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Technical Report

Hermosa Property Mineral Resource and Taylor Deposit PEA update

Arizona Mining Inc.

Santa Cruz County, Arizona, USA

In accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators.

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1 Summary

1.1 Introduction

AMC Mining Consultants (Canada) Ltd. (AMC) was commissioned by Arizona Mining Inc. (AZ) to prepare an updated Mineral Resource estimate and updated Preliminary Economic Assessment (PEA) and report according to National Instrument 43-101 Technical Report (NI 43-101 Technical Report or Report) for the Taylor Zn-Pb-Ag deposit located on the Hermosa Property, (Property). The Property is located in Santa Cruz County near the town of Patagonia, southern Arizona, USA.

The Property hosts two known mineral deposits, the Taylor Deposit and the Central Deposit. The latest Technical Report for the Property was completed by AMC, dated 11 April 2017 and titled "Hermosa Property, Taylor Zn-Pb-Ag Deposit Preliminary Economic Assessment" for Arizona Mining Inc., with an effective date of 29 March 2017, (March 2017 Technical Report). Prior to that the Taylor Deposit was also the subject of an NI 43-101 report dated 29 November 2016 and reported additional Mineral Resources for the Taylor Deposit, (November 2016 Technical Report). The Property is 100% owned by Arizona Minerals Inc. (AMI) a wholly owned subsidiary of AZ. This Technical Report provides an update of the Mineral Resource estimate for both the Central and Taylor Deposits and reports the updated results of the PEA.

AMC are responsible for managing and preparing the Technical Report with inputs from Ms D. Nussipakynova AMC, Mr G. Methven AMC, Mr C. Kottmeier AMC, Mr Q. Jin, SGS North America Inc., Mr R. Michael Smith, Newfields Mining Design and Technical Services, Mr E. Christenson of WestLand Resources Inc., Mr D. Bartlett of Clear Creek Associates, Ms L. Bloom, Analytical Solutions Ltd., and Mr C. Kaye, Mine and Quarry Engineering Services Ltd.

The economic analysis in the PEA is preliminary in nature and is based, in part, on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

All currency amounts and commodity prices are in United States (US) dollars unless stated otherwise. Quantities are stated in both Imperial and SI units. Commodity weights of measure are in ounces per short ton (oz/ton) or percent (%) unless stated otherwise.

1.2 Location, ownership, and history

The Property is located approximately 50 miles (81 km) southeast of Tucson, Arizona; 15 miles (24 km) northeast of Nogales in Santa Cruz County, Arizona, and eight miles (13 km) north of the international border with Mexico. The area has a semi-arid climate. Available water well information and preliminary hydrological analysis suggests adequate water supplies are available for project requirements. Experienced, skilled workers are readily available within a reasonable commuting distance. All major services and supplies are available in Tucson.

AZ holds 100% ownership interest in the Property through its wholly owned subsidiary AMI, a Nevada corporation, which was registered on 4 October 2005 with the Arizona Corporation Commission to do business within the State of Arizona.

The Property was explored by ASARCO intermittently from 1940 through 1991. Pan American Silver held the Property between 1994 and 2002 but confined their activity to internal economic evaluations. AZ has been active on the Property since 2006.

1.3 Geology and mineralization

Southeastern Arizona lies within a belt of 1600 to 1700 Ma-age Proterozoic rocks. Late Precambrian-Early Paleozoic rifting split the Proterozoic basement into a number of separate continental blocks with passive continental margins. Phanerozoic shelf-type sediments overlie the Precambrian basement.

The oldest rocks in the Patagonia Mountains are Proterozoic-age granodiorite that are overlain by Cambrian-age sedimentary rocks. Most of Arizona was above sea level during the Ordovician and Silurian. Widespread sedimentary deposition resumed in the upper Devonian. Pennsylvanian-Permian-age sandstones, shales and carbonates were deposited during a time of shifting and cyclical environments. The Pennsylvanian Naco Group of southeastern Arizona is comprised of Pennsylvanian Horquilla Limestone, the Pennsylvanian-Permian Earp Formation and the Permian Colina Limestone, Epitaph Dolomite, Scherrer Formation, and Concha Limestone. The Epitaph Formation, Scherrer Formation, and the Concha Limestone underlie the Hermosa project and are disconformably overlain by Jurassic rhyolites.

Mesozoic-age volcanic, sedimentary and intrusive rocks lie disconformably above the Paleozoic stratigraphic sequence. Cretaceous-age intermediate and felsic volcanic and intrusive rocks cover much of the Property and surrounding areas. In the northwestern Patagonia Mountains, Jurassic granite intrudes Triassic to Jurassic volcanic and sedimentary rocks. Most of the central and southern parts of the range consist of Laramide-age (64 to 58 Ma), medium to coarse-grained hornblende granodiorite batholithic rocks. The batholith is bounded by northwest-striking faults and its emplacement is thought to have been structurally controlled.

Nine stratigraphic domains have been recognized within the Property: three carbonate units of Paleozoic age (in ascending order, Epitaph, Scherrer, and Concha) that are overlain by two volcanic units; the Hardshell (Jurassic age) and Meadow Valley (Cretaceous age) that dip gently to the northwest. A listric thrust that dips to the southwest and predates the two younger volcanic units places the Epitaph, Scherrer and Concha over an older volcanic unit (Older Volcanics Triassic / Jurassic age) and a repeated section of Paleozoic carbonates (Concha, Scherrer, and Epitaph) which were modeled separately and comprise the sixth, seventh, eighth and ninth domains respectively.

The Property hosts two stratigraphically controlled mineral deposits, the Taylor Deposit and the Central Deposit. The Taylor Deposit is predominantly a carbonate replacement deposit (CRD) which permeates downward, to significant depth of 3,600 ft (1,100 m), into three recognized sedimentary formations on the Property and is comprised of Zn-Pb-Ag-Cu sulphides. The Central Deposit is a Manto-style replacement deposit which is confined to the upper 100 to 500 ft of the Concha limestone and the overlying Jurassic rhyolites. The Central Deposit is comprised of Mn oxides with accessory silver and zinc carbonate and silicate minerals. The host rocks (Jurassic Rhyolites and Concha Limestone) strike approximately southwest-northeast and dip ± 25° to the northwest. They do not appear to be significantly disrupted by post-mineralization faulting at deposit scale.

1.4 Exploration and data management

AZ has been active on the Property since 2006; the work carried out has been predominantly exploration and delineation drilling.

Drill programs conducted by AZ on the Property between 2007 and 2017 are summarized in Table 1.1. The last drilling program was conducted after the PEA estimation in March 2017 and consists of 65 new drillholes added to the database after 16 February 2017. This was the cut-off date of drilling for the previous Mineral Resource.

Table 1.1 AZ drill programs

Year	Type	Number	Length (ft)	Length (m)	Target
2007	Core	4	4,450	1,356	Central Deposit
2007 & 2008	Core	3	7,928	2,416	Central Deposit
2009	Core	6	12,005	3,659	Central Deposit
2010 – 2012	Core	57	81,846	24,947	Central Deposit
2012	RC	6	2,480	756	Central Deposit
2010 – 2012	RC	159	101,813	31,033	Central Deposit
2007 – 2012	Core	16	32,846	10,011	Taylor Deposit
2014 – 2015	Core	8	29,337	8,942	Taylor Deposit
2016	Core	35	144,010	43,894	Taylor Deposit
2016 – February 2017	Core	37	151,483	46,172	Taylor Deposit
Post February 2017	Core	65	234,514	71,480	Taylor Deposit
Total		**396**	**802,712**	**244,666**	

1.5 Mineral Resource estimates

AZ commissioned AMC to prepare an independent estimate of the Mineral Resources of the Property compiled using exploration data available up to 17 October 2017. Table 1.2 is a summary of the Mineral Resources for the Taylor Deposit stated at 30 November 2017.

Table 1.2 Taylor Deposit Mineral Resources

Classification	Million tons	Zn (%)	Pb (%)	Ag (ozpt)	ZnEq (%)
Measured	15.2	4.0	4.0	1.6	9.6
Indicated	85.8	4.2	4.3	2.2	10.5
Measured and Indicated	101.0	4.1	4.3	2.1	10.4
Inferred	43.6	3.9	4.8	3.4	11.9

D. Nussipakynova, P.Geo. from AMC, is the Qualified Person under NI 43-101 and takes responsibility for the Mineral Resource estimate.
CIM Definition Standards (2014) were used for reporting the Mineral Resources.
Mineral Resources are reported as of 30 November 2017.
Stated at a cut-off grade of 4% ZnEq based on prices, recovery, and costs as follows:
Prices of $1.00/lb for zinc, $0.95/lb for lead, and $20.00/oz for silver.
Average processing recovery factors of 92% for zinc, 95% for lead, and 90% for silver.
Total operating costs are estimated to be of the order of $60/ton.
Numbers are rounded and may not match later detailed tables.
Source: AMC Mining Consultants (Canada) Ltd.

Grades of silver, lead, and zinc for the Taylor Deposit have been estimated for the sulphide domains and the resource has been tabulated on the basis of Zinc Equivalency (ZnEq). Copper was not used as a component of the ZnEq formula because of its relatively low abundance and uncertainty pertaining to mineral processing and recovery and therefore to its value.

The ZnEq formula to equate lead and silver to zinc is:

ZnEq = [((Pb%/100)*2000*$0.95*95%) + ((Zn%/100)*2000*$1.00*92%) + (Ag ounces/short ton*$20.00*90%)] /((2000*$1.00*92%)/100)

The price and recovery inputs to the equation are given in the footnotes of Table 1.2.

Table 1.3 is a summary of the Mineral Resources for the Central Deposit stated at 30 November 2017.

Table 1.3 Central Deposit Mineral Resources

Classification	Million tons	Zn (%)	Ag (oz/ton)	Mn (%)	Oxval ($/ton)
Measured	21.8	1.9	3.3	9.2	262
Indicated	41.7	2.3	1.7	9.8	257
Measured and Indicated	63.5	2.2	2.3	9.6	259
Inferred	1.8	2.6	1.6	7.4	207

D. Nussipakynova, P.Geo. from AMC, is the Qualified Person under NI 43-101 and takes responsibility for the Mineral Resource estimate.
CIM Definition Standards (2014) were used for reporting the Mineral Resources.
Mineral Resources are reported as of 30 November 2017.
Stated at a cut-off grade of $100/ton Oxval based on prices, recovery, and costs as follows:
Prices of $1.00/lb for zinc, $20.00/oz for silver, and $0.91/lb for manganese.
Average processing recovery factors of 55% for zinc, 72% for silver, and 86% for manganese.
Total operating costs are estimated to be on the order of $100/ton.
Numbers are rounded and may not match later detailed tables.
Source: AMC Mining Consultants (Canada) Ltd.

Silver, zinc, and manganese grades have been estimated for the Central Deposit. Although manganese is generally the most valuable metal of the three, it was decided to tabulate the resource on the basis of the combined monetary value of the three metals rather than as a manganese equivalency because a manganese equivalency is considered an unconventional concept. The dollar value is based on metal grade times metal price times metal recovery. The combined metal value is termed Oxval (oxide value) and the formula is:

Oxval = ((Mn grade (%)* $0.91*86%)+(Zn grade (%)*$1.00*55%)+(Ag ounces / short ton*$20.00*72%)) where the recovery rate for manganese is 86%, for zinc 55%, and for silver 72%.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Neither deposit is materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, political, or other relevant issues. The estimates of Mineral Resources may be affected if mining, metallurgical, or infrastructure factors change from those currently anticipated at the Property.

1.6 Metallurgy

The Hermosa Taylor Deposit is a lead-zinc-silver deposit with relatively simple mineralogy. Upon review of the metallurgical testing data, it is clear that Hermosa Taylor mineralization responded well to a conventional sequential lead / silver – zinc flotation. The Hermosa Taylor mineralization does not require the addition of lime to achieve good process recovery.

Most of the composites tested for Bond ball mill work index were in the medium to moderately hard range. The Bond abrasion indices indicate mild to medium abrasiveness.

The projected final lead concentrate graded 69.7% Pb and 1,072 g/t Ag at a lead recovery of 95.4% and a silver recovery of 69.2%. The final zinc concentrate graded 56.1% Zn at a zinc recovery of 92.7%. The overall silver recovery was 92.4%.

Mercury and fluorine levels of cycle F concentrates from all locked cycle tests were below levels deemed problematic to smelters. The manganese content of the final zinc concentrate was 1.35% Mn. Zinc smelters in particular could start with penalty rates as low as 0.5% Mn.

1.7 Processing

The project processing facility is designed to treat 10,000 tpd of lead, zinc, and silver material at an operational availability of 92%. The processing flow sheet for the project is a standard flow sheet that is commonly used in the mining industry, including conventional flotation recovery methods typical for lead-zinc material. Figure 1.1 below is a process plant overall flowsheet. SGS completed the process design based on the results of 2017 SGS Lakefield metallurgical testing programs.

Run-of-mine (ROM) material will be crushed in a primary jaw crusher that is located adjacent to the underground mine portal. From there it will be conveyed to the processing facilities where it will be ground to 80 percent finer than 150 microns in a semi-autogenous grinding (SAG) and ball milling circuit.

The mineralized material is further processed in a flotation circuit consisting of lead flotation followed by zinc flotation. The majority of the silver will be recovered in the lead flotation circuit and some silver will also be collected in the zinc flotation circuit.

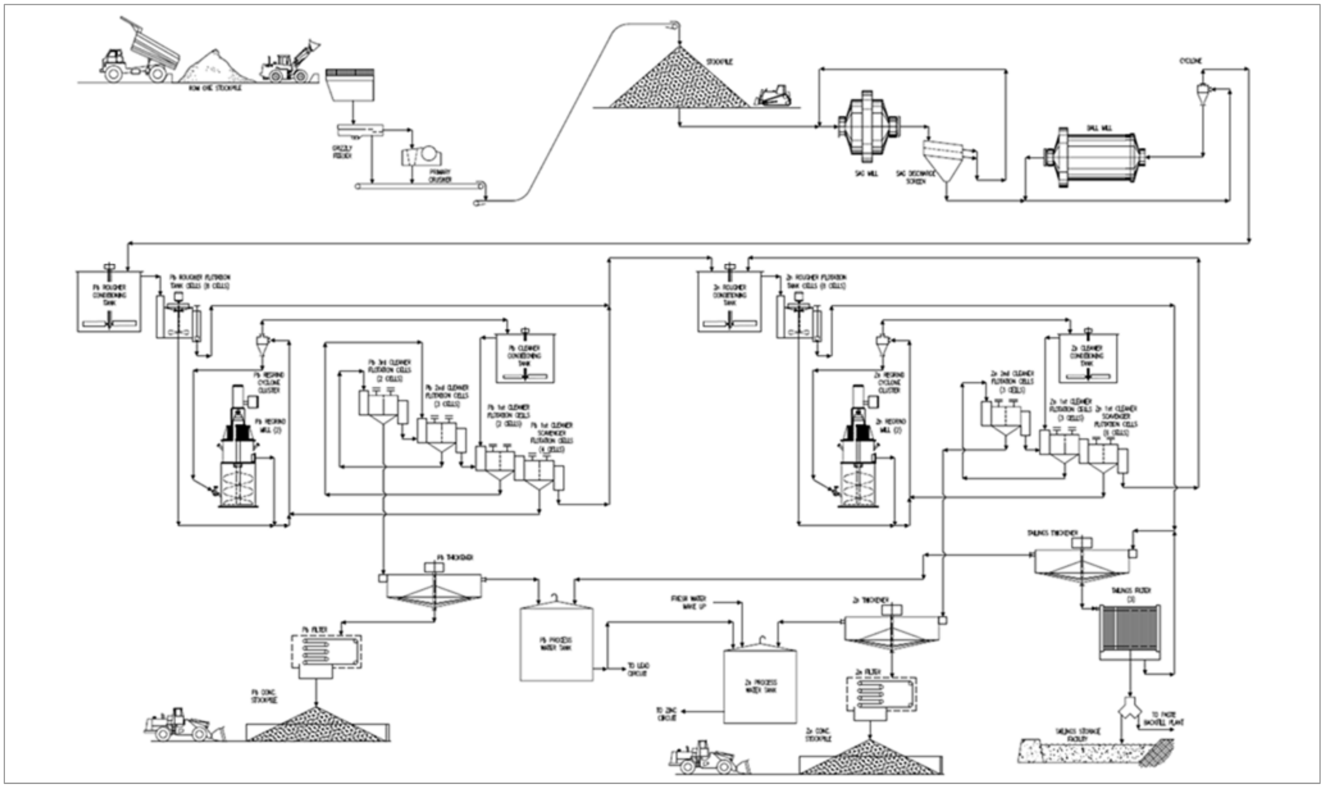
Lead sulphide will be recovered in a rougher flotation bank, producing a concentrate that will be upgraded to smelter specifications in three stages of cleaning. Tails from the lead flotation section will then be conditioned for zinc sulphide flotation. The process scheme for zinc flotation also includes a rougher bank and two stages of cleaning to produce smelter-grade zinc concentrates. For both lead and zinc sections, the rougher flotation concentrates will be reground to 80 percent finer than 38 microns prior to cleaner flotation to liberate the sulphides for further upgrading.

Tailings from the flotation circuit will be thickened, filtered and conveyed to a splitter at the plant. From there, normally 40% of the filtered tailings will be conveyed to tailing storage facility and the remainder will be disposed of as backfill into the underground mine.

Water will be reclaimed from the tailing thickener overflow and from the tailing filtrate. Process make-up water will be pumped from the water wells.

Lead and zinc concentrates will be thickened, filtered, and discharged to a covered stockpile. They will then be reclaimed by front-end loader onto highway haulage trucks for ocean shipment to smelters.

Figure 1.1 Flowsheet for lead and zinc concentrate production



1.8 Surface infrastructure

Infrastructure near the project site is either available or readily upgradable as described in this report. There is reasonably modern infrastructure surrounding the Hermosa project and it is in close proximity to a qualified work force for construction and also operating the mine site. The infrastructure includes a paved road between Patagonia and the project mine site. The existing power line to the site is adequate for temporary facilities only and will need to be upgraded to supply power for the project.

The water source for the project has been investigated through the installation and testing of three test wells with total depths ranging from 965 ft to 3,680 ft (294 m to 1,121 m). Constant rate pumping tests were performed at rates ranging from 321 gpm to 1,103 gpm (1,215 LPM to 4,175 LPM). Water level data collected in each pumping well as well as numerous surrounding wells were used to evaluate the hydraulic characteristics of the bedrock system. Results were incorporated into a regional groundwater model which was used to assess the water production potential of the bedrock. These field data and computer modelling indicate that there is sufficient, accessible groundwater to supply mining operations at the currently planned source supply of 650 gpm (2,460 LPM). A reasonable amount of infrastructure has been included for development of this project.

1.9 Tailings storage facility

The Hermosa project will include an underground mining operation where minerals will be extracted through a milling process. After mineral extraction, approximately 60% of the tailings will be sent back underground as backfill and the remaining tailings will be filtered and placed in a dry stack tailings storage facility (TSF) on the surface. In addition to tailings, mine development rock will be generated during the mining process. It is anticipated that approximately half of the mine development rock will contain sulphide minerals and will be classified as potentially acid generating (PAG) rock and the other half classified as non-PAG. All PAG rock will be stored within the dry stack TSF on the surface. The PAG rock will be co-mingled with the tailings, thereby encapsulating the PAG rock within the dry stack tailings. The non-PAG rock will be utilized as construction material for the dry stack TSF and related infrastructure.

Two dry stack TSF locations, Trench Camp and Hermosa, will be designed to contain the dry stack tailings and PAG development rock produced from the mining operation. Additionally, the Trench Camp TSF will store historic tailings which currently reside within the proposed TSF footprint. The TSFs will utilize the majority of the non-PAG development rock as armoring on the exposed face of the tailings to prevent water and wind erosion. The design of each TSF consists of a perimeter road which fully encloses a composite lined TSF basin, consisting of a low permeability soil layer (prepared subgrade) and geomembrane liner. Located directly on the geomembrane liner, a protective layer augmented by drainage pipes sited in the topographic lows will be used to provide cover over the geomembrane liner. The protective layer (granular material) will also limit hydraulic head and hydraulic gradient on the liner system by promoting drainage. Each TSF is designed as a “zero discharge” (non-discharge) facility where water liberated from the tailings will be re-used in the process circuit along with any meteoric water collected from precipitation events, falling directly on the TSF footprint.

1.10 Underground mining and infrastructure

The climate in the project area varies from high desert in the Sonoita Valley to the steppe-like climate of the higher elevation grasslands and scrub area. Average rainfall is 17 in (432 mm) per year, with the majority of precipitation occurring between June and October. The Project area is located within the Middle Sonoita Creek and Harshaw Creek watersheds.

Groundwater flows in bedrock fractures at the site. There is little to no alluvium present. Groundwater is recharged from precipitation at higher elevations and in the washes and drainages which carry surface flows from rain events north and northwest out of the basins.

Porosity of fractured bedrock aquifers is generally low, on the order of 1% to 2%. However, mineralization can result in higher porosities. Based on initial aquifer testing results at selected locations, it is estimated that groundwater inflows to the underground mine will be low, possibly less than 5 l/s, depending on the geometry of the underground workings.

Call & Nicholas, Inc. (CNI) undertook the preliminary geotechnical study for the underground works on the Project. The recommended mining method is sub-level open stoping (SLOS). Mining will take place initially from the primary stopes followed by secondary stopes. The recommended maximum stope dimensions for mining parallel to strike in the Concha are 148 ft H by 69 ft L by 50 ft W (45 m H by 21 m L by 15 m W) and in the Epitaph are 100 ft H by 45 ft L and 50 ft W (30 m H by 14 m L by 15 m W). While CNI recognize a third rock type, the Scherrer, is rich in mineralization and is planned for mining, it was not separated as a distinct geotechnical domain. Any mining that occurs within the Scherrer should follow the criteria of the Epitaph rock type. Smaller stopes (20 ft high) were assumed to allow recovery of fringes and irregular shaped portions of the deposit to increase resource recovery; the smaller stopes attracted a higher operating cost.

Conceptual stope dimensions were optimized for height, rather than length. In both domains, because of the geologic joint fabric, mining perpendicular to the strike of the deposit allows for greater achievable dimensions. Analyses were limited to a depth of 4000 ft (1,219 m).

The Concha rock type was identified as the superior mining host rock. The rock quality designation (RQD = 93%), joint conditions, and intact rock strength qualify this rock to be of good quality per Barton's Q' classification system. The Epitaph rock type was identified as the lesser quality mining host rock. While the Epitaph has an identical rock quality designation (RQD = 93%), the joint conditions were of significantly less quality than those from within the Concha rock type.

In order to achieve nearly full mineral recovery at the project, paste backfill is projected to be used to fill open stopes following their excavation. By filling these stopes with paste backfill, pillars will be established that will subsequently become the walls of later stage (secondary) stopes. In order to stand at heights up to 147 ft (45.0 m) when mining in the Concha, a paste fill strength of 967 kPa is calculated to be required (Mitchell, et al.). When mining in the Epitaph, in which stope heights are less 100 ft (30.0 m), a paste fill strength of 645 kPa is calculated to be required.

Early access to the deposits will be by means of the decline, which will provide exposure to the mineralization by late 2020, and allow drilling from underground, bulk sampling, and test mining.

Development drifts include all decline drifting and level access drifts, with assumed dimensions of 18 ft by 18 ft (5.5 m by 5.5 m). Due to the good quality of the rock, no support beyond spot bolting should be required in the development drifts although a standard bolting pattern for all development is recommended. AZ should anticipate the presence of infrequent faults that may require some support. Surficial support in the form of fibre-reinforced shotcrete (fibrecrete), or shotcrete in conjunction with pattern bolting may be needed when mining through these faults.

Production drifts include all stope accesses; bottom cuts, middle cuts, and top cuts, with dimensions of 14.8 ft by 14.8 ft (4.5 m by 4.5 m). Stope bottom cuts will not generally require any support beyond infrequent spot bolting. However, to account for faulting and areas of lesser quality ground, CNI recommend using fibrecrete or shotcrete with systematic bolting 6 ft lengths, 5.2 ft spacing (1.8 m lengths; 1.6 m spacing) in approximately 20% of all production drifting.

The proposed shaft dimensions are 21 ft diameter (6.5 m). The total shaft length is 3,625 ft (1,105 m). The temporary support requirements consist of 7.8 ft (2.4 m) friction bolts and welded-wire mesh. Permanent support includes concrete lining with a design compressive strength of 418,000 lbs/ft^2 to 585,000 lbs/ft^2 (20-28 MPa), minimum lining thickness of 17.7 in (450 mm).

Several options exist to access the Taylor Deposit. Based on previous trade-off studies to evaluate the mineralized material and waste handling system, a combination of a decline and vertical shaft system was selected as the optimum methodology based on economics and operability. Development of the access decline is projected to commence at the same time as sinking of the vertical shaft. Once the sub-levels are established, development mineralization is extracted via the decline. Stope production is projected to commence in Year 4 when the shaft is ready to begin hoisting.

The method that best supports low operating cost, high productivity with good recovery and low dilution is sub level open stoping (SLOS). Mining activities will be fully mechanized and large modern trackless mobile equipment will be employed throughout. Ground conditions are generally expected to be fair to good, with a relatively small proportion of poor ground anticipated.

AMC used a function of the Datamine™ software, Mine Stope Optimizer (MSO) to evaluate preliminary stope wireframes for the SLOS mining method. Varying stope heights between 60 ft and 100 ft (18 m and 30 m) were generated. The stope height of 100 ft (30 m) was selected as the optimum when considering planned dilution. The amount of dilution in a stope is a trade-off with the additional development required to access stopes with less height. The selected stope height of 100 ft (30 m) is within the maximum stope size recommendations from the geotechnical stope design criteria.

On completion of the stope shape optimization, it was identified that there remained flat dipping relatively thin lens shaped mineralized zones with reasonable grade between the larger stope shapes. Alternative stope shapes with a height of 20 ft (6.1 m) that would be applicable to a more selective cut and fill mining method were generated in these areas.

As part of the economic optimization process, a high grade core of mineralization was identified above a cut-off grade of 15% ZnEq, that is located between 2900 L and 3320 L. The high grade material is accessible from each level independently, with associated stopes having the possibility of being mined simultaneously, using more selective longhole type mining methods over stope heights of 60 ft (18 m) floor to floor. A mine plan and mine design were developed to allow early access of the high grade core between Year 4 and Year 7 (inclusive) of the Life of Mine (LOM) plan. The use of pastefill ensures that lower grade material is not sterilized but is extracted during a second pass.

Stope wireframes were generated above a cut-off grade of 6% ZnEq in order to determine the potential material for mining. The potential material for mining is the Mineral Resource above the cut-off grade that includes the application of mining factors such as recovery and dilution. AMC has applied a dilution factor of 5% at zero grade to the Mineral Resource and a mining recovery factor of 95% has been applied to the stopes. For the small cut and fill stopes, AMC has applied a dilution factor of 10% at zero grade to the Mineral Resource and a mining recovery factor of 80%.

Once stope wireframes were generated, a check was made to remove any outlying stopes that would not be economic when the cost of access development was included. The cost of access development was determined for each level and each level was evaluated to determine if the value was sufficient to pay for its access. The potential material for mining associated with the potentially economic stopes are summarized in Table 1.4.

Table 1.4 Potential material for mining

Tons (M)	Zn (%)	Pb (%)	Ag (oz/t)	ZnEq (%)
96.7	4.0	4.3	2.22	10.4

In order to maximize the Net Present Value (NPV) of the project, higher grade material is targeted as the optimal starting position. The lower grade material will be extracted as a second pass.

Mining panels consist of five 60 ft (18 m) levels that will be mined in a bottom up mining sequence. Once the higher grade material is extracted, the mine will extract mineralized material using primary and secondary stopes that are backfilled with cemented pastefill. The primary stopes will be mined and backfilled prior to mining secondary stopes on a level sequence. As the level advances towards the south of the deposit, the level above can commence primary stoping. The cut and fill stopes are extracted at the completion of SLOS on each level.

In order to determine an appropriate production rate which can be supported by the deposit, AMC has used a combination of Taylor`s rule of thumb and vertical tons per metre to determine anticpated production ranges. Production rate based on Taylors rule of thumb is estimated at approximately 4.8 Mtpa (4.4 Mtonnes pa).

Most successful mines do not exceed 40 vertical metres / annum (vmpa). The deposit has approximately 105 kt/vm of mineralization and this would support a production rate of approximately 4.2 Mtpa (3.8 Mtonnes pa).

AMC has completed a high level schedule of the conceptual mineralized material production aimed at meeting the target production rate of 10,000 tons per day. Based on this production schedule, the targeted throughput of 3.6 Mtpa (3.3 Mtonnes pa) is achievable. AMC considers that this production could be increased to 12,500 tpd, given the potential to mine from multiple fronts on each level as well as over multiple levels at a time. For this study AMC has scheduled production at a rate of 3.6 Mtpa (3.3 Mtonnes pa).

Underground layouts were prepared for the shaft and decline design layout and the development quantities determined by type for cost estimation and scheduling. Vertical development is generally associated with vertical ventilation raises or passes. All waste access development was assumed to be 18 ft by 18 ft (5.5 m by 5.5 m) and all development in mineralization to be 14.8 ft by 14.8 ft (4.5 m by 4.5 m). The total development required over the LOM is summarized in Table 1.5.

Table 1.5 Development quantities by type

Description	Units	Value	Units	Value
Decline	(ft)	39,138	(m)	11,929
Lateral waste development	(ft)	363,147	(m)	110,687
Vertical raise development	(ft)	20,160	(m)	6,145
Vertical shaft development	(ft)	3,625	(m)	1,105
Total lateral development	**(ft)**	**402,285**	**(m)**	**122,616**
Total vertical development	**(ft)**	**23,785**	**(m)**	**7,250**

AMC also determined a shaft sinking schedule based on an average blind sinking rate of 8.2 ft/d (2.5 m/d). The schedule assumes that once the shaft has been sunk to the 2600 L, hoisting can commence, a six-month delay between sinking and hoisting to allow for fitting out the loading station was assumed.

The function of the ventilation system is to dilute / remove airborne dust, diesel emissions, explosive gases, and to maintain temperatures at levels necessary to ensure safe production throughout the life of the mine. AMC has undertaken a preliminary estimate of the ventilation requirements in consideration of the production rate, mineralized material handling system, and mining method. This methodology provides an estimate of total mine airflow for a consistent production rate of 3.6 Mtpa (3.3 Mtonnes pa). It is estimated that for a mine with SLOS and shaft hoisting including some ramp haulage, the total mine airflow should be 1,983,272 cfm (936 m^3/s).

The mine will be ventilated by a “Pull” or exhausting type ventilation system. That is, the primary mine ventilation fans will be located at the primary exhaust airways of the mine. Fresh air will enter each mine via the main intake raises or shaft with exhaust to the surface via dedicated return airways. Most production activities will require auxiliary fans and ducting with level airflows managed through regulators located at raise accesses.

Intake air will be provided via the 21 ft (6.5 m) diameter shaft, the decline and one fresh air raise 18 ft (5.5 m) in diameter. Air will be exhausted via three return air raises that are 14.8 ft (4.5 m) in diameter.

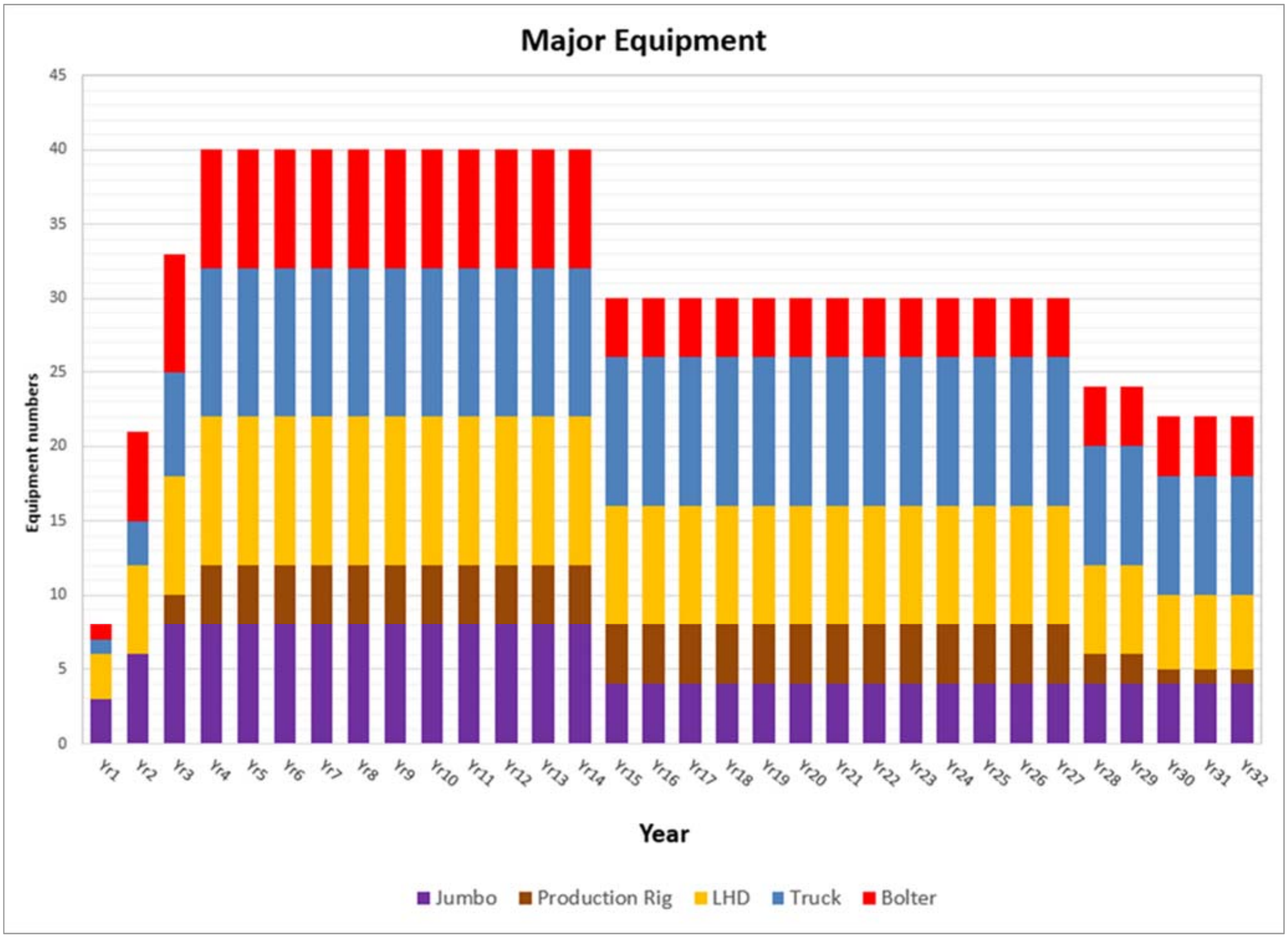
The stopes will be mined in a primary then secondary sequence. All stopes will be backfilled with cemented paste fill. Paste fill will be reticulated underground via boreholes and pipelines placed adjacent to the return air raise to the active mining level and then extended as mining progresses. Paste fill will flow under gravity to the active level and to the respective stope for filling. Fill delivery to all sublevels below each main level will be made via a series of inter-linked boreholes that connect to the perimeter drive on each sublevel.

AMC has conducted a high-level evaluation of the paste fill strength required and estimates a cement dosage of approximately 4.5% will generate a paste fill strength of 645 kPa, which is in line with geotechnical design specifications. Based on the production rate of 10,000 t/d (9.1 ktonnes pd) and the selected stope sizes, approximately 1,177,155 yards^{3}pa (900,000 m^3pa) of paste fill will be required.

AMC has generated high level capital cost estimates for the paste fill plant as well as the distribution system of US$12M and an operating cost of US$4.35 per ton of mineralized material (US$4.80/tonne).

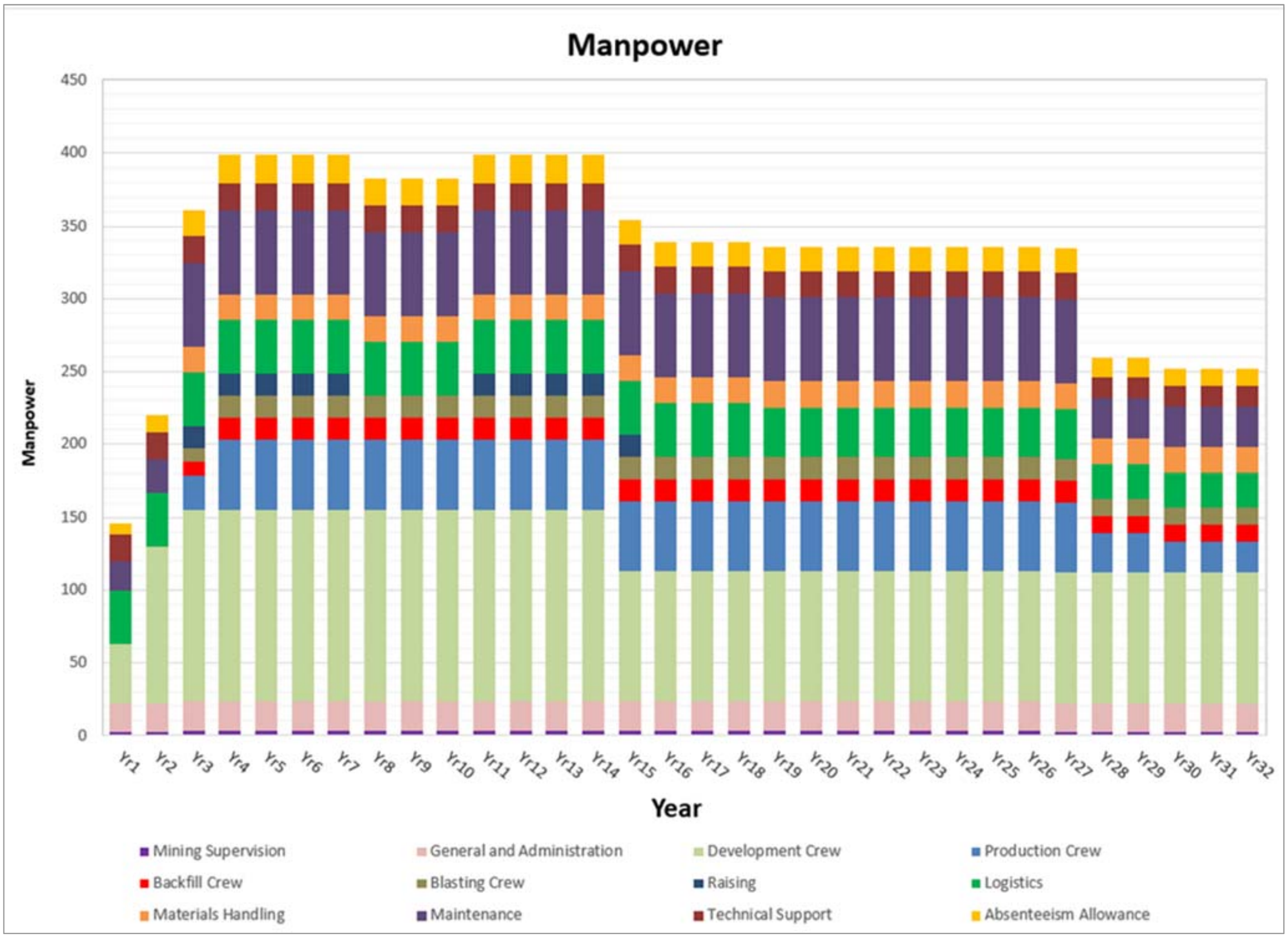
AMC has completed an estimate of the quantity of major equipment required to meet the production rate. The equipment numbers are based on average haul distances for trucks, number of active crews for development and the number of active stopes required to meet production. AMC has not selected specific equipment models, however recommended equipment includes twin boom Jumbos with 16 ft (4.9 m) feeds, long hole drills capable of 148 ft (45 m) holes, 50 t underground trucks and 12.5 t loaders. Equipment numbers are summarized in Figure 1.2.

Figure 1.2 Primary underground equipment



Based on the primary equipment requirements, AMC undertook an estimate of the expected labour required to meet the projected development and production schedules. A maximum of 408 personnel will be required for the mine, the workforce will operate on a three shift basis, and crews will rotate between day shift, night shift and rostered days off. The mine is assumed to be owner operated and a maximum of 272 underground personnel will be on site each day. A summary of the projected workforce is provided in Figure 1.3.

Figure 1.3 Underground work force



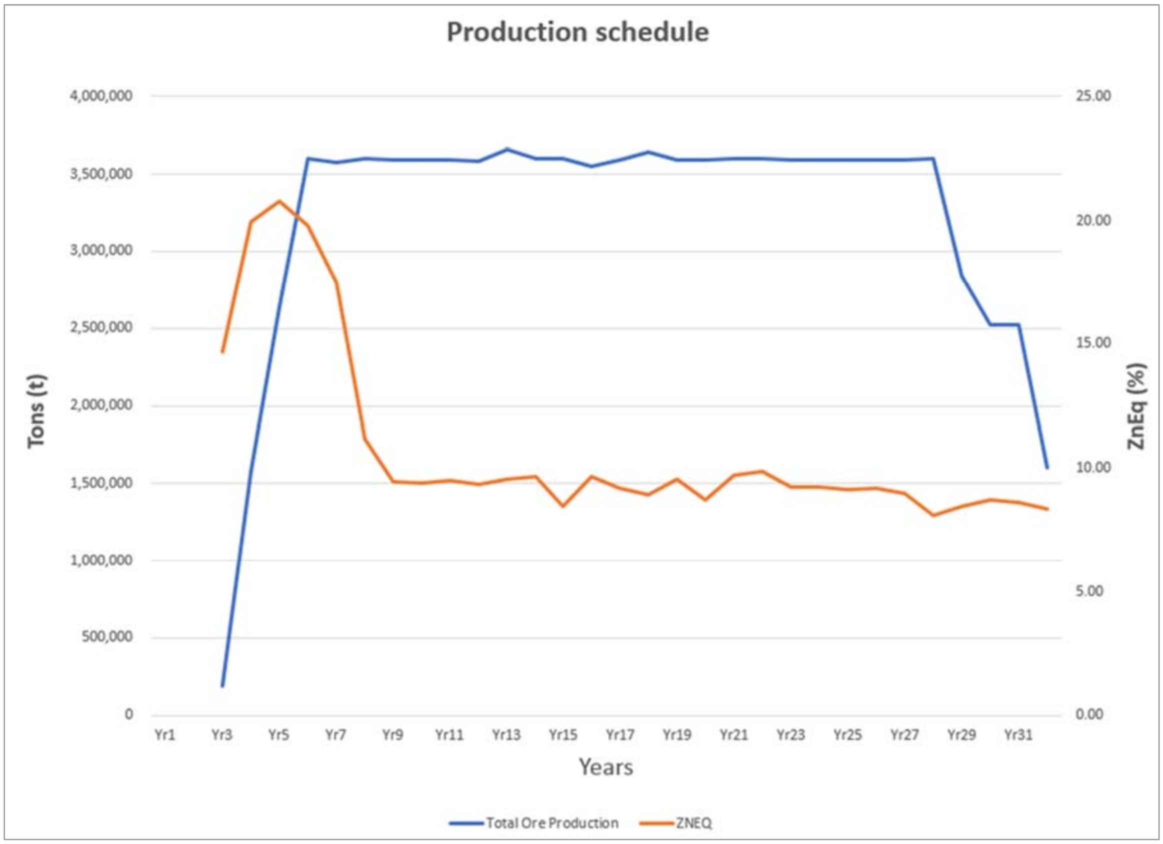
The SLOS stopes are mined at a rate of 1,000 tpd, with the target being 10,000 tpd (9.1 Ktonnes pd). A minimum of 42 stopes are required to be in operation to meet the production rate. A total of 14 stopes per level and an additional level to allow for any unscheduled production delays was considered necessary to meet the production rate.

The cut and fill stopes are mined towards the end of each production level; a production rate of 400 tpd is used for scheduling purposes.

A focused approach was adopted to mine higher grade in the initial production years using selective Longhole stoping and leaving any low grade material to be extracted in a second pass. The projected production schedule reflects this strategy.

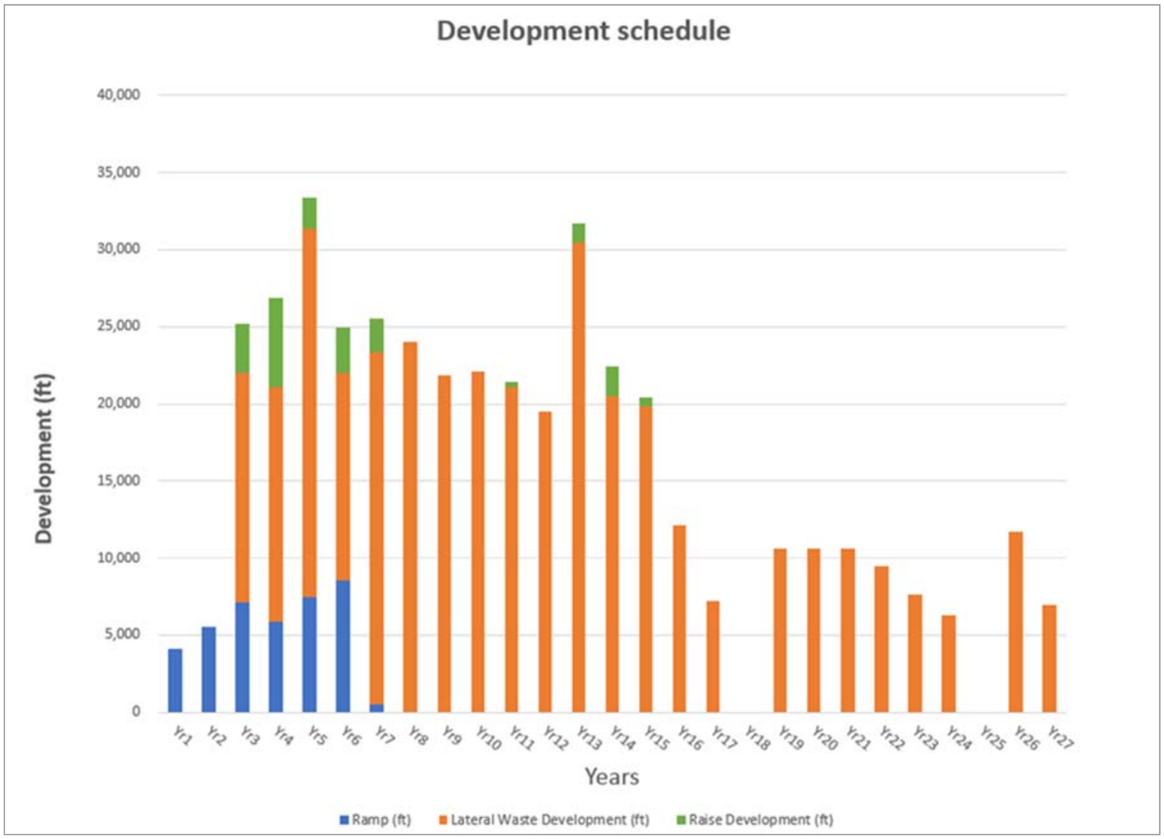
A summary of the production and ZnEq grade is shown in Figure 1.4.

Figure 1.4 Conceptual production schedule and ZnEq (%)



Decline development is scheduled at an advance rate of 460 ft/month (140 m/month) with the focus aiming at developing to the selected higher grade levels on 2900 L through to 3320 L. The development is projected to take two and a half years to access these levels, with mineralized material production from development commencing in Year 3. The projected development schedule by type is summarized in Figure 1.5.

Figure 1.5 Conceptual development schedule by type



1.11 Underground infrastructure

Twin 13.8 kV electrical distribution cables will be installed in the decline during development. One line will run from surface down the main zone decline and the second line will run down the Taylor Deeps decline which joins the main zone decline at approximately the 3720 L. Portable 600 kVA 4160 / 480 V / 120 V sub-stations will be established on each active level to power auxiliary fans and pumps, mining equipment and lighting panels respectively.

When the main dewatering sump and loading pocket are established a permanent 900 kVA sub-station will be located near shaft bottom to drive the peak power of the pump station as well as the loading pocket. Once the operating shaft and ventilation raises are established the 13.8 kV feeders will be redistributed to vertical routes to mitigate voltage drops and provide a loop distribution system for redundancy.

The primary power demand for the underground mine is associated with the main fans located on surface at the top of the exhaust raises, the main shaft, and the secondary fans and mining equipment. A maximum demand of 13.5 MW will be required for the underground mine.

During development of the main zone decline, seventeen staged submersible high-head low-flow pump stations will be established to dewater the mine. Each pump will transfer up the decline through a 4” steel grooved pipe line to the next sump. The pumps are sized so that the nominal 80 gpm (5 l/s) ground water and the 80 gpm (5 l/s) drilling and utility water can be handled by one pump; two pumps are installed in each sump in case of failure. Smaller pumps on the level can be used to transfer water from the face to the decline sumps. A similar arrangement is proposed for the second decline.

When the main zone decline is established near the shaft bottom, the main dewatering sump station will be developed. The main sump will consist of three horizontal multi-stage 400 hp pumps each capable of 160 gpm (10 l/s).

For the mines service water, a three-inch HDPE line will be installed in stages down the decline to provide fresh water for use in the mine. Every 100 vertical feet, a head tank or a pressure reducing valve will be installed to control the pressure in the line.

Four portable air compressors (one for each level) will be moved together with the primary mining equipment. The compressors will be sized so that they will be able to supply four operating drills.

A leaky feeder system will provide means for communication underground. All vehicles will be fitted with radios. A call bell and emergency system will be used when signalling the main production shaft.

The main production shaft will have a 21-foot (6.5 m) finished diameter and the production hoist will be a conventional double drum hoist with two skips discharging into the bins on surface in the headframe. Loading pockets will be on the 2600 L and 1600 L. The cycle times were estimated using 32.8 ft/s (10 m/s) velocity for the conveyance and allowing for creep in / creep out and decking time. The hoist is designed to accommodate the mine’s full production target of 10,000 tons per day, and the capacity of each skip is 27.6 tons (25 tonnes).

Underground mine services will include lunchrooms, a small maintenance shop for minor and urgent repairs, fuel and lubricant storage, and small magazines for high explosives and detonators.

1.12 Environmental

A variety of permits and approvals from state and federal agencies may be required in order to open and operate the project. The project will be built on Patented land, and does not envisage encroaching on Federal Land (USFS – Coronado National Forest).

By far, the most involved permitting effort will be the preparation of an Environmental Assessment (EA) or Environmental Impact Statement (EIS) by the U.S. Forest Service (USFS), in order to comply with the National Environmental Policy Act (NEPA). A NEPA-compliant analysis may be required of the potential involvement of the USFS Coronado National Forest (CNF) in the permitting process for the project. Another major permitting effort is the Aquifer Protection Permit (APP) from the Arizona Department of Environmental Quality (ADEQ), which covers any facility that discharges a pollutant either directly to an aquifer or the land surface or the vadose zone in such a manner that there is a reasonable probability that the pollutant will reach an aquifer. A third major permitting effort will be an Air Permit under the Clean Air Act, which is administered by ADEQ with oversight from the U.S. Environmental Protection Agency (EPA). A fourth permitting effort that is administered by ADEQ with oversight from the EPA will be an Arizona Pollutant Discharge Elimination System Permit (AZPDES), which covers any facility that discharge from a point source to receiving waters. Based on recent data, the time required for the USFS to prepare an EIS ranges significantly, from less than one to ten years or more, with a mean of approximately 4 years. The APP, Air Permit, and other permit actions can all be performed coincident with the EIS and may generally be timed to be completed at approximately the same time as the EIS. To minimize the environmental permitting timeline, a Plan of Operations (POO) should be submitted to the CNF, should it be necessary, as soon as possible after completion of a Pre-Feasibility Study or Feasibility Study.

Baseline studies to obtain background environmental data have been initiated and should be continued in the coming months. Results from exploration, geotechnical and hydrogeological investigations will be used to develop a POO to submit to the CNF as well as provide data to support the NEPA, APP, and other permitting processes.

1.13 Capital cost

Capital costs for the project were estimated by AMC for mining, SGS for the processing and associated plant infrastructure and Newfields for the tailings facility and ponds. Preproduction capital includes capital costs for Years 1 to 3; all capital from Year 4 to the end of mine life is termed sustaining capital.

The initial and sustaining capital cost estimates are summarized in Table 1.6 and include direct and indirect costs and a variable contingency that depends on the individual accuracy of the various components of the estimate. Contingency averages 14.1% of pre-production capital. Owner’s costs, Engineering, Procurement and Construction Management (EPCM) and contingency are spread equally over the three year pre-production period.

Table 1.6 Total estimated pre-production and sustaining capital costs

Item	Total (US$)	Pre-production capital (US$)		
		Year 1	Year 2	Year 3
Underground Development	582,498,200	5,670,194	7,560,259	35,052,409
Mine Equipment	138,800,000	10,000,000	16,000,000	18,000,000
Shaft	176,220,000	42,105,000	42,105,000	42,105,000
Backfill plant	10,000,000			10,000,000
Water to site	3,000,000	3,000,000		
Power	42,016,707	12,039,987	20,919,976	9,056,744
Roads	16,090,240	4,022,560	8,045,120	4,022,560
TSF - Trench and Hermosa	43,320,000	8,913,000	8,913,000	1,000,000
Processing	106,533,836		32,674,544	65,358,892
UG Infrastructure	23,441,825	3,365,304	3,365,304	3,365,304
EPCM	30,678,575	10,226,192	10,226,192	10,226,192
Owners Cost	900,000	300,000	300,000	300,000
Capitalized opex	6,724,522			6,724,522
Contingency	63,995,607	21,331,869	21,331,869	21,331,869
Total	**1,244,219,512**	**120,974,107**	**171,441,264**	**226,543,492**
Pre-production capital				**518,958,863**
Sustaining capital				**725,260,650**

Note: Totals do not necessarily equal the sum of the components due to rounding.

1.14 Operating cost

The total operating cost is estimated to be US$50.56/t for the mine. The total operating cost includes an average mining cost (US$38.02/t of mineralized material), processing cost (US$10.01/t of mineralized material), material placement at the TSF (US$0.53/t of mineralized material) and General and Administration cost (US$2/t of mineralized material). Operating cost estimates are based on a combination of bench mark costs for similar operations with equivalent production throughput and validated by first principle estimates for labour, power, reagents, and consumables.

1.15 Economic assessment

All currency is in US dollars (US$) unless otherwise stated. The cost estimate was prepared with a base date of Year 1 and does not include any escalation beyond this date. For net present value (NPV) estimation, all projected costs and revenues are discounted at 8% from the base date. Metal prices were selected after discussion with AZ and referencing current markets and forecasts and reports in the public domain. A regular corporate tax rate of 21% for federal tax and 4.9% for Arizona State tax is applied as the mining income would be earned in Arizona, USA. It is assumed that 3% of the NSR value would be the royalties to be paid.

AMC conducted a high level economic assessment of the conceptual underground operation of the Taylor Deposit. The underground mine is projected to generate approximately US$2,406M pre-tax NPV and US$1,979M post-tax NPV at 8% discount rate, pre-tax IRR of 54% and post-tax IRR of 48%. Pre-production capital is estimated at $519M and total project capital is estimated at US$1,244M. The project has a projected payback period of 1.5 years (discounted pre-tax cash flow from start of production). Key assumptions and results of the underground mine economic assessment are provided in Table 1.7

The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

Table 1.7 Taylor Deposit underground mine – key economic assumptions and results

Arizona Taylor Deposit	Unit	Value
Total mineralized rock	kton	96,671
Total waste production	kton	11,992
Zinc grade (1)	%	4.01%
Lead grade (1)	%	4.34%
Silver grade (1)	oz/ton	2.22
Zinc recovery (1)	%	92.7%
Lead recovery (1)	%	95.4%
Silver recovery (1)	%	92.4%
Zinc price	US$/lb	1.10
Lead price	US$/lb	1.00
Silver price	US$/oz	20.00
Zinc payable (2)	%	85%
Lead payable (2)	%	95%
Silver payable - Pb con(2)	%	95%
Silver payable - Zn con(2)	%	70%
Payable Zn metal	klbs	6,112,710
Payable Pb metal	klbs	7,608,117
Payable Ag metal	koz	162,566
Revenue split by commodity	Zinc	38%
Revenue split by commodity	Lead	43%
Revenue split by commodity	Silver	19%
Total revenue	US$ ($ 000)	17,583,425
Capital costs	US$ ($ 000)	1,244,220
Operating costs (Total) (3)	US$ ($ 000)	4,880,781
Mine operating costs (4)	US$/ton	38.02
Process and tails storage operating costs	US$/ton	10.54
Operating costs (Total) (3)	US$/ton	50.56
Operating cash cost (ZnEq)	US$/lb ZnEq	0.57
C1 Zn co-product costs (8)	US$/lb	0.49
C1 Pb co-product costs (8)	US$/lb	0.37
Total all-in sustaining cost (ZnEq)	US$/lb ZnEq	0.61
Payback Period pre-tax(5)	(Yrs)	1.51
Cumulative net cash flow (6)	US$ ($ 000)	7,260,841
Pre-tax NPV (7)	US$ ($ 000)	2,405,888
Pre-tax IRR	%	54%
Post-tax NPV (7)	US$ ($ 000)	1,979,101
Post-tax IRR	%	48%

1. LOM average.
2. Overall payable % includes treatment, transport, refining costs and selling costs.
3. Includes mine operating costs, milling, and mine G&A.
4. Underground mining costs only.
5. Values are pre-tax and discounted at 8%, from production start date Year 4.
6. Pre-tax and undiscounted.
7. At 8% discount rate.
8. Silver treated as a by-product.
9. The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

1.16 Conclusions and / or recommendations

1.16.1 Geology

Approximately 70% of the Taylor Deposit Mineral Resource has been classified as Measured and Indicated. The tons of Measured and Indicated show an increase of 39% from the March 2017 estimate. The Inferred portion of the Taylor Deposit is largely located on the periphery of the deposit and, therefore, the Qualified Person sees little benefit in AZ conducting additional surface drilling to upgrade the remaining 30% of the deposit as currently defined.

The calculation used to estimate bulk density and tonnage factors for the Taylor Deposit may be refined by the inclusion of pyrite content and possibly by inclusion of a term to account for porosity as well as other elements. Some of this data is currently available and it is recommended that AZ investigates the possibility of obtaining a calculated bulk density that is in closer agreement with measured values than has been achieved to date.

The Mineral Resource for the Central Deposit was estimated using fixed bulk density values; it is probable that these single values can be improved upon by using an approach similar to that advocated for the Taylor Deposit.

Geological and mineral resource risks associated with the Property are those attributable to any mineral exploration property at a comparable stage of exploration, namely the uncertainty attached to the continuity, grade, and tonnage of the mineral resource that has been estimated. Additional drilling to enhance the level of confidence that can be placed on the estimate, and the refinement of the bulk density equation will both help to mitigate this risk.

No further drilling is required for the Central Deposit at this time. However, desktop studies should be undertaken to determine the full resource, non-pit constrained, should it be decided that the most appropriate way to mine the Central Deposit is through common underground infrastructure developed for the Taylor Deposit.

1.16.2 Exploration

AZ should continue to aggressively explore the Hermosa project for additional zinc / lead / silver / copper resources. This is especially true for the near vertical vein sets extending across the Trench claim block and for the Taylor Deeps zone. The Trench Vein domain has the potential to impact the early production of the mine with higher than average grade zinc / lead / silver material. Additionally, the Taylor Deeps zone should be drilled to its extents as it could significantly increase the overall size of the deposit.

1.16.3 Mining

Additional work on the structural geology of the deposit is recommended. This will assist with better definition of the expected groundwater inflows and a more accurate estimate of the implications of faulting on ground conditions and ground support requirements. In the next level of study, AMC recommends obtaining geotechnical information regarding the shaft, portal and decline locations.

AMC recommends evaluating an alternative option that considers an increased production rate aimed at targeting high grade material in the early stages on mine life. Given the length of mine life, increased throughput would likely have a positive impact on the project economics.

The primary issues remain around permitting of the mine, including permitting of access roads and power supply upgrades. The underground deposit shows good potential for an economic mine with a relatively simple mining method and accessibility.

More work should be done to improve the extraction of the Mineral Resource considering more selective mining methods such as cut and fill. The production schedules completed for the PEA are level-based schedules, a more detailed schedule on a stope basis is recommended for the next level of study.

Operating cost estimates for mining have largely been based on benchmark costs for similar type of mining method and throughput. Operating cost estimates have also been estimated based on labour schedules and labour numbers and then split into cost categories for North American costs for a mining operation. The latter cost estimates are within 5% of the benchmark data, but it was decided to use the more conservative mining cost of US$35.35 (60 foot and 100 foot stopes) and US$60/t (20 foot stopes). On a weighted average basis, the total mine operating cost is estimated at $38.02/t. The backfill costs were estimated seperately and are based on costs for labour, cement, and consumables from local vendors.

1.16.4 Metallurgical

Additional lock cycle testing is recommended for each Domain, this will allow for validation of the final estimated recoveries and the selected concentrate grades.

It should be verified that potential smelters have the capacity and ability to accept the proposed quantity and quality of produced lead and zinc concentrates. Transportation, treatment charges, and refinery charges should be confirmed.

1.16.5 Surface infrastructure

Further studies to potentially improve the project economics include the following:

- Further review the topography and geotechnical conditions to minimize earthwork, foundation and conveying costs.
- Utilize on-site mining equipment to supplement the contractor equipment for rough grading required for the access roads to the site.
- Coordinate with the local power company to optimize the power line routing and connection to the electrical power grid.

1.16.6 Environmental permitting

AZ should continue baseline studies that will support the permitting processes expected to be required to develop the project. These include:

- Biological resources
- Cultural resources
- Hydrogeological studies
- Geochemical studies
- Air and weather monitoring
- Storm water quality
- Geotechnical (soil and rock) investigations

The estimated cost for additional baseline studies is $2.5M.

1.16.7 Project economics

The economic assessment results show that the projected pre-tax NPV is robust and remains positive for the range of sensitivities evaluated. The projected post-tax NPV performs similarly and also remains positive for the range of sensitivities evaluated. The sensitivity analysis examined the impact on pre-tax and post-tax NPV (at 8% discount rate) of a 15% positive or negative change in metal prices, operating costs and capital costs. The project is seen to be most sensitive to changes in zinc and lead prices, followed next by changes in operating costs.

Contents

Tables

Figures

Acronyms and abbreviations

Acronyms	Description
1000 L	1000 feet Level
AAS	Atomic absorption spectroscopy
ACC	Arizona Corporation Commission
ADEQ	Arizona Department of Environmental Quality
Ag	Silver
Ai	Abrasion index
AMC	AMC Mining Consultants (Canada) Ltd.
AMI	Arizona Minerals Inc.
ANFO	Ammonium Nitrate fuel oil
APP	Aquifer Protection Permit
ARD	Acid rock drainage
ASARCO	ASARCO LLC is a mining, smelting, and refining company based in Tucson
ASMI	Arizona State Mine Inspector
Au	Gold
AWQS	Aquifer Water Quality Standards
AZ	Arizona Mining Inc.
AZPDES	Arizona Pollutant Discharge Elimination System
BADCT	Best available demonstrated control technology
BC	British Columbia
BTU	British thermal unit
BWI	Bond ball mill work index
CAA	Clean Air Act
Capex	Capital expenditure (also Opex)
CEC	Certificate of Environmental Compatibility
CEET	Comminution Economic Evaluation Tool
Ci	Crusher index
Clear Creek	Clear Creek Associates
CLF	Chiricahua leopard frog
Clnr	Cleaner
CMC	Carboxymethyl cellulose
CNF	Coronado National Forest
CNI	Call & Nicholas, Inc.
CO	Carbon monoxide
CO_2	Carbon dioxide
COG	Cut-off grade
Con	Concentrate
Corps	U.S. Army Corps of Engineers
CRD	Carbonate replacement deposit
CRM	Certified reference material
Cu	Copper

Acronyms	Description
$CuSO_4$	Copper Sulphate
CWA	Clean Water Act
EA	Environmental Approval
EIA, EIS	Environmental impact assessment / statement
EIS	Environmental impact statement
EL	Exploration Licence
EMT	Emergency Medical Technician
EPCM	Engineering, Procurement and Construction Management
EPNG	El Paso Natural Gas
ESA	Endangered Species Act
F	Fluorine
Fe	Iron
FERC	Federal Energy Regulatory Commission
FOS	Factor of Safety
FW	Footwall
GPS	Global positioning system
H	High or Horizontal
H_2O	Water
HDPE	High-density polyethylene
HDS	High Definition Surveying
HW	Hanging Wall
ICP	Inductively Coupled Plasma
ID^2	Inverse distance squared
ISO	International Organization for Standardization
K value	Hydraulic conductivity
L	Long
LAG	Upper Silver zone
LCT	Locked cycle test
LHD	Load-haul-dump
LHOS, LHS	Long-hole open-stoping or long-hole stoping
LOM	Life-of-Mine
LVL	Level
M	Millions
Ma	Mega-annum (million years)
MIBC	Methyl isobutyl carbinol
Mn	Manganese
MOA	Memorandum of Agreement
MOX	Manto Oxide zone
MSO	Mine Stope Optimizer or Mexican spotted owl
NAAQS	National Ambient Air Quality Standards
NaCN	Sodium cyanide

Acronyms	Description
NAD 83	North American Datum of 1983
NAEP	National Association of Environmental Professionals
NAG	Net acid generation
NEC	National Electric Code
NEPA	National Environmental Policy Act
Newfields	Newfields Mining Design and Technical Services
NHPA	National Historic Preservation Act
NI 43-101	National Instrument 43-101
NPV	Net Present Value
NSR	Net Smelter Return
OK	Ordinary Kriging
Opex	Operating expenditure
Oxval	Oxide value
P&ID	Process and Instrumentation Diagram
P_{100}	100% Passing
P_{80}	80% Passing
PAC	Protected Activity Center
PAG	Potentially acid generating
Pb	Lead
PEA	Preliminary Economic Assessment
pH	pH is a measure of hydrogen ion concentration; a measure of the acidity or alkalinity of a solution
POC	Point-of-compliance
POO	Plan of operations
Property	Hermosa Property
PSD	Prevention of Significant Deterioration
PTE	Potential-to-emit
Q	Q-system (rock mass quality)
QA/QC	Quality assurance and quality control
QP	Qualified Person as defined by NI 43-101
RAR	Return air raise
RC	Reverse circulation drilling
RDi	Resource Development Inc.
RL	Reduced level or relative level
RMR	Rock mass rating (Bieniawski)
ROM	Run-of-Mine
RPMs	Reasonable and prudent measures
RQD	Rock quality designation
RWI	Bond rod mill work index
SAG	Semi-autogenous grinding
SGS	SGS North America Inc.
SHPO	State Historic Preservation Office

Acronyms	Description
SI units	SI (Système International d'Unités) is a globally agreed system of units
SIPX	Sodium Iso-Propyl Xanthate
SLOS	Sub-level open stoping
SPI	SAG Power Index
TCs	Terms and conditions
TMDL	Total Maximum Daily Load
TSF	Tailings storage facility
UCS	Unconfined Compressive Strength
US	United States
US$/t	US dollar per ton
USA	United States of America
USFWS	U.S. Fish and Wildlife Service
V	Vertical
W	Wide
Westland	WestLand Resources Inc.
Zn	Zinc
ZnEq	Zinc equivalent
Zn-Pb-Ag	Zinc-lead-silver
Zn-Pb-Ag-Cu	Zinc-lead-silver-copper
$ZnSO_4$	Zinc sulphate

Unit abbreviations	Description
%	Percentage
°	Degree (angle of dip)
°C	Degrees Celsius
μm	Micrometre
A	Amps
BTU	British Thermal Units
cfm	Cubic feet per minute
cm	Centimetre
d	Days
dmt	Dry metric tonne
dst	Dry short ton
ft	Feet
ft/month	Feet per month
g	Gallon
g/t	Grams per ton
g/tonne	Grams per tonne
gpm	Gallons per minute
ha	Hectare
hp	Horsepower
hr	Hours
in	Inch
kg	Kilogram
km	Kilometre
koz	Thousand ounces
kPa	Kilopascal
kt	Thousand (short) tons
ktonne	Kilotonne
kV	Kilo volts
kVA	Kilovolt-Ampere
kW	Kilowatts
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per ton
L	Litre
L/s	Litres / second
lb/ton	Pound per ton
lbs	Pounds
LPM	Litres per minute
M	Metre
M tonnes pa	Million tonnes per annum
m/d	Metre per day
m^3	Cubic metre (cu. m)
m^3/s	Cubic metre per second

Unit abbreviations	Description
mg	Milligram
mi	Mile
mil	A thousandth of an inch
mm	Millimetre
MPa	Megapascal
Mt	Million tons
Mtpa	Million tons per annum
MVA	Mega volt amperes
MW	Megawatt
oz	Ounces
oz	Troy ounce
oz/ton	Troy ounces per short (US) ton
Pa	Pascal
pa	Per annum
pcf	Per cubic foot
ppm	Parts per million (equivalent to g/t)
t	Short ton
ton	Short (US) ton = 2,000 lb
tonne	Tonne = 1,000 kg
tonnes pa	Tonnes per annum
tonnes pd	Tonnes per day
tpa	Tons per annum
tpd	Tons per day
V	Volt
W	Watt
wt	Wet ton
$yards^3pa$	Cubic yards per annum

Distribution list

1 e-copy to John Barber
1 e-copy to AMC Vancouver office

2 Introduction

AMC Mining Consultants (Canada) Ltd. (AMC) was commissioned by Arizona Mining Inc. (AZ) to prepare an updated Mineral Resource estimate and updated Preliminary Economic Assessment (PEA) and report according to a National Instrument 43-101 Technical Report (NI 43-101 Technical Report or Report) for the Taylor Zn-Pb-Ag deposit located on the Hermosa Property, (Property). The Property is located in Santa Cruz County near the town of Patagonia, southern Arizona, USA.

The Property hosts two known mineral deposits, the Taylor Deposit and the Central Deposit. The latest Technical Report for the Property was completed by AMC, dated 11 April 2017 and titled "Hermosa Property, Taylor Zn-Pb-Ag Deposit Preliminary Economic Assessment" for Arizona Mining Inc., with an effective date of 29 March 2017, (March 2017 Technical Report). Prior to that the Taylor Deposit was also the subject of an NI 43-101 report dated 29 November 2016 and reported additional Mineral Resources for the Taylor Deposit, (November 2016 Technical Report). The Property is 100% owned by Arizona Minerals Inc. (AMI) a wholly owned subsidiary of AZ. This Technical Report provides an update of the Mineral Resource estimate for both the Central and Taylor Deposits and reports the updated results of the PEA.

AMC are responsible for managing and preparing the Technical Report with inputs from Ms D. Nussipakynova, AMC, Mr G. Methven AMC, Mr C. Kottmeier, AMC, Ms L. Bloom, Analytical Solutions Ltd., Mr C. Kaye, Mine and Quarry Engineering Services Ltd., Mr Q. Jin, SGS North America Inc., Mr R. M. Smith, Newfields Mining Design and Technical Services, Mr E. Christenson, WestLand Resources Inc., and Mr D. Bartlett of Clear Creek Associates.

Persons who contributed to the report and assume responsibility for Sections are listed in Table 2.1.

Table 2.1 Persons who prepared or contributed to this technical report

Qualified Persons responsible for the preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of AZ?**	**Date of last site visit**	**Professional designation**	**Sections of report**
Mr G. Methven	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	13 July 2016	P.Eng. (BC)	1 (part), 2, 3, 15, 16, 21 (part), 24, 25 (part), 26 (part).
Ms D. Nussipakynova	Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	6 - 7 Sept. 2017	P.Geo. (BC)	1 (part), 4 - 10 (exc. 5.3.1), 11.3, 12, 14, 23, 25 (part), 26 (part), 27 (part).
Ms L. Bloom	President	Analytical Solutions Ltd.	Yes	No visit	P.Geo. (ON)	11 (exc. 11.3).
Mr Q. Jin	Senior Process Engineer	SGS North America Inc.	Yes	4 Oct. 2016	P.E.	1 (part), 17, 18 (part), 19, 21 (part), 25 (part), 26 (part), 27 (part).
Mr C. Kaye	Principal Process Engineer	Mine and Quarry Engineering Services Inc.	Yes	No visit	FAusIMM	1 (part), 13, 25 (part), 26 (part), 27 (part).
Mr R. M. Smith	Principal Engineer	Newfields Mining Design and Technical Services	Yes	19 Jan. 2017	P.E.	1 (part), 18 (part), 21 (part), 25 (part), 26 (part), 27 (part).
Mr C. Kottmeier	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng. (BC)	1 (part), 18 (part), 21 (part), 22, 25 (part), 26 (part).
Mr D. Bartlett	Principal and President	Clear Creek Associates	Yes	4 Oct. 2016	CPG AIPG, RG AZ	5.3.1, 20.3.3.
Mr E. Christenson	Senior Engineer	WestLand Resources Inc.	Yes	2 March 2017	P.E. AZ	1 (part), 20 (exc. 20.3.3), 25 (part), 26 (part), 27 (part).

Other Experts who assisted the Qualified Persons					
Expert	**Position**	**Employer**	**Independent of AZ?**	**Visited site**	**Sections of report**
Mr J. Barber	VP Mining	Arizona Minerals Inc.	No	Yes	All
Mr D. Taylor	Chief Operating Officer	Arizona Minerals Inc.	No	Yes	1 - 12
Mr S. Burkett	Senior Geologist	Arizona Minerals Inc.	No	Yes	1 - 12
Mr J. Pappas	Director Environmental and Permitting	Arizona Minerals Inc.	No	Yes	20

The key information used in this report is listed in Section 27, References.

All currency amounts and commodity prices are in United States (US) dollars unless stated otherwise. Quantities are stated in both Imperial and SI units. Commodity weights of measure are in ounces per short ton (oz/ton) or percent (%) unless stated otherwise.

This Report includes the tabulation of numerical data which involves a degree of rounding for the purpose of Mineral Resource estimation. AMC does not consider any rounding of the numerical data to be material to the Property.

This Report has been produced in accordance with the Standards of Disclosure for Mineral Projects as contained in NI 43-101 and accompanying policies and documents. NI 43-101 utilizes the definitions and categories of Mineral Resources and Mineral Reserves as set out in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves adopted 10 May 2014, (CIM Definition Standards).

A draft of the NI 43-101 Technical Report was provided to AZ to check for factual accuracy. The report has an effective date of 1 January 2018.

3 Reliance on other experts

The Qualified Persons have relied, in respect of legal and taxation aspects, upon the work of the Experts listed below. To the extent permitted under NI 43-101, the Qualified Persons disclaim responsibility for the relevant section of the Technical Report.

The following disclosure is made in respect of this Expert:

- Marian C. LaLonde, attorney-at-law of the firm Quarles & Brady LLP.

Report, opinion, or statement relied upon:

- Status of Mineral Properties report dated 8 November 2017. The report provides a legal description of all patented and unpatented mining claims associated with Arizona Minerals, Inc.'s Hermosa Property located in Santa Cruz County, Arizona.
- Marian C. LaLonde stated that her office confirmed on 8 November 2017 during a phone call with the Treasurer of Santa Cruz County, that all taxes due on the patented mining claims have been timely paid or are being paid as of the week ending 10 November 2017.
- In addition, on 8 November 2017, her office researched the validity of the unpatented claims on the Bureau of Land Management's Legacy Rehost System (LR2000) and confirmed that all unpatented claims for the Hermosa Project are of record, have the current maintenance fees timely paid, and are in good standing.

Extent of reliance:

- Full reliance following a review by the Qualified Person(s).

Portion of Technical Report to which disclaimer applies:

- Section 4.

The following disclosure is made in respect of this Expert:

- Tom Whelan, Chief Financial Officer, Arizona Mining Inc. (with input from external chartered professional accountants).

Report, opinion or statement relied upon:

- Information on the application of corporate taxation for Arizona, USA and applicable royalties.

Extent of reliance:

- Full reliance following a review by the Qualified Person(s).

Portion of Technical Report to which disclaimer applies:

- Section 22.

4 Property description and location

4.1 Location

The Taylor and Central Deposits are located on the Hermosa Property, which is part of the Harshaw and Patagonia Mining Districts located in the Patagonia Mountains of Santa Cruz County, Arizona (Figure 4.1 and Figure 4.2). The Property is located 6 miles (9.6 km) southeast of the town of Patagonia, which has a population of approximately 1,000 people.

Figure 4.1 Arizona map showing Property

The Property is located 15 miles (24.1 km) northeast of the Santa Cruz county seat at Nogales and 50 miles (80.5 km) southeast of Tucson, in adjacent Pima County. The international border with Mexico is approximately 8 miles, (12.9 km) to the south.

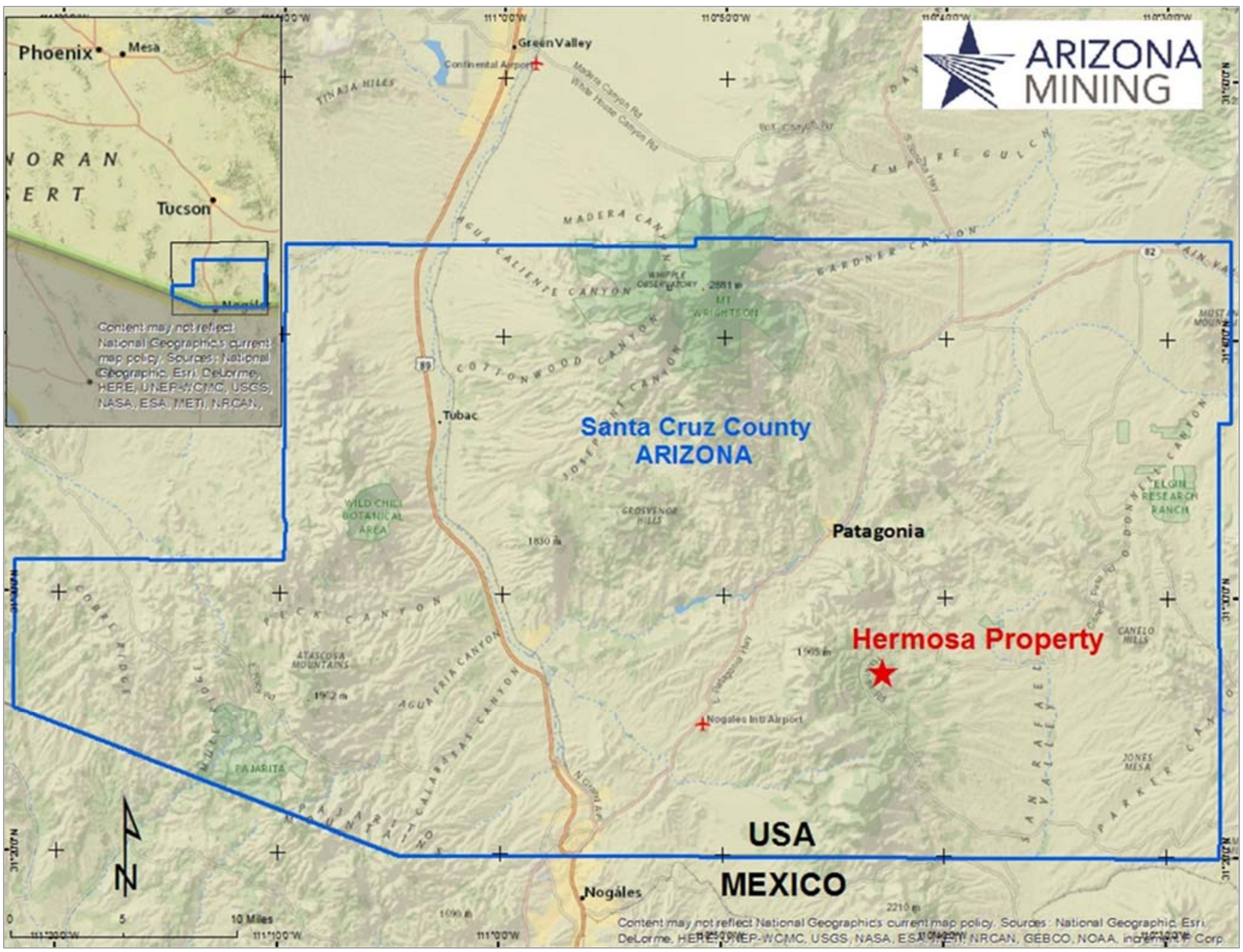
The Property encompasses an area of approximately, 21 square miles, (54.4 square kms) and lies within the surveyed and protracted unsurveyed lines of the Gila & Salt River (G&SR) Meridian, Santa Cruz County, Arizona shown in Table 4.1.

Table 4.1 Surveyed and protracted unsurveyed lines of the G&SR Meridian

Township	Range	Section(s)
22 South	15 East	13, 24-26, 35-36
22 South	16 East	17-20, 27-36
23 South	15 East	13, 24, 25, 36
23 South	16 East	1-5, 8-23, 26-31
23 South	17 East	6, 7, 18
24 South	15 East	1
24 South	16 East	6

General property coordinates are 31° 28’ North latitude and 110° 43’ West longitude (NAD 83, Geographic, North America).

Figure 4.2 Property location map



4.2 Property description

The Property is located on the northern end of the Patagonia Mountains. Elevations on the property range from 4,050 ft to 6,500 ft, (1,234 m to 1,981 m) above sea level. The area is sparsely populated, and livestock grazing is the dominant land use. The Property is located within several US Forest Service (USFS) Grazing Allotments under the management of the United States Department of Agriculture.

The Property is composed of approximately 532.98 acres (215.69 hectares) of fee simple surface and mineral rights ownership on patented mining claims. These patented mining claims are surrounded by unpatented lode mining claims held by AMI. These unpatented mining claims are federal lands where the subsurface is administered by the Bureau of Land Management (BLM) and the surface is administered by the USFS, Coronado National Forest (CNF). The Sierra Vista Ranger District of the Coronado National Forest is the responsible agent.

The Property contains shafts, trenches and other surface openings from historic mining and exploration activities. The area is accessed through a series of interconnected low maintenance County roads and trails.

4.3 Property ownership

AZ holds majority interest in the Property through its wholly owned subsidiary AMI, a Nevada corporation, which was registered on 4 October 2005 with the Arizona Corporation Commission to do business within the State of Arizona. AZ is incorporated in British Columbia, Canada and listed on the Toronto Stock Exchange with its common shares trading under the symbol "AZ". On 28 October 2005, AMI entered into an agreement with ASARCO, LLC to purchase the Property. At that time, the property consisted of eight patented mining claims in three separate tax parcels acquired by a combination of patents in 1961 and purchases in 1968 and 1978; in addition, 26 unpatented “Shell No.” lode mining claims located in 1965 and 1968 by American Smelting and Refining Company. American Smelting and Refining Company later changed its name to ASARCO Incorporated and was subsequently merged into ASARCO, LLC. On 17 February 2006, the US Bankruptcy Court, Southern District of Texas, Corpus Christi Division in Case 05-21207 approved the sale of the Hardshell Group of Mining Claims by ASARCO, LLC to AMI. This acquisition closed on 14 March 2006, with the final payment made to ASARCO, LLC on 14 March 2007. AMI has no royalty or other obligations due to ASARCO, LLC or any predecessor claim owners.

In January of 2016, AZ closed the acquisition of 16 patented claims “Trench” (approximately 300 acres or 121 hectares) from the ASARCO Multi-State Environmental Custodial Trust. Consideration for the acquisition comprised $10 and the assumption of the environmental liabilities relating to the site that resulted from historic mining activity. AMI has an approved remediation plan to address the environmental liabilities that includes a plan for a passive water treatment system. AZ subsequently transferred the claims to AMI. These claims are directly adjacent (northwest) to the original eight patented claims.

As part of the purchase agreement with ASARCO, LLC, AMI also acquired all available original or copies of data, documents and reports pertaining to the property including information on land, geology, previous drilling, assays, engineering, groundwater and metallurgical studies. ASARCO, LLC also transferred the remaining drill core, samples and assay pulps to AMI.

Additional patented claims were acquired by AMI in August 2017 to add to the property footprint. The Allis, Garfield, California, and Bonnie Carrie patented claims were purchased on 18 August 2017 and are also situated in the Harshaw Mining District. Those claims are in Table 4.2 and shown in Figure 4.4.

The combined AMI property holdings now consist of 28 patented mining claims totaling approximately 532.98 acres (215.69 hectares) with the surface and mineral rights owned fee simple. The patented land is surrounded by 1,104 unpatented lode mining claims approximately 19,012 acres (7,694 hectares). Under the terms of United States mining law, the unpatented mining claims can be held as long as the federal annual maintenance fee is paid (no expiration date) to the United States Department of the Interior, Bureau of Land Management. Data on the individual patented claims is shown in Table 4.2.

4.4 Mineral tenure

The core of the Property is comprised of 28 patented mining claims totalling about 532.98 acres (215.69 hectares) with the surface and mineral rights owned fee simple. The patented land is surrounded by 1,104 unpatented lode mining claims totalling approximately 19,012 acres (7,694 hectares). Title to the mineral rights is vested in AZ and its wholly-owned subsidiary AMI. A map of the claims is shown as Figure 4.3.

The wholly-owned, patented land parcels with full surface and mineral rights are subject to annual real property tax payments to Santa Cruz County, Arizona. The mineral rights for the unpatented mining claims are held by the payment of federal annual maintenance fees to the BLM and record of such must also be filed with the Santa Cruz

County Recorder. The unpatented mining claims can be held as long as the federal annual maintenance fee is paid to the BLM. The surface rights of the unpatented mining claims are administered by the USFS under multiple-use regulatory provisions.

Marian C. LaLonde, attorney-at-law of the firm Quarles & Brady LLP issued a 34-page Status of Mineral Properties report dated 8 November 2017. The report provides a legal description of all patented and unpatented mining claims associated with Arizona Minerals, Inc.'s Hermosa Property located in Santa Cruz County, Arizona. Marian C. LaLonde stated that her office confirmed on 8 November 2017 during a phone call with the Treasurer of Santa Cruz County, that all taxes due on the patented mining claims have been timely paid or are being paid as of the week ending 10 November 2017. In addition, on 8 November 2017, her office researched the validity of the unpatented claims on the Bureau of Land Management's Legacy Rehost System (LR2000) and confirmed that all unpatented claims for the Hermosa Project are of record, have the current maintenance fees timely paid, and are in good standing.

All Mineral Resources disclosed in this report are fully contained within the claims as listed in Table 4.2 and Table 4.3.

Table 4.2 Unpatented Mining Claims held by AMI

Campaign	Number of claims staked	Area (acres)	Area (hectares)	Year	Description
1	26	486	197	1965	Staked by ASARCO in the immediate vicinity of Hardshell deeded land package in 1965.
2	276	5,021	2,032	2005-2006	Staked by White Cloud Resources (WCR) in 2005 and 2006, 35 of these were re-located / papered by AMI in 2014.
3	52	1,012	410	2006-2007	Staked to expand ASARCO claim package to Hermosa Canyon, the Bender Mine, and the American Mine in 2006 and 2007.
4	72	1,372	555	2007-2008	Staked to expand south to Mowry and east to Goldbaum Canyon in 2007 and 2008.
5	16	318	129	2008	Staked to cover area between Harshaw townsite and Northern Goldbaum Canyon in 2008.
6	85	1,654	669	2011	Staked to cover Corral Canyon, Willow Springs Canyon, and the remainder of Goldbaum Canyon in 2011.
7	149	3,051	1,235	2012-2013	Staked to expand north to the Lead Queen and east to the edge of the San Rafael Valley 2012 and 2013.
8	48	595	231	2013	Staked to cover Mowry area as well as mineral fractions in various locations in 2013 including 24 acres of fractions.
9	49	802	325	2015	Staked northward to the foot of Red Mountain and west to the World's Fair in 2015.
10	144	2,143	867	2016	Staked at additional locations near Red Mountain, west of the World's Fair and to cover mineral fractions in 2016.
11	171	2,370	959	2016	Staked southward to Finley and Adams Canyon and Sycamore Canyon in 2016.
12	16	188	76	2017	Acquired Bronco Creek Claims. Located directly south of and adjoining the Trench Patented Claims
Total	**1,104**	**19,012**	**7,694**		

Table 4.3 Patented claims owned by AMI

Un-surveyed Sections 3, 4, and 5, Township 23 South, Range 16 East and Surveyed Sections 19, 30, and 32, Township 22 South, Range 16 East, and Sections 24 and 25 of Township 22 South, Range 15 East, G&SRM, Santa Cruz County, Arizona

Patented claim name	BLM recorded patent No.	Patent grant date	Mineral survey		Mineral survey approved / record of survey record	Claim acreage**	Quadrant of section	Santa Cruz County records document	County assessor parcel No.
			No.	Lot					
Camden Mine	1211192	8/5/1960	4460	*	05/13/1959/02/14/2008	20.64	Sec. 4: All	Doc. 25, Page 30 / Seq. 2008-01675	105-49-001A
Camden No. 2	1211192	8/5/1960	4460	*	05/13/1959/02/14/2008	20.63	Sec. 4 NE/4, NW/4	Doc. 25, Page 30 / Seq. 2008-01675	105-49-001A
Hardshell No. 1	1211192	8/5/1960	4460	*	05/13/1959/02/14/2008	20.64	Sec. 4 NE/4, NW/4	Doc. 25, Page 30 / Seq. 2008-01675	105-49-001A
Hardshell No. 15	1211192	8/5/1960	4460	*	05/13/1959/02/14/2008	17.08	Sec. 4 NE/4, NW/4	Doc. 25, Page 30 / Seq. 2008-01675	105-49-001A
Bluff	10279	12/04/1885	500	50	06/05/1883/02/14/2008	19.4	Sec. 3 SW/4	Book 88, Page 476 / Seq. 2008-01672	105-52-001
Hermosa	10278	12/04/1885	499	49	06/05/1883/02/14/2008	20.23	Sec. 3: SW/4, Sec.4: SE/4	Book 88, Page 469 / Seq. 2008-01674	105-52-001
Salvador	10614	06/11/1886	498	48	06/05/1883/02/14/2008	13.75	Sec. 4: SE/4, SW/4	Book 88, Page 482 / Seq. 2008-01676	105-52-001
Alta	8653	01/10/1884	84	38A	02/16/1877/02/14/2008	19.87	Sec. 4: NW/4	Book 17, Page 213 & Doc. 182, Page 616 / Seq. 2008-01673	105-49-002
January	25015	12/04/1894	745	51	12/24/1885/12/1/2015	20.6	Sec. 5: NE/4, Sec. 32: SE/4 T22S, R16E,	Seq. 2010-03552(QCD), Seq. 2016-00445(QCD)	105-50-001B
Norton	19644	2/06/1892	929	52	7/25/1890/12/1/2015	19.63	Sec. 5: NE/4	Seq. 2010-03552(QCD), Seq. 2016-00445(QCD)	105-50-001B
Trench	2837	5/11/1878	28	37A	1/07/1874/3/23/2011	10.73	Sec. 5: NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Trench No. 2	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	20.66	Sec. 5: NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Trench No. 3	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	20.66	Sec. 5: NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Trench No. 4	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	20.45	Sec. 5: NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A

Un-surveyed Sections 3, 4, and 5, Township 23 South, Range 16 East and Surveyed Sections 19, 30, and 32, Township 22 South, Range 16 East, and Sections 24 and 25 of Township 22 South, Range 15 East, G&SRM, Santa Cruz County, Arizona

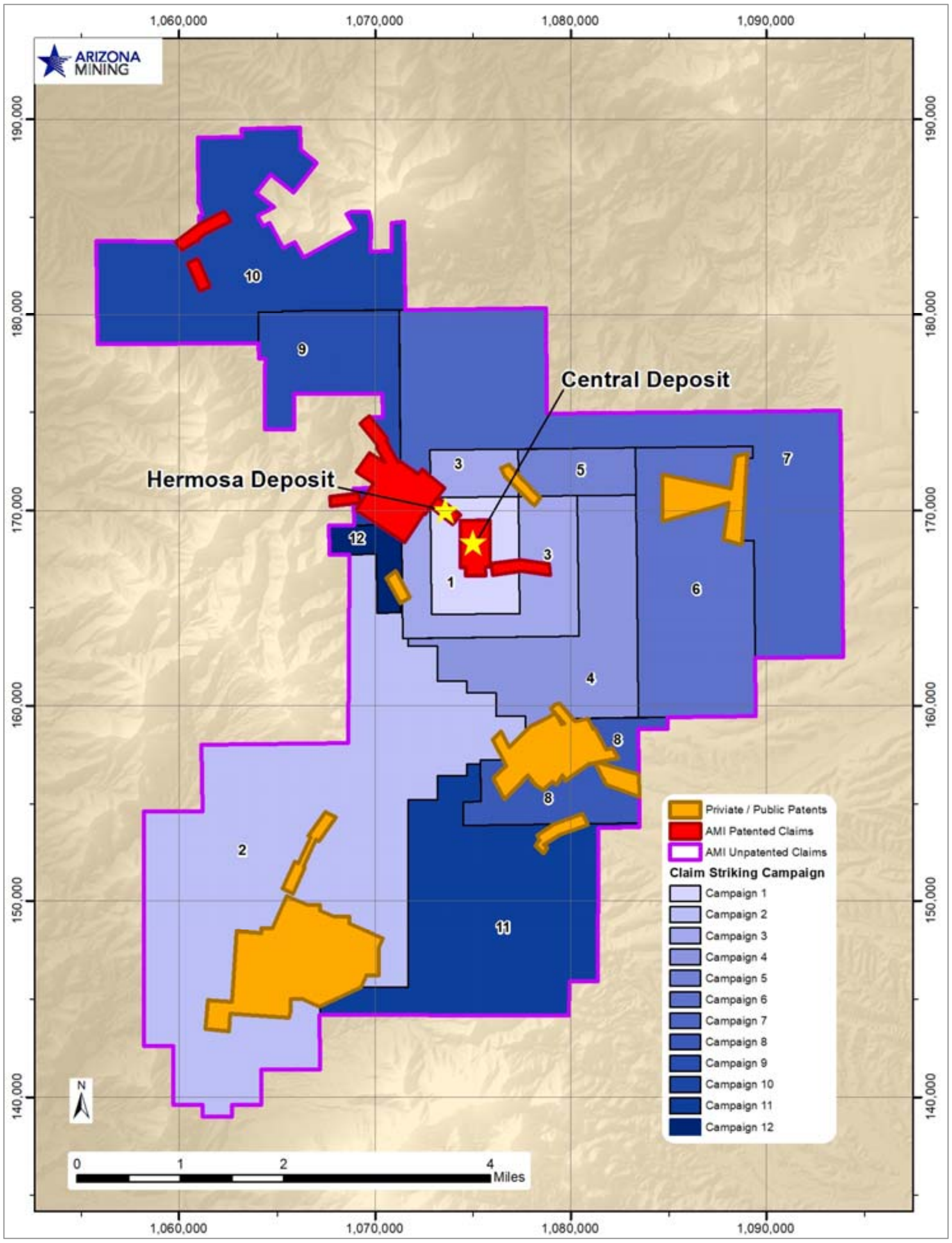
Patented claim name	BLM recorded patent No.	Patent grant date	Mineral survey		Mineral survey approved / record of survey record	Claim acreage**	Quadrant of section	Santa Cruz County records document	County assessor parcel No.
			No.	Lot					
Trench No. 5	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	20.45	Sec. 5: SE/4, NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Trench No. 6	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	20.66	Sec. 5: SE/4, NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Trench No. 7	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	20.4	Sec. 5: NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Trench No. 8	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	18.7	Sec. 5: NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Trench Ext. No. 1	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	14.2	Sec.4: NW/4	Doc. 119/393, Seq. 2009- 11239(QCD), Seq. 2010-03552(QCD), Seq. 2016-00443(QCD), Seq. 2011-02069(Survey)	105-49-003
Trench Ext. No. 2	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	20.32	Sec. 5: NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Trench Ext. No. 3	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	19.4	Sec. 5: NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Trench Ext. No. 4	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	18.71	Sec.4: NW/4	Doc. 119/393, Seq. 2009- 11239(QCD), Seq. 2010-03552(QCD), Seq. 2016-00443(QCD), Seq. 2011-02069(Survey)	105-49-003
Hardshell No. 7	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	15.97	Sec.4: NW/4	Doc. 119/393, Seq. 2009- 11239(QCD), Seq. 2010-03552(QCD), Seq. 2016-00443(QCD), Seq. 2011-02069(Survey)	105-49-003
Josephine	1107723	4/10/1940	4222	*	5/5/1939/3/23/2011	20.65	Sec. 5: NE/4	Seq. 2009-11239(QCD), Re-recorded 2010-03552(QCD), Seq. 2016-00443(QCD (6.00)), Seq. 2016-00444(QCD), Seq. 2011-02069(Survey)	105-50-001A
Allis	9306	5/24/1884	353	*	04/14/1882/06/02/2017	20.66	Sec. 24: SE/4 Sec 25: NE/4	Document no. 2017-00058666	106-20-003

Un-surveyed Sections 3, 4, and 5, Township 23 South, Range 16 East and Surveyed Sections 19, 30, and 32, Township 22 South, Range 16 East, and Sections 24 and 25 of Township 22 South, Range 15 East, G&SRM, Santa Cruz County, Arizona

Patented claim name	BLM recorded patent No.	Patent grant date	Mineral survey		Mineral survey approved / record of survey record	Claim acreage**	Quadrant of section	Santa Cruz County records document	County assessor parcel No.
			No.	Lot					
Garfield	8892	3/6/1884	352	*	04/14/1882/06/02/2017	20.48	Sec 19: SW/4	Document no. 2017-00058666	106-20-003
California	9164	4/26/1884	351	*	04/14/1882/06/02/2017	19.80	Sec 25: NE/4 Sec 30: NW/4	Document no. 2017-00058666	106-52-001
Bonnie Carrie	437958	10/24/1914	2952	*	10/24/1914	17.61	Sec 5: NE/4 Sec 5: NW/4	N/A	106-53-001
Total						**532.98**			

Note: Filed with the Official Records of Santa Cruz County, Nogales, Arizona and U.S. Bureau of Land Management, Phoenix, Arizona. The Bluff, Hermosa, Salvador and Alta claims, when surveyed and patented, were part of Pima County, Arizona Territory. Early records with Pima County, Tucson.
These sections in T23S, R16E are non-standard, un-surveyed and protracted.
*No lot number assigned. (QCD) Quit claim deed.
**Record of Survey Total Acreage 532.98.

Figure 4.3 Property claim status map



4.5 Options agreements

On 7 October 2015 AMI entered into an agreement with Bronco Creek Exploration, Inc. granting it permission to explore on 16 unpatented mining claims with an option to acquire the claims on fulfilment of the terms of the agreement. On 18 September 2017 AMI fulfilled the terms of the agreement and exercised the option to acquire the claims, making AMI the owner of these 16 unpatented mining claims located in Sections 5, 8, and 9 of Township 23 South, Range 16 East of the G&SR Meridian, Santa Cruz County, Arizona. This transaction resulted in conveying a Net Smelter Return (NSR) Royalty interest of 2% of production returns from those claims to the seller. Figure 4.3 displays the optioned claims as Campaign 12 and are indicated in dark blue.

4.6 Agreements, royalties, and agreements

There is a 2% NSR Royalty payable by AMI to a private Canadian Company controlled by AZ's Executive Chairman, from any future production extracted from the original eight patented mining claims and 26 unpatented mining claims acquired in 2005. There are no underlying royalties, fees or other obligations due to ASARCO, LLC, or previous claim holders.

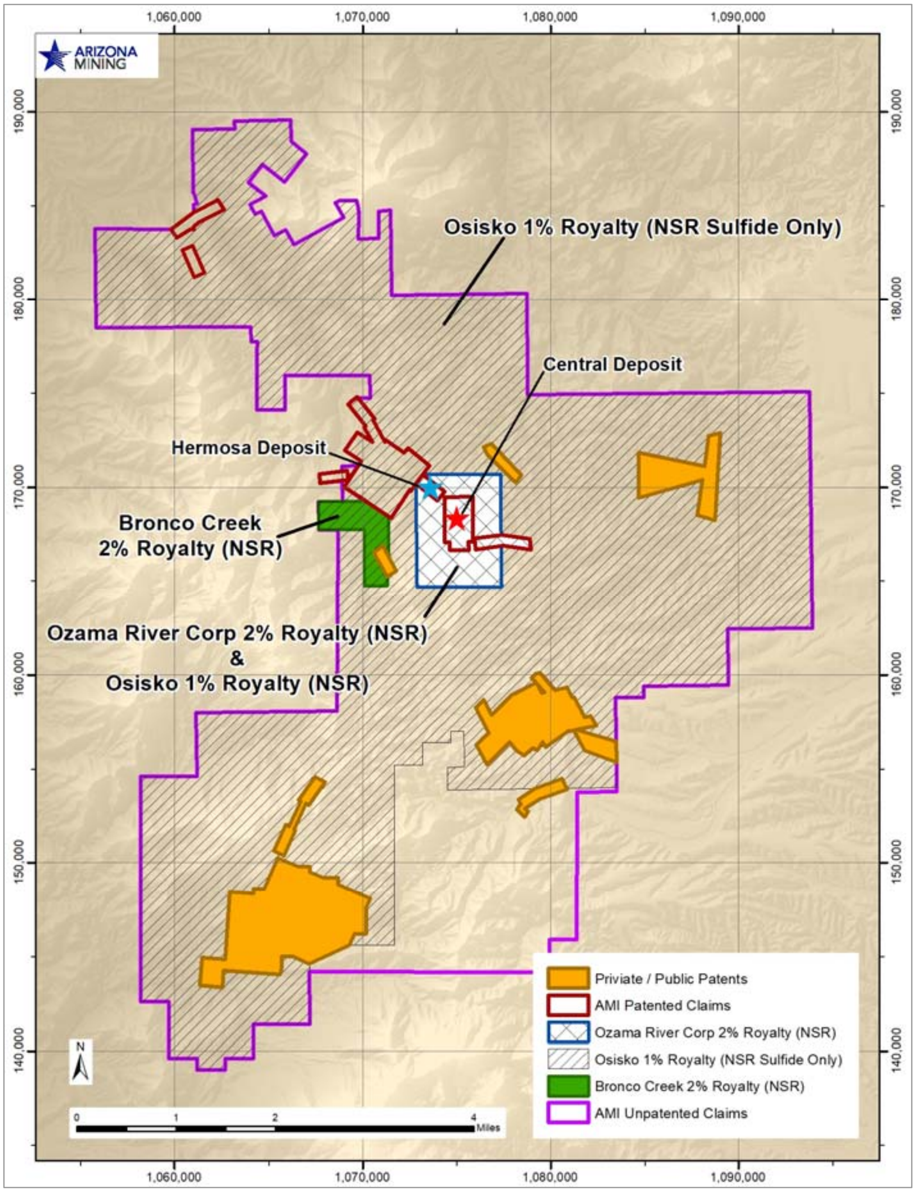
On 25 April 2016 the Company mentioned above closed the sale of a 1% NSR Royalty to Osisko Gold Royalties Ltd. ("Osisko") on all sulphide mineralized material of lead and zinc (and any copper, silver or gold recovered from the concentrate from such mineralized material) in, under or upon the surface or subsurface of the Hermosa Property which are mined, excavated, extracted or recovered from the Hermosa Property.

As discussed under 4.5, above, AMI exercised its option to acquire the Bronco Creek claims, therefore a 2% NSR Royalty is payable from any production from those claims totalling approximately 188 acres, located approximately three quarters of a mile south west of the Taylor Deposit.

See Figure 4.4 for the location of these royalties and other landholdings.

Santa Cruz County has a 60 ft (20 m) wide road easement centred on the mid-line of Harshaw Road. About 400 ft (122 m) of Harshaw Road crosses the northwest end of the Alta patented claim, where an access road to the property is located. The local power company, UniSource Energy Services, also has a high voltage power line with easements along the Harshaw Road, through the Alta patented claim. A branch of this power line also extends through the Harshaw townsite owned by the Hale Ranch and continues into the San Rafael Valley.

Figure 4.4 Agreements and royalties map



4.7 Environmental liabilities

In January 2016, AMI acquired the patented "Trench" claims from the ASARCO Multi-State Environmental Custodial Trust. Consideration for the acquisition included the assumption of the environmental liabilities relating to the trust site that resulted from historic mining activity. AMI has submitted a workplan to ADEQ and is working with ADEQ's Voluntary Remediation Program and Water Quality Division on the active treatment system, which will effectively manage and treat discharge from ASARCO's January Mine Adit and seepage from the Tailings Storage Facilities (TSF).

4.8 Permits and others

The issue of permits is addressed in Section 20.

5 Accessibility, climate, local resources, etc.

5.1 Accessibility

The Property is accessed via Harshaw Road, a Santa Cruz county road, leading 6 miles (10 km) southeastward from Patagonia, Arizona to the Harshaw townsite. An interconnecting system of USFS numbered roads, originally constructed largely for exploration, mining and ranching, exist around Harshaw and the district. The Property extends southward for approximately 3 miles (5 km) from Harshaw townsite and approximately 1 mile (1.6 km) southeast and southwest from Harshaw townsite. Access around the Property is by unimproved two-track roads. See Figure 4.2 for location and access information.

5.2 Climate

The Harshaw-Patagonia area is mountainous and has a semi-arid climate characteristic of the Arizona Uplands. Daytime temperatures seldom remain above 90°F (32°C) in the summer with warm to moderately cool nights. Winter days are usually mild with periodic frosts at night. Light snowfall is not uncommon but seldom remains for more than a few days. Cooler temperatures and higher winds occur at higher elevations in the area.

Precipitation, characteristic of this upland desert region, is variable and cyclic. Annual precipitation averages 17 in (43 cm) and ranges from 8 in to 36 in (20 cm to 91 cm) per year with higher amounts of precipitation occurring at higher elevations in the range. More than 50% of the rainfall occurs during the period from late June to early October in cyclonic, often torrential “monsoonal” thunderstorms, which are often accompanied by strong, destructive winds.

5.3 Infrastructure

5.3.1 Water

The local base level of the water table is approximately 4,950 ft (1,509 m) elevation at Harshaw town site. The project area and the local Harshaw Creek drainage are not part of an Arizona Department of Water Resources Active Management Area. Pump testing of new and existing wells conducted in 2017 has indicated the presence of a productive aquifer within the bedrock at the site that is sufficiently large to meet the water supply needs of the mining operation.

5.3.2 Workforce

Southern Arizona hosts several major mining districts and the local area has several large active mines. Experienced, skilled workers are readily available within a reasonable commuting distance.

5.3.3 Commercial resources and services

Resources in the town of Patagonia are limited. The town has a high school, a motel, several restaurants, a small grocery store and a gas station. Nogales, 15 miles (24 km) southwest of Patagonia, has a population of approximately 25,000 people and is large enough to serve as a supply and service centre for most needs. Nogales has rail freight service, and a small commercial airport.

Tucson, just over 50 miles (80 km) to the north, is the commercial and service / supply centre for one of the world’s largest mining districts. Tucson has a full-service commercial airport and is a large rail centre.

5.3.4 Social services and security

Patagonia has K-12 schools and a well-stocked town library. In addition, the community has a small family medical facility. Emergency Medical Technician (EMT) services are associated with the Volunteer Fire Department. Medical helicopter landing facilities are available. Patagonia has a small police force which is supplemented by the Santa Cruz County Sherriff and the Arizona Highway Patrol. The U.S. Border Patrol has a strong presence in the area. Nogales has a small regional hospital. Tucson’s large hospitals are easily accessible by ambulance or helicopter.

5.3.5 Power

A 13.2 kV power line follows Harshaw Creek from west of Patagonia to the old town site of Harshaw and continues on to the San Rafael Valley. Higher capacity power lines traverse the Sonoita Creek Valley from Huachuca City to Sonoita-Elgin and Patagonia from the east. A major regional natural gas pipeline, owned and operated by El Paso Natural Gas extends from Nogales to the northeast through the Sonoita Valley and to localities to the east. A trunk phone line follows the Harshaw Creek Road with phone service available in Harshaw. Cellular telephone service is good in the Patagonia-Harshaw area.

5.3.6 Transportation

The Property is accessed via state and county hard surfaced roads and USFS secondary and tertiary roads, constructed largely for exploration, mining and ranching needs around Harshaw townsite and the district. A major rail hub is located approximately 15 miles (24 km) south near the city of Nogales.

5.4 Physiography

The Property lies on the eastern pediment flank of the Patagonia Mountains, a portion of the northwestern edge of the Mexican Highlands section of the Basin and Range Physiographic Province of the southwestern United States. Elevations in the mountains range up to 7,200 ft (2,195 m) above sea level, while elevations on the Property range from 4,800 ft to 6,200 ft (1,460 m to 1,890 m) above sea level. The Property is dominated by the western San Rafael Valley pediment plateau at about 5,400 ft (1,646 m), which on-laps the higher foothills of the Patagonia range to the west. The plateau is deeply incised by tributaries of Harshaw Creek which drain to the north.

The Property is located in an area of moderate to rugged topography, with numerous arroyos and canyons incised through volcanic and sedimentary stratigraphy. The arroyos and canyons contain streams which flow intermittently in response to rainfall events. Vegetation is typical of the Pinyon-Oak-Juniper woodland and is characterized by short evergreen trees and scrub oaks mixed with a variety of desert and upland shrubs. Lower slope faces are covered by open grasslands.

6 History

6.1 Prior ownership of the Property

Ownership of the Property prior to its acquisition by Asarco is not known. Asarco began operating the nearby Trench Mine in 1939 and continued ownership of the Property until it was acquired by AMI which was subsequently acquired by AZ.

6.2 Previous exploration and development work

ASARCO explored the Property with intermittent drill programs from 1940 through 1991. The early program diamond drilling, spurred by WWII metal prices, failed to find significant extensions of Hardshell Incline lead-silver minerals. Nonetheless, several thousand tons of moderate grade lead-silver oxide mineralization was shipped from the lower levels of the Hardshell Incline Mine from 1943 to 1948 and from 1963 and 1964. Second pass diamond drilling programs, undertaken from 1946 to 1953, located thick Ag-Pb-Zn bearing, manganese oxides of the Main Manto to the southeast of the Hardshell Incline.

Rising silver prices in the mid-1960s led to renewed interest in the Hermosa mineralization. Re-evaluation of the geological data led to staking of additional claims in the district and the three patented claims of the Hermosa Group were acquired between 1965 and 1968. ASARCO used the newly developed, air-hammer rotary drilling equipment to drill the silica jasperoid cap and the vuggy Main Manto zone. Diamond drilling was used successfully in some outlying stratigraphic holes but attempts to deepen air-hammer drillholes in vuggy, silicified limestone often failed when drill fluid circulation was lost.

Recovery by weight or footage, water levels and volumes, lithology, alteration, mineralization, and miscellaneous comments were logged in the field for most ASARCO drillholes and posted to graphic logs and cross-sections. Most air-hammer holes were drilled dry or with minimal water injection for dust control. They were usually lost after the water table or significant fracture zones or voids were encountered. Most of this drilling did not penetrate the static water table. Down-hole deviation was not measured for any of the ASARCO drillholes at the Property.

Geophysical surveying, detailed geological, and metallurgical studies on the manganese oxide mineralization began in the late 1960s and continued through 1991. Close-spaced, rotary hammer drilling partially defined heap leach amenable, low-grade manganese, low-grade silver resource located near the historic Hermosa mine workings. Three shallow rotary air-hammer drillholes were completed in 1989 for metallurgical samples and a 1,500 ft (457 m) deep diamond drillhole in 1990-91 explored for deeper mineralization. ASARCO drilled 114 air-hammer and core holes, with an aggregate of approximately 46,000 ft (14,021 m) on the Property and surrounding area.

ASARCO conducted beneficiation tests to determine silver recovery processes. Bench scale, high-tension magnetic separation, electrostatic separation, reduction and segregation kilning, SO_2 and thio-sulphate leaching and various cyanidation processes, in both company and commercial laboratories, were tested. Little consideration was given to recovering other metals, including Mn, Zn, Cu, Au, and potential co-products silica or clays. Minor test consideration was given to heap-leaching non-manganese low-grade silver mineralization.

Pan American Silver had a minimal lease / option / first right of refusal on most of ASARCO's Hardshell Property from 1994 to 2002, Pan American Silver did not undertake any significant exploration work, confining their activity to internal economic evaluations.

6.3 Historical Mineral Resource and Mineral Reserve estimates

ASARCO made a number of historical resource and reserve estimates for the Property. A 1968, open pit resource of 6.5 million tons at 5 oz/ton silver; 1% to 2% lead and zinc and 15% MnO_2 was calculated and used in a number of older publications. An updated, open pit resource was calculated by ASARCO in 1975 to contain 20 million tons at an average grade of 3.33 oz/ton silver with 8% manganese, with a stripping ratio of 2:1. A 1979 ASARCO estimate reported a range of resources, and the median was 6,586,500 tons at an average grade of 7.92 oz/ton silver, at a cut-off grade of 5 oz/ton silver. A mineral inventory estimate calculated by ASARCO in 1984 estimated a resource of 9,596,000 short tons with an average grade of 6.9 oz/ton silver, at a cut-off grade of 1.5 oz/ton silver.

These estimates pre-date the inception of National Instrument 43-101 (NI 43-101) and are included here only for the purpose of completeness of the historical record. These estimates do not conform to the categories set out in Sections 1.2 and 1.3 of the Instrument and a qualified person has not done sufficient work to classify these historical estimates as current Mineral Resources and Reserves and AMI is not treating these historical estimates as current Mineral Resources or Mineral Reserves.

6.4 Prior production from the Property

Mining in the Harshaw District dates from mid-18th century Spanish Colonial times, but is poorly documented before the 1870s. Initially, an oxide lead-silver vein was mined from the Trench Property, located approximately 1 mile (1.6 km) northwest of Hermosa and from the Mowry Property located approximately two miles to the south. This work continued intermittently until the late 19th century. Historical information from the late 1800s and early 1900s has been well documented (Schrader, 1915: USGS Bulletin 582 and Keith, 1975: AZ Geol. Survey Bulletin 191). The district's historic production is poorly reported but is believed to be around 250,000 tons, yielding approximately two million ounces of silver with by-product lead, zinc, copper, and manganese.

Early, unnamed, small-scale miners in the Hermosa area developed small tonnages of milling and direct-shipping oxidized mineralization in a number of small individual mines.

Production from the district was dominated by the Trench-area mines, small mines on the Alta claim, the Hardshell Incline and the Hermosa mine. The Trench area mines and sulphide flotation custom mill, located a mile northwest of the Property, produced primarily silver with minor by-product lead, but important production of direct-shipping manganese was recorded during World Wars I and II and the Korean War. The bulk of the production was from small underground operations in the area. Approximately half of the production was direct-shipping oxide mineralization and the balance was milling mineralization. The Trench mill produced both lead and zinc concentrates, with copper, silver, and minor gold by-product production.

The Alta Claim, staked in 1877, produced several thousand tons of oxidized high-grade lead-silver material from a northeastward-dipping vein. The Hardshell Incline Mine, discovered in 1879, produced approximately 35,000 tons with an average grade of about 8 oz/ton silver and 6% to 8% lead between 1896 and 1964.

The Hermosa Mine located one-half mile to the southeast of the Hardshell Incline Mine and discovered about the same time, produced high-grade silver halide mineralization from a 30° north-dipping stratiform vein, averaging approximately 20 oz/ton silver. Approximately 70,000 tons (63,490 tonnes) of material was processed in a 100 t/day mill over an 18 to 24 month period, producing 1.4 million ounces of silver, as confirmed by Wells Fargo shipping records. Scavenging secondary production from 1902 to 1943 yielded an additional 600,000 ounces of silver with by-product lead and copper.

ASARCO operated the nearby Trench Mine, located approximately 1 mile northwest of Hermosa, between 1939 and 1949 and produced lead, zinc, silver, and copper from a fissure vein sulphide deposit. The 150 t/day Trench lead-zinc flotation mill also treated district mineralization between 1939 and 1964 on a custom basis.

A summary of the historic production of the Hermosa area mines is presented in Table 6.1, derived from the Arizona Bureau of Mine Data (Bulletin 191, 1975) and ASARCO company files (Fleetwood Koutz, personal communication, 2006).

Table 6.1 Historic production from Hardshell area mine

Mine name	Production period	*Tons produced	Average grades						Comments
			Ag (oz/ton)	Zn (%)	Pb (%)	Cu (%)	Ag (oz/ton)	Mn (%)	
Alta Mine	Before 1905	3,500	10	NA	35	1	minor	NA	Direct shipping
Hardshell Incline	1896-1905	20,000	unknown	unknown	unknown	unknown	unknown	NA	Direct shipping and milling
Hardshell Mine	1921-1927	900	20	NA	20	NA	NA	NA	
Hardshell Mine	1905-1940	Several 000s	unknown	unknown	unknown	unknown	unknown	unknown	Direct shipping, with some Mn in WWI
Hardshell Incline	1943-1948	2,500	8	NA	6	NA	NA	NA	ASARCO production, direct shipping
Hardshell Mine	1963-1964	2,900	8	NA	6	NA	NA	NA	McFarland lease from ASARCO, smelter flux
Hardshell Mine	1964 to present	None							
Hermosa Mine	1880-1902	70,000	20	unknown	unknown	unknown	unknown	unknown	About 1.4 million ounces Ag produced in period
Salvador Mine	1880s	Unknown	unknown	unknown	unknown	unknown	unknown	unknown	About 30,000 ounces Ag produced in period
Black Eagle Mine	1880s	4,900	22	NA	NA	NA	NA	unknown	Direct shipping Mn-Ag mineralization
Black Eagle Mine	WWII	Few hundred	unknown	NA	NA	NA	NA	unknown	Direct shipping Mn
Bender Mine	Prior to WWI	50	20	NA	NA	NA	NA	NA	Mn smelter fluxing
Bender Mine	WWI, WWII, 1952-55	6,000	unknown	unknown	unknown	unknown	unknown	unknown	Direct shipping Mn – US Gov't Purchase
Trench Mine	1850-1890; 1918-1945	237,000	13	6.3	8.5	unknown	unknown	unknown	Operated 150 ton/day Pb-Zn floatation mill

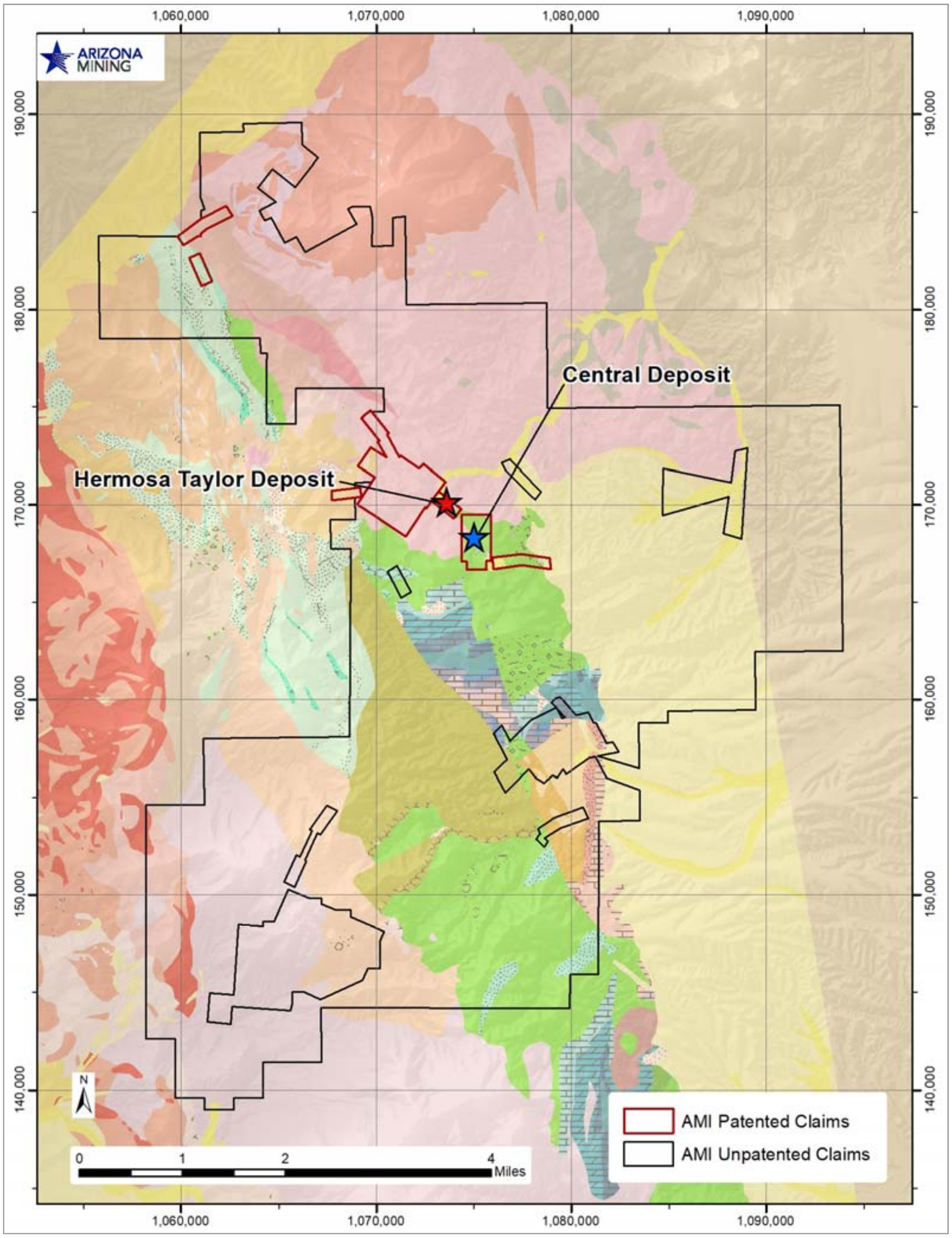
Source: AZ Bureau of Mines, Bulletin 191, 1975.

7 Geological setting and mineralization

7.1 Regional geology

The regional geology of the area is shown in Figure 7.1 obtained from the Geological Map of the Patagonia Mountains, Santa Cruz County, Arizona published by the USGS in 2015. The locations of the Hermosa Taylor Deposit and Central Deposit are indicated by red and blue stars, respectfully. Note the legend is shown on the following page.

Figure 7.1 Regional geology map




Legend
Map units
Symbol, Unit name
Qal—Younger alluvium and talus
QTal—Older alluvium
QTg—Gravel and conglomerate
Tl—Limestone
Tt—Biotite rhyolite tuff
si—Silicification
Tv—Volcaniclastic rocks of middle Alum Gulch
Tib—Intrusive breccia of middle Alum Gulch
Tqp—Quartz feldspar porphyry of middle Alum Gulch
Tqpx—Xenolithic quartz feldspar porphyry of middle Alum Gulch
Tqmp—Quartz monzonite porphyry, in granodiorite of the Patagonia Mountains
Tqmpb—Breccia, in quartz monzonite porphyry (unit Tqmp) of granodiorite of the Patagonia Mountains
Tg—Granodiorite, in granodiorite of the Patagonia Mountains
Tgb—Breccia, in granodiorite (unit Tg) of granodiorite of the Patagonia Mountains
Tlp—Latite porphyry, in granodiorite of the Patagonia Mountains
Tbq—Biotite quartz monzonite, in granodiorite of the Patagonia Mountains
Tbqb—Breccia, in biotite quartz monzonite (unit Tbq) of granodiorite of the Patagonia Mountains
Tbg—Biotite granodiorite, in granodiorite of the Patagonia Mountains
Tibx—Intrusion breccia, in granodiorite of the Patagonia Mountains
Tsy—Syenodiorite or mangerite, in granodiorite of the Patagonia Mountains
Tag—Biotite augite quartz diorite, in granodiorite of the Patagonia Mountains
Tmp—Quartz monzonite porphyry of Red Mountain
TKr—Rhyolite of Red Mountain
TKggt—Gringo Gulch Volcanics
Ka—Trachyandesite
r—Rhyolite or latite, in trachyandesite (unit Ka)
Km—Pyroxene monzonite
Kl—Biotite quartz latite(?)
Kv—Silicic volcanics
la—Biotite latite(?), in silicic volcanics (unit Kv)
Kpg—Porphyritic biotite granodiorite
Kb—Bisbee Formation
Kbc—Conglomerate, in Bisbee Formation (unit Kb)
Jtg—Granite of Three R Canyon, in granite of Cumero Canyon
Jtgb—Breccia, in granite of Three R Canyon (unit Jtg) of granite of Cumero Canyon
Jcm—Porphyritic granite, in granite of Cumero Canyon
Jcs—Equigranular alkali syenite, in granite of Cumero Canyon
Jcsb—Breccia, in equigranular alkalik syenite (unit Jcs) of granite of Cumero Canyon
Jcg—Equigranular granite, in granite of Cumero Canyon
Jcgb—Breccia, in equigranular granite (unit Jcg) of granite of Cumero Canyon
Jhm—Hornblende monzonite of European Canyon
JTRv—Volcanic rocks, in silicic volcanic rocks
ha—Hornblende andesite dike and (or) plug, in volcanic rocks (unit JTRv)
b—Volcanic breccia, in volcanic rocks (unit JTRv)
s—Sedimentary rocks, in volcanic rocks (unit JTRv)
cg—Limestone conglomerate, in volcanic rocks (unit JTRv)
qz—Quartzite, in volcanic rocks (unit JTRv)
ls—Exotic blocks of upper Paleozoic limestone, in volcanic rocks (unit JTRv)
w—Rhyolitic welded(?) tuff, in volcanic rocks (unit JTRv)
lp—Latite(?) porphyry, in volcanic rocks (JTRv)
JTRvs—Volcanic and sedimentary rocks, in silicic volcanic rocks
TRm—Mount Wrightson Formation
q—Quartzite, in Mount Wrightson Formation (unit TRm)
a—Biotite(?)-albite andesite lava(?), in Mount Wrightson Formation (unit TRm)
t—Coarse volcaniclastic beds, in Mount Wrightson Formation (unit TRm)
TRms—Sedimentary rocks, in the Mount Wrightson Formation (unit TRm)
Pcn—Concha Limestone
Ps—Scherrer Formation
Pe—Epitaph Dolomite
Pc—Colina Limestone
PPe—Earp Formation
Ph—Horquilla Limestone
Me—Escabrosa Limestone
Dm—Martin Limestone
Ca—Abrigo Limestone
Cb—Bolsa Quartzite
pCq—Biotite or biotite-hornblende quartz monzonite
pCh—Hornblende-rich metamorphic and igneous rocks
pCm—Biotite quartz monzonite
pCd—Hornblende diorite
USA Topo Maps

7.2 Stratigraphy

The southeastern third of Arizona lies within a belt of 1600 to 1700 Ma-age Proterozoic rocks, dominated by the Pinal Schist, a greenschist-grade metamorphosed argillaceous quartz wacke (Anderson, 1989). The continental crust below these rocks is believed to consist of batholiths appended to the craton during the early Proterozoic. These rocks were then intruded by granitic stocks and batholiths at about 1450 Ma (Silver and others, 1977).

Late Precambrian-Early Paleozoic rifting split the Proterozoic basement into a number of separate continental blocks with passive continental margins (Dickinson, 1989). Phanerozoic shelf-type sediments overlie the Precambrian basement.

The oldest rocks in the Patagonia Mountains are Proterozoic-age granodiorite with subordinate amounts of pelitic schist, diorite and gabbro. Cambrian units in southern Arizona include the Bolsa Quartzite and the Abrigo Formation limestones, dolostones and clastic interbeds. Most of Arizona was above sea level during the Ordovician and Silurian; the Ordovician El Paso limestone, present only in southeastern Arizona, is the only significant unit of this age (Middleton, 1989).

Widespread sedimentary deposition resumed in the upper Devonian. The Martin Formation carbonates are the prevalent Devonian units in the southern part of the state, along with the Percha Formation. They are overlain by the Mississippian Escabrosa Limestone, the dominant Mississippian unit in southern Arizona (Beus, 1989).

Pennsylvanian-Permian-age sandstones, shales and carbonates were deposited during a time of shifting and cyclical environments (Blakey and Knepp, 1989). The Pennsylvanian Naco Group of southeastern Arizona is comprised of Pennsylvanian Horquilla Limestone, the Pennsylvanian-Permian Earp Formation and the Permian Colina Limestone, Epitaph Dolomite, Scherrer Formation and Concha Limestone (Gilluly and others, 1954).

The Epitaph Formation, Scherrer Formation and the Concha Formation (Paleozoics) underlie the Property and are disconformably overlain by Cretaceous andesites and Triassic / Jurassic rhyolites. The carbonate replacement deposit (CRD), known as the Taylor Deposit, is comprised of lead-zinc-silver sulphide mineralization that was predominantly deposited along this disconformable contact and also occurs intermittently throughout the three carbonate formations. The Manto mineralization, known as the Central Deposit, is comprised of manganese-silver-zinc oxides and was also deposited along the Jurassic rhyolites Paleozoic carbonate contact, however the Manto mineralization is limited to this contact and does not extend below the Concha into the underlying Paleozoic formations.

Mesozoic-age volcanic, sedimentary and intrusive rocks lie disconformably above the Paleozoic stratigraphic sequence. Cretaceous-age intermediate and felsic volcanic and intrusive rocks cover much of the Property and surrounding areas. In the northwestern Patagonia Mountains, Jurassic granite intrudes Triassic to Jurassic volcanic and sedimentary rocks. Most of the central and southern parts of the range consist of Laramide-age (64 Ma to 58 Ma), medium to coarse-grained hornblende granodiorite batholithic rocks. The batholith is bounded by northwest-striking faults and its emplacement is thought to have been structurally controlled.

Laramide felsic volcanic and intrusive stocks are prevalent at Red Mountain and west of the historic Trench mining camp in the Chief-Sunnyside Diatreme area. Intrusive rocks and alteration at Sunnyside are thought to be coeval with alteration at the Property.

Late Oligocene to Miocene conglomerates, sandstones, ash flow tuffs and lakebed sedimentary rocks onlap the Property and fill the San Rafael Basin to the east of the Patagonia Mountains and the northeastward-trending Sawmill Creek Basin.

7.3 Regional structural geology

The structural character of Arizona was largely established during the late Mesozoic and Tertiary, although there is evidence that older (Precambrian) structures were reactivated during this time (Krantz, 1989). Laramide, Mid-Tertiary, and Late Tertiary tectonic phases are recognized in southern Arizona.

7.3.1 Laramide

The Laramide orogeny in southern Arizona generated north-south and northeast-southwest compressional stresses that resulted in regional tectonic fabrics and thrust faulting. Structures in the southeast Arizona province have been particularly controversial. Interpretation of a regional over thrust terrane has been advocated by numerous workers, most prominently and recently by Harald Drewes (1981). The Drewes model proposes low-angle reverse and thrust faults as a response to southwest-northeast Laramide compressional stresses, with a thrust slip of perhaps 62 miles (100 km). This model has been of particular interest in petroleum exploration circles.

In contrast, the basement uplift model views the same nearly flat faults as normal in sense, with considerable lateral displacement of thrusted, folded basement rocks. In this manner, these “detachment faults” react to the same southwest-northeast stresses recognized by the overthrust model (Rehrig and Heidrick, 1976; Heidrick and Titley, 1982). This core complex–detachment theory is now widely viewed as the preferred structural model for southern Arizona, part of a pattern extending to the Canadian border. Detachment structures are now recognized as important hydrothermal metallic deposit hosts in the southwestern US.

7.3.2 Middle Tertiary

Laramide deformation was followed by a relative structural and magmatic respite during the Eocene epoch and then by renewed tectonism and magmatic activity during the Oligocene to mid-Miocene. Middle Tertiary tectonism was characterized by crustal extension, with stresses directed in an ENE-WSW axis, plus attendant magmatism dominated by intermediate to silicic melts. Extension resulted in normal faulting and rotation of fault blocks over much of Arizona.

Menges and Pearthree (1989) summarize mid-Tertiary extensional features as follows:

1. Calc-alkalic rhyolitic to basaltic volcanism.
2. Emplacement of shallow plutons.
3. Basin development and filling by sediments.
4. Rotation of sediments and volcanics on low-angle normal faults and detachment faults.
5. Shear zones and cataclastic fabrics at deeper levels.
6. Northeast-trending folds with amplitudes of several km.

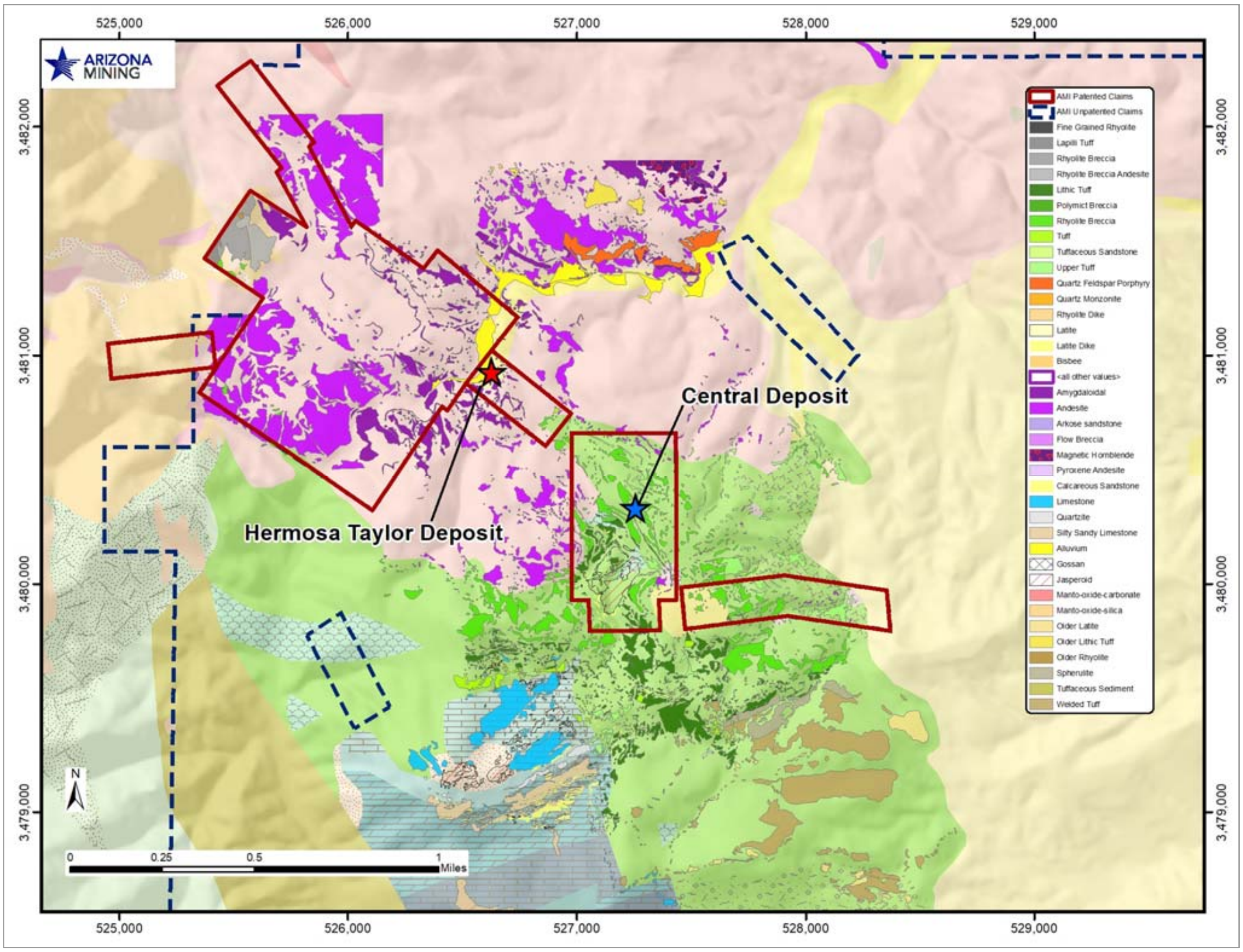
Detachment faults were the most important structural features active during the mid-Tertiary in Arizona. In contrast to detachment faults engendered by compression in Laramide times, mid-Tertiary detachments gained their low-angle normal displacement by means of crustal thinning. Isostatic uplift of crustal segments denuded by erosion is the preferred mechanism for this phase of detachment faulting. Evidence points to mid-Tertiary low-angle normal faulting accounting for 85% to 95% of Tertiary crustal extension, with late Tertiary normal faulting accounting for the remainder.

7.4 Project geology

The Property hosts two stratigraphically controlled mineral deposits; Taylor Deposit (Taylor Sulphide and Taylor Deeps) and the Central Deposit. The two Taylor domains are separated by a low angle thrust fault. The Taylor Sulphide is predominantly a Carbonate Replacement Deposit (CRD) which extends downward, to significant depth (3,600 ft or 1,100 m), and principally occurs in three recognized sedimentary formations on the property and is comprised of Zn-Pb-Ag-Cu sulphides. The Taylor Deeps is a CRD which is confined to a relatively flat contact, below the thrust fault, between the Older Volcanics and Permian carbonates at a depth of 3,400 ft (1,036 m) below the surface. The Central Deposit is a manto style deposit which is confined to the contact between Permian carbonates and the overlying Jurassic rhyolites and does not permeate below the Concha limestone (100 ft - 500 ft or 30 m – 150 m). The Central Deposit is comprised of Mn oxides with accessory silver and zinc. The host rocks (Jurassic Rhyolites and Permian sediments) strike approximately southwest-northeast and dip ± 30° to the northwest. They do not appear to be significantly disrupted by post-mineralization faulting at deposit scale.

An outcrop geological map for the Property is shown in Figure 7.2.

Figure 7.2 Taylor Property geology



7.4.1 Lithology and stratigraphy

The Cretaceous, Triassic / Jurassic and Jurassic volcanic rocks and the underlying Permian sedimentary rocks of the Property are divided into the following units (Figure 7.6), recognized across the property and in the drillholes. They are listed and described from youngest to oldest, with features shown in millimeters (mm).

Intrusive Rocks (Tk – Tertiary – Cretaceous): Two types of intrusives have been identified in the drilling. The first intrusive has been classified as a quartz-feldspar-porphyry (QFP). The intrusive has primarily been intercepted in the Taylor Sulphide domain as narrow dikes following high angle structures through the Paleozoic sequence and along the contact between the Concha and Scherrer formations as narrow sills. The second intrusive identified in drilling is a diorite. The diorite intrusive is most common below the thrust fault and Taylor Deeps Sulphide domain at depths greater than 3,400 ft (1,036 m). It's possible that these are feeders for the Meadow Valley Andesite (Kmv). The intrusives have not been age dated.

Trachyandesite of Meadow Valley (Kmv-Cretaceous): It is an approximately conformable, complex flow unit that overlies the Hardshell Volcanic Sequence on the western and northern margins of the Property. Drilling shows local dikes of similar composition. The trachyandesite is variably described as dark gray to brown, fine to medium-grained with 1 mm to 3 mm euhedral-subhedral plagioclase phenocrysts and sparse 2 mm to 5 mm square K-feldspar phenocrysts in a fine-grained plagioclase-pyroxene-amphibole groundmass. It may contain interstitial magnetite and is generally fresh to weakly propylitized, especially on fractures.

Hardshell Volcanic Sequence (Jh-Jurassic): Five distinct rhyolitic volcanic units have been identified in the Hardshell sequence in addition to a basal Tuffaceous Sandstone, and have been correlated between surface mapping and drillholes.

Rhyolite crystal tuff (Jhct): Appears to be the uppermost unit in the Hardshell volcanic sequence and is conformable with underlying rhyolite breccia unit (Khb). Described as white to gray to buff to locally pale pink, fine- to medium-grained, and crystal-rich. Rare, thin, relict bedding planes. Abundant 1 mm to 3 mm plagioclase crystals and rare 0.5 mm, broken quartz eyes. Rare patches and zones of 5 mm to 15 mm, angular to subrounded lithic clasts.

Rhyolite Breccia (Jhb): Prominent outcrop former in the Hardshell Ridge zone. Clast-supported or nearly clast-supported fragmental unit with abundant 1 mm to 5 m (15 ft), angular, unsorted, rhyolite clasts in very-fine-grained rhyolitic groundmass. Contains abundant clasts with diameters greater than core diameter.

Rhyolite Lithic Tuff (Jhlt): Gray to gray-green, locally crystal-rich tuff with common 5 mm to 25 mm rhyolitic lithic fragments. Abundant 1 mm to 25 mm, partially-collapsed and flatted pumice fragments in very-fine-grained, partially welded groundmass give the rock a distinctive, eutaxitic texture.

Rhyolite Polymict Breccia (JhHZ): This is the unit that comprises the Hardshell Zone, and is interpreted to be the primary host to the deposits exploited by the old Hardshell Incline workings. Rhyolite volcanoclastic and fragmental unit with abundant 1 mm to 25 mm, angular, rhyolite lithic clasts in a welded, eutaxitic matrix. Distinguished from the Jhlt unit by the presence of sparse to very abundant sedimentary clasts derived from underlying Paleozoic rocks. Contains limestone clasts, up to 10 ft (3 m) or more in diameter. Commonly mineralized with Mn-oxide as 1 mm to 10 mm blebs and larger pods up to complete replacements, as well as in veins / veinlets and fracture coatings. Limestone clasts are replaced by Zn-Pb-Ag sulphides, at depth, in the northwest area of the property. Local zones of gray, vuggy, pervasive silicification.

Rhyolite Tuff (Jht): Basal unit in the Hardshell Volcanic Sequence. Light gray, massive, rhyolite tuff with rare, fine-grained plagioclase phenocrysts and rare, < 10 mm lithic clasts in very-fine-grained, tuffaceous groundmass. Local irregular, faint relict bedding and weak, hematite-limonite liesegang banding. Lies directly on Paleozoic sedimentary rock in the western part of the property, and on the spherulite unit (JoSP) of the Older Volcanic Sequence to the east.

Tuffaceous Sandstone (Jhtss): Tan to reddish-brown, granular, fine-grained, massive to thin bedded, reworked, partially silicified tuffaceous-sandstone. Composed of fine-grained quartz and feldspar with sparse lithic fragments. Unconformably overlies the Paleozoics.

Older Volcanic Sequence (Jo-Triassic / Jurassic): The Older Volcanic Sequence is a predominantly rhyolitic volcanic package that underlies the Hardshell Sequence in the southeastern part of the property and contains lithologies that occur as clasts in the Hardshell Volcanic Sequence, especially in the Khb and Khlt units. The Older Volcanic Sequence has not been mapped in detail, and relatively few core holes penetrate the unit. The following units have been recognized and placed in a tentative stratigraphic sequence.

Rhyolite Spherulite Zone (JoSP): Abundant, crowded, 1 mm to 100 mm, semi-spherical, zoned, partially devitrified spherulites in very-fine-grained partially welded groundmass.

Rhyolite Welded Tuff (JoT): Light reddish-gray to purple, densely welded crystal tuff with strong to subtle laminar eutaxitic texture. Abundant, 0.1 mm to 3 mm, subhedral to euhedral, plagioclase phenocrysts in shard-bearing, eutaxitic, very-fine-grained groundmass. Laminated to thin-bedded, locally contorted due to flowage. This rock type is the most common clast lithology in the Khb of the Hardshell Volcanic Sequence.

Latite Porphyry (JoLA): Distinctly porphyritic intrusive and / or flow unit with prominent, abundant, 1 mm to 5 mm, subhedral to euhedral, white, prismatic plagioclase phenocrysts and less common 1 mm to 5 mm, euhedral, white, approximately equant K-feldspar phenocrysts. Rare, relict, 0.1 mm to 1 mm, rotten, biotite books in fine to medium-grained, red-brown groundmass.

Lithic Tuff (JoLT): Greenish-gray, fragmental. Rare, 1 mm to 3 mm, subhedral plagioclase phenocrysts, Common 1 mm to 25 mm, angular, lithic clasts in fine-grained, partially silicified, tuffaceous groundmass.

Basal Breccia (Jobb): Structure / basal breccia of the older volcanic sequence. Abundant, angular to sub rounded, 50 mm to greater than core diameter, clasts of lithic fragments of older volcanics, tuffaceous sandstone, limestone and sparse sulphide clasts replacing limestone. Common to abundant quartz and calcite veins.

Tuffaceous Sandstone (Jotss): Tan to reddish-brown, granular, fine-grained, massive to thin bedded, reworked, partially silicified tuffaceous-sandstone. Composed of fine-grained quartz and feldspar with sparse lithic fragments. Unconformably overlies the Paleozoics.

Concha Formation (Pzlc-Paleozoic): Gray, massive, fine-grained, recrystallized limestone-marble with common 1 cm by 5 cm to 10 cm by 25 cm, irregular dark gray to black chert pods. Local 1 mm to 5 mm wide, irregular, discontinuous calcite veinlets. Prominent chert nodules and complete absence of sandy detritus distinguish the Concha Formation limestone-marble from the underlying Scherrer Formation.

Scherrer Formation (Paleozoic): Three lithologic members comprise the Scherrer formation stratigraphy:

Upper Member (Pzcs): A calcareous sandstone. Light gray, massive. 30% to 60%, fine-grained, well-rounded, well-sorted quartz sand in calcareous matrix. Sparse, relict thin bedding.

Middle Member (Pzl): Massive to irregular thin-bedded limestone which includes variations of silty and cherty limestone.

Lower Member (Pzq): Gray quartzite, massive thin-bedded. 60% fine-grained, well-rounded, well-sorted quartz and is non-calcareous.

Epitaph Formation (Paleozoic): Three lithological members comprise the Epitaph Formation:

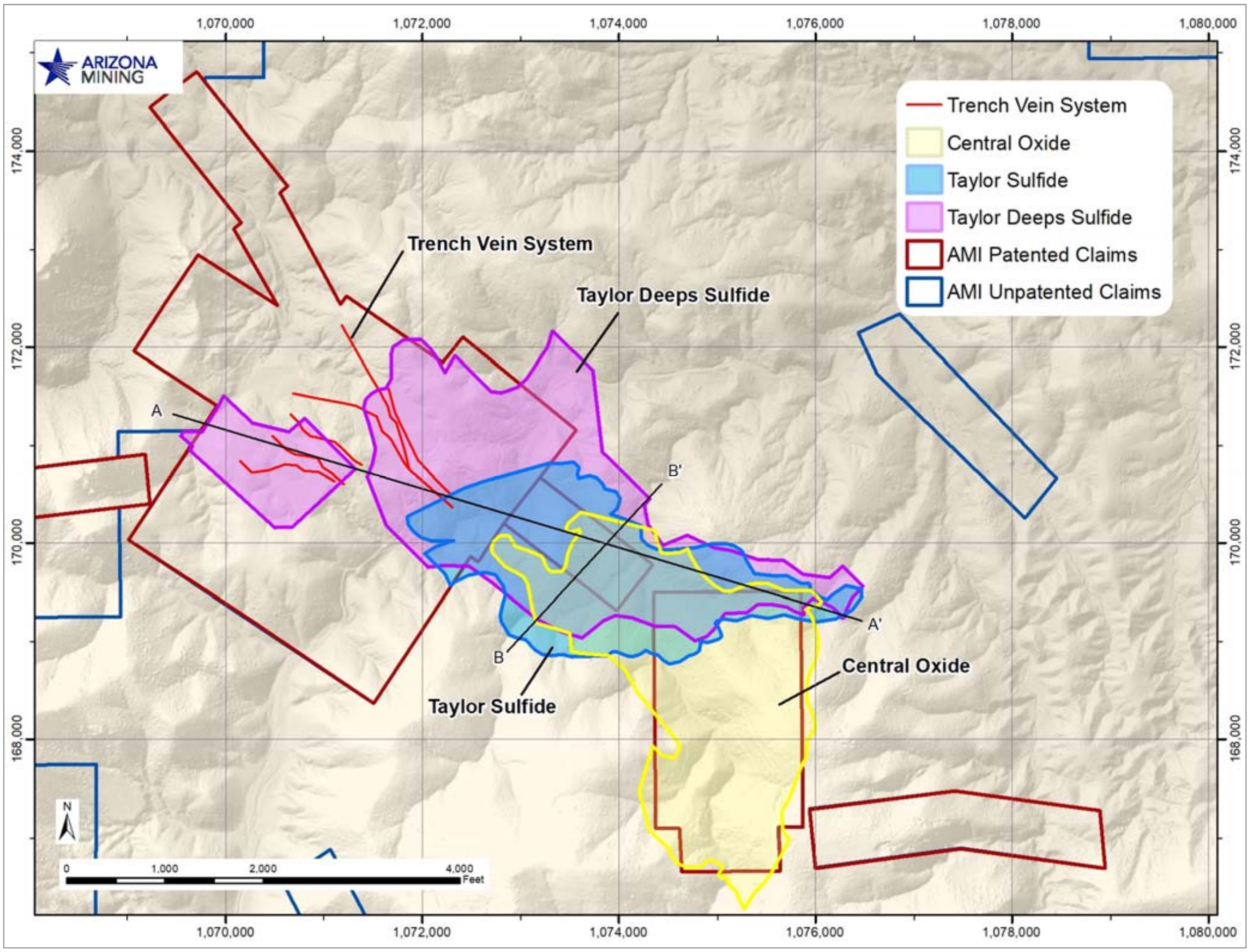
Limestone (Pzls): Gray, bleached, massive to irregularly thin-bedded, very-fine-grained limestone with rare, 1 cm by 5 cm to 10 cm^3 by 25 cm^3, irregular dark gray to black chert pods with 1 mm to 10 mm, talc selvages. Common 1 mm^3 to 25 mm^3, spots, pods and ovals of white calcite after gypsum.

Silty Limestone (Pzst). Gray: thin-bedded, very-fine-grained, silty. Well preserved, 0.1 mm to 1 mm, regular, thin-beds. Common carbonaceous slips and partings. Common, very-fine-grained, pyritic partings. Common, short intervals without thin-bedding. Reactive to hydrochloric acid.

Carbonaceous Limestone (Pzcl): Massive to thin-bedded, dark gray to black carbonaceous limestone.

The projected outlines of the Taylor and Central Deposits along with the relationships to the mineral claims are shown in Figure 7.3, and generalized long and cross sections of the Taylor Deposit and Central Deposit are shown in Figure 7.4 and Figure 7.5. Figure 7.6 is a stratigraphic column for the Property.

Figure 7.3 Plan of outlines for Taylor and Central Deposit



Note the section lines for sections illustrated in Figure 7.4 and Figure 7.5.

Figure 7.4 Generalized long section (looking northeast) of the Taylor and Central Deposits

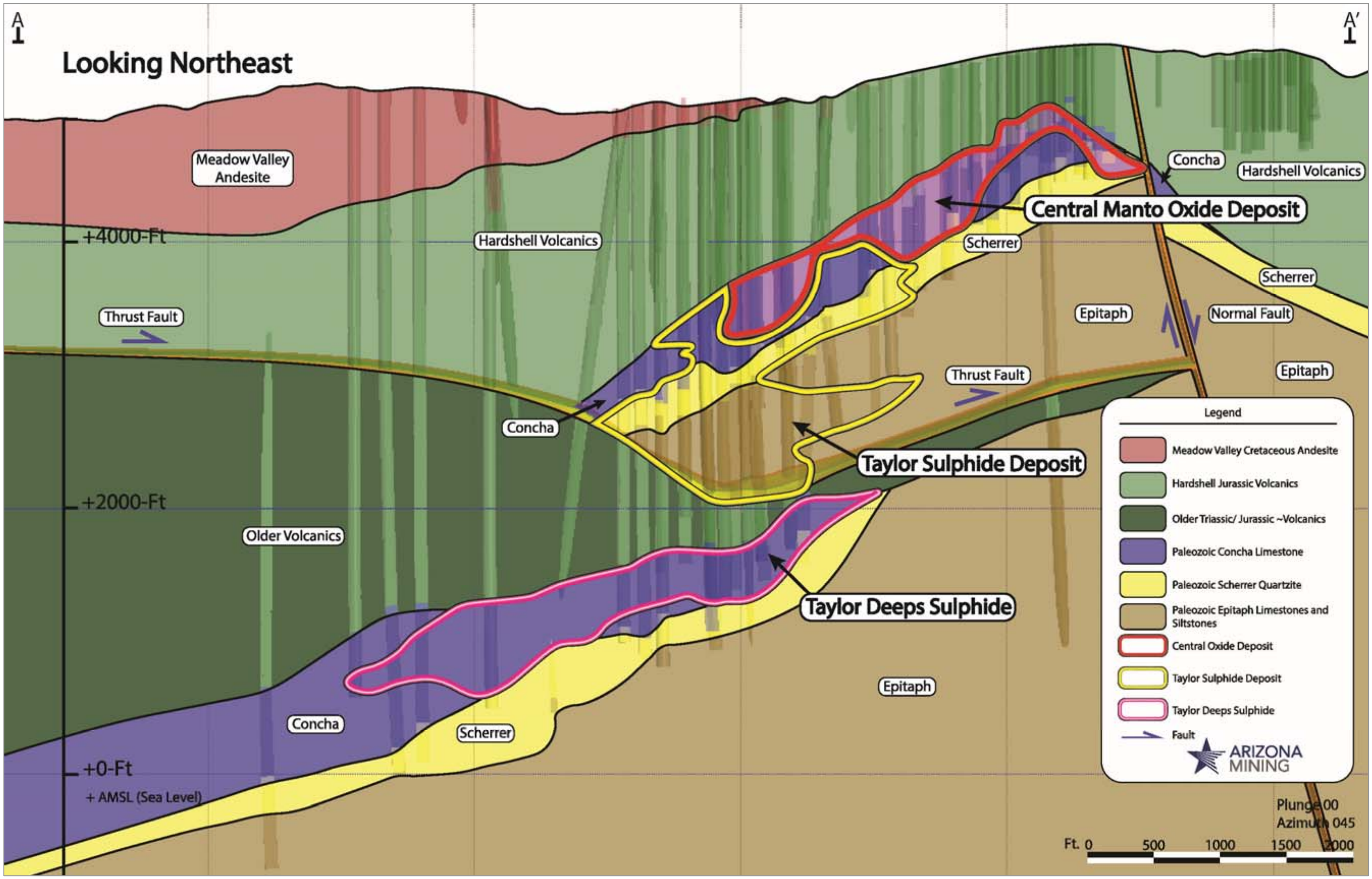


Figure 7.5 Generalized cross section (looking northwest) of the Taylor and Central Deposits

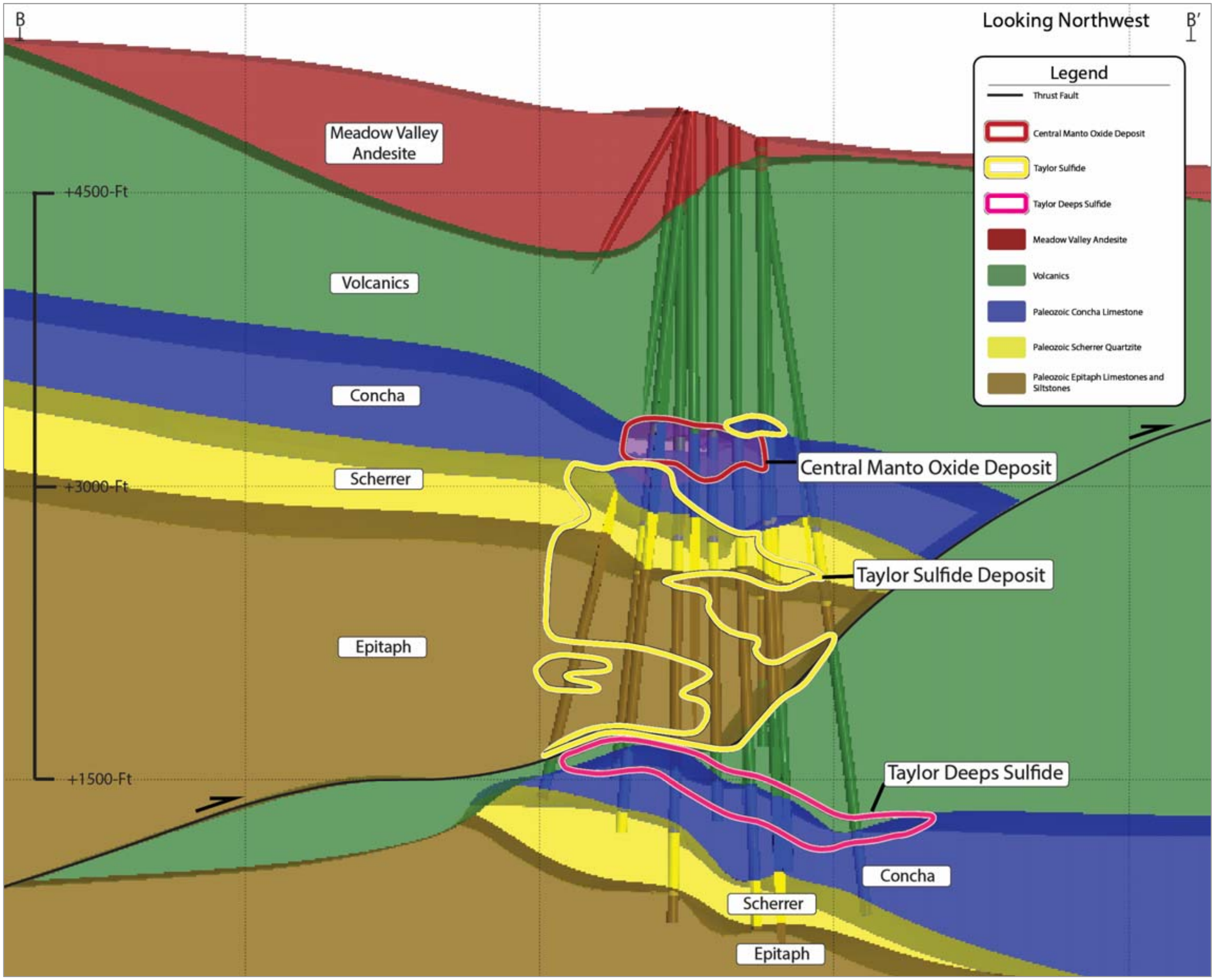
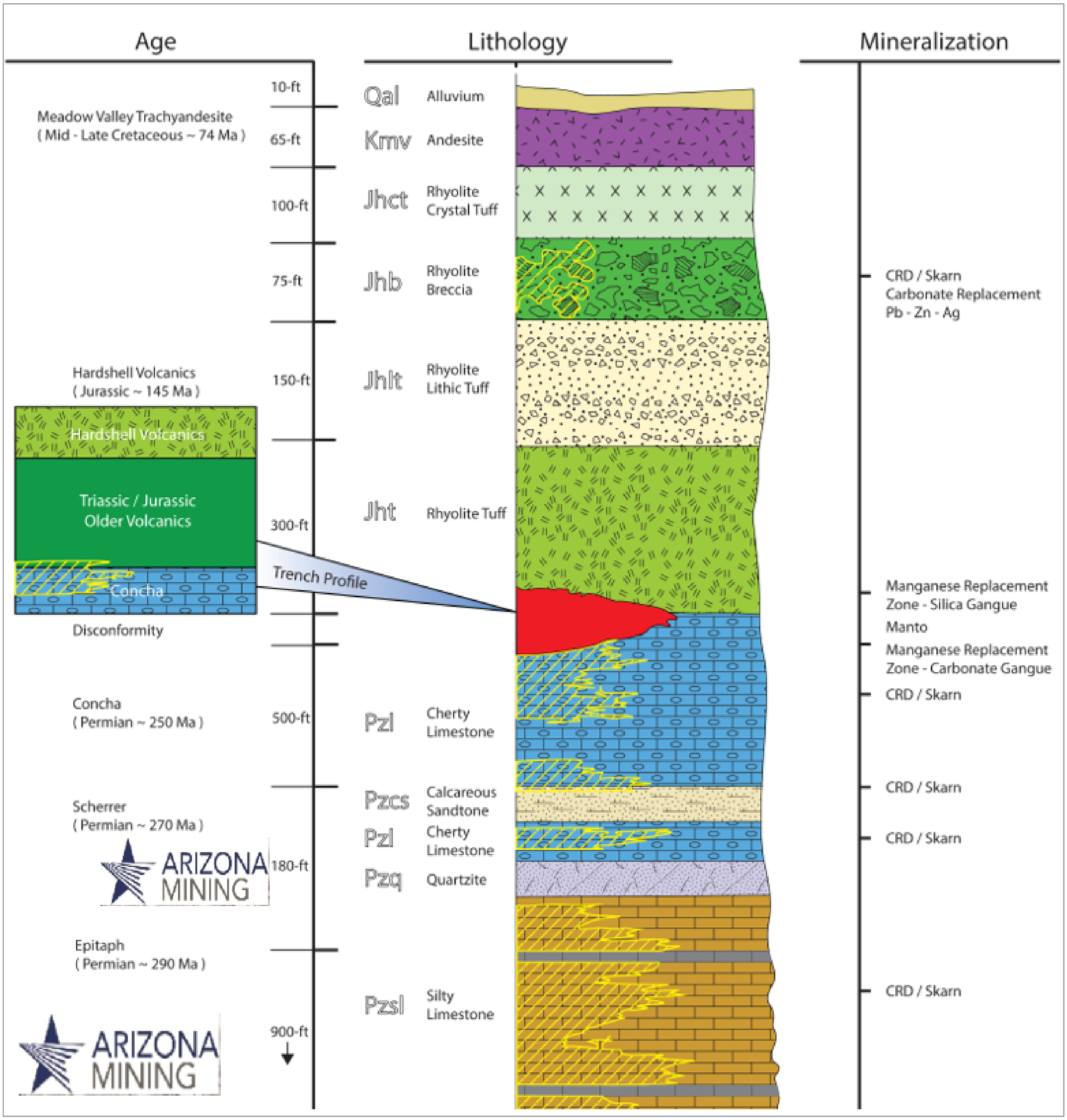


Figure 7.6 Stratigraphic column for the Property



Note: Individual units have not been measured for true stratigraphic extent. Their thicknesses are represented relative to one another.

7.4.2 Structural geology

A northeast-southwest trending structure corridor divides the Project into two structural domains. This zone intersects a northwest-southeast trending conjugate set that lies south of the main Hermosa patented claim block and runs through the Black-Eagle and Bender mine areas. A second northwest-southeast trending structural zone runs through the centre of the patented claim block and has been known by previous workers as the Hudson Fault Zone.

Outcrops, old workings and road cuts are commonly disrupted by irregular, discontinuous, complex structural zones. These zones are characterized by rubbly, broken, brecciated and sheared features that do not typically displace either lithologic contacts or alteration or mineralization zones at map or cross-section scale (typically 1:2400).

7.4.2.1 High angle structures

There are two main fault orientations observed on the property, a northeast-southwest trending structural zone and a northwest-southeast.

One of the main fault orientations on the Property is a northeast-southwest trending structural zone that runs through the southeastern corner of the main patented claim block. This zone intersects a northwest-southeast trending conjugate set that lies south of the main Hermosa patented claim block and runs through the Black-Eagle and Bender mine areas. A second northwest-southeast trending structural zone runs through the centre of the patented Alta and Trench claim blocks and has been known by previous workers as the Hudson Fault Zone (Alta claim block), Trench-Josephine Fault Zone and January-Norton Fault Zone (Trench claim block). This structural zone is interpreted as controls for mineralization in the Volcanics (Cretaceous through Triassic) and Paleozoic Sediments.

Outcrops, old workings and road cuts are commonly disrupted by irregular, discontinuous, complex structural zones. These zones are characterized by rubbly, broken, brecciated and sheared features that do not typically displace either lithologic contacts or alteration or mineralization zones at map or cross-section scale (typically 1:2400).

7.4.2.2 Low angle structure

A low angle thrust fault has been identified on the property through drilling. The fault is assumed to be Mesozoic in age and has emplaced three members of the Paleozoic sequence (Concha, Scherrer and Epitaph) over the Triassic - Jurassic Older Volcanics creating a wedge of “Older” Volcanics below the Paleozoic sequence. The primary direction of movement along the thrust is from the south to north. Due to the lack of surface expression in the Cretaceous Meadow Valley Andesite, there is no evidence that the thrust fault propagates through the entire Jurassic - Hardshell Volcanic Sequence. It’s likely that the thrusting occurred during the deposition of Hardshell Volcanics.

7.4.3 Alteration

Rhyolitic rocks, particularly Jhb, across the Property are uniformly light gray to tan, with primary volcanic and clastic textures generally well preserved. The same rocks are generally shades of purple to maroon where they crop out at a distance from known mineralization. Locally, in otherwise unaltered rhyolite outcrops, small patches of fine-grained secondary K-feldspar have been noted. These observations suggest that the tan coloration proximal to mineralization may be pervasive and moderately-strong potassic alteration. This alteration appears to form a broad background upon which later alteration more directly associated with the Property mineralization has been imposed. The clasts within Hardshell volcanic sequence lithic tuff and breccia are commonly selectively overprinted by white kaolinite-sericite veinlets and patches. The fine-grained, tuffaceous, matrix to the lithic tuff, polymict breccia and lower rhyolite tuff are pervasively overprinted by very-fine, disseminated kaolinite-sericite. In both cases, primary textures are generally very well preserved, and the rock remains competent and hard.

Where mineralization occurs at the contact between Jurassic and Permian rocks, it exhibits an asymmetric envelope of pervasive and strong silicification, referred to in the past as “jasperoid”. The greatest volume and the most massive expression of this silicification is within the rhyolite tuff in the hanging wall of the mineralization where it commonly penetrates more than 30 ft (10 m) above mineralization. In the footwall carbonates, silicification is less complete and penetrates only a few metres below the volcanic-carbonate contact into the Concha limestone. Primary minerals and textures in these rocks are completely replaced by grey, fine-grained quartz. Rare, small patches or pods of ghostly relict volcanic texture have been noted. Where quartz sulphide veins are present in the Jurassic volcanics, pervasive silicification of the host rock is associated with the vein-forming event.

Concha, Scherrer and Epitaph Formation carbonate rocks are weakly to moderately recrystallized and contain fine to coarse, irregular and discontinuous calcite veinlets. These rocks are commonly bleached to a light gray color. Fossils are normally well preserved along with fine primary sedimentary textures. Drillholes in the northwestern part of the property intersected increasingly pervasive and stronger recrystallization of the carbonate rocks that ultimately grades into diopside-wollastonite-rhodonite calc-silicate skarn with associated base metal sulphide mineralization. Calcareous sandstone intervals contain fewer calcite veinlets but they are still present. Quartzite only rarely host calcite veinlets.

Andesite drill intercepts and outcrops typically contain fine, thin, irregular and discontinuous calcite veinlets and may also contain finely-distributed groundmass calcite. Biotite, where present, is typically degraded with greenish chlorite selvages. Magnetite is occasionally noted, and pyrite is not uncommon.

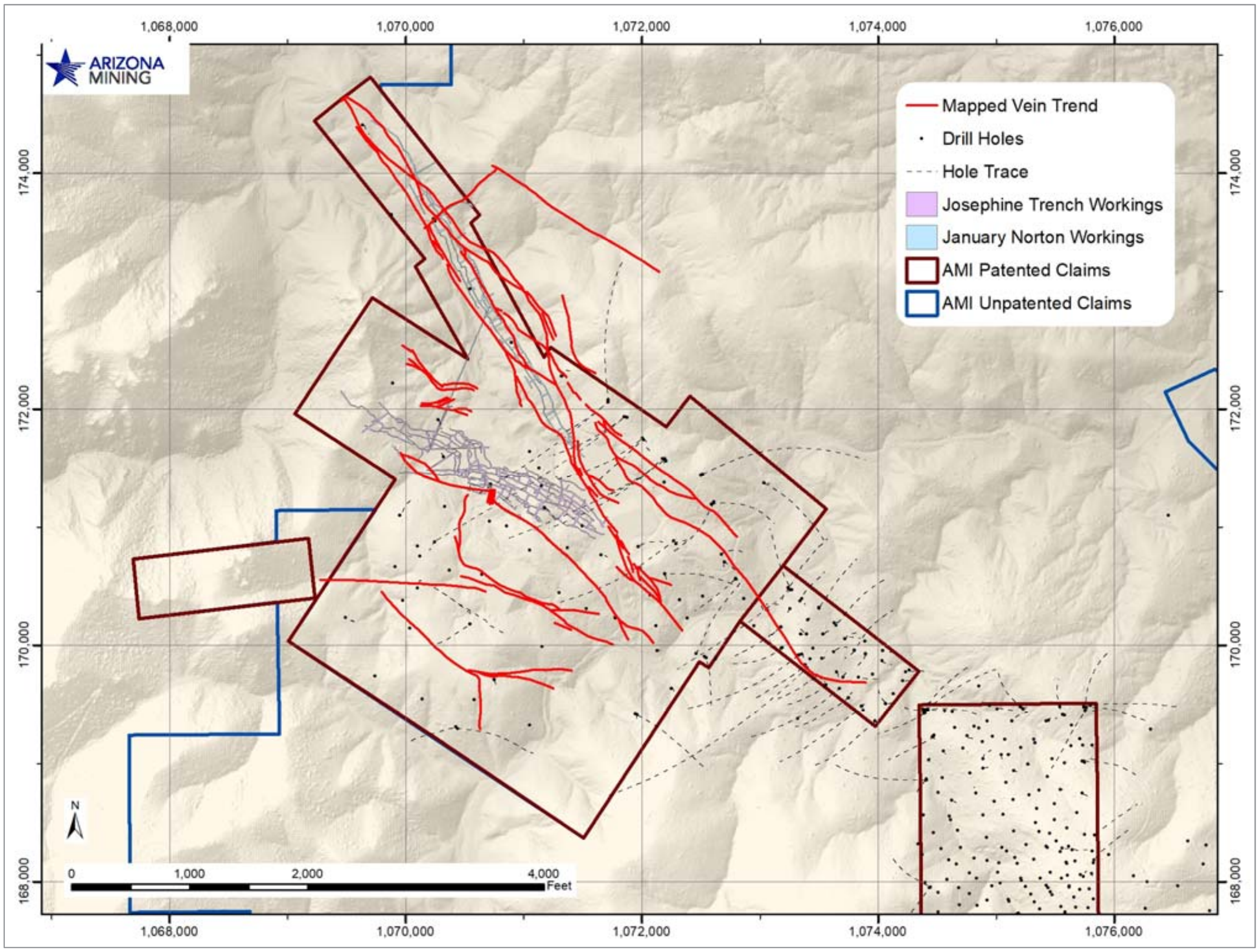
7.4.4 Mineralization

Mineralization has been subdivided in to two mineral-types, sulphide (CRD, Skarn and vein) and oxide (Manto). The Taylor Deposit sulphide CRD mineralization is developed within two domains. The upper mineralized domain consists of the Concha Formation, Scherrer Formation and Epitaph Formation of the Paleozoic sequence around the patented Alta claim block. Continuity of CRD mineralization in the Paleozoic-age carbonate formations extends for 4,400 ft (1,340 m) along strike (Northwest 310°) and 1,500 ft (457 m) laterally (Northeast 40°) beneath the eastern edge of the Hardshell claim extending across the entire Alta claim to the Southeast edge of the Trench claim block. Thickness of mineralization varies depending on the stratigraphic horizon. The average thickness of mineralization, on the basis of drillhole intercepts, for each stratigraphic host is: Concha – 200 ft (61 m), Scherrer – 60 ft (18 m) and Epitaph 300 ft (91 m). The lower domain of mineralization is characterized by calc-silicate mineralogy and occurs between contact of the Older Volcanics and Paleozoic sediments at a depth of 3,400 ft (1,036 m) below the surface. The average thickness of mineralization in this zone is 75 ft (23 m) and extends for 6,900 ft (2,100 m) from the east edge of the Hardshell claim to the west edge of the Trench claim (Northwest 310°). Laterally (Northeast 40°), the mineralization extends 2,600 ft (790 m).

Sulphide mineralization in the Taylor Deposit also occurs as calc-silicate skarn type mineralization that contains patches and massive, wholesale replacements of carbonate by very-fine-grained, massive, wollastonite-diopside and rhodonite, generally white to pink, very-fine-grained to aphanitic, hard and massive. Significant, sparse zones with coarse-grained, radiating crystal aggregates up to 2 cm and common coarse-grained, euhedral-subhedral galena, sphalerite, chalcopyrite, and pyrite are present. Massive replacements up to 20 ft (6 m) thick of carbonate by galena, sphalerite, chalcopyrite and pyrite are not uncommon. Light green, massive, coarse-grained garnet with abundant sulphides as disseminations, pods, masses and interstitial replacements are sparsely noted, deep within the Epitaph Formation and is directly related to intrusive dikes and sills. This style of sulphide mineralization is not as common but is present.

Vein-hosted sulphide mineralization occurs in northwest trending structural zones (azimuth 310°) (Figure 7.7) and is interpreted as being high-angle (75° - 85°) and dipping to the northeast. Vein thicknesses vary from 1.5 ft (0.5 m) up to 6 ft (2 m) and can occur as single veins or vein zones up to 20 ft (6 m) thick with a strike length of 5,000 ft (1,524 m). The veins are comprised of white, massive quartz with open-space, growth-zoned quartz crystals and contain coarse grained sulphides (pyrite, galena and sphalerite). Quartz–sulphide veins have been noted in all stratigraphic formations on the Property and are believed to be related to CRD mineralization in the Paleozoic sequence, “Hardshell Zone” and the veins exploited by ASARCO in the Meadow Valley Andesite on the Trench Claims.

Figure 7.7 Structural trends with quartz sulphide vein mineralization



The Central Deposit is comprised of oxide manganese / zinc / silver mineralization (Manto). The oxidized rhyolites overlying the manto-style mineralization and the carbonate units contain irregular patches and zones of veinlet-controlled hematite-limonite and sooty Mn-oxide with accessory zinc and silver mineralization. Manto-style mineralization in rocks of rhyolitic composition is dominated by black, sooty cryptomelane, with or without yellowish orange secondary lead-oxides and with quartz-dominant gangue mineralogy. Manto-style mineralization in carbonate rocks does not typically contain lead-oxides. Strong, pervasive gray, silicification is also present and calcite occurs as veinlets, vugs, and fracture fillings. Drill core intercepts containing rhodochrosite and pink calcite are not uncommon and rarer intercepts of hard pinkish rhodonite-bustamite have also been noted.

A separate, noteworthy horizon in the Jurassic rhyolitic volcanics has been designated the “Hardshell Zone” contains both sulphide and oxide mineralization. This zone supported historic mining at the Hardshell Incline mine and is composed of a 10 ft to greater than 100 ft (3 m to 30 m) thick polymict rhyolite breccia with a minor portion of clasts of carbonate sedimentary provenance. This horizon is the locus of partial to massive Mn-oxide replacement mineralization in the southeastern drillholes and partial to massive Pb-Zn sulphide replacement mineralization in the northwestern drillholes beneath the Alta and Trench.

8 Deposit types

The Property hosts two stratigraphically-controlled mineral deposits which are described below.

8.1 Taylor Deposit

The Taylor Deposit closely conforms to the genetic class of polymetallic CRD. The salient characteristics of this class of deposit are described below. This description is taken, with modification, from Nelson, 1996.

Geological characteristics

Irregularly shaped, conformable to crosscutting bodies, such as massive lenses, pipes and veins, of sphalerite, galena, pyrite and other sulphides and sulphosalts in carbonate hosts; distal to skarns and to small, high-level felsic intrusions.

Tectonic setting: Intrusions emplaced into miogeoclinal to platformal, continental settings.

Depositional environment / geological setting: In northern Mexico, most are hosted by Cretaceous limestones. In Colorado, the principal host is the Devonian-Mississippian Leadville limestone; in Utah, the Permian Torweap Formation hosts the Deer Trail Deposit. The most favourable hosts in the Canadian Cordillera are massive Lower Cambrian and Middle Devonian limestones, rather than impure carbonates and dolostone-quartzite units.

Age of mineralization: In the southern Cordillera deposits of this class are typically Tertiary in age.

Host / associated rock types: Hosted by limestone and dolostone. The carbonates are typically within a thick sediment package with siliciclastic rocks that is cut by granite, quartz monzonite and other intermediate to felsic hypabyssal, porphyritic lithologies. There may be volcanic rocks in the sequence, or more commonly above, which are related to the intrusive rocks.

Deposit form: Irregular: mantos (cloak shaped), lenses, pipes, chimneys, veins; in some deposits the chimneys and / or mantos are stacked.

Texture / structure: Massive to highly vuggy, porous mineralization. In some cases, fragments of wallrock are incorporated into the mineralization. Some deposits have breccias: fragments of wallrock and also of sulphide mineralization within a sulphide matrix.

Mineralogy (principal and subordinate): Sphalerite, galena, pyrite, chalcopyrite, marcasite; arsenopyrite, pyrargyrite / proustite, enargite, tetrahedrite, geocronite, electrum, digenite, jamesonite, jordanite, bournonite, stephanite, polybasite, rhodochrosite, sylvanite, calaverite. Chimneys may be more Zn-rich, Pb-poor than mantos.

Gangue mineralogy (principal and subordinate): Quartz, barite, gypsum; minor calc- silicate minerals.

Alteration mineralogy: Limestone wallrocks are commonly dolomitized and / or silicified, whereas shale and igneous rocks are argillized and chloritized. Jasperoid occurs in some U.S. examples.

Weathering: In some cases, a deep oxidation zone is developed. Mexican deposits have well developed oxide zones with cassiterite, hematite, Cu and Fe carbonates, cerussite, and smithsonite.

Mineralization controls: The irregular shapes of these deposits and their occurrence in carbonate hosts emphasize the importance of ground preparation in controlling fluid channels and depositional sites. Controlling factors include faults, fault intersections, fractures, anticlinal culminations, bedding channel ways (lithologic contrasts), karst features, and pre-existing permeable zones. In several districts karst development associated with unconformities is believed to have led to development of open spaces subsequently filled by mineralization. Some deposits are spatially associated with dikes.

Genetic model: Manto deposits are high-temperature replacements as shown by fluid inclusion temperatures in excess of 572°F (300°C), high contents of Ag, presence of Sn, W, and complex sulphosalts, and association with skarns and small felsic intrusions. They are the product of pluton-driven hydrothermal solutions that followed a variety of permeable pathways, such as bedding, karst features and fracture zones.

Associated deposit types: There is probably an overall outward gradation from granite- hosted Mo-Cu porphyries, endoskarns and possibly W- and Sn mineralization, through exoskarns and into Ag-Pb-Zn veins, mantos and possibly Carlin-type sediment-hosted Au-Ag deposits. Only some, or possibly one, of these types may be manifest in a given district. Ag-Pb-Zn vein, manto and skarn deposits belong to a continuum which includes many individual occurrences with mixed characteristics.

8.2 Central Deposit (Oxide)

The Central Deposit is also a CRD manto style deposit comprised predominately of cryptomelane-type manganese oxide minerals. Silver and base metals occur predominately in the lattice-structure of cryptomelane. Accessory silver-bearing sulphides as well as lead oxides and sulphate minerals are present as well.

9 Exploration

AZ has been active on the Property since 2006. A re-assay program of all remaining ASARCO assay pulps verified the silver and manganese assay data and added high-quality Pb, Zn, Cu, and Au values to the database. Rock types, alteration, and mineral codes from paper drill logs and cross sections were added to the electronic assay database. All available ASARCO drill assays and supplemental 16 element X-ray fluorescence analyses were captured electronically as well. Preliminary SO_2 leach tests were run on two composite samples of assay pulps at Hazen Laboratories. A Mineral Resource estimate and preliminary economic evaluation was included in a 7 February 2007 Preliminary Economic Assessment report written by Pincock, Allen, and Holt.

A mapping program at a 1:50 (metric) scale is in progress. The primary focus of the program is to generate outcrop and structure maps, on the newly acquired Trench claims and the surrounding unpatented lode mining claims, in close proximity to the Taylor Sulphide Deposit. An emphasis is being placed on mapping structures and trying to identify any post-mineral faults or mineralizing controls that can be used to generate blind drill targets.

10 Drilling

10.1 Introduction

The following discussion is an overview of the drill programs conducted on the Property by AZ. Drilling carried out by ASARCO is discussed in Section 6.2. The drilling completed by AZ was initially focused on the Manto (oxide) of the Central Deposit (2006-2014) and subsequently on the Taylor Deposit (2010-present) to determine the extents of the CRD mineralization.

10.2 Drilling summary

Drill programs conducted by AZ on the Property between 2007 and 2017 are summarized in Table 10.1.

Table 10.1 AZ drill programs

Year	Type	Number	Length (ft)	Length (m)	Target	Resource statement
2007	Core	4	4,450	1,356	Central Deposit	Pre-2016
2007 & 2008	Core	3	7,928	2,416	Central Deposit	Pre-2016
2009	Core	6	12,005	3,659	Central Deposit	Pre-2016
2010 -2012	Core	57	81,846	24,947	Central Deposit	Pre-2016
2012	RC	6	2,480	756	Central Deposit	Pre-2016
2010 -2012	RC	159	101,813	31,033	Central Deposit	Pre-2016
2007 - 2012	Core	16	32,846	10,011	Taylor Deposit	Feb 1, 2016
2014 - 2015	Core	8	29,337	8,942	Taylor Deposit	Feb 1, 2016
2016	Core	35	144,010	43,894	Taylor Deposit	Oct 31, 2016
2016-2017	Core	37	151,483	46,172	Taylor Deposit	PEA - March 29, 2017
2017	Core	65	234,514	71,480	Taylor Deposit	PEA Update - November 2017
Total		**396**	**802,712**	**244,666**		

The objective of the drill programs has evolved over time. The programs carried out between 2007 and 2012 were designed to assess and define the near-surface, silver-manganese oxide mineralization that was historically referred to as the Hardshell deposit. Drilling since 2014 has focused on the sulphide mineralization that is located stratigraphically below and down-dip of the oxide mineralization and that forms the basis of the current and previous resource estimates.

The sulphide mineralization (Taylor Deposit) is a carbonate replacement type deposit of lead, zinc, and silver with subordinate copper content. Manganese is generally present as a carbonate or silicate in similar content to the oxide deposit. The sulphide mineralization occurs both as stratiform (manto) bodies that dip at generally less than 30°, and as steep-dipping, crosscutting bodies (chimneys). The manto-type mineralization is generally constrained within the host carbonate units; the chimney-type mineralization cuts across formational boundaries. The near-surface manganese-silver oxide mineralization (Central Deposit) has been interpreted to be a stratiform (manto) type replacement-style deposit that may be the oxidized upper portion of the deeper sulphide deposit.

In addition to the carbonate-hosted mineralization, lead-zinc-silver-copper sulphide mineralization occurs at the tectonized contact between underlying carbonates and overlying volcanic rocks. Another minor type of mineralization that has been identified occurs within agglomerate horizons in the Hardshell Volcanics.

10.3 Taylor Deposit

10.3.1 Procedures

Only the procedures for the drill core collected in 2016-2017 are discussed here as only diamond drilling has been carried out in that period. Procedures for earlier programs including RC programs are discussed in the March 2016 Technical Report.

Drill core is washed by the drill helper and transferred from core barrel to the core box. Core is collected from the rig by an AZ field helper and brought to the on-site facility by an AZ truck where it is washed, photographed, logged and sampled. Drill core is cut lengthwise by a 5 hp diamond saw using a 14 in diamond impregnated blade. Typical sample intervals lengths were nominally set at 5 ft (1.5 m). In areas of mineralogical or geological interest, sample intervals range from one to seven feet in order to honor lithological boundaries.

After a sample is cut, one half core was returned to the original core box for reference and long-term storage. The remaining half core was placed in a heavy gauge plastic bag marked with drillhole number and interval labels. Duplicate samples were collected for QA / QC purposes by cutting the half core into two quarters; these samples were collected at the same time as the normal samples and the resultant assay results were treated as if the underlying sample was a full half-core. The sample bags were closed with a wire tie, weighed and consolidated in shipping boxes or bulk shipping bags. They are transported by ALS Minerals to their laboratory in Tucson Arizona for sample preparation and analysis.

Drill collars are preserved with a 10 ft (3 m) section of drill steel with a steel cap and cemented in place. The drillhole number is inscribed in the metal cap for identification (Figure 10.1). Collar coordinates are surveyed by a licensed Arizona registered land surveyor. Collar locations are recorded using the Arizona State Plane coordinate system.

Figure 10.1 Drill collar for drillhole HDS-475

There are no drilling, sampling, or recovery factors identified that materially impact the accuracy and reliability of the results.

10.3.2 Relevant sample results

Figure 10.2 shows in plan, the location of drillholes that have intersected sulphide mineralization, and Table 10.2 contains some of the relevant drillhole intercepts of sulphide mineralization from the 2016 and 2017 program. Much of the mineralization is stratabound and dips at less than 30° although some is steep-dipping to vertical. Most of the drillholes are vertical although some drillholes are inclined at dip angles between approximately 70° to 80°. This combination of variable dips of mineralization and variable drillhole dips means that most of the drillhole intercepts are greater than true thickness although it is not possible to accurately determine this variance.

Grades of mineralization vary significantly throughout the deposit; however, transitions from higher to lower grades appear to be generally gradual and does not juxtapose mineralization of highly contrasting tenor and therefore does not represent a significant risk that during the resource estimation process higher grades will be inappropriately assigned to blocks that are located in areas of lower grade mineralization.

Measurements on core from the 2010 - 2017 drilling campaigns show the average core recovery was 91.5%. The rate of recovery from the 2017 drill program was 92%.

Figure 10.2 Locations of drillholes intersecting sulphide mineralization

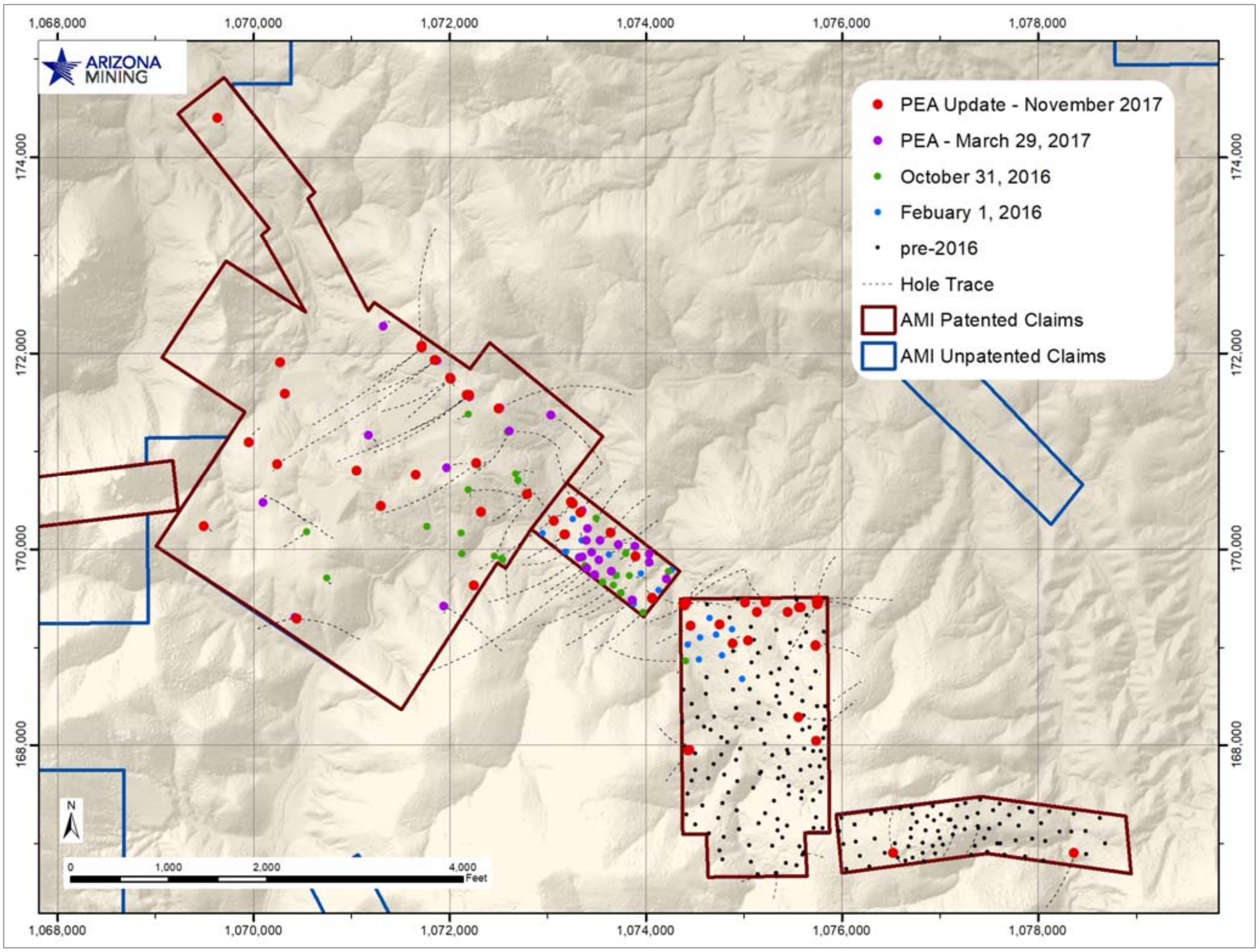


Table 10.2 Taylor Deposit 2016-2017 CRD drilling results summary

DH_ID	From (ft)	To (ft)	Interval (ft)	From (m)	To (m)	Interval (m)	Zn (%)	Pb (%)	Ag (oz/ton)	Cu (%)	Zone
HDS-343	3493	3495.5	2.5	1064.6	1065.4	0.8	0.80	8.59	28.06	1.76	TVS
HDS-343	3597	3673	76	1096.3	1119.5	23.2	1.11	1.95	2.46	0.13	TDS
Including	3664	3673	9	1116.7	1119.5	2.7	6.24	9.89	15.58	0.87	TDS
HDS-343	4946	4970	24	1507.5	1514.8	7.3	0.02	0.03	2.61	3.08	TDS
HDS-493	No significant mineralization										
HDS-394	3237	3244	7	986.6	988.7	2.1	1.08	0.71	6.20	0.33	TVS
HDS-394	3821.5	3846	24.5	1164.7	1172.2	7.5	2.70	2.40	2.34	0.16	TVS
HDS-394	3920.5	3985	64.5	1194.9	1214.6	19.7	0.45	2.75	4.88	0.02	TDS
HDS-394	4315	4330	15	1315.1	1319.7	4.6	1.80	1.36	0.54	0.02	TDS
HDS-395	1360	1367	7	414.5	416.6	2.1	3.35	1.23	4.17	0.08	TVS
HDS-395	1472	1490.5	18.5	448.6	454.3	5.6	3.25	3.70	1.61	0.08	TVS
HDS-395	5086.5	5089	2.5	1550.3	1551.1	0.8	13.60	2.91	1.92	0.67	TDS
HDS-410	3301	3317	16	1006.1	1011.0	4.9	0.75	6.25	27.82	1.55	TVS
HDS-410	4244	4253.5	9.5	1293.5	1296.4	2.9	1.03	2.32	1.29	0.59	TVS
HDS-412	671	683	12	204.5	208.2	3.7	2.38	1.38	2.88	0.09	TVS
HDS-412	722.5	746.5	24	220.2	227.5	7.3	5.42	3.10	2.61	0.02	TVS
HDS-412	824	829	5	251.1	252.7	1.5	5.56	1.65	6.15	0.23	TVS
HDS-412	929	950	21	283.1	289.5	6.4	3.89	1.67	6.06	0.37	TVS
HDS-412	3504	3511	7	1068.0	1070.1	2.1	0.07	2.59	8.69	0.04	TVS
HDS-414	370	376	6	112.8	114.6	1.8	1.09	0.20	3.53	0.02	TVS
HDS-414	1393.5	1400	6.5	424.7	426.7	2.0	9.77	4.44	3.53	0.05	TVS
HDS-414	1657	1662	5	505.0	506.6	1.5	0.74	5.04	2.91	0.03	TVS
HDS-414	3811.5	3902	90.5	1161.7	1189.3	27.6	0.35	1.53	2.74	0.07	TDS
HDS-415	No significant mineralization										
HDS-416	406	412	6	123.7	125.6	1.8	1.69	6.25	5.28	0.06	TVS
HDS-416	2439	2485	46	743.4	757.4	14.0	3.00	2.41	0.93	0.04	TS
HDS-416	2595	2658	63	790.9	810.1	19.2	2.05	2.10	0.82	0.06	TS
HDS-416	2723	2728	5	829.9	831.5	1.5	5.14	3.92	1.65	0.11	TS
HDS-416	2751	2767	16	838.5	843.3	4.9	3.72	7.18	2.55	0.06	TS
HDS-416	2826	2862	36	861.3	872.3	11.0	6.44	8.22	3.14	0.15	TS
Including	2849	2862	13	868.3	872.3	4.0	8.89	11.84	5.21	0.28	TS
HDS-416	2882	3002	120	878.4	915.0	36.6	3.35	3.29	2.97	0.10	TDS
Including	2913	2941	28	887.8	896.4	8.5	10.33	11.20	6.39	0.27	TDS
HDS-419	1697	1707	10	517.2	520.3	3.0	1.64	5.83	5.69	0.19	TVS
HDS-419	2192	2214	22	668.1	674.8	6.7	10.94	11.32	5.69	0.24	TS
HDS-419	2254	2278	24	687.0	694.3	7.3	1.23	1.53	0.53	0.02	TS
HDS-419	2387	2399	12	727.5	731.2	3.7	2.23	1.64	0.51	0.02	TS
HDS-419	2558.5	2672	113.5	779.8	814.4	34.6	3.24	3.20	0.93	0.02	TS
HDS-419	2697	2737	40	822.0	834.2	12.2	5.96	5.02	1.46	0.04	TS
Including	2697	2707	10	822.0	825.1	3.0	10.03	8.04	2.34	0.12	TS
HDS-419	2762	2858	96	841.8	871.1	29.3	1.22	1.13	0.34	0.01	TS
HDS-419	2967	3007	40	904.3	916.5	12.2	2.15	3.88	4.22	0.15	TS
HDS-419	3034	3036.5	2.5	924.7	925.5	0.8	19.65	13.80	38.21	2.01	TS
HDS-419	3252.5	3255.5	3	991.3	992.2	0.9	0.09	10.90	4.11	0.27	TVS
HDS-419	3333.5	3384	50.5	1016.0	1031.4	15.4	1.49	4.54	1.59	0.12	TDS

DH_ID	From (ft)	To (ft)	Interval (ft)	From (m)	To (m)	Interval (m)	Zn (%)	Pb (%)	Ag (oz/ton)	Cu (%)	Zone
HDS-419	3411	3424	13	1039.6	1043.6	4.0	3.63	2.28	0.87	0.18	TDS
HDS-420	1157	1167	10	352.6	355.7	3.0	2.14	0.92	1.10	0.13	TVS
HDS-420	1472	1526.5	54.5	448.6	465.3	16.6	2.03	0.91	1.91	0.10	TVS
HDS-420	2276.5	2291.5	15	693.8	698.4	4.6	3.12	2.49	1.40	0.40	TS
HDS-420	2911.5	2931	19.5	887.4	893.3	5.9	4.11	2.42	0.91	0.01	TS
HDS-421	403	407	4	122.8	124.0	1.2	1.49	2.43	10.18	0.53	TVS
HDS-421	445.5	448.5	3	135.8	136.7	0.9	0.55	5.47	4.00	0.11	TVS
HDS-421	1347	1404.5	57.5	410.5	428.1	17.5	18.03	10.36	4.78	0.25	TS
HDS-421	1954	1993.5	39.5	595.6	607.6	12.0	9.24	4.97	2.11	0.17	TS
HDS-421	2069.5	2091.5	22	630.8	637.5	6.7	5.63	2.23	0.77	0.02	TS
HDS-421	2749.5	2794.5	45	838.0	851.7	13.7	3.71	3.37	1.05	0.02	TS
HDS-421	2887	2912	25	879.9	887.5	7.6	4.26	6.87	2.34	0.01	TS
HDS-421	2937	3019.5	82.5	895.2	920.3	25.1	0.96	1.92	0.67	0.01	TS
HDS-421	3234	3268	34	985.7	996.0	10.4	1.60	2.77	3.10	0.20	TVS
HDS-421	3366	3462	96	1025.9	1055.2	29.3	3.88	5.54	4.50	0.35	TDS
Including	3373	3395	22	1028.0	1034.7	6.7	8.81	16.59	7.10	0.93	TDS
Including	3452.5	3462	9.5	1052.3	1055.2	2.9	13.35	10.12	26.50	1.22	TDS
HDS-422B	1062	1067	5	323.7	325.2	1.5	7.63	3.64	2.65	0.06	TVS
HDS-422B	1785	1841	56	544.0	561.1	17.1	4.41	4.04	1.41	0.06	TS
HDS-422B	2592	2622	30	790.0	799.1	9.1	3.81	3.20	0.92	0.09	TS
HDS-422B	2766	2773	7	843.0	845.2	2.1	14.90	11.80	4.43	0.07	TS
HDS-422B	2842	2927	85	866.2	892.1	25.9	2.38	3.33	1.02	0.01	TS
Including	2857.5	2882	24.5	870.9	878.4	7.5	4.88	7.54	2.29	0.01	TS
HDS-422B	2957	2995	38	901.2	912.8	11.6	0.47	1.48	0.49	0.00	TS
HDS-422B	3044	3116	72	927.8	949.7	21.9	0.90	2.52	1.42	0.08	TS
HDS-422B	3162	3165	3	963.7	964.6	0.9	2.35	1.62	10.18	0.47	TVS
HDS-422B	3212	3217	5	979.0	980.5	1.5	2.64	1.39	3.18	0.11	TVS
HDS-422B	3332.5	3413.5	81	1015.7	1040.4	24.7	7.01	10.22	4.26	0.39	TDS
Including	3332.5	3354	21.5	1015.7	1022.2	6.6	17.12	22.44	9.17	0.76	TDS
HDS-423	964	982	18	293.8	299.3	5.5	1.22	1.10	5.69	0.32	TVS
HDS-423	1208	1235	27	368.2	376.4	8.2	2.38	2.12	2.06	0.03	TVS
HDS-423	1867	1872	5	569.0	570.6	1.5	1.37	4.46	3.27	0.02	TVS
HDS-423	3865.5	3873.5	8	1178.1	1180.6	2.4	1.81	1.89	5.03	0.43	TVS
HDS-423	4206	4227	21	1281.9	1288.3	6.4	0.59	0.95	1.38	0.03	TDS
HDS-423	4451	4470.5	19.5	1356.6	1362.5	5.9	1.49	0.86	0.70	0.13	TDS
HDS-424	2184.5	2211	26.5	665.8	673.9	8.1	4.23	2.67	1.06	0.09	TVS
HDS-424	2242	2248	6	683.3	685.2	1.8	9.42	10.85	6.45	0.87	TVS
HDS-424	3543	3545.5	2.5	1079.9	1080.6	0.8	1.64	8.08	3.91	0.28	TVS
HDS-424	3568.5	3571	2.5	1087.6	1088.4	0.8	0.24	10.65	3.47	0.17	TVS
HDS-424	3653	3681	28	1113.4	1121.9	8.5	2.34	6.96	3.11	0.14	TDS
HDS-424	3726	3796	70	1135.6	1157.0	21.3	1.05	1.21	0.40	0.10	TDS
HDS-424	3821	3876	55	1164.6	1181.3	16.8	2.48	2.21	0.85	0.42	TDS
HDS-424	3946	3991	45	1202.7	1216.4	13.7	1.23	1.03	0.26	0.05	TDS
HDS-425	1755	1761	6	534.9	536.7	1.8	1.48	1.06	4.23	0.28	TVS
HDS-425	2037	2060	23	620.8	627.9	7.0	2.91	2.71	1.44	0.08	TVS
HDS-425	2242	2273	31	683.3	692.8	9.4	6.47	6.12	5.57	0.22	TVS

DH_ID	From (ft)	To (ft)	Interval (ft)	From (m)	To (m)	Interval (m)	Zn (%)	Pb (%)	Ag (oz/ton)	Cu (%)	Zone
HDS-425	2295	2420	125	699.5	737.6	38.1	1.74	13.99	8.20	0.12	TS
Including	2351	2391	40	716.5	728.7	12.2	0.09	36.72	19.72	0.05	TS
HDS-425	2459	2487	28	749.5	758.0	8.5	1.38	0.96	1.54	0.08	TVS
HDS-425	3287	3327	40	1001.8	1014.0	12.2	1.41	4.38	2.73	0.10	TDS
HDS-425	3357	3374	17	1023.2	1028.3	5.2	4.72	6.81	2.57	0.12	TDS
HDS-426	No significant mineralization										
HDS-427	No significant mineralization										
HDS-428	874	883.5	9.5	266.4	269.3	2.9	3.64	1.24	3.14	0.01	TVS
HDS-428	2052	2059	7	625.4	627.6	2.1	0.49	2.03	3.97	0.25	TVS
HDS-428	4946.5	4975.5	29	1507.6	1516.5	8.8	1.52	1.78	2.25	0.19	TDS
HDS-429	642	647	5	195.7	197.2	1.5	6.27	4.10	3.70	0.04	TVS
HDS-429	1850	1856.5	6.5	563.9	565.8	2.0	1.54	7.27	4.32	0.07	TVS
HDS-430	1725	1752	27	525.8	534.0	8.2	2.70	3.21	1.33	0.10	TS
HDS-430	1789	1795	6	545.3	547.1	1.8	10.40	3.71	1.53	0.14	TVS
HDS-430	2497	2517	20	761.0	767.1	6.1	4.26	3.87	1.33	0.06	TS
HDS-430	2625	2743	118	800.1	836.0	36.0	1.39	1.15	0.40	0.01	TS
HDS-430	3002	3084	82	915.0	940.0	25.0	2.31	3.84	2.53	0.15	TDS
Including	3032	3047	15	924.1	928.7	4.6	8.33	12.56	5.41	0.48	TDS
HDS-431	1237	1258	21	377.0	383.4	6.4	8.33	6.08	4.06	0.08	TVS
HDS-431	1892	1902	10	576.7	579.7	3.0	2.15	1.25	5.25	0.24	TS
HDS-431	1987	1992	5	605.6	607.1	1.5	1.35	0.37	13.88	0.30	TVS
HDS-431	2105.5	2185	79.5	641.7	666.0	24.2	2.15	1.34	1.26	0.09	TS
HDS-431	2245	2270	25	684.2	691.9	7.6	1.15	1.23	0.47	0.05	TS
HDS-431	2488	2490.5	2.5	758.3	759.1	0.8	4.13	27.17	25.61	0.95	TVS
HDS-431	2559	2672.5	113.5	779.9	814.5	34.6	7.92	11.41	3.34	0.06	TS
Including	2559	2602	43	779.9	793.1	13.1	19.70	24.60	7.05	0.14	TS
HDS-431	2702	2760	58	823.5	841.2	17.7	4.07	3.95	1.19	0.01	TS
HDS-431	2859.5	2907	47.5	871.5	886.0	14.5	1.16	1.42	0.46	0.01	TS
HDS-431	3133	3137.5	4.5	954.9	956.3	1.4	6.13	7.15	5.37	0.33	TVS
HDS-431	3179.5	3182	2.5	969.1	969.8	0.8	0.57	23.07	8.14	0.42	TVS
HDS-431	3274	3297	23	997.9	1004.9	7.0	3.62	8.10	3.11	0.52	TDS
HDS-432	No significant mineralization										
HDS-433	No significant mineralization										
HDS-434	755	768	13	230.1	234.1	4.0	2.51	2.11	2.21	0.04	TVS
HDS-434	2046	2051	5	623.6	625.1	1.5	3.81	2.06	11.32	1.17	TVS
HDS-434	2206.5	2239	32.5	672.5	682.4	9.9	2.04	1.45	0.94	0.10	TS
HDS-434	3515	3656	141	1071.3	1114.3	43.0	4.00	6.36	2.16	0.28	TDS
Including	3515	3552	37	1071.3	1082.6	11.3	11.43	20.00	6.77	0.97	TDS
HDS-434	3843.5	3863	19.5	1171.4	1177.4	5.9	1.73	2.27	0.66	0.21	TVS
HDS-435	2121	2182	61	646.4	665.0	18.6	16.47	13.81	9.84	0.03	TDS
Including	2121	2152	31	646.4	655.9	9.4	27.25	22.36	14.85	0.04	TDS
HDS-436	3444	3449	5	1049.7	1051.2	1.5	0.14	2.25	4.64	0.54	TVS
HDS-436	3485.5	3488	2.5	1062.3	1063.1	0.8	0.81	1.50	2.33	0.98	TVS
HDS-436	3670	3676	6	1118.6	1120.4	1.8	0.23	0.97	2.08	5.42	TVS
HDS-436	3695.5	3753	57.5	1126.3	1143.9	17.5	3.30	2.31	2.89	0.29	TDS
HDS-436	3821	3824.5	3.5	1164.6	1165.7	1.1	7.93	4.07	2.80	0.98	TVS

DH_ID	From (ft)	To (ft)	Interval (ft)	From (m)	To (m)	Interval (m)	Zn (%)	Pb (%)	Ag (oz/ton)	Cu (%)	Zone
HDS-436	3864	3893	29	1177.7	1186.5	8.8	4.86	7.37	6.29	0.69	TDS
Including	3884.5	3893	8.5	1183.9	1186.5	2.6	13.16	22.90	20.26	2.09	TDS
HDS-436	3948.5	4049	100.5	1203.4	1234.1	30.6	1.65	2.52	2.85	0.10	TDS
Including	4032.5	4045	12.5	1229.0	1232.9	3.8	9.87	12.25	6.55	0.48	TDS
HDS-437	2077	2087	10	633.0	636.1	3.0	2.34	2.56	2.58	0.01	TS
HDS-437	2367	2460	93	721.4	749.8	28.3	3.85	3.31	1.17	0.17	TS
HDS-437	2537	2571	34	773.2	783.6	10.4	14.84	13.17	4.33	0.09	TS
HDS-437	2602	2619	17	793.1	798.2	5.2	6.90	5.49	2.16	0.16	TS
HDS-437	2734	2737	3	833.3	834.2	0.9	17.20	17.45	5.37	0.02	TVS
HDS-437	3035	3157	122	925.0	962.2	37.2	6.40	6.47	10.73	0.60	TDS
Including	3120	3147	27	950.9	959.2	8.2	19.82	13.32	41.17	2.51	TDS
HDS-438	967	1022.5	55.5	294.7	311.6	16.9	1.53	0.72	1.13	0.04	TVS
HDS-438	1047	1068	21	319.1	325.5	6.4	1.74	1.68	7.34	0.40	TVS
HDS-438	1086	1112	26	331.0	338.9	7.9	1.28	0.36	1.47	0.05	TVS
HDS-438	1163	1169.5	6.5	354.5	356.4	2.0	0.80	1.57	4.40	0.20	TVS
HDS-438	1211	1265.5	54.5	369.1	385.7	16.6	5.40	1.78	1.39	0.03	TVS
Including	1211	1235	24	369.1	376.4	7.3	11.74	3.71	2.24	0.01	TVS
HDS-438	1303	1309	6	397.1	399.0	1.8	5.33	28.06	18.46	0.24	TVS
HDS-438	1390	1476	86	423.7	449.9	26.2	14.45	8.74	7.18	0.36	TVS
HDS-439	No significant mineralization										
HDS-440	2060	2128	68	627.9	648.6	20.7	0.30	0.11	3.45	0.04	TVS
HDS-440	2172	2180	8	662.0	664.4	2.4	0.13	0.11	7.89	0.08	TVS
HDS-441	864	910.5	46.5	263.3	277.5	14.2	1.86	0.89	2.16	0.06	TVS
HDS-441	1195	1197	2	364.2	364.8	0.6	7.51	4.84	1.35	0.06	TVS
HDS-441	1458.5	1461	2.5	444.5	445.3	0.8	12.25	8.54	9.16	0.57	TVS
HDS-441	1501	1505.5	4.5	457.5	458.9	1.4	1.21	2.03	9.28	0.44	TVS
HDS-441	1619.5	1656	36.5	493.6	504.7	11.1	1.31	1.11	4.34	0.26	TVS
HDS-441	1734.5	1787	52.5	528.6	544.7	16.0	2.07	5.06	2.84	0.09	TVS
HDS-441	1825	1914	89	556.2	583.4	27.1	4.31	5.49	3.67	0.10	TVS
Including	1825	1860	35	556.2	566.9	10.7	9.46	11.80	6.19	0.09	TVS
HDS-441	2210	2234	24	673.6	680.9	7.3	1.76	1.21	1.01	0.04	TVS
HDS-442	1426.5	1481	54.5	434.8	451.4	16.6	3.87	2.34	1.36	0.05	TVS
HDS-443	1050	1055	5	1.5	0.5	0.1	0.86	1.30	1.25	0.05	TVS
HDS-443	1206.5	1215	8.5	2.6	0.8	0.2	0.76	0.36	0.84	0.03	TVS
HDS-444	612	622	10	186.5	189.6	3.0	1.95	0.69	1.82	0.07	TVS
HDS-444	772	780	8	235.3	237.7	2.4	2.92	2.29	1.18	0.07	TVS
HDS-444	1202	1217	15	366.4	370.9	4.6	0.70	0.57	2.04	0.09	TVS
HDS-444	2267	2351	84	690.9	716.5	25.6	1.42	0.65	0.44	0.04	TVS
HDS-444	2592	2602	10	790.0	793.1	3.0	2.53	1.23	7.98	0.55	TVS
HDS-444	2635	2656	21	803.1	809.5	6.4	1.53	2.05	2.00	0.18	TVS
HDS-445	759.5	770.5	11	231.5	234.8	3.4	0.08	0.01	0.98	0.01	TVS
HDS-445	1827	1837	10	556.8	559.9	3.0	1.13	0.30	0.35	0.03	TVS
HDS-446	2393	2455	62	729.4	748.2	18.9	2.89	2.05	1.58	0.26	TS
HDS-446	2656	2661	5	809.5	811.0	1.5	10.45	11.10	4.35	0.10	TS
HDS-446	2701	2768	67	823.2	843.6	20.4	20.46	18.09	7.48	1.25	TDS
HDS-447	470	485	15	143.2	147.8	4.6	0.07	1.54	2.13	0.08	TVS

DH_ID	From (ft)	To (ft)	Interval (ft)	From (m)	To (m)	Interval (m)	Zn (%)	Pb (%)	Ag (oz/ton)	Cu (%)	Zone
HDS-447	520	545	25	158.5	166.1	7.6	0.02	0.16	2.83	0.01	TVS
HDS-447	2152	2213	61	655.9	674.5	18.6	2.35	1.80	1.97	0.10	TS
HDS-447	2237	2247	10	681.8	684.9	3.0	3.12	6.14	3.54	0.07	TS
HDS-447	2307	2456	149	703.1	748.6	45.4	4.66	6.21	4.29	0.11	TDS
Including	2307	2350	43	703.1	716.2	13.1	11.42	18.59	12.33	0.33	TDS
HDS-447	2683	2688	5	817.7	819.3	1.5	13.35	5.44	14.00	1.29	TVS
HDS-448	1540	1542.5	2.5	469.4	470.1	0.8	4.69	2.44	1.23	0.06	TVS
HDS-448	1719.5	1730	10.5	524.1	527.3	3.2	0.44	2.47	1.68	0.08	TVS
HDS-448	1792	1815	23	546.2	553.2	7.0	1.33	0.93	1.62	0.09	TS
HDS-448	1910	2000	90	582.1	609.6	27.4	1.57	0.95	0.81	0.03	TS
HDS-448	2215	2240	25	675.1	682.7	7.6	6.71	5.17	1.62	0.02	TS
HDS-448	2295	2352.5	57.5	699.5	717.0	17.5	3.93	2.83	4.00	0.24	TS
HDS-448	2495.5	2513	17.5	760.6	765.9	5.3	7.36	7.12	2.21	0.11	TS
HDS-449	873	881	8	266.1	268.5	2.4	4.45	2.07	1.76	0.11	TVS
HDS-449	2330	2468	138	710.1	752.2	42.1	7.37	9.28	3.28	0.15	TS
Including	2353	2415	62	717.2	736.1	18.9	14.14	18.39	5.82	0.31	TS
HDS-449	2666	2670	4	812.6	813.8	1.2	3.52	2.82	3.76	0.16	TVS
HDS-449	3344	3362.5	18.5	1019.2	1024.8	5.6	2.26	2.65	3.49	0.19	TVS
HDS-449	3418	3441	23	1041.8	1048.8	7.0	1.60	2.72	0.84	0.01	TDS
HDS-449	3699.5	3707.5	8	1127.6	1130.0	2.4	2.77	1.61	2.50	0.27	TVS
HDS-450	2178	2207	29	663.8	672.7	8.8	0.63	0.54	1.44	0.08	TS
HDS-450	2454	2537	83	747.9	773.2	25.3	3.53	3.85	2.17	0.07	TS
HDS-450	2543	2588.5	45.5	775.1	788.9	13.9	11.99	11.81	4.84	0.24	TDS
HDS-450	3002	3012	10	915.0	918.0	3.0	1.50	1.22	5.54	0.36	TVS
HDS-451	456	460	4	139.0	140.2	1.2	2.82	1.32	9.28	0.04	TVS
HDS-451	889.5	893.5	4	271.1	272.3	1.2	4.61	1.23	1.90	0.02	TVS
HDS-451	1135	1192	57	345.9	363.3	17.4	1.18	0.47	0.34	0.00	TVS
HDS-452	2367	2395	28	721.4	730.0	8.5	2.04	1.48	1.26	0.13	TS
HDS-452	2517	2531	14	767.1	771.4	4.3	1.00	0.98	0.78	0.11	TS
HDS-452	2612	2635.5	23.5	796.1	803.3	7.2	1.15	1.34	0.89	0.18	TS
HDS-452	2747	2852	105	837.2	869.2	32.0	1.65	1.51	1.23	0.12	TS
Including	2835	2852	17	864.1	869.2	5.2	4.71	4.67	3.77	0.44	TS
HDS-452	2892	2929	37	881.4	892.7	11.3	15.15	11.29	10.19	1.47	TDS
HDS-453	1883	1897.5	14.5	573.9	578.3	4.4	2.08	1.80	3.03	0.14	TVS
HDS-453	2115.5	2118	2.5	644.8	645.5	0.8	2.29	3.75	10.88	1.09	TVS
HDS-453	2607	2612	5	794.6	796.1	1.5	0.08	4.70	2.31	0.01	TVS
HDS-453	3378	3391.5	13.5	1029.6	1033.7	4.1	1.43	3.13	1.32	0.33	TVS
HDS-453	3470.5	3540.5	70	1057.8	1079.1	21.3	7.21	19.21	6.41	0.32	TDS
Including	3480	3507	27	1060.7	1068.9	8.2	8.32	29.77	9.46	0.36	TDS
HDS-454	849	852	3	258.8	259.7	0.9	4.03	1.68	1.21	0.08	TVS
HDS-454	901	908	7	274.6	276.7	2.1	3.74	1.51	0.84	0.03	TVS
HDS-454	2497	2502	5	761.0	762.6	1.5	1.20	6.22	2.55	0.03	TVS
HDS-455	2560	2610	50	780.2	795.5	15.2	5.39	3.05	1.29	0.09	TS
HDS-455	2652	2675	23	808.3	815.3	7.0	1.40	1.59	0.63	0.02	TS
HDS-455	2705	2710	5	824.4	826.0	1.5	2.67	7.96	2.39	0.04	TVS
HDS-455	2858.5	2947	88.5	871.2	898.2	27.0	1.72	2.18	1.21	0.07	TDS

DH_ID	From (ft)	To (ft)	Interval (ft)	From (m)	To (m)	Interval (m)	Zn (%)	Pb (%)	Ag (oz/ton)	Cu (%)	Zone
HDS-456	2457	2482	25	748.9	756.5	7.6	2.17	1.11	1.29	0.01	TDS
HDS-457	1333	1338	5	406.3	407.8	1.5	2.99	1.53	1.66	0.03	TVS
HDS-457	1421	1427	6	433.1	434.9	1.8	5.17	2.68	1.22	0.03	TVS
HDS-458	144	146.5	2.5	43.9	44.7	0.8	3.16	2.93	1.37	0.02	TVS
HDS-458	206	216	10	62.8	65.8	3.0	1.48	1.79	1.87	0.03	TVS
HDS-459	1414	1450.5	36.5	431.0	442.1	11.1	10.18	6.95	3.11	0.10	TS
HDS-459	1497	1512	15	456.3	460.8	4.6	2.93	2.18	2.84	0.53	TS
HDS-459	2255.5	2325.5	70	687.4	708.8	21.3	2.55	1.81	0.84	0.27	TS
Including	2283.5	2305	21.5	696.0	702.5	6.6	5.08	3.19	1.46	0.39	TS
HDS-459	2419	2482.5	63.5	737.3	756.6	19.4	1.54	1.64	0.64	0.06	TS
HDS-459	2798	2802	4	852.8	854.0	1.2	0.99	1.77	7.99	0.08	TS
HDS-459	2837	2872	35	864.7	875.3	10.7	10.79	13.25	21.00	1.15	TDS
HDS-461	2302	2362	60	701.6	719.9	18.3	1.88	1.35	0.76	0.18	TS
HDS-461	2437	2470	33	742.8	752.8	10.1	2.13	2.15	0.88	0.07	TS
HDS-461	2628	2684	56	801.0	818.0	17.1	0.70	1.65	0.60	0.01	TS
HDS-461	2752.5	2792	39.5	838.9	851.0	12.0	1.48	4.58	1.84	0.04	TDS
HDS-462	1061	1067	6	323.4	325.2	1.8	4.11	1.39	1.55	0.01	TVS
HDS-462	1407	1418	11	428.8	432.2	3.4	0.92	1.34	1.39	0.06	TVS
HDS-463	2303	2315	12	701.9	705.6	3.7	4.72	2.90	3.66	0.25	TS
HDS-463	2427	2521	94	739.7	768.4	28.6	0.82	0.94	1.75	0.06	TS
HDS-463	2572	2587	15	783.9	788.5	4.6	2.02	1.62	1.14	0.04	TDS
HDS-464	2525	2544	19	769.6	775.4	5.8	5.32	2.53	2.75	0.05	TDS
HDS-465	784	788	4	239.0	240.2	1.2	3.17	1.58	10.94	0.31	TSV
HDS-465	1058.5	1076	17.5	322.6	327.9	5.3	1.59	0.64	4.53	0.20	TSV
HDS-466	945	995	50	288.0	303.3	15.2	1.84	0.85	1.51	0.07	TVS
HDS-466	1370	1381	11	417.6	420.9	3.4	0.74	3.14	1.08	0.03	TVS
HDS-467	735	740	5	224.0	225.5	1.5	0.15	13.20	16.13	0.54	TSV
HDS-467	2729	2734	5	831.8	833.3	1.5	2.47	2.86	1.95	0.10	TDS
HDS-467	2796	2802	6	852.2	854.0	1.8	1.61	1.10	10.15	0.16	TDS
HDS-469	2317	2328	11	706.2	709.5	3.4	3.10	2.69	1.89	0.13	TS
HDS-469	2441	2452	11	744.0	747.3	3.4	2.79	1.63	0.74	0.03	TS
HDS-470	2476	2481	5	754.6	756.2	1.5	14.20	7.78	5.45	0.02	TS
HDS-470	2560	2587	27	780.2	788.5	8.2	5.67	1.56	4.79	0.05	TDS
HDS-471	2416	2427	11	736.4	739.7	3.4	2.42	1.64	0.73	0.10	TS
HDS-471	2674	2732	58	815.0	832.7	17.7	4.37	6.65	2.44	0.26	TDS
HDS-472	2311.5	2345	33.5	704.5	714.7	10.2	2.00	1.15	1.54	0.18	TS
HDS-472	2531	2557	26	771.4	779.3	7.9	1.99	2.47	1.05	0.09	TS
HDS-472	2595	2673	78	790.9	814.7	23.8	2.36	2.29	1.20	0.07	TS
HDS-472	2691	2727	36	820.2	831.1	11.0	6.70	4.62	1.97	0.57	TDS
HDS-473	617	631	14	188.1	192.3	4.3	2.14	1.08	0.95	0.07	TSV
HDS-473	1145.5	1148	2.5	349.1	349.9	0.8	9.69	12.70	8.20	0.28	TSV
HDS-473	1822	1882	60	555.3	573.6	18.3	0.89	1.14	1.38	0.07	TSV
HDS-473	1925.5	1933	7.5	586.9	589.1	2.3	1.90	1.12	2.84	0.14	TSV
HDS-473	3651	3670.5	19.5	1112.8	1118.7	5.9	2.38	22.23	21.67	0.75	TDS
HDS-473	3823	3887	64	1165.2	1184.7	19.5	1.66	1.93	0.93	0.03	TDS
HDS-474	686	691.5	5.5	209.1	210.8	1.7	3.36	6.57	6.46	0.38	TVS

DH_ID	From (ft)	To (ft)	Interval (ft)	From (m)	To (m)	Interval (m)	Zn (%)	Pb (%)	Ag (oz/ton)	Cu (%)	Zone
HDS-474	2357	2717	360	718.4	828.1	109.7	5.01	4.98	1.65	0.07	TS
Including	2447	2566	119	745.8	782.1	36.3	10.38	10.87	3.50	0.13	TS
HDS-474	2931	2972	41	893.3	905.8	12.5	7.23	14.97	6.09	0.25	TDS
HDS-474	3103	3175	72	945.7	967.7	21.9	7.72	5.08	11.94	0.94	TDS
HDS-477	2303	2362	59	701.9	719.9	18.0	5.17	3.29	4.14	0.28	TS
HDS-477	2402	2422.5	20.5	732.1	738.3	6.2	2.80	2.15	2.24	0.13	TS
HDS-477	2474	2531	57	754.0	771.4	17.4	2.82	2.03	1.32	0.22	TS
HDS-477	2586.5	2953	366.5	788.3	900.0	111.7	7.51	7.70	4.99	0.19	TDS
Including	2737	2949.5	212.5	834.2	899.0	64.8	10.18	11.07	7.34	0.26	TDS
HDS-478	2047	2052	5	623.9	625.4	1.5	23.00	16.90	30.04	1.05	TS
HDS-478	2075	2095	20	632.4	638.5	6.1	15.59	11.31	15.93	0.58	TS
HDS-478	2138	2163	25	651.6	659.3	7.6	13.05	6.92	5.29	0.08	TDS

*TS – Taylor Sulphide
*TDS – Taylor Deeps Sulphide
*TVS – Trench Vein System
**Sulphide drill intervals are down-the-hole drill widths but are considered to be within +10% of true width based on the dip of the mineralized stratigraphy at 22 degrees.

Table 10.3 shows the details of the drilling carried out in late 2016 and mid 2017 since the March 2016 Technical Report.

Table 10.3 Taylor Deposit 2016-2017 drilling details

Drillhole	Easting (ft)	Northing (ft)	Elevation (ft)	Azimuth	Dip	Length (ft)
HDS-343	1069953.62	171095.77	5184.64	0	-90	5567
HDS-393	1075559.44	168282.11	5460.54	0	-90	4502
HDS-394	1069493.71	170234.18	5260.98	0	-90	5736
HDS-395	1069636.71	174405.92	4929.14	0	-90	5387
HDS-410	1070275.47	171906.68	5035.34	0	-90	4956
HDS-412	1073646.82	170169.40	5035.61	30	-82	3997
HDS-414	1072014.28	171739.44	5149.24	230	-82	4767
HDS-415	1078365.56	166898.99	5123.93	180	-82	4706.5
HDS-416	1074400.25	169425.32	5087.10	350	-87	3287
HDS-419	1073173.34	170148.26	5048.99	233	-87	3672
HDS-420	1070440.58	169290.60	5225.88	85	-82	4504
HDS-421	1073897.79	169925.12	5067.53	0	-90	3640.5
HDS-422B	1074066.39	169501.90	5117.16	320	-87	3680
HDS-423	1072503.35	171435.91	5140.90	0	-90	4512.5
HDS-424	1073242.62	170481.03	5005.26	30	-82	4205
HDS-425	1072321.21	170381.68	5068.71	0	-90	3757
HDS-426	1075742.66	168041.13	5498.11	0	-90	2287
HDS-427	1076525.62	166899.61	5460.25	340	-82	3548
HDS-428	1071715.23	172079.75	5177.41	355	-75	5359.5
HDS-429	1072249.45	169634.19	5060.02	0	-90	4437
HDS-430	1074066.76	169501.70	5117.22	0	-90	3640
HDS-431	1073181.69	170151.88	5047.76	240	-82	3703
HDS-432	1074436.43	167945.63	5369.40	220	-82	3517
HDS-433	1076525.62	166899.61	5460.25	0	-90	3805
HDS-434	1073339.32	170374.09	5038.75	30	-86	3983

Drillhole	Easting (ft)	Northing (ft)	Elevation (ft)	Azimuth	Dip	Length (ft)
HDS-435	1075746.67	169470.84	5229.34	85	-82	3213
HDS-436	1070246.07	170871.93	5211.04	0	-90	4626
HDS-437	1074385.54	169437.72	5087.03	35	-82	3467
HDS-438	1072198.33	171575.24	5170.49	230	-75	2835
HDS-439	1075754.31	169440.98	5227.39	0	-82	3550
HDS-440	1074448.11	167950.41	5369.82	300	-80	3277
HDS-441	1072195.63	171573.07	5171.63	230	-55	2526
HDS-442	1072794.01	170559.70	5045.01	230	-75	1766.5
HDS-443	1073255.31	170465.59	5006.23	230	-45	1617
HDS-444	1072273.20	170884.49	5138.57	230	-65	2707
HDS-445	1071715.55	172058.26	5179.09	230	-55	2447
HDS-446	1075229.13	169460.03	5188.63	0	-90	3177
HDS-447	1075754.33	169441.21	5228.75	0	-90	3281
HDS-448	1073063.99	170291.43	4982.92	0	-90	3020
HDS-449	1072795.52	170561.41	5043.95	0	-90	4322
HDS-450	1075581.03	169407.29	5269.39	0	-90	3367
HDS-451	1071718.61	172061.24	5181.81	230	-75	2154.5
HDS-452	1074757.80	169233.08	5227.91	0	-90	3339.5
HDS-453	1071654.51	170761.09	5111.44	0	-90	4227
HDS-454	1071299.75	170442.36	5144.46	0	-90	4358
HDS-455	1075228.34	169460.72	5188.16	310	-82	3242
HDS-456	1075734.06	169018.67	5275.50	90	-82	3247
HDS-457	1071852.57	171930.88	5165.70	230	-55	2705
HDS-458	1070319.92	171590.69	5046.28	0	-90	4493
HDS-459	1074460.60	169223.38	5131.90	0	-90	3454
HDS-461	1074419.05	169457.06	5087.53	90	-82	3216
HDS-462	1071855.28	171933.32	5166.01	230	-75	2601
HDS-463	1075447.81	169362.60	5213.01	0	-90	2839
HDS-464	1075732.32	169018.48	5275.01	0	-90	4117
HDS-465	1072007.21	171751.49	5150.13	230	-75	2714
HDS-466	1072175.96	171576.25	5171.99	230	-65	2465
HDS-467	1075047.93	169070.30	5268.45	0	-90	3151
HDS-469	1074890.75	169039.37	5278.26	0	-90	3368
HDS-470	1075767.02	169465.90	5227.59	210	-80	3424.5
HDS-471	1075012.84	169455.90	5126.86	0	-90	3365
HDS-472	1075134.98	169357.76	5141.45	0	-90	3137.5
HDS-473	1071052.13	170804.94	5213.58	0	-90	4364
HDS-474	1074402.98	169444.60	5086.47	53	-82	3673
HDS-477	1075568.26	169410.70	5268.98	330	-84	3517
HDS-478	1075748.51	169478.31	5229.00	80	-75	2987

10.4 Central Deposit

10.4.1 Procedures

Competent, intact core samples were divided with a hydraulic splitter. Spatulas and trowels were used for splitting the sample in clayey or rubbly intervals. Splitter and sample trays were carefully cleaned between samples. Typical, standard sample interval length was nominally set at 5 ft. In areas of mineralogical or geological interest, sample intervals ranged from 1.5 ft to 7 ft.

One split was returned to the original core box for reference and long-term storage. The other split was placed in a heavy gauge plastic bag marked with drillhole number and interval labels. These bags were closed with a wire tie, weighed, and consolidated in shipping boxes or bulk shipping bags.

Reverse circulation holes were drilled wet. The holes were cleaned and blown by the driller between each nominal 5 ft sample interval. A cyclone and wet rotary splitter were set up to obtain two identical splits, weighing approximately 10 lb to 15 lb. The original and duplicate samples were placed in Tyvek sample bags, collected on pallets, shrink wrapped and transported to the project sample processing facility.

The samples were then inventoried and weighed. Standards, blanks and duplicates were inserted in the sample stream. Shipment of samples to Skyline Laboratory of Tucson, Arizona for sample preparation and analyses occurred at regular intervals throughout the drilling campaign.

Figure 10.2 shows in plan, the location of drillholes that have intersected Manto mineralization. Much of the mineralization is stratabound and dips at less than 30°. All of the drillholes are vertical so this combination of variable dips of mineralization and vertical drillholes dips means that most of the drillhole intercepts are slightly greater than true thickness although it is not possible to accurately determine this variance. This is also shown in Figure 7.4 and Figure 7.5.

10.4.2 Relevant sample results

In addition to the true thickness, the results from drilling results and calculated intercepts are shown in long and cross sections in Figure 7.4 and Figure 7.5.

11 Sample preparation, analyses, and security

The QP for this section is Lynda Bloom of Analytical Solutions Ltd. except for the observation in Section 11.3 which is made by the QP for Section 10.

11.1 Background

ASARCO drill programs generated chip samples derived from air rotary hammer drilling and core samples from diamond drillholes. It is assumed that sampling conformed to standard industry practices of the time. AMI inventoried the ASARCO samples in 2006 and re-analyzed 4,272 ASARCO pulp samples. Sample preparation and copper, lead, zinc, and manganese analyses were conducted by Skyline Laboratories in Tucson, Arizona using inductively-coupled plasma and atomic absorption methods. A split of each pulp was then sent to Assayers Canada in Vancouver, British Columbia for silver and gold fire assays.

11.2 Sample preparation and analysis

For the 2010 to 2012 drilling campaign, Skyline Laboratory prepared two 250-gram pulps from each sample. One pulp was retained by Skyline Laboratory and the second pulp was sent to Inspectorate Laboratories of Sparks, Nevada.

Pulps were analyzed by ICP at Skyline for percent copper, lead, zinc, and manganese after a multi-acid digestion. Inspectorate Laboratory determined silver values by gravimetric fire assay with gold values determined by AA finish on the same dissolved doré bead. Remaining portions of the core and all assay pulps are stored in locked, steel shipping containers on property owned or controlled by AZ.

Since 2014, all samples were prepared and analyzed at ALS Minerals, an ISO 17025 accredited laboratory. Samples have been submitted to ALS Minerals, Tucson for crushing and pulverizing (Method Code Prep-31) that includes:

- Drying
- Crushing entire sample to more than 70% passing 2 mm
- Riffle splitting to achieve a 250-g subsample
- Pulverizing the 250-g subsample to better than 85% passing 75 micron

Drill core samples used for Mineral Resource estimation have been analyzed for 33-elements using a 4-acid digestion followed by Inductively Coupled Plasma (ICP) determination.

The analytical methods are summarized in Table 11.1.

Table 11.1 Summary of preparation and assay methods

Element	Method code	Detection limit	Digest	Instrumentation
33 elements, see below	ME-ICP61	Varies; see below	0.25 grams four-acid: HNO_3 + $HClO_4$ +HF + HCl digest plus HCl leach	ICP-AES
Au	Au-ICP21	0.001 ppm	30 grams Fire Assay	ICP-AES
Reanalysis when initial analysis is greater than 1% for lead and zinc and 100 g/ton for Silver				
Ag	Ag-OG62	1 ppm	0.25 grams four-acid: HNO_3 + $HClO_4$ +HF + HCl	ICP-AES
Pb	Pb-OG62	0.001%	0.25 grams four-acid: HNO_3 + $HClO_4$ +HF + HCl	ICP-AES
Zn	Zn-OG62	0.001%	0.25 grams four-acid: HNO_3 + $HClO_4$ +HF + HCl	ICP-AES

The lower and upper limits for the 4-acid digest method (ME-ICP61) are shown Table 11.2.

Table 11.2 Upper and lower limits for 4 acid ICP method

Element	Lower limit	Upper limit	Element	Lower limit	Upper limit	Analyte	Lower limit	Upper limit
Ag	0.5 ppm	100 ppm	Fe	0.01%	50%	S	0.01%	10%
Al	0.01%	50%	Ga	10 ppm	10,000 ppm	Sb	5 ppm	10,000 ppm
As	5 ppm	10,000 ppm	K	0.01%	10%	Sc	1 ppm	10,000 ppm
Ba	10 ppm	10,000 ppm	La	10 ppm	10,000 ppm	Sr	1 ppm	10,000 ppm
Be	0.5 ppm	1,000 ppm	Mg	0.01%	50%	Th	20 ppm	10,000 ppm
Bi	2 ppm	10,000 ppm	Mn	5 ppm	100,000 ppm	Ti	0.01%	10%
Ca	0.01%	50%	Mo	1 ppm	10,000 ppm	Tl	10 ppm	10,000 ppm
Cd	0.5 ppm	500 ppm	Na	0.01%	10%	U	10 ppm	10,000 ppm
Co	1 ppm	10,000 ppm	Ni	1 ppm	10,000 ppm	V	1 ppm	10,000 ppm
Cr	1 ppm	10,000 ppm	P	10 ppm	10,000 ppm	W	10 ppm	10,000 ppm
Cu	1 ppm	10,000 ppm	Pb	2 ppm	10,000 ppm	Zn	2 ppm	10,000 ppm

High grade samples, for Ag greater than 100 g/ton and base metals over 1%, are analyzed a second time using ICP methods optimized for high grade samples. The same sample weight and acids are used for the repeat analysis.

11.3 Security

Core and samples are stored in secure shipping containers, owned by AZ, at the project and at the office located in Patagonia, Arizona. The on-site storage location also has facilities for core logging, core cutting and core sampling. Core is stored in wax cardboard boxes and organized in shipping containers by drillhole number which is shown in Figure 11.1. This has been validated by the QP for Section 10.

Figure 11.1 Core storage container

11.4 Quality control

The 2010-2012 QC program included the use of one standard inserted with every 20th sample. Five in-house standards were prepared for the Hermosa project and certified by Mineral Exploration Group of Reno, Nevada using a six laboratories round-robin analytical program.

Blank samples were used to check the integrity of sample preparation procedures and were inserted at the beginning and end of every sample batch run. Blank samples were prepared and certified by Mineral Exploration Group of Reno, Nevada from limestone, silica sand and volcanic rocks. Systematic contamination during sample preparation was not detected in the analyses of blanks.

Field duplicates from core and chips were taken at intervals of approximately 50 ft (15 m). Core duplicates were quarter-splits, chip duplicates were nominally full sample weight. The duplicate pairs show acceptable reproducibility.

AZ retained the services of Analytical Solutions Ltd to assess the effectiveness of the quality assurance, quality control program used by the company for the 2010-2012 drilling campaign. Lynda Bloom's report to AZ was included in the 31 October 2012 Preliminary Economic Assessment document. Analytical Solutions Ltd concluded that the QA / QC program was adequate to ensure a reliable resource level estimate.

The in-house standards were re-submitted for analysis and showed a low bias for Ag assays by fire assay with a gravimetric finish; Au, Cu, and Zn concentrations were confirmed. Based on additional test work, including check assays at TSL Laboratories (Saskatoon), it was determined that Ag assays for samples were also biased low and a re-assay program was initiated.

Skyline recommended the use of a multi-acid digestion and AAS finish for all samples with the stipulation that all assays above a certain limit trigger a fire-assay, gravimetric Ag assay. That recommendation has been adopted as standard protocol for the project.

Pulps for samples with assay grades between 0.4 oz/ton and 7 oz/ton Ag were selected for re-assay. Pulps were retrieved from the sample archive and re-submitted to Inspectorate Laboratories. In total 6,735 sample pulps, for intervals included in the resource estimation, and 1,343 standards and duplicates were assayed from 188 holes. The re-assay results for Ag using the 4-acid, AAS finish were inserted into the assay database, replacing all assays between 0.02 oz/ton and 5.0 oz/ton Ag. Samples with assay grades greater than 5.0 oz/ton Ag retained the 1 ton, fire-assay, gravimetric Ag finish, were judged to be a more accurate and precise for higher grade materials.

Most of the replaced assays were from the Upper Silver, Hardshell, and Manto Oxide areas.

Since 2014, quality control samples are inserted at a rate of approximately 10% which exceeds industry standards. Quality control measures include laboratory preparation duplicates, pulp duplicates and core duplicates.

Samples for the 2014 - 2015 drilling campaign were submitted to ALS Minerals, Tucson for the same methods as described in Table 11.1. The quality control (QC) results and discussion for these programs can be found in the March 2016 Technical Report. All data for the 2014-2015 drilling campaign were acceptable for use in resource estimation.

Samples for the 2016 drill program, specifically holes HDS-330 to HDS-369, were submitted to ALS Minerals, Tucson for the same methods as described in Table 11.1. The quality control (QC) results and discussion for these programs can be found in the October 2016 Technical Report. All data for the first phase of the 2016 drilling campaign were acceptable for use in resource estimation.

Samples for the second phase of the 2016 drill program and early 2017, covering holes HDS-347, 353, 359, 372, HDS-378 to 409, 411, 413, and HDS-417 to HDS-418, were submitted to ALS Minerals, Tucson for the same methods as described in Table 11.1. The quality control (QC) results and discussion for these programs can be found in the May 2017 Technical Report. All data for the second phase of the 2016 drilling campaign were acceptable for use in resource estimation.

Approximately 45,000 samples (including Quality Control "QC" samples) have been collected and assayed for Arizona Minerals' Hermosa Project from February 2017 to September 2017. The following information refers to the analytical program related to holes HDS-393 to 395, HDS-410, HDS-412, HDS-414 to HDS-416, HDS-419 to HDS-467, HDS-469 to HDS-474, HDS-477, and HDS-478.

11.4 Blanks

Barren fine-grained silica ("blank") was submitted with samples to determine if there has been contamination or sample cross-contamination. Elevated values for blanks may indicate sources of contamination in the analytical procedure (contaminated reagents or crucibles) or sample solution carry-over during instrumental finish.

Blanks were inserted with samples 525 times and coarse-grained granite blanks were inserted with samples 131 times. The granite material has background levels of the elements that are monitored, and the allowed upper limit was adjusted accordingly.

Table 11.3 Upper limit for blanks

Element	Fine blank	Coarse blank
Ag (ppm)	5	5
Cu (ppm)	10	40
Pb (%)	20	100
Zn (%)	20	200
Mn (ppm)	50	1200

Two percent of the blanks had Mn contamination of 50 ppm or more. The fine-grained blanks are not pulverized and therefore the Mn contamination is likely associated with solution carry-over at the ICP. The contamination was mostly apparent in mineralized intervals with more than 1% Mn. This is evident of nominal solution carry-over and is not actionable.

In only one case out of 131 insertions of coarse blanks was there evidence of sample cross-contamination. Sample HDS-446 1087-1092B assayed 298 ppm Pb, 601 ppm Zn, and 6,930 ppm Mn and is likely evidence of sample carry-over in sample preparation and / or analysis. The samples before and after the contamination event are very high grade.

The very low rate of Ag, Pb, or Zn QC failures for blanks indicates that sample cross-contamination in preparation and analysis is well controlled and not a risk for the project.

11.4.1 Reference materials

The three certified reference materials (CRMs) inserted for the QC program are purchased from a third-party supplier, OREAS. The CRMs were analyzed at more than 15 laboratories to determine expected values and tolerances. The materials are matrix-matched for the Hermosa deposit style with the source rock from SEDEX deposits within carbonaceous dolomitic sediments. Expected values for the CRMs are based on 4-acid digest Inductively Coupled Plasma (ICP) analyses.

There were 1,664 insertions of CRMs with drill core samples. A low proportion of quality control failures were identified for Pb, Zn, Cu, and Ag; these are cases where the results were outside the tolerance of 3 standard deviations or there were consecutive RMs outside +2 or -2 standard deviations. AZ staff identified QC failures when results were received and requested repeat assays as required.

All acceptable data were plotted on control charts and are summarized in Table 11.4 to Table 11.10. For the purposes of these calculations, samples were labelled as "outliers" where an outlier is identified by having a 'Z' score greater than 5, where Z=(Measured-Expected) / Tolerance. The "outliers" are usually mislabelled CRMs and not relevant to a discussion of laboratory performance.

The observed average values for Ag, Pb, Zn, and Cu in CRMs fall within ± 3% of expected values. There is no consistent bias for the reference materials with respect to Ag, Pb, Zn, and Cu. Iron, Mn, and As are also monitored as elements representative of laboratory performance for the ICP suite of elements.

Table 11.4 Performance of silver in CRMs

RM	N	Outliers excluded	Failures excluded	Ag ppm		Observed Ag ppm		Percent of accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS 133b	474	3	7	104	2.00	103	1.90	99%
OREAS 132a	555	-	1	57	3.04	58	1.87	102%
OREAS 131b	604	2	3	33	1.21	34	1.01	103%
Total	**1633**	**5**	**11**	**Weighted average**				**101%**

Table 11.5 Performance of arsenic in CRMs

RM	N	Outliers excluded	Failures excluded	As ppm		Observed As ppm		Percent of accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS 133b	484	-	-	144	13	136	6	95%
OREAS 132a	556	-	-	146	16	140	6	96%
OREAS 131b	609	-	-	82	7	80	4	98%
Total	**1649**			**Weighted average**				**96%**

Table 11.6 Performance of copper in CRMs

RM	N	Outliers excluded	Failures excluded	Cu ppm		Observed Cu ppm		Percent of accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS 133b	481	-	3	320	14	327	10	102%
OREAS 132a	556	-	-	461	23	468	15	101%
OREAS 131b	606	-	3	216	11	225	7	104%
Total	**1643**		**6**	**Weighted average**				**103%**

Table 11.7 Performance of iron in CRMs

RM	N	Outliers excluded	Failures excluded	Fe %		Observed Fe %		Percent of accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS 133b	477	-	7	8.16	0.345	7.78	0.222	95%
OREAS 132a	547	-	9	7.73	0.324	7.35	0.213	95%
OREAS 131b	608	-	1	5.71	0.328	5.53	0.156	97%
Total	**1632**		**17**	**Weighted average**				**96%**

Table 11.8 Performance of lead in CRMs

RM	N	Outliers excluded	Failures excluded	Pb %		Observed Pb %		Percent of accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS 133b	481	2	1	5.1	0.098	5.1	0.082	101%
OREAS 132a	556	-	-	3.6	0.135	3.6	0.061	99%
OREAS 131b	609	-	-	1.9	0.086	1.9	0.031	98%
Total	**1646**	**2**	**1**	**Weighted average**				**99%**

Table 11.9 Performance of zinc in CRMs

RM	N	Outliers excluded	Failures excluded	Zn %		Observed Zn %		Percent of accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS 133b	483	1	-	11.4	0.347	11.3	0.186	100%
OREAS 132a	555	-	1	5.0	0.107	5.0	0.084	99%
OREAS 131b	609	-	-	3.0	0.119	3.1	0.050	101%
Total	**1647**	**1**	**1**	**Weighted average**				**100%**

Table 11.10 Performance of manganese in CRMs

RM	N	Outliers excluded	Failures excluded	Mn ppm		Observed Mn ppm		Percent of accepted
				Accepted	Std. Dev.	Average	Std. Dev.	
OREAS 133b	483	-	1	1165	58	1177	39	101%
OREAS 132a	553	-	3	2000	100	2085	66	104%
OREAS 131b	605	-	4	1671	84	1730	53	104%
Total	**1641**		**8**	**Weighted average**				**103%**

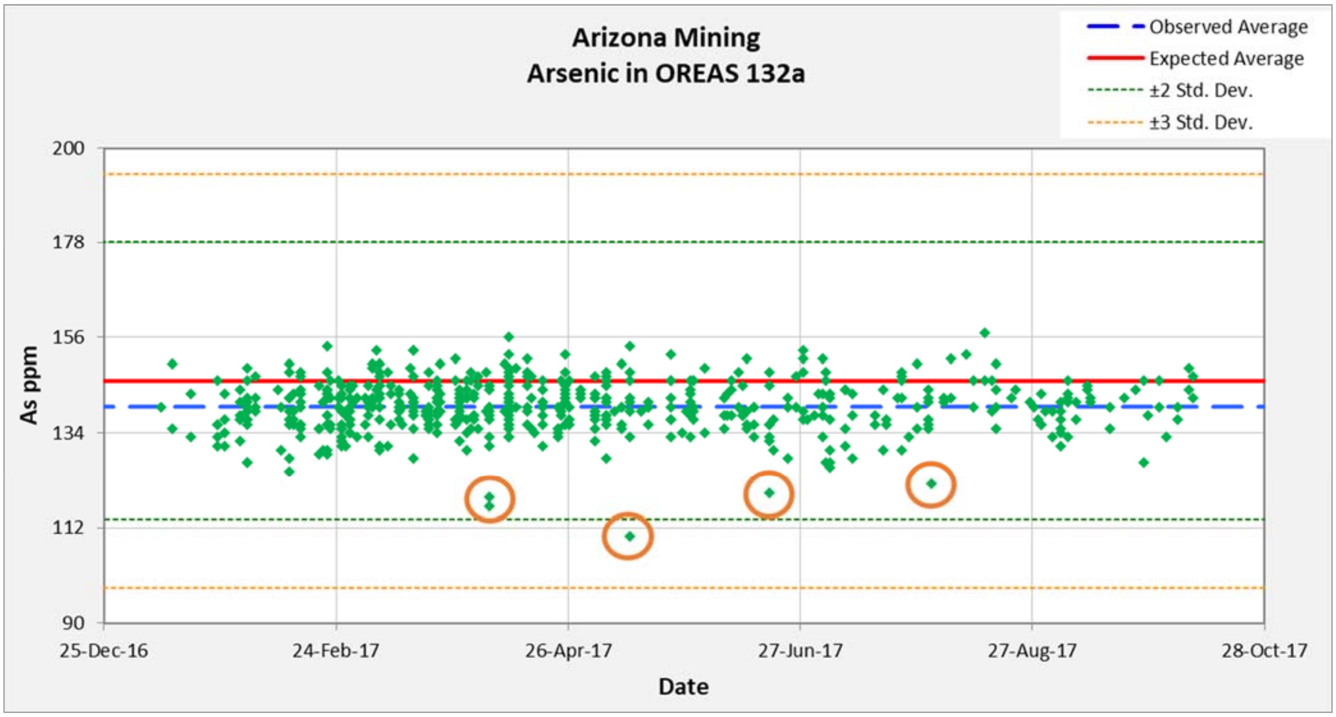
Laboratory performance, based on blanks and standards, was excellent and Ag, Cu, Pb, and Zn analytical data are considered acceptable for use in resource estimation.

It was noted that there were 6 cases where ALS reports Ag for RM OREAS 133b at 6% to 8% low. This is a significant trend and noteworthy because RM OREAS 133b has a relatively high failure rate of 2% and an overall low bias (although within an allowed 2%). The expected Ag value is 104 ppm which is just above the upper limit of the ICP61 method. It is possible that the instrument calibration is not stable at this upper limit and sometimes reports low. This means that samples that are reporting around 100 ppm Ag by ICP61 might also sometimes report low. In the cases where Ag reports low by ICP61, Fe also reported low by 13% to 15% which may indicate a further complication with incomplete dissolution.

It is recommended that in the future all samples that have Ag greater than 90 ppm by method ICP61 be automatically re-analyzed by the over-range method (OG62). Currently only samples with Ag greater than 100 ppm are re-analyzed by the over-range analytical method. It is not expected that there would be a significant impact on the overall resource estimate, but it is preferable to have the correct Ag values at the 100 ppm concentration level.

The observed As values are generally lower than the expected values by approximately 4%. It is not uncommon for there to be differences between laboratories for analytical methods that require acid digestion, and this may explain the low bias for As at ALS relative to the laboratories that participated in the certification program for the CRMs. Figure 11.2 shows a few outliers have been circled in orange which may represent a problem separate from the overall low bias seen in As for OREAS 132a. The low outliers could be caused by insipient dryness. Samples are dissolved in multiple acids and the solutions are heated and taken almost to dryness. If the mixture allowed to thoroughly dry salts start to crystallize and some salts, particularly for As, are known to not dissolve in the final HCl solution. As a result of salts not dissolving, results will report low. A low bias for As is not significant for resource modelling but should be taken into account if As is used to model environmental impact.

Figure 11.2 Arsenic control chart CRM 132a



11.4.3 Reproducibility of laboratory preparation and pulp duplicates

Commercial laboratories routinely assay a second aliquot of the sample pulp, usually for one in ten samples. The data are used by the laboratory for their internal quality control monitoring. The data are provided at no additional cost and were requested by AZ. ALS Minerals provided the quality control data from a query of its QC database system.

Results for pulp and preparation duplicates that were reviewed fall within an expected range for base metal assays. Only the duplicate pairs above 10 times the lower detection limit are considered significant and are included in calculations.

Duplicate pairs for pulps report within ±10% for more than 81% of the cases considered. Duplicates for 921 duplicates pairs for Pb have the lowest rate of results within ± 10% at 81%, increasing to better than 97% for Ag, Mn, Fe, and Mn. Similarly, duplicate pairs for splits of the crushed material (i.e., preparation duplicates) report within ± 10% for 62% to 90% of duplicate pairs.

The results are consistent with the results from previous reports. The data are suitable for use in mineral resource estimation.

11.4.4 Core duplicates

The second half of a drill core sample is assayed to determine:

- The reproducibility of assays for different halves of the core.
- If there is any sampling bias.

To make core duplicate samples, AZ splits the primary half-core sample into two quarter-core samples. One quarter-core was submitted as the primary sample and the other quarter-core was submitted as the duplicate. This is common industry practice but alters the sampling statistics.

A total of 1,259 quarter-core duplicates were collected and submitted for analyses. The summary of the quarter core duplicates can be found in Table 11.11.

Table 11.11 Summary of quarter core duplicate results

Analyte	No. of pairs above 10x d.l.	% of sample pairs (>10x d.l.) reporting within		
		±10%	±25%	±50%
Pb	930	29%	64%	87%
Zn	1,227	51%	78%	92%
Cu	810	38%	74%	91%
Ag	122	56%	80%	91%
As	297	48%	84%	96%
Fe	1,225	71%	94%	99%
Mn	1,259	64%	91%	99%
S	878	64%	88%	97%
Sb	97	35%	64%	88%

Between 65% to 80% of the quarter core duplicates agree within +/- 25% for Pb, Zn, and Ag.

The correspondence between duplicates improves for higher grade samples, likely because core is more pervasively mineralized. Large differences between assays for quarter core samples have been previously reported. It is suspected that the reason for the extreme variability is related to the style of mineralization. Within CRD mineralization zones, it was noted that galena, sphalerite, pyrite, and chalcopyrite mineralization can be irregular. A duplicate sample from a quarter split could simply have abundant CRD mineralization in duplicate-A and moderate mineralization in the duplicate-B sample. Similarly, an interval within a "Hardshell" mineralization zone may include a mineralized limestone clast, which occurs intermittently. The variability due to geology has been described by AMC in the 2014-2015 technical report for a low percentage of core duplicates.

These observations are consistent with previous technical reports. The variation for core duplicates is within the expected range for the deposit style.

There is no quality expectation for core duplicates and therefore the results are not actionable.

There is sufficient data for quarter-core duplicates that core duplicates should not be collected for the remainder of resource drilling, unless new styles of mineralization are encountered.

11.5 Summary

No aspect of the sample preparation process was conducted by an employee, officer, director, or associate of AZ. All samples were prepared and analysed at ALS Minerals, an ISO 17025 accredited laboratory.

In the opinion of the QPs, the sample preparation, security and analytical procedures for all assay data since 2010 are adequate for use in Mineral Resource estimation.

12 Data verification

12.1 Taylor Deposit

Over time, QPs reviewed procedures and process to assure that the data used for the Mineral Resource estimate described in Section 14 of this report was adequate for the purpose of that estimate. A visit was carried out by Dinara Nussipakynova during 6 - 7 September 2017, and data was also verified in the data import process.

12.1.1 Drill core examination

Drill core was examined on three occasions for the identification of the major lithotypes described in the drill logs generated by AZ, and for the presence of lead-zinc mineralization as reported in the drill logs and in assays. The major rock types, carbonate and volcanic, that were observed in the core, were consistent with the descriptions employed in the drill logs. Lead and zinc mineralization, as galena and sphalerite, were abundant in the core from a number of drillholes that were being logged at the time of the site visits. No samples were collected as the presence of this mineralization was obvious. Further, samples of split core can be expected to display variations in metal content because of the coarse and commonly disseminated nature of the mineralization.

12.1.2 Drillhole collar location

The locations of several drillholes (HDS-342 and HDS-354) were verified using a hand-held GPS. The datum employed by the GPS was WGS 84; the datum used by the Taylor project is NAD 83, Arizona Central State Plane 0202. AZ provided coordinates for the two holes converted to WGS 84. The converted coordinates were within one foot (0.3 m) of those recorded by the GPS. During the most recent site visit the locations of the majority of the infill holes were examined and their locations noted with respect to previously-drilled holes.

12.1.3 Assay data verification

A comparison of assay certificates, as received by AZ from ALS Minerals in Tucson, Arizona, with the corresponding assay values included in the drillhole dataset received by AMC from AZ was carried out. Approximately 1,500 samples (6,000 assays) were checked for lead, zinc, copper and silver values from six drillholes. No discrepancies were found. It should be noted that silver grades in the dataset are reported in ounces per short ton whereas in the assay certificates silver grades are reported in grams per metric tonne. A conversion of 34.2857 grams per metric tonne = one ounce per short ton was used to compare the two data sets for silver. Lead, zinc and copper grades were recorded in the AZ dataset in percent whereas the assay certificates report grades in parts per million (ppm). A conversion of 10,000 ppm = 1% was used to compare the two data sets.

12.2 Central Deposit

The following text has been copied or abridged from the January 2014 PFS report, which describes the work pertaining to the Central Deposit, that is not covered in Section 12.1.

12.2.1 AZ comparative drilling

Table 12.1 Comparison drillhole pairs

New core hole number	Core hole TD (ft)	Previous air- hammer number	Air-hammer hole TD (ft)	Separation distance (ft)
HDS-99	1,257.00	HDS-83	480	6.6
HDS-98	1,016.00	HDS-40	570	10.5
HDS-100	1,127.00	HDS-62	385	27.8
HDS-101	**1,058.50**	**HDS-81**	**500**	**13.4**

Note: All drillholes are vertical. TD = top down.

The results of the twin drilling are summarized in Table 12.2. Twinned holes intercepted similar material at equivalent depths down hole with similar interval lengths.

Table 12.2 Twinned drillhole comparisons

Hole	Interval (ft)	Thickness (ft)	Silver (oz/ton)	Manganese (%)	Zinc (%)	Copper (%)
HDS-98 vs. HDS-40						
HDS-98 (Core)	390-567	177	5.09	17.75	1.83	0.2
HDS-81 (Air-hammer)	380-565	185	6.4	18.65	1.93	0.21
HDS-99 vs. HDS-83						
HDS-99 (Core)	350-470	120	6.34	17.6	1.44	0.13
HDS-81 (Air-hammer)	350-470	120	7.08	14.52	1.55	0.17
HDS-100 vs. HDS-62						
HDS-100 (Core)	222-373	151	7.09	12.35	2.43	0.21
HDS-81 (Air-hammer)	220-370	150	6.52	8.57	2.09	0.34
HDS-101 vs. HDS-81						
HDS-101 (Core)	272-508	236	5.64	5.07	1.86	0.11
HDS-81 (Air-hammer)	**265-500**	**235**	**9.36**	**5.87**	**2.4**	**0.1**

The analytical results for all metals show grade values behave similarly. Pincock, Allen & Holt (2008) concluded that the analytical variability reflects natural short range grade differences in the deposit rather than drilling method bias.

12.2.2 Additional data validation

Logging procedures and protocols, re-logs of chips and core and field checks have also been used to validate data sources. Drillhole collar locations have been resurveyed by a licensed Arizona registered land surveyor.

In the QP's opinion, the data is suitable for the purposes used in the Technical Report.

13 Mineral processing and metallurgical testing

13.1 Introduction

Metallurgical test work programs have been performed towards developing a processing route to recover sulphide material from the Taylor deposit. This test work has focused on recovering lead, zinc, and silver mineralized materials via production of a lead concentrate containing payable amounts of silver followed by a zinc concentrate. Additional test work programs are either in progress or are planned to more definitively define the processing route.

Mineralized sulphide material contained within the Taylor deposit has been classified into five main types. They are:

1. Concha
2. Scherrer
3. Epitaph
4. Deeps
5. Veins

Metallurgical test work has been performed to date on samples of sulphide material from the project by:

- Resource Development, Inc. (RDI) of Wheat Ridge, Colorado.
- SGS Lakefield (SGS) of Ontario, Canada.
- FLSmidth Minerals Testing and Research Center (FLS) of Salt Lake City, Utah.

The results of the RDI and SGS test work programs are reported in the November 2016 Technical Report and the March 2017 Technical report and the reader is directed to those reports for additional information. This Technical Report incorporates recent results from an FLS test work program. Information contained in the previous Technical reports has been copied verbatim and is identified in Italics. The Italicised text has been taken from the March 2017 Technical Report where the text was authored by Mr Qinghua Jin, P.E. of SGS Minerals. The recent FLS test work was performed on a sample identified as Master Composite (MC) which was a mix of materials from the five sulphide mineralized types. The results are summarized below and are presented along with the metallurgical test results reported in the March 2017 Technical Report. The current results add to the knowledge of the metallurgical response of sulphide material from the project and supplement the data from earlier testing programs. Additional testing is in progress.

FLS test work results incorporated into the capital and operating costs for this PEA are limited to a revision of the primary grind size and to the elimination of lime as a flotation reagent. The primary grind size increases from an 80% passing size (P_{80}) of 105 µm to P_{80} of 150 µm. Additional data produced by the FLS test program is preliminary in nature and will be developed more completely for use in future studies. Table 13.1 provides the basic metallurgical performance parameters used in this PEA.

Table 13.1 Metallurgical performance

Products	Weight %	Assays			% distribution		
		Zn (%)	Pb (%)	Ag (g/t)	Pb (%)	Zn (%)	Ag (g/t)
3rd Lead Cl Concentrate	6.1	3.40	69.7	1,072	95.4	5.2	69.2
2nd Zinc Cl Concentrate	6.7	56.1	1.03	331	1.5	92.7	23.2
Zinc Ro Tailings	87.2	0.10	0.16	8.27	3.1	2.1	7.6
Primary Grind P_{80}, µm	150						

Ro = Rougher, Cl = Cleaner.

Hazen Research, Inc. (Hazen) of Golden, Colorado is currently performing test work on oxide material from the project. This test work is not part of this Technical Report.

13.2 Summary of historical test work

"RDI located in Wheat Ridge, Colorado completed a scoping level metallurgical study for zinc / lead / silver mineralization for the Taylor deposit which has been reported in detail in the November 2016 Technical Report. This work was on a composite sample representative of the mineralization known at that time and was prepared from split drill core samples which were projected to have the average grade of the deposit. The composite assayed 9.14% Pb, 7.99% Zn, 0.24% Cu, 126 g/ton Ag, and 0.29 g/ton Au which was higher grade than the average projected grade of ± 5% Pb and ± 5% Zn. The concentrates produced, are not expected to pose any issue for contract sales or smelting of the concentrates; it is recognized that the Mn content of the Zn concentrates produced occurs at levels, about 1.3-1.4% Mn, where a penalty will usually occur, but this amount to about 1% of the total value of the Zn concentrate. The composite sample had a Bond's mill work index of 14.03 kwh/ton.

A summary of the prior metallurgical testwork performed on the Central Deposit mineralization is also contained in the November 2016 Technical Report. However, testwork has continued and average processing recovery factors of 55% for zinc, 72% for silver, and 86% for manganese are substantiated in a report from Hazen."

Additional testwork is in progress, but has not been completed.

13.3 Metallurgical test work samples

13.3.1 Samples provided to SGS

"Two skids of pails containing Taylor deposit material types, weighing approximately 680 kg, were used to generate 12 different metallurgical composites. A subsample of each composite was submitted for assay.

The pertinent assay results from each of the 12 composites are shown in Table 13.2".

Table 13.2 Composite head assays

Composite		Head assays (%)					Head assays (g/t)	
		Pb	Zn	Cu	Fe	S	Ag	Au
Comp 1	Concha_2.5%	1.21	1.36	0.06	1.67	2.09	25.0	0.03
Comp 2	Concha_5%	2.27	3.02	0.08	2.07	3.38	41.0	0.04
Comp 3	Concha_10%	4.35	5.74	0.09	4.81	6.89	59.9	0.09
Comp 4	Concha_15%	6.57	8.10	0.22	4.53	8.34	115	0.05
Comp 5	Scherrer_5%	3.02	2.67	0.09	2.24	3.09	40.4	0.05
Comp 6	Epitaph_2.5%	1.26	1.47	0.05	2.07	2.40	16.8	0.02
Comp 7	Epitaph_5%	2.62	2.68	0.10	2.91	3.95	32.4	0.04
Comp 8	Epitaph_10%	5.12	5.69	0.14	3.43	6.39	74.8	0.17
Comp 9	Epitaph_15%	8.25	6.76	0.19	3.64	6.77	192	0.06
Comp 10	High Copper	25.7	20.1	1.05	5.69	18.8	314	0.07
Comp 11	High Lead / Low Zinc	18.1	2.26	0.02	2.31	5.00	187	0.10
Comp 12	Low Lead / High Zinc	4.29	10.8	0.07	2.87	1.60	<10.0	0.02

13.3.2 Samples provided to FLS

Ten barrels containing mineralized material from the five different mineral types were supplied to FLS. The mineralized material was one-quarter HQ sized core. A summary of the material received are presented in Table 13.3.

Table 13.3 Summary of sulphide samples used in preparing the MC

Mineralized	Mass		Interval	Number
Classification	**lbs**	**Kg**	**m**	**Samples**
Concha	746.0	337.6	582.5	124
Scherrer	786.0	355.7	588	124
Epitaph	780.4	344.1	572.5	122
Deeps	808.0	365.6	632	125
Veins	708.6	320.6	522	117

Source: Table 6 of FLS report.

The material was stage crushed to 100% minus 10 mesh. A 300 Kg composite sample was prepared and identified as MC. This sample was then sub-split into batch sizes of 2 Kg and 25 Kg and stored in a freezer. A head grade analysis of each mineralized classification as well as for the MC was performed. The results are shown in Table 13.4.

Table 13.4 Head assay results for mineralized classifications

Element	Units	Domain					
		MC	**Concha**	**Scherrer**	**Epitaph**	**Deeps**	**Veins**
Ag	ppm	89.3	79.2	43.8	37.1	85.5	186.7
Au	ppm	0.058	0.063	0.06	0.033	0.047	0.036
Pb	wt%	4.52	5.79	3.17	3.66	3.59	5.21
Cu	wt%	0.120	0.202	0.064	0.055	0.132	0.169
Zn	wt%	4.70	7.40	3.21	3.58	2.45	6.24
Fe	wt%	5.04	6.98	5.72	2.85	3.64	5.62
S =	wt%	7.87	12.00	8.60	3.98	3.69	10.50
S (t)	wt%	8.33	12.40	8.63	4.27	3.92	10.85
As	ppm	50.7	169.2	<20	<20	<20	216.3
Sb	ppm	263.4	304.2	80.7	<10	103.8	748.3
Bi	ppm	86.0	82.0	70.0	67.0	120.0	74.5
Cd	ppm	157.6	309.4	95.3	84.2	42.9	227.8
Tl	ppm	<10	<10	<10	<10	<10	<10
Al	wt%	1.9	0.8	1.8	1.5	1.7	4.4
Ca	wt%	9.9	7.7	10.4	16.3	14.5	0.8
Si	wt%	16.6	14.8	16.5	14.1	18.5	23.1
Mg	wt%	1.34	0.51	1.59	3.83	0.52	0.29
Mn	wt%	3.91	6.82	4.77	3.31	3.04	1.83
K	wt%	1.33	0.31	1.15	0.71	0.71	4.04
Na	wt%	0.10	0.03	0.09	0.05	0.04	0.09
C(t)	%	2.99	2.95	3.06	4.23	3.94	0.558
C(org)	%	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Ba	wt%	0.03	<0.02	0.02	<0.02	<0.02	0.09
Co	ppm	<10	<10	<10	<10	12.1	<10
Cr	ppm	47.4	102.6	86.8	23.9	78.4	103.3
Li	ppm	19.4	22.9	30.6	26.8	35.1	11.0
Mo	wt%	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002
Ni	wt%	< 0.003	< 0.003	< 0.003	< 0.003	< 0.003	< 0.003
P	wt%	0.09	0.13	0.12	0.06	0.10	0.04
Sn	wt%	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005
Sr	ppm	40.4	22.8	37.2	48.8	50.3	46.1

Element	Units	Domain					
		MC	Concha	Scherrer	Epitaph	Deeps	Veins
Te	ppm	<10	<10	<10	<10	<10	<10
Ti	wt%	0.10	0.03	0.09	0.11	0.09	0.17
V	ppm	25.2	30.1	27.8	<25	40.3	39.4
W	wt%	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002	< 0.002
Zr	wt%	< 0.003	< 0.003	< 0.003	< 0.003	< 0.003	0.0115

Source: Table 7 of FLS Report.

13.4 Mineralogy

13.4.1 Mineralogy – SGS

"The samples include various silicates and sulphides; sphalerite ranged from 2.2% to 32.6% and averaged 8.5%, galena from 1.2% to 31.1% and averaged 8.5%. Cu-sulphides included mainly chalcopyrite from traces to 1.8% and averages 0.2%, tetrahedrite averaged 0.1%, and there were traces of chalcocite, bornite, and other Cu-sulphides. The main gangue sulphide was pyrite that ranged from 0.1% to 9.6% and averaged 4.6%. A number of other minerals included quartz (24.4%), rhodonite (10.7%), rhodochrosite (10.6%), and calcite (8.3%). A number of Zn-bearing silicates were also present.

Mineralogy data indicated that the average zinc concentration was similar at 62% to 63% by weight in Composites 2, 4, 8, and 10 and 65% by weight in Composite 12. Sphalerite from Composite 12 had the lowest concentrations of iron (0.14% by weight), manganese (0.19% by weight), and cadmium (0.01% by weight). It is apparent that sphalerite in Composite 12 was different than that in the other samples.

Sphalerite hosted most of the zinc at 72% to 99% in all samples. Composite 12 was the exception in which sphalerite accounted for 29% of the total zinc in the sample and Zn-bearing silicates for the remainder.

Liberated sphalerite at a P_{80} grind size of approximately 212 µm accounts for between 78% to 93% in Composites 1 to 11, and 28% in Composite 12. Liberated galena ranged from 29% to 89% and averages 70%. Liberated Cu-sulphides (which comprise chalcopyrite, chalcocite, bornite, tetrahedrite, and other Cu-sulphides as one mineral group) ranged from 9% to 74% and averaged 49%. Liberated pyrite accounted for between 7% and 94% and averages 82%."

13.4.2 Mineralogy – FLS

A mineralogy study of the five rock types and the MC were not completed. Mineralogy work completed by SGS was used as a reference for the six mineralized samples including the MC. FLS mineralogy work has been limited to examining a zinc rougher tailing to identify zinc mineral losses.

It was suspected zinc losses were associated with smithsonite ($ZnCO_3$) that is not floatable with the current flotation reagent scheme. A Zn deportment by QEMSCAN analysis was performed on a Zn rougher tails sample to better understand this hypothesis.

Polished sections were prepared and analyzed by QEMSCAN. It was identified that the zinc bearing phases are sphalerite (locked in gangue and as free <10 µm particles), willemite (Zn_2SiO_4), and in zinc bearing carbonates. Some of the zinc bearing carbonates have compositions similar to smithsonite, while many are intermediate compositions between rhodochrosite ($MnCO_3$), siderite ($FeCO_3$) and smithsonite ($ZnCO_3$). The Zn deportment is presented in Table 13.5.

Table 13.5 Zinc deportment in zinc rougher tails

Mineral phase	Zn (weight %)	Zn (% distribution)
Sphalerite	0.05	16
Carbonates	0.00	1
Smithsonite (Zn, Mn, Fe) CO_3	0.14	49
Willemite	0.10	34

Source: Table 19 in FLS report.

13.5 Comminution – SGS

"Composites 1-10 were submitted for SAG Power Index (SPI) and Bond ball mill work index testing (BWI). Composites 1, 2, 4, 7, and 8 were also tested for Bond rod mill work index (RWI) and Bond abrasion index. The results achieved are shown in Table 13.6. There is a good relationship between hardness and head grade. As the head grade of zinc and lead increase, the hardness of the mineralization decreases.

Composites 6 and 7 were deemed hard to very hard with respect to SPI rating. Composite 10 was deemed soft, while the rest of the composites obtained a medium hardness SPI rating.

Composites 1 and 7 were hard to very hard in testing for RWI at 18.3 kWh/t and 17.7 kWh/t, respectively. The three other composites were deemed moderately hard.

Most of the composites tested for BWI were in the medium to moderately hard range. Composites 1 and 2, at 17.9 kWh/t and 17.8 kWh/t respectively, were deemed hard to very hard. Composite 10, at 8.9 kWh/t, was soft in nature.

The Bond abrasion indices ranged from 0.211 g to 0.340 g, indicating mild to medium abrasiveness."

Table 13.6 Comminution test results

Composite		CEET	SPI®	Work indices (kWh/t)		AI
		Ci	(Min)	RWI	BWI	(g)
Comp 1	Concha_2.5%	9.8	102.6	18.3	17.9	0.340
Comp 2	Concha_5%	8.0	97.9	15.9	17.8	0.221
Comp 3	Concha_10%	15.3	64.4	-	15.9	-
Comp 4	Concha_15%	9.8	95.1	15.6	14.3	0.277
Comp 5	Scherrer_5%	5.6	87.5	-	14.9	-
Comp 6	Epitaph_2.5%	9.5	126.3	-	16.6	-
Comp 7	Epitaph_5%	5.6	121.5	17.7	15.8	0.310
Comp 8	Epitaph_10%	7.5	74.2	14.8	15.4	0.299
Comp 9	Epitaph_15%	4.5	60.5	-	14.1	-
Comp 10	High Copper	7.9	14.7	-	8.9	-

CEET = Comminution Economic Evaluation Tool
Ci = Crusher index
AI = Bond abrasion index
SPI = SAG Power Index

13.6 Flotation tests

13.6.1 Locked cycle flotation tests – SGS

"One locked cycle flotation test was completed for composites Epitaph_10%, Concha_10%, and Scherrer_5%. Test conditions were based on those of comparable batch rougher and cleaner flotation tests. The flowsheet followed for each test is shown in Figure 13.1 and the metallurgical projections are summarized in Table 13.7.

Each 2 kg sample for the locked cycle tests were ground to a target P_{80} of 500 µm, in the presence of $ZnSO_4$ and NaCN, and subjected to flash flotation. The ground material was treated with 3418A collector, lime to reach a pH of 9.0 and then floated for 1 minute. The flash rougher concentrate was then reground to a target P_{80} of 75 µm and cleaned twice with the addition of more $ZnSO_4$ and NaCN, to help reject contained zinc. The 2nd flash cleaner concentrate was a final product. The tailings from each stage were reverted back to the head of the previous circuit in the next cycle. The flash tailing was ground to a target P80 of 106 µm and floated for 6 minutes in the presence of $ZnSO_4$, NaCN and 3418A. The lead rougher concentrate was then reground to a target P_{80} of 38 µm and cleaned three times. The third lead cleaner concentrate was a final product. The tailings from each stage were reverted back to the head of the previous circuit in the next cycle.

The lead rougher tailings were treated with $CuSO_4$ and SIPX and floated for 6 minutes at a pH of 11. The zinc rougher concentrate was then reground to a target P_{80} of 38 µm, treated with additional $CuSO_4$ and SIPX and then cleaned twice to generate the final zinc concentrate. The first zinc cleaner tailing and the zinc rougher tailing were final products at each stage."

Figure 13.1 Locked cycle flowsheet

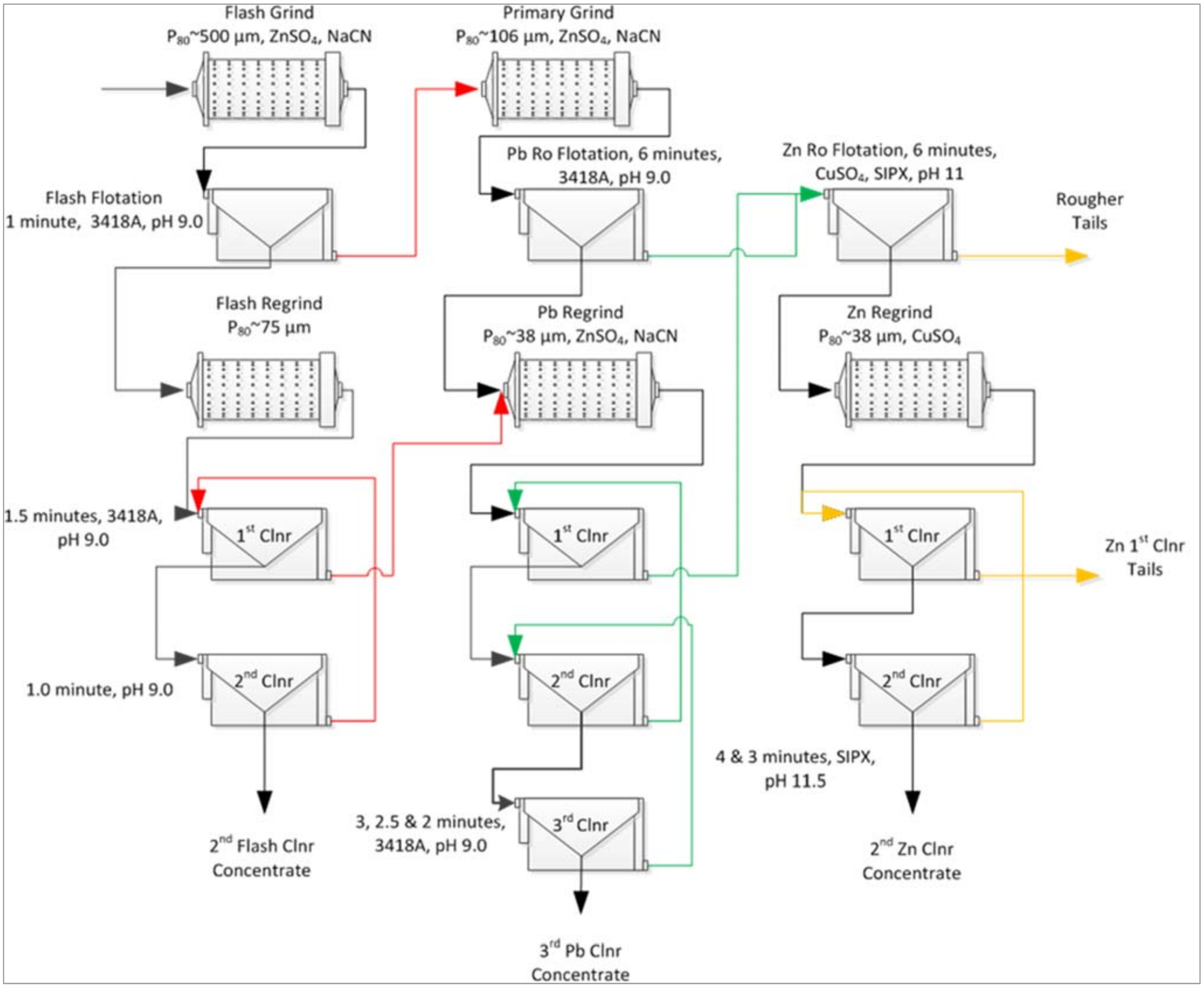


Table 13.7 Locked cycle metallurgical results

Composite	Products	Weight %	Assays, %		Assays, g/t	% distribution		
			Pb	Zn	Ag	Pb	Zn	Ag
Epitaph_10%	2nd Flash Cl Con	4.0	81.0	1.45	894	69.2	1.1	50.5
	3rd Lead Cl Con	2.3	50.2	5.61	806	24.3	2.4	25.7
	2nd Zinc Cl Con	8.7	0.97	57.4	138	1.8	93.3	16.8
Concha_10%	2nd Flash Cl Con	3.2	78.2	2.33	879	59.2	1.4	44.5
	3rd Lead Cl Con	2.6	46.0	10.4	758	28.5	5.0	31.4
	2nd Zinc Cl Con	7.1	1.11	56.8	103	1.9	74.1	11.6
Scherrer_5%	2nd Flash Cl Con	2.1	80.1	1.38	806	53.8	1.1	42.4
	3rd Lead Cl Con	2.5	44.4	7.31	587	34.3	6.8	35.6
	2nd Zinc Cl Con	3.3	1.83	57.5	114	1.9	70.9	9.2

"The flash cleaner flotation concentrates were very good across all tests with final grades all greater than 78.2% Pb. The final lead cleaner concentrate grades were poorer than expected across all tests at between 44.4% Pb and 50.2% Pb. This can be improved by rejecting more gangue material reporting to the concentrate by the use of a silicate dispersant or depressant, such as CMC or dextrin, and by generating a coarser regrind product. The final zinc concentrate from each locked cycle test was of very high quality with final grades ranging from 56.8% Zn to 57.5% Zn. Zinc recovery was high for the Epitaph_10% test at 93.3%. Zinc recovery was lower for the Concha_10% and Scherrer_5% composites, at 74.1% and 70.9% respectively, due to the presence of Zn-bearing silicates and oxides in the composites. Silver grades averaged between 550-900 g/t for the flash and lead concentrates and 100-150 g/t Ag for the zinc concentrate. The overall silver recovery ranged from 87.2% to 93.0%.

A series of batch cleaner tests were undertaken on composites 3, 4, 5, and 8 to prove the concentrate grade of the lead cleaner flotation circuit can be upgraded by coarsening the regrind size and adding the silicate depressant CMC. Two tests were also undertaken on composites 9 and 11 without regrind size optimization and depressant addition. The results are shown in Table 13.8.

Table 13.9 and Table 13.10 provide projected metallurgical results for Zn and Ag. The projected results are calculated by comparing the locked cycle test results to the best cleaner batch test results and applying those trends back to the individual composites of that mineral type. Full locked cycle tests under optimized conditions would be required to confirm results. Results from composites 1, 2, 6, and 7 are based on batch cleaner tests without lead cleaner optimization and the results are projected based on the above locked cycle results.

The flash concentrate returned Pb grades of over 79% in the optimized tests, with the highest grade at 81.2% Pb for Concha_15%. Coarsening the regrind and adding CMC to the last lead cleaner improved Pb grades to greater than 69% Pb. This helped to improve average Pb grade in the combined flash and lead concentrates to over 76% in each composite. Tests on Epitaph_10% and Composite 11 achieved combined concentrates of well over 79% Pb. Lead recovery was highest for Composite 11 at 97.6%, with the Epitaph composites ranging from 93.4% to 96.6% lead recovery. The high-grade Concha composite recovered 91.2% of the available lead while the average grade Concha and Scherrer composites recovered 87.7% and 88.1%, respectively.

The tests requiring further lead cleaner optimization graded between 75.7% and 82.2% Pb in the flash concentrate, while recovering 39% to 58% of the contained lead. The lead cleaner concentrates were all lower than expected ranging from 45.1% Pb to 56.2% Pb. The combined lead concentrates graded between 60.4% Pb and 71.6% Pb. Overall lead recoveries for the batch cleaner tests were low for the lower grade Concha composites, but over 91% for the low-grade Epitaph composites."

Table 13.8 Projected lead metallurgical results

Composite		Head grade	Flash con		Lead con		Combined Pb cons		
		Pb (%)	Grade	Recovery	Grade	Recovery	Mass	Grade	Recovery
			Pb (%)	Pb (%)	Pb (%)	Pb (%)	(%)	Pb (%)	Pb (%)
Composite 1	Concha_2.5%	1.21	79.1	39.2	55.4	43.4	1.7	67.3	82.6
Composite 2	Concha_5%	2.27	75.7	50.8	45.1	35.5	3.1	60.4	86.3
Composite 3	Concha_10%	4.35	81.0	59.2	69.4	28.5	4.8	76.8	87.7
Composite 4	Concha_15%	6.57	81.2	49.9	73.0	41.3	6.6	77.8	91.2
Composite 5	Scherrer_5%	3.02	80.1	53.7	70.8	34.3	3.7	76.2	88.1
Composite 6	Epitaph_2.5%	1.26	79.0	45.7	56.2	45.4	1.6	67.6	91.1
Composite 7	Epitaph_5%	2.62	82.2	58.0	55.2	36.8	3.4	71.6	94.8
Composite 8	Epitaph_10%	5.12	81.0	69.6	73.9	23.8	5.5	79.1	93.4
Composite 9	Epitaph_15%	8.25	79.0	90.7	48.2	5.9	8.9	77.7	96.6
Composite 11	High Pb - Low Zn	18.4	83.7	88.3	51.6	9.3	21.9	80.0	97.6

"Zinc results were very good for all of the average grade and high-grade composites. The Epitaph zinc concentrate grades ranged from 57.4% Zn to 59.8% Zn, with recoveries of 94.5% and 94.8%. The Concha and Scherrer composites also produced very high zinc concentrate grades ranging from 56.8% to 60.3%. However, the zinc recoveries were all in the mid 70%'s due to the presence of Zn-bearing silicates and oxides in those composites.

The zinc concentrates also performed well for the batch cleaner test results. All grades were over 55% Zn with the low-grade Concha composites recovering 71% and 71.2% of the zinc, respectively, and the low-grade Epitaph composites recovering over 90% of the contained zinc."

Table 13.9 Projected zinc metallurgical results

Composite		Head grade	Zinc concentrate		
		Zn (%)	Mass	Grade	Recovery
			(%)	Zn (%)	Zn (%)
Composite 1	Concha_2.5%	1.36	1.7	55.0	71.0
Composite 2	Concha_5%	3.02	3.7	56.6	71.2
Composite 3	Concha_10%	5.74	7.1	56.8	75.8
Composite 4	Concha_15%	8.10	6.3	60.3	76.7
Composite 5	Scherrer_5%	2.67	3.9	1.5	2.2
Composite 6	Epitaph_2.5%	1.47	2.4	58.9	90.3
Composite 7	Epitaph_5%	2.68	4.3	56.0	92.8
Composite 8	Epitaph_10%	5.69	8.8	57.4	94.5
Composite 9	Epitaph_15%	6.76	8.1	59.8	94.8

"The combined lead concentrate averaged 785 g/t Ag to 1,492 g/t Ag across the six optimized composites, while recovering between 73.4% and 85.3% of the available silver. The silver grade in the zinc concentrate ranged from 59 g/t Ag to 562 g/t Ag, while recovering an additional 7.6% to 17.9% of the available silver after the flash and lead flotation circuits. The overall silver recovery was good ranging from 86.2% to 95.3%.

The batch cleaner tests on the low-grade Concha and Epitaph composites were lower than expected at less than 77% in the combined lead concentrates. However, overall silver recovery was over 80% for each test."

Table 13.10 Projected silver metallurgical results

Composite		Head grade	Combined Pb con		Zinc con		Overall
		Ag (g/t)	Grade	Recovery	Grade	Recovery	Recovery
			Ag (g/t)	Ag (%)	Ag (g/t)	Ag (%)	Ag (%)
Composite 1	Concha_2.5%	25.0	1015	60.2	292	24.9	85.1
Composite 2	Concha_5%	41.0	989	76.7	75.6	8.8	85.5
Composite 3	Concha_10%	59.9	872	73.4	103	12.8	86.2
Composite 4	Concha_15%	115	973	84.2	59.2	7.9	92.1
Composite 5	Scherrer_5%	40.4	785	76.3	114	9.9	86.2
Composite 6	Epitaph_2.5%	16.8	658	62.7	99.2	20.0	82.7
Composite 7	Epitaph_5%	32.4	770	72.5	52.2	8.4	80.9
Composite 8	Epitaph_10%	74.8	911	74.7	138	17.9	92.6
Composite 9	Epitaph_15%	192	1,492	85.3	170	7.6	92.9
Composite 11	High Pb - Low Zn	195	852	84.7	562	10.6	95.3

13.6.2 Flotation tests – FLS

The objective of the FLS flotation test work was to further refine the primary grind size and flotation parameters. The master composite was used for all tests. Any refinements identified by this and future testing will be used to advance the project into the next stage of studies. The parameters evaluated were:

- Primary grind size
- Reagent type and dosages
- Regrind sizes

13.6.2.1 Reagent tests

Lead and zinc flotation tests were performed to assess reagents and dose rates. The tests were conducted at a targeted primary grind P_{80} 106 µm and natural pH (7.5 to 8). Lime was not used as a pyrite depressant in any of the tests all with positive results. Therefore, with respect to this PEA, lime was removed from the reagent scheme. Five g/t NaCN and 150 g/t $ZnSO_4{\bullet}7H_2O$ were added as part of grinding. Collectors evaluated as part of lead flotation were:

- Sodium ethyl xanthate (SEX).
- Sodium isopropyl xanthate (SIPX).
- Cytec 3418A (dialkyl dithiophosphinate).
- Cytec 242 (dithiophosphate).

Reagents evaluated as part of Zinc flotation were:

- Copper sulphate ($CuSO_4$ to $5H_2O$) sphalerite activator.
- Sodium cyanide (NaCN) pyrite depressant.
- Sodium ethyl xanthate (SEX) collector.
- Sodium isopropyl xanthate (SIPX) collector.
- Cytec 3418A (dialkyl dithiophosphinate) collector.

Based on the results, the following reagent scheme was selected:

- To the grind: NaCN / $ZnSO_4{\bullet}7H_2O$.
- Lead flotation (5 minutes): SEX + Cytec 242.
- Zinc flotation (6 minutes): NaCN + $CuSO_4{\bullet}5H_2O$ + Cytec 3418A.

13.6.2.2 Primary grind size

Using the above reagent scheme primary grind tests were performed to assess grind sizes ranging from P_{80} = 106 µm to 262 µm. The purpose of the grinding tests was to determine at what particle size distribution did mineral losses to the final tailing begin to appear. The results of these tests were used to modify the primary grind size for this PEA and future test work. As part of the grind size evaluations further modifications to reagent selection and doses were made. At completion of the primary grind size testing the following criteria was selected for performing additional testing on the MC sample:

- Primary grind P_{80} = 150 µm to 160 µm.
- 25 g/t NaCN and 150 g/t $ZnSO_4•7H_2O$, 30 g/t SEX (stage added to lead flotation).
- 5 g/t NaCN, 125 g/t $CuSO_4•5H_2O$ and 15 g/t Cytec 3418A for the zinc flotation stage.

13.6.2.3 Lead and zinc rougher kinetic tests

A series of rougher lead and zinc kinetic tests were performed at the test conditions determined above. The results show that 5 minutes for lead rougher flotation and 6 minutes zinc rougher flotation are sufficient for the MC sample.

13.6.2.4 Locked cycle test

Using the test conditions determined above a six-cycle locked-cycle test was conducted on a sample of MC. A metallurgical balance around the last three cycles (4 - 6) show No. 2 lead concentrate assayed 1,235 g/t silver and 73% lead, which corresponded to recoveries of 83% silver and 92% lead. The No. 2 zinc cleaner concentrate assayed 57% zinc which corresponded to 84% zinc recovery. Acid insol in the final lead and zinc concentrates was less than 4%.

A sample of No. 2 zinc cleaner concentrate was submitted for fluorine and chlorine analysis and trace element analysis. These assays / results are pending.

A summary of the locked cycle results is presented in Table 13.11. The flowsheet is shown in Figure 13.2.

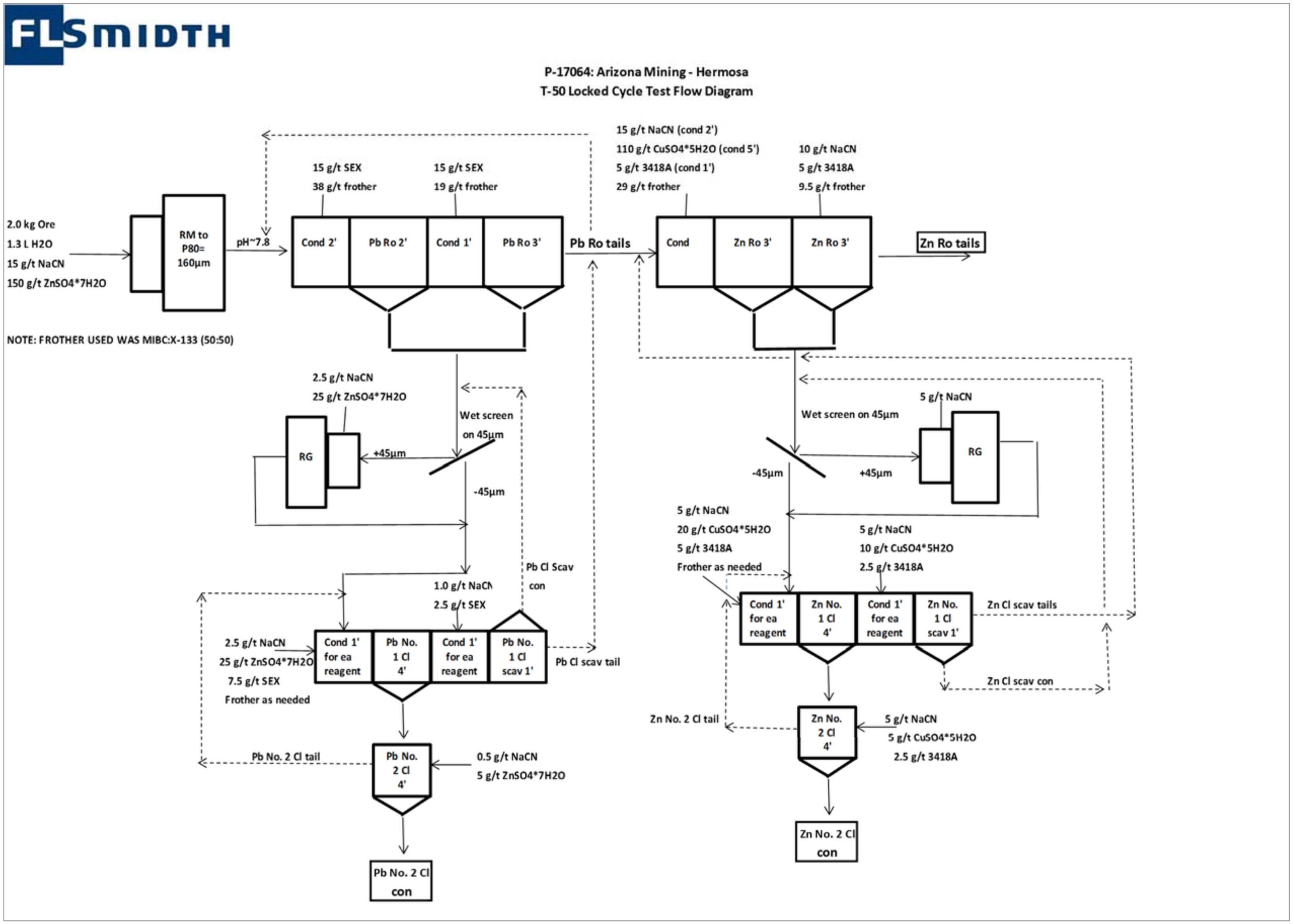
As only one locked cycle test has been completed the results should be considered very preliminary. Additional locked cycle testing is required.

Table 13.11 Locked cycle test results

Balanced cycles (4-6)			Ag, g/t	Pb, %	Cu, %	Zn, %	Fe, %	Ag	Pb	Cu	Zn	Fe
No. 2 Pb Cleaner Con	350.2	5.88	**1235.3**	**72.8**	1.29	5.5	2.55	**83.33**	**92.15**	60.27	6.63	2.89
No. 2 Zn Cleaner Con	428.4	7.20	112.3	2.21	0.46	**57.26**	3.58	9.26	3.42	26.42	**84.41**	4.97
Zn Ro / Scav Tails	5172.4	86.92	7.4	0.24	0.02	0.50	5.49	7.41	4.43	13.31	8.96	92.13
Total (calc)	5951.0	100.00	87.2	4.65	0.13	4.88	5.18	100.00	100.00	100.00	100.00	100.00
Total (assay)	6000.0		89.3	4.52	0.12	4.70	5.04					
			% Acid Insol	**Au, g/t**	**% S(T)**	**% Mn**						
No. 2 Pb Cleaner Con (4-6)			2.85	<0.2	16.0	-						
No. 2 Zn Cleaner Con (4-6)			3.73	<0.2	31.5	2.16						

Source: Table 15 in FLS report.

Figure 13.2 Locked Cycle flow diagram



Source: FLS report.

13.6.3 Thickening and filtration tests – FLS

Thickening test work shows that an ionic polyacrylamide flocculant with a very high molecular weight and medium charge density produces the best overflow clarity and settling velocities for the tailings sample. The recommended flocculant dosage is 25 grams of flocculant per metric tonne of dry solids (0.05 lb/ton). Optimum feedwell suspended solids concentration for flocculation is 10 to 12 weight%.

Additional testing is required in these areas.

13.7 Simplified flowsheet for the Taylor deposit – SGS

"The main objective of this test work program was to simplify the mineral processing flowsheet presented in Figure 13.2. The main change was to eliminate the flash flotation circuit which involved a rougher flash flotation step followed by a regrind on the flash flotation concentrate. The reground flash concentrate was then subjected to two stages of cleaning to generate a lead concentrate that consistently graded ~80% Pb.

The new test work has allowed the flowsheet to be reduced to just a lead circuit and a zinc circuit. Throughout the course of this program the main parameters investigated were lead circuit collector type and dosage, depressant dosage, and regrind applicability to the lead circuit.

Table 13.12 shows the head assays of the main composites used in the simplified flowsheet test work program."

Table 13.12 Composite head assays

Composite blend	Head assays (%)					Head assays (g/t)	
	Zn	Pb	Cu	Fe	S	Ag	Au
Composite 7	2.68	2.62	0.10	2.91	3.95	32.4	0.04
50:50 Composite 8 / Composite 9	5.97	5.81	0.15	3.42	6.60	109	0.15
20:40:40 Composite 6 / Composite 7 / Composite 9	3.97	4.36	0.12	2.89	4.77	93.1	0.04
10:45:45 Composite 1 / Composite 2 / Composite 4	4.96	3.79	0.13	3.16	4.55	72.5	0.04

"After several rougher and cleaner flotation tests, followed by a number of locked cycle flotation tests, and an extensive mineralogical analysis on some the lead and zinc concentrates produced, a simplified flowsheet was developed as shown in Figure 13.3.

The results obtained in the final test, locked cycle test 15 (LCT15), using the flowsheet and conditions shown in Figure 13.3 are tabulated in Table 13.13".

Figure 13.3 Simplified locked cycle flowsheet and conditions

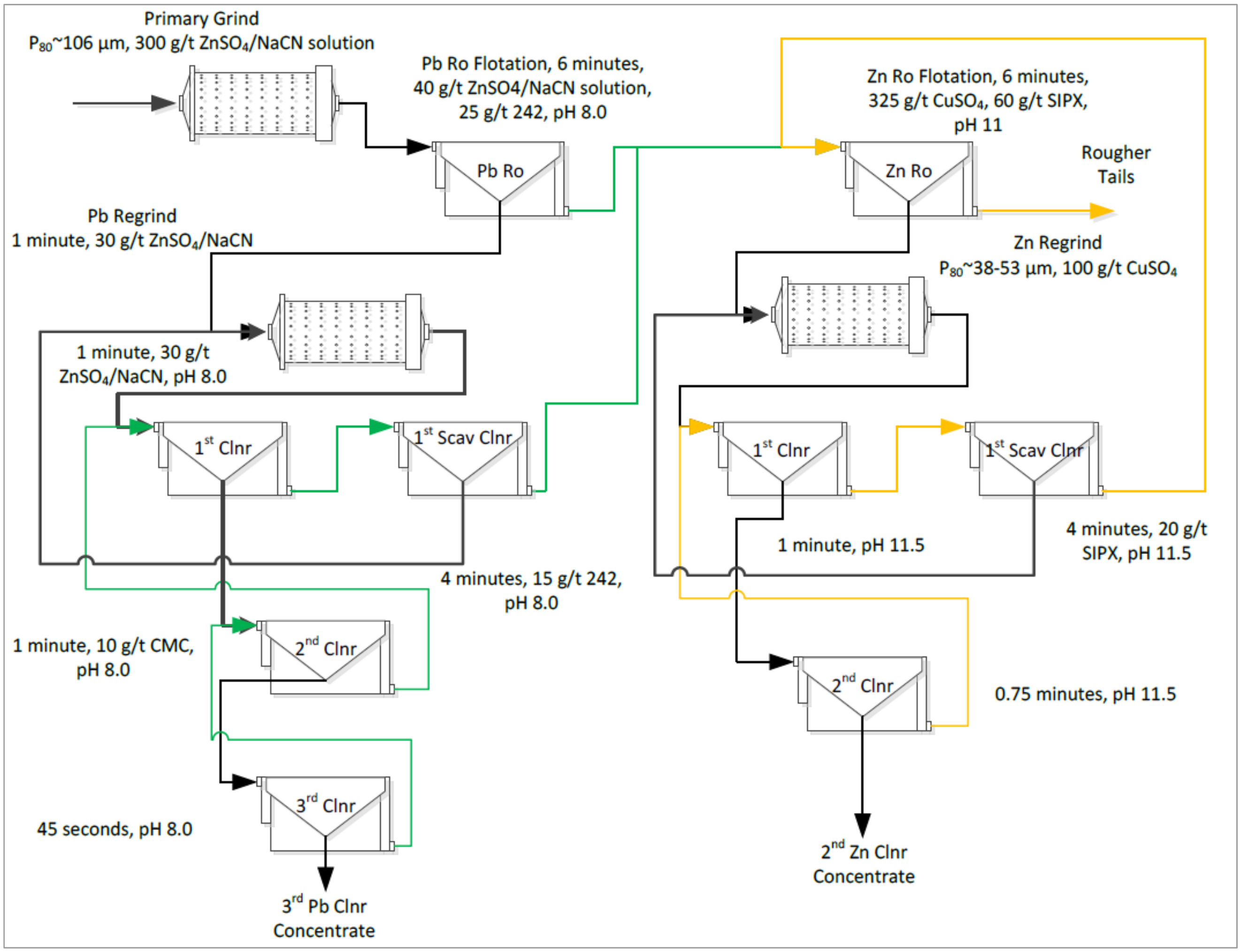


Table 13.13 Optimized locked cycle results

Products	Weight	Assays, %, g/t			% distribution		
	%	Zn	Pb	Ag	Pb	Zn	Ag
3rd Lead Cl Con	6.1	3.4	69.7	1,072	95.4	5.2	69.2
2nd Zinc Cl Con	6.7	56.1	1.03	331	1.5	92.7	23.2
Zinc Ro Tail	87.2	0.1	0.16	8.27	3.1	2.1	7.6
Head (Calc.)	100.0	4.03	4.49	95.0	100.0	100.0	100.0
Head (Direct)		3.97	4.36	93.1			

"The projected final lead concentrate graded 69.7% Pb and 1,072 g/t Ag at a lead recovery of 95.4% and a silver recovery of 69.2%. The final zinc concentrate graded 56.1% Zn at a zinc recovery of 92.7%. The overall silver recovery was 92.4%. The manganese content of the final zinc concentrate was 1.35% Mn."

13.8 Concentrate analysis – SGS

"The potential penalty elements associated with the lead and zinc concentrates produced in cycle F of LCT15 are shown in Table 13.14. The manganese content of 1.35% Mn in the zinc concentrate is of potential concern."

Table 13.14 Potential penalty element analysis for LCT15 concentrates

Element	Unit	3rd Pb Cl Con F	2nd Zn Cl Con F
Hg	g/t	3.3	6.0
Fe	g/t	17,100	31,900
Mn	g/t	4,530	13,500
Cd	g/t	-	1,540
Si	g/t	4.42	2.92

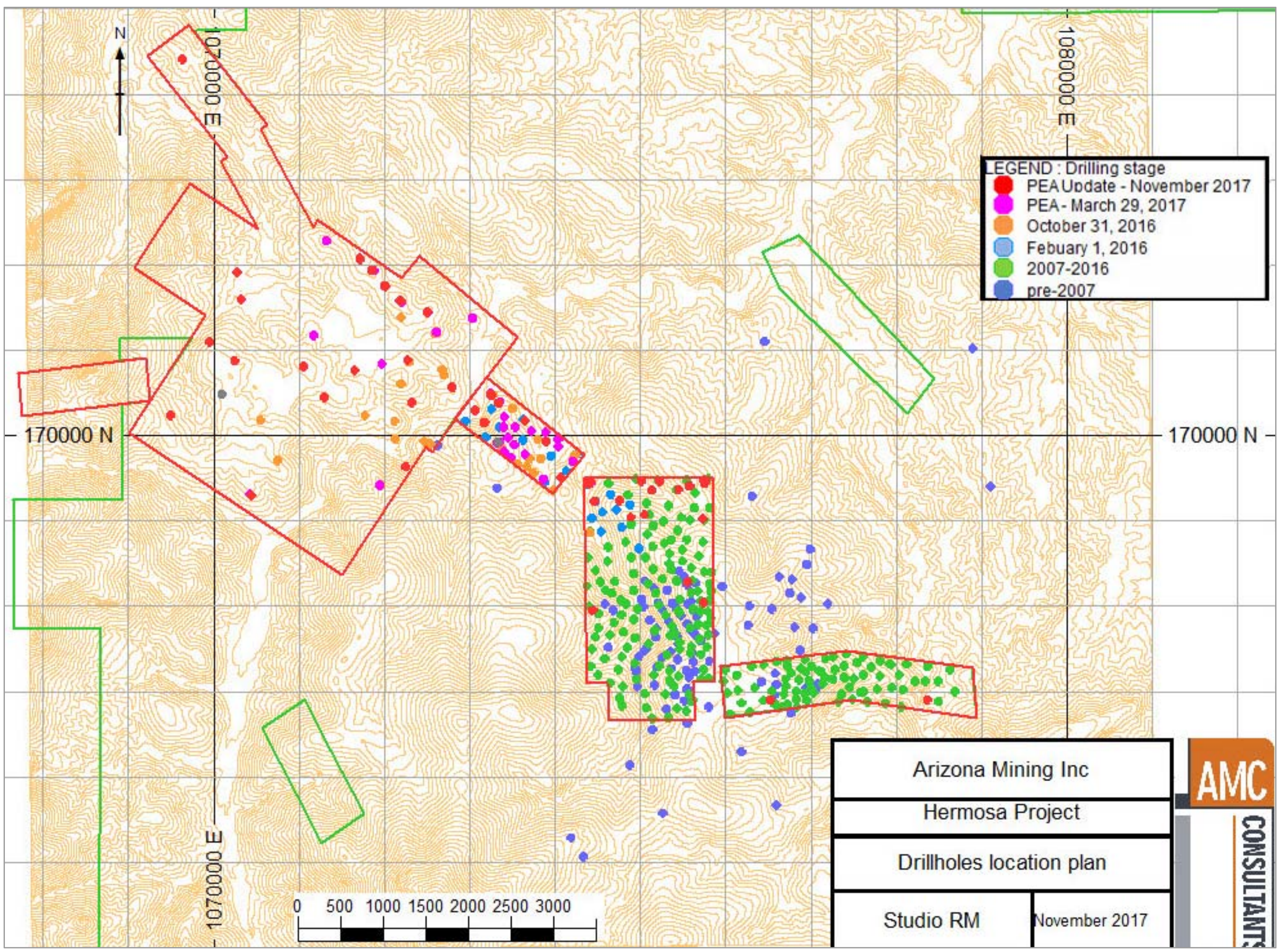
14 Mineral Resource estimates

14.1 Introduction

This Mineral Resource estimate is an update of the estimate reported in the March 2017 Technical Report. The database provided for the Property, for this estimate including historic holes, contains 474 drillholes with 160,654 samples. The updated Mineral Resource estimation is based on 429 surface drillholes with 144,439 samples. AZ provided wireframes of major lithological units, and gradeshells of the main mineralized domains, in dxf format, together with drillhole locations, downhole surveys, assays, and geology as csv data files. Ms Dinara Nussipakynova, P.Geo., a Principal Geologist of AMC, completed the Mineral Resource estimate using Datamine^{TM} software.

The dataset upon which the current Mineral Resource is based includes additional data from 65 holes (234,513.5 aggregate feet), that were drilled since the March 2017 Mineral Resource estimate. Figure 14.1 is a plan view of all drillhole locations and highlighting those holes drilled subsequent to the March 2017 estimate.

Figure 14.1 Plan view of Taylor and Central Deposit drillholes



The Mineral Resource is comprised of sulphide and oxide domains; the sulphide domains comprise the Taylor Deposit and include the Concha, Scherrer, Epitaph and Taylor Deeps Zones, Trench Vein System, and Hardshell domains. The oxide domains comprise the Central Deposit which consists of the Upper Silver zone (LAG), and the Manto Oxide zone (MOX).

Table 14.1 is a summary of the Mineral Resources for the Taylor Deposit stated at 30 November 2017. Table 14.2 is a summary of the Mineral Resources for the Central Deposit stated at 30 November 2017. Detailed tables for both deposits follow in Section 14.8.

Table 14.1 Taylor Deposit Mineral Resources

Classification	Million tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Measured	15.2	4.0	4.0	1.6	9.6
Indicated	85.8	4.2	4.3	2.2	10.5
Measured and Indicated	101.0	4.1	4.3	2.1	10.4
Inferred	43.6	3.9	4.8	3.4	11.9

CIM Definition Standards (2014) were used for reporting the Mineral Resources.
Mineral Resources are reported as of 30 November 2017.
Stated at a cut-off grade of 4% ZnEq based on prices, recovery, and costs as follows:
- Prices of $1.00/lb for zinc, $0.95/lb for lead, and $20.00/oz for silver.
- Average processing recovery factors of 92% for zinc, 95% for lead, and 90% for silver.
- Total operating costs are estimated to be of the order of $60/ton.

ZnEq calculation is discussed in Section 14.6.3.
Numbers are rounded and may not match later detailed tables.

Table 14.2 Central Deposit Mineral Resources

Classification	Million tons	Zn (%)	Ag (oz/ton)	Mn (%)	Oxval ($/ton)
Measured	21.8	1.9	3.3	9.2	262
Indicated	41.7	2.3	1.7	9.8	257
Measured and Indicated	63.5	2.2	2.3	9.6	259
Inferred	1.8	2.6	1.6	7.4	207

CIM Definition Standards (2014), were used for reporting the Mineral Resources.
Mineral Resources are reported as of 30 November 2017.
Stated at a cut-off grade of $100/ton Oxval based on prices, recovery, and costs as follows:
- Prices of $1.00/lb for zinc, $20.00/oz for silver, and $0.91/lb for manganese.
- Average processing recovery factors of 55% for zinc, 72% for silver, and 86% for manganese.
- Total operating costs are estimated to be on the order of $100/ton.

Oxval calculation is discussed in Section 14.6.3.
Numbers are rounded and may not match later detailed tables.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Neither deposit is materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, political or other relevant issues. The estimates of Mineral Resources may be affected if mining, metallurgical, or infrastructure factors change from those currently anticipated at the Property.

14.2 Exploratory data analysis

14.2.1 Assays

AMC received a dataset in csv format from AZ with a closing date of 17 October 2017, that included data for drillhole collars, downhole surveys, assays for copper, lead, zinc, silver and manganese, and lithology. The dataset as received contained assay data for a total of 160,654 samples from 474 drillholes. However, only 429 of those drillholes with 24,849 samples are contained in the domains within which the Mineral Resource has been estimated. Descriptive statistics of the assays employed in the Mineral Resource estimate are presented in Table 14.3

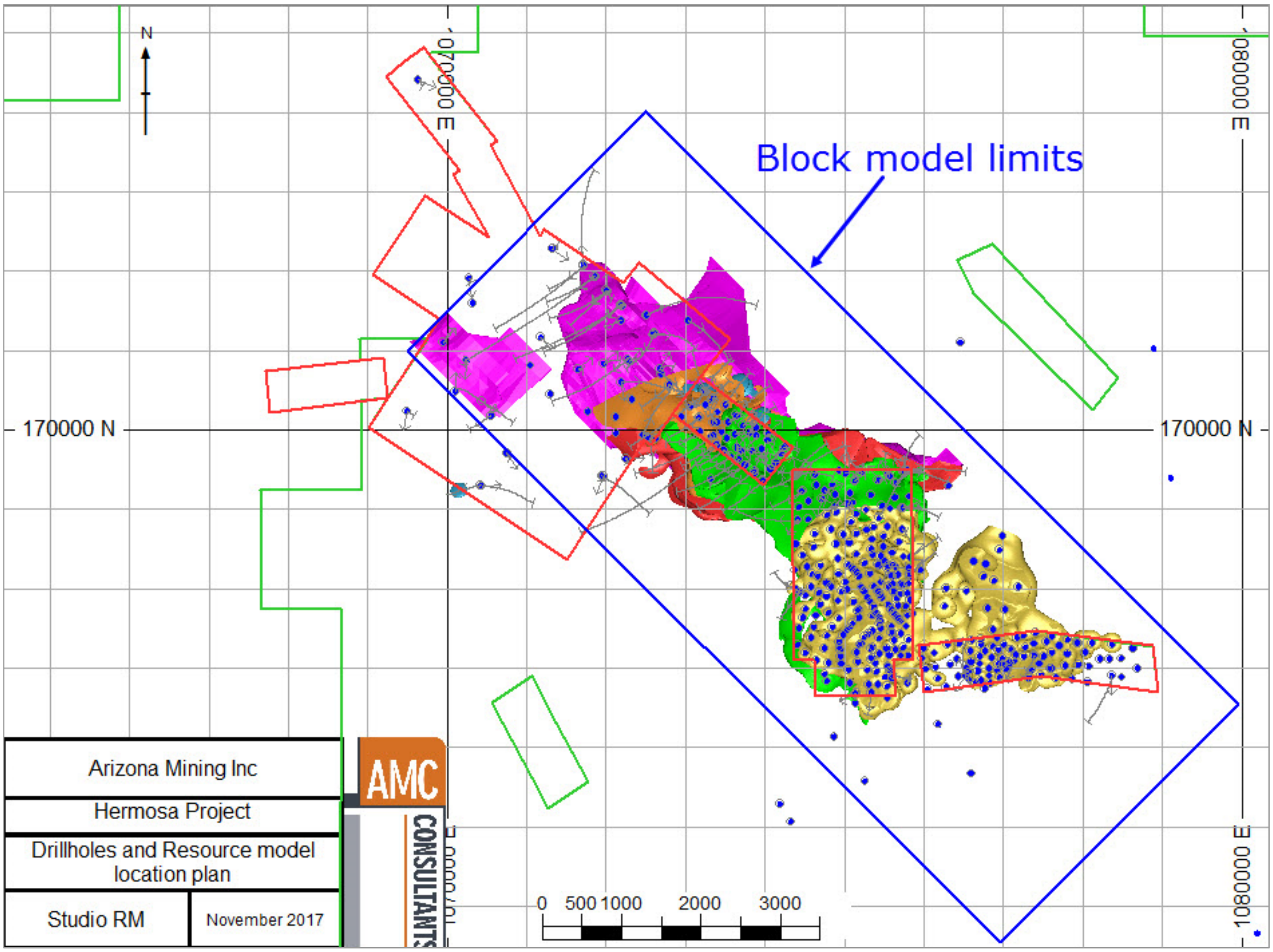
Table 14.3 Statistics of selected raw drillhole data

LAG					
Field	**Pb (%)**	**Zn (%)**	**Ag (oz/ton)**	**Cu (%)**	**Mn (%)**
NSamples	6,177	6,177	6,177	6,177	6,177
Minimum	0.00	0.00	0.00	0.00	0.00
Maximum	41.02	10.71	56.2	1.14	27.38
Mean	0.23	0.05	1.35	0.02	0.49
Variance	0.71	0.07	3.86	0.00	3.46
Standard deviation	0.84	0.27	1.96	0.04	1.86
CV	3.59	5.03	1.45	2.10	3.82
MOX					
Field	**Pb (%)**	**Zn (%)**	**Ag (oz/ton)**	**Cu (%)**	**Mn (%)**
NSamples	7,400	7,400	7,400	7,400	7,400
Minimum	0.00	0.00	0.00	0.00	0.00
Maximum	27.85	43.84	93.33	1.82	39.8
Mean	1.19	1.91	2.51	0.08	8.14
Variance	4.57	10.96	26.84	0.02	61.93
Standard deviation	2.14	3.31	5.18	0.14	7.87
CV	1.70	1.52	1.67	4.77	0.66
Concha					
Field	**Pb (%)**	**Zn (%)**	**Ag (oz/ton)**	**Cu (%)**	**Mn (%)**
NSamples	1,902	1,902	1,902	1,902	1,902
Minimum	0.00	0.00	0.01	0.00	0.01
Maximum	64.95	45.17	34.56	16.50	18.32
Mean	3.72	4.81	1.74	0.13	5.86
Variance	39.95	53.29	8.46	0.36	14.78
Standard deviation	6.32	7.30	2.91	0.60	3.84
CV	1.70	1.52	1.67	4.77	0.66
Sherrer					
Field	**Pb (%)**	**Zn (%)**	**Ag (oz/ton)**	**Cu (%)**	**Mn (%)**
NSamples	1,305	1,305	1,305	1,305	1,305
Minimum	0.000	0.003	0.007	0.000	0.002
Maximum	82.79	45.03	55.71	5.02	22.27
Mean	2.20	2.40	1.13	0.08	3.64
Variance	24.58	22.43	11.49	0.09	10.40
Standard deviation	4.96	4.74	3.39	0.30	3.22
CV	2.25	1.98	3.00	3.55	0.89
Epitaph					
Field	**Pb (%)**	**Zn (%)**	**Ag (oz/ton)**	**Cu (%)**	**Mn (%)**
NSamples	3,892	3,892	3,892	3,892	3,892
Minimum	0.0001	0.0011	0.0015	0.0001	0.0026
Maximum	55.94	35.1	45.06	1.93	11.1
Mean	2.30	2.32	0.90	0.04	2.16
Variance	17.01	17.76	3.67	0.01	4.37
Standard deviation	4.12	4.21	1.92	0.10	2.09
CV	1.79	1.82	2.12	2.73	0.97

Taylor Deeps					
Field	**Pb (%)**	**Zn (%)**	**Ag (oz/ton)**	**Cu (%)**	**Mn (%)**
NSamples	3,175	3,175	3,175	3,175	3,175
Minimum	0.0002	0.001	0.0073	0.0001	0.0113
Maximum	56.96	41.61	201.54	6.63	11.7
Mean	2.83	2.01	1.94	0.14	2.20
Variance	36.28	18.02	37.54	0.16	5.58
Standard deviation	6.02	4.25	6.13	0.40	2.36
CV	2.13	2.11	3.16	2.96	1.07
Trench Vein System					
Field	**Pb (%)**	**Zn (%)**	**Ag (oz/ton)**	**Cu (%)**	**Mn (%)**
NSamples	998	998	998	998	998
Minimum	0.0003	0.0016	0.0073	0.0001	0.0151
Maximum	33.1	39	30.77	1.735	12.93
Mean	1.66	2.37	1.79	0.06	1.71
Variance	12.16	24.75	10.93	0.02	6.04
Standard deviation	3.49	4.97	3.31	0.14	2.46
CV	2.10	2.10	1.84	2.24	1.43

The outline of the block model, surface expression of the domains and location of the drillholes are shown in plan view in Figure 14.2.

Figure 14.2 Plan view of drillholes and boundary of block model



14.2.2 Capping

Log probability plots of copper, lead, zinc, silver, and manganese assays were examined for evidence of statistical outliers. Capping was applied as required. Capping levels are shown in Table 14.4 below. If a domain is not listed it was decided that capping was not warranted.

Table 14.4 Grade capping summary

Domain	Top cut	Original mean	New mean	Number of samples top cut	% difference
Silver (oz/ton)					
LAG	30	1.35	1.35	30	0.0
MOX	40	2.51	2.46	20	2.0
Concha	20	1.74	1.72	6	1.1
Sherrer	20	1.13	1.04	11	8.0
Epitaph	20	0.9	0.89	5	1.1
Taylor Deeps	40	1.94	1.82	17	6.2
Copper (%)					
LAG	1.0	0.02	0.02	21	0.0
MOX	1.2	1.82	1.2	7	34.1
Concha	2.0	0.13	0.11	5	15.4
Sherrer	1.5	0.08	0.08	14	0.0
Epitaph	0.8	0.04	0.04	14	0.0
Taylor Deeps	1.8	0.14	0.12	38	14.3
Manganese (%)					
LAG	na	0.49	0.49	26	0.0
MOX	35	8.14	8.14	4	0.0
Concha	17	5.86	5.86	6	0.0
Sherrer	17	3.64	3.63	2	0.3
Epitaph	10	2.16	2.16	3	0.0
Taylor Deeps	na	2.2	2.2	0	0.0
Lead (%)					
LAG	20	0.23	0.23	34	0.0
MOX	22	1.19	1.19	4	0.0
Concha	30	3.72	3.65	15	1.9
Sherrer	22	2.2	2.06	13	6.4
Epitaph	28	2.3	2.28	11	0.9
Taylor Deeps	40	2.83	2.81	10	0.7
Zinc (%)					
LAG	27	0.05	0.05	19	0.0
MOX	35	1.91	1.9	9	0.5
Concha	23	4.81	4.8	8	0.2
Sherrer	26	2.4	2.3	17	4.2
Epitaph	30	2.32	2.3	13	0.9
Taylor Deeps	3	2.01	2.01	3	0.0

14.2.3 Composites

The majority of samples are five (5) feet in length but because the anticipated stope height is on the order of 60 or 100 ft, resolution of data at a scale of five feet in the vertical direction was considered unnecessarily fine. For that reason, samples from the LAG, (Upper Silver Zone), MOX (Mantos Oxide Zone), Concha, Scherrer Epitaph, and Taylor Deeps domains were composited to 10 ft in length. In comparison to the other domains, the Trench Vein System and Sub-Taylor Deeps domains are relatively narrow for which reason samples from these domains were composited to a nominal five-foot length. In practice, the length of these composites was adjusted to completely fill the sample length so that the exact five-foot length was obtained only in cases in which the samples spanned a distance evenly divisible by five feet. Compositing honoured the mineralized domain boundaries. The 24,849 samples within the volume of the mineralized domains were reduced to 13,259 composites.

It should also be noted that as a result of older drillholes in the Central Deposit only being assayed for silver, there are about 300 missing assays of manganese and zinc. In this estimate absent data has been treated conservatively and has been replaced by zeros.

The raw, capped, and composited assay data for the mineralized domains are shown in Table 14.5.

Table 14.5 Statistics of raw, capped, and composited assay data

Domain	Statistic	Ag (oz/ton)			Cu (%)			Mn (%)			Pb (%)			Zn (%)		
		Raw	Capped	Composite	Raw	Capped	Composite	Raw	Capped	Composite	Raw	Capped	Composite	Raw	Capped	Composite
LAG	No. samples	6,177	6,177	3,472	6,177	6,177	3,472	6,177	6,177	3,472	6,177	6,177	3,472	6,177	6,177	3,472
	Minimum	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
	Maximum	56.20	30.00	22.26	1.14	1.00	0.88	27.38	27.38	24.30	41.02	20.00	18.49	10.71	8.00	6.87
	Mean	1.35	1.35	1.35	0.02	0.02	0.02	0.49	0.49	0.49	0.23	0.23	0.23	0.05	0.05	0.05
	Stand.Dev	1.96	1.86	1.62	0.04	0.04	0.03	1.86	1.86	1.67	0.84	0.72	0.64	0.27	0.26	0.23
	CV	1.92	1.67	1.20	2.27	2.15	1.81	2.08	2.08	3.44	2.47	2.26	2.75	3.09	2.87	4.26
MOX	No. samples	7,400	7,400	3,804	7,400	7,400	3,804	7,400	7,400	3,804	7,400	7,400	3,804	7,400	7,400	3,804
	Minimum	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
	Maximum	93.33	40.00	40.00	1.82	1.20	1.11	39.80	35.00	33.69	27.85	22.00	19.33	43.84	27.00	25.80
	Mean	2.51	2.46	2.46	0.08	0.08	0.08	8.14	8.14	8.14	1.19	1.19	1.19	1.91	1.90	1.90
	Stand.Dev	5.18	4.57	4.09	0.14	0.13	0.12	7.87	7.87	7.23	2.14	2.11	1.92	3.31	3.22	2.95
	CV	2.05	1.84	1.66	1.75	1.73	1.58	0.95	0.94	0.89	1.77	1.75	1.62	1.71	1.67	1.55
Concha	No. samples	1,902	1,902	941	1,902	1,902	941	1,902	1,902	941	1,902	1,902	941	1,902	1,902	941
	Minimum	0.01	0.01	0.01	0.00	0.00	0.00	0.01	0.01	0.03	0.00	0.00	0.01	0.00	0.00	0.03
	Maximum	34.56	20.00	18.56	16.50	2.00	1.69	18.32	17.00	16.61	64.95	30.00	30.00	45.17	35.00	34.45
	Mean	1.74	1.72	1.72	0.13	0.11	0.11	5.86	5.86	5.86	3.72	3.65	3.65	4.81	4.80	4.80
	Stand.Dev	2.91	2.79	2.37	0.60	0.21	0.16	3.84	3.84	3.44	6.32	5.88	5.20	7.30	7.25	6.52
	CV	1.67	1.62	1.37	4.77	1.97	1.49	0.66	0.66	0.59	1.70	1.61	1.42	1.52	1.51	1.36
Sherrer	No. samples	1,305	1,305	641	1,305	1,305	641	1,305	1,305	641	1,305	1,305	641	1,305	1,305	641
	Minimum	0.007	0.007	0.009	0.000	0.000	0.000	0.002	0.002	0.076	0.000	0.000	0.002	0.003	0.003	0.008
	Maximum	55.71	20.00	15.88	5.02	1.50	1.42	22.27	17.00	15.77	82.79	22.00	20.79	45.03	23.00	19.98
	Mean	1.13	1.04	1.04	0.08	0.08	0.08	3.64	3.63	3.63	2.20	2.06	2.07	2.40	2.30	2.31
	Stand.Dev	3.39	2.44	1.86	0.30	0.21	0.16	3.22	3.20	2.84	4.96	3.87	3.04	4.74	4.13	3.25
	CV	3.00	2.36	1.79	3.55	2.82	2.14	0.89	0.88	0.78	2.25	1.88	1.47	1.98	1.80	1.41
Epitaph	No. samples	3,892	3,892	1,885	3,892	3,892	1,885	3,892	3,892	1,885	3,892	3,892	1,885	3,892	3,892	1,885
	Minimum	0.002	0.002	0.007	0.000	0.000	0.000	0.003	0.003	0.019	0.000	0.000	0.001	0.001	0.001	0.002
	Maximum	45.06	20.00	17.56	1.93	0.80	0.74	11.10	10.00	10.00	55.94	28.00	28.00	35.10	26.00	26.00
	Mean	0.90	0.89	0.89	0.04	0.04	0.04	2.16	2.16	2.16	2.30	2.28	2.28	2.32	2.30	2.30
	Stand.Dev	1.92	1.71	1.39	0.10	0.09	0.07	2.09	2.09	1.81	4.12	3.90	3.20	4.21	4.10	3.38
	Coeff. var	2.12	1.92	1.57	2.73	2.54	2.01	0.97	0.97	0.84	1.79	1.71	1.41	1.82	1.78	1.47

Domain	Statistic	Ag (oz/ton)			Cu (%)			Mn (%)			Pb (%)			Zn (%)		
		Raw	Capped	Composite	Raw	Capped	Composite	Raw	Capped	Composite	Raw	Capped	Composite	Raw	Capped	Composite
Taylor Deeps	No. samples	3,175	3,175	1,555	3,175	3,175	1,555	3,175	3,175	1,555	3,175	3,175	1,555	3,175	3,175	1,555
	Minimum	0.007	0.007	0.007	0.000	0.000	0.000	0.011	0.011	0.019	0.000	0.000	0.000	0.001	0.001	0.002
	Maximum	201.54	40.00	40.00	6.63	1.80	1.80	11.70	11.70	10.00	56.96	40.00	35.91	41.61	30.00	28.36
	Mean	1.94	1.82	1.82	0.14	0.12	0.12	2.20	2.20	2.20	2.83	2.81	2.81	2.01	2.01	2.01
	Stand.Dev	6.13	4.42	3.65	0.40	0.30	0.25	2.36	2.36	2.04	6.02	5.87	5.14	4.25	4.21	3.60
	CV	3.16	2.43	2.01	2.96	2.46	2.02	1.07	1.07	0.93	2.13	2.09	1.83	2.11	2.09	1.79
Trench veins system	No. samples	697		660	697		660	697		660	697		660	697		660
	Minimum	0.01		0.01	0.00		0.00	0.02		0.02	0.00		0.00	0.00		0.00
	Maximum	30.77		23.45	1.74		1.13	10.00		10.00	33.10		30.24	39.00		37.71
	Mean	1.74		1.74	0.07		0.07	1.00		1.00	1.69		1.69	2.14		2.14
	Stand.Dev	3.37		2.91	0.14		0.12	1.54		1.47	3.72		3.51	4.87		4.57
	CV	1.94		1.68	2.12		1.80	1.53		1.47	2.20		2.07	2.27		2.13
Hardshell	No. samples	301		301	301		301	301		301	301		301	301		301
	Minimum	0.01		0.01	0.00		0.00	0.03		0.03	0.00		0.00	0.00		0.00
	Maximum	21.06		21.06	1.24		1.24	12.93		12.93	18.85		18.85	32.80		32.80
	Mean	1.92		1.92	0.06		0.06	3.29		3.29	1.58		1.58	2.87		2.87
	Stand.Dev	3.17		3.17	0.15		0.15	3.25		3.25	2.91		2.91	5.17		5.17
	CV	1.65		1.65	2.54		2.54	0.99		0.99	1.84		1.84	1.80		1.80

14.3 Bulk density

AZ has collected a total of 1,462 bulk density measurements from both mineralized and un-mineralized intervals from 93 drillholes throughout the deposit. These measurements were made on pieces of whole drill core and provide an indication of expected values and potential range of values for un-mineralized rock and a range of concentrations of mineralization. However, because the bulk density of mineralized rock varies significantly in proportion to the abundance of galena and sphalerite, it is not possible to apply fixed values when computing tonnage. For that reason, a formula to estimate bulk density during the resource tabulation process was devised on the basis of abundance of galena, sphalerite and chalcopyrite. Table 14.6 sets out the parameters used for the bulk density estimation. This formula produces bulk density values within approximately 10% of the measurements carried out on the drill core. Because the estimation was carried out in Imperial units, it was necessary to convert bulk density to tonnage factor (cubic feet/ton). That conversion is also included in Table 14.6. The formula, in its reduced form is:

TF= (((Pb%*0.0862)+(Zn%*0.0597)+(Cu%*0.12))+((100-Pb%-Zn%-Cu%)*0.027)*0.031)

Table 14.6 Tonnage factor calculation

Element	% of mineral	Mineral	SG of mineral
Pb	87	Galena	7.5
Zn	67	Sphalerite	4.0
Cu	35	Chalcopyrite	4.2
Hostrock			2.7

SG units g/cm³
Bulk density to ft³/ton = 62.43 lbs/ft³/2000 lbs
Example of calculation of formula terms:
SG of Galena = (Pb%/0.87)*(7.5/100) = Pb%*0.0862
TF = (((Pb%*0.0862)+(Zn%*0.0597)+(Cu%*0.12)+((100-Pb%-Zn%-Cu%)*0.027))*0.031)

For the Central Deposit oxide domains, a fixed tonnage factor was used. For the LAG domain it was 0.0821t/ft^3 and for the MOX domain it was 0.0843 t/ft^3.

14.4 Geological interpretation and mineralization domains

The estimation has been carried out within 30 mineralization domains. They have been grouped based on lithological location and mineralization type into eight areas. Vertical sulphide veins within Mesozoic volcanics are termed the Trench Vein System and consist of 10 separate mineralization domains (veins). Two horizontal sulphide domains hosted in volcanics are referred to as the Hardshell domains. There are three main carbonate units of Paleozoic age; from deepest to shallowest they are Epitaph, Scherrer and Concha. The mineralized domains have been given the same names. There is an underlying thrust contact between the Epitaph and overthrust younger volcanics, and the zones below that contact are termed the Taylor Deeps and are modelled separately. In that geological area, there are 14 separate mineralization domains.

All the previously mentioned areas (Trench Vein System, Hardshell, Epitaph, Scherrer, Concha, and Taylor Deeps) comprise the sulphide portion of the deposit collectively termed the Taylor Deposit. The Central Deposit, which lies up-dip of the Taylor Deposit and contains oxide mineralization, is comprised of the domains: LAG and MOX.

The Mineral Resource estimate has been constrained by gradeshells for the eight areas listed in the preceding paragraph. The gradeshells, with the exception of Taylor Deeps, Trench Veins, and Hardshell were constructed using Leapfrog software and were constrained as follows (from uppermost to lowermost):

- LAG: Oxide, within the Hardshell volcanics, 0.5 ounces per ton silver, and clipped against the MOX.
- MOX: Oxide: 1% manganese equivalent, within the lower contact of the Concha unit to constrain the base of the domain. Manganese equivalent = ((Mn%/100)*1.22*0.95)+(Zn%/100)*1*0.95)+(Ag oz/st*20*0.95)) /(1.22*0.95).
- Concha, Scherrer, and Epitaph: Sulphide, lithological domain, 0.5% zinc equivalent.

The Taylor Deeps mineralization domains were built by AMC using Datamine[TM] software. Domains were constrained using, 0.5% zinc equivalent, plus +/- 150 ft of the thrust contact between Epitaph and lower volcanic package.

Mineralization domains are shown in plan and long section in Figure 14.3 and Figure 14.4.

The Trench Vein System and Hardshell domains, provided by AZ, were modelled visually using conventional wireframes. Wireframes were reviewed and accepted by AMC. It should be noted that Mineral Resources of both the Trench Veins and Hardshell domains have been incorporated with those of the Taylor Deeps domain for reporting purposes in Table 14.15.

Figure 14.3 3D view of mineralization domains coloured by area

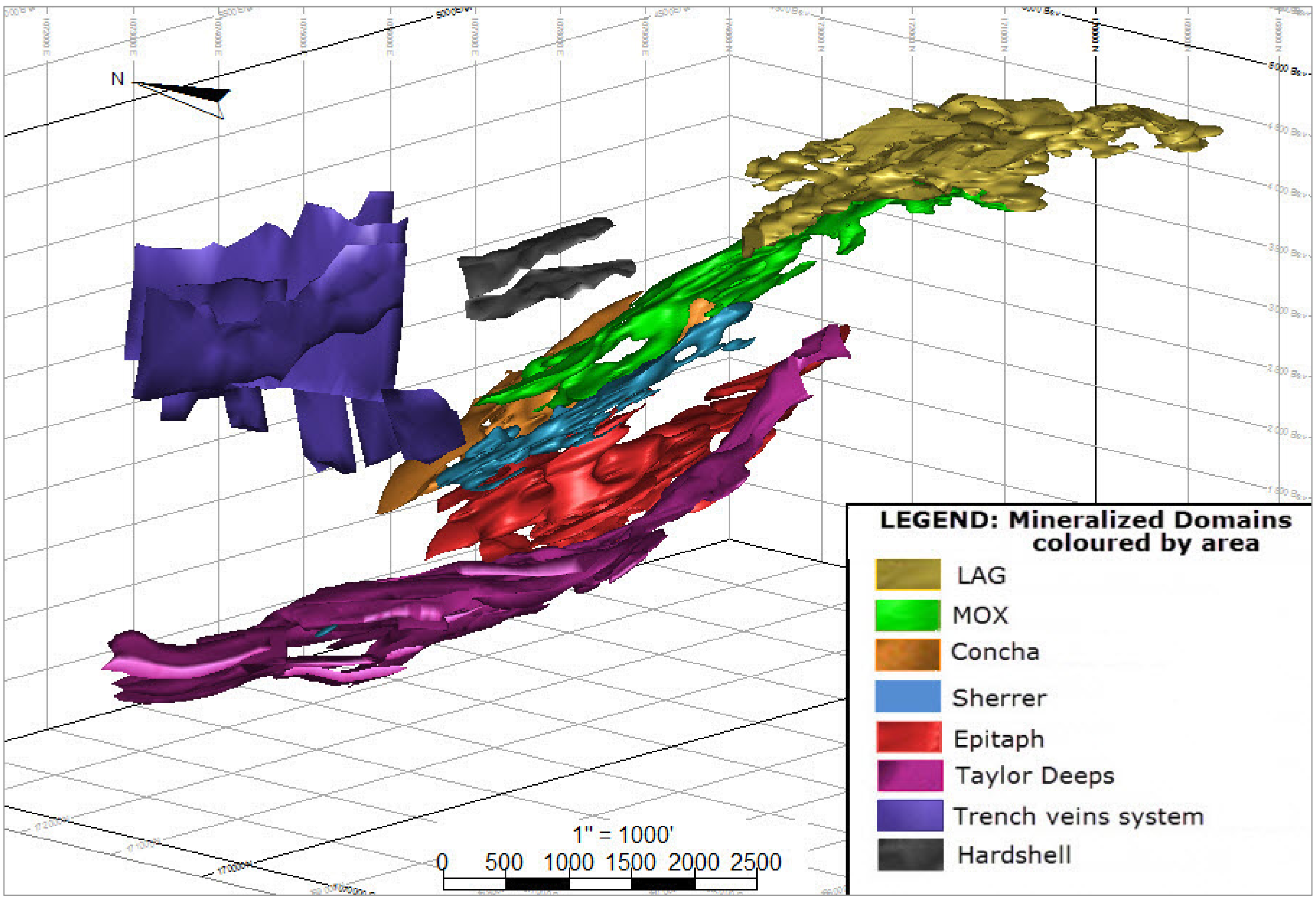
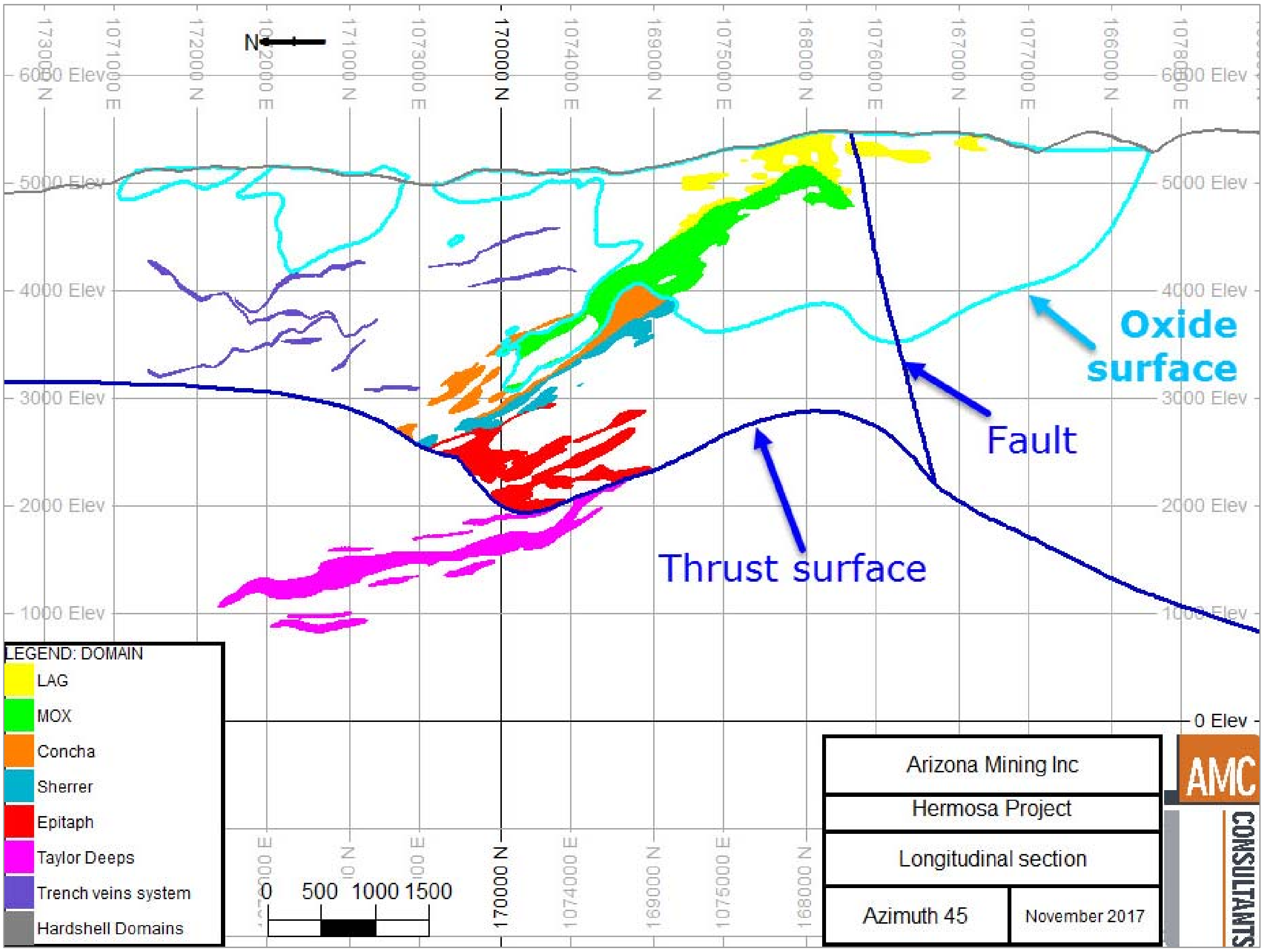


Figure 14.4 Mineralized domains in long section coloured by area



Note: Hardshell domains are off section and not shown on the figure.

14.5 Spatial analysis

Spatial continuity of mineralization (assays of silver, copper, lead, zinc, and manganese) was assessed using Datamine™ variographic software, with results shown in Table 14.7. The models are all spherical and have two structures.

Table 14.7 Variograms by area

Domain	Metal	Rotation Z	Rotation Y	Rotation X	Nugget	Range X1	Range Y1	Range Z1	C1	Range X2	Range Y2	Range Z2	C2
LAG	Ag	135	25	-15	0.263	83.3	83.3	7.7	1.18	228.8	260.0	20.0	1.18
	Cu	135	25	-15	0.001	116.7	155.6	18.1	0.00	233.3	309.7	34.7	0.01
	Mn	135	25	-15	0.299	136.4	171.2	24.2	0.11	221.2	292.4	34.8	2.58
	Pb	135	25	-25	0.123	56.1	72.7	13.6	0.43	218.2	262.1	47.0	0.68
	Zn	135	25	-15	0.131	120.8	142.5	15.5	0.74	283.4	320.6	31.0	0.44
MOX	Ag	135	25	-25	1.81	84.0	101.2	7.0	6.83	265.0	330.8	23.3	9.46
	Cu	135	25	-25	1.81	56.9	82.6	8.9	4.85	163.0	235.6	24.4	11.44
	Mn	135	25	-25	5.295	108.5	151.6	16.0	14.99	219.4	319.2	35.7	32.67
	Pb	135	25	-25	0.369	116.7	124.7	9.9	1.22	376.9	382.6	29.8	2.10
	Zn	135	25	-25	0.885	86.3	143.9	11.5	4.01	176.2	300.5	21.9	3.96
Concha	Ag	135	15	-25	0.56	129.0	175.0	18.0	0.77	251.0	350.0	34.0	4.24
	Cu	135	15	-25	0.003	116.7	157.2	19.3	0.01	233.3	301.0	35.1	0.01
	Mn	135	15	-25	1.173	116.7	153.4	13.2	3.24	233.3	291.3	27.6	7.32
	Pb	135	15	-25	2.69	73.6	115.9	10.9	11.42	202.8	302.9	26.6	12.75
	Zn	135	15	-25	4.247	77.1	109.4	13.7	18.25	233.3	333.2	46.0	19.97
Sherrer	Ag	135	15	-25	0.331	70.2	107.1	9.2	1.07	226.8	298.2	33.4	1.91
	Cu	135	15	-25	0.003	81.0	102.1	11.2	0.01	232.9	302.7	31.1	0.01
	Mn	135	15	-25	0.801	81.2	106.3	7.0	4.45	232.9	290.3	20.3	2.76
	Pb	135	15	-25	0.91	71.7	96.7	11.4	4.35	218.5	262.9	37.6	3.86
	Zn	135	15	-25	4.247	77.1	109.4	13.7	18.25	233.3	333.2	46.0	19.97
Epitaph	Ag	105	20	-25	0.195	112.6	127.1	13.4	0.36	247.5	287.6	30.1	1.40
	Cu	105	20	-25	0.001	49.2	75.9	7.0	0.00	211.6	344.1	29.9	0.01
	Mn	105	20	-25	0.329	116.7	175.0	17.0	1.48	233.3	350.0	33.3	1.48
	Pb	105	20	-25	1.06	106.0	124.0	13.0	4.78	233.0	264.0	31.0	4.78
	Zn	105	20	-25	1.245	116.7	152.5	14.2	2.42	221.1	290.9	28.4	8.79
Taylor Deeps	Ag	135	15	-25	1.172	125.9	159.5	32.5	4.75	325.5	327.7	65.1	5.80
	Cu	135	15	-25	0.006	100.0	104.9	13.3	0.00	223.0	245.6	23.9	0.05
	Mn	135	15	-25	0.413	58.5	73.2	9.0	0.62	208.3	246.5	29.3	3.10
	Pb	135	15	-25	1.665	72.6	92.7	10.5	5.85	232.1	266.4	34.3	9.13
	Zn	135	15	-25	1.115	88.5	116.7	12.5	5.02	228.1	272.9	29.2	5.02

14.6 Block model

14.6.1 Parameters

The block model parameters are tabulated in Table 14.8.

Table 14.8 Block model parameters

Parameter	X	Y	Z
Origin*	1,069,500	170,200	0
Maximum block size (ft)	50	50	20
Minimum block size (ft)	5	5	1
Number of blocks	211	85	311

*Block model bottom left coordinate.

A search ellipse was created for each domain based on the distribution and orientation in space, of the composites within the domain. Note Trench Vein System is abbreviated to TVS in these tables. The parameters for the eight search ellipse areas are tabulated in Table 14.9.

Table 14.9 Search parameters summary

Domain	Range X (m)	Range Y (m)	Range Z (m)	Rotation Z	Rotation X	Rotation Y	Minimum composites	Maximum composite	Maximum composite per drillhole	Dynamic anisotropy
Pass 1										
LAG	220	250	30	0	0	0	4	10	2	Yes
MOX	220	250	30	0	0	0	4	10	2	Yes
Concha	250	250	30	135	15	-25	4	10	2	No
Sherrer	250	250	30	135	15	-25	4	10	2	No
Epitaph	250	250	30	0	0	0	4	12	2	Yes
Taylor Deeps	250	250	30	0	0	0	4	10	2	Yes
TVS	300	300	40	0	0	0	4	16	2	Yes
Hardshell	300	300	40	0	0	0	4	16	2	Yes
Pass 2										
LAG	440	500	60	0	0	0	4	10	2	Yes
MOX	440	500	60	0	0	0	4	10	2	Yes
Concha	500	500	60	135	15	-25	4	10	2	No
Sherrer	500	500	60	135	15	-25	4	10	2	No
Epitaph	500	500	60	0	0	0	4	12	2	Yes
Taylor Deeps	500	500	60	0	0	0	4	10	2	Yes
TVS	600	600	80	0	0	0	4	16	2	Yes
Hardshell	600	600	80	0	0	0	4	16	2	Yes
Pass 3										
LAG	2200	2500	300	0	0	0	3	12	2	Yes
MOX	2200	2500	300	0	0	0	3	12	2	Yes
Concha	1000	1000	120	135	15	-25	3	10	2	No
Sherrer	1000	1000	120	135	15	-25	3	10	2	No
Epitaph	1000	1000	120	0	0	0	3	12	2	Yes
Taylor Deeps	1500	1500	180	0	0	0	3	10	2	Yes
TVS	1500	1500	200	0	0	0	3	10	2	Yes
Hardshell	300	1500	200	0	0	0	3	10	1	Yes

*All domains were interpolated using dynamic anisotropy except for Concha and Sherrer.

14.6.2 Interpolation plan

Lead, zinc, copper, silver, and manganese grades were estimated for the all domains. However, for reporting purposes, the economic elements, lead, zinc, and silver are reported for the eight sulphide areas Taylor Deeps, Concha, Scherrer, Epitaph, Taylor Deeps, Trench Vein System, and Hardshell. The economic elements, silver, zinc, and manganese were reported for the LAG and MOX domains.

The Taylor Deeps, Concha, Scherrer and Epitaph domains were estimated by Ordinary Kriging when there were a sufficient number of composites; the Trench Vein System and Hardshell domains were estimated by Inverse Distance Squared (ID^2) weighting as were the small domains in Taylor Deeps.

The LAG and MOX domains were estimated using Ordinary Kriging. Grades were interpolated in three passes of increasing search ellipse dimensions, as shown in Table 14.9.

In all three passes, for all domains, a maximum of two (2) composites per drillhole was permitted thereby ensuring that at a minimum each block was informed by two holes.

Because of their variable orientation, grades were interpolated in all domains using the dynamic anisotropy module in Datamine™ except for the Concha and Scherrer domains. Each domain was estimated separately and boundaries between domains were treated as hard, i.e. the estimation of grades within one domain could not be influenced by grades of composites in adjacent domains.

14.6.3 Metal equivalency formula

Grades of silver, lead and zinc have been estimated for the six sulphide domains and the resource has been tabulated on the basis of Zinc Equivalency (ZnEq). Copper was not used as a component of the ZnEq formula because of its relatively low abundance and uncertainty pertaining to mineral processing and recovery and therefore to its value.

The ZnEq formula to equate lead and silver to zinc is:

ZnEq = [((Pb%/100)*2000*$0.95*95%) + ((Zn%/100)*2000*$1.00*92%) + (Ag oz/ton*$20.00*90%)]
/((2000*$1.00*92%)/100)

The price and recovery inputs to the equation are given in Table 14.10.

Table 14.10 Zinc equivalent parameters

Metal	Price ($)	Recovery (%)
Lead	0.95/lb	95
Zinc	1.00/lb	92
Silver	20.00/oz	90

Silver, zinc, and manganese grades have been reported for the LAG and MOX Domains. Although manganese is generally the most valuable metal of the three, it was decided to tabulate the Mineral Resource on the basis of the combined monetary value of the three metals rather than as a manganese equivalency because a manganese equivalency is considered an unconventional concept. The dollar value is based on metal grade times metal price times metal recovery. The combined metal value is termed Oxval (oxide value) and the formula is:

Oxval = ((Mn grade (%)* $0.91*86%)+(Zn grade (%)*$1.00*55%)+(Ag oz/ton*$20.00*72%)) where the recovery for manganese is 86%, for zinc 55% and for silver 72%.

The price and recovery inputs to the equation are given in Table 14.11.

Table 14.11 Oxval parameters

Metal	Price ($)	Recovery (%)
Manganese	0.91/lb	86
Zinc	1.00/lb	55
Silver	20.00/oz	72

14.7 Mineral Resource classification

Mineral Resources were classified as Measured, Indicated and Inferred. Classification was carried out using data support as a main criterion, with a manual review creating volumes based on sample density and distance to inform a block, thus removing outliers. For a block to be classified as Measured, it was necessary that a minimum of 16 composites from a minimum of 8 drillholes were located within 250 ft of the block centroid.

For Indicated Mineral Resources, if blocks were estimated by pass 1 and pass 2 but did not meet the Measured Resource criteria, they were classified as Indicated Resources except for domains estimated using ID^2. These were classified as Inferred Resources due to the small number of composites. The Trench vein system was also classified as an Inferred Resource based on a lack of geological confidence.

14.8 Mineral Resource tabulation

Mineral Resources for the Taylor Deposit (sulphide domains) are summarized in Table 14.12 at a cut-off grade of 4% zinc equivalent. Mineral Resources for the Central Deposit (oxide domains) are summarized in Table 14.13 at an Oxval cut-off value of US$100.

Table 14.12 Taylor Deposit Mineral Resource summary

Classification	Million tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Measured	15.2	4.0	4.0	1.6	9.6
Indicated	85.8	4.2	4.3	2.2	10.5
Measured and Indicated	101.0	4.1	4.3	2.1	10.4
Inferred	43.6	3.9	4.8	3.4	11.9

CIM Definition Standards (2014), were used for reporting the Mineral Resources.
Mineral Resources are reported as of 30 November 2017.
Stated at a cut-off grade of 4% ZnEq based on prices, recovery, and costs as follows:
- Prices of $1.00/lb for zinc, $0.95/lb for lead, and $20.00/oz for silver.
- Average processing recovery factors of 92% for zinc, 95% for lead, and 90% for silver.
- Total operating costs are estimated to be of the order of $60/ton.

ZnEq calculation is discussed in Section 14.6.3.
Numbers are rounded and may not match later detailed tables.

Table 14.13 Central Deposit Mineral Resource summary

Classification	Million tons	Zn (%)	Ag (oz/ton)	Mn (%)	Oxval ($/ton)
Measured	21.8	1.9	3.3	9.2	262
Indicated	41.7	2.3	1.7	9.8	257
Measured and Indicated	63.5	2.2	2.3	9.6	259
Inferred	1.8	2.6	1.6	7.4	207

CIM Definition Standards (2014), were used for reporting the Mineral Resources.
Mineral Resources are reported as of 30 November 2017.
Stated at a cut-off grade of $100/ton Oxval based on prices, recovery, and costs as follows:
- Prices of $1.00/lb for zinc, $20.00/oz for silver, and $0.91/lb for manganese.
- Average processing recovery factors of 55% for zinc, 72% for silver, and 86% for manganese.
- Total operating costs are estimated to be on the order of $100/ton.

Oxval calculation is discussed in Section 14.6.3.
Numbers are rounded and may not match later detailed tables.

In Table 14.14 through Table 14.19, the Taylor Deposit Mineral Resources are tabulated for each domain at a range of zinc equivalent cut-off grades. Note that resources estimated for the Trench Veins domain and Hardshell domains are all classed as Inferred Resources and the figures have been incorporated into the Inferred portion of the Taylor Deeps Resource. In Table 14.18 and Table 14.19, the Mineral Resources for the Central Deposit are stated at a range of Oxval cut-off values.

Table 14.14 Taylor Deposit Concha Domain Mineral Resources

Cut-off ZnEq (%)	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Measured					
25	379,798	15.1	10.7	4.2	29.7
20	733,205	13.2	9.6	3.7	26.2
15	1,174,719	11.5	8.4	3.3	22.9
10	1,915,381	9.3	6.9	2.7	18.8
5	3,506,880	6.6	5.0	2.0	13.5
4	4,020,663	6.1	4.5	1.8	12.3
3	4,701,071	5.4	4.1	1.7	11.0
2	5,570,615	4.8	3.5	1.5	9.7
1	6,284,044	4.3	3.2	1.4	8.8
0.5	6,343,626	4.3	3.2	1.4	8.7
Indicated					
25	1,895,951	16.7	12.0	5.0	33.3
20	3,391,581	13.9	10.4	4.4	28.5
15	6,011,959	11.1	8.9	3.9	23.6
10	9,743,197	9.0	7.3	3.3	19.3
5	15,010,917	7.0	5.6	2.6	15.1
4	16,575,388	6.6	5.2	2.4	14.1
3	18,291,711	6.1	4.8	2.3	13.1
2	20,278,090	5.6	4.4	2.1	12.0
1	21,926,893	5.2	4.1	2.0	11.3
0.5	22,319,514	5.1	4.1	2.0	11.1
Measured and Indicated					
25	2,275,749	16.4	11.8	4.9	32.7
20	4,124,786	13.8	10.3	4.3	28.1
15	7,186,678	11.1	8.8	3.8	23.5
10	11,658,577	9.0	7.2	3.2	19.2
5	18,517,797	7.0	5.5	2.5	14.8
4	20,596,051	6.5	5.1	2.3	13.7
3	22,992,782	6.0	4.7	2.2	12.7
2	25,848,704	5.4	4.2	2.0	11.5
1	28,210,938	5.0	3.9	1.9	10.7
0.5	28,663,140	4.9	3.9	1.8	10.5

Cut-off ZnEq (%)	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Inferred					
25	449,948	16.2	14.4	6.3	36.5
20	1,006,792	11.6	11.7	5.6	28.6
15	1,523,557	9.5	10.4	5.1	24.7
10	2,315,136	7.4	9.0	4.5	20.6
5	2,599,960	6.7	8.4	4.2	19.1
4	2,687,480	6.6	8.2	4.1	18.7
3	2,903,616	6.2	7.7	3.9	17.5
2	3,140,118	5.8	7.2	3.7	16.4
1	3,161,167	5.8	7.1	3.6	16.3
0.5	3,161,167	5.8	7.1	3.6	16.3

Table 14.15 Taylor Deposit Scherrer Domain Mineral Resources

Cut-off ZnEq (%)	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Measured					
25	5,691	7.2	11.5	8.8	27.1
20	10,970	6.4	10.1	8.0	24.1
15	68,102	5.4	7.6	5.3	18.0
10	396,753	3.6	6.0	3.8	13.3
5	1,460,472	2.9	3.8	2.0	8.6
4	1,884,614	2.7	3.4	1.7	7.7
3	2,435,947	2.4	2.9	1.5	6.7
2	2,878,374	2.2	2.7	1.3	6.1
1	3,234,188	2.0	2.4	1.2	5.6
0.5	3,323,504	2.0	2.4	1.2	5.5
Indicated					
25	27,667	12.4	10.0	5.7	27.8
20	124,966	10.1	9.0	4.5	23.3
15	561,051	8.6	6.9	3.3	18.5
10	1,678,783	6.6	5.2	2.5	14.2
5	6,706,173	3.9	3.2	1.7	8.7
4	8,597,152	3.5	2.9	1.5	7.8
3	10,620,082	3.1	2.6	1.4	7.0
2	12,787,797	2.8	2.3	1.2	6.2
1	14,858,936	2.5	2.0	1.1	5.5
0.5	15,180,691	2.4	2.0	1.1	5.4
Measured and Indicated					
25	33,358	11.5	10.2	6.2	27.6
20	135,937	9.8	9.1	4.8	23.4
15	629,153	8.2	7.0	3.5	18.5
10	2,075,536	6.0	5.4	2.8	14.0
5	8,166,645	3.7	3.3	1.7	8.7
4	10,481,765	3.3	3.0	1.5	7.7
3	13,056,030	3.0	2.6	1.4	6.9
2	15,666,171	2.7	2.4	1.2	6.2
1	18,093,124	2.4	2.1	1.1	5.6
0.5	18,504,194	2.4	2.1	1.1	5.4

Cut-off ZnEq (%)	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Inferred					
20	712	12.0	8.1	2.8	22.7
15	52,381	8.5	6.3	2.3	16.9
10	125,809	6.9	5.3	2.1	14.1
5	265,310	4.8	4.1	1.7	10.5
4	360,087	4.0	3.5	1.4	8.9
3	451,177	3.4	3.1	1.3	7.8
2	540,406	3.0	2.8	1.2	6.9
1	650,242	2.6	2.4	1.0	6.0
0.5	658,103	2.6	2.4	1.0	5.9

Table 14.16 Taylor Deposit Epitaph Domain Mineral Resources

Cut-off ZnEq %	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Measured					
20	41,732	10.6	8.9	2.8	22.1
15	294,427	8.3	7.0	2.3	17.4
10	1,315,717	6.2	5.4	1.9	13.3
5	4,835,097	3.9	3.6	1.3	8.6
4	6,329,184	3.4	3.2	1.1	7.7
3	8,283,534	2.9	2.8	1.0	6.7
2	10,633,700	2.5	2.5	0.9	5.7
1	12,737,619	2.2	2.2	0.8	5.0
0.5	13,031,132	2.1	2.1	0.7	5.0
Indicated					
25	451,062	13.3	13.2	4.6	30.8
20	1,247,609	11.3	10.6	3.7	25.4
15	2,966,711	9.2	8.5	3.0	20.5
10	8,061,353	6.6	6.3	2.3	15.1
5	24,439,470	4.2	4.0	1.6	9.7
4	30,382,160	3.8	3.6	1.4	8.7
3	37,166,917	3.3	3.2	1.3	7.7
2	45,259,773	2.9	2.8	1.1	6.8
1	52,837,340	2.6	2.5	1.0	6.0
0.5	54,168,170	2.5	2.5	1.0	5.9
Measured and Indicated					
25	451,062	13.3	13.2	4.6	30.8
20	1,289,341	11.3	10.6	3.7	25.3
15	3,261,138	9.1	8.4	3.0	20.2
10	9,377,071	6.6	6.2	2.3	14.8
5	29,274,567	4.2	4.0	1.5	9.5
4	36,711,343	3.7	3.5	1.4	8.5
3	45,450,451	3.3	3.1	1.2	7.6
2	55,893,473	2.8	2.8	1.1	6.6
1	65,574,959	2.5	2.4	1.0	5.9
0.5	67,199,302	2.5	2.4	0.9	5.7

Cut-off ZnEq %	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Inferred					
25	352,073	8.6	12.9	9.2	30.2
20	694,782	8.6	10.9	7.0	26.1
15	1,477,995	7.4	8.9	5.2	21.3
10	2,893,617	6.1	7.2	3.8	16.8
5	5,697,053	4.7	5.1	2.5	12.1
4	6,464,984	4.3	4.8	2.3	11.2
3	7,337,355	4.0	4.4	2.1	10.3
2	9,012,002	3.4	3.7	1.8	8.8
1	11,134,857	2.9	3.1	1.5	7.4
0.5	11,383,538	2.8	3.1	1.5	7.3

Table 14.17 Taylor Deposit Taylor Deeps Domain Mineral Resources

Cut-off ZnEq (%)	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Measured					
25	182,647	7.9	18.0	6.1	31.6
20	311,877	7.4	15.3	5.4	27.8
15	603,681	6.5	12.0	4.6	22.7
10	1,198,701	5.3	8.9	3.7	17.6
5	2,552,162	3.8	5.8	2.6	12.1
4	2,973,452	3.5	5.3	2.4	11.0
3	3,489,448	3.2	4.7	2.2	9.9
2	3,948,097	2.9	4.3	2.0	9.1
1	4,243,490	2.7	4.0	1.9	8.5
0.5	4,356,817	2.7	3.9	1.9	8.3
Indicated					
25	1,599,209	8.7	13.7	8.6	30.6
20	3,467,576	7.9	11.4	7.3	26.2
15	6,571,668	6.8	9.6	5.9	22.0
10	13,497,509	5.3	7.5	4.4	17.0
5	26,236,997	3.8	5.4	3.2	12.2
4	30,195,685	3.5	5.0	2.9	11.2
3	35,040,839	3.1	4.5	2.7	10.1
2	40,323,894	2.8	4.0	2.4	9.1
1	45,492,893	2.5	3.7	2.2	8.3
0.5	47,538,150	2.4	3.5	2.1	7.9
Measured and Indicated					
25	1,781,856	8.6	14.1	8.4	30.7
20	3,779,452	7.9	11.7	7.1	26.3
15	7,175,349	6.8	9.8	5.8	22.1
10	14,696,209	5.3	7.6	4.4	17.0
5	28,789,159	3.8	5.5	3.1	12.2
4	33,169,137	3.5	5.0	2.9	11.2
3	38,530,287	3.1	4.5	2.6	10.1
2	44,271,991	2.8	4.1	2.4	9.1
1	49,736,383	2.6	3.7	2.2	8.3
0.5	51,894,967	2.5	3.5	2.1	8.0

Cut-off ZnEq (%)	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
Inferred					
25	2,830,149	9.4	13.1	10.5	32.5
20	5,211,160	8.5	11.2	8.5	27.9
15	8,411,847	7.6	9.6	7.0	23.9
10	13,554,374	6.2	7.9	5.7	19.5
5	28,015,564	4.1	5.1	4.0	13.0
4	34,096,543	3.6	4.5	3.5	11.5
3	42,211,955	3.1	3.9	3.1	10.0
2	51,785,757	2.7	3.3	2.7	8.6
1	65,398,468	2.2	2.7	2.2	7.1
0.5	71,468,437	2.1	2.5	2.1	6.6

Table 14.18 Central Deposit LAG Domain Mineral Resources

Cut-off Oxval (US$)	Tons	Zn (%)	Ag (oz/ton)	Mn (%)
Measured				
500	0	0	0	0
400	0	0	0	0
300	4,105	0.3	10.1	12.1
200	27,657	0.2	7.1	9.4
100	618,962	0.2	3.9	4.7
50	2,693,701	0.1	3.0	2.4
Indicated				
500	0	0	0	0
400	0	0	0	0
300	18,555	2.4	1.4	18.2
200	85,979	2.1	1.5	14.0
100	844,758	0.7	2.3	6.2
50	4,187,346	0.3	2.4	2.8
Measured and Indicated				
500	0	0	0	0
400	0	0	0	0
300	22,660	2.1	3.0	17.1
200	113,637	1.6	2.8	12.9
100	1,463,720	0.5	3.0	5.6
50	6,881,047	0.2	2.7	2.6
Inferred				
500	0	0	0	0
400	0	0	0	0
300	0	0	0	0
200	0	0	0	0
100	0	0	0	0
50	246,392	0.1	2.9	1.1

Table 14.19 Central Deposit MOX Domain Mineral Resources

Cut-off Oxval (US$)	Tons	Zn (%)	Ag (oz/ton)	Mn (%)
Measured				
500	121,223	2.5	15.5	18.3
400	677,856	2.5	10.8	17.3
300	3,551,854	2.6	6.1	15.6
200	10,484,465	2.4	4.4	12.4
100	21,134,316	2.0	3.3	9.3
50	26,319,767	1.8	2.8	8.1
Indicated				
500	26,238	6.6	7.1	22.3
400	275,271	5.2	4.7	19.6
300	4,521,125	3.6	2.7	16.7
200	19,751,511	3.0	2.1	13.0
100	40,888,208	2.4	1.7	9.9
50	49,448,072	2.1	1.6	8.8
Measured & Indicated				
500	147,462	3.2	14.0	19.0
400	953,127	3.3	9.0	18.0
300	8,072,979	3.2	4.2	16.2
200	30,235,976	2.8	2.9	12.8
100	62,022,524	2.2	2.3	9.7
50	75,767,839	2.0	2.0	8.6
Inferred				
500	0	0	0	0
400	0	0	0	0
300	6,628	3.0	2.7	16.0
200	419,740	3.6	1.9	10.1
100	1,837,150	2.6	1.6	7.4
50	2,186,278	2.3	1.5	6.8

14.9 Block model validation

The block model was validated in three ways: 1) by visual comparison of composite and block grades to check for similarity of magnitude and to identify any anomalous relationships, 2) by comparison of assay, composite, and block model grades, and 3) by swath plots.

Figure 14.5 is a long section view through the block model showing the correspondence of composite and block grades to demonstrate the general correspondence between the two.

Figure 14.5 Long section through a portion of the Taylor block model

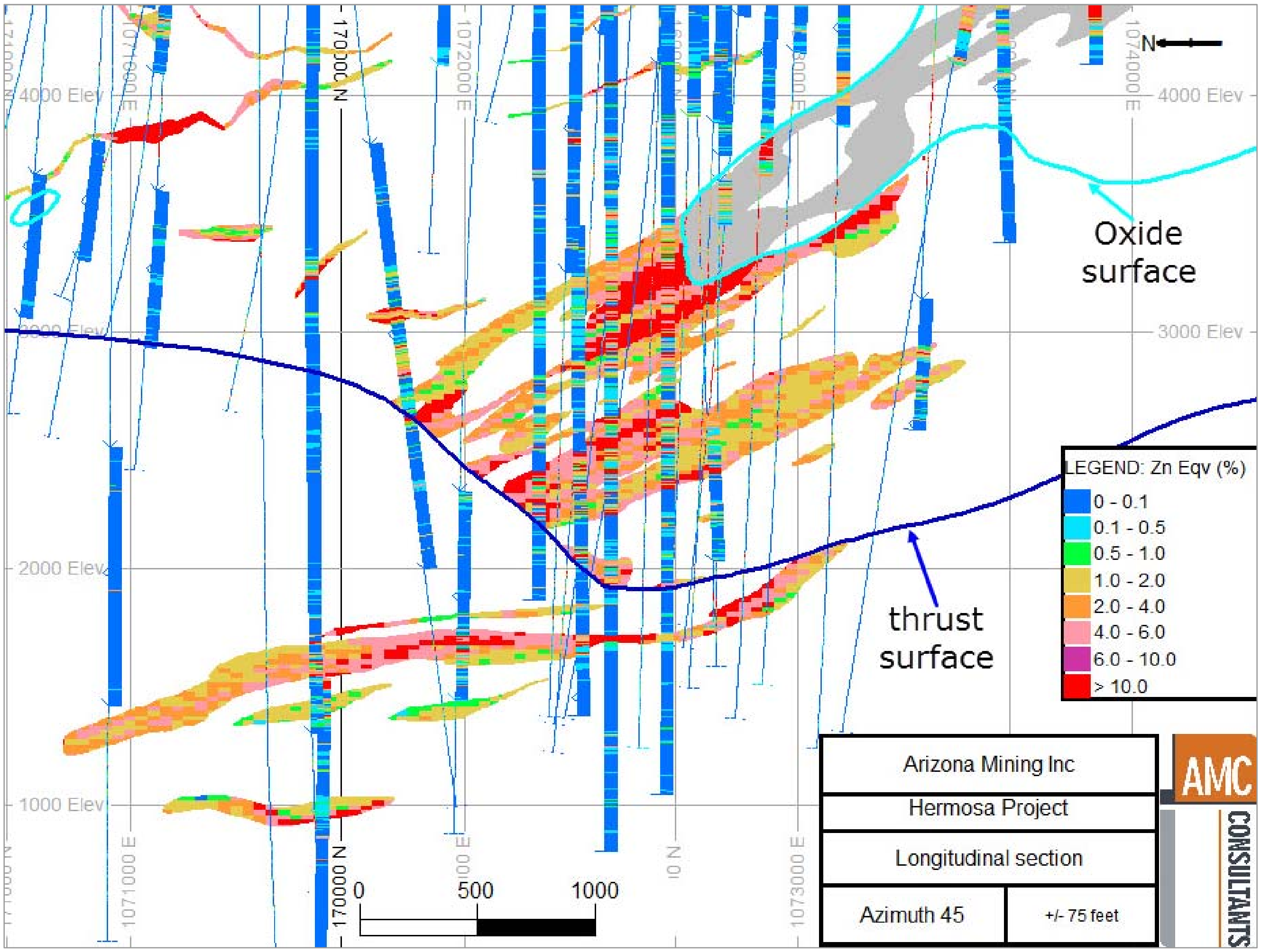


Table 14.20 shows the comparison of composite and block model grades for each domain. The assay and composite values are in close agreement. Block model grades are generally in good agreement with the drillholes with the exception of local areas where the block model grade is higher. AMC notes that the block model statistics below are weighted by tons which may be artificially increasing the emphasis on higher grade samples (due to tonnage factor). In comparison, the SWATH below where weighted by volume and agreement with drillholes is better.

Table 14.20 Statistical comparison of composited assay data and block model

Domain	Statistic	Ag (oz/ton)		Cu (%t)		Mn (%)		Pb (%)		Zn (%)	
		Model	Composite	Model	Composite	Model	Composite	Model	Composite	Model	Composite
LAG	No. samples	1,023,989	3,472	1,023,989	3,472	1,023,989	3,472	1,023,989	3,472	1,023,989	3,472
	Minimum	0.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Maximum	10.12	22.26	0.54	0.88	21.11	24.30	7.77	18.49	4.32	6.87
	Mean	**1.32**	**1.35**	**0.02**	**0.02**	**0.49**	**0.49**	**0.22**	**0.23**	**0.06**	**0.05**
	Stand.Dev	0.73	1.62	0.02	0.03	1.01	1.67	0.36	0.64	0.13	0.23
	CV	0.55	1.20	0.96	1.81	2.06	3.44	1.67	2.75	2.28	4.26
MOX	No. samples	1,160,739	3,804	1,160,739	3,804	1,160,739	3,804	1,160,739	3,804	1,160,739	3,804
	Minimum	0.0	0.0	0.0	0.0	0.2	0.0	0.0	0.0	0.0	0.0
	Maximum	14.50	40.00	0.38	1.11	25.87	33.69	5.96	19.33	10.60	25.80
	Mean	**1.79**	**2.46**	**0.07**	**0.08**	**8.21**	**8.14**	**1.14**	**1.19**	**1.77**	**1.90**
	Stand.Dev	1.67	4.09	0.05	0.12	4.58	7.23	0.96	1.92	1.37	2.95
	CV	0.93	1.66	0.81	1.58	0.56	0.89	0.85	1.62	0.77	1.55
Concha	No. samples	477,769	941	477,769	941	477,769	941	477,769	941	477,769	941
	Minimum	0.06	0.01	0.00	0.00	0.58	0.03	0.11	0.01	0.18	0.03
	Maximum	16.91	18.56	3.99	1.69	15.72	16.61	30.12	30.00	31.12	34.45
	Mean	**2.07**	**1.72**	**0.13**	**0.11**	**5.54**	**5.86**	**4.35**	**3.65**	**5.04**	**4.80**
	Stand.Dev	1.95	2.37	0.19	0.16	2.59	3.44	4.22	5.20	4.89	6.52
	CV	0.94	1.37	1.49	1.49	0.47	0.59	0.97	1.42	0.97	1.36
Sherrer	No. samples	361,629	641	361,629	641	361,629	641	361,629	641	361,629	641
	Minimum	0.020	0.009	0.001	0.000	0.447	0.076	0.049	0.002	0.095	0.008
	Maximum	14.51	15.88	1.14	1.42	12.28	15.77	20.88	20.79	19.47	19.98
	Mean	**1.16**	**1.04**	**0.09**	**0.08**	**3.80**	**3.63**	**2.21**	**2.07**	**2.45**	**2.31**
	Stand.dev	1.23	1.86	0.11	0.16	1.87	2.84	1.92	3.04	2.04	3.25
	CV	1.06	1.79	1.26	2.14	0.49	0.78	0.87	1.47	0.83	1.41
Epitaph	No. samples	967,919	1,885	967,919	1,885	967,919	1,885	967,919	1,885	967,919	1,885
	Minimum	0.035	0.007	0.001	0.000	0.088	0.019	0.024	0.001	0.005	0.002
	Maximum	15.34	17.56	0.51	0.74	8.48	10.00	23.48	28.00	22.15	26.00
	Mean	**1.04**	**0.89**	**0.05**	**0.04**	**2.17**	**2.16**	**2.53**	**2.28**	**2.54**	**2.30**
	Stand.Dev	1.17	1.39	0.06	0.07	1.26	1.81	2.32	3.20	2.34	3.38
	CV	1.13	1.57	1.15	2.01	0.58	0.84	0.91	1.41	0.92	1.47

Domain	Statistic	Ag (oz/ton)		Cu (%t)		Mn (%)		Pb (%)		Zn (%)	
		Model	Composite	Model	Composite	Model	Composite	Model	Composite	Model	Composite
Taylor Deeps	No. samples	1,693,000	1,555	1,693,000	1,555	1,693,000	1,555	1,693,000	1,555	1,693,000	1,555
	Minimum	0.009	0.007	0.000	0.000	0.024	0.019	0.004	0.000	0.004	0.002
	Maximum	39.47	40.00	2.26	1.80	8.67	10.00	30.16	35.91	20.67	28.36
	Mean	**2.19**	**1.82**	**0.14**	**0.12**	**2.06**	**2.20**	**3.15**	**2.81**	**2.15**	**2.01**
	Stand.Dev	2.81	3.65	0.19	0.25	1.32	2.04	3.55	5.14	2.50	3.60
	CV	1.28	2.01	1.33	2.02	0.64	0.93	1.13	1.83	1.16	1.79
Trench veins system	No. samples	1,609,332	660	1,609,332	660	1,609,332	660	1,609,332	660	1,609,332	660
	Minimum	0.01	0.01	0.00	0.00	0.02	0.02	0.00	0.00	0.01	0.00
	Maximum	16.64	23.45	0.79	1.13	7.74	10.00	25.31	30.24	31.39	37.71
	Mean	**1.72**	**1.74**	**0.06**	**0.07**	**0.99**	**1.00**	**1.81**	**1.69**	**2.30**	**2.14**
	Stand.Dev	1.78	2.91	0.07	0.12	1.12	1.47	2.45	3.51	3.26	4.57
	CV	1.03	1.68	1.13	1.80	1.13	1.47	1.35	2.07	1.42	2.13
Hardshell	No. samples	197,405	301	197,405	301	197,405	301	197,405	301	197,405	301
	Minimum	0.02	0.01	0.00	0.00	0.08	0.03	0.00	0.00	0.00	0.00
	Maximum	12.51	21.06	0.98	1.24	10.77	12.93	15.28	18.85	25.89	32.80
	Mean	**1.90**	**1.92**	**0.05**	**0.06**	**3.18**	**3.29**	**1.56**	**1.58**	**2.87**	**2.87**
	Stand.Dev	1.91	3.17	0.08	0.15	1.91	3.25	1.80	2.91	3.23	5.17
	CV	1.00	1.65	1.43	2.54	0.60	0.99	1.16	1.84	1.13	1.80

Block model grades weighted by tons, drillhole grades weighted by length.

Figure 14.6 and Figure 14.7 are swath plots showing the distribution of Zn and Pb grades respectively east-west, north-south and vertical swath plots through the Epitaph Domain. The analysis of all the swath plots for each domain shows a good match of the composite grade distribution to that in the model for all metals. As noted above, model grades are weighted by volume and drillholes are weighted by length.

Figure 14.6 Swath Plot for Epitaph Domain – Zn

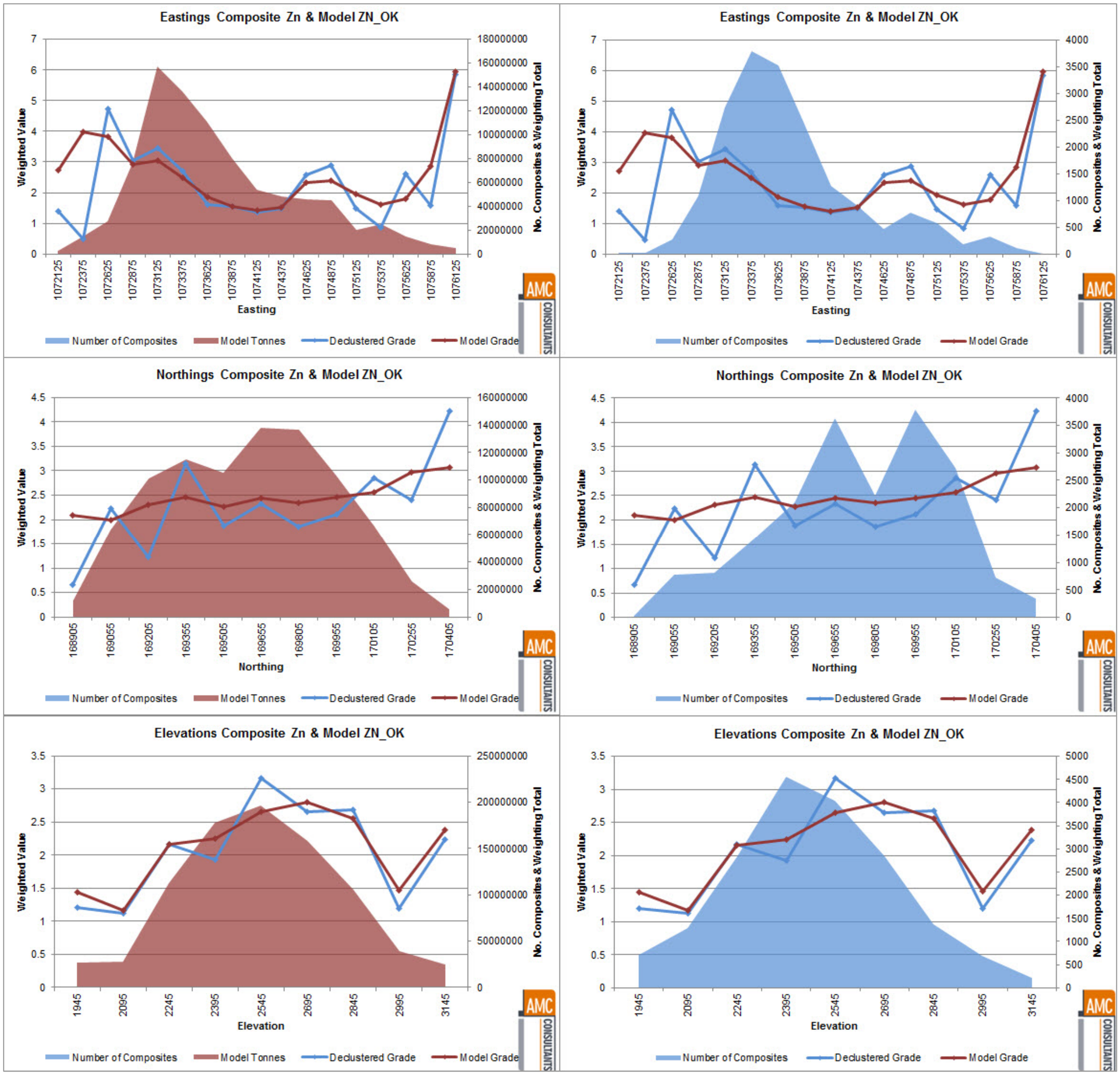
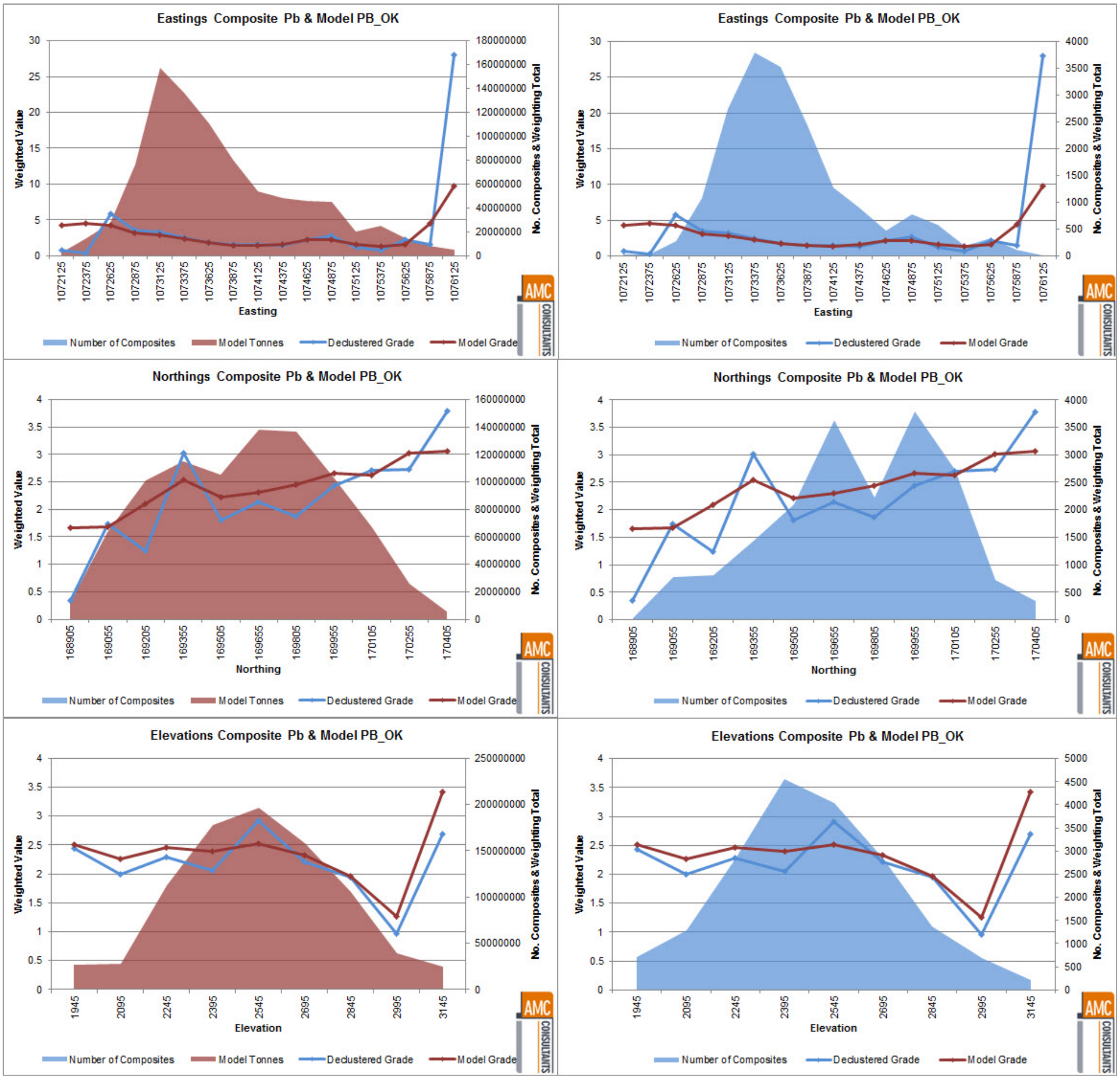


Figure 14.7 Swath Plot for Epitaph Domain – Pb



14.10 Comparisons

14.10.1 Taylor Deposit estimates

The comparison of the March 2017 and November 2017 Mineral Resource estimates for the Taylor Deposit are shown in Table 14.21. The significant changes are that the Measured Resource tons increased by 76.6%, the Indicated Resource tons increased by 34.3% and the Inferred Resource tons increased by approximately 12%. The silver grades increased in the Measured Resource by 2.2% and by 13.6% in the Indicated Resource. However, the zinc grade decreased in all categories. The lead grade increased slightly in the Measured Resource, but decreased by 5.3% in the Indicated Resource.

Table 14.21 Comparison of March 2017 and November 2017 Mineral Resource estimates for Taylor Deposit

March 2017					
Classification	**M tons**	**Zn (%)**	**Pb (%)**	**Ag (oz/ton)**	**ZnEq (%)**
Measured	8.6	4.2	4.0	1.6	9.7
Indicated	63.8	4.5	4.4	1.9	10.6
Measured and Indicated	72.5	4.5	4.4	1.9	10.5
Inferred	38.6	4.4	4.2	3.1	11.6
November 2017					
Classification	**M tons**	**Zn (%)**	**Pb (%)**	**Ag (oz/ton)**	**ZnEq (%)**
Measured	15.2	4.0	4.0	1.6	9.6
Indicated	85.8	4.2	4.3	2.2	10.5
Measured and Indicated	101.0	4.1	4.3	2.1	10.4
Inferred	43.6	3.9	4.8	3.4	11.9
% difference					
Classification	**M tons**	**Zn (%)**	**Pb (%)**	**Ag (oz/ton)**	**ZnEq (%)**
Measured	76.6	-4.1	--0.1	2.2	-1.5
Indicated	34.3	--7.4	-1.6	13.6	-0.7
Measured and Indicated	39.3	-7.8	-2.8	9.5	-1.2
Inferred	12.9	-11.1	13.1	8.6	2.3

Notes for the AMC November Estimate: See footnotes under Table 1.2.
Notes for the AMC March Estimate:
1 Mineral Resources are reported as of 29 March 2017.
2 Stated at a cut-off grade of 4% ZnEq based on prices, recovery, and costs as follows:
- Prices of $1.00/lb for zinc, $0.95/lb for lead, and $20.00/oz for silver.
- Average processing recovery factors of 90% for zinc, 95% for lead, and 85% for silver.
- Total operating costs are estimated to be of the order of $60/ton.
- ZnEq calculation is discussed in Section 14.6.3.
- Numbers are rounded and may not match later detailed tables.

Both Mineral Resource estimates are reported at a Zn equivalent cut-off of 4%.
Source: AMC Mining Consultants (Canada) Ltd.

A considerable amount of new drilling was carried out on the deposit since the March 2017 Mineral Resource. Primarily this drilling was focused on upgrading Mineral Resources to Measured and Indicated categories. In addition to new drilling, the domaining threshold for gradeshells dropped from 1% Zn equivalent to 0.5%. The increase of tons is a result of new drilling in addition to the expansion of the domains. The overall decrease of grades, as seen in the Zn equivalency, can be explained by the use of a lower cut-off grade of 0.5% of ZnEq for building of the mineralization domains.

14.10.2 Central Deposit estimates

Table 14.22 shows the comparison between the March 2017 and November 2017 Mineral Resource estimates for the Central Deposit at $100 Oxval cut-off. In the updated Mineral Resource, there is a decrease in tonnage of Measured plus Indicated Resource of about 10%. The manganese price used in the derivation of the Oxval dropped from $1.22/lb to $0.91/lb thus reducing value and hence tons above the cut off.

Table 14.22 Comparison of March 2017 and November 2017 Mineral Resource estimates for Central Deposit

March 2017				
Category	**M tons**	**Zn (%)**	**Ag (oz/ton)**	**Mn (%)**
Measured	20.7	1.8	4.1	9.2
Indicated	49.9	2.3	1.9	9.6
Measured and Indicated	70.6	2.2	2.5	9.5
Inferred	0.4	3.2	2.7	7.2
November 2017				
Category	**M tons**	**Zn (%)**	**Ag (oz/ton)**	**Mn (%)**
Measured	21.8	1.9	3.3	9.2
Indicated	41.7	2.3	1.7	9.8
Measured and Indicated	63.5	2.2	2.3	9.6
Inferred	1.8	2.6	1.6	7.4
% difference				
Category	**M tons**	**Zn (%)**	**Ag (oz/ton)**	**Mn (%)**
Measured	5.1	7.2	-19.3	0.1
Indicated	-16.4	0.9	-8.4	2.5
Measured and Indicated	-10.1	-0.5	-9.0	1.3
Inferred	424.9	-19.4	-39.3	2.8

Notes for the AMC November 2017 Estimate: See footnotes under Table 1.3.
Notes for the AMC March 2017 Estimate:
1 Mineral Resources are reported as of 29 March 2017.
2 Stated at a cut-off grade of $100/ton Oxval based on prices, recovery, and costs as follows:
- Prices of $1.00/lb for zinc, $20.00/oz for silver, and $1.22/lb for manganese.
- Average processing recovery factors of 55% for zinc, 72% for silver, and 86% for manganese.
- Total operating costs (mining and processing) are estimated to be on the order of $100/ton.

Source: AMC Mining Consultants (Canada) Ltd.

Note that the Manganese price used has declined from $1.22/lb to $0.91/lb since the March 2017 estimate

14.11 Conclusions and recommendations

While both the Taylor and Central Mineral Resources may be materially affected by constraints placed by the various responsible government agencies with respect to the granting of environmental and other permits to AZ; there are no known legal, title, taxation, socio-economic, marketing, political or other relevant factors that may materially affect the Mineral Resource estimates for the Taylor and Central deposits at this time.

The new drilling program resulted in positive increases in the sulphide Taylor Deposit. The Measured and Indicated Resources have increased by about 40% though the grades of lead and zinc decreased by about 5%. The Inferred tonnage increased by about 13%.

The updated Measured and Indicated Resources of the Central Deposit decreased for about 10% in tons and grades due to the change in prices affecting the Oxval calculation.

The calculation used to estimate bulk density and tonnage factors for the Taylor Deposit may be refined by the inclusion of pyrite content and possibly by inclusion of a factor to account for porosity as well as other elements. Some of this data is currently available and it is recommended that AZ investigates the possibility of obtaining a calculated bulk density that is in closer agreement with measured values than has been achieved to date.

The Mineral Resource for the Central Deposit was estimated using fixed bulk density values; it is probable that these single values can be improved upon by using an approach similar to that Limerick advocated for the Taylor Deposit.

15 Mineral Reserve estimates

There are no Mineral Reserve estimates to report for the Property.

16 Mining methods

16.1 Hydrological parameters

The climate in the project area varies from high desert in the Sonoita Valley to the steppe-like climate of the higher elevation grasslands and scrub area. Average rainfall is 17 in (432 mm) per year, with the majority of precipitation occurring between June and October. The project area is located within the Middle Sonoita Creek and Harshaw Creek watersheds.

Groundwater flows in bedrock fractures at the site. There is little to no alluvium present. Groundwater is recharged from precipitation at higher elevations and in the washes and drainages which carry surface flows from rain events north and northwest out of the basins.

Porosity of fractured bedrock aquifers is generally low, on the order of 1% to 2%. However, mineralization can result in higher porosities. Based on initial aquifer testing results at selected locations, K values in the upper-1,640 ft. (500 m) of the aquifer appear to range from about 0.03 ft./d (0.01 m/d) to 14.8 ft/d (4.5 m/d). Below 1,640 ft. (500 m), K values tend to be significantly lower, and may be less than 0.0003 ft/d (0.0001 m/d) in many locations. Based on this hydraulic conductivity value, it is estimated that groundwater inflows to the underground mine will be low, AMC has assumed 1.3 g/s (5 l/s), depending on the geometry of the underground workings.

16.2 Geotechnical parameters

Call & Nicholas, Inc. (CNI) undertook the preliminary geotechnical study for the project. The geotechnical work undertaken provides recommendations for:

- Excavation dimensions by rock type, depth, and orientation.
- Paste backfill strength.
- Ground support recommendations for development drifting, mineralization production drifting, and shaft support.
- Placement of critical mine infrastructure.

16.2.1 Excavation dimensions

The recommended stope dimensions for mining in the Concha and Epitaph \ Scherrer rock types to be used in the PEA study are provided in Table 16.1. These recommendations are based on stability at depths above which 80% of mineralization occurs for the different domains:

- Concha - 80% of mineralization is less than 2,296 ft (700 m) in depth.
- Epitaph / Scherrer - 80% of mineralization is less than 3,034 ft (925 m) in depth.

While CNI recognize a third rock type, the Scherrer, that is rich in mineralization and is planned for mining, it was not separated as a distinct geotechnical domain. Any mining that occurs within the Scherrer should follow the criteria of the Epitaph rock type.

Table 16.1 Key assumptions for the production and development schedules

Rock type	Mining parallel to strike max stope dimensions			Mining perpendicular to strike max stope dimensions		
	Height (ft)	Strike length (ft)	Width (ft)	Height (ft)	Strike length (ft)	Width (ft)
Concha	150	70	50	150	80	50
Epitaph	100	45	50	100	53	45
Rock type	Mining parallel to strike max stope dimensions			Mining perpendicular to strike max stope dimensions		
	Height (m)	Strike length (m)	Width (m)	Height (m)	Strike length (m)	Width (m)
Concha	45	21	15	45	25	15
Epitaph	30	14	15	30	16	14

Recommendations are based on Stable Dimension Criteria at the 80% Mineralization Depth Reliability.

Stope dimensions were optimized for height, rather than length. In both domains, because of the geologic joint fabric, mining perpendicular to the strike of the deposit allows for greater achievable dimensions. Analyses were limited to a depth of 4,000 ft (1,219 m).

16.2.2 Mining in the Concha

The Concha rock type was identified as the superior mining host rock. The rock quality designation (RQD = 93%), joint conditions, and intact rock strength qualify this rock to be of good quality per Barton's Q' classification system.

16.2.3 Mining in the Epitaph / Scherrer

The Epitaph rock type was identified as the lesser quality mining host rock. While the Epitaph has an identical rock quality designation (RQD = 93%), the joint conditions were of significantly less quality than those from within the Concha rock type. Observations from the drilling indicate that there are continuous zones of 30 m to 60 m of predominately slicken-sided joints with carbonaceous infill material. The joint conditions used for analysis reflect this. Due to these joint conditions, the Epitaph falls within the fair quality classification per Barton's Q' classification system.

Further to being of a lesser rock quality than the Concha, the Epitaph is found at greater depths where stresses more significantly influence stability. Mining at an orientation perpendicular to strike allows for greater lengths to be achieved at the optimal height.

16.2.4 Paste backfill strength

In order to achieve nearly full mineralization recovery at the project, paste backfill will be used to fill open stopes following their excavation. By filling these stopes with paste backfill, pillars will be established that will subsequently become the walls of later stage (secondary) stopes.

The stability of the paste backfill wall is directly related to the amount of cement binder used in the paste mixture. The strength of the paste backfill mixture must be great enough to hold a vertical face of a backfilled primary stope at the full stope height during mining of secondary stopes. In order to stand at heights up to 148 ft (45 m) when mining in the Concha, a backfill strength of 140 psi (967 kPa) is required (Mitchell, et al.). When mining in the Epitaph, in which stope heights are less at 98 ft (30 m), a backfill strength of 94 psi (645 kPa) is required. These values include a 1.15 safety factor to compensate for the natural variability in the paste backfill quality and potential for binder separation due to long pumping distances. AMC has assumed a cement binder rate of 4.5%. Further details on the pastefill distribution and quantities are provided in Section 16.10.

16.2.5 Ground support recommendations

16.2.5.1 Development drifts

Development drifts include all decline drifting and level access drifts. CNI have assumed dimensions of 18 ft (5.5 m) height and 18 ft (5.5 m) width for all development drifts. Due to the good quality of the rock at the project, no support beyond spot bolting should be required in the development drifts although a standard bolting pattern for all development is recommended.

Despite not needing patterned ground control in development drifting, AZ should anticipate the presence of infrequent faults that may require some support. Surficial support in the form of fibre-reinforced shotcrete (fibrecrete), or shotcrete in conjunction with pattern bolting may be needed when mining through these faults. Because of the scarcity of drilling data and the absence of a rock quality model, the frequency of these faults is difficult to predict. CNI recommend that AZ anticipate using fibrecrete or shotcrete with systematic bolting 6 ft in length with a spacing of 5.3 ft (1.8 m lengths; 1.6 m spacing) in approximately 20% of all drifting.

16.2.5.2 Production drifts

Production drifts include all stope accesses; bottom cuts, middle cuts, and top cuts. CNI have assumed dimensions of 14.8 ft (4.5 m) height and 14.8 ft (4.5 m) width for all production drifts.

Stope bottom cuts will not generally require any support beyond infrequent spot bolting. However, to account for faulting and areas of lesser quality ground, CNI recommend that AZ anticipate using fibrecrete or shotcrete with systematic bolting 6 ft in length with a spacing of 5.3 ft (1.8 m lengths; 1.6 m spacing) in approximately 20% of all production drifting.

16.2.5.3 Stope top and middle cuts in the Concha

Ground support requirements when stoping in the Concha are on the boundary of unsupported stability and requiring systematic bolting. Consequently, two forms of support, a minimum and maximum support, should be anticipated when mining in the Concha to be applied at an assumed 50% occurrence for each:

- At the minimum 50%, no top cut or middle cut support is required.
- At the maximum 50%, pattern bolting will be required in backs and ribs and will include the installation of 8 ft (2.4 m) friction bolts on 6.6 ft (2.0 m) nominal spacing in conjunction with a welded wire mesh.

16.2.5.4 Stope top and middle cuts in the Epitaph

Support of the top cuts and middle cuts when stoping in the Epitaph rock type will require systematic bolting and regular (50% occurrence) shotcrete application as specified below:

- Pattern bolting of the backs and sills will include the installation of 8 ft (2.4 m) friction bolts on 5.3 ft (1.6 m) nominal spacing in conjunction with a welded wire mesh.
- In approximately 50% of headings, regular (unreinforced) shotcrete should be applied to a thickness of 1.6 in to 3.9 in (40 mm to 100 mm).

16.2.5.5 Shaft ground support recommendations

The proposed shaft dimensions are: 21 ft inner diameter (6.5 m). The total shaft depth is 3,625 ft (1,105 m). The support requirements are as follows:

- Temporary support consists of 8 ft (2.4 m) friction bolts and welded-wire mesh.
- Final support consists of steel reinforced concrete that meets the following criteria:
  - Concrete Design compressive strength of 20-28 MPa.
  - Minimum lining thickness of 18 in (450 mm).

16.2.5.6 Placement of critical mine infrastructure

Recommendations for the placement of critical mine infrastructure is based on the knowledge of the regional fault geology and spatial drill hole data. Often, these types of large scale regional faults can inhibit the transfer of mining-induced stresses and as a result, these stresses will concentrate on the edges of the faults. To mitigate complications of placing infrastructure into such a high-stress environment, any infrastructure should be planned to the south of all mine workings.

16.3 Underground access

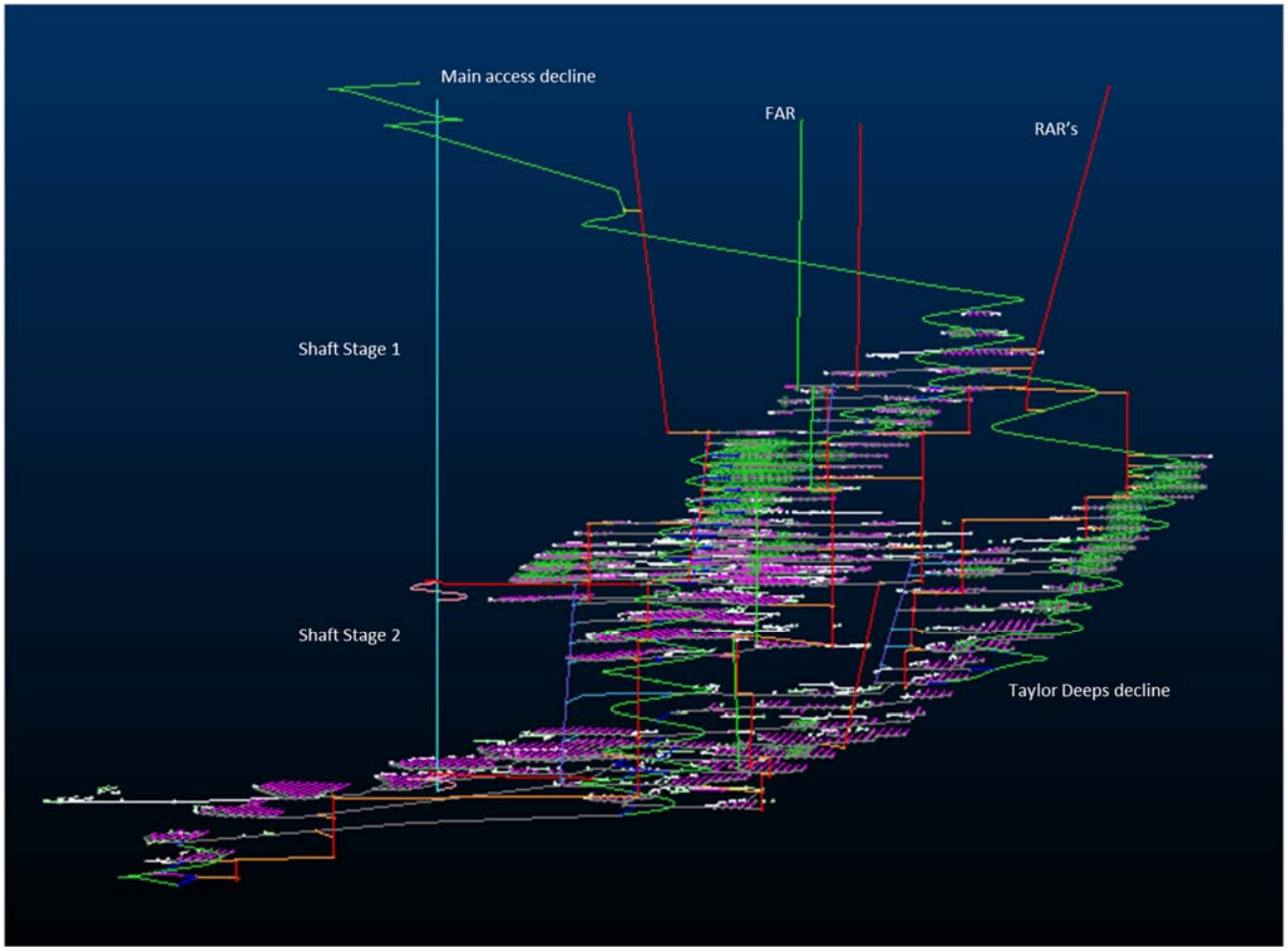
Based on previous trade-off studies to evaluate the mineralized material and waste handling system, a combination of a decline and vertical shaft system was selected as the optimum methodology based on economics and operability. Development of the access decline commences at the same time as sinking of the vertical shaft. Once the sub-levels are established, development mineralization is extracted via the decline. Stope production commences in Year 4 when the shaft is ready to commence hoisting.

It is assumed that the shaft is developed using the blind sinking method. Average advance rate for sinking the shaft is 8.2 ft/d (2.5 m/d). The shaft is assumed to be sunk in two stages, the first stage will allow mineralization to be mined from the middle of the deposit and be hoisted to surface. A second stage will be sunk directly below stage one to a depth of 3,625 ft (1,105 m) below surface. Stage one will be equipped, and a loading station constructed on 2600 L. All mineralization will then be dropped to the interim haulage level via passes and then trucked to the shaft for loading and hoisting.

Once the stage two shaft sinking is complete, hoisting will be suspended while the stage two shaft is fully equipped to the shaft bottom. All mineralization will then be dropped via the pass to the 1600 L and trucked via the main haulage level for hoisting via the shaft.

The general layout showing the underground access is provided in Figure 16.1.

Figure 16.1 Underground access general layout



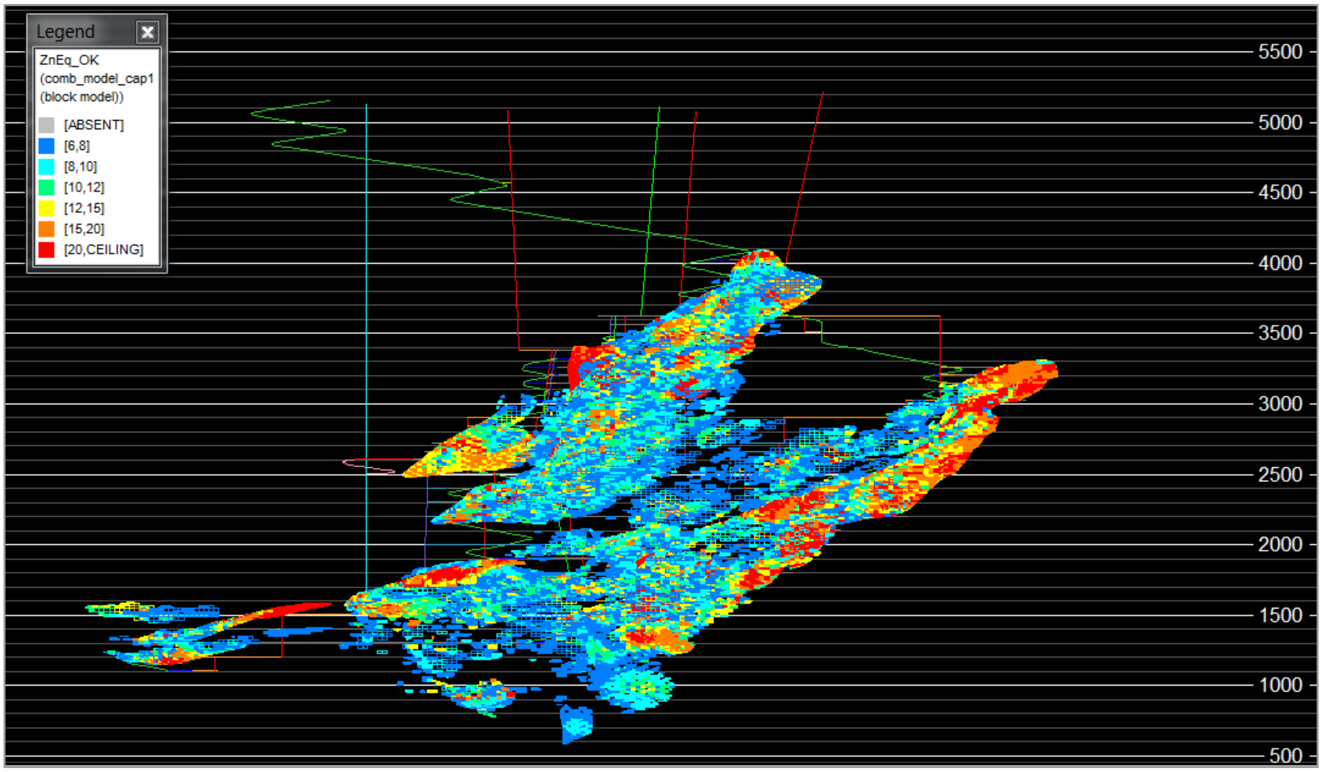
16.4 Mining method selection

The mining method selection criteria was based on:

- Deposit geometry – depth, shape, thickness, plunge.
- Rock quality – mineral zone and host rock competency (structures, stress, and stability).
- Mineralization variability – mineral uniformity, continuity, and grade distribution.
- Economics – mineral recovery, mineral value, productivity, capital and operating costs, safety.

The mineralization above a cut-off grade of 6% ZnEq is shown in Figure 16.2. Operating costs are approximately US$51/ton on mineralized material, however in line with the high-grade strategy a higher cut-off grade was selected.

Figure 16.2 Block model showing mineralization above cut-off grade of 6% ZnEq



The method that best supports low operating cost, high productivity with good recovery and low dilution is Sub level open stoping (SLOS). Mining activities will be fully mechanized and large modern trackless mobile equipment will be employed throughout. Ground conditions are generally expected to be fair to good, with a relatively small proportion of poor ground anticipated.

In order to optimize the mine economics, a high-grade core of mineralization was identified above a cut-off grade of 15% ZnEq, that is located between 2900 L and 3320 L. The high-grade material is accessible from each level independently and could be mined simultaneously, using more selective Longhole type mining methods over stope heights of 60 ft (18 m) floor to floor. A mine plan and mine design were developed to allow early access of the high-grade core between Year 4 and Year 7 (inclusive) of the LOM plan. The use of pastefill ensures that lower grade material is not sterilized but is extracted as a second pass.

AMC used a function of the Datamine™ software, Mine Shape Optimizer (MSO) to evaluate preliminary stope wireframes for the SLOS mining method. Varying stope heights between 60 ft and 100 ft (18 m and 30 m) were generated. The larger stope size ensures greater productivity and lower costs. This is in line with the geotechnical stope design criteria. The following parameters were adopted to generate stope wireframes (Table 16.2).

Table 16.2 MSO parameters (large stopes)

MSO parameter	Unit	Value	Unit	Value
Stope height 1	ft	60	M	18
Stope width 1	ft	40	M	12
Stope length	ft	100	M	30
Stope height 2	ft	100	M	30
Stope width 2	ft	50	M	15
Operating cost	US$/ton	49	US/tonne	54
Cut-off grade	% ZnEq	6.0	% ZnEq	6.0
Hangingwall / footwall dilution thickness	ft	0	M	0
Hangingwall / footwall dip angle	°	90	°	90
Drive height in mineralization	ft	14.8	M	4.5
Drive width in mineralization	ft	14.8	M	4.5

On completion of the stope shape optimization, it was identified that there remained flat dipping relatively thin lens shaped mineralized zones with reasonable grade between the larger stope shapes. Alternative stope shapes with a height of 20 ft (6.1 m) that would be applicable to a more selective cut and fill mining method were generated in these areas. The smaller stope shapes for the more selective method were generated from the depleted block model after consideration of the larger 100 ft and 60 ft stopes (30 m and 18 m). Only those stopes readily accessible from the respective interlevel spacing for the larger stopes were considered in the mine plan. The MSO parameters used for cut and fill stoping are summarized in Table 16.3.

Table 16.3 MSO parameters (small stopes)

MSO parameter	Unit	Value	Unit	Value
Stope height	ft	20	M	6.1
Operating cost	US$/ton	74	US/tonne	82
Cut-off grade	% ZnEq	6.0	% ZnEq	6.0
Hangingwall / footwall dilution thickness	ft	0	M	0
Hangingwall / footwall dip angle	°	90	°	90
Drive height in mineralization	ft	14.8	M	4.5
Drive width in mineralization	ft	14.8	M	4.5

The wireframes generated above the cut-off grade were then used to determine the potential mining inventory. The potential mining inventory is the Mineral Resource above the cut-off grade that includes the application of mining factors such as recovery and dilution.

16.5 Dilution and mining recovery factors

There are two main sources of dilution in underground stopes:

- Planned dilution. This is the dilution required to achieve the designed stope shape. Designed dilution can result from waste included:
  - To achieve minimum mining width.
  - To achieve a viable mining shape.
- Unplanned dilution. This is dilution that is outside of the designed stope shape. Depending on the mining method, it may include both overbreak and floor dilution.
  - Overbreak is typically a result of blasting practices and geotechnical conditions, and can include backfill from the adjacent stope walls.
  - Floor dilution is the result of mucking backfill from the stope floor.

AMC has applied a dilution factor of 5% at zero grade to the Mineral Resource and a mining recovery factor of 95% has been applied to the large stopes (SLOS). For the small stopes, AMC has applied a dilution factor of 10% at zero grade to the Mineral Resource and a mining recovery factor of 80%.

16.6 Stope design and selection

Stope wireframes were generated using MSO, a check was made to remove any outlying stopes that would not be economic when the cost of access development was included. The cost of access development was determined for each level and each level was evaluated to determine whether it was economic to develop. The potential mining inventory associated with the potentially economic stopes is summarized in Table 16.4.

Table 16.4 Potential mining inventory

Tons (M)	Zn (%)	Pb (%)	Ag (oz/t)	ZnEq (%)
96.7	4.0	4.3	2.22	10.4

Stoping commences on 2960 L, through to 3500 L. Mining panels consist of five 60 ft levels that will be mined in a bottom up mining sequence. All stopes are assumed to be 60 ft H by 40 ft W by 50 ft L (18 m H by 12 m W by 15 m L). Outside of the high-grade core larger stopes of 100 ft H by 50 ft W and 50 ft L (30 m H by 15 m W by 15 m L) are mined on a level by level basis. Once the high-grade material is extracted, the mine will extract mineralized material using primary and secondary stopes that are filled with cemented pastefill. The cut and fill stoping of the smaller stopes is assumed to be the last activity on each level. The primary stopes will be mined and backfilled prior to mining secondary stopes on a level sequence. As the level advances towards the south of the deposit, the level above can commence primary stoping, this will be repeated over the operating levels.

A summary of the tons and ZnEq grade by level as well as individual grades is provided in Table 16.5.

Table 16.5 Tons and grade by level

Level	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
4020	289,451	5.2	3.1	1.4	9.6
3920	941,195	6.3	3.6	1.7	11.4
3820	1,417,374	4.2	2.3	1.2	7.6
3720	1,425,729	4.2	2.6	1.4	8.1
3620	1,493,666	4.3	3.1	1.6	8.9
3560	723,950	4.6	3.3	1.6	9.4
3500	819,813	5.2	3.9	1.8	10.7
3440	746,561	5.1	4.1	1.7	10.7
3380	730,525	4.3	3.5	1.5	9.1
3320	1,369,516	7.3	5.3	2.1	14.6
3260	2,122,893	7.5	5.5	2.2	15.1
3200	2,667,530	7.7	6.1	3.0	16.5
3140	2,749,701	7.2	5.9	3.2	16.1
3080	2,494,313	6.9	5.4	3.1	15.2
3020	1,876,061	7.0	5.1	2.8	14.7
2960	1,432,732	6.8	5.1	2.8	14.5
2900	1,816,044	4.7	4.3	2.4	11.3
2840	2,098,300	4.1	3.9	2.1	10.0
2780	2,903,977	4.4	4.1	2.0	10.3
2720	4,049,335	4.0	4.3	2.1	10.2
2660	5,115,675	4.0	4.4	2.1	10.4
2600	5,006,333	4.1	4.4	2.0	10.4
2500	7,037,545	3.9	4.1	1.7	9.6

Level	Tons	Zn (%)	Pb (%)	Ag (oz/ton)	ZnEq (%)
2400	6,169,940	4.2	4.3	1.7	10.0
2300	6,267,636	3.5	3.9	1.7	9.0
2200	4,454,443	3.4	4.1	2.4	9.8
2100	1,271,131	2.7	3.5	2.8	8.9
2000	1,196,075	2.6	3.5	3.1	9.1
1900	1,182,211	2.3	3.2	3.0	8.3
1800	2,322,723	2.6	4.1	2.5	9.1
1700	6,979,410	2.2	4.1	2.4	8.5
1600	5,297,186	2.6	5.4	2.2	10.0
1500	5,792,809	1.9	4.9	2.9	9.6
1400	2,401,025	1.8	4.2	2.6	8.5
1300	1,427,349	2.1	3.0	2.3	7.3
1200	446,778	2.0	2.8	2.5	7.1
1100	133,612	1.9	2.6	2.3	6.7
Total	**96,670,545**	**4.0**	**4.3**	**2.2**	**10.4**

16.7 Production rate

In order to determine an appropriate production rate which can be supported by the deposit, AMC has used a combination of Taylor`s rule of thumb and vertical tons per metre to determine expected production ranges.

Production rate based on Taylors rule of thumb, is estimated at approximately 4.8 Mtpa (4.4 Mtonnes pa)

Annual Production Rate = 5 * Potential mining inventory $^{0.75}$

Annual production rate = 5 * (96.7 Mt) $^{0.75}$

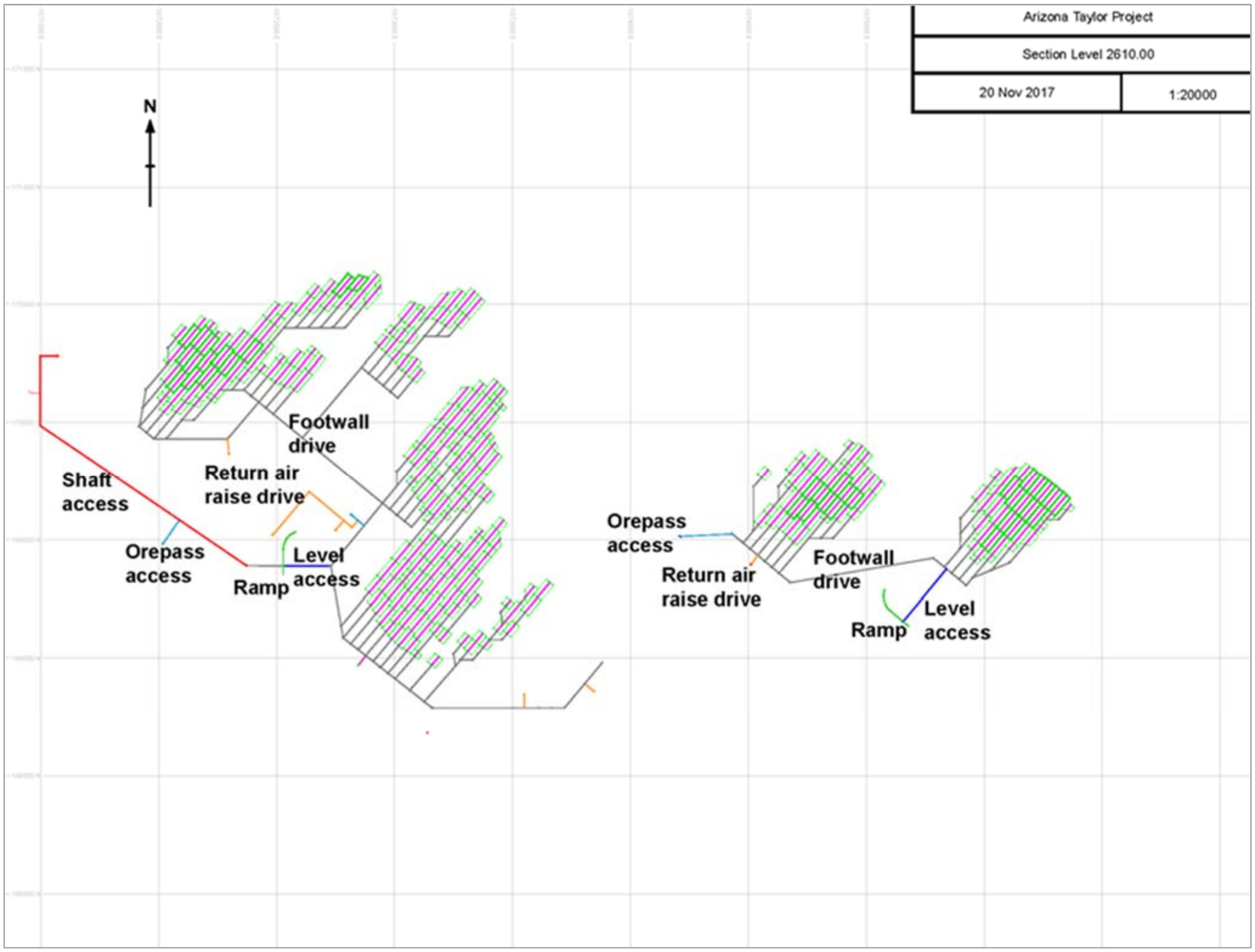
Most successful mines do not exceed 40 vertical metres / annum (vmpa). The deposit has approximately 105 kt/vm of mineralization this would support a production rate of approximately 4.2 Mtpa (3.8 Mtonnes pa).

AMC has completed a high-level schedule of the mineralization production aimed at meeting the target production rate of 10,000 tons per day. Based on this production schedule, the targeted throughput of 3.6 Mtpa is achievable. AMC considers that this production rate is low for the deposit, given the potential to mine from multiple fronts on each level as well as over multiple levels at a time. AMC considers that there is opportunity to explore an increased throughput of 12,500 tpd. For this PEA study AMC has scheduled production at a rate of 3.6 Mtpa.

16.8 Underground development

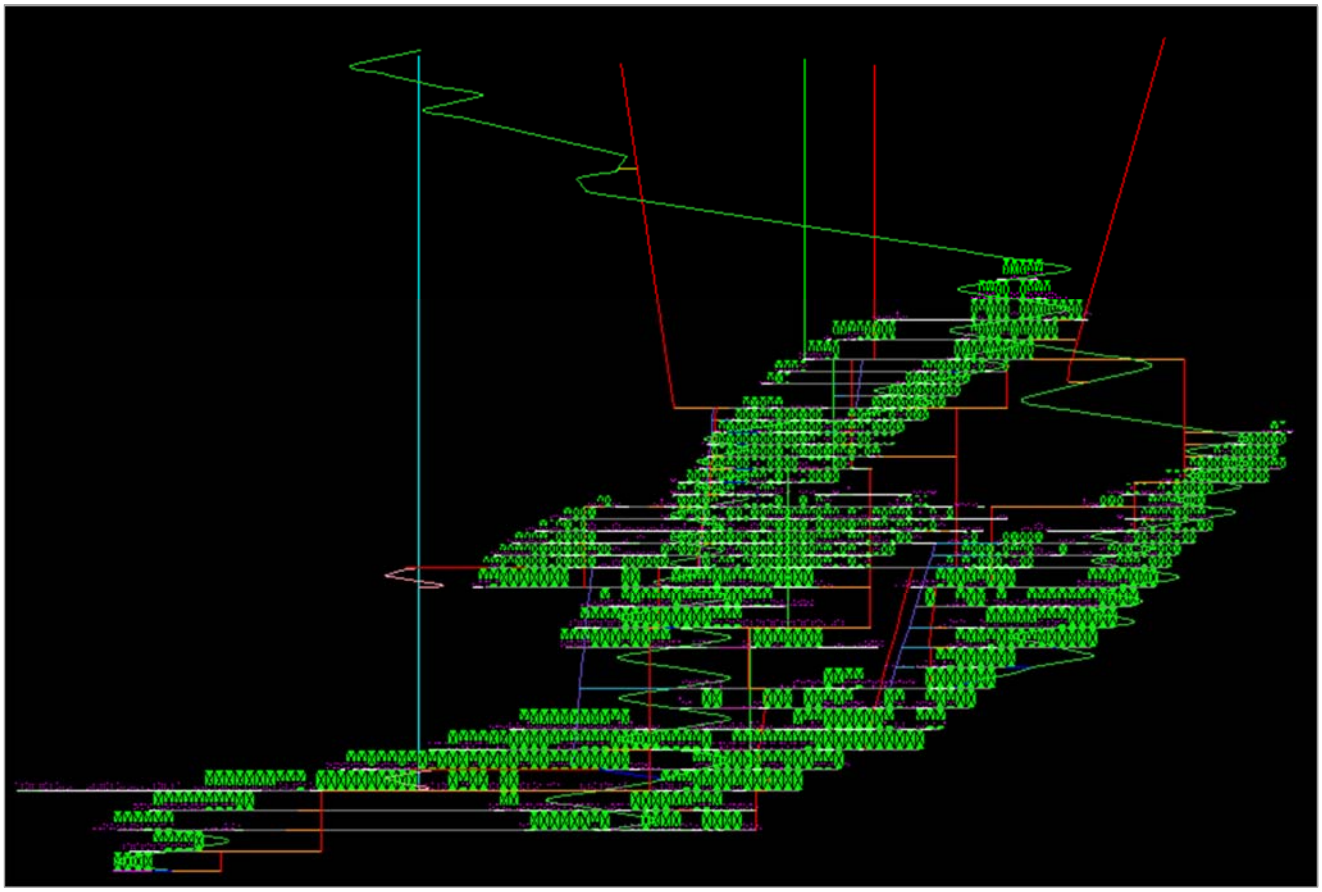
Underground layouts were prepared for the shaft and decline design layout and the development quantities determined by type for cost estimation and scheduling. A typical level design is shown in Figure 16.3 for the 2600 L.

Figure 16.3 Design layout for the 2600 L (plan view)



The long section view of the complete mine design is provided in Figure 16.4.

Figure 16.4 Underground mine design



Vertical development is generally associated with ventilation raises or mineralized material passes. All waste access development was assumed to be 18 ft by 18 ft (5.5 m by 5.5 m) and all development in mineralization to be 14.8 ft by 14.8 ft (4.5 m by 4.5 m). A summary of the development by type is provided in Table 16.6.

Table 16.6 Development quantities by type

Description	Units	Value	Units	Value
Decline	(ft)	39,138	(m)	11,929
Lateral waste development	(ft)	363,147	(m)	110,687
Vertical raise development	(ft)	20,160	(m)	6,145
Vertical shaft development	(ft)	3,625	(m)	1,105
Total lateral development	**(ft)**	**402,285**	**(m)**	**122,616**
Total vertical development	**(ft)**	**23,785**	**(m)**	**7,250**

16.9 Ventilation

The function of the ventilation system is to dilute / remove airborne dust, diesel emissions, explosive gases, and to maintain temperatures at levels necessary to ensure safe production throughout the life of the mine. AMC has undertaken a preliminary estimate of the ventilation requirements in consideration of the production rate, mineralized material handling system, and mining method. This methodology[1] provides an estimate of total mine

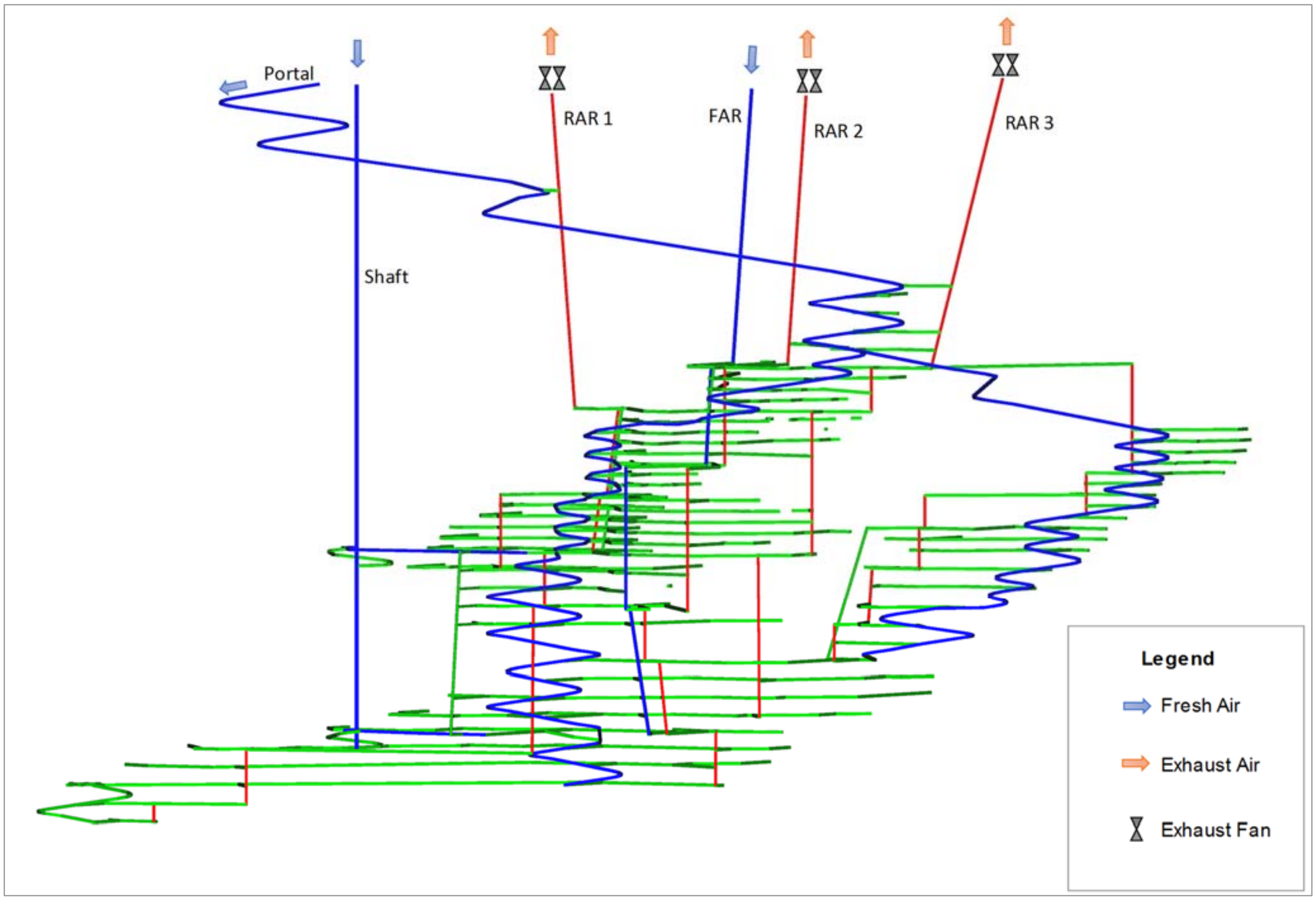
[1] Howes, M.J. (2011) Ventilation and Cooling in Underground Mines. 74. Mining and Quarrying, Armstrong, James R. Menon, Raji, Editor, Encyclopedia of Occupational Health and Safety, Jeanne Mager Stellman, Editor-in-Chief. ILO, Geneva.

airflow for a consistent production rate of 3.6 Mtpa (3.3 Mtonnes pa). It is estimated that for a mine with small open stopes and shaft hoisting including some ramp haulage, the total mine airflow should be 1,983,272 cfm (936 m³/s).

The mine will be ventilated by a "Pull" or exhausting type ventilation system. That is, the primary mine ventilation fans will be located at the primary exhaust airways of the mine. Fresh air will enter the mine via the main intake airways with exhaust to the surface via dedicated return airways. Most production activities will require auxiliary fans and ducting with level airflows managed through regulators located at raise accesses.

Intake air will be provided via the 21 ft (6.5 m) diameter shaft, the decline and one fresh air raise 18 ft (5.5 m) in diameter. Air will be exhausted via three return air raises that are 14.8 ft (4.5 m) in diameter. See Figure 16.5 which shows the ventilation strategy for the mine.

Figure 16.5 Ventilation strategy



16.9.1 Fan selection

Fan sizing estimate was based on:

- Raise diameter and length
- Maximum raise airflow
- Estimated frictional resistance assuming raisebore development
- Estimated fan efficiencies

Table 16.7 shows the primary fan requirements.

Table 16.7 Primary fan selection

Description	RAR 1	RAR 2	RAR 3
Airflow per raise (cfm)	661,090 (312 m^3/s)	661,090 (312 m^3/s)	661,090 (312 m^3/s)
No of fans per raise	2	2	2
Arrangement	Parallel	Parallel	Parallel
Each fan motor size (hp)	800 (597 kW)	1000 (746 kW)	700 (522 kW)

16.10 Backfill

The stopes will be mined in a primary then secondary sequence. All stopes will be backfilled with cemented paste fill. Mining will progress from the centre of the deposit towards the extremities. Paste fill will be distributed underground via pipelines in boreholes that are placed adjacent to the return air raise to the active mining level and then extended as mining progresses. Paste fill will flow under gravity to the active level and to the respective stope for filling. Fill delivery to all sublevels below each main level will be made via a series of inter-linked boreholes that connect to the perimeter drive on each sublevel.

Paste fill will be retained in each stope using a structural arched shotcrete barricade constructed in the stope drawpoint. The barricade will be designed to take the anticipated load from the curing paste fill that will enable a stope to be filled in one continuous pour, subject to paste fill being available. AMC has conducted a high-level evaluation of the paste fill strength required and estimates a fill strength of 400 kPa (60 ft stopes) and 645 kPa (100 ft stopes). A curing time of approximately 21 days prior to mining secondary stopes is recommended.

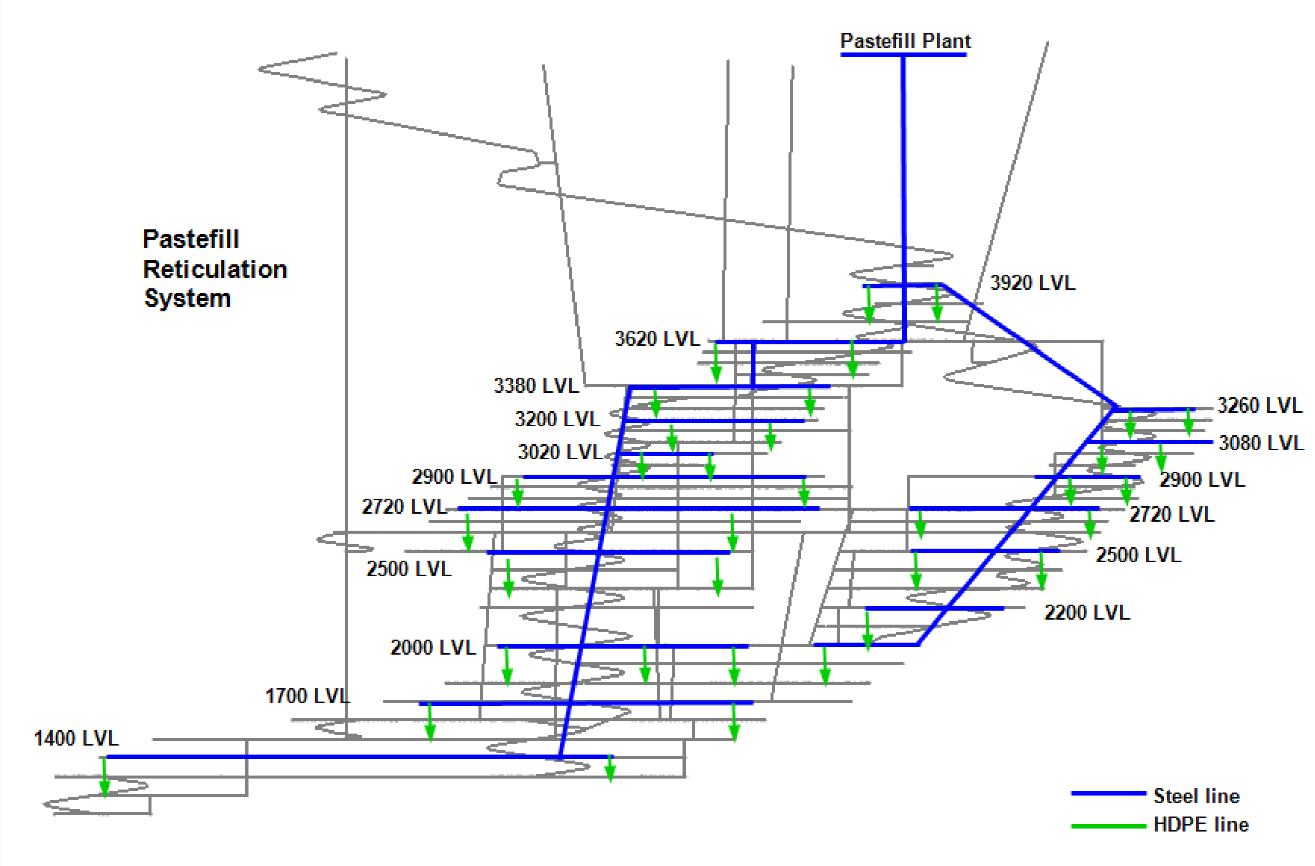
Based on the production rate of 10,000 t/d (9.1 ktonnes pd) and the selected stope sizes, approximately 1,177,155 yards^{3}pa (900,000 m^{3}pa) of paste fill will be required. Key assumptions are summarized in Table 16.8.

Table 16.8 Key assumptions for paste fill

Assumption	Unit (imperial)	Value
Production rate	Mtpa	3.6
Density of the mineralization	t/yd^3	2.40
Volume of mineralization mined	Myd^{3}pa	1.31
Paste fill	Myd^{3}pa	1.18
Mass pull to concentrates	%	14
Tails density	t/yd^3	2.07
Cement dosage	%	4.5
Backfill plant utilization factor	%	55
Tailings produced	Mtpa	2.8
Tailings to paste fill	%	50%
Cement required	ktpa	63.9
Operating cost	US$/t mineralization	4.35
Capital cost of plant	US$M	12

AMC has undertaken high level capital cost estimates for the paste fill plant as well as the distribution system and operating cost of US$4.35 per ton of mineralization (US$4.80/tonne). A schematic of the paste fill distribution system is provided in Figure 16.6. The capital cost estimate for the paste fill plant (US$12M) including EPCM (US$1M) and contingency (US$1M) and the cost for distribution.

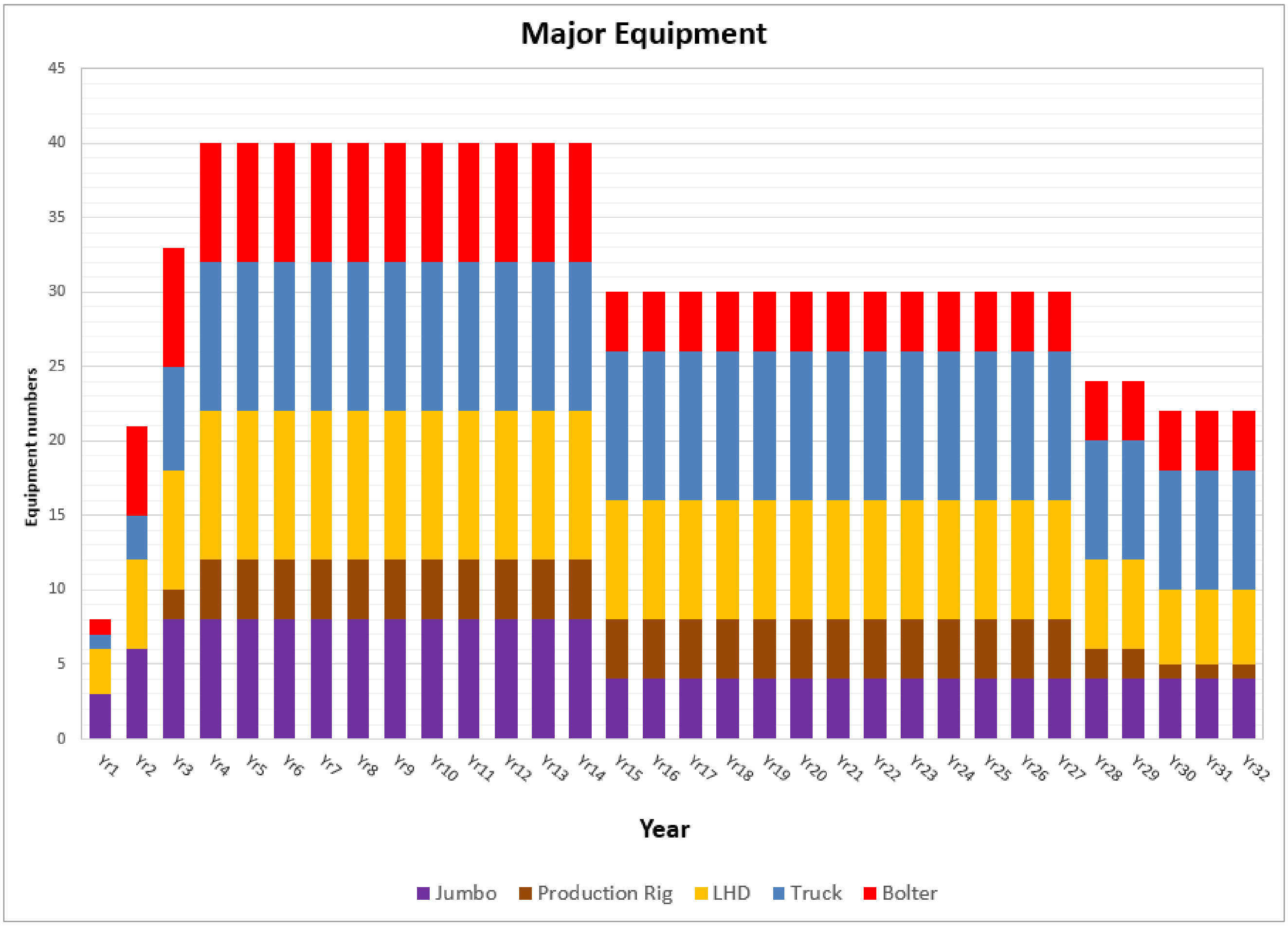
Figure 16.6 Paste fill distribution system



16.11 Underground mining equipment

AMC has completed an estimate of the quantity of major equipment required to meet the production rate of 3.6 Mtpa (3.3 Mtonnes pa). The equipment numbers are based on average haul distances for trucks, number of active crews for development and the number of active stopes required to meet production. Development headings were scheduled at 140 m/month, which drives the crew and equipment numbers. Major equipment numbers are summarized in Figure 16.7. AMC has not selected specific equipment models however recommended equipment includes 2 boom Jumbos with 16 ft (4.9 m) feeds, long hole drills capable of 148 ft (45 m) holes with 50 t underground trucks and 12.5 t loaders.

Figure 16.7 Primary underground equipment

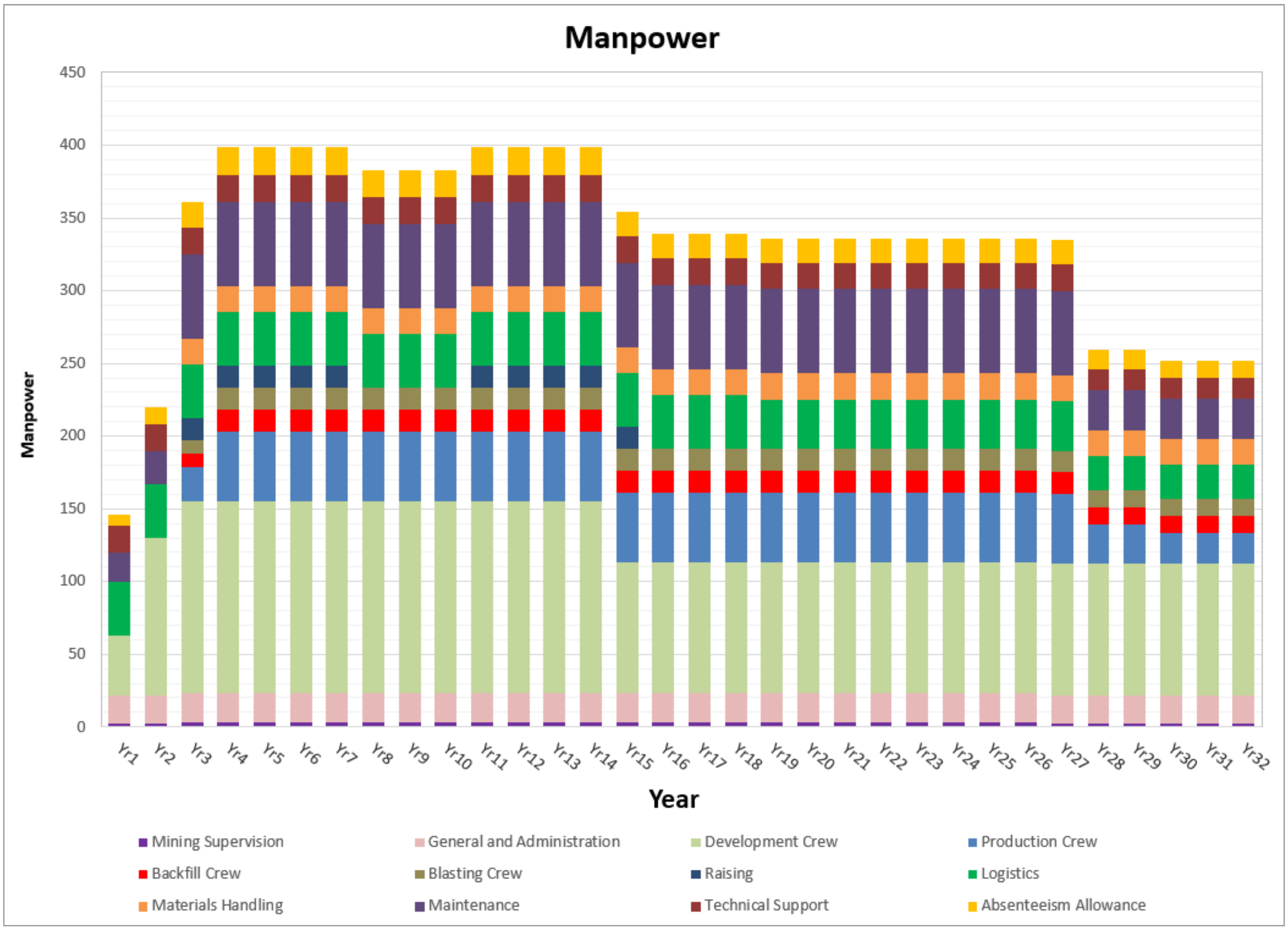


16.12 Underground mining personnel

Based on the primary equipment requirements, AMC undertook an estimate of the expected labour required to meet the development and production schedules. A maximum of 408 personnel will be required for the mine, the workforce will operate on a three-shift basis, and crews will rotate between day shift, night shaft and rostered days off. The mine is assumed to be owner operated and a maximum of 272 underground personnel will be on site each day.

A summary of the workforce is provided in Figure 16.8.

Figure 16.8 Underground work force



16.13 Underground production and development schedule

AMC generated a production and development schedule for the life of mine plan. Key assumptions for the production and development schedules are provided in Table 16.9.

Table 16.9 Key assumptions for the production and development schedules

Assumption	Unit	Value
Development advance rate per end	ft/month	460
Stope size	98 ft (30 m) H by 49 ft (15 m) W by 49 ft (15 m) L = cubic yards	8,829
Tons per stope	t	22,664
Tonnes of mineralization in development	t	1,069
Mineralization density	t/yd^3	2.40
Drilling rate	ft/shift	787
Mucking rate	t/shift	1,378
Backfill rate	cubic yards/hr	589
Curing time	days	21
Effective production rate / stope	t/day	1,000

AMC also determined the shaft sinking schedule based on an average blind sinking rate of 8.2 ft/d (2.5 m/d). The schedule assumes that once the shaft has been sunk to the 2600 L, hoisting can commence, a six-month delay between sinking and hoisting to allow for fitting out the loading station was assumed. During the development stage any mineralization produced will be trucked to surface via the access decline. Single heading development was scheduled at an advance rate of 460 ft/month (140 m/month).

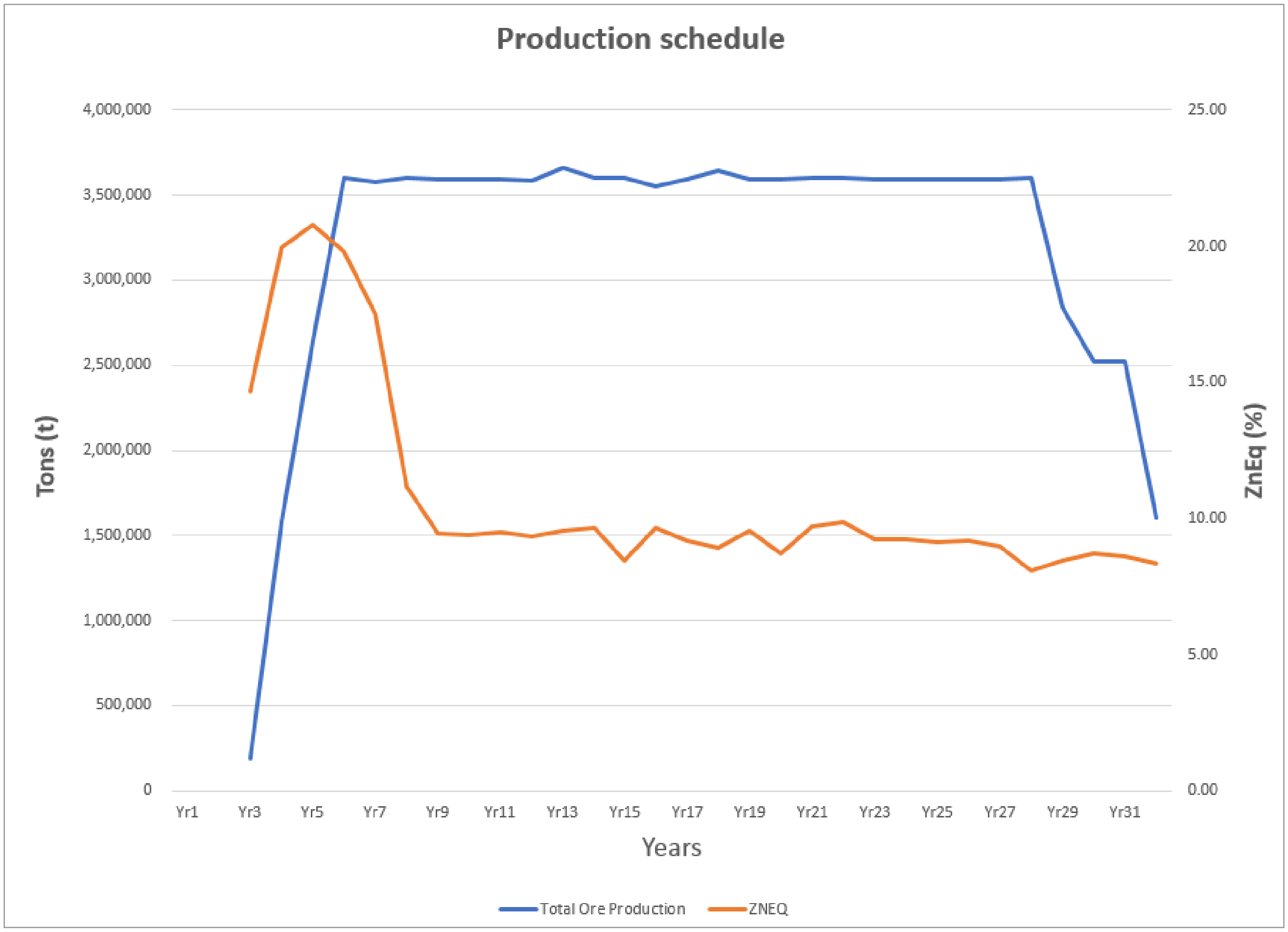
The large stopes are mined at an average rate of 1,000 tpd, with the overall target for the mine being 10,000 tpd. A minimum of 42 stopes are required to be in operation to meet the production rate. A total of 14 stopes per level and an additional level to allow for any unscheduled production delays was considered necessary to meet the production rate.

The cut and fill stopes are mined towards the end of each production level, a production rate of 400 tpd is used for scheduling purposes.

A focused approach was adopted to high grade the initial production years using selective Longhole stoping and filling the stopes with pastefill, lower grade material is extracted as primary and secondary stopes in a second pass. The production schedule reflects this strategy.

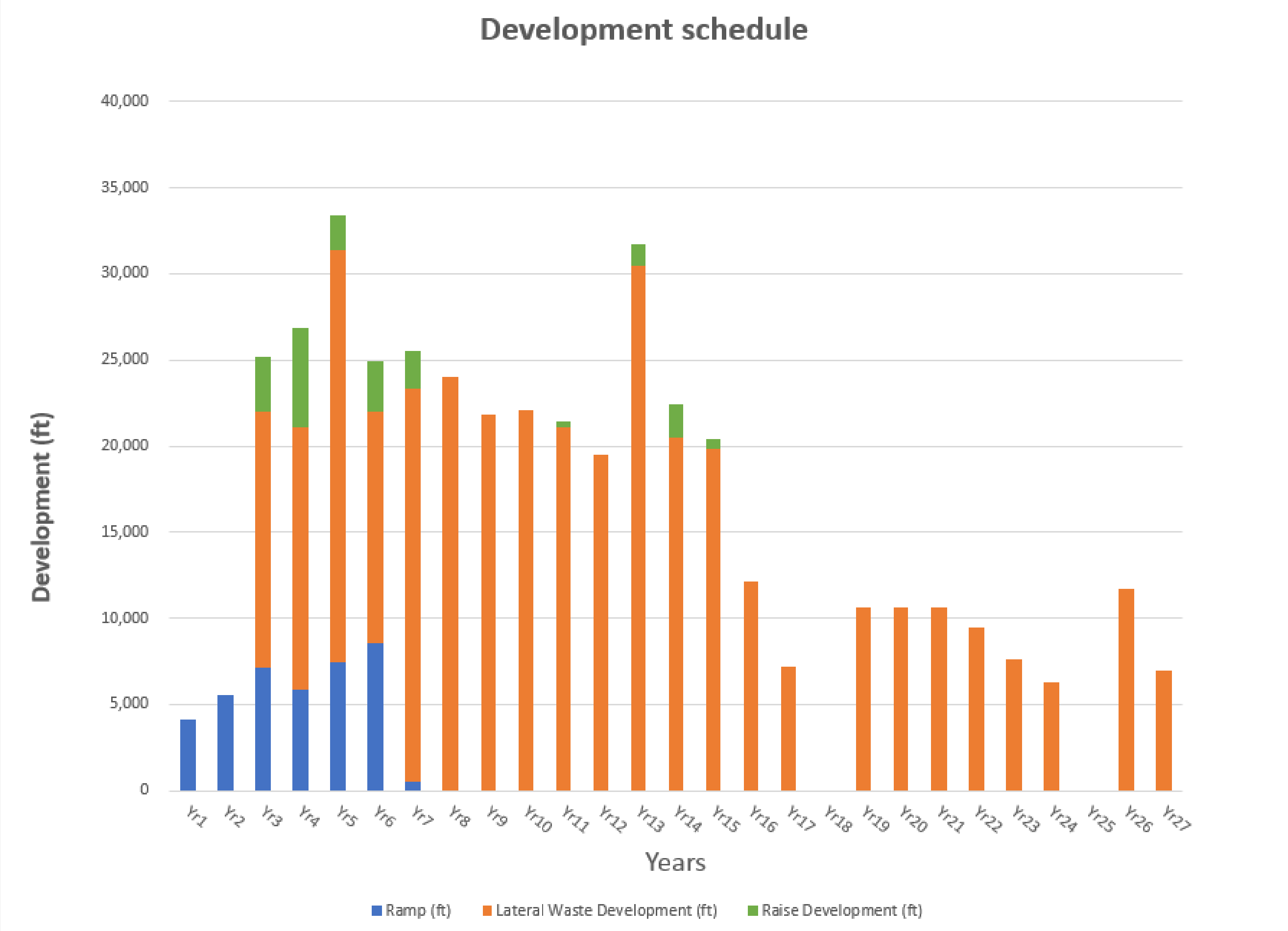
A summary of the production and ZnEq grade is shown in Figure 16.9.

Figure 16.9 Production schedule and ZnEq (%)



Production of mineralization from development commences in Year 3. The development schedule by type is summarized in Figure 16.10.

Figure 16.10 Development schedule by type



The production and development schedule by year is summarized in Table 16.10.

Table 16.10 Production and development schedule

	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17
Tons	0	0	190,227	1,581,878	2,636,000	3,604,186	3,579,876	3,600,988	3,596,374	3,596,692	3,592,262	3,586,242	3,659,867	3,597,797	3,598,004	3,547,947	3,596,736
ZnEq	0.00	0.00	14.66	19.97	20.79	19.77	17.49	11.17	9.44	9.39	9.50	9.35	9.54	9.66	8.45	9.67	9.18
Ag	0.00	0.00	2.38	3.43	3.85	3.88	3.60	2.30	1.81	1.76	1.72	1.63	1.79	1.81	1.61	1.88	1.88
Pb	0.00	0.00	5.43	7.25	7.42	7.31	6.81	4.44	3.69	3.69	3.68	3.61	3.69	3.68	3.41	4.16	3.90
Zn	0.00	0.00	7.01	9.50	9.75	8.81	7.30	4.56	4.05	4.06	4.21	4.22	4.17	4.28	3.54	3.75	3.52
Ramp (ft)	4,134	5,512	7,100	5,875	7,487	8,544	485	0	0	0	0	0	0	0	0	0	0
Level (ft)	0	0	14,927	15,240	23,849	13,463	22,847	24,043	21,818	22,084	21,101	19,479	30,490	20,545	19,824	12,120	7,251
Raise (ft)	0	0	3,176	5,728	2,051	2,905	2,225	0	0	0	301	0	1,256	1,918	600	0	0
Shaft (ft)	899	899	899	928	0	0	0	0	0	0	0	0	0	0	0	0	0
Waste (tons)	115,537	154,050	733,548	802,834	951,942	722,926	734,711	671,955	609,771	617,205	600,909	544,400	898,776	645,415	576,323	338,724	202,651
Pastefill (tons)	0	0	86,458	718,964	1,198,062	1,638,102	1,627,054	1,636,649	1,634,552	1,634,696	1,632,683	1,629,947	1,663,409	1,635,199	1,635,293	1,612,542	1,634,717
	Yr18	**Yr19**	**Yr20**	**Yr21**	**Yr22**	**Yr23**	**Yr24**	**Yr25**	**Yr26**	**Yr27**	**Yr28**	**Yr29**	**Yr30**	**Yr31**	**Yr32**	**Yr33**	**Totals**
Tons	3,646,487	3,595,810	3,595,831	3,597,967	3,598,226	3,595,791	3,595,839	3,596,577	3,596,006	3,596,245	3,598,163	2,843,596	2,524,386	2,524,386	1,600,157		96,670,545
ZnEq	8.91	9.55	8.73	9.69	9.85	9.22	9.24	9.12	9.17	8.97	8.10	8.48	8.73	8.58	8.36		10.44
Ag	1.97	2.05	2.11	2.18	2.34	2.13	2.30	2.30	2.35	2.26	1.94	2.09	2.17	2.21	1.83		2.22
Pb	3.85	4.26	3.89	4.66	4.55	4.34	4.30	4.20	4.25	4.17	3.56	3.63	3.60	3.66	3.80		4.34
Zn	3.21	3.36	2.85	2.99	3.09	2.87	2.77	2.75	2.70	2.66	2.71	2.86	3.08	2.83	2.84		4.01
Ramp (ft)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	39,138
Level (ft)	0	10,673	10,664	10,673	9,507	7,610	6,296	0	11,715	6,929	0	0	0	0	0	0	363,147
Raise (ft)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	20,160
Shaft (ft)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3,625
Waste (tons)	0	298,290	298,038	298,290	265,702	212,685	175,961	0	327,412	193,652	0	0	0	0	0		11,991,707
Pastefill (tons)	1,657,328	1,634,296	1,634,305	1,635,276	1,635,394	1,634,287	1,634,309	1,634,644	1,634,385	1,634,494	1,635,365	1,292,415	1,147,333	1,147,333	727,271	0	43,936,763

16.13.1 Proposed underground infrastructure

The proposed underground mine services will include a small maintenance shop for minor and urgent repairs, fuel and lubricant storage, and a small explosives magazine.

Compressed air will be supplied by mobile electric compressors. The compressors will be relocated to active mining levels as needed.

During development the decline will be equipped with power for distribution underground as well as pipelines for mine service water and dewatering. Telecommunications will be provided by a conventional leaky feeder system.

Details of the underground infrastructure are provided in Section 18.

17 Recovery methods

17.1 Introduction

This section defines the process design criteria to be applied to the crushing, grinding, flotation, and dewatering facilities for a 10,000 stpd (9,072 tonnes per day) lead, zinc and silver mineral processing facility for the Taylor Deposit, to be located 50 miles (80 km) southeast of Tucson, Arizona, and 8 miles (13 km) north of the USA border with Mexico.

The crushing plant will process the run-of-mine (ROM) material by using a primary jaw crusher to reduce the material from a nominal 20 inch to a 100% Passing (P_{100}) size of 243 mm (P_{80} of 117 mm).

The grinding circuit will be a semi-autogenous (SAG) mill - ball mill grinding circuit with subsequent processing in a differential flotation circuit. The SAG mill will operate in closed circuit with a vibrating screen. The ball mill will operate in closed circuit with hydrocyclones.

Cyclone overflow, the grinding circuit product, is fed to the flotation plant. The flotation plant will consist of separate lead and zinc flotation circuits. The lead flotation circuit will consist of rougher flotation and three-stage cleaner flotation. The zinc flotation circuit will consist of rougher flotation and two-stage cleaner flotation.

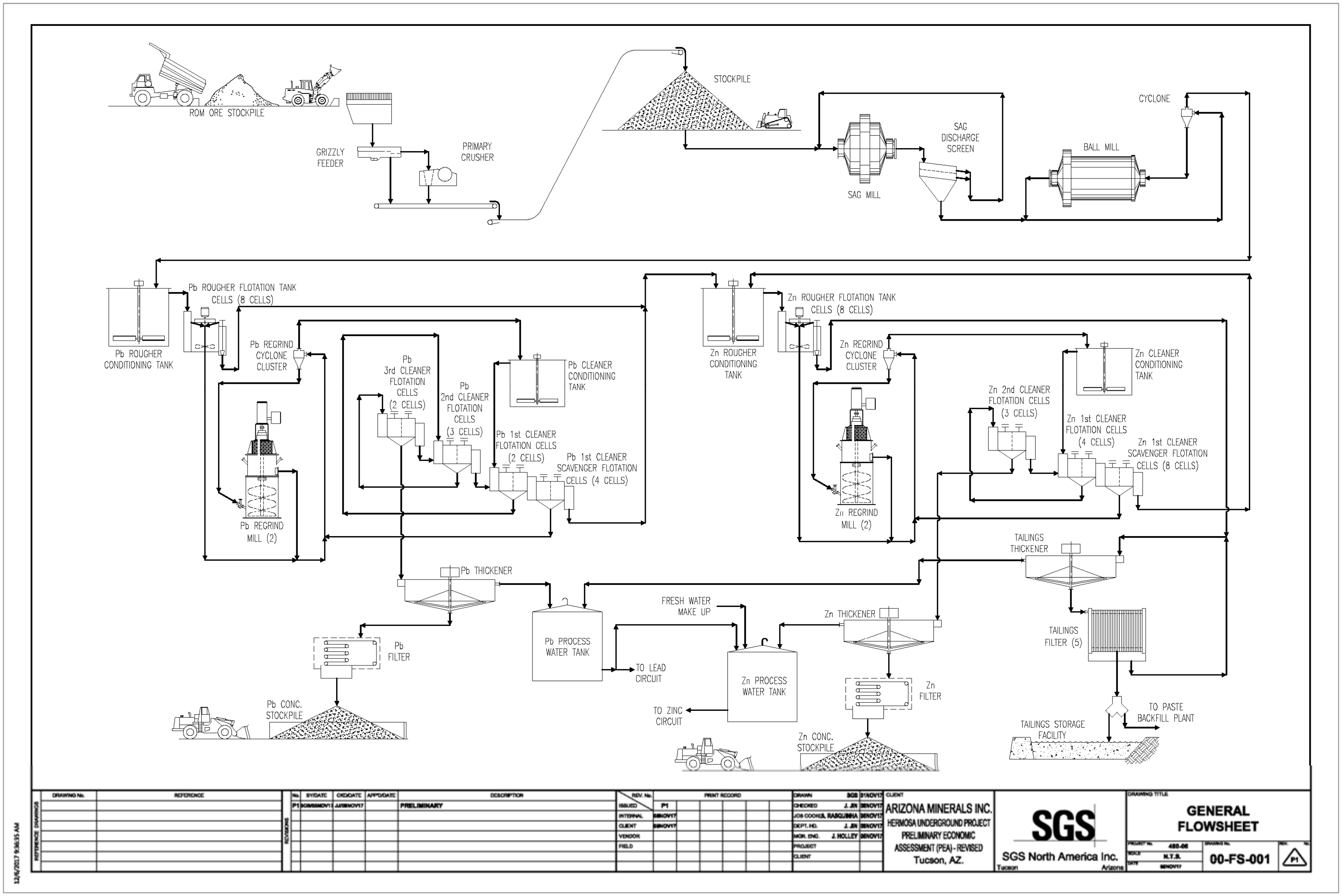
Both lead and zinc concentrates are thickened, filtered, and stored in concentrate storage facilities prior to loading onto trucks for shipment.

Zinc rougher flotation tailing will be the final tailing. Tailing thickener underflow will be pumped to a tailing filtration facility. After filtration, sixty percent (60%) of final tailing will be transferred to the backfill plant and the remainder will be transferred to a dry stack tailings storage facility (TSF).

Plant water stream types include: lead process water, zinc process water, fresh water, and potable water.

The overall flowsheet is shown in Figure 17.1.

Figure 17.1 Process plant overall flowsheet



17.2 Primary crushing

ROM material is transported to the crushing plant area by rear-dump trucks. The primary crushing line consists of a dump hopper, grizzly screen, rock breakers, crusher and associated dust collection and transfer equipment. ROM material is dumped into the dump hopper using a front end loader. The grizzly screen oversize feeds the jaw crusher. Two mobile rock breakers are available, one to service the crusher or screen and another one to service the ROM area stockpile. The crusher reduces the ROM size from a maximum of 19.7 in (500 mm) to approximately P_{100} of 9.6 in (243 mm). Crushed material drops onto a belt conveyor that transports it to a stockpile.

The crushing production rate will be monitored by a belt scale mounted on the conveyor. Tramp iron will be removed using a magnet that will be located at the discharge of the primary crusher discharge conveyor. A metal detector will be installed over the conveyor. Dust is controlled in the dump pocket with water sprays and dust collector vents positioned at the conveyor transfer points. An air compressor and instrument air dryer will be installed for operation and maintenance. A mobile crane will also be installed for maintenance of the primary crusher.

17.3 Crushed material conveying, transport, and storage

Primary crushed material will be stockpiled on the ground. A reclaim tunnel will be installed beneath the stockpile. The stockpile will contain approximately 10,000 tons of “live” storage (9,072 tonnes). When required, the material will be moved from the “dead” storage area to the “live” storage area by a front-end loader.

Material will be withdrawn from the coarse reclaim stockpile by variable speed belt feeders. The feeders will discharge to the transfer conveyor belt. The transfer conveyor will discharge to the SAG mill in the grinding circuit. The reclaim rate will be monitored by a belt scale mounted on the conveyor.

Dust control in the stockpile area will be achieved using a wet type dust collection system. One of the two dust collector systems will be installed to control dust at the discharge of the stockpile feed conveyor and another one will be installed to control dust in the reclaim tunnel.

17.4 Grinding

The mineralized material will be ground in a SAG mill primary grinding circuit and a ball mill secondary grinding circuit.

The SAG mill will operate in closed circuit with a vibrating screen. Water is added to the SAG mill to produce a slurry and the material feed size is reduced as it traverses the SAG mill. The SAG mill discharges onto a double deck screen with 10.0 mm sized bottom openings. Screen oversize is recirculated to the SAG mill feed chute by a series of conveyors. Screen undersize will flow by gravity to the cyclone feed pump box. A belt scale mounted on the recycle conveyor will monitor the SAG mill recycle rate. The target SAG grind is P_{80} of 2,191 microns.

Secondary grinding will be performed in a ball mill. The ball mill will operate in closed circuit with hydrocyclones. Ball mill discharge will be combined with vibrating screen undersize in the cyclone feed pump box and will be pumped to a hydrocyclone cluster. Combined slurry will be pumped using variable speed horizontal centrifugal slurry pumps (one operating and one standby) to the cyclone clusters.

Hydrocyclone overflow (final grinding circuit product at 80% minus 150 microns) will flow by gravity to the tramp oversize screen positioned prior to the flotation circuit.

Cyclone overflow will be sampled by a primary sampler and analysed by the lead and zinc on-stream analyser for metallurgical control prior to flotation. Cyclone overflow from the cyclone cluster will also be monitored for particle size distribution by a particle size monitor.

Zinc sulfate ($ZnSO_4$) and sodium cyanide (NaCN) will be added into the ball mill.

Grinding balls will be added to the SAG mill and ball mill by ball loading systems. Air compressors and an instrument air dryer will provide service and instrument air for operations and maintenance. An overhead crane will be installed for maintenance of the grinding mills.

17.5 Lead flotation and regrind

Hydrocyclone overflow will flow by gravity to the lead flotation circuit. The lead flotation circuit will consist of one row of rougher cells and one row of cleaner cells. The rougher row will consist of eight (8) 1,766 ft^3 (50 m^3) tank type rougher flotation cells with a drop between each cell. The lead rougher concentrate will be sampled by a rougher concentrate primary sampler and pumped (one operating pump and one spare) to the lead regrind mill circuit. Reground lead rougher concentrate will flow by gravity from the lead cleaner conditioning tank to the lead first cleaner flotation cells. The lead cleaner row consists of eleven (11) flotation cells; two (2) 11 $yard^3$ (8.5 m^3) forced air first cleaner cell, four (4) 300 ft^3 (8.5 m^3) forced air first cleaner scavenger cells, three (3) 100 ft^3 (2.8 m^3) forced air second cleaner cells, and two (2) 100 ft^3 (2.8 m^3) forced air third cleaner cells. The lead first cleaner concentrate is pumped (one operating pump and one spare) into the second cleaner flotation cells. Lead rougher tailing and lead first cleaner scavenger tailing will flow by gravity into the zinc rougher conditioning tank. The lead second cleaner concentrate will be pumped to the lead third cleaner flotation cells. The lead third cleaner concentrate will flow by gravity to the lead concentrate thickener.

The concentrate samples cut by the samplers will be analysed for process control by the lead and zinc on-stream analyser. Tailing from rougher flotation cells and first cleaner scavenger cells will be combined together and sampled with primary samplers and analysed by the lead and zinc on-stream analyser.

Lead rougher concentrate will be pumped to the lead regrind cyclone feed pump box and combined with the regrind mill discharge. The combined slurry will be pumped using horizontal centrifugal slurry pumps (one operating and one spare) to a hydrocyclone cluster. Overflow from the regrind cyclone cluster (final regrind circuit product) will be sampled for particle size distribution analysis by the lead regrind cyclone particle size monitor. It will then be analysed by the lead and zinc on-stream analyser and flow by gravity to the lead cleaner conditioning tank. The cyclone underflow will flow by gravity to the lead regrind mill. Product from the regrind mill will report to the lead regrind cyclone feed pump box.

Air compressors, air receivers, and instrument air dryer will be installed for general plant operation and maintenance.

A bridge crane will be installed for maintenance of the flotation and regrind equipment.

17.6 Zinc flotation and regrind

Lead rougher tailing and lead first cleaner scavenger tailing will flow by gravity to a zinc rougher conditioning tank. The zinc flotation circuit will consist of one row of rougher cells and one row of cleaner cells. The rougher row will consist of eight (8) 1,766 ft^3 (50 m^3) tank type rougher flotation cells. The zinc rougher concentrate will be sampled by the zinc rougher concentrate primary sampler and pumped (one operating pump and one spare) to the zinc regrind mill circuit. The zinc cleaner row consists of fifteen (15) flotation cells; one bank of four (4) 300 ft^3 (8.5 m^3) forced air first cleaner flotation cells, eight (8) 300 ft^3 (8.5 m^3) forced air first cleaner scavenger flotation cells, and three (3) 100 ft^3 (2.8 m^3) forced air second cleaner flotation cells. Tailing from zinc rougher cells will flow by gravity to the tailing sample box, then to the tailing thickener.

Reground zinc rougher concentrate will flow by gravity from the zinc cleaner conditioning tank to the zinc first cleaner flotation cells. The zinc first cleaner concentrate will be pumped (one operating pump and one spare) into the zinc second cleaner flotation cell. The zinc secondary cleaner flotation concentrate will be pumped to the zinc concentrate thickener.

The concentrate samples cut by the samplers will be analysed for process control by the lead and zinc on-stream analyser. Tailing from rougher flotation cells and first cleaner scavenger cells will be sampled with primary samplers and then analysed by the lead and zinc on-stream analyser.

Zinc rougher concentrate will be pumped to a zinc regrind hydrocyclone feed pump box and combined with the zinc regrind mill discharge. The combined slurry will be pumped using horizontal centrifugal slurry pumps (one operating and one spare) to the zinc regrind hydrocyclone cluster. Overflow from the zinc regrind cyclone cluster will be sampled for particle size distribution analysis by the zinc regrind cyclone particle size monitor. It will then be analysed by the lead and zinc on-stream analyser and will flow by gravity to the zinc cleaner conditioning tank. The underflow will flow by gravity to the zinc regrind mill. Product from the regrind mill will report to the zinc regrind cyclone feed pump box.

17.7 Lead concentrate dewatering

Concentrate from the lead third cleaner flotation cells will be pumped to a lead concentrate thickener. The concentrate thickener overflow will be pumped back to the thickener feed for dilution and the thickener spray bar; to control froth, or to the lead process water tank. The concentrate thickener underflow will be pumped (one operating pump and one spare) to an agitated storage tank and then to a pressure filter. Filter cake will discharge to a covered lead concentrate stockpile.

Concentrates, both lead and zinc, will be reclaimed by front-end loader onto highway haulage trucks. A truck scale will be located near the concentrate load out area.

17.8 Zinc concentrate dewatering

Concentrate from the zinc secondary cleaner flotation cell will be pumped to a zinc concentrate thickener. The concentrate thickener overflow will be pumped back to the thickener feed for dilution and the thickener spray bar; to control froth, or to the zinc process water tank. The concentrate thickener underflow will be pumped to an agitated storage tank and then to a pressure filter. Filter cake will discharge to a zinc concentrate covered stockpile.

17.9 Tailing dewatering

Tailings from the zinc rougher flotation circuit will flow by gravity to a high rate tailings thickener. Thickener overflow will flow by gravity from the tailings thickener overflow tank to the process water tank. Thickener underflow will be pumped by variable speed horizontal centrifugal slurry pumps (one operating and one stand-by) to the tailing filter feed tank.

Tailings slurry will be pumped from the tailing filter feed tank by horizontal centrifugal pumps to feed slurry to five (5) tailing filters (four filters will normally be in operation with one (1) filter on stand-by.) Tailing filter cake from the filters will discharge to a series of conveyor belts. After filtration, sixty percent (60%) of final tailing will be transferred to the backfill plant and the remainder will be discharged to a mobile / stacking conveyor system to build dry stack tailings disposal area.

Filtrate will flow by gravity to a filtrate surge tank. The filtrate transfer pumps (one operating and one stand-by) will return filtrate from the filtrate surge tank to the tailing thickener distribution box.

17.10 Tailing deposition

Damp tailings from the tailings filters will be transported to a tailings disposal area. Filtered tailings will be delivered by conveyors and placed behind a tailings buttress with a radial stacker similar to that used for some heap leach operations. A dozer will be used to spread the filtered tailings and provide sufficient compaction for trafficability of the conveyors and stacker. The active stacking area will be limited to minimize dust and erosion.

Advantages of the dry tailings disposal over conventional tailings disposal is that it eliminates the need for an engineered embankment and seepage containment system, maximizes water conservation and minimizes water makeup requirements. Dry tailings disposal also results in a very compact site and limited ground disturbance.

17.11 Reagents

Reagents requiring receiving, handling, mixing, and distribution systems include:

- Zinc sulfate ($ZnSO_4 \cdot 7H_2O$)
- Aerofloat 242 (promoter)
- Carboxymethyl cellulose (CMC)
- Copper sulfate ($CuSO_4 \cdot 5H_2O$)
- Sodium isopropyl xanthate (SIPX)
- Methyl isobutyl carbinol (MIBC, frother)
- Sodium cyanide (NaCN) .
- Flocculant

17.12 Water system

17.12.1 Fresh water

Fresh water will be supplied from wells located on the property. Fresh water from the wells will be pumped to a fresh water tank (also used for fire suppression). The fresh water distribution system provides fresh water for process requirements such as process water makeup, reagent mixing and gland water. Controls will be installed to ensure flow to the process water system when the raw water system is operating. From the fresh water tank, low pressure process water will flow to the systems that do not require high pressure. Booster pumps will be installed to provide high pressure water to the systems that require it; including pump gland water. Gland water is provided for sealing each pump without return. Pumps and control systems will be installed at the fresh water tank to provide pressure to the fire suppression system.

17.12.2 Process water

17.12.2.1 Process water – lead circuit

The lead process water tank will receive overflow from the lead concentrate thickener, and tailing thickener. The lead process water will be used as makeup water in the primary cyclone feed sump. Fresh water can be added to the lead process water tank if necessary. This lead process water is not suitable for general distribution throughout the process plant.

17.12.2.2 Process water – zinc circuit

Overflow from the zinc concentrate thickener and lead process water excess overflow will be recycled to the zinc process water tank, and will be used as makeup water in the zinc flotation circuit. Fresh water can be added to the zinc process water tank.

18 Project infrastructure

18.1 Surface infrastructure

The Project is located approximately 9 miles (14.5 km) from Patagonia Arizona. The first 6 miles (9.7 km) of this road is paved and the last 3 miles (4.8 km) to the mine property is a dirt road. Alternate routes to the site from the county seat of Nogales are a mixture of paved roads and dirt roads that would require upgrading. A major rail hub is located approximately 15 miles (24 km) south, near the city of Nogales. There are water wells on the property and an overhead electrical power line to the property. However, these will also need to be upgraded for the Hermosa project.

The following paragraphs discuss the proposed upgrades required for the proposed mining and processing equipment and associated infrastructure.

A general arrangement drawing of surface infrastructure is provided in Figure 18.1 and a detailed surface layout of the key infrastructure in Figure 18.2.

Figure 18.1 General arrangement drawing of the site infrastructure

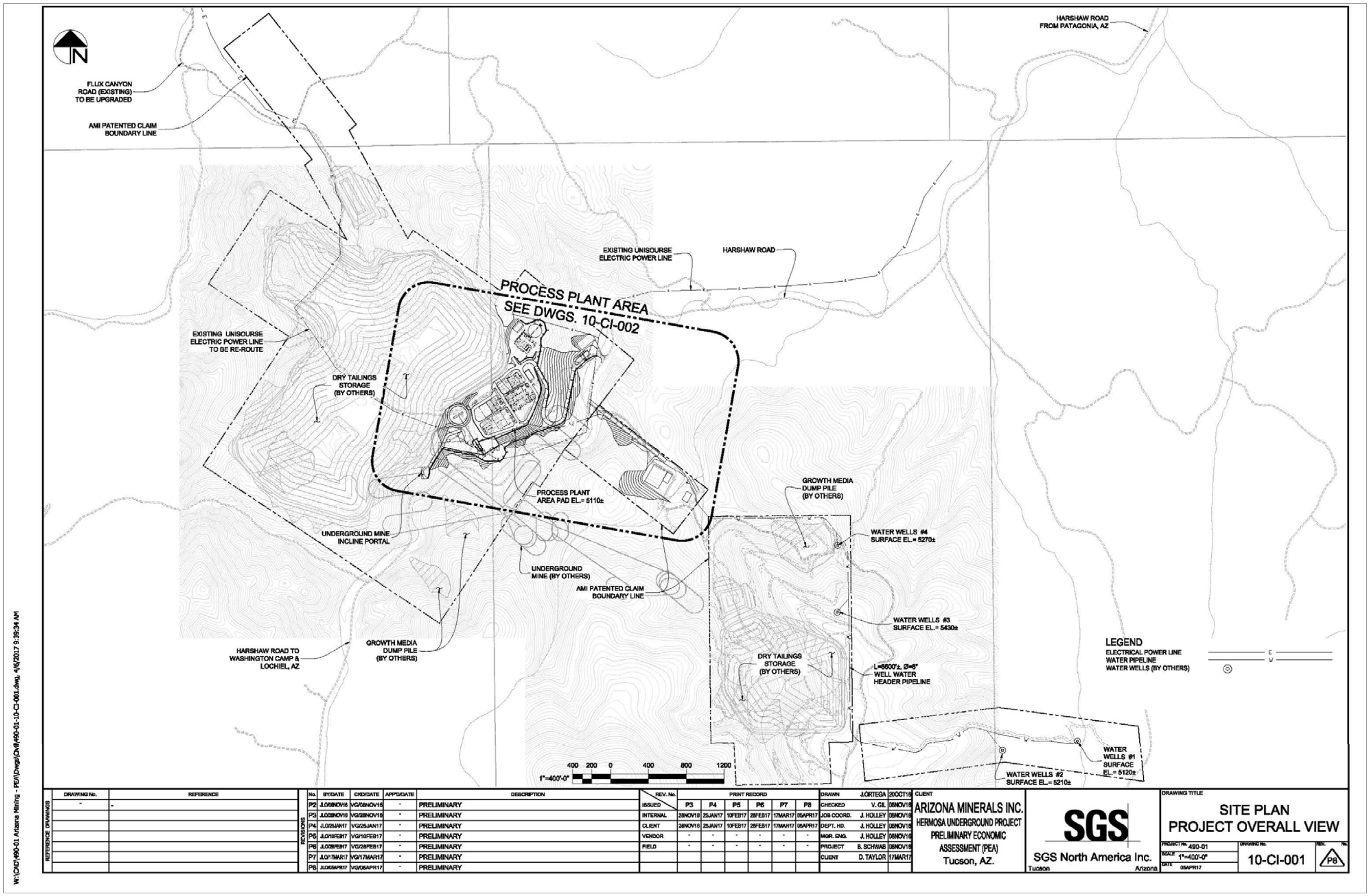
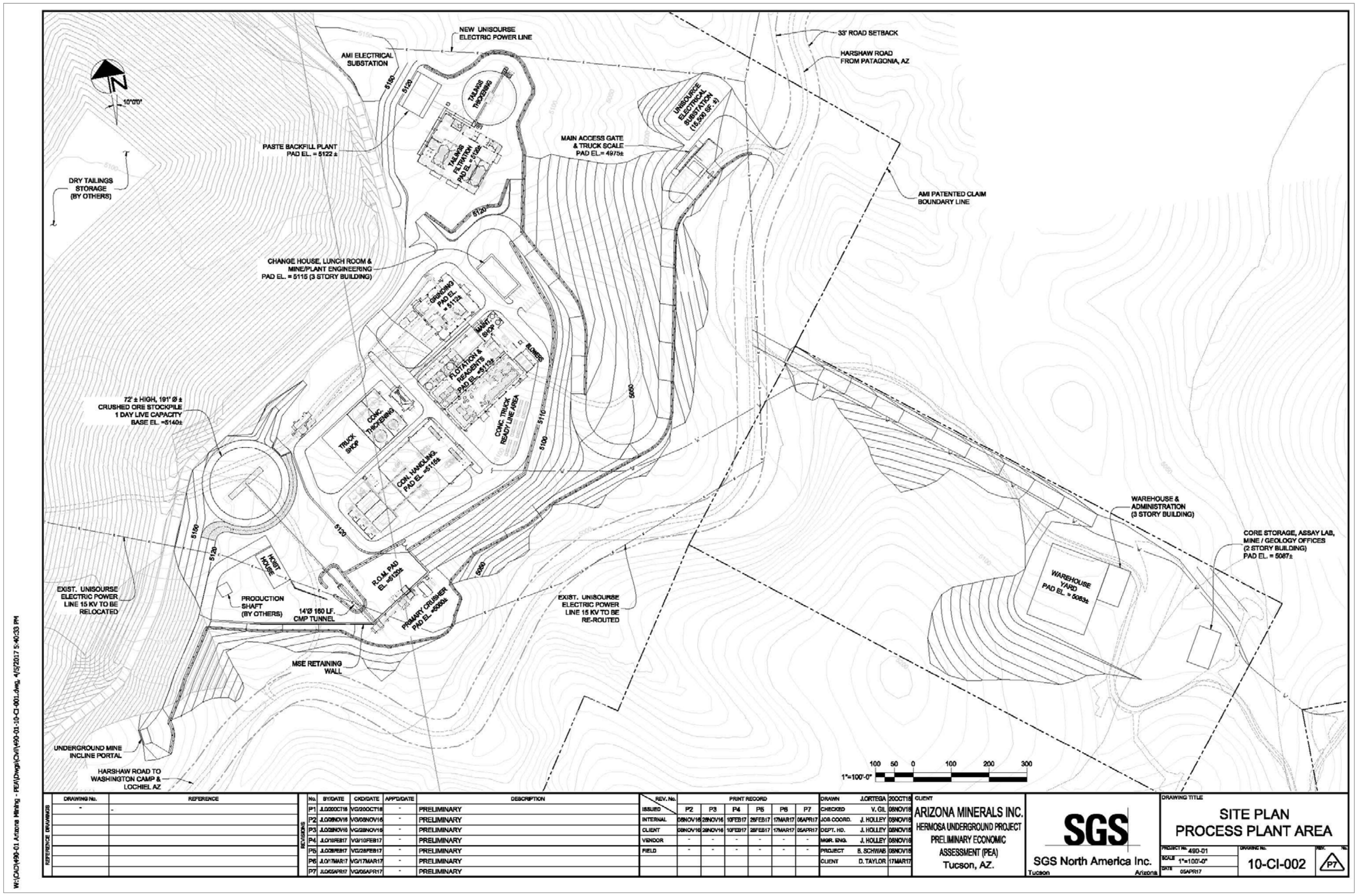


Figure 18.2 Detailed surface drawing of key infrastructure



18.1.1 Power

There is an existing power line to site that runs parallel to the Flux Canyon road. However, this is not adequate for development of the mine site. A new electrical line will be required for the proposed electrical loads required for the project.

18.1.1.1 Power supply

Electric power to the site will be supplied via an overhead utility (UniSource Energy Services [UES]) transmission line rated 69 kV. There is an existing 13.2 kV distribution line along Flux Canyon Road that will be used to supply power during the construction period. The 69 kV line will be 23 miles (37 km) long, originating in Rio Rico.

The utility transmission line will include a new switchyard near the Sonoita substation in Rio Rico and terminate in the mine main substation yard. The new Sonoita switchyard will include connections to the existing 138 kV transmission line and associated switchgear for installing the new transmission line to the mine site. The utility connection in the mine main substation will include 69 kV terminations, circuit protection, metering and connections to the main substation transformers. This utility connection will be the boundary limit between AMI and UES. The 69 kV line will satisfy all the power requirements of the project based on preliminary discussions with the utility.

Electrical grid power supplied by UES is assumed for this study, however, an alternative source of power supply was explored which included natural gas-powered generation on the mine property. In this alternate option natural gas would be supplied to the mine site by El Paso Natural Gas (EPNG) that would include connecting into an existing transmission pipeline, a new compressor station and a new distribution pipeline to the mine. Electrical power would be generated by a natural gas gen-set system that would include gen-sets operating in parallel, a cooling system, other ancillary systems and controlling switchgear enclosed in a building. This alternative will be further explored at the next level of engineering to provide the best option for supplying power to the mine.

18.1.1.2 Power distribution

The mine's main power substation will step down the utility transmission line voltage to 24.9 kV via two (2) 40 MVA transformers and distribute power on site using 24.9 kV switchgear to the crusher area, mill building, flotation building, filtration and concentrate handling area, tailings thickening and filtration area, underground mine, ancillary buildings and fresh water pump stations. The power distribution will be via underground duct-bank to nearby process plant and ancillary facilities and overhead power lines to remote facilities. The main substation transformers will be sized to handle the entire plant load with 100% redundancy. Thus, if one transformer fails, the other can pick up the entire load of the project to allow continued operation of the plant. The power requirements of the project are listed in Table 18.1. The installed power requirements were established using the process design criteria, equipment list (by SGS) and underground mine design plan (by AMC).

Table 18.1 Electrical power requirements for each area of the project

Area description	MW
Underground (U/G) mine	16.7
Crushing	0.4
Grinding	11.4
Flotation	4.0
Filtration & concentrate handling	1.2
Tailings	2.2
Ancillary buildings	0.3
Water supply	0.3
Contingency	3.5
Total electrical load	40.0

The total power demand for the project is 40 MW. This includes a 10% contingency for future additions during feasibility and detail engineering. The process plant includes crushing, grinding, flotation, filtration, concentrate handling, tailing, air and water supply, and process and ancillary buildings.

18.1.2 Water

There are currently several small to large fresh water wells on the property. Additional fresh water capacity is continuing to be developed.

18.1.2.1 Fresh water

Based on preliminary review field studies by Clear Creek and evaluation of pumping rates and water-level drawdown data from two existing on-site supply wells, there appears to be adequate groundwater supply available at the project site. Well WW-1 was pumped for three days (72 hours) at a rate of 1,103 gpm (4,175 LPM). The pumping well had a maximum drawdown of 110 ft (33 m) while a monitor well 32 ft (10 m) away had only 8 ft (2.4 m) of drawdown. Most of the drawdown in the pumping well was due to well inefficiency (turbulent flow). Other wells ranging in distance up to about 1,350 ft (411 m) from the pumping well exhibited similar drawdowns of around 8 ft (2.4 m).

A pumping test at well HT-1 was also conducted for a three-day period at a pumping rate of 140 gpm (530 LPM). Drawdowns ranged from 0.8 to 3.1 ft (0.2 m to 0.9 m) in observation wells ranging in distance from 151 to 3,059 ft (46 m to 932 m) from the pumping well. HT-1 had 132 ft (40 m) of drawdown; again, mostly as a result of inefficient turbulent flow near the well. These results indicate the presence of a well-connected fractured rock aquifer with significant water storage. The preliminary evaluation included a simple analytical model that simulated drawdown for a 20-year period. This model was based on an aquifer transmissivity value estimated from the supply well's specific capacity.

SGS has included costs for pumps, water distribution pipelines and storage tanks for fresh, process water, and fire water required for the Hermosa project. For this study it is assumed that there will be four wells of which any two will provide the required fresh water for the project. The groundwater supply system is designed to provide operational flexibility and water storage capacity while utilizing conventional equipment and construction materials.

Fresh water and fire water pumps to distribute water as required on the project site, have been determined, and fresh water will be utilized for the following:

- Fire suppression system
- Process system (lead and zinc process water tank make-up)
- Potable water treatment system
- Reagent mixing and seal water

18.1.2.2 Process water and distribution

Process water storage tanks will be included in the process plant area and placed on a concrete containment curbed area. Reclaim water from the lead and zinc process system will be recycled into lead and zinc process water tanks for distribution to the processing facilities.

18.1.2.3 Potable water

A packaged potable water treatment system for the process plant operation and non-process buildings is included in the process plant design.

18.1.3 Access roads

Three access routes to the mine property are being reviewed. Each route is along existing improved and unimproved roadways. For purposes of this document, the cost of the most conservative option was used. The proposed improvements for all access road options will be most easily constructed within existing roadway right-of-way and easements. Additionally, all proposed routing and upgrades will allow for a higher design speed and the ability to maintain the posted operating speeds. It is noted that this preferred access route may result, if implemented, in additional improvement requirements. These potential new improvements or any operating restrictions could arise through the necessary coordination with the town of Patagonia, Santa Cruz County, and possibly others. Issues such as these are routinely identified and mitigated during the Feasibility Study.

On the mine property there are currently exploration access roads for the drills; however, these will be extended and upgraded for the project. These roads will allow access to the on-site processing facilities and non-process buildings.

18.1.4 Process and associated buildings and structures

The following non-process buildings are included in the Project:

- Core shed
- Mill change room
- Process area lunch rooms
- Warehouse
- Truck shop
- Truck wash area
- Maintenance shops
- Plant engineering building
- Assay laboratory
- Concrete batch plant
- Multi-purpose room / training room
- Infirmary / ambulance area
- Emergency generators
- Reagent storage
- Truck scale and guard gate
- Electrical substations

The following process buildings are included in the Project:

- Grinding
- Flotation
- Concentrate filtering
- Tailings filter and pastefill plant
- Lead and zinc concentrate storage / load out building

18.1.5 Process plant site development

During the study, various process plant site locations were considered and, as a result, the current process plant site adjacent to tailings impoundment was selected as the preferred location. The advantages of this plant site location are:

- The primary crusher was located adjacent to the mine shaft to minimize haulage.
- The crushed mineralization stockpile was located to reduce the overland conveyor length.
- The primary crusher elevation was set to balance cuts versus fill, and to place the jaw crusher on cut. In addition, the location was selected to minimize conveyor lengths to the coarse mineralization stockpile.
- The process facility was oriented to optimize the natural area sloping direction and cut and fill requirements.
- The fresh and firewater storage tank was located at an elevation to utilize gravity flow to the processing plant areas.

Preliminary geotechnical information was available for the recommended process plant site earthwork and depth to bedrock. Additional geotechnical investigations will need to be carried out for all major equipment locations prior to finalizing the process plant layout and location.

18.1.6 Administration buildings and mine dry

The main buildings and offices are constructed and sited near the processing plant. Offices will be provided for all technical service and management personnel. A double storey structure with an equivalent 2,500 ft^2 (232 m^2) footprint is estimated for the mine buildings, offices, meeting rooms and change rooms. A 260 cap lamp station will be required for mine personnel. A mine dry with 380 lockers will be required for the mine and 117 for the plant; mill employees will have their own change facility.

18.1.7 Surface workshop

The underground mine will be supported by a centrally located maintenance facility near the offices; a workshop fitted with a storage warehouse. The maintenance workshop will consist of a pre-engineered steel structure placed on a slab cast on grade. The building will have three maintenance bays and one wash bay. The shop will be sufficient to handle major maintenance and repairs that will be needed by the underground mining operation. Smaller repairs and routine maintenance will be handled underground.

The maintenance shop will provide administration space and will be attached to the warehouse. Both structures will be fitted with sprinklers and fire alarms. The fire water pumps will be installed in the wash bay mechanical room.

The warehouse will provide enough inventory space for daily operations as well as for critical maintenance spares. Stock levels for routine and minor maintenance will be set at a one-week supply which will provide enough buffer given the direct access to the mine site and the proximity of local suppliers. Other major stock items for planned maintenance will be brought in via the main highway from Tucson, AZ.

18.1.8 Surface magazine

The mine is expected to be relatively dry and therefore the primary explosive being used for development and stoping will be Ammonium Nitrate Fuel Oil (ANFO). The production blasting powder factor, not including slot raising, is 0.88 lb/ton (0.44 kg/tonne). The lateral development has a powder factor of 2.20 lb/ton (1.10 kg/tonne). At 10,000 tons/day, the average explosive consumption was determined to be 1,710 tons (1,550 tonnes) of ANFO in a year for all stoping and lateral development. This would require approximately 143 tons/month (130 tonnes/month) of explosives requiring 7 transport deliveries per month. Peak consumption is in Year 6 and the maximum quantity of explosives required is 2,460 tons (2,232 tonnes) per year, or 205 tons (186 tonnes) per month, requiring 10 transport deliveries per month.

The surface magazines should be placed in a remote location near the access road but away from main buildings and mine infrastructure. A fenced and gated facility will be required with suitable storage to meet requirements and separate storage facilities will be required for high explosives and detonators.

18.1.9 Surface mobile equipment

The mine will designate an emergency vehicle (ambulance) for evacuation to medical care via the access road. A fire truck will also be located near the ambulance. A vacuum truck, flat deck, and mechanics vehicles will also be required.

A telehandler and small forklift for moving equipment and supplies around the processing plant and warehouse will be required.

A small 38.6 ton (35 tonne) crane for maintenance tasks in the processing plant, surface handling, and other tasks can be procured with an operator on rent as needed for shutdown maintenance tasks. The low usage of such equipment would not necessitate having a permanent qualified operator on site.

An 18.7 ton (17 tonne) LHD will be required to move mineralization from the shaft to the feeder apron of the primary crusher.

18.1.10 Accommodation

The work force will be encouraged to live near the mine and a daily bus service will be provided to drive them from Nogales, Sierra Vista, and Tucson. There is no allowance for a mine camp on site. Kitchen facilities will be available for dispensing tea and coffee in the mine offices. AMC has estimated an underground workforce of a maximum of 380 personnel in addition there will be approximately 120 processing employees for a total of 500. The workforce will operate on a three-shift basis: one-week night shift, one week day shift and one week rostered days off. It was assumed that the mine will be owner operated.

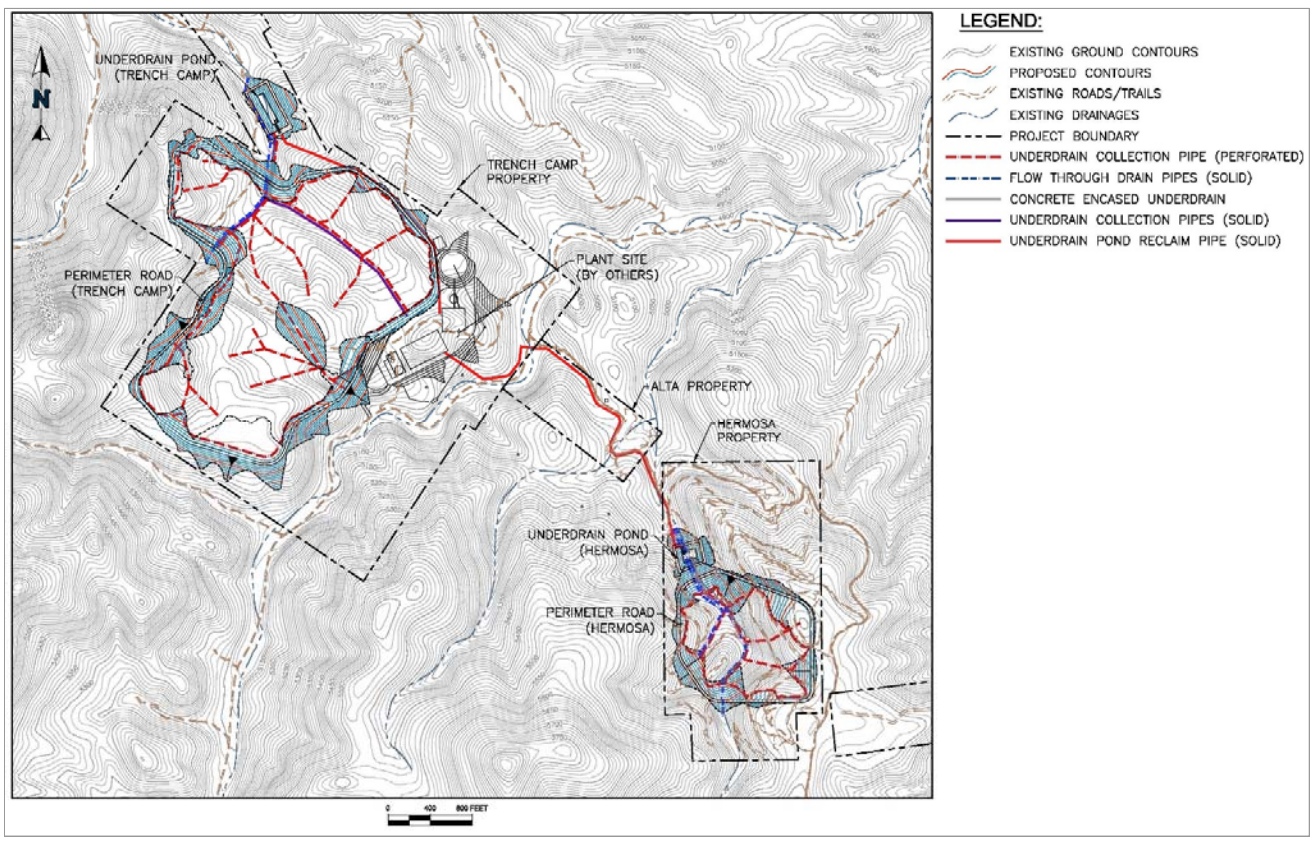
18.2 Fuel storage

Fuel storage will consist of two tanks that will have the capacity to support two weeks' consumption at peak production. The tank system will be enclosed by a lined berm of sufficient capacity to contain 110% of the contents of a full tank in the event of a major leak or spillage. Fuel will be trucked to site on a year-round basis.

18.3 Tailings storage facilities (TSFs)

Two "dry stack" tailings storage facilities (TSF) have been designed and located on private land that is wholly owned by AZ at the sites known as Trench Camp and Hermosa. The Trench Camp TSF was sized to contain historic tailings currently located on the Trench Camp site, tailings produced from the mineral recovery process and potentially acid generating (PAG) development rock. It was designed to be near the processing facilities for easy access and will be the first TSF to be constructed. The Hermosa TSF was developed to contain additional tailings and PAG development rock after the Trench Camp TSF is full. Both TSFs have been designed with the capacity to contain the projected extracted mineral resource. A plan view of the Trench Camp and Hermosa overall site layout can be referenced on Figure 18.3.

Figure 18.3 Tailings storage facilities



18.3.1 Tailings and development rock storage requirements

It is anticipated that the LOM mineralized material of 96.67 million tons (87.70 million tonnes) will produce 83.88 million tons (76.09 million tonnes) of tailings given 13.23% of the total tonnage processed will be removed as part of the mineral recovery process. After mineral recovery, it is estimated that 60% of the tailings will be utilized as paste backfill in the underground mine workings stopes and the remaining 40% will be stored on the surface in the form of dry stack tailings. In addition to tailings, the mining process will create 11.99 million tons (10.88 million tonnes) of development rock. It is anticipated that approximately 34.5% of the development rock with be stored underground and the remainder will be stored in the TSF or utilized as construction material. Additionally, it is assumed that the development rock stored underground will have a porosity of 30% and two-thirds of the voids will be filled with paste backfill. Utilizing the porosity of the development rock backfill increases the tailings percentage reporting underground as paste backfill from 60% to 60.8%.

Based on geologic data, approximately half of the development rock, 6.00 million tons (5.44 million tonnes), contains sulphide mineralization and as a result is classified as PAG rock. The remaining 6.00 million tons (5.44 million tonnes) is considered non-PAG rock and suitable as construction material. Geochemical analysis of the waste rock material will be carried out in more advanced stages of this project to better define waste characterization. The current plan is to co-mingle the PAG development rock reporting to the surface with the dry stack tailings, thereby encapsulating the PAG material within the tailings. By encapsulation of the PAG development rock material in the fine grained dry stack tailings, oxygen and moisture ingress will be effectively cut off, which in turn will minimize the potential for acid rock drainage.Table 18.2 shows the amount of LOM mineralized material removed during mineral recovery, paste tailings used as mine backfill, filtered dry stack tailings placed within the two TSFs, and development rock distribution.

Table 18.2 LOM tailings and development rock distribution

Description	Material quantity (tons / tonnes)	Percentage (%)	Tailings / development rock (tons/tonnes)
Tailings produced during mineral recovery from mineralized material	96,670,545 tons LOM mineralized material (87,698,000 tonnes)	86.77%	83,877,000 tons (76,092,000 tonnes)
Paste tailings used as mine backfill	83,877,000 tons LOM tailings (76,092,000 tonnes)	60.78%	50,977,000 tons (46,245,000 tonnes)
Filtered tailings to be placed in dry stack TSFs	83,877,000 tons LOM tailings (76,092,000 tonnes)	39.22%	32,901,000 tons (29,847,000 tonnes)
Development rock (PAG / non-PAG) used as mine backfill	11,992,000 tons LOM development rock (10,879,000 tonnes)	34.48%	4,134,000 tons (3,751,000 tonnes)
Development rock (PAG/non-PAG) directed to TSF or construction use	**11,992,000 tons LOM development rock (10,879,000 tonnes)**	**65.52%**	**7,858,000 tons (7,128,000 tonnes)**

18.3.2 Trench Camp existing tailings piles

Four historic tailings deposits containing tailings and waste rock (existing tailings piles 1 through 4) exist within the proposed Trench Camp TSF footprint (Figure 18.4). A geotechnical investigation was completed in January 2017 that consisted of borings, test pits and geophysical surveys which focused on the Trench Camp existing tailings piles 1 through 4, to define the tailings and waste rock volumes within each facility as well as determine tailings and waste rock material properties (sections are shown in Figure 18.5). Boreholes were placed along the geophysics lines in order to correlate known depths of the logged materials to seismic velocities. Using the depth of tailings and waste rock identified in the boreholes in combination with the velocities generated during the geophysical survey, a velocity band was identified that correlated with the bottom of the tailings and waste rock material within the existing tailings piles. The tailings depth data was used to estimate the volume of tailings and / or PAG waste rock within each pile. The estimated tailings and PAG waste rock volumes to be relocated onto the lined TSF are based on an in-situ density of 96 pcf and 115 pcf, respectively (1.54 tonnes/m^3 and 1.84 tonnes/m^3) and are presented in the Table 18.3.

Table 18.3 Trench Camp existing tailings piles volume estimates

Existing tailings piles	Estimated volume (tons/tonnes)
1	352,000 tons (319,000 tonnes)
2 / 4	684,000 tons (621,000 tonnes)
3	227,000 tons (206,000 tonnes)
Total	**1,263,000 tons (1,146,000 tonnes)**

Figure 18.4 Plan view of existing tailings piles 1 through 4

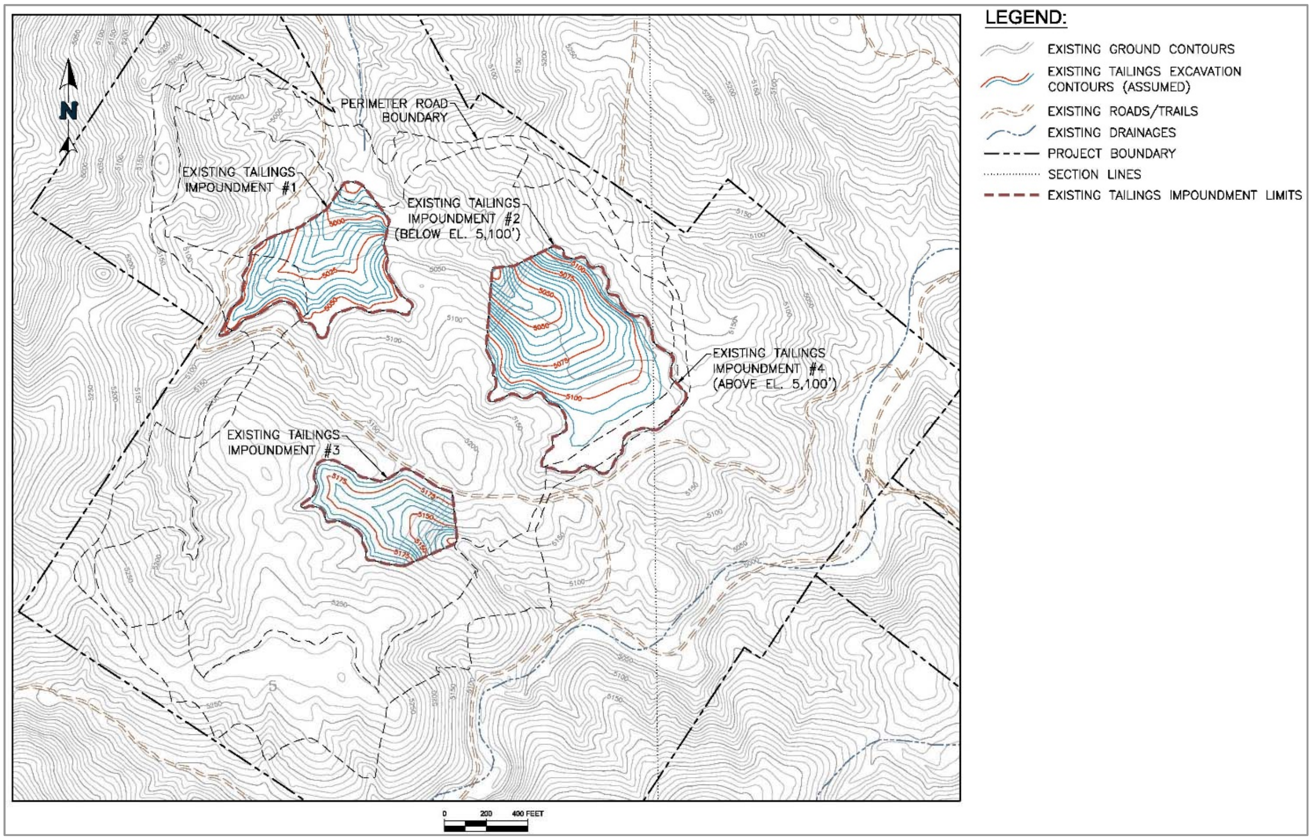
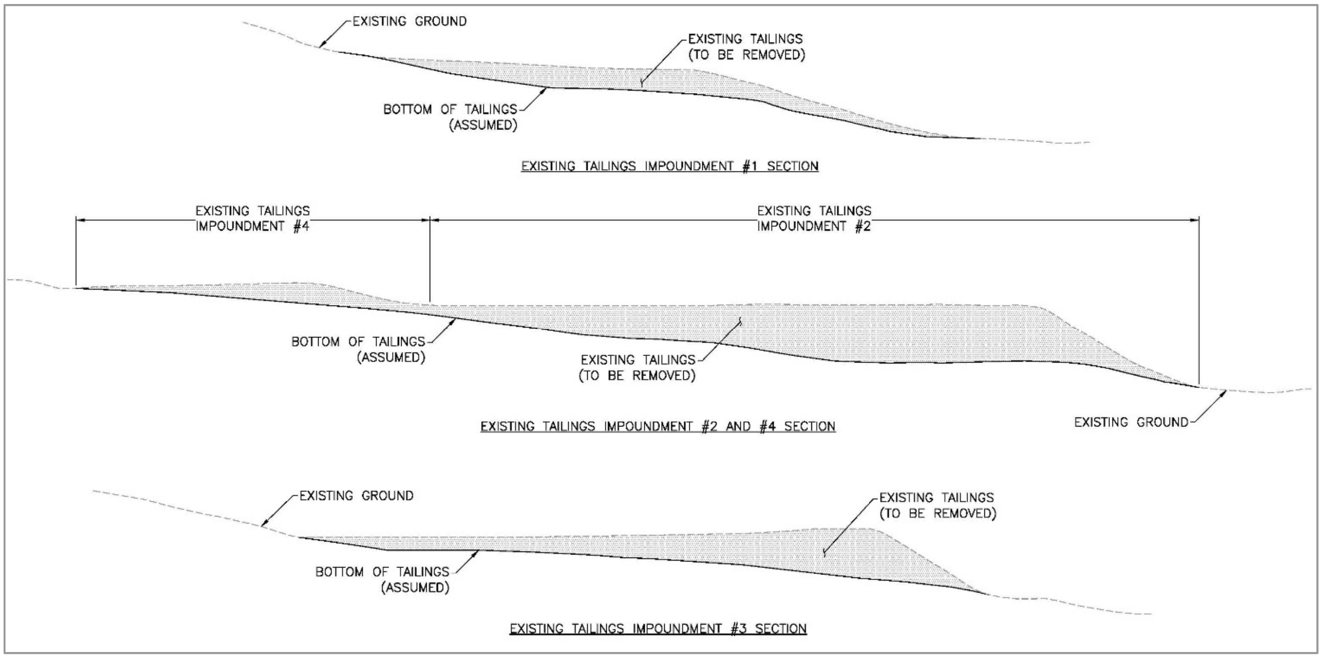


Figure 18.5 Sections of existing tailings piles 1 through 4



Based on data collected during the geotechnical field investigation, standard penetration test blow counts in the existing tailings deposits show the tailings are relatively soft and have generally increasing moisture with depth. If left in place the historic tailings may result in questionable founding conditions for the proposed Trench Camp TSF for the following reasons:

- Low tailings strength do not provide adequate slope stability.
- Variable tailings depths will result in differential settlement which is a concern in geomembrane lined facilities given the geomembrane has finite, albeit robust, allowable deformation properties.
- Wet conditions near the bottom of the existing tailings mass will produce seepage when surcharged with new tailings.

Utilizing the results of the geotechnical investigation, the current design approach to historic tailings is to remove the tailings and relocate them on the proposed geomembrane lined Trench Camp TSF. The existing tailings in pile 1 will be double handled as they will be excavated, hauled and temporarily placed on existing tailings pile 2 and 4 to allow access to the proposed starter TSF footprint for construction of the line facility. After tailings from pile 1 are removed, the northern portion of the Trench Camp TSF will be constructed and upon completion, the tailings from pile 1, 2, and 4 will be relocated to the geomembrane lined Trench Camp TSF. By relocating the existing tailings onto a lined facility, the environmental issues currently associated with the existing tailings that are located in unlined facilities will be effectively mitigated.

18.3.3 TSF storage capacity

The TSF development plan is to stage construction of the facility to spread capital costs over the life of the facility. The TSFs were developed with a 6-year starter (Phase 1) containing approximately 7.5 M tons (6.8 M tonnes) of tailings. The starter TSF has the capability to hold existing tailings from Tailings Piles 1, 2, and 4, dry stack tailings and development rock (PAG and non-PAG) from 6 years of mining production.

The TSFs were designed with a combined storage capacity of approximately 41 million tons (37.19 million tonnes). The capacities were determined based on an expected in-place filtered tailings density of 106.3 pcf (1.70 tonnes per cubic metre) for new tailings, 118 pcf (1.84 tonnes per cubic metre) for development rock, 104 pcf (1.67 tonnes per cubic metre) for historic tailings and 115 pcf (1.84 tonnes per cubic metre) for waste rock.

The in-place filtered tailings, development rock, historic tailings and waste rock densities within the TSF were assumed to be 90 percent of the maximum dry density as determined by a standard proctor compaction test (ASTM D698). The moisture density characteristics of the tailings (ASTM D-698) used in the design of the Dry Stack TSF were derived from laboratory testing while the development rock and waste rock densities were assumed. All capacity calculations are based on the densities listed above with dry stack tailings making up the majority of the stored material. The storage capacities of Trench Camp and Hermosa TSFs are shown in Table 18.4.

Table 18.4 TSF storage capacities

Material	Trench Camp starter TSF (tons/tonnes)	Trench Camp ultimate TSF (tons/tonnes)	Hermosa ultimate TSF (tons/tonnes)
Existing tailings	1,036,000 tons (940,000 tonnes)	1,263,000 tons (1,146,000 tonnes)	0 tons (0 tonnes)
Dry stack tailings / development rock	6,464,000 tons (5,860,000 tonnes)	33,179,000 tons (30,099,000 tonnes)	6,962,000 tons (6,316,000 tonnes)
TSF design storage capacity	**7,500,000 tons (6,800,000 tonnes)**	**34,442,000 tons (31,245,000 tonnes)**	**7,000,000 tons (6,350,000 tonnes)**

Note: Total TSF storage capacity includes relocation of the existing tailings from the Trench Camp property to the dry stack.

The ultimate Trench Camp and Hermosa TSF will provide the capacity to contain approximately 41,404,000 tons (37,561,000 tonnes) of material. The combination of the two TSFs will store approximately 1,263,000 tons (1,146,000 tonnes) of existing tailings, 32,901,000 tons (29,847,000 tonnes) of dry stack tailings and 7,858,000 tons (7,128,000 tonnes) of PAG/non-PAG mine development rock. This accounts for all the existing tailings, dry stack tailings, PAG mine development rock and the majority of non-PAG development rock. The remaining non-PAG development rock will be used as construction material for the TSF embankments or surrounding infrastructure.

Plan views of the Trench Camp starter, Trench Camp ultimate and Hermosa ultimate TSF locations can be referenced on Figure 18.6, Figure 18.7, and Figure 18.8, respectively. Each figure shows the property boundary, perimeter road, underdrain pond, and expanded dry stack TSF.

Figure 18.6 Trench Camp starter TSF plan view

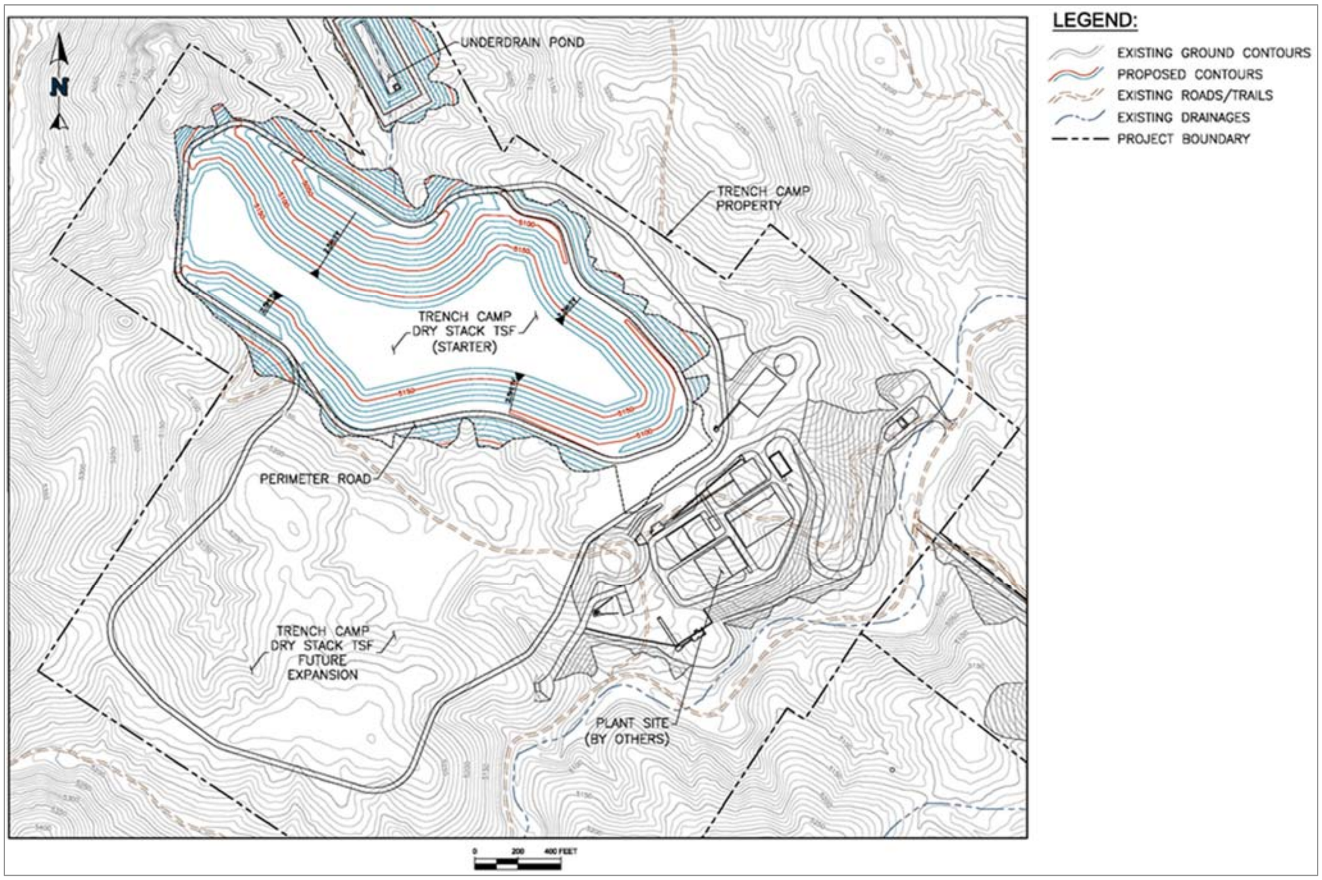


Figure 18.7 Trench Camp ultimate TSF plan view

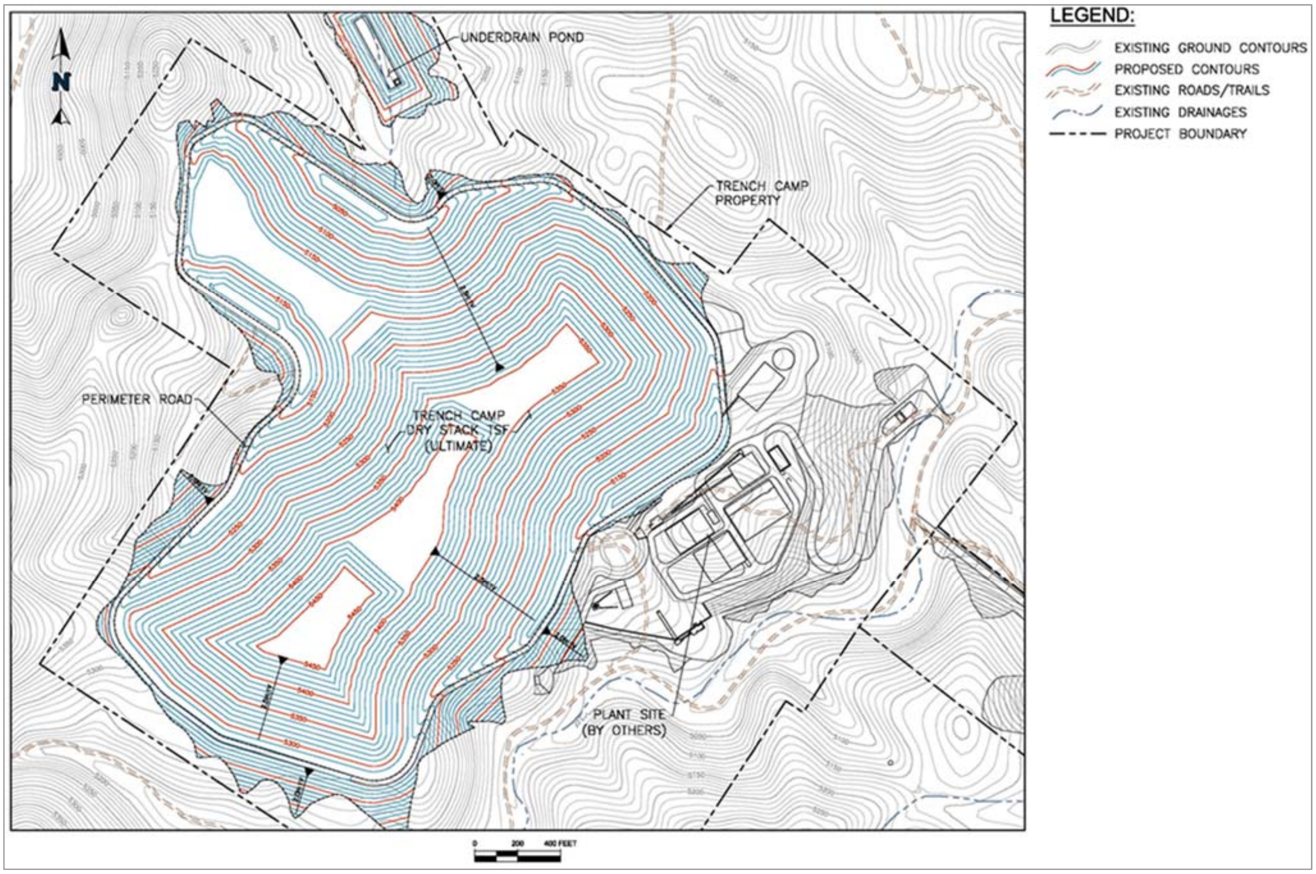
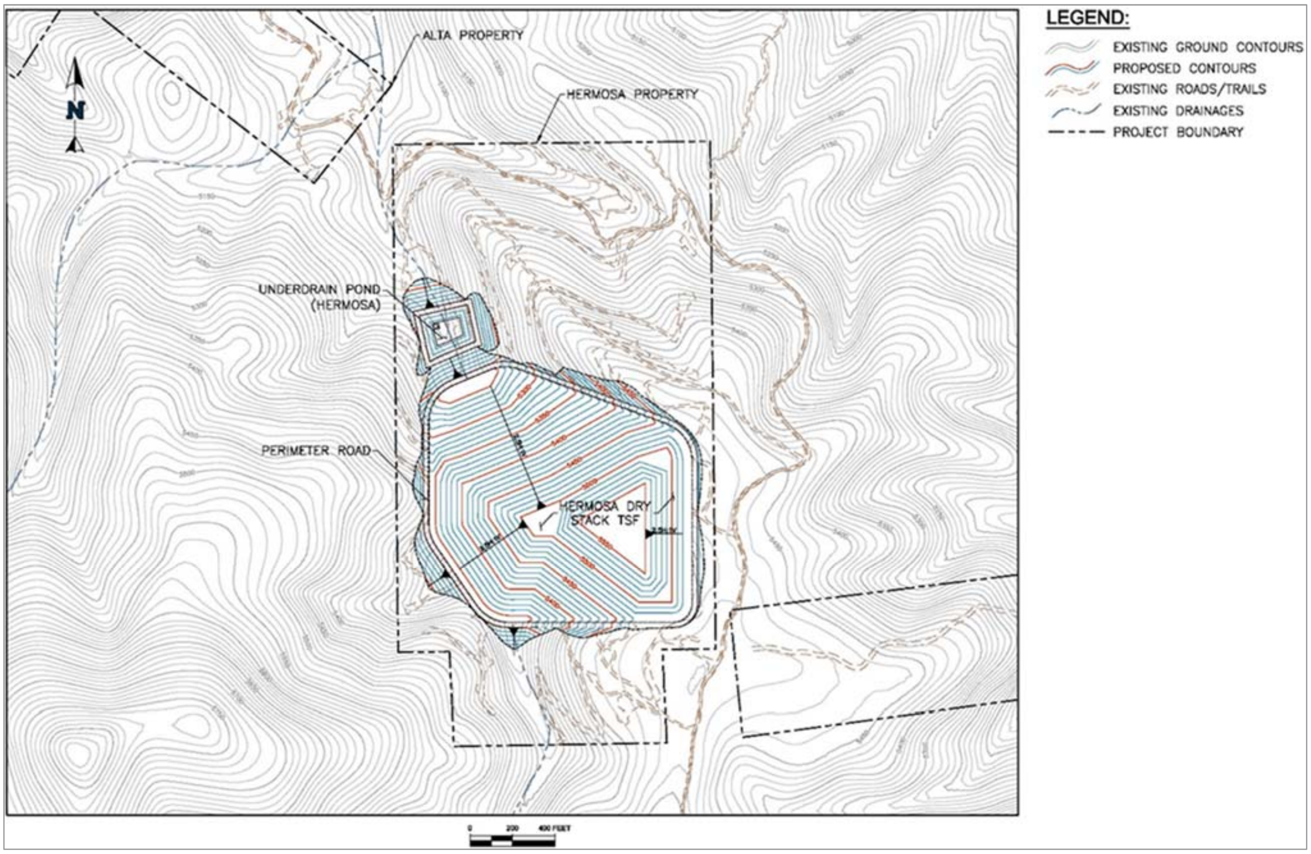


Figure 18.8 Hermosa ultimate TSF plan view

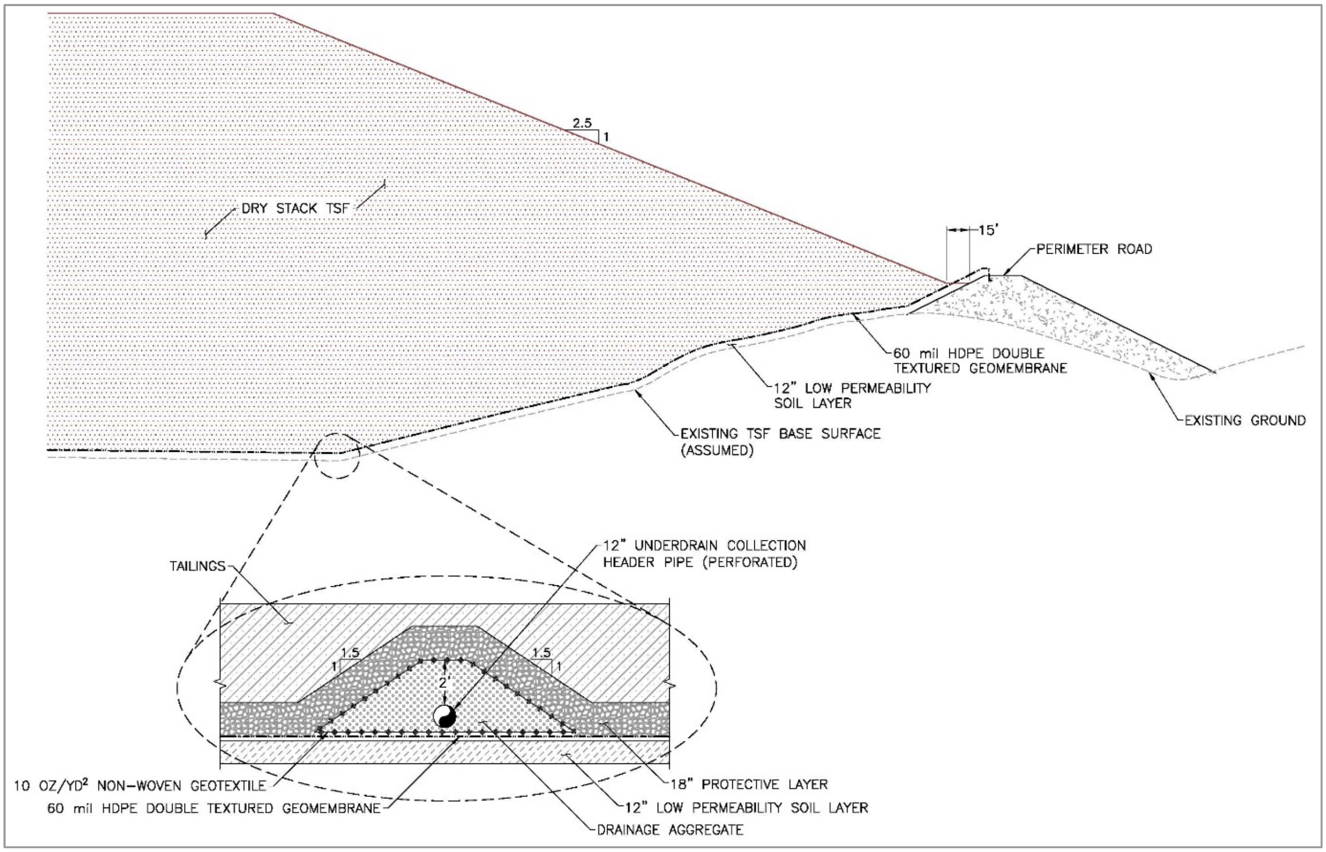


18.3.4 TSF design

In general, the TSF design concepts for Trench Camp and Hermosa are similar. Each TSF consists of a perimeter road which fully encompasses a basin area. The perimeter road is designed with upstream slopes of 2.5 H (horizontal):1 V (vertical), downstream side slopes of 2.0 H:1 V and a crest width of 25 ft (7.6 m). In areas where the downstream slope is in cut, the daylight slope is increased to 1.5 H:1 V since the cut is assumed to be in rock. The perimeter road will provide light vehicle access, containment of surface water runoff from the dry stack surface and passive resistance at the toe of the dry stack TSF slope. The passive resistance component generally requires that the perimeter road be elevated above existing ground or constructed in conjunction with existing ground in a manner which provides an internal slope toward the facility. The perimeter roads will be constructed using standard cut / fill operations within the TSF basin and plant site area as well as non-PAG material produced from mining operations. A typical perimeter road section and detail can be referenced on Figure 18.9.

The dry stack TSF basins and upstream slopes of the perimeter roads are designed with a composite liner system consisting of 12 in (305 mm) of low permeability soil with a coefficient of permeability (k) ≤ 1×10^{-6} cm / sec overlain by a 60 mil (1.5 mm) high-density polyethylene (HDPE) double sided textured geomembrane. To protect the geomembrane, reduce head and facilitate long-term drainage of the tailings, an 18 in (457 mm) protective layer consisting of crushed gravel will be placed over the geomembrane liner. In addition, perforated corrugated polyethylene piping will be placed in the topographic lows within the TSF basin to augment collection and conveyance of underdrainage flow from the tailings. Underdrain flows will be directed via gravity to underdrain collection ponds which are located downstream of the TSFs. The underdrain collection outlets are routed through the perimeter roads via a reinforced concrete encased outlet pipe. The underdrain collection ponds have been sited near the north side end of the Trench Camp and Hermosa Dry Stack tailings storage facilities. The TSFs are considered zero discharge facilities, given the underdrain flow collected in the pond will be pumped back to the plant site and ultimately reused in the processing circuit. Typical TSF basin and underdrain system details can be referenced on Figure 18.9.

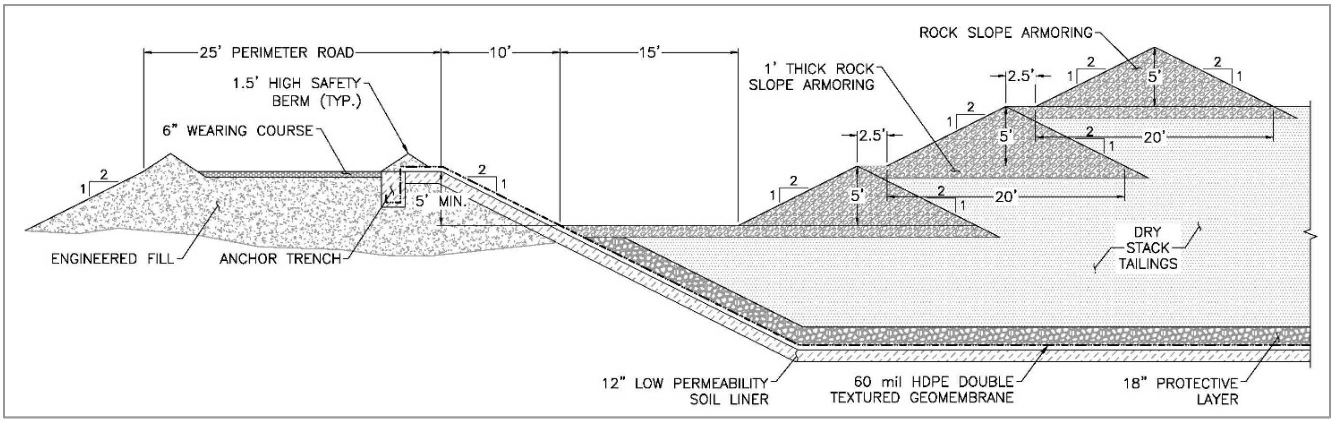
Figure 18.9 TSF typical section



18.3.5 Rock armouring and tailings placement

Prior to filtered tailings placement in the Dry Stack TSF a rock armouring berm will be constructed from non-PAG development rock to protect the external face of the filtered tailings from stormwater and wind erosion. The rock armoring will be placed in a manner to maintain an overall 2.5 H:1 V composite slope with 2.0 H:1 V berm side slopes and benches. The benches will serve as energy dissipaters to slow runoff water velocities down the slopes of the TSF meteoric storm events. After the rock armouring berm is in place, tailings will be placed against the berm and compacted. Near the external areas of the TSF, sited between the initial rock armouring berm and the perimeter road internal slope, is an internal diversion channel directing flow to the underdrain collection system. Generally, the internal diversion channel has a bottom width of approximately 15 ft (4.57 m), a depth of 5 ft (1.52 m) and a minimum slope of 1%. Details of the rock armouring and internal diversion channel can be referenced on Figure 18.10.

Figure 18.10 Typical rock armouring and dry stack tailings section 3



18.3.6 Stormwater control

The TSFs are sized to safely convey and / or contain direct precipitation from the 100 year / 24-hour storm event while maintaining 2 ft (0.6 m) of freeboard. Direct precipitation on the TSF footprints will be collected by the internal diversion channels, directed to the underdrain collection headers and ultimately the underdrain collection ponds. The underdrain collection ponds are sized to contain underdrainage flow, direct precipitation runoff from the filtered dry stack tailings and direct precipitation on the pond footprint from the 100 year / 24-hour storm event while maintaining 2 ft (0.6 m) of freeboard. If empty at the time of the storm the underdrain collection pond has the capability to hold the 500 year / 24-hour storm event. The total pond capacity will allow the operator time to react to issues such as a power outage with redundant generator sets to restore pumping capacities at the underdrain collection pond. External stormwater reporting to the TSFs will be routed around the facilities through engineered diversion channels. Where external diversion channels are not practical, a stormwater flow through drain will be located under the TSFs to transmit and release stormwater from the upstream to downstream side of the TSFs.

18.4 Underground infrastructure

18.4.1 Power demand and distribution

Twin 13.8 kV 500MCM electrical distribution cables will be installed in the decline during development. One line will run from surface down the main zone decline and the second line will run down the Taylor Deeps decline which joins the main zone decline at approximately the 3720 L. Portable 600 kVA 4160 / 480V / 120V sub-stations will be established on each active level to power auxiliary fans and pumps, mining equipment and lighting panels respectively. The sub-stations will be connected via a disconnect such that they can be removed from service without interrupting power in the remainder of the mine. Once a level is complete, the sub-station can be moved to the next active heading. A total of 24 portable sub-stations will be required during peak mining activities.

When the main dewatering sump and loading pocket are established a permanent 900 kVA sub-station will be located near shaft bottom to drive the peak power of the pump station as well as the loading pocket. Once the operating shaft and ventilation raises are established the 13.8 kV feeders will be redistributed to vertical routes to mitigate voltages drops and provide a loop distribution system for redundancy.

The primary power demand for the underground mine is associated with the main fans located on surface at the top of the exhaust raises, the main shaft and the secondary fans and mining equipment. A maximum demand of 11.2 MW will be required for the underground mine (including the hoist which requires 3.5 MW supplied power directly on surface).

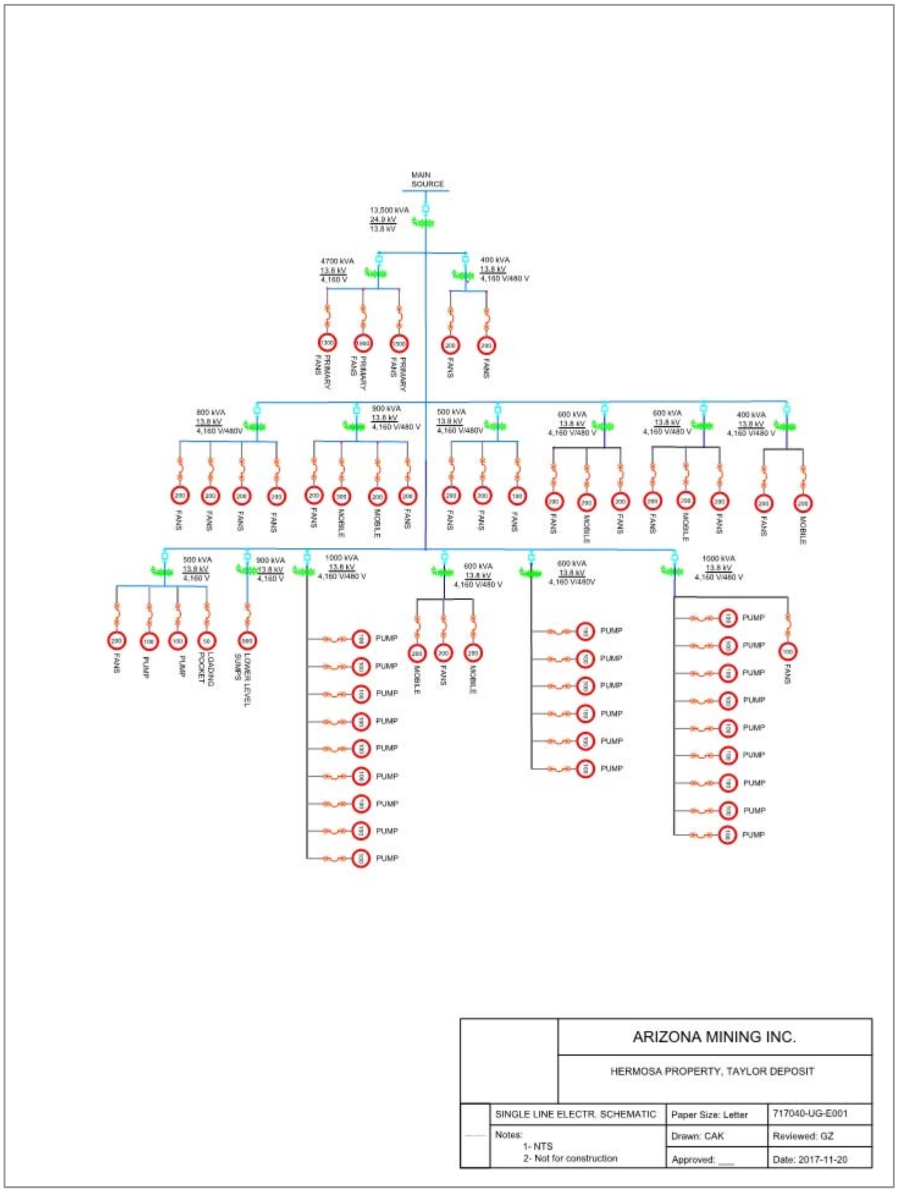
A summary of the peak and average power demand by activity is provided in Table 18.5.

Table 18.5 Summary of underground power demand

Description	Peak power (kW)	Average power draw (kW)
Hoist	3,470	2,144
Main dewatering pumps	1,893	826
Main fans	5,222	4,776
Level distribution fans	598	280
Other (compressors, lights, etc.)	3,470	2,144
Totals	**11,183**	**8,026**

Figure 18.11 shows the conceptual plan for UG power distribution on site via the 24.9 kV, 3 phases, 60 Hz feeders and transformers stepping down 13.8 kV to 4,160 V or 480 V as needed for mining equipment, underground fans, pumps, etc. Lighting and control voltage will be rated at 120 V. Emergency diesel generators will be installed at the process plant and underground mine for backup power in case the utility power fails.

Figure 18.11 Single line diagram showing underground power distribution



18.5 Dewatering

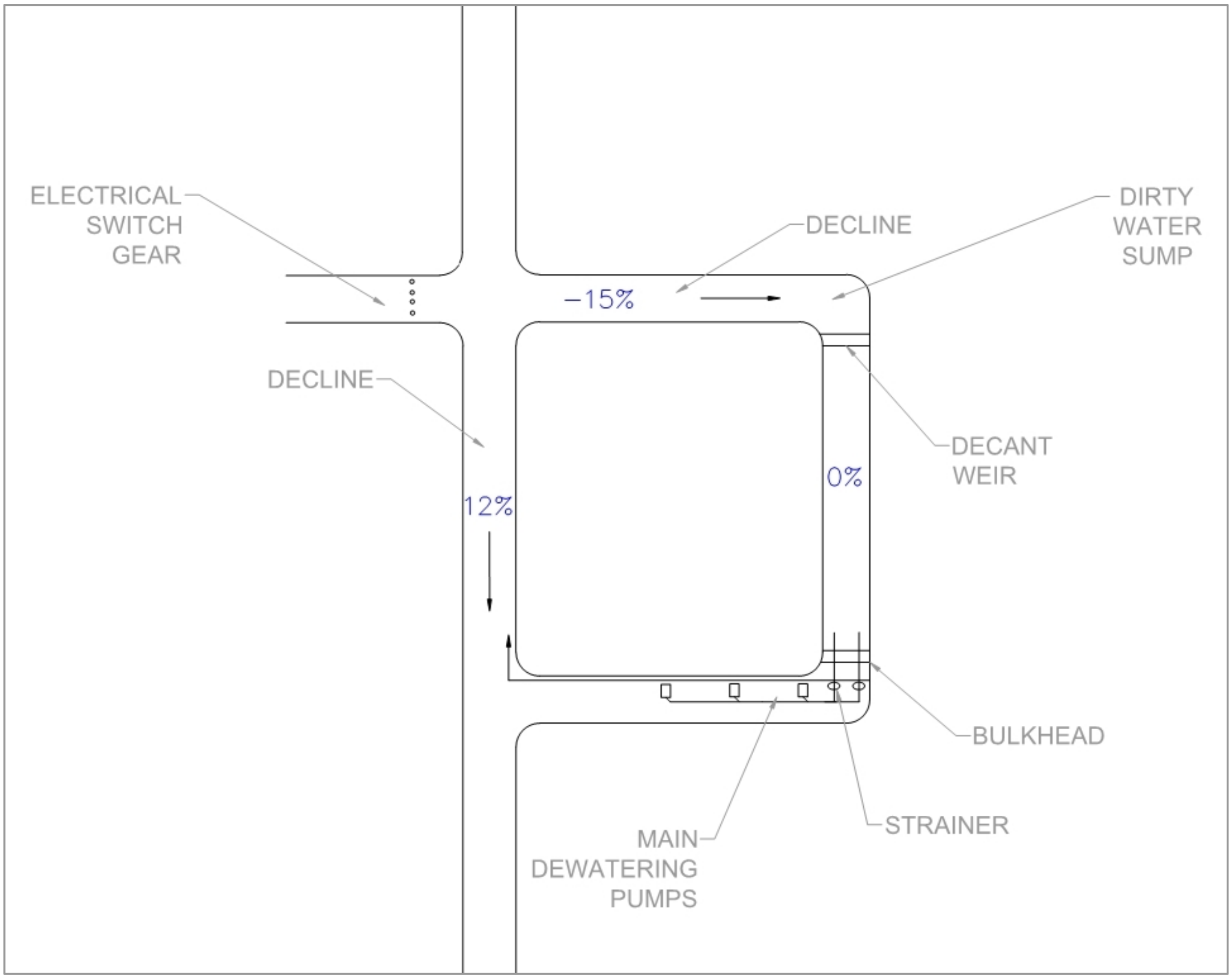
During development of the main zone decline, seventeen staged submersible high-head low-flow pump stations will be established. Each pump will transfer up the decline through a 4” steel grooved pipe line to the next sump. The pumps are sized so that the nominal 80 gpm (5 l/s) ground water and the 80 gpm (5 l/s) drilling and utility water can be handled by one pump; two pumps are installed in each sump in case of failure. Smaller pumps on the level can be used to transfer water from the face to the decline sumps.

During development of the Taylor Deeps decline, nine staged submersible high-head low-flow pump stations will be established. Each pump will transfer water up the decline through a 4” steel grooved pipe line to the next sump. At the top of the Taylor Deeps decline, the dewatering line will tie-in to the main zone decline. The pumps are sized so that the nominal 80 gpm (5 l/s) ground water and the 80 gpm (5 l/s) drilling and utility water can be handled by one pump, two pumps are installed in each sump in case of failure. Smaller pumps on the level can be used to transfer water from the face to the decline sumps.

When the main zone decline is established near the shaft bottom, the main dewatering sump station will be developed. The main sump will consist of three horizontal multi-stage 400 hp pumps each capable of 160 gpm (10 l/s). Once the main dewatering sump is ready, the 4” steel dewatering line in the main zone decline will be connected together bypassing the staged pumps altogether. The decline sumps in the main zone will then be connected together, via drain holes and water will flow via gravity to the dirty water side of the main sump station. In a similar manner, the decline sumps in the Taylor Deeps zone will be connected together via drain holes, and water will flow via gravity to the lowest decline sump in the Taylor Deeps zone. On the 2000 L, the Taylor Deeps dewatering line will tie-in to the main zone’s dewatering system.

In the main dewatering sump, water overflowing the intermediate weir will enter a holding sump that feeds the main dewatering pumps. Two dewatering pumps will normally operate to provide enough capacity to drain the clean water sump, which will be sized to provide a duty cycle of no more than 25% to prevent frequent starting of the pumps. The third pump provides an online spare, and in an emergency upset condition can provide additional capacity to the system. Additional pumps can be added to the system should water in-flows exceed initial estimates. Figure 18.12 shows a plan view of the main dewatering sump.

Figure 18.12 Plan view of the main dewatering sump



As the decline progresses below the shaft bottom, and after the main sump is in service, existing pumps can be relocated to extend the staged dewatering system into deeper levels in the mine.

18.5.1 Service water

A three-inch HDPE line will be installed in stages down the decline to provide fresh water for use in the mine. Every 100 vertical feet, a head tank and/or a pressure reducing valve will be installed to control the pressure in the line. A combination of hoses and HDPE piping will extend out onto the levels to provide utility water. Service water will be required for drilling and watering down access routes. A minimum requirement of 70 gpm of service water is required to operate the underground equipment.

18.5.2 Waste water

Underground mine water from operations and grey water from the office and mine dry will be routed via HDPE piping systems, partially or completely buried, to the plant for processing as part of the tailings system.

18.6 Compressed air

Four portable air compressors (one for each level in operation) will be moved together with the primary mining equipment. The compressors will be sized so that they will be able to supply four operating drills.

18.7 Communications

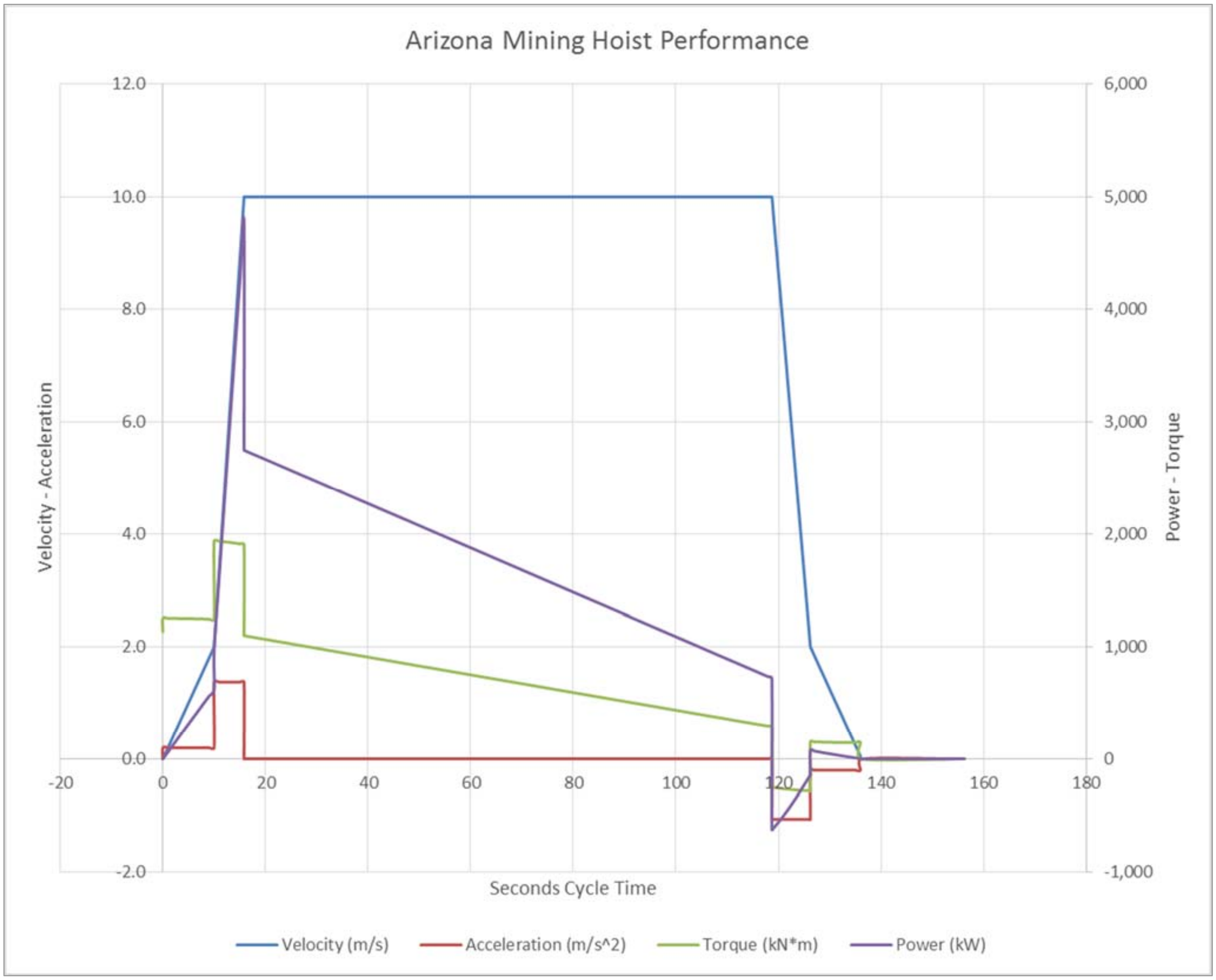
A leaky feeder system will provide means for communication underground. All vehicles will be fitted with radios. A call bell and emergency system will be used when signalling the main production shaft.

18.8 Main production hoist

The shaft will have a 21-foot (6.5 m) finished diameter and the production hoist will be a conventional double drum hoist with two skips discharging into the bins on surface in the headframe. Loading pockets will be on the 2600 L and 1600 L. The cycle times were estimated using 10 m/s velocity for the conveyance and allowing for creep in / creep out and decking time. The hoist is designed to accommodate the mine's full production target of 10,000 tons per day (achieved in Year 6), and the capacity of each skip is 27.6 tons (25 tonnes) and the total weight of each skip is 47.4 tons (43 tonnes) when fully loaded.

The cycle time was used to estimate the peak and average power requirements (including acceleration loads for the sheaves and drums), this is illustrated in Figure 18.13.

Figure 18.13 Estimated hoist performance curve



The peak power demand is during the acceleration phase just as the conveyance reaches peak velocity. Average annual power draw includes time allowances for regular and unplanned maintenance downtime.

Skips will be provided with decks for inspection purposes and the conveyance will have bails and cage-heads for lowering heavy slung loads if required. The skips themselves will be bottom dump and activated by scrolls in the headframe.

The loading pocket will consist of a conveyor feeding a diversion chute that alternately charges two weigh flasks. Each flask is loaded during a skipping cycle so that it is ready when a skip returns to the loading pocket. Allowance has been made in the skip production schedule for 10% additional capacity for waste to be hoisted to surface if required.

18.8.1 Fire detection and suppression systems

The mine ventilation systems will be provided with an ethyl mercaptan (stench gas) system (activated manually or remotely) to warn underground personnel in the event of an emergency. Radio contact via the leaky feeder system provides an alternative method of communication. The main ventilation fans can be shut down or adjusted to assist with fire control systems in the mine.

If the automatic stench system fails to release, two back-up measures will be in place. Back-up measures include manual firing of the system at the unit, allowing the stench gas to be distributed as above and release of a gas cylinder by hand into the fresh air intake.

Once stench is released, underground mine personnel would report immediately to the nearest mine refuge station or surface, whichever is closer.

18.8.2 Underground facilities

Underground mine services will include lunchrooms, a small maintenance shop for minor and urgent repairs, fuel and lubricant storage, and small magazines for high explosives and detonators.

The lunchrooms will provide a clean space with potable water, tables, and chairs. They will also be used as mine refuge areas. The mine rescue team will be able to use the space for training and to store equipment and supplies. A lunchroom will be provided on each of the main operating areas.

A single bay maintenance shop with a jib crane will be provided. The intent of this bay is to enable routine tasks such as lubrication and changing of filters, and minor repairs to keep the equipment in a serviceable condition and return it quickly into service. Any significant maintenance will be conducted on surface in the main workshop.

A fuel and lubrication area will be provided underground. Fuelling will be conducted via tankers from surface. Storage will also be provided for lubricants and waste oil. A small location equipped with fire doors, fire detection, and air operated pumps will dispense the products near the maintenance bay.

The explosives magazines will be a few rounds deep and equipped with lockable doors and wooden benches. The magazines will be ventilated and kept cool. The intent is to provide a small stockpile of detonators, cord, and high velocity explosive for daily blasting activities. Explosives handling and delivery from surface will be accomplished using mobile loading equipment drawing from the surface magazines.

18.8.3 Mine escape and rescue

Portable refuge stations will be located appropriately relative to operating levels. Lunchrooms near the maintenance area will also serve as refuge stations. Self-rescue storage will be provided in the lunchrooms as well as first aid kits at the refuge stations.

Four portable refuge stations will provide refuge for up to 40 persons each during an emergency. MineARC was approached to provide a refuge station design and specifications. MineARC has estimated the finished refuge area to be 25 ft x 60 ft x 10 ft high (7.6 m x 18.3 m x 3 m). This size chamber will provide:

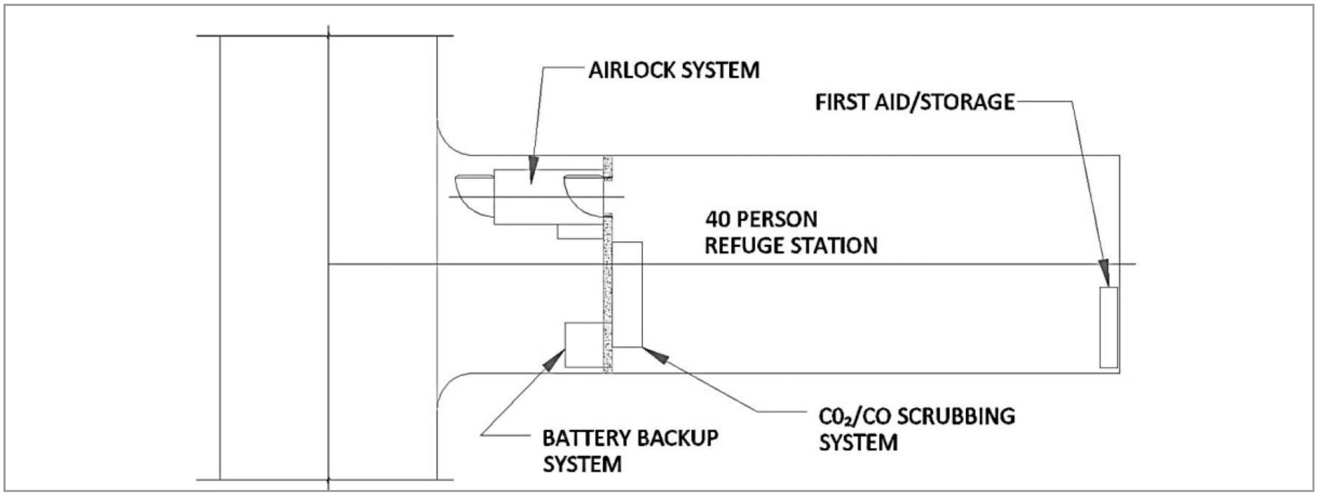
- Approximately 38 ft^2 (3.5 m^2) unobstructed floor space per occupant (less when figuring in tables, furnishings, etc.).

- Useable floor space area per occupant with nominal furnishings and equipment installed, should remain above the minimum recommended of 15 ft^2 (1.4 m^2) per occupant.
- Refuge chamber volume of approximately 15,000 ft^3 (423 m^3).
- Refuge volume of 375 ft^3 per occupant (10.6 m^3 per occupant) – well above the minimum recommendation of 60 ft^3 (1.7 m^3) per occupant.

For a 15,000 ft^3 refuge chamber, a 495 cfm scrubbing unit will provide two complete air exchanges per hour (above the minimum 1.5 x system recommendation), ensuring efficient air flow across both the CO_2 and CO scrubber chemicals and effective removal of CO_2 and CO within the chamber, under full occupancy (40 persons).

It is recommended to install a battery backup / air conditioning system sized correctly for 40 persons that will provide backup electrical supply for all emergency equipment for a minimum 36 hours of duration, under full occupancy. A 22,000 British Thermal Unit (BTU) air conditioning system will manage heat and humidity build-up from 40 occupants and additional heat sources within the chamber (i.e. lighting, electrical equipment, host rock temperature) and will operate off the main power supply or via the battery backup system for a minimum of 36 hours during emergencies. The general layout of a 40-person refuge station is shown in Figure 18.14.

Figure 18.14 40-person refuge station



Main egress is provided by the decline and a second means of egress via the main production shaft which will be equipped with an emergency hoisting cage.

19 Market studies and contracts

An independent marketing and logistics study for the concentrate products to be produced from the Hermosa deposit was undertaken by Exen Consulting Services (Exen).

19.1 Overview

The Hermosa Taylor project will produce relatively large quantities of both zinc and lead concentrates. The longer term outlook for demand for each of the concentrate products is favourable, with growing demand for the payable metals matched by only limited mine supply growth. Based on transportation logistics, the concentrates will likely be loaded in bulk into ocean-going vessels at the port of Guaymas, Mexico for shipment to buyers in Asia, Europe, and elsewhere. There are currently no sales contracts for this project.

Long term forecast metal prices used in cash flow model were as follows (all in US$):

- Zn - $1.10/lb
- Pb - $1.00/lb
- Ag - $20.00/oz

Copper mineralization is present in low concentrations. It was assumed that there would be no value for the copper and it is not included in the project economics. These penalties have been estimated and accounted for in the financial model.

19.2 Concentrate terms

19.2.1 Zinc concentrates

The project is expected to produce an average of approximately 221,000 dmt of zinc concentrate per year over the 29 year mine life. During the first five and 10 years of production, the project is expected to produce approximately 384,000 and 315,000 dmt of zinc concentrate per year, respectively. Based on indicated grades, the zinc concentrate should be suitable for most zinc smelters; however, elevated levels of manganese may result in the imposition of minor penalties for AMI.

19.2.2 Commercial terms

For the purposes of project evaluation, the following terms were used in derivation of the zinc concentrate NSRs (all figures in US dollars).

Payable metals:

- Zinc - 85% of the Zn content, subject to a minimum deduction of 8 units.
- Silver - deduct 3.0 ozs/dmt and pay for 70% of the balance of Ag content.
- Treatment charge - US$210.00/dmt ($190.51/dst).
- Penalties - All inclusive, US$12.60/dmt (Mn - 0.50% free; US$1.50 per dmt for every 0.10% above 0.50%) – this has been included in the Zinc TC's in the Financial Model.

19.2.3 Lead concentrates

The project is expected to produce an average of approximately 198,000 dmt of lead concentrate per year over the 29 year mine life. During the first five and 10 years of production, the project is expected to produce approximately 270,000 and 250,000 dmt of lead concentrate per year, respectively. Based on the expected analysis, the concentrates can be considered 'clean', high grade with valuable levels of payable silver and no deleterious elements which might affect their marketability.

19.2.4 Silver

The project is expected to produce approximately 214 million ounces of silver (or 163 million ounces of payable silver) over the 29 year mine life. Approximately 75% of the recovered silver is expected to report to the lead concentrate with the remaining 25% of the recovered silver to the zinc concentrate. The payable factor for silver in lead concentrate is much higher than the payable factor for silver in zinc concentrate. The project is expected to annually average 5.6 million ounces of payable silver over the 29 year mine life. During the first five and 10 years of production, the average annual production is expected to be approximately 7.6 million and 6.1 million ounces of payable silver respectively.

19.2.5 Commercial terms

For the purposes of project evaluation, the following terms were used in derivation of the lead concentrate NSRs (all figures in US dollars):

Payable metals:

- Lead - 95% of the Pb content, subject to a minimum deduction of 3 units.
- Silver - 95% of the Ag content, subject to a minimum deduction of 50 grams/dmt (1.46 ozs/dst).
- Treatment charge - US$190.00/dmt (US$172.37/dst).
- Silver refining charge - US$1.25/payable oz.
- Penalties - Assumed Mn penalty of $12.00/dmt on Zinc Concentrate.

19.3 Concentrate transportation logistics

The project is well located with nearby infrastructure available for both bulk rail and truck shipments to the load port alternatives evaluated by Exen. Although other port options may be considered, Guaymas, Mexico, located approximately 440 kms from the mine, is in regular use by other concentrate producers in the US and Mexico and likely offers the best load port alternative to AMI. Although railing concentrates to the port appears competitive, trucking will likely prove to be the most flexible and cost competitive option available to AMI.

Exen recommends that an all-inclusive transportation cost average for the two products of US$97.20/dmt (US$88.18/dst) be used for evaluation purposes.

20 Environmental studies, permitting, and social or community impact

The purpose of this section is to identify and discuss those environmental permits and approvals that are most likely to drive the permitting schedule for the Project. The following sections explain the various permitting programs and the estimated time required to secure permits and approvals. Erik Christenson of WestLand is the qualified person for Section 20 with the exception of Section 20.3.2. Doug Bartlett of CCA is the qualified person for Section 20.3.2. Details on tailings and waste rock disposal, site monitoring, and water management are discussed in Section 18.

The format of this section is as follows:

- A brief overview of the social and community setting within which the Project will be developed.
- The USA federal permitting processes that may drive the permitting schedule for Project development.
- Overview of the key permits administered by the State of Arizona that are likely to be required to develop the Project. These state permits are separate from the federal permitting processes, but analyses, modelling, and baseline data collected for state permits can be used to provide baseline information for federal evaluation under the National Environmental Policy Act (NEPA) and other federal permitting processes.

Several federal agencies may have a role in the review and approval of the Project. If required, the U.S. Forest Service (USFS) must approve a mine plan of operations (POO) that will be prepared and submitted to the Coronado National Forest (CNF) by AMI to develop the Project. If the U.S. Army Corps of Engineers (Corps) determines that the Project will impact surface water features that are considered waters of the US, a permit issued by the Corps in accordance with the requirements of Section 404 of the Clean Water Act (CWA) and its implementing regulations will be required. Off-site utility infrastructure improvements needed to develop the Project (power and possibly water supply, as well as access to the Project Area) may, upon final design, cross public lands administered by the CNF or impact water features defined by the Corps to be waters of the US. Approval of the Project by these agencies will require compliance with NEPA.

NEPA is the centrepiece of USA federal environmental policy. NEPA provides a process that federal agencies must follow to ensure that environmental effects of federal actions (e.g., the approval of a POO or CWA Section 404 Permit) are disclosed to the public, offer the public opportunity to provide input during the review process, and ensure that environmental resources are considered in the decision-making process. Considering the federal agencies likely to be involved in the review and approval of the Project, it is anticipated that the CNF will take the lead for federal agencies for implementation of the NEPA review process, and that the other federal agencies (e.g., the Corps) will act as cooperating agencies for the purpose of NEPA compliance. Even if the CNF is not involved in the permitting process, the Corps may require a NEPA review for any impacts to waters of the US on private land. It is anticipated that the development of the Project may require, at a minimum, an Environmental Assessment (EA) and possibly an Environmental Impact Statement (EIS).

Other key federal permits required to develop the Project may include the Endangered Species Act (ESA) and the National Historic Preservation Act (NHPA). Elements of the ESA are applicable even on private lands absent of any other federal nexus. NHPA would only apply if there is a discretionary federal nexus. As with the NEPA process, if USFS land and authorization of the action is required, it is anticipated that the CNF will be the lead agency for ESA and NHPA compliance for the Project.

Primary state environmental permits that are likely to be required to develop the Project are an Air Quality Permit pursuant to the Clean Air Act (CAA), an Aquifer Protection Permit (APP), 401 Water Quality Certification, a permit to discharge treated wastewater under the Arizona Pollutant Discharge Elimination System (AZPDES), a permit to discharge storm water under AZPDES, and a Mined Land Reclamation Plan, which includes financial assurance.

The Environmental Protection Agency (EPA) has granted Arizona Department of Environmental Quality (ADEQ) authority over the CAA and Section 401 and 402 of the CWA in relation to water quality standards and treated wastewater and storm water discharge permits, respectively. These permitting processes are expected to proceed concomitantly with any NEPA process, and any data analysis, collection, and modelling performed to support these permits will be used to disclose and analyze effects during the NEPA process, if required. In the balance of this section, a more detailed description of Social and Community functions that have the potential to affect permitting process (Section 20.1), key USA federal permits and approvals (Section 20.2), and key Arizona state permits and approvals (Section 20.3) are provided.

20.1 Social and community

The Project Area is located in a relatively remote area, approximately eight miles north of the international border with Mexico in Santa Cruz County, Arizona. Nogales, the Santa Cruz county seat, is located approximately 20 miles by road to the southwest, with a 2015 estimated population of approximately 20,250.[2] The second largest community in the county is Rio Rico, also approximately 20 miles away from the Project Area, with a 2010 population of approximately 19,000.[3] Both of these communities are located along Interstate 19, the principal interstate highway connecting Nogales to Interstate 10 in Tucson, Arizona. Santa Cruz County also includes several small towns and communities, of which Patagonia, with approximately 900 residents, is the closest to the Project Area.[4] Patagonia straddles State Route (SR) 82 and is located about 6 miles (10 km) northwest of the Project Area. In addition to Nogales, other major population and economic centres in the region include Sierra Vista, with a 2015 estimated population of approximately 43,350, located approximately 45 miles to the east, and Tucson, with a 2015 estimated population of approximately 531,650, located approximately 65 miles (105 km) to the north.[5] Pima County, where Tucson is located, had a 2015 estimated population of approximately 1,010,000.[6]

Patagonia has limited social and economic infrastructure. The Town has a public elementary and middle school and a high school serving grades 9 through 12. There are several commercial lodging locations, several restaurants, a small grocery store and a gas station. Patagonia has a Police Department with a small, fully-staffed force. The Santa Cruz County Sheriff and the Arizona Department of Public Safety Highway Patrol Division, patrol the area around Patagonia and the Project Area. Medical facilities in Patagonia include a small family medical clinic and the Patagonia Fire Department’s Emergency Medical Technician (EMT) service. The Fire Department also has helicopter landing facilities for transporting serious medical cases to larger hospitals in Nogales or Tucson. Nogales has a regional hospital. The Tucson metropolitan area of eastern Pima County has historically been the commercial and service / supply centre for the mining industry in southern Arizona. Tucson has a commercial airport and large rail centre.

Although the Patagonia area has historically been a mining, ranching, and railroad community that would generally be favorable to development of a major mining operation with the attendant economic benefits and increase in employment opportunities, the Project, as well as past and current drilling activities by AMI, have already attracted the attention of local and national environmental organizations, and the community appears to be divided in its support of the Project. In recent years, the Patagonia and nearby Sonoita areas have attracted artists and upscale, well-educated, professional/technical individuals who have either retired to the area or commute to work elsewhere. Sonoita is also home to a nascent wine industry. Many local businesses cater to the tourist and outdoor sporting industry. The Patagonia Mountains, in which the Project Area is located, have been noted internationally as a bird-watching destination to observe numerous species of rare and exotic birds. The area is also popular for other outdoor recreational activities, including hiking, biking, horseback riding, and off-road four-wheel driving within the CNF lands. As a result, it is expected that the Project may attract similar levels of opposition as has other recent mine permitting efforts in the region.

20.2 Biological and cultural resource work completed to date

Since 2012, AMI has conducted biological and cultural studies and surveys in portions of the Property. These efforts have included multiple years of survey for species listed under the ESA, Forest Service sensitive species, and full pedestrian surveys for cultural resources adjacent to and in portions of the Project Area. This section

[2] United States Census Bureau American Fact Finder. 2015 Population Estimates (as of 1 July 2015).
[3] United States Census Bureau American Fact Finder. 2010 Population Estimates (as of 1 July 2015).
[4] Town of Patagonia. General Plan 2009. Available online at: https://issuu.com/seagoedd/docs/patagonia_general_plan?layout=http://skin.issuu.com/v/light/layout.xml&showFlipBtn=true&e=3005223/4148168.
[5] United States Census Bureau American Fact Finder. 2015 Population Estimates (as of 1 July 2015).
[6] Ibid.

summarizes the results of environmental surveys completed to date. Sections 20.2.3 and 20.2.4, describe the permitting implications of survey results and findings.

In 2012 and 2013, AMI commissioned surveys for Sonoran tiger salamander, a species listed as endangered without critical habitat under the ESA. The surveys determined that the closest known observation of the Sonoran tiger salamander is approximately 2 miles (3.2 km) away and across several topographic ridges from the Project Area.[7,8]

Lesser long-nosed bats, a species listed under the ESA without critical habitat, are known to forage within the Property,[9,10] but an extensive search of the known abandoned mine features and monitoring of select features detected no evidence that this species was using these features as day-roosting habitat in 2012, 2013, and 2017.[8,9,11] The closest known lesser long-nosed day-roost to the Project Area is approximately 5 miles (8 km) away.

Surveys were conducted for yellow-billed cuckoo in 2012, 2013, 2016, and 2017. Yellow-billed cuckoos, a species listed as threatened under the ESA with proposed critical habitat, have been detected along Harshaw Creek and other drainages in the vicinity of the Project Area.[12,13,14,15] There are no areas of proposed critical habitat within or adjacent to the Project Area.

Mexican spotted owl (MSO) is listed as threatened with critical habitat under the ESA. One pair of MSO and their associated Protected Activity Center (PAC) is located in Alum Gulch, within approximately 0.5 miles (0.6 km) of the Project Area. The MSO pair has historically been reported from this PAC and breeding was confirmed by surveys in 2016.[16] Widespread surveys for MSO in the areas within and adjacent to the Project Area in 2012, 2013, 2016, and 2017 have detected two additional roost locations for MSO.[16,17,18,19] One of these roost locations is approximately 1 mile (1.6 km) south and the other approximately 5 miles (8 km) south of the Project Area.[19]The Project Area is located in designated critical habitat for the species.

The jaguar and ocelot, both listed as endangered under the ESA, are known from the mountainous regions of southeastern Arizona, they have historically been detected in the Patagonia Mountains in the past, but are not known to currently occupy them. The Project Area is located in designated critical habitat for the jaguar; critical habitat has not been designated for ocelot. Over four years ago, a male ocelot originally detected in the Huachuca Mountains in 2011 was detected once in the Patagonia Mountains, but has since returned to the Huachuca Mountains over 15 miles (24 km) east of the Project Area and has not been detected since in the Patagonia

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[7] WestLand Resources, Inc. 2013. 2012 Surveys for the Sonora tiger salamander (*Ambystoma movortium stebbinsi*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. April.

[8] WestLand Resources, Inc. 2013. 2013 Surveys for the Sonora tiger salamander (*Ambystoma mavortium stebbinsi*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. December.

[9] WestLand Resources, Inc. 2013. Summary of 2013 Survey for lesser long-nosed bat (*Leptonycteris yerbabuenae*) in the Patagonia Mountains near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. December.

[10] WestLand Resources, Inc. 2017. 2017 Survey of potential bat roost habitat in support of long-range planning efforts. Prepared for Arizona Minearls, Inc. November.

[11] WestLand Resources, Inc. 2013. 2012 Surveys for lesser long-nosed bat (*Leptonycteris curasoae yerbabuenae*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. April.

[12] WestLand Resources, Inc. 2013. Revised 2012 Survey for yellow-billed cuckoo (*Coccyzus americanus*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. April.

[13] WestLand Resources, Inc. 2013. 2013 Survey for yellow-billed cuckoo (*Coccyzus americanus*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. December.

[14] WestLand Resources, Inc. 2016. 2016 yellow-billed cuckoo (*Coccyzus americanus*) in support of the Hermosa Taylor Drilling Plan of Operations. November.

[15] WestLand Resources, Inc. 2017. 2017 Yellow-billed cuckoo survey in support of long-range planning efforts. Prepared for Arizona Minerals, Inc. November.

[16] WestLand Resources, Inc. 2016. 2016 Surveys for Mexican spotted owl (*Strix occidentalis lucida*) in Support of the Hermosa Taylor Drilling Plan of Operations. November.

[17] WestLand Resources, Inc. 2013. 2012 Survey for Mexican spotted owl (*Strix occidentalis lucida*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. April.

[18] WestLand Resources, Inc. 2013. Summary of 2013 Survey for Mexican spotted owl (*Strix occidentalis lucida*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. December.

[19] WestLand Resources, Inc. 2017. 2017 Survey for Mexcan spotted owl (*Strix occidentalis lucida*) in support of long-range planning efforts. Prepared for Arizona Minerals, Inc. November.

Mountains.[20] There has been extensive survey for jaguar and ocelot by the University of Arizona and, no other ocelots currently known to occur in Arizona have been detected in the Patagonia Mountains. There are only two jaguars known to occur at present in the USA; one has been observed in the Huachuca Mountains and one in the Dos Cabezas Mountains in eastern Arizona. No jaguars have been detected in the Patagonia Mountains in the past 50 years.[21,22]

In 2012 and 2013, AMI commissioned surveys for Chiricahua leopard frog (CLF; *Lithobates* [*Rana*] *chiricahuensisi*), a species listed as threatened under the ESA with designated critical habitat, in the vicinity of the Project Area. No CLF were detected and the areas within and adjacent to the Project Area, and there is no designated critical habitat in the area. During these surveys, no other special-status species that might inhabit aquatic systems in the vicinity of the Project Area were observed. These species include Arizona treefrog (*Hyla wrightorum*), northern Mexican garter snake (*Thamnophis eques megalops*), springsnails (*Pyrgulopsis* spp.), and Huachuca water umbel (*Lilaeopsis schaffneriana* ssp. *recurva*).

Gila topminnow (*Poeciliopsis occidentalis occidentalis*) is listed as an endangered species without critical habitat under the ESA. Surveys for this species were conducted in 2013 in a 0.5 mile (0.8 km) perennial reach of Harshaw Creek, approximately 4.5 miles (7.2 km) downstream from the Project Area. Gila topminnow were not detected during these surveys.[23]

In addition to analyses of species listed as threatened or endangered under the ESA, the CNF also evaluates the effects to CNF designated sensitive and rare species as part of the NEPA process, (see Section 20.2.3). Surveys for USFS sensitive plant and animal species were commissioned by AMI between 2012 and 2016 in the vicinity of the Project Area. These sensitive species include beardless chinchweed (*Pectis imberbis*), Bartram stonecrop (*Graptopetalum bartramii*), Sonoran noseburn (*Tragia laciniata*), and *Hexalectris* orchid species. Neither beardless chinchweed nor Bartram stonecrop were detected in the area surrounding the Project Area in 2013 and 2016.[24] Hexalectris species have been detected in the vicinity of the Project Area.[25,26] Surveys conducted in 2016 have also detected Sonoran noseburn (*Tragia laciniata*) in areas adjacent to the Project Area (WestLand unpublished data).

In 2012 and 2013, surveys for two grassland avian species listed as sensitive by Region 3 of the USFS: Arizona grasshopper sparrow (*Ammodramus savannarum ammolegus*) and Baird's sparrow (*Ammodramus bairdii*) were conducted in the vicinity of the Project Area. Neither the Arizona grasshopper sparrow nor Baird's sparrow were detected.[27]

Surveys for CNF sensitive small mammals in the vicinity of the Project Area were conducted in 2012. No small mammals currently considered sensitive by the CNF were captured.

In 2013, 2016, and 2017, AMI commissioned surveys for northern goshawk (*Accipiter gentilis*), a species that is considered sensitive by Region 3 of the USFS in the CNF, within portions of the Property. No goshawks were

[20] USFWS. 2016. Recovery Plan for the Ocelot (*Leopardus pardalis*), First Revision. U.S. Fish and Wildlife Service, Southwest Region, Albuquerque, New Mexico.
[21] Culver, M., Malusa, S., Childs, J.L., Emerson, K., Fagan, T., Harveson, P.M., Haynes, L.E., Sanderson, J.G., Sheehy, J.H., Skinner, T., Smith, N., Thompson, K., and Thompson, R.W., 2016, Jaguar surveying and monitoring in the United States: U.S. Geological Survey Open-File Report 2016–1095, 228 p., http://dx.doi.org/10.3133/ofr20161095.
[22] USFWS. 2016. Amended Final Reinitiated Biological and Conference Opinion for the Rosemont Copper Mine, Pima County, Arizona.
[23] WestLand Resources, Inc. 2013. 2013 Survey for Gila Topminnow (*Poeciliopsis occidentalis occidentalis*), in the Patagonia Mountains, Near Harshaw, Arizona.
[24] WestLand Resources, Inc. 2013. 2012 Survey for Bartram's Stonecrop (*Graptopetalum bartramii*) and Beardless Chinchweed (*Pectis imberbis*), in the Patagonia Mountains, Near Harshaw, Arizona.
[25] WestLand Resources, Inc. 2012. Survey for *Hexalectris colemanii* and *Hexalectris arizonica* across southeastern Arizona – 2012. [*publically available at http://www.rosemonteis.us/technical-reports/all*].
[26] WestLand Resources, Inc. 2013 [revised]. 2012 Survey for *Hexalectris colemanii* and *H. arizonica* in the Patagonia Mountains, Near Harshaw, Arizona.
[27] WestLand Resources, Inc. 2013. 2012-2013 Surveys for Grassland Bird Species in the Patagonia Mountains, Near Harshaw, Arizona.

observed during these surveys.[28,29,30] In general, habitat selection by nesting northern goshawks in southern Arizona is poorly understood, but potential habitat for goshawks, particularly in patches of dense riparian woodland, exists in the vicinity the Project Area.

Based on the results of these biological surveys, the development of the Project is not expected to result in a trend towards federal listing under the ESA for any CNF designated sensitive species, and their presence in the vicinity of the Project Area is not expected to preclude development of the Project.

Between 2012 and 2017, a large portion of the Property was surveyed for cultural resources, and a number of historic and pre-historic cultural resources were identified.[31,32,33] Surveys have not been conducted on the majority of private lands owned by AMI that will be used for development of the Project. On the portions of AMI's private land surveyed for cultural resources, historic resources have been identified.

20.2.1 Forest Service approval of a POO

Even if facilities and operations for the Project are located on private land, there may be off-site improvements for access and utilities (power and water) that cross land administered by the CNF. Pursuant to USA mining laws, AMI is entitled to conduct operations that are reasonably incident to the exploration and development of mineral deposits on its unpatented mining claims, i.e., those claims for which the surface right is still held by federal government. Pursuant to regulations of the U.S. Secretary of Agriculture, AMI must conduct mining operations on public lands administered by the USFS in accordance with the requirements found at 36 Code of Federal Regulations (CFR) Part 228A and in accordance with a POO that has been approved by the USFS. Pursuant to USFS regulations AMI has assumed that the planned activities on CNF lands will require approval of a POO. If required, AMI will prepare a proposed POO and submit that to the CNF. Once the POO is submitted and determined sufficient to initiate environmental review, the CNF will conduct an environmental review of the plan in accordance with the requirements of the NEPA and USFS implementing regulations and policy.

NEPA requires the federal government to involve the public in its planning / decision making activities, consider the effects of its decisions on the environment, and to disclose the effects of its activities to the public. There are three levels of NEPA review:

1. Categorical Exclusion, for groups or categories of actions that are relatively minor and have been determined by the action agency not to have significant impacts to the human environment.
2. Environmental Assessment for actions that are not categorically excluded from NEPA analysis and are not expected to have significant affects to the human environment.
3. Environmental Impact Statement for actions that result in significant effects to the human environment.

When multiple federal agencies are involved in a project, one of the agencies normally will act as the lead agency, and the other federal action agencies as cooperating agencies for the purpose of NEPA compliance can rely on the lead agencies NEPA analysis provided it fully covers the actions of the cooperating agencies. If NEPA compliance is required for the Project, it is anticipated that the CNF will act as the Lead Agency for NEPA and that

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[28] WestLand Resources, Inc. 2013. Summary of 2013 Survey for Northern Goshawk (*Accipiter gentilis*) in the Patagonia Mountains, Near Harshaw, Arizona.
[29] WestLand Resources, Inc. 2016. 2016 Northern Goshawk (*Accipiter gentilis*) survey in support of the Hermosa Taylor Drilling Plan of Operations. December.
[30] WestLand Resources, Inc. 2017. 2017 Northern Goshawk (*Accipiter gentilis*) survey in support of long-range planning efforts. Prepared for Arizona Minerals, Inc. November.
[31] WestLand Resources, Inc. 2013. A Cultural Resources Inventory of 2,634 Acres of Private and Federal Lands in Support of the Hermosa Drilling Project Plan of Operations Within the Coronado National Forest, Arizona.
[32] WestLand Resources, Inc. 2016. A Cultural Resources Inventory 160 Acres of Coronado National Forest Land for Possible Mineral Exploration Activities in Santa Cruz County, Arizona.
[32] WestLand Resources, Inc. 2017. A Cultural Resources Inventory of Approximately 19.4 Acres of Coronado National Forest Land near Harshaw, in Santa Cruz County, Arizona.
[32] WestLand Resources, Inc., 2017. A Cultural Resources Inventory of Approximately 9.8 Acres of Coronado National Forest Land near Harshaw, in Santa Cruz County, Arizona.
[33] WestLand Resources, Inc. unpublished data.

the Corps of Engineers, if a CWA Section 404 permit is required, would be a cooperating agency. The USFWS does not have a NEPA obligation associated with any ESA Section 7 consultation that may be required.

Under 36 CFR Part 228.5, the CNF must determine whether to approve the POO as submitted by AMI, as proposed, or to require changes or additions deemed necessary to meet the requirements of the regulations for environmental protection. The purpose of the CNF's evaluation of the proposed action and the evaluation of alternatives to the proposed action during NEPA is to determine if changes or additions to the POO are necessary to meet the requirements of the regulations for environmental protection set forth in 36 CFR Part 228.8. The CNF cannot select the no action alternative that would be analyzed as part of the NEPA review. A CNF NEPA review of the POO is also expected to provide the NEPA review required for the Corps (if a 404 permit is required).

As a general practice, completion of the NEPA process, if required will ultimately determine the permitting timeline for the Project. All other required federal and state environmental permits are expected to be completed within the time frame anticipated for NEPA compliance. The time to complete required permitting efforts can vary substantially depending on the level of NEPA review and other factors outside of AMI's control. A recent study published by the National Association of Environmental Professionals (NAEP) evaluated EIS preparation and review times in 2016 (Table 20.1). The median time to complete an EIS was approximately 4.5 years, ranging from 0.3 to 16 years. Over the past number of years this average time has been increasing approximately 1 month per year.[34] The completion of an EA is typically much shorter, but national statistics to inform an expected range of preparation times are not available. Previous experience on permitting of mining projects suggests that 1.5 to 3 years is a reasonable time range, but can vary significantly depending upon public interest, the availability of agency resources, and nature of the resources likely to be affected. While every project has its own unique circumstances that can affect compliance schedules for the NEPA, it is currently anticipated that completion of NEPA review for development of the Project will take between 2 and 6 years. Once the environmental review of the Project is complete, revisions to the submitted POO will likely be necessary to incorporate changes during the NEPA process and additions required by the USFS. Note that the USFS will also require that closure and reclamation plans meet performance standards and have financial assurance. Table 20.1 shows the duration of USFS, BLM, and Corps EIS process for Final and Final Supplemental EIS's completed during 2016.

Table 20.1 Duration of USFS, BLM, and Corps EIS process completed during 2016.

Agency	Number of EIS completed in 2016	Preparation time				
		Units	Mean	Median	Min	Max
USFS	30	Calendar days	1,581	1,208	406	4,676
		Approximate months	53	40	14	156
BLM	21	Calendar days	2,257	2,130	693	5,706
		Approximate months	75	71	23	190
Corps	18	Calendar days	2,421	2,555	918	5,706
		Approximate months	81	85	31	190
All agencies (including those above)	168	Calendar days	1,864	1,615	128	5,706
		Approximate months	62	54	4	190

[a] Defined as the timeframe between the publication of a Notice of Intent to prepare and EIS and the publication of the Notice of Availability for the EIS.
Source: National Association of Environmental Professions. 2017. Annual NEPA Report 2016 of the National Environmental Policy (NEPA) Practice.

20.2.2 Clean water act Section 404 permit

At this time, it is not certain that surface water features within the likely footprint of the Project are subject to jurisdiction under Section 404 of the CWA. The presence or absence of drainage features subject to Corps jurisdiction under the CWA is determined by a jurisdictional waters delineation (a Delineation) by the Corps. A Delineation has not been completed for all of AZ's private land or adjacent USFS lands. A portion of AMI's private

[34] National Association of Environmental Professions. 2016. Annual NEPA Report 2015 of the National Environmental Policy (NEPA) Practice.

land has been evaluated by the Corps and the small drainage features within that area have been determined not to be waters of the US. If the Corps determines that any remaining surface water features within the Project Area are subject to their jurisdiction under the CWA, a CWA Section 404 permit will be required for the Project activities that propose discharges of fill to these features. Based upon preliminary review of the Project, some drainages in the Project Area may be considered waters of the U.S. under the current Corps regulations.

If the Project requires a CWA Section 404 permit, it is not likely to qualify for the simpler general permit program administered by the Corps, but rather will likely require an Individual CWA Section 404 permit. Securing this permit will require completion of an alternatives analysis to identify the least environmentally damaging practicable alternative, and development of mitigation measures in accordance with applicable Corps regulations to offset unavoidable impacts to waters of the US. A State Water Quality Certification from ADEQ will also be required, and the Corps must meet its NEPA obligations. Consistent with past practices, the Corps is likely to seek cooperating agency status with the CNF rather than prepare their own NEPA review document. Unlike the CNF, however, the Corps can select the no action alternative when they prepare their separate decision document permit for the Project.

20.2.3 Endangered species act

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 species and their critical habitat. If a federal agency with authority over the Project determines that the Project may affect a listed species or designated critical habitat, consultation with the U.S. Fish and Wildlife Service (USFWS) will be required. Based on experience with other mining projects in southern Arizona, it is anticipated that formal Section 7 consultation may be required. During this consultation the USFWS is required to determine if any listed species will be harmed or harassed (collectively referred to as 'take') by the Project and determine if adverse impact to critical habitat will occur. USFWS will also determine, during this consultation, if the proposed action is likely to jeopardize the continued existence of any listed species[35] or adversely modify critical habitat.[36] Should the USFWS make a jeopardy or adverse modification determination, they are required to identify reasonable and prudent alternatives to the proposed action that meet the purpose and need of the proposed activity. If an incidental take permit is required, the USFWS is likely to identify binding reasonable and prudent measures (RPMs) and terms and conditions (TCs) of 'take' to offset the impacts. Importantly, the ESA does not necessarily preclude development of projects with potential impacts to federally listed species.

Regardless of whether the Project will require a federal agency action, Section 9 of the ESA will be applicable and the 'take' of listed species is prohibited without a permit. Should the Project have no federal nexus and require a permit for 'take' of listed species, AMI must obtain a Section 10 permit under the ESA. The Section 10 permitting process is an applicant-driven process, is often complex, requires mitigation to offset 'take' of listed species, and can take several years to develop in coordination with the USFWS.[37]

The Project Area is within designated critical habitat for jaguar and Mexican spotted owl. In addition, Sonora tiger salamander and lesser long-nosed bat, both listed as endangered under the ESA, the threatened Mexican spotted owl, and yellow-billed cuckoo, of which the populations in western North America are listed as threatened, are known to occur in the vicinity of the Project Area (see Section 20.2). It is anticipated that the Project may trigger Section 7 consultation if a federal permitting process is required. While ultimately to be determined by the USFWS, the Project is unlikely to jeopardize any listed species or to adversely modify designated critical habitat, and as such, the ESA is not anticipated to preclude development of the Project. The USFWS is likely to authorize 'take' listed species by the Project. As part of this authorization, it is anticipated that USFWS will issue RPMs and T&Cs of 'take'. Often these conditions are determined during consultation and are part of negotiated conservation

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[35] Jeopardizing the continued existence is defined as directly or indirectly affecting a species' numbers, reproduction, or distribution in such a way as to considerably reduce the species' ability to survive and recover in the wild. (50 CFR Part 402)

[36] Adversely modifying critical habitat is defined as "Destruction or adverse modification means a direct or indirect alteration that appreciably diminishes the value of critical habitat for the conservation of a listed species. Such alterations may include, but are not limited to, those that alter the physical or biological features essential to the conservation of a species or that preclude or significantly delay development of such features." (50 CFR Part 402)

[37] Because Section 10 permits are discretionary decisions by the USFWS or National Marine Fishers Service, these permits generally require NEPA review and independent ESA compliance by these agencies.

measures proposed by the project proponent. These conservation measures will ultimately be incorporated into the final POO.

20.2.4 National historic preservation act

As stated in Section 20.2, a large portion of the Property has been surveyed for cultural resources and a number of historic and pre-historic cultural resources have been identified. If the Project will have a federal nexus, any adverse effects to cultural properties will require consultation and mitigation in the form of data recovery and research. Should impacts to cultural resources eligible for registration on the National Register of Historic Places[38] (Historic Properties) be unavoidable, authorization to mitigate the impacts to these resources is obtained through implementation of Section 106 consultation under the NHPA.

The consultation is typically conducted between the federal action agencies and the State Historic Preservation Office (SHPO). The Advisory Council on Historic Preservation will also be asked if they would like to participate in the consultation but typically they decline. The National Historic Preservation Act also requires that federal action agencies consult with tribes having cultural affinity to the Project Area, development of an historic properties treatment plan, and development and execution of a Memorandum of Agreement (MOA). Signatories to the MOA could be the SHPO, CNF, Corps (if a 404 permit is required), concurring parties to the agreement can include interested Native American groups and AMI. Concurring parties are not obligated to sign the MOA but will be given opportunity to review and comment. It is not anticipated that effects to cultural resources will preclude development of the Project.

20.2.5 Natural Gas Act Section 7

The Project may include connecting into an existing transmission pipeline, a new compressor station and a new distribution pipeline to the mine. Under Section 7 of the Natural Gas Act, Federal Energy Regulatory Commission (FERC) is charged with evaluating whether interstate natural gas pipeline projects proposed by private companies should be approved and, if determined appropriate for approval, issues certificates of public convenience and necessity of natural gas facilities engaged in interstate natural gas transportation by pipeline. The FERC decision to approve a project may require NEPA compliance. It is uncertain whether or not FERC involvement will be required (depending on whether or not the pipeline work will constitute an interstate project) however, should a NEPA process be required through FERC, the process is not expected to preclude development of the Project.

20.3 State environmental permitting

A variety of state permits and approvals may be necessary to develop the Project. A summary of the expected state permits / approvals, the lead agency for each permit/approval, and comments relevant to each are provided in Table 20.2, at the end of this section. This list has been prepared based on the current understanding of the Project approach and the regulations currently in effect. The list may be subject to change as Project development continues forward. The timeframes described are based on recent projects in Arizona, but are subject to change depending on the complexity of the project, public opinion, agency capabilities and priorities and other factors outside of AMI's control.

Discussion of the most significant state environmental permits and approval actions is provided in the following subsections. These processes are anticipated to be completed concurrent with the NEPA analysis for the Project and none of these permitting processes are expected to preclude development of the Project.

20.3.1 Arizona state cultural resource regulations

The Arizona Antiquities Act (ARS §41-841 through §41-847) was enacted in 1927 and subsequently has been amended. The law provides for the protection and regulation of archaeological and paleontological sites on lands owned or controlled by the State of Arizona, or an agency of the State. The Arizona State Historic Preservation Act (ARS §41-861 through §41-866) was passed in 1982. The Act places the responsibility for historic properties on the head of state agencies, requires state agencies to identify properties meeting the criteria of the Arizona Register of Historic Places (Administrative Code R12-8-302). It also establishes a responsibility for agencies to

[38] The official list of the Nation's historic places considered worthy of preservation.

actively manage historic properties, provides for the State Historic Preservation Officer to review agency plans involving an Arizona Register of Historic Places Property, and establishes an acquisition and preservation fund. Importantly, the law criminalizes the intentional disturbance of human remains or funerary objects on private land within the state. For unintentional disturbance on private land, the statute defines a process for reporting, treatment, and disposition of human remains. No Arizona state regulations that require systematic survey or treatment of cultural resources within the Project Area are known.

20.3.2 Air quality permit

Air quality is regulated at the federal level by the EPA under the CAA, although authority for air quality permitting has been delegated by the EPA (Region IX) to the Arizona Department of Environmental Quality (ADEQ), with the EPA retaining oversight. Prevention of Significant Deterioration (PSD) is a program established under the CAA to maintain ground-level concentrations of regulated air pollutants within National Ambient Air Quality Standards (NAAQS), which have been established for a variety of pollutants, including ozone, carbon monoxide, nitrogen dioxide, sulphur dioxide, particulate matter, and lead. Areas of the USA in compliance with NAAQS are designated as “attainment areas”. A PSD permit allows a facility to be constructed and operated within an attainment area.

The Project Area is presently located in an attainment area for all regulated pollutants. A relatively small non-attainment area for particulate matter is located in the vicinity of Nogales, Arizona, such that any facilities proposed beyond the current Project Area should be reviewed for potential effects to this non-attainment area. PSD review is triggered for proposed emissions of a regulated pollutant greater than 250 tpa or for proposed emissions greater than 100 tpa, if the proposed facility includes a “categorical source”.

The PSD program also provides special protection for designated Class I areas, which are areas of special national or regional natural, scenic, recreational, or historic value. Generally, these additional analyses come into play for proposed facilities planned to be constructed within 6.2 miles (10 km) of a Class I area. Currently, there are no Class I areas within 6.2 miles (10 km) of the Project Ares.[39]

ADEQ has a Unitary Permit Program wherein construction permits and operating permits are combined into one application and subsequently one air quality control operating permit is issued. ADEQ has two air quality permit classifications: Class I (major source) and Class II (minor source). A Class I air quality operating permit is required for emissions of regulated pollutant exceeding 100 tpa (not to be confused with the PSD threshold). An assessment of the potential-to-emit (PTE) of regulated air pollutants allows the determination of the source classification for an air quality control permit application as a Class I or a Class II.

Development and issuance of a Class I permit may take 18 months to over 2 years, based on complexity and level and nature of public comment, whereas a Class II permit generally takes about 9 to 12 months. In either case, it is anticipated that ADEQ will require atmospheric dispersion modelling to demonstrate compliance with NAAQS for the Project. Obtaining a Clean Air Act permit is likely to occur within the timeframe of any NEPA process and information collected to support the permit will be used to support air permitting and the NEPA process. It is not anticipated that obtaining an air permit will preclude development of the Project.

20.3.3 Aquifer protection permit

ADEQ is responsible for issuing an Aquifer Protection Permit (APP) to facilities that discharge pollutants which may have the potential to adversely impact groundwater quality. The following types of facilities fall under APP regulations: surface impoundments (process water ponds, holding ponds, settling pits or ponds, etc.), tailings storage facilities, waste rock stockpiles subjected to leaching, mine leaching facilities, wastewater treatment facilities, septic tanks (sewage treatment facilities, including on-site wastewater treatment facilities), injection wells, and point-source discharges to “navigable waters”.

[39] EPA Region 9 – Air. 2011. Class I Areas in EPA Region 9. Available online at: https://www3.epa.gov/region9/air/maps/pdfs/air1100018-9.pdf.

In order to obtain an APP, applicants must demonstrate to the satisfaction of the ADEQ:

1 That the facilities are designed and will be constructed and operated according to "best available demonstrated control technology (BADCT)."
2 That the facility will not "cause or contribute to" an exceedance of Aquifer Water Quality Standards (AWQS) at designated point(s) of compliance, or if AWQS for a pollutant has already been exceeded in an aquifer (pre-existing condition), that no additional degradation will occur.
3 That the applicant is technically capable of carrying out the conditions of the permit.
4 That the applicant is financially capable of constructing, operating, closing and assuring post-closure care of the facility.
5 That the facility complies with applicable municipal or county zoning ordinances and regulations (however, mines are exempted from local zoning ordinances).

In the BADCT demonstration, two general approaches are available:

1 The use of "prescriptive" BADCT criteria.
2 The use of "individual" BADCT criteria.

Prescriptive BADCT include pre-approved control technologies for tailings impoundments and certain types of ponds and generally represent conservative approaches that are relatively independent of site conditions. As such, the site characterization requirements are generally less rigorous.

In order to characterize pre-existing conditions and demonstrate that the regulated facilities will not cause exceedances of AWQSs at points of compliance, an extensive hydrogeological characterization of the site, including the characterization of subsurface water levels, groundwater flow direction(s), groundwater quality, and other parameters, is required. Baseline data are required to establish normal seasonal fluctuations in subsurface conditions and background water quality. Several existing wells are currently being monitored by AMI, and additional hydrogeological investigations have been conducted in 2017. One or more point-of-compliance (POC) wells will be identified during the APP permitting process. To the extent possible, one or more of the wells identified in the POO will be used as POC wells.

In addition to the hydrogeological characterization, characterization of representative samples of materials representing waste rock and tailings is a permit requirement. These samples assist in identifying the BADCT approaches for waste rock and tailings accumulations on site.

The permitting process for an APP on a project of this size typically requires 12 to 18 months. ADEQ has an expedited program for accelerating the APP process in which an additional fee is paid to use an ADEQ-approved consultant.

20.3.4 Arizona pollutant discharge elimination system

The ADEQ's AZPDES program was developed out of Section 402 of the CWA, of which the EPA has ceded administration to ADEQ. AMI currently maintains stormwater compliance coverage under the AZPDES Multi-Sector General Permit (MSGP-2010) industrial stormwater program. Mine facilities, which can include associated pre-mining exploration and construction, are required to obtain coverage for discharges of stormwater from their operations. This program requires a project proponent to prepare a Stormwater Pollution Prevention Plan (SWPPP), submit a Notice of Intent (NOI) to discharge stormwater, install appropriate Best Management Practices (BMPs), and conduct regular inspections of the site and analytical monitoring during exploration, construction and operations, in accordance with the approved SWPPP.

Stormwater discharges to Harshaw Creek and Alum Gulch, drainages in the immediate vicinity of the Project Area that are classified by ADEQ as impaired waters, will require a demonstration that the discharges are not expected to cause or contribute to an exceedance of a water quality standard. Harshaw Creek and Alum Gulch have an approved Total Maximum Daily Load (TMDL), a calculation of the amount of a pollutant that a waterbody can receive and still safely meet water quality standards. Because discharges will be to impaired water, ADEQ will require additional limits, controls, or monitoring necessary to be consistent with the assumptions of any available waste load allocation in the TMDL.

It is anticipated that the Project will require coverage under an individual AZPDES permit to discharge treated waste water. This will include effluent limitations, usually consisting of both numeric and narrative standards. The numeric limitations typically restrict quantities, rates, and concentrations of pollutants that may be present in the discharge, and can be either technology or water quality-based. Technology-based standards require usage of available pollution control technology, while water quality-based standards protect ambient water quality by requiring the discharger to achieve the applicable numeric standard (as mentioned above). If both technology and water quality-based standards exist for a particular constituent, the more restrictive standard applies. It is not anticipated that obtaining an AZPDES individual permit will preclude development of the Project.

20.3.5 Arizona 401 water quality certification

The ADEQ's Section 401 certification is issued to ensure that federally permitted or licensed activities do not cause a violation of state water quality standards when an activity may result in a discharge to water of the state. In Arizona, this certification is almost exclusively required only when a 404 permit is also required. Each review is specific to the proposed project and the project's site. A State Water Quality Certification is necessary before a permit may be issued by a federal agency. The certification process runs concurrently with the NEPA process. It is not anticipated that obtaining a 401 Water Quality Certification will preclude development of the Project.

20.3.6 Arizona state mine inspector

The Arizona State Mine Inspector (ASMI) has jurisdiction over reclamation plans, associated costs, and financial assurance mechanisms. The amount of financial assurance is based on the actual estimated costs of reclamation. These costs and financial assurance mechanisms will be developed concurrent with all other permitting and will not preclude development of the Project.

20.3.7 Arizona corporation commission

The current design of the Project will not require a Certificate of Environmental Compatibility (CEC) issued by the Arizona Corporation Commission (ACC). For transmission lines, a CEC is required only for lines 115 kV or larger. Note that voltage from multiple circuits on a line is not cumulative for the purposes of the CEC requirement.

20.3.8 Class V Injection Well Permit

Arizona's UIC program is a direct implementation program, meaning it is overseen directly by the EPA's Underground Injection Control (UIC) Program. The purpose of the UIC Program is to ensure that Underground Sources of Drinking Water (USDW) are not going to be adversely affected by the Class V operations, such as structural cemented paste backfill.

20.3.9 Arizona department of water resources application for approval to construct a dam

The project will include construction of the tailings storage facility underdrain collection pond embankment. The application for construction of this facility has been approved by ADWR as of 9 November 2017.

20.3.10 Mine Safety and Health Administration

The Mine Safety and Health Administration (MSHA) sets forth mandatory health and safety standards for surface and underground metal and non-metal mines. The purpose of these standards is the protection of life, the promotion of health and safety, and the prevention of accidents.

Table 20.2 Environmental permits and approvals

Lead agency	Permit, approval, or other action	Described in section	Comment
Federal permits, approvals, and actions			
USFS	Plan of Operations (POO) Approval for Mining	20.2.1	POO for mining operations may be required to be submitted after completion of a favourable Feasibility Study, incorporating the results of the drilling activities. USFS needs to comply with NEPA before making a decision on the POO.
USFS	NEPA Compliance and Decision pursuant to NEPA for mining operations	20.2.1	An EIS may be required for the mining operation. The NEPA process, including obtaining the record of decision (ROD), is expected to take 2 to 6 years or more to complete, if required. EPA has review authority of EISs under the Clean Air Act, Section 309.
U.S. Army Corps of Engineers (Corps)	CWA Section 404 Permit	20.2.2	Permit(s) required for discharge of fill material to waters of the U.S, including jurisdictional wetlands. An individual permit may likely to be required, unless affected tributaries on the site are determined by the Corps to be "non-jurisdictional". An individual permit requires NEPA compliance and a Record of Decision (ROD), which is expected to be performed in coordination with the CNF NEPA process. Timeline is generally coincident with the CNF NEPA process. EPA has authority to review the CWA 404 permit public notice, elevate concerns, and require restrictions related to the discharge area.
US Fish and Wildlife Service (USFWS)	Endangered Species Act Section 7 Consultation	20.2.3	USFWS review and consultation is likely to be required for CNF POO decision and Section 404 permit. Consultation documentation and process generally occurs in coordination with NEPA.
Consultation with the State Historic Preservation Office (SHPO)	Section 106 of the National Historic Preservation Act (NHPA)	20.2.4	Consultation with the SHPO and consulting Native American tribes is required for CNF POO decision and Section 404 permit. Consultation documentation and process generally occurs in coordination with NEPA.
Federal Energy Regulatory Commission (FERC)	Approval of interstate natural gas pipeline projects proposed by private companies and issuance of certificates of public convenience and necessity of natural gas facilities engaged in interstate natural gas transportation by pipeline.	20.2.5	If the natural gas pipeline features that may be used for the project are determined to be considered an interstate natural gas pipeline project, FERC would evaluate whether the pipelines should be approved and, if determined appropriate for approval, FERC will also issue certificates of public convenience and necessity of natural gas facilities engaged in interstate natural gas transportation by pipeline. The FERC decision to approve a project may require NEPA compliance.
State permits, approvals, and actions			
Arizona Department of Environmental Quality (ADEQ)	Air Quality Permit	20.3.1	EPA has granted air permitting primacy to the ADEQ. Required for mobile and stationary emission sources, including any source that may emit air pollutants (e.g. dust, listed air pollutants). Usually requires baseline studies and monitoring of weather and ambient air conditions. EPA may exercise authority to review the air permit. As an agency of the state ADEQ should comply with the Arizona State Historic Preservation Act and review the permit area for impacts to cultural resources. Will defer to federal agencies and the State Historic Preservation Office.
ADEQ	Individual Aquifer Protection Permit (APP)	20.3.2	Required for waste dumps, tailings storage, leaching facilities, process-water ponds and reservoirs, or any other facility that has the potential to "discharge" to the aquifer or vadose zone. Requires hydrogeological study and the submission of construction plans for the proposed facilities.
ADEQ	Aquifer Protection Permit (APP) Sewage Collection System	20.3.2	Individual On-Site Wastewater Treatment System, Sewage Collection System.
ADEQ	Arizona Pollutant Discharge Elimination System (AZPDES-MSGP) for Stormwater Discharges Associated with Industrial Activity-Mineral Industry General Stormwater Permit	20.3.3	EPA has granted ADEQ administration authoring of permits associated with Section 402 of the CWA. Regulates discharge to receiving waters. Substantive requirements are development and implementation of a SWPPP, best management practices, and regular inspections and monitoring.

Lead agency	Permit, approval, or other action	Described in section	Comment
ADEQ	Arizona Pollutant Discharge Elimination System (AZPDES) for Point Sources Waste Water Treatment Permit	20.3.3	EPA has granted ADEQ administration authoring of permits associated with Section 402 of the CWA. Regulates discharge to receiving waters. Substantive requirements are development and implementation of a SWPPP, best management practices, and regular inspections and monitoring.
ADEQ	401 Water Quality Certification	20.3.4	This certification is issued to ensure that federally permitted or licensed activities do not cause a violation of state water quality standards when an activity may result in a discharge to waters of the state. A State Water Quality Certification is necessary before a permit may be issued by a federal agency.
ASMI	Arizona State Mine Inspector Mined Land Reclamation Plan	20.3.6	Mined Land Reclamation Plan. Reclamation plans, associated costs and financial assurance for all metalliferous mining units and exploration operations with surface disturbance.
ACC	Arizona Corporation Commission Certificate of Environmental Compatibility	20.3.7	Not required for the Project. The utility provider for the transmission line would be required to apply for a CEC according to A.R.S. (reference: 40-3 60) for transmission lines 115 kv or greater.
EPA	Class V Injection Well Permit	20.3.8	Class V authorization is presumptively by rule; i.e., generally, an individual permit is not required. Required for injecting cemented paste backfill for structural support underground.
ADWR	Application for Approval to Construct a Dam	20.3.9	Approval to construct Tailings Storage Facility Underdrain Collection Pond Embankment.
MSHA	Mine Safety and Health Administration Mine Identification Number	20.3.10	MSHA sets forth mandatory health and safety standards for surface and underground metal and non-metal mines.

21 Capital and operating costs

This section is written discussing and tabulating the capital costs followed by the operating costs for each major area.

The total LOM capital cost estimate for the mine is US$1.2B that includes US$519M in pre-production capital and US$752M in sustaining capital.

The total LOM operating cost is estimated to be US$4.9B with an average unit cost US$50.56/t of mineralized material for the mine. The total operating cost includes mining (US$38.02/t), processing cost (US$10.01/t), material placement at the TSF (US$0.53/t) and General and Administration cost (US$2/t).

21.1 Underground mine capital cost estimate

A capital cost estimate for the underground mine was undertaken by AMC. Key areas include underground development, underground mining equipment, shaft and infrastructure. Equipment numbers were estimated to meet the production target of 3.6 Mtpa (3.3 M tonnes pa). Underground infrastructure costs are based on estimated quantities and some supplier quotes. If no direct quotes were obtained, costs were derived from benchmark construction costs, and assumptions and quotes from recent projects undertaken by AMC. Total underground capital cost is estimated to be US$955.6M with pre-production capital of US$263.4M and sustaining capital of US$692.3M.

The total underground mine capital cost estimate considering shaft and decline access is provided in Table 21.1. Pre-production capital (capital spent prior to Year 4) as well as the sustaining capital (total capital less Pre-production capital) is also provided. The individual figures in Table 21.1 do not include a contingency, EPCM or owner's costs and these are shown for all items together. Where applicable they are individually allocated in Table 21.2 to Table 21.5 as separate line items.

Table 21.1 Underground capital cost

Capital cost	Pre-production capital (US$M)	Sustaining capital (US$M)	Total capital (US$M)
Underground development	48.3	534.2	582.5
Mine equipment (sustain cap incl.)	44.0	94.8	138.8
Shaft	126.3	49.9	176.2
Underground infrastructure	10.1	13.3	23.4
Backfill plant	10.0	0	10.0
Engineering, Procurement and Construction Management (EPCM)	3.4	0	3.4
Owner's cost	0.9	0	0.9
Contingency	20.4	0	20.4
Total	**263.4**	**692.3**	**955.6**

Totals do not necessarily equal the sum of the components due to rounding.

21.1.1 Underground development capital cost

Cost for development is estimated at US$1,372/ft (US$4,500/m) for lateral waste development and US$1,524/ft (US$5,000/m) for vertical development assuming raisebored ventilation raises and passes. The underground capital cost estimate for development is US$582.5M and is summarized in Table 21.2.

Table 21.2 Underground development cost estimate

Capital development costs	Length (ft)	Unit cost (US$/ft)	Pre-production capital (US$M)	Sustaining capital (US$M)	Total capital (US$M)
UG Lateral development (waste)	402,285	1,372	43.4	508.3	551.8
UG Vertical development	20,160	1,524	4.8	25.9	30.7
Total	**422,445**		**48.3**	**534.2**	**582.5**

Totals do not necessarily equal the sum of the components due to rounding.

21.1.2 Underground mobile equipment capital cost

The underground capital cost estimate for mobile equipment is US$143.2M including contingency and is summarized in Table 21.3.

Table 21.3 Underground mobile equipment cost estimate

Description	Capital cost (US$M)
Pre-production capital*	44.0
Sustaining capital	94.8
Contingency	4.4
Total	**143.2**

*Years 1 to 3 inclusive.

The estimate for mobile equipment icludes the following:

- Longhole production drill (4)
- 2-boom development jumbo (8)
- Scoops (4 production, 6 development)
- 50-tonne trucks (6 production, 4 waste)
- Bolter (8)
- Ancillary equipment

21.1.3 Main production shaft capital cost

The main production shaft capital cost estimate is US$184.9M including owners costs and contingency and is summarized in Table 21.4. The cost estimate is based on a 2017 contractor quote for this project.

Table 21.4 Main production shaft cost estimate

Description	Capital cost (US$M)
Pre-production capital*	126.3
Sustaining capital	49.9
EPCM, owner’s costs, and contingency	8.7
Allowances for delays and bad ground	Included in contingency
Total	**184.9**

*Years 1 to 3 inclusive

21.1.4 Underground infrastructure capital cost

The underground infrastructure capital cost estimate is US$33.0M and is summarized in Table 21.5. The costs are based upon supplier quotations, pricing in the public domain, and unit rates from previous experience. The underground infrastructure costs largely consist of electrical distribution, ventilation, and dewatering system costs.

Table 21.5 Underground infrastructure cost estimate

Description	Total cost (US$M)
Mine dewatering	5.6
Service water	Included in mine development rates
Electrical distribution	7.3
Workshop, magazine, and refuge stations	1.6
Communications	0.9
Primary fans and facilities	4.9
Indirect costs and contingency	9.6
Sustaining capital	3.3
Total	**33.0**

21.2 Underground mine operating cost

AMC has used benchmark operating costs for mining from its underground database of mining costs. Benchmark costs indicate that for a production rate of 3.6 Mtpa (3.3 Mtonnes pa), the mine operating cost averages approximately US$31/t of mineralization. The benchmark data includes all mining methods, however approximately a third of the data represents SLOS or Longhole stope data. The database has costs for backfill included in some of the operations but not all.

AMC considers a cost of US$31/t of mineralization to be a reasonable estimate for the production rate in 60 ft and 100 ft high longhole stopes, with an additional cost of US$4.35/t added for paste backfill. The total mining cost for 60 ft and 100 ft high longhole stopes was therefore assumed to be US$35.35/t. For the 20 ft high stopes, a total cost of $60/t was used, including backfill costs. On a weighted average basis, the total mine operating cost is $38.02/t. Table 21.6 shows the cost per ton for the different stope heights.

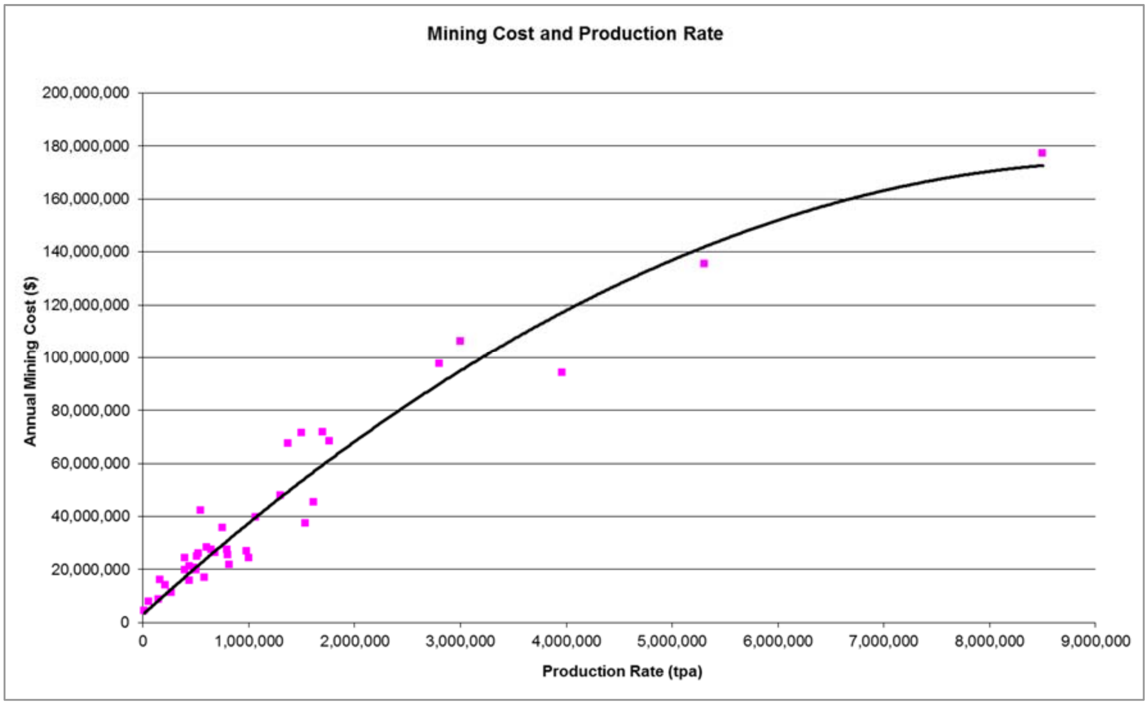
Table 21.6 Mine operating cost by stope height

Description	(US$) cost per ton	LOM tons	(US$) LOM cost
Cost for 20 ft stopes	60.00	10,464,639	627,878,327
Cost for 60 ft stopes	35.35	35,312,995	1,248,314,370
Cost for 100 ft stopes	35.35	50,892,912	1,799,064,430
Total	38.02	96,670,545	3,675,257,128

Totals do not necessarily equal the sum of the components due to rounding.

The benchmark data for mining costs is provided in Table 21.7.

Figure 21.1 Benchmark data for mining costs



AMC validated the operating costs based on labour schedules and labour numbers and then split into cost categories for North American costs for a mining operation. The split by cost area is summarized in Table 21.8 and Table 21.9. The validated costs are within 5% of the benchmark data, it was decided to use the more conservative mining costs of US$35.35/t and US$60/t. The backfill costs were determined seperately and are based on costs for labour, cement and consumables from local vendors.

Table 21.7 Mine operating cost by area (60 foot and 100 ft high stopes)

Item	Percentage of total	Total (US$) / ton of mineralized material
Labour	33%	11.69
Power	17%	6.02
Consumables	25%	8.86
Services	7%	2.48
Other	6%	1.95
Backfill	12%	4.35
Total	**100%**	**35.35**

Totals do not necessarily equal the sum of the components due to rounding.

Table 21.8 Mine operating cost by area (20 ft high stopes)

Item	Percentage of total	Total (US$) / ton of mineralized material
Labour	45%	27.00
Power	13%	7.80
Consumables	19%	11.55
Services	10%	6.00
Other	6%	3.30
Backfill	7%	4.35
Total	**100%**	**60.00**

Totals do not necessarily equal the sum of the components due to rounding.

21.3 Processing capital cost estimate

A summary of the initial pre-production capital costs for the process plant and associated infrastructure is shown in Table 21.9. This table includes direct costs, indirect costs, and a 25% contingency. This capital cost was based on equipment cost and multiplied by factors for installation. An additional sustaining cost of US$12.2M was noted below this table for additional conveyors that must be purchased in year 4 of the mine operation.

Table 21.9 Summary process plant - initial capital cost for 10,000 TPD process plant

Direct costs	**US$M**
Process plant	
Area 10 - Crushing, conveying, stockpile	2.8
Area 15 – Grinding	15.3
Area 25 - Lead flotation	8.2
Area 26 - Zinc flotation	9.4
Area 27 – Multiplexor	1.2
Area 30 - Concentrate thickening and filtration	13.4
Area 60 - Tailings thickening and filtration	14.2
Area 65 – Reagents	0.6
Area 66 - Water distribution on-site	1.3
Area 67 – Plant Air	0.9
Installed equipment cost	**67.2**
Site development	
General site development	4.8
Process and overland piping on site	3.4
Buildings (process and non-process)	10.6
Electrical power distribution on site	9.1
Site development cost	**27.8**
Infrastructure	**US$M**
Water source and distribution to site	3.0
Infrastructure cost	**3.0**
Total direct costs	**98.0**
Plant indirect costs	
EPCM	10.8
Construction indirect costs incl:	4.9
Spare parts	2.4
Initial fill and reagents	1.0
Equipment insurance and freight cost	3.4
Total indirect costs	**22.4**
Total direct and indirect	**120.5**
Contingency - 25%	**30.1**
Total	**150.6**

Totals do not necessarily equal the sum of the components due to rounding.

A sustaining cost of US$12.0M is also requred one year prior to operating at 10 ktpd to purchase additional tailings filters and additional conveyors to transport tailings to the tailings storage facility. The sustaining cost total includes $8.5M in direct costs, $1.3M in indirect costs, and $2.4M of contingency costs.

21.3.1 Direct costs

The direct capital costs were based on the following list of documents prepared by SGS:

- Design criteria
- Equipment list
- Mining cost service source quote data for minor equipment
- SGS engineering equipment database for recent similar projects
- Budget quotations from vendors for major equipment
- Miscellaneous: Engineering drawings performed by SGS

The direct costs exhibited in this estimate include, but are not limited to, labour, equipment and materials for the detailed construction activities set forth below:

21.3.2 Equipment costs

An equipment list was developed and incorporated into the cost estimate. The estimate for equipment was developed from the following sources:

- Written or emailed budgetary estimates from vendors for major equipment.
- Historical data and budget costs from recent similar projects for miscellaneous equipment.

The cost for “Installed equipment” was estimated using a factor of forty percent (40%) of purchased equipment costs. This factor reflects typical costs to install equipment and covers labour, concrete foundations, steel, and other services and construction materials associated with equipment foundations, erection, and placement.

21.3.3 Process piping

Process piping costs include materials and installation of all piping within the process plant. The initial process piping cost was estimated using a seven percent (7%) factor of purchased equipment costs. The factored costs were based on the size of the plant, as well as the distribution of piping within the plant area.

21.3.4 Electrical main power supply

Main electrical power supply costs include utility transmission line costs for high voltage circuit protection, power transformers, poles, conductors, insulators, labour, and other miscellaneous costs associated with utility transmission and connections for bringing power to site. No additional costs have been estimated for future installations under the assumption that all main electrical supply be installed during the initial construction, and sized to accommodate the future equipment. The electrical cost was developed from budgetary estimates from the utility and historical data from recent similar projects. The cost also includes a non-refundable tax gross up estimated by the utility at twenty-two percent (22%) of the electrical capital cost for installing the utility transmission line. The utility will require the following main electrical power supply items for this project:

- Utility Connection Switchyard (138 kV).
- Utility Termination Facility – 138 kV circuit protection and connection to mine substation.
- Utility Transmission Line (138 kV) – Overhead pole-line, including permitting and right-of-way.

21.3.5 Electrical distribution

Electrical distribution costs include transformation and service, wiring, cable tray, instrumentation, lighting and grounding within the process plant. The initial electrical cost was estimated using seventeen percent (17%) of purchased equipment costs. The factors were selected based on preliminary equipment power requirements of 36 MW (plus 10% contingency), and latest National Electric Code (NEC) standards. The project will require the following electrical power distribution items:

- Main substation (37.5 MVA transformers, circuit protection, switchgear).
- 24.9 kV distribution lines (on site) – underground duct-bank and overhead line.

- Pad-mounted distribution transformers (process plant and ancillary buildings).
- Pad / Pole-mounted transformers (remote facilities).
- Medium voltage (4160 V) switchgear.
- Low voltage (480 V) motor control centre.
- Back-up diesel generators.

21.3.6 Site development for process plant and associated infrastructure

General site development costs include excavations, backfills, grading, roads, and fencing. The initial construction site development cost was estimated using a ten percent (10%) factor of the purchased plant equipment cost. The factor was selected based on the mountainous nature of the proposed project site, and the type of native soils in the area. The project will require development at the following major locations:

- On site access roads
- Primary crusher area
- Overland conveyor and stockpile area
- Process plant areas

Process and overland piping on site is included and was estimated based on a seven percent (7%) of the purchased plant equipment cost.

Building costs include materials, labour, and other miscellaneous costs associated with erecting covered structures within the project site. The initial construction building cost was estimated using a 22% factor of the purchased plant equipment cost. The factor was selected to reflect the projected costs of the buildings based on building type and square footage. The project will require the following buildings:

- Grinding and flotation area
- Control rooms and offices
- Tailings filter area
- Mill area offices
- Mill area change rooms
- Reagent storage area
- Warehouse
- Laboratory

21.3.7 Access road to project site

Harshaw road is proposed to be a paved, two lanes, all weather access road. Approximately 6 miles (9.7 km) of this road are paved and the remaining 3 miles (4.8 km) of Harshaw road is unpaved. Based upon the proposed increase in traffic on this road due in large part to mine worker commutes and concentrate delivery trucks, it was decided that Harshaw road be paved to the project site. Costs to pave the 3 miles (4.8 km) of dirt road were included in the cost estimate. In addition, upgrades to the remaining 6 miles (9.7 km) and required culverts and bridges were also included in the estimate. The mine property access roads were included in the site development costs.

21.3.8 Fresh water source and distribution to head tank on site

The Project site is located at an elevation of 5,195 ft above sea level. There are existing wells on the property however the current capacity is not adequate for the mine and associated process facilities. Additional water sources are being evaluated within the project site by Clear Creek. For the PEA it was assumed an adequate water resource from ground water wells is available on the mine property. The water system included pumping and piping to distribute approximately 650 gpm (2,460 LPM) on a continuous basis. Fresh, potable and process water pumps, storage tanks and distribution pipelines were included in the capital cost estimate. Sourcing the fresh water and drilling and casing wells is included in the Clear Creek cost estimate.

21.3.9 Indirect costs

Certain indirect costs exhibited in this estimate include, but are not limited to, labour, equipment, and materials for the detailed activities set forth below:

- **EPCM** for the process facilities and associated infrastructure was estimated using 11% of the direct costs and includes the following:
  - Feasibility study
  - Detailed engineering
  - Procurement
  - Construction management
- **Construction indirect costs** for the initial construction and mill expansion were estimated using a five percent (5%) factor of the total direct costs and includes:
  - Construction supervision
  - Equipment rental
  - Field office expenses
  - Mobilization / demobilization
  - Consumables
- **Spare parts costs** were estimated using a five percent (5%) factor of the installed plant equipment cost.
- **Initial fill and reagents costs** were estimated using a one percent (1%) factor of the installed plant equipment costs.
- **Equipment insurance and freight costs** were estimated using a seven percent (7%) factor of the installed plant equipment costs.

21.3.10 Process plant contingency and accuracy

The SGS crushing and process plant portion of the cost estimate includes a 25% contingency for project unknowns and identified risks. Contingency is a necessary part of the cost estimate and is based on the fact that less than three percent (< 3%) of the engineering is completed to date. SGS believes the estimated contingency amount will be spent during the construction period of the plant site and associated infrastructure for identified risks and unknown items.

While SGS has not performed a statistical analysis of the crushing plant and process plant accuracy of the capital cost estimate, SGS has a high confidence, based on previous experience with similar projects, that the accuracy of the process portion of the PEA capital cost estimate will end up between minus ten percent and plus thirty percent (-10 / +30%) of the SGS capital cost estimate.

21.3.11 Exclusions from process plant cost estimate

SGS has excluded the following cost items from the process plant estimate and assume these are included in other sections of the report:

- Owners costs
- Geotechnical
- Mining
- Reclamation and closure
- Metallurgical testing
- Property acquisition
- Permitting
- Environmental
- Permits, royalties, and licenses
- Taxes, duty and import fees

- Local sales and import taxes
- Hazardous waste removal
- Other consultants

21.4 Processing operating cost estimate

Annual and unit process operating cost estimates for a 10,000 stpd (9,072 tonnes per day) milling operation are summarized in the following Table 21.10. Support tables for the cost estimates are shown in Table 21.11 through Table 21.17.

Table 21.10 Summary of plant operating cost by cost item

Item	Annual	Cost	
	Cost (US$)	(US$/short ton)	(US$/metric tonne)
Power	8,696,021	2.42	2.66
Labor	8,749,405	2.43	2.68
Reagents	9,454,022	2.63	2.89
Grinding media	4,960,207	1.38	1.52
Repair materials and operating supplies	1,454,901	0.40	0.45
Liners and wear materials	2,736,758	0.76	0.84
Total	36,051,314	10.01	11.04

The detailed plant power consumption estimate is based on the installed power with estimates of the operating power draft and operating time, and power unit cost of US$ 0.8/kWh. The process power consumption and power cost calculation are summarized in Table 21.11 and Table 21.12 respectively.

Table 21.11 Plant power consumption summary

Area	kWh/tonne
Area 10 - Primary crushing	0.65
Area 15 - Grinding	18.04
Area 25 - Lead flotation	2.78
Area 26 - Zinc flotation	3.46
Area 27 - Multiplexer	0.03
Area 30 - Thickening and filtration	1.76
Area 60 - Tailings thickening and filtration	4.09
Area 65 - Reagents	0.09
Area 66 - Water	1.28
Area 67 - Plant air	1.16
Total	**33.28**

Table 21.12 Plant power cost

Usage	Value
kWh per tonne	33.28
Power cost, US$ per kWh	0.08
Power cost, US$ per tonne	2.66
Power cost, US$ per year	8,696,021

The labour cost estimate for mill operations is shown in Table 21.13. The labour rates and burden are based on the rates for a similar mill operation.

Table 21.13 Labour cost

Function	Per crew	Total	Total hrs/yr	Rate (US$)	Total (US$)
Operations shift crews (4 crews req'd)					
Control room operator	1	4	8,760	42.00	367,920
Crusher operator	1	4	8,760	39.20	343,392
Grinding operator	1	4	8,760	36.40	318,864
Zinc flotation operator	1	4	8,760	35.00	306,600
Lead flotation operator	1	4	8,760	35.00	306,600
Filter operator (concentrate)	1	4	8,760	35.00	306,600
Tailings / water operator	1	4	8,760	35.00	306,600
Training / vacation relief	1	4	8,760	28.70	251,412
Labor	3	12	8,760	28.70	251,412
Zinc conc handling (loading trucks)	1	4	8,760	28.70	251,412
Lead conc handling (loading trucks)	1	4	8,760	28.70	251,412
Sub total		**52**			**2,759,400**
Operations day crew					
Reagent mixing (10 and 4)	2	2	4,160	28.70	119,392
Tailings storage operation	4	4	8,320	28.70	238,784
General cleanup; ball charging	3	3	6,240	28.70	179,088
Sub total		**9**			**537,264**
Maintenance					
Mechanics	10	10	20,800	41.30	859,040
Shift electrician	1	4	8,760	42.42	371,599
Day electrician / inst. Tech	6	6	12,480	42.42	529,402
Laborers	2	2	4,160	28.70	119,392
Sub total		**22**			**1,879,433**
Technical					
Shift sample prep / sampler	1	4	8,760	$28.00	$245,280
Day sample prep	2	2	4,160	$28.00	$116,480
Assayers (day only)	4	4	8,320	$36.40	$302,848
Sub total		**10**			**$664,608**
Salaried personnel					
Mill superintendent	1			142,000	142,000
General foreman	1			130,600	130,600
Maintenance foreman	3			133,500	400,500
Plant foreman	3			125,000	375,000
Senior metallurgist	1			139,200	139,200
Metallurgist	3			127,800	383,400
Process technician	3			96,600	289,800
Instrument technician	3			99,400	298,200
Process foreman	6			125,000	750,000
Sub total	**24**				**2,908,700**
Grand total		**117**			**8,749,405**

Reagent cost estimates are shown in Table 21.14. The reagent consumption rates are based on SGS Lakefield metallurgical test work data in 2017.

Table 21.14 Reagent cost

Reagents	Usage	Quantity	Reagent cost	Cost	Cost
	kg/mt of mineralized material	kg/year	$/kg	$/year	$/tonne
Sodium cyanide (NaCN)	0.100	326,592	3.02	985,001	0.30
Zinc sulfate ($ZnSO_4$)	0.300	979,776	1.24	1,214,922	0.37
Aerofloat 242	0.040	130,637	6.66	870,041	0.27
Carboxymethyl cellulose (CMC)	0.010	32,659	2.50	81,648	0.03
Copper sulfate ($CuSO_4$)	0.425	1,388,016	2.72	3,775,404	1.16
Sodium isopropyl xanthate (SIPX)	0.080	261,274	4.30	1,123,476	0.34
Methyl isobutyl carbinol (MIBC)	0.073	238,412	3.75	894,046	0.27
Flocculant	0.040	130,637	3.90	509,484	0.16
Total				**9,454,022**	**2.89**

The grinding media and liner and wear material cost estimates are provided in Table 21.15 Table 21.17. The consumption estimates are based on abrasion index (Ai).

Table 21.15 Wear material operating cost estimates

	Bond wear equations	Usage, kg/kWh	Power consumption kWh/tonne	Usage, kg/tonne	Cost, $/kg	Cost, $/tonne	Cost, $/year
Crusher liners	=(Ai + 0.22)/11	0.020	0.186	0.004	5.71	0.022	71,091
SAG mill liners				0.069	5.71	0.394	1,287,595
Ball mill liners	=0.026 x (Ai - 0.015)^0.3	0.008	7.9	0.062	5.71	0.356	1,162,514
Regrind mill liners	=0.026 x (Ai - 0.015)^0.3	0.008	1.948	0.015	5.71	0.088	286,649
Total wear material						**0.838**	**2,736,758**

Table 21.16 Grinding media operating cost estimates

	Bond wear equations	Usage, kg/kWh	Power consumption kWh/tonne	Usage, kg/tonne	Cost, $/kg	Cost, $/tonne	Cost, $/year
SAG mill balls				0.786	0.85	0.668	2,183,023
Ball mill balls	=0.35 x (Ai - 0.015)^(1/3)	0.102	7.900	0.803	0.85	0.682	2,227,849
Regrind mill balls	=0.35 x (Ai - 0.015)^(1/3)	0.102	1.948	0.198	0.85	0.168	549,335
Total grinding media						**1.519**	**4,960,207**

The repair materials and operating supplies is estimated using empirical factor based on total equipment installed cost, SGS recommends to use 3.0 percent for this 10,000 ton per day plant.

Table 21.17 Repair materials and operating supplies

Item	Value
New equipment capital estimate	US$ 48,496,685
Repair materials and supplies (percentage of equip)	3.00%
Annual maintenance cost	US$ 1,454,901
Cost per tonne	US$ 0.45

21.5 Tailings storage facility capital cost

The TSF capital cost estimation was developed for the Trench Camp starter (6 year production), Trench Camp ultimate (Table 21.18) and Hermosa TSF (Table 21.19). Capital costs were generated using unit rates assuming contractor work for major construction components. Unit rates were developed based on (1) equipment rental rates, prevailing wages and fringes and estimated fuel prices, (2) cost data from previous similar projects, and

(3) vendor supplied quotes. The costs are PEA level with an inherent accuracy of +35% to -15%. See Table 21.18 and Table 21.19 for the Trench Camp and Hermosa TSF capital cost estimate summaries, respectively.

21.5.1 Trench Camp dry stack TSF capital cost

Table 21.18 Trench Camp dry stack TSF capital cost estimate summary

Construction item	Cost (starter) (US$M)	Cost (ultimate) (US$M)
Mobilization / demobilization	$0.85	$1.48
Site preparation / remove existing tailings piles	$3.86	$5.33
Rock excavation	$1.25	$2.88
Perimeter road construction	$2.02	$4.47
Low permeability soil layer	$0.59	$1.49
Geomembrane liner	$0.98	$2.44
Protective layer	$1.55	$3.90
TSF underdrain collection system	$2.60	$2.80
Underdrain collection pond (including reclaim system)	$0.68	$0.68
External stormwater management – flow through drain	$0.60	$0.60
Contingency (20%)	$3.00	$5.22
Estimated direct costs (including contingency)	$17.98	$31.30
Estimated indirect costs	$2.85	$4.96
Total cost	**$20.82**	**$36.25**

21.5.2 Hermosa dry stack TSF

Table 21.19 Hermosa dry stack TSF capital cost estimate summary

Construction item	Cost (ultimate) (US$M)
Mobilization / demobilization	$0.52
Site preparation	$0.23
Rock excavation	$0.31
Perimeter road construction	$3.31
Low permeability soil layer	$0.39
Geomembrane liner	$0.65
Protective layer	$1.03
TSF underdrain collection system	$0.87
Underdrain collection pond (including reclaim system)	$0.75
External stormwater management – flow through drain	$1.05
Contingency (20%)	$1.82
Estimated direct costs (including contingency)	$10.93
Estimated indirect costs	$1.73
Total cost	**$12.66**

21.5.3 Cost estimate basis for major construction items

- Mobilization / demobilization
  - Unit rate
    - Earthworks contractor is assumed to be 5% of total direct cost.
    - Geosynthetics contractor is assumed to be 1% of total direct cost.
- Site preparation / relocating existing tailings piles

- Measurement
  - Assumed to be approximately 18 in to 24 in (457 mm – 610 mm) thick over the disturbance footprint of the TSF.
  - Volume based on assumed base topography of existing tailings.
- Unit rate
  - Clearing and stripping based on scraper and dozer fleet with material located to stockpile within 1 mile (1,609 m).
  - Existing tailings relocation based on haul, place and compaction of tailings using trucks, a loader, dozers, a compactor and a tractor with a disc considering an average haul of approximately 0.75 mile (1,208 m) round trip.

- Rock excavation
  - Measurement
    - Rock excavation assumed to be approximately 25% of cut to fill excavation value.
  - Unit rate
    - Drill and blast based on work completed by a contractor.
- Perimeter road construction
  - Measurement
    - Engineered fill volume based on a crest width of 25 ft (7.6 m) with 2.5 H:1 V upstream and 2.0 H:1 V downstream side slopes.
    - Wearing course volume is based on 6 in (152 mm) thick across the width of the perimeter road.
  - Unit rate
    - Engineered fill material sourced as a cut to fill from the basin and plant site areas in combination with non-PAG mine development rock.
    - Wearing course material is produced by a crushing and screening operation and placed using trucks and dozers.
- Low permeability soil liner
  - Measurement
    - Low permeability soil to be 12 in (305 mm) thick (prescriptive BADCT requirement) over the basin and upstream slope of the perimeter road.
  - Unit rate
    - Assumes borrow source located on site with material placed using dozers, scrapers and compactor.
    - Assumes low permeability surface is prepared for geosynthetics placement.
- Geomembrane liner
  - Measurement
    - Area of basin and upstream slope of the perimeter road.
  - Unit rate
    - Supply and install for 60 mil class (1.5 mm) HDPE geomembrane (prescriptive BADCT requirement).
    - 10 percent increase for supply to account for wastage and overlap.
- Protective layer
  - Measurement
    - Protective layer to be 18 in (457 mm) thick (prescriptive BADCT requirement) over the basin area and upstream slope of the perimeter road.
  - Unit rate

        - Assumes material is produced from non-PAG mine development rock using a crushing and screen operation and placed using trucks and dozers.
- TSF underdrain collection system
    - Measurement
        - Underdrain collection pipe - linear foot measurement.
        - Concrete encasement - linear foot measurement.
    - Unit rate
        - Supply and install
        - Pipe fittings were estimated at 10% of pipe supply and install cost.
        - Supply and install of non-woven geotextile.
        - Cross sectional area of select gravel (crushing and screening operation).
        - Cross sectional area of reinforced concrete.
- Underdrain collection pond
    - Measurement
        - Volume based on a crest width of 25 ft (7.6 m) with 2.5 H:1 V upstream and 2.0 H:1 V downstream side slopes.
        - Area of geomembrane.
        - Reclaim pipe linear foot measurement.
        - Pump, support and instrumentation.
    - Unit rate
        - Material sourced as a cut to fill from the basin area in combination with non-PAG mine development rock.
        - Supply and install for 80 mil (2.0 mm) HDPE geomembrane and geonet.
        - 10% increase for supply of geomembrane and geonet to account for wastage and overlap.
        - Supply and install of pump system and pipe.
- External storm water management
    - Measurement
        - Flow through drain linear foot measurement.
    - Unit rate
        - Based on supply and install of flow through drain pipe as well as cross sectional area of pipe bedding and pipe backfill.
- Direct costs
    - Summation of costs listed above.
- Indirect costs
    - Engineering cost is assumed to be 5% of direct costs.
    - Construction management is assumed to be 6% of direct costs.
    - Quality assurance / quality control (QA/QC) is assumed to be 7% of direct costs.
    - Surveying is assumed to be 1% of direct costs.
- Filtered dry stack tailings (operating cost)
    - Measurement
        - Volume based on tailings capacity of dry stack TSF.
    - Unit rate
        - Based on cost to spread and compact tailings after placed by conveyor using a dozer, compactor and tractor with disc (conveyor cost captured by others).
- Mine development rock (includes rock armoring) (operating cost).

- Measurement
  - Volume based on mine development rock capacity of dry stack TSF including the 5.3 ft (1.6 m) thick rock armour exterior.
- Unit Rate
  - Based on cost to haul and place of mine development rock using trucks and dozers.

- Growth media cover (reclaim) (operating cost)
  - Measurement
    - Volume based on 2 ft (600 mm) depth of growth media over dry stack surface.
  - Unit rate
    - contractor cost Based on to haul, place and re-vegetate growth media from a stockpile using trucks and dozers.

21.6 Tailings storage facility operating cost

Assuming dry stack tailings are placed by AMI, an operating cost of US$0.34 per ton of mineralized material was estimated for spreading and compacting the filtered tailings (US$1.00 per ton of placed material). The operating cost assumes a medium size dozer, vibratory compactor and tractor with disc would be utilized to place the tailings from the conveyor system. Additional operating costs include placement of mine development rock. Assuming mine development rock is hauled and placed by AMI, an overall operating cost of US$0.083/ton is estimated to haul and place the development rock with articulated haul trucks, medium size dozer and vibratory compactor (US$2.20 per cubic yard of placed material).

Capital expenditure considerations should include an increase for mine haul trucks due to increased haul distances but spreading and compacting equipment could be covered by equipment purchased for tailings placement. A decrease in cost for placement of mine development rock of approximately 30% will be realized for starter construction as the haul distance is reduced. Last, it is assumed that growth media will be hauled, placed and hydroseeded by AMI at an approximate operating cost of US$0.016 per ton of mineralized material using 40 ton articulated haul trucks, medium size dozer and loader (US$3.60 per cubic yard of placed material). See Table 21.20 and Table 21.21 for dry stack TSF operational unit cost summary and operational total cost summary.

Table 21.20 Dry stack TSF operational unit cost summary

Construction item	Mine placed (US$/ton)
Tailings placement (spread and compact)	0.340
Mine development rock (haul and place)	0.083
Growth media cover (haul, place and hydroseed)	0.016
Contingency	0.089
Total operational cost	0.53

Table 21.21 Dry stack TSF operational total cost summary

Construction item	Trench camp cost (starter) (US$M)	Trench camp cost (ultimate) (US$M)	Hermosa cost (US$M)
Tailings placement	$3.37	$26.66	$6.24
Mine development rock (including rock armoring)	$2.50	$7.18	$0.83
Growth media cover	$0.27	$1.30	$0.28
Estimated operating costs (not including contingency)	$6.14	$35.14	$7.35
Total cost (including 20% contingency)	$7.36	$42.17	$8.81

21.7 Total capital cost estimate for the mine

The total LOM capital cost estimate for the mine is provided in Table 21.22. Pre-production capital (capital spent prior to Year 4) as well as the project capital (total capital less pre-production capital) is also provided.

Table 21.22 Total mine capital cost estimate

Item	Total (US$)	Pre-production capital (US$)		
		Year 1	Year 2	Year 3
Underground Development	582,498,200	5,670,194	7,560,259	35,052,409
Mine Equipment	138,800,000	10,000,000	16,000,000	18,000,000
Shaft	176,220,000	42,105,000	42,105,000	42,105,000
Backfill plant	10,000,000			10,000,000
Water to site	3,000,000	3,000,000		
Power	42,016,707	12,039,987	20,919,976	9,056,744
Roads	16,090,240	4,022,560	8,045,120	4,022,560
TSF - Trench and Hermosa	43,320,000	8,913,000	8,913,000	1,000,000
Processing	106,533,836		32,674,544	65,358,892
UG Infrastructure	23,441,825	3,365,304	3,365,304	3,365,304
EPCM	30,678,575	10,226,192	10,226,192	10,226,192
Owners Cost	900,000	300,000	300,000	300,000
Capitalized opex	6,724,522			6,724,522
Contingency	63,995,607	21,331,869	21,331,869	21,331,869
Total	**1,244,219,512**	**120,974,107**	**171,441,264**	**226,543,492**
Pre-production capital				**518,958,863**
Sustaining capital				**725,260,650**

Totals do not necessarily equal the sum of the components due to rounding.

21.8 General and administration operating cost estimate

Cost estimates for General and administration (G&A) were provided by AZ at a unit cost of US$2.00/ton. G&A costs are those that support the overall management and operation of the business and include rent, utilities, insurance and managerial, procurement, environment, safety and administrative salaries.

21.9 Total mine operating cost

The total operating cost is estimated to be US$50.56/t of mineralized material for the mine. The total operating cost includes mining (US$38.02/t of mineralized material), processing cost (US$10.01/t of mineralized material), material placement at the TSF (US$0.53/t of mineralized material) and General and Administration cost (US$2/t mineralized material).

Table 21.23 summarizes the operating costs.

Table 21.23 Total operating cost

Item	Units	Cost
Mine operating cost	$US/ton	38.02
Processing cost	$US/ton	10.01
Tailings placement cost	$US/ton	0.53
General and administration	$US/ton	2.00
Total	$US/ton	50.56

22 Economic analysis

22.1 Assumptions

All currency is in US dollars (US$) unless otherwise stated. The cost estimate was prepared with a base date of year 1 and does not include any escalation beyond this date. For net present value (NPV) estimation, all costs and revenues are discounted at 8% from the base date. Metal prices were selected after discussion with AZ and referencing current markets and forecasts in the public domain. A regular corporate tax rate of 21% for federal tax and 4.9% for Arizona State tax is applied as the mining income will be earned in Arizona, USA. It is assumed that 3% of the NSR value would be the royalties to be paid.

22.2 Economic analysis

AMC conducted a high level economic assessment of the conceptual underground operation of the Taylor Deposit. The underground mine is projected to generate approximately US$2,406M pre-tax NPV and US$1,979M post-tax NPV at 8% discount rate, pre-tax IRR of 54% and post-tax IRR of 48%. Project capital is estimated at US$1,244M with a payback period of 1.5 years (discounted pre-tax cash flow from start of production in Year 4). Key assumptions and results of the underground mine economics are provided in the Table 22.1. The LOM production schedule, average metal grades, recovered metal, and cash flow forecast is shown in Table 22.2.

The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

Table 22.1 Taylor Deposit underground mine – key economic assumptions and results

Arizona Taylor Deposit	Unit	Value
Total mineralized rock	kton	96,671
Total waste production	kton	11,992
Zinc grade[1]	%	4.01%
Lead grade[1]	%	4.34%
Silver grade[1]	oz/ton	2.22
Zinc recovery[1]	%	92.7%
Lead recovery[1]	%	95.4%
Silver recovery[1]	%	92.4%
Zinc price	US$/lb	1.10
Lead price	US$/lb	1.00
Silver price	US$/oz	20.00
Zinc payable[2]	%	85%
Lead payable[2]	%	95%
Silver payable - Pb con[2]	%	95%
Silver payable - Zn con[2]	%	70%
Payable Zn metal	klbs	6,112,710
Payable Pb metal	klbs	7,608,117
Payable Ag metal	koz	162,566
Revenue split by commodity	Zinc	38%
Revenue split by commodity	Lead	43%
Revenue split by commodity	Silver	19%
Total revenue	US$ ($ 000)	17,583,425
Capital costs	US$ ($ 000)	1,244,220
Operating costs (total)[3]	US$ ($ 000)	4,880,781
Mine operating costs[4]	US$/ton	38.02
Process and tails storage operating costs	US$/ton	10.54

Arizona Taylor Deposit	Unit	Value
Operating costs (total)[3]	US$/ton	50.56
Operating cash cost (ZnEq)	US$/lb ZnEq	0.57
C1 Zinc co-product costs (8)	US$/lb	0.49
C1 Lead co-product costs (8)	US$/lb	0.37
Total all-in sustaining cost (ZnEq)	US$/lb ZnEq	0.61
Payback Period pre tax[5]	(Yrs)	1.51
Cumulative net cash flow[6]	US$ ($ 000)	7,260,841
Pre-tax NPV[7]	US$ ($ 000)	2,405,888
Pre-tax IRR	%	54%
Post-tax NPV[7]	US$ ($ 000)	1,979,101
Post-tax IRR	%	48%

1. LOM average
2. Overall payable % includes treatment, transport, refining costs and selling costs
3. Includes mine operating costs, milling, and mine G&A
4. Underground mining costs only
5. Values are pre-tax and discounted at 8%, from production start date Year 4
6. Pre-tax and undiscounted
7. At 8% discount rate.
8. Silver treated as a by-product.
9. The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the results of the PEA will be realized.

Table 22.2 Taylor Deposit production and cash flow forecast – year 1 to 14

Mine production	**Unit/yr**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Total mined - mineralized rock	kton	-	-	190	1,582	2,636	3,604	3,580	3,601	3,596	3,597	3,592	3,586	3,660	3,598
Total mined - waste	kton	116	154	734	803	952	723	735	672	610	617	601	544	899	645
Total waste development - lateral	m	1,260	1,680	6,714	6,436	9,551	6,708	7,112	7,328	6,650	6,731	6,432	5,937	9,293	6,262
Total waste development - vertical	m	-	-	968	1,746	625	885	678	-	-	-	92	-	383	585
Total mill feed	kton	-	-	190	1,582	2,636	3,604	3,580	3,601	3,596	3,597	3,592	3,586	3,660	3,598
ZnEq	%	0.00	0.00	14.66	19.97	20.79	19.77	17.49	11.17	9.44	9.39	9.50	9.35	9.54	9.66
Ag	oz/ton	0.00	0.00	2.38	3.43	3.85	3.88	3.60	2.30	1.81	1.76	1.72	1.63	1.79	1.81
Pb	%	0.00	0.00	5.43	7.25	7.42	7.31	6.81	4.44	3.69	3.69	3.68	3.61	3.69	3.68
Zn	%	0.00	0.00	7.01	9.50	9.75	8.81	7.30	4.56	4.05	4.06	4.21	4.22	4.17	4.28
Recoveries															
Overall Ag recoveries	%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%
Overall Pb recoveries	%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%
Overall Zn recoveries	%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%
Total payable metal															
Ag	koz	-	-	-	4,254	7,472	10,408	9,693	6,235	4,841	4,684	4,530	4,254	4,836	4,825
Pb	klb	-	-	-	226,712	354,655	477,748	441,611	289,937	240,475	240,352	239,855	234,386	244,912	239,975
Zn	klb	-	-	-	257,881	404,911	500,555	411,746	258,814	229,480	229,851	238,175	238,540	240,520	242,674
Overall Ag payable in Zn Con	%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
Overall Ag payable in Pb Con	%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Overall Pb payable	%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Overall Zn payable	%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
Total net revenue	US $ '000	-	-	-	595,460	949,507	1,236,517	1,088,399	699,342	589,728	586,878	592,445	581,863	606,204	603,410
Operating costs															
Mining	US $ '000	-	-	-	55,919	93,183	127,408	126,549	127,295	127,132	127,143	126,986	126,774	129,376	127,182
Processing & tailing storage	US $ '000	-	-		18,678	27,783	37,988	37,732	37,954	37,906	37,909	37,862	37,799	38,575	37,921
General & administration	US $ '000	-	-		3,544	5,272	7,208	7,160	7,202	7,193	7,193	7,185	7,172	7,320	7,196
Smelter costs	US $ '000	-	-	-	131,161	206,478	264,418	228,523	146,296	125,648	125,581	127,933	126,657	130,004	129,665
Royalty	US $ '000	-	-	-	13,929	22,291	29,163	25,796	16,591	13,922	13,839	13,935	13,656	14,286	14,212
Mine development	US $ '000	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Other costs	US $ '000	-	-	-	500	500	500	500	500	500	500	500	500	500	500
Severance tax	US $ '000	-	-	-	3,006	5,901	8,495	7,335	3,782	2,747	2,708	2,898	2,941	2,951	3,119
Salvage value	US $ '000	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reclamation & closure	US $ '000	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total operating cost	US $ '000	-	-	-	226,737	361,407	475,180	433,594	339,621	315,048	314,873	317,299	315,500	323,012	319,795
Capital costs															
Project capital	US $ '000	120,974	171,441	226,543											
Sustaining capital	US $ '000				104,662	51,121	42,177	45,390	44,874	39,126	34,640	32,651	31,867	52,934	32,603
Total capital cost	US $ '000	120,974	171,441	226,543	104,662	51,121	42,177	45,390	44,874	39,126	34,640	32,651	31,867	52,934	32,603
Undiscounted cash flows (pre-tax)	US $ '000	(120,974)	(171,441)	(229,543)	252,757	525,304	704,264	597,560	312,305	235,803	237,725	242,803	235,110	230,437	251,034
Income tax	US $ '000	-	-	-	7,982	55,408	128,263	109,365	49,987	32,943	32,285	36,133	37,576	36,872	40,335
Undiscounted cash flows (post-tax)	US $ '000	(120,974)	(171,441)	(229,543)	244,775	469,896	576,001	488,195	262,318	202,859	205,440	206,670	197,534	193,565	210,699
Discounted cash flows (pre-tax)	US $ '000	(112,013)	(146,983)	(182,219)	185,784	357,513	443,805	348,670	168,728	117,960	110,113	104,134	93,365	84,731	85,467
Discounted cash flows (post-tax)	US $ '000	(112,013)	(146,983)	(182,219)	179,917	319,803	362,978	284,857	141,722	101,480	95,159	88,637	78,444	71,173	71,735

Table 22.3 Taylor Deposit production and cash flow forecast – year 15 to 28

Mine production	**Unit/yr**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**
Total mined - mineralized rock	kton	3,598	3,548	3,597	3,646	3,596	3,596	3,598	3,598	3,596	3,596	3,597	3,596	3,596	3,598
Total mined - waste	kton	576	339	203	-	298	298	298	266	213	176	-	327	194	-
Total waste development - lateral	m	6,042	3,694	2,210	-	3,253	3,250	3,253	2,898	2,320	1,919	-	3,571	2,112	-
Total waste development - vertical	m	183	-	-	-	-	-	-	-	-	-	-	-	-	-
Total mill feed	kton	3,598	3,548	3,597	3,646	3,596	3,596	3,598	3,598	3,596	3,596	3,597	3,596	3,596	3,598
ZnEq	%	8.45	9.67	9.18	8.91	9.55	8.73	9.69	9.85	9.22	9.24	9.12	9.17	8.97	8.10
Ag	oz/ton	1.61	1.88	1.88	1.97	2.05	2.11	2.18	2.34	2.13	2.30	2.30	2.35	2.26	1.94
Pb	%	3.41	4.16	3.90	3.85	4.26	3.89	4.66	4.55	4.34	4.30	4.20	4.25	4.17	3.56
Zn	%	3.54	3.75	3.52	3.21	3.36	2.85	2.99	3.09	2.87	2.77	2.75	2.70	2.66	2.71
Recoveries															
Overall Ag recoveries	%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%
Overall Pb recoveries	%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%
Overall Zn recoveries	%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%
Total payable metal															
Ag	koz	4,303	4,999	5,086	5,476	5,622	5,871	6,018	6,527	5,913	6,457	6,457	6,621	6,333	5,403
Pb	klb	222,412	267,491	254,229	254,197	277,545	253,698	303,649	296,540	283,088	280,018	273,706	277,169	272,068	232,029
Zn	klb	200,468	209,610	199,716	184,464	190,314	161,537	169,757	175,258	162,778	157,027	155,863	152,814	151,001	153,723
Overall Ag payable in Zn Con	%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
Overall Ag payable in Pb Con	%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Overall Pb payable	%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Overall Zn payable	%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
Total net revenue	US $ '000	528,978	598,045	575,635	566,632	599,328	548,809	610,735	619,865	580,394	581,885	574,289	577,684	564,829	509,182
Operating costs															
Mining	US $ '000	127,189	125,420	127,145	128,903	127,112	127,113	127,188	127,197	127,111	127,113	137,344	130,871	136,107	175,731
Processing & tailing storage	US $ '000	37,923	37,395	37,910	38,434	37,900	37,900	37,923	37,925	37,900	37,900	37,908	37,902	37,904	37,925
General & administration	US $ '000	7,196	7,096	7,193	7,293	7,192	7,192	7,196	7,196	7,192	7,192	7,193	7,192	7,192	7,196
Smelter costs	US $ '000	112,343	125,066	119,360	114,969	121,690	108,082	120,948	121,744	114,479	112,605	110,958	110,872	108,977	100,759
Royalty	US $ '000	12,499	14,189	13,688	13,550	14,329	13,222	14,694	14,944	13,977	14,078	13,900	14,004	13,676	12,253
Mine development	US $ '000	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Other costs	US $ '000	500	500	500	500	500	500	500	500	500	500	500	500	500	500
Severance tax	US $ '000	2,479	3,328	3,193	3,228	3,392	2,931	3,545	3,666	3,311	3,360	3,285	3,212	3,103	2,139
Salvage value	US $ '000	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reclamation & closure	US $ '000	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total operating cost	US $ '000	300,129	312,994	308,990	306,877	312,114	296,939	311,993	313,173	304,471	302,748	311,088	304,553	307,459	336,503
Capital costs															
Project capital	US $ '000														
Sustaining capital	US $ '000	31,505	21,773	17,304	2,958	19,647	19,735	18,197	16,298	13,688	13,786	3,800	17,268	11,254	2,450
Total capital cost	US $ '000	31,505	21,773	17,304	2,958	19,647	19,735	18,197	16,298	13,688	13,786	3,800	17,268	11,254	2,450
Undiscounted cash flows (pre-tax)	US $ '000	199,001	262,945	249,653	257,313	267,092	232,619	279,549	289,090	261,947	264,847	259,915	255,698	246,754	175,738
Income tax	US $ '000	29,863	44,503	42,383	43,116	45,438	37,674	47,953	49,992	44,226	45,010	43,551	42,089	40,294	23,223
Undiscounted cash flows (post-tax)	US $ '000	169,138	218,442	207,270	214,197	221,654	194,945	231,596	239,098	217,721	219,838	216,364	213,610	206,461	152,515
Discounted cash flows (pre-tax)	US $ '000	62,733	76,751	67,474	64,392	61,889	49,908	55,534	53,175	44,614	41,766	37,952	34,571	30,890	20,370
Discounted cash flows (post-tax)	US $ '000	53,319	63,761	56,019	53,603	51,360	41,825	46,008	43,980	37,081	34,668	31,593	28,880	25,846	17,679

Table 22.4 Taylor Deposit production and cash flow forecast – year 29 to 35

Mine production	**Unit/yr**	**29**	**30**	**31**	**32**	**33**	**Total**
Total mined - mineralized rock	kton	2,844	2,524	2,524	1,600		96,671
Total mined - waste	kton	-	-	-	-		11,992
Total waste development - lateral	m	-	-	-	-		122,616
Total waste development - vertical	m	-	-	-	-		6,145
Total mill feed	kton	2,844	2,524	2,524	1,600		96,671
ZnEq	%	8.48	8.73	8.58	8.36		10.44
Ag	oz/ton	2.09	2.17	2.21	1.83		2.22
Pb	%	3.63	3.60	3.66	3.80		4.34
Zn	%	2.86	3.08	2.83	2.84		4.01
Recoveries							
Overall Ag recoveries	%	92.4%	92.4%	92.4%	92.4%	92.4%	92.4%
Overall Pb recoveries	%	95.4%	95.4%	95.4%	95.4%	95.4%	95.4%
Overall Zn recoveries	%	92.7%	92.7%	92.7%	92.7%	92.7%	92.7%
Total payable metal							
Ag	koz	4,624	4,243	4,351	2,231		162,566
Pb	klb	187,250	164,522	167,649	110,236		7,608,117
Zn	klb	128,307	122,703	112,678	71,543		6,112,710
Overall Ag payable in Zn Con	%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
Overall Ag payable in Pb Con	%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Overall Pb payable	%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Overall Zn payable	%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
Total net revenue	US $ '000	420,863	384,356	378,608	233,552		17,583,425
Operating costs							
Mining	US $ '000	149,821	138,537	138,537	95,146		3,668,533
Processing & tailing storage	US $ '000	29,972	26,607	26,607	16,866		1,018,908
General & administration	US $ '000	5,687	5,049	5,049	3,200		193,341
Smelter costs	US $ '000	82,913	76,165	73,767	47,063		3,625,124
Royalty	US $ '000	10,139	9,246	9,145	5,595		418,749
Mine development	US $ '000	-	-	-	-		-
Other costs	US $ '000	500	500	500	500		14,500
Severance tax	US $ '000	1,746	1,578	1,542	801		95,720
Salvage value	US $ '000	-	-	-	-		-
Reclamation & closure	US $ '000	-	-	-	-	20,000	20,000
Total operating cost	US $ '000	280,778	257,681	255,147	169,171	20,000	9,054,873
Capital costs							-
Project capital	US $ '000						518,959
Sustaining capital	US $ '000	1,800	1,500	250			725,261
Total capital cost	US $ '000	1,800	1,500	250			1,244,220
Undiscounted cash flows (pre-tax)	US $ '000	143,050	129,627	127,704	75,157	(20,000)	7,260,841
Income tax	US $ '000	19,017	17,175	16,732	8,629	-	1,208,014
Undiscounted cash flows (post-tax)	US $ '000	124,033	112,453	110,971	66,527	(20,000)	6,052,826
Discounted cash flows (pre-tax)	US $ '000	15,353	12,882	11,751	6,403	(1,578)	2,405,888
Discounted cash flows (post-tax)	US $ '000	13,312	11,175	10,211	5,668	(1,578)	1,979,101

22.3 Sensitivity analysis

AMC has carried out a sensitivity analysis of the projection for underground mine economics. The sensitivity analysis examined the impact on post-tax NPV (at 8% discount rate) of a 15% positive or negative change in metal prices, operating costs, and capital costs. The results of the sensitivity analysis are summarized in Table 22.5 and in Figure 22.1.

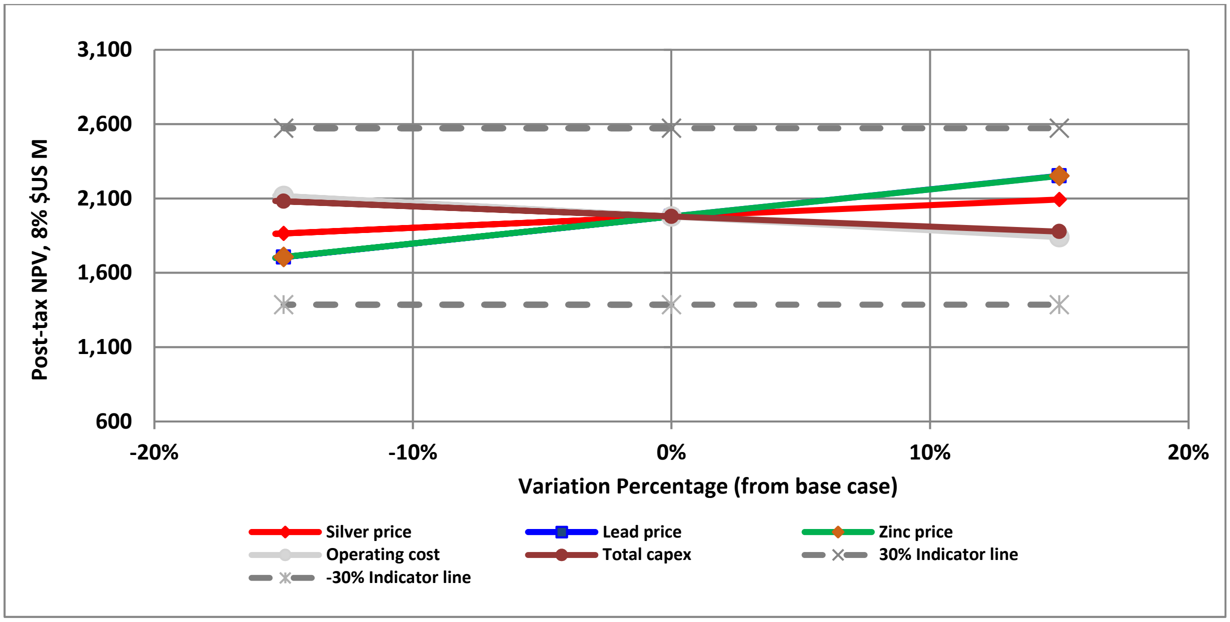
The results show that the post-tax NPV is robust and remains positive for the range of sensitivities evaluated.

Post-tax NPV is most sensitive to changes in the zinc and lead prices. The NPV is moderately sensitive to changes in operating costs. Changes in the total capital cost and in the price of silver have the least impact on NPV. Note in Figure 22.1, lead price and zinc price follow the same line.

Table 22.5 Taylor Deposit economic sensitivity analysis (post-tax)

Item	Value	Unit	Post-tax NPV (US$M)	Post-tax IRR %
Base case (NPV @ 8%)			**1,979**	**48%**
Silver price - fall of 15%	17.00	US$/oz	1,865	47%
Silver price - increase of 15%	23.00	US$/oz	2,093	50%
Lead price - fall of 15%	0.85	US$/lb	1,706	45%
Lead price - increase of 15%	1.15	US$/lb	2,252	52%
Zinc price - fall of 15%	0.94	US$/lb	1,706	44%
Zinc price - increase of 15%	1.27	US$/lb	2,252	52%
Operating cost - fall of 15%	42.97	US$/ton	2,116	50%
Operating cost - increase of 15%	58.14	US$/ton	1,840	47%
Total capex - fall of 15%	1,057,587	US$M	2,081	56%
Total capex - increase of 15%	1,430,852	US$M	1,877	43%

Figure 22.1 Sensitivity analysis – post-tax NPV at 8% discount rate



22.4 Taxation assumptions

The following assumptions (Table 22.6) have been applied in determining the US taxation cash flows incorporated into the financial model used for the updated 2017 Preliminary Economic Assessment for the Taylor Zinc-Lead-Silver sulphide project (Project).

The United States Congress gave its final approval to the House and Senate conference committee of tax reform legislation (HR1) that lowered business and individual tax rates, modernize international tax rules and provided the most significant overhaul of the US tax code in more than 30 years on 20 December 2017. The bill was signed into law by the President of the United States on 22 December 2017.

Table 22.6 Assumptions for taxation purposes in PEA Financial Model

Corporate structure	• For both Federal and Arizona State tax purposes the project is owned and operated by Arizona Minerals Inc., a US C-Corporation and is prepared on a single entity basis. The Tax Model does not take into account the impact of any other affiliated US entities.
Tax authorities	• Applicable tax jurisdictions are US Internal Revenue Code (IRC) and Arizona income tax codes and Arizona code substantially follows IRC for income tax purposes.
Financing	• Assuming all financing of project is through 100% equity funding.
Tax rates	• Assumes US statutory federal income tax rate at 21%, the, Arizona State income tax is at 4.9% and the Arizona State severance tax remains at 2.5%.
Losses and carryover attributes	• All historical tax attributes such as any loss carry forwards, recapture, mineral property, exploration costs or net tax basis of capital assets are ignored. • Net operating losses generated after 31 December 2017 are limited to 80% of taxable income for the year.
Depletion	• For regular tax purposes, only percentage depletion has been calculated – not cost depletion (see assumption above, assuming mineral property basis is nil). The proportionate profits method is used to calculate percentage depletion, and estimated non-mining costs. The rate to be used applied to adjusted gross income from mining property is based on the applicable rates to be applied for each metal based on the total revenue for each metal over the LOM and not calculated on a year by year basis assuming the mix stays relatively the same, year-by-year.
Other taxes	• Property taxes have been included in the model at a flat $500,000 per annum. • No consideration has been given to any other forms of taxation such as the Arizona State transaction privilege tax and municipal taxes.
Other timing difference and others	• The model currently shows the Project under construction for three years, which is considered development and then in production for the balance of the projected cash flows, which is considered operating. • Commercial production is assumed to commence in year 4, the year that operations commence. • Capital equipment acquisitions in years 1, 2, and 3 are considered put into service in year 4 and are depreciated from that time using the 7 year MACRS tables.. Bonus depreciation has not been taken into consideration in the model. • Development costs incurred in years 1, 2, and 3 are deducted as to 70% in the year incurred and the balance over 5 years beginning in the year incurred. As a result of the forgoing, tax losses are generated in the respective years and carried forward to set off against taxable income commencing in year 4. • The cost of buildings included in capital costs are considered immaterial and are ignored. • Roads total $16 million and are treated as development costs and deducted as development costs, as per above. • Minerals produced in a year are considered sold in the same period with no inventories of work-in-process or finished goods. • Cash is collected from sales 15 days after production. • Cash is paid to vendors 30 days after production. • Underground development costs incurred after the commencement of production are assumed to relate to mining in the year the costs are incurred and are expensed in the same year. • Reclamation costs are assumed to be incurred at the end of the mine life. For tax calculation, the cash method is used. • As stated in the royalty agreements, the royalty payments are not grossed up for any withholding taxes that may be deductible. • Assume all taxes are paid in the year incurred. • Withholding taxes on repatriation to Canadian Parent are not considered as all after tax profits are assumed to remain in the US subsidiary. • Uniform capitalization rules 263A will be disregarded as the adjustment gives rise to 1 year timing differences.

23 Adjacent properties

Currently there are no significant operating mines in the Harshaw or nearby mining districts. Properties adjacent to the Hermosa Property have had limited or no recent exploration. The mineralization on adjacent properties is hosted in various types of deposits that are not directly related to AZ's Hermosa Taylor Deposit sulphide CRD mineralization nor a projection of the mineralization types found on the Hermosa Property, and this information is not intended to indicate that such mineralization might be present on the Hermosa Property.

24 Other relevant data and information

There is no additional information or explanation to add at this time to make the technical report understandable and not misleading.

25 Interpretation and conclusions

25.1 Geology and Mineral Resources

The Property hosts two known mineral deposits, the Central Deposit and the Taylor Deposit. The Central Deposit is a siliceous, oxide, silver-manganese manto that was the subject of extensive exploration by AZ. The Taylor Deposit contains zinc-lead-silver sulphide mineralization with subordinate copper, and is comprised of both strataform replacements and chimney-type zones of mineralization. This deposit is the down-dip extension of the Central deposit. Drilling on these deposits has resulted in a reasonable understanding of the nature of the mineralization and its morphology.

Seven stratigraphic domains have been recognized within the Property: three carbonate units of Paleozoic age (in ascending order, Epitaph, Scherrer and Concha) that are overlain by two volcanic units; the Hardshell (Jurassic age) and Meadow Valley (Cretaceous age). An undivided carbonate unit (Lower Paleozoic Carbonate) and an older volcanic unit (Older Volcanics Triassic / Jurassic age) comprise the sixth and seventh domains. All units dip gently to the northwest but stratigraphic relationships are complicated by the presence of a listric thrust that dips to the southwest, predates the two youngest Mesozoic volcanic units, and places the Epitaph, Scherrer, and Concha over the undivided Lower Paleozoic Carbonate unit. A near-vertical, northeast striking fault, that may comprise a portion of the thrust, also predates the two youngest Tertiary volcanic units and separates the carbonate sequence to the southeast from a volcanic sequence to the northwest that includes the Older Volcanic unit.

Gradeshells have been used to constrain the current Mineral Resource Estimate. The sulphide domains within the Taylor Deposit have been constrained on the basis of the lithological domain and minimum zinc equivalent grade. The oxide domains in the Central Deposit have been constrained by lithological domain and either minimum silver or manganese grades.

The majority of new drilling since the last Mineral Resource estimate has been on the Taylor Deposit. Approximately 70% of the Taylor Deposit Mineral Resource has been classified as Measured and Indicated. There has been a substantial increase in Measured and Indicated Resource tons (39%) since the March 2017 estimate and the Inferred Resource portion of the Taylor Deposit has increased in tons by 13%.

The ability to model bulk density has improved over time but despite the collection of measurements of a broad range of types of mineralization and host rocks the difference between calculated and measured bulk density is still 10%. It is recommended that further investigation is carried out with the goal of obtaining an even more accurate formula with which bulk density can be estimated on the basis of metal content.

The Mineral Resource for the Central Deposit was estimated using fixed bulk density values; it is probable that these single values can be improved upon by using an approach similar to that advocated for the Taylor Deposit.

Geological and mineral resource risks associated with the Property are those attributable to any mineral exploration property at a comparable stage of exploration, namely the uncertainty attached to the continuity, grade, and tonnage of the mineral resource that has been estimated. Additional drilling to enhance the level of confidence that can be placed on the estimate, and the refinement of the bulk density equation will both help to mitigate this risk.

25.2 Mining

Additional work on the structural geology of the Taylor Deposit is required. This will assist with better definition of the expected groundwater inflows and a more accurate estimate of the implications of faulting on ground conditions and ground support requirements. Additional geotechnical sampling and testwork is required particularly in areas of critical infrastructure including the main shaft, decline and portal locations.

The underground mine is relatively deep and has a large mining footprint. Given the extent of the Mineral Resource and the potential for multiple access points, there is an opportunity to explore a higher production rate. AMC considers a production rate of 12,500 ktpa (11,250 ktonnes pa) to be achievable.

The primary issues remain around permitting of the mine, including permitting of access roads and power supply upgrades. The underground deposit shows good potential for an economic mine with a relatively simple mining method and accessibility.

Additional work was carried out to evaluate the potential for smaller stopes associated with a more selective cut and fill mining method. There is additional potential to increase extraction of the Mineral Resource through more selective mining methods.

The production schedules completed for the PEA are level based schedules, a more detailed schedule on a stope basis is required for the next level of study. The more detailed schedule should take into account opportunities to further defer capital development expenditure.

Operating cost estimates for mining have largely been based on benchmark costs for similar types of mining methods and throughput. These costs were validated based on first principles costs for labour and benchmark distribution of costs for North America. Mine capital costs are largely based on recent estimates for similar projects for other studies, vendor quotes for equipment and unit rates from previous experience. A first principles estimate and vendor or contractor quotes should be obtained for the next level of study.

25.3 Metallurgical testing and Mineral processing

The conclusions from the test-work carried out on the Taylor Deposit concluded that most of the composite samples tested for BWI were in the medium to moderately hard range, and a conventional process flowsheet for Pb / Zn minerals and standard suite of reagents produced marketable-grade concentrates of lead and zinc. From the work carried out the metal recoveries are projected to be 95.4% of lead and 69.2% of silver in lead concentrate and 92.7% of zinc and 23.2% of silver in zinc concentrates.

Processing of the material will be by conventional flotation recovery methods. The material will be crushed close to the underground mine portal and the material conveyed to the processing plant. The material will be ground to 80% passing 150 microns in a SAG / Ball mill circuit. The material will then be floated with the rougher concentrates being reground to 80% passing 38 microns prior to cleaning to produce high-value separate lead-silver and zinc concentrates. Concentrates will be trucked to the port for ocean shipment to smelters.

25.4 Tailings storage facility

The conclusions relate to the TSF:

- Cost for removing existing tailings piles are based on quantities developed from geotechnical investigation involving test borings and geophysics in the existing tailings piles. Existing ground base grades beneath the tailings piles are reasonably well defined. An increase in the amount of tailings or unsuitable material under the tailings assessed during this effort will result in an increase in cost. However, the risk of significant volumetric increases are relatively low given drilling and geophysics were performed to develop the volumetrics presented herein. No further work is required to de-risk the tailings and waste rock materials that exist in tailings piles 1, 2, 3, and 4.
- Costs for lining the TSF is based on conformance with the prescriptive BADCT criteria which states “Tailings Impoundments will be designed with a composite liner consisting of single geomembrane of at least 30 mil thickness (60 mil if HDPE) over, a minimum, 12 in (placed in two 6 in lifts) of 3/8 inch minus native or natural materials compacted to achieve a saturated hydraulic conductivity (k) no greater than 10^{-6} cm/sec.”
  - Unit costing could be impacted depending on identification and location of a suitable borrow source for the soil component of the liner system. Costing presented herein assumes a clay source that is proximate to the TSF area; cost could increase if the borrow area is remote to the TSF locations.
  - Dependent upon groundwater depth (if shallow in the area of the TSF), the prescriptive BADCT approach may be altered.
- Cost for the protective layer is based on production of a material that conforms to the prescriptive BADCT criteria which states “The geomembrane will be covered by a protective / drainage layer consisting of 3/4 inch minus, well-draining material with a minimum thickness of 18 inches.”

  - Costing for the protective layer assumes mine development rock is crushed and screened to prescriptive BADCT standards and placed on the TSF liner system using trucks and low contact pressure dozers. If a portion of the protective layer is to be sourced from a surface borrow, costing could be impacted depending on identification and location of a suitable borrow source as well as the processing requirements to develop the material within specifications. A decrease in cost may be realized with a borrow source proximate to the TSF area but conversely may increase if the borrow area is remote to the TSF locations.
- Costing for the TSF underdrain system assumes an individual BADCT approach where underdrain pipes are constructed in topographic drainages instead of 3 inch (76 mm) diameter corrugated perforated HDPE pipe at 20 foot (6.1 m) spacing for hydraulic relief over the liner. Additional cost would be required if a full underdrain piping system is required within the TSF basin.
- Costing for the rock excavation has been based on a volume estimated as 25 percent of the basin cut to fill. Should more areas require drilling and blasting, costs for this item may increase.
- Cost for construction of the rock slope armoring assumes the entirety of the rock slope armoring is constructed from non-PAG development rock. If there is a decrease in the availability of non-PAG development rock, additional cost may be required for construction of the perimeter road and rock slope armoring.

25.5 Surface infrastructure

Additional work has been devoted to infrastructure components including power acquisition, road upgrading and fresh water development Electric power to the site will be supplied via an overhead utility (UniSource Energy Services [UES]) transmission line rated 69 kV. There is an existing 13.2 kV distribution line along Flux Canyon Road that will be used to supply power during the construction period. The 69 kV line will be 23 miles (37 km) long, originating in Rio Rico.

There is currently a paved road from Patagonia to within a few miles of the mine property. This road will be upgraded for the project. The cost for this effort was reviewed with local engineering and construction companies and an allowance included in the cost estimate.

Fresh water is available on site from groundwater wells that have been tested. Groundwater modelling indicates that there is sufficient groundwater available to supply the mine operations at a rate of 650 gpm (2,460 LPM) for the life of the project. A water distribution system delivers fire and raw water on site to the processing facilities.

25.6 Environmental permitting

Numerous permits and approvals from state and federal agencies may be required in order to develop the project. The most involved permitting efforts could include the preparation of an EA or EIS for the USFS to comply with NEPA, an APP from the ADEQ, and an Air Permit, also from the ADEQ. The preparation of an EIS, should it be required, will certainly be the most complex, costly, and time-consuming permitting effort. The time to prepare an EA or EIS is expected to take 2 to 6 years or more after submission of a POO to the USFS. Should a NEPA process be required, a POO should be submitted as soon as possible after completion of a Pre-Feasibility Study or Feasibility Study. Baseline studies to obtain background data on environmental and cultural resources have been initiated and should be continued in the coming months.

25.7 Project economics

The results show that the pre-tax NPV of US$2,406M and the pre-tax IRR of 54% is robust and remains positive for the range of sensitivities evaluated. The post-tax NPV of US$1,979M at 8% discount rate and post-tax IRR of 48%, performs similarly, and also remains positive for the range of sensitivities evaluated. The sensitivity analysis examined the impact on pre-tax and post-tax NPV (at 8% discount rate) of a 15% positive or negative change in metal prices, operating costs and capital costs. The project is most sensitive to changes in zinc and lead prices, followed next by changes in operating costs.

26 Recommendations

26.1 Geology and Mineral Resources

In some areas of the Taylor Deposit, the understanding and definition could be improved by additional drilling and through building more refined mineralization domains. Specifically, AMC recommends infill drilling on the Taylor Deposit, including Taylor Deeps, to convert Inferred material to Indicated classification prior to conducting more advanced studies on the deposit (example: prefeasibility and feasibility studies). In addition to this, AMC recommends a continuation of the step out drilling to test extents / continuity of mineralization in the Taylor Deposit. The cost of this work is estimated to be US$15.0M.

For the Central Deposit, additional drilling is not recommended, most of the surface locations from which holes can be drilled in a practical and efficient fashion have already been exploited during the initial and infill drill programs. Therefore, no further drilling is recommended until underground access becomes available at which time drill stations can be located to effectively test those portions of the deposit that are material to near-term mining plans. However, AMC recommends investigating the possibility of re-assaying pulps that were originally only assayed for silver. This would avoid having absent data set to zero grade when doing the resource estimate. AMC also recommends that the existing mineralization domains be refined prior to the next Mineral Resource update.

AMC recommends that additional bulk density measurements be made on the core. The ability to model bulk density has improved but despite the collection of measurements of a broad range of types of mineralization and host rocks the difference between calculated and measured bulk density is still 10%. It is recommended that further investigation is carried out with the goal of obtaining an even more accurate formula with which bulk density can be estimated on the basis of metal content.

The Mineral Resource for the Central Deposit was estimated using fixed bulk density values; it is probable that these single values can be improved upon by using an approach similar to that advocated for the Taylor Deposit.

26.2 Exploration

AZ should continue to aggressively explore the Hermosa project for additional zinc / lead / silver / copper resources.

26.3 Mining

AMC recommends further work be done to define the structural geology with the aim of better defining ground water ingress and ground conditions. This will allow a more precise interpretation of the ground control requirements and related costs.

AMC recommends obtaining additional geotechnical sampling and testwork particularly in areas of critical infrastructure including the main shaft, decline, and portal locations.

AMC recommends evaluating an alternative option that considers an increased production rate aimed at targeting high grade material in the early stages on mine life. Given the length of mine life increased throughput would likely have a positive impact on the project economics.

More work should be done to improve the extraction of the Mineral Resource considering more selective mining methods such as cut and fill. The production schedules completed for the PEA are level based schedules, a more detailed schedule on a stope basis is recommended for the next level of study.

Opportunity to defer capital development expenditure to an as needed basis should be evaluated in the next level of study.

A first principles estimate of operating and capital costs based on actual vendor or contractor quotes should be carried out for the next level of study.

The cost of this work is estimated to be US$1.0M.

26.4 Metallurgical and Mineral processing

Further studies to improve the economics include the following:

- It should be verified that potential smelters have the capacity and ability to accept the proposed quantity and quality of produced lead and zinc concentrates. As part of the program, additional concentrate analysis should be completed to further define the concentrate qualities. Transportation, treatment charges, and refinery charges should be confirmed.
- Additional lock cycle testing is recommended for each deposit, this will allow for validation of the final estimated recoveries and the selected concentrate grades.
- Once additional testing has been performed, and samples representing optimized test conditions are available, the statistical model should be applied to ensure estimated recoveries represent optimal conditions.
- Perform a Feasibility Study to provide additional project definition. This will provide basic engineering in adequate detail to obtain a +/- 15% capital and operating cost estimate for the process plant and infrastructure.

The cost of this work is estimated to be US$1.0M.

26.5 Geotechnical investigation and evaluation

Additional geotechnical investigation and engineering evaluation should be performed on the TSF design elements presented in this PEA to develop a basis for design of all engineered structures. The investigation should focus on defining geotechnical and construction related design parameters for use in engineering analyses to be performed in future phases of the project. A brief summary of investigation and evaluation required is presented below.

- Basin preparation assessment (US$100,000)
  - Conduct a geotechnical investigation including borings, test pits, geotechnical and laboratory testing on samples collected from borings/test pits as well as geologic mapping of the proposed TSF areas. These investigations will be completed to assess subsurface conditions for the purpose of quantifying rock excavation required, identifying construction borrow sources and estimating surface preparation requirements to form a uniform and smooth basin for placement of geomembrane and construction of appurtenant civil structures. The geotechnical laboratory testing will include but not necessarily be limited to engineering characterization, strength testing of soils and rock, one dimensional consolidation test work and permeability testing of soils and rock encountered in the geotechnical investigation. The geotechnical investigation can be split evenly into two phases for Prefeasibility and Feasibility level assessments. The Prefeasibility level assessment would focus on de-risking the project by addressing any areas of geotechnical concern such as differential settlement, foundation strength and / or construction borrow source identification. If the results of the Prefeasibility work are satisfactory, a Feasibility level geotechnical investigation would then be undertaken to focus on augmentation of the Prefeasibility work to add confidence to the previous findings and to shore up quantification and qualification issues that might remain.
- Perimeter road assessment (US$25,000)
  - Conduct geotechnical drilling within the proposed perimeter road foundation where fills are greatest to define overburden depth and assess strength parameters (in particular identifying low strength areas), define bedrock conditions such as identifying karst conditions that may affect foundation treatment requirements. This activity would also be split evenly between the Prefeasibility and Feasibility phases of the project with continued Feasibility level work if Prefeasibility results are positive.
- Borrow area assessment (US$35,000)
  - Undertake a borrow investigation and laboratory test work to confirm assumptions in the cost estimate with respect to the material suitability and haul distance for construction. This would typically be completed as part of the Feasibility level effort.

- Topographic survey (US$30,000)
  - Complete a ground / aerial survey of the area for accurate contour generation and determine areas of localized steep topography and overhangs. The survey should be completed to develop a topographic base map accurate to a one ft contour interval and would be completed as part of the Prefeasibility level work.
- Tailings testwork (US$10,000)
  - Complete additional testing on samples of the proposed tailings to obtain information regarding the strength and drainage characteristics of the material. This work should be completed as part of the Prefeasibility Level effort.

26.6 Surface infrastructure

Further studies to improve the economics include the following:

- Further review the topography and geo-technical conditions to minimize earthwork, foundation and conveying costs.
- Utilize on-site mining equipment to supplement the contractor equipment for rough grading required for the access roads to the site. This same philosophy could be evaluated for the bulk of the cut and fill required at the leach pad and ponds.
- In conjunction with UES, the utility company, complete a study to optimize the power line routing and options for connection to the electrical power grid.
- Drill, install, and test one or more additional groundwater wells to add hydrological data to expand the range and understanding of the ground water regime.

26.7 Environmental permitting

AMI should continue baseline studies that will support the permitting processes expected to be required to develop the project. These include:

- Biological resources
- Cultural resources
- Hydrogeological studies
- Geochemical studies
- Air and weather monitoring
- Storm water quality
- Geotechnical (soil and rock) investigations

The estimated cost for additional baseline studies is US$2.5M.

26.8 Project economics

Given the robust economics of the project, AMC recommends taking the project to the next study level of accuracy. The cost to complete a Feasibility Study is provided under the mining and processing recommendations. This PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. The next level of study should consider only Measured and Indicated Resources.

27 References

AMC report titled “Hermosa Property, Taylor Zn-Pb-Ag Deposit Preliminary Economic Assessment” for Arizona Mining Inc., with an effective date of 29 March 2017, and dated 11 April 2017.

AMC report titled “Taylor Deposit Zn-Pb-Ag Mineral Resource Update” for Arizona Minerals Inc., with an effective date of 22 October 2016, and dated 16 November 2016.

Metal Mining Consultants Inc. report titled “NI43101 Technical Report Mineral Resource Estimate” for Arizona Mining Inc., with an effective date of 1 February 2016, and dated 17 March 2016.

Geological

Arizona Department of State, Administrative Code, Chapter 15. Department of Water Resources.

Arizona Mining Guidance Manual (BADCT), Aquifer Protection Program, Arizona Department of Environmental Quality.

Beus, S.S., 1989, Devonian and Mississippian geology of Arizona, in Jenney, J.P., and Reynolds, S.J., eds., Geologic evolution of Arizona: Arizona Geological Society Digest 17, p. 287-311.

Bloom, L., 2012. Review of Wildcat Silver Corporation’s Assay Quality Control Program. Unpublished report, Analytical Solutions Ltd. 152 pages.

Dickinson, W.R., 1989, Tectonic setting of Arizona through geologic time, in Jenney, J.P., and Reynolds, S.J., eds., Geologic evolution of Arizona: Arizona Geological Society Digest 17, p. 1-16.

Drewes, H.D., 1981, Tectonics of southeastern Arizona: U.S. Geological Survey Professional Paper 1144, 96p., 10 plates.

Drielick, Thomas L., Easton, George, Timothy J., Christopher L., McNulty, Terance P., Osterberg, Mark W., Smith, Michael R., Snider, Joshua W., Taylor, Mark F., 2014 Hermosa Project, Form 43-101F1 Technical Report, Pre-Feasibility Study, Santa Cruz County, Arizona, M3, January 17, 2014.

Gilluly, James, Cooper, J.R., and Williams, J.S., 1954, Late Paleozoic stratigraphy of central Cochise County, Arizona: U.S. Geological Survey Professional Paper 266, 49 p., 1 sheet, scale 1:62,500.

Heidrick, T.L., and Titley, S.R., 1982, Fracture and dike patterns in Laramide plutons and their structural and tectonic implications, American southwest, in Titley, S.R., ed., Advances in geology of the porphyry copper deposits, southwestern North America: Tucson, University of Arizona Press, p. 73-91.

Krantz, R.W., 1989, Laramide structures of Arizona, in Jenney, J.P., and Reynolds, S.J., eds., Geologic evolution of Arizona: Arizona Geological Society Digest 17, p. 463-483.

Malhotra, 2015. Scoping Metallurgical Testing of Hermosa NW Polymetallic Project, Unpublished report, Resource Development Inc. 43 pages.

Middleton, L.T., 1989, Cambrian and Ordovician depositional systems in Arizona, in Jenney, J.P., and Reynolds, S.J., eds., Geologic evolution of Arizona: Arizona Geological Society Digest 17, p. 273-286.

Nelson, J.L. (1996): Polymetallic Mantos Ag-Pb-Zn, in Selected British Columbia Mineral Deposit Profiles, Volume 2 - Metallic Deposits, Lefebure, D.V. and Hõy, T., Editors, British Columbia Ministry of Employment and Investment, Open File 1996-13, pages 101-104.

Rehrig, W.A., and Heidrick, T.L., 1976, Regional tectonic stress during the Laramide and late Tertiary intrusive periods, Basin and Range province, Arizona, in Wilt, J.C., and Jenney, J.P., eds., Tectonic digest: Arizona Geological Society Digest, v. 10, p. 205-228.

Silver, L. T., Bickford, M. E., Van Schmus, W. R., Anderson, J. L., Anderson, T. H., and Medaris, L. G. Jr., 1977. The 1.4-1.5 b.y. transcontinental anorogenic plutonic perforation of North America: Geological Society of America Abstracts with Programs, v. 9, no. 7, p. 1176-1177.

Smith, CDM, "Slope Stability Evaluation (Updated), Jan Adit Tailings Impoundment, Patagonia, Arizona" dated 16 February 2009.

Wilson, S.E. (2016) NI 43-101 Technical Report Mineral Resource Estimate for Arizona Mining Inc. Hermosa Taylor Deposit, Santa Cruz County, Arizona. Prepared by Metal Mining Consultants Inc. 17 March 2016.

Environmental

Arizona Department of State, Administrative Code, Chapter 15. Department of Water Resources.

Arizona Mining Guidance Manual (BADCT), Aquifer Protection Program, Arizona Department of Environmental Quality.

CDM Smith, "Slope Stability Evaluation (Updated), Jan Adit Tailings Impoundment, Patagonia, Arizona" dated 16 February 2009.

Culver, M., Malusa, S., Childs, J.L., Emerson, K., Fagan, T., Harveson, P.M., Haynes, L.E., Sanderson, J.G., Sheehy, J.H., Skinner, T., Smith, N., Thompson, K., and Thompson, R.W., 2016, Jaguar surveying and monitoring in the United States: U.S. Geological Survey Open-File Report 2016–1095, 228 p., http://dx.doi.org/10.3133/ofr20161095.

EPA Region 9 – Air. 2011. Class I Areas in EPA Region 9. Available online at: https://www3.epa.gov/region9/air/maps/pdfs/air1100018-9.pdf.

National Association of Environmental Professions. 2016. Annual NEPA Report 2015 of the National Environmental Policy (NEPA) Practice.

Town of Patagonia. General Plan 2009. Available online at:
https://issuu.com/seagoedd/docs/patagonia_general_plan?layout=
http://skin.issuu.com/v/light/layout.xml&showFlipBtn=true&e=3005223/4148168.

United States Census Bureau American Fact Finder. 2015 Population Estimates (as of 1 July 2015).

United States Census Bureau American Fact Finder. 2010 Population Estimates (as of 1 July 2015).

USFWS. 2016. Amended Final Reinitiated Biological and Conference Opinion for the Rosemont Copper Mine, Pima County, Arizona.

USFWS. 2016. Recovery Plan for the Ocelot (Leopardus pardalis), First Revision. U.S. Fish and Wildlife Service, Southwest Region, Albuquerque, New Mexico.

WestLand Resources, Inc. 2012. Survey for Hexalectris colemanii and Hexalectris arizonica across southeastern Arizona – 2012. [publicly available at http://www.rosemonteis.us/technical-reports/all].

WestLand Resources, Inc. 2013. 2012 Survey for Bartram's Stonecrop (Graptopetalum bartramii) and Beardless Chinchweed (Pectis imberbis), in the Patagonia Mountains, Near Harshaw, Arizona.

WestLand Resources, Inc. 2013 [revised]. 2012 Survey for *Hexalectris colemanii* and *H. arizonica* in the Patagonia Mountains, Near Harshaw, Arizona.

WestLand Resources, Inc. 2013. 2012 Surveys for lesser long-nosed bat (*Leptonycteris curasoae yerbabuenae*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. April.

WestLand Resources, Inc. 2013. 2012 Surveys for the Sonora tiger salamander (*Ambystoma movortium stebbinsi*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. April.

WestLand Resources, Inc. 2013. 2012-2013 Surveys for Grassland Bird Species in the Patagonia Mountains, Near Harshaw, Arizona.

WestLand Resources, Inc. 2013. 2013 Survey for Gila Topminnow (*Poeciliopsis occidentalis occidentalis*), in the Patagonia Mountains, Near Harshaw, Arizona.

WestLand Resources, Inc. 2013. 2012 Survey for Mexican spotted owl (*Strix occidentalis lucida*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. April.

WestLand Resources, Inc. 2013. 2013 Surveys for the Sonora tiger salamander (*Ambystoma mavortium stebbinsi*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. December.

WestLand Resources, Inc. 2013. 2013 Survey for yellow-billed cuckoo (*Coccyzus americanus*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. December.

WestLand Resources, Inc. 2013. A Cultural Resources Inventory of 2,634 Acres of Private and Federal Lands in Support of the Hermosa Drilling Project Plan of Operations Within the Coronado National Forest, Arizona.

WestLand Resources, Inc. 2013. Revised 2012 Survey for yellow-billed cuckoo (*Coccyzus americanus*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. April.

WestLand Resources, Inc. 2013. Summary of 2013 Survey for lesser long-nosed bat (*Leptonycteris yerbabuenae*) in the Patagonia Mountains near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. December.

WestLand Resources, Inc. 2013. Summary of 2013 Survey for Mexican spotted owl (*Strix occidentalis lucida*) in the Patagonia Mountains, near Harshaw, Arizona. Prepared for Arizona Minerals, Inc. December.

WestLand Resources, Inc. 2013. Summary of 2013 Survey for Northern Goshawk (*Accipiter gentilis*) in the Patagonia Mountains, Near Harshaw, Arizona.

WestLand Resources, Inc.2016. 2016 yellow-billed cuckoo (*Coccyzus americanus*) in support of the Hermosa Taylor Drilling Plan of Operations. November.

WestLand Resources, Inc. 2016. A Cultural Resources Inventory 160 Acres of Coronado National Forest Land for Possible Mineral Exploration Activities in Santa Cruz County, Arizona.

WestLand Resources, Inc. WestLand Resources, Inc. 2016. Northern Goshawk (*Accipiter gentilis*) Survey in Support of the Hermosa Taylor Drilling Plan of Operations.

WestLand Resources, Inc. 2016. 2016 Surveys for Mexican spotted owl (*Strix occidentalis lucida*) in Support of the Hermosa Taylor Drilling Plan of Operations. November.

WestLand Resources, Inc. unpublished data, 2017. A Cultural Resources Inventory of Approximately 19.4 Acres of Coronado National Forest Land near Harshaw, in Santa Cruz County, Arizona.

WestLand Resources, Inc. unpublished data, 2017. A Cultural Resources Inventory of Approximately 9.8 Acres of Coronado National Forest Land near Harshaw, in Santa Cruz County, Arizona.

Tailings storage facility

Arizona Department of State, Administrative Code, Chapter 15. Department of Water Resources.

Arizona Mining Guidance Manual (BADCT), Aquifer Protection Program, Arizona Department of Environmental Quality.

CDM Smith, “Slope Stability Evaluation (Updated), Jan Adit Tailings Impoundment, Patagonia, Arizona” dated February 16, 2009.

28 QP Certificates

CERTIFICATE OF GARY METHVEN, P.ENG.

I, Gary Methven, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as Underground Manager / Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd. with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4;
2. This certificate applies to the technical report titled “Hermosa Property Mineral Resource and Taylor Deposit PEA update”, with an effective date of 1 January 2018, (the “Technical Report”) prepared for Arizona Mining Inc. (“the Issuer”);
3. I am a graduate of the University of Witwatersrand in Johannesburg, South Africa (Bachelors of Science in Mining Engineering in 1993). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #180019), a member of Registered Professional Engineers of Queensland (License #06839), and a member of the Australian Institute of Mining and Metallurgy (CP). I have experience in narrow-vein gold deposits, flat and steeply dipping, bulk and selective mining methods for base metals, mine infrastructure, design and planning, mine production and financial evaluation, reserve estimation, technical reviews, feasibility and pre-feasibility studies, project and construction management, contracts management and cost estimation.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I have visited the Property on 13 July 2016, for 1 day;
5. I am responsible for Sections 2, 3, 15, 16, and 24, and parts of 1, 21, 25, and 26 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1 January 2018
Signing Date: 16 January 2018

(original signed and sealed)

Gary Methven, P.Eng.
Underground Manager / Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF DINARA NUSSIPAKYNOVA, P.GEO.

I, Dinara Nussipakynova, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Principal Geologist with AMC Mining Consultants (Canada) Ltd. with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4;
2. This certificate applies to the technical report titled “Hermosa Property Mineral Resource and Taylor Deposit PEA update”, with an effective date of 1 January 2018, (the “Technical Report”) prepared for Arizona Mining Inc. (“the Issuer”);
3. I am a graduate of Kazakh National Polytechnic University (B.Sc. and M.Sc. in Geology, 1987). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #37412) and the Association of Professional Geoscientists of Ontario (License #1298). I have practiced my profession continuously since 1987, and have been involved in mineral exploration and mine geology for a total of 28 years since my graduation from university. This has involved working in Kazakhstan, Russia and Canada. My experience is principally in database management, geological interpretation and Mineral Resource estimation.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I have visited the property from 6 - 7 September 2017;
5. I am responsible for Sections 4, 6, 7, 8, 9, 10, 12, 14, and 23, and parts of 1, 5, 11, 25, 26, and 27 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1 January 2018
Signing Date: 16 January 2018

(original signed and sealed)

Dinara Nussipakynova, P.Geo.
Principal Geologist
AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF LYNDA BLOOM, P.GEO.

I, Lynda Bloom, P.Geo., of Mulmur, Ontario, do hereby certify that:

1. I am currently employed as President of Analytical Solutions Ltd. with an office at 878213 5th Line East, Mulmur, ON, Canada, L9V 0L1;
2. This certificate applies to the technical report titled “Hermosa Property Mineral Resource and Taylor Deposit PEA update”, with an effective date of 1 January 2018, (the “Technical Report”) prepared for Arizona Mining Inc. (“the Issuer”);
3. I am a graduate of Carleton University, Ottawa Ontario (B.Sc. Honours, Combined Geology and Chemistry, 1977) and Queen’s University, Kingston, Ontario (M.Sc., Geological Sciences, 1981). I am a registered geologist of the Association of Professional Geologists of Ontario (No. 0019).

   I have practiced as a professional geochemist for 30 years. I have experience working in the field of assaying and sampling for mining companies that include projects worldwide for a wide variety of commodities. I have published over 40 publications and presented over 25 workshops and short courses on these topics.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I have not visited the Property;
5. I am responsible for parts of Section 11 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had prior involvement with the property that is the subject of the Technical Report; I prepared an assay quality control report in 2011, assisted with organization of the 2013 silver re-assay Program and contributed to the “Technical Report for the Taylor Deposit Mineral Resource Update for Arizona Minerals Inc.”, with an effective date of 14 November 2016;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1 January 2018
Signing Date: 16 January 2018

(original signed and sealed)

Lynda Boom, P.Geo.
President
Analytical Solutions Ltd.

CERTIFICATE OF QINGHUA JIN, P.E.

I, Qinghua Jin, P.E., of Tucson, Arizona, do hereby certify that:

1. I am currently employed as a Senior Process Engineer with SGS North America Inc. with an office at 3845 N. Business Center Drive, Suite 111, Tucson, AZ 85705;
2. This certificate applies to the technical report titled “Hermosa Property Mineral Resource and Taylor Deposit PEA update”, with an effective date of 1 January 2018, (the “Technical Report”) prepared for Arizona Mining Inc. (“the Issuer”);
3. I am a graduate of Northeastern University in Shenyang, China (Bachelor of Engineering in Mineral Processing Engineering in 1990) and West Virginia University, USA (two Master of Science degrees in Mining Engineering and Statistics in 2002 and 2006). I am a member in good standing of the Association of Arizona State Board of Technical Registration (License #53463), and a registered member of the Society for Mining, Metallurgy & Exploration (04138753RM). I have experience over 26 years in scoping, prefeasibility and feasibility studies for mining projects in the North America, South America, Europe, and Asia, as well as having worked on the design phases of some of these projects.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I have visited the Property on 4 October 2016 for 1 day;
5. I am responsible for Sections 17 and 19, and parts of 1, 18, 21, 25, 26, and 27 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1 January 2018
Signing Date: 16 January 2018

(original signed)

Qinghua Jin, P.E.
Senior Process Engineer
SGS North America Inc.

CERTIFICATE OF CHRISTOPHER KAYE, FAusIMM

I, Christopher Kaye, FAusIMM, of San Mateo, CA, USA do hereby certify that:

1. I am currently employed as Principal Process Engineer with Mine and Quarry Engineering Services, Inc. (MQes) with an office at 635 Mariner's Island Blvd. Suite 202, San Mateo, CA 94404, USA;
2. This certificate applies to the technical report titled "Hermosa Property Mineral Resource and Taylor Deposit PEA update", with an effective date of 1 January 2018, (the "Technical Report") prepared for Arizona Mining Inc. ("the Issuer");
3. I am a graduate of the University of Queensland, Australia, (B. Eng. in Chemical Engineering in 1984). I am a Fellow of Australasian Institute of Mining and Metallurgy in Australia. I have worked as a process engineer in the minerals industry for over 35 years. I have been directly involved in the mining, exploration and evaluation of mineral properties internationally for gold and base metals;

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I have not visited the Property;
5. I am responsible for Section 13 and parts of Sections 1, 25, 26, and 27 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1 January 2018
Signing Date: 16 January 2018

(original signed)

Christopher Kaye, FAusIMM
Principal Process Engineer
Mine and Quarry Engineering Services Ltd.

CERTIFICATE OF R. MICHAEL SMITH, P.E.

I, R. Michael Smith, P.E., of Lone Tree, Colorado, do hereby certify that:

1. I am currently employed as a Principal Engineer with Newfields Mining Design and Technical Services with an office at 9400 Station Street, Suite 300, Lone Tree, CO 80124;
2. This certificate applies to the technical report titled "Hermosa Property Mineral Resource and Taylor Deposit PEA update", with an effective date of 1 January 2018, (the "Technical Report") prepared for Arizona Mining Inc. ("the Issuer");
3. I am a graduate of The University of Colorado in Denver, Colorado (Bachelors of Civil Engineering in 1983). I am a member in good standing of the Society of Mining Engineers (SME), the American Society of Civil Engineers (ASCE), Colorado Board of Registration for Profession Engineers (License Number 28114), Alaska Board of Registration for Professional Engineers (License Number CE8785) and Nevada Board of Registration of Professional Engineers (License Number 16194). My primary areas of expertise are design, construction and capital / operation cost estimation of Tailings Storage Facilities and Heap Leach Pad Facilities.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I have visited the property multiple times, the last time being 19 January 2017;
5. I am responsible for parts of Sections 1, 18, 21, 25, 26, and 27, of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had prior involvement with the property that is the subject of the Technical Report; in 2014 I lead a study to develop a tailing storage option on the Trench Camp site and in 2012 / 13 I lead an effort to complete a PFS on the Hermosa Project;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1 January 2018
Signing Date: 16 January 2018

(original signed)

R. Michael Smith, P.E.
Principal Engineer
Newfields Mining Design and Technical Services

CERTIFICATE OF CARL KOTTMEIER, P.ENG.

I, Carl Kottmeier, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd. with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4;
2. This certificate applies to the technical report titled "Hermosa Property Mineral Resource and Taylor Deposit PEA update", with an effective date of 1 January 2018, (the "Technical Report") prepared for Arizona Mining Inc. ("the Issuer");
3. I am a graduate of the University of British Columbia in Vancouver, British Columbia, Canada (Applied Science – Mining and Mineral Process Engineering, 1989). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #18702), and a member of the Canadian Institute of Mining, Metallurgy and Petroleum. I have practiced my profession continuously since 1989, and have been involved in mine engineering for a total of 28 years since my graduation from university. This has involved working primarily in Canada and in the United States. My experience is principally in coal, base metals, gold, and silver.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I have not visited the Property;
5. I am responsible for Section 22 and parts of Sections 1, 18, 21, 25, and 26 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1 January 2018
Signing Date: 16 January 2018

(original signed and sealed)

Carl Kottmeier, P.Eng.
Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF DOUG BARTLETT, CPG AIPG, RG AZ

I, Doug Bartlett, CPG AIPG, RG AZ, of Scottsdale, Arizona, do hereby certify that:

1. I am currently employed as a Principal and President with Clear Creek Associates with an office at 6155 E. Indian School Rd., Suite 200, Scottsdale, AZ 85251;
2. This certificate applies to the technical report titled “Hermosa Property Mineral Resource and Taylor Deposit PEA update”, with an effective date of 1 January 2018, (the “Technical Report”) prepared for Arizona Mining Inc. (“the Issuer”);
3. I am a graduate of Colorado State University in Fort Collins, Colorado, USA (Bachelors/Masters of Geology in 1977/1984). I am a member in good standing of the American Institute of Professional Geologists (CPG #8433), and a registered geologist in the states of Arizona (RG#25059), California (PG#8809; CHG#965), Oregon (RG#2305), Washington (PG#2879), and Pennsylvania (PG#4995). I have experience in mining hydrogeology, groundwater production, and hydrogeologic permitting.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I have visited the Property on 4 October 2016 for 1 day;
5. I am responsible for parts of Sections 5 and 20 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1 January 2018
Signing Date: 16 January 2018

(original signed)

Doug Bartlett, CPG AIPG, RG AZ
Principal and President
Clear Creek Associates

CERTIFICATE OF ERIK CHRISTENSON, P.E. AZ

I, Erik Christenson, P.E. AZ, of Tucson, Arizona, do hereby certify that:

1. I am currently employed as a Senior Engineer with WestLand Resources Inc. with an office at 4001 E. Paradise Falls Drive, Tucson, AZ 85712;
2. This certificate applies to the technical report titled "Hermosa Property Mineral Resource and Taylor Deposit PEA update", with an effective date of 1 January 2018, (the "Technical Report") prepared for Arizona Mining Inc. ("the Issuer");
3. I am a graduate of the University of Arizona in Tucson, Arizona (Bachelors of Civil Engineering in 2012). I am a registrant in good standing of the Arizona Board of Technical Registration (Civil Engineering License #57421), and a member of the AZ Water Association. I have experience in mining process design, leaching system design and operation, milling design and operation, Plan of Operation development, water balance studies, surface hydrology, and dewatering operations.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
4. I have visited the Property on 2 March 2017 for 1 day;
5. I am responsible for parts of Sections 1, 20, 25, 26, and 27 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1 January 2018
Signing Date: 16 January 2018

(original signed)

Erik Christenson, P.E. AZ
Senior Engineer
WestLand Resources Inc.

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