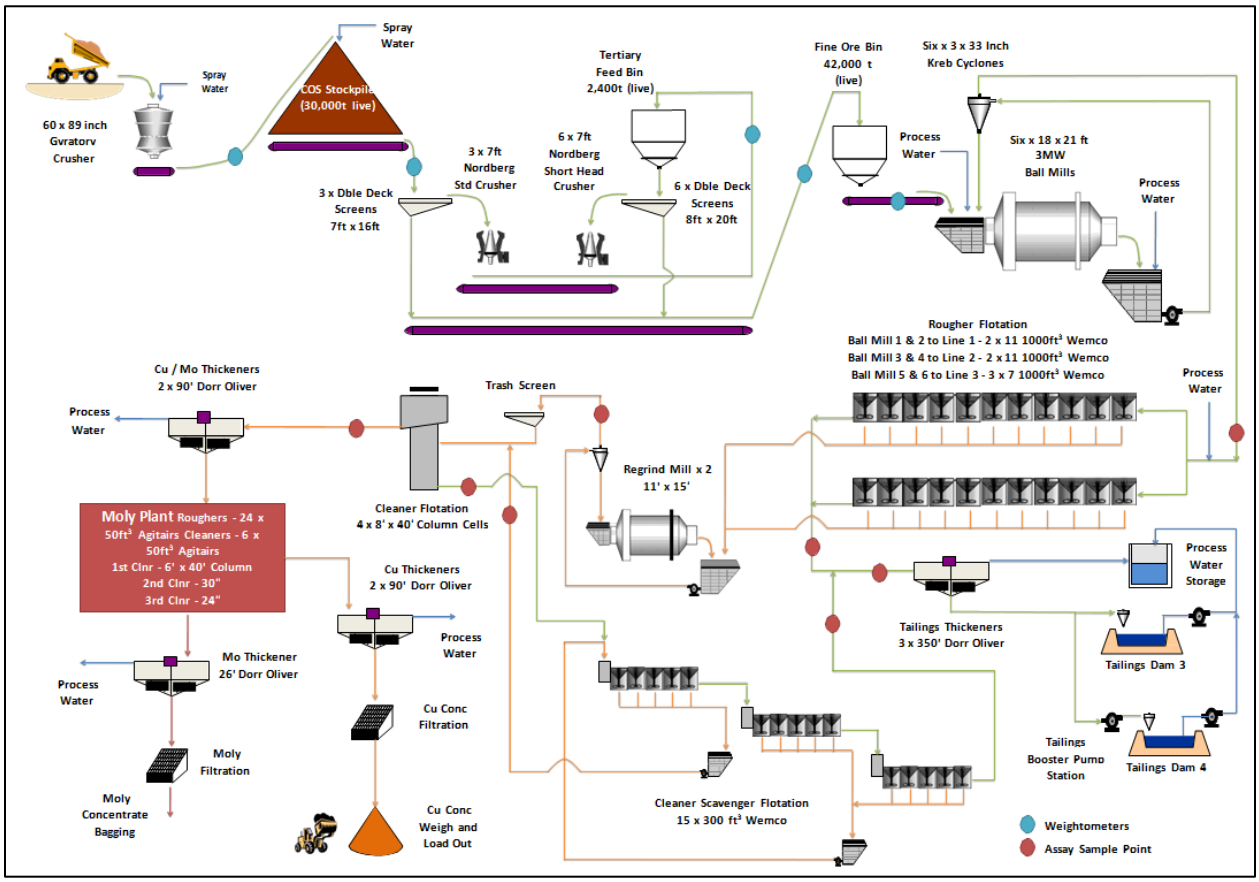
- Secondary / Tertiary Crusher
  - Crushing Circuit Availability: 75% (projected / historical)
  - Crushing Circuit Product, $P_{80}$: 11 mm (operating data)
  - Standard Cone Crusher Feed Rate: 3,000 tpoh (operating data)

- Grinding Circuit
  - BWi: 14.7 (metric) (testwork)
  - Grinding Circuit Availability: 95% (projected / historical)
  - Daily Throughput, Nominal: 55,100 tpd (model / operating data)
  - Cyclone Overflow, $P_{80}$: 280 µm (operating data)

- Copper Flotation
  - Copper Rougher Feed Density: 37% solids
  - Rougher Flotation Lab Residence Time: 8 minutes (ALS kinetic tests)
  - Plant Residence Time: 24.8 minutes
  - Scale-Up Factor: 3.1 (target > 2.0)

Figure 17-1 is a simplified process flow sheet representing the main unit operations applied in the Pinto Valley mill.

**Figure 17-1. Pinto Valley Simplified Mill Flow Sheet**




## 17.2 Plant Design – Mill

The following are the descriptions of the primary unit processes in the Pinto Valley mill. Based on the evaluation of the existing plant equipment, current operating results, and process models, and maintaining the design criteria identified in Section 17.1, it has been determined that the Pinto Valley mill has a nominal capacity of 55,100 tpd. With circuit optimization and tuning, it is believed that this can be increased to nominal 57,300 tpd.

### 17.2.1 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 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 30,000 tons. Based on digital size analysis of the primary crusher discharge, the $P_{80}$ of the primary crushed ore is approximately 3.5 inches. The fine product distribution is attributed to ore fragmentation and mine blasting practices. The discharge product averages 66% –2 inch and 45% –1 inch. The amount of fines in the feed has a significant influence on the production capacity of the fine-crushing facility.

### 17.2.2 Secondary and Tertiary Crushing

The coarse 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 a 7 ft × 16 ft Simplicity double-deck vibrating screen. Each screen oversize feeds a secondary 7 ft Nordberg standard cone crusher. Screen undersize from the secondary screens is sent to the FOB, with a nominal live capacity of 44,000 tons.

The secondary crushers operate in 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 ft × 20 ft Simplicity 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 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, so that tertiary crushing is in closed circuit. At the current plant throughput, the $P_{80}$ of the fine-crushing plant has been determined to be about 11 mm.

### 17.2.3 Grinding

Fine ore is reclaimed from the FOB and fed directly to six 18 ft × 21 ft, 4,000 hp Allis-Chalmers overflow ball mills. Each ball mill is an independent circuit consisting of discharge sump, pump, and cyclone nest. A review of recent operating data indicated that the $P_{80}$ of the cyclone overflow has been approximately 270 µm. 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 manage the

operation of the cyclones. Each circuit is equipped with three 33-inch inclined cyclones. Cyclone overflow slurry gravity feeds the rougher flotation banks, while the underflow discharges back to the ball mill feed. The ball mills operate in closed circuit with the cyclones, with a circulating load estimated at 300%. Xanthate, dithiophosphate, diesel, and lime are added to the grinding circuit to prepare the slurry for flotation.

### 17.2.4 Flotation

The flotation circuit operates as a staged process designed for the recovery of copper and molybdenum to individual concentrates. The primary focus of the rougher flotation circuit is to optimize recovery of the primary minerals from the ore into a reduced mass for economic downstream processing. Cleaner flotation delivers economic concentrate grades for marketing while maintaining high recoveries.

The flotation reagents used include Flomin C-3430 (Xanthate), C-2420 (dithiophosphate), and F-171 (frother). Some of the flotation reagents have been added to the grinding circuit for slurry conditioning, with addition rates augmented in the flotation circuit where required.

Regrinding of the rougher concentrate is required to provide the mineral liberation to allow high concentrate grades and recoveries from the cleaners. The molybdenum in the PVM ore is recovered to the bulk cleaner concentrate with the copper. The molybdenum flotation circuit provides for separation of the copper and molybdenum into respective concentrates for marketing.

#### Rougher Flotation

The rougher flotation circuit consists of sixty-five 1,000 $ft^3$ Wemco cells configured in three trains, with cyclone overflow from two ball mills feeding each train. Frother is added to the head of the rougher flotation cells, with other reagents (that were added at grinding) added as required. The rougher section is operated in open circuit, with the rougher tailings going directly to the final tailings.

#### Regrind

The rougher concentrate is delivered to the regrind ball mill circuit. Rougher concentrate is combined with the regrind ball mill discharge and pumped to the closed-circuit cyclones. The target product for regrind cyclone overflow is $P_{80}$ of 50 µm. The cyclone overflow is fed to the cleaner flotation circuit, while the underflow is sent to two regrind mills. The regrind mills operate in closed circuit with the cyclones.

### Cleaner Flotation

The cleaner circuit consists of four 8 ft diameter × 40 ft tall column flotation cells operated in parallel. The column cell concentrate, the final copper-molybdenum bulk concentrate, contains 27%–29% copper and 0.35%–0.7% molybdenum. The column cell tails are sent to the cleaner scavenger flotation bank. The cleaner scavenger bank comprises 15-300 $ft^3$ Wemco flotation cells. The concentrate from the first five cells is recirculated back to the column cells. The concentrate from cells 6 to 15 is recirculated back to the head of the cleaner scavenger bank. The tails of the cleaner scavenger bank are sent to final tailings.

### Molybdenum Plant

The copper-molybdenum concentrate from the cleaner columns is thickened before being sent to the molybdenum plant. The plant comprises four banks of Agitair rougher cells of six 50 $ft^3$ cells each and three stages of cleaning using column cells. Sodium hydrosulfide is added to the slurry to provide depression of copper and iron sulfides, thus enabling molybdenum to float. Fuel oil is added as a molybdenum promoter.

## 17.2.5 Concentrate Dewatering

The molybdenum rougher tailing is the final copper concentrate reporting to two 90 ft copper thickeners. The final molybdenum product is thickened in a 26 ft molybdenum thickener, filtered on a disk filter, dried, 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 $m^3$ copper slurry storage tanks. The slurry is pumped from the storage tanks to the filter plant. Previously, copper concentrate was pumped off-site to a filter plant. The process flow sheet has been modified to include an Eimco pressure filter and on-site copper concentrate storage shed to manage concentrate shipments.

## 17.2.6 Tailings Thickening

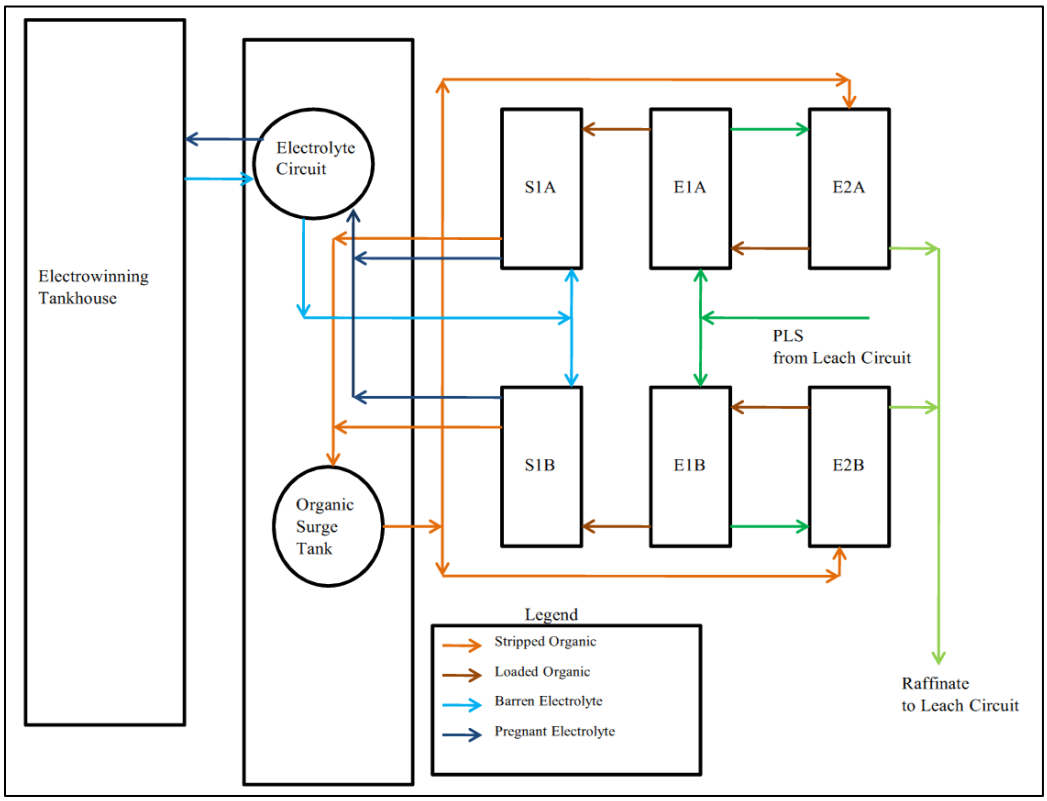
Tailings from the three rougher banks and the cleaner scavenger bank are combined and feed three 350 ft tailings thickeners where water is reclaimed, and the tails are thickened and sent on to the TSFs.

TSF4 is the primary location for the disposal of tailings from the Pinto Valley mill. TSF3 is used only as an emergency disposal point if a problem arises with the TSF4 conveyance system. It is also used for initial start-up.

## 17.3 Solvent Extraction and Electrowinning Process Flow Sheet

Figure 17-2 provides a schematic flow sheet identifying the primary unit operations in the SX-EW process facility.

**Figure 17-2. Solvent Extraction and Electrowinning Schematic Flow Sheet**



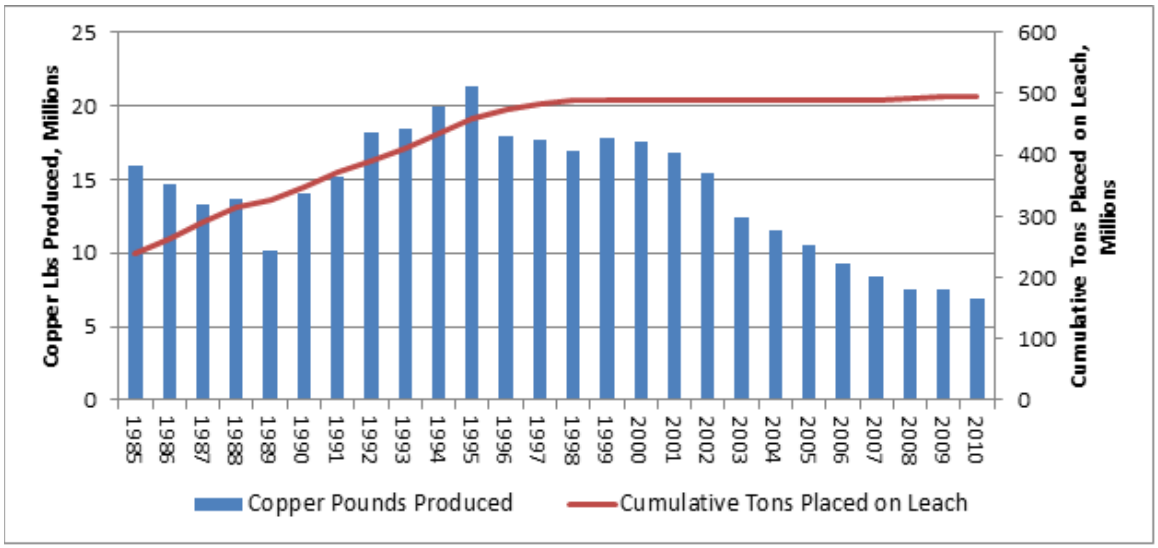

## 17.4 Plant Design – Solvent Extraction and Electrowinning

The PVM SX-EW plant was built and commissioned in 1981 by Holmes and Knarver to process solutions from the leach-grade material placed on the leach dumps.

### 17.4.1 Leach Dumps

Through 1998, approximately 500 M tons of leach material grading 0.13% TCu was placed in the dump leaching area. Chalcopyrite is the predominant mineral, with very small occurrences of chalcocite. In 1998, the mine and mill were placed on care and maintenance, and no new leach material was placed between 1998 and 2007. In late 2007, the sulfide operations were restarted, only to be shut down again in January 2009. An additional 6 M tons of leach material grading 0.18% TCu was placed on the dumps during this timeframe. To date, approximately 25% of the recoverable copper has been leached from the material placed on the leach dumps. Figure 17-3 shows historical SX-EW production at PVM.

**Figure 17-3. Solvent Extraction and Electrowinning Production from 1985 to 2010**




### 17.4.2 Solvent Extraction

The plant was designed to process 6,000 gallons per minute (gpm) of PLS through two solvent extraction trains (3,000 gpm/train). Each train contains a standard two extractor / one stripper (2 × 1) solvent extraction circuit, in which the copper is transferred by an organic solution to the electrolyte. The extractors are set up in a counter-current design. The organic contains roughly 3% LIX 984N, an extractive reagent produced by Cognis, now BASF. The remainder of the organic consists of kerosene, presently Phillips SX-12. This cut of kerosene was specifically developed for use in solvent extraction plants. The depleted leaching solution, known as raffinate, is acidulated and sent back through the leach dumps to recover more copper. Typically 7,000 gpm of PLS is processed through solvent extraction on a daily basis.

### 17.4.3 Electrowinning

Pregnant electrolyte from the solvent extraction plant is advanced to the electrowinning tankhouse, where the copper is plated on starter sheets, which are harvested on a 7-day cycle. The electrowinning tankhouse has 52 commercial cells and 8 starter cells. Cobalt sulfate and guar gum are added to the electrolyte to ensure cathode quality. Presently, the cathode produced in the electrowinning plant is 99.999% pure ("5 nines") and is COMEX-certified.

The current budget has funds allocated for repairs to process equipment based on the planned PV2 cathode copper production projections. If these increase to historical production levels, there may be a requirement for significant process repairs.

# 18.0 PROJECT INFRASTRUCTURE

The forward work plan for infrastructure at PVM has considered mining, processing, support, and concentrates handling activities. This section examines the maintenance facilities, warehouses, TSFs, roads, offices, change houses, explosives, and power and water distribution systems. Costs are discussed in Sections 21 and 22.

## 18.1 Site Plan and Layout

Most buildings and facilities on the property were erected from 1972 to 1974, when PVM was developed. The mine and mill went into production in 1974, producing a copper concentrate and a molybdenum concentrate. Beginning in 1981, copper cathode was produced from leach solutions at an SX-EW plant (E&MJ News 2012). The SX-EW plant, mill, and processing infrastructure are addressed in Section 17.

### 18.1.1 Existing Mine

The existing infrastructure is fully functional, with plans to improve and augment the infrastructure as necessary for future operations.

#### North Barn

The North Barn (indicated in Figure 18-1) is an approximately 15,000 $ft^2$ mining fleet maintenance facility containing a lift, a 10-ton bridge crane, compressed air, and lubrication services. Adjacent to the North Barn are a two-bay wash pad and a fueling station.

#### Main Shop

The Main Shop has eight primary service bays, three vehicle access doors, and three secondary service bays. All primary bays are serviced by twin 20-ton bridge cranes that traverse the length of the shop. The Main Shop also houses the General Tool Room, Machine Shop Tool Room, Pipe Shop, Plate and Iron Weld Shop, Shovel and Drill Repair Shop, Light Vehicle Repair Shop, other ancillary repair areas, and offices for engineering, maintenance services, and procurement.

The work plan calls for the relocation of light-vehicle repair to the North Barn facility and for the consolidation of primary maintenance activities on the mine haulage fleet and support equipment to the Main Shop area.

#### Explosives Plant

The primary explosives facility includes two ammonium nitrate silos, an emulsion plant, and two powder magazines. They are located on the southeast perimeter of the pit, near the remains of the abandoned Castle Dome processing facility.

#### Pit Dewatering

The El Niño pit dewatering system is primarily used to handle storm events and consists of in-pit pumps, a booster pump station, and piping that exits the pit on the southern wall and follows the Castle Dome road on the southern perimeter of the pit.

### 18.1.2 Concentrate Dewatering and Load-Out

Two 90 ft Dorr-Oliver thickeners are used to settle final copper concentrate. One 26 ft Dorr-Oliver thickener handles molybdenum concentrate. All thickeners are open conical tanks. A 50-chamber Dorr-Oliver horizontal filter press performs batch concentrate dewatering and is housed in a dedicated building. The concentrate cake is conveyed to an adjacent covered building with a 10,000-ton storage capacity. Empty concentrate trucks enter at the north end of the building, are loaded, and exit through a door to the south.

### 18.1.3 Support Facilities

Key support facilities include the administration offices, two warehouses, a core storage building, two change houses, and an assay / sample preparation laboratory.

#### Warehousing

The Main Warehouse is integrated with the Main Shop. The pallet storage area is a 20,468 $ft^2$ high-volume space with a height of 22 ft (at low side). The 4,896 $ft^2$ small-parts storage area is located underneath the office mezzanine.

The Heavy Duty Warehouse, located east of the main gate, is 26,000 $ft^2$ and has a height of 20 ft (at low side).

Additional warehousing is available at the North Barn and at several laydown yards.

#### Change Houses

The Mine Change House and the Mill Change House service both male and female staff and are equipped with banks of dry lockers (connected with a plenum), hooks for hanging soiled garments and personal protective equipment, and showers. The combined locker space is over 600 units. Additional lockers and showers at the lower (main) change house will be recovered by completing renovations in unused areas of the building.

**Figure 18-1. Pinto Valley Mine Site Plan and Layout**

Google, USDA Farm Service Agency 2013

## 18.2 Roads and Logistics

PVM is located in a mountainous region in southeastern Arizona, 6 miles west of the town of Miami and 13 miles northeast of the town of Superior.

### 18.2.1 Roads

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 (4,600 ft) occurs just west of PVM. A three-lane tunnel on US 60 is located 2 miles east of Superior. The tunnel can accommodate mid-size haulage and construction equipment. Extremely large loads can be brought to the site using alternate routes through Tonto Basin (US Highways 87 and 188) from the west, or Safford (US Highways 70 and 191) to the east.

#### Access

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.

#### Property

Roads in proximity to the processing area are generally paved and well maintained. Roads near the pit, leaching area, and TSFs are dirt and are accessible throughout the year. The well-maintained Haunted Canyon Road runs north and south on the eastern perimeter of the property and is used to support site and facility access. All roads on the property are maintained by PVM. Significant modifications are required only for haulage roads in the pit, as scheduled in the mine plan. Mine haulage is addressed in Section 16.

### 18.2.2 Rail and Ports

SMARRCO, a wholly owned subsidiary of Capstone, operates a trans-shipment facility in San Manuel, Arizona; a public carrier railroad; and a 250-car-capacity switching yard in Hayden, Arizona. Concentrate is typically loaded and trucked by contractor from PVM to San Manuel and loaded into 100-ton-capacity railcars. Approximately twice per week, a trainload of copper concentrate is hauled by SMARRCO approximately 30 miles to Hayden. Capstone uses the Class 1 carrier Union Pacific Railroad and the connecting carrier Ferromex to transport concentrate from Hayden to Ciudad de Guaymas, Sonora, Mexico, where it is loaded onto ocean freighters for delivery to international markets. An agreement is in place for

30,000 tons of concentrate storage in Guaymas. Upgrades to the San Manuel facility were completed in 2012–2013 and are not required for the next phase of work.

In coordination with PVM operations, SMARRCO schedules railcar interchanges with the Union Pacific Railroad. Capstone uses its 80,000 tons of covered storage (40,000 tons at San Manuel, 30,000 tons at Guaymas, and 10,000 tons at PVM) and temporary in-car storage at Hayden to optimize inventories and shipments.

## 18.3 Waste and Marginal Ore Storage

The Northside Dumps (9.1, 9.11, 9.12, and 9.3), the Gold Gulch Dump, the North Dump (briefly used in past), and the East Dump (future) are located in the Gold Gulch drainage sub-basin. Waste rock below 0.10% TCu or below the 0.18% TCu stockpile cutoff that is high in clay content or calcareous will be delivered to the Gold Gulch Dump. The Westside Dump is in the Miller Gulch sub-basin. A number of others, including the Southside Dump 13, Southside Dump 14, 19 Dump, 19.1 Dump, and 19 Extension Dump (future) are in the Cottonwood Canyon sub-basin.

The majority of the waste rock storage facilities overlie the local basin-fill formations (Gila conglomerate, Whitetail conglomerate, dacite). Three dumps overlie the Pinal schist and/or granite (19 Dump, 19 Extension Dump, and East Dump), and two others (Westside and Southside Dump 14) overlie the consolidated TSF1/2.

As part of the PV2 stage of operations, marginal-grade material will be mined and stored at either the Castle Dome Marginal Dump or the North Barn Marginal Dump. The Castle Dome Marginal Dump will be built on the south side of the pit in an area within the open-pit drainage sub basin. The North Barn Marginal Dump is an extension of the Northside Dump 9.3, west of the open-pit. This marginal-grade material will be placed on the Low-Grade Ore Leaching Pile or processed in the mill, should it be economical to do so.

## 18.4 Tailings Storage

PVM currently operates two TSFs. TSF4 is the primary storage facility, and TSF3 is used when maintenance is required on the TSF4 tailings distribution system or during plant upset conditions. Previously used TSFs at PVM were the Cottonwood and TSF1/2. These previously used facilities have been closed and partially reclaimed.

TSF4 has been evaluated and was determined to be capable of storing 246.0 M tons (dry weight) of tailings at a settled density of 92 lb/ft$^3$. TSF4 would rise from the current ~3,800 ft crest elevation to the planned 4,005 ft crest elevation at an average 17 ft-per-year rate of rise.

TSF3, once the USFS Plan of Operations is amended, will be capable of storing 23.0 M tons of tailings at a settled density of 90 lb/f$^3$. Within the current patented land boundary, TSF3 is capable of storing 7.1 M tons of tailings. TSF3 is designated to be used intermittently for a maximum of 32 days per year, or 10 ft of raise per year.

### 18.4.1 Tailings Characteristics

The average annual daily mill throughput in 2014 and 2015 is expected to be 55,100 tpd (dry weight), increasing to 57,300 tpd in 2016 through April 2026. The total tailings storage requirement is 253.6 M tons for the 12.3-year mine life. The design tailings pipeline flow rate is calculated to be 12,011 gpm, with 55% solids. The solids percentage can vary between 48% and 60%.

The tailings gradation test results performed by AMEC indicate that approximately 10% of the particles are finer than 4.8 µm, approximately 50% of the particles are finer than 112 µm, and approximately 80% of the particles are finer than 360 µm.

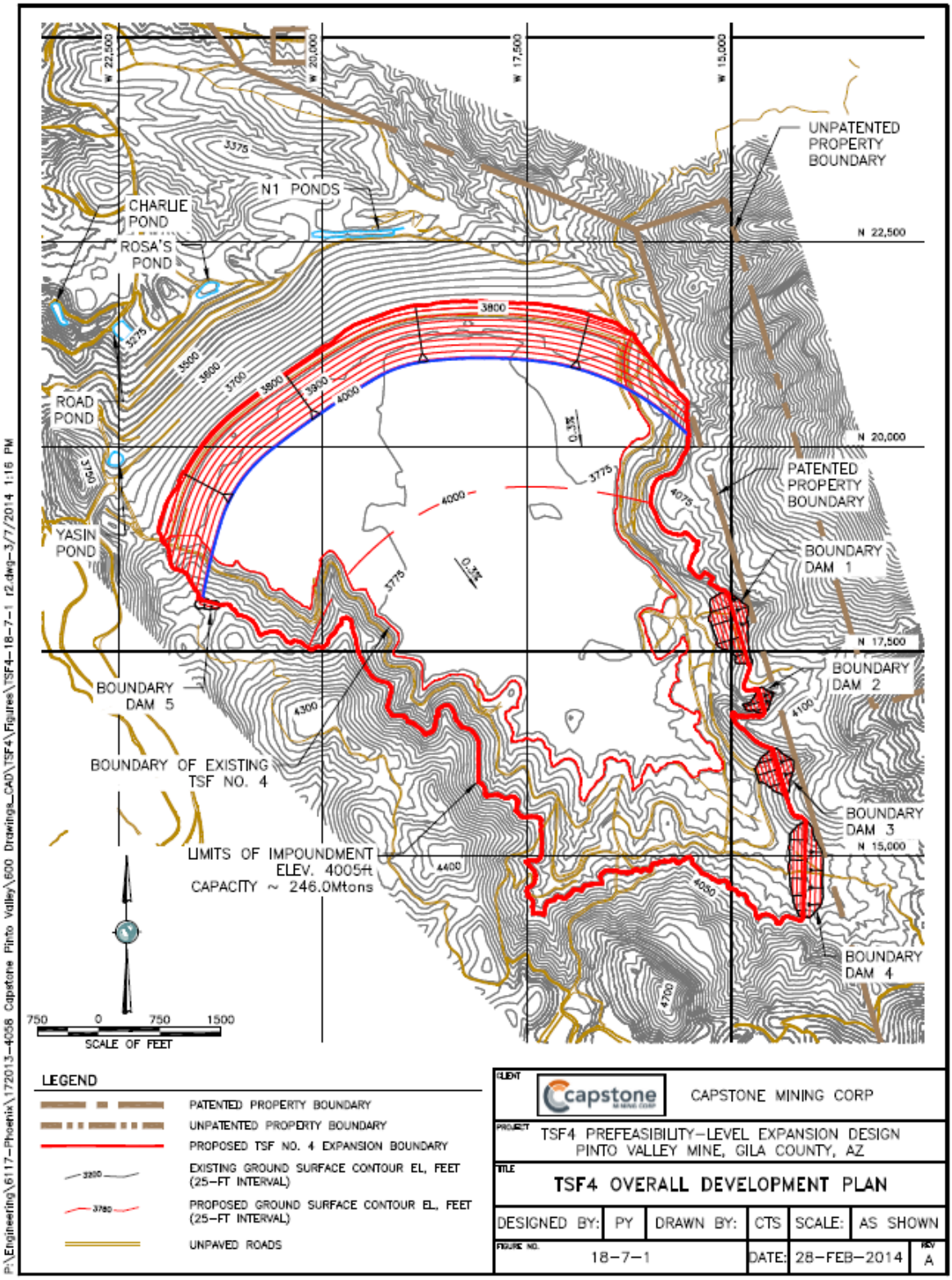
### 18.4.2 Facility Design

#### Design of Tailings Storage Facility No. 4

Figure 18-2 presents the layout of TSF4. TSF4 was placed in service in 1977. The crest of the TSF was at elevation 3,800 ft at the end of 2013. The mine plan mill production will result in TSF4 raised to elevation 4,005 ft over 12.3 years, with an average raise rate of 16.7 ft/yr. The dam is constructed in an upstream manner with a cycloned sand shell.

The raise of TSF4 is constrained by the patented land boundary. Boundary dams will be required at four locations along the eastern edge of the facility to prevent the migration of tailings to the adjacent USFS land. The boundary dams can be built in a staged manner, beginning in 2016.

**Figure 18-2. Tailings Storage Facility No. 4 Overall Development Plan**



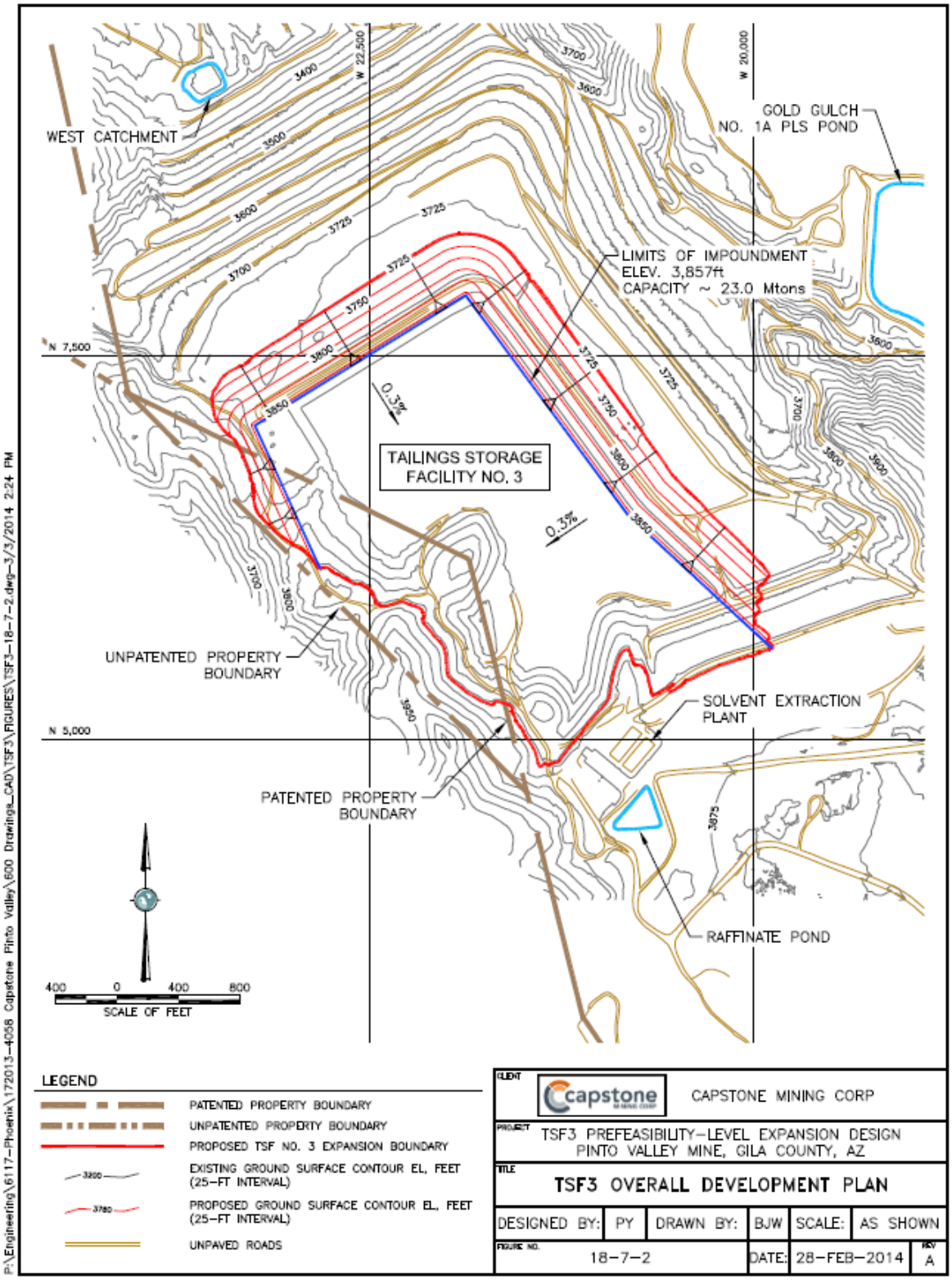
### Design of Tailings Storage Facility No. 3

Figure 18-3 presents the layout of TSF3. The facility was placed in service in 1973 and was operated until early 2009 to a crest elevation of 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 TSF starter embankment was constructed to an elevation of 3,756 ft. A boundary dam was constructed in this area in late 2013, and tailings that had crossed into the adjacent USFS land in mid-2013 were removed.

The raise of TSF3 is also currently constrained by the patented land boundary. A raise of boundary dam will be required at the southwestern portion of the facility to prevent the migration of tailings to the adjacent USFS land. The mine plan includes raising of the boundary dam an additional 24 ft in two 12 ft-high embankments constructed of borrow materials in an upstream manner. The main embankment will be raised in a centerline manner using cyclone sand construction.

Modifications to the current USFS Plan of Operations are ongoing. Once the agreement is in place, the plan is for TSF3 to be raised to elevation 3,857 ft over 12.3 years, with a maximum raise rate of 10 ft/yr.

**Figure 18-3. Tailings Storage Facility No. 3 Overall Development Plan**



### 18.4.3 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 conditions associated with a seismic event of a 1,000-year recurrence interval over the planned 12.3-year mine life. AMEC 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 reclaimed in downgradient wells.

TSF3, while having historical issues with seepage on embankment slopes, will be used intermittently and with a revised configuration with deposition of the tailings inset from the prior dam crest. Recommendations have been developed to limit the number of days per year that TSF3 should be operated and for the maximum yearly raise rate. A work plan has been developed to evaluate the performance of TSF3 and TSF4 under dynamic earthquake loading.

### 18.4.4 Tailings Storage Facilities Operation

The mill tailings are thickened to approximately 50%–57% solids by weight and are conveyed from the thickener collection box to the TSF3 junction 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 AMEC 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 of this study. 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.

#### Tailings Storage Facility No. 4 Operation

From the TSF3 junction, the tailings are conveyed via gravity pressure flow through a 28-inch diameter Naylor (carbon steel) polyurethane-lined pipeline for approximately 16,080 ft, transitioning to 34-inch diameter DR 15.5 HDPE pipe for the final 3,160 ft, terminating at the TSF4 Booster Pump Station.

The existing TSF4 Tailings Booster Pump Station consists of three parallel trains, each with two pumps in series. All pumps are rubber-lined Warman Model AH 14/12 slurry pumps. The second-stage pumps in each train are controlled by variable frequency drives (VFDs). Normal operation consists of running two trains, with the third train on standby. Pump speed is controlled by maintaining a set pressure level in the suction pipeline, as measured by pressure gauges on the vent pipeline immediately to the south of the pump station. The booster station piping generally consists of 24-inch rubber-lined steel piping for the suction and discharge manifolds, with nominal 16-inch rubber-lined suction and discharge lines to each pump. Appropriate reducers and increasers change the size of this piping to match the pump suction and discharge flanges. The first-stage pump motors are 600 hp, 1,780 rpm, 4,160 volt (V) synchronous motors with 4.06:1 belt-driven speed reducer giving a constant speed of approximately 450 rpm. The second-stage pump motors are 400 hp, 1,780 rpm, 480 V synchronous motors with VFDs and 4.06:1 belt-driven speed reducer giving a 100% speed rating of approximately 450 rpm.

### *Tailings Storage Facility No. 4 Third-Stage Pump*

The next raise of TSF4 will require that a third-stage pump be added to the Booster Pump Station to provide additional energy to reach the far cyclones. The third-stage pumps would be Warman Model AH 14/12 slurry pumps with 400 hp, 1,780 rpm synchronous motors with 4.06:1 belt-driven speed reducer giving a 100% speed rating of approximately 450 rpm. The new pumps would be installed between the existing pumps and the Motor Control Center building. The discharge pipes from the existing second-stage pumps would be replaced with new pipes directly connected to the new third-stage pump suction. The discharge from the new third-stage pumps would be piped to the existing discharge header with appropriate valves. The new third-stage pump would allow for an increase of the TSF4 crest height to 3,940 ft.

### *Tailings Storage Facility No. 4 Fourth-Stage Pump*

Prior to exceeding the TSF4 crest height of 3,940 ft, a fourth pump stage would be added. The fourth-stage pumps would be Warman Model AH 14/12 slurry pumps with 400 hp, 1,780 rpm synchronous motors with 4.06:1 belt driven speed reducer giving a 100% speed rating of approximately 450 rpm. The new fourth-stage pumps would be installed between the existing pumps and the Motor Control Center building. The discharge pipes from the third-stage pumps would be replaced with new pipes directly connected to the new fourth-stage pump suction. The discharge from the new fourth-stage pumps would be piped to the existing discharge header with appropriate valves. The new fourth-stage pump would allow for an increase of the TSF4 crest height to the ultimate elevation of 4,005 ft.

The discharge from the TSF4 Tailings Booster Pump Station consists of a 24-inch polyurethane-lined steel pipeline up the south side of the TSF4 embankment to the dam crest. The pipeline continues across the dam crest to the far abutment. Taps for cyclones are spaced approximately 50 ft apart. Up to 16 cyclones may be in operation at any one time. The fine-grained fraction from the cyclone overflow is piped to the TSF beach. The coarse cyclone underflow material is used to construct the dam embankment.

Concurrent with the addition of the new third-stage pumps, the pipeline from the Booster Pump Station to the dam crest should be replaced with a new 34-inch DR 15.5 HDPE pipeline to lower pumping requirements and associated energy costs.

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. In addition to supplying reclaimed water to the mill, the system is also configured to provide a backup supply of gland water to the TSF4 Booster Pump Station in the event that Peak Well water is not available.

The existing dual barge pumps are Hazleton Model 12DA, Type VNCC with 21.5-inch impellers. The drivers are Westinghouse 186 kilowatt (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. These back-pressure-valve-controlled pump drives will be replaced with new VFDs to increase energy efficiency.

When the TSF4 decant pool reaches an elevation of 3,8oo ft, the first booster pump station will no longer be needed and will be abandoned. At that time, the second booster pump station will be relocated to elevation 4,035 ft.

Because of the deep draft of the existing barge pumps, higher reclaim return rates cannot be sustained without a potential for mud to be pulled into the pump suction. The barge pumps will be replaced with landside, skid-mounted Godwin HL260 Dri-Prime centrifugal pumps. Suction for the pumps will be provided by a floating suction intake, allowing for pools as shallow as 2 ft deep. The landside pumps will be controlled with VFDs to minimize energy consumption. As the pool elevation rises, the skid-mounted pumps are pulled to higher elevation.

#### Tailings Storage Facility No. 3 Operation

From the TSF3 junction, the tailings are conveyed by a 24-inch diameter Naylor Schedule-40 Steel and 24-inch DR 11 HDPE pipeline for 1,700 ft to the 18-inch Naylor Steel distribution ring, with 6-inch rubber hose laterals to the cyclones spaced at roughly 40 ft apart. TSF3 will be raised to elevation 3,857 ft using a cyclone deposition system similar to that described in the previous section. As deposition increases past elevation 3,763 ft, judgment will have to be exercised in the selection of which cyclones to operate. When the far 8 cyclones are used, the near cyclones should also be used to minimize the friction loss in the pipeline to the far cyclones. Deposition above elevation 3,795 ft at full mill production rates will require the construction of a pump station at TSF3.

The TSF3 Booster Pump Station will be similar to the TSF4 Booster Pump Station, but only one stage will be required. The new TSF3 Tailings Booster Pump Station consists of three parallel pumps. All pumps will be Warman Model AH 14/12 slurry pumps controlled by VFDs. Normal operation will consist of running two trains with the third train on standby. Pump speed will be controlled by maintaining a set pressure level in the suction pipeline. The station piping will consist of 24-inch rubber-lined steel piping for the suction and discharge manifolds, with nominal 16-inch rubber-lined suction and discharge lines to each pump. Appropriate reducers and increasers will change the size of this piping to match the pump suction and discharge flanges. The pump motors will be 600 hp, 1,780 rpm synchronous motors with 3.18:1 belt-driven speed reducer giving a 100% speed rating of approximately 560 rpm.

The existing reclaim water system at TSF3 consists of a single trailer-mounted 8 × 6 self-priming centrifugal pump with an engine drive. The pump conveys water from the southern end of the TSF3 to the Mill Water Supply Tank. Eight-inch rubber suction hose is connected to an intake float. A 6-inch rubber discharge hose connects to a larger steel pipeline. The capacity of the existing system was not determined.

### 18.4.5 Monitoring

A series of piezometers has been installed to monitor groundwater fluctuations at the current and previously operated TSFs. The piezometers consist of open standpipes, pneumatic, and vibrating wire piezometers. Several of the open and pneumatic piezometers were converted to the electronic vibrating wire style in 2009 and are recorded daily remotely via telemetry. The remaining piezometers are measured monthly. Quarterly monitoring reports of the piezometer readings are prepared and reviewed by the designated PVM engineer to verify that the TSFs are operated as designed. An annual monitoring report, which includes a summary and graphs of

the piezometer measurements, is submitted to ADEQ. A work plan has been developed to replace inoperative piezometers.

TSF1/2, TSF3, and TSF4 are inspected quarterly and after rainfall events greater than 3 inches in 24 hours, for evidence of crest failure, toe slippage, substantial cracks, and erosion features. The active and former TSFs are annually inspected by a registered consulting geotechnical engineer. Feedback is provided by the geotechnical engineer on the operation of the TSFs, and recommendations, if necessary, are provided to the PVM engineer.

## 18.5 Water Management

### 18.5.1 Water Management Systems

Total water demand is met by seven systems: fresh, process, service, fire, reclaim, pit dewatering, and potable. The ultimate use of a particular water source is generally determined by its dissolved solids content and pH level.

PVM is located in a semi-arid region where rainfall is variable, with drought-like conditions possible. Water is a valuable resource, and PVM is sensitive to the demands of the local businesses and residents on the available supply. PVM has ongoing programs in place to manage water. An overall balance for the property is complex and is being updated. Alternative sources for water and programs for water conservation have been investigated. Rainfall runoff is captured and diverted into the process water supply. Continued use of water pumped from the pit and recycled from the TSFs helps to minimize the need for new water. The TSFs will continue to receive attention, as some of the largest water losses are caused by evaporation, entrainment, and transpiration in the TSFs.

### 18.5.2 Sources of Water

Fresh water is drawn from the Peak Wells system and is used as service water for heat exchangers, pump glands, and fire water, but can also be added to the process water distribution systems.

Process water is supplied by the Burch / Old Dominion system, east wells, recycled thickener overflow, the Diamond-H pit pumping system, and the Pinto Valley pit dewatering system. A portion of the process water is diverted to the mine reservoir for use in equipment washing and dust control. Service water and fresh water supply the fire system, which has a separate distribution network.

The process water supply is augmented by the reclaim water system to meet site total water demand. Reclaim water is discussed in detail in Section 18.4.4.

Water is pumped out of the bottom of the pit (pit dewatering) to be used as make-up water for the SX-EW leaching circuit or added to the process water system. PVM's dewatering system consists of barge pumps, an 18-inch HDPE pipeline, a booster pump with a 600 hp motor, and associated switchgear, power cabling, and transformers.

The potable water system consists of four on-site groundwater wells (Miller Springs No. 2, Peak No. 1, Peak No. 37, and Domestic No. 1), two of which are currently in operation. The water is disinfected using a chlorination system.

### 18.5.3 Water Storage

The service water storage facility consists of two 1-million-gallon tanks and one 300,000-gallon head tank. The bottom portion of the head tank also services the fire water system. Process water is stored at Cottonwood Reservoir and in two 2-million-gallon mill tanks. Reclaimed water may also be pumped to the mill tanks or Cottonwood Reservoir for storage. Storage for the potable water system consists of one 110,000-gallon storage tank and a 10,000-gallon tank that equalizes flows between the wells and the booster pumps.

### 18.5.4 Reclaim Water System

Reclaimed water is a recycle stream from TSF3 and TSF4 and is primarily used as mill process water, but may also be pumped to the mill tanks or Cottonwood Reservoir. The operation of the reclaim water system is discussed in Section 18.4.4.

### 18.5.5 Wastewater Treatment

Serving the majority of PVM employees, a gravity sewer collection system conveys the sewage to a 25,000-gallon-per-day extended aeration treatment system. This central treatment plant, a factory-built Smith & Loveless Inc. OXIGEST System Model 20-B-25, utilizes primary and secondary treatment technologies. The effluent is disinfected with calcium hypochlorite tablets within the effluent box prior to being pumped to Tule Pond (just south of the plant) or to the TSFs. Independent septic systems are used for supplemental wastewater treatment in several remote areas.

## 18.6 Mine Infrastructure Modifications

PVM will utilize existing infrastructure with minor upgrades to support the work plan. The infrastructure work plan has identified two primary areas that will require sustaining capital: (1) the consolidation of mine operations and maintenance into the central operations area of PVM, and (2) ensuring continued development of the ore body.

### 18.6.1 Mine

PV2 mine development pushbacks are planned for the south and east sides of the open pit. Optimum development of the mine plan requires the relocation of the East Power Loop and explosives plant and the modification of the pit dewatering system. Additionally, the work plan considers construction of a drainage channel on the south perimeter of the pit.

#### East Power Loop

The location of the existing overhead power lines limits mine development to the east and north. This power loop provides primary power to the TSF4 reclaim pumps and provides a backup power source for booster pumps serving the leach pads and other equipment on the pit perimeter. Approximately 8,000 ft of cabling will be moved outside of the planned pit perimeter, requiring 32 new poles and modification to approximately 6 existing poles. An existing line isolator (circuit breaker) will be relocated, and a new line isolator will be installed to permit a staged implementation and improved access to the power loop.

#### Pit Dewatering (modifications staged to meet the needs of operations)

The El Niño booster station and piping limit mine development to the south. The work plan considers the installation of new 18-inch HDPE piping, supplemented with 18-inch stainless steel in areas subject to high stresses. The corrosive (acidic) nature of the pit water precludes the use of cheaper alloys and mild steels. Upgrades and modifications to the electrical infrastructure, in-pit pumping network, and booster pump station are included in the plan.

#### Explosives Plant

The two magazines will be moved to an isolated area on the western edge of the pit with easy access to the pit.

The ammonium nitrate silos and the emulsion plant will also be moved to an area 900 ft south of the powder magazines, on the pit's west side, to ensure convenient access and continued safety for mine operations and support personnel.

#### South Perimeter Drainage

PVM uses catchments, wells, and diversions to capture or divert stormwater before it enters the pit. The South Hill catchment, Rick's Catchment, Rick's Caisson, and drainage to L Catchment will be disturbed by mine development. A channel is under consideration to divert the drainage flows that would otherwise be captured by the catchments.

### 18.6.2 Maintenance

To increase the effectiveness and efficiency of the operation, all work on mine equipment will be moved to the Main Shop. To accommodate this, upgrades will be necessary to the lube distribution system, truck wash, tire handling facility, and welding shop. Capital expenditures for these projects are expected to be required from 2015 to 2017.

#### Main Shop

The upgrades to the Main Shop support the relocation of the maintenance and operations activities to the central processing area of PVM. Upgrades required for this consolidation include the following.

- Installation of a new efficient air compressor system that allows the existing compressor to operate in standby mode.
- Construction of new offices to house mine operations staff.
- Relocation of the existing light-vehicle fueling station from below the primary crusher to the Main Shop area.
- Installation of steel rails in one maintenance bay to facilitate work on tracked equipment.
- Construction of a new lubrication facility, modifying the northwest corner of the Main Shop, to include the following.
  - Relocation of one lube tank from the North Barn
  - Installation of new lube hose reels in two aisles (to support four bays)
  - Installation of a pumping, storage, and distribution system

#### Truck Wash

The existing wash pad near the Main Shop will undergo earthwork improvements to widen the surrounding area and concrete wash pad and to improve mining fleet access. Light-vehicle traffic will be limited in this area. Diesel-powered high-pressure sprays (water cannons), an enclosed boiler, and platforms will be added to the existing facility.

### Tire Maintenance

A new tire maintenance facility (tire pad) with an approximately 3,100 $ft^2$ concrete pad will be built between the Main Shop and its wash pad. Compressed air and power lines will be trenched to the tire pad from the Main Shop. Tires and rims will be stored nearby. An air dryer and a nitrogen receiver will be installed at the tire pad.

## 18.7 Power

SRP, a major electrical utility serving residential and business customers in central Arizona, provides electrical power to PVM. A substation is located near the mill. SRP's main hydroelectric generating station on the Salt River operates at Mormon Flat, 20 miles northwest of the mine. Main distribution loops within the mine property are 13.8 kV or 15 kV, with service distribution at 480 V typical.

Natural gas service is not available at the mine. Gasoline and diesel fuel supplies are trucked to the property.

## 18.8 Community Support Services

The nearby tri-city communities of Globe, Miami, and Claypool, Arizona, have a mature support services infrastructure. Additional services are available within 100 miles in the Phoenix and Tucson metropolitan areas.

Globe is the Gila County government seat. Emergency Management, an office of the county government, is responsible for the coordination of county resources and responsibilities for the protection of life, property, and the environment in the event of an emergency or disaster.

The Cobre Valley Regional Medical Center is a 25-bed Critical Access Hospital located in Globe. It is a modern, full-service, 24-hour emergency service, acute-care facility that has served the local area for 100 years (Cobre Valley 2014).

In addition to fire protection and public education, the Tri-City Fire District also provides emergency medical services, technical rescue services, and ambulance transport within the community and to Phoenix and Tucson (Tri-City 2014).

Normal communications channels using land-based facilities, radio, cell phone, microwave, and satellite are available.

# 19.0 MARKET STUDIES AND CONTRACTS

## 19.1 Pinto Valley Marketing

### Copper

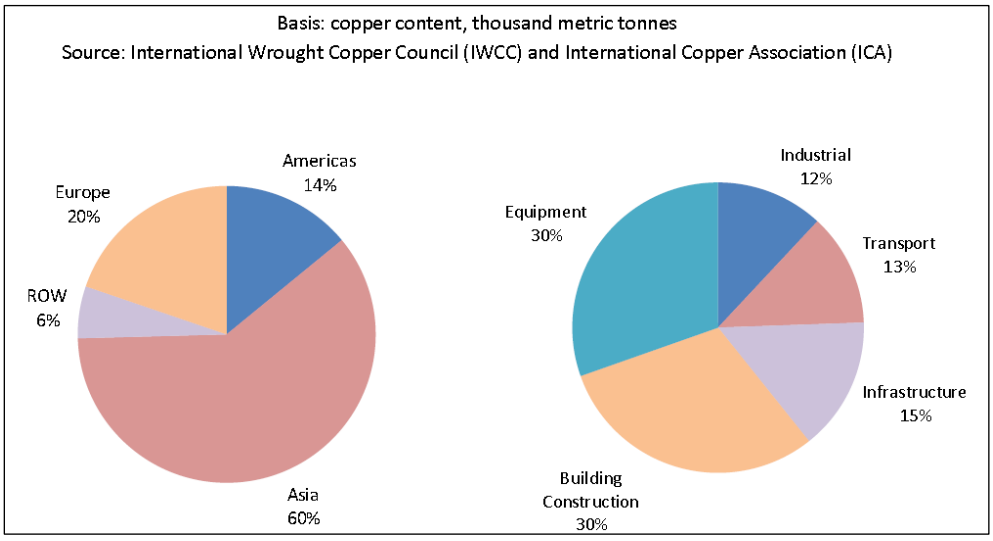
Copper and copper-based alloys are used in a variety of applications that are used to increase standards of living. Its continued production and use is essential for society's economic development.

Copper is an important contributor to the national economies of mature, newly developed, and developing countries. Mining, processing, recycling, and the transformation of metal into a multitude of products creates jobs and generates wealth. These activities contribute to building and maintaining a country's infrastructure and create trade and investment opportunities. Copper will continue to contribute to society's development well into the future.

### Global Production

Since 1900, demand for refined copper increased from less than 0.5 Mt to over 20 Mt in 2012 at a compound annual growth rate of 3.4% per year. 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 – 2012**



### Copper Concentrate Market

Overall, demand for copper concentrates is forecast to increase from 12.9 Mt in 2012 to 20.5 Mt in 2025. This represents a compound annual growth rate of 3.7%.

### Supply

After several years of slow growth, copper mine production recovered in 2012, advancing by 4% to 16.7 Mt contained copper. High prices have encouraged mine project development, and as a result, mine supply is set to enter a period of strong growth due to new projects scheduled to come on stream over the next 5 years. However, supply disruptions continue to arise unexpectedly, which will impact forecasted increases in production. For example, government legislation recently has halted the export of copper concentrate from Indonesia.

### Copper Prices

The 2012 long-run marginal cost (LRMC) of copper equals $6,595/t or $2.99/lb. LRMC is determined by the operating costs of the marginal project at a given point in the future. It is assumed the supply will be equal to demand in the long run, because it is not possible to accurately forecast cyclical surpluses and deficits beyond a 5-year time horizon. As such, the operating costs of the producer required to fulfill the final proportion of demand are equal to the price at that point. By modeling the operating costs of the project as if they were producing in 2012, the LRMC can be calculated in real 2012-dollar terms. Thus, this provides the long-run point toward which prices will converge in the future.

### Treatment Charge / Refining Charge

China is the leading importer of copper concentrate and has the greatest capacity for smelting and refining. As a result, China will set the marginal cost for smelting and refining. This is due to a desire to reduce the country's reliance on imports of value-added copper products by installing domestic processing capacity.

The annual treatment charge / refining charge TC/RC in 2013 was $70/7 or 18 cents/lb on a combined basis. For 2014, the annual TC/RC has moved up to $92/9.2, or 23.6 cents/lb, on a combined basis and sufficient to attract investment in smelting and refining capacity in China and to sustain continued demand for copper concentrate. TC/RC costs used by Capstone are $85/8.5.

### Pinto Valley Mining Corporation

PVM produces copper concentrate (about 240,000 tons in 2014) and produces about 1.4 M lb Mo and approximately 6.3 M lb Cu cathode through SX-EW.

### 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. Contracting strategy will be based on the existing contracts of varying tenure. 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 elements (copper and silver), less applicable TC/RC.

For the molybdenum sulfide concentrate, interested buyers have submitted terms for the full annual production. Negotiations are currently underway to finalize the sales agreement. 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.

The full production of copper cathode has been sold through a competitive tendering process to a single buyer. Cathode pricing is based on the London Metals Exchange pricing for copper cathode at the applicable quotational period for the monthly deliveries.

# 20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

Section 20 describes factors related to the project and the results of environmental studies. The section summarizes the environmental issues identified on site, existing permit status and environmental monitoring, and permitting efforts that may be required during the PV2 phase. PVM's health and safety, and social and community programs are also discussed.

## 20.1 Current Environmental and Regulatory Context

PVM has well-established health, safety, and environmental procedures and protocols that adhere to regulatory requirements and internal corporate guidance to reduce impacts to the environment and to provide a safe work environment for employees and contractors.

PVM currently operates within the terms and conditions associated with its many environmental permits, licenses, and approvals. While the key permits are all in place, PVM is currently undergoing a review of a Plan of Operations with the USFS, which is essentially a consolidation of prior authorizations and encroachments on federal lands. As such, the permits required for the execution of the PV2 mine plan are in place, although some amendments will be required, as described in Section 20.3.

No upgrades or major processing changes are foreseen to the milling or leaching circuits, and no new infrastructure is proposed, including any disturbance or improvement that may impact drainage features. The existing water sources are capable of supplying adequate water, and no substantive changes are anticipated to surface water management.

### 20.1.1 Key Environmental Statutes and Regulations

The principal permitting agencies providing oversight of the operation and closure of PVM are ADEQ and the USFS – Tonto National Forest (TNF). Other agencies include the Arizona Department of Water Resources (ADWR), the ASMIO, and potentially the US Army Corps of Engineers (USACE). A brief overview of specific environmental programs and permits / actions required by the federal government is shown in Table 20-1; those required by the State of Arizona are shown in Table 20-2.

The development, operation, and closure of mining and processing facilities in Arizona are subject to environmental regulations addressing groundwater, surface water, and stormwater management; air quality; well installation; water withdrawal from state aquifers; waste handling and disposal; handling and storage of toxic substances; surface reclamation; and cultural and biological resources. Specific operational, maintenance, and closure / post-closure requirements are detailed in general or individual permits.

### 20.1.2 Site Environmental Permits and Other Licenses

The current key environmental permits are shown in Table 20-1 and Table 20-2. These permits were transferred by the regulatory agencies from BHP to Capstone following the property sale and submittal of required information. The permits are in effect for the life of mine (or until substantive changes are needed), with the exception of the Arizona Pollutant Discharge Elimination System (AZPDES) Multi-Sector General (MSG) discharge permit, which is included in a permit program renewed by ADEQ every 5 years. Also shown in Table 20-1 and Table 20-2 are other regulatory actions, approvals, and requirements that were evaluated during the review of PV2 activities.

Other minor permits, registrations, and licenses required for operation are in effect and include a federal MSHA registration number, an Arizona state mining ID number, hazardous materials and hazardous waste ID numbers, a solid waste management inventory number, waste tire cell registration, wastewater septic permits, radio and other communications licenses, blasting operator registrations, exploration / well drilling permits, and laboratory / nuclear instrumentation licenses and registrations.

### 20.1.3 Site Environmental 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 is coordinated by the environmental manager. All employees receive environmental orientation training related to air quality, groundwater, stormwater, and regulatory programs; environmental department staff receive specialized environmental and regulatory training from internal and external sources.

### 20.1.4 Site Health, Safety, and Industrial Hygiene Program

PVM is subject to health and safety regulations under the supervision of MSHA, ASMIO, Arizona Department of Transportation, and other federal and state agencies. New-miner training, annual refresher training, and training certification to operate specialized equipment is handled internally at site.

Safety incidents and accidents are reported via an integrated internal notification system. Incidents, property damage, and injuries are investigated by PVM's HSEC department and other relevant internal personnel to review the causes and develop preventative and remedial plans.

Safety procedures and personal protective equipment requirements for routine work, emergency reporting and response, and other protocols and training, such as risk review procedures and environmental training, are well documented and reinforced by the HSEC department as part of employee training. The HSEC department also oversees industrial hygiene-related programs, such as hearing conservation programs that test employee hearing, monitor noise and particulate levels, and work with operations, as required, to mitigate excessive noise and dust. The department activities include a respiratory fitness program, fatigue management, and vibration analyses for equipment operators.

Health-related programs managed primarily by PVM's Human Resources department include new-employee medical screenings, annual medical examinations, the "back-to-work" assessment following an employee illness or injury, and the drug and alcohol program.

### 20.1.5 Site Characterization Data and Studies

Numerous site characterization studies have been completed at PVM to support operations, environmental permitting efforts, closure planning, and other investigations. A summary is provided below.

#### Climate

PVM has two on-site weather stations. Data monitored includes daily precipitation, maximum and minimum temperatures, maximum and minimum percent relative humidity, wind speed, and pan evaporation.

#### Groundwater

Characterization of groundwater and seeps / springs began during the late 1970s to mid-1990s to support operations and permitting activities and is an ongoing part of the routine compliance monitoring program associated with the APP.

### Surface Water, Process Water, Stormwater, and Wastewater

The surface water characteristics of Pinto Creek upgradient and downgradient of PVM have been established. The characteristics of process water, stormwater, and water from the wastewater treatment plant (WWTP) are well known as a result of routine monitoring.

### Air Quality

Ambient and operations air quality characteristics have been established.

### Geotechnical

The characteristics and stability of all engineered facilities at PVM have been reviewed at various periods over the life of operations by qualified professionals. This includes the benches and ramps in the open pit, the Low-Grade Ore Leaching Pile, the TSFs, and waste rock storage facilities. Geotechnical evaluations have also been performed to evaluate potential sources of cover material.

### Geochemical

Studies evaluating the geochemical characteristics of the geologic formations, borrow sources, pit walls, waste rock, and tailings materials have been performed.

### Biological Resources

Studies evaluating the presence of threatened and endangered species listed for protection under the Endangered Species Act (ESA) have been performed. Previously surveyed areas include patented mine property and those areas of unpatented claims adjacent to the mine where future expansion may potentially occur.

### Cultural Resources

Cultural resources surveys have been completed on both private and public lands in the vicinity of PVM, including inventories completed in support of application to the TNF for prior and anticipated encroachments on federal lands. Surveyed areas include patented mine property and those areas of unpatented claims adjacent to the mine where future expansion may potentially occur.

**Table 20-1. Federal Permits, Approvals, and Requirements**

| Item | Regulatory Agency | Description |
|---|---|---|
| Plan of Operations | US Forest Service | Ongoing review of the Plan of Operations will remain unchanged, but it is considered here should opportunities related to PV2 be pursued by amending the scope of the current review. A Plan of Operations is required for mining activities on federal lands (within valid unpatented mining claims). The Plan of Operations, under current USFS review, incorporates previously authorized federal land uses covered under prior right-of-way authorizations, special-use permits, and prior Plan of Operations. New disturbance or changes to existing activities on federal lands require that a new Plan of Operations be filed or that the Plan of Operations currently under review be amended with the TNF. The final Plan of Operations is prepared after USFS completion of appropriate NEPA review. |
| Clean Water Act (CWA) Section 404 Permit | US Army Corps of Engineers | May be required for new canyon fills for tailings, leach stockpile, or waste rock deposition and road or utility crossings of washes. |
| NEPA Review (Environmental Impact Statement [EIS] or Environmental Assessment [EA]) | Federal agencies in administrative branch of the federal government | Ongoing NEPA review for previous federal land use authorizations is underway. Further NEPA review is not required for PV2 as presently configured, but is considered here should opportunities to further optimize PV2 require federal authorization. NEPA review is required for federal actions, including plans of operations, special-use permits, and/or CWA Section 404 permits. Activities not categorically excluded from NEPA require preparation of an EA or EIS. NEPA compliance is frequently the critical path for project timeframes. |
| Endangered Species Act | US Fish and Wildlife Service | The ESA applies to both public and private lands. On private lands, compliance with Section 9 of the ESA is required and prohibits the “take” of any federally listed threatened or endangered species (excluding plant species). Although not currently contemplated for PV2, should opportunities to further optimize PV2 require a federal permit, the applicable federal agency must comply with Section 7 of the ESA. |
| National Historic Preservation Act Section 106 Consultation | Lead federal agency and Arizona State Historic Preservation Office (SHPO) | Not required for PV2 as presently configured, but considered here should opportunities to further optimize PV2 require a federal permit and potentially have impacts to register-eligible historic properties. Under these circumstances, Section 106 consultation between the federal action agency and the SHPO is required. These projects require approval, usually from the SHPO, of a treatment or mitigation plan prior to impacts to archaeological sites. As part of this process, consultation with interested Native American tribal entities is also required. In the absence of a federal nexus, state laws related to burials and funerary objects would apply. |
| Federal Lands Reclamation Plan and Reclamation Bond | Tonto National Forest | A federal lands reclamation plan and reclamation bond are not required for PV2 as presently configured, but are considered here should opportunities to further optimize PV2 require additional federal land use. Reclamation requirements related to Plan of Operations currently under NEPA review will be addressed following final approval of the Plan of Operations. |

Compiled by WRI and SRK in 2014

**Table 20-2. State Permits, Approvals, and Requirements**

| Item | Regulatory Agency | Description |
|---|---|---|
| Aquifer Protection Permit | Arizona Department of Environmental Quality | Required for facilities that have the potential to discharge pollutants to an aquifer. Compliance monitoring and reporting obligations are in place. Permit program attempts to minimize discharges to the best extent possible through siting selection process, engineered controls, and best-management structures and practices. The APP will require modification if additional discharging facilities are proposed or if the assumptions or parameters for existing facilities change. |
| Arizona Pollutant Discharge Elimination System | Arizona Department of Environmental Quality | Required for industrial facilities that discharge from any point or non-point source into waters of the United States. Existing permits are in place. |
| AZPDES Multi-Sector General Stormwater Permit | Arizona Department of Environmental Quality | Required for categorical sites (Sector G1: Copper ore mining and dressing). This is a general permit that requires a site-specific Stormwater Pollution Prevention Plan. |
| Clean Water Act Section (CWA) 401 Certification | Arizona Department of Environmental Quality | ADEQ is required to certify that all CWA Section 404 permits comply with state water quality standards. Required only if a Section 404 permit is needed. |
| Applicable Arizona Antiquities Regulations: State Historic Preservation Act | Arizona State Historic Preservation Office | State regulation does not require proactive treatment of cultural sites by a landowner on private lands, but they do prescribe specific obligations with regard to the discovery of human remains during ground-disturbing activities. Any such discoveries must be treated in accordance with Arizona Revised Statutes. The law criminalizes the intentional disturbance of human remains or funerary objects on private land. For unintentional disturbance on private land, the statute defines a process for reporting, treatment, and disposition. |
| Clean Air Act Permit | Arizona Department of Environmental Quality | Required for stationary sources of air pollution that permit priority pollutants above regulated thresholds. This permit is already in place. Any modifications to equipment or pollution controls would likely require a permit modification. Permit modification is not anticipated for PV2. |
| Dam Safety Permit | Arizona Department of Water Resources | Required for dams of jurisdictional size (excluding TSFs). Permit requires construction-level (100%) plans. |
| Water Withdrawal Permits | Arizona Department of Water Resources | Covers the Gold Gulch 1A Dam. Required for water withdrawal for potable water and mineral processing; permits required to install pumps in groundwater monitoring wells. |
| Mined Land Reclamation Plan | Arizona State Mine Inspector's Office | Required for mine disturbance on private and state land. Existing plan will need to be modified to accommodate any changes in mine operations associated with PV2. |

Compiled by WRI and SRK in 2014.

## 20.2 Environmental Issues, Monitoring, and Management

The monitoring and management at PVM for various environmental issues is driven by federal and state regulatory requirements and Capstone's commitment to environmental stewardship and reducing impacts of the operation on the environment. A brief summary of the PVM monitoring and management program is presented next.

### 20.2.1 Groundwater

Water quality analyses of various constituents are required on a quarterly, annual, biennial, and triennial basis from designated seeps / springs, point-of-compliance (POC) and alert-level groundwater wells, and discharge monitoring points. Routine self-monitoring report forms are submitted to ADEQ with the results of water quality monitoring and site inspections.

Compliance APP reporting requires an annual demonstration of the adequacy of pit containment and that the open pit is a terminal sink with groundwater flowing inwards toward the open pit. Cross sections demonstrating the inward hydraulic gradient of groundwater are prepared annually using measurements of water levels from monitor, dewatering, and alert-level and POC wells. Additionally, a more comprehensive groundwater report is required by ADEQ every 5 years to assess adequacy of POC wells and the hydraulic capture in the open pit.

### 20.2.2 Surface Water, Stormwater, Process Water, and Wastewater

Routine sampling of surface water in Pinto Creek was performed for many years as part of the AZPDES compliance monitoring, but is no longer required by ADEQ.

Monitoring and sampling of stormwater discharges at specific outfall locations and seeps that enter Pinto Creek is required by AZPDES Permit No. AZ0020401. Discharge limits are set for trace metal constituents, pH, and hardness; other trace substances are also monitored with monthly average and daily maximum assessment levels established for each outfall. Whole effluent toxicity is monitored at Outfall 005. Seepage zones are mapped, and site facility maps are updated annually to show constructed and natural drainage structures, pipelines, pump stations, pump capacities, and all monitoring points associated with Clean Water Act (CWA) compliance activities. Best-management practices have been established in the permit to inspect the surge capacity and freeboard on a routine basis at process ponds and impoundments designated as "no-discharge," and to conduct routine

inspections and maintenance of berms, ditches, dikes, dams, ponds, containment structures, pipelines, and pump stations.

No compliance water quality monitoring requirements exist for process solution.

The 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 Dust Abatement

As part of the mine's synthetic minor air quality permit, specific controls, including scrubbers 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 mine's 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 HSEC department maintains a Hearing Conservation program and works with operations, as appropriate, to mitigate excessive noise.

### 20.2.5 Hazardous, Regulated, and Solid Wastes

Hazardous and regulated wastes, such as motor oil, transformer oil, paints, aerosols, solvents, fluorescent tubes / bulbs, batteries, spent lead anodes, and lead anode flake, are collected and stored on site prior to shipment to a recycling facility or permitted waste disposal facility, as appropriate. Chemicals brought on site to be used by vendors or contractors must have approved material safety data sheets on file and approved in advance of bringing them on site by PVM HSEC personnel.

A solid-waste landfill facility is located within the footprint of Northside Dump 9.3, located west of the open pit and north of the North Barn, to dispose of construction debris. PVM personnel use 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 nonhazardous shop refuse (food, paper, maintenance, trash) is collected and deposited in large waste bins and roll-off containers for weekly collection by a local hauler for transport to an off-site disposal facility.

### 20.2.6 Waste Rock Storage

There are eight potentially active waste rock storage facilities at PVM, some of which have been consolidated over time into single facilities. One facility (Southside Dump 14) was closed and is in post-reclamation status, and three facilities (North Dump 19, Extension Dump, East Dump) have been permitted, but not yet constructed. Each facility has a specific design storage capacity limit and is to be managed to reduce acid rock drainage. Acid-base accounting and synthetic precipitation leaching procedures analyses are required for every 10 M tons per active dump. The results are reported to ADEQ on an annual basis (if sufficient dumping occurred) and are intended to be used to support future closure evaluations.

### 20.2.7 Tailings Storage

A maximum disposal of 32 M tons of tailings material 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. TSF1/2, TSF3, and TSF4 are inspected quarterly and after rainfall events greater than 3 inches in 24 hours for evidence of crest failure, toe slippage, substantial cracks, and erosion features. Associated seepage to drains and caissons are to be kept free of debris, sediments, vegetation, and other obstructions.

### 20.2.8 Process Ponds and Stormwater Ponds / Catchment Basins

The PLS and raffinate ponds are inspected daily and weekly, respectively, for evidence of cracks, holes in liners, erosion, subsidence, sloughing, overtopping, seepage, or other structural damage to pond sides and berms. Stormwater ponds, basins, and catchments are inspected weekly and after strong storm events for evidence of seepage, damage to berms, erosion, subsidence, sliding, or overtopping of freeboard requirements. The inspections results and required repairs are documented in a log book and reported to ADEQ on an annual basis.

### 20.2.9 Remaining Evaluations

There are no outstanding evaluations required to complete PV2. Capstone may elect to further optimize operations that require additional permitting. An ongoing assessment of cost and schedule implications will determine if PV2 is completed on private lands as presently planned or if it will include additional TNF authorizations.

## 20.3 Environmental Permit Review Related to PV2

The following analysis is a brief review of current federal and state permits and requirements with respect to operations and closure planning. Any potential actions or permit amendments that will be required by the updated LOM changes and/or changes to waste disposal management are noted.

### 20.3.1 Federal Permitting

#### Forest Service Approval of a Mine Plan of Operations

As presently configured, PV2 will not require use of federal lands beyond existing authorizations and encroachments that are currently included in the Plan of Operations (No. 03-12-02-10) under NEPA review by the TNF. The authorizations and encroachments in the Plan of Operations include tailings impoundments, waste rock dump, power lines, well sites, water lines, and other appurtenances to the ongoing PVM operations. Pinto Valley Mining Corp. is currently evaluating the opportunity to modify the PV2 configuration to include minor additional expansions of TSF3 and TSF4 onto USFS lands; these modifications would be incorporated into the Plan of Operations currently under NEPA review.

As previously stated, Capstone may elect to further optimize operations by adding minor encroachments on lands administered by the TNF. Minor encroachments onto public lands could reasonably be requested under one of two avenues: (1) an amendment to the proposed Plan of Operations No. 03-12-02-010, or (2) as a separate request to the TNF.

### National Environmental Policy Act

PV2 can be developed without federal approvals, licenses, or permits that would require NEPA compliance review. However, Capstone is considering opportunities that would result in encroachments onto public lands for PV2 to enhance operational efficiencies that would trigger these laws. NEPA review, if applicable, may be completed as part of ongoing NEPA review for the proposed Plan of Operations No. 03-12-02-010.

### Clean Water Act

Drainage features that may be considered waters of the United States are subject to regulation by USACE under the authority of Section 404 of the CWA. Surface water features within the planned footprint of PV2 are likely not subject to jurisdiction under the CWA, though a formal waters of the United States determination has not been submitted to USACE for verification.

Should the drainage features in the areas of land required be considered jurisdictional, then discharge of fill material in these areas will likely require a CWA Section 404 permit, which would trigger an NEPA review by USACE, as well as consultation requirements under Section 106 of the National Historic Preservation Act.

### Endangered Species Act

Undisturbed areas at and immediately adjacent to PVM are not known to support any species currently listed as threatened or endangered, nor are any of these areas designated as critical habitat for federally listed species. More broadly, listed species are known to occur in the general vicinity of PVM (e.g., Arizona hedgehog cactus), and the list of species protected by the ESA continues to expand. Based on previous survey and screening efforts, PV2 is not expected to result in adverse impacts to any species currently listed by the federal ESA; therefore, the obligations of the ESA are not triggered. Absent a federal nexus, there is no requirement for federal agency consultation with the US Fish and Wildlife Service (USFWS) under Section 7 of the ESA.

Although not currently contemplated for PV2, should opportunities to further optimize PV2 require a federal permit, the applicable federal agency must comply with Section 7 of the ESA. Section 7 of the ESA requires that, for any federal agency action, the permitting authority must evaluate the potential impact of a project to federally listed threatened and endangered species and their critical habitat (includes plants). If a federal agency with authority over PV2 determines that the project may affect a listed species or critical habitat, consultation with the USFWS will be required to ensure that the project will not jeopardize the continued existence of any listed species or adversely modify critical habitat.

### 20.3.2 State Permitting

Significant Arizona state permits issued by the three divisions of ADEQ (Water Quality, Waste, and Air) and the MLRP approved by the ASMIO were reviewed to assess whether amendments will be required as a result of PV2 phase of activities. The findings on the key permits are presented below.

#### Clean Air Act Permit

PVM currently operates under a Clean Air Act (CAA) Class II "synthetic minor" air quality control permit. A synthetic minor permit requires that operations stay within voluntarily accepted operating constraints so that emissions do not exceed established major-source thresholds. The PVM air permit was renewed in 2012, extending the term of the permit for 5 years through 2017, and would be renewed prior to expiration. The operation and development assumptions identified for PV2 are not expected to trigger modification of the CAA permit. PVM will continue to operate within voluntarily accepted limits. There is no replacement of equipment anticipated for processing components considered for PV2. Therefore, no additional baseline or modeling studies have been necessary. Additional air permitting efforts, such as modeling, may be required as part of the NEPA review, should opportunities to further optimize PV2 require use of additional federal lands.

#### Aquifer Protection Permit

An amendment to the existing APP will likely be required to address facility expansions within the private property boundary, as shown in Figure 20-1. This includes the following facilities.

*Low-Grade Ore Leaching Pile*

This will include disposal of very low-grade material on the eastern edge of the Low-Grade Ore Leaching Pile that will enlarge the eastern footprint of this existing facility by approximately 32 acres. The foundation geology in this area is crystalline igneous basement rock similar to that found on the eastern side of the existing facility. The materials to be placed on this expanded footprint area are identical in rock type, mineralization (copper and iron minerals), and associated gangue minerals (i.e., rock-forming minerals and vein minerals such as quartz and calcite) to those currently being placed on the facility. A physical survey of this area by WRI in March 2014 found no seeps, springs, adits, or underground workings.

*Northside Dump 9.3*

This existing waste rock facility is located west of the open pit, east of the consolidated TSF1/2, and north of the North Barn area, which is a disturbed laydown area. The existing Northside Dump 9.3 and expansion area lies primarily within the No.1-2 Tailings Basin as designated on the MSG Stormwater Pollution Prevention Plan (SWPPP) map. Expansion of this facility, if completed, may require the closure of the existing small solid-waste landfill located within the southern portion of Northside Dump 9.3. An extension of the Northside Dump 9.3 to be used as a marginal ore stockpile is expected to be needed by January 2016.

*Castle Dome Marginal Dump*

A new 35-acre marginal ore storage facility is proposed to be located on the southern rim of the open pit, partially overlying the closed historic Castle Dome plant facility area. The Castle Dome Marginal Dump would be located within the Open Pit Basin so that stormwater runoff would report to the Open Pit Basin. Disposal of material to this facility is expected to be needed in early 2015. A physical survey by WRI in March 2014 found no seeps, springs, adits, or underground workings in this area.

Preparation of an application for a “significant amendment” to the existing APP will begin in the early second quarter of 2014 and be submitted to ADEQ Groundwater Division in the second quarter. The application review process by ADEQ is expected to take approximately 9 to 12 months and will require a public notice and 30-day comment period.

**Arizona Pollutant Discharge Elimination and Multi-Sector General Permits**

No changes to the existing surface water and stormwater management permits, including new outfall monitoring points, will be required related to the PV2 phase of mining. PVM’s MSG SWPPP is updated annually to update the location of seepage zones, pump locations, stormwater berms / diversion ditches, and any new or expanded waste management facilities, as needed.

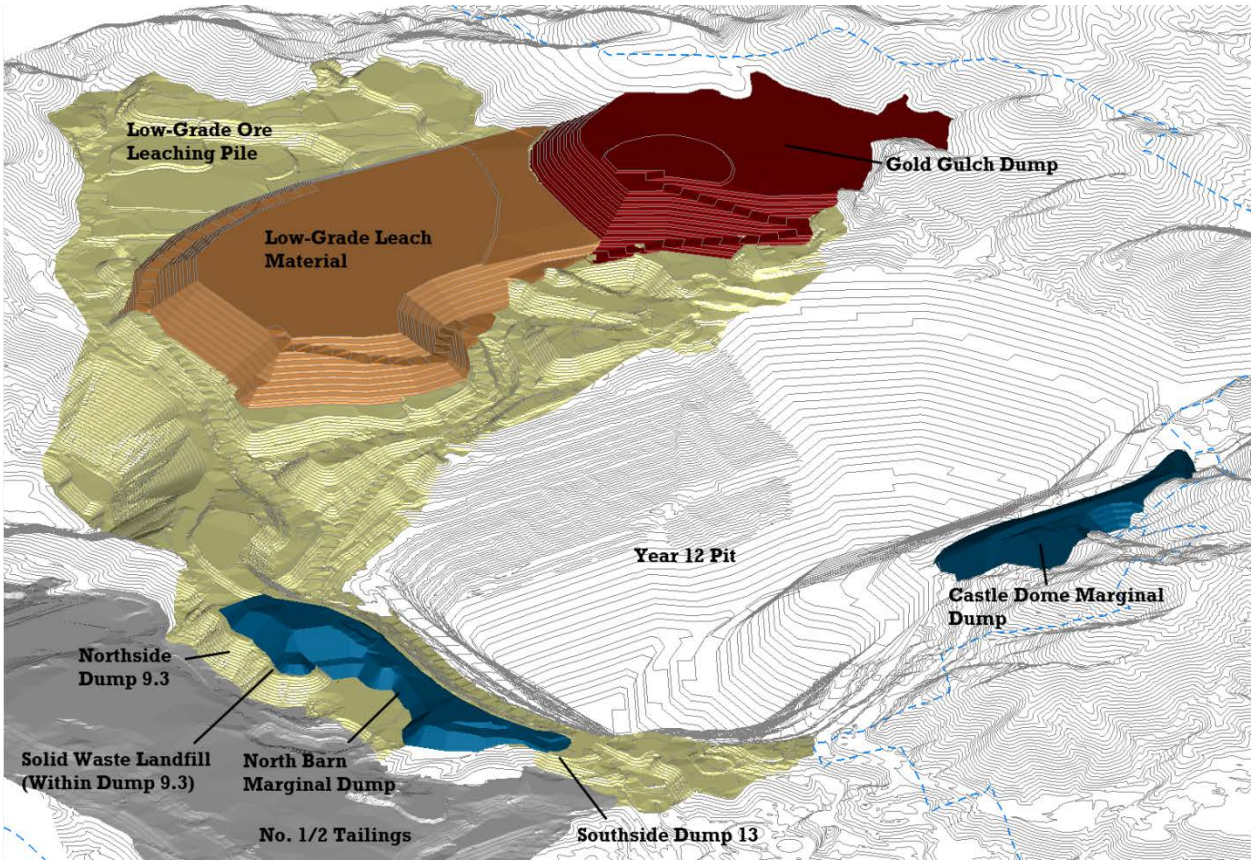
Stormwater runoff from the new planned Castle Dome waste rock storage facility will fall within the non-discharging Open Pit Basin as designated on the SWPPP map; stormwater run-on flows to this new dump will be minimized through use of berms and/or ditches. Stormwater runoff flows related to the expansion of the Low-Grade Ore Leaching Pile along the eastern portion of the facility will be managed within the Open Pit Basin and Gold Gulch PLS Pond No. 1, as is currently done with the stormwater flows and PLS draindown from this facility. Stormwater flows from the planned dump expansion of Northside Dump 9.3, west of the open pit, will report to the non-discharging open pit watershed and/or be managed downgradient of the No. 1-2 Tailings Basin, depending on placement location.

### State Historic Preservation Act

State regulations for archaeological features that may be affected for PV2 are limited to the Arizona State Historic Preservation Act. Applicable elements of state regulation do not require proactive treatment of cultural sites by a private landowner; however, they do prescribe specific obligations with regard to the discovery of human remains during ground-disturbing activities.

Previous cultural resources surveys have been completed to support ongoing planning on both private and public lands in the vicinity of PVM. Additional cultural resources surveys are not required for PV2. There are known archaeological sites located on private lands within the footprint of the PV2 expansion. Covering these sites is not commonly considered a violation of state statutes; however, if any excavation work will be done in preparation for construction of the dams or other activity and human remains are encountered, work in these areas must be stopped until the remains can be properly recovered and repatriated. A detailed assessment of these cultural resources was conducted, and a treatment plan was developed to provide the opportunity to conduct voluntary recovery at these sites.

**Figure 20-1. Perspective View of Life-of-Mine Leach and Waste Rock Facilities**




Source: View generated in MineSight looking N15E using LOM dump and leaching plan.
Yellow = existing facilities including leaching and waste rock facilities
Orange = new ROM leach material placed within current facility footprint
Blue = marginal grade stockpile material
Red = proposed new waste facility overlying a retired portion of existing leaching facility

### 20.3.3 Mined Land Reclamation Plan

PVM is subject to closure and post-closure reclamation requirements by ASMIO. The original MLRP for PVM was approved in 1998; reports tabulating new disturbance and new reclamation completed during the year have been filed annually with ASMIO. The MLRP was updated by BHP in June 2013 to update the reclamation techniques used, revise the estimate of the mine life and LOM disturbance, identify several new facilities built with existing disturbed acreage since 1998, and provide reclamation costs that were escalated from 1998 to 2011 dollars.

## 20.4 Mine Closure and Reclamation

The plan for end-of-life operations, closure, and post-closure activities consists of a general approach to the methods and goals for site closure and reclamation. Conceptual closure designs and cost estimates are based on the current LOM plan for mining and waste disposal facilities. Although there are some general aspects in the regulatory agencies' approach to site closure and reclamation, the agencies (USFS, ADEQ, ADWR, and ASMIO) also focus on their specific regulatory responsibilities. The sections below briefly summarize the existing closure and reclamation plan and conceptual-level closure and post-closure costs.

### 20.4.1 Closure and Reclamation Plan

Updated conceptual closure plans were submitted to ADEQ (BHP 2013b, MWH 2013) and to ASMIO in 2013 (BHP 2013a). The general goal of the ADEQ APP closure plan is to prevent and/or minimize post-closure degradation of groundwater. General closure activities include removing unneeded buildings, structures, pipelines, ponds, or sumps; preventing the accumulation of water in buried, closed structures, ponds and sumps; regrading and covering discharging facilities to prevent continued infiltration of precipitation; and management of the draindown of spent PLS in the open pit. The closure period, including demolition, regrading, revegetation, and earthworks construction, is estimated to take 3 years. Post-closure maintenance is estimated to take a minimum of 10 years until site conditions are stable; post-closure sampling and monitoring is required by the APP to last for 30 years.

The general goal of the MLRP submitted to ASMIO is to ensure public safety and reclamation of disturbed land. A public-access county road crosses a portion of the PVM property and would likely remain open for fire control access and recreational use by the public. Access by the public to the open pit area, however, will be restricted by use of berms, gates, and/or fencing. Disturbed areas would be regraded to shed stormwater, covered, and revegetated with an approved seed mix. Conventional revegetation methods would be used, including a combination of dry seed dispersal with soil amendments (organic materials and/or fertilizer) and hydroseeding.

Closure considered for land use on USFS lands is also discussed in plans of operations. A federal lands reclamation plan and reclamation bond is not required for PV2 as presently configured, but could be prepared as part of the final USFS approval following NEPA review should opportunities to further optimize PV2 require additional federal land use.

### 20.4.2 Closure and Post-Closure Costs

Updated closure and post-closure costs were submitted by BHP to ADEQ and to ASMIO in 2013 prior to the sale of PVM to Capstone. The costs were itemized by permit program, including the APP regulated by ADEQ and the MLRP regulated by ASMIO.

The APP-related closure costs of $60,242,837 (submitted in August 2013) 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 are not needed for post-closure monitoring would be abandoned.

The MLRP-related closure costs of $26,943,837 (submitted June 2013) address closure activities related to ensuring public safety, facility regrading, and revegetation.

The capital costs associated with closure and reclamation of PVM are estimated at $129.1 M and are spread out from 2026 to 2055. This is based on the $87.1 M currently indicated by the APP and MLRP for the current mine, and an additional $42 M to cover an additional reclamation bond related to the approved Plan of Operations currently under review by the USFS; owner's costs, including internal G&A and labor to support the 30-year post-closure period; and the studies to support permit amendments, increased land disturbances, and other impacts of executing the 12-year mine plan. Capstone and SRK have reviewed the submitted closure and post-closure costs of $129.1 M 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. 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. All levels of management and staff are involved in community involvement initiatives, though the HSEC department manages and tracks communication with stakeholders, ensuring timely responses to community needs.

# 21.0 CAPITAL AND OPERATING COSTS

## 21.1 Initial Capital Cost Estimate

PVM is an existing operating mine and therefore does not require initial capital costs. All capital expenditures starting from 2014 will be treated as sustaining capital costs. Costs are in fourth-quarter 2013 dollars.

## 21.2 Sustaining Capital Cost Estimate

Sustaining capital for PVM over the 12-year mine life totals $187.9 M. This capital is allocated to mine equipment, mine infrastructure and maintenance, mill, SX-EW plant, tailings and water, site infrastructure, and miscellaneous upgrades and engineering studies. Owners' costs for 2014 and 2015 have been estimated to be $1.5 M for the execution of the project. This cost is in addition to ongoing costs defined in Section 21.4.4 – General and Administrative Operating Costs.

### 21.2.1 Mine Equipment Requirements

PVM currently has a fleet of 15 Caterpillar 789D and 3 Caterpillar 994 front-end loaders and associated auxiliary equipment. The mine plan indicates that additional equipment will be required to move waste material during the mine life. The equipment requirements are discussed in Section 16.5. Equipment prices are based on analysis of recent sales in the mining industry. The capital costs associated with the mining equipment are detailed in Table 21-1.

**Table 21-1. Mine Equipment Capital Costs**

| Mine Equipment | Unit Cost (US$ M) | Units Purchased | | | LOM Requirement (US$ M) |
|---|---|---|---|---|---|
| | | 2014 | 2015 | 2016 | |
| Haul Truck | 3.4 | | 3 | 1 | 13.7 |
| Excavator | 12.1 | 1 | 1 | | 18.1 |
| Dozer | 1.8 | 1 | 1 | | 3.7 |
| Rubber-Tired Dozer | 1.6 | 1 | | | 1.6 |
| Grader | 1.3 | | 1 | | 1.3 |
| Water Cart | 1.8 | | 1 | | 1.8 |
| Fuel Truck | 1.7 | 1 | | | 1.7 |
| Lube Truck | 1.9 | | 1 | | 1.9 |
| Rotary Drill | 2.0 | 1 | | | 2.0 |
| 120-Ton Crane | 0.9 | 1 | | | 0.9 |
| Buses for Shift Change | 0.1 | 3 | | | 0.2 |
| Tow Truck for Mine Equipment | 0.8 | 1 | | | 0.8 |
| **Total Mine Equipment Purchased** | - | **10** | **8** | **1** | **47.6** |

### 21.2.2 Infrastructure Capital Requirements

PVM will require capital to replace or upgrade equipment through 2026. The various areas and summary costs are detailed below and in Table 21-2.

- Mine Infrastructure and Mine Maintenance
  - Pit dewatering pumps and piping, pit water diversions, maintenance shop upgrades, electrical power loop relocation, and mine offices.
- Mill and SX-EW
  - Maintenance and upgrades of current equipment.
- Tailings and Water
  - Construction of tailings dams, tailings line replacement, tailings, and reclaim water pumping booster stations upgrades.
- Other Infrastructure
  - On-site and off-site infrastructure and an estimate for future sustaining capital.
- Miscellaneous Upgrades and Engineering Studies
  - Accounting system upgrades, IT infrastructure, light vehicles, and engineering studies.

**Table 21-2. Infrastructure Capital Requirements**

| Infrastructure | LOM Requirement (US$ M) |
|---|---|
| Mine Infrastructure and Mine Maintenance | 13.0 |
| Mill and SX-EW | 23.5 |
| Tailings and Water | 41.3 |
| Other Infrastructure | 51.2 |
| Miscellaneous Upgrades and Engineering Studies | 11.3 |
| **Total** | **140.3** |

## 21.3 Closure and Reclamation Cost Estimate

The capital costs associated with closure and reclamation of PVM are estimated at $129.1 M and are spread out from 2026 to 2055. This is based on the $87.1 M currently indicated by the APP and MLRP for the current mine, and an additional $42 M to cover an additional reclamation bond related to the approved Plan of Operations currently under review by the USFS; owner's costs, including internal G&A and labor to support the 30-year post-closure period, and the studies to support permit amendments; increased land disturbances; and other impacts of executing the 12-year mine plan.

## 21.4 Operating Cost Estimate

The LOM operating cost for PVM is projected to be $9.40/ton milled (excluding SX-EW, freight, and TC/RC costs). Costs for SX-EW production of copper are projected to be $1.80/lb Cu. These costs do not include concentrate transportation or treatment / refining charges at smelters and refineries. The operating costs are detailed in Table 21-3.

**Table 21-3. Operating Cost Summary**

| Operating Cost | Imperial | Metric |
|---|---|---|
| Mining Cost | $1.98/ton moved | $2.18/tonne moved |
| Mining Cost | $3.16/ton milled | $3.48/tonne milled |
| Milling Cost | $4.85/ton milled | $5.35/tonne milled |
| G&A Cost | $1.39/ton milled | $1.53/tonne milled |
| **Total** | **$9.40/ton milled** | **$10.37/tonne milled** |
| SX-EW Cost | $1.80/lb Cu cathode | - |

### 21.4.1 Mine Operating Cost

The mine operating costs were prepared by Joe Fernandez of AJF Mining Consultants and are based on historical, actual, and budgetary PVM mining costs, with input from PVM staff, and have been reviewed by John Marek, QP. The average LOM cost is $1.98/ton of material moved, and $3.16/ton of ore milled.

The following assumptions were made in calculating the mine operating costs.

- Costs are in fourth-quarter 2013 dollars.
- Diesel – $3.38/gallon for 2014, $2.92/gallon long-term.
- Explosives – emulsion $30.35/100 lb, ANFO price of $29.15/100 lb.
- Equipment operating costs are based on PVM and benchmark information.
- Maintenance costs are estimated based on equipment manufacturer component replacement schedules and include allowances for major component rebuilds.
- Hourly labor and salary costs are based on PVM costs.

The breakdown of mining costs by area can be seen in Table 21-4.

**Table 21-4. Mine Operating Costs by Area**

| Area | LOM Cost (US$ M) | Cost per Ton Moved (US$) | Cost per Ton Milled (US$) |
|---|---|---|---|
| Drilling and Blasting | 179.7 | 0.44 | 0.70 |
| Loading | 89.1 | 0.22 | 0.35 |
| Hauling | 337.8 | 0.83 | 1.32 |
| Support | 105.6 | 0.26 | 0.41 |
| Pit Dewatering | 18.1 | 0.04 | 0.07 |
| Mine G&A | 78.1 | 0.19 | 0.31 |
| **Total Mine Operating Cost** | **808.5** | **1.98** | **3.16** |

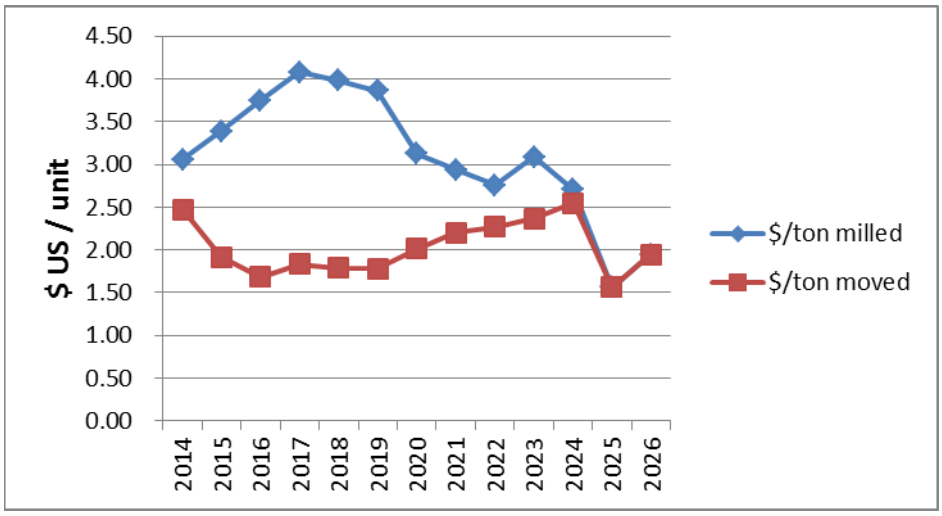
Note: Maintenance costs (labor and parts) are allocated to each area accordingly.

The mining costs by year are shown in Table 21-5. The first and last few years have higher-than-normal hauling costs, as a significant portion of the ore is mined from the bottom of the pit, resulting in longer haul cycles. Figure 21-1 compares the cost per ton of ore milled versus cost per ton moved. In the last 2 years, the cost per ton moved is equal to the cost per ton milled, due to very little waste present in the bottom of the pit and mining the stockpiles.

**Table 21-5. Mining Costs per Year**

| Type | Annual Cost (US$ M) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | **2014** | **2015** | **2016** | **2017** | **2018** | **2019** | **2020** | **2021** | **2022** | **2023** | **2024** | **2025** | **2026** |
| Drilling and Blasting | 13.5 | 16.3 | 20.6 | 20.6 | 20.6 | 20.1 | 14.9 | 13.3 | 12.3 | 13.1 | 11.1 | 3.3 | 0.0 |
| Loading | 6.7 | 9.0 | 10.0 | 9.8 | 9.8 | 9.6 | 7.0 | 5.8 | 5.3 | 5.7 | 4.7 | 4.5 | 1.3 |
| Hauling | 25.9 | 26.2 | 30.1 | 37.1 | 35.1 | 33.4 | 25.9 | 24.7 | 24.6 | 30.4 | 26.7 | 13.6 | 4.3 |
| Support | 7.0 | 8.7 | 9.8 | 9.8 | 9.8 | 9.8 | 9.7 | 9.7 | 7.5 | 7.6 | 6.1 | 5.1 | 4.9 |
| Pit Dewatering | 2.0 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.2 | 0.0 |
| Mine G&A | 6.4 | 6.4 | 6.4 | 6.4 | 6.4 | 6.4 | 6.4 | 6.4 | 6.4 | 6.4 | 6.4 | 5.1 | 2.2 |
| **Total Mine Operating Cost** | **61.5** | **68.2** | **78.4** | **85.3** | **83.3** | **80.8** | **65.4** | **61.4** | **57.7** | **64.6** | **56.5** | **32.7** | **12.7** |
| **Type** | **Unit Cost (US$ per ton moved)** | | | | | | | | | | | | |
| | **2014** | **2015** | **2016** | **2017** | **2018** | **2019** | **2020** | **2021** | **2022** | **2023** | **2024** | **2025** | **2026** |
| Drilling and Blasting | 0.54 | 0.46 | 0.44 | 0.44 | 0.44 | 0.44 | 0.46 | 0.48 | 0.49 | 0.48 | 0.50 | 0.16 | 0.00 |
| Loading | 0.27 | 0.25 | 0.21 | 0.21 | 0.21 | 0.21 | 0.22 | 0.21 | 0.21 | 0.21 | 0.21 | 0.21 | 0.20 |
| Hauling | 1.04 | 0.74 | 0.65 | 0.80 | 0.75 | 0.74 | 0.80 | 0.89 | 0.97 | 1.11 | 1.20 | 0.65 | 0.65 |
| Support | 0.28 | 0.25 | 0.21 | 0.21 | 0.21 | 0.22 | 0.30 | 0.35 | 0.30 | 0.28 | 0.27 | 0.24 | 0.75 |
| Pit Dewatering | 0.08 | 0.04 | 0.03 | 0.03 | 0.03 | 0.03 | 0.05 | 0.05 | 0.06 | 0.05 | 0.07 | 0.06 | 0.00 |
| Mine G&A | 0.26 | 0.18 | 0.14 | 0.14 | 0.14 | 0.14 | 0.20 | 0.23 | 0.25 | 0.24 | 0.29 | 0.24 | 0.34 |
| **Total Mine Operating Cost** | **2.47** | **1.91** | **1.68** | **1.83** | **1.79** | **1.78** | **2.01** | **2.20** | **2.27** | **2.37** | **2.55** | **1.56** | **1.94** |

**Figure 21-1. Mine Operating Cost**




## 21.4.2 Mill Operating Cost

The mill operating cost estimates include all costs related to the Pinto Valley metallurgical facilities, including the crusher, mill, SX-EW, tailings pumping, tailings dam, and reclaim water. As an operating mine, the cost review completed by Kenneth W. Major, QP, focused on the operating budgets provided by PVM. The budget has been based on current operating conditions, with details included for power consumption and costs, consumables (including wear materials and reagents), and direct and indirect labor costs. The average LOM operating cost for the mill based on a production rate of 55,100 tpd has been estimated to be $5.01/ton milled. Throughput has been projected to increase to 57,300 tpd in Year 3, resulting in an estimated operating cost of $4.82/ton milled.

The following assumptions were made in calculating the mill operating costs.

- Costs are in fourth-quarter 2013 dollars.
- Mill throughput is 55,100 tpd (first 2 years) and 57,300 tpd (into 2026).
- Power cost is $0.065/kWh.
- Grinding balls cost is $0.50/lb.
- Lime cost is $220/ton.
- Hourly labor and salary costs are based on PVM costs.
- Maintenance and component change costs are based on PVM costs.

The breakdown of mill costs can be seen in Table 21-6 and Table 21-7.

**Table 21-6. Mill Operating Costs**

| Type | 55,100 tpd (2014–2015) | | 57,300 tpd (2016–2026) | |
|---|---|---|---|---|
| | US$ M | US$/Ton Milled | US$ M | US$/Ton Milled |
| Labor | 28.41 | 1.41 | 28.41 | 1.36 |
| Consumables | 32.69 | 1.62 | 33.46 | 1.60 |
| Power | 22.08 | 1.10 | 22.88 | 1.09 |
| Maintenance* | 17.67 | 0.88 | 16.17 | 0.77 |
| **Total** | **100.85** | **5.01** | **100.93** | **4.82** |

*Maintenance decreases in later years (when mill ramps up to 57,300 tpd) as external contractor usage is reduced.

**Table 21-7. Mill Operating Costs – Life-of-Mine Average**

| Type | LOM Cost (US$ M) | LOM Cost per Ton Milled (US$) |
|---|---|---|
| Labor | 347.1 | 1.36 |
| Consumables | 408.6 | 1.60 |
| Power | 280.2 | 1.09 |
| Maintenance | 206.1 | 0.81 |
| **Total Mill Operating Cost** | **1,241.9** | **4.85** |

### 21.4.3 Solvent Extraction and Electrowinning Operating Costs

SX-EW operating costs include all costs related to the leaching of ore and the production of copper cathodes. Based on the nominal production of 6.3 M lb Cu/yr, the LOM costs have been estimated at $1.80/lb Cu produced. A breakdown of the operating costs is detailed in Table 21-8.

**Table 21-8. SX-EW Operating Costs – Life-of-Mine Average**

| Type | LOM Cost (US$ M) | Cost/lb Cu Cathode (US$) |
|---|---|---|
| Labor | 51.1 | 0.48 |
| Consumables and Maintenance | 105.8 | 0.99 |
| Power | 36.5 | 0.33 |
| **Total** | **193.3** | **1.80** |

### 21.4.4 General and Administrative Operating Costs

G&A costs include administration, finance, human resources, HSEC, engineering services, IT, procurement, warehousing, insurance, and site infrastructure. These costs were prepared by PVM staff and reviewed by Adam Majorkiewicz, QP. The average LOM G&A cost is $1.39/ton milled.

The G&A costs were calculated using the current costs from PVM. A breakdown of these costs can be seen in Table 21-9.

**Table 21-9. General and Administrative Operating Costs**

| G&A Area | LOM Cost (US$ M) | LOM Cost per Ton Milled (US$) |
|---|---|---|
| Management and Operating Supplies | 165.0 | 0.64 |
| Engineering Services and Site infrastructure | 114.6 | 0.45 |
| HSEC | 55.7 | 0.22 |
| IT | 21.0 | 0.08 |
| **Total G&A Operating Cost** | **356.4** | **1.39** |

Note: Summation errors due to rounding.

### 21.4.5 Transport and Shipping

PVM produces a copper concentrate, molybdenum concentrate, and copper cathode. All products have different destinations and cost associated with that transportation.

Copper concentrate will be sold internationally and to smelters in North America (domestic). Domestic sales of copper concentrate are moved from PVM to the US-based smelters by truck. Each truck carries approximately 25 tons. The trucking rate and carrier is contracted by competitive tendering. Export sales of copper concentrate are more complex, requiring multiple handling and multi-modal transport. The truck haul is from the Pinto Valley mill load-out to Capstone's San Manuel storage facility, where the concentrate is stored under cover and accessible by railcars. The concentrate is then loaded on railcars owned by SMARRCO. Each railcar is loaded with approximately 87 tons of concentrate, which is then moved to the port of Guaymas, Mexico. When sufficient stocks have been accumulated, the concentrate is transferred to chartered ocean vessels and transported to final destinations.

The costs associated with selling concentrate internationally and domestically can be seen in Table 21-10.

**Table 21-10. Concentrate Transport and Shipping Costs**

| Type | Copper Concentrate Sold Internationally | Copper Concentrate Sold Domestically |
|---|---|---|
| Transport Cost | $143.71/wet tonne | $77.60/wet tonne |
| Percentage of Concentrate Sold | 90% | 10% |
| **Average Realized Concentrate Shipping Cost** | **$137.10/wet tonne** | - |

Molybdenum concentrate and copper cathode are sold locally, and both are produced at low quantities compared to the copper concentrate.

### Copper Cathode

Copper cathode is sold on a Free Carrier basis. As such, PVM is responsible to only load the bundles of cathode on the buyer's hired trucks when they arrive at the load-out.

### Molybdenum Concentrate

Molybdenum concentrate is placed in one-tonne super sacks and stored on pallets. With the terms of sale being Free Carrier or Ex Works, PVM is only required to load the pallets into containers when the containers are delivered to the load-out by the buyer.

# 22.0 ECONOMIC ANALYSIS

A financial evaluation of PVM was performed to assess the economic viability of continuing mill operations until 2026 and SX-EW operations until 2030. The determination of economic viability was carried out by calculating a pre-tax and after-tax NPV. The NPV is calculated using annual cash flow projections over the life of the mine, based on estimates of capital expenditures, production costs, sales revenue, and taxes. The sales revenue is based on the sale of copper concentrate, molybdenum concentrate, and copper cathode. The estimates for capital expenditures and operating costs are presented in Section 21. Costs are in fourth-quarter 2013 dollars.

## 22.1 Cash Flow Basis

### 22.1.1 Mine Production Statistics

The mill will operate until the first quarter of 2026 (12.3 years), and SX-EW operations will continue until 2030 (16 years). The mill will process 256 M tons of ore at an average head grade of 0.33% TCu, producing a total of 1.418 billion pounds of copper, along with 16.8 M lb Mo and 2.6 M oz of silver. The Low-Grade Ore Leaching Pile will receive 78 M tons of material grading 0.14% TCu and produce 107 M lb Cu cathode. Further production statistics are detailed in Table 22-1.

**Table 22-1. Mine Production Statistics (imperial units)**

| Item | Production Details |
|---|---|
| Mine Life | 12.3 years |
| Average Plant Throughput (ore) – 2014 | 55,100 tpd |
| Average Plant Throughput (ore) – 2016+ | 57,300 tpd |
| Total Ore Processed (milled) | 256 M tons |
| TCu Grade | 0.33% |
| Mo Grade | 0.008% |
| Average Cu Recovery | 88% |
| Total Leach | 78 M tons |
| Total Waste | 54 M tons |
| Strip Ratio | 0.65 |
| Payable Cu | 1,418 M lb |
| Payable Cu (SX-EW) | 107 M lb |
| Payable Mo | 17 M lb |
| Payable Ag | 2.6 M oz |

### 22.1.2 Transportation, Treatment, and Refining Costs

Transportation costs and details are discussed in Section 21. Copper concentrate transportation averages $132.5 per wet tonne. No transportation cost is incurred by PVM for molybdenum concentrate or copper cathode.

Market studies and contracts are detailed in Section 19. The concentrate smelting and refining terms assumed are shown in Table 22-2.

**Table 22-2. Concentrate Smelting and Refining Terms**

| Item | Details |
|---|---|
| **Concentrate Details** | |
| Copper Concentrate Grade – TCu | 27.5% |
| Copper Concentrate Grade – Ag | 1.2 oz/dry tonne |
| Moisture Content | 9.0% |
| Copper Payable | 96.5% |
| Copper Treatment Charge | $85.00/dry tonne |
| Copper Refining Charge | $0.085/lb |
| Silver Payable Threshold | 1.00 oz/dry tonne |
| Silver Payable When Grade Above Threshold | 90% |
| **Molybdenum Concentrate** | |
| Molybdenum Concentrate Grade – $MoS_2$ | 80% |
| Molybdenum Concentrate Grade – Mo | 48% |
| Molybdenum Concentrate Moisture Content | 4.5% |
| Molybdenum Payable | 97% |
| Molybdenum Roasting Cost | 12% Mo price |

### 22.1.3 Royalties and Taxes

PVM is wholly owned by Capstone and has no royalty payments.

Taxes applicable to the project include property tax, severance tax, state tax, and federal tax. Tax payments have been estimated for the life of the project based on net earnings, depreciation, depletion, and taxable rates. Over the life of the project, taxes are estimated to total $273.8 M.

### 22.1.4 Capital and Operating Costs

Sustaining and closure capital costs are discussed in Section 21. Capital costs for the project total $188 M for sustaining capital and $129 M for closure and reclamation.

Operating costs are discussed in Section 21. The average LOM cost is $9.40/ton of ore milled and $1.80/lb Cu produced at the SX-EW plant. The breakdown of the operating costs is detailed in Table 22-3.

**Table 22-3. Operating Cost Summary**

| Operating Cost | Imperial | Metric |
|---|---|---|
| Mining Cost | $1.98/ton moved | $2.18/tonne moved |
| Mining Cost | $3.16/ton milled | $3.48/tonne milled |
| Milling Cost | $4.85/ton milled | $5.35/tonne milled |
| G&A Cost | $1.39/ton milled | $1.53/tonne milled |
| **Total** | **$9.40/ton milled** | **$10.37/tonne milled** |
| SX-EW Cost | $1.80/lb Cu cathode | - |

### 22.1.5 Salvage Value

No allowance for salvage value has been included in the cash flow analysis.

### 22.1.6 Working Capital

PVM is an existing operating mine with working capital. No additional working capital has been identified.

## 22.2 Project Cash Flow

The evaluation period begins 01 January 2014. The base case financial indicators have been determined with 100% equity financing of the sustaining capital. Any acquisition costs or expenditures prior to the start of the project period have been treated as “sunk” costs and have not been included in the analysis. Costs are fourth-quarter 2013 dollars, and no escalation is applied. The detailed project cash flow used to evaluate the project is presented in Table 22-4.

## Table 22-4. Project Cash Flow

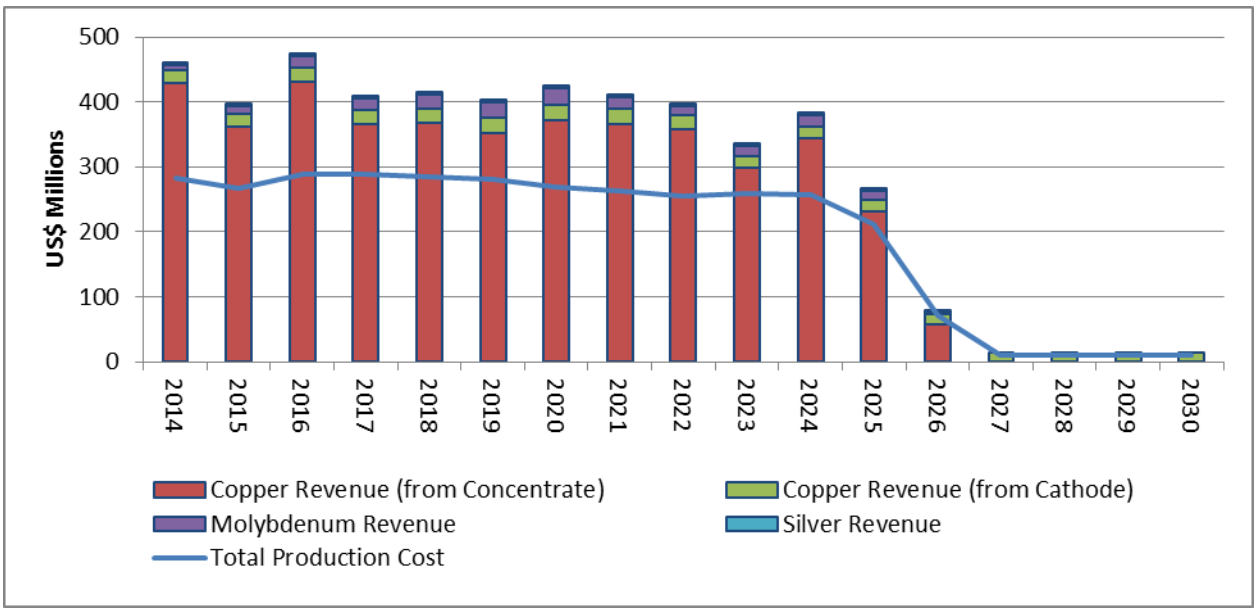
| Year | Unit | LOM | 2014 | 2015 | 2016 | 2017 | 2018 | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 | 2037 | 2038 | 2039 | 2040 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **MINING** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| **Open Pit Mine Schedule** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Ore mined to concentrator | *Million tons* | 235.44 | 20.12 | 20.12 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 6.91 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Ore mined to stockpile | *Million tons* | 20.54 | 1.20 | 4.74 | 1.99 | 0.93 | 4.17 | 2.95 | 2.59 | 1.96 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Ore from stockpile to concentrator | *Million tons* | 20.54 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 14.01 | 6.52 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Low Grade Material mined to heap leach | *Million tons* | 77.85 | 2.41 | 5.99 | 10.04 | 10.08 | 9.42 | 16.79 | 8.11 | 4.34 | 4.13 | 5.31 | 1.25 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Waste mined | *Million tons* | 54.10 | 1.16 | 4.78 | 13.65 | 14.67 | 12.09 | 4.81 | 0.87 | 0.67 | 0.34 | 1.08 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Total Material Mined | *Million tons* | 387.92 | 24.88 | 35.62 | 46.60 | 46.60 | 46.60 | 45.47 | 32.49 | 27.89 | 25.39 | 27.31 | 22.17 | 6.91 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Total Material Moved | *Million tons* | 408.46 | 24.88 | 35.62 | 46.60 | 46.60 | 46.60 | 45.47 | 32.49 | 27.89 | 25.39 | 27.31 | 22.17 | 20.92 | 6.52 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| **MILLING & CONCENTRATING** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Days/year | *days* | 4494 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 114 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Mill Capacity - per Day | *stpd* | 56,962 | 55,116 | 55,116 | 57,320 | 57,320 | 57,320 | 57,320 | 57,320 | 57,320 | 57,320 | 57,320 | 57,320 | 57,320 | 57,320 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Ore to Concentrator | *million tons* | 256 | 20.12 | 20.12 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 20.92 | 6.52 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Copper grade processed | *% Cu* | 0.33% | 0.39% | 0.34% | 0.38% | 0.33% | 0.33% | 0.32% | 0.33% | 0.33% | 0.32% | 0.31% | 0.35% | 0.24% | 0.19% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| Molybdenum grade processed | *% Mo* | 0.008% | 0.008% | 0.008% | 0.008% | 0.008% | 0.009% | 0.010% | 0.011% | 0.008% | 0.006% | 0.006% | 0.007% | 0.006% | 0.006% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% | 0.000% |
| Copper recovery | *%* | 88% | 88.5% | 88.1% | 88.7% | 87.8% | 88.1% | 87.9% | 88.2% | 88.1% | 87.9% | 87.9% | 88.3% | 87.3% | 86.6% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% |
| Molybdenum recovery | *%* | 43% | 20.0% | 30.0% | 40.0% | 45.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% | 47.0% |
| **Recovered Copper to concentrate** | ***M lbs*** | **1469.9** | **140.01** | **118.69** | **141.85** | **120.51** | **120.93** | **115.91** | **122.18** | **120.51** | **118.00** | **112.56** | **129.72** | **87.36** | **21.63** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** |
| **Recovered Moly to concentrate** | ***M lbs*** | **17.3** | **0.66** | **0.98** | **1.39** | **1.45** | **1.79** | **1.99** | **2.07** | **1.55** | **1.16** | **1.22** | **1.40** | **1.26** | **0.37** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** |
| **CONCENTRATE** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| **Copper Concentrate** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Copper concentrate grade - Cu | *% Cu* | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 27.5% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% |
| Copper concentrate grade - Ag | *oz/dmt* | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 1.20 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Copper concentrate produced | *dmt X'1000* | 2,425 | 231.0 | 195.8 | 234.0 | 198.8 | 199.5 | 191.2 | 201.6 | 198.8 | 194.7 | 185.7 | 214.0 | 144.1 | 35.7 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Copper concentrate moisture content | *% wt* | 9% | 9% | 9% | 9% | 9% | 9% | 9% | 9% | 9% | 9% | 9% | 9% | 9% | 9% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% |
| Copper concentrate produced | *wmt X'1000* | 2,665 | 253.9 | 215.2 | 257.2 | 218.5 | 219.3 | 210.1 | 221.5 | 218.5 | 213.9 | 204.1 | 235.2 | 158.4 | 39.2 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Contained silver production | *k oz* | 2,910 | 277.2 | 235.0 | 280.8 | 238.6 | 239.4 | 229.5 | 241.9 | 238.6 | 233.6 | 222.9 | 256.8 | 173.0 | 42.8 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| Copper payable | *%* | | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% | 96.5% |
| Silver payable when grade above threshold | *%* | | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% | 90.0% |
| **Molybdenum Concentrate** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Moly concentrate grade | *% Mo* | 48% | 48% | 48% | 48% | 48% | 48% | 48% | 48% | 48% | 48% | 48% | 48% | 48% | 48% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% |
| Moly concentrate produced | *dmt x 1000* | 16.35 | 0.62 | 0.93 | 1.31 | 1.37 | 1.69 | 1.88 | 1.95 | 1.47 | 1.10 | 1.15 | 1.32 | 1.19 | 0.35 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Moly concentrate moisture content | *%* | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | 0% |
| Moly concentrate produced | *wmt x 1000* | 17.12 | 0.65 | 0.97 | 1.38 | 1.44 | 1.77 | 1.97 | 2.05 | 1.54 | 1.15 | 1.21 | 1.38 | 1.25 | 0.37 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Moly Payable | *%* | | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% | 97.0% |
| **Payable Metals** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| **Payable Copper Produced** | ***M lbs*** | **1,418** | **135.11** | **114.54** | **136.89** | **116.29** | **116.70** | **111.85** | **117.91** | **116.29** | **113.87** | **108.62** | **125.18** | **84.30** | **20.87** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** |
| **Payable Moly Produced** | ***M lbs*** | **16.8** | **0.64** | **0.95** | **1.35** | **1.41** | **1.74** | **1.93** | **2.00** | **1.51** | **1.13** | **1.18** | **1.35** | **1.22** | **0.36** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** |
| **Payable Silver Produced** | ***k oz*** | **2,619** | **249.5** | **211.5** | **252.8** | **214.7** | **215.5** | **206.5** | **217.7** | **214.7** | **210.3** | **200.6** | **231.1** | **155.7** | **38.5** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** |
| **LEACHING** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Low Grade Material mined to heap leach | *Million tons* | 77.85 | 2.41 | 5.99 | 10.04 | 10.08 | 9.42 | 16.79 | 8.11 | 4.34 | 4.13 | 5.31 | 1.25 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Copper grade of material to leach dump | *% Cu* | 0.14% | 0.15% | 0.14% | 0.15% | 0.13% | 0.15% | 0.15% | 0.14% | 0.15% | 0.14% | 0.13% | 0.14% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| **Total Copper Cathode (Payable)** | ***M lbs*** | **107.28** | **6.2** | **6.3** | **6.7** | **6.8** | **7.1** | **7.5** | **7.7** | **7.3** | **6.8** | **6.6** | **6.4** | **6.0** | **5.5** | **5.3** | **5.2** | **5.0** | **4.9** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** |
| | ***ktonnes (metr*** | **48.68** | **2.8** | **2.9** | **3.1** | **3.1** | **3.2** | **3.4** | **3.5** | **3.3** | **3.1** | **3.0** | **2.9** | **2.7** | **2.5** | **2.4** | **2.3** | **2.3** | **2.2** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** |
| **REVENUE** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| **Commodity Price Forecast** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Copper | *$/lb Cu* | | 3.17 | 3.16 | 3.15 | 3.15 | 3.14 | 3.14 | 3.14 | 3.14 | 3.14 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 |
| Copper Cathode Premium | *$/lb Cu* | | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 |
| Molybdenum | *$/lb Mo* | | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 | 12.50 |
| Silver | *$/oz Ag* | | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 |
| **Revenue** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Copper Revenue (from Concentrate) | *$ M* | 4,333.8 | 428.73 | 361.85 | 431.64 | 366.17 | 366.92 | 351.67 | 370.80 | 365.72 | 358.10 | 298.71 | 344.23 | 231.83 | 57.39 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Copper Revenue (from Cathode) | *$ M* | 322.2 | 19.69 | 20.05 | 21.35 | 21.65 | 22.42 | 23.72 | 24.43 | 23.05 | 21.53 | 18.24 | 17.73 | 16.51 | 15.37 | 14.75 | 14.32 | 13.87 | 13.47 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Molybdenum Revenue | *$ M* | 209.5 | 8.00 | 11.85 | 16.84 | 17.58 | 21.70 | 24.08 | 25.04 | 18.84 | 14.07 | 14.78 | 16.93 | 15.28 | 4.51 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Silver Revenue | *$ M* | 52.4 | 4.99 | 4.23 | 5.06 | 4.29 | 4.31 | 4.13 | 4.35 | 4.29 | 4.21 | 4.01 | 4.62 | 3.11 | 0.77 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| **Total Revenue** | ***$ M*** | **4,917.8** | **461.4** | **398.0** | **474.9** | **409.7** | **415.3** | **403.6** | **424.6** | **411.9** | **397.9** | **335.7** | **383.5** | **266.7** | **78.0** | **14.7** | **14.3** | **13.9** | **13.5** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** |

## Table 22-4. Project Cash Flow (continued)

| Year | Unit | LOM | 2014 | 2015 | 2016 | 2017 | 2018 | 2019 | 2020 | 2021 | 2022 | 2023 | 2024 | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 | 2037 | 2038 | 2039 | 2040 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **PRODUCTION COSTS** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| **Concentrate Costs (Offsite)** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Copper Treatment charge | $/dmt | 85.0 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 | 85.00 |
| Copper Refining charge | $/lb | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 | 0.085 |
| Percentage of Concentrate sold internati | % | 83% | 83% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% | 90% |
| Average concentrate shipping cost | $/wmt | 132.5 | 132.5 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 | 137.1 |
| Moly concentrate transportation cost | $/wmt | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| Moly Roasting cost | % Mo price | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% | 12% |
| Cathode Freight Cost | $/lb | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 |
| Treatment Charges | $/M | 206.1 | 19.64 | 16.65 | 19.89 | 16.90 | 16.96 | 16.26 | 17.14 | 16.90 | 16.55 | 15.79 | 18.19 | 12.25 | 3.03 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Refining Charges | $/M | 145.7 | 12.44 | 11.16 | 13.66 | 11.99 | 12.52 | 12.40 | 13.03 | 12.15 | 11.37 | 11.01 | 12.67 | 9.00 | 2.32 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Concentrate Transport | $/M | 364.2 | 33.63 | 29.50 | 35.26 | 29.96 | 30.06 | 28.81 | 30.37 | 29.96 | 29.33 | 27.98 | 32.24 | 21.72 | 5.38 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| **Total Concentrate (Off-site) Costs** | **$/M** | **716.0** | **65.7** | **57.3** | **68.8** | **58.9** | **59.5** | **57.5** | **60.5** | **59.0** | **57.2** | **54.8** | **63.1** | **43.0** | **10.7** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** |
| Net Smelter Return | $/M | 4,201.8 | 395.7 | 340.7 | 406.1 | 350.8 | 355.8 | 346.1 | 364.1 | 352.9 | 340.7 | 281.0 | 320.4 | 223.8 | 67.3 | 14.7 | 14.3 | 13.9 | 13.5 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| **Mine Site Operating Costs** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Mining Cost | $/M | 808.5 | 61.51 | 68.20 | 78.40 | 85.31 | 83.27 | 80.84 | 65.38 | 61.38 | 57.68 | 64.63 | 56.51 | 32.72 | 12.68 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Milling Cost | $/M | 1,241.9 | 102.91 | 100.85 | 100.74 | 100.99 | 100.49 | 100.93 | 100.51 | 100.99 | 100.45 | 100.88 | 100.61 | 100.55 | 31.02 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Leach Costs | $/M | 193.3 | 12.88 | 11.57 | 12.47 | 15.12 | 13.15 | 12.81 | 13.50 | 12.01 | 10.74 | 10.47 | 10.64 | 10.13 | 9.57 | 9.57 | 9.57 | 9.57 | 9.57 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Site G&A | $/M | 356.4 | 39.11 | 28.83 | 28.89 | 28.89 | 28.89 | 28.89 | 28.89 | 28.89 | 28.89 | 27.47 | 26.11 | 24.82 | 7.78 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| **Total Mine Site (On-site) Costs** | **$/M** | **2,600.1** | **216.4** | **209.4** | **220.5** | **230.3** | **225.8** | **223.5** | **208.3** | **203.3** | **197.8** | **203.5** | **193.9** | **168.2** | **61.1** | **9.6** | **9.6** | **9.6** | **9.6** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** |
| **Total Production Cost** | **$ M** | **3,316.2** | **282.1** | **266.8** | **289.3** | **289.2** | **285.4** | **280.9** | **268.8** | **262.3** | **255.0** | **258.2** | **257.0** | **211.2** | **71.8** | **9.6** | **9.6** | **9.6** | **9.6** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** |
| **Operating Cashflow** | **$ M** | **1,601.7** | **179.3** | **131.2** | **185.6** | **120.5** | **130.0** | **122.7** | **155.8** | **149.6** | **142.9** | **77.5** | **126.5** | **55.6** | **6.3** | **5.2** | **4.8** | **4.3** | **3.9** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** |
| **MINESITE COST DATA** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Mining Cost | $/ton moved | 1.98 | 2.47 | 1.91 | 1.68 | 1.83 | 1.79 | 1.78 | 2.01 | 2.20 | 2.27 | 2.37 | 2.55 | 1.56 | 1.94 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Mining Cost | $/ton milled | 3.16 | 3.06 | 3.39 | 3.75 | 4.08 | 3.98 | 3.86 | 3.12 | 2.93 | 2.76 | 3.09 | 2.70 | 1.56 | 1.94 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Milling Cost | $/ton milled | 4.85 | 5.12 | 5.01 | 4.81 | 4.83 | 4.80 | 4.82 | 4.80 | 4.83 | 4.80 | 4.82 | 4.81 | 4.81 | 4.76 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Leach Costs | $/lb cathode | 1.80 | 2.09 | 1.83 | 1.85 | 2.21 | 1.86 | 1.71 | 1.75 | 1.65 | 1.58 | 1.59 | 1.66 | 1.70 | 1.72 | 1.80 | 1.85 | 1.91 | 1.97 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Site G&A | $/ton milled | 1.39 | 1.94 | 1.43 | 1.38 | 1.38 | 1.38 | 1.38 | 1.38 | 1.38 | 1.38 | 1.31 | 1.25 | 1.19 | 1.19 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Total Production Cost (C1 Cost) includes M | $/lb Cu | 2.00 | 1.90 | 2.07 | 1.86 | 2.17 | 2.10 | 2.12 | 1.91 | 1.94 | 1.96 | 2.08 | 1.79 | 2.14 | 2.52 | 1.80 | 1.85 | 1.91 | 1.97 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| **CAPITAL EXPENDITURES** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Mine Equipment Purchases | $/M | 47.6 | 21.14 | 23.04 | 3.41 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Mine Infrastructure, Mill, Tailings, SX-EW, G | $/M | 89.1 | 23.06 | 34.61 | 5.78 | 8.91 | 2.91 | 5.31 | 8.00 | 0.51 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Site Infrastructure, Other | $/M | 51.2 | 2.57 | 4.76 | 3.30 | 3.00 | 5.80 | 5.50 | 5.80 | 5.50 | 5.00 | 5.00 | 3.00 | 2.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| TOTAL CAPITAL COSTS | $/M | 187.9 | 46.77 | 62.41 | 12.49 | 11.91 | 8.71 | 10.81 | 13.80 | 6.01 | 5.00 | 5.00 | 3.00 | 2.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| TOTAL CLOSURE COSTS | $/M | 129.1 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 18.83 | 18.83 | 18.83 | 4.48 | 3.22 | 17.22 | 16.22 | 16.22 | 1.22 | 1.22 | 1.22 | 1.22 | 1.22 | 0.54 | 8.61 |
| **CASH FLOW** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Total Revenue (excluding TC/RC/Transport | $/M | 4,917.8 | 461.41 | 397.99 | 474.89 | 409.70 | 415.35 | 403.61 | 424.62 | 411.90 | 397.91 | 335.74 | 383.52 | 266.74 | 78.05 | 14.75 | 14.32 | 13.87 | 13.47 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Total Concentrate (Off-site) Costs | $/M | -716.0 | -65.71 | -57.31 | -68.81 | -58.85 | -59.54 | -57.46 | -60.53 | -59.00 | -57.25 | -54.77 | -63.11 | -42.97 | -10.72 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Total Mine Site (On-site) Costs | $/M | -2,600.1 | -216.42 | -209.44 | -220.50 | -230.31 | -225.81 | -223.47 | -208.28 | -203.28 | -197.76 | -203.45 | -193.87 | -168.22 | -61.05 | -9.57 | -9.57 | -9.57 | -9.57 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| **Operating Cash flow** | **$/M** | **1,601.7** | **179.29** | **131.24** | **185.58** | **120.53** | **129.99** | **122.67** | **155.81** | **149.62** | **142.90** | **77.52** | **126.55** | **55.55** | **6.27** | **5.18** | **4.76** | **4.30** | **3.90** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** | **0.00** |
| Capital Costs | $/M | -187.9 | -46.77 | -62.41 | -12.49 | -11.91 | -8.71 | -10.81 | -13.80 | -6.01 | -5.00 | -5.00 | -3.00 | -2.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Closure Costs | $/M | -129.1 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | -18.83 | -18.83 | -18.83 | -4.48 | -3.22 | -17.22 | -16.22 | -16.22 | -1.22 | -1.22 | -1.22 | -1.22 | -1.22 | -0.54 | -8.61 |
| Net Pre-Tax Cash Flow | $/M | 1,284.7 | 132.52 | 68.82 | 173.09 | 108.62 | 121.29 | 111.86 | 142.01 | 143.62 | 137.90 | 72.52 | 123.55 | 53.55 | -12.56 | -13.65 | -14.07 | -0.17 | 0.68 | -17.22 | -16.22 | -16.22 | -1.22 | -1.22 | -1.22 | -1.22 | -1.22 | -0.54 | -8.61 |
| NPV Pre-tax - 8% Discount | $/M | 930.9 | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Taxes Payable | $/M | 273.8 | -33.20 | -20.69 | -32.56 | -17.06 | -19.15 | -16.68 | -29.30 | -31.10 | -30.26 | -10.88 | -25.79 | -5.98 | -0.35 | -0.27 | -0.23 | -0.18 | -0.13 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Net Post-Tax Cash Flow | $/M | 1,010.9 | 99.32 | 48.13 | 140.53 | 91.57 | 102.13 | 95.18 | 112.71 | 112.51 | 107.64 | 61.63 | 97.76 | 47.58 | -12.91 | -13.92 | -14.30 | -0.35 | 0.55 | -17.22 | -16.22 | -16.22 | -1.22 | -1.22 | -1.22 | -1.22 | -1.22 | -0.54 | -8.61 |
| NPV Post-tax - 8% Discount | $/M | 738.4 | | | | | | | | | | | | | | | | | | | | | | | | | | | |

Revenue by metal and total operating costs by year are shown in Figure 22-1. The revenue is shown as a stacked bar graph, while the operating costs are shown as a solid line. Revenue is generated by 95% copper, 4% molybdenum, and 1% for silver.

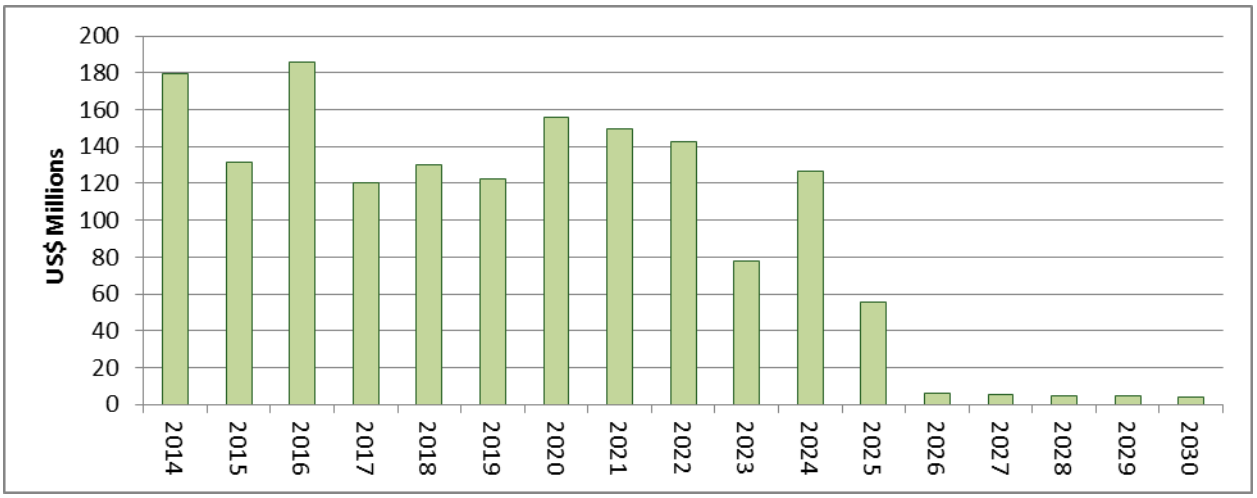
**Figure 22-1. Revenue and Total Production Cost by Year**




Note: Total production cost includes mine, mill, G&A, SX-EW, concentrate transportation costs, treatment and refining charges.

Operating cash flow (revenue minus total production costs) is shown in Figure 22-2.

**Figure 22-2. Operating Cash Flow**



## 22.3 Cash Flow Analysis

### 22.3.1 Economic Results Summary

The cash flow analysis shows an operating cash flow of $1,602 M, which results in a pre-tax NPV (8%) of $931 M and a post-tax NPV (8%) of $738 M, as shown in Table 22-5.

**Table 22-5. Financial Analysis**

| Item | US$ M |
|---|---|
| Copper Revenue | 4,656 |
| Molybdenum Revenue | 210 |
| Silver Revenue | 52 |
| **Total Revenue** | **4,918** |
| Site Costs | 2,600 |
| Concentrate Transport, TC/RCs | 716 |
| Operating Cash Flow | 1,602 |
| Sustaining Capital Costs | 188 |
| Closure Costs | 129 |
| Net Cash Flow – Pre-Tax, Undiscounted | 1,285 |
| **NPV (8%) Pre-Tax** | **931** |
| Taxes Payable | 274 |
| Net Cash Flow – Post-Tax, Undiscounted | 1,011 |
| **NPV (8%) Post-Tax** | **738** |

## 22.4 Sensitivity Analysis

A sensitivity analysis has been performed on discount rate, copper price, operating costs, and sustaining capital costs. The result of the discount rate analysis is detailed in Table 22-6 and the remaining parameters in Table 22-7 and Figure 22-3.

**Table 22-6. Post-Tax Net Present Value Sensitivity**

| Net Present Value | US$ M |
|---|---|
| NPV (0%) Post-Tax | 1,011 |
| NPV (5%) Post-Tax | 828 |
| **NPV (8%) Post-Tax** | **738** |
| NPV (10%) Post-Tax | 687 |
| NPV (12%) Post-Tax | 640 |

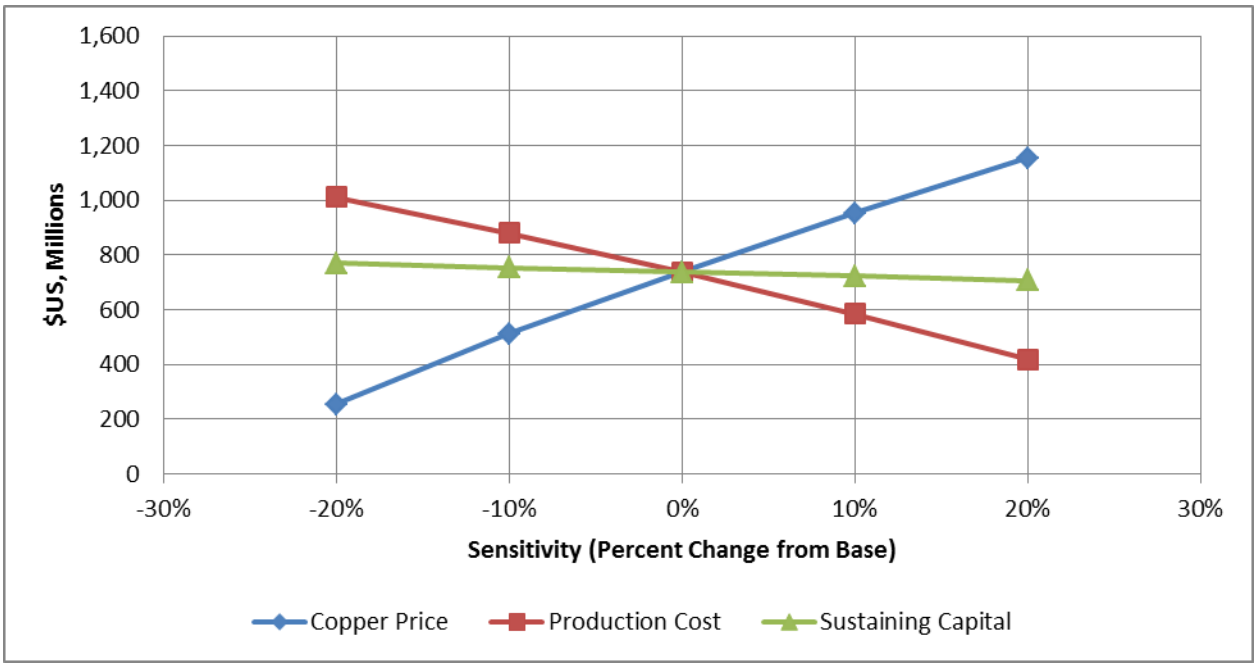
**Table 22-7. Sensitivity Analysis Parameters**

| Parameter or Variation | Pre-Tax | | Post-Tax | |
|---|---|---|---|---|
| | NPV @ 0% (US$ M) | NPV @ 8.0% (US$ M) | NPV @ 0% (US$ M) | NPV @ 8.0% (US$ M) |
| **Copper Price** | | | | |
| –20% | 354 | 291 | 306 | 254 |
| –10% | 819 | 611 | 682 | 512 |
| Base Case | 1,285 | 931 | 1,011 | 738 |
| 10% | 1,750 | 1,251 | 1,324 | 952 |
| 20% | 2,215 | 1,571 | 1,620 | 1,155 |
| **Total Production Costs*** | | | | |
| –20% | 1,948 | 963 | 1,415 | 1,011 |
| –10% | 1,616 | 1,156 | 1,221 | 880 |
| Base Case | 1,285 | 931 | 1,011 | 738 |
| 10% | 953 | 706 | 785 | 585 |
| 20% | 621 | 481 | 537 | 418 |
| **Sustaining Capital Costs** | | | | |
| –20% | 1,322 | 963 | 1,048 | 770 |
| –10% | 1,303 | 947 | 1,030 | 754 |
| Base Case | 1,285 | 931 | 1,011 | 738 |
| 10% | 1,266 | 915 | 992 | 723 |
| 20% | 1,247 | 899 | 973 | 707 |

*Total production costs include mine, mill, G&A, SX-EW, concentrate transportation, and treatment and refining charges.

The analysis shows that the project is most sensitive to copper prices, followed by operating cost, and then sustaining capital costs. A 10% change in metal price can impact the post-tax NPV (8%) by approximately $225 M, while a 10% change in operating cost can impact the post-tax NPV (5%) by approximately $145 M. A 10% fluctuation in sustaining capital costs has a minor impact on NPV.

**Figure 22-3. Sensitivity: Net Present Value – Post-Tax – 8% Discount**



# 23.0 ADJACENT PROPERTIES

The PVM site is in proximity to the following mining properties: Carlota Mine, Copper Cities Mine area, and Miami Operations.

The sources of the information included in this section are public websites, as well as publically disclosed information by KGHM International Ltd. and Freeport-McMoRan Copper & Gold Inc. (FMI).

## 23.1 Carlota Mine

The Carlota Mine is adjacent to PVM and is under sole ownership 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 the mine has produced an average of 25 M lb of cathode copper annually for the last 4 years, using ROM heap 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 in reclamation, the Carlota Mine is implementing a staged mine closure plan 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.

## 23.2 Copper Cities Mine Area

The Copper Cities Mine area is located approximately 3 miles east of PVM on the south flank of Sleeping Beauty Peak. It consists of three properties: Sleeping Beauty Mine, Copper Cities Mine, and Diamond-H Mine. The Copper Cities Mine and Diamond-H Mine are owned by BHP. Mine workings in this area date back to 1896, and all three mines are currently closed.

The Sleeping Beauty Mine, known for the high quality and unique coloration of its turquoise deposit, had a low production rate. Open-pit production for the Copper Cities Mine totaled over 61 Mt of sulfide ore from 1954 to 1971, with an additional 18.2 ktons Cu produced by heap leach (Burgin 1976, 40). The smaller Diamond-H ore deposit, 1 mile west of the Copper Cities pit area, was discovered in 1965 and operated during the 1970s. The Copper Cities and Diamond-H pits currently serve as reservoirs for supplemental process water pumped from the Miami area to PVM under an agreement with BHP.

## 23.3 Miami Operations

Miami Operations, located east of PVM, is under sole ownership of FMI. Its leaching facilities, SX-EW plant, and Live Oak pit are the nearest FMI operating areas to PVM. The Live Oak deposit was discovered in 1872 and is thought to occupy the western section of the Miami-Inspiration disseminated copper deposit, along with the Bluebird, Oxhide, and Joe Bush pits. The Miami mine is a porphyry copper deposit that has leachable oxide and secondary sulfide mineralization. The predominant oxide copper minerals are chrysocolla, copper-bearing clays, malachite, and azurite (mindat.org 2014). Chalcocite and covellite are the most economically important secondary copper sulfide minerals (Peterson 1962, 83–8). The mine produces cathode copper from the mine ore, using heap leach and solvent extraction methods.

Miami Operations includes a smelter, which was modernized in 1974 to meet CAA standards and was further modernized and expanded in 1992. The SX-EW plant was commissioned in 1979. A continuous-cast copper rod mill, commissioned in 1966, accepts cathode feed from multiple Arizona sources. The refinery was closed in 2005 (Freeport 2014).

# 24.0 OTHER RELEVANT DATA AND INFORMATION

There are no additional data that are relevant to this report.

# 25.0 INTERPRETATION AND CONCLUSIONS

## 25.1 Risk Assessment

PVM has developed and maintains a comprehensive risk register of its operations. Capstone conducted a risk workshop specifically related to the impact of extending the mine life through 2025. Appropriate controls are in place, and no risks have been identified that would have a material effect on meeting the objectives set forth in this technical report.

## 25.2 Key Outcomes

- Ore mining and milling capacities can be sustained at 55,100 tpd for the first 2 years, increasing to 57,300 tpd into 2026, while meeting copper production forecasts defined in the operating plan.
- The proven and probable mineral reserves of 256 M tons (232 Mt) at 0.33% TCu, at a cutoff grade of 0.18% TCu, will sustain mining operations into 2026.
- PVM has the existing infrastructure to support the operational requirements of the site for PV2. Minor upgrades to the mine maintenance facilities have been identified to increase operational efficiency.
- No upgrades or major processing changes are foreseen to the milling or leaching circuits.
- The existing water sources are capable of supplying adequate water, and no substantive changes are anticipated to surface water management.
- PV2 will not require use of federal lands beyond existing authorizations and encroachments that are currently included in the Plan of Operations. PVM has the necessary permits to operate the mine into 2026 and ancillary operations (SX-EW) through 2030.

## 25.3 Conclusions

PVM has been in operation since 1974, developing a porphyry copper ore body using open-pit mining techniques and processing technology to produce a copper concentrate that is well accepted by the market. The PV2 study extends the mine life by 8 years into 2026, while maintaining current fundamental mine and mill operational status. The conclusions of the QPs are summarized in Section 1 and detailed in the preceding sections of the report. Based on the site technical and financial data evaluated by the QPs for the technical report, the planned PV2 extension of the mine life into 2026 is justified.

# 26.0 RECOMMENDATIONS

PVM is an established, operating mine and mill complex that has recently undergone a major renovation as part of the 2012 restart by BHP. This technical report substantiates the conclusions of the QPs, which state that operational objectives will be achieved by implementing the $187.9 M sustaining capital and minor operational initiatives outlined in this report.

PVM has significant mineral resource that has not been evaluated in the PV2 study due to permitted tailings storage area and property boundaries. The evaluation of the options to develop this resource may offer significant opportunity to Capstone.

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