PULACAYO PROJECT
FEASIBILITY STUDY

Pulacayo Pb – Ag – Zn Project
Phase I, 1,000 t/d
Pulacayo, Potosi, Bolivia

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1 SUMMARY

1.1 Introduction


The study has been managed by TWP with input from various consulting groups engaged by Apogee.

The conclusions and recommendations in this report reflect the author’s best judgment in light of the information available at the time of the writing of this report.

1.2 Location and Property Description

The Pulacayo prospect is located 18 km north east of the city of Uyuni (Canton of Pulacayo, Quijarro Province) in the Department of Potosí in south western Bolivia, 460 km south east of the capital city, La Paz, and 130 km south west of Potosí, the department capital (Figure 1.1). The property is located at 740 450 mE and 774 4695 mN WGS84 Zone 19, south datum, and at an elevation of 4,305 m ASL.

Pulacayo is accessible by a good quality, newly paved highway from La Paz via Oruro and Potosí (715km). A shorter, alternative route to the mine exists from Oruro via Challapata and Uyuni which reduces the distance by 150 km. Unpaved road sections are generally passable during the whole year although they may present some level of difficulty during the rainy season. The tourist town of Uyuni, on the edge of the large Salar de Uyuni (salt lake) provides limited local services. The town has unpaved road and railway connections with the cities of Oruro, Potosí, Villazon, and to the borders with Argentina and Chile.

Uyuni has a newly developed airport, in operation since 2011; with an asphalt runway, which can now accommodate turbo props and a regional jet service (only small aircrafts are currently under operation, with less than 40 passengers). Daily flights are available from La Paz with an average flying time of one hour. There are also several small hotels, hostels, restaurants, schools, medical and dental facilities and internet cafes in Uyuni. San Cristobal Mining Company has constructed a gravel road from San Cristóbal, approximately 100 km south west of Uyuni, to the border with Chile.
Figure 1.1: Location Map - Pulacayo Project
1.3 Geology and Mineral Resources

1.3.1 Geological Setting and Mineralization

The Pulacayo deposit is located on the western flank of the Cordillera Oriental, near the Cordillera-Altiplano boundary. The area is underlain by folded sedimentary and igneous rocks of Silurian, Tertiary and Quaternary ages that locally host low sulphidation epithermal polymetallic vein and stock work styles of mineralization. The Pulacayo deposit is representative of such mineralization, which developed in association with a Tertiary volcanic center at Pulacayo. Development of this center was controlled by the major, north trending Uyuni-Khenayani Fault that parallels the Cordillera-Altiplano boundary.

Polymetallic (Ag-Pb-Zn) vein and disseminated wall rock mineralization at Pulacayo are controlled by east west trending secondary faults. The main mineralized trend was emplaced on the southern side of the Pulacayo dome complex and is best exemplified by the Tajo Vein System (TVS) that is the subject of this Feasibility Study. The TVS trends east west, dips 75° to 90° to the south along 2,700 meters (m) of defined surface strike length, and is present in mine workings at a depth of 1,000 m below surface. In the upper levels of the deposit, where volcanic strata host mineralization, the TVS consists of a stock work vein system that locally reaches 120 m in mineralized width. Sedimentary strata host the vein system at depth, where narrower widths of one to 3 m are apparent. Mineralization of economic interest is comprised of sphalerite, galena and tetrahedrite in sulphide-rich veins that are accompanied by locally abundant quartz, barite and pyrite. Disseminated mineralization is preferentially developed around and between veins hosted by andesite.

1.3.2 Deposit Type

The Pulacayo deposit has been classified as an epithermal vein deposit of low to intermediate sulphidation state association. Veins commonly contain semi-massive to massive sulphide and show internal features such as compositional banding, crustiform texture, and drusy character.

1.3.3 Exploration

Apogee has carried out detailed geological mapping and sampling programs at surface and in underground workings, a topographic survey, Induced Polarization (IP) geophysical surveying, four diamond drilling programs, four mineral resource estimates (including that used in this Feasibility Study) and a Preliminary Economic Assessment (PEA) at Pulacayo. In addition, underground workings have been rehabilitated to support bulk sampling and metallurgical testing of bulk sample materials has been carried out.

1.3.4 Drilling

Modern era drilling at Pulacayo was initiated in 2002 and subsequent drilling programs were undertaken between 2006 and 2012. A total of 69,739 m of diamond drilling by Apogee in 226 surface drill holes and 42 underground drill holes support mineral resource and reserve estimates pertinent to the current Feasibility Study. HQ (65.3 mm diameter) size core was recovered in these programs, except where poor drilling conditions required reduction in
coring size to NQ (47.6 mm diameter). Calculated core recoveries for all programs are very high and core loss is not considered a problem for this project.

1.3.5 Sample Preparation, Analyses and Security

Apogee has instituted various protocols with respect to handling, logging and sampling of drill core and for sample preparation, transportation, laboratory analysis, quality control and assurance (QAQC), and security. Samples are shipped to ALS Chemex (ALS) in Oruro, Bolivia for preparation. Analysis of prepared samples was initially carried out by ALS Chemex in Vancouver, BC, Canada and since 2006 has been carried out by ALS Chemex in Lima, Peru. Silver, lead and zinc levels are analyzed using an Aqua Regia digestion and Atomic Absorption Spectroscopy (AAS). ALS is an internationally accredited laboratory with National Association of Testing Authorities (NATA) certification and complies with standards of ISO 9001:2000 and ISO 17025:1999. Apogee’s internal QAQC program includes blind insertion of reference standards, blanks and duplicates in each analytical shipment and analysis of check samples at an independent accredited laboratory.

1.3.6 Data Verification

Core sample records, lithologic logs, laboratory reports and associated drill hole information for all drill programs were digitally compiled by Apogee staff and made available to Mercator for checking and validation to support resource estimation programs. Mercator completed two field visits to the Pulacayo site, (03 August to 10 August 2011 and 26 April to 28 April 2012) and in each case completed reviews of Apogee drill program components, checked drill collar positions using GPS instrumentation, and conducted drill core check sampling programs. Detailed review and assessment of all Apogee drilling program QAQC program results was carried as part of the resource estimation process. All checking and validation procedures carried out by Mercator produced acceptable results and the project drilling database was determined to be acceptable for resource estimation purposes.

1.3.7 Mineral Resource Estimates

The Mineral Resource Estimate used in the current Feasibility Study was prepared by Mercator in accordance with Canadian Securities Administrators National Instrument 43-101 ("NI 43-101") and is based on Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards on Mineral Resources and Mineral Reserves (CIM Standards). The effective date of the mineral resource estimate is 28 September 2012.

Modeling was performed using Gemcom Surpac® 6.3 modeling software with silver, lead and zinc grades estimated independently by inverse distance squared (ID2) interpolation from 1 m down hole assay composites, capped at 1,500 g/t, 15%, and 15% respectively. Block size is 5 m (x) by 3 m (y) by 3 m (z), with one unit of standard sub-blocking allowed. The modeled deposit has a strike length of 1,500 m oriented at 280° and a 600 m sub-vertical dip extent. A specific gravity model was also interpolated using ID2 methodology. Net Smelter Return (NSR) values were calculated from block grade parameters and reflect a 36 month trailing average silver price of $25.00 USD/oz and long term prices of $0.86 USD/lb lead and $1.00 USD/lb zinc. Open pit resources to an elevation of 4,159 m ASL (top of proposed crown pillar) were determined within a Whittle optimized pit shell. Only silver
derived NSR values were used for oxide zone material, while silver, lead and zinc derived NSR values were used for sulphide zone material. Estimated mineral resources at the 28 September 2012 effective date are tabulated below.

**Table 1.1: Pulacayo Deposit Mineral Resource, Prepared by Mercator – Effective 28 September 2012**

<table>
<thead>
<tr>
<th>Resource Class</th>
<th>Type</th>
<th>Tons</th>
<th>Ag g/t</th>
<th>Pb %</th>
<th>Zn %</th>
<th>Ag Oz</th>
<th>Pb M lbs.</th>
<th>Zn M Lbs.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Open Pit Resources (Base case 42° Average Pit Wall Slope Angle)</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Open Pit Indicated Oxide</td>
<td></td>
<td>1,500,000</td>
<td>95.9</td>
<td>0.96</td>
<td>0.13</td>
<td>4,626,000</td>
<td>NA</td>
<td>NA</td>
</tr>
<tr>
<td>Open Pit Inferred Oxide</td>
<td></td>
<td>248,000</td>
<td>71.2</td>
<td>0.55</td>
<td>0.31</td>
<td>569,000</td>
<td>NA</td>
<td>NA</td>
</tr>
<tr>
<td>Open Pit Indicated Sulphide</td>
<td></td>
<td>9,283,000</td>
<td>44.1</td>
<td>0.66</td>
<td>1.32</td>
<td>13,168,000</td>
<td>135.9</td>
<td>269.54</td>
</tr>
<tr>
<td>Open Pit Inferred Sulphide</td>
<td></td>
<td>2,572,000</td>
<td>33.4</td>
<td>0.92</td>
<td>1.36</td>
<td>2,765,000</td>
<td>51.99</td>
<td>76.88</td>
</tr>
<tr>
<td>Waste Rock (Waste/Ore 5.3:1)</td>
<td></td>
<td>71,679,000</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Underground Resources (All blocks below 4,159 m ASL with NSR &gt; $58 USD)</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Underground Indicated Sulphide</td>
<td></td>
<td>6,197,000</td>
<td>213.6</td>
<td>0.86</td>
<td>1.74</td>
<td>42,547,000</td>
<td>117.5</td>
<td>237.72</td>
</tr>
<tr>
<td>Underground Inferred Sulphide</td>
<td></td>
<td>943,000</td>
<td>193.1</td>
<td>0.43</td>
<td>1.61</td>
<td>5,853,000</td>
<td>8.94</td>
<td>43.47</td>
</tr>
<tr>
<td>Total Indicated Oxide + Sulphide</td>
<td></td>
<td>16,980,000</td>
<td>110.5</td>
<td>0.74</td>
<td>1.49</td>
<td>60,341,000</td>
<td>253.4</td>
<td>507.26</td>
</tr>
<tr>
<td>Total Inferred Oxide + Sulphide</td>
<td></td>
<td>3,763,000</td>
<td>75.9</td>
<td>0.79</td>
<td>1.43</td>
<td>9,187,000</td>
<td>60.93</td>
<td>120.35</td>
</tr>
</tbody>
</table>

**Notes to Table 1.1**

1. Tonnages have been rounded to the nearest 1,000 tonnes. Average grades may not sum due to rounding.
2. Metal prices used were $25.00 USD/Oz silver, $0.89USD/lb lead, and $1.00 USD/lb zinc. Lead and zinc do not contribute to revenue in the oxide zone.
3. Open Pit Sulphide Resources are reported a $13.20 USD NSR cut-off. Underground Sulphide Resources are reported at a $58USD NSR cut-off. Open Pit oxide resources are reported at a $US 23.10 revenue/tonne cut-off.
4. Contributing 1.0 meter assay composites were capped at 1500 g/t Ag, 15% Pb, and 15% Zn.
5. Specific gravity is based on an interpolated inverse distance squared model.
6. The Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves.
7. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
1.4 Mining Methodology and Reserve Statement

Pulacayo is an existing mine with a long history of development, however in more recent years on a much reduced scale. This project plans to increase production significantly and exploit previously unmined material utilizing a combination of two methods. Shrinkage stoping will be used on zero level and longhole open stoping will be used on all levels below zero level. The stope widths vary from 1 m to 6 m, depending on the mining method and width of mineralization.

The mine has an existing tunnel (San Leon Tunnel) located on zero level which starts in the town of Pulacayo and exits on the northern side of the mountain that hosts the mineralized rock. This tunnel will continue to be used but a new decline ramp system will be constructed to allow access with trackless equipment and to exit closer to the mineral processing plant. This decline ramp system will be developed from surface (at an inclination of 8 degrees from the horizontal) using conventional drill and blast techniques.

For the shrinkage stoping planned or zero level, conventional drill and blast techniques will be used to advance existing ore and waste drives, and to develop new shrinkage stope. Air loaders will be used to clean development faces. The broken rock will be loaded into locomotives and trammed out of the mine. Development of the required tunnels for access to the long hole stopes will be done with a drill jumbo and blasting using Anfex. These tunnels are the main haulage, crosscuts and ore drives and are planned with dimensions 4.0 m high and 4.0 m wide. An LHD will be used to muck out, and the broken rock will be hauled out of the mine using the existing 15 tonne truck.

1.4.1 Backfill

Backfilling of mined out long hole stopes is planned to facilitate both safe mining and to provide for better grade and dilution control. Cemented backfill will provide permanent support to the stope excavation, particularly with regard to the mining of adjacent areas. A backfill plant will be constructed on surface near an existing ventilation shaft and will be fed with tailings from the concentrator plant. The backfill will be mixed on surface with cement and pumped underground in a pump column and distributed to the stopes.

1.4.2 Mine Production

The mine has been designed for an ore production of 1,000 tons (t) per day. The mine plan indicates a total of 3,557 Million tons of ore and 0.839 Million tons of waste. These will be extracted over a planned 12.5 year Life of Mine (LOM). The average grades over the LOM are 239 g/t for silver, 1.09% for lead and 1.91% for zinc.
Table 1.2: Mineral Reserve Statement prepared by TWP - Effective 17 January 2013

<table>
<thead>
<tr>
<th>Class</th>
<th>Million Tons (Mt)</th>
<th>AG (G/T)</th>
<th>PB (%)</th>
<th>ZN (%)</th>
<th>AG (MOz)</th>
<th>PB (T)</th>
<th>ZN (T)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Probable Reserve</td>
<td>3.558</td>
<td>239.4</td>
<td>1.09</td>
<td>1.91</td>
<td>27.385</td>
<td>38,927</td>
<td>67,905</td>
</tr>
</tbody>
</table>

Notes to table 1.2

1. Probable Reserves are modified resources included in the Company’s Resource statement (see Table 1.1) and are not in addition to these Mineral Resources.
2. Only Indicated Resources were modified to determine the Probable Reserve.
3. Modifying factors include mining losses (2%), lashing losses (2%), and void losses (5%) and mining dilution of 8%.
4. An NSR cutoff of US $70/t or 150g/t was considered. Metal prices used were US$28/oz for silver, US$0.86/lb for lead and US$1.00/lb for zinc. Metal recovery curves using locked cycle flotation tests (LCTs) completed by Maelgwyn Minerals Services Africa, were applied to all blocks used for calculating recovered metal. Average payables on lead, zinc and silver were estimated at USD240/t concentrate and a mass pull of 5%.
5. Process Cost (1000tpd) 12.00 USD/ton processed
6. General and Administration 2.50 USD/ton processed
7. Extra out of Mine Haulage 1.00 USD/ton processed
8. Selling costs & payables 12 USD/ton processed ($240/tonne concentrate x 5% mass pull)
9. Operating costs per ore ton = (12.00 USD +2.50 USD +1.00 USD+ 42.50 USD + 12.00USD) = 70.00 USD/ton processed

The mineral reserves were developed by TWP Projects (TWP) with Jim Porter (FSAIMM) acting as the qualified person. Metal price changes or significant changes in costs or recoveries could materially change the estimated mineral reserves in either a positive or negative way. At this time, there are no unique situations relative to environmental or socio-economic conditions that would put the Pulacayo mineral reserves at a higher level of risk than any other developing resource in Bolivia.

1.5 Mineral Processing and Metallurgical Testing

The interpretation of results from metallurgical test work carried out in four stages between 2009 and 2012 has been used to guide the process plant design. Testwork suggested that conventional crushing and milling (to P80 of 74 µm) circuits followed by lead and zinc differential flotation and concentrates dewatering can be used to attain saleable lead and zinc concentrates containing silver credits.

Flotation tailings will be routed to a paste plant to produce cemented paste fill when backfill is required in the underground mine. When backfill is not required, tailings will be routed to a tailings storage facility (TSF) on the project site.

Plant design includes a water treatment plant to produce potable water for human consumption. A simplified process flow sheet is shown in Figure 1.2.
Figure 1.2: Consolidated Process Flowsheet
1.6 Infrastructure

The Project site infrastructure is shown in Figure 1.3 and Figure 1.4. Administrative and auxiliary buildings will be located near to the plant and mining area.

The tailings storage facility will be located to the south east of the main operation area. The initial tailings storage facility construction will provide 13 years of storage capacity, and a subsequent raise will provide sufficient tailings storage capacity for the remainder of the Life of Mine.

The electrical power will be provided by constructing a 61 km long 115 kV overhead transmission line that will tie in to a substation in Pulacayo.

Water will be supplied to the Project by connecting to a regional raw water supply system. A 1.4 km pipeline will be installed by the Project to deliver water from the tie-in point to the Pulacayo processing operation. The tie-point is located near the San Leon mine portal to the west of the process plant. A pipeline running 3.8 km through the mountain along the San Leon mine tunnel originates from the Yana Pollera reservoir, a total of 9.5 km to the north of the project.

A handling facility for the concentrates will be constructed; it will include a storage warehouse, weigh scales and loading equipment. Concentrates will be transported by truck from the Pulacayo Project to the final destination by road.
Figure 1.3: Project Infrastructure Site Plan
Figure 1.4: Site Plot Plan
1.7 Social and Environmental Aspects

1.7.1 Environmental Impact

A wide range of environmental and socioeconomic studies have been conducted since 2007 for the Pulacayo Mining Project to guide exploration, development and operational decisions. These studies were completed by MINCO and more recently by MEDMIN S.A. Both are independent environmental consulting companies operating in Bolivia.

During the various studies, baseline information was collected on water quality, air quality, soil quality, sediment quality, drinking water quality and the condition of flora and fauna. In many cases, the baseline conditions exceeded either Bolivian requirements, or, where no local requirements existed, accepted international standards.

The Pulacayo Mining Project is not within any protected areas of Bolivia.

The characterization of eco-biological components indicates that the project area lies within two major sub-Eco regions:

- The semi-arid Puna, and
- The Puna desert (semi-arid) in the Andean region.

Various main families of fauna were identified, including mammals, birds and reptiles. The greatest threat identified was habitat destruction, but Apogee seeks to address those concerns as set out below.

1.7.2 Environmental Management Plans

Articles 29 and 30 of the Regulation on Prevention and Environmental Control, states that it is necessary to define and describe a set of protective, corrective, and compensatory measures that will serve to prevent, reduce, and eliminate or compensate for the expected changes. Plans have been developed for the impacts on air, soil, water, fauna, and flora. Plans have been developed to minimize impact on social and cultural areas.

1.7.3 Closing and Rehabilitation Plan

The closure of the mining operation involves the implementation of activities that are necessary to generate levels of security and the rehabilitation of areas impacted by mining operations. The area of greatest impact on surface will be the Tailings Storage Facility.

1.7.4 Environmental Permits and Licenses

Three permits have been acquired for exploration phase activities and two permits have been obtained for exploitation phase activities. There are still a number of permits required before full scale project development and operations can start.
1.8 Capital and Operating Costs

The capital cost estimate has been developed in Q2 2012 United States Dollars (USD). The estimates were developed to a Feasibility Study level with a target accuracy of ±15%.

1.8.1 Capital Costs

The pre-production (initial) capital cost (Capex) for the Project was estimated at USD 45.9M. A contingency of 10% is included in the capital cost estimate (direct plus indirect costs) before contingency and taxes. Table 1.2 summarizes the initial capital cost estimate.

Sustaining capital required over the life of the operation has been estimated to be in the order of USD 41.1M (over 12.5 full production years), see Table 1.3.

Table 1.3: LOM Initial Capital Estimate

<table>
<thead>
<tr>
<th>Startup Capex</th>
<th>LOM Capex, USD$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process Plant</td>
<td>12,790,234</td>
</tr>
<tr>
<td>Tailings Storage Facility (TSF)</td>
<td>923,700</td>
</tr>
<tr>
<td>Mining</td>
<td></td>
</tr>
<tr>
<td>Mining Development</td>
<td>5,642,924</td>
</tr>
<tr>
<td>Mining Equipment</td>
<td>3,633,881</td>
</tr>
<tr>
<td>Mining Services</td>
<td>2,833,416</td>
</tr>
<tr>
<td>Mining Backfill System</td>
<td>2,107,109</td>
</tr>
<tr>
<td>Site Development</td>
<td>2,207,287</td>
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<tr>
<td>Power line</td>
<td>4,229,439</td>
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<tr>
<td>EPCM</td>
<td>2,753,265</td>
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<tr>
<td>Freight Taxes and Insurance</td>
<td>738,877</td>
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<tr>
<td>Inventory and Commissioning</td>
<td>1,349,745</td>
</tr>
<tr>
<td>Owners Costs</td>
<td>3,022,500</td>
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<tr>
<td><strong>Sub Total Startup Capex</strong></td>
<td><strong>42,232,377</strong></td>
</tr>
<tr>
<td>Contingency, 10%</td>
<td>3,716,401</td>
</tr>
<tr>
<td><strong>Total Startup Capex</strong></td>
<td><strong>45,948,779</strong></td>
</tr>
<tr>
<td>Startup Capex, USD/ore t processed</td>
<td>13.07</td>
</tr>
</tbody>
</table>
Table 1.4: LOM Sustaining Capital Estimate

<table>
<thead>
<tr>
<th>Sustaining Capex</th>
<th>LOM Capex, USD</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process Plant</td>
<td>1,484,581</td>
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<tr>
<td>Tailings Storage Facility (TSF)</td>
<td>5,248,294</td>
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<tr>
<td>Mining</td>
<td></td>
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<tr>
<td>Mining Development</td>
<td>16,599,077</td>
</tr>
<tr>
<td>Mining Equipment</td>
<td>8,450,907</td>
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<td>Mining Services</td>
<td>2,682,431</td>
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<tr>
<td>Mining Backfill System</td>
<td>1,318,427</td>
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<tr>
<td>Site Development</td>
<td>10,000</td>
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<tr>
<td>EPCM</td>
<td>258,119</td>
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<tr>
<td>Freight Taxes and Insurance</td>
<td>670,074</td>
</tr>
<tr>
<td>Inventory &amp; Commissioning</td>
<td>155,947</td>
</tr>
<tr>
<td>Owner Costs</td>
<td>3,332,000</td>
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<tr>
<td><strong>Sub Total Sustaining Capex</strong></td>
<td><strong>40,209,855</strong></td>
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<tr>
<td>Contingency</td>
<td>929,100</td>
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<tr>
<td><strong>Total Sustaining Capex</strong></td>
<td><strong>41,138,956</strong></td>
</tr>
<tr>
<td>Total Sustaining Capex, USD/ore t processed</td>
<td>11.7</td>
</tr>
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</table>

1.8.2 Operating Costs

Operating costs (OPEX) were estimated in constant 2Q USD. The LOM operating cost is USD 54.9/ton of ore processed. The OPEX includes costs for mining, processing and general and administration. Table 1.4 presents a summary of the life-of-mine total operating costs.

Table 1.5: LOM Total Operating Costs

<table>
<thead>
<tr>
<th>Total OPEX Cost</th>
<th>LOM OPEX, USD</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining</td>
<td>107,768,344</td>
</tr>
<tr>
<td>Processing</td>
<td>53,464,419</td>
</tr>
<tr>
<td>General and Administration</td>
<td>31,729,025</td>
</tr>
<tr>
<td><strong>Total OPEX Costs</strong></td>
<td><strong>192,961,787</strong></td>
</tr>
<tr>
<td>OPEX, USD/ore t mined</td>
<td>54.2</td>
</tr>
<tr>
<td>OPEX, USD/ore t processed</td>
<td>54.9</td>
</tr>
</tbody>
</table>
1.9 Financial Analysis

A Technical-Financial Model (TFM) has been developed to evaluate the economic viability of Pulacayo mine. The key assumptions and results for the study are shown in table 1.5.

<table>
<thead>
<tr>
<th>Financial Analysis</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Internal Rate of Return (IRR), pre-tax</td>
<td>47.1%</td>
</tr>
<tr>
<td>Average annual cash flow, pre-tax</td>
<td>$ 27.0 million</td>
</tr>
<tr>
<td>Annual pre-tax cash flow at production maturity</td>
<td>$ 39.3 million</td>
</tr>
<tr>
<td>Annual after-tax cash flow at production maturity</td>
<td>$ 24.0 million</td>
</tr>
<tr>
<td>Net Present Value (NPV@8%), pre-tax</td>
<td>$ 126 million</td>
</tr>
<tr>
<td>Internal Rate of Return (IRR), after-tax</td>
<td>32.1%</td>
</tr>
<tr>
<td>Average annual cash flow, after-tax</td>
<td>$ 17.8 million</td>
</tr>
<tr>
<td>Net Present Value (NPV8%), after-tax</td>
<td>$ 72.1 million</td>
</tr>
<tr>
<td>Silver price assumption</td>
<td>$ 28/oz</td>
</tr>
<tr>
<td>Lead price assumption</td>
<td>$ 0.86/lb</td>
</tr>
<tr>
<td>Zinc price assumption</td>
<td>$ 1.00/lb</td>
</tr>
<tr>
<td>After tax capital payback period</td>
<td>3.9 years</td>
</tr>
<tr>
<td>NSR ($/ton Milled)</td>
<td>$171/t</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Capital Costs</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Pre-production capital</td>
<td>$ 45.9 million</td>
</tr>
<tr>
<td>Maximum cash funding (incl. working capital)</td>
<td>$ 55.4 million</td>
</tr>
<tr>
<td>Sustaining capital (Life of Mine)</td>
<td>$ 41.1 million</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Operating Costs (Average over the life of mine)</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining ($/t milled)</td>
<td>$30.65/t</td>
</tr>
<tr>
<td>Processing ($/t milled)</td>
<td>$15.21/t</td>
</tr>
<tr>
<td>G &amp; A ($/t milled)</td>
<td>$9.02/t</td>
</tr>
<tr>
<td>Mine operating cost ($/oz milled)</td>
<td>$54.88/oz</td>
</tr>
<tr>
<td>Cash operating cost ($/ozAgEq.)</td>
<td>8.44/oz</td>
</tr>
<tr>
<td>Cash operating cost ($/oz)</td>
<td>11.20/oz</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Production Data</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>LoM - Life of Mine (@ 1,000tpd production)</td>
<td>12.5 years</td>
</tr>
<tr>
<td>Number of years at steady state (1,000tpd)</td>
<td>9 years</td>
</tr>
<tr>
<td>Ore tonnes milled</td>
<td>360,000 tpa / 1,000 tpd</td>
</tr>
<tr>
<td>LoM metallurgical recovery (silver)</td>
<td>86.3%</td>
</tr>
<tr>
<td>LoM metallurgical recovery (lead)</td>
<td>85.6%</td>
</tr>
<tr>
<td>LoM metallurgical recovery (zinc)</td>
<td>85.8%</td>
</tr>
<tr>
<td>Silver produced over the LoM</td>
<td>19.5 million oz</td>
</tr>
<tr>
<td>Lead produced over the LoM</td>
<td>67.021 t_{Znconc} / 70.9 M lbs_{Znmetal}</td>
</tr>
<tr>
<td>Zinc produced over the LoM</td>
<td>104.903 t_{Znconc} / 117.9 M lbs_{Znmetal}</td>
</tr>
<tr>
<td>Average annual silver produced (ozAgEq. steady state production)</td>
<td>2.56 million ozEquiv.</td>
</tr>
<tr>
<td>Average annual silver produced (oz steady state production)</td>
<td>1.94 million oz.</td>
</tr>
<tr>
<td>Average annual equivalent silver produced (oz Ag Eq over LOM)</td>
<td>2.11 million ozEquiv</td>
</tr>
<tr>
<td>Average annual silver produced (oz over LoM)</td>
<td>1.63 million ozEquiv</td>
</tr>
</tbody>
</table>
Other than the metal price, the project does not seem to be sensitive to any specific parameter. Figure 22.1 depicts the sensitivity of NPV and IRR to the Silver Price.

1.10 Conclusions and Recommendations

TWP concludes that the Pulacayo Project is technically and economically viable under the conditions stipulated in the FS and recommends that the project proceed with the basic/detailed engineering stage. An investment of $US 45.9 million will yield a post-tax NPV of $71.2 million at an 8% discount rate over an operating life of 12.5 years in January 2013 money terms.

Although the existing mineral resources at Pulacayo could support a significantly higher production profile, it was the Company’s preference at this time to build a robust underground operation with a conservative footprint and reduced capital cost. In this way, technical execution risk is reduced while allowing subsequent growth to take place at a pace that respects the needs and concerns of local communities. The initial mining scenario forms the foundation for future production growth, and could provide the Company with the opportunity to fund future expansion from internally generated cash flow.

Opportunities to further improve economic value of the Pulacayo Project include the consideration of the existing open pit resources in the study, updating the resources and reserves through additional drill campaigns, both at Pulacayo and the Paca deposit immediately to the north, optimization of the mine schedule and additional optimization of process plant design after additional test work on the lower grade silver ores once they are accessible through mine development, the incorporation of the open pit resources at Pulacayo into the mining plan, incorporation of the adjacent Paca resource, situated 6km to the north of Pulacayo into the mining plan.
INTRODUCTION

Apogee Silver Ltd. (Apogee) is an international exploration and development company with head office in Toronto with a strategic focus on advanced stage silver, zinc, and lead deposits in world class mineral districts in South America. Apogee is listed on the TSX Venture Exchange under the symbol, “APE”.

In June 2012, Apogee commissioned TWP Sudamérica to conduct a Bankable Feasibility Study (FS) on the viability of mining the deposit from underground and processing ore in a 1,000 t/d facility on its primary focus property “Pulacayo" to produce lead – silver and zinc – silver concentrates. The property is located in south western Bolivia as shown in Figure 2.1.

![Figure 2.1: Pulacayo Property Location](image-url)
2.1 Terms of Reference

This report is to comply with disclosure and reporting requirements set forth in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 23-101). This independent technical report provides information in support of the Company’s press release dated 17 January 2013 (press released by Apogee titled “Apogee Silver’s Pulacayo Project Demonstrates Positive Feasibility Study Results”.

No author of this report has any material interest in Apogee or related entities. This report is prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

2.2 Qualified Persons

Qualified Persons (QPs) responsible for the content of this technical report are:

- Ryan Illingworth, Pr. Eng, TWP Sudamerica, Technical Director
- Michael Cullen, P.Geo; M.Sc., Mercator Geological Services Limited, Chief Geologist
- Jim Porter, FSAIMM, JPMC, Owner and Principal Mining Consultant
- Graeme Farr, FSAIMM, TWP RSA, Senior Process Consultant
- Mark Smith, P.E., G.E., S.E., RRD International Corp, President
- Peter Webster, P Geo, Mercator Geological Services Limited
- Eugene Puritch, P. Eng., P & E Mining Consulting INC.

Specific sections of the report that the QPs are responsible for are provided in Table 2.1.

<table>
<thead>
<tr>
<th>Name</th>
<th>Responsibility</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ryan Illingworth</td>
<td>Chapter 1 Summary</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 2 Introduction</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 3 Reliance on Other Experts</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 4 Property Description and Location</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 5 Accessibility, Climate, Local Resources, Infrastructure and Physiography</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 6 History</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 19 Market Studies and Contracts</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 20 Environmental, Permitting and Social or Community Impact – Arvind Dalpatram will sign off on the water study section</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 21 Capital and Operating Costs</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 23 Adjacent Properties</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Chapter 24 Other Relevant Data and Information</td>
<td></td>
</tr>
</tbody>
</table>
2.3 Site Visits and Scope of Personal Inspection

From 06 June to 08 June 2012, a site visit was held to review the project layout, platforms for process plant and surface infrastructure, the underground mine, the metallurgical/chemical laboratory and also to carry out a field reconnaissance for the prospective tailings storage facility and location. Mining, processing, environmental and waste disposal aspects were also covered during the visit by TWP.

From 29 May to 30 May 2012, Jim Porter visited the Pulacayo underground mine to review the mining conditions for the project. Mark Smith took part on another site visit and reviewed the layout and terrain conditions to outline the location and technology to be used for the plant tailings storage facility (TSF) on 12 June 2012.

On 15 May and 16 May 2012, a site visit was undertaken by Michael Dabiri and Lisbeth Pimentel from Kohln Crippen Berger (KCB). Arvind Dalpatram did not visit the site.

Peter Webster and Mr. Harrington (resource geologist) from Mercator visited the site (period from 02 August to 11 August 2012) and completed a review of Apogee drill program components, including protocols for drill core logging, storage, handling, sampling and security. An independent core check sampling program was completed by KCB, drill sites were visited and various trench and channel sampled bedrock exposures were examined.

Michael Cullen (Mercator) carried out a site visit during the period 26 April to 28 April 2012 to complete technical reviews plus a core check sampling program with respect to the 2011 oxide zone drilling by Apogee. In both cases, Mercator staff were accompanied by Mr. Chris
Collins, Apogee President, and met with Apogee Exploration Manager, Mr. Hernán Uribe Zeballos.

From 24 October to 26 October, TWP Sudamérica conducted a field visit to review the underground workings, ventilation shafts, underground services and maintenance facilities to obtain information in support of the underground mine engineering study.

Ryan Illingworth visited the Pulacayo site on the 13th of April 2011.

2.4 Effective Dates

The effective date of this report is 17 January 2012.

There were no modifications on the processing route, engineering design, mine design, capital and operating costs and financial model between the signature date of this report and the publication of the Press Release announcing positive Feasibility Study results dated on 17 January 2013 titled: “Apogee Silver’s Pulacayo Project Demonstrates Positive Feasibility Study Results”.

2.5 Information Sources and References

In addition to site visits undertaken to the Pulacayo Project in Bolivia, the authors of this report have relied on information provided by Apogee, discussions with John Grewar (Consulting Metallurgist) and a number of studies completed by other internationally recognized independent consulting and engineering groups. The authors have made all reasonable enquiries to establish the completeness and authenticity of the information provided. TWP has also relied upon other experts. A full listing of the principal sources of information is included in Section 27 of this report.

2.6 Previous Technical Reports

Apogee has previously filed the following technical reports for the Project as follows:

- Revised Mineral Resource Estimate, Technical Report for the Pulacayo Ag-Pb-Zn Deposit, Pulacayo Township, Potosí District, Quijaro Province, Bolivia (Webster, P., Cullen, M., Mercator Geological Services Limited, 23 May 2012)

3 RELIANCE ON OTHER EXPERTS

The QPs state that they are qualified persons for those areas as identified in the appropriate QP “Certificate of Qualified Person” attached to this Report. The authors have relied upon and disclaim responsibility for information derived from the reports, opinions and statements of other experts, most of whom are not qualified persons, pertaining to mineral concession tenure, surface rights agreements, permitting, environmental, social impacts and political issues. All reasonable endeavors have been made to ensure the accuracy and
reasonableness of the information supplied by other experts. No warranty or guarantee, be it express or implied, is made by TWP with respect to the completeness or accuracy of such information provided.

While exercising all reasonable diligence in checking, confirming and testing data pertaining to the mineralization found on the Pulacayo project located in Pulacayo Township, Potosí District, Bolivia. TWP and its authors have relied upon data provided by Apogee public domain documents and its consultants, and has drawn its own conclusions therefrom. TWP has not carried out any independent exploration work, drilled any holes or carried out any sampling and assaying other than described in this report.

The status of the mineral concessions under which Apogee holds title to the surface and mineral rights for these properties has not been investigated or confirmed by TWP, and TWP offers no legal opinion as to the validity of the mineral title claimed by Apogee.

The description of the property, and ownership thereof, as set out in this report, is provided for general information purposes only. As well, the substance of the various option agreements has not been investigated or confirmed by TWP, and TWP offers no legal opinion as to the validity of the terms set out there in. The essential terms of these agreements outlined in this report are provided for general information purposes only.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Pulacayo prospect is located 18 km east of the city of Uyuni (Canton of Pulacayo, Quijarro Province) in the Department of Potosí in south western Bolivia, 460 km south east of the capital city, La Paz, and 130 km south west of Potosí, the department capital (Figure 4.1). The property is located at 740 450 mE and 774 695 mN WGS84 Zone 19, south datum, and at an elevation of 4,305 m ASL.

Pulacayo is accessible by a good, newly paved highway from La Paz via Oruro and Potosí (715km). A shorter alternative route to the mine exists from Oruro via Challapata and Uyuni also exists reduces the distance by 150 km, includes a 199km unpaved section. Unpaved road sections are generally passable during the whole year although they may present some level of difficulty during the rainy season. The tourist town of Uyuni, on the edge of the large Salar de Uyuni (salt lake) provides limited local services. The town has unpaved road and railway connections with the cities of Oruro, Potosí, Villazon, and to the borders with Argentina and Chile.

Uyuni has a newly developed airport, in operation since 2011; with an asphalt runway, which can now accommodate turbo props and regional jet service (only small aircraft are currently under operation, with less than 40 passengers). Daily flights are available from La Paz with an average flying time of one hour.
There are also several small hotels, hostels, restaurants, schools, medical and dental facilities and internet cafes. San Cristóbal Mining Company has constructed a gravel road from San Cristóbal, approximately 100 km south west of Uyuni, to the border with Chile.

4.2 Property Status

The following is an excerpt from Pressacco et al. (2010) that outlines the mineral title position and governance regime applicable to the Pulacayo property at the effective date of this report.

4.2.1 Overview of Bolivian Mining Law

The granting of mining concessions in Bolivia is governed by the Constitution (Constitución Política del Estado), the Mining Code (Código de Minería) enacted by Law No. 1 777 of March 17, 1997, supplemented by certain Supreme Decrees that rules taxation, environmental policies, and administrative matters, etc.

Ground and underground resources are from original domain of the Bolivian state and the resources can be granted for its exploitation; but the Bolivian state is prohibited to transfer them, according to the Article 349.1 of the Constitution.

Bolivian or foreign companies or individual persons may have mining concessions; with the exception of minors, governments agents, armed forces member, policemen, as well as relatives of the mentioned persons, etc., according to the Second Chapter of the First Title of the First Book of the Mining Code (Articles 16 to 23).
Foreigners, according to the Article 262.I of the Constitution and Article 17 of the Mining Code are not authorized to own mining concessions or real estate property within a buffer zone of 50 km surrounding the Bolivian international borders, but they are authorized to enter into Joint Venture agreements on the frontier regions.
Once the mining concession has been obtained, the title holder is able to explore and exploit the minerals within the mining concession including tailings and residual material. To retain the rights to the concession, the title holder must maintain the concession in good standing through the annual “patentes” payment, the cost of the “patentes” per “cuadrícula” being about USD 40.00 per year (USD 80.00 if the concession has more than five years). If the title holder continues to make the “patentes” payment on time, the mining concession is indefinite, according to current legislation.

“Cuadrícula” is the mining measure unit, which is an inverted pyramid with the inferior vertex pointing to the earth’s core, with an exterior perimeter equal to 25 hectares. Some existing mining concessions have been applied for and granted according to the system governed by the old Mining Code, which has not been in effect since 1997. However, the concessions are totally legal. The measure unit of the mining concessions obtained according to the old Mining Code system is the “pertenencia minera”, which is an inverted pyramid with the inferior vertex pointing to the earth’s core, with an exterior perimeter equal to one hectare.

Mining concessions cannot be transferred, sold or mortgaged. Joint Venture agreements are permitted.

The Bolivian Constitution passed and enacted in February 2009, on its article eighth section 3, of the Transitory Provisions Chapter, establishes that all the mining concessions must be adapted to the new constitutional regime and then must be converted to “mining contracts” in a term of one year from December 6th 2009.

Since 06 December 2010, a new Mining Law compliant with the Constitution enacted in February 2009 was not in effect, the Bolivian government issued the Supreme Decree No. 726/2010. The mentioned Decree on its first article establishes that all the mining concessions (while the new Mining Law is being prepared and then enacted) will have the category of “Autorizaciones Transitorias Especiales” or Special Transitory Authorizations. The second article of the Supreme Decree No. 726/2010 also establishes that the pre-established rights of the Special Transitory Authorizations are guaranteed.

The Codigo de Mineria (1997) is available in an official Spanish-English side-by-side version, which facilitates understanding the Bolivian mining code. Key features are:

- There is only one type of mining license, a “La Concesion Minera” (currently known as “Special Transitory Authorization STA”), which is comprised of 25 ha units, named “cuadrícula minera”. A maximum of 2,500 units is allowed for a mining concession.
- There is no limitation to the number of concessions that can be held by a company or an individual.
- Field staking is not required; concessions are applied for on 1:50 000 scale base maps.
- The concessionaire has exclusive rights to all minerals within the STA.
- If the title holder continues to make the “patentes” payment on time the term of the mining concession is indefinite.
- Mining concessions cannot be transferred, sold or mortgaged.
• Provision is made for surface access, compensation and arbitration with private land owners, if any. (NB: private ownership of surface lands outside of major cities is limited).

• Mining concessions, both “cuadrículas” and “pertenencias” must have their “Título Ejecutorial” registered with the “Mining Registry” that is part of the SERGEOTECMIN and before the Real State Registration Office.

• Simultaneous with the introduction of the new mining code in 1997 were a number of taxation reforms. Bolivian taxes are now fully deductible by foreign mining companies under US corporate income tax regulations.

Taxes applicable are:

• Mining Royalty (Regalía Minera) equivalent to 1% to 7% of the gross sales value of the mineral. The tax is paid before the mineral is exported or sold in the local market (in this case, only 60% of the tax is paid).

• Profits tax of 25% on net profits [Gross income – (expenses plus costs)]; losses can be carried forward indefinitely. An additional 12.5% is paid when metals/minerals reach extraordinary market prices.

• Mineral production is subject to a Value Added Tax of 13%.

The Ministry of Mining and Metallurgy is responsible for mining policy. Servicio Geológico y Técnico Minero de Bolivia (SERGEOTECMIN) – the Bolivian Geological Survey, a branch of the Ministry, is responsible for management of the mineral titles system. SERGEOTECMIN also provides geological and technical information and maintains a USGS-donated geological library and publications distribution center. In addition, tenement maps are available from SERGEOTECMIN, which has a GIS based, computerized map system.

Exploration and subsequent development activities require various degrees of environmental permits, which various company representatives have advised are within normal international standards. Permits for drill road construction, drilling and other ground disturbing activities can be readily obtained in 2 – 4 months, or less, upon submission of a simple declaration of intent and plan of activities."

4.2.2 Project Ownership

Details of property ownership of the Pulacayo project properties are complicated by multilayered option and joint venture agreements. Apogee Minerals Ltd. (renamed Apogee Silver Ltd. in March 2011) currently controls 100% of the Pulacayo Ag-Pb-Zn deposit through an agreement with Golden Minerals Company (GMC), the successor of Apex Silver Company. Golden Minerals Company’s Bolivian subsidiary, ASC Bolivia LDC (“ASC”), holds the mining rights to the concessions through a series of option and lease agreements with the Pulacayo Mining Cooperative and COMIBOL, The Mining Corporation of Bolivia. On 21 January 2011, Apogee entered into a definitive agreement with GMC to acquire all of the issued share capital of an indirectly held subsidiary of GMC known as ASC, which holds a 100% interest in the Pulacayo Project.

Pursuant to the agreement, Apogee acquired all of the issued and outstanding shares of a subsidiary from GMC. In consideration, Apogee issued 5 000 000 common shares of the
Company upon closing of the transaction and issued an additional 3,000,000 Common Shares and shall pay GMC a cash fee for USD 500,000 eighteen (18) months following closing of the transaction.

Mr. Gustavo A. Miranda Pinaya, Executive President and in-house legal counsel for Apogee Minerals Bolivia S.A. provided Mercator with the following property ownership report, dated August 26, 2011, that details Concession, Lease and Joint Venture agreements that pertain to the company’s involvement with the Pulacayo and Paca properties. On 23 May 2012, Mr. Gustavo A. Miranda Pinaya also confirmed that conditions described in this report remained in place at the effective date of this report. Mercator has relied upon this information for report purposes and has not independently verified related content.

Comibol/Pulacayo Ltda. Lease Agreement

- The Bolivian Mining Corporation (COMIBOL) and the Pulacayo Ltda. Mining Cooperative have executed a Lease Agreement (Contrato de Arrendamiento) according to the “Testimonio” No. 235/97 dated 08/01/1997 granted by the Mining and Petroleum Special Notary from La Paz (María Esther Vallejos). The Testimonio was registered before Mining Registry under the No. 253 Book “B” dated 09/05/1997. There is no reference on the “Testimonio” about the registration before the Potosí Real State Office. However, the amendment referred ahead was duly registered which implies that the Lease Agreement was registered.

- The lease contract includes the mining concessions — Pulacayo (1,031 hectares), Porvenir (1,099), Huanchaca (460 hectares), Galería General (76 hectares) and Rochschild (3 hectares). Santa Bárbara (149 hectares), La Esperanza (148 hectares), Flora (60 hectares) and Victoria (40 hectares). Cholita (10 hectares) and Tolentino (220 hectares). All the concessions (a total of 3,296 hectares) are owned by the COMIBOL.

- As described on the agreement, contained on the Testimonio 65/2002 dated 13 May 2002, granted by Notary Public No. 003, executed between the Pulacayo Cooperative and COMIBOL the mining concessions (property of COMIBOL) “Real del Monte” and “Temeridad” have been included on the agreement mentioned on previous paragraphs. Both concessions are located on the main area of PACA (where most of the drilling was performed).

- The term of the Lease Agreement is 15 years, starting June 1997, there is no “day” mentioned on the Testimonio, only the month and the year are described. The contract is valid until June 2012. The term could be extended.

- The Lease establishes a rent equal to 1% of the net production value. If the payment has a 3 month delay then the contract is terminated.

- The Scope of the Lease Agreement is the Mining, Development, Milling and Marketing of ore from Pulacayo, Ubina and Cholita Chaquiri areas in the Province Quijarro, Potosí District. Exploration is permitted under the Lease Agreement.

- An amendment to the aforementioned contract was executed between COMIBOL and the Pulacayo Ltda. Mining Cooperative according to the “Testimonio” No.
115/2002 dated 07/30/2002 granted by Public Notary from La Paz No. 003 (Nelly Alfaro de Maldonado).

- The Testimonio was registered before Mining Registry under the No. PT-195 File No. 195 dated 08/02/2002, and finally registered before the Potosí Real State Office under the Partida No. 61-26 File No. 50 vta.– 20 Book No. 8-49, dated 08/08/2002.

- The amendment extends the term of the Lease Contract until 23 June 2025 under the condition to execute a Joint Venture Agreement with a “strategic partner”. If the Joint Venture is not executed or is terminated then the term of the Lease Contract will return to the original term (June 2012).

- An exploration period of five years is allowed. During this period the “strategic partner” must pay USD 1 000 per month to COMIBOL as rent fee. A minimum investment of USD 300 000 in exploration costs is compromised, a USD 30 000 bond (Boleta de Garantía Bancaria) must be granted on behalf of COMIBOL to guarantee the investment during the exploration period.

- A third party could be part of the "Pulacayo/Strategic Partner” Joint Venture Agreement, but under permission from COMIBOL's Board, previous written request. COMIBOL's Board may deny permission. The third party must have a renowned name and capacity on the mining industry.

**Pulacayo Ltds./ASC Bolivia LDC (Sucursals Bolivia) Joint Venture Agreement**

- The Pulacayo Ltda. Mining Cooperative and ASC BOLIVIA LDC (Sucursal Bolivia) (a subsidiary of APEX) have executed a Joint Venture Agreement (Contrato de Riesgo Compartido) according to the “Testimonio” No. 116/2002 dated 07/30/2002 granted by Public Notary from La Paz No. 003 (Nelly Alfaro de Maldonado). The Testimonio was registered before Mining Registry under the No. PT-197 File No. 144 dated 08/02/2002, and finally registered before the Potosí Real State Office under the Partida No. 62-27 File No. 54 vta.– 24 Book No. 8-49, dated 08/12/2002.

- COMIBOL's Board through Board's Resolution No. 2594/2002 dated 25 July 2002 has authorized the execution of the Joint Venture Agreement.

- The Joint Venture Agreement only includes the Pulacayo Group of mining concessions: Pulacayo (1,031 hectares), Porvenir (1,099), Huanchaca (460 hectares), Galería General (76 hectares), Roschild (3 hectares), Temeridad (10 hectares) and Real del Monte (24 hectares).

- The term of the Joint Venture Agreement is 23 years starting 30 July 2002 the first five years are for exploration period.

- The Joint Venture Agreement could be terminated at any time if results from exploration are not satisfactory to ASC Bolivia LDC.

- ASC Bolivia LDC is committed to pay to COMIBOL USD 1 000 during the exploration period. During the mining period ASC Bolivia LDC will pay to
COMIBOL the equivalent of 2.5% of the Net Smelter Return (NSR) and 1.5% of the Net Smelter Return (NSR) to the Pulacayo Cooperative.

- The First Stage of Exploration implies the investment of USD 500,000 or at least a minimum of USD 300,000 during the sixth to eighth month.

- As defined on the clause 21 of the Joint Venture Agreement a third party could be integrated to the Joint Venture under permission of COMIBOL’s Board.

**ASC Bolivia LDC (Sucursal Bolivia)/Apogee Minerals Bolivia S.A. Option Agreement to be Part of a “Joint Venture Agreement”**

- Apogee Minerals Bolivia S.A. and ASC BOLIVIA LDC (Sucursal Bolivia) have executed an Option Agreement to be Part of a “Joint Venture Agreement” (Contrato de Opción para la Incorporación a un Contrato de Riesgo Compartido) according to the “Testimonio” No. 68/2006 dated 03/08/2006 granted by Public Notary from La Paz No. 038 (Daysi Benito Pozzo). The contract establishes the conditions that Apogee Minerals Bolivia S.A. must fulfill to vest 60% participation on the Pulacayo Cooperative/ASC Bolivia LDC Joint Venture Agreement and also to have 60% participation on the Paca Group of mining properties through a Joint Venture Agreement.

- The inclusion of Apogee Minerals Bolivia S.A. to the Pulacayo Cooperative/ASC Bolivia LDC Joint Venture Agreement was authorized by the Board of COMIBOL according to the Board Resolution dated 06 November 2005.

- The inclusion of Apogee Minerals Bolivia S.A. to the Pulacayo Cooperative/ASC Bolivia LDC Joint Venture Agreement was also approved by the Board of the Cooperative according to the letter dated 17 August 2005.

- The agreement includes the Pulacayo and the Paca Group of mining concessions. The Paca Group of Mining Concessions is property or is under the control of ASC Bolivia LDC.

- On 23 October 2007 and amendment to the Option Agreement to be Part of a “Joint Venture Agreement” was executed, extending the term of the mentioned agreement until 30 July 2009, and in consequence extending the deadline to prepare a Feasibility Study until such date.

- On 19 May 2009, COMIBOL has approved Apogee’s request to extend the deadline to deliver a Feasibility Study until 30 November 2009. In consequence the deadline mentioned on previous paragraph was extended to 30 November 2009.

- If the Feasibility Study establishes the existence of a critical mass of resources, ASC will have the chance to dilute Apogee’s participation from 60% to 40% if ASC in a term of 90 days notifies to Apogee that will develop Pulacayo to production.

- However, in order to avoid the disclosure of the commitments between Apogee Minerals Bolivia and ASC Bolivia LDC on regard of the incorporation of Apogee Minerals Bolivia S.A. to the Joint Venture Agreement, a simply separate
agreement was executed by Apogee Minerals Bolivia S.A. and ASC Bolivia LDC. The mentioned agreement just details the incorporation of Apogee Minerals Bolivia S.A. to the Joint Venture Agreement and specifies the approval given by COMIBOL and the Pulacayo Cooperative. The mentioned agreement is detailed on the “Testimonio” No. 21/2006 dated 01/24/2006 granted by Public Notary from La Paz No. 038 (Daysi Benito Pozzo) and registered before the Mining Registry under No. PT-16 dated 06/22/2006.

- On 20 January 2011, Golden Minerals Company ("Golden") entered into a Purchase and Sale Agreement (the "Purchase Agreement") with Apex Silver Mines. Apex Mining Partners Limited and Apogee Silver Ltd. ("Apogee") pursuant to which Apogee purchased through other Golden subsidiaries, all of Golden's interest in ASC Bolivia LDC, which holds a 100% interest in the Pulacayo Cooperative/ASC Bolivia LDC Joint Venture Agreement through ASC Bolivia LDC Sucursal Bolivia, the Bolivian branch of ASC Bolivia LDC.

- As consequence of the aforementioned and as described on the “Testimonio” No. 438/2011 dated 07/15/2011 granted by Public Notary from La Paz No. 051 (Katherine Ramirez Calderon) on 31 January 2011, Apogee Minerals Bolivia S.A. and ASC Bolivia LDC Sucursal Bolivia have executed an agreement separating Apogee Minerals Bolivia S.A. from the Pulacayo Cooperative/ASC Bolivia LDC Joint Venture Agreement. On the third clause of the mentioned agreement it has been established that Apogee Minerals Bolivia S.A. will continue with the exploration duties, even a small scale pilot production, until all the environmental permits have been issued on behalf of ASC Bolivia LDC Sucursal Bolivia. Current environmental permits are under the name of Apogee Minerals Bolivia S.A.

- As a consequence of the aforementioned, the Pulacayo Cooperative/ASC Bolivia LDC Joint Venture Agreement has returned to its original scheme with the Pulacayo Cooperative and ASC Bolivia LDC Sucursal Bolivia as parties.

The Figure 4.2 shows the distribution of Apogee licenses on the property.

4.2.3 Environmental Considerations

The project’s current environmental operating requirements are set out in compliance with the Environment Law (Law Nº 1333) and the Environmental Regulation for Mining Activities. A certificate of exemption has been obtained for the exploration phase and an audit of the Environmental Base Line (ALBA) was carried out between December 2007 and July 2008 by Mining Consulting and Engineering “MINCO S.R.L.”, a Bolivian based professional consulting firm with broad exposure to the mining industry. Its audit report summarizes the work carried out during the Environmental Assessment by Apogee and includes:
Figure 4.2: Outline of Mineral Concessions, Pulacayo Project

Note: Interest In Property 22494 was relinquished By Apogee In 2010.
A compilation of information on the local vegetation, animals, soil, water, air, etc., including collection of more than 500 samples in the area of interest to support the conclusions and recommendations of the report.

An evaluation of the social impact of the project.

An evaluation of the area contaminated during previous mining activities, including tailings, abandoned facilities, acid waters, scrap, etc.

An evaluation of other environmental liabilities.

On 25 May 2011, Apogee was awarded an environmental license by the Bolivian authorities sanctioning mining operations at its Pulacayo project. The permit (Certificado de Ispsensación Categoría 3 Para Exploración y Actividades Mineras Menores/EMAP) allows for the extraction of up to 200 tons per day from underground for stockpiling and transporting for off-site processing.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

Bolivia is a landlocked country located in central South America and includes diverse geographic and climatic conditions that range from snow-capped peaks and high altitude plateaus to vast, low-lying grasslands and rainforests. The country is accessible by international air travel from Miami (American Airlines), Mexico City, Brazil, Chile (LAN), Argentina and Peru (Taca).

In addition, local Bolivian airlines fly regular internal flights between major cities, with three flights a week to a newly paved runway at Uyuni city, which is located 18 km from the Pulacayo property.

The principal highways are generally paved and heavy trucks and buses dominate road traffic outside of the major cities. For the most part, road freight service functions adequately even to small remote villages.

The Pulacayo project is accessed from La Paz by means of a paved road, which runs to the area of Huari, passing through Oruro. It can also be accessed by the road between Oruro (gravel) and Potosí (paved) and from Potosí to Uyuni by a good quality gravel road. Paving of the road from Potosí to Uyuni began in 2007 and at the time of the 2011 site visit by Mercator was almost completed to Pulacayo. Secondary roads can be best described as “tracks” and winding, single lane roads are often precariously carved out of steep slopes.

There is also a reasonably well developed rail system with connections south to Argentina, east to Brazil and west to Chile and the port of Antofagasta. Rail service from Uyuni connects with Oruro, Atocha, Tupiza, and Villazon (on the border with Argentina). Uyuni is also connected by railway to Chile through Estación Abaroa. Disused rail lines exist between Uyuni-Potosí and Oruro-La Paz.
The Figure 5.1 presents major highway and rail routes of Bolivia relative to the Pulacayo project’s location.

5.2 Climate and Physiography

Two Andean mountain chains run through western Bolivia, with many peaks rising to elevations greater than 6,000 m. The western Cordillera Occidental Real forms Bolivia’s western boundary with Peru and Chile, extending south east from Lake Titicaca and then south across central Bolivia to join with the Cordillera Central along the country’s southern border with Argentina.

Between these two mountain chains is the Altiplano, a high flat plain system at elevations between 3,500 m and 4,000 m above sea level. East of the Cordillera Central a lower altitude region of rolling hills and fertile basins having a tropical climate occurs between elevations of 300 m and 400 m above sea level. To the north, the Andes adjoin tropical lowlands of Brazil’s Amazon Basin (Figure 5.1).

Climate within Bolivia is altitude related. The rainy period lasts from November to March and corresponds with the southern hemisphere’s summer season. Of the major cities, only Potosí receives regular snowfalls, with these typically occurring between February and April at the end of the rainy season. La Paz and Oruro occasionally receive light snow. On the Altiplano and in higher altitude areas, sub-zero temperatures are frequent at night throughout the year. Snow-capped peaks are present year round at elevations greater than approximately 5,200 m.

The Pulacayo project area is located immediately south west of the Cosuño Caldera and local topographic relief is gentle to moderate, with elevations ranging between 4,000 m and 4,500 m above sea level. The Paca and Pulacayo Domes are volcanic structures that exist as prominent topographic highs in this area. Pulacayo has a semi-arid climate, with annual rainfall of approximately 100 mm and a mean summer temperature of 12°C between October and March.

During winter, minimum temperatures reach the -20 to -25 degree C range and summer maximums in the 18 to 20°C range occur between June and July. The yearly mean temperature is 5.5°C.
5.3 Local Resources and Infrastructure

Bolivia has a long history as a significant primary producer of silver and tin, with associated secondary production of gold, copper, antimony, bismuth, tungsten, sulphur and iron. The
country also contains sizeable reserves of natural gas that have not been fully developed to date, due to export issues and limited access to required infrastructure.

Exploration in Bolivia by international companies has been minimal in recent years and Coeur d’Alene Mines Corporation (San Bartolome), Pan American Silver Ltd. (San Vicente), Glencore International plc (AR Zinc, Sinchi Wayra) and Apex Silver Mines Ltd - now Golden Minerals Company (San Cristóbal) have been the most significant international companies present in recent years. To a substantial degree, this reflects political instability and threatened changes to mining taxation. Basic exploration services are available in Bolivia and include several small diamond core drilling contractors, ALS Group, which operates a sample preparation facility in Oruro, SGS Group that has operations in La Paz, and several locally owned assay facilities.

The Bolivian National School of Engineering operates a technical college in Oruro (Universidad Técnica de Oruro) that includes a mineral processing department and laboratory facilities that provide commercial services to the mining industry. In general, an adequate supply of junior to intermediate level geologists, metallurgists, mining engineers and chemists is currently considered to be present in the country.

Approximately 600 people currently live in Pulacayo on a permanent basis and many are associated with the Cooperativa Minera Pulacayo Ltda. (Pulacayo Mining Cooperative). The village has a state-run school and medical services are provided by the state’s Caja Nacional de Seguros (National Insurance Fund). A hospital and clinic function independently. Numerous dwellings and mining related buildings in Pulacayo are owned by COMIBOL some of these have been donated to the Pulacayo Mining Cooperative. Under the Shared Risk Contract, COMIBOL makes some mining infrastructure available for use by Apogee (Figure 5.2).

5.3.1 Utilities

The country has an abundance of hydroelectric power and transmission lines, which parallel the road system, provide service to most major settlements. Remote villages generally have diesel generators, which run infrequently during evening hours. Transmission lines from the hydroelectric plants of Landara, Punutuma, and Yura that were reconditioned by a joint venture between COMIBOL and the Valle Hermoso Electrical Company and pass within a few km of Pulacayo.

An adequate supply of potable water for the town is supplied by pipeline from a dam and reservoir (Yana Pollera) facility located 28 km from Pulacayo in the Cerro Cosuño.

The project area lies between two climatic zones in the Andean region: the semi-humid Puna and the arid Puna. The wet season, when about 80% of the approximately 1.80 mm (average) annual precipitation occurs, is between December and March. Elevated wind velocity and solar radiation result in evapotranspiration rates of approximately 1,700 mm per year, of which the higher rates are encountered during the dry season of April to November. The discrepancy between magnitudes and seasonality of precipitation and evaporation result in a short period of excess water during the wet season, and water shortages during the dry season.
Currently, the main supply of water in the region is the Yanapollera reservoir. The Yanapollera reservoir stores water from the snowmelt, runoff and groundwater springs in the upstream Cosuño nevado (snow-capped mountain peak) and distributes them to local users. Storage and release of water in the nevado snowpack, the wetlands upstream of the reservoir and in the reservoir itself somewhat counteracts the acute nature of the seasonal runoff by providing a slow release of stored water during periods of little to no recharge (i.e., the dry season). Flow measurements taken in 2011 and 2012, supported by hydrologic modeling, indicate that inflows to the reservoir may range from 8 L/s to 30 L/s during the dry season, and as high as 2,600 L/s during the wet season.

At present, the reservoir has three users: a mining cooperative located in the Pulacayo town; the Pulacayo population and the town of Uyuni. The actual water demand of these users from Yanapollera is still somewhat uncertain, and their water demands are supplemented by groundwater and other sources. The water demand from Yanapollera has been estimated based on recorded flow values, as well as population and usage projections, as approximately 16,000 m$^3$/month for the town of Uyuni, and between approximately 2,000 m$^3$/month and 7,500 m$^3$/month for Pulacayo, which includes the water demand of the local mining cooperative.

Klohn Crippen Berger, S.A. developed a hydrologic model to evaluate the likelihood of a water deficit over the Life of Mine, considering climatic conditions as well as the demands of the mine and the other water users. This model accounts for variability of climate parameters and includes a sensitivity analysis on key input parameters to account for the lack of long-term site data normally recommended for model validation. The average water demand of the mine are estimated to be between 15.93 m$^3$/hour and 26.79 m$^3$/hour (values from TWP process water balance), depending on process conditions. The results of the model indicate that there is a probability of approximately 20% to 50% that the Yanapollera reservoir will not be able to consistently meet the demands of the mine and the other users during a given year.

The results of this work indicate that the Yanapollera reservoir alone should not be relied upon for process and potable water related to mine operation, as this would leave a statistically significant chance of a water deficit during the life of the mine. This probability can be reduced or eliminated by constructing a water storage reservoir, and/or by supplementing the water supply with groundwater sources. Investigation and design of the aforementioned items is currently underway.

5.3.2 Communications

Telephone service and internet access are available in most areas and cellular telephone service is widespread. However, coverage is not complete and international connectivity is not ensured.

Local communication services in the area are good and consist of an ENTEL-based long distance telephone service, a GSM signal for cell phones and two antennae for reception and transmission of signals from national television stations. Apogee has installed a satellite receiver to provide internet access for its operation and this service is shared with the Cooperative Social del Riesgo Compartido (Shared Risk Cooperative).
6 HISTORY

6.1 Introduction

The following description of mining history is modified after Pressacco et al. (2010) and is directly based on an internal Apogee report written in Spanish. A more detailed description of exploration history is presented in Spanish by Iriondo et al. (2009). Report section 5.2 is presented without change from Cullen and Webster (2012).

6.2 Summarized Exploration History

Mining of silver deposits at Pulacayo began in the Spanish Colonial Period (c. 1545) but production details do not exist. The first work formally recorded on the property was carried out in 1833 when Mariano Ramírez rediscovered the Pulacayo deposit. In 1857 Aniceto Arce founded the Huanchaca Mining Company of Bolivia with support of French investors and subsequently pursued development and production at Pulacayo. Revenue from the mine funded the first railway line in Bolivia which in 1888 connected Pulacayo to the port of Antofagasta, Chile. In 1891, reported annual silver production reached 5.7 Million ounces and mining operations at Pulacayo at that time were the second largest in Bolivia.
Highest production is attributed to the nearby Cerro Rico de Potosi deposit. Pulacayo production was predominantly from the rich Veta Tajo (Tajo Vein System) which had been defined along a strike length of 2.5 km and to a depth of more than 1000 meters. In 1923, mining operation ceased due to flooding of the main working levels.

In 1927, Mauricio Hochschild bought the property and re-started mine development. The Veta Cuatro vein was the focus of this work and was intersected at a mine elevation of approximately -266 m. It was proven to continue down-dip to the -776 m elevation where it showed a strike length of 750 m. During this time, the 2.8 km long San Leon access tunnel was developed to facilitate ore haulage and the first recorded exploration work in the area was undertaken.

Work continued through the intervening years, and in 1952, the Bolivian government nationalized the mines and administration of the Pulacayo deposit and management was assumed by the state mining enterprise COMIBOL. Operations continued under COMIBOL until closure in 1959 due to exhaustion of reserves and rising costs. COMIBOL also imposed cutbacks on exploration at this time.

The total production from the Pulacayo mine during this period as estimated by the National Geological and Mineral Service of Bolivia (SERGEO TECHMIN) is 678 Million ounces of silver, 200,000 tons of zinc and 200,000 tons of lead (SERGEO TECHMIN Bulletin No. 30, 2002, after Mignon 1989).

In 1962, a local cooperative group named Cooperativa Minera Pulacayo (the “Cooperative”) was founded and this group leased the Pulacayo mine from COMIBOL. The Cooperative has operated small scale mining in the district since that time and continues to do so at present.

Efforts are directed toward exploitation of narrow, very high grade silver mineralization in upper levels of the old mining workings, above the San Leon tunnel level.

Modern exploration of the Pulacayo area began to a limited degree in the 1980’s when various mining and exploration companies targeted epithermal Ag and Au mineralization within the volcanic-intrusive system of the Pulacayo area. In 2001, ASC initiated an exploration program in the district and signed agreements with the Cooperative and completed regional and detailed geological mapping, topographic surveying and sampling of historical workings. Subsequently ASC completed three drill campaigns at Pulacayo, totaling 3,130 m of diamond drilling, and concluded that Ag-Pb-Zn mineralization and hydrothermal alteration in the district are controlled by a strong east-west fracturing system developed in the andesitic rocks hosting the Tajo Vein.

Apogee acquired the property in 2006 under option from ASC and since then, has actively pursued exploration and economic assessment of the property. Details of Apogee programs carried out since 2006 are presented in chapter 10.0 of this report. [See corresponding chapter].
7 GEOLOGICAL SETTING

7.1 Regional Geology

The regional geology of Bolivia is well described in various Bolivian government reports and is summarized in technical reports completed for Apogee by Micon in 2008, 2009, and 2010. The following regional geology description was extracted from the 2010 Micon report (Pressacco et al., 2010) that in turn is based on translated content from an Apogee company report (Iriondo et al. 2009) and US Geological Survey Bulletin (1975). The following text is presented without change from Cullen and Webster (2012).

“In southwest Bolivia, the Andes Mountains consist of three contiguous morphotectonic provinces, which are, from west to east, the Cordillera Occidental, the Altiplano, and the Cordillera Oriental. The basement beneath the area, which is as thick as 70 km, is believed to be similar to the rocks exposed immediately to the east, in the Cordillera Oriental, where a polygenic Phanerozoic fold and thrust belt consists largely of Paleozoic and Mesozoic marine shales and sandstones (Figure 7.1). Deposited mostly on Precambrian basement, the rocks of the Cordillera Oriental were deformed during at least three tectonic-orogenic cycles, the Caledonian (Ordovician), the Hercynian (Devonian to Triassic), and the Andean (Cretaceous to Cenozoic). The Altiplano is a series of high, intermontane basins that formed primarily during the Andean cycle, apparently in response to folding and thrusting. Its formation involved the eastward underthrusting of the Proterozoic and Paleozoic basement of the Cordillera Occidental, concurrent with the westward overthrusting of the Paleozoic miogeosynclinal rocks of the Cordillera Oriental. These thrusts resulted in continental foreland basins that received as much as 15,000 m of sediment and interlayered volcanic rocks during the Cenozoic.

Igneous activity accompanying early Andean deformation was primarily focused further west, in Chile. During the main (Incaico) pulse of Andean deformation, beginning in the Oligocene and continuing at least until the middle Miocene, a number of volcano-plutonic complexes were emplaced at several localities on the Altiplano, particularly along its eastern margin with the Cordillera Oriental, and to the south.

In Pleistocene time, most of the Altiplano was covered by large glacial lakes. The great salars of Uyuni and Coipasa are Holocene remnants of these lakes. The Cordillera Occidental consists of late Miocene to Recent volcanic rocks, both lava flows and ash flow tuffs, primarily of andesitic to dacitic composition, that have been erupted in response to the subduction of the Nazca plate beneath the continent of South America. This underthrusting continues, and many of the volcanoes that form the crest of the Andes and mark the international border with Chile are presently active”. 
### 7.2 Local Geology

As described earlier, Pulacayo is a low sulphidation epithermal polymetallic deposit hosted by sedimentary and igneous rocks of Silurian and Neocene age (Pressacco et al., 2010). The Silurian sediments underlie the volcanic rocks and include diamictites, sandstones and shales. The Neocene rocks are predominantly volcano-sedimentary in origin and include
conglomerates, sandstones, rhyolitic tuffs, dacitic-rhyolitic domes, andesitic porphyries and andesitic flows.

The Pulacayo project is located on the western flank of a regional anticline that affects sedimentary and igneous rocks of Silurian, Tertiary and Quaternary ages on the western flank of the Cordillera Oriental, near the Cordillera-Altiplano boundary. Figure 7.2 presents an interpretation of local geology and the structures and features discussed below are considered to be particularly important with respect to localization of mineralization in the Pulacayo district.

The Uyuni-Khenayani Fault is a reverse fault, which is believed to have controlled localization of volcanic center complexes at Cuzco, Cosuño, Pulacayo and San Cristóbal and related mineralized areas at Pulacayo, Cosuño, El Asiento, Carguaycollu and San Cristóbal. This fault brings Tertiary sediments in contact with Paleozoic formations at surface and is located about 4 km west of Pulacayo (Figure 7.2 and Figure 7.3).

Figure 7.2: Regional Geology Map (From Iriondo Et Al., 2009)
The mineralized zones at Pulacayo, Paca Mayu and Paca all occur on the west flank of a north-south striking anticline primarily comprised of Silurian sediments overlain by Tertiary lacustrine formations. Local topographic highs define Lower Miocene dacitic-andesitic...
domes and stocks associated with caldera resurgence that intrude the folded section. A younger Miocene-Pliocene phase of volcanism is also superimposed on the anticlinal trend and is marked by pyroclastic deposits and flows of andesitic and rhyolitic composition. Ignimbrites associated with the Cosuño Caldera are the youngest volcanic deposits in the area. A dacitic to andesitic dome complex at the Pulacayo property intruded the folded sediment section and forms the main topographic highs that occur on the property (Figure 7.3).

7.3 Structure

The Pulacayo, Paca Mayu and Paca volcanic dome complexes occur along a north-south corridor defined by two parallel, north-south trending regional faults that are separated by about 2.7 km. The domes occur over an interval measuring approximately 10 km in length and polymetallic vein and wall rock mineralization at Pulacayo is controlled by east-west trending secondary faults that cut Tertiary sediments and volcanic rocks of the Pulacayo dome (Figure 7.4). The stock work vein system was emplaced on the southern side of the Pulacayo dome complex and is best exemplified by the Tajo Vein System (TVS). The TVS bifurcates in andesitic rocks to form separate veins that collectively form a dense network or stock work of veinlets along strike. The bifurcating, polymetallic veins are commonly separated by altered andesitic rock that contains disseminated sulphide mineralization.

Figure 7.4: Structural Interpretation For Tajo Vein System At Pulacayo (Iriondo Et Al.,2009)

The TVS is almost 2,700 m in strike length at surface and is still present at a depth of 1,000 m below surface, the lowest level in the underground mine. In the upper levels of the mine, the stock work vein system locally reaches approximately 120 m of mineralized width. The polymetallic veins exhibit a sigmoidal geometry along strike, which is generally
interpreted to be the result of sinistral movement along the two north-south oriented bounding faults mentioned earlier (Figure 7.4).

7.4 Alteration

Wallrock alteration is spatially associated with the main vein system trends at Pulacayo and includes propylitic, sericitic, moderate-advanced argillic, and siliceous assemblages. Host rock composition exerts a strong local influence on both the nature of alteration assemblages present and their relative intensity of development. On this basis, spatial distribution of hydrothermal alteration assemblages in the district is a useful indicator of proximity to mineralized structures. Moderate argillic alteration is observed throughout the Pulacayo area and transitions to intense argillic alteration in close proximity to veins and disseminated-stock work zones. Haloes of silicification are developed around vein contacts and measure up to several cm in width in some cases. Silicification grades into advanced argillic alteration as distance into the wall rock increases from the vein contact and this gradually grades to argillic and propylitic zones with greater distance from the contact.

7.5 Mineralization

As referenced by Pressacco et al. (2010) the Pulacayo deposit is considered an example of a sub-volcanic epithermal mineralization system showing well developed vertical metal zonation. The main mineralized vein and stock work system developed on the southern flank of a dacitic intrusive dome and shows a surface strike dimension of 2 700 m. At Pulacayo, east-west striking faults are interpreted to have acted as a conduit system for mineralizing fluids, with sulphide precipitation in open spaces to form veins and along fractures or by replacement to form zones of disseminated mineralization. Changes in temperature, pressure and redox state between the wall rock and fluid are thought to have influenced the style and intensity of mineralization. As such, Ag-Pb-Zn lead mineralization at Pulacayo is typical of a high level epithermal system that in this case is hosted by sedimentary and intrusive rocks of Silurian and Neocene age.

The principal mineralized structure at Pulacayo is the TVS, which has historically been the main Ag producer of the mine. The TVS is a large structural stock work system that trends east-west and dips 75° to 90° to the south (Figure 7.4). The high grade parts of TVS were historically mined as single veins over widths of 1 m to 3 m but transitions from this setting into zones of complex quartz-sulphide or sulphide vein arrays that include conjugate veins, veinlets, stock works and disseminated sulphides that occur over widths ranging from less than a meter up to 120 m.

Mineralization of economic interest at Pulacayo is predominantly comprised of sphalerite, galena and tetrahedrite in sulphide-rich veins that are accompanied by locally abundant quartz, barite and pyrite. These veins range from a few centimeters to greater than 1 m in thickness and disseminated sphalerite, galena and tetrahedrite typically occur in wallrock between the veins. Disseminated mineralization is preferentially developed around and between veins hosted by andesite. To date, the TVS system has been continuously proven by mining and/or surface exposure along a strike length of 2,700 m and to a vertical depth of 1,000 m below surface and was open in both strike and dip components at the effective date of this report. The first 450 vertical meters of the TVS is hosted by andesitic volcanic rocks...
and the remaining 550 vertical meters is hosted by underlying Silurian sedimentary strata (Figure 7.5).

![Figure 7.5: Section PY-740200](image)

Veins at Pulacayo commonly contain semi-massive to massive sulphide and show internal features such as compositional banding, crustiform texture and drusy character (Figure 7.6).
They also frequently exhibit vuggy texture and have local infillings of quartz and barite (Figure 7.7).

Figure 7.6: Crustiform Texture In Nq Core (~47.6 MM Width)

Figure 7.7: Vuggy Texture With Quartz And Barite Infilling In Nq Core (~47.6 mm Width)
8 DEPOSIT TYPE

The Pulacayo deposit has been classified as an epithermal deposit of low to intermediate sulphidation state or association. Deposits of this type have been extensively researched and various summary publications that document specifics of the association are available. Examples of these include Lindgren (1922), White and Hedenquist (1994), Corbett and Leach, (1998) and Corbett, G.J., (2002). The following discussion of this deposit type is presented without change from Cullen and Webster (2012), who cited the work completed earlier by Presasacco et al. (2010).

Presasacco et al. (2010) highlighted the following key geological characteristics of Pulacayo that support classification as a low to intermediate sulphidation epithermal deposit and Figure 8.1 provides a schematic summary of the general deposit model:

- The vein and disseminated sulphide mineralization is hosted by Tertiary volcanic rocks of intermediate composition that form part of an outcropping dome complex.
- The mineralized body is composed of narrow veins, veinlets, stock works and disseminations in argillicly-altered host rock that are controlled by an east-west oriented fault system. Width of the mineralized zone varies from a few m or less to 120 m.
- Sedimentary rocks intruded by the dome complex host high grade veins such as TVS that are typically less than 3 m in width but transition to stock work and disseminated zones in overlying andesitic volcanic rocks that reach as much as 120 m in width.
- The sulphide mineralization has been proven to continuously occur along strike for 2 700 m and to a depth of approximately 100 m below surface.
- The vein system mineral assemblages are relatively simple and in combination are diagnostic of an epithermal setting. They consist of galena, sphalerite, tetrahedrite, and other silver sulfo-salts that form the main assemblage of economic interest and barite, quartz, pyrite and calcite that are present as gangue phases. Chalcopyrite and jamesonite are present in minor amounts locally.
- Internal texture of veins is typically banded and drusy with segments containing almost massive sulphides. This is typical of epizonal veins that have been subjected to multiple pulses of mineralizing fluid.
- Vertical metal zonation exists within the deposit that includes a mid-elevation zone of highest silver values that transition with depth to progressively increasing total base metal concentrations.
9 EXPLORATION

9.1 Introduction

The following sections provide summary descriptions of exploration programs carried out by Apogee during the period 2006 through 2012. Program descriptions for the 2006 to 2009 period presented below were previously presented by Pressacco et al. (2010) and are based on a detailed internal Apogee company document prepared in June 2009 in Spanish that is referenced herein as Iriondo et al. (2009). These descriptions were presented in Cullen et al. (2012) and are presented below without substantive change from that source. Descriptions of Apogee programs carried out in 2010 through 2012 are based on subsequent review by Mercator of internal and public company documents plus discussions with Apogee staff.

9.2 Work Programs During 2006 to 2012 Period

9.2.1 Introduction

In 2005, Apogee signed a joint venture agreement with ASC and subsequently commenced exploration on the Pulacayo property. Since then, Apogee has completed detailed geological mapping and sampling of surface exposures and underground workings, completed a topographic survey of the area, completed an IP geophysical survey, completed 4 diamond
drilling programs and completed 2 mineral resource estimates. Exploration work by Apogee was generally focused on the Pulacayo property but IP, diamond drilling and a mineral resource estimate were also completed on the Paca prospect located 10 km to the north of Pulacayo. Although the Paca deposit is included within the limits of the exploration licenses, it was not part of the current mineral resource estimate.

9.2.2 Topographic Survey

In 2006, Apogee contracted Geodesia y Topografia of La Paz, Bolivia to complete a topography survey of the Pulacayo-Paca areas using four LEICA Total Stations, models TCR 407, TC 703, TC 605L, and TC 600. The survey covered a total area of 24 km² and survey points were collected in WGS84, Zone 19 South Datum and the coordinates were referenced to known government control points including GCP CM-43 obtained from the IGM (Instituto Geografico Militar).

The survey points allowed the construction of a detailed topographic map for the Pulacayo and Paca areas and two meter contour intervals were established. The new topographic map was used as a base to establish road access, geological mapping and surface sampling as well as for locating drill collars.

As part of the field work, Eliezer Geodesia y Topografia also surveyed the collars of all completed drill-holes and established 12 surveyed grid lines for an Induced Polarization survey. Seven IP survey lines were located in the Pulacayo area and 5 were located in the Paca area. Surveyed stations were established at 50 m intervals along each line.

9.2.3 Geological Mapping and Sampling

Apogee initiated a surface mapping and sampling program at Pulacayo in 2005 and initially utilized preliminary geological maps completed by ASC in 2003. The company completed detailed 1:1 000 scale surface mapping that covered all exploration licenses, including both the Pulacayo and Paca areas. The sampling consisted mostly of rock chip samples taken from outcrops and the objective of the mapping program was to characterize the alteration patterns and locate sulphide mineralization both at surface and also within accessible underground mine workings. A total of 549 samples were collected from Andesita, Ramales, Paisano, TVS and Veta Cuatro. The Andesitas and Ramales areas are located to the east of the TVS and the Paisano area is located to the south of the TVS. Summary results of the geological mapping program appear in Figure 9.1.

During 2006 Apogee also initiated development of a detailed, three dimensional digital model of the historic underground mine workings based on available historic records. The workings solid model was completed by EPCM Consultores S.R.L. (EPCM) and was subsequently modified by Apogee through transformation of the model from the historic mine grid to the current datum plus adjustment to include a +1% incline grade of the San Leon tunnel.

9.2.4 Geophysical Surveys

An Induced Polarization geophysical survey was carried out by Apogee between November and December 2007. The survey covered grid lines on both the Pulacayo and Paca areas and the IP survey was completed by Fractal S.R.L (Fractal), a geophysical consulting
company based in San Cruz, Bolivia and independent of Apogee. The survey used a dipole-dipole electrode configuration along 400 m spaced lines. A total of 29 line km of IP surveying was completed on the Pulacayo and Paca properties and data were recovered using a 50 m dipole spacing to n=6. Results are summarized by Iriondo et al. (2009).
Figure 9.1: Property Geology Map
Seven geophysical survey lines oriented north-south were completed in the Pulacayo area and these were oriented approximately perpendicular to the east-west strike of the TVS (Figure 9.2). At Paca a total of 5 similarly oriented survey lines were completed.

The IP survey was successful in outlining several areas of anomalously low apparent resistivity that, based on correlation in an area of known bedrock geology, were interpreted to represent weakly altered rocks. On the same basis, high apparent resistivity zones were interpreted to represent zones of siliceous alteration.

Chargeability results were seen to vary between 2 and 20 mV/m. Chargeability values below 7 mV/m were interpreted to represent the background values. Highest chargeability values are seen along lines LPY4, LPY5 and LPY6 at Pulacayo between stations 0 and -900, and these are coincident with high apparent resistivity values. The TVS zone occurs within this anomaly between -750 m and -900 m. Fractal interpreted these responses as representing an area of silicification containing disseminated sulphide mineralization. Combined results of the survey show that an east-west oriented zone of anomalous apparent resistivity and chargeability responses measuring some 450 m in width and extending over the length of the survey grid that contains the TVS (Figure 9.3). Moderately anomalous values in chargeability located at the edges of the main anomalous zone were interpreted as altered rocks that could be related to a mineralized vein system at depth.
Figure 9.2: IP Survey Coverage
Figure 9.3: Representative IP Responses Line LPY-5
9.2.5 Diamond Drilling

Details of diamond drilling carried out by Apogee on the Pulacayo property are reported in detail in Section 10.0 of this report and are summarized below.

Since originally accessing the property in 2002, until the effective date of this report, ASC and Apogee have completed 69,739.15 m of drilling from surface and underground on the Pulacayo property. ASC drilling totaled 5,009.2 m between 2002 and 2006 in 24 holes and Apogee drilling was completed in 4 subsequent phases. Phase I was undertaken between January and June of 2006 and included 19 holes totaling approximately 4,718 m. Phase II drilling was initiated in November, 2007 and consisted of 14 holes totaling 3,442.18 m. Phase III drilling was carried out between January and August of 2008 and included 84 drill holes totaling approximately 20,758.91 m. Phase IV drilling that was carried out between January 2010 and December 2011 and consisted of 35,810.81 m in 149 holes. The last 45 holes (6,254 m) of Phase IV were focused on oxide zone definition.

Between June 2006 and February 2007, Apogee also completed 13,631.2 m of diamond drilling on the Paca deposit that is located approximately 10 km north of Pulacayo. Results of the program are reported in a mineral resource estimate for the Paca deposit completed in 2007 by Micon and reported by Pressacco and Gowans (2007). This drilling program is not material to the current Pulacayo resource estimate and is mentioned here for information purposes only.

9.2.6 Mineral Resource Estimates by Micon

Two mineral resource estimates for the Pulacayo property were prepared on behalf of Apogee by Micon, with the first having an effective date of October 28, 2008 and the second having an effective date of October 14th, 2009. Pressacco and Shoemaker (2009) reported on the first estimate and Pressacco et al. (2010) presented results of the second. In 2007, Micon also prepared a resource estimate for the nearby Paca deposit, also held by Apogee, but, as noted above, that deposit is not included in the current or past Pulacayo estimates. Both Pulacayo estimates are now historic in nature and should not be relied upon. They were superseded by subsequent estimates prepared by Mercator.

Tabulated results of the 2008 and 2009 Micon resource estimates are presented below in Table 9.1 and Table 9.2 and both were prepared in accordance with Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines (the CIM Standards) and disclosure requirements of National Instrument 43-101. Technical reports for each have been filed on SEDAR.

Both Micon estimates were based on geostatistical block models developed using Gemcom-Surpac Version 6.1.1 software with metal grade interpolation carried out using Ordinary Kriging (OK) methods checked by Nearest Neighbour (NN) and Inverse Distance Squared (ID2) methods. The deposit was modeled using a wireframed solid developed from drilling cross section interpretations of calculated Net Smelter Return (NSR) values. Resource reporting included all blocks within the respective wireframed solids and metal capping values of 1800 g/ for Ag, 11.5% Zn and 15% Pb were applied.

<table>
<thead>
<tr>
<th>Class</th>
<th>Rounded Tons</th>
<th>Ag g/t</th>
<th>Pb %</th>
<th>Zn %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Inferred</td>
<td>9,556,000</td>
<td>75</td>
<td>0.61</td>
<td>1.46</td>
</tr>
<tr>
<td>Indicated</td>
<td>7,003,000</td>
<td>53</td>
<td>0.63</td>
<td>1.42</td>
</tr>
</tbody>
</table>

(1) Tonnages have been rounded to the nearest 1,000 tons. Average grades may not sum due to rounding.

(2) Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

(3) The quantity and grade of reported inferred resources in this estimation are conceptual in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource.

(4) And it is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.

(5) Metal prices for the estimate are $14.38 USD/oz Ag, $0.86 USD/lb Zn and $0.92 USD/lb Pb.

(6) Grade capping of Ag at 1,800 g/t, Zn at 11.5% and Pb at 15% was applied.

Table 9.2: Historic MICON Mineral Resource Estimate - Effective 14 October 2009

<table>
<thead>
<tr>
<th>Class</th>
<th>Rounded Tons</th>
<th>Ag g/t</th>
<th>Pb %</th>
<th>Zn %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Inferred</td>
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<tr>
<td>Indicated</td>
<td>4,892,000</td>
<td>79.96</td>
<td>0.79</td>
<td>1.64</td>
</tr>
</tbody>
</table>

(1) Tonnages have been rounded to the nearest 1,000 tons. Average grades may not sum due to rounding.

(2) Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

(3) The quantity and grade of reported inferred resources in this estimation are conceptual in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource.

(4) It is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resource category.

(5) Metal prices for the estimate are $13.81 USD/oz Ag, $0.86USD/lb Zn and $0.86 USD/lb Pb.

(6) Grade capping of Ag at 1,800 g/t, Zn at 11.5% and Pb at 15% was applied.
Preliminary Economic Assessment

A Preliminary Economic Assessment (PEA) dated 25 June 2010 was completed by Micon for Apogee and reported by Pressacco et al. (2010). The assessment forms part of Apogee’s continuous public disclosure record and is available on SEDAR. However, as disclosed in the revised Mercator resource estimate Technical Report dated 09 May 2012 (Cullen and Webster, 2012a) the conclusions and analysis presented in the Micon PEA were no longer considered to be current with respect to the Pulacayo project. This reflected Apogee’s initiation of current project Feasibility Study programs that include changes in project scope and approach. Since Apogee no longer considers the Micon 2010 PA to be current, details of that analysis are not presented in this report and were not used to support the 09 May 2012 revised resource estimate by Mercator.

9.2.7 Mineral Resource Estimates by Mercator

Mercator prepared a mineral resource estimate for the Pulacayo deposit in 2011 after completion of additional diamond drilling on the property. Table 9.3 presents details of this estimate, which has an effective date of 19 October 2011. The current estimate supersedes this resource statement, which is now considered historic and should not be relied upon.

<table>
<thead>
<tr>
<th>Class</th>
<th>Rounded Tons</th>
<th>Ag g/t</th>
<th>Pb %</th>
<th>Zn %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Inferred</td>
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</tr>
<tr>
<td>Indicated</td>
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<td>153.14</td>
<td>0.91</td>
<td>2.04</td>
</tr>
</tbody>
</table>

(1) Mineral Resources are reported above a $ 40 USD NSR cut-off
(2) Metal prices used were $ 24.78 USD/oz Ag, $ 1.19 USD/lb Pb, and $ 1.09 USD/lb Zn
(3) Tonnages have been rounded to the nearest 10,000
(4) Contributing 1 m composites were capped at 1,500 g/t Ag, 15% Pb, and 15% Zn
(5) Specific gravity is based on an interpolated ID2 model
(6) Mineral resources that are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

The 19 October 2011 Mercator estimate was based on validated results of 59,352 m of diamond drilling from 174 surface drill holes and 41 underground drill holes carried out by ASC Bolivia and Apogee through various drill programs between 2002 and 2011. Modeling was performed using Gecomm Surpac(R) 6.2.1 modeling software with silver, lead and zinc grades estimated by inverse distance squared (ID2) interpolation from 1 m down hole assay
composites. Block size was 5 m (x) by 3 m (y) by 3 m (z) with no sub-blocking. Block model results were checked using ordinary Krigeing and Nearest Neighbor interpolation methods.

The polymetallic nature of the mineralization was evaluated by a Net Smelter Return (NSR) value considering an underground mining scenario below the oxide surface, constrained by a minimum operating cost of $ 40 NSR/t. The $ 40 NSR/t peripheral constraining solid was created from 50 m spaced interpreted sections of assay sample NSR values. NSR values were determined from a calculator developed by John Starkey, P. Eng., of Starkey and Associates Inc., Consulting Metallurgical Engineers. The modeling used a 24 month trailing average silver price and a 27 month forward seller contract price for both lead and zinc as of 31 August 2011, which correspond to prices of $ 24.87 USD/oz silver, $ 1.19 USD/lb lead, and $ 1.09 USD/lb zinc respectively. The $ 40 NSR solid model has a 25 to 30 m average thickness with a 1,100 m strike length oriented at 280 degrees and a 425 m average sub-vertical dip extent. This reflects the orientation and geometry of the principal TVS mineralized structure and associated secondary structures.

Grade interpolation was peripherally constrained by the $40NSR solid and carried out using multiple independent search ellipsoid passes for silver, lead and zinc. Contributing silver values were capped at 1,500 g/t and interpolated using an ellipsoid oriented at 280 degrees with a 30 degrees major axis plunge and 80 degrees N dip, contributing lead values were capped at 13.5% and interpolated using an ellipsoid oriented at 280 degrees with a 45 degrees major axis plunge and a 75 degrees N dip, and contributing zinc values were capped at 13.5% and interpolated using an ellipsoid oriented at 280 degrees with a 45 degrees major axis plunge and an 80 degrees N dip. A 60 m ellipsoid major axis, 30 m semi-major axis and 5 m minor axis were applied for all primary interpolation passes, with secondary and tertiary passes applied at 2 times and 3 times these ranges to completely fill the peripheral wireframe model.

A specific gravity model was interpolated by ID2 methodology from 1 m down hole specific gravity composites using an ellipsoid oriented at 280 degrees with a 45 degrees major axis plunge, 80 degrees N dip and the ranges specified above.

A solid model of historic underground mining and stoping was used to remove previously mined blocks and an interpolated 5 m marginal envelope to historic workings was applied to assign intersecting resource blocks to the Inferred category. This reflects uncertainty in local accuracy of the underground solid model. Indicated resources occur outside this envelop and have an interpolated grade for each metal derived from primary interpolation ellipsoid passes and have at least two or more contributing drill holes within at least one of the passes. All other interpolated blocks were categorized as Inferred resource blocks.

The Mercator deposit model included additional new Apogee drilling results and differed from previous Micon models in its use of a specific block grade cutoff value for statement of resources rather than inclusion of all blocks within an interpreted assay-based peripheral constraint, as carried out by Micon. This difference manifests as inclusion of sub cut-off grade blocks using the Micon approach.

Mercator prepared a subsequent mineral resource estimate (Cullen et al., 2012) for the Pulacayo deposit with an effective date of 28 September 2012. This estimate included oxide zone material not included in previous resource estimates and forms the basis of the current Feasibility Study. Full documentation for the estimate appears in this Feasibility Study.
technical report and was previously disclosed in the recent technical report by Cullen et al. (2012).

9.2.8 Bulk Sampling and Metallurgical Testing

Apogee carried out a trial mining program during the first half of 2012 that was focused on two hangingwall veins accessed at the 4 275 m level of the mine. This program produced approximately 7,000 tons of mineralized material that was stockpiled at the Pulacayo site. Both veins targeted for trial mining occur within the limits of the current mineral resource estimate. Apogee holds the necessary environmental permits issued by the Bolivian authorities to conduct mining and processing operations of up to 200 tons per day, which includes the transport of mineralized material to an established concentrator outside the project area.

During the second half of 2012, Apogee processed two bulk samples from the trial mining at separate toll milling operations in the district. The first was carried out at Tatasi and the second was carried out at facilities operated in Potosi by the Federación de Cooperativas Mineras de Potosi (FEDECOMIN) and located approximately 180 road kilometres from the Pulacayo site. Results of testing at both sites are discussed below in report Section 12. In addition, a series of controlled laboratory bench and pilot scale tests in support of the ongoing Feasibility Study (Apogee Press Release dated 29 May 2012) are underway at Maelgwyn Minerals Services Africa (Pty) Ltd. in South Africa in order to replicate the process flow and optimizing reagent recipes utilized during the FEDECOMIN bulk test (Apogee Press Release dated 13 September 2012).

9.2.9 Commissioning of Project Feasibility Study

On 29 May 2012 Apogee announced that a contract for completion of a Feasibility Study of an underground mine and concentrator plant at its 100% controlled Pulacayo Project had been awarded to TWP. TWP is a member of the Basil Read Group, a leading engineering and construction company in the world, with offices across South Africa, Australia, and Peru. Feasibility study programs began in the middle of 2012 and results of these are presented in this technical report. In light of initiation of the Feasibility Study programs, Apogee also disclosed by press release on 29 May 2012 that it was no longer relying on results of the previously completed PEA carried out by Micon.

10 DRILLING

10.1 Introduction

The first modern era drilling program at Pulacayo was initiated in 2002 by ASC and subsequent drilling programs were undertaken by Apogee between 2006 and 2012. Data from all programs is used in the current resource estimate and details of each program are present below under separate headings.
10.2 ASC Bolivia LDC Drilling (2002-2005)

ASC completed 14 diamond holes totaling 3,095 m in length between July 2002 and November 2003 (holes PUD001-PUD017). Eleven holes were drilled from surface and another three from drill stations located in the Pulacayo underground workings. Drilling was completed by Leduc Drilling S.R.L. of La Paz, Bolivia, using two Longyear LF-140 and LY-44 drill rigs and HQ (63.5 mm diameter) core was recovered.

A second phase of drilling was initiated in February 2003 and although 10 holes were planned only 2 underground drill holes were subsequently completed for a total of 554 m (holes PUD025 and PUD026). Drilling was performed by Drilling Bolivia Ltd. and HQ core was recovered.

ASC continued the drilling program in September 2003 and completed eight additional holes totaling 1,302 m (holes PUD018 to PUD024 and PUD027). Six holes were completed from surface and two holes were completed from drill stations located in the Pulacayo underground workings. Drilling was contracted to Maldonado Exploraciones S.R.L. of La Paz, Bolivia and they used long year model LY-44 and LF-70 drilling rigs recovering HQ size core.

The 2002 though 2005 drilling programs outlined disseminated, veinlet and stock work style mineralization occurring between previously mined high grade veins. Cullen and Webster (2012a) presented a tabulation of selected significant drill intercepts in that previously disclosed resource estimate technical report along with a plan for drill holes and tabulations of collar coordination, hole orientation and length data.

10.3 Apogee Drilling (January 2006 – May 2008)

Following the acquisition of the Pulacayo property in 2005 Apogee initiated a Phase I drill program that consisted of 19 holes totaling 4,150 m in length (PUD028 to PUD042). Four of the holes were completed from drill stations located in the Pulacayo underground workings and 15 were completed from surface locations. The Apogee program objective was to confirm mineralization defined by earlier ASC drilling results and the program was successful in demonstrating the presence of significant amounts of disseminated, veinlet, and stock work sulphide mineralization located between the high grade veins that were exploited by historic, narrow underground mine workings (Pressacco et al., 2010). A tabulation of significant 2006 drilling intercepts appears in Appendix 1 along with drill hole locations and a tabulation of associated collar coordination, hole orientation and length data.

Apogee switched their focus to the Paca deposit, located approximately 10 km north of Pulacayo, in 2006 and completed approximately 13 631.2 m of diamond drilling in three drill programs. Between February 2006 and April 2006, Apogee completed a total of 2 301.5 m in 23 drill holes (PND031 to PND053). A second phase of diamond drilling was carried out from June 2006 to November 2006 and a total of 10,443.70 m were completed in 46 drill holes (PND054 to PND099). Seven additional holes totaling 886 m were drilled in a third phase of drilling completed late in 2006 (PND100 to PND106). The results of this exploration drilling program formed the basis of a 2007 mineral resource estimate on the Paca deposit completed by Micon and are described in detail by Pressacco and Gowans (2007). As noted earlier in this report, the Paca deposit is not included in the current resource estimate and the preceding description has been included for completeness.
In November 2007, Apogee started Phase II drilling at Pulacayo and completed 14 holes totaling 3,745 m (PUD043 to PUD056). All holes were drilled from surface locations and results showed that the TVS consisted of disseminated, veinlet, and stock work sulphide mineralized material measuring up to 120 m in width within which high grade mineralized shoots were present that had not been exploited by previous operators of the mine.

Phase III drilling was undertaken by Apogee between January and May, 2008 at which time 54 holes totaling 14,096 m were completed (PUD057 to PUD110). Of these, 8 holes were drilled from underground and the balance from surface. A tabulation of significant 2007 and 2008 drilling intercepts, drill hole locations and associated collar coordination, hole orientation and length data appear in the previously disclosed resource estimate technical report by Cullen and Webster (2012a).

Phase I drilling was completed by the Leduc Drilling S.R.L of La Paz, Bolivia and subsequent Phase II and III programs were completed by the Fujita Core Drilling Company of Bolivia. The companies used Longyear model LF44, LM-55, LF-90 and LM-90 drilling rigs for the surface and underground programs and core size was generally HQ (65.3 mm diameter). In certain instances, ground conditions around old workings or other issues required reduction in core size to NQ (47.6 mm diameter).

10.4 Apogee Drilling (January 2010 – December 2011)

Phase IV drilling was initiated by Apogee in January of 2010. The surface program continued until the end of 2011 and underground drilling was carried out on a limited basis during that time in support of test mining activities within the current resource outline (PUD-111-PUD266). The current resource estimate includes all Phase IV surface holes up to and including hole PD266, for a total of 42,522 m of Phase IV drilling to the end of 2011. The last 45 holes of the surface drilling program (6,253.83 m) were directed toward evaluation of oxide zone resource potential above the main sulhide zone of the TVS. Drilling program results received since the last resource estimate by Mercator improved the level of confidence within certain sulphide zone mineral resource areas and also allowed estimation and evaluation of oxide zone resources.

Cullen et al. (2012) presented a tabulation of selected significant drill intercepts in that previously disclosed technical report, along with a plan for drill holes and tabulations of collar coordination, hole orientation and length data. Figure 10.1 presents a typical drilling plan and longitudinal section for the project.

The Fujita Core Drilling Company continued to provide drilling services at Pulacayo through the current reporting period, using Longyear models LF44, LM-55, LF-90 and LM-90 rigs for surface and underground drilling. The core size has been HQ except where ground conditions around old workings or other issues have required reduction in core size to NQ.
Figure 10.1: Plan and longsection of Pulacayo Drilling
10.5 Apogee Drilling Logistics

Planning for drill holes is based on the logging and interpretation of geological cross sections generated by Apogee staff geologists. Drillhole coordinates are established from digital maps and surface drill hole collars are located on the ground by field geologists using a hand-held GPS. Hole azimuth and inclination are established using a compass and clinometer. Collar coordinates for underground drilling are established by company surveyors and hole azimuth and inclination are set by transit. Downhole deviation is determined for both surface and underground holes at approximately 50 m intervals using either Tropari or Reflex down hole survey tools.

Drill core is initially stored at the drill site in wooden core boxes, which hold approximately 3 m of core. Boxes are marked with the hole identification, box number and the included depth interval of the hole. Drilling staff mark core depth, generally in 3 m intervals, with a wooden tag indicating down hole depth at that point in meters. Once the drill hole is completed, hole collar coordinates are surveyed by staff surveyors and PVC pipe is inserted into the hole with a portion left exposed approximately 0.5 m above ground to define hole inclination and azimuth (Figure 10.2). A concrete monument is established around the PVC pipe and a metal plate is attached that records the company name, hole number, grid easting and northing coordinates, elevation, final depth and the start and end drilling dates (Figure 10.3).

Overall core recovery reported by Apogee for its programs exceeds 90% in most cases regardless of the type of rock being recovered. However, proximity to old mine workings reduces recovery potential due to associated bedrock instability. Mercator staff reviewed core from various positions within the deposit during the two field visits and confirmed that good recovery through mineralized zones is generally obtained. However, some intervals of strong fracturing and reduced core recovery were also inspected.
Figure 10.2: Typical Apogee Drill Collar after Completion of Hole

Figure 10.3: View of Apogee Drill Collar Description Plate
11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Sample Preparation from 2006 to 2012 Apogee Programs

The following description of sample preparation and core handling protocols applies to all drilling programs at Pulacayo in which Apogee has participated. Program details were discussed with Apogee staff during the April 2012 site visit by Mercator and were found to be unchanged from the time of Mercator’s earlier July 2011 site visit. Pressacco et al. (2010) previously outlined the same general conditions as being applicable for work programs carried out by Apogee prior to the effective date of that report.

Apogee staff are responsible for transport of core boxes by pick-up truck from drill sites to the company’s locked and secure core storage and logging facility located in the town of Pulacayo (Figure 11.1). The core is initially examined by core technicians and all measurements are confirmed. Core is aligned and repositioned in the core box where possible and individual depth marks are recorded at 1 m intervals on the core box walls. Core technicians photograph all core, measure core recovery between core meterage blocks, complete magnetic susceptibility readings and specific gravity measurements and record information on hard copy data record sheets. This information is initially entered into Excel digital spreadsheets and then incorporated into the project digital database.

Drill geologists initially complete a written quick log of drill hole lithologies along with a graphical strip log that illustrates lithologies. They subsequently complete a detailed written description of lithologies alteration styles and intensities, structural features, mineralization features such as occurrences and orientations of quartz veins, and the style, amount and distribution of sulphide minerals. Drillholes are drawn on paper cross sections when logging is completed and lithologies are graphically correlated from drill hole to drill hole.

Mineralized intervals are marked for sampling by the logging geologist using colored grease pencils and intervals plus associated sample numbers are recorded on a hardcopy sample record sheet. All paper copy information for each hole, including quick logs, detailed logs, graphical logs, sample record sheets and assay certificates are secured together in a drill hole file folder to provide a complete archival record for each drill hole. Subsequent to logging and processing, down hole litho coded intervals, sample intervals and drill hole collar and survey information are entered into digital spreadsheets and then incorporated into the project digital database.

Sample intervals are marked by the logging geologist and core technicians then cut sample intervals in half using a diamond saw. Friable core is cut in half with a knife. Each half core sample is assigned a unique sample tag and number and placed in a correspondingly numbered 6 mm plastic sample bag. A duplicate tag showing the same number is secured to the core box at the indicated sample interval. As noted earlier, all sample intervals and corresponding numbers are recorded on a hardcopy sample date sheet and are subsequently entered into a digital spreadsheet for later incorporation in the project database.
The secured 6 mm plastic sample bags are grouped in batches of 6 to 10 samples and secured in a larger plastic mesh bag in preparation for shipment to the ALS Chemex (ALS) preparation laboratory located in Oruro, Bolivia. All bagged samples remain in a locked storage facility until shipment to the laboratory. Samples are transported from the core storage area to the ALS facility by either Apogee personnel or a reputable commercial carrier. Sample shipment forms are used to list all samples in each shipment and laboratory personnel crosscheck samples received against this list and reported any irregularities by fax or email to Apogee. Apogee advised Mercator that it has not encountered any substantial issues to date with respect to sample processing, delivery or security for Pulacayo programs.

11.2 Sample Preparation from 2002 to 2003 ASC Programs

Site procedures pertinent to ASC were not documented in support information reviewed by Mercator. However, Apogee staff familiar with the earlier program indicated that procedures were generally similar to those employed by Apogee with respect to core logging, sampling, transport of samples and security. All ASC drill core samples were processed at the Oruro, Bolivia laboratory of ALS Chemex, with those from the first phase of drilling being analyzed at ALS Chemex (formally Bondar-Clegg) facilities in Vancouver, BC, Canada. In both instances, standard core preparation methods were used prior to elemental analysis.

11.3 Drill Core Analysis from 2006 to 2012 Apogee Programs

Apogee staff log and sample drill core and carry out immersion method specific gravity determinations but do not carry out any form of direct sample preparation or analytical work.
on project samples. Project analytical work has been completed by ALS at its analytical facility in Lima, Peru after completion of sample preparation procedures at the ALS facility located in Oruro Bolivia. ALS is an internationally accredited laboratory with National Association of Testing Authorities (NATA) certification and also complies with standards of ISO 9001:2000 and ISO 17025:1999. The laboratory utilizes industry standard analytical methodology and utilizes rigorous internal Quality Assurance and Quality Control (QAQC) procedures for self-testing.

All samples are weighed upon receipt at the lab and prepared using ALS preparation procedure PREP-31B that consists of crushing the entire sample to >70% -2 mm, then splitting off 1 kg and pulverizing it to better than 85% passing 75 micron. The coarse reject materials from this processing are returned to Apogee for storage on site at Pulacayo. Similar processing was used by ASC.

Silver, lead and zinc concentrations for Apogee programs were analyzed by ALS using an Aqua Regia digestion and Atomic Absorption Spectroscopy (AAS) following ALS method codes AA46 and AA62. Samples returning assay values greater than 300 g/t Ag were further analyzed using quantitative method Ag-GRAV22, which uses a Fire Assay pre-concentration and Gravimetric Finish on a 50 g sample aliquot. Gold values were determined using the Au-AA26 analytical method provided by ALS that employs a Fire Assay pre-concentration followed by Atomic Absorption finish on a 50 g sample aliquot. A 35 element multi-element analysis was also completed on samples using method code ME-ICP41 that uses Aqua Regia digestion and ICP-AES analysis.

11.4 Drill Core Analysis from 2002 to 2003 ASC Programs

Samples from the ASC drilling programs carried out in 2002 and 2003 were also prepared and analyzed by ALS. However, after preparation at the facility in Oruro, Bolivia under the same protocols as noted above for Apogee, analytical work was carried out at the company’s laboratory in Vancouver, BC, Canada. This facility is fully accredited, as described earlier and analytical protocols were the same as those described above for Apogee.

11.5 Quality Assurance and Control from 2006 to 2012 Apogee Programs

Apogee developed an internal QAQC program that includes blind insertion of reference standards, blanks and duplicates in each analytical shipment. A blank is inserted at the beginning of each sample batch, standards are inserted at random intervals throughout each batch of 50 samples and duplicates are analyzed at the end of each batch. All data gathered for QAQC purposes is captured, sorted and retained in the QAQC database. Apogee purchased two commercial reference standards from Western Canadian Minerals Ltd. (WCM), these being PB-128 and PB-124, created an in-house standard and also purchased commercial prepared blank materials. Coarse field blanks were prepared by Apogee from locally sourced un-mineralized quartzite outcropping near Pulacayo.

Three quality control samples that include a field or commercial blank, a commercial standard and a duplicate sample are inserted with a minimum frequency of 1 in every 50 samples. Review by Mercator showed that in practice, an analytical standard typically occurs within every 20 samples submitted for analysis. Analysis of duplicate samples of quarter core is
accommodated through their blind inclusion in the sample stream and analysis of duplicate 
prepared pulp splits are also requested for each batch.

Apogee’s protocol also includes a check sampling program based on analysis of sample 
splits at a second accredited laboratory. Check samples submitted to the check laboratory 
are processed using the same preparation and analytical techniques as used by ALS in Lima. 
The ALS laboratory in La Serena, Chile provided initial second laboratory analytical services 
for the Apogee programs, results of which are discussed in report Section 13. Since January 
2010, starting with drill hole PUD 140 samples, all check analyses are done at SGS in Lima, 
Peru.

In addition to chemical analysis, bulk density measurements (specific gravity) were 
systematically collected by Apogee staff using standard water emersion methods and 
unsealed core samples. Micon (Pressacco and Shoemaker, 2009) reviewed ASC and early 
Apogee density determination methods and recommended that these be modified to include 
more representative sampling methods in non-mineralized areas, and be carried out 
continuously through mineralized zones. Procedures were subsequently modified and 
measurements thereafter were collected for continuous 10 m sampled intervals where no 
observed sulphide mineralization was present and for all sampled intervals within recognized 
mineralized zones. Micon also noted that samples were not being wax sealed and advised 
that sealing would be advisable for some samples due to higher porosity and low 
permeability seen in some samples. Characteristics of lithology and alteration were also 
recorded as part of the density program and all information was assembled in digital 
spreadsheets. Results of the density measurement programs appear in report section 13.

The authors are satisfied that Apogee’s sample preparation, analysis and security 
methodologies are sufficient for a project of this size and that suitable precautions are being 
taken to identify irregularities in sample analytical results.

11.6 Quality Control and Assurance for ASC 2002-2003 Programs

QAQC procedures pertinent to the ASC programs were not documented in support 
information reviewed by Mercator for this report. However, the first drilling program carried 
out by Apogee in 2006 consisted of 19 holes totaling approximately 4,150 m drilled to confirm 
earlier ASC analytical data. Full QAQC protocols instituted by Apogee were applied to this 
program and results of an Apogee re-drill program correlate well with those of ASC. Such 
correlation suggests that acceptable standards were being met by ASC.
12 DATA VERIFICATION

12.1 Review and Validation of Project Data Sets

Core sample records, lithologic logs, laboratory reports and associated drill hole information for all drill programs completed by Apogee and ASC were digitally compiled by Apogee staff and made available to Mercator for resource estimation purposes. Information pertaining to the exploration history in the property area was also provided by Apogee and was reviewed to assess consistency and validity of Apogee results.

Digital drill hole records supplied by Apogee were checked against original hard copy source documents during both project site visits by Mercator to assess consistency and accuracy of such records. Parameters reviewed in detail include collar coordinates, down hole survey values, hole depths, and lithocodes. This was followed by review and validation of approximately 10% of the compiled core sample dataset against original source documents. Review of logging and sample records showed consistently good agreement between original records and digital database values.

Micon noted in the Pressacco et al. (2010) Pulacayo Preliminary Economic Assessment (PEA) that minor inconsistencies were in some instances present between digital project datasets and original source documents reviewed at the field site. More specifically, disagreement was noted in drill hole coordinates between original and digital datasets for such parameters as hole survey data, core recovery information on paper logs, incomplete digital representation of specific gravity analysis in the project database, and paper logs not having original sample documents appended. Apogee staff followed up on associated recommendations laid out by Micon and now prepare a summary file at the field site for each drill hole that contains updated drill collar coordinates, complete down hole survey results, graphical and tabulated quick logs, geological logs, updated cross sections, original sample records, summary assay results, specific gravity analyses and core recoveries. Examples of these summary drill hole files were reviewed by Mercator.

After completion of all manual record checking procedures, the drilling and sampling database records were further assessed through digital error identification methods available through the Gemcom-Surpac Version 6.2.1® software. This provided a check on items such as sample record duplications, end of hole errors, survey and collar file inconsistencies and some potential lithocode file errors. The digital review and import of the manually checked datasets through Surpac provided a validated Microsoft Access® database that Mercator considers to be acceptable with respect to support of the resource estimation program described in this report.

12.2 Site Visits by Mercator

12.2.1 03 August to 10 August 2011

Author and qualified person Peter Webster and Mercator resource geologist Matthew Harrington carried out a site visit to the Pulacayo deposit during the period of 03 August to 10 August 2011. At that time, they completed a review of all Apogee drill program components, including discussion of protocols for lithologic logging plus storage, handling, sampling and
security of drill core. A core check sampling program consisting of 9 quarter core samples, 2 duplicate split samples, 2 quality control samples, and 4 reject material samples was also carried out. A drill collar coordinate check program was also completed during the visit, with collar coordinates for 7 Apogee drill holes collected using a hand-held GPS device for comparison against database records. Apogee President, Mr. Chris Collins, P. Geo, and Exploration Manager Mr. Hernan Uribe provided technical assistance and professional insight during the site visit.

During the core inspection and review process, several previously sampled core intervals representative of the Ag, Pb and Zn grade ranges of the Pulacayo deposit were selected from drill holes PUD111, PUD134, PUD140, PUD144, PUD175, PUD188, and PUD203 for use in the Mercator check sampling program. After mark-up and photographing of sampled core intervals by Mercator, Apogee staff carried out quarter core sampling of the designated core samples under Mercator supervision. Resulting bagged, labeled and sealed core samples were securely stored at the Apogee facility until being transported by commercial courier to SGS del Peru S.A.C for analysis.

Observations regarding the character of the landscape, vegetation, site elevations, surface drainage, road/drill pad features, drill sites, mine accesses, exploration conditions, and core logging and handling facilities were noted during the site visit. Based on observations made during the site visit and discussions with Apogee staff and consultants, Mercator determined that, to the extent reviewed during the visit, evidence of work programs carried out to date on the property is consistent with descriptions reported by the company and that procedures employed by Apogee staff are consistent with current industry standards and of good quality.

12.2.2 26 April to 28 April, 2012

Author Cullen visited the Pulacayo site during the period 26 April to 28 April 2012 the purpose being to carry out additional review of on-going drilling and resource estimation program work pertaining to oxide zone mineralization not included in the previous resource estimate. Reviews of site drill program components, including discussion of protocols for lithologic logging plus storage, handling, sampling and security of drill core were carried out at that time, with specific focus on the 45 drill holes completed late in 2011 that targeted oxide zone mineralization of the TVS.

Trenched areas, TVS surface exposures and drilling locations associated with assessment of the oxide zone were visited and a general overview of the site and facilities was obtained (Figure 12.1 and Figure 12.2). A core check sampling program consisting of 24 quarter core samples, 1 quality control reference material sample, and 1 blank sample was also carried out, along with field location checking of drill collar coordinates for 14 oxide zone drill holes. Collar locations were checked using a hand-held GPS device for comparison against database records and acceptable results were returned from both programs. Details of these are presented below in report section 12.3.5.

An active test mining area developed by Apogee on a subsidiary hanging-wall vein (ramale) interval of the TVS near the San Leon tunnel was also visited during the April 2012 trip. Stoping of the vein was being carried out at that time over a horizontal width of approximately 1.5 m to 2.0 m and stoped material was being stockpiled on surface for use in future metallurgical testing programs (Figure 12.3).
Apogee President, Mr. Chris Collins, P. Geo. accompanied the author throughout the visit and Exploration Manager, Mr. Hernan Uribe, discussed the project during meetings held on 24 April at Apogee’s La Paz office. Discussions on site were held with Apogee mine staff and mine manager, Mr. Wouter Erasmus, led an underground tour of the test mining area and adjacent historic workings (Figure 12.4). Apogee site geological staff under direction of Mr. Freddy Mayta, Senior Geologist, plus core facility technicians facilitated review and sampling of the oxide zone drill core that was a primary focus of this site visit (Figure 12.4).

Figure 12.1: Oxide Zone Exposure – April, 2012
Figure 12.2: View of the Pulacayo Town Site – April, 2012

Figure 12.3: Bulk Sample Stockpile Area – April, 2012
12.3 Quality Control and Quality Assurance (QAQC)

12.3.1 Apogee Programs 2006 – 2012

The Apogee drilling programs covered in the October 2011 mineral resource estimate by Mercator occurred between 2006 and July of 2011, with the last drill hole included in that resource being PUD214. QAQC results for the Apogee programs that contributed to the previous estimate were presented by Cullen et al. (2012) and the discussions presented below that pertain to these programs are taken directly from that source with a minimum of modification. QAQC results that pertain to oxide zone drilling carried out between July 2011 and December 2011 by Apogee are also described below.

Apogee Programs 2006 to 2012

Drill core sampling carried out by Apogee during the 2006 through 2012 programs on the Pulacayo property were subject to a QAQC program administered by the company. This included submissions of blank samples, duplicate split samples of quarter core and half core, certified analytical standards, Apogee field standards and analysis of check samples at a third party commercial laboratory. Additionally, internal laboratory reporting of quality control and assurance sampling was monitored by Apogee on an on-going basis during the course of the project. Details of the various components are discussed below under separate headings.

QAQC discussions for certified and in-house standards, blanks and duplicate splits pertaining to Apogee programs completed prior to January 2010 were previously disclosed in the Micon PEA (Pressacco et al. 2010) for Pulacayo. After review of these results, Mercator concurs with the opinion expressed by Micon that drill core sample data associated with the earlier...
Apogee QAQC programs is of quality acceptable for resource estimation use. The Micon report should be consulted if access to more detailed information pertaining to drilling programs carried out by Apogee prior to January 2010 is required. QAQC discussions presented below in this report pertain only to drilling carried out by Apogee since January 2010.

12.3.2 Certified Reference Material Programs

January 2010 to July 2011

The following description of certified reference material program results is taken from Cullen et al (2012) with only minor local modification of text to match the current reporting context.

Apogee has used three certified reference standards since the Phase IV drilling program commenced in January of 2010. These are CDN-SE-1, obtained from CDN Resource Laboratories (CDN) of Burnaby, BC and PB128 and PB138, obtained from WCM Minerals Ltd. (WCM) of Burnaby, BC. CDN-SE-1 has been used since the start of the Phase IV drill program, beginning with drill hole PUD141, and remained in use at the effective date of this report. PB128 was used in pre January 2010 programs, beginning with drill hole PUD061, and was replaced by PB138 at drill hole PUD207. Descriptions for all certified reference materials appear in the previously disclosed resource estimate technical report by Cullen et al. (2012) and Table 12.1 presents their certified mean values.

In total, results for 178 certified reference samples submitted for analysis were reviewed for this project period. This includes all certified reference samples used during the Apogee Phase IV drilling program during the period plus those pertaining to Phase III drill holes PUD134 through PUD138 for which assay results had not be received at the time of the Micon PEA in 2010. Reference samples were systematically inserted into the laboratory sample shipment sequence by Apogee staff that ensured that at least one standard was submitted for every 50 samples. Records of reference standard insertion were maintained as part of the core sampling and logging protocols.

<table>
<thead>
<tr>
<th>Reference Material</th>
<th>Certified Mean Value ±2 Standard Deviations</th>
<th>Number</th>
</tr>
</thead>
<tbody>
<tr>
<td>CDN-SE-1</td>
<td>712 ±57 1.92 ±0.09 2.65 ±0.20</td>
<td>82</td>
</tr>
<tr>
<td>PB128</td>
<td>181 ±16.41 4.43 ±0.342 2.25 ±0.18</td>
<td>91</td>
</tr>
<tr>
<td>PB138</td>
<td>199 ±8.958 2.04 ±0.149 2.08 ±0.124</td>
<td>5</td>
</tr>
</tbody>
</table>

The CDN-SE-1 standard was used exclusively during the Phase IV drill program initiated in January 2010 and is still in use by Apogee. In total, 82 samples of the material were analyzed during the drilling period covered in this report section, with samples submitted in association with drill holes PUD141 through PUD214. Returned Ag values fall within a +15 g/t and -55 g/t range of the 95% confidence interval certified mean range and the average value of 698.75 g/t falls within the mean ±2 standard deviations control limits. One sample value falls below
the control limits (Figure 12.6). A total of 12 Pb values fall below the ±2 standard deviations control limits for that metal, with returned values within a +0.025% and -0.135% range of the certified mean. However, the average Pb value of 1.87% falls within the control limits (Figure 12.7). Returned zinc values are more closely distributed around the certified value than those of Ag and Pb, with the average returned value of 2.65% being the same as the certified value. All values fall within +0.32% and -0.13% of the certified mean, with one result above the control limits (Figure 12.8).

The PB128 standard was used throughout the Phase III drill program, beginning in January 2008 and continued throughout most of the Phase IV drill program. Use began with drill hole PUD061 and finished with drill hole PUD208. Results for a total of 91 samples collected since January, 2010 were reviewed for this report, with these corresponding to drill holes PUD134 to PUD208, exclusive of hole PUD207. All samples returned results for Pb and Zn and 89 of the 91 samples returned results for Ag. Ag values fall within a range of +13 g/t and -8 g/t of the certified mean value and the average value of 181.57 g/t very closely approximates the 181.08 g/t certified mean value (Figure 12.8). Pb values fall within +0.15% and -0.27% of the certified mean range and average 4.36%, all of which fall within mean ±2 standard deviations control limits (Figure 12.10). One Zn value falls above the control limits, but others fall within +0.20% and -0.09% of the certified mean range. The average returned value of 2.29% falls within the mean ±2 standard deviations control limits (Figure 12.10).
Figure 12.8: Certified Standard CDN-SE-1 Results - Zn % (N=82)

Figure 12.9: Certified Standard Pb128 Results - Ag G/T (N=89)
The PB138 certified reference material was introduced during the Phase 4 drill program to replace PB128 and a total of 5 samples of the material were analyzed in association with drill holes PUD207, PUD210, PUD211A, and PUD214. Ag values returned fell within -9 g/t of the certified mean value range and average 194.20 g/t, all of which fall within mean ±2 standard deviations control limits (Figure 12.11). The average Pb value of 1.91% falls within mean ±2 standard deviations control limits, with all but one value falling within -0.25% of the certified mean range. One value occurs below the lower control limit (Figure 12.12). Zn results fall within +0.11% and -0.20% of the certified mean value with 2 falling below mean ±2 standard deviations control limits. The mean value of 2.00% falls within control limits (Figure 12.13).

Based on results presented above, it is apparent that a low bias exists in Ag and Pb results for CDN-SE-1. This is most pronounced in the Pb data set where 15% of samples returned values below mean ±2 standard deviations control limits. In contrast, Zn results for CDN-SE-1 closely track the certified mean value. A low bias may also be present for Ag, Pb and Zn in the PB138 data set but the limited number of samples (5) prevents further comment. PB128 results for all three metals typically fall within mean ±2 standard deviation project control limits.

After review of all results, and notwithstanding the possible low bias trends noted above, Mercator considers combined data of all certified reference material programs by Apogee to be sufficiently consistent to support use of associated datasets for current resource estimation purposes. However, it is recommended that potential low bias trends be investigated further.
July 2011 to January 2012 Programs

The oxide zone diamond drilling program was carried out between September 2011 and December 2011 and some oxide zone resampling of earlier holes was carried out in January of 2012. Certified reference material insertion protocols for this period were the same as those described above for the earlier Apogee program. The CDN SE-1 and PB138 reference materials were used during this period and results for both typically fall within the mean ±2 standard deviations control limits for the project. Slight low bias within the control limits is notable for Pb in CDN-SE-1 and Pb -138 results show similar slight low bias within control limits for all three metals. Results for the two reference materials are interpreted as indicating an acceptable degree of accuracy in the associated data set. Figures 12.14 to Figure 12.19 present certified reference material Ag, Pb and Zn results for this program.

Figure 12.11: Certified Standard Pb138 Results - Ag G/T (N=5)
Figure 12.12: Certified Standard Pb138 Results - Pb % (N=5)

Figure 12.13: Certified Standard Pb138 Results - Zn % (N=5)
Figure 12.14: Certified Standard Cdn-Se-1 Results - Ag G/T (N=25)

Figure 12.15: Certified Standard CDN-SE-1 Results - Pb % (N=25)
Figure 12.16: Certified Standard CDN-SE-1 Results - Zn % (N=25)

Figure 12.17: Certified Standard Pb138 Results - Ag G/T (N=54)
Blank Sample Programs
January 2010 to July 2011

Both field and commercial blank materials were systematically inserted into the laboratory sample stream by Apogee staff during this reporting period and total 174 associated...
samples. The insertion rate was at least 1 blank per 50 samples submitted. Blank samples used by Apogee consisted of blank reference material BL107 obtained from WCM, standard reference material CDN-BL-7 obtained from CDN and a field “coarse blank” that was prepared from a source of barren quartzite outcropping near the Pulacayo deposit. Blank results from all three blank materials are reviewed together and pertain to drill holes PUD134 through PUD214.

The blank results reviewed are deemed acceptable and no significant or systematic cross-contamination effect is interpreted to be present. Average blank values of 0.879 g/t Ag, 0.0014% Pb, and 0.0053% Zn are below expected values for both the field and commercial blanks. Returned blank Ag results are consistently near expected values, with 4 results greater than 2 g/t and a maximum value of 4 g/t (Figure 12.20). Pb blank results are more variable, with 8 results greater than 0.003% and a maximum value of 0.007% (Figure 12.21). A total of 6 Zn blank samples returned a value greater than 0.010%, with three results returning values of 0.018%, 0.021%, and 0.021% (Figure 12.22). The majority of the blank sample anomalies are associated with field blanks prepared by Apogee and may reflect heterogeneity of such material.

![Figure 12.20: Blank Samples Values Ag G/T (N=174)](image-url)
July 2011 to January 2012 Programs

The oxide zone diamond drilling program was carried out between September 2011 and December 2011 and some oxide zone resampling of earlier holes was carried out in January of 2012. The blank sample insertion protocol for this period was the same as described above for the earlier Apogee program and Figure 12.23, Figure 12.24 and Figure 12.25 present Ag, Pb and Zn results, respectively. These results are interpreted as indicating that
no problematic level of sample material cross-contamination exists within the associated dataset. However, increasing incidence of spiking Ag between 2 and 4 g/t is present in the dataset and should be reviewed for cause.

Figure 12.23: Blank Sample Values Ag G/T (N=102)

Figure 12.24: Blank Sample Values Pb % (N=102)
12.3.3 Quarter and Half Core Duplicate Split Check Sample Programs

January 2010 to July 2011

In addition to scheduled analysis of duplicate splits of core sample pulps by the laboratories, Apogee carried out a program of quarter core and half core sampling to check on sample variability and lab consistency during this report period. A total of 149 duplicate samples were processed by ALS in Lima, Peru, during the period, including 107 quarter core samples associated with drill holes PUD134 through PUD211A and 42 half core samples associated with drill holes PUD176 through PUD214. Ag results are presented in Figure 12.26 for half core duplicates and in Figure 12.27 for quarter core duplicates and have correlation coefficients of 0.84 and 0.83 respectively. Although the duplicate samples in both cases returned higher values in general than the original results, duplicates of high grade original samples show greater variability. Pb and Zn results of duplicate samples show a higher degree of correlation with the original result. Pb half core duplicate samples (Figure 12.28) and quarter core duplicate samples (Figure 12.29) have correlation coefficients of 0.97 and 0.94 respectively and Zn half core duplicate samples (Figure 12.30) and quarter core duplicate samples (Figure 12.31) have correlation coefficients of 0.96 and 0.95 respectively.
Figure 12.26: 1/2 Core Duplicate Samples - Ag G/T (N=42)

Figure 12.27: 1/4 Core Duplicate Samples - Ag G/T (N=107)
**Figure 12.28**: 1/2 Core Duplicate Samples - Pb % (N=42)

**Figure 12.29**: 1/4 Core Duplicate Samples - Pb % (N=107)
12.3.4 Pulp Split and Reject Duplicate Split Check Sample Programs

January 2010 to July 2011

Apogee incorporated collection of third party check samples through all drill programs, including the Phase IV exploration program initiated in January 2010, with prepared pulp
splits and rejects selected from various holes for this purpose. In total, results from 442 data pairs were reviewed for this period and are presented in Figure 12.32: Pulp Splits and Reject Duplicate Samples - Ag G/T (N=442), Figure 12.33: Pulp Splits and Reject Duplicate Samples - Pb % (N=442) and Figure 12.34: Pulp Splits and Reject Duplicate Samples – Zn % (N=442) for Ag, Pb and Zn respectively. A high degree of correlation exists between sample pairs for all three metals and all show 0.999 correlation coefficients. Analytical results included in the check sample program were determined at the ALS facility in La Serena, Chile and original project results were from the ALS facility in Lima, Peru. Since January 2010 and starting at PUD 140 all second laboratory cross check analysis are done at SGS in Lima Peru.

![Figure 12.32: Pulp Splits and Reject Duplicate Samples - Ag G/T (N=442)](image-url)
July 2011 to January 2012 Program

The oxide zone diamond drilling program carried out between September 2011 and December 2011 and the oxide zone core resampling program carried out in January of 2012.
included analysis of 524 check samples under the same protocols described above for the immediately preceding Apogee program. Figure 12.35, Figure 12.36 and Figure 12.37 present Ag, Pb and Zn results, respectively. As in the earlier program, a high degree of correlation exists between sample pairs for all three metals, these being 0.99 in each case.

Figure 12.35: Pulp Splits and Reject Duplicate Samples – Ag G/T (N=524)

Figure 12.36: Pulp Splits and Reject Duplicate Samples – Pb % (N=524)
Figure 12.37: Pulp Splits and Reject Duplicate Samples – Zn% (N=524)

12.3.5 Mercator Program

August 2011 Site Visit Program

During the August 2011 site visit by Mercator, quarter core samples were obtained from Apogee drill core for purposes of independent check sample analysis. In total, 9 quarter core samples were collected to provide sample coverage across the Ag, Pb and Zn grade ranges represented in the deposit. Sample record details pertaining to the program appear in Appendix 1. The quarter core samples were collected from drill holes PUD111, PUD134, PUD140, PUD144, PUD175, PUD188, and PUD203 and were submitted for analysis to SGS del Peru S.A.C. A sample of certified reference material CDN-SE-1, a commercial blank sample, and 4 reject samples were added to the batch of core samples submitted by Mercator for quality control and quality assurance purposes.

Sample intervals of archived drill core were selected and marked by Mercator and then photographed prior to being placed in labeled plastic bags for shipment to the laboratory. Core intervals taken for check sample purposes were clearly identified by explanatory tags secured in the core boxes for archival reference purposes. All core sampling work was carried out at the Apogee core logging facility on the Pulacayo property by Mercator and Apogee field staff carried out sample cutting and bagging activities under Mercator supervision.

After standard crushing and pulverization, Ag, Pb, Zn and Cu levels were determined using SGS code ASS11B element analysis, which incorporates Aqua Regia digestion followed by AAS determination, and a Fire Assay – FAG313 finish for samples with Ag values greater than 300 g/t. Specific gravity measurements for all prepared sample pulps were also completed using pycnometer instrumentation (PHY03V Code).
Mercator core check sample results are compared to original Apogee results in Figure 12.38, Figure 12.39 and Figure 12.40 for Ag, Pb and Zn respectively. A correlation coefficient of 0.78 applies to the Ag data set but removal of one sample having an original value of 529 g/t and a check result of 21 g/t moves the correlation coefficient to 0.98. Pb results show good agreement between data sets, with a correlation coefficient of 0.96, while Zn data show higher variability between samples that is reflected in a correlation coefficient of 0.83.

Results of the Mercator check sample program are interpreted as showing that reasonable correlation exists between the original and check sample data sets in all but one instance of an anomalous Ag result. The value in question may suggest spatial heterogeneity of Ag within the sample interval but could also be a result of sample contamination or analytical error. Despite the lower correlation coefficient for Zn results, all original samples that returned a grade of greater than 2.00% also returned a check value greater than 2.00% and samples below that threshold provide a correlation coefficient of 0.99. These results confirm the anomalous character of mineralization within the sections sampled, particularly at grades below 2.00%, and indicate that lower correlation at higher grades may be related to heterogeneity of sulphide mineral distribution at the core sample scale rather than analytical error or sample contamination.

Figure 12.41, Figure 12.42, and Figure 12.43 show good correlation between original sample values and the check sample values for coarse reject materials submitted by Mercator and support correlation coefficients of 0.99 for all three metals. Blank sample results for the Mercator sample suite returned values below detection limits for all three metals and results for certified reference material CDN-SE-1 all fall within the mean ±2 standard deviations control limits for the material. CDN-SE-1 returned grades of 714.83 g/t Ag, 1.83% Pb, and 2.67% Zn.

Mercator considers results of the August 2011 independent QAQC program to be acceptable and to reflect consistency with the results reported by Apogee.
Figure 12.39: Mercator 1/4 Core Check Samples - Pb % (N=9)

Figure 12.40: Mercator 1/4 Core Check Samples - Zn % (N=9)
Figure 12.41: Mercator Reject Check Samples - Ag G/T (N=4)

Figure 12.42: Mercator Reject Check Samples – Pb % (N=4)
April 2012 Site Visit Program

During the April 2012 one site visit by Mercator, quarter core samples were obtained from Apogee oxide zone drill core for purposes of independent check sample analysis. In total, 23 quarter core samples were collected to provide sample coverage across the Ag, Pb and Zn grade ranges represented in the oxide zone. The quarter core samples were collected from drill holes PUD266, PUD230, PUD220, PUD251 and PUD256 and were submitted for analysis to SGS del Peru S.A.C.. A sample of certified reference material CDN-SE-1 and a commercial blank sample were added to the batch of core samples submitted by Mercator for quality control and quality assurance purposes.

Sample intervals of archived drill core were selected and marked by Mercator and then photographed prior to being placed in labeled plastic bags for shipment to the laboratory. Core intervals taken for check sample purposes were clearly identified by explanatory tags secured in the core boxes for archival reference purposes (Figure 12.44). All core sampling work was carried out at the Apogee core logging facility on the Pulacayo property by Mercator and Apogee field staff carried out sample cutting and bagging activities under Mercator supervision.

After standard crushing and pulverization, Ag, Pb, Zn and Cu levels were determined using the same preparation and analytical methods that were used in the previous Mercator program described above.

Mercator core check sample results are compared to original Apogee results in Figure 12.45, Figure 12.46 and Figure 12.47 for Ag, Pb and Zn respectively. Results are interpreted as showing that acceptable correlation exists between the original oxide zone analyses and the...
check sample data set. Results confirm the anomalous character of mineralization within the sections sampled but also show that lower correlation between sample pairs exists at higher grade levels. This may be related to heterogeneity of oxide and sulphide mineral assemblage distribution at the core sample scale rather than analytical error or sample contamination. Visual core inspection during the April 2012 site visit showed that higher Zn and Pb grades within the oxide zone often correspond with irregular zones in which remnants of sulphide zone mineralogy are preserved.

Figure 12.44: Photo of Mercator April 2012 Core Sample Interval
Figure 12.45: Mercator 1/4 Core Check Samples 2012 - Ag G/T (N=24)

Figure 12.46: Mercator 1/4 Core Check Samples 2012 - Pb % (N=24)
Collar coordinates for 14 drill holes completed during the 2011 oxide zone drilling program were checked by Mercator during the April 2012 site visit. A Garmin Map 60 handheld GPS unit was used to collect collar coordinate check values and these were then compared to validated resource database collar file values. Excellent correlation exists between the two data sets with respect to UTM easting and Northing values, with the total range in Easting variation being -2.1 m to +2.9 m and the total range for Northing being -4.5 m to +1.8 m. Values in the site visit elevation data set range between +7.9 m and +14.5 m above corresponding database collar values. This variation is systematic and project database values acquired using differential GPS equipment are considered to be more accurate than hand held GPS values obtained during the site visit. Tabulated results of the drill collar checking program are presented in Table 12.2 and Mercator is of the opinion that these results adequately confirm database collar locations.

**Table 12.2: Collar Coordinate Check Program Results**

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<tr>
<th>Drillhole ID</th>
<th>GPS East (m)</th>
<th>GPS North (m)</th>
<th>GPS Elev. (m)</th>
<th>Database East (m)</th>
<th>Database North (m)</th>
<th>Database Elev. (m)</th>
<th>East Change (m)</th>
<th>North Change (m)</th>
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<td>4,403.3</td>
<td>-0.7</td>
<td>1.8</td>
<td>11.4</td>
</tr>
<tr>
<td>11PUD239</td>
<td>740,047.9</td>
<td>7,744,595.6</td>
<td>4,409.7</td>
<td>740,048.6</td>
<td>7,744,596.8</td>
<td>4,394.0</td>
<td>-0.8</td>
<td>-1.2</td>
<td>15.7</td>
</tr>
<tr>
<td>PUD224</td>
<td>740,098.6</td>
<td>7,744,858.2</td>
<td>4,383.5</td>
<td>740,100.0</td>
<td>7,744,862.1</td>
<td>4,376.0</td>
<td>-1.4</td>
<td>-3.9</td>
<td>7.5</td>
</tr>
<tr>
<td>11PUD241</td>
<td>740,197.2</td>
<td>7,744,580.3</td>
<td>4,344.5</td>
<td>740,199.3</td>
<td>7,744,583.1</td>
<td>4,333.0</td>
<td>-2.1</td>
<td>-2.8</td>
<td>11.5</td>
</tr>
<tr>
<td>PUD217</td>
<td>740,248.2</td>
<td>7,744,349.9</td>
<td>4,345.3</td>
<td>740,250.0</td>
<td>7,744,350.0</td>
<td>4,335.4</td>
<td>-1.8</td>
<td>-0.1</td>
<td>9.9</td>
</tr>
<tr>
<td>PUD045</td>
<td>740,094.7</td>
<td>7,744,434.1</td>
<td>4,313.5</td>
<td>740,094.3</td>
<td>7,744,435.7</td>
<td>4,305.6</td>
<td>0.4</td>
<td>-1.6</td>
<td>7.9</td>
</tr>
</tbody>
</table>
13 MINERAL PROCESSING AND METALLURGICAL TESTING

To date, four metallurgical programs are complete and a fifth (variability testing) is underway. Resource Development Inc. (RDi), Denver, USA in 2003, conducted the first program, the second by UTO (Universidad Técnica de Oruro), Oruro, La Paz, Bolivia in 2009. ED&ED Ingeniería y Servicios, Lima, Peru completed the third program in 2011 and the fourth program was conducted by UTO and Maelgwyn Mineral Services (MMSA) Laboratory in South Africa during the 2012. Maelgwyn Mineral Services (MMSA) is currently conducting the fifth variability program

In summary, the following test work programs were undertaken and completed:

- Program by RDi (2003) on the first sample set (stage 1).
- Program by UTO (2011) on the second sample set (stage 2).
- Program by ED&ED (2011) on the third sample set (stage 3).
- Program by UTO and Maelgwyn (2012) on the fourth sample set and the fifth sample set, respectively (stage 4).
- Paste thickening test work by FLSmidth on stage 3 tailings samples (2011).
- Lead and zinc concentrate assays by SGS (2012).
- Concentrate filtration test work by Andritz (2012).

This section of the report summarizes the outcomes and findings of the metallurgical programs. The test work results used in the Technical-Financial Model

13.1 Resource Development Inc (RDi) Metallurgical Test Program (Stage 1)

In 2002, RDi in USA, received material from the Pulacayo Project (nine boxes, 120 kg). Individual drill intervals were combined to generate a single bulk composite sample. This sample was prepared out of a set of drill core intervals provided (drill holes PuD005 and PuD006).

RDi undertook preliminary metallurgical test work to evaluate the silver and sulfide base metals recovery potential utilizing the composite sample from the Pulacayo property in Bolivia. This test work included:

<table>
<thead>
<tr>
<th>11PUD226</th>
<th>740,190.7</th>
<th>7,744,472.6</th>
<th>4,331.8</th>
<th>740,189.3</th>
<th>7,744,472.2</th>
<th>4,318.9</th>
<th>1.5</th>
<th>0.4</th>
<th>12.9</th>
</tr>
</thead>
<tbody>
<tr>
<td>11PUD249</td>
<td>740,647.5</td>
<td>7,744,387.0</td>
<td>4,294.3</td>
<td>740,644.6</td>
<td>7,744,388.6</td>
<td>4,282.8</td>
<td>2.9</td>
<td>-1.6</td>
<td>11.5</td>
</tr>
<tr>
<td>11PUD247</td>
<td>740,701.4</td>
<td>7,744,397.1</td>
<td>4,294.8</td>
<td>740,699.6</td>
<td>7,744,396.9</td>
<td>4,263.1</td>
<td>1.8</td>
<td>0.2</td>
<td>11.7</td>
</tr>
<tr>
<td>11PUD250</td>
<td>740,794.4</td>
<td>7,744,452.0</td>
<td>4,298.4</td>
<td>740,794.8</td>
<td>7,744,452.8</td>
<td>4,265.6</td>
<td>-0.4</td>
<td>-0.7</td>
<td>12.8</td>
</tr>
<tr>
<td>11PUD257</td>
<td>740,901.8</td>
<td>7,744,320.2</td>
<td>4,260.9</td>
<td>740,900.6</td>
<td>7,744,321.7</td>
<td>4,248.1</td>
<td>1.2</td>
<td>-1.5</td>
<td>12.8</td>
</tr>
<tr>
<td>11PUD260</td>
<td>740,951.2</td>
<td>7,744,301.3</td>
<td>4,250.1</td>
<td>740,951.8</td>
<td>7,744,302.0</td>
<td>4,239.0</td>
<td>-0.6</td>
<td>-0.7</td>
<td>11.1</td>
</tr>
<tr>
<td>11PUD261</td>
<td>740,949.8</td>
<td>7,744,257.9</td>
<td>4,245.5</td>
<td>740,949.1</td>
<td>7,744,262.4</td>
<td>4,234.5</td>
<td>0.8</td>
<td>-4.5</td>
<td>11.0</td>
</tr>
</tbody>
</table>
13.1.1 First Set of Samples

Several pieces of rock from the composite were selected for in-place bulk density determination and for mineralogical study. The remaining composite sample was crushed to nominal 6 # (3.36 mm), blended and split into 2 kg and 10 kg charges for metallurgical test work.

A representative portion of the composite sample was pulverized and submitted for chemical XRF/XRD analyses. A summary of the estimated core grades and assay values for this single composite (first sample composite set) is shown in Table 13.1. The head analyses of the sample were significantly higher than the values calculated from drill core data.

### Table 13.1 Sample Composite Grades and Head Assays

<table>
<thead>
<tr>
<th>Sample ID</th>
<th>Estimated Core Grades</th>
<th>Head Assays</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Ag, g/t</td>
<td>Pb, %</td>
</tr>
<tr>
<td>Bulk Composite</td>
<td>415</td>
<td>1.84</td>
</tr>
</tbody>
</table>

The sample contained 2,120 ppm (0.212%) Copper and 910 ppm (0.09%) Arsenic, which may have detrimental effects on the concentrates.

13.1.2 Mineralogy

Mineralogical studies of pieces of drill core from drill holes PuD005 and PuD007 were investigated microscopically in polished sections for characterization of sulphide minerals with the highlights following:

- The predominant sulfide mineral in the samples was pyrite with minor amounts of galena, sphalerite and marcasite.
- Sphalerite and galena was locked in pyrite-marcasite mineralization.
- Native silver was not detected in the samples.
- Replacement rimming of sphalerite by late galena and tennanite associated with galena was observed in the drill core sample. This could result in product contamination during froth flotation recovery (intertwined species).
- The host rock contained significant amount of clay material which resulted in problems in settling of tailings and flotation pulp rheology.
13.1.3 Bulk Density

Two rock samples were selected from drill core samples for in-place bulk densities. Each sample was weighed, waxed and re-weighed, then immersed in water to determine volume of the sample. The in-place bulk density of the sample ranged from 2.2 t/m$^3$ (-73.5 #) to 3.4 t/m$^3$ (-85 #).

13.1.4 Leach Tests

Silver Minerals in the ore were not amenable to leaching, even at the fine grind size. The silver extraction was extremely poor at the fine grind size of 150 µm, less than 20%, indicating that silver was not present in the ore in the native form. The NaCN and lime consumptions were high in the leach tests.

A representative sample of the solid was submitted for determination of silver content by fire assay technique.

13.1.5 Gravity Concentration Tests

A gravity test was performed to determine the potential of recovering silver and other sulfide minerals by gravity concentration. A 2 kg sample was stage ground to P100 of 48 # (0.297 mm) and processed over a Deister table. The Deister concentrate was re-processed on a Gemini table.

The results indicated that the sample was not amenable to gravity concentration for silver recovery since the recovery of silver minerals was only 17% in the Gemini concentrate.

13.1.6 Flotation Tests

13.1.6.1 Grindability

A grind study was initiated to determine the laboratory grind time vs. particle size relationship. The grind time requirement to achieve a P80 of 150 # (104 µm) was determined from the grind data, this being 20 minutes for the composite sample.

13.1.6.2 Bench Scale Open Circuit Flotation Tests

Bench scale open circuit flotation tests using the composite samples was performed using the flotation reagent suite developed for the San Cristobal Project. The flotation process consisted of grinding the ore with lime (pH > 8) and ZnCN followed by conditioning the ground pulp with more ZnCN, AP-238 collector, sodium isopropyl xanthate (SIPX) and frother, methyl isobutyl carbinol (MIBC). The lead minerals were floated and the rougher concentrate was cleaned once. The lead rougher tails were conditioned with lime and copper sulfate and zinc rougher concentrate was collected using SIPX and MIBC. The zinc rougher concentrate was again cleaned once.

The overall silver recovery in the lead and rougher concentrates was 97.1%.

The lead cleaner concentrate recovered 2.8% of the weight, 84.6% of lead, 3.1% of zinc and 46.9% of silver. The lead concentrate assayed 60.8% Pb, 4.22% Zn and 8,440 g/t Ag.
The zinc cleaner concentrate recovered 7.8% of weight, 1.3% of lead, 84.7% of zinc and 38.8% of Ag. The concentrate assayed 0.324% Pb, 41.2% Zn and 2,463 g/t Ag.

13.1.6.3 Locked Circuit Flotation Tests

Large scale two cycle locked cycle flotation tests were performed using the process flowsheet similar to that developed for San Cristobal deposit. The lead concentrate assaying 62.2% Pb, 4.46% Zn and 10,891 g/t Ag, recovered 3.1% weight, 88.8% of lead, 3.9% of zinc and 63.4% of silver. The zinc concentrate assaying 61.5% Zn, 0.9% Pb and 3,303 g/t Ag, recovered 5% weight, 87.6% of zinc, 2.1% of lead and 31.3% of silver.

The tailings were very difficult to settle due to high proportions of clay in the ore, which will impact the process flow sheet and overall plant design.

The lead and zinc third cleaner concentrates were analyzed for impurities. The results indicate that penalties may be incurred on the concentrates for several impurities, as they are higher than the norm for smelter contracts.

13.2 UTO Metallurgical Test Program (Stage 2)

UTO conducted a metallurgical test work program on these three samples comprising comminution (only Bond Ball Work Index), open circuit flotation tests (OCT), locked cycle flotation tests (LCT), OCT tailings (non-float) size by size analyses and OCT tailings (non-float) sedimentation tests.

Clays mineralogy studies were not carried out to determine the presence of shrinking-swelling clays (Smectites) and other type of clays that may produce very fine slimes. However, during the course of the metallurgical test work, slimes were produced affecting the flotation performance. The host rock contained significant amount of clay material, which resulted in problems in settling of tailings and flotation pulp rheology.

13.2.1 Second Set of Samples

In 2009, the laboratory facility of Universidad Nacional de Oruro (UTO) in Bolivia, received material from the Pulacayo Project. Individual drill intervals were combined by UTO as specified by Micon and ASL to generate three (3) composite samples.

These three composites represent a higher grade, a medium grade and a lower grade composite sample (roughly 100 kg each). All the samples were metallurgical tested by UTO and the products were assayed by ALS Chemex in Lima, Peru. Silver, zinc, lead and copper concentrations were determined using an aqua regia digestion followed by spectroscopy. For samples with silver value exceeding 300 ppm the fire assays (ALS code AA26) method was used.

A summary of the estimated core grades and assay values for all three composites (first sample composite set) is shown in Table 13.2. Core intervals were chosen that would result in low, medium and high grade composites. The spatial distribution of the intercepts used to prepare the bulk samples is well documented.
Table 13.2 First Sample Composite Set Grades and Head Assays

<table>
<thead>
<tr>
<th>Sample ID</th>
<th>Core Size, kg</th>
<th>Details</th>
<th>Estimated Core Grades</th>
<th>Head Assays</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Ag, g/t</td>
<td>Pb, %</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Zn, %</td>
<td>Ag, g/t</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Pb, %</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Zn, %</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Cu, %</td>
</tr>
<tr>
<td>High Grade (LA)</td>
<td>105.7</td>
<td>Half Core</td>
<td>301.4</td>
<td>1.57</td>
</tr>
<tr>
<td>Medium Grade (LM)</td>
<td>93.9</td>
<td>Half Core</td>
<td>204</td>
<td>0.69</td>
</tr>
<tr>
<td>Low Grade (LB)</td>
<td>102.2</td>
<td>Half Core</td>
<td>41</td>
<td>0.75</td>
</tr>
</tbody>
</table>

13.2.2 Comminution Test Work

All samples were submitted for a Bond Ball Mill Work Index (BWI) test. The results are summarized in Table 13.3.

In the determination of BWI, a sample of representative ore at 100% +3.35 mm is stage crushed to 100% -6.35 mm.

Table 13.3 Bond Ball Work Index Tests Results Summary

<table>
<thead>
<tr>
<th>Sample ID</th>
<th>Cut Mesh, µm</th>
<th>Work Index (BBWi)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>UTO, kW-h/t</td>
</tr>
<tr>
<td>Average</td>
<td>212</td>
<td>10.3</td>
</tr>
<tr>
<td></td>
<td>150</td>
<td>11.4</td>
</tr>
<tr>
<td></td>
<td>106</td>
<td>12.9</td>
</tr>
<tr>
<td>Standard Deviation (SD)</td>
<td>212</td>
<td>0.49</td>
</tr>
<tr>
<td></td>
<td>150</td>
<td>0.88</td>
</tr>
<tr>
<td></td>
<td>106</td>
<td>1.51</td>
</tr>
<tr>
<td>Design (Average + 1xSD)</td>
<td>212</td>
<td>10.8</td>
</tr>
<tr>
<td></td>
<td>150</td>
<td>12.3</td>
</tr>
<tr>
<td></td>
<td>106</td>
<td>14.4</td>
</tr>
</tbody>
</table>

The Bond ball mill grindability test was performed at 212 µm (65 #), 150 µm (100 #) and 106 µm (150 #). The test results categorized the samples as medium to hard. Abrasion index, crushing work index and rod work index tests were not performed.

13.2.3 Specific Gravity

A set of specific gravity test work was conducted on various samples as summarized below.
Table 13.4 Specific Gravity Testwork Summary

<table>
<thead>
<tr>
<th>Sample ID</th>
<th>ED&amp;ED Lab SG Tests (*)</th>
<th>UTO Lab BBWi Tests</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Grind Size P100, mm</td>
<td>Cut Mesh, µm</td>
</tr>
<tr>
<td>Min</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Max</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Average</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Standard Deviation (SD)</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Design (Average + 1.8xSD)</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

(**) Method of pycnometer; (*) results taken from ED&ED metallurgical program which will be addressed after.

13.2.4 Flotation Test work

The focus of the flotation test work was on lead (and therefore silver) recovery and both batch open circuit and closed circuit flotation tests were conducted to derive the metallurgical performance predictions. Investigations into copper-lead separation were not carried out.

Determination of likely recoveries in an actual industrial scale flotation plant with recycling of intermediate streams requires closed circuit flotation testing. These tests were carried out in accordance to the flowcharts shown in Figure 13.1 and Figure 13.2.
The locked cycle flotation (LCT) test results, corresponding to the High Grade (LA) sample (grind size of d80 150 µm), indicate that conventional selective lead-silver and zinc-silver flotation techniques are able to recover almost 56% of silver in the lead concentrate (with a mass pull of 2.39%) and 26.7% of silver in the zinc concentrate (with a mass pull of 3.76%). The lead and zinc concentrate grades are 52.4% and 57%, respectively. Also, the lead and zinc recoveries are estimated as 78.8% and 81.1%, respectively (Test no 1). The silver grades in the lead and zinc concentrates were reported as 6,620 g/t and 2,010 g/t, respectively.

These results are further improved by proceeding with a cyclone desliming step ahead rougher flotation (see Figure 13.) to reduce the amount of slimes in the feed (Test N° 2). However, around 8.5% Ag is lost in the slimes fraction or discarded overflow (18.7% in weight). Native silver occurs as fine particles of size smaller than 20 µm. Lime consumption is reported to be as high as 19.3 kg/t.

Similarly, the locked cycle flotation (LCT) tests results corresponding to the Medium Grade (LM) sample (grind size of 150 µm), indicated that it is possible to recover almost 33.6% of silver in the lead concentrate (with a mass pull of 1.2%) and 49.8% of silver in the zinc concentrate (with a mass pull of 3.7%). The lead and zinc concentrate grades are 51% and 58.3%, respectively. Also, the lead and zinc recoveries are estimated as 74.3% and 82.6%, respectively (Test no 3). The silver grades in the lead and zinc concentrates were reported as 6,220 g/t and 2,990 g/t, respectively.
These results are further improved by proceeding with a cyclone desliming step ahead rougher flotation (see Figure 13.2) to reduce the amount of slimes in the feed (Test N° 2). However, around 9.9% Ag is lost in the slimes fraction or discarded overflow (20% in weight). Native silver occurs as fine particles of size smaller than 20 µm (see section 13.3.2).

The Low Grade (LB) sample with a grind size of d80 150 µm, the LCT results shows that the differential flotation is able to recover almost 30% of silver in the lead concentrate (with a mass pull of 0.53%) and 20.9% of silver in the zinc concentrate (with a mass pull of 1.28%). The lead and zinc concentrate grades are 46.7% and 44.8%, respectively. Also, the lead and zinc recoveries are estimated as 34.6% and 50.9%, respectively. The silver grades in the lead and zinc concentrates were reported as 2,600 g/t and 749 g/t, respectively (Test N° 1). Lead recovery was low with a substantial amount of the zinc tied up in the lead circuit cleaner concentrate.

These results are further improved by proceeding with cyclone desliming step ahead rougher flotation to reduce the amount of slimes in the feed (Test N°2 and Test N° 3). However, 5% to 11% Ag is lost in the slimes fraction or discarded overflow (27% and 26% in weight). Native silver occurs as fine particles of size smaller than 20 µm (see section 13.3.2). The lime consumption is reported to be as high as 14.3 kg/t.

It is not clear from the test work data and reports the extent to which this closed circuit test approached the locked cycle test standards commonly used in the industry. There is no information on how many cycles were performed (usually six) and whether circuit stability was reached as the circulating loads of middlings approached a sort of equilibrium. The most likely consequence of not attaining equilibrium is that concentrate grades may be over-estimated and recoveries under-estimated.

However, the results seem to be reasonable and in accordance with expectations from the mineralogy of the ore. Further testing was undertaken by ED&ED and their findings are shown later. These results constitute the design basis for the flow sheet.

Full open-circuit tests (OCT) of sulphide minerals flotation were conducted initially on each sample as a proof of concept of the overall circuit and to establish a workable set of flotation conditions and reagents. These test demonstrated that sulphide flotation to saleable lead and zinc concentrates at acceptable (for batch tests) recoveries was possible.

OCT flotation flow sheet was developed incorporating a moderate grind size of 150 µm (P80). The pH in the lead rougher circuit was elevated to a value in the range of 9 to 10 using lime. The pH in the zinc rougher circuit was maintained in the range of 10 to 11. AF-242 and Z-11 were used as the lead and zinc collectors, respectively. A regrind step to produce high grade concentrate was not considered. In the lead flotation circuit, Sodium Cyanide (NaCN) and Zinc Sulphate (ZnSO₄·7H₂O) were used as pyrite and sphalerite depressors, respectively. The pH of the cleaning stages in the lead concentrate was maintained at the range of 9 to 10 using lime. The pH in the zinc circuit cleaning stages was elevated to activate spharelite and lime was used to raise the pH to above pH 10 to depress iron (pyrite).

According to the mineralogy studies carried out by ED&ED (see section 13.3.2), galena-sphalerite assemblage (intertwined specimens) is present at some extend. This may be reason for some not desired zinc content in the lead concentrates. In addition, unwanted
activation of sphalerite and pyrite by copper ions (Cu$^{2+}$) dissolved from copper minerals (mainly chalcopyrite) hampers lead flotation selectivity.

13.2.5 OCT Tailings and Flotation Feed Size by Size Analyses

Results indicated that, most of the valuable metal losses occur at mesh sizes less than 74 µm (200 #) for all the OCT tailings samples tested.

13.2.6 Sedimentation Test work

A set of thickening tests were run on tailings slurry samples of the OCT flotation tests. The results indicated that higher sedimentation velocities (lower unit areas) are obtained for the tests carried out after a cycloning step (desliming).

13.3 ED&ED Lab Metallurgical Test Program (Stage 3)

In 2011, the laboratory facility of ED&ED Ingeniería y Servicios S.A.C. (ED&ED) in Peru, received material from the Pulacayo Project. Two (2) head samples were sent to ASL who were contracted to undertake mineralogical analyses (PUL-HG and PUL-LG) and a series of flotation tests.

The initial ED&ED flotation test work was not successful as the zinc floated with the lead in the lead differential float. ED&ED then pre-conditioned with activated carbon and subsequent differential flotation was moderately successful.

13.3.1 Third Set of Samples

ED&ED lab received material from the Pulacayo Project. Individual drill cores were combined to generate a high grade composite (PUL-HG) and a low grade composite (PUL-LG). A summary of the estimated core grades and assays are shown in Table 13.6.

<table>
<thead>
<tr>
<th>Sample ID</th>
<th>Core Size, m</th>
<th>Estimated Core Grades</th>
<th>Head Assays (*)</th>
</tr>
</thead>
<tbody>
<tr>
<td>PUL-HG</td>
<td>23.22</td>
<td>Ag, g/t: 705.6 Pb, %: 3.45 Zn, %: 7.98 Cu, %: 0.19</td>
<td>Ag, g/t: 225.1 Pb, %: 1.58 Zn, %: 2.24 As, %: 0.11</td>
</tr>
<tr>
<td>PUL-LG</td>
<td>250</td>
<td>Ag, g/t: 132.9 Pb, %: 1.36 Zn, %: 2.77 Cu, %: 0.03</td>
<td>Ag, g/t: 178.23 Pb, %: 0.91 Zn, %: 1.79 As, %: 0.09</td>
</tr>
</tbody>
</table>

13.3.2 Mineralogy

The results of this study indicate that the main mineral specimens present in both samples were sphalerite, galena, pyrite and quartzite gangue. Galena-sphalerite assemblages (intertwined specimens) are present to some extend. The latter may be the reason for some not desired zinc content in the lead concentrates.
13.3.3 Flotation Test work

Twelve (12) open circuit flotation tests (OCT were conducted on each of the samples PUL-HG and PUL-LG. These tests were carried out to confirm the previous flotation results (UTO) and to evaluate the effect of flotation response at a finer grind sizes as seen in the flowcharts.

The slurry samples were pre-treated with activated carbon to adsorb the metallic ions (Cu$^{2+}$) present in the mineral. The Cu$^{2+}$ ions activate sphalerite and pyrite to float in the lead circuit and the carbon pre-treatment removes the Cu$^{2+}$ ions in order to obtain higher lead concentrate grades. It was also reported that the lead grade was constrained by the presence of mixed, intertwined galena-sphalerite assemblage.

The flotation tests, carried out on the PUL-HG samples indicated that it is possible to obtain commercial lead and zinc concentrates with grades of lead and zinc of 42.1% and 43%, respectively. The concentration of silver in the lead and zinc concentrates were reported as 7,010 g/t and 198.2 g/t, respectively (Test no 5, PUL-HG-5 sample). The straightforward conventional selective lead-silver and zinc-silver flotation techniques after carbon pre-treatment are able to recover 85.7% of silver in the lead concentrate (with a mass pull of 3.1%) and 2.93% of silver in the zinc concentrate (with a mass pull of 3.75%). The lead and zinc recoveries are estimated as almost 80% and 77.8%, respectively.

The lime consumption for the open circuit flotation test PUL-HG-5 is 28.3 kg/t.

The flotation tests, carried out on the PUL-LG samples indicated that it is possible to obtain commercial lead and zinc concentrates with grades of lead and zinc of 41% and 43.1%, respectively. The concentration of silver in the lead and zinc concentrates were reported as 6,734 g/t and 207 g/t, respectively (Test no 6, PUL-LG-6 sample). The straightforward conventional selective lead-silver and zinc-silver flotation techniques after carbon pre-treatment are able to recover 74% of silver in the lead concentrate (with a mass pull of 1.95%) and 3.27% of silver in the zinc concentrate (with a mass pull of 2.8%). The lead and zinc recoveries are estimated as almost 77.6% and 71.9%, respectively.

The lime consumption for the open circuit flotation test PUL-LG-6 is 26.2 kg/t.

In overall, better flotation (open circuit tests) performances are obtained at a grind size of P80 of 74 µm. Locked cycle tests at this grind size will be necessary to confirm these results.

13.4 FLSmidth Paste Thickening Testwork

A set of paste thickening tests were run on dry samples of the flotation test (tailings) carried out by ED&ED (PUL-HG-5 and PUL-LG-6. These samples were received by FLSmidth from Apogee Silver staff.

Thickening tests were conducted on dry flotation tailings samples provided by the Client to investigate the performance of FLSmidth Deep Cone Paste thickening technology. Screening flocculent tests were carried out. Anionic flocculent (Floenger PHP 50 Plus) was selected to improve sedimentation performance based on settling rates and observed visual supernatant clarity.

Experience has shown that it is difficult to scale paste flow characteristics from small-scale tests to full-scale pipeline conditions, pilot-scale pumping tests are usually necessary. PLC -
control is essential because only slight changes in moisture content cause wide variations in viscosity and pipe friction. Some water release (bleed) was observed in all of the samples tested.

### 13.5 SGS Lead and Zinc Concentrate Assays

The ED&ED lab flotation concentrates (open circuit tests) were submitted to SGS lab to get an idea of the deleterious elements in the concentrate. These values were used in the Net Smelter Return (NSR) calculations.

Mineralogical analyses of concentrate samples from composites PUL-HG (high grade sample) and PUL-LG (low grade sample) have been included.

The results showed that the lead concentrate assayed 47.2% Pb and 6,273 g/t with 1.3% Cu, 1.45% As and 1.23% Sb. The zinc concentrate assayed 53.8%Zn with negligible copper, arsenic or antimony. The lead, silver and zinc concentrate grades are in agreement with the LCT carried out before.

Concentrations of deleterious elements appear below typical smelter penalty thresholds, with arsenic appearing as the principal penalty element.

### 13.6 Concentrate Filtration Test work

Concentrate test work results were not available at the time of writing this report. It is recommended that test work be performed on a representative sample provided by the Client in order to confirm the pressure filter sizing (Andritz).

### 13.7 UTO Metallurgical Test program (Stage 4)

In 2012, UTO received a single composite sample and conducted a metallurgical test work program comprising:

- A single collective flotation test
- A series of open circuit differential flotation (with a de-sliming step)
- A single locked cycle flotation test (with de-sliming step)
- PORCO flow sheet testing

This test work was designed to explore the flotation response of the ore to conventional differential flotation and to establish the operating conditions, reagent scheme and consumptions.

#### 13.7.1 Fourth Set of Samples

Apogee (ASL) provided Universidad Técnica de Oruro (UTO) with a bulk composite sample prepared by ASL out of drill cores with grain sizes up to 76.2 mm (3’"). A summary of the head core grades and assays are summarized in Table 13.7.
Table 13.6 Fourth Sample Composite Head Core Grades and Assays

<table>
<thead>
<tr>
<th>Sample ID</th>
<th>Core Size, kg</th>
<th>Details</th>
<th>Estimated Core Grades</th>
<th>Head Assays</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Ag, g/t</td>
<td>Pb, %</td>
</tr>
<tr>
<td>UTO Sample</td>
<td>68.6</td>
<td>Half Core</td>
<td>225.8</td>
<td>0.63</td>
</tr>
</tbody>
</table>

13.7.2 Sample Preparation and Testing Procedure

The bulk sample preparation and testing procedure was as follows:

- Crushing
- Sample homogenization, quartering, and sample sizing for flotation tests
- Single bulk flotation test (1)
- Open circuit differential flotation tests (4)
- Single locked cycle differential flotation test (1)
- Bulk flotation tests under PORCO flowsheet (6)

13.7.3 Open Circuit Differential Flotation with a De-slime Step

A single bulk flotation test (Test N° 1) was carried out to identify the floating fraction of the ore (sulphides) and to have an indication of metal losses. In addition, a series of open circuit flotation circuit were carried out to determine the flotation performance (range of concentrates grades and recoveries obtained by differential flotation). Then, a locked cycle test was carried out under the most suitable conditions (Test N°6).

The first exploratory test indicated that silver recovery to bulk concentrate is about 72%, while the lead and zinc recoveries are approximately 66% and 78%. The floating fraction account for about 13% w/w while the slimes fraction is 18%, the rest is lost as final tailings. Lead and silver losses are up to 23% and 13%, respectively.

A series of open batch flotation tests were performed. The results indicated that lead recovery is in between of 48% and 54%, while zinc recovery is in the range from 50.1% to 72%. Total silver recovery to both lead and zinc concentrates is in between 30% and 68%. Lead concentrate grades range from 33.5% to 59%, zinc concentrate grades is in between of 49% and 55%. Similarly, silver grades in both concentrates range from 9,875 g/t to 15,333 g/t.

A single locked cycle test (LCT), a repetitive batch used to simulate a continuous circuit where all the intermediate material added to the appropriate location in the flowsheet, was conducted to produce a metallurgical projection of the sample tested and to assess if the flowsheet and reagent suite is stable.

A good locked cycle test achieves steady state, typically over the last three cycles. Steady state implies both stability and mass conservation. Stability implies constancy.

It was not indicated whether the test reached stability or whether mass conservation was achieved. Assuming that steady state was reached, the results indicated that lead and zinc
recoveries were 60.1% and 76.5%, respectively. Lead concentrate assayed 11,114 g/t Ag, 49.1% Pb and 4.81% Zn. Additionally, the metal values in the zinc concentrate were: 2,220 g/t Ag, 2.29% Pb and 48.6% Zn. Concentrates account for about 2.9% w/w of the feed (0.81% lead and 2.1% zinc). Silver metal loss in the slimes is as high as in the tailings. Lead and silver losses in the final tails are 23.1% and 9.12%. Some factors affecting selectivity in the differential flotation of lead – zinc ores needed to be further explored.

The LCT reagents consumption is as follows: 150 g/t ZnSO₄, 9 kg/t lime, 50 g/t Z to 11, 60 g/t MIBC; 75 g/t ZnSO₄: NaCN complex, 40 g/t AP3418 and 250 g/t CuSO₄.

The PORCO flowsheet is basically a bulk flotation followed by lead and zinc flotation, this processing route should be carried out at high pH (12.2) intended to depress pyrite at the outset. However, the Pulacayo ore did not respond well mainly because of lead and silver selectivity issues and high consumption of acid (H₂SO₄) to drop the pH to a level suitable for lead flotation after the bulk stage. Herein, the results of these tests are not present and will not be further discussed.

13.8 MAELGWYN Metallurgical Test program (Stage 4)

Maelgwyn Mineral Services Africa (MMSA) carried out laboratory flotation optimization test work on ore samples from the Pulacayo property.

The objectives of the test work were:

Initially to test the flotation conditions supplied by Apogee on the core samples to determine the metal recoveries and grades achievable by differential flotation of the Pb and Zn minerals

Subsequently to optimize the flotation conditions for effective differential of the Pb and Zn minerals and to achieve saleable grades Pb and Zn concentrates.

Locked Cycle testing of the optimized flotation conditions on selected variability core samples (7 cycles, steady state reached over the last three cycles).

13.8.1 Fifth Set of Samples (Variability Samples)

Initially a bulk sample of approximately 300 kg was delivered to MMSA for the test work. After limited test work on this sample it was realized that the head grades were lower than expected. Apogee Silver then supplied another higher head grade bulk sample of approximately 500 kg. Later, during the test work Apogee Silver also supplied 9 variability core samples.

Each of the bulk and variability samples were crushed to <3 mm, blended and split into representative portions.

13.8.2 Head Assays

A sample portion of each was submitted to Setpoint Laboratories for head assays, the results are shown in Table 13.8. The results show a large variation in the head grades of the different samples.
### Table 13.7 Samples Assays

<table>
<thead>
<tr>
<th>Sample ID</th>
<th>Grades</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Pb, %</td>
<td>Zn, %</td>
</tr>
<tr>
<td>Low Grade Composite</td>
<td>0.45</td>
<td>1.32</td>
</tr>
<tr>
<td>High Grade Composite</td>
<td>2.18</td>
<td>2.7</td>
</tr>
<tr>
<td>PUD 147</td>
<td>2.72</td>
<td>2.02</td>
</tr>
<tr>
<td>PUD 093</td>
<td>3.07</td>
<td>4.34</td>
</tr>
<tr>
<td>PUD 001</td>
<td>4.3</td>
<td>5.96</td>
</tr>
<tr>
<td>PUD 106</td>
<td>1.52</td>
<td>1.86</td>
</tr>
<tr>
<td>PUD 146</td>
<td>1.44</td>
<td>2.38</td>
</tr>
<tr>
<td>PUD 037</td>
<td>2.15</td>
<td>1.63</td>
</tr>
<tr>
<td>PUD 170</td>
<td>1.65</td>
<td>1.37</td>
</tr>
<tr>
<td>PUD 203</td>
<td>0.78</td>
<td>0.8</td>
</tr>
<tr>
<td>PUD 159</td>
<td>1.74</td>
<td>0.81</td>
</tr>
</tbody>
</table>

#### 13.8.3 Grindability

Laboratory Rod Milling curves were produced for all the samples to determine the milling times required to achieve different grinds. This was done by milling 1 kg sample portions for different time periods. The milled products were wet screened at 53 μm and dry screened at 150 μm, 106 μm, 75 μm and 53 μm.

The main target grind for the flotation tests was d80 75 μm. From this information, the milling times to achieve this target were determined. The milling times required for the samples indicated a high degree of variability in hardness between the sample types.

#### 13.8.4 Flotation Experimental Procedures and Results

In total 65 open circuit flotation tests (exploratory test work) were carried out and 4 Locked Cycle flotation tests. These investigated:

- Initial Flotation conditions supplied by apogee Silver (11 tests)
- Initial Variations of known Pb/Zn differential flotation (10 tests)
- Differential Pb/Zn sulphide rougher flotation (26 tests)
- Pb rougher and cleaner rate flotation tests (10 tests)
- Pb and Zn differential cleaner flotation tests (4 tests)
- Pb only cleaner flotation tests (4 tests)

All the flotation products were assayed by XRF. Later, in the test work, the high grade concentrates were assayed by chemical titration methods, which are more accurate for high grade products.
Locked Cycle flotation test work was conducted on selected ore samples to determine the concentrate grades and recoveries achievable in a closed circuit. The samples selected were:

- High grade composite:
  - PUD 147
  - PUD 001
  - PUD 106

The conditions for the locked cycle tests were as follows:

- A Pb concentrate of 54.6% Pb at a recovery of 91.9% was achieved. A Zn concentrate of 40.8% Zn at a recovery of 82.9% was achieved. The overall silver recovery was 89.2% on the High Grade Composite Sample.

- A Pb concentrate of 65.8% Pb at a recovery of 90.5% was achieved. A Zn concentrate of 36.8% Zn at a recovery of 78.6% was achieved. The overall silver recovery was 67.8% on sample PUD-147.

- A Pb concentrate of 69.1% Pb at a recovery of 92.6% was achieved. A Zn concentrate of 56.4% Zn at a recovery of 89.6% was achieved. The overall silver recovery was 93.5% on sample PUD-001.

- A Pb concentrate of 62.6% Pb at a recovery of 87.8% was achieved. A Zn concentrate of 38.5% Zn at a recovery of 86.2% was achieved. The overall silver recovery was 76.9% on sample PUD-106.

- The locked cycle results show that good Pb recoveries of between 88% and 93% can be achieved with a large variation in head grade from 1.5% Pb to 4.3% Pb.

- The silver recoveries ranged between 67% and 93.5% with a variation in head grade of between 136 g/t Ag and 375 g/t Ag.

In order to produce head grade versus recovery curves for Pb and Ag, a low Pb head grade point was needed. Sample PUD 203 with a head grade of 0.78% Pb was then chosen and a full open circuit flotation test carried out on it. It resulted in a Pb concentrate grade of 52.9% Pb at a recovery of 75.3% Pb. The outlier Ag result for PUD 147 was not used.
14 MINERAL RESOURCE ESTIMATE

14.1 General

The definition of mineral resource and associated mineral resource categories used in this report are those recognized under National Instrument 43-101 and set out in the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves Definitions and Guidelines (the CIM Standards). Assumptions, metal threshold parameters and deposit modeling methodologies associated with the current Pulacayo resource estimate are discussed.

14.2 Geological Interpretation Used In Resource Estimation

The Pulacayo deposit is considered a low sulphidation epithermal deposit hosting both precious and base metals, of which silver, lead and zinc are of economic significance. Mineralization included in the current resource is associated with the TVS that were emplaced on the southern side of the Pulacayo dome complex that contains Tertiary sediments of the Quenhua Formation and Miocene intrusive andesitic volcanic rocks of the Rothchild and Megacristal units. The dome complex is tens of km in length and commonly hosts polymetallic sulphide mineralization within east-west trending fault systems. The TVS is the most prominent mineralized structure on the property and portions of the trend have been mined for several hundred years. The structure strikes east-west and has near vertical dip in most areas.

TVS mineralization is hosted by both volcanic and sedimentary host rocks, with stock works of narrow veins and veinlets plus disseminations that aggregate up to 120 m in width being typical of volcanic hosted sections. Sections hosted by sedimentary rocks show much narrower high grade vein structures that typically measure a few meters or less in width. These generally bifurcate transitionally upward into the stock work style systems and zones of dissemination seen in the overlying volcanic rocks.

Mineralization is known to extend along the TVS strike for 2,700 m and to a depth of almost 1,000 m below surface. The latter reflects the depth of historic mining, of which approximately 450 m occur within overlying volcanic host rocks and 550 m occur within the sedimentary sequence. The portion of the TVS defined by Apogee drilling and considered in this resource estimate extends along strike for approximately 1,500 m and to an average depth of 450 m below surface. As a result, most of the resource area is hosted by andesitic volcanic lithologies.

Contributing minerals of economic significance include galena, sphalerite, tetrahedrite and other silver sulfosalts, along with minor occurrences of chalcopyrite and jamesonite. These minerals are accompanied by barite, quartz, pyrite, and calcite. Local occurrences of Au have also been noted but not extensively assessed to date. Veins generally have a banded texture, with segments containing semi-massive to massive sulphides. The TVS shows vertical metal zonation with increasing base metal concentrations with depth and higher Ag content at middle elevations (350 m to 450 m depth).
14.3 Methodology of Resource Estimation

14.3.1 Overview of Estimation Procedure

This updated Pulacayo deposit mineral resource estimate is based on a three dimensional block model developed using GemcomSurpac ® Version 6.3.1 modeling software. The estimate is based on validated results of 69,739 m of diamond drilling from 226 surface drill holes and 42 underground drill holes completed by Apogee and ASC through various drill programs between 2002 and the end of 2011. The final drill hole completed and included in the resource is 11PUD266. A total of 28,807 drill core samples have been assayed from these programs with 24367 samples occurring within the limits of the current resource model. In addition, 627 trench samples completed in 2011 were included in the resource estimate.

Prior to deposit modeling, a complete set of vertical cross sections through the deposit were produced from the project database and used to develop north-south geological section interpretations at a nominal section spacing of 50 m. Raw analytical results were represented on the interpreted sections to understand TVS geometry and grade distribution trends, along with corresponding calculated Net Smelter Return (NSR) values determined using a calculator developed for Apogee by Starkey and Associates Inc., Consulting Metallurgical Engineers. The NSR calculator is discussed in more detail in report section 14.3.3 below.

To better represent the individual grade trends and distribution of silver, lead and zinc, each metal was modeled independently for the current resource estimate. On this basis, nine domain solid models defining zones of veining and higher grade mineralization were created for silver, lead and zinc from the sectional interpretation of drill core assays and geological data. The resulting 27 metal domain solid models range from a few meters to tens of meters in thickness and have varying continuity over a 1,500 m strike length oriented at Az. 280°, and a 600 m sub-vertical total dip extent. This reflects the orientation and geometry of the principal mineralized TVS structure and associated secondary structures. A peripheral constraining wireframe was developed to include all the metal domains and digital terrain models were created for both the oxide-sulphide zone boundary and the topographic surface.

Inverse distance squared (ID²) grade interpolation was first constrained within the interpreted metal domain wireframes for silver, lead and zinc using multiple independent search ellipsoid passes and independent 1 m down hole assay composites. Search ellipsoid orientations for the interpolation passes conformed to the general trend of Az. 280° with a 30° major axis plunge and sub-vertical dip, but were modified to accommodate local variations in the distribution of mineralization. Contributing 1 m assay composites were capped at 1,500 g/t silver, 15% lead, and 15% zinc. ID² grade interpolations for each metal were subsequently carried out and constrained within the peripheral wireframe envelope, outside the interpreted metal domain wireframes. A 150 m ellipsoid major axis, 75 m ellipsoid semi-major axis and 25 m ellipsoid minor axis were applied for all interpolation passes with the exception of a restricted range pass using 30 m major and semi-major axes and a 5 m minor axis for all values greater than 50 g/t silver, 5% lead, or 5% zinc within the peripheral domain and external to the main metal wireframes. Contributing 1 m down hole composites were constrained to a minimum of two and a maximum of nine with no more than three from a single drill hole. The oxide-sulphide zone digital terrain model functioned as a hard boundary in all interpolation passes.
A specific gravity model was interpolated by ID$^2$ methodology from 18236 sulphide zone and 3131 oxide zone specific gravity values normalized to 1 m down hole composites. Interpolation procedures were the same as those used for metal grade assignment and the highest interpolated specific gravity value calculated for each block was selected as the valid block value.

The Pulacayo deposit has an extensive history of mining activity represented by shafts, winzes, level development, and stoping. Digital modeling of stopes and underground development by EPCM Consultants S.R.L (EPCM) for Apogee was upgraded and validated by TWP by digitally adding historic underground mining and stoping from archived mine engineering plans. The revised digital model of underground development and stopes shows good visual correlation with the drill hole database.

Indicated Resources are all interpolated blocks with at least 7 contributing assay composites with a maximum average distance of 70 m from the block centroid, and having the nearest contributing composite at 60 m or less. Inferred Resources are all other interpolated blocks within the peripheral constraint or blocks that otherwise meet Indicated Resources parameter but are interpolated through drill hole intersections that lack sampling.

Net Smelter Return (NSR) values for the blocks occurring within the sulphide zone were determined by means of a net smelter return calculator developed for the deposit by John Starkey, P. Eng., of Starkey and Associates Inc., Consulting Metallurgical Engineers of Oakville Ontario. The modeling used a 36 month trailing average silver price of 25.00 USD/oz. The base metal prices used in the modeling were 0.86 USD/lb lead and 1.00 USD/lb zinc. The base metal prices were supplied by Exen Consulting Ltd. of Oakville, Ontario and are based upon the average long-term analyst lead and zinc price projections from a number of metal broker and mining investment banking sources. Exen Consulting Ltd. is a commercial concentrate marketing business providing consulting services to producers, metal traders and smelters around the world.

Open pit resources to an elevation of 4,159 m ASL (top of crown pillar) were determined within a Whittle optimized maximum NPV pit shell utilizing 1.80 USD/ton mining cost, 1.60 USD/ton surface haulage cost, 2.50 USD /ton G&A, and 19.0 USD/ton and 9.10 USD/ton respectively for oxide and sulphide processing costs. Pit slopes varied from 42 to 43 degrees. In the pit optimization process, only silver derived NSR values were used in the oxide zone, while silver, lead and zinc derived NSR values were used in the sulphide zone. P&E Mining Consultants Inc. of Brampton, Ontario completed the Whittle open pit optimization program and provided statements of associated oxide and sulphide zone mineral resources.

Resource block model results were validated through comparison with separate deposit models based on grade interpolation using Ordinary Kriging (OK) and Nearest Neighbor (NN) methodologies. Statistical results of the ID$^2$ interpolation were determined as most appropriately representing contributing assay populations and associated sectional deposit interpretations.
14.3.2 Data Validation

Results from 268 surface and underground drill holes completed by Apogee and ASC between 2002 and 2011, totaling 69,739 m of diamond drilling and 28807 core samples, were received by Mercator in digital spreadsheet format from Apogee and were subsequently compiled and imported into GemcomSurpac ® Version 6.3.1 software. The final drill hole accepted for the current resource estimate from the 2011 drill programs by Apogee was 11PUD266.

Validation checks on overlapping intervals, inconsistent drill hole identifiers, improper lithological assignment, unreasonable assay value assignment, and missing interval data were performed. Checking of database analytical entries was also carried out against laboratory records supplied by Apogee. Of the 28,807 assay results in the drilling data set for the property, 24,366 occur within the current resource outline and associated peripheral interpolation domain.

14.3.3 Metal Pricing and Net Smelter Return (NSR) Calculation

The previous resource estimate by Mercator completed in October, 2011 used a NSR factor and NSR shell wireframe to evaluate the polymetallic nature of mineralization at Pulacayo. This was consistent with methods used in preceding historical resources estimates for the deposit prepared by Micon. For the purpose of the current resource, silver, lead and zinc grades were interpreted and modeled independently to better assess individual grade distribution and continuity and to improve overall spatial confidence in grade interpolation. A NSR factor was subsequently calculated from interpolated block grades within the sulphide zone for reporting purposes.

NSR incorporates consideration of metallurgical, milling, and mining inputs. More specifically, NSR is the calculated potential revenue that is returned from the smelter for the sale of concentrate products. The NSR method recognizes that more than one metal, in this circumstance Ag, Pb and Zn, can contribute to a potential revenue stream. It derives a potential revenue value that accounts for such items as recovery to concentrate, metal prices, payable fractions of the metals treatment, and refining charges, penalties, freight and handling. By this means, in situ metal grades can be converted to potential revenues and a cut-off grade can be identified as the estimated cost of all activities related to mining, mineral processing and general administration.

John Starkey, P. Eng., of Starkey and Associates Inc., Consulting Metallurgical Engineers, was retained by Apogee to develop a digital spreadsheet-based NSR calculator for the current Pulacayo resource estimate. This calculator was reviewed by Mercator and accepted for use in the resource estimation program. The following description of the NSR calculator is a direct excerpt from the Starkey (2010) report to Apogee.

“The NSR calculation method used for Apogee Minerals’ Pulacayo Project was done using basic principles and standard concepts taken from typical smelter contracts in the silver lead zinc industry, to calculate the value of one ton of lead or zinc concentrate, shipped to either a lead or a zinc smelter respectively.

To do this in a meaningful way, a metallurgical balance, based on the grade of ore in the mine and metallurgical research to determine mill recoveries, was first done in order to
calculate the ratios of concentration for the lead and zinc concentrates respectively. In this way, the value for each concentrate could be related to a value per ton of ore, based on the grade of the ore being evaluated and the current price of the metals deemed to be appropriate for this evaluation.

Standard deductions were made from the metal content in each concentrate to represent what payment a smelter would calculate in a concentrate purchase transaction. Penalties for deleterious elements were not deducted because it was not possible to predict the exact amount of arsenic or antimony in any given ore block, and these are considered to have minimal impact on the values so calculated. However, care should be taken in the final ore reserve assessment to ensure that pockets of deleterious metals will not interfere with the profitability of the operation.”

NSR values for model blocks occurring within the sulphide zone were determined by means of the Starkey net smelter calculator. The modeling used a 36 month trailing average silver price of 25.00 USD/oz. The base metal prices used in the modeling were 0.89 USD/lb lead and 1.00 USD/lb zinc. The base metal prices were supplied by Exen Consulting Ltd. of Oakville, Ontario and are based upon the average long-term analyst lead and zinc price projections from a number of metal broker and mining investment banking sources. Exen Consulting Ltd. is a commercial concentrate marketing business providing consulting services to producers, metal traders and smelters around the world. Missing or null assay values prevented calculation of a NSR value and in such circumstances values of silver, lead or zinc were replaced with values reflecting one half of their respective analytical detection limits. Details of smelter contract terms used in the calculator are confidential and cannot be publicly disclosed.

14.3.4 Data Domains and Solid Modeling

14.3.4.1 Topographic Surface

Apogee carried out a detailed topographic survey in 2008 that generated a high quality topographic map for an area that measures approximately 2,600 m east-west and 1,600 m north south over the Pulacayo deposit. The survey used total station survey methods and a series of reflecting prisms to generate a two meter elevation data set and to pick up additional important features such as roads and shafts. The topographic map is represented as a Gemcom Surpac DTM model and is applied as the topographic constraint for resource modeling (Figure 14.1).
14.3.4.2 Oxide Surface

The Pulacayo deposit is capped by a layer of oxide material where the original volcanic host rocks and sulphide mineralization has been altered by deep weathering effects. Economic mineralization of the TVS is observed to continue through the oxide-sulphide transition and was the focus of 45 drill holes from the 2011 Apogee drill program.

Sectional interpretations from drill hole data on nominal 50 m spaced sections were used to develop an oxide-sulphide surface DTM model in Surpac and subsequently used to code oxide blocks within the block model (Figure 14.2). The oxide zone ranges from less than 5 m to 50 m or more in thickness across the Pulacayo deposit area, but averages 20 m to 30 m thick above the mineralized zone evaluated in the current resource.
Sectional interpretation showed that although silver and lead grade continues across the oxide-sulphide boundary, the metals show depletion in the oxide zone. Zinc assay core results are low to anomalous within the oxide zone and are interpreted to be almost completely depleted. On this basis, the oxide-sulphide surface DTM model was used as a hard boundary in grade interpolation, with assay composites unable to contribute to the interpolation of blocks on opposite sides of the surface.

14.3.4.3 Domain Modeling

The spatial distribution of volcanic host rocks and sulphide-silver mineralization contributes directly to variability in grade distribution within the Pulacayo deposit as defined by Apogee drilling. Stringer and disseminated style mineralization with locally massive to semi-massive zones are typical of the TVS within the intruding andesitic volcanics. Despite observed deposit scale zonation of all three metals throughout the history of mining, with higher grade silver occurring at middle elevations and increasing base metal values with depth, the TVS can locally be defined by any one metal or various combinations of the three.

The previous resource estimate by Mercator used an NSR-based spatial domain model to define that portion of the TVS having the greatest economic potential. Although this method accommodates spatial grade variability within the deposit and adequately constrains interpolation, a smearing effect of each metal can potentially occur by diluting higher-grade vein based mineralization into lower-grade adjacent tons from the lack of hard individual grade boundaries for high grade structures. Detailed sectional geological modeling subsequent to the previous resource allowed development of a more precise correlation of veining and associated mineralization, resulting in a transition from the NSR-domain methodology to the individual metal domain methodology used in the current resource.

Assay results for silver, lead and zinc were displayed on drill hole traces at nominal 50 m section spacing and sectional interpretations for each metal based on these values were used to develop metal-specific, independent, three dimensional wireframe solid models. The
solid model domains were limited up-dip by the topographic surface or half the distance to a constraining drill hole and were limited by either a down-dip extension of 25 m from the last intercept, half the distance to a constraining drill hole, or to a depth that was continuous with the depth of the domain defined on adjacent sections. Along strike, the solid models were projected 25 m from the last defined section that showed continuity and definition based on the current drill hole database, or half the distance to a constraining drill hole. For silver, lead, and zinc, nine domain solid models defining zones of veining and higher grade mineralization were created. The 27 resulting metal domain solid models range from a few meters to tens of meters in thickness and have varying continuity over a 1,500 m strike length oriented along Az. 280°. All occur within a 600 m sub-vertical dip extent (Figure 14.3, Figure 14.4 and Figure 14.5).

A peripheral constraining wireframe enveloping all the metal domains was developed for the purpose of interpolating grade for each metal outside the respective metal domain solid models. Limits of the peripheral constraint were defined by the limits of the domain solid models and presence/absence of sampling results.

Figure 14.3: Longitudinal and Plan View of the Silver Domain Solid Models
14.3.4.4 Underground Workings Model

The underground workings solid model used for the current resource estimate is based on a solid model completed by EPCM Consultants S.R.L. (EPCM) that was validated and upgraded by TWP. The EPCM workings model was developed from digitally scanned archived plans and section and focused on the area of the old USD 40 NSR domain solid used in previous resource estimates (Figure 14.6). TWP subsequently expanded this model into other areas of the deposit by digitally adding data acquired from archived mine engineering plans of historic underground mining and stoping (Figure 14.7 and Figure 14.8). Mercator validated the upgraded workings model against the drill hole database and found acceptable correlation between the two.
Figure 14.6: Isometric View of the Previous SURPACUSD40NSR Domain Solid Model – View to NW (This Model is No Longer Current and is Included for Comparative Purposes Only)

Figure 14.7: Longitudinal View of the Current Workings Solid Model (View to N)
Compatibility issues arose between the workings model and common commercial mining modeling software such as the Gemcom-SurpacRsuite, since a significant portion of the tunnels and stopes existed in an incompatible solid model format and other tunnels existed only as 2D CAD files. To deal with this issue, the perimeter of the workings existing in solid model format was converted to level plan string files at 1 m increments and these were subsequently populated with a point every meter along each string file. A vertical ellipsoid oriented along the strike of the deposit with a 2.5 m major axis and 1.25 m semi-major and minor axes was then passed through the block model to identify intersecting blocks that define the workings. Additionally, the 2D CAD format tunnels were populated with a point every meter along each file and a vertically dipping ellipsoid oriented along the strike of the deposit, with a 2.5 m major axis, 1.67 semi-major axis and 1.50 m minor axis, was then passed through the block model to identify intersecting blocks that define the workings.

Interpolated blocks intersecting level plan string files of the upgraded workings model were considered to have been previously mined and were assigned null values for all attributes after block model grade interpolation procedures were completed (Figure 14.9).
14.3.4.5 Drill Core Assay Composites and Statistics

The drill core assay data set used in the updated resource estimate contains 28,807 core sample records exclusive of quality control and quality assurance samples, including 24,367 core samples occurring within the peripheral resource domain solid. Sample lengths range between 0.3 m and 6.0 m for the peripheral domain core sample subpopulation, with over 90% of samples measuring 1.0 m in length. Frequency histograms and cumulative frequency distribution plots for the sample lengths for these were presented by Cullen et al. (2012). Based on these results, down hole assay composites at 1.0 m support were developed for silver, lead and zinc for drill hole intervals intersecting the respective metal domain solid models. In addition, 627 trench samples completed in 2011 were included in the resource estimate. Trench samples were converted into horizontal drill holes and included in the down hole compositing process. No lithological constraints were imposed on down hole assay compositing.

Descriptive statistics were calculated for each metal from 1.0 m assay composite datasets constrained within the respective metal domains and results are presented in Table 14.1. Distribution histograms, cumulative frequency plots and probability plots for the 1.0 m composites are included in the previously disclosed resource estimate technical report by Cullen et al. (2012).
### Table 14.1: Silver, Lead and Zinc Statistics for Uncapped 1 m Composites

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Ag Metal Domains</th>
<th>Pb Metal Domains</th>
<th>Zn Metal Domains</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean Grade</td>
<td>122.10 (g/t)</td>
<td>1.04 (%)</td>
<td>1.54 (%)</td>
</tr>
<tr>
<td>Maximum Grade</td>
<td>8,279.2 (g/t)</td>
<td>26.7 (%)</td>
<td>28.5 (%)</td>
</tr>
<tr>
<td>Minimum Grade</td>
<td>0.14 (g/t)</td>
<td>0.01 (%)</td>
<td>0.01 (%)</td>
</tr>
<tr>
<td>Variance</td>
<td>90,402.59</td>
<td>2.53</td>
<td>3.49</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>300.67</td>
<td>1.59</td>
<td>1.86</td>
</tr>
<tr>
<td>Coefficient of Variation</td>
<td>2.46</td>
<td>1.52</td>
<td>1.21</td>
</tr>
<tr>
<td>Number of Composites</td>
<td>8,068</td>
<td>6,952</td>
<td>11,398</td>
</tr>
</tbody>
</table>

14.3.4.6 High Grade Capping Of Assay Composite Values

A grade cap for each metal was applied to 1 m assay composites to limit influence of high grade anomalous results having limited demonstrated continuity. Composites were capped at levels of 1,500 g/t for silver and 15% for both lead and zinc that correspond with the 99.3, 99.9, and 99.8 percentiles for each metal respectively. A subjective check on applicability of capping factors was carried out on the basis of logged geology and mineralization styles and it was concluded that the presence of 1 m intervals of mineralization at the selected capping grades were geologically reasonable, with local potential for both strike and dip continuity at such levels. Descriptive statistics were calculated for the 1 m capped assay composites and are presented in Table 14.2. As expected, mean composite grades for each metal decrease relative to the mean grades of the raw composite values presented above. The Coefficient of Variation for capped datasets is also lower, indicating that they are more statistically acceptable for resource estimation purposes. Distribution histograms, cumulative frequency plots and probability plots for capped 1 m composites for all metals are included in the previously disclosed resource estimate technical report by Cullen et al. (2012).

### Table 14.2: Silver, Lead and Zinc Statistics for Capped 1.0 m Composites

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Ag Metal Domains</th>
<th>Pb Metal Domains</th>
<th>Zn Metal Domains</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean Grade</td>
<td>114.87 (g/t)</td>
<td>1.04 (%)</td>
<td>1.53 (%)</td>
</tr>
<tr>
<td>Maximum Grade</td>
<td>1,500 (g/t)</td>
<td>15.00 (%)</td>
<td>15 (%)</td>
</tr>
<tr>
<td>Minimum Grade</td>
<td>0.14 (g/t)</td>
<td>0.01 (%)</td>
<td>0.01 (%)</td>
</tr>
<tr>
<td>Variance</td>
<td>47,015.29</td>
<td>2.41</td>
<td>3.12</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>216.83</td>
<td>1.55</td>
<td>1.77</td>
</tr>
<tr>
<td>Coefficient of Variation</td>
<td>1.89</td>
<td>1.49</td>
<td>1.15</td>
</tr>
<tr>
<td>Number of Composites</td>
<td>8,068</td>
<td>6,952</td>
<td>11,398</td>
</tr>
</tbody>
</table>
14.3.4.7 Variography

Manually derived models of geology and grade distribution provided definition of the primary east-west sub-vertical trend associated with the TVS. Mineralization is characterized by narrow vein style occurrences in tuffaceous sandstone host rocks at depth that bifurcate into stock work vein arrays and disseminated mineralization in overlying andesitic volcanics. To assess spatial aspects of grade distribution within this recognized orientation corridor, experimental variograms for silver, lead and zinc were individually created based on the capped 1 m down hole composite data set from the respective metal domain.

Experimental down hole variograms were developed to assess the global nugget value for each metal (Figure 14.10) with good spherical model results obtained. Experimental directional variograms were developed for several Pulacayo data sets, including individual metal domains and the global deposit defined by limits of the peripheral domain solid. The experimental directional variogram results presented below were developed from a global dataset consisting of all 1 m grade-capped assay composites occurring within the peripheral domain, including all composites within contained individual metal domain solids. Data were assessed within a vertical, east-west oriented plane defined in the modeling software by a 90° plunge towards 0° azimuth.

Experimental variogram results for silver were fitted with spherical model attributes that define a primary axis (major axis) of continuity of 150 m along an azimuth of 270° and a plunge of 40°, a secondary axis (semi-major) of continuity of 75 m along an azimuth of 90° and a plunge of 40°, and a third axis (minor axis) of continuity of 25 m along an azimuth of 0° and a plunge of 0° (Figure 14.11). The anisotropic variogram results for silver combined with sectional interpretations of geology and grade distributions contributed to development of interpolation ellipsoids that can generally be defined as plunging 30° along a strike of 280° and dipping 90°, with a 150 m major axis, 75 m semi-major axis, and a 25 m minor axis. Interpolation ellipsoid orientation was modified along dip and strike to better fit local trends and geometries of the deposit.

Experimental variogram results for lead were fitted with spherical model attributes that define a primary axis of continuity of 150 m along an azimuth of 280° and a plunge of 35°, a secondary axis of continuity of 100 m along an azimuth of 90° and a plunge of 55°, and a third axis of continuity of 25 m along an azimuth of 0° and a plunge of 0° (Figure 14.12). The anisotropic variogram results for lead, combined with sectional interpretations of geology and grade distributions contributed to the development of interpolation ellipsoids that can generally be defined as plunging 30° along a strike of 280° and dipping 90°, with a 150 m major axis, 75 m semi-major axis, and a 25 m minor axis. As in the case for silver noted above, interpolation ellipsoid orientation was modified along dip and strike to better fit local trends and geometries of the deposit.

Experimental variogram results for zinc were fitted with spherical model attributes that define a primary axis of continuity of 150 m along an azimuth of 280° and a plunge of 35°, a secondary axis of continuity of 100 m along an azimuth of 90° and a plunge of 56°, and a third axis of continuity of 25 m along an azimuth of 0° and a plunge of 0° (Figure 14.13). The anisotropic variogram results for zinc, combined with sectional interpretations of geology and grade distributions contributed to the development of interpolation ellipsoids that can generally be defined as plunging 30° along a strike of 280° and dipping 90°, with a 150 m
major axis, 75 m semi-major axis, and a 25 m minor axis. As in the cases for silver and lead noted above, interpolation ellipsoid orientation was modified along dip and strike to better fit local trends and geometries of the deposit.

Figure 14.10: Downhole variograms of Silver, Lead and Zinc
Figure 14.11: Anisotropic Variograms for Silver Capped Composite Values
Figure 14.12: Anisotropic Variograms for Lead Capped Composite Values
14.3.4.8 Setup of Three Dimensional Block Model

The Pulacayo block model was developed using WGS84 (Zone 19, South Datum) grid coordination and a sea level elevation datum. The block model is oriented along Az. 100° with no dip rotation applied and model extents are further defined in Table 14.3. Standard
block size for the model is 5 m x 3 m x 3 m (X, Y, Z); with one unit of sub-blocking to a minimum block size of 2.5 m x 1.5 m x 1.5 m allowed. As discussed above in Section 14.3.3, the nominal topographic surface as defined by a digital terrain model functions as the upper deposit constraint.

14.3.4.9 Resource Estimation

Inverse distance squared (ID²) grade interpolation was used to assign block grades within the Pulacayo block model. As reviewed earlier, interpolation ellipsoid orientation and range values used in the estimation reflect a combination of trends determined from the variography and sectional interpretations of geology and grade distributions for the deposit. The trends and ranges of the major, semi-major, and minor axes of grade interpolation ellipsoids used to estimate silver, lead and zinc grades were described previously in report section 14.3.7.

All three metals were evaluated independently using respective assay composite data sets. Block model grade interpolation was a multiphase process for each metal that consisted of sequential interpolation within (1) the interpreted metal domain solids below the oxide surface DTM, (2) the interpreted metal domain solids above the oxide surface DTM, (3) below the topographic surface DTM within the peripheral domain solid and below the oxide surface DTM, and (4) within the peripheral domain solid above the oxide surface DTM and below the topographic surface DTM. Individual metal domain solids, nine for each metal for a total of 27, required multiple interpolation passes to accommodate minor orientation adjustments to the azimuth and dip of ellipsoids. This was done to better accommodate local variations in orientation and geometry of respective domains. The minimum number of contributing assay composites used to estimate a block grade for all metal domains was set at 2 and the maximum number of contributing composites was set at 9, with no more than 3 contributing assay composites allowed from a single drill hole. Block discretization was set a 1Y x 1X x 1Z. Model blocks identified as occurring within the underground workings solid model were removed from the resource estimate after grade interpolation was completed. Interpolated silver, lead, and zinc block grades were used to calculate NSR block values within the sulphide zone by means of the net smelter calculator discussed in section 14.3.3. This calculator considers metallurgy, economics, and costs relevant to sulphide zone resources.
and underground mining methods and is not applicable to oxide zone resources because recovery of lead and zinc is not anticipated from the oxide zone.

**Bulk Density**

Density determinations were performed systematically by Apogee staff using the Archimedes method on selected core samples. Results were compiled with corresponding lithologies in a digital spreadsheet and a total of 29344 analyses were available for use in this updated resource estimate. Mercator imported these results into the Surpac resource database and normalized data by developing 1 m down hole composites for drill hole intervals occurring within the metal domain and peripheral domain solids. Descriptive statistics for the 21,331 specific gravity composites that occur within the resource estimate peripheral constraint solid are presented in Table 14.4 (sulphide zone) and Table 14.5 (oxide zone). Corresponding distribution histograms and cumulative frequency plots for the volumetrically dominant sulphide zone are presented in Figure 14.14.

**Table 14.4: Density Statistics for 1 m Composites in Peripheral Domain Sulphide Zone**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Specific Gravity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>2.35 g/cm³</td>
</tr>
<tr>
<td>Maximum</td>
<td>6.20 g/cm³</td>
</tr>
<tr>
<td>Minimum</td>
<td>1.02 g/cm³</td>
</tr>
<tr>
<td>Variance</td>
<td>0.10</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>0.32</td>
</tr>
<tr>
<td>Coefficient of Variation</td>
<td>0.13</td>
</tr>
<tr>
<td>Number of Composites</td>
<td>18,200</td>
</tr>
</tbody>
</table>

**Table 14.5: Density Statistics For 1 m Composites in Peripheral Domain Oxide Zone**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Specific Gravity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>2.16 g/cm³</td>
</tr>
<tr>
<td>Maximum</td>
<td>5.54 g/cm³</td>
</tr>
<tr>
<td>Minimum</td>
<td>1.00 g/cm³</td>
</tr>
<tr>
<td>Variance</td>
<td>0.07</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>0.27</td>
</tr>
<tr>
<td>Coefficient of Variation</td>
<td>0.13</td>
</tr>
<tr>
<td>Number of Composites</td>
<td>3,131</td>
</tr>
</tbody>
</table>

Mercator considers variation in model space bulk densities to be primarily dependent on variation of contained sulphide levels in core samples. Sulphide content is also closely related to overall metal grades within the resource peripheral constraint. Assignment of the average specific gravities of 2.35 g/cm³ and 2.16 g/cm³ for the sulphide and oxide zone, respectively, across the entire deposit was not considered appropriate, due to these
recognized spatial density inhomogeneities. Density values were therefore interpolated into block model space using ID² methodology and constant parameters for each metal. This produced three independently interpolated density values for each block derived from separate interpolations based on the three metal sets and interpolation routines. The highest interpolated density value for each block was selected as the database block density attribute.

![Image]

Figure 14.14: Cumulative Frequency and Distribution Histogram of Density Results in Peripheral Domain Sulphide Zone

Extremely low density values in the database were flagged and plotted on the drill hole traces along with geology and drill hole assay grades to assess their validity. Values less than 1.50 g/cm³ were found to most commonly occur in samples from drill hole PUD005. Core from this hole is characterized by competent lithologies that would be expected to have density values more closely approximating the data set mean. This issue was further assessed by comparing PUD005 density values for core intervals for similar metal grades in similar rocks in adjacent drill holes. These intervals showed values close to or greater than the mean data set value and it was concluded that results below 1.50 g/cm³ were erroneous where no logged physical core attributes could explain the low value present. These suspect results were excluded from the density interpolation process for the resource block model.

14.3.4.10 Resource Category Definitions

Definitions of mineral resource and associated mineral resource categories used in this report are those recognized under National Instrument 43-101 (NI 43-101) and set out in the Canadian Institute of Mining, Metallurgy and Petroleum Standards On Mineral Resources and Reserves Definitions and Guidelines (the CIM Standards).
14.3.4.11 Resource Category Parameters Used in Current Estimate

Mineral resources presented in the current updated estimate have been assigned to Inferred and Indicated resource categories that reflect increasing levels of confidence with respect to spatial configuration of resources and corresponding grade assignment within the deposit. Several factors were considered in defining resource category assignments, including drill hole spacing, geological interpretations, and number and range of informing composites. Apogee has significantly improved the quality of the underground workings model since the previous resource estimate by Mercator that incorporated an Inferred category buffer zone around the digital workings model. A similar Inferred category envelope was not included in the current estimate.

Additional specific definition parameters for each resource category applied in the current estimate are set out below and Figure 14.15 through Figure 14.18 illustrate spatial distribution of these categories and the underground workings within the block model.

**Measured Resource:** There are no interpolated resource blocks with the certainty of definition suitable for classification in this category present in the current estimate.

**Indicated Resources:** All interpolated blocks that occur within the peripheral deposit constraint with one or more of the interpolated metals qualifying to have at least 7 contributing assay composites with a maximum average distance of 70 m to a block centroid and the nearest contributing composite at 60 m or less from a block centroid are categorized as Indicated mineral resources.

**Inferred Resources:** All interpolated blocks that occur within the peripheral deposit constraint that do not meet Indicated resource category requirements, or for which grades were interpolated through drill intervals without sampling are categorized as Inferred mineral resources.

![Figure 14.15: Isometric View of the Mineral Resource Categories – View to NW](image-url)
14.3.4.12 Pit Optimization and Cut-off Values

Oxide and sulphide zone resources potentially amenable to economic development by means of an open pit were modeled by P & E Mining Consultants Inc. (P&E) by applying Net Smelter Return (NSR)/ton cut-off values to the block model and reporting the resulting tonnage and grade of silver, lead and zinc for potentially mineable areas within a Whittle optimized pit shell to El 4159 ASL. Resources potentially amenable to economic development by means of underground extraction were estimated below EL4159ASL using an NSR cut-off per ton value as described below.

The NSR cut-off grade calculations for resource reporting of open pit and underground potentially economic portions of the mineralization were founded on operating costs provided by Apogee. P&E has reviewed these costs and consider them to be reasonable given P&E’s recent resource/reserve estimating experience in Peru, Guyana, Ecuador and Argentina.
Oxide NSR Value Calculation Parameters

- Ag Price: 25 USD/oz
- Ag Process Recovery: 80%
- Ag Smelter Payable: 90%
- Royalty: 13%

Sulphide NSR Value Calculation Parameters

- Ag Price: 25 USD/oz
- Pb: 0.89 USD/lb
- Zn: 1.00 USD/lb
- Ag Process Recovery: 92%
- Pb Process Recovery: 79%
- Zn Process Recovery: 79%
- Ag Smelter Payable: 100%
- Pb Smelter Payable: 100%
- Zn Smelter Payable: 100%
- Royalty: Nil

Oxide NSR Open Pit Cut-Off Calculation USD

<table>
<thead>
<tr>
<th>Cost Item</th>
<th>Cost (USD/ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process Cost (10,000 tpd)</td>
<td>19.00</td>
</tr>
<tr>
<td>General and Administration</td>
<td>2.50</td>
</tr>
<tr>
<td>Extra Out of Pit Haulage</td>
<td>1.60</td>
</tr>
<tr>
<td>Operating costs per ore ton</td>
<td>23.10 USD/ton</td>
</tr>
</tbody>
</table>

Sulphide SR Open Pit Cut-Off Calculation USD

<table>
<thead>
<tr>
<th>Cost Item</th>
<th>Cost (USD/ton processed)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process Cost (10,000 tpd)</td>
<td>9.10</td>
</tr>
<tr>
<td>General &amp; Administration</td>
<td>2.50</td>
</tr>
<tr>
<td>Extra Out of Pit Haulage</td>
<td>1.60</td>
</tr>
<tr>
<td>Operating costs per ore ton processed</td>
<td>13.20 USD/ton processed</td>
</tr>
</tbody>
</table>

Sulphide NSR Underground Cut-Off Calculation USD

<table>
<thead>
<tr>
<th>Cost Item</th>
<th>Cost (USD/ton processed)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process Cost (1500 tpd)</td>
<td>12.00</td>
</tr>
<tr>
<td>General and Administration</td>
<td>2.50</td>
</tr>
<tr>
<td>Extra out of Mine Haulage</td>
<td>1.00</td>
</tr>
<tr>
<td>Underground Mining</td>
<td>42.50</td>
</tr>
<tr>
<td>Operating costs per ore ton processed</td>
<td>58.00 USD/ton processed</td>
</tr>
</tbody>
</table>

Pit Optimization Parameters

The Pulacayo resource model was further investigated with a Whittle pit optimizations to ensure a reasonable stripping ratio was applied and a reasonable assumption of potential economic extraction could be made (See Figure 14.19 below). The following parameters were utilized in the pit optimizations:
In addition to the open pit resource estimate, mineralization below the EL 4,159 m level (top of crown pillar) was quantified at the 58 USD/ton NSR cut-off value described above to define underground sulphide resources.

**Statement of Mineral Resource Estimate**

Block grade, block density and block volume parameters for the Pulacayo deposit were estimated using methods described in preceding sections of this report. Subsequent application of resource category parameters and pit optimization parameters set out above resulted in the mineral resource estimate statement presented below in Table 14.6. Results are in accordance with Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: *Definitions and Guidelines* (the CIM Standards) as well as disclosure requirements of National Instrument 43-101.

Table 14.7 presents a re-statement of the underground resource calculated on the basis of uncapped drill core assay composites and shows an increase of approximately 22% in contained Inferred category Ag ounces and 10% in contained Indicated category Ag ounces at the USD 58 NSR cut-off value. Open pit resources remained essentially unchanged between capped and uncapped drill core assay composite methodologies. The relationship between deposit silver grade and tonnage at various cutoff values is presented in sensitivity charts for blocks within the sulphide zone (Figure 14.19) and within the oxide zone (Figure 14.20).

<table>
<thead>
<tr>
<th>Resource Category</th>
<th>Cost Parameter</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore and Waste Mining Cost</td>
<td>1.80 USD/ton mined</td>
</tr>
<tr>
<td>Oxide Process Cost</td>
<td>19.00 USD/ton processed</td>
</tr>
<tr>
<td>Sulphide Process Cost</td>
<td>9.10 USD/ton processed</td>
</tr>
<tr>
<td>General/Administration</td>
<td>2.50 USD/ton processed</td>
</tr>
<tr>
<td>Extra Out of Pit Haulage</td>
<td>1.60 USD/ton processed</td>
</tr>
<tr>
<td>Pit Slopes</td>
<td>42 degree average</td>
</tr>
</tbody>
</table>

**Figure 14.18:** Isometric View of Whittle Pit Shell Developed by P&E – View to Nw
Table 14.6: Pulacayo Deposit Mineral Resource – Effective 28 September 2012

<table>
<thead>
<tr>
<th>Resource Class</th>
<th>Type</th>
<th>Tons</th>
<th>Ag</th>
<th>Pb</th>
<th>Zn</th>
<th>Ag</th>
<th>Pb</th>
<th>Zn</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>g/t</td>
<td>%</td>
<td>%</td>
<td>%</td>
<td>g/t</td>
<td>%</td>
<td>%</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Oz</td>
<td>M lbs.</td>
<td>M lbs.</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Open Pit Resources (Base case 42° Average Pit Wall Slope Angle)</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Open Pit Indicated Oxide</td>
<td>1,500,000</td>
<td>95.9</td>
<td>0.96</td>
<td>0.13</td>
<td>4,626,000</td>
<td>NA</td>
<td>NA</td>
<td></td>
</tr>
<tr>
<td>Open Pit Inferred Oxide</td>
<td>248,000</td>
<td>71.20</td>
<td>0.55</td>
<td>0.31</td>
<td>569,000</td>
<td>NA</td>
<td>NA</td>
<td></td>
</tr>
<tr>
<td>Open Pit Indicated Sulphide</td>
<td>9,283,000</td>
<td>44.10</td>
<td>0.66</td>
<td>1.32</td>
<td>13,168,000</td>
<td>135.90</td>
<td>269.54</td>
<td></td>
</tr>
<tr>
<td>Open Pit Inferred Sulphide</td>
<td>2,572,000</td>
<td>33.40</td>
<td>0.92</td>
<td>1.36</td>
<td>2,765,000</td>
<td>51.99</td>
<td>76.88</td>
<td></td>
</tr>
<tr>
<td>Waste Rock (Waste/Ore 5.3 : 1)</td>
<td>71,679,000</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Underground Resources (All blocks below 4,159 m ASL with NSR&gt;USD 58)</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Underground Indicated Sulphide</td>
<td>6,197,000</td>
<td>213.60</td>
<td>0.86</td>
<td>1.74</td>
<td>4,254,000</td>
<td>117.50</td>
<td>237.72</td>
<td></td>
</tr>
<tr>
<td>Underground Inferred Sulphide</td>
<td>943,000</td>
<td>193.10</td>
<td>0.43</td>
<td>1.61</td>
<td>5,853,000</td>
<td>8.94</td>
<td>43.47</td>
<td></td>
</tr>
<tr>
<td>Total Indicated Oxide +Sulphide</td>
<td>16,980,000</td>
<td>110.50</td>
<td>0.74</td>
<td>1.49</td>
<td>60,341,000</td>
<td>253.40</td>
<td>507.26</td>
<td></td>
</tr>
<tr>
<td>Total Inferred Oxide +Sulphide</td>
<td>3,763,000</td>
<td>75.90</td>
<td>0.79</td>
<td>1.43</td>
<td>9,197,000</td>
<td>60.93</td>
<td>120.35</td>
<td></td>
</tr>
</tbody>
</table>

Notes:

1. Tonnages have been rounded to the nearest 1,000 tons. Average grades may not sum due to rounding.
2. Base case 42° average pit wall slope angle for open pit resources.
3. Open pit sulphide resources are reported at a 13.20 USD NSR cut-off. Open pit oxide resources are reported at a 23.10 USD revenue/ton cut-off.
4. Underground resources are reported at a 58USDNSR cut-off below 4,159 m ASL.
5. Metal prices used were 25.00 USD/Oz silver, 0.89USD/lb lead, and 1.00 USD/lb zinc. Lead and zinc do not contribute to revenue in the oxide zone.
6. Contributing 1 m assay composites were capped at 1,500 g/t Ag, 15% Pb, and 15% Zn.
7. Specific gravity is based on an interpolated inverse distance squared model.
8. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
Table 14.7: Pulacayo Uncapped Underground Resource – 28 September 2012

<table>
<thead>
<tr>
<th>Class</th>
<th>Rounded Tons</th>
<th>Ag g/t</th>
<th>Pb %</th>
<th>Zn %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Indicated</td>
<td>6,197,000</td>
<td>236.14</td>
<td>0.87</td>
<td>1.76</td>
</tr>
<tr>
<td>Inferred</td>
<td>943,000</td>
<td>234.95</td>
<td>0.43</td>
<td>1.66</td>
</tr>
</tbody>
</table>

Figure 14.19: Resource Sensitivity - Block Ag G/T Grade and Tonnage

Figure 14.20: Resource Sensitivity - Oxide Zone Block Ag (G/T) Grade and Tonnage
14.3.4.13 Model Validation

Results of block modeling were reviewed in three dimensions and compared on a section by section basis with corresponding manually interpreted sections prepared prior to model development. This showed block model grade patterns within the sulphide zone to generally conform to a near-vertical east-west striking deposit system with a moderate westerly grade plunge (Figure 14.21). This is consistent with the recognized east-west normal fault controlled systems of the Pulacayo dome complex, locally represented as stock work, narrow vein and associated disseminations in the overlying volcanic rocks and high grade vein style mineralization of intruded sedimentary rocks.

![Figure 14.21: Isometric View of the Sulphide Zone Block NSR Value Distribution](image)

Oxide zone silver grade distribution showed discrete zones of mineralization that have good correlation with higher grade mineralization within the underlying sulphide zone (Figure 14.22). Overall, results of the visual inspection of both the sulphide and oxide zones for all
metals show an acceptable degree of consistency between the block model and the independently derived interpretations of the deposit.

Descriptive statistics were calculated for the drill hole composite values used in block model grade interpolations and these were compared to values calculated for the individual blocks in the block model (Table 14.8). The mean drill hole composite grades were found to compare acceptably with corresponding grades of the block model, thereby providing a general check on the model with respect to the underlying assay composite population. Descriptive statistics for Ag grades show a decrease in mean grade between the block model and the contributing sample dataset that is a result of the substantial number of low silver grade blocks occurring along the western limit and along the top 100 m of the model compared to the number of drill hole composites that represent these areas of the deposit.
### Table 14.8: Comparison of Drillhole Composite Grades and Block Model Grades

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Metal Domain Composites</th>
<th>Metal Domain Blocks</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Ag</td>
<td>Pb</td>
</tr>
<tr>
<td>Mean Grade</td>
<td>114.87 g/t</td>
<td>1.04 %</td>
</tr>
<tr>
<td>Minimum Grade</td>
<td>0.14 g/t</td>
<td>0.01 %</td>
</tr>
<tr>
<td>Maximum Grade</td>
<td>1,500 g/t</td>
<td>15 %</td>
</tr>
<tr>
<td>Variance</td>
<td>47015</td>
<td>2.41</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>216.83</td>
<td>1.55</td>
</tr>
<tr>
<td>Coefficient of Variation</td>
<td>1.89</td>
<td>1.49</td>
</tr>
<tr>
<td>Count</td>
<td>8,068</td>
<td>6,952</td>
</tr>
</tbody>
</table>

The ID² resource model for the Pulacayo deposit was checked using Ordinary Kriging (OK) interpolation methodology. Search ellipse and other parameters were the same as those used in the ID² model where applicable. Results of the OK modeling showed that average grades and tonnage closely match those of the preferred ID² model. Results of the two methods are considered sufficiently consistent to provide an acceptable check on the preferred ID² methodology. Nugget and sill values were determined from the variogram analysis discussed in section 14.3.6 and are presented in Table 14.9.

### Table 14.9: Additional Resource Parameters for Ordinary Kriging Check Model

<table>
<thead>
<tr>
<th>Item</th>
<th>Ag</th>
<th>Pb</th>
<th>Zn</th>
</tr>
</thead>
<tbody>
<tr>
<td>Nugget (C0)</td>
<td>0.60</td>
<td>0.65</td>
<td>0.65</td>
</tr>
<tr>
<td>Sill (C1)</td>
<td>0.40</td>
<td>0.58</td>
<td>0.52</td>
</tr>
</tbody>
</table>

14.3.4.14 Comment on Previous Resource Estimates

Mercator reviewed each of the previous resource estimates and has concluded that these provide valuable assessments of the Pulacayo deposit as defined by drilling results available at the time of reporting. The current resource is based on geological and mineralization models of discrete domains for each metal that differ significantly from the combined NSR domain modeling of all contributing metals from previous estimates. While both methodologies provide an evolving assessment of grade distribution of the TVS and associated secondary structures, the current resource methodology aims at improving grade correlation and minimizing potential dilution of each metal with adjacent lower grade tons. Results from the current resource estimate differ from those of the previous resource. Outside of the stated methodology change, this reflects impact of new drilling results, upgraded confidence in the underground workings digital model, inclusion of oxide zone and sulphide zone optimized pit resources and revised (higher) NSR cutoff values for underground resources.
15 MINERAL RESERVES

15.1 Introduction

To convert Mineral Resources to Mineral Reserves, minable stopes were designed within the mineralized zone. These stopes have sufficient widths to be accessed and practically mined with the selected mining method and equipment fleet. A mining recovery and dilution factor has been included. Only Indicated Mineral Resource material has been converted to Reserves. This deposit does not contain Measured Mineral Resource material. Any Inferred material within the designed stopes is treated as waste and allocated a mineralization grade of zero.

15.2 Methodology

Two mining methods will be utilized at Pulacayo, shrinkage stoping and longhole open stoping. The stope widths vary from 3 m to 6 m, depending on the mining method and width of mineralization. Mining losses total 9% for the ore body. This includes material lost to the existing mine workings. Mining dilution is 8%. It is anticipated that 8% of the material mined will be unplanned waste from blasting overbreak and possible contamination from the existing stopes.

The geological block model was provided by Mercator Geological Services, based in Dartmouth Canada. The model was created in Surpac and converted into Datamine format by TWP for mine planning. The mine planning model is based on the original file received from Mercator called ‘pulacayo2012.mdl’. The model is rotated by ten degrees in the clockwise direction and contains 4,098,653 blocks. The parent block size is 5 m in the X direction by 3 m in the Y direction by 3 m in the Z direction. The minimum sub-cell size is half of the parent cell dimensions. Rock density has been interpolated in the block model.

15.3 Stope Generation and Mining Cut-off Grades

Potentially economic portions of the mineralization were founded on operating costs provided by Apogee. TWP Sudamerica S.A has reviewed these costs and consider them to be reasonable given TWP’s recent reserve estimating experience in Peru and across Africa.

Several factors were used to determine if a stope was viable and economic to mine. The first criterion was the average grade for the stope must be above 150 g/t silver. Because the Pulacayo deposit is polymetallic, there is potential for the stope optimisation software (MSO) to draw revenue from stopes that have a low silver grade, but high lead and zinc grades. Using the silver grade cut-off of 150 g/t forces the model to remain silver focused by first seeking to exploit the stopes with higher silver grades.

The second criterion is that the stope must have an NSR greater than US$ 70/t. US$ 70/t is the economic breakeven value of a block. See the calculation of NSR in the table below. Using the NSR instead of an individual grade allowed flexibility in the ratios of silver, lead and zinc, while still keeping silver as the main commodity through the silver cut-off grade.
Sulphide NSR Underground Cut-Off Calculation USD

- Process Cost (1000 tpd) 12.00 USD/ton processed
- General and Administration 2.50 USD/ton processed
- Extra out of Mine Haulage 1.00 USD/ton processed
- Underground Mining 42.50 USD/ton mined
- Selling costs & payables 12 USD/ton processed ($240/tonne concentrate x 5% mass pull)
- Operating costs per ore ton (12.00 USD + 2.50 USD + 1.00 USD + 42.50 USD + 12.00USD) = 70.00 USD/ton processed

The third criterion was based on practical mining widths. The shrinkage stopes widths are between 0.8m and 1.5m. The long-hole open stopes are between 3.0 and 6.0 meters wide.

The forth criterion for stope generation was that the ore within the stopes on each level must pay for the development to reach and exploit that level. Due to there being less Indicated Resource below level 147, the stopes that qualified under the previous three criteria were minimal. These stopes were not able to carry the costs of development below level 147 and were removed from the design.

Recovery curves have been generated to determine the recoverable metal content of the blocks. These curves are based on locked cycle flotation tests (LCFTs) completed by Maelgwyn Minerals Services Africa. Metal prices used were US$28/oz for silver, US$0.86/lb for lead and US$1.00/lb for zinc. Average payables on lead zinc and silver were estimated at US$240/t concentrate.

15.4 Modifying Factors

The selected mining method will require backfilling of stopes and no pillars will be left behind. Based on this, the mining recovery was estimated to be 91%.

- Mining Loss: 2%
- Loss due to lashing: 2%
- Loss due to existing voids: 5%

Mining dilution is estimated to be 8%. It is anticipated that 8% of the material mined will be unplanned waste from blasting overbreak and possible contamination from the existing stopes.

15.5 Mineral Reserves

The Pulacayo Mineral Reserves are estimated in Table 15.1 below.

<table>
<thead>
<tr>
<th>Class</th>
<th>Million Tons (Mt)</th>
<th>AG (G/T)</th>
<th>PB (%)</th>
<th>ZN (%)</th>
<th>AG (MOz)</th>
<th>PB (T)</th>
<th>ZN (T)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Probable</td>
<td>3.558</td>
<td>239.4</td>
<td>1.09</td>
<td>1.91</td>
<td>27.385</td>
<td>38,927</td>
<td>67,905</td>
</tr>
</tbody>
</table>
The mineral reserves were developed by TWP Projects (TWP) with Jim Porter (FSAIMM) acting as the qualified person. Metal price changes or significant changes in costs or recoveries could materially change the estimated mineral reserves in either a positive or negative way. At this time, there are no unique situations relative to environmental or socio-economic conditions that would put the Pulacayo mineral reserves at a higher level of risk than any other developing resource in Bolivia.

15.6 Comparison of Resource to Reserve

Table 15.2 compares the resource to the reserve.

<table>
<thead>
<tr>
<th>Item</th>
<th>Indicated Resource</th>
<th>Probable Reserve</th>
<th>Variation (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tons</td>
<td>5,960,000</td>
<td>3,557,683</td>
<td>-40.3</td>
</tr>
<tr>
<td>Ag Grade</td>
<td>235.94 g/t</td>
<td>239.4 g/t</td>
<td>1.4</td>
</tr>
<tr>
<td>Pb Grade</td>
<td>1.11 %</td>
<td>1.09 %</td>
<td>-1.4</td>
</tr>
<tr>
<td>Zn Grade</td>
<td>1.85 %</td>
<td>1.91 %</td>
<td>3.7</td>
</tr>
<tr>
<td>In-situ Ag</td>
<td>45,210,423 oz</td>
<td>27,385,190 oz</td>
<td>-39.4</td>
</tr>
<tr>
<td>In-situ Pb</td>
<td>66,156 t</td>
<td>38,927 t</td>
<td>-41.1</td>
</tr>
<tr>
<td>In-situ Zn</td>
<td>110,260 t</td>
<td>67,905 t</td>
<td>38.4</td>
</tr>
</tbody>
</table>

Due to the exclusion of mineralised material that is uneconomic to mine, probable reserves are 40.3% less than the indicated resources considered in this study. Consequently, the average grade of silver is slightly higher.

16 MINING

16.1 Mining Method Selection

A trade-off analysis of potential mining methods at Pulacayo mine has been undertaken to determine the most efficient, safe and economic operational arrangement under the existing conditions. The analysis was based on the local geology, geotechnical properties of the rock mass, a cut-off grade of 150 g/t silver (Ag) and a Net Smelter Revenue (NSR) of United States Dollars (USD) 70 per ton mill feed. A variety of conventional, caving and self-supporting methods were considered during the trade-off analysis and the deposit was found amenable to three options; shrinkage stoping, long hole stoping and sub-level caving.

The mine is currently in operation and extracting the upper portions of the deposit along Level Zero using shrinkage stoping. The mine has fully equipped access tunnels, mining equipment and the skills required to accomplish this mining method. Consequently, the
remaining mineral reserves along Level Zero should continue to be extracted using shrinkage stoping.

Long hole stoping, supported by a fleet of trackless equipment, is recommended for extracting the mineral reserves below Level Zero. Owing to the low strength of the rock mass, the stopes will require backfilling to ensure stability post ore extraction. The reasons for selecting this method instead of sub-level caving are listed below:

- The mining method allows for selective mining of high grade veins,
- The mining method will make use of process plant tailings in the backfill, thus minimizing the footprint of the tailings dam on surface and associated environmental liability,
- The use of cemented backfill is required for geotechnical stability. This will allow for maximum ore extraction without compromising on run-of-mine (ROM) grade while mining stopes adjacent to each other,
- Better control of dilution,
- A higher degree of flexibility in the mining sequence.

Mining will be undertaken using a top down approach (i.e. access and infrastructure will be developed to the ore body as required). The main objective of using this approach will be to minimize capital expenditure in the early years of the operation.

16.2 Mine Design and Layout

16.2.1 Shrinkage Stoping

The shrinkage stopes will have a design stoping height of 30 m and a stope length of 30 m. The stope width will depend on the thickness of the payable vein, which varies between 0.8 m and 2 m. Figure 16.1 shows a typical layout of a shrinkage stope at Pulacayo mine.
Ore will be mined in horizontal slices from the bottom to the top of a stoping block. Stopers will be used to drill 41 millimeters (mm) diameter blast holes into the hangingwall of the stope, and Anfex will be used to charge the blast holes. During mining, only 30% of the blasted ore (swell) will be drawn out after each blast to create enough headroom for the drilling of the next slice of ore.

The remaining broken ore will be stored in the stope to support the sidewalls and to serve as a working platform. Once the entire stoping block has been mined out, the remaining ore will be drawn out. The broken ore will be drawn out via boxholes. The boxholes will be located along the ore drive and they will be equipped with pneumatically controlled chutes to allow for safe and easy loading. The ore will be loaded into hoppers and it will be trammed out of the mine and tipped at the ore stockpile at the process plant. The mine will use one 4 tonne locomotive pulling five 3 tonne hoppers to accomplish this. At the end of a stope’s life, it will be sealed off accordingly. No backfill will be placed in the stopes.
16.2.2 Long Hole Stoping

The long hole stopes will have an average length of 10 m, a stope width that varies between 3 m and 6 m depending on the payable vein thickness and an average stoping height of 23.4 m. The level spacing of the old mine has been maintained, and as a result, the spacing varies from 21.5 m to 32.5 m. Figure 16.2 shows a typical layout of a long hole stope.

![Figure 16.2: Top and Bottom Stope Accesses in Long Hole Stoping](image)

The bottom access will be used as a drilling and mucking level. Stoping will be done on retreat using parallel hole, long hole drilling techniques. A trackless drill rig will be used to drill 64 mm diameter blast holes into the hangingwall of the stope and the blast holes will be charged with Anfex.

Remote-controlled LHDs (load-haul-dumper), each with a 3.1 m$^3$ bucket, will be used to tram broken ore from the stopes to the mucking bays. The mucking bays will be located along the haulage drive and they will be spaced at a maximum distance of 120 m. The LHDs will also be used to load rock at the mucking bays into trucks. The trucks will haul the ore out of the mine via the decline ramp system and tip at the stockpile at the process plant. Due to the tonnages involved, a 20 tonne haul truck has been recommended for this operation. The 20 tonne truck will be required to supplement the 15 tonne haul truck that the mine already
owns. Once all the broken ore has been removed from the stope, it will be filled with cemented backfill, as recommended in the Backfill Section of this report.

Historic mining has left voids throughout the planned operation. The voids that have been identified are flagged in the block model and tend to be located along the high grade portions of the ore body. This risk of encountering them will be mitigated by implementing an exploratory drilling program (cover drilling) ahead of production. This will allow for evaluation of ground conditions and the quality of the deposit prior to mining.

The information obtained from this exercise will: verify and identify the location and width of the previously mined areas (voids); determine if the voids are filled with old waste material; allow an economic estimation of the mineralization in each block to ensure that the area is profitable to mine; allow planning for the extraction of each stope block to be made based on ground conditions; allow planning to prevent ore losses by backfilling voids that are located adjacent to stope blocks; and put in suitable measures to minimize dilution and ensure the safety of people in the case where stope blocks are located next to voids that are filled.

There are several possibilities of void condition that could be encountered:

- They could be completely empty, or partially filled with material that has caved from the sidewalls of the stope. If such a void is encountered, it would need to be filled with cemented backfill in order to safely mine adjacent areas.

- They could be filled with water and mud. In such a case, it is almost certain that the mud would have settled and solidified at the bottom of the void. In such a case, the water would need to be drained through cover drilling holes. Once this has been achieved, the void would need to be filled with cemented backfill as recommended in the Backfill Section of this report.

- If the void is filled with loose material, it will not be safe to remove the waste due to the risk of losing equipment. In such a case, a pillar would need to be left around the void before mining adjacent areas.

A potential risk to the project is a deficit in backfill material in the early years on the mine. The problem will worsen if a large number of unplanned and unfilled voids are encountered. To reduce the risk of having a deficit of fill material, the mine will have to use development waste and/or waste rock from a borrow pit on surface. The waste rock will have to be combined with cement as recommended in the Backfill Section of this report.

16.2.3 Mine Access

The shrinkage stopes will be accessed via the San Leon tunnel, which is an existing haulage that was holed through the mountain during historic mining. The tunnel has a width and height of 2.5 m and a total length of about 1.57 km. The long hole stopes will be accessed via a 4 m (h) x 4 m (w) decline ramp system that will be developed from surface to allow rock to be trucked out of the mine. The ramp system will be developed at an inclination of 8 degrees from the horizontal and it will comprise the east decline ramp and the west spiral ramp. Figure 16.3 shows the decline ramp system and Figure 16.4 shows the designed mine layout.
Figure 16.3: The Ramp System for Long Hole Stoping
Figure 16.4: Layout of the Mine
A second outlet from the mine has been included in the design and will be available for people in case of emergencies. This outlet will be located in the vent raise adjacent to the west ramp. The vent raise measures 3 m (h) × 3 m (w). It will have fresh air intake from surface and it will be equipped with a ladder system. The raise intersects the west ramp at each level and hence will be accessible from all levels, as shown in Figure 16.5 below.

![Image](image-url)

**Figure 16.5: A Secondary Mine Outlet (Vent Raise)**

### 16.3 Mine Production

The mine has been designed for an ore production of 1,000 tons (t) per day. The mine plan indicates a total of 3,557 Million tons of ore and 0.839 Million tons of waste. These will be extracted over a planned 12.5 Life of Mine (LOM). The average grades over the LOM are 239 g/t for silver, 1.09% for lead and 1.91% for zinc. A summary of the annual mine production plan is outlined in the table below.
Table 16.1: Annual Mining Schedule

<table>
<thead>
<tr>
<th>Year</th>
<th>Ore Mined (Tonnes)</th>
<th>Silver Grade (G/T)</th>
<th>Lead Grade (%)</th>
<th>Zinc Grade (%)</th>
<th>Silver Production (MIL OZ)</th>
<th>Lead Production (Tonnes)</th>
<th>Zinc Production (Tonnes)</th>
<th>Dev Waste (Tonnes)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>11,316</td>
<td>129</td>
<td>0.63</td>
<td>1.37</td>
<td>0.047</td>
<td>71</td>
<td>155</td>
<td>36,072</td>
</tr>
<tr>
<td>2</td>
<td>97,886</td>
<td>244</td>
<td>1.71</td>
<td>2.61</td>
<td>0.767</td>
<td>1,673</td>
<td>2,552</td>
<td>162,058</td>
</tr>
<tr>
<td>3</td>
<td>170,872</td>
<td>332</td>
<td>1.66</td>
<td>3.27</td>
<td>1.826</td>
<td>2,828</td>
<td>5,581</td>
<td>119,344</td>
</tr>
<tr>
<td>4</td>
<td>259,160</td>
<td>365</td>
<td>1.77</td>
<td>3.05</td>
<td>3.045</td>
<td>4,576</td>
<td>7,916</td>
<td>38,137</td>
</tr>
<tr>
<td>5</td>
<td>283,979</td>
<td>341</td>
<td>1.44</td>
<td>2.18</td>
<td>3.117</td>
<td>4,083</td>
<td>6,182</td>
<td>106,003</td>
</tr>
<tr>
<td>6</td>
<td>405,150</td>
<td>272</td>
<td>1.28</td>
<td>1.89</td>
<td>3.548</td>
<td>5,202</td>
<td>7,663</td>
<td>378,273</td>
</tr>
<tr>
<td>7</td>
<td>361,355</td>
<td>262</td>
<td>1.53</td>
<td>2.13</td>
<td>3.050</td>
<td>5,519</td>
<td>7,702</td>
<td>0</td>
</tr>
<tr>
<td>8</td>
<td>365,009</td>
<td>198</td>
<td>0.80</td>
<td>1.40</td>
<td>2.327</td>
<td>2,928</td>
<td>5,124</td>
<td>0</td>
</tr>
<tr>
<td>9</td>
<td>363,243</td>
<td>199</td>
<td>0.77</td>
<td>1.56</td>
<td>2.320</td>
<td>2,809</td>
<td>5,650</td>
<td>0</td>
</tr>
<tr>
<td>10</td>
<td>361,982</td>
<td>191</td>
<td>0.72</td>
<td>1.63</td>
<td>2.222</td>
<td>2,615</td>
<td>5,912</td>
<td>0</td>
</tr>
<tr>
<td>11</td>
<td>366,580</td>
<td>191</td>
<td>0.69</td>
<td>1.66</td>
<td>2.251</td>
<td>2,528</td>
<td>6,093</td>
<td>0</td>
</tr>
<tr>
<td>12</td>
<td>367,580</td>
<td>173</td>
<td>0.81</td>
<td>1.65</td>
<td>2.043</td>
<td>2,979</td>
<td>6,057</td>
<td>0</td>
</tr>
<tr>
<td>13</td>
<td>143,572</td>
<td>178</td>
<td>0.78</td>
<td>0.92</td>
<td>0.822</td>
<td>1,115</td>
<td>1,319</td>
<td>0</td>
</tr>
<tr>
<td>Overall</td>
<td>3,557,684</td>
<td>239</td>
<td>1.09</td>
<td>1.91</td>
<td>27.39</td>
<td>38,926</td>
<td>67,906</td>
<td>839,887</td>
</tr>
</tbody>
</table>

16.4 Hydrogeology

No detailed hydrogeological studies have taken place at the modern Pulacayo mine. Despite this, water is not expected to be a problem on the first seven levels of the mine (Level Zero to Level 147). This conclusion is based on the information obtained from the existing workings at the mine. TWP recommends that a hydrological study be done before mining beyond level 147.

16.5 Geotechnical

TWP conducted a geotechnical investigation at Pulacayo mine. The purpose of the investigation was to define geotechnical conditions of the rock mass, identify geological features that will affect stability and determining mining spans, pillar dimensions and support requirements for the different excavations. The investigation consisted of the engineering geological description, assessment and geotechnical logging of borehole core and underground mapping. TWP Projects’ Geotechnical Engineer, Mr. Xolisani Ndlovu, visited site from 13 May to 27 May 2012 to carry out the geotechnical investigation. Dr Michael Roberts oversaw Xolisani’s entire project work.

16.5.1 Quality of the Rock Mass

A total of 29 boreholes with ore intersections were selected for the evaluation. Eighteen boreholes previously logged by Geomecanica Latina S.A were also included in the analysis in an attempt to increase the database of rock mass characterization. On plotting both sets of
boreholes, it was found that 8 boreholes did not intersect or plot close to the proposed stopes. These boreholes were excluded from the database and were not used in the design of the underground workings.

Two rock mass rating systems were used, namely Laubscher’s Mining Rock Mass (RMRL) System (1990) and Barton’s Q System (1974). The ore body and country rock were found to be generally in the poor to fair range of rock mass quality. The average RMRL values were estimated to be 44 (fair) for the hangingwall and 42 (fair) for the footwall. The average Q values were estimated to be 2.3 (poor) and 1.8 (poor) for the hangingwall and footwall, respectively. The results of the mapping exercise that was completed in one of the tunnels (drift OB7) show similar rockmass conditions in the country rock. The Q values average 2.6 (poor) with a minimum of 1.3 (poor) and a maximum of 4.2 (fair). These values translate to equivalent RMR values of minimum of 47 (fair), average of 53 (fair) and maximum of 57 (fair).

An effort was made to demarcate the different sections of the mine into different geotechnical zones based on rock mass characterization. This would have enabled design of different stope dimensions for the different geotechnical zones based on rock mass quality. This analysis was not conclusive due the limited number of boreholes in some of these areas. It is therefore recommended that additional geotechnical logging of boreholes that intersect the proposed stopes should be carried out to increase the database of rock mass quality. For design purposes, a single geotechnical zone was assumed.

16.5.2 Mining Spans

Analysis of the geotechnical data described above indicate that the recommended stope dimensions of maximum 30 m (height) by maximum 10 m (length) and maximum 6 m (width) will be stable. Stopes are expected to remain free standing for a reasonable time with possible sidewall slabbing issues for stopes that plot in the transition zone on Potvin’s Stability Graph. The slabbing material will generally come from the mineralized zones thus the dilution of ore should be minimized, as the diluting material will have some grade in it. Cemented backfill placed after the broken ore has been drawn out will provide permanent support to the stope back.

16.5.3 Support of Excavations

16.5.3.1 Pillars

A sill pillar will be required below zero level to protect the San Leon tunnel from the proposed underground workings and a crown pillar will be required to protect the underground workings from the planned future open pit mine. The study proposes a crown/sill pillar thickness of 15 m. This was determined based on a Safety Factor of 1.8 and an average Q value of 2.6. The possibility of having voids in the pillars was also taken into consideration. The design and stability of the proposed pillar width was assessed using empirical formulae and deemed sufficient.

Crosscut protection pillars will also be required to support crosscuts during mining. Once all the stopes being serviced by the crosscut have been mined out, the crosscut pillar will be recovered on retreat. A 15 m wide crosscut protection pillar is deemed sufficient. This dimension was checked using numerical modeling code Map3D.
16.5.3.2 Other Support Types

Tendons and Other Support Types: Support tendons will be in form of 20 mm diameter rock bolts except where specified otherwise. The tendons will be full column resin grouted with a fast setting capsule installed in the back of the hole. The rest of the hole will be filled with medium setting resin and the tendons will be spun to stall. The lengths of the tendons are tabulated below. The spacing of the tendons will vary depending on ground conditions.

In good ground conditions (as per Rock Engineer’s assessment) support spacing can be opened up to 1.5 m x 1.5 m while in poor ground conditions, support spacing can be reduced to 1.0 m x 1.0 m. Weld mesh or chain link wire mesh or shotcrete should be used to support excavations and keep loose rocks intact in poor ground conditions as per the Rock Engineer’s assessment. The mesh should be galvanized to minimize the rate of corrosion, and its aperture size should be 100 mm x 100 mm with wire strand diameter of 4 mm.

In exceptionally poor ground conditions (as per Rock Engineer’s assessment), timber sets should be used for both the hangingwall and sidewalls.

Table 16.2: Tendon Support Requirements for Different Excavations

<table>
<thead>
<tr>
<th>Excavation dimensions</th>
<th>Excavations</th>
<th>Tendon length</th>
<th>Comment</th>
</tr>
</thead>
<tbody>
<tr>
<td>2.5 m (w) x 2.5 m (h)</td>
<td>Ore and waste drives for shrinkage stopes</td>
<td>2.1 m</td>
<td>Weld mesh or 50 mm thick plain shotcrete will be required in poor ground conditions</td>
</tr>
<tr>
<td>3.0 m (w) x 3.0 m (h)</td>
<td>Waste pass, vent raise and RAW</td>
<td>2.1 m</td>
<td>Weld mesh or 50 mm thick plain shotcrete will be required in poor ground conditions</td>
</tr>
<tr>
<td>4.0m (w) x 4.0m (h)</td>
<td>Ramp, access ramp, haulage, crosscuts for the longhole stopes, mucking bays, orepass access, vent raise access</td>
<td>2.1 m</td>
<td>Weld mesh or 50 mm thick plain shotcrete will be required in poor ground conditions</td>
</tr>
<tr>
<td>4.0m (w) x 6.0m (h)</td>
<td>Loading bay</td>
<td>2.1 m</td>
<td>Weld mesh or 50 mm thick plain shotcrete will be required in poor ground conditions</td>
</tr>
<tr>
<td>5.0m (w) x 6.0m (h)</td>
<td>Workshop</td>
<td>2.4 m</td>
<td>Locate workshop in good ground and avoid poor areas. 5 m long, 18 mm diameter 38 ton cable anchors will be required at 2 m x 2 m spacing.</td>
</tr>
<tr>
<td>6.0m (w) x 4.0m (h)</td>
<td>Long hole stope ore drives</td>
<td>2.4 m</td>
<td>Rock bolts will be 2.4 m long and 20 mm diameter at 1.2 m x 1.2 m spacing. 50 mm thick plain shotcrete should be used in poor ground conditions.</td>
</tr>
</tbody>
</table>

16.6 Mine Development

The decline ramp system will be developed from surface (at an inclination of 8 degrees from the horizontal) using conventional drill and blast techniques. Broken rock from the ramp development will be mucked out using the 3.1 m³ LHD that the mine already owns. Mucking bays will be developed every 100 m along the decline ramp. These will serve as temporary
storage of broken rock as the ramp system deepens. Once the ramp development has been completed, the mucking bays will be used as passing bays.

Conventional drill and blast techniques will also be used to advance existing ore and waste drives, and to develop new shrinkage stopes. Air loaders with a bucket capacity of 0.5 tonne will be used to clean development faces. The broken rock will be loaded into locomotives and trammed out of the mine. A face jumbo will be used to develop tunnels that are required to access the long hole stopes. These tunnels are the main haulage, crosscuts and ore drives. An LHD will be used to muck out, and the broken rock will be hauled out of the mine using the 15 tonne truck that the mine already owns. Utilities including compressed air and water pipelines, ventilation ducts and power cables will be installed, supported from the hangwall, as the ramp system advance.

The development will take place by drilling a planned round of 4.2 m (at a blasthole diameter of 45mm) for the trackless section and 2.4 m (at a blasthole diameter of 38mm) for the conventional section of the mine to achieve an effective face advance of 3.8 m and 2.2 m respectively per blast. The blastholes will be charged with Anfex. The total development meters for the mine are shown in the Figure 16.6 below. The lack of development in the conventional section between the third and fourth year is due to the need to rehabilitate and open up old existing workings to access shrinkage stopes. In the trackless section of the mine, there is a high increase in ore development between years five and six. This is due to the compression of the tail end of the development year seven into year six.

![Ore & Waste Development](image)

Figure 16.6: Development metres at Pulacayo mine

Roof bolts will be installed in the tunnels by means of a roof bolter in the trackless section and hand-held drills in the conventional section.
16.7 Backfill

Backfilling of mined out long hole stopes is planned to facilitate safe mining. Cemented backfill will provide permanent support to the stope excavation, particularly with regard to the mining of adjacent areas. Mitchell's equation was used for the determination of the required strengths of a cemented backfill block with freestanding heights of 20 m and 30 m. The results indicate a UCS strength in the range of 0.35 MPa to 0.40 MPa.

16.7.1 Test Work

The backfill system presented in this study is conceptual. Appropriate detailed site specific project engineering work for detailed system design is recommended on process flow diagrams, process and instrument diagrams, mass balances, detailed equipment selection, detailed site specific equipment locations, and plant and equipment general arrangement drawings etc. Furthermore, only limited test work has been done on the hydraulic transportation properties of Pulacayo tails as part of the Feasibility Study. As a result, the pump, pipeline and equipment suggestions made in the Feasibility Study are based purely on industrial experience.

16.7.2 Stope Backfilling Process

*Backfill bottom reinforcement:* The sequence of mining, as proposed for Pulacayo mine, requires backfilled stopes to be undermined. As a result, the backfill in the lower portion of each stope will be reinforced to prevent the possible collapse of backfill into the lower stope. Mesh reinforcement, as practiced at *Lucky Friday Mine* in the United States, has therefore been suggested. F2416.7 shows the suggested reinforcement design.
The reinforcement will be installed remotely to ensure the safety of people. It is proposed that two operators position themselves on either side of the top of the stope (wearing PPE), or on a suitable portable steel gantry placed across the top of the stope. The reinforcement will be attached to ropes and lowered to the bottom of the stope. This operation may be facilitated by two additional operators, one at the top and one at the bottom of the stope in the entrance, using portable radios to direct the operators in lowering the strips.

**Bulkhead construction:** Once the reinforcement has been installed, the stope will be sealed off with a bulkhead (see Figure 16.8 below). The outside of the bulkhead will be supported by a temporary hardwood timber structure. The timber support structure can be dismantled and removed after the backfill has cured for 28 days; thereafter, it can be re-used at another bulkhead.
**Backfill placement:** All stopes will be filled from the level above (top access) to facilitate pipeline flushing. Backfilling will be conducted at filling rates in the range from 30-40 m$^3$/hour. This flow rate will provide a satisfactory rate of rise of the uncured pastefill against the fill barricades and allow operations within time constraints of sequenced mining and backfilling. The pastefill will comprise tailings from the process plant and about 6% by mass, cementitious binder. A high cement content is required due to the high clay content of the ore, as well as the tailings derived from it.

An initial lower backfill plug will be placed into each stope. The plug will extend vertically to about 1 m above the top of the ore drive tunnel hanging wall. The initial lower plug will be left to cure undisturbed for about seven days, after which, the remainder of the stope will be filled with cemented backfill paste. In the case where there is a shortage of tailings, development waste can be used. The waste has to be dump into the stopes simultaneously with cemented backfill paste to prevent uncemented voids from forming between rocks! In addition, the development waste cannot be dumped into the stopes during the initial plug filling, as this would knock over the upright backfill anchors.

### 16.8 Ventilation

The planned ventilation system for Pulacayo mine is complicated in some respects by a number of unknowns with respect to the existing workings. The planned operations cover a vertical extent of some 150 m over seven levels. The ventilation system is intended to ultimately establish through-ventilation on the main haulages of all levels, from which fans and ducts can draw air for stopes and ore access development. There will not be any ‘main fan’ installations on surface; instead, a number of relatively small (30 m$^3$/s each) ‘booster’ fans will be installed underground at positions leading to surface connections. The intent is to ensure that a positive flow of air [from whichever source, including those that may be unknown] will report underground to the position of the fan inlets. Ventilation controls [walls,
regulators, and small fans] will be used to ensure that the air circulates through the production haulages on its way to the booster fans.

The ventilation design is to route intake air to the centre of the mine and to the various levels via the production ramp and central shaft. Air will then flow out from the centre of the mine towards the extremities of each level, then to a ‘gathering level’, proposed to be Level 25. On the East side of the mine, a connection will be made to surface from Level 25, with four fans located at the bottom. This exhaust shaft will handle 120 m$^3$/s and will be approximately 10 m$^2$ in cross-sectional area. On the West side, return air will be routed to several existing ramps that connect to the existing Candelaria 1 and Rothchild shafts. A further 120 m$^3$/s will be exhausted from the mine via these routes. Air will be routed from level to level via dedicated ventilation passes, or through old mining connections, where they exist close to where they are required. Figure 16.9 below, illustrates the tonnage and ventilation requirement of the mine with time.

![Figure 16.9: Stoping Tonnage versus Annual Ventilation Requirements with Time](image)

Figure 16.10 below shows what the ultimate ventilation layout of Pulacayo mine will be. Light blue airways denote main intake airways, with red denoting return airways. Dark blue are the production levels. The black colored level denotes the workings on Level Zero, which will be mined as conventional shrinkage stopes. The layout incorporates some risk – the routes to the surface connections are small and possibly highly restricted. Existing plans and documents have been scrutinized to estimate the size of these airways, and ventilation modeling software [VUMA and Ventsim] were used to determine the required fan capacities. Discussions with mine staff have indicated that there would be a willingness to re-raise some connections if necessary.
16.9 Mining Productivity

16.9.1 Mine Operating Schedule

The production elements of the Pulacayo mine will operate on a 24/7 basis. The mine is scheduled to operate two 10 hour shifts per day over a period of 345 days per annum. This results in a total of 6,900 gross operating hours per annum.
Figure 16.10: Pulacayo Complete Footprint - View Looking Southwest
16.9.2 Shift Utilization

Shift utilization is defined as the time that available equipment is put to use. It takes into account operator inefficiency (including queuing and comfort breaks for operators) and the time that is lost due to non-productive activities such as stope and development end inspections, safety meetings and equipment pre-use inspections. Shift utilization was estimated to be 82.5%, as shown in the mining chapter of the Feasibility Study Report (Document No: 090644-3-0000-20-IFI-116).

16.9.3 Mechanical Availability

Mechanical availability is defined as the time that a unit of equipment is mechanically available to operate. According to Kennedy B.A. (1990), it only takes the time that is lost due to planned maintenance (e.g. weekly maintenance and major overhauls) into account. Mechanical availability was calculated to be 87.6%, as shown in the mining chapter of the Feasibility Study Report (Document No: 090644-3-0000-20-IFI-116). This includes a 5% allowance that has been made for unforeseen stoppages. The table below shows the proposed maintenance plan for the mining fleet:

<table>
<thead>
<tr>
<th>Maintenance</th>
<th>hours/week</th>
<th>hours/annum</th>
</tr>
</thead>
<tbody>
<tr>
<td>Weekly maintenance</td>
<td>4.0</td>
<td>208.0</td>
</tr>
<tr>
<td>Monthly maintenance</td>
<td>8.0</td>
<td>96.0</td>
</tr>
<tr>
<td>Major overhauls (5 days per 5 000 hours)</td>
<td>6.9</td>
<td>138.0</td>
</tr>
<tr>
<td>Total mechanical unavailability time</td>
<td></td>
<td>442.0</td>
</tr>
</tbody>
</table>

16.9.4 Net Operating Hours

Taking shift utilization and mechanical availability into account, effective operating hours were calculated as follows:

<table>
<thead>
<tr>
<th></th>
<th>hours/annum</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gross operating hours</td>
<td>6,900</td>
</tr>
<tr>
<td>Mechanical availability</td>
<td>87.6%</td>
</tr>
<tr>
<td>Shift utilization</td>
<td>82.5%</td>
</tr>
<tr>
<td>Maximum net operating hours</td>
<td>4,988</td>
</tr>
<tr>
<td>Overall availability</td>
<td>72%</td>
</tr>
</tbody>
</table>
16.9.5 Swell Factor

The swell factor is the ratio of the bank density of material to the loose density of the same material. It is used to calculate the weight and volume of material that has been blasted and is to be loaded and hauled to its destination. The average in-situ density of ore and waste is 2.47 t/m$^3$ and 2.2 t/m$^3$ respectively. The swell factor for both ore and waste is estimated to be 30% and the loose densities were estimated to be 1.90 t/m$^3$ for ore and 1.69 t/m$^3$ for waste.

16.10 Mine Equipment Fleet Requirements

A trade-off study has been done to estimate the fleet of equipment required to meet the mine’s production targets. To avoid double accounting, the study took into account the equipment that the mine already owns. The mine is currently extracting the upper portions of the deposit using shrinkage stoping and the primary equipment used to accomplish this is listed below:

- 4 locomotives
- 4 flat cars
- 10 x 1 tonne Granby cars
- 5 x 4 tonne hoppers
- 2 air loaders (each with a 0.5 tonne bucket)
- 12 stopers
- 12 jackleg hand-drills

In addition to the equipment listed above, the mine also owns a fleet of refurbished trackless equipment, which is not currently in use (see Table 16.5 below).

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Model</th>
<th>Quantity</th>
</tr>
</thead>
<tbody>
<tr>
<td>LHD</td>
<td>Atlas Copco ST3.5 with a bucket capacity of 3.1 m$^3$ and a trammimg capacity of 6.0 tonne</td>
<td>2</td>
</tr>
<tr>
<td>Truck</td>
<td>Atlas Copco MT416 Wagner with a dump box volume of 7.5 m$^3$ and a rated payload of 15 tonne</td>
<td>1</td>
</tr>
<tr>
<td>Development drill rig</td>
<td>Sandvik secona quasar 14B</td>
<td>1</td>
</tr>
</tbody>
</table>

16.10.1 Primary Equipment

Table 16.6 summarizes the number of primary equipment required to supplement the equipment that the mine already has. These were determined based on the mine calendar and the production schedule:
Table 16.6: Additional Primary Trackless Equipment

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Typical model</th>
<th>Quantity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Truck</td>
<td>Atlas Copco MT2010 with a dump box volume of 8.4 m³ and a rated payload of 20 tonne</td>
<td>1</td>
</tr>
<tr>
<td>Production drill rig</td>
<td>Resemin raptor 55 drill rig (51 to 89 mm diameter)</td>
<td>1</td>
</tr>
<tr>
<td>Roof bolter</td>
<td>Resemin bolter 77D</td>
<td>1</td>
</tr>
<tr>
<td>Shotcreter</td>
<td>Atlas Copco Uni-grout miniflex E with a 200L capacity for the cemag agitator and 100L for the cemax mixer</td>
<td>1</td>
</tr>
</tbody>
</table>

16.10.2 Ancillary Equipment

The ancillary equipment required to support the primary fleet of the mine was also determined. These are tabulated below.

Table 16.7: Additional Primary Trackless Equipment

<table>
<thead>
<tr>
<th>Equipment type</th>
<th>Typical model</th>
<th>Quantity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cassette carrier</td>
<td>Fermel liberator RoRo cassette handler</td>
<td>2</td>
</tr>
<tr>
<td>Cassette (attachment)</td>
<td>Fermel flatbed</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td>Fermel scissors lift</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td>Fermel fuel and oil lube cassette with 2 x 500L oil tanks, 1 x 250L oil tank and 1 x 500L diesel tank</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td>Fermel general purpose cassette</td>
<td>1</td>
</tr>
</tbody>
</table>

16.11 Manpower Requirements

Table 16.8 below summarizes the manpower requirements of Pulacayo mine per annum. Underground workers will be operating on an 8 days on, 4 days off, roster and they will be alternating between morning and night shifts. All the other departments (engineering, technical services and stores) will be operating during the day shift only. A 12% provision (for leave) has been made on underground workers that are fundamental to mine production and development.
Table 16.8: Annual Manpower Requirements for Pulacayo Mine

<table>
<thead>
<tr>
<th>Project year</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
<th>7</th>
<th>8</th>
<th>9</th>
<th>10</th>
<th>11</th>
<th>12</th>
<th>13</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Mine Management</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
<td>5</td>
</tr>
<tr>
<td>2. Engineering</td>
<td>15</td>
<td>15</td>
<td>15</td>
<td>20</td>
<td>20</td>
<td>15</td>
<td>15</td>
<td>15</td>
<td>15</td>
<td>15</td>
<td>15</td>
<td>15</td>
<td>15</td>
</tr>
<tr>
<td>3. Technical Services</td>
<td>6</td>
<td>8</td>
<td>12</td>
<td>12</td>
<td>13</td>
<td>13</td>
<td>13</td>
<td>13</td>
<td>13</td>
<td>13</td>
<td>13</td>
<td>13</td>
<td>13</td>
</tr>
<tr>
<td>4. Underground Workers</td>
<td>80</td>
<td>126</td>
<td>126</td>
<td>90</td>
<td>135</td>
<td>135</td>
<td>101</td>
<td>101</td>
<td>101</td>
<td>101</td>
<td>65</td>
<td>46</td>
<td>40</td>
</tr>
<tr>
<td>4.2 Production</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
<td>16</td>
</tr>
<tr>
<td>4.1 Conventional Mining</td>
<td>22</td>
<td>48</td>
<td>48</td>
<td>6</td>
<td>48</td>
<td>48</td>
<td>36</td>
<td>36</td>
<td>36</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>4.3 Development</td>
<td>27</td>
<td>41</td>
<td>41</td>
<td>41</td>
<td>41</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td>19</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>4.4 Rock Transportation</td>
<td>15</td>
<td>21</td>
<td>21</td>
<td>27</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>24</td>
</tr>
<tr>
<td>5. Stores</td>
<td>4</td>
<td>4</td>
<td>6</td>
<td>6</td>
<td>6</td>
<td>6</td>
<td>6</td>
<td>6</td>
<td>6</td>
<td>6</td>
<td>6</td>
<td>6</td>
<td>6</td>
</tr>
<tr>
<td>5.1 Material store</td>
<td>2</td>
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<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
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<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>5.2 Explosives store</td>
<td>2</td>
<td>2</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Total</td>
<td>110</td>
<td>158</td>
<td>164</td>
<td>133</td>
<td>179</td>
<td>179</td>
<td>140</td>
<td>140</td>
<td>140</td>
<td>140</td>
<td>104</td>
<td>85</td>
<td>79</td>
</tr>
</tbody>
</table>
16.12 Underground Infrastructure and Mine Services

This section summarizes underground infrastructure and mine services required to support the extraction of the ore body at Pulacayo. The designs have been made for Levels Zero, 25, 50, 75, 90, 116 and 147.

16.12.1 Existing Underground Infrastructure and Installations

The following infrastructure and services currently exist at Pulacayo mine:

- Two vertical shafts, namely Shaft No. 1 and Shaft No. 2 (Candelaria 1). Of these two, Candelaria 1 has been selected for the placement of water and air distribution equipment for the mine. It has a diameter of 2.5 m and it is still in good condition.

- A fresh water line (150mm steel pipe) which transports water from Yanapollera Dam (located 13 km away from the mine) along the Pacamayu and the San Leon tunnel entrances.

- A fresh water reservoir where the water from the dam is stored. The reservoir is placed near the San Leon tunnel entrance and is used to supply water to the mine and the village of Pulacayo.

16.12.2 Proposed Mine Services

Compressed air supply and distribution: Compressed air will be provided by two compressors, each with a capacity of 0.823m³/s (1744 cfm) and a compressed air storage vessel with a capacity of 10 m³. The compressors will be connected in parallel to allow one compressor working on normal operation conditions and two compressors when there is peak demand. The second compressor will also act as a standby in case of breakdown or maintenance. Compressed air will be supplied via a 150mm HDPE pipeline. The pipeline will run from surface to Level Zero (through Candelaria 01) where it will be diverted to the workshop and ramp access on Level 25, and towards east and west on Level Zero. A smaller pipeline (100mm) will be used to transport the compressed air along Level Zero. From the ramp access, the pipeline (150mm) will run down the ramp to the various levels. All the pipelines will be fixed on the side walls of the tunnels, and supported by chains placed every two meters and connected to roof bolts.

Water supply and distribution: A 300 m³ capacity tank will be used to supply water to the mine. The tank will supply water by gravity through a 100mm HDPE pipeline at a rate of 29.3 m³/h. The pipeline will run from surface to Level Zero (through Candelaria 1) where it will be diverted to the workshop and ramp access on Level 25, and towards east and west on Level Zero. From the ramp access, the pipeline will run down the ramp to the various levels. All the pipelines will be fixed on the side walls of the tunnels, and supported by chains placed every two meters and connected to roof bolts.

Electrical equipment: A sub-station and an encapsulated power generator will be placed on surface on Level +147. The sub-station will consist of a switchgear (400 kW, 400 V, 3φ, 50 Hz, at 4 200 m). The substation will consist: switchgear MT (25 kV), power transformer (2 MVA, 25/0.4 kV), main switchgear (0.4 kV, 3 phases, 3H, 50 Hz) and a distribution transformer (100 kVA 380/220 V) amongst others. Three mini sub-stations will be placed...
underground on Levels Zero, 75 and 145 along the access ramp. Each mini sub-station will consist of LT switchgear (0.4 kV), MT switchgear (3.3 kW) and a transformer (3.3/0.4 MVA 0.5 kV). In case of power failure, the power generator will be used.

16.12.3 Proposed Underground Infrastructure

_Underground workshop:_ The workshop will be located in an old hoist chamber on Level Zero (see Figure 16.11). It will be used to do daily and minor maintenance on the underground mining fleet. The services will include a wash bay, fuel and lube bay and a tyre change bay. Major services will be handled in the existing main workshop on surface. The surface workshop will be equipped with all the machinery and tools required to perform the tasks.

_Collection ponds:_ Collection ponds will be installed along the access ramp on Level Zero, Level 75 and Level 147, to collect the water used for drilling and other mining activities. The ponds will have a length of 30 m, a width of 4m and a height of 2m and they will be able to store water for 6 hours without the need for pumping (as a contingency). In addition, each pond will have three centrifugal pumps for dewatering. Two pumps will be used for pumping the water stored in the pond (one pump will be on standby) and the other pump will be used to recirculate the fluid and thus prevent any settling of sludge or solids. The pumping system has been designed in a cascade configuration. The benefits of this configuration include low energy consumption and reduced pressure in pipelines.

Submersible pumps will also be installed in _Candelaria 1_ and various other centrifugal pumps will be installed on surface. The water will be pumped out of the mine via a 150mm HDPE pipeline to a settler on surface (Level +147). Once the water has been treated, it will be pumped back to the 300 m³ storage tank.

![Figure 16.11: Location of the Underground Workshop](image)
17 RECOVERY METHODS

The Pulacayo mineral processing plant design is based on metallurgical test results undertaken by various laboratories on behalf of Apogee Silver Ltd. Where the test work results were unavailable, assumed data from similar projects were used. The plant has been designed to process 1,000 t/d of polymetallic bearing mineral ores. The overall process plant and unit operations therein are designed to produce base metal concentrates of lead and zinc with silver reporting predominantly to the lead concentrate.

The processing plant will include circuits for conventional crushing, screening, clays washing and screening, grinding, lead flotation, zinc flotation, concentrate filtration and storage, reagents storage and handling, tailings storage and disposal facility, backfill, auxiliary services, carbon columns to remove copper ions in solution and organic flotation reagents from the recirculated process water.

The processing plant is a conventional design and comprises equipment of the type and size that has been proven in the plants processing similar mineral ores in the industry.

17.1 Process Design Criteria

Apogee Silver Ltd. tasked TWP Sudamérica S.A. to design a process plant for the Pulacayo project with a design capacity of 1,000 t/d. For the design, TWP used an availability factor of 86.2% for the crushing and milling areas. These design availabilities are fairly common for current and recent projects at TWP.

The annual ore tonnage of 360,000 metric tons can be processed through the mill in 360 days.

A summary of the design parameters are given below in Table 17.1

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unit</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore Characteristics</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Silver, Ag</td>
<td>Design g/t</td>
<td>240</td>
</tr>
<tr>
<td></td>
<td>Range g/t</td>
<td>135.6 - 487.3</td>
</tr>
<tr>
<td></td>
<td>Mean g/t</td>
<td>302.3</td>
</tr>
<tr>
<td>Lead, Pb</td>
<td>Design %</td>
<td>1.1</td>
</tr>
<tr>
<td></td>
<td>Range %</td>
<td>0.66 - 4.3</td>
</tr>
<tr>
<td></td>
<td>Mean %</td>
<td>2.28</td>
</tr>
<tr>
<td>Zinc, Zn</td>
<td>Design %</td>
<td>1.92</td>
</tr>
<tr>
<td></td>
<td>Range %</td>
<td>1.34 - 5.96</td>
</tr>
<tr>
<td>Parameter</td>
<td>Unit</td>
<td>Value</td>
</tr>
<tr>
<td>---------------------------------</td>
<td>------------</td>
<td>---------</td>
</tr>
<tr>
<td>Mean</td>
<td>%</td>
<td>2.78</td>
</tr>
<tr>
<td>Specific Gravity</td>
<td>Design</td>
<td>2.65</td>
</tr>
<tr>
<td>Range</td>
<td>-</td>
<td>2.65 - 3.02</td>
</tr>
<tr>
<td>Mean</td>
<td>-</td>
<td>2.85</td>
</tr>
<tr>
<td>Crushed Ore Bulk Density</td>
<td>-</td>
<td>1.67</td>
</tr>
<tr>
<td>Moisture Content</td>
<td>%</td>
<td>8-10</td>
</tr>
<tr>
<td>Abrasion Index (Ai)</td>
<td>Design</td>
<td>-</td>
</tr>
<tr>
<td>Clays Content</td>
<td>-</td>
<td>Yes</td>
</tr>
<tr>
<td>Bond Rod Mill Work Index</td>
<td>Design</td>
<td>kW-h/t</td>
</tr>
<tr>
<td>Bond Ball Mill Work Index</td>
<td>Design</td>
<td>kW-h/t</td>
</tr>
<tr>
<td>Range</td>
<td>kW-h/t</td>
<td>13 - 14.6</td>
</tr>
<tr>
<td>Mean</td>
<td>kW-h/t</td>
<td>14</td>
</tr>
<tr>
<td>Bond Crusher Work Index</td>
<td>Design</td>
<td>kW-h/t</td>
</tr>
<tr>
<td>Angle of Repose Coarse Ore</td>
<td>°</td>
<td>35</td>
</tr>
<tr>
<td>Angle of Repose Fine Ore</td>
<td>°</td>
<td>42</td>
</tr>
<tr>
<td>Angle of Drawdown</td>
<td>°</td>
<td>70</td>
</tr>
<tr>
<td>OPERATING PHILOSOPHY</td>
<td></td>
<td></td>
</tr>
<tr>
<td>General</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Plant Throughput (Maximum Throughput)</td>
<td>t/y</td>
<td>360,000</td>
</tr>
<tr>
<td>Life Of Mine (LOM)</td>
<td>y</td>
<td>12.5</td>
</tr>
<tr>
<td>Operating Days Per Year</td>
<td>Nominal</td>
<td>d/y</td>
</tr>
<tr>
<td>Operating Hours Per Day</td>
<td>Nominal</td>
<td>h/d</td>
</tr>
<tr>
<td>Operating Hours Per Year</td>
<td>Nominal</td>
<td>h/y</td>
</tr>
<tr>
<td>Crushing Operating Schedule</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Crushing Operating Hours Per Year</td>
<td>Design</td>
<td>h/y</td>
</tr>
<tr>
<td>Crushing Overall Availability</td>
<td>%</td>
<td>86.2</td>
</tr>
<tr>
<td>Shifts</td>
<td>-</td>
<td>2</td>
</tr>
<tr>
<td>Design</td>
<td>h/shift</td>
<td>12</td>
</tr>
<tr>
<td>Concentrator Operating Schedule</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Milling Operating Hours Per Year</td>
<td>Design</td>
<td>h/y</td>
</tr>
<tr>
<td>Milling Availability</td>
<td>%</td>
<td>86.2</td>
</tr>
<tr>
<td>Shifts</td>
<td>-</td>
<td>3</td>
</tr>
<tr>
<td>Design</td>
<td>h/shift</td>
<td>8</td>
</tr>
</tbody>
</table>
The Pulacayo concentrator plant is designed to operate based on three 8 hour shifts per day. The overall availability design for is 86.2%, this will allow sufficient downtime for planned and unplanned maintenance process plant equipment.

The processing plant is designed to process 1,000 t/d and recover the base metals through the process described in Figure 17.1.
Figure 17.1 Overall Process Flow Diagram
17.1.1 Crushing Area

Run of mine ore will be delivered from the underground mine to the process plant via Gramby self-dumping cars through a railway system. The crushing area is situated on surface 1.43 km from the adit. The ore is moved from the adit to the crushing area and dumped onto a fixed grizzly with 400 mm bar spacings and into the coarse ore bin. Oversize rock is broken by a JCB backhoe with a rock breaker attachment. Since the ore comes from relatively small stopes, most of the rock is expected to be smaller than 500 mm.

From the 40 t coarse ore bin, ore is moved by a 6,600 mm by 760 mm apron feeder directly into the primary jaw crusher. The primary crusher, a 500 mm by 750 mm single toggle jaw type with a close side setting gap of 50 mm and installed with a 75 kW de-rated motor. The crushed ore is discharged to a belt conveyor. A self-cleaning moving belt magnet and fixed/static magnet on this belt removes tramp iron and a metal detector trips the belts protecting the secondary cone crusher from tramp metal such as rock bolts or steel balls.

The conveyor belt dumps directly onto a 4,800 mm by 1,800 mm double deck primary vibrating screen. The primary screen is provided with an upper deck screen of 25 mm and a lower deck screen of 12 mm. Both screens oversize fractions pass through a secondary cone crusher with a close side setting of 12 mm and equipped with a 200 kW de-rated motor. Both screen decks have polyurethane screen panels, polyurethane weir bars spaced at 900 mm intervals and four bar pressure spray nozzles wash the clays off the rock particles. The 1,300 mm diameter cone crusher discharges onto conveyor belt from where it is combined with the jaw crusher discharge and is returned to the screen (i.e. the cone crusher is in closed circuit with the vibrating screen). Another alternative has also been considered in which the primary screen is being replaced by a scrubber with a trommel at the discharge end. The scrubber discharge goes to the secondary double deck screen while the oversize material goes to the cone crusher. This option will be studied further during the detail engineering stage.

The primary screen undersize bypasses the second stage cone crusher and feeds directly onto the 4,800 mm by 1,800 mm double deck vibrating screen. Both screen decks have polyurethane screen panels, polyurethane weir bars spaced at 900 mm intervals and four bar pressure spray nozzles wash the clays off the rock particles. The dewatering screen will have an upper deck screen of 6.5 mm and a lower deck screen of 0.4 mm. Oversize fractions from the screen are transported to the two 1,000 t fine ore storage bins by a belt conveyor. This crushed product has a nominal top size of 12 mm, 85% +6.5 mm and 14.5% +0.4 mm.

Undersize fraction is directed to an in-line strainer (Tekleen filter, 600 kPa in-line pressure) to retain the clayish material to clarify the water that will be used again in the circuit (high pressure spray water and hosing water). The clayish fraction is directed to the ball mill discharge pump box (or back to the dewatering screen if required).

The design operating rate for the crushing plant is of 83.3 t/h. At 1,000 t/d, the crushing plant requires 12 hours per day operating time. The crushing circuit feeds (2) parallel milling circuits (1 and 2) of the same design capacity.
17.1.2 Milling Circuit

Each of the two parallel ball mills circuits receives ore from its fine ore bin via an electromagnetic vibrating feeder discharging onto the mill feed conveyor belt. Each ball mill (2,700 mm by 3,600 mm ball mill with rubber liners) is in closed circuit with its cluster cyclone. De-rated power is 560 kW.

Milled ore slurry at 70.9 percent solids by weight discharges from the ball mill through a trommel screen into a pump box and pumped with Wilfley centrifugal pumps to the cyclone cluster. The slurry is diluted with process water, and then classified in a D-6 (4 operating, 2 stand-by) Weir cyclone cluster with 150 mm diameter cyclones. Floor spillage from the sump pump is pumped to the mill discharge pump box.

Oversize particles report to the cyclone underflow apex and return to the ball mill feed for further size reduction. Particles that have met the eighty percent passing size of 74 microns report to the cyclone overflow via the vortex finder and move on to the rotospiral trash screen (2 mm) for waste fibre removal (woodchips, plastics, etc.) and then onto the flotation section.

The following reagent suite will be added to the ball mill feed:

\[ \text{ZnSO}_4 \cdot 7\text{H}_2\text{O} : \text{NaCN Complex (3:1)} \]

Each milling circuit (1 & 2) feeds each of the two (2) parallel flotation circuits (lead and zinc)

17.1.3 Lead Flotation Circuit

Milled slurry is directed to the high intensity conditioning tank (0300-TK-001/3) in the lead flotation circuit where the iron pyrite depressor (NaCN), Sodium Ethyl Xanthate (SEX) collector, Sasfroth 200 and pH modifier (Ca(OH)\text{2}) are added. After conditioning, the slurry is directed to the lead rougher cell flotation feed pump box (0300-PB-001/007) where Sodium Ethyl Xanthate (SEX) collector and Sasfroth 200 are added. The slurry is then pumped to the lead rougher cell (Imhoflot G cell 1.4, 0300-CL-001/6); the tailings are directed to the lead rougher/scavenger cell (Imhoflot G cell 1.4, 0300-CL-002/7). Similarly, the lead rougher/scavenger cell tailings are sent to the lead scavenger cell (Imhoflot G cell 1.4, 0300-CL-003/8). The lead scavenger tailings are pumped to the zinc flotation whilst the lead rougher, rougher/scavenger and scavenger concentrates are directed to the lead cleaner circuit. SEX collector, KU5 (CMC, Carboxymethyl Cellulose) and ZnSO\text{4} \cdot 7\text{H}_2\text{O} : \text{NaCN complex are added into the cleaner feed pump boxes (0300-PB-004/5 and 0300-PB-010/11).}

The lead cleaner circuit has two flotation cells in series (Imhoflot G cell 0.8, 0300-CL-004/9, 0300-CL-005/10). The final lead concentrate flows to the lead concentrate surge tank (0300-TK-002) and the final lead concentrate is pumped to the lead recessed plate pressure filter for dewatering.

17.1.4 Zinc Flotation Circuit

The lead circuit tailing is directed to the high intensity conditioning tank (0400-TK-001/3) in the zinc flotation circuit where pH modifier (Ca(OH)\text{2}), CuSO\text{4} \cdot 5\text{H}_2\text{O} ( sphalerite activator), Aero 3418A collector and Sasfroth 200 are added. The slurry is then pumped to the zinc
rougher cell (Imhoflot G cell 1.4, 0400-CL-001/6) and the tailings are directed to the zinc rougher/scavenger cell (Imhoflot G cell 1.4, 0400-CL-002/7). Similarly, the zinc rougher/scavenger cell tailings are sent to the zinc scavenger cell (Imhoflot G cell 1.4, 0400-CL-003/8).

The zinc scavenger tailings are pumped to the paste thickener (backfill area). The zinc rougher, rougher/scavenger and scavenger concentrates are directed to the zinc cleaner circuit. Hydrated lime and Aero 3418A collector are added into the cleaner feed pump boxes (0400-PB-006/8 and 0400-PB-014/16). The zinc cleaner circuit has two flotation cells in series (two Imhoflot G cell 0.8, 0400-CL-004/9, 0400-CL-005/10).

The final zinc concentrate flows to zinc concentrate surge tank (0400-TK-002) and it is pumped to the zinc recessed plate pressure filter for dewatering.

17.1.5 Carbon Columns Area

The lead concentrate filter filtrate, zinc concentrate filter filtrate, paste thickener overflow and run-off water report to three carbon in column (CIC) tanks in series where the copper ions and organic flotation reagents is adsorbed onto 6 x 12 mesh granular coconut carbon. The train of CIC tanks is made up of three 2 m diameter x 1 m high columns. The CIC treated solution water is pumped to the process water tank (1200-TK-006).

17.1.6 Concentrate Filtration Area

Lead concentrate slurry is pumped to the lead concentrate pressure filter, the filter size is 1.4 m x 7.59 m (W x L). The dewatered concentrate (< 10% moisture) is discharged through a chute onto a belt conveyor and conveyed to the lead concentrate stockpile.

The filtrate solution drains to a pump box and is pumped to the CIC system to adsorb dissolved metal ions that might be present in solution. The lead concentrate filter section floor spillage is pumped with the sump pump to the mill discharge pump box.

Zinc concentrate slurry is pumped to the zinc concentrate pressure filter, the filter size is 1.4 m x 8.58 m (W x L). The dewatered concentrate (< 10% moisture) is discharged through a chute onto a belt conveyor and conveyed to the zinc concentrate stockpile. The filtered solution is recovered in the pump box, and sent to the CIC system to adsorb the dissolved metal ions that might be present in solution.

The zinc concentrate filter section floor spillage is pumped with a pump sump to the zinc section conditioning tank. A vehicle weighbridge will be used to determine the mass of concentrates dispatched.

17.1.7 Backfill Area

The final zinc flotation tailings slurry is pumped through a series of pumps to an 8.4 m diameter paste thickener. The thickener overflow reports to flushing water tank (1500-TK-001), this tank will provide water for piping flushing, to the high shear mixing tank (1500-TK-004) and to the plant process water tanks. The excess water is pumped to a series of the carbon columns to adsorb dissolved metal ions and organic flotation reagents before reporting to the plant process water tanks. The paste thickener underflow is either pumped
to the tailings storage facility (TSF) (at 70% solids) or it is pumped to the backfill plant. Paste is prepared by mixing thickened tailings (65% solids) with cement (6% w/w) to produce a flowing material to be deposited underground. The paste plant is designed to treat 100% tailings.

17.1.8 Reagents Area

The reagents used within the plant are:

<table>
<thead>
<tr>
<th>Reagent</th>
<th>Unit</th>
<th>Consumption</th>
<th>Observations</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hydrated Lime (Ca(OH)2)</td>
<td>kg/t</td>
<td>3.83</td>
<td>pH modifier</td>
</tr>
<tr>
<td>Sodium Cyanide (NaCN)</td>
<td>kg/t</td>
<td>0.03</td>
<td>Iron pyrite depressor</td>
</tr>
<tr>
<td>Copper Sulphate Pentahydrate (CuSO4.5H2O)</td>
<td>kg/t</td>
<td>0.88</td>
<td>Sphalerite activator</td>
</tr>
<tr>
<td>Sodium Ethyl Xanthate (SEX)</td>
<td>kg/t</td>
<td>0.03</td>
<td>Collector, lead and zinc separation</td>
</tr>
<tr>
<td>KU5: Carboxymethyl Cellulose (CMC)</td>
<td>kg/t</td>
<td>0.15</td>
<td>Gangue depressor</td>
</tr>
<tr>
<td>Zinc Sulphate Heptahydrate : NaCN Complex</td>
<td>kg/t</td>
<td>0.08</td>
<td>Depressant for zinc, improve selective flotation of lead mineral</td>
</tr>
<tr>
<td>Coagulant</td>
<td>kg/t</td>
<td>0.1</td>
<td>Improve thickening performance</td>
</tr>
<tr>
<td>Sasfroth 200</td>
<td>kg/t</td>
<td>0.02</td>
<td>Frother</td>
</tr>
<tr>
<td>Aero 3418A</td>
<td>kg/t</td>
<td>0.01</td>
<td>Collector</td>
</tr>
<tr>
<td>Flocculant</td>
<td>kg/t</td>
<td>0.02</td>
<td>Improve thickening performance</td>
</tr>
<tr>
<td>Activated Carbon</td>
<td>kg/t</td>
<td>0.004</td>
<td>Used in packed carbon columns for process water treatment</td>
</tr>
</tbody>
</table>

Reagents will be delivered by road transport to the reagent store. Reagents are then withdrawn from the store, diluted in stirred mixing tanks, drained into the reagent dosing tanks and piped to the flotation sections.

17.1.9 Process Water System Area

The process water system is made of two (2) fresh water tanks and four (4) process water tanks. The fresh water tanks (1200-TK-001/002) receive water from the Yana Pollera reservoir. The reagent area, general services, potable water treatment plant, camp and the firewater pump receive water from these tanks.

The process water tanks receive make-up water from the fresh water tanks and also recovered water from the Carbon Columns (CIC). Process water is used in the plant and also make-up water for the spray water tank (0100-TK-001) located in the crushing area. In addition, these tanks provide water for hosing.

Water treatment facilities to produce potable water will be required at the plant site. The plant spillage will be contained in a dam and pumped back to the mill discharge sump.
nature of the composition of the drainage and fresh water are not well understood at this stage of the project and the technology required for water treatment will need to be defined during the next stage of engineering studies. The plant sewerage will drain to a septic tank.

17.1.10 Auxiliary Services Area

17.1.10.1 Compressed Air

High pressure air will be provided in the plant by the concentrates pressure filter compressor which provides air to operate the lead and zinc concentrates pressure filters. High pressure plant air will also be used for instrumentation in the plant. High pressure air will also be used for instrumentation air in the plant after it has passed through a dryer.

17.1.10.2 Diesel

Diesel fuel will be required for underground and surface mobile equipment and on-site emergency power generation equipment. A fuel storage container will be located at the plant site and a fuel rail tanker will be used to distribute fuel underground. The emergency generator will have a dedicated diesel storage tank.

17.1.10.3 Emergency Generator

During power outage, one diesel generator (75 kW) will supply energy for the paste thickener rake and pumps at the solid liquid separation area and lighting. The critical loads are the thickener rake motor, the thickener underflow pump and general lighting.

17.1.10.4 Tailings Disposal Facility (TSF) Area

Paste thickener underflow is directed to the paste tailings storage (1400-TSF-001) facility by gravity or by pumping. The TSF deposit is provided with a water drainage system that allows for water recover. The reclaimed water, which includes the water from the TSF surface, will be pumped to the Paste Thickener to remove any suspended solids (if any) in the water and the Paste Thickener overflow then reports the carbon columns and onto the process water tank.

17.1.10.5 Plant Spillage Containment Area

All excessive plant spillage from the crushing, milling and flotation sections will drain to the plant spillage containment dam. The dam has a sloped base, is lined with a geomembrane and equipped with a vertical spindle pump suspended from a frame and the pump is lowered into the slurry with a chain block. Any spillage in the dam will be hosed to the pump and pumped to the mill discharge sump.

17.1.11 Site Layout Considerations

The project site is steep, high-altitude terrain that has limited flat space. Due to these considerations, particular attention is required to develop acceptable sites for the facility. The general site arrangement is show in Figure 17.2 and the plant layout is presented in Figure 17.3.
The development of the site layout was based on maximizing the ease of the operation and minimizing both the capital and operating costs.

17.1.11.1 Plant Site Selection

The positioning of the plant also had to take the following additional factors into consideration.

- Site access for construction and operations.
- The supply of bulk services, water, electricity, and others, as well as environmental considerations, in particular visual impacts, dust and noise.
- Animal and plant systems, drainage lines and the presence of heritage resources.

No sensitive flora or fauna occur on the selected site position.

To ensure that the selected site would be acceptable from a construction aspect, preliminary geotechnical drilling and evaluation was also conducted.
Figure 17.2 Overall Site Layout
Figure 17.3 Concentrator Area Layout
17.1.12 Plant Layout

The plant layout was developed from the basic process design criteria. These criteria defined the required steps to treat the ore received from the mining operations to produce acceptable concentrates for dispatch.

The process plant steps were set out for the given site conditions, and the required infrastructure and services were added. The plant layout was constrained by a series of factors:

- The conveyor lengths required to reach the required heights of their respective discharge points
- Ground conditions.
  - The layout was then optimized in respect of the following:
  - To reduce piping distances; and to position unit process steps, taking advantage of the site contours, to minimize the bulk earthworks and terracing required
  - Evaluated for maintenance requirements, and the necessary access provided for all the process equipment.

The support facilities for operations were then integrated into the plant design. The change house facilities are located to the entrance to the concentrator area.

The support functions for workshop, stores, reagent make-up and laboratory are located on the east side of the process area to allow easy and close access to the main plant sections.

The collection and delivery of the products have been optimized in terms of the in-plant road systems and layouts.

Consideration was also made in the plant layout for future expansions and increases in the Pulacayo plant production. Consequently, these planned future plant expansions can be constructed without interference to the operations, except for final plant tie-ins and hot commissioning.

17.1.13 Control Philosophy

17.1.13.1 Process Plant Control Philosophy

Standard process control equipment such as flow meters, level detectors, density gauges will provide important information on performance of the process plant. The mill will have a variable speed drive for start-up and speed adjustment and the float section pumps will also have variable speed drives to control the flow through the section. The paste thickener will be furnished with a dedicated PLC.

17.1.13.2 Control Systems

For all instruments located in the process plant, control system will be locally, there is no central control system. Main equipment such as cone crusher, jaw crusher, ball mills and the
thickener have its own control system (vendor package) and shall have the ability to be integrated with other systems, facilitating the operation thereof.

18 INFRASTRUCTURE

18.1 Site Roads

18.1.1 Site Access Road

Arica or Antofagasta ports in Chile are the preferred locations to receive equipment imported from outside South America and from there by road to Uyuni and Pulacayo.

The road from Potosi to Uyuni was paved recently during 2012 and will be used for local supplies coming from that area of the country. Uyuni and Pulacayo are connected by unpaved, good quality road.

Equipment coming from Brazil will arrive in Puerto Suarez and be transported to the mine through Santa Cruz (975 km), then to Oruro (471 km) and from Oruro to Pulacayo (210 km). The same road from Santa Cruz will be used to bring equipment, materials and supplies from the east of the country.

The road coming from the capital city, La Paz, passing through Oruro and Santiago de Huari is being improved by the government and is the main route to transport goods from the capital and Oruro, to Uyuni, Pulacayo and Potosi.

18.1.2 Plant Site Roads

Apogee has developed site roads from the mine adit (San León Adit) to the primary crusher, access from the plant to the Tailing Storage Facility (TSF) and Fresh/Process Water System and access to the various components of the processing plant.

This access will be surfaced with granular materials. Drainage ditches and culverts will be placed in accordance with the site drainage requirements. Access road from Pulacayo to the camp site has not been developed as yet.

18.2 Power Supply

A significant operating cost for the Pulacayo Project will be the power required to run the operations. Permanent, on-site diesel generation is not an option due to diesel cost in Bolivia and availability.

EPCM Consultants, Bolivian local consultant was appointed to develop the power supply assessment for the mining and process operations. The most suitable option to get power to the Pulacayo project is the construction of a 115 kV power line, connecting to the existing Punutuma-Atocha 115 kV line, implementing a switching substation in Tazna town, 61 km from Pulacayo and a transformer substation close to the operation to get 25 kV. The route for the line is defined. The definition of the route allowed the company to start the permitting
process for the construction of the power line with the Bolivian authorities. The request for categorization has already been filed to Government.
Figure 18.1: 115 kV Power Line Trazo Tazna-Pulacayo
18.3 Power Distribution

A 115 kV overhead power line will be taken from the national grid to connect the plant substation. This substation 115/25 kV will deliver three lines in 25 kV, two of them will be of exclusive use of the project.

The first line will give power to the process plant and will service a number of step-down and distribution transformers prior to distribution to particular MCC's.

The process building and power system modules will generally include outdoor oil-filled transformers, motor control centers (MCC’s), dry-type transformers, three-phase distribution panels and local control devices.

The process and plant site ancillary facilities switchgear and electrical equipment will be installed in modular electrical rooms adjacent, near or within the respective buildings where economically feasible.

Motor control centers will be complete with motor starters, contactors, disconnect switches, transformers, panels, circuit breakers and fuses. All electrical distribution will be in cable trays using copper coated cables. The Pulacayo Project will purchase its electrical power from Empresa Nacional de Electricidad de Bolivia (ENDE).

The second 24.9kV line will be dedicated and will bring power to the underground mine services and equipment. A substation in surface, in the proximities of the mining activities will be also required for 25/0.38kV transformation.

The third derivation will be available to service power to Coseu, Cooperative and Pulacayo town, and it is not part of the scope of this project.
Figure 18.2 Power Distribution
18.4 Site Buildings

The process facility and ancillary buildings include the following:

18.4.1 Process Plant

- Receiving Crushing and Screening
- Milling
- Lead Flotation
- Zinc Flotation
- Separation Solid/Liquid
- Filtration of Concentrates
- Reagents
- Process Water
- Auxiliary Services
- Tailings Storage Facility Area
- Backfill Plant

18.4.2 General Site Development

- Main Access Road
- Main Tension (MT) Electrical Supply
- Main Low Tension (LT) Sub station
- Waste Dump Area
- Railway
- Camp

18.4.3 Process Plant Site Development

- Internal Plant Roads
- Plant Spillage Containment Dam
- Electrical Generators
- Septic Tanks
- Process Plant Access Gate
- Entrance First Aid Facilities
- Visitors Parking
- Internal Plant Parking
18.4.4 Mine Infrastructure and Ancillary Buildings

- Magazine
- Warehouse
- Yard Storage
- Truck Wash
- Truck Fuel Storage and Fueling Station
- Tire Change Station

At the end of the operating life of the processing plant, with the exception of those components to be utilized in the post-closure water treatment system, the plant would be decommissioned and demolished. A small amount of infrastructure would be left in place for post-closure use, including conversion of portions of the plant for water treatment. The main power line and ancillary equipment utilized during operations would be removed and the cable salvaged. Once the off-site man camp is no longer required, the camp would be converted to an alternative beneficial use by a future custodian or local community representative.

18.5 Water Storage and Distribution

Klohn Crippen Berger (KCB) was requested to develop the Surface Water Feasibility Assessment for Pulacayo Project. KCB visited the site in May 2012. Fundacion Medmin, a Bolivian consulting
company was appointed to supply data collection to KCB. TWP Sudamérica delivered the water balance to be considered in modeling the water supply. KCB conclusions are as follows:

Water balance calculations indicate that while the Yanapollera reservoir should be able to satisfy demands of the mine in addition to users from Pulacayo and Uyuni under average conditions, the extreme climatic conditions of the region result in a statistically significant likelihood that the mine will experience water deficits during the projected 12-year mine life. Seasonal variation inflows combined with the limited storage capacity of this reservoir is a significant contributor to this potential lack of water – the reservoir exhibits a significant annual surplus which is largely discharged either in Pulacayo or downstream of the impoundment. Constructing additional water storage and supplementing mine water supply with groundwater is expected to significantly improve water security. Sensitivity analyses indicate that the results presented in this report are sensitive to a number of variables; most notably the water demand of Uyuni and Pulacayo and the behavior of groundwater springs in the wetlands upstream of Yanapollera. Further calibration of data inputs and validation of model outputs is required for reliable model predictions. Due to the variability of climate conditions and the inherent uncertainties of this predictive analysis, KCB recommends that an integrated system including groundwater supply and water storage be considered to reduce the chance of temporary mine shutdown due to water shortage. KCB recommends that a groundwater supply investigation be undertaken, in addition to on-going monitoring of site surface water flows and local and regional climate data, in order to improve the level of confidence of subsequent water supply analyses.

Apogee requested KCB to develop two complementary studies to reinforce water supply: Design of reservoir to increase water storage capacity, and a Shallow Groundwater Supply Investigation. Both, at the time of writing this report are being developed. The intention to develop these two studies is to address the best option suggested by the consultant which is a combined one, increasing water storage capacity and sourcing from somewhere other than the Yanapollera reservoir.

18.5.1 Surface Water Management

Surface water runoff that has not come into contact with tailings ("non-contact" water) shall be captured by perimeter diversion channels and routed around the dry stack to discharge to the natural drainage down gradient of the facility. Efforts should be made to minimize water run-on to the TSF. Since the TSF footprint will increase over the operations lifetime, construction of temporary diversion channels for different expansion phases of the TSF will be required.

A fresh-water pipeline will deliver the water to the project Raw Water/Fire Water Tank, located at the Project site at an elevation of approximately 4,500 m. Any water in excess of the plant raw water requirements would be directed to the Process Water Tank located adjacent to the process facilities.

18.5.2 Firewater Distribution

A fresh water pipeline will deliver fresh water by gravity for firefighting services.

Individual hand-held fire extinguishers will also be located throughout the offices and work areas.
The administration building, security office, located close to the main access road and entrance to the mine, will be provided with hand-held fire extinguishers. These extinguishers will be distributed throughout the offices and common areas in accordance with Bolivian Fire Codes.

18.5.3 Potable Water Distribution

Potable water will be supplied to the ancillary facilities at the plant site. Fresh water will be treated and stored in a potable water self-contained treatment plant adjacent to the process water tanks. Potable water treatment will consist of carbon filtering, hypochlorate addition system (inside a container) and ultraviolet (UV) disinfection.

18.6 Sewage Collection

Sewage collection at the plant and mine site, will be comprised of buried gravity septic tanks (with carbon filters). There will be no surface disposal to the environment.

18.7 Communications

There is a basic communication system in the project office in Pulacayo. Cell phone communication is the key instrument for surface communication. A radio system is also used by the geology team. Internet service depends on a wireless system that is obtained from a phone and data repeater in Pulacayo town and distributed through the offices.

The services will be improved for the construction stage with a base radio. The laboratory will also be served with internet and radio signal.

All vehicles will be equipped with radios and essential personnel will have hand held radios. Key personnel will also be equipped with mobile telephones. Cellular phones will have coverage to Pulacayo as a safety precaution.

18.8 Fuel and Lubricant Storage and Distribution

Diesel will be delivered to mine by rail tankers (3 m3) and will be off-loaded into the UG fuel distribution tank from where it will be pumped to various areas of use on the mine.

Plant Diesel fuel and oil distribution will be limited to loading and unloading facilities and metering equipment at the diesel fuel tank at the plant site.

Hydraulic and engine oils will be delivered into the mine by suitable utility vehicles.

Lubricants will be delivered to the site in drums and stored in a secure area.

18.9 Tailings Disposal Facility

18.9.1 Tailing Management Background

The final flotation tailings will be filtered and dewatered in a paste plant (see Section 17.1.7) to between 63% and 70% solids by weight. Part of the paste will be pumped to and distributed within the underground workings (see Section 17.1.7) and the balance will be pumped to and distributed within the tailings storage facility (TSF). The TSF will be located in a gentle, natural value immediately down gradient of the paste plant at a nominal elevation of 4,030 m. The
pumps, piping and electrical systems are included in the paste plant and discussed in that section of this report as well as the related costs reported in those cost areas. The tailings production by year and the distribution between underground backfilling and discharge to the TSF are summarized in Table 18.1. The allowance for underground disposal has used a series of conservative assumptions and, as such, the quantity going to the TSF (and the resulting size of the TSF) should be somewhat overstated. This approach ensures that there will always be adequate TSF capacity and that the TSF operating cost estimates are conservative. On the other hand, the basin has ample room for more tailings should additional capacity be needed.

**Table 18.1: Tailings to the TSF by Year**

<table>
<thead>
<tr>
<th>Year</th>
<th>Annual Production (x1,000 t)</th>
<th>Cumulative Production (x1,000 t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>2</td>
<td>43</td>
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</tr>
<tr>
<td>3</td>
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<td>5</td>
<td>118</td>
<td>247</td>
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<tr>
<td>6</td>
<td>152</td>
<td>400</td>
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<tr>
<td>7</td>
<td>144</td>
<td>543</td>
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<td>8</td>
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<td>593</td>
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<td>9</td>
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<td>11</td>
<td>81</td>
<td>806</td>
</tr>
<tr>
<td>12</td>
<td>360</td>
<td>1,166</td>
</tr>
<tr>
<td>13</td>
<td>180</td>
<td>1,346</td>
</tr>
</tbody>
</table>

**18.9.2 TSF Design Concepts**

The TSF will be developed by first constructing four small starter dikes, one upstream of the other and within the same drainage. Construction of the starter facility will start before commissioning of the paste plant, and will continue concurrently with the paste plant during Year 1. This is to allow placement of cement-treated tailings along the valley floor to seal the basin. The facility will be expanded annually during the dry season by raising two dikes each year by 4 m. Below the first dike (Dike 1) will be a small emergency pond and monitoring ports for the subdrain system. The emergency pond is to manage surface waters that enter the TSF and has a working capacity of 1,750 m$^3$. The subdrain monitoring ports are to collect the discharge from the subdrains (to allow testing and return to the TSF or emergency pond, as needed) with a working capacity of 15 m$^3$ Figure 18.2 shows the ultimate general arrangement of the facility.

The tailings handling circuit, from the discharge of the paste plant to the TSF, will have the following equipment:
- Paste pump,
- Cement silo (this will be the same silo as used for underground distribution),
- Pipeline to the TSF,
- Distribution pipes and discharges points within the TSF,
- Control valves and meters,
- Subdrain pipes to collect consolidation water from the tailings, and
- Pumps and pipes to reclaim the water from the two ponds.
Figure 18.3 General Arrangement of TSF Dikes, Emergency Pond and Monitoring Ports
18.9.3 TSF Design Criteria

Table 18.2 summarizes the design criteria for the TSF along with the source of each value. This criteria was developed from industry standards, requirements to meet the plant criteria (such as operating life and capacity), calculations and modeling of the paste plant, data developed by the owner’s team, and the design engineers’ recommendations and experience.

<table>
<thead>
<tr>
<th>Item Description</th>
<th>Units</th>
<th>Value</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tailings production</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total</td>
<td>t</td>
<td>3,519,000</td>
<td>A</td>
</tr>
<tr>
<td>Total to TSF</td>
<td>t</td>
<td>1,346,000</td>
<td>A</td>
</tr>
<tr>
<td></td>
<td>m³</td>
<td>1,121,700</td>
<td>D</td>
</tr>
<tr>
<td>Nominal Minimum average daily discharge to TSF (Yr 4)</td>
<td>t/d</td>
<td>83</td>
<td>A</td>
</tr>
<tr>
<td>Maximum average daily discharge to TSF (Yr 12)</td>
<td>t/d</td>
<td>986</td>
<td>A</td>
</tr>
<tr>
<td>Density (terminal bulk specific gravity, dry wt basis)</td>
<td>t/m³</td>
<td>1.2</td>
<td>D</td>
</tr>
<tr>
<td>Water content (total wt basis)</td>
<td>%</td>
<td>30 - 37</td>
<td>D</td>
</tr>
<tr>
<td>As discharged</td>
<td>%</td>
<td>33</td>
<td>C, D</td>
</tr>
<tr>
<td>Long-term (as retained, average)</td>
<td>%</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Project life</td>
<td>years</td>
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<td>A</td>
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<tr>
<td>Design Rainfall</td>
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<td></td>
<td></td>
</tr>
<tr>
<td>Average annual rainfall</td>
<td>mm</td>
<td>Average of record</td>
<td>B</td>
</tr>
<tr>
<td>Maximum annual rainfall</td>
<td>mm</td>
<td>Maximum on record</td>
<td>B</td>
</tr>
<tr>
<td>Peak storm event, diversion works</td>
<td>mm</td>
<td>50 - year, 24 - hour</td>
<td>B</td>
</tr>
<tr>
<td>Peak storm event, impoundments &amp; ponds</td>
<td>mm</td>
<td>100 - year, 24 - hour</td>
<td>B</td>
</tr>
<tr>
<td>Impoundment design (closure)</td>
<td>mm</td>
<td>50% of PMP</td>
<td>B</td>
</tr>
<tr>
<td>Slope stability</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Minimum FoS, static (short term)</td>
<td>-</td>
<td>1.3</td>
<td>B, C</td>
</tr>
<tr>
<td>Minimum FoS, static (long term)</td>
<td>-</td>
<td>1.5</td>
<td>B, C</td>
</tr>
<tr>
<td>Minimum FoS, pseudo-static</td>
<td>-</td>
<td>1</td>
<td>B, C</td>
</tr>
<tr>
<td>Design seismic event</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Return interval (operating life)</td>
<td>years</td>
<td>475</td>
<td>B, C</td>
</tr>
<tr>
<td>Design seismic acceleration</td>
<td>g</td>
<td>0.20</td>
<td>D</td>
</tr>
<tr>
<td>Maximum allowable seismic-induced displacement</td>
<td>m</td>
<td>1</td>
<td>B, C</td>
</tr>
<tr>
<td>Containment systems</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Tailings impoundment</td>
<td></td>
<td>Geologic &amp; cement - treated tailings</td>
<td>B, D, C</td>
</tr>
<tr>
<td>Emergency pond</td>
<td></td>
<td>Geomembrane</td>
<td>B, C</td>
</tr>
<tr>
<td>Capping system for TSF</td>
<td></td>
<td>Cement - treated</td>
<td>B, C</td>
</tr>
</tbody>
</table>
### Item Description

<table>
<thead>
<tr>
<th>Item Description</th>
<th>Units</th>
<th>Value</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>tailings, clay &amp; organic vegetative layers</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

A: Provided by owner  
B: Standard industry practice  
C: Provided by consultant (see the Anddes design report)  
D: Criteria based on calculations and analysis

#### 18.9.4 Design and operational Components

During normal operations, tailings will be pumped from the paste plant, without cement addition, and distributed around the impoundment using conventional moveable pipes and discharge points (pipe “tees” or, if practical, spigots).

Annually, usually two dikes will be raised each year. The four starter dikes will be constructed with locally available soils (Dikes 1 & 2 with structural fill, Dikes 3 & 4 with random or mass fill).

Each raise will be constructed using paste tailings with Portland cement added at 3.0% by dry weight; other binders such as pozzolanic fly ash or polymer amendments may be used, subject to testing, to reduce the cement demand or improve performance. Each raise will be nominally 4 meters of elevation gain such that the rate of rise is maintained at a very low rate (4 m per year in the early years, then declining as the working impoundment area increases), allowing the tailings to dry and consolidate and avoiding the development of excess pore pressures.

Dike raises will use modified upstream methods, as shown in the following figure. The area between the two downstream dikes (Dikes 1 & 2) will be filled with cement-treated tailings to provide robust global and post-closure stability and erosion protection.

The tailings behind the other dikes will not generally contain cement, except for the first layer to seal the basin. The total quantity of tailings to be treated with cement is 584,300 t, or about 36% of the total tailings to be deposited in the TSF.
Figure 18.4 Cross-Section of Dikes 1 & 2 with Raises
18.9.5 Foundation Preparation

The site is overlain by a very thin layer of topsoil, which will be removed from the dike foundations as the first step in construction (topsoil within the impoundment and outside of the dike foundations may be either removed or left in place at the owner’s election. The stripped topsoil will be stockpiled for use in closure and reclamation. The topsoil layer is underlain by alluvial soil in and near the bottom of the valley, and residual soils up slope. There are a few areas where moraine deposits remain, but these are limited in area and generally thin. The entire site is underlain by a sandstone basement rock.

The alluvial soils have been placed by the seasonal flow within the tailings valley, and (more importantly) the seasonal flow of the adjacent Rio Negro, which has its confluence with the tailings valley just downstream of the facility. Since the tailings valley is very near the top of the watershed, with a very small catchment area (about 45 ha), the extent and depth of the alluvium is very limited within the TSF area.

The alluvium consists of well-graded gravels and sands, generally dense. The residual soils are those that have weathered in situ from the basement rock and are dense to very dense. These soils are poorly graded silty gravels and sands with cobbles. The glacial moraines are generally composed of silty gravels with some cobbles, and are very dense. The basement rock consists of principally of competent, medium grained sandstone. The surface of the basement rock is lightly weathered and fractured to a shallow depth, and it gains competency with increasing depth.

Except for the thin layer of topsoil, the site soils are considered suitable for use as structural fill (Dikes 1 & 2), random or mass fill (Dikes 3 & 4) and as foundations for all of the dikes. Some removal (less than 3,000 m$^3$) of inadequate soils will be required beneath the first dike, emergency pond and subdrain monitoring ports area. This soil will be stockpiled with the topsoil.

18.9.6 Foundation Underdrains

A system of underdrains or subdrains will be installed to capture shallow subsurface water and seepage from the tailings. This system will drain the entire TSF basin and will also act to monitor and provide an early indication if there is detrimental impacts to water quality from the TSF. This subdrainage system will consist of a network of perforated double wall polyethylene pipes (generally 100 mm and 300 mm diameter), which will be installed in trapezoidal ditches nominally 700 mm deep and filled with drainage gravel (conventionally called French drains). These French drains will slope at a minimum of 2% and the flow will be collected in the small monitoring ports located below the emergency pond.

The preliminary location of the drainage network is shown on the design drawings (by Anddes), but the final locations will be determined and fit in the field based on the observed ground conditions at the time of construction. That is, the network may be adjusted or expanded to accommodate actual as-built conditions. The discharge from the subdrains is expected to range from negligible to a peak of between 1.0 and 2.2 m$^3$/h (based on the peak storm event).

18.9.7 Containment System

The facility is located in a tight, closed basin underlain by shallow, tight sandstone. This basin drains to a single outlet just below the TSF. The tailings will be discharged at high solids contents
(up to 70% by total weight), and the TSF in located in an arid climate (very low precipitation and high evaporation, with a very high ratio of evaporation to precipitation); thus, the amount of water available is limited. As such, it was determined that an engineered liner system such as a geomembrane is not required for the TSF impoundment and that natural, geologic containment is sufficient to properly protect the environment, both to Bolivian and International standards. However, an additional barrier will be provided; cement - treated tailings (3%) will be placed as a continuous cover over the entire base of the TSF area. This will perform better than the average compacted clay barrier and when combined with the natural containment should provide excellent containment.

The emergency pond, located immediately below the toe of the lowest TSF dike, will be lined with 1.5 mm thick high density polyethylene (HDPE) over and in direct contact with at geosynthetic clay liner (GCL). The pond is being lined because it can develop important hydraulic heads, which will be focused just below the toe of Dike 1, and therefore a very high level of containment is warranted.

18.9.8 Starter Dikes & Dikes Raises

The TSF includes four separate dikes, each supporting the toe of the corresponding deposit. These are located sequentially upstream from one another, within the same small valley and numbered Dike 1 to Dike 4, from the lowest elevation working up gradient. The starter dikes range in nominal height from 6 to 8 meters. The reason for four dikes are:

1) To allow initial filling of the valley floor,
2) To allow placement of cement-treated tailings (3%) along the entire bottom area, to act as a seepage barrier,
3) To provide sufficient working area for the placement of cement-treated tailings to be used in the construction of the subsequent raises to each dike,
4) To create a system of paddocks which will allow seasonal drying of the tailings, by moving the discharge from one pond to the other at intervals to be determine based on the season and nature of the produced tailings, and
5) To provide sufficient storage capacity in the early years, while the storage is principally in the bottom of the valley (that is, very little capacity per meter of elevation gain).

As the TSF is filled with tailings the four separate impoundments will merge into one larger impoundment, as depicted in Figure 18.5.

All four of the starter dikes will be constructed as part of the initial facility, and the four impoundments filled with cement-treated tailings during the first 12 months of operation. Starter Dikes 1 & 2 will be constructed with high quality, compacted structural fill; dikes 3 & 4 will be constructed with random fill since they will be quickly supported by the downstream tailings and do not form part of any permanent slope. Each dike will be raised about 4 meters at a time using compacted, cement-treated tailings (3% Portland cement by dry weight). The volume created between Dikes 1 & 2 (that is, the impoundment behind Dike 1) will be filled with non-compacted cement-treated tailings for its entire depth (to the ultimate crest). The other 3 impoundments will be filled with non-cement-treated tailings. As the crest elevation increases and the four separate
impoundments merge into one larger impoundment, only the lower two dikes (Dikes 1 & 2) will be raised. The detailed sequencing is shown in the design report (Anddes).
Figure 18.5 Ultimate TSF Plan Vie
18.9.9 Progressive Slope Reclamation

The working face of the tailings as deposited will be sloped at a relatively gentle 3 horizontal to 1 vertical overall (toe-to-crest), which will reduce the propensity for erosion and allow faster establishment of native vegetation. Horizontal benches will be provided at each 4 m raise in the dike, which will help limit erosion during operations. Progressive reclamation will begin in about the third year of operations, once sufficient final slope area has been established.

The final slope may require some re-contouring, depending on how well operations manages the slope geometry and the degree of differential settlement that results. The current closure concept includes light grading of the ultimate constructed slope of the TSF to result in 3 benches spaced 12 m to 16 m vertically and of nominally 6 m to 8 m in width. The bench face slopes will be nominally 2.5 horizontal to 1 vertical, and the overall slope (toe to crest) will remain 3:1. Native species will be used to provide the highest long-term survival rate and in the early years test plots will be established to find the best combination of plants and methods of planting and starting. The first plantings will occur on the flat benches, where it is easier to place vegetative soils and establish the first growths. This, then, will aid in establishing growth on the slopes by both providing annual seeds and further limiting erosion.

The closure design criteria include more robust storm event (the PMP) and seismic standards (maximum probable event, MPE). These will require some upgrades to the surface water diversions. Since the impoundment will be dry in the long term, and the tailings fully consolidated, long-term stability will be substantially better than that during the operating life, even considering the more robust standards. The diversion works will either be upgraded to a 500-year return interval (still a relatively small event, given the small the drainage basin area) or the impoundment will be contoured to safely store this event (depending on the final configuration of the tailings, which will depend on how closely they are managed relative to the design, the final production volume, and the amount of consolidation).

The final layer of tailings, nominally 1 m thick, will also contain 3% cement to provide a firm, low permeability cap below the growth medium. If this cap is not continuous, or if the actual physical properties are determined to be less than idea for a cap, then an additional layer of low permeability soil will be spread and compacted to a thickness of 300 mm. This will then be capped with a 300 mm thick layer of organic soils (topsoil) recovered during construction to provide a growth media. Since organic soils are sparse and the recovered topsoil volume will probably not be adequate for this closure demand, non-organic soils will be upgraded with nutrients and then mixed with the natural topsoil.

18.9.10 Surface Water Management

Meteorological data was obtained from the national station located in Uyuni, 13 km from the site and at an elevation of 3,680 m (the TSF average elevation is 4,030 m). The database included the following years: 1943 to 1948 and 1975 to 2011. This data was then adjusted for elevation to estimate the climate at the site. This produced the following estimates of rainfall and evaporation for the TSF site (see Table 18.3 & Table 18.4). Two important results of this are: 91% of the rainfall occurs between December and March (and thus no dike construction will occur in this time period); and the average ratio of evaporation to rainfall is 8.5 : 1.
Table 18.3: Precipitation (mm)

<table>
<thead>
<tr>
<th>Month</th>
<th>Average</th>
<th>Maximum</th>
<th>Minimum</th>
<th>% of Annual</th>
</tr>
</thead>
<tbody>
<tr>
<td>January</td>
<td>78</td>
<td>271.6</td>
<td>3.1</td>
<td>39</td>
</tr>
<tr>
<td>February</td>
<td>44.1</td>
<td>149.1</td>
<td>0</td>
<td>22</td>
</tr>
<tr>
<td>March</td>
<td>28.1</td>
<td>137</td>
<td>0</td>
<td>14</td>
</tr>
<tr>
<td>April</td>
<td>2.1</td>
<td>14.3</td>
<td>0</td>
<td>1</td>
</tr>
<tr>
<td>May</td>
<td>1.3</td>
<td>9.6</td>
<td>0</td>
<td>1</td>
</tr>
<tr>
<td>June</td>
<td>1.8</td>
<td>39</td>
<td>0</td>
<td>1</td>
</tr>
<tr>
<td>July</td>
<td>0.3</td>
<td>13.2</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>August</td>
<td>1.9</td>
<td>20.6</td>
<td>0</td>
<td>1</td>
</tr>
<tr>
<td>September</td>
<td>2.5</td>
<td>25.4</td>
<td>0</td>
<td>1</td>
</tr>
<tr>
<td>October</td>
<td>2.2</td>
<td>36.4</td>
<td>0</td>
<td>1</td>
</tr>
<tr>
<td>November</td>
<td>5.7</td>
<td>45</td>
<td>0</td>
<td>3</td>
</tr>
<tr>
<td>December</td>
<td>31.7</td>
<td>104.7</td>
<td>0</td>
<td>16</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>199.8</strong></td>
<td><strong>465.8</strong></td>
<td><strong>33.6</strong></td>
<td><strong>100</strong></td>
</tr>
</tbody>
</table>

Table 18.4: Evaporation (mm) at the TSF

<table>
<thead>
<tr>
<th>Month</th>
<th>Average</th>
<th>Maximum</th>
<th>Minimum</th>
<th>% of Annual</th>
</tr>
</thead>
<tbody>
<tr>
<td>January</td>
<td>163.9</td>
<td>223.5</td>
<td>107.5</td>
<td>10</td>
</tr>
<tr>
<td>February</td>
<td>139.4</td>
<td>221.2</td>
<td>104.8</td>
<td>8</td>
</tr>
<tr>
<td>March</td>
<td>148</td>
<td>190.6</td>
<td>99.1</td>
<td>9</td>
</tr>
<tr>
<td>April</td>
<td>142.1</td>
<td>185.4</td>
<td>103.8</td>
<td>8</td>
</tr>
<tr>
<td>May</td>
<td>102</td>
<td>128.7</td>
<td>74.8</td>
<td>6</td>
</tr>
<tr>
<td>June</td>
<td>90.8</td>
<td>107.8</td>
<td>83.3</td>
<td>5</td>
</tr>
<tr>
<td>July</td>
<td>85.1</td>
<td>104.6</td>
<td>63.8</td>
<td>5</td>
</tr>
<tr>
<td>August</td>
<td>111.9</td>
<td>160.8</td>
<td>78.5</td>
<td>7</td>
</tr>
<tr>
<td>September</td>
<td>148.5</td>
<td>206.3</td>
<td>90.4</td>
<td>9</td>
</tr>
<tr>
<td>October</td>
<td>197.7</td>
<td>221.5</td>
<td>152.6</td>
<td>12</td>
</tr>
<tr>
<td>November</td>
<td>185.2</td>
<td>232.8</td>
<td>127.1</td>
<td>11</td>
</tr>
<tr>
<td>December</td>
<td>185.2</td>
<td>251.9</td>
<td>130</td>
<td>11</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>1700</strong></td>
<td><strong>2195.6</strong></td>
<td><strong>1355.2</strong></td>
<td><strong>100</strong></td>
</tr>
</tbody>
</table>

Surface run-off will be diverted away from the TSF by a series of three diversion ditches, identified as West, East and South. Because the drainage basin is very small (45 ha) and the peak storm events are small, the size of the ditches and the predicted peak flows are both modest. The ditches were designed to safely divert up to the 50-year, 24-hour storm. Larger events will breach the ditches and be contained within the TSF or routed to the emergency pond.
During the operating life of the TSF, the TSF and emergency pond are designed to safely contain the 100-year event. The closure design will provide for one-half of the probable maximum precipitation (PMP).

All three of the diversion ditches will be constructed without armouring, as the drainage areas are small and the predicted flow velocities low; further, the soils are dense and gravely and thus erosion resistant. The cross sections will be trapezoidal with 1:1 slopes Table 18.5.

### Table 18.5: Diversion Ditch Design Summary

<table>
<thead>
<tr>
<th>Diversion Ditch</th>
<th>Base Width (m)</th>
<th>Depth (m)</th>
<th>Drainage Area (Ha)</th>
</tr>
</thead>
<tbody>
<tr>
<td>East</td>
<td>0.6</td>
<td>0.6</td>
<td>2.9</td>
</tr>
<tr>
<td>West</td>
<td>1.2</td>
<td>1.2</td>
<td>41</td>
</tr>
<tr>
<td>South</td>
<td>0.6</td>
<td>0.6</td>
<td>1.2</td>
</tr>
</tbody>
</table>

18.9.11 Management of TSF Effluent

The TSF will be a net consumer of water, given the low water content of the paste tailings and the low rainfall and very high ratio of evaporation to rainfall. However, during rain events some water will be produced. This water will be subjected to routine testing for quality; if it contains constituents above the permitted discharge standards then it will be returned to the plant for use in the processing circuit, or recycled to the TSF. Otherwise, it will be discharged to the Rio Negro unless needed by the plant.

An analysis of the water balance within the TSF has been performed using the average rainfall and evaporation conditions, with allowance for the 100-year, 24-hour storm event, along with the properties of the tailings; the results are summarized in the following table. The emergency pond provides a residence time of 195 to 215 hours (8 to 9 days) during the 100-year event; when that event is increased to 50% of the PMP, the residence time is reduced to 104 hours (4 days). The residence time is the time to fill the ponds, during which the operators must test the water, decide to where it will be discharged, and then set the discharge accordingly. Four days is generally considered ample time to take such actions; many operating mines have much shorter action times, often in the 1-day or less range.
Table 18.6: TSF Water Balance Results with 100-Year, 24 Hour Storm Event

<table>
<thead>
<tr>
<th>Month</th>
<th>Subdrain Monitoring Ports</th>
<th>Emergency Pond</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Flow (m³/h)</td>
<td>Residence Time (h)</td>
</tr>
<tr>
<td>January</td>
<td>2.2</td>
<td>7</td>
</tr>
<tr>
<td>February</td>
<td>1.9</td>
<td>8</td>
</tr>
<tr>
<td>March</td>
<td>1.4</td>
<td>10</td>
</tr>
<tr>
<td>April</td>
<td>1.1</td>
<td>14</td>
</tr>
<tr>
<td>May</td>
<td>1</td>
<td>15</td>
</tr>
<tr>
<td>June</td>
<td>1.1</td>
<td>14</td>
</tr>
<tr>
<td>July</td>
<td>1</td>
<td>15</td>
</tr>
<tr>
<td>August</td>
<td>1</td>
<td>15</td>
</tr>
<tr>
<td>September</td>
<td>1.1</td>
<td>14</td>
</tr>
<tr>
<td>October</td>
<td>1</td>
<td>14</td>
</tr>
<tr>
<td>November</td>
<td>1.1</td>
<td>13</td>
</tr>
<tr>
<td>December</td>
<td>1.5</td>
<td>10</td>
</tr>
</tbody>
</table>

18.9.12 TSF Stability Evaluation

The dikes and tailings deposit were analyzed for slope stability under static (normal) and pseudo-static (earthquake) conditions. Additionally, the earthquake-induced deformations were estimated. A seismicity study was completed to determine the design peak ground acceleration (PGA). Based on the available literature and other studies performed in the area, and based on a return interval of 475 years, the design PGA is 0.2 g. This is the value that was used in the deformation analysis. Based on standard convention and the guidelines published by the US Army Corps of Engineers, for the purposes of the pseudo-static analysis only, the PGA was reduced by 50% to 0.1 g. The geotechnical parameters used and the results of these analyses are summarized in the following tables.

The yield acceleration, Ky, is the ground acceleration, which causes initial or incipient yield within the considered mass. When the required acceleration to produce yield (Ky) is greater than the design acceleration to be imparted to the TSF (PGA), the expected deformations are very small. That is the case here, with predicted deformations of under 20 mm. For these types of facilities, it is generally accepted that deformations of under 1.0 m will not seriously affect the performance of the facility. Therefore, the predicted deformations should cause no damage or performance problems within the TSF and will likely be hard even to detect.
### Table 18.7: Summary of Geotechnical Properties used in Satbility and Deformation Analyses

<table>
<thead>
<tr>
<th>Properties</th>
<th>Material</th>
<th>Tailings</th>
<th>Tailings with 3% cement</th>
<th>Structural Fill</th>
<th>Basement Rock</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density, total (kN/m³)</td>
<td></td>
<td>17</td>
<td>18</td>
<td>18</td>
<td>21</td>
</tr>
<tr>
<td>Density, saturated (kN/m³)</td>
<td></td>
<td>18</td>
<td>19</td>
<td>19</td>
<td>22</td>
</tr>
<tr>
<td>Cohesion, effective</td>
<td></td>
<td>0</td>
<td>10</td>
<td>0</td>
<td>80</td>
</tr>
<tr>
<td>Friction angle, effective</td>
<td></td>
<td>25</td>
<td>28</td>
<td>35</td>
<td>30</td>
</tr>
<tr>
<td>Cohesion, total</td>
<td></td>
<td>0</td>
<td>0</td>
<td>na</td>
<td>na</td>
</tr>
<tr>
<td>Friction angle, total</td>
<td></td>
<td>0</td>
<td>17</td>
<td>na</td>
<td>na</td>
</tr>
</tbody>
</table>

Note: na = not applicable to this layer or material

### Table 18.8: Static & Pseudo-Static Factors of Safety for the TSF (Slide 6.0)

<table>
<thead>
<tr>
<th>Section</th>
<th>Type of Failure</th>
<th>Factor of Safety</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Static (0.1 g)</td>
<td>Pseudo-Static (0.1 g)</td>
</tr>
<tr>
<td>Section 3-3 (local)</td>
<td>Circular</td>
<td>1.31</td>
</tr>
<tr>
<td>Section 1-1 (global)</td>
<td>Circular</td>
<td>2.09</td>
</tr>
<tr>
<td>Section 2-2 (global)</td>
<td>Circular</td>
<td>2.05</td>
</tr>
<tr>
<td>Section 3-3 (global)</td>
<td>Circular</td>
<td>1.87</td>
</tr>
<tr>
<td>Section 4-4</td>
<td>Circular</td>
<td>1.96</td>
</tr>
</tbody>
</table>

### Table 18.9: Earthquake-Induced Deformations, PGA= 0.2 g (Bray & Travasarou, 2007)

<table>
<thead>
<tr>
<th>Section</th>
<th>Yield Acceleration, Ky</th>
<th>Permanent Deformation, Average (Range), mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>Section 1-1</td>
<td>0.255</td>
<td>15 (13 to 15)</td>
</tr>
<tr>
<td>Section 2-2</td>
<td>0.277</td>
<td>13 (12 to 15)</td>
</tr>
<tr>
<td>Section 3-3</td>
<td>0.247</td>
<td>17 (15 to 19)</td>
</tr>
<tr>
<td>Section 4-4</td>
<td>0.263</td>
<td>15 (13 to 17)</td>
</tr>
</tbody>
</table>
18.10 Waste Rock Management

There are 2 areas for mine waste disposal for the current activities and will continue being used when operations increase. Both are located south of the railroad on its way to the crushing building. The use of some of the existing voids inside the mine has also been considered.

18.11 Security

The entire plant site will be surrounded by a 2 m high fence. Access to the plant will be restricted to one access at the main gate, which will include a gatehouse manned 24 hours per day.

The main mine gate house and security office will be located near the administration building at the main property entrance. This small building will house security offices, toilet and small reception area. A Bolivian security firm will be contracted to provide on-site security services to the Project starting during construction.

19 MARKETING AND LOGISTICS

19.1 Introduction

At the request of Apogee Silver Ltd. (“Apogee”), Mr A. Falls of Exen Consulting Services was asked to prepare a Marketing and Logistics Study for the zinc and lead concentrates projected to be produced from Apogee’s Pulacayo Project.

Although selected parties were contacted to assess their interest in and/or their ability to treat these concentrates, no terms indications were sought due to the early-stage nature of the project. Instead, NSRs were prepared based on the author’s knowledge of the zinc and lead concentrate markets and metal price and treatment charge assumptions as set out in this report.

Also due to the early stage of the project, only a limited number of service providers were contacted to assess potential concentrate transportation and logistics costs to be used in the derivation of these NSRs. Although information as provided is believed to provide a reasonable estimate of what such costs might be, further work should be undertaken to verify the figures contained in this report once a decision is made to approach the market with these concentrates as several different concentrate distribution options will likely be available, including, but not limited to:

- Direct offshore sale with concentrates either (a) loaded in containers (in bags or bulk) either at the mine site or at an intermediate facility, or (b) shipped in bulk in an ocean going vessel hold, to a buyer or buyers (Apogee manages/oversees all logistics through to final destination).
- Delivery to a warehouse/blending facility in either Chile or Peru where concentrates can be combined with other materials and shipped either (a) in bulk in vessel holds, or (b) in bags or bulk in containers, to smelters (Apogee manages trucking to the warehouse/blending facility; all onward logistics from delivery in-warehouse managed by buyer).
- Delivery to a warehouse/blending facility in either Chile or Peru where concentrates can be combined with other materials and shipped in containers or in bulk to smelters (Apogee manages trucking to the warehouse/blending facility; all onward logistics from delivery in-warehouse managed by buyer)

- Delivery of concentrates ex Works (loaded in trucks or containers) with buyer(s) managing all logistics from site to destination.

The above options will need to be refined based on potential buyer interest and capabilities and Apogee’s appetite for managing the logistics activities. Regardless of which avenue is selected, the net cost to Apogee can be assumed to be approximately the same as if Apogee manages the logistics itself, as buyers taking delivery anywhere along the continuum will look for allowances to cover the cost to deliver the concentrates from that point of delivery to the ultimate market destination, which, for the purposes of this report, is assumed to be Asia.

19.2 Payable Metal Price Assumptions

19.2.1 Market Outlook: Zinc and Lead

Global demand for zinc is forecast to be approximately 13.4 Million tons in 2012, up some 4.4% on 2011, while that for lead is expected to grow by close to 5% to 10.8 Million tons (see Figure 19.1 and Figure 19.2). Despite these relatively strong growth rates, inventory levels remain stubbornly high, with zinc stocks, in particular, estimated to be close to 2 Million tons (producer +consumer +merchants +exchanges) representing some 8 weeks supply. Global lead stocks have also crept up since hitting lows in 2007 to 2008 but, at an estimated 650 kt (total), they still represent just 3 weeks of supply.
As with most metals, China has become the “story” for both zinc and lead demand globally. According to International Lead and Zinc Study Group data, China currently accounts for some 40% and 45% of global annual zinc and lead metal demand, respectively, increasing since 2007, by roughly 50% and 80%, against global demand growth of 13% and 22%, respectively, over this same period. While such levels are not deemed sustainable longer-term, China is still expected to be the main driver to growth in demand for both metals for several years to come.

Although Chinese mine production accounts for a large proportion of domestic supply, China remains a significant importer of both zinc and lead, in both refined and unrefined form. In addition, with its anticipated expanding demand profile, China can be expected to continue to play a major role with respect to future demand.

As we look forward, global usage of both zinc and lead is expected to continue growing strongly. By 2020, for example, CRU forecasts annual zinc metal consumption of close to 18 Million tons, representing approximately 30% growth from current levels.

Source: ILZSG

However, with growth in the developed world expected to remain flat to 2%, most of this expansion is projected to emanate from China where a compound annual growth rate of approximately 5% is forecast over the balance of the decade (see Figure 19.3).
Although CRU projects mine and scrap production to roughly keep pace with demand through to mid-decade, a shortfall in zinc metal supply is anticipated thereafter as rising demand outpaces growth in mine output where production from expansions and new projects (e.g. MacArthur River, Mt Isa, Dugald River, BraceMac, Perkoa, Lalor) will fail to fully offset the combination of demand growth and losses from scheduled mine closures (Perseverance, Brunswick, Lisheen, Century) and attrition elsewhere in the coming years (e.g. Antamina, Zyrano, Golden Grove) (Figure 19.4).
Should such supply shortfalls materialize as projected, stocks could be severely squeezed in the second half of the decade (see Figure 19.5).

Source: CRU
The lead market, on the other hand, is unique in that over half of lead supply is derived from secondary sources (largely from spent batteries, which consume about 80% of refined lead production). With recycle activities effectively a closed loop, an on-going supply of lead from that source can be expected. While primary supply will be required to make up the balance, lead mine production is forecast to remain essentially flat over the coming few years, with lead from new operations and/or expansions largely offset by mine closures and attrition elsewhere. Beyond 2014 however, projections indicate that lead supply – mine and secondary will not keep pace with demand, and annual shortfalls of 50,000 to 100,000 tons are projected (see Figure 19.6).

Figure 19.6: Forecast Lead Supply Vs Demand to 2016

19.2.2 Metal Price Review and Outlook

The dramatic improvement in both zinc and lead prices since the late 1990s/early 2000s can be largely attributed to improved market sentiment associated with the onset of the commodities ‘super-cycle’. With the base metals complex in large part led by copper, where improved supply demand fundamentals driven as much by underinvestment in new mine capacity as they were by favorable global economic conditions which invigorated demand, zinc prices rose five fold from levels below USD 880/mt (USD 0.40/lb) in the early part of the decade to peak at around the USD 4,400/mt (USD 2.00/lb) level in late 2006 when exchange inventories dropped below 100,000
tons (see Figure 19.7). At these historically high prices, numerous zinc mine reactivations occurred which, combined with ever-expanding output in China, rapidly reversed the market imbalances.

Exacerbated by the global financial crisis, zinc inventories on the LME began their relentless increase in 2007 to the point where they now sit above the 1 Million mt mark, a 17 year high and alone representing close to four weeks supply (Figure 19.8).
In the face of rising stocks and slower economic growth, prices have largely retreated from their lofty levels but they continue to be sustained by warehousing finance activities. In addition, although zinc prices dropped below the USD 2,000/mt (USD 0.90/lb) mark in May 2012, they appear to have found support around the USD 1,800/mt (USD 0.80/lb) level.

After peaking about one year after the zinc price highs, lead metal prices have followed a not too dissimilar trend to zinc. With LME lead stocks hovering below 50,000 mt, lead prices reached a high in October 2007 above USD 3,700/mt (approx. USD 1.70/lb), but they too retraced along with the rest of the metals complex, hitting lows in late 2008 under USD 970/mt (USD 0.45/lb) at the height of the global financial crisis. With relatively strong fundamentals however, lead prices rebounded fairly rapidly in 2009, peaking in April 2011 above USD 2,900/mt (USD 1.30/lb) before trending back down with the rest of the metals complex. Although prices have held up relatively well since, they are currently trading around the USD 1,800/mt (USD 0.80/lb) level where support appears to have been established.

Looking forward, support for zinc and lead prices remains sound with most pundits pointing towards limited downside risk as prices trade into the mine cost curve for both metals. However, over time, the outlook for the two metals diverges somewhat due to their respective fundamental outlooks.

19.2.2.1 Zinc Price Outlook

As noted, the zinc metal market remains mired in significant surplus. This oversupply situation is expected to persist at least through 2012 and 2013, with on-going projected surpluses over the next two years forecast to add over 500,000 tons to the 1 Million tons currently in inventory on the
LME alone (and with the SHFE sitting on another 330,000 tons as of 13 July 2012). Although a share of this inventory will likely continue to be used in financing deals, the scope of these deals is less than that for aluminum, the contango offered is lower, and the surplus is larger as a percentage of the market, fundamental features that can be expected to cap any price rallies. As a result, any gains foreseen in zinc prices nearby can largely be expected to be the result of broader macro-economic sentiment, which will drive the entire LME base metals complex.

Zinc consumption has been strong, with global gains of almost 8% in 2011 and growth for 2012 to 2015 projected to average around 5% p.a. mainly due to expansion in galvanizing applications. China will continue to dominate this market, accounting for more than 40% of global consumption going forward, but other emerging markets are expected to support demand as well. In the developed world however, growth is forecast to be flat to marginal.

Despite the zinc metal market being in considerable surplus, zinc mine/smelter demand remains relatively well balanced, with the consequential low concentrate treatment charges adding support to most miners who are still operating in a cash positive environment, and discouraging any production scale back. This relatively low treatment charge environment is expected to persist with, as noted earlier, the impending closures of some major mines in the coming years. And while growth in Chinese mine production has been substantial over the years, most of its mines are small (less than 30 kt/y) and, with diminishing by-product credits and combined zinc-lead grades typically less than 10% (and often below 5%), there may prove to be limits to this growth profile. Nonetheless, China’s smelting capacity continues to expand and, even after accounting for secondary production, the gap between domestic mine supply and primary refined production in China will continue to require significant concentrate imports (expected to account for some 20% of 2012 to 2015 smelter requirements). This is expected to keep the concentrate market in relative balance and hold treatment charges in a tight range around current levels.

With low treatment charges and comparatively high metal prices supportive of continued mine supply, the two “markets” – zinc concentrate and zinc metal – are forecast to remain in divergence until at least 2014 to 2015. By then however, the impact of the various scheduled mine closures is forecast to begin feeding through into tightness in the metal market – an extended period to wait and unfortunately, a similar outlook that has disappointed several times in the past.

Further detailed discussions of the zinc supply-demand projections will not be undertaken for the purposes of this report. Nonetheless, the outlook is based on market fundamentals largely in line with those set out in Table 19.1, views consistent with those shared by many zinc market analysts.
In an effort to attain zinc price projections independent of any individual outlook, zinc price forecasts were drawn from a number of different sources to achieve a consensus view. Detailed data is set out in Appendix I of this report however, to summarize, year by year projected consensus average annual zinc prices through 2016 are as set out in Table 19.2.

The above projections compare favorably against a forward curve (as of 19 July 2012) in contango but relatively flat through to December 2015 (see Figure 19.10) which shows a spot (cash) price of USD 1,883/mt Zn (USD 0.85 per pound) and a forward price of USD 1,968/mt (USD 0.90/lb), well below the projections of the various analysts.
Nonetheless, consistent with the above market outlook, and for the purposes of this report, long-term zinc prices are projected to average USD 2 200 per mt, or approximately USD 1.00/lb.

19.2.2.2 Lead

As with zinc, the lead metal market remains in surplus in 2012 however, medium-term, lead appears to show some of the best fundamentals of any base metal, with strong support for higher prices coming from both the demand and supply sides. On the demand side, with motor vehicles accounting for roughly fifty percent of lead usage, demand will continue to benefit from strong growth in the global automotive industry, particularly in China and, to a somewhat lesser extent, India and other emerging markets. Elsewhere, while European figures are dismal in the face of the ongoing economic crisis there, United States and Japanese auto output has rebounded strongly from the disruption caused by the Japanese earthquake in March 2011. In other applications, backup power batteries, which are benefitting from the growth of data centers and the build-out of electrical grids in China and India, account for a significant portion of the remainder of lead consumption.

On the supply side, 2012 has witnessed several events, which have affected production. In China, a cadmium spill in Hechi in January and lead poisoning cases in three separate provinces have affected production at several mines and smelters, while a fire at Doe Run’s 120 kt/y Herculaneum smelter in the United States saw production disrupted there for nearly 6 weeks in the second quarter. Despite these events, the lead market remains in surplus in 2012, the International Lead-Zinc Study Group recently reported a small 32 kt refined metal surplus to the end of May, but is forecast to move into deficit starting as early as 2013 on the back of closures of major mines (Brunswick: 50 to 70 kt/y lead, closure set for 2013; Lisheen: 20 kt/y, closure set
for 2014; and, Century: 40 to 70 kt/y, closure now set for 2016) and lack of any significant new projects outside of China. Having said that however, this transition to deficit may prove more gradual than some anticipate as China continues to invest in both new lead mines (driven also by relatively attractive zinc and silver prices) and in its currently underdeveloped secondary smelting industry. Furthermore, environmental issues and concerns can be expected to continue limiting the ability to expand both primary and secondary supply, with such challenges no longer confined to developed economies, as China too has experienced considerable environmental scrutiny of its lead industry over the past few years and, as a result, can be expected to implement stricter regulation on projects going forward (particularly with respect to secondary smelters).

Elsewhere, the market continues to watch developments at several other western operations, all of which will have medium term supply side consequences:

- **At Doe Run’s La Oroya smelter in Peru, which was shuttered in early 2009, discussions surrounding the recommencement of operations there were initiated with the Humala government in early 2012 however, several significant issues remain unresolved and a restart date remains elusive. While it is viewed as likely that these issues will ultimately be worked out, this is not expected to happen in the near future, keeping important western smelting capacity and in particular that capable of handling complex, high precious metal concentrates out of the market.**

- **There are ‘on again/off again’ plans to restart Glencore’s 80,000 t/y lead smelter in Port Vesme, Italy, which was placed on care and maintenance in 2009. Although expected to start up earlier this year, this decision has been postponed with the latest rumors focusing on plans to treat complex, precious metal bearing concentrates there. In conjunction with this decision however, Glencore is apparently discussing with Xstrata the future for the latter is 60,000 ton Belledune, New Brunswick lead smelter.**

- **Original plans had Belledune scheduled for closure in tandem with the Brunswick mine shutdown in 2013. However, the smelter was given an unofficial reprieve in 2011 when it was unofficially chosen to treat residues from the future Albion process (residue produced from treatment of Xstrata’s McArthur River bulk concentrates).**

- **More recent plans apparently have Porto Vesme handling the bulk of these residues. Any deferral of the Porto Vesme restart or an advancement/delay of the Belledune closure will significantly impact the lead market, but particularly the market for concentrates.**

- **In the United States, Doe Run announced in late June that it will not be proceeding with the investment in a hydrometallurgical process for treating lead concentrates at its Herculaneum, Missouri operation but will instead shut that facility at year end 2013 and sell concentrates into the custom market, thereby removing the last primary lead smelter in the United States.**

- **On the mine supply side, the low lead content of many of the larger zinc mine projects (e.g. Bracemac, Perkoa, Lalor Lake, Bisha) and the poor economics of developing lead-only mines anywhere outside of China, will restrict growth in primary lead output and will be supportive of lead prices.**
Although further detailed discussions of lead supply-demand projections will not be undertaken for this report, Credit Suisse shows a four-year outlook for lead per Table 19.3, which is consistent with views generally shared by most market analysts.

Table 19.3: Lead Market Outlook (‘000 Mt Contained Pb)

<table>
<thead>
<tr>
<th>Source: Credit Suisse</th>
</tr>
</thead>
</table>

Accordingly, while the market is forecast to remain in a balanced to surplus position in 2012 and 2013, it is forecast to trend towards deficit starting in 2014 and widen thereafter as demand growth continues to outpace supply.

As with zinc, in an effort to attain lead price projections independent of individual views and opinions, lead price forecasts were drawn from different sources to form a consensus outlook.

Detailed data is set out in Appendix II of this report however, year by year projected consensus lead prices can be summarized as set out in Table 19.4 above.
Table 19.4: Lead Price Outlook

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Price, USD</td>
<td>1.09</td>
<td>1.00</td>
<td>1.07</td>
<td>1.08</td>
<td>1.12</td>
<td>1.05</td>
</tr>
</tbody>
</table>

These projections compare with a forward curve (as of 19 July 2012) in contango through to December 2015 which shows a spot price of USD 1,920.50/mt Pb (USD 0.87 per pound) and a forward price of USD 2,060/mt (USD 0.93/lb) which, although higher than zinc prices, still remains well below most analysts’ projections (see Figure 19.11).

Nonetheless, nearby lead prices, as noted above, are forecast to rebound and remain well above the long-term average lead price, which, for the purposes of this report, is projected at USD 1,962 per mt, or approximately USD 0.89/lb.

19.2.2.3 Silver Price Outlook

No detailed analysis of the silver market was undertaken for this study and available analyst projections were deemed to be limited to be considered as representative of a consensus outlook. However, a review of London Bullion Market Association (LBMA) silver prices since January 2009 (see Figure 19.12) shows a 3 year trailing average price of USD 25.44/oz. Accordingly, for purposes of the Feasibility Study, it has been recommended that a silver price of USD 25.00/oz be used.

Source: LME, July 20, 2012
19.2.3 Zinc and Lead Concentrates Terms Outlook

19.2.3.1 Zinc Concentrates

Zinc provides a perfect example of how different segments of a particular market can completely diverge. Although the metal market is mired in surplus, zinc mines are struggling to keep pace with smelter demand for concentrates. The result of this has been a tight concentrate market and low treatment charges for smelters, which may only worsen with the closure of the aforementioned mines in the coming years i.e. Perseverance, Brunswick, Lisheen, Century, Skorpion.

The importance of concentrate supply and the pressure placed on growth in mine capacity is especially great for zinc due to a lack of growth in the secondary smelting sector. Although a large proportion of refined lead and copper, for example, is derived from scrap, zinc is almost entirely dependent on production from primary sources. This is a result of the difficulty and unprofitability of recycling zinc in most of its principal applications. Most notably, while galvanized steel is by far the largest use of zinc, is widely recycled, it is used as steel scrap rather than contributing to secondary zinc production.

While growth in Western World mine and smelter capacity is projected to stagnate, Chinese output will likely be key to determining the extent of market balances in any given year. Although China has invested heavily in zinc mining, most of its mines are small (less than 30 kt/y) and have combined zinc lead grades typically less than 10%. Furthermore, a substantial portion of newly discovered reserves lie in the remoter western provinces. Nevertheless, China continues to expand its zinc smelter capacity the result of which will be an on-going gap between domestic...
mine supply and primary refined production, which will continue to necessitate significant concentrate imports. Combined with steady demand from western refiners, this strong global demand for concentrates can be expected to keep treatment charges in favor of the mines.

2012 zinc benchmark treatment charges were established in February with a settlement reached between Teck and Korea Zinc at a level of USD 191/dmt at a basis price of USD 2,000/t zinc, and escalated/de-escalated above/below this level. This represents a net drop of about USD 18/dmt from the 2011 levels which were set at USD 229/dmt at a USD 2,500/t basis price.

As we look forward, despite the several planned mine closures in the coming years, most analysts are projecting a zinc concentrate market in a deficit to balanced position nearby but with a slight trend towards oversupply towards the middle of the decade. A key risk to this projection is, of course, Chinese production, which has proven to be capable of rapidly delivering into any supply gap particularly when price conditions provide the necessary support:

Table 19.5: Zinc Concentrate Market Projected

<table>
<thead>
<tr>
<th>000 mt cont'd Zn</th>
<th>2008</th>
<th>2009</th>
<th>2010</th>
<th>2011</th>
<th>2012</th>
<th>2013</th>
<th>2014</th>
<th>2015</th>
</tr>
</thead>
<tbody>
<tr>
<td>Concentrate Balance</td>
<td>0.32</td>
<td>0.28</td>
<td>-0.40</td>
<td>-0.13</td>
<td>0.19</td>
<td>-0.10</td>
<td>0.15</td>
<td>0.16</td>
</tr>
</tbody>
</table>

While treatment charges will vary year to year depending on market balances, going forward, consensus appears to be forming around long-term TCs in the range of USD 180 to USD 200 per dmt concentrate at a basis price USD 2,200 per ton Zn (USD 1.00/lb). Although there has been a movement over the years to eliminate the escalators/de-escalators, there does not appear to be enough sustained momentum behind this movement to suggest they will be eliminated any time soon.

Having trended down to the current levels as agreed in 2012, based on the above balance projections, annual treatment charges are expected to trend within a relatively narrow band over the coming years. Accordingly, it is recommended that the following benchmark charges be used for the purposes of the Feasibility Study:

- Treatment Charge USD 190 per dmt concentrate at a basis price USD 2,200 per ton Zn
- TC Escalator/De-escalator +/-USD 0.03 per dmt for each USD 1.00/mt Zn price above/below USD 2,200/mt

19.2.3.2 Lead Concentrates

Unlike the zinc and copper markets, where treatment and refining charges (both spot and benchmark) are generally consistent across a range of qualities at any given point in time, lead concentrate terms are considerably more variable, a factor which has led to a much less consistent and transparent market. In part this can be attributed to a wider range of qualities in the lead concentrate market but, more broadly, it can be attributed to the smaller overall size of the market and, in particular, the influence of China which currently accounts for approximately 45% of global refined lead production and which has become the single largest buyer of custom lead concentrates.
Historically, quality differentiation in the lead concentrate market was largely addressed by the smelters through the imposition of penalties (as it continues to be with zinc and copper) and, occasionally, adjustments to payable metals. China’s expanded influence however, has had a notable effect on the market, particularly for high silver-bearing lead concentrates, which are not well suited to Chinese smelters for the following reasons:

- 17% VAT on silver bearing lead concentrate imports
- No VAT rebate on silver metal exports
- Negative London/Shanghai silver price differential

Combined, these three factors have resulted in a negative arbitrage in the silver price, which is currently in the order of USD 4 to 5/oz.

As a consequence of changing dynamics, today’s lead concentrate market can effectively be segregated into three sub-categories:

1) Clean, high grade (+/-60% Pb) concentrates (e.g. Doe Run/Missouri)
2) Medium-grade (+/- 50% Pb), complex concentrates (e.g. Red Dog)
3) Complex, high precious metal bearing concentrates (e.g. Cannington, San Cristobal, various Peruvian/Mexican, Greens Creek)

With precious metal prices and, in particular, silver prices projected to remain strong, the greatest growth in output of lead concentrates is expected to be seen in the latter complex, high precious metal bearing category, although the expected closure of Doe Run’s Herculaneum, Missouri smelter at the end of 2013 can be expected to free up some 120,000 tons of contained lead in clean concentrates (approximately 200,000 tons concentrates). While declining silver prices have somewhat supported the economics of importing silver-bearing concentrate into China, the decline has been nowhere near adequate to offset the effect of Chinese tax policies and the continuing, although narrowing, negative London/Shanghai silver arb noted above. Nonetheless, unless and until regulatory and tax policy changes are forthcoming in China, these qualities will remain best suited to western smelters, the result of which may be an ongoing smelter ‘bottleneck’ for such materials.

As with the zinc and copper markets, annual contract negotiations take place between the major miners (e.g. Cannington, San Cristobal, Red Dog, etc.) and western smelters in late- Q4/early Q1 of each year and such settlements typically set a frame of reference for other market transactions. However, unlike these other markets, as a result of the aforementioned market segregation in lead, there is no longer any single identifiable lead benchmark, with mines and smelters agreeing terms (treatment charges, escalators/de-escalators, if any, and silver refining charges) based on the specific characteristics of the individual concentrates (notably silver levels) and the market outlooks and objectives of the individual players involved in the discussions.

Historically refining charges for silver in lead concentrates of USD 0.30 to 0.35 per payable ounce were standard in the market. However, as western smelting capacity migrated to China and new high silver mines such as San Cristobal and Penasquito came into production, the aforementioned bottleneck for higher silver lead concentrate emerged due to the Chinese limitations. This was amplified with the 2009 closure of Doe Run’s La Oroya, Peru lead smelter, a facility specialized in the treatment of complex concentrates. As a result, silver refining charges jumped from the USD 0.30 to USD 0.35 per payable ounce level to USD 0.50 per payable ounce.
and then to USD 1.50 to +USD 2.50 per payable ounce, the latter typically being won on spot or distress business.

With regards to overall settlements, two of the recognized “benchmark” agreements set in 2012 were as follows:

<table>
<thead>
<tr>
<th>Table 19.6: Two of The Recognized “Benchmark” Agreements set in 2012</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>San Cristobal- Korea Zinc/Nyrstar</strong></td>
</tr>
<tr>
<td>TC</td>
</tr>
<tr>
<td>Esc/De-esc</td>
</tr>
<tr>
<td></td>
</tr>
<tr>
<td>Silver RC</td>
</tr>
<tr>
<td>Esc/De-esc</td>
</tr>
<tr>
<td></td>
</tr>
</tbody>
</table>

While both qualities can be similarly categorized vis a vis lead and silver grades and both settlements represent increases over the 2011 agreements in the order of USD 75 to 100 per ton of concentrate, the owners (Mitsubishi Corp and BHP, respectively) selected different means of achieving their pricing objectives in 2012. As we look forward, Western smelters are expected to remain well supplied, supporting the argument for sustained high charges. Nonetheless, there are several factors, which may influence the direction of charges, and particularly those for complex, high precious metal bearing concentrates:

- Restart plans for the La Oroya smelter have stalled and the company is now under government controlled liquidation. Both the plant’s owner (Doe Run) and the government refuse to invest the money necessary to upgrade the facility and, specifically, construct a sulphuric acid plant. The extraordinarily high spot silver refining charges witnessed in recent years (USD 2.50 to USD 5.00/oz) have been extremely profitable for most Western lead smelters. This has led to an investment in plant infrastructure and specifically with numerous increases in silver refining capacity (e.g. Korea Zinc-Onsan; Teck-Trail; Berzelius-Stolberg) which can be expected to result in increased consumption of complex precious metal bearing concentrates by these facilities thereby somewhat offsetting the impact on the market of any ongoing La Oroya closure.

- As noted earlier, Glencore appears to be moving forward with the restart of its Porto Vesme, Italy lead plant with the goal being to treat complex, precious metal bearing concentrates. However, they are also rumored to be pushing for the closure of 35% owned Xstrata’s Belledune lead smelter in New Brunswick. Any deferral to the restart at

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1 While western smelter silver refining charges have risen, they have remained somewhat more constant and considerably lower, particularly under long-term contracts.
Porto Vesme or an advancement/delay of the Belledune closure will have a significant effect on the market.

- There has been an explosion of secondary lead smelter capacity over the last year or so with the resultant increase in capacity tightening the spent battery market significantly. As a consequence, some primary plants are being forced back into the concentrate market for lead units.
- The aforementioned and recently announced closure of Doe Run’s Herculaneum smelter will release some 220,000 dmt clean concentrates into the custom market.

Taking all of the above into consideration, consensus appears to be forming around long-term treatment charges for complex, high silver-bearing lead concentrates in the following range and it is recommended that these base levels be used for the purposes of the Feasibility Study:

<table>
<thead>
<tr>
<th>Treatment Charge</th>
<th>USD 225 per dmt @ basis price of USD 2,000/mt</th>
</tr>
</thead>
<tbody>
<tr>
<td>Escalator</td>
<td>+USD 0.10/dmt for each USD 1.00 Pb price &gt; USD 2,200/mt</td>
</tr>
<tr>
<td>De-escalator</td>
<td>-USD 0.05/dmt for each USD 1.00 Pb price &lt; USD1,800/mt</td>
</tr>
<tr>
<td>Ag Refining Charge</td>
<td>USD 1.50/payable oz</td>
</tr>
</tbody>
</table>

19.3 Pulacayo-Paca Project

19.3.1 Marketing Pulacayo Concentrates

The securing of long-term contracts by Apogee and the terms, which can be fixed, will be influenced principally by the balance of supply and demand for custom zinc and/or lead concentrates during the period of contract negotiation, and the willingness, ability, and capacity of the counterparties to receive and process these concentrates based on their specific characteristics.

Because of the magnitude and history of the custom smelting/refining trade, the habits of the industry and the manner in which custom miners and custom smelters deal with each other is well established. For many new mines, off take contracts are frequently required which will ensure that the mine can deliver its concentrate to the market through the period during which debt financing is being repaid. For the purposes of this study, it is assumed that no specific restrictions will be placed on Apogee with respect to the sale of its concentrate products.

19.3.2 Concentrate Production Schedule and Quality

Over the currently projected 12.5 year LOM, annual production is projected to average approximately **21,200** dry metric tons zinc concentrates and **10,250** dry metric tons lead concentrates, grading approximately as set out in Table 19.7.
Table 19.7: Pulacayo Projected Concentrate Grades

<table>
<thead>
<tr>
<th>Element</th>
<th>Units</th>
<th>Grade</th>
<th>Grade</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Zinc Concentrates</td>
<td>Lead Concentrates</td>
</tr>
<tr>
<td>Pb</td>
<td>%</td>
<td>1.12</td>
<td>48</td>
</tr>
<tr>
<td>Zn</td>
<td>%</td>
<td>51</td>
<td>6.53</td>
</tr>
<tr>
<td>Ag</td>
<td>gms/mt</td>
<td>411</td>
<td>7438</td>
</tr>
<tr>
<td>Au</td>
<td>gms/mt</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Fe</td>
<td>%</td>
<td>2.19</td>
<td>N.A.</td>
</tr>
<tr>
<td>As</td>
<td>%</td>
<td>na</td>
<td>1.48</td>
</tr>
<tr>
<td>Sb</td>
<td>%</td>
<td>0.04</td>
<td>1.25</td>
</tr>
<tr>
<td>Bi</td>
<td>ppm</td>
<td>na</td>
<td>0.03</td>
</tr>
<tr>
<td>H₂O</td>
<td>%</td>
<td>7 to 10</td>
<td>7 to 11</td>
</tr>
</tbody>
</table>

Based on prevailing market, environmental and technological conditions, potential Pulacayo concentrate quality issues should be limited to the following:

19.3.2.1 Zinc Concentrates

Based on the limited specifications available at the time of preparation of this report, no significant issues are foreseen with the zinc concentrates. Having said that, although projected zinc levels are in line with most other standard qualities, the high silver levels may make them unsuitable for smelters with no or limited silver recovery. Nonetheless, as there are several potential smelter homes in Asia, Europe, Canada and Mexico that could likely handle these concentrates, this should not have a material impact on saleability, treatment terms or net realizations.

Elements commonly found in zinc concentrates, which are typically subject to penalties, and their thresholds are:

Table 19.8: Elements Commonly Found in Zinc Concentrates Which are Typically Subject to Penalties and their Thresholds

<table>
<thead>
<tr>
<th>Element</th>
<th>Typical Penalty Thresholds</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fe</td>
<td>8 to 9%</td>
</tr>
<tr>
<td>Pb</td>
<td>3 to 6%</td>
</tr>
<tr>
<td>Cd</td>
<td>0.3 to 0.4%</td>
</tr>
<tr>
<td>Hg</td>
<td>100 ppm</td>
</tr>
<tr>
<td>F+Cl</td>
<td>500 to 800 ppm</td>
</tr>
<tr>
<td>SiO₂</td>
<td>3 to 4%</td>
</tr>
</tbody>
</table>

Although none of the above are expected to be an issue in the Pulacayo concentrates based on early expectations, penalties may be incurred should any of these thresholds be breached.
Should levels significantly exceed any of the thresholds certain smelters may choose not to consider such concentrates for their feed mix.

19.3.2.2 Lead Concentrates

At around 48% Pb, the Pulacayo concentrates can be considered medium-grade with several internationally traded qualities running both higher and lower than these levels. The silver level in the concentrates can be considered relatively high, which will likely make these, concentrates less attractive to Chinese smelters because of the prevailing negative arbitrage between the LBMA and Shanghai prices.

Elements commonly found in lead concentrates, which are typically subject to penalties, and their thresholds are:

Table 19.9: Elements Commonly Found in Lead Concentrates Which are Typically Subject to Penalties and their Thresholds

<table>
<thead>
<tr>
<th>Element</th>
<th>Typical Penalty Thresholds</th>
</tr>
</thead>
<tbody>
<tr>
<td>As</td>
<td>0.2 to 0.5%</td>
</tr>
<tr>
<td>Sb</td>
<td>0.2 to 0.5%</td>
</tr>
<tr>
<td>Zn</td>
<td>5 to 7%</td>
</tr>
<tr>
<td>F+Cl</td>
<td>500 to 800 ppm</td>
</tr>
</tbody>
</table>

Based on the information available, arsenic and antimony levels are both seen to be well above typical penalty thresholds and, on a standalone basis, might make these concentrates less attractive to certain buyers for, at a combined 2.6 to 2.8% arsenic +antimony, the level is quite high. While Chinese smelters can typically handle antimony at these levels - in fact, many smelters in southern China like antimony unfortunately, this is not the case at many Western plants, as the overall load of arsenic and antimony will affect product and by-product quality. As such, although most plants in theory could handle the concentrate the quantity that each plant might accept will vary dramatically.

In addition to the arsenic and antimony, the silver content may determine the final homes for this material for, as highlighted in this report; high silver-bearing concentrates are a very difficult fit for the Chinese market. This can be expected to lead to additional charges being incurred via higher silver refining charges, higher treatment charges, or both.

In addition to these elements, other potential quality issues with the lead concentrates include:

- The lead grade at 46 to 50% is acceptable for China and in many cases preferable, although this is not the case in the West where most plants generally look for higher grade concentrates;
- Zinc at 3.5% would be attractive to the Chinese however, it is generally an undesirable element at most Western plants, and although at these levels, it is unlikely to be a significant issue.

Despite the above, it is not believed that quality issues would preclude sales of these concentrates, as there are other means of managing the impurities. Accordingly, regardless of
quality (unless, of course, impurities proved to be at extreme levels), placing the Pulacayo lead (or zinc) is not seen to be an issue.

19.3.3 Proposed Concentrate Sales Strategy and Distribution

For large scale projects where project financing is required, lenders generally look to have 70 to 80% of a mine’s output committed to smelters (or other acceptable buyers) under multi-year sales contracts with the balance of production available for sales under short- (spot) and medium-term contracts. Because of the size of and anticipated investment required for the Pulacayo project it is not expected that such restrictions will apply. Accordingly, sales of concentrates to buyers can likely be made at Apogee’s discretion.

Based on annual production volumes in the 10,000 and 20,000 ton ranges for lead and zinc concentrates, respectively, it is likely best to distribute the full volume of each product to a single buyer (or buyers if the zinc and lead products are sold separately). Seeing, as smelters will be the ultimate destination for both these products, consideration should be given to contacting selected operations to determine interest in these products. Both Korea Zinc and Teck, for example, are known to buy concentrates in South America and are prepared to handle all logistics from the ports of loading and, in the case of Korea Zinc, may also consider handling some internal logistics. While they are certainly alternatives for consideration, it is more than likely that traders operating in South America will present better opportunities to Pulacayo, both with respect to commercial terms and overall logistics.

Traders active in the zinc and lead markets include the following:

- Trafigura (CorMin)
- Glencore
- Ocean Partners
- MRI
- Transamine
- Louis Dreyfus

Most, if not all traders will take delivery of concentrates in warehouse at or near port facilities and will manage all logistics including packaging, if required (bags, lined containers, etc.), documentation and shipment to destination, subject to the provision of suitable allowances in consideration of the costs incurred. Some traders (e.g. Trafigura at Callao, Peru) operate extensive concentrate blending operations which are specifically designed to handle small- to medium sized quantities of concentrates (lead, zinc and copper) which are blended (e.g. to control Ag grade and impurity levels) to produce qualities particularly suited to specific buyers. As such, deleterious elements in complex qualities of concentrates can be diluted to levels acceptable to different smelters. Because of their ability to produce specific products from these blends, traders are frequently able to offer more aggressive purchase terms than will smelters, including early payment and financing provisions.

As noted above, the zinc concentrates are clean and can likely be placed direct with most smelters. Nonetheless, traders are regular buyers of clean concentrates, which they can either use as a diluent for their blend or for direct sale opportunities and will frequently bid aggressively.
to secure supplies. The lead concentrates on the other hand, being more complex and high in silver may be best suited for the trade, particularly where arsenic and antimony levels can be managed, either through mechanical or ‘paper’ blends. Furthermore, the small annual production volumes of each product make them more amenable to trader sales as they are frequently better equipped to handle small parcel sizes, thereby obviating the need to build shippable (typically 5,000 ton) lots.

Regardless of buyer, contracts can be entered into for any duration, from as short as spot sales (single or multiple sales but less than one-year contract duration), to short (one year) or medium term (2 to 4 year) contracts, to Life of Mine deals. Spot sales will typically provide fixed terms for the duration of the contract whereas other transactions can provide for a combination of terms including:

- Fixed terms (for all or a portion of the contract).
- Benchmarked terms in which base commercial terms (payable metals, penalties, payment terms, etc.) may be agreed but treatment charges, escalators, and refining charges and freight allowances (if applicable), may be adjusted annually or bi-annually, for example, using industry referenced benchmarks.
- Annually negotiated terms within a multi-year frame agreement.

In order to ensure a home for their concentrates many miners will typically enter into medium-term contracts with smelters and/or traders with terms agreed for two or three years and then subject to renegotiation thereafter.

There is no single ‘right way’ to manage concentrate sales in terms of contract duration and distribution. A marketing strategy needs to be developed and implemented which meets the specific requirements of the mine (e.g. comfort with/confidence in counterparty’s abilities; assurance of off take under potentially difficult market conditions; risk tolerance vs. a vis terms fluctuations, etc.) while taking into consideration prevailing market conditions at the time contract discussions are entered into (e.g. under tight market conditions a mine may want to lock in lower treatment and refining charges over a period of time rather than risk having terms move against them; or, in a soft market, it may be preferable to do short-term [spot or one-year] deals to ride out unfavorable conditions).

The strategy ultimately implemented for the sale of the Pulacayo concentrates will need to factor in such conditions and opportunities, as well as others, prevailing at the time discussions are entered into.

### 19.4 Concentrate Contract Sales Terms

#### 19.4.1 Delivery Terms

There are several different delivery options available to consider when selling concentrates but the most commonly used are (all basis Incoterms nomenclature):

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2 Traders are frequently able to provide smelters with paper blends of concentrates whereby a parcel of complex concentrates can be delivered with another parcel of clean concentrates. On a stand-alone basis, the smelter may not consider more complex parcels but if suitable volumes of clean feeds can be provided, they might be encouraged to take in such materials.
- CIF, named discharge port: for most offshore concentrate sales transactions, the seller delivers concentrates to the buyer’s discharge port with all costs of loading, transport, insurance and freight to destination for the seller’s account (CIF = Carriage, Insurance, Freight). Under CIF contracts, the buyer/receiver is responsible for all vessel discharge costs.

- INW named destination: in many instances the buyers, be they traders or in some instances smelters, will take delivery of the concentrates In Warehouse at or near the load port where they can be blended and/or packaged (e.g. bagged) for onward shipment to a smelter buyer. In this instance, the seller is responsible for the costs of delivering the concentrates to the buyer’s warehouse and for the insurance covering this move. The buyer is responsible for all handling costs from the warehouse to delivery to the receiving smelter’s discharge port, including handling, vessel loading, ocean freight and insurance after delivery to the warehouse.

- FOB named load port: as an alternative to an INW delivery, buyers will sometimes take delivery of concentrates loaded onto the ocean vessel at the load port. In this case, the seller is responsible for the costs of delivering the concentrates to the buyer on board the vessel (so including any warehousing and vessel loading costs) and for the insurance up to the vessel’s rail. The buyer is responsible for all ocean shipping costs from the load port to the discharge port, including insurance coverage from delivery to the carrying vessel’s rail along with all discharge costs.

- EXW or FCA: less frequently used, buyers will sometimes take delivery of the concentrates at Seller’s site loaded into a truck or railcar. In this instance, seller is responsible for simply loading the concentrates into the carrying conveyance; the buyer is responsible for all further costs to destination, including insurance.

Where concentrates are to be delivered to a domestic or inland smelter then other terms and conditions would apply (comparable to INW above); however this is not likely to be an option for Pulacayo unless the La Oroya smelter were to reopen.

In the case of anything other than a CIF delivery, the seller will typically provide an allowance (“Freight Allowance”) to the buyer to cover the equivalent costs of getting the concentrates to the ultimate market destination i.e. the cost that the seller were to otherwise incur were it to be selling concentrates direct to the end user. This allowance can either be included as part of the treatment charge or appear as a separate charge.

19.4.2 Payable (Accountable) Metals

Based on the anticipated grades as set out in Section 3, payable, or accountable metals in the Pulacayo zinc and lead concentrates can be expected to be as follows:
There have occasionally been efforts to institute changes to the zinc payable formula, as it is not representative of smelter recoveries, which run closer to 95-97%. While there is constant talk of a need for change, there does not appear to be enough momentum behind the miners' efforts to implement it, so for the foreseeable future no change is anticipated. If however, such a change were to occur, smelters would demand compensation for the lost value (commonly referred to as 'free zinc') as it is a vitally important contributor to their profit margins.

19.4.3 Metal Prices

Metal prices to be used in determining the payable value of the accountable metals in the concentrate (settlement prices) will be based on metal exchange quotations established during the applicable Quotational Period. Standard industry pricings for lead, zinc, silver and gold are based on the following:

<table>
<thead>
<tr>
<th>Payable Metal</th>
<th>Accountability</th>
</tr>
</thead>
<tbody>
<tr>
<td>Zinc</td>
<td>85%, min. 8 units&lt;sup&gt;3&lt;/sup&gt;</td>
</tr>
<tr>
<td>Lead</td>
<td>Nil</td>
</tr>
<tr>
<td>Silver</td>
<td>Deduct 3 oz/dmt, pay for 70% of balance</td>
</tr>
<tr>
<td>Gold</td>
<td>Nil</td>
</tr>
</tbody>
</table>

There have occasionally been efforts to institute changes to the zinc payable formula, as it is not representative of smelter recoveries, which run closer to 95-97%. While there is constant talk of a need for change, there does not appear to be enough momentum behind the miners' efforts to implement it, so for the foreseeable future no change is anticipated. If however, such a change were to occur, smelters would demand compensation for the lost value (commonly referred to as ‘free zinc’) as it is a vitally important contributor to their profit margins.

19.4.4 Treatment and Refining Charges

As noted in Section 3, treatment charges, escalators/de-escalators and silver refining charges, if applicable, can be established by various means:

**Long-Term Frame Contracts**

Under long-term agreements, contract terms are usually established as part of a frame agreement, with annual negotiations taking place to establish the appropriate treatment and refining charges (TC/RCs) to apply to a given contractual year’s deliveries. Under most such

<sup>3</sup> To be read as pay for the lesser of (a) 85% of the contained Zn in concentrates and (b) the Zn grade less 8%
agreements, very few terms are left undefined, with typically only the treatment charges, escalators/de-escalators and refining charges, as applicable, and any Quotational Periods subject to annual adjustment. Other terms, such as metal accountabilities and payment terms are typically fixed for the life of the contract.

The annual TC/RC adjustment to be agreed under such long-term agreements is typically referenced off market benchmarks. As the zinc concentrate market tends to be relatively transparent with information regarding settlements readily available via market sources (merchants, press reports, industry analysts such as Brook Hunt or CRU, etc.), it has traditionally been fairly easy to establish the applicable annual benchmarks. Although these sources can also be used for establishing reference lead TC/RCs, as discussed above, the market for lead concentrates has become somewhat more opaque over the past few years so careful consideration should go into which references are chosen and how such references might apply under evolving market conditions.

There is no consistent market structure for the duration of the TC/RCs agreed – they can be set according to various formats. For example, TC/RCs can be negotiated annually to cover 100% of the contractual year’s deliveries. Alternatively, terms can be fixed for two or three year periods (or longer).

For annually negotiated agreements, and particularly where the agreement is tied to project financing, clear definitions as to how terms are to be determined must be set out in the contract. Under such agreements, failure by the parties to agree in any given year will subject the parties to a referee mechanism whereby an independent third-party referee (or referees) will determine the appropriate treatment and refining charges to apply to a given year’s contractual deliveries. Where there is no such referee mechanism, failure by the parties to agree terms in any given year will typically lead to a “holiday” i.e. the tonnage contracted for that year’s (or those years) deliveries will be cancelled.

Medium-Term (2-5 Year) Contracts

Essentially, TC/RCs under medium-term contracts can be established using TC/RC conventions as per the long-term agreements. However, it is uncommon to have a referee mechanism available for failure to agree on annually negotiated terms, such that failure to agree will lead to a contract holiday which will result in cancellation of that year’s (or those years) deliveries.

Short Term/Spot Contracts

Shorter term or spot contracts are usually negotiated basis prevailing spot market conditions. All contractual terms (TC, RCs, escalators/de-escalators, if any, Quotational Period, payment terms, etc.) are up for negotiation in such contracts. While many of these contracts are entered into with merchants or brokers, it is not entirely uncommon for a mine to deal direct with a smelter. Short-term/spot contracts can be for single or multiple shipments but would typically be defined as deliveries, which are to occur within no more than a year of the agreement date.

For purposes of the Feasibility Study, it is recommended that the following charges be used:
Table 19.12: Treatment Charges/Refining Charges/Escalators/De-Escalators (USD)

<table>
<thead>
<tr>
<th></th>
<th>Zinc Concentrates</th>
<th>Lead Concentrates</th>
</tr>
</thead>
<tbody>
<tr>
<td>TC</td>
<td>$190.00 per dmt</td>
<td>$225.00 per dmt</td>
</tr>
<tr>
<td>Basis price</td>
<td>$2,200 per mt Zn</td>
<td>$2,000 per mt Pb</td>
</tr>
<tr>
<td>Escalator</td>
<td>+$0.03/dmt for each $1.00 Zn price &gt; $2200/mt</td>
<td>+$0.10/dmt for each $1.00 Pb price &gt; $2200/mt</td>
</tr>
<tr>
<td>De-escalator</td>
<td>-$0.03/dmt for each $1.00 Zn price &lt; $2,200/mt</td>
<td>-$0.05/dmt for each $1.00 Pb price &lt; $1,800/mt</td>
</tr>
<tr>
<td>RC – Ag</td>
<td>Nil</td>
<td>$1.50 per payable oz</td>
</tr>
<tr>
<td>RC – Au</td>
<td>Nil</td>
<td>$8.00 per payable oz</td>
</tr>
</tbody>
</table>

If sales are to be made to a trader or traders, then mines should typically expect to receive a discount to the benchmark treatment charge to entice the mine to sell to them. In exchange for such discounts, traders will seek contract flexibility, including options on Quotational Periods (discussed below), as well as possible destination and/or quantity options. The more options provided, the greater the value to the trader and thus the higher the level of the likely discount provided. Other factors can also come into play when determining discounts such as prevailing market conditions (e.g., a softer market would likely command smaller discounts) and concentrate quality considerations.

As it is assumed that sales will ultimately be made to the trade, for the purposes of the Feasibility Study it is recommended that the following discounts to the zinc and lead TCs are used:

Table 19.13: Discounts to the Zinc and Lead TCs

<table>
<thead>
<tr>
<th>TC Discount</th>
<th>Zinc Concentrates</th>
<th>Lead Concentrates</th>
</tr>
</thead>
<tbody>
<tr>
<td>TC Discount</td>
<td>$15.00 per dmt</td>
<td>$15.00 per dmt</td>
</tr>
</tbody>
</table>

19.4.5 Penalties

As noted, based on indicated assays, the Pulacayo zinc concentrates are to be considered “clean” and thus not subject to any penalties. Nonetheless, at the time of preparation of this report a specification for cadmium, an element commonly found in zinc concentrates, was not available. Typical cadmium penalties for zinc concentrates are:

\[
\text{Cd} \quad \text{USD 1.50/dmt for each 0.10\% Cd > 0.35\% Cd}
\]

The Pulacayo lead concentrates are to be considered somewhat complex due to the elevated levels of arsenic and antimony. While these levels are unlikely to create significant challenges to placing this material, penalties will apply. As such, for purposes of the Feasibility Study, it is recommended that the following penalty schedule be used:

\[
\begin{align*}
\text{As} & \quad \text{USD 1.50/dmt for each 0.10\% As > 0.30\% As} \\
\text{Sb} & \quad \text{USD 1.50/dmt for each 0.10\% Sb > 0.30\% Sb}
\end{align*}
\]
Based on the above penalties and projected grades, Pulacayo could expect to incur penalties per ton of lead concentrate as set out below.

<table>
<thead>
<tr>
<th>Element</th>
<th>Estimated Grade</th>
<th>Penalty</th>
</tr>
</thead>
<tbody>
<tr>
<td>Arsenic</td>
<td>1.45%</td>
<td>USD 17.25/dmt</td>
</tr>
<tr>
<td>Antimony</td>
<td>1.23%</td>
<td>USD 13.95/dmt</td>
</tr>
</tbody>
</table>

19.4.6 Freight Allowance

For deliveries of concentrates to the buyer’s/receiving smelter’s discharge port (CIF), there is typically no freight allowance provided to the buyer as the miner covers all costs to deliver the concentrates to destination (vessel loading, ocean freight, insurance). For deliveries to an intermediate destination (e.g. in warehouse at or near the load port, loaded on a vessel [FOB] or delivered to the buyer at the mine [FCA/EXW]), the miner will typically provide a freight allowance to the buyer to cover the typical costs which would otherwise be incurred to move the concentrates to their ultimate destination. This allowance is typically based on prevailing rates for movement of the concentrates either in bulk in vessels or by container and may be adjusted based on changes to prevailing market conditions. As noted above, the freight allowance may be rolled into the treatment charge or may be a standalone item but, in either case, the value of the freight allowance should be roughly equivalent to the actual costs associated with moving the concentrates from the point of delivery to their agreed upon destination.

Similarly, if concentrates were to be delivered to a domestic smelter, or a destination other than an agreed upon market destination (typically China/Japan/Korea ex west coast South America but may be a main European port), the buyer may seek an allowance to cover the cost differential versus delivery to such other market destination. In this instance, the freight allowance is effectively a freight capture or locational advantage, which the buyer is expecting to benefit from either in full or in part.

19.4.7 Quotational Period

The Quotational Period (QP) is the month (typically, although it can be any period of time) during which the payable metals in the concentrates are priced when the concentrate is sold to a buyer. The QP is typically tied to the month of shipment or the month of vessel arrival at the discharge port when concentrate is being shipped by vessel by the seller, or tied to month of delivery when concentrates are being delivered to a buyer at a named destination (e.g. INW at or near the load port), and can range from month of shipment (and, in some instances, even prior to shipment) to several months after arrival at the delivery point. In long-term and medium-term contracts, the QP is usually defined although buyers frequently seek an option which allows them to change the QP month for a particular metal (or metals), which they exercise under certain market conditions. Traders will always seek QP options and the greater the range or flexibility that can be offered, the higher the value to the buyer, which can typically be translated into higher discounts to applicable charges. For zinc and lead, the QP will typically range from month of delivery/shipment to third or fourth month after month of delivery/vessel arrival at destination, although even more
extended durations can be agreed. For gold and silver, the QP may have a similar range, but it is more likely that buyers will look for an early Quotational Period to allow them to earn contango.

19.4.8 Payment Terms

Standard industry terms allow for a first provisional payment by the buyer for 90% of the value of the concentrates at a fixed date ‘X’ days after vessel departure from the load port or ‘Y’ days after vessel arrival at the discharge port, with final payment due when all final information (prices, assays, weights, etc.) is known. In several instances, a second provisional payment may also be provided as an interim payment prior to when all final details (weights, assays and prices) are available.

Most traders will offer some form of financing with many frequently offering to pay for concentrates as they are delivered to them at the port or in their warehouses. While most will offer this service for ‘free’, some will look to charge extra for this although rates tend to be relatively reasonable and below most miners’ cost of capital (e.g. LIBOR +2 to 3%).

For purposes of the Feasibility Study, it is recommended that the following payment terms be used:

**Direct Smelter Sale**

**First Provisional** 90% of invoice value\(^4\), 15 days after vessel arrival at discharge port.

**Final Payment** 100% of the invoice value, less the value of the first provisional, on final settlement (assume 100 days after vessel arrival)

**Trader Sale**

**First Provisional:**
- 90% of invoice value, 5 days after the end of the month of delivery of each monthly quota of concentrates.

**Second Provisional:**
- 100% of invoice value, less the value of the first provisional payment, 45 days after the date of the first provisional payment.

**Final Payment:**
- 100% of the invoice value, less the value of the first and second provisionals, on final settlement (assume 120 days after the end of the month of delivery of the monthly quota).

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\(^4\) The invoice value equals the net value of the payable metals less all charges (treatment charge, refining charges, escalators/de-escalators (if any), penalties, freight allowance (if any))
19.4.2 Weight Franchise

For sales to buyers where delivery is made to the receiving smelter’s port of discharge, no weight franchise will typically apply unless the concentrates are being shipped to a destination where internationally recognized weighing and sampling procedures and facilities are not available (e.g. most destinations in China). If delivery is made to one of these destinations or, in the case where delivery is made to an intermediate destination (e.g. to a warehouse or basis an FOB delivery), a weight franchise will typically apply, with such franchise intended to represent the handling losses which are typically incurred in transit to the destination. Weight franchises can range from 0.15% to as high as 0.50% but are typically in the 0.20 to 0.30% range5.

19.5 Concentrate Logistics

19.5.1 General

The Pulacayo-Paca project is located some 7,800 km by road from the Chilean ports of Arica, Antofagasta and Mejillones and some 2,200 km from the Peruvian port of Callao. Although the zinc concentrates can very likely be exported through one of the Chilean ports6, with rising environmental concerns, the Chileans have reportedly been looking to restrict bulk lead concentrate flows through their ports and, for this reason, unless the lead concentrates can be containerized, exports may have to go through Peru where a substantial lead business already exists. At this point in time, Arica and Antofagasta do apparently still permit lead concentrate exports in containers (in bags in containers or in bulk in lined containers) although some additional charges may apply (including additional International Maritime Organization – IMO – charges).

As noted in previous sections, because of the volumes of concentrates projected to be produced, it is most likely that they will either be sold to a trader and (a) exported “as is”, loaded in containers (in bags or in bulk in lined containers), or (b) delivered to a warehouse in Bolivia, Chile or Peru for blending or accumulation with other small parcels after which the concentrates will either be loaded and shipped in containers (as per (a)) or shipped in bulk lots of approximately 5,000 wmt (typical minimum bulk shipment sizes).

Bulk shipments of concentrate in ocean-going vessels remain the standard means of delivering concentrates to the market. Most major mines ship concentrates in 10 to 15,000 wmt parcel sizes, although shipments of smaller lots (approximately 5,000 wmt), while more expensive, are not entirely uncommon.

As with many of the commodity markets, the ocean freight market has witnessed severe volatility since the early part of this decade. After years of relative stability, surging demand and a shortage of vessels drove ocean rates for bulk commodities to their highest levels in history with the Baltic Dry Index (BDI) – an overall measure of the spot dry bulk cargo market which captures daily bookings for capsize, panamax and handy size vessels - hitting all-time highs above 11,000

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5 Weight franchises will tend to be higher if concentrates are to be blended as losses due to handling and dilution can expected to be greater.
6 Iquique, Chile, some 600 kms by road from the mine, may also be an alternative load port
in 2007 and 2008, up more than five-fold since the beginning of the decade. In a dramatic retracement however, the market gave up all of these gains by the end of 2008 as the global financial crisis took hold and only a marginal recovery has been witnessed since (see Figure 19.13).

These market movements translate into ocean freight rates for shipment of 10,000 ton parcels from Chile/Peru to Japan/Korea/China reaching as high as USD 120 to 130 per ton at their peak, with current rates being seen in the USD 50 to 60/wmt range. While a gradual rise in rates is expected going forward as current levels are simply not sustainable, the return to the highs of 2007 to 2008 are not foreseen anytime soon.

**Figure 19.13: Baltic Dry Index: 2000 – 2012**

*Source: InvestmentTools.com*

Container shipments of concentrates typically present an expensive alternative to bulk shipments, however for some movements they have become a competitive option and, in certain instances, have in fact become a cheaper alternative. Rates to China in particular have come down to very low levels as container lines look for backhaul opportunities to fill otherwise empty containers returning to China. Additionally, reduced in-transit losses and the ability to ship more frequently and in smaller lot sizes (as low as approximately 20 tons) thereby reducing working capital, can further add to the value of shipping by this means.

SDV, part of the diversified transportation and logistics group Bolloré, and which focuses on international ocean freight movements (including container and inland logistics management), provided the following indicative rates for the movement of concentrates in containers from Uyuni to delivery China via Antofagasta, Chile:
Table 19.15: Rates for the Movement of Concentrates in Containers from Uyuni to Delivery China via Antofagasta, Chile

<table>
<thead>
<tr>
<th>US$</th>
<th>Zn Concs via Antofagasta, Chile</th>
<th>Pb Concs via Antofagasta, Chile</th>
</tr>
</thead>
<tbody>
<tr>
<td>Truck Uyuni to Port</td>
<td>USD 3,100/truckload</td>
<td>USD 3,100/truckload</td>
</tr>
<tr>
<td>Handling Costs, including container stuffing</td>
<td>USD 760/cont.</td>
<td>USD 935/cont.</td>
</tr>
<tr>
<td>Add’l Terminal Costs/Customs</td>
<td>USD 170/cont.</td>
<td>USD 170/cont.</td>
</tr>
<tr>
<td>Ocean Freight, basis 20’ cont</td>
<td>USD 1,980/cont.</td>
<td>USD 2,230/cont.</td>
</tr>
<tr>
<td>Total</td>
<td>USD 6,010/cont.</td>
<td>USD 6,435/cont.</td>
</tr>
<tr>
<td>Total per wmt 1</td>
<td>USD 325/wmt</td>
<td>USD 348/wmt</td>
</tr>
</tbody>
</table>

Note: 1. incl. estimated IMO charges of $250/container 2. Basis max payload of 22.0 mt/container

As an alternative to the above movement, SDV also advised that concentrates could be shipped in bulk in trucks to a facility owned and operated by their affiliate Grupo Empresarial Nautilius in Patacamaya, Bolivia (La Paz district), some 450 kms from the project, where they can be transloaded into bags and stuffed in containers for export. According to SDV, this facility regularly handles concentrates in Bolivia and employing this option could reduce costs as issues associated with Chilean regulations and weight limits on the roads can be negated. Indicated costs for this movement to China are as follows:

Table 19.16: Costs to Export the Concentrates to China

<table>
<thead>
<tr>
<th>US$</th>
<th>Zn or Pb Concs via Patacamaya, Bolivia/ Arica, Chile</th>
</tr>
</thead>
<tbody>
<tr>
<td>Truck Uyuni to Patacamaya (est.) 1</td>
<td>USD 1,500/truckload</td>
</tr>
<tr>
<td></td>
<td>USD 75/wmt</td>
</tr>
<tr>
<td>Handling Costs, incl stuffing (est.) 1</td>
<td>USD 15/wmt</td>
</tr>
<tr>
<td>Container to Shenzhen China, basis 20’ cont. 2</td>
<td>USD 4,600/cont.</td>
</tr>
<tr>
<td></td>
<td>USD 210/wmt</td>
</tr>
<tr>
<td>Total per wmt 2</td>
<td>USD 300/wmt</td>
</tr>
</tbody>
</table>

Note: 1. Exen estimate 2. basis 22 wmt/container

Because of the production volumes and the concentrate quality expected from Pulacayo, for the purposes of this report, it is assumed that zinc concentrates will be shipped via container ex Arica to China at a cost as indicated above, although if concentrates can be shipped as part of a bulk movement, total realized savings may be in the order of USD 40/mt, based on estimated port costs and ocean freight rates.

For the lead concentrates, because it is expected that they will have to be blended and because of potential Chilean export restrictions, is it is assumed that they will be trucked to a facility in
Patacayama for packaging before being onward shipped to China. However, if the concentrates need to be blended and suitable facilities do not exist in Bolivia, trucking to Callao, Peru where they will either be blended for export or shipped ‘as is’ in containers, again with destination China (or parity) may be an option. This movement may actually prove to be competitive based on estimated trucking, port costs and ocean freight rates and would offer a greater degree of commercial flexibility due to the number of traders operating out of this port.

In addition to the various trucking alternatives, FCAB (owned by the Luksic Group in Chile) operates a rail line from which runs relatively near to the project to Antofagasta. Early indications were that rates for concentrate movements to the port may potentially be very low. However, it is understood that this line is already operating at capacity and may not be able to handle any additional concentrates.

As noted earlier, any costs associated with these movements will either be borne directly by Pulacayo or be charged back to the mine via a freight allowance imposed by the buyer. In either case, the net cost is assumed to be the equivalent.

19.5.2 Additional Logistics Issues

19.5.2.1 Handling Losses

During the transportation of concentrates from the mine to the customer, it is inevitable that some losses will occur due to spillage and exposure to the elements. While there is no exact figure as to what handling losses can be expected, common industry practice suggests losses of 0.1 to 0.2% at each handling point. Based on the movement of concentrate from the mine to the receiving smelter the handling points would be as follows:

1. Storage shed at mine loaded into truck
2. Truck discharged into storage shed at port
3. Storage shed at port conveyed into container/onto vessel
4. Vessel discharged onto quay/into trucks

Depending on the handling procedures prior to weighing and sampling at the receiving end, there could be an additional handling point (or points) after discharge. Based on the above noted rule of thumb, handling losses could be in the order of 0.3% to as much as 1.0%. In order to minimize handling losses, precautions should be taken at each transfer point (e.g. aprons on the sides of vessels at loading and discharge) and during transit (e.g. covered trucks).

19.5.2.2 Insurance

Ocean marine cargo insurance can be obtained for all concentrates shipped by vessel. Under CIF contracts, marine insurance is taken out by the seller in the name of the buyer for 110% of the estimated value of the concentrates in each shipment. Risk of loss, excluding normal handling losses, passes to the buyer as concentrates are progressively loaded onto the carrying vessel. Insurance rates typically average around 0.05 to 0.065% of the estimated invoice value (adjusted to 110%) i.e. the payable metal value, less all treatment and refining charges, as well as any penalties and escalators/de-escalators which may apply (equivalent to the “Net Invoice Value”, or...
“NIV”). For purposes of the Feasibility Study, it is recommended that insurance costs of 0.06% of the Net Invoice Value (adjusted to 110%) be used for evaluation purposes.

19.5.2.3 Representation

Inspection/representative services are typically employed at the vessel discharge or at an intermediate delivery point during the weighing and sampling procedures to ensure that the mine’s interests with respect to the handling of the concentrates are fully respected. There are a number of companies, which offer these services, including such international firms as Alfred H. Knight International, SGS and Stewart Group. Costs for representation services are typically in the order of USD 0.30 to 0.50/wmt concentrate.

19.6 Conclusions

Although the Pulacayo lead concentrates may face some additional challenges to marketing due to their high silver grades and impurity levels (notably arsenic and antimony), both Pulacayo products are viewed as being readily saleable to buyers under normal market conditions. Although mine-smelter supply-demand balances will ultimately drive the market for concentrates, because of the relatively small projected annual production volumes of each material, it is assumed that sales will be pursued with traders who can generally offer more favorable purchase terms and conditions and have better means of dealing with any quality issues.

For purposes of the Feasibility Study, it is recommended that the following charges be used:

<table>
<thead>
<tr>
<th></th>
<th>Zinc Concentrates</th>
<th>Lead Concentrates</th>
</tr>
</thead>
<tbody>
<tr>
<td>TC</td>
<td>USD 190 per dmt</td>
<td>USD 225 per dmt</td>
</tr>
<tr>
<td>Basis price</td>
<td>USD 2,200 per mt Zn</td>
<td>USD 2,000 per mt Pb</td>
</tr>
<tr>
<td>Escalator</td>
<td>+USD 0.03/dmt for each $1.00 Zn price &gt; USD 2 200/mt</td>
<td>+USD 0.10/dmt for each USD 1.00 Pb price &gt; USD 2 200/mt</td>
</tr>
<tr>
<td>De-escalator</td>
<td>-USD 0.03/dmt for each USD 1.00 Zn price &lt; USD 2 200/mt</td>
<td>-USD 0.05/dmt for each $1.00 Pb price &lt; USD 1,800/mt</td>
</tr>
<tr>
<td>Trader TC Discount</td>
<td>USD 15.00 per dmt</td>
<td>USD 15.00 per dmt</td>
</tr>
<tr>
<td>RC – Ag</td>
<td>Nil</td>
<td>USD 1.50 per payable oz</td>
</tr>
<tr>
<td>RC – Au</td>
<td>Nil</td>
<td>USD 8.00 per payable oz</td>
</tr>
<tr>
<td>Penalties:</td>
<td>Cd: USD 1.50/dmt/0.10% Cd &gt; 0.35% Cd</td>
<td>As: USD 1.50/dmt/0.10% As &gt; 0.30% As</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Sb: USD 1.50/dmt/0.10% Sb &gt; 0.30% Sb</td>
</tr>
</tbody>
</table>

Based on the above, net smelter return calculations (NSRs) were derived for the Pulacayo zinc and lead concentrates basis sales to traders under the following assumptions:
Zinc concentrates:

1. Delivery CIF China equivalent, basis truck to Antofagasta, Chile and shipment in bulk in ocean vessels ex Antofagasta
2. Delivery CIF China equivalent basis truck to Antofagasta, Chile and shipment in bags in containers ex Antofagasta, Chile
3. Delivery CIF China equivalent basis truck to Patacayama, Bolivia and shipment in bags in containers ex Arica, Chile

In each case, terms are as per benchmark less a trader discount on the treatment charge of USD 15/dmt.

Lead concentrates:

1. Delivery CIF China equivalent, basis truck to Callao, Peru and shipment in bulk in ocean vessels ex Callao.
2. Delivery CIF China equivalent basis truck to Antofagasta, Chile and shipment in bags in containers ex Antofagasta, Chile.
3. Delivery CIF China equivalent basis truck to Patacayama, Bolivia and shipment in bags in containers ex Arica, Chile.

In each case, terms are as per benchmark less a trader discount on the treatment charge of USD 15/dmt.

The NSRs, based on the above noted assumptions and metal prices, treatment terms and logistics costs as set out in this report are set out in Schedules III and IV in the Appendix and can be summarized as follows:

### Table 19.18: NSRs for Sale of Pulacayo Zinc and Lead Concentrates

<table>
<thead>
<tr>
<th>USD/dmt</th>
<th>Zn Concentrates</th>
<th>Pb Concentrates</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Bulk via Antofagasta, Chile, USD</td>
<td>Containers via Antofagasta, Chile, USD</td>
</tr>
<tr>
<td>Payable Metals:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Contained Metal Value</td>
<td>1,999.81</td>
<td>1,999.81</td>
</tr>
<tr>
<td>Payable Metal Value</td>
<td>1,508.31</td>
<td>1,508.31</td>
</tr>
<tr>
<td>Value of Metal Deductions</td>
<td>491.50</td>
<td>491.50</td>
</tr>
<tr>
<td>Less:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Treatment Charges, incl Esc/De-esc</td>
<td>175.14</td>
<td>175.14</td>
</tr>
<tr>
<td>Ag and Au Refining Charges</td>
<td>0.00</td>
<td>0.00</td>
</tr>
<tr>
<td>Freight Allowance</td>
<td>0.00</td>
<td>0.00</td>
</tr>
<tr>
<td>Penalties and Other Charges</td>
<td>0.00</td>
<td>0.00</td>
</tr>
<tr>
<td>Net Invoice Value</td>
<td>1,333.17</td>
<td>1,333.17</td>
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</table>
Pulacayo 1 000 t/d Phase I Feasibility Study - NI 43-101 Technical Report
090644-3-0000-20-IFI-100

20.1 General Description of the Project

The Pulacayo Mining Project area is located 18 km by road north east of the town of Uyuni, as a territorial part of the first section Uyuni of the province Antonio Quijarro in the Department of Potosi.

The Pulacayo mineral deposit comprises a reservoir of zinc, lead and silver concentrates.

For activities of exploitation and complex mineral concentration, the Pulacayo Mining Project considers a process of treatment capacity of 1,000 tons per day of ore, incorporating underground mining methods and ore treatment through flotation process.

20.2 Description of Available Information

For the Pulacayo Mining Project a wide range of environmental and socioeconomic studies has been made (see Chart 20.1), these studies have been developed for the exploration stage as well as for the future operation on the field. All information on environmental and socio-economic studies, has been generated by the Consultant MEDMIN through contracts made with Apogee Minerals Bolivia Company and ASC Bolivia LDC.

This information has been collected from works performed in the project area since year 2009.

<table>
<thead>
<tr>
<th>Title</th>
<th>Author</th>
<th>Year</th>
</tr>
</thead>
<tbody>
<tr>
<td>Exploration Stage</td>
<td></td>
<td></td>
</tr>
<tr>
<td>EMAP (Exploracion Minera y Actividades Pequeñas – Mining Exploration and Small Activities)</td>
<td>MINCO</td>
<td>2007</td>
</tr>
<tr>
<td>Updated EMAP</td>
<td>MEDMIN</td>
<td>2010</td>
</tr>
<tr>
<td>Updated PMA (Plan de Manejo Ambiental – Environmental Management Plan) – PASA (Plan de Aplicación y Seguimiento Ambiental – Execution and Follow up Environmental Plan)</td>
<td>MEDMIN</td>
<td>2010</td>
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<tr>
<td>First Report of Environmental Monitoring (IMA I/2010)</td>
<td>MEDMIN</td>
<td>2010</td>
</tr>
<tr>
<td>Title</td>
<td>Author</td>
<td>Year</td>
</tr>
<tr>
<td>----------------------------------------------------------------------</td>
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<td>------</td>
</tr>
<tr>
<td>Updated EMAP</td>
<td>MEDMIN</td>
<td>2011</td>
</tr>
<tr>
<td>License for Activities with Hazardous Substances</td>
<td>MEDMIN</td>
<td>2011</td>
</tr>
<tr>
<td>Fifth Report of Environmental Monitoring (IMA I/2012)</td>
<td>MEDMIN</td>
<td>2012</td>
</tr>
<tr>
<td>Sixth Report of Environmental Monitoring (IMA II/2012)</td>
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<td>2013</td>
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**Exploitation Stage**

<table>
<thead>
<tr>
<th>Title</th>
<th>Author</th>
<th>Year</th>
</tr>
</thead>
<tbody>
<tr>
<td>Environmental Baseline Audit</td>
<td>MINCO</td>
<td>2007</td>
</tr>
<tr>
<td>Environmental Record - Project of extraction and concentration of complex minerals and Pulacayo tailings dam construction</td>
<td>MEDMIN</td>
<td>2009</td>
</tr>
<tr>
<td>Environmental Baseline Study - Physical Environment, dry season.</td>
<td>MEDMIN</td>
<td>2010</td>
</tr>
<tr>
<td>Environmental Baseline Study – Biotic Environment - dry season</td>
<td>MEDMIN</td>
<td>2010</td>
</tr>
<tr>
<td>Environmental Baseline Study.- Socioeconomic</td>
<td>MEDMIN</td>
<td>2010</td>
</tr>
<tr>
<td>Environmental Baseline Study.- Archeologic</td>
<td>MEDMIN</td>
<td>2010</td>
</tr>
<tr>
<td>Environmental Baseline Study.- Agriculture</td>
<td>MEDMIN</td>
<td>2010</td>
</tr>
<tr>
<td>Environmental Baseline Study (Updated – Dry season)</td>
<td>MEDMIN</td>
<td>2010</td>
</tr>
<tr>
<td>Environmental Record, Rehabilitation Project inside the Mine for the Mineral Extraction and Transportation for External Metallurgical Testing - Pulacayo.</td>
<td>MEDMIN</td>
<td>2011</td>
</tr>
<tr>
<td>Environmental Record, Project of Mineral Extraction and Concentration of Complex minerals and Pulacayo tailing dam construction</td>
<td>MEDMIN</td>
<td>2011</td>
</tr>
<tr>
<td>Environmental Baseline Study - Physical Environment – wet season</td>
<td>MEDMIN</td>
<td>2011</td>
</tr>
<tr>
<td>Environmental Baseline Study - Biotic Environment - wet season</td>
<td>MEDMIN</td>
<td>2011</td>
</tr>
<tr>
<td>Environmental Baseline Study – Drinking water – wet season</td>
<td>MEDMIN</td>
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<tr>
<td>Environmental Baseline Study – Drinking water – Dry season</td>
<td>MEDMIN</td>
<td>2011</td>
</tr>
<tr>
<td>Environmental Baseline Audit (Updated - wet season)</td>
<td>MEDMIN</td>
<td>2011</td>
</tr>
<tr>
<td>Environmental Baseline Study - Socioeconomic (Updated)</td>
<td>MEDMIN</td>
<td>2012</td>
</tr>
<tr>
<td>Environmental Baseline Study - Flow Measurement and Measuring of water storage volumes.</td>
<td>MEDMIN</td>
<td>2012</td>
</tr>
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<td>Socialization MEDMIN Pulacayo Mining Project</td>
<td>MEDMIN</td>
<td>2012</td>
</tr>
<tr>
<td>Public Consultation Pulacayo Mining Project</td>
<td>MEDMIN</td>
<td>2013</td>
</tr>
<tr>
<td>Study of Environmental Impact Assessment - Specific Analytical(EEIA-AE)</td>
<td>MEDMIN</td>
<td>2013</td>
</tr>
</tbody>
</table>

**Source:** MEDMIN, 2013.
20.3 Potential Environmental Impacts and Proposed Mitigation Measures

20.3.1 Water Quality

Within the study area is the water spring called Yanapollera, which has a small dam, and supplies water to the mining centers of Pulacayo and Uyuni. Watersheds of Negro, Huanchaca and Escalera rivers flow into the plain of Kasapampa following, in most cases, from a north west direction to south east, these two rivers are considered the largest within the study area.

Water sector bodies have a close relationship between surface and groundwater. Although the area is dissected by several valleys, some rivers are of perennial type. The water flow in most of these sources increases in the rainy season, some of them leading to dryness in times of drought.

For the assessment of water quality, samples were obtained through the certified laboratory Spectrolab, in charge of collecting the samples.

The sample points for the development of the Environmental Baseline Study (ELBA)–Physical, were collected in dry and wet seasons. The assessment took the standards for defining the permissible limits from Chart 1-A of the Regulation on Water Pollution (RMCH) of the Environmental Law 1333, that belongs to the Bolivian regulation.

The results of water quality analysis indicate that the sample with the highest number of non-standard parameters (a total of 15) was obtained in the study of the dry season and corresponds to the tails of the Pulacayo Mining Cooperative Ltd. This is because these tails are not treated to reduce its content of contaminants before coming into contact with the Negro river.

Other samples that stand out for the number of parameters outside the norm are the waters from tailings in the Kasapampa sector, with a total of 14 physical-chemical and multi-elemental parameters. This situation is given because in that zone there is tailings accumulation, product of activities of ore concentration conducted by the Pulacayo Works until 1959, these tailings were washed away by the Negro river flow to be deposited in this zone and contributing to water contamination mainly associated to heavy metals.

The parameters that are outside the norm, most recurrent in all samples and in all studies are cadmium, uranium, iron, pH, antimony, arsenic, copper and lead.

20.3.2 Air Quality

Regarding air quality, 6 measurements of particles were performed (PM₁₀ - PST) at different points of the Pulacayo Mining Project area, these points were selected according activities, locations and regulatory aspects. These measurements were taken both in the dry and wet season. The results indicate that in almost all the sampled points, the allowable limits for PM₁₀ and TSP are exceeded.

20.3.3 Soil Quality

To assess the quality of soil a total of 16 samples were collected. The samples were analyzed and compared between the Environmental Baseline Study (ELBA)-Physical ones, for the dry season and the wet season. Due to the lack of Bolivian bylaws for evaluating soil quality, Medmin
used the Environmental Quality Standard for Soil Resources and Criteria for Contaminated Soil Remediation of the Republic of Ecuador.

The sample with the highest number of non-standard parameters, was obtained in the ELBA of the dry season from an area of land in the area called Tajo, where there is the presence of mining environmental threats in the area of sampling and also in the surrounding ones, that coupled with the action of climatic factors (mainly rain and wind), generated pollution due to dragging of material.

The parameters that are beyond the norm, recurrent in all samples and in all studies are arsenic, zinc, lead and cadmium.

20.3.4 Sediment Quality

To evaluate the quality of sediments in the area of the Pulacayo Mining Project, 7 sampling points were collected for studies of dry season and wet season respectively. To compare with the reference values, Medmin used the Canadian Sediment Guidelines for the Protection of Aquatic Life, because Bolivia does not have its own regulations.

The sample with the highest number of parameters beyond the Canadian standard is the one taken from the Huanchaca River, with a total of 5 exceeding parameters. This situation is mainly due to the presence of mining environmental threats in the sampling area and surrounding areas, which together with the action of climatic factors (mainly rain and wind) generated pollution thereof due to entrainment of material in the basin of the river aforesaid.

The parameters that are beyond the norm, more recurrent in all samples, in all studies, are Arsenic, Lead, Zinc and Cadmium.

20.3.5 Drinking Water Quality (Drinking Water Distribution System)

The Pulacayo mining center uses surface water from different sources, such as the snow thaws of the Cosuño mountain, the streams of Huayllas and Juskuni, these water sources are stored in the dam Yanapollera, Huayllas pond and Juskuni pond respectively. The distribution of water in the town of Pulacayo is made by plumbing and is done directly from the source.

To study the quality of drinking water two sample collection campaigns were made, one in the dry season and one in the wet season, in order to evaluate and compare the data and get a better interpretation of the results and variations throughout the year.

The results of quality of drinking water shows that the water quality is good in the ponds Huayllas and Juskuni throughout the year, the latter being the best quality. The water quality of the Yanapollera dam is low during the wet season; however, during the dry season this water improves its quality but still presents contamination by whole coliforms contamination, reducing its concentration relative to the sample obtained in wet season.

20.3.6 Condition of Flora and Fauna

The Pulacayo Mining Project is not within any protected areas of Bolivia. The characterization of eco-biological components according to the classification by eco-regions and sub-Eco regions (Ibisch et al. 2003), indicates that the project area lies within two major sub-Eco regions:
• The semi-arid Puna
• The Puna desert (semi-arid) in the Andean region.

Moreover, its correlation with the classification of Navarro (2002), places the project in the Puna-Peruvian province, bio-geographical Chicas district. These are areas where the ombroclimate is dry; the dominant vegetation consists of xeric shrub lands. With six sets of vegetation ranging from Tholar-grassland, high-Andean to Tolillar semiarid of the Altiplano.

20.3.6.1 Flora

Within the two characterized sub-eco regions are different types of vegetation, where the most representative are:

a) Tarapacana Polylepis thickets (keñua) related to compact Azorella (yareta)
b) Distichia cushions muscosoides-Werneria pygmaea (cienegos) in the wetlands
c) Cristatum Tetracochlin thorny scrub, Adesmia spinossissima and Adesmia sp.
d) Low shrubs of Baccharis boliviensis (pescollanta) and Fabiana densa (tarallanta)
e) Lampaya castellani Scrub (Lampaya)
f) Trollii Oreocereus Scrub (branched cactus).
g) Orthophylla Festuca grasslands (ichu)
h) Leptostachya Stipa Grasslands (sicuya) and scrub Parastrephia lucida (yakullanta).
i) Secondary vegetation (in environmental threats).
j) Secondary vegetation (ex-crops)

Within the study area there have been recorded two families with two endangered species in different categories being the most threatened the Polylepis tarapacana "keñua" (Rosaceae) (see Chart 20.2). On the other hand, there have been four cactus species that are regulated under Appendix II of CITES (Convention on International Trade in Endangered Species of Wild Fauna and Flora). There are no endemic species.

<table>
<thead>
<tr>
<th>FAMILY</th>
<th>SPECIES</th>
<th>COMMON NAME</th>
</tr>
</thead>
<tbody>
<tr>
<td>Apiaceae</td>
<td>Azorella compacta</td>
<td>yareta</td>
</tr>
<tr>
<td>Rosaceae</td>
<td>Polylepis tarapacana</td>
<td>keñua</td>
</tr>
<tr>
<td>Cactaceae</td>
<td>Opuntia soehrensii (*)</td>
<td></td>
</tr>
<tr>
<td>Cactaceae</td>
<td>Oreocereus trollii (*)</td>
<td></td>
</tr>
<tr>
<td>Cactaceae</td>
<td>Trichocereus atacamensis (*)</td>
<td>pasacana</td>
</tr>
<tr>
<td>Cactaceae</td>
<td>sp.FZR-9819 (*)</td>
<td>achacama</td>
</tr>
</tbody>
</table>

Source: MEDMIN, 2011.

20.3.6.2 Fauna

The southern part of the Bolivian highlands is one of the most arid and inhospitable of Bolivia, which determines the presence of populations and species of animals. Wildlife species that were
recorded in the work area were mostly identified by direct observation, recording evidence and local references.

The main families of fauna are:

a) Mammals
The mammals are the Andean ones, located in a system of gullies and rocky gorges known as the D gorge of the Yanapollera river and the Mercedes gorge, which has regular weather and milder habitability. The recorded species of importance are: Cougar (Puma concolor), Fox (Lycalopex culpaeus), vizcacha (L. viscacia) and the presence of llamas (Lama glama).

b) Birds
The most common birds in the area include: land birds such as partridges (Tinamotis pentlandii) and scavengers like the condor (Vultur gryphus). In the work area was seen the pichitanka (Zonotrichia capensis) and Geositta punensis.

c) Herpetofauna and Ichthy Fauna
Some common species include the high Andes, including spinolosus Bufo (toad), and Tachymenis armoratus Telmatobius peruviana (steel or Andean snake).

20.3.6.3 Condition

The current pressure of Pulacayo on the vegetation is intended for wood fuel, overgrazing and enabling new farmland. The only category of threatened species in the IUCN are Polylepis tarapacana (keñua) and Azorella compacta (yareta).

Environmental threats, represent the greatest danger from habitat destruction. In addition, since this activity continues on a smaller scale, there are still remnants of these environmental threats, which could still affect large segments of the biota, in general, as well as any agricultural and even human activity.

The observation of the State of Conservation (EC) shows evidence of highly relevant well preserved areas (flora - fauna, habitats, landscape/approx. 57 km²) associated with this are the areas with EC medium (approximately 345 km²) occupying most of the surface and whose EC value is affected by its traditional use, use of water sources and foraging resources shared by native and domestic species. Less than a quarter of the area (77 km² approximately) Is highly degraded by human disturbance (mining pollution and landscape change), representing the areas of greatest environmental concern, with direct and residual effects on the biota, with highly modified landscape.

20.4 Environmental Management Plans

20.4.1 Environmental Mitigation Plan
Pursuant to Articles 29 and 30 of the Regulation on Prevention and Environmental Control, states that it is necessary the definition and description of a set of protective, corrective, and
compensatory measures properly implemented that will serve to prevent, reduce, and eliminate or compensate the expected changes.

20.4.1.1 Air

From the analysis conducted, it is established that the activities of the Pulacayo Mining Project will have the following main environmental impacts:

- Alteration of air quality by the generation of particulate matter.
- Change in air quality by generating combustion gases.

These environmental impacts have negative characteristics with an assessment of moderate to critical, therefore specific measures need to be done, whether preventive, corrective or compensatory according to each case for mitigation.

In the case of particulate matter generated by drilling and blasting, the effects can be minimized by implementing a ventilation system inside the mine, which must be consistent with the engineering of the project, can be from simple chimneys to systems of injection and extraction of air. In any case, it should be verified that the final output does not exceed the permissible limits of quality and aspects such as population and downwind direction should be considered. There is also the application of wet drilling techniques and the implementation of dust traps at the mouth of the drills.

20.4.1.2 Soil

Among the most significant environmental impacts that would affect the ground, we have:

- Changes to soil quality by inappropriate disposal of domestic solid waste.
- Changes to soil quality from spills of fuels and lubricants.
- Alteration of soil levels structure.
- Soil compaction.
- Alteration of soil quality due to the accumulation of waste material in dumps.
- Alteration of soil quality for tailings disposal.
- Changes to soil quality from spills of hazardous substances.
- Changes to soil quality by improper final disposal of special solid waste (accessories and spare parts for vehicles and machinery, discarded tires, non-hazardous solid sanitary waste, debris).
- Possible soil erosion due to loss of vegetation layer.
- Alteration of soil quality due to sedimentation of heavy metals from DAM (Mine Acid Drainage) and DAR (Rock Acid Drainage).

Negative impacts previously mentioned, only three reach the Critical category.

Among the preventive measures, it is intended to do:
• Implementation of the Plan of Assimilable Domestic Solid Waste Management, considering the implementation and operation of a manual sanitary landfill, implement recycling programs and reducing the generation of solid waste.

• Implementation of an accumulation Plan of smaller volume waste, considering the use of deposits or fills of sterile material with all appropriate environmental control measures and promoting re-vegetation in the waste deposit, once the deposit is closed.

• Application of closure and rehabilitation plan that considers encapsulation of sterile material and tails with topsoil after completion of the project life.

• Canalization and DAM directness to water reservoir.

20.4.1.3 Water

The water factor identified 40 negative impacts, of which only one is marked as Critical being the remainder as mostly Moderate.

The impacts rated as important are:

• Generation of acid mine water.

• Alteration of surface water regime.

• Generation of acidic water from rock dumps.

• Infiltration of rainwater through the waste rock dumps to the underground.

• Risk alteration of the quality of ground water by filtering pollutants in the tailings dam area (PDC).

Preventive measures are aimed at:

• Perform acidic water canalization to the water reservoir: In case of producing mine acid water by drilling by layers or water mass or when it reaches 100 meters of depth, which is considered the water table, this Water must be channeled to the water reservoir where it can be re-circulated for the concentration process in the plant.

• A recirculating water system will be implemented for the plant, this would promote the reuse of the process water and the water captured by the reservoir.

• The material from cuts and/or sterile materials should be stored in dried deposits arranged at an appropriate distance of 300 meters surrounding watercourses.

• The tailings disposal area must be an area with low permeability characteristics. Before tailings facility starts receiving paste, a compaction should be performed and if necessary waterproofing the sector with clay, obtaining the material from nearby sources. This measure will be analyzed during the design of the tailings dam facility.

20.4.1.4 Fauna

The major impacts identified for this factor are:

• Disturbance to terrestrial and aquatic fauna.

• Destruction and/or modification of habitat.
• Increased barrier effect (removal of wildlife, disruption of migration processes, and subdivision of populations).

• Establish of disease vector species.

Significantly, no impact on this factor is critical. Therefore, we present the following measures:

• Develop programs of importance on local wildlife.

• Construction of access roads that do not interfere with important existing ecosystems, home to the terrestrial and aquatic fauna, and signaling traffic areas of wild and domestic animals.

• Only the necessary vegetation should be removed and animal nests and local ecosystems preserved. Programs on the importance of local and regional biodiversity will be developed.

20.4.1.5 Vegetation

The impacts of this factor are:

• Modification of the floristic composition.

• Decrease of species of local use.

• Fragmentation of vegetation layer.

• Introduction of invasive plant species.

• Altered states of vegetation succession.

In this subject, none of the impacts was rated as Critical. Therefore, the following measures are presented:

• Training programs focused on the importance of local vegetation will be made.

• A plan for re-vegetation of intervened areas will be established.

• Plant barriers should be introduced and the reshaping of land forms.

• Non-native species will be forbidden in the area.

• Access roads shall not interfere with important existing ecosystems with peculiar vegetation types.

20.4.1.6 Social Media

The negative impacts associated with the social factor are:

• Damage to the lifestyle, routine, habits.

• Alteration of current land use

• Health effects of population

• Increased uncertainty (risk of accidents)
These impacts are quantified with critical values where the measures are preventive and/or corrective-compensatory

- In case of transfer of property or use of cropping or grazing areas for exploitation activities within the area of the concession, it will be negotiated with the affected the type of compensation.
- Signaling and isolation of risk pathways.
- Implementation and monitoring of good practices of defensive driving.

20.4.1.7 Economic Environment

No negative impact was identified on this factor therefore, no measures are proposed.

20.4.1.8 Cultural Environment

In the archeological factor it was identified an impact of negative character concerning the involvement of archaeological sites, for which it intends to undertake training to workers with the general objective of preventing the potential impact to archaeological and cultural heritage and, in the event of discovering archaeological remains (pottery, bones, metals, etc.), during mining, immediately inform it to the authorities of Archaeology and Museums Unit (UDAM) dependent on the Bolivian State.

20.4.2 Training Plan to Workers in Environmental Topics

The plan aims to inform all workers about the impact of the project and how it has been planned to prevent or mitigate. In this sense, training will be provided to educate on preventive and mitigation measures.

It is also in the process of work, a plan that includes the community members, so that they can be selected as mining operators for future mining activities. The inclusion of community members plan ranges from training in mining issues to the development of self-environmental monitoring activities.

20.4.3 Plan of Common Solid Waste Management

It is considered necessary and appropriate to assess the generation, handling and disposal of solid waste from domestic properties that as a product of the Mining Operation and establish steps for its proper management.

Actions shall be executed to minimize the negative impact of solid waste generated in the operating fronts, also should minimize adverse impacts on the environment and limit the exposure of humans and animals caused by solid waste.

20.4.4 Closing and Rehabilitation Plan

The closure of the mining operation involves the implementation of activities that are necessary to generate levels of security and protecting long-term environmental factors that were somehow affected by the mining and complex ore concentration and construction of the tails dam.
After completion of the project lifetime, the company will proceed to the cessation of mining activities and the concentrator plant. The work stoppage refers also to activities involving the evacuation of staff and working machinery.

20.5 Legislation Applicable

The Environment Law came into force in April 1992, its regulations became effective in December 1995. The new Mining Code was enacted in March 1997 and the Environmental Regulations for Mining Activities (RAAM) in August 1997.

The General Rules of Environmental Management (RGGA), approved by Supreme Decree No. 24176 provides in Article 59 that the Environmental Permit (LA – Licencia Ambiental) is the legal administrative document granted by the Competent Environmental Authority (CAA) to the Legal Representative (RL) of an activity, work or project, which guarantees compliance with all requirements under the Environment Law and the corresponding regulations on regards to procedures for prevention and environmental control.

The Article 60 of the aforementioned RGGA, determines that for legal and administrative purposes, have the status of environmental license: the Environmental Impact Statement (DIA), the Certificate of Dispensation (CD) and the Declaration of Environmental Adjustment (DAA).

The Mining Code in Title VII, Chapter I of the Environment (Arts. 84 to 90), establishes the concept of Environmental License (LA) only for mining and of indefinite character. This means that the environmental license, whether DIA, the CD or the DAA, must be included in an integrated manner, all licenses, permits or environmental requirements applicable to mining activities.

The environmental licenses are:

- The Certificate of Dispensation Category 4 (CD-C4)
- The Certificate of Dispensation Category 3 (CD-C3)
- The Environmental Impact Statement (DIA)
- The Declaration of Environmental Adjustment (DAA)

20.5.1 Certificate of Dispensation Category 4 (Cd-C4)

The Certificate of Dispensation Category 4 (CD-C4) is the corresponding environmental License to mining activities listed below:

a) Existing mining activities or new of: Land survey, Prospecting, Geological mapping, Geochemical prospecting, Aerial survey.

The Environment Authority is the Departmental one.

20.5.2 Certificate of Dispensation Category 3 (CD-C3)

The Certificate of Dispensation Category 3 (CD-C3) is the corresponding environmental license to mining activities listed below:

a) Mining exploration activities new or existing outside of protected areas, which comprise:
• Exploration geophysics
• Drilling and boring
• Exploration by wells, boxes, shafts and trenches (ditches and pits)
• Other screening methods that do not cause deforestation and whose activity involves opening paths, setting up camps, site preparation for the construction of drilling platforms, warehouses and deposits.

The CD-C3 environmentally authorizes only the implementation of the activities identified in the EMAP Form. The Environmental Authority for such Licenses is the Departmental.

20.5.3 Environmental Record (FA)

The Environmental record (Ficha Ambiental) is a technical document that marks the beginning of the process of Environmental Impact Assessment (EIA). From the Environmental record, it is defined the study category of the Assessment of Environmental Impact (EEIA) of the project, work or metallurgical mining activity, and therefore the type of study to be followed for the processing of the environmental license. The Environment Record displays information about the project, work or activity, the identification of key impacts and possible solutions to the negative impacts. It is advisable to start the process of categorization through the Environmental Record in the pre-feasibility stage of the project of the new activity, which was already addressed for this specific project.

20.5.4 Environmental Impact Statement (DIA)

The Environmental Impact Statement (DIA) is the corresponding Environmental License to new mining activities that are categorized one or two For the methodology of the Environmental Record. The procedure for obtaining the Environmental License involves the Environmental Impact Assessment (EIA) that begins with the categorization by the Environmental Record to define the class or category of Study of Environmental Impact Assessment (EEIA) required for the new activity. The process for obtaining the Environmental Impact Statement (EIS) involves Sectorial Environmental Authority (Ministry of Mining and Metallurgy) and a Competent Environmental Authority (Ministry of Environment and Water) who is in charge of issuing the license.

20.5.5 Environmental Permits and Licenses

The permits obtained are divided for the Exploration phase and the Operational phase. Chart 20.3 details the current, valid permits held by the Pulacayo Mining Project.
Table 20.3: Currently Valid Permissions

<table>
<thead>
<tr>
<th>Title</th>
<th>Author</th>
<th>Year</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>Exploration Stage</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Updated EMAP</td>
<td>MEDMIN</td>
<td>2011</td>
<td>The environmental permit for the Pulacayo Exploration project has the license No. 05120102 C3 CD EMAP 42/2011.</td>
</tr>
<tr>
<td>License for Activities with Hazardous Substances</td>
<td>MEDMIN</td>
<td>2011</td>
<td>The License for Activities with Hazardous Substances (LASP) for Pulacayo Exploration Project has the license Nº051201_LASP_019/2011.</td>
</tr>
<tr>
<td>Sixth Report of Environmental Monitoring (IMA II/2012)</td>
<td>MEDMIN</td>
<td>2013</td>
<td>In January 2013, delivery was made of the Sixth Report of Environmental Monitoring (IMA) for the second half of 2012 Pulacayo Exploration Project to the Autonomous Government of the Potosí Department.</td>
</tr>
<tr>
<td>Exploitation Stage</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Environmental Record Draft of Extraction and Concentration of Complex Minerals and Pulacayo tailings dam construction</td>
<td>MEDMIN</td>
<td>2011</td>
<td>In August of 2011 it was obtained the categorization of the Project Environmental Tab by MMAYA Report - VMA - DGMACC - FA No. 4011 (b)/11. Indicating that the Study of Environmental Impact Assessment (IAES) corresponds to a Specific Analytical Study (AE).</td>
</tr>
</tbody>
</table>

Source: MEDMIN, 2013.

20.6 Environmental Achievements

The environmental achievements for Pulacayo Mining Project include:

- Updating the EMAP Form in 2011, this document contains information concerning the mineralogical material extraction from Pulacayo hill, in order to make external metallurgical testing with a maximum of 2000 tons per month.

- License for Activities with Hazardous Substances (LASP) in 2011 A license authorizing the transport, storage, preparation, use and disposal of hazardous substances involved in the stage of Mining Exploration.
Acceptance of the Public Consultation, on the month of December 2012. The Pulacayo Mining Project has been endorsed for implementation by the 22 communities and two existing committees in the project area.

Delivery of the Study of Environmental Impact Assessment - Specific Analytical (EEIA-AE) to the Ministry of Mining and Metallurgy as the first requirement for obtaining the environmental license of Pulacayo Mining Project. It is important to note that the authorization to be received will be updated in the future to process 1000 tons of ore per day.

20.7 Social, Community and Cultural Aspects

The Article 162 of the Regulations for the Prevention and Environmental Control (RPCA) indicates that at the stage of identification of impacts to be considered in an EEIA, a Public Consultation shall be organized to take into account comments, suggestions and recommendations of the public who may be affected by the implementation of the project, work or activity.

In October of 2012 the socialization of the Pulacayo Mining Project was conducted, performing the query at 2 ayllus where the project Pulacayo Mining is located, within the ayllus, the query was made to 22 communities and two commissions, the detail of the commissions and communities visited is:

a) Ayllu Aransaya: (Community Huanchaca, Comunity Tolapampa, Challa-Jankoyo, Totora K, Sivingani, Chifluyo, Ubina, Parkajisi, Arislaka, Vila Vililque, Cerdas, Sullchi, Noel Mariaca, Calerias y Jamachuma)

b) Ayllu Urinsaya: (Escara, Kulla, Lequepata, Jaruma, Chita, Oro Ingenio, Colchani y Chacala)

Both in the socialization and in the process of Public Consultation, it was explained the scope of Pulacayo Mining Project, potential environmental impacts that could arise with the implementation of the mining operation and how these will be mitigated to prevent damage to the surrounding communities.

In the public consultation process undertaken in December 14th of 2012, which has received the support of all the communities involved, having been present for validation the original authorities and authorities representing ayllus ministries and municipalities involved.

20.8 Agreements with Institutions

ASC Bolivia LDC has a joint venture agreement with Minera Pulacayo Cooperative Ltd. who is the lessee by the Mining Corporation of Bolivia (COMIBOL) of mining concessions Pulacayo, Galeria General, Porvenir and Huanchaca concessions where the Pulacayo Mining Project will operate.

Under the joint venture agreement between ASC Bolivia LDC and Pulacayo Mining Cooperative Ltd., the incorporation of the Company Apogee Minerals Bolivia S.A. was agreed, having the express permission of the Cooperative and the Mining Corporation of Bolivia (COMIBOL).
20.9 Recommendations

The recommendations on the Pulacayo Mining Project are aimed at:

- Keeping the communities and native authorities and the ones of the ayllus informed regularly about the progress and gradual changes of the mining project.
- Comply with the Section 151 of Regulation on Prevention and Environmental Control (RPCA) which states that the legal representative shall submit to the Competent Environmental Authority, technical annual reports that show the progress and the environmental situation of the project.
- Conduct a thorough inspection to the Environmental Mitigation Plans in order to reduce the probabilities of environmental incidents within areas of influence of the mining project, also make changes and updates to the plans when required.

21 CAPITAL AND OPERATING COSTS

The Pulacayo Pb-Zn-Ag Project scope covered in this Feasibility Study (FS) is based on the construction of a greenfield facility having a nominal ore processing capacity of 1,000 t/d. The Capital and Operating Cost estimates related to the underground mine, concentrator, and site infrastructure, have been developed by a number of independent consultants and integrated by TWP. The capital and operating costs for the Pulacayo Project have been prepared in accordance with standard industry practices for this level of study and to a level of definition and intended accuracy of ±15%.

21.1 Capital Cost Estimate

21.1.1 Summary of Total Capital Cost

The capital cost estimates have been split into two categories: Initial (upfront) and sustaining (ongoing) capital costs. The split between initial capital and sustaining capital was chosen as month 26 of the project. Hence all capital required from month 27 is classified as sustaining capital. This was the suggestion of the Mining QP, J Porter, with the justification that at this point the cumulative losses (earnings before taxes and depreciation) became zero. In month 26, the concentrator is not complete and so there is approximately $1.4 million of capital outstanding to complete the process plant. The sustaining capital is carried over operating year 3 through year 13 (6 months, 12.5 years). Table 21.1 presents the total estimated initial and sustaining capital costs for the Project. All cost estimates are expressed in constant US Dollars.

The estimated capital cost (upfront and sustaining) to design, construct, install and commission the facilities is USD 87 087 734 (see Table 21.1). This amount covers the direct costs of executing the project, plus indirect costs associated with construction management and commissioning. Base pricing is second quarter (Q2), 2012 American dollars (USD), with no allowance for inflation and escalation beyond that time.
<table>
<thead>
<tr>
<th>Operating Year</th>
<th>Calendar Year</th>
<th>-</th>
<th>y-1</th>
<th>y-2</th>
<th>y-3</th>
<th>y-4</th>
<th>y-5</th>
<th>y-6</th>
<th>y-7</th>
<th>y-8</th>
<th>y-9</th>
<th>y-10</th>
<th>y-11</th>
<th>y-12</th>
<th>y-13</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>TOTAL CAPEX,USD</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
| Process Plant, USD |              | 14,274,815 | 5,938,323 | 5,881,224 | 2,455,268 | - | - | - | - | - | - | - | - | -  
| Tailings Storage Facility (TSF), USD | | 6,171,994 | 85,708 | 716,683 | 372,245 | 130,322 | 472,276 | 609,053 | 574,956 | 197,037 | 168,147 | 359,584 | 325,984 | 1,440,000 | 720,000 |
| Mining Development, USD | | 22,242,001 | 2,070,440 | 3,053,562 | 2,865,535 | 2,549,050 | 2,395,078 | 3,486,731 | 1,438,803 | 973,956 | 973,956 | 973,956 | 973,956 | 486,976 | -  
| Mining Equipment , USD | | 12,084,787 | 2,347,570 | 602,874 | 1,173,553 | 1,487,532 | 554,645 | 863,733 | 862,865 | 1,450,865 | 1,001,442 | 413,442 | 862,865 | 281,165 | 192,233 |
| Mining Services , USD | | 5,151,849 | 2,125,062 | 607,161 | 579,141 | 551,121 | 551,121 | 551,121 | 551,121 | - | - | - | - | - | -  
| Mining Backfill System, USD | | 3,425,535 | 1,229,147 | 877,962 | 146,166 | 292,331 | 217,478 | 142,625 | 145,687 | 148,748 | 112,128 | 75,509 | 37,754 | - | -  
| Site Development, USD | | 2,217,287 | 2,207,287 | - | 8,571 | 1,429 | - | - | - | - | - | - | - | - | -  
| Powerline, USD | | 4,229,439 | 4,229,439 | - | - | - | - | - | - | - | - | - | - | - | -  
| EPCM, USD | | 3,011,384 | 1,927,286 | 653,900 | 430,198 | - | - | - | - | - | - | - | - | - | -  
| Freight Taxes & Ins.,USD | | 1,408,951 | 369,439 | 323,259 | 145,490 | 60,080 | 60,080 | 60,080 | 60,080 | 60,080 | 60,080 | 60,080 | 60,080 | 30,040 | -  
| Inventory & Commissioning, USD | | 1,505,692 | 219,010 | 1,109,226 | 177,457 | - | - | - | - | - | - | - | - | - | -  
| Owners Costs, USD | | 6,354,500 | 1,395,000 | 1,395,000 | 840,850 | 286,700 | 286,700 | 286,700 | 286,700 | 286,700 | 286,700 | 286,700 | 286,700 | 143,350 | -  
| Sub Total CAPEX, USD | | 82,442,233 | 24,143,709 | 15,220,851 | 9,194,473 | 5,358,565 | 4,537,378 | 6,000,043 | 3,920,211 | 3,117,867 | 2,602,455 | 2,169,272 | 2,547,341 | 2,554,924 | 1,075,623 |
| Contingency, USD (*) | | 4,845,502 | 464,550 | 2,787,301 | 1,393,651 | - | - | - | - | - | - | - | - | - | -  
| Life of Mine (LOM) CAPEX, USD | | 87,087,734 | 24,608,259 | 18,008,152 | 10,588,124 | 5,358,565 | 4,537,378 | 6,000,043 | 3,920,211 | 3,117,867 | 2,602,455 | 2,169,272 | 2,547,341 | 2,554,924 | 1,075,623 |
| CAPEX, USD/ore processed | | 24.8 | - | 216.97 | 58.82 | 21.52 | 14.93 | 16.67 | 10.89 | 8.66 | 7.23 | 6.03 | 7.08 | 7.10 | 5.98 |
| **Initial (upfront) CAPEX, USD** | | | | | | | | | | | | | | | |
| Process Plant, USD | | 12,790,234 | 5,938,323 | 5,881,224 | 970,687 | - | - | - | - | - | - | - | - | - | -  
| Tailings Storage Facility (TSF), USD | | 923,700 | 85,708 | 716,683 | 121,310 | - | - | - | - | - | - | - | - | - | -  
| Mining Development, USD | | 5,842,924 | 2,070,440 | 3,053,562 | 518,922 | - | - | - | - | - | - | - | - | - | -  
| Mining Equipment, USD | | 3,633,881 | 2,347,570 | 602,874 | 683,436 | - | - | - | - | - | - | - | - | - | -  

*This table represents the project capital costs for Pulacayo 1 000 t/d Phase I Feasibility Study.*
<table>
<thead>
<tr>
<th>Operating Year</th>
<th>-</th>
<th>y-1</th>
<th>y-2</th>
<th>y-3</th>
<th>y-4</th>
<th>y-5</th>
<th>y-6</th>
<th>y-7</th>
<th>y-8</th>
<th>y-9</th>
<th>y-10</th>
<th>y-11</th>
<th>y-12</th>
<th>y-13</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining Services, USD</td>
<td>2,833,416</td>
<td>2,125,062</td>
<td>607,161</td>
<td>101,193</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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</tr>
<tr>
<td>Mining Backfill System, USD</td>
<td>2,107,109</td>
<td>1,229,147</td>
<td>877,962</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Site Development, USD</td>
<td>2,207,287</td>
<td>2,207,287</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Powerline, USD</td>
<td>4,229,439</td>
<td>4,229,439</td>
<td>-</td>
<td>-</td>
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<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>EPCM, USD</td>
<td>2,753,266</td>
<td>1,927,286</td>
<td>653,900</td>
<td>172,079</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Freight Taxes &amp; Ins., USD</td>
<td>738,877</td>
<td>369,439</td>
<td>323,259</td>
<td>46,180</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Inventory &amp; Commissioning, USD</td>
<td>1,349,745</td>
<td>219,010</td>
<td>1,109,226</td>
<td>21,510</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Owners Costs, USD</td>
<td>3,022,500</td>
<td>1,395,000</td>
<td>1,395,000</td>
<td>232,500</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Sub Total Initial CAPEX, USD</td>
<td>42,232,377</td>
<td>24,143,709</td>
<td>15,220,851</td>
<td>2,867,818</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Contingency, USD (*)</td>
<td>3,716,401</td>
<td>464,550</td>
<td>2,787,301</td>
<td>464,550</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Total Initial CAPEX, USD</td>
<td>45,948,779</td>
<td>24,435,692</td>
<td>18,180,719</td>
<td>3,332,368</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Total Initial CAPEX, USD / ore t processed</td>
<td>13.1</td>
<td>-</td>
<td>216.97</td>
<td>18.51</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

### Sustaining Capex, USD

<p>| Process Plant, USD | 1,484,581 | - | - | 1,484,581 | - | - | - | - | - | - | - | - | - |
| Tailings Storage Facility (TSF), USD | 5,248,294 | - | - | 250,935 | 130,322 | 472,276 | 609,053 | 574,955 | 197,037 | 168,147 | 359,584 | 325,984 | 1,440,000 | 720,000 |
| Mining Development, USD | 16,599,077 | - | - | 2,346,613 | 2,549,050 | 2,395,078 | 3,486,731 | 1,438,803 | 973,956 | 973,956 | 973,956 | 973,956 | 486,978 | - |
| Mining Equipment, USD | 8,450,907 | - | - | 490,117 | 1,487,532 | 554,645 | 863,733 | 862,865 | 1,450,865 | 1,001,442 | 413,442 | 862,865 | 281,165 | 182,233 |
| Mining Services, USD | 2,882,431 | - | - | 477,947 | 551,121 | 551,121 | 551,121 | 551,121 | - | - | - | - | - | - |
| Mining Backfill System, USD | 1,318,427 | - | - | 146,166 | 292,331 | 217,478 | 142,625 | 145,687 | 148,748 | 112,128 | 75,509 | 37,754 | - | - |
| Site Development, USD | 10,000 | - | - | 8,571 | 1,429 | - | - | - | - | - | - | - | - | - |
| Powerline, USD | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| EPCM, USD | 258,119 | - | - | 258,119 | - | - | - | - | - | - | - | - | - | - |
| Freight Taxes &amp; Ins, USD | 670,074 | - | - | 99,310 | 60,080 | 60,080 | 60,080 | 60,080 | 60,080 | 60,080 | 60,080 | 60,080 | 30,040 | - | - |</p>
<table>
<thead>
<tr>
<th>Operating Year</th>
<th>y-1</th>
<th>y-2</th>
<th>y-3</th>
<th>y-4</th>
<th>y-5</th>
<th>y-6</th>
<th>y-7</th>
<th>y-8</th>
<th>y-9</th>
<th>y-10</th>
<th>y-11</th>
<th>y-12</th>
<th>y-13</th>
</tr>
</thead>
<tbody>
<tr>
<td>Calendar Year</td>
<td></td>
<td></td>
<td></td>
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<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Inventory &amp; Commissioning, USD</td>
<td>155,947</td>
<td>-</td>
<td>-</td>
<td>155,947</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Owner Costs, USD</td>
<td>3,332,000</td>
<td>-</td>
<td>-</td>
<td>608,350</td>
<td>286,700</td>
<td>286,700</td>
<td>286,700</td>
<td>286,700</td>
<td>286,700</td>
<td>286,700</td>
<td>286,700</td>
<td>286,700</td>
<td>286,700</td>
</tr>
<tr>
<td>Sub Total Sustaining CAPEX, USD</td>
<td>40,209,855</td>
<td>-</td>
<td>-</td>
<td>6,326,655</td>
<td>5,358,565</td>
<td>4,537,378</td>
<td>6,000,043</td>
<td>3,920,211</td>
<td>3,117,387</td>
<td>2,602,455</td>
<td>2,169,272</td>
<td>2,547,341</td>
<td>2,554,924</td>
</tr>
<tr>
<td>Contingency, USD</td>
<td>929,100</td>
<td>-</td>
<td>-</td>
<td>929,100</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Total Sustaining CAPEX, USD</td>
<td>41,138,956</td>
<td>-</td>
<td>-</td>
<td>7,255,756</td>
<td>5,358,565</td>
<td>4,537,378</td>
<td>6,000,043</td>
<td>3,920,211</td>
<td>3,117,387</td>
<td>2,602,455</td>
<td>2,169,272</td>
<td>2,547,341</td>
<td>2,554,924</td>
</tr>
<tr>
<td>Total Sustaining CAPEX, USD/ton processed</td>
<td>11.7</td>
<td>-</td>
<td>-</td>
<td>35.15</td>
<td>21.52</td>
<td>14.93</td>
<td>16.67</td>
<td>10.89</td>
<td>8.66</td>
<td>7.23</td>
<td>6.03</td>
<td>7.08</td>
<td>7.10</td>
</tr>
</tbody>
</table>

(*) LOM CAPEX contingency is 10% of the LOM initial CAPEX
21.1.2 Initial (Upfront) Capital Cost Estimate

The initial capital cost estimate includes direct costs, indirect costs, contingency and owner’s costs covering all of the traditional items typical of this type of project. Owner’s costs were provided by Apogee Silver Ltd.

The capital cost estimate was developed for this Project by:

- TWP Sudamérica: Process plant and related surface infrastructure and site services
- TWP Projects (RSA): Mining and mine services
- BBE Consulting: Mine ventilation
- RRD International Corporation: Tailings Storage Facility (TSF)
- EPCM Consultores: Power supply and distribution
- Apogee Silver Ltd: Owner’s costs
- Klohn Crippen Berger: Water storage dam

The estimated capital cost to design, construct, install and commission the facilities is USD 45,948,779 (see Table 21.1).

21.1.1.1 Basis of Estimate

The basis of estimate is summarized in Table 21.2

<table>
<thead>
<tr>
<th>Item</th>
<th>Description</th>
<th>Detail</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Budget Quotations for Process Equipment, Mine Equipment Fleet, Mine Ventilation Equipment and Underground Workshop Equipment</td>
<td>The vendors supplied an equipment price, delivery lead times, freight costs and spare parts allowance. When no detailed information was available, TWP’s database was used. These costs, if appropriate, are covered in the indirect section of the estimate.</td>
</tr>
<tr>
<td>2</td>
<td>Labor Rates</td>
<td>Based on quotes from Bolivian contractors</td>
</tr>
<tr>
<td>3</td>
<td>Labor Productivity</td>
<td>Based on weather conditions, altitude, access to working area, availability of labor, etc. This is included in Bolivian contractor’s quotations. Labor rates have an associated productivity factor that reflects the nature of the work and the working conditions. Productivity factors (or loss of productivity in the field) were established based on TWP’s database, contract information and benchmarking.</td>
</tr>
<tr>
<td>4</td>
<td>Infrastructure/Vendor Packages</td>
<td>They are based on the cost of infrastructure (water treatment plant, septic tank, camp, etc.) for construction and operation. When quotations were not available, they were estimated by using ratios according to the TWP database of similar projects</td>
</tr>
<tr>
<td>5</td>
<td>Civil Works</td>
<td>The volumes of concrete and formwork areas come directly from the general arrangements drawings (plan and elevation views). The steel reinforcement is obtained by ratios based on volume of concrete and the item (footings, walls, slabs, columns, walls, beams, sumps). Concrete unit rates were developed using supply and installation information provided</td>
</tr>
<tr>
<td>Item</td>
<td>Description</td>
<td>Detail</td>
</tr>
<tr>
<td>------</td>
<td>-------------</td>
<td>--------</td>
</tr>
<tr>
<td>6</td>
<td>Structural Steel</td>
<td>The weight of structural steel (metallic structures) is obtained from design models (SAP 2000) and the bolts and connections are estimated according to weight ratios. Handrails and grating panels are measured from the plant layout and the grout is calculated according to the section of the column.</td>
</tr>
<tr>
<td>7</td>
<td>Tanks and Platework (Chutes and Pump Boxes)</td>
<td>Quantities for platework and metal liners for tanks, pump boxes and chutes have been calculated from sketches and provided in kilograms of steel. Unitary prices used were obtained from provider and from the TWP’s database, as appropriate.</td>
</tr>
<tr>
<td>8</td>
<td>Piping/Valves</td>
<td>Piping and fittings quantities are based in quantity take-off with prices (USD/m) budgetary quotes. The quantity take-offs are developed from pipe routing drawings based on the general arrangement drawings and the P&amp;ID’s which identifies pipe size. The list of materials (MTO’s) have been developed for the main pipes only (as shown in the piping route GA). These pipes have been costed based on this information and the available quotes for materials/accessories. The rest of the pipes for each processing area will be costed based on a percentage of the total pipes capital costs.</td>
</tr>
<tr>
<td>9</td>
<td>Electrical</td>
<td>Power cables costs have been estimated by using rates (USD/m). Electrical costs have been developed from the related study deliverables of others disciplines and are based on the single line diagram (substations, switchgear, switchboards, power transformers and motor control centers), drawings and sketches. The mechanical equipment list was used to estimate plant loading and site power requirements. The equipment list in conjunction with the site plan was used to determine electrical building locations by centralizing electrical infrastructure to minimize cable runs. Major electrical rooms are located within the large process buildings and infrastructure. Electrical equipment and cable (caliber length) prices are supplied by providers. TWP’s database has been used as appropriate. Field instrumentation costs are derived from in-house pricing data base and recent pricing obtained from recent similar projects.</td>
</tr>
<tr>
<td>10</td>
<td>Material/Quantity Take-offs (MTO’s)</td>
<td>Material take-offs were prepared for all facilities. In addition to the equipment list and other engineering discipline deliverables, the engineering disciplines are also responsible to generate bulk materials take-offs for estimating. A material take-off is the process of analyzing drawings and determining all the materials required to complete the design (list of materials, quantities, types, grades of materials and the weights of such materials).</td>
</tr>
<tr>
<td>11</td>
<td>Instrumentation</td>
<td>It is based on the list of Instruments and Control Equipment, quotations from suppliers and/or ratios that are accordance with the TWP database of similar projects. Fields instrumentation costs are derived from in-house pricing database and recent pricing obtained from recent similar project.</td>
</tr>
<tr>
<td>Item</td>
<td>Description</td>
<td>Detail</td>
</tr>
<tr>
<td>------</td>
<td>-------------</td>
<td>--------</td>
</tr>
<tr>
<td>12</td>
<td>EPCM</td>
<td>It is based on quotes from contractor's and/or ratios in accordance with the TWP database of similar projects</td>
</tr>
<tr>
<td>13</td>
<td>Indirect Costs</td>
<td>It is based on quotes from suppliers and/or ratios that are according to the TWP database of similar projects</td>
</tr>
</tbody>
</table>

21.1.1.2 Direct Costs

Direct costs include all new equipment, material, and installation of permanent facilities associated with the following:

- Mine equipment fleet
- Mine ventilation equipment
- Underground workshop equipment
- Backfill plant/system equipment
- Crushing, material handling and processing facilities
- Process buildings
- Plant mobile equipment
- Infrastructure roads and site preparation
- Power supply and distribution
- Concentrate load out
- Pre-production mining
- Tailings storage area
- Warehousing
- Administration
- Truck shop (underground mine)
- Yard services and other utilities
- Control and communications systems
- Fuel storage
- Magazine (explosives storage)

The direct costs are referred to the expenses directly incurred during the physical development of the project, including the processing equipment, materials take-off (MTO's), rates for construction (labor for construction and supervision), construction equipment, contract services, transportation costs, assembly and installation of infrastructure, civil works metered (concrete, structural steel, coverage), piping, electric equipment (substations, transmission and distribution lines).
a) Direct Labor
The labor cost was calculated using the Bolivian regulations on social laws and benefits established for employees in civil and mining works. All of the labor costs are based in the following criteria:

- Cost based on a working day
- Payroll burdens and overtime shift premium rates
- Social laws
- Working week of 48 hours
- Tools and supplies
- Indirect costs of the contractor on site and at head office
- Contractor profits
- Labour productivity

The labor rates assumed that all contractor craft personnel will be Bolivian. Temporary power and catering were excluded from the labour rates and changed to the project indirect costs.

b) Materials and Rates for Construction
The prices for materials and rates at unitary prices for the execution of diverse works in the project construction are determined using the available information in the TWP’s database and the rates used for contractors in the execution of the different works that TWP contracts at present time.

c) Civil Works
The list of materials drifts from the plans and schemes of project engineering. The estimations are based in the usually accepted practice in this type of study.

Earth Movement Works
The estimates of the amounts for earth movements include the effect of volume lost by the compaction of the material. In general, earthworks quantities were generated from site layouts (cut and fill, drilling and blasting, etc.).

The estimation of the volume quantity of earth moving is subject to two parameters; the influence area and depth of the foundation structure. These parameters are calculated based on the structure weight and the ground bearing capacity.

The measurement unit of the earth movement cost is based on the ratio USD/m³. These ratios are in agreement with the TWP database for similar projects.

The earth movement costs and platform construction for the plant and plant infrastructure, as well as the internal access construction have been assumed by the client (Sunk Costs).
The corresponding budget for these works have been already disbursed by the Client and are therefore Sunk Costs for the present project and will not be considered in the calculation of the capital cost in the feasibility study.

The earth movement cost and platform construction that has not been built has been estimated from drawings and quantities derived from the drawings.

**Concrete Works**

Concrete and formwork quantities were determined based on the mechanical arrangements – plan/elevation view drawings and sketches. No allowances for over-pour, wastage or rework were considered. The unitary concrete prices are all-in (concrete reinforcing steel and formwork). Additionally, the data of the Soil Mechanics Study (Geotecnia, Tarifa Fernandez S.R.L., Bolivia, Julio 2011) was taken into consideration to calculate the bearing capacity and depth of the concrete foundations.

To size the foundations for vibrating equipment, a ratio in the range of 1 to 3 (for each ton of equipment, three tons of concrete is used) was used.

For foundations for structures, the concrete dimensioning is subject to a weight ratio of the structure (kg) per foundation area (m²) given that this relationship should not be below the soil bearing capacity.

Reinforcing steel is obtained by ratios (kg/m³) according to the volume of concrete and structural elements (footings, slabs, columns, walls, beams). The unit of measure of the cost of concrete is based on ratios in USD/m³. These ratios are in according to the TWP database of similar projects.

The concrete price included supply of concrete from an on-site batch plant.

For slabs, the quantity of concrete to be used depends on the application. For pedestrian traffic, concrete is 15 cm thick, for light traffic, concrete is 20 cm thick. If the slab is to be used for heavy machinery traffic, the concrete thickness is in the range of 25 cm to 30 cm.

d) **Structural Steel Works**

Structural steel quantities were developed from general arrangement drawings and experience from previous projects of similar nature. Unit rates (USD/kg) have been provided by regional contractors. The weights include allowances for connections, stiffeners, clips, base plates, etc.

e) **Equipment (Process and Mechanical)**

The equipment was classified and budgeted from the Equipment List. The providers quotations are a fundamental part in the selection of the equipment considered in this study (main and secondary equipment). As is necessary, most of the equipment and material will be imported due to a lack of availability in Bolivia. Where equipment is available in Bolivia, it will be procured locally. Imported equipment will be shipped by ocean, rail and trucks. Transport costs have been considered as indirect costs and have been estimated from quotations from logistics companies operating in South America. Equipment already purchased by Apogee was excluded (Maelgwyn flotation cells, Andritz press filters, and Pengfei comminution equipment).
f) Tanks, Chutes and Platework

Tanks, chutes and platework were included in the Equipment List. Platework was priced on a cost per kilogram basis. Field fabricated tank costs were based on budget quotations for supply and erection from a fabricator. These ratios are in agreement with TWP’s database of other similar projects.

g) Piping

Piping material quantities were based on the process flowsheets and general piping arrangements. The unit prices include acquisition, transport and handling and storage in the project warehouses. Material pricing and installation man-hours and productivity were provided by local contractors. The unitary prices are in USD/m. These ratios are in agreement with the TWP database of other similar projects.

h) Electrical Equipment

Electrical costs were based on the electrical equipment list (including substations, switchgear and switchboards) the general single line diagram, and the cable list. The mechanical equipment list was used to cross check for equipment or packages requiring power. Electrical equipment and cables pricing was based on vendor quotations.

i) Instrumentation Equipment

The cost estimate for instrumentation was based on the Instrumentation List and the Control Equipment Schedules derived from P&ID drawings. The main prices of the instrumentation equipment were taken from supplier quotations and/or ratios that are in agreement with the TWP database of other similar projects (the cost of cables, fibre optics and other minor materials is considered in the estimated cost per instrument).

j) On-site Infrastructure Capital Cost Estimate

Project primary and auxiliary buildings lists and sketches were used to identify the size of the facilities. Costs for construction of the buildings were supplied by contractors (quotations). An additional factor was added to the construction materials quotation to account for furnishing, where required.

k) Off-site Infrastructure Capital Cost Estimate

Off-site infrastructure consists of:

- Incoming high voltage power line (priced by EPCM Consultores)
- Water line, sourced from public water main

l) Tailings Storage Facility (TSF) Capital Cost Estimate

The TSF capital cost consists of mainly costs for earthworks associated with the preparation and the construction of the facility. These costs have been estimated by RRD International Corporation.

m) Taxes and Duties

The Value Added Tax (VAT) is excluded from Capital Cost Estimate. Import duties were allowed for in the estimate by taking and average percentage of the equipment value and adding an average duty amount. VAT has been included in the cost estimate because in Bolivia it can take two years to claim VAT back from the government.
21.1.1.3 Indirect Costs

Includes all expenditures for Engineering, Procurement and Construction Management (EPCM) services, third party consultants, construction site operation, vendor support and spares.

In summary, the indirect costs of the project comprise the following:

- Engineering, procurement and construction management (EPCM) services
- Construction indirect costs
  - Construction Management/Services
  - Temporary Construction Buildings/Facilities
  - Temporary Construction Equipment/Tools
  - Catering
  - Vendor Representatives
- Commissioning and start-up allowance
- Car/trucks/light vehicles rental
- IT equipment and software
- Transport costs (freight and duties)
- Consumables (first fills and capital spare parts)
- Erection (construction)
- Quality assurance
- Owner’s costs: The owner’s cost typically include: pre-operations personnel and training, mine equipment, mine development, owner’s project team, initial fills, insurance, housing, permitting, commissioning, general and administrative costs for project support and management, environmental and permitting activities and corporate and owner’s contingency. An estimate of owner’s costs to support the Project from project initiation to plant commissioning has been provided by Apogee based on their current head office operating costs (La Paz offices). It is assumed that these owners costs will cover both underground mine construction and construction of surface installations for the project.

21.1.1.4 Contingency

Project contingency is an integral component of the cost estimated of a project. It is required to cover the unwanted aspects (inherent error or omissions and undefined cost elements), lack of precision in design, changes in estimates for equipment purchases, variation in labour rates and any unforeseeable costs within the scope of the estimate.

Contingency does not cover changes in the scope of studies and services or project exclusions. Additionally, contingency is not intended to take into account items such as labour disruptions, weather related impediments, price escalation, currency fluctuation and allowances for force majeure.
The percentage considered for the present study has been set as 10% over the LOM Initial CAPEX.

21.1.1.5 Assumptions and Exclusions

Assumptions

The following assumptions were made in preparing the estimate:

- Construction work is based on unit and fixed price contracts (no cost plus or time and materials arrangements)
- Budget quotes from vendors for equipment and materials are valid to within ± 5% of the purchase price
- Cement, aggregate and sand will be available locally to produce concrete
- Soil conditions will be adequate for foundation bearing pressures
- Construction activities will be carried out in a continuous program
- Labor productivities have been validated with input from experienced contractors
- Bulk materials such as rebar, structural steel and plate, cable, cable tray and piping are all readily available in the scheduled timeframe.
- Capital equipment is available in the timeframes shown since availability has been verified by suppliers

Project Currency

All capital costs are expressed in Q2, 2012 United States Dollars (USD). Costs submitted in other currencies have been converted to USD by using the rate of exchange quoted by Vendors. No provision has been made for any taxes or fees applicable to currency charges or for forward cover insurance during the project.

Accuracy

The capital cost estimate for the underground mine, process plant, tailings storage facility and infrastructure has been prepared to a level of ± 15%.

Exclusions

The following items are not included or covered by the estimates of the capital cost estimates:

- Owner sunk costs (acquired equipment, platforms built, exploration works performed, exploration tunnel and drifts constructed, etc.)
- Changes in the design and technical specifications
- Modifications in the land conditions
- Costs of complementary studies
- Additional requirements for permits and licenses
• Exploration or sampling costs
• Working Capital
• Costs to accelerate the construction schedule (additional costs for accelerated deliveries of equipment, materials and services resultant from a change in project schedule)
• Royalties
• Financial costs and interests
• Refundable taxes and duties
• Land acquisition
• Warehouse inventories (other than these supplied in first fills)
• Risk due to the political and social situation, change in the government politics, work conflicts, delays in obtaining operating permit, etc.
• Mine reclamation costs
• Mine closure costs
• Salvage values
• Permitting costs
• Currency fluctuations
• Allowance for lost time due to force majeure
• Owner’s cost (except as provided by the Client)
• Community relations
• Escalation
• Schedule delays and associated costs

21.1.3 Sustaining Capital Cost Estimate

Sustaining capital over the mine life totals 41,138,956 USD and includes:
• On-going tailings storage facility development (capacity upgrade)
• Mine capital development
• Mine capital equipment replacement
• Mine auxiliary services
• Backfill system (underground reticulation system)
• Transport costs (freight, taxes and insurance)
• Purchasing management and capital spares
21.2 Operating Cost Estimate

21.2.1 Summary of Total Operating Costs

The operating cost expenditure (OPEX) for the Project is defined as all the operating costs incurred from the start of the Project, after the initial capital spend, until the last ore tonne has been mined out. OPEX costs include labour costs, running costs for equipment, consumables costs, utilities and any other overhead costs incurred during the life of the operation.

The operating cost estimate includes operating costs for the underground mine and cemented backfill plant, mineral processing plant, surface infrastructure including the tailings disposal facility and general and administrative costs (G&A) costs for the integrated operation. No contingency for the Project OPEX has been included. The integrated life-of-mine (LoM) OPEX amounts to 192,961,787 USD (54.9 USD/t ore processed). The OPEX cost is presented in Table 21.3. Mining operating costs account for 56% of the total OPEX cost, whilst process and G&A account for 28% and 16%, respectively (see Figure 21.1).

Figure 21.1 provides the composition of the OPEX cost expressed as a percentage of the total operating cost.

![Figure 21.1 Breakdown of LOM Operating Costs](image-url)
Table 21.3: Project Operating Costs

<table>
<thead>
<tr>
<th>Operating Year</th>
<th>y-1</th>
<th>y-2</th>
<th>y-3</th>
<th>y-4</th>
<th>y-5</th>
<th>y-6</th>
<th>y-7</th>
<th>y-8</th>
<th>y-9</th>
<th>y-10</th>
<th>y-11</th>
<th>y-12</th>
<th>y-13</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tonnes Mined to Stockpile, t</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Development Ore, t</td>
<td>448,830</td>
<td>9,762</td>
<td>56,178</td>
<td>78,702</td>
<td>80,717</td>
<td>74,058</td>
<td>194,414</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Total Waste, t</td>
<td>839,887</td>
<td>36,072</td>
<td>162,058</td>
<td>119,344</td>
<td>38,137</td>
<td>106,003</td>
<td>378,273</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Metal Production (Ex mine)</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ag Grade, g/t</td>
<td>239</td>
<td>129</td>
<td>244</td>
<td>332</td>
<td>365</td>
<td>341</td>
<td>272</td>
<td>262</td>
<td>198</td>
<td>199</td>
<td>191</td>
<td>191</td>
<td>173</td>
</tr>
<tr>
<td>Pb Grade, %</td>
<td>1.09</td>
<td>0.63</td>
<td>1.71</td>
<td>1.66</td>
<td>1.77</td>
<td>1.44</td>
<td>1.28</td>
<td>1.53</td>
<td>0.80</td>
<td>0.77</td>
<td>0.72</td>
<td>0.69</td>
<td>0.81</td>
</tr>
<tr>
<td>Zn Grade, %</td>
<td>1.91</td>
<td>1.37</td>
<td>2.61</td>
<td>3.27</td>
<td>3.05</td>
<td>2.18</td>
<td>1.89</td>
<td>2.13</td>
<td>1.40</td>
<td>1.56</td>
<td>1.63</td>
<td>1.66</td>
<td>1.65</td>
</tr>
<tr>
<td>Ag, Oz</td>
<td>27,385,190</td>
<td>47,075</td>
<td>767,242</td>
<td>1,826,019</td>
<td>3,044,585</td>
<td>3,116,811</td>
<td>3,547,892</td>
<td>3,044,585</td>
<td>2,362,971</td>
<td>2,319,851</td>
<td>2,222,524</td>
<td>2,251,479</td>
<td>2,042,721</td>
</tr>
<tr>
<td>Pb, t</td>
<td>38,927</td>
<td>71</td>
<td>1,673</td>
<td>2,828</td>
<td>4,576</td>
<td>4,083</td>
<td>5,202</td>
<td>5,519</td>
<td>2,928</td>
<td>2,809</td>
<td>2,615</td>
<td>2,528</td>
<td>2,979</td>
</tr>
<tr>
<td>Zn, t</td>
<td>67,905</td>
<td>155</td>
<td>2,552</td>
<td>5,581</td>
<td>7,916</td>
<td>6,182</td>
<td>7,663</td>
<td>7,702</td>
<td>5,124</td>
<td>5,650</td>
<td>5,912</td>
<td>6,093</td>
<td>6,057</td>
</tr>
<tr>
<td>Total Tons Milled from Stockpiles, t</td>
<td>3,516,000</td>
<td>83,000</td>
<td>180,000</td>
<td>249,000</td>
<td>304,000</td>
<td>360,000</td>
<td>360,000</td>
<td>360,000</td>
<td>360,000</td>
<td>360,000</td>
<td>360,000</td>
<td>360,000</td>
<td>360,000</td>
</tr>
<tr>
<td>Silver grade milled, g/t</td>
<td>240</td>
<td>-</td>
<td>240</td>
<td>326</td>
<td>366</td>
<td>340</td>
<td>272</td>
<td>264</td>
<td>207</td>
<td>199</td>
<td>192</td>
<td>191</td>
<td>176</td>
</tr>
<tr>
<td>Pb Grade milled, %</td>
<td>1.10</td>
<td>-</td>
<td>1.64</td>
<td>1.67</td>
<td>1.77</td>
<td>1.45</td>
<td>1.28</td>
<td>1.50</td>
<td>0.90</td>
<td>0.78</td>
<td>0.73</td>
<td>0.69</td>
<td>0.79</td>
</tr>
<tr>
<td>Zn Grade milled, %</td>
<td>1.92</td>
<td>-</td>
<td>2.44</td>
<td>3.25</td>
<td>3.14</td>
<td>2.18</td>
<td>1.89</td>
<td>2.10</td>
<td>1.50</td>
<td>1.53</td>
<td>1.62</td>
<td>1.66</td>
<td>1.65</td>
</tr>
<tr>
<td>Silver Metal Processed (not recovered), Oz</td>
<td>27,173,397</td>
<td>-</td>
<td>639,945</td>
<td>1,887,227</td>
<td>2,933,347</td>
<td>3,323,087</td>
<td>3,153,548</td>
<td>3,052,631</td>
<td>2,392,163</td>
<td>2,298,889</td>
<td>2,223,948</td>
<td>2,211,034</td>
<td>2,037,556</td>
</tr>
<tr>
<td>Operating Year</td>
<td>y-1</td>
<td>y-2</td>
<td>y-3</td>
<td>y-4</td>
<td>y-5</td>
<td>y-6</td>
<td>y-7</td>
<td>y-8</td>
<td>y-9</td>
<td>y-10</td>
<td>y-11</td>
<td>y-12</td>
<td>y-13</td>
</tr>
<tr>
<td>----------------</td>
<td>-----</td>
<td>-----</td>
<td>-----</td>
<td>-----</td>
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<td>-----</td>
<td>------</td>
<td>------</td>
<td>------</td>
<td>------</td>
</tr>
<tr>
<td>Pb Metal Processed (not recovered), t</td>
<td>38,633</td>
<td>-</td>
<td>1,365</td>
<td>3,013</td>
<td>4,413</td>
<td>4,412</td>
<td>4,623</td>
<td>5,387</td>
<td>3,229</td>
<td>2,800</td>
<td>2,629</td>
<td>2,502</td>
<td>2,842</td>
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<td>Zn Metal Processed (not recovered), t</td>
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<td>-</td>
<td>2,024</td>
<td>5,841</td>
<td>7,813</td>
<td>6,641</td>
<td>6,809</td>
<td>7,563</td>
<td>5,395</td>
<td>5,522</td>
<td>5,836</td>
<td>5,967</td>
<td>5,941</td>
</tr>
</tbody>
</table>

### Mining Costs, USD

| | Development, USD | 15,780,102 | 1,388,618 | 1,859,631 | 2,062,834 | 1,652,911 | 2,105,519 | 2,062,834 | 887,556 | 734,355 | 734,355 | 653,715 | 446,153 | 172,081 |
| | Production, USD | 48,807,116 | 1,459,962 | 2,687,597 | 3,524,569 | 4,162,930 | 4,467,090 | 4,704,235 | 4,664,940 | 4,714,153 | 4,680,675 | 4,014,227 | 4,004,323 | 1,399,534 |
| | Technical Services, USD | 17,545,815 | 1,675,568 | 1,742,165 | 1,814,000 | 1,990,334 | 2,035,947 | 2,035,947 | 964,932 | 964,553 | 964,932 | 965,128 | 965,603 | 477,599 |
| | Backfill System, USD | 22,620,903 | - | 676,515 | 1,653,777 | 2,187,200 | 2,229,912 | 2,235,498 | 2,294,600 | 2,999,718 | 2,667,901 | 2,726,139 | - | - |
| | Mine Administration, USD | 3,014,408 | 214,961 | 214,961 | 246,142 | 246,142 | 246,142 | 246,142 | 246,142 | 246,142 | 246,142 | 246,142 | 123,071 | - |
| Total OPEX Cost, USD | 107,768,344 | 4,739,109 | 7,180,869 | 9,301,321 | 10,239,517 | 10,924,905 | 11,332,697 | 9,097,007 | 9,560,100 | 9,659,427 | 9,293,626 | 8,605,351 | 5,662,220 | 2,172,284 |
| OPEX, USD/ore t processed | 30.65 | - | 86.5 | 51.7 | 41.1 | 35.9 | 31.5 | 25.3 | 26.6 | 26.8 | 25.8 | 23.9 | 15.7 | 12.1 |

### Processing Cost, USD

<p>| | Operating Supplies, USD | 36,330,645 | - | 857,635 | 1,859,931 | 2,572,904 | 3,141,216 | 3,719,861 | 3,719,861 | 3,719,861 | 3,719,861 | 3,719,861 | 1,859,931 |
| | Maintenance Supplies, USD | 147,821 | - | 3,490 | 7,568 | 10,469 | 12,781 | 15,135 | 15,135 | 15,135 | 15,135 | 15,135 | 7,568 |
| | Manpower, USD | 7,196,725 | - | 169,889 | 368,433 | 509,666 | 622,242 | 736,866 | 736,866 | 736,866 | 736,866 | 736,866 | 368,433 |</p>
<table>
<thead>
<tr>
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<th>y-1</th>
<th>y-2</th>
<th>y-3</th>
<th>y-4</th>
<th>y-5</th>
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<th>y-11</th>
<th>y-12</th>
<th>y-13</th>
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</thead>
<tbody>
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<td>693,264</td>
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<td>1,002,310</td>
<td>1,002,310</td>
<td>1,002,310</td>
<td>501,155</td>
</tr>
<tr>
<td>Total OPEX Cost, USD</td>
<td>53,464,419</td>
<td>1,262,101</td>
<td>2,737,086</td>
<td>3,786,303</td>
<td>4,622,635</td>
<td>5,474,173</td>
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<td>5,474,173</td>
<td>5,474,173</td>
<td>5,474,173</td>
<td>2,737,086</td>
</tr>
<tr>
<td>OPEX, USD/ore t processed</td>
<td>15.2</td>
<td>15.2</td>
<td>15.2</td>
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<td>15.2</td>
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<td>15.2</td>
</tr>
</tbody>
</table>

**General & Admin**

- **G&A Labour, USD** | 6,679,875 | 534,390 | 534,390 | 534,390 | 534,390 | 534,390 | 534,390 | 534,390 | 534,390 | 534,390 | 534,390 | 534,390 | 267,195 |
- **G&A Power, USD** | - | - | - | - | - | - | - | - | - | - | - | - | - |
- **Third Party Services: Catering, Camp Cleaning, USD** | 2,828,750 | 226,300 | 226,300 | 226,300 | 226,300 | 226,300 | 226,300 | 226,300 | 226,300 | 226,300 | 226,300 | 226,300 | 113,150 |
- **External Assays, USD** | - | - | - | - | - | - | - | - | - | - | - | - | - |
- **External Consulting And Software, USD** | 7,750 | 620 | 620 | 620 | 620 | 620 | 620 | 620 | 620 | 620 | 620 | 620 | 310 |
- **Equipment And Vehicle Rental (Heavy And Light), USD** | - | - | - | - | - | - | - | - | - | - | - | - | - |
- **Safety/Protective Clothing (EPP), USD** | 187,500 | 15,000 | 15,000 | 15,000 | 15,000 | 15,000 | 15,000 | 15,000 | 15,000 | 15,000 | 15,000 | 15,000 | 7,500 |
- **Postage, Courier And Light Freight, USD** | 250,000 | 20,000 | 20,000 | 20,000 | 20,000 | 20,000 | 20,000 | 20,000 | 20,000 | 20,000 | 20,000 | 20,000 | 10,000 |
- **Office/Computer Supplies/Maintenance** | 125,000 | 10,000 | 10,000 | 10,000 | 10,000 | 10,000 | 10,000 | 10,000 | 10,000 | 10,000 | 10,000 | 10,000 | 5,000 |
<table>
<thead>
<tr>
<th>Operating Year</th>
<th>y-1</th>
<th>y-2</th>
<th>y-3</th>
<th>y-4</th>
<th>y-5</th>
<th>y-6</th>
<th>y-7</th>
<th>y-8</th>
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<th>y-11</th>
<th>y-12</th>
<th>y-13</th>
</tr>
</thead>
<tbody>
<tr>
<td>OPEX Cost, USD</td>
<td></td>
<td></td>
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<td>20,000</td>
<td>20,000</td>
<td>20,000</td>
<td>20,000</td>
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<td>20,000</td>
<td>20,000</td>
<td>20,000</td>
</tr>
<tr>
<td>Travel/ Accommodation/ Camp, USD</td>
<td>625,000</td>
<td>50,000</td>
<td>50,000</td>
<td>50,000</td>
<td>50,000</td>
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<td>50,000</td>
<td>50,000</td>
<td>50,000</td>
</tr>
<tr>
<td>Access And Internal Roads Maintenance, USD</td>
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<td>1,000</td>
<td>1,000</td>
<td>1,000</td>
<td>1,000</td>
<td>1,000</td>
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<td>40,000</td>
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<td>10,000</td>
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</tr>
<tr>
<td>Site Security (External), USD</td>
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<td>50,000</td>
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<td>44,520</td>
<td>44,520</td>
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</tr>
<tr>
<td>Total OPEX Cost, USD</td>
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<td>2,538,322</td>
<td>2,538,322</td>
<td>2,538,322</td>
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</tr>
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<td>y-4</td>
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<td>y-9</td>
<td>y-10</td>
<td>y-11</td>
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<tr>
<td>OPEX, USD/ore t mined</td>
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<td>14.9</td>
<td>9.79</td>
<td>8.94</td>
<td>6.27</td>
<td>7.02</td>
<td>6.95</td>
<td>6.99</td>
<td>7.01</td>
<td>6.92</td>
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<td>-</td>
<td>30.6</td>
<td>14.1</td>
<td>10.2</td>
<td>8.35</td>
<td>7.05</td>
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<tr>
<td>Mining, USD</td>
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<td>7,180,869</td>
<td>9,301,321</td>
<td>10,239,517</td>
<td>10,924,905</td>
<td>11,332,697</td>
<td>9,097,007</td>
<td>9,560,010</td>
<td>9,659,427</td>
<td>9,293,626</td>
<td>8,605,351</td>
<td>5,662,220</td>
</tr>
<tr>
<td>Processing, USD</td>
<td>53,464,419</td>
<td>-</td>
<td>1,262,101</td>
<td>2,737,086</td>
<td>3,786,303</td>
<td>4,622,635</td>
<td>5,474,173</td>
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<td>5,474,173</td>
<td>5,474,173</td>
<td>5,474,173</td>
<td>2,737,086</td>
</tr>
<tr>
<td>General &amp; Admin., USD</td>
<td>31,729,025</td>
<td>2,538,322</td>
<td>2,538,322</td>
<td>2,538,322</td>
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<td>2,538,322</td>
<td>1,269,161</td>
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<tr>
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<td>7,277,431</td>
<td>10,981,292</td>
<td>14,576,729</td>
<td>16,564,142</td>
<td>18,085,862</td>
<td>19,345,191</td>
<td>17,109,502</td>
<td>17,572,505</td>
<td>17,671,921</td>
<td>17,306,121</td>
<td>16,617,845</td>
<td>13,674,714</td>
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<td>OPEX, USD/ore t mined</td>
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<td>112.2</td>
<td>85.3</td>
<td>63.9</td>
<td>63.7</td>
<td>47.7</td>
<td>47.3</td>
<td>48.1</td>
<td>48.6</td>
<td>47.8</td>
<td>45.3</td>
<td>37.2</td>
</tr>
<tr>
<td>OPEX, USD/ore t processed</td>
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<td>-</td>
<td>132.3</td>
<td>81.0</td>
<td>66.5</td>
<td>59.5</td>
<td>53.7</td>
<td>47.5</td>
<td>48.8</td>
<td>49.1</td>
<td>48.1</td>
<td>46.2</td>
<td>38.0</td>
</tr>
</tbody>
</table>
21.2.2 Basis of Estimate

The underground mine operating cost estimate is determined from multiple modeling approaches, life cycle costing for mining fleet, zero based costing for the mining consumables, first principles costing for the labour and an activity base costing approach for the remainder of the cost elements. The estimate is derived from two models, the mining fleet model, which forecasts the maintenance and replacement costs for the entire mining fleet and all other vehicles owned by the operation, and the consolidated cost model, which consolidates all other cost components.

The mining OPEX costs consist of the following:

- Mining and backfill consumables and materials (drill steel, drill bits and explosives, backfill reagents: flocculant, cement, wear parts, repair parts, etc.)
- Utilities (power, diesel, air, etc.)
- Ventilation maintenance schedule
- Mining fleet maintenance (Predicted Operating Costs (POC), Ground Engaging Tools (GET) replacements, etc.
- Underground infrastructure maintenance (maintenance costs for piping, electrical equipment, cable, lighting, instrumentation, etc.)
- Technical service consumables (fixed monthly costs for the consumables required for the maintenance of gas detection equipment incurred every three years)
- Labour (derived based on the production requirement for each time period, a leave allowance of 12% was catered for over and above the labour complement)
- Mine administration consumables and transport (fixed monthly costs based on costs rates provided by Apogee)

The processing OPEX costs consist of:

- Operating supplies
  - Wear parts (grinding media, liners, filter cloths, etc.)
  - Reagents and consumables
  - Services and utilities (energy, engine oil, diesel, etc.)
- Maintenance supplies, engine oil and lubricants
- Labour
- General and administration costs include the personnel costs for management and administrative support functions, insurance, legal service office expenses, external assays, camp catering and maintenance, accommodation and transportation (crew, staff flights, etc.) costs. Environmental testing and studies are also considered. G&A costs are in accordance with the Owner’s Costs by Apogee. The following items are included:
  - Third Party Services: Catering, Camp Cleaning, Maintenance and Refuse Removal
The following sources were used to derive the mining and processing OPEX estimate:

- Design Criteria
- Vendor data (quotations received from Bolivian and overseas suppliers)
- Client data
- Operating practice and industry standards
- Engineering Handbooks
- Equipment capital-based factors (direct capital cost) for maintenance supplies cost estimation
- Labour Survey
- Allowances for supplies (operating and maintenance), general and administration cost estimation
- Mechanical Equipment List
- Plant Power Demand
- Plant Water Mass Balance
- Mechanical Drawings and General Arrangements
21.2.3 Assumptions & Exclusions

Assumptions

a) Project Currency

Operating cost estimates were prepared on a money basis with all cash flows expressed in second quarter (2Q) 2012 terms. Results are expressed in United States dollars (USD) with underlying items estimated in the applicable source currency. Costs estimated in different currencies other US dollars, were converted to USD at an exchange rate quoted by Vendors.

b) Accuracy

The expected accuracy range of the operating cost estimates for this study is ± 15%

Exclusions

The operating costs related to the following items have been excluded:

- Exploration and assessment of the viability of other potential ore resources
- Mining activities (including exploration and pre-production mining costs)
- In-country or overseas corporate head office, financing, legal, banking, insurance, accounting costs and charges. However, La Paz (Bolivia) office costs were included.
- Insurance, shipping costs and refining charges for the concentrates
- Royalties, Value Added Tax (VAT), income taxes or similar imposts
- Enterprise fees, licenses, land use and water use
- Activities covered by the sustaining capital and closure/rehabilitation provisions
- Accuracy provisions to account for changes in currency exchange rate variations
- Tailings Storage Facility (TSF) operating costs

The off-reef development consumable costs for drilling, blasting, ducting and rock bolts as well as development fleet maintenance and fuel costs have been excluded from OPEX and have been considered in the CAPEX estimate. Fleet replacement and overhaul costs form part of the Sustaining Capital estimate.
22 ECONOMIC ANALYSIS

22.1 Introduction

A Technical-Financial Model (TFM) has been developed to evaluate the economic viability of Pulacayo mine. To complete the model, information was collected from the mine to determine the value chain that its products will follow from the mine to the market. This information covered commercial information, engineering, production, process and cost data such as on-mine costs and sales terms.

22.2 Assumptions

The following assumptions have been made:

- All the costs included in the TFM model and this report are expressed in terms of 2012 quarter four US dollars (USD) and do not include escalation or inflation (i.e. the model is in constant real money terms).
- Revenue is recognized as the concentrate is produced. No allowance has been made for time delays between shipment and payment for the concentrate. It is assumed that the concentrate will be sold as it is produced or an inventory arrangement be made.
- Ocean freight, concentrate containerization and terminal costs have not been accounted for.
- The initial capital expenditure will be financed by equity and revenue.
- Operating cash flows will be used to finance sustaining and working capital once the mine has reached full production.
- The metal price estimates used in the model are tabulated below:

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<th>Metal</th>
<th>Price and Price Participation</th>
</tr>
</thead>
<tbody>
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<td>0.86 $/lb</td>
</tr>
<tr>
<td>Zn</td>
<td>1.00 $/lb</td>
</tr>
<tr>
<td>Ag</td>
<td>28.0 $/oz</td>
</tr>
</tbody>
</table>

22.3 Financial Modeling

22.3.1 The Mine Value Chain of Pulacayo mine

The purpose of the Technical-Financial Model (TFM) is to evaluate a mine in the context of the mine value chain that its products follow from the mine to the market. The value chain that the products of Pulacayo mine will follow is described below:
• The Run-of-Mine (ROM) will be trucked out of the mine and tipped at the stockpile at the process plant. The average grades of the RoM product delivered to the plant have been estimated to be 240 g/t for silver, 1.10% for lead and 1.92% for zinc over the life of mine.

• From the stockpile, the ROM material will pass through a beneficiation plant comprising a crushing circuit, a milling and cycloning circuit, two sequential flotation stages (the first for lead and the second for zinc) and a filtration plant, producing a lead concentrate and a zinc concentrate. The contents and grade of each concentrate are tabulated below:

<table>
<thead>
<tr>
<th></th>
<th>Lead Concentrate</th>
<th>Zinc Concentrate</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb (%)</td>
<td>48</td>
<td>1.12</td>
</tr>
<tr>
<td>Zn (%)</td>
<td>6.53</td>
<td>51</td>
</tr>
<tr>
<td>Ag (g/t)</td>
<td>7.438</td>
<td>411</td>
</tr>
<tr>
<td>Fe (%)</td>
<td>-</td>
<td>2.19</td>
</tr>
<tr>
<td>Si (%)</td>
<td>-</td>
<td>1.93</td>
</tr>
<tr>
<td>As (%)</td>
<td>1.48</td>
<td>0.03</td>
</tr>
<tr>
<td>Sb (%)</td>
<td>1.25</td>
<td>0.04</td>
</tr>
<tr>
<td>Bi (%)</td>
<td>0.03</td>
<td>-</td>
</tr>
</tbody>
</table>

• The two concentrates will then pass through the sales process. The quantities of concentrate and the cost per equivalent silver ounce over the mine life are shown in the table below:
Table 22.3: Concentrate produced and the Cost per Equivalent Silver Ounce Over the Mine Life

<table>
<thead>
<tr>
<th>Project Year</th>
<th>Units</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
<th>7</th>
<th>8</th>
<th>9</th>
<th>10</th>
<th>11</th>
<th>12</th>
<th>13</th>
</tr>
</thead>
<tbody>
<tr>
<td>Lead Concentrate</td>
<td>tonnes</td>
<td>0</td>
<td>2,064</td>
<td>5,761</td>
<td>8,147</td>
<td>8,236</td>
<td>8,523</td>
<td>9,975</td>
<td>5,808</td>
<td>4,296</td>
<td>3,860</td>
<td>3,501</td>
<td>4,284</td>
<td>2,569</td>
</tr>
<tr>
<td>Zinc Concentrate</td>
<td>tonnes</td>
<td>0</td>
<td>2,479</td>
<td>9,629</td>
<td>12,977</td>
<td>10,545</td>
<td>10,637</td>
<td>11,887</td>
<td>8,203</td>
<td>8,057</td>
<td>8,844</td>
<td>9,202</td>
<td>9,088</td>
<td>3,355</td>
</tr>
<tr>
<td>Silver in Concentrate</td>
<td>Oz'000</td>
<td>0</td>
<td>416</td>
<td>1,552</td>
<td>2,368</td>
<td>2,583</td>
<td>2,463</td>
<td>2,429</td>
<td>1,671</td>
<td>1,425</td>
<td>1,337</td>
<td>1,278</td>
<td>1,284</td>
<td>723</td>
</tr>
<tr>
<td>Silver Base Metal Credits ¹</td>
<td>ozequiv.'000</td>
<td>0</td>
<td>167</td>
<td>574</td>
<td>786</td>
<td>691</td>
<td>704</td>
<td>802</td>
<td>518</td>
<td>463</td>
<td>483</td>
<td>504</td>
<td>218</td>
<td></td>
</tr>
<tr>
<td>Silver produced with Base Metal Credits ¹</td>
<td>ozequiv.'000</td>
<td>0</td>
<td>583</td>
<td>2,125</td>
<td>3,154</td>
<td>3,274</td>
<td>3,167</td>
<td>3,231</td>
<td>2,189</td>
<td>1,888</td>
<td>1,818</td>
<td>1,761</td>
<td>1,788</td>
<td>941</td>
</tr>
<tr>
<td>Cost per Equivalent Silver Ounce</td>
<td>US$/oz</td>
<td>19.24</td>
<td>7.85</td>
<td>6.20</td>
<td>6.36</td>
<td>7.02</td>
<td>6.38</td>
<td>8.99</td>
<td>10.30</td>
<td>10.55</td>
<td>10.51</td>
<td>8.87</td>
<td>8.73</td>
<td></td>
</tr>
</tbody>
</table>

Notes:
1) The application of "silver equivalent ounces", (OzAgEq) means the US dollar value of lead & zinc metals divided by the price of silver and added to the pure silver ounces in any applicable category. Economic calculations used the following factors: lead equivalent ounces is Lead AgEq. = (Lead Tonnes x 2204lbs/t x $0.86/lb) / $28/oz and for zinc equivalent ounces is Zinc AgEq. = (Zinc Tonnes x 2204lbs/t x $1.00/lb) / $28/oz.
22.3.3 The Methodology of Modeling (after the Mine Value Chain)

Starting with a Free-on-Board (FoB) price, the off-mine cost and the concentrate transportation cost were deducted to arrive at a Free-on-Truck (FoT) mine gate price.

The steady state operating cost varies between about $35 and $53 per tonne of the mill feed over the mine life. The difference between the FoT price and the operating cost is the Operating Margin (before Royalties). The Operating margin was used to calculate the Earnings Before Interest, Tax and Dividends (EBITD). Royalties were calculated for each concentrate based on the rates shown in Table 22.3 below and the metal contained in the concentrate as well as the Net Smelter Revenue (NSR) as may be applicable.

<table>
<thead>
<tr>
<th>Table 22.4: Royalty Rates</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Lead Concentrate</strong></td>
</tr>
<tr>
<td>Regalia Minera - Pb</td>
</tr>
<tr>
<td>Regalia Minera - Zn</td>
</tr>
<tr>
<td>Regalia Minera - Ag</td>
</tr>
<tr>
<td>Comibol</td>
</tr>
<tr>
<td>Co-Operative</td>
</tr>
</tbody>
</table>

The Operating Margin or EBITD (after Royalties) was calculated. The capital expenditure was then deducted to yield a Cash Flow before Tax. The taxable amount was calculated taking depreciation into account. Two taxation rates were applied in the TFM. The first is a corporate tax of 25% which is payable once both corporate tax and the cumulative corporate tax become positive. The other is a mining tax rate of 12.5%. It becomes payable when the local income turns positive. The Cash Flow after Tax was then calculated. The final result of the TFM are the Net Present Value (NPV) and Internal Rate of Return (IRR) calculations of the Operating Margin and Cash Flows before and after Taxes.

22.4 Results of the Technical Financial Model

The TFM is based on a mine using conventional mining on Level Zero and trackless mining on the other levels. The Capital Cost associate with Ore Reserve Development has been taken into account in the TFM. Ore Reserve Development is a Sustaining Capital Investment that is made to accommodate future low commodity prices permitting ongoing development to be temporarily halted without loss of production. This investment enables a mine to maintain a positive cash flow when commodity prices are low. The TFM model is attached as Appendix K in the Feasibility Study Report (Document No: 090644-3-0000-20-IFI-116). A summary of the results obtained from the model is shown in Table 22.5 below:
Table 22.5: Output Values from the TFM

<table>
<thead>
<tr>
<th>LOM Results</th>
<th>8%</th>
</tr>
</thead>
<tbody>
<tr>
<td>Constant Money Discount Rate- % pa</td>
<td>8%</td>
</tr>
<tr>
<td>Operating Margin</td>
<td>MUS$ 194</td>
</tr>
<tr>
<td>Cash before tax</td>
<td>MUS$ 126</td>
</tr>
<tr>
<td>Cash after tax</td>
<td>MUS$ 72.5</td>
</tr>
<tr>
<td>Maximum Month for NPV and IRR</td>
<td>174</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Notes:</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>1. (Operating costs plus [selling]shipping costs) / silver in concentrate produced, excluding royalties.</td>
<td></td>
</tr>
<tr>
<td>2. (Operating costs plus [selling]shipping costs) / (silver in concentrate produced + silver base metal credits). The application of &quot;silver equivalent ounces&quot;, (OzAgEq.) means the US dollar value of lead &amp; zinc metals divided by the price of silver and added to the pure silver ounces in any applicable category. Economic calculations used the following factors: lead equivalent ounces is Lead AgEq. = (Lead Tonnes x 2204lbs/t x $0.86/lb) / $28/oz and for zinc equivalent ounces is Zinc AgEq. = (Zinc Tonnes x 2204lbs/t x $1.00/lb) / $28/oz</td>
<td></td>
</tr>
</tbody>
</table>

The payback period for the mine has been calculated to be 20 months. This period is based on the metal prices shown in Table 22.1. It does not take into account interest on loans.

The cash flow of the project over the mine life is tabulated below:
Table 22.6: CashFlow of the Project Over the Mine Life

<table>
<thead>
<tr>
<th>Project Year</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
<th>7</th>
<th>8</th>
<th>9</th>
<th>10</th>
<th>11</th>
<th>12</th>
<th>13</th>
<th>14</th>
<th>15</th>
<th>16</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gross Revenue Before Royalties</td>
<td>US$'000</td>
<td>0</td>
<td>13,432</td>
<td>49,002</td>
<td>73,087</td>
<td>76,822</td>
<td>74,174</td>
<td>75,188</td>
<td>51,013</td>
<td>43,746</td>
<td>41,780</td>
<td>40,274</td>
<td>40,993</td>
<td>21,979</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Royalties</td>
<td>US$'000</td>
<td>0</td>
<td>1,470</td>
<td>5,370</td>
<td>8,002</td>
<td>8,380</td>
<td>8,091</td>
<td>8,210</td>
<td>5,574</td>
<td>4,793</td>
<td>4,591</td>
<td>4,434</td>
<td>4,503</td>
<td>2,400</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Gross Revenue After Royalties</td>
<td>US$'000</td>
<td>0</td>
<td>11,962</td>
<td>43,632</td>
<td>65,085</td>
<td>68,442</td>
<td>66,083</td>
<td>66,978</td>
<td>45,440</td>
<td>38,953</td>
<td>37,189</td>
<td>35,839</td>
<td>36,490</td>
<td>19,580</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Shipping (Selling) costs</td>
<td>US$'000</td>
<td>0</td>
<td>697</td>
<td>2,289</td>
<td>3,155</td>
<td>2,864</td>
<td>2,929</td>
<td>3,357</td>
<td>2,117</td>
<td>1,818</td>
<td>1,836</td>
<td>1,815</td>
<td>1,946</td>
<td>902</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Operating cost</td>
<td>US$'000</td>
<td>6,648</td>
<td>10,514</td>
<td>14,405</td>
<td>16,387</td>
<td>17,958</td>
<td>19,314</td>
<td>17,270</td>
<td>17,559</td>
<td>17,270</td>
<td>17,559</td>
<td>17,270</td>
<td>17,559</td>
<td>17,270</td>
<td>17,559</td>
<td>17,270</td>
</tr>
<tr>
<td>Earnings Before Income Tax and Dividends (EBITB)</td>
<td>US$'000</td>
<td>-6,648</td>
<td>752</td>
<td>26,939</td>
<td>45,543</td>
<td>47,621</td>
<td>43,840</td>
<td>46,350</td>
<td>25,764</td>
<td>19,495</td>
<td>18,016</td>
<td>17,324</td>
<td>20,627</td>
<td>11,365</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Income Tax</td>
<td>US$'000</td>
<td>0</td>
<td>515</td>
<td>2,683</td>
<td>4,995</td>
<td>14,259</td>
<td>14,404</td>
<td>15,353</td>
<td>7,736</td>
<td>5,510</td>
<td>5,067</td>
<td>4,915</td>
<td>6,212</td>
<td>3,399</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Net cash after Tax (before capital)</td>
<td>US$'000</td>
<td>-6,648</td>
<td>236</td>
<td>24,256</td>
<td>40,548</td>
<td>33,361</td>
<td>29,436</td>
<td>30,998</td>
<td>18,028</td>
<td>13,985</td>
<td>12,949</td>
<td>12,410</td>
<td>14,415</td>
<td>7,967</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Mining Capital Expenditure</td>
<td>US$'000</td>
<td>21,880</td>
<td>19,610</td>
<td>11,232</td>
<td>5,140</td>
<td>4,487</td>
<td>5,981</td>
<td>4,139</td>
<td>3,042</td>
<td>2,380</td>
<td>2,040</td>
<td>2,433</td>
<td>3,099</td>
<td>1,624</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Plant Capital Expenditure</td>
<td>US$'000</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Vat Tax</td>
<td>US$'000</td>
<td>2,967</td>
<td>3,205</td>
<td>-63</td>
<td>-638</td>
<td>-272</td>
<td>368</td>
<td>-56</td>
<td>-573</td>
<td>-304</td>
<td>-156</td>
<td>-93</td>
<td>-234</td>
<td>-1,155</td>
<td>-1,972</td>
<td>-1,023</td>
</tr>
<tr>
<td>Capital Expenditure &amp; VAT Tax</td>
<td>US$'000</td>
<td>24,847</td>
<td>22,816</td>
<td>11,170</td>
<td>4,502</td>
<td>4,215</td>
<td>6,349</td>
<td>4,082</td>
<td>2,469</td>
<td>2,076</td>
<td>1,884</td>
<td>2,341</td>
<td>2,864</td>
<td>468</td>
<td>-1,972</td>
<td>-1,023</td>
</tr>
<tr>
<td>Project Cash flow</td>
<td>US$'000</td>
<td>-31,495</td>
<td>-22,579</td>
<td>13,086</td>
<td>36,047</td>
<td>29,146</td>
<td>23,086</td>
<td>26,915</td>
<td>15,558</td>
<td>11,910</td>
<td>11,065</td>
<td>10,069</td>
<td>11,550</td>
<td>7,498</td>
<td>1,972</td>
<td>1,023</td>
</tr>
</tbody>
</table>
22.5 Sensitivity Analysis

Other than the metal price, the project does not seem to be sensitive to any specific parameter. Figure 22.1 depicts the sensitivity of NPV and IRR to the Silver Price.

![NPV & IRR Sensitivity to Silver Price](image)

**Figure 22.1: Silver Price Sensitivity**

23 ADJACENT PROPERTIES

There are no adjacent properties as defined by NI43-101 that are pertinent to the Pulacayo mineral resource estimate described in this report.

24 PROJECT EXECUTION PLAN

24.1 Project Outline

24.1.1 Introduction

The purpose of the Project Execution Plan (PEP) is to describe how the project will be developed, built and delivered to the production team through a project implementation process. The PEP also includes engineering, procurement, project management, project closure and risks mitigation plans.
The project will be designed and constructed to industry and regulatory standards, with emphasis on addressing environmental and safety issues. Adherence to the Project Execution Plan will ensure timely and cost effective completion, while ensuring quality is maintained.

The PEP is a comprehensive plan that includes the following:

- Engineering (Basic and Detailed)
- Procurement
- Construction
- Pre-commissioning and commissioning
- Start-up/ramp-up
- Handover to operator team (operational readiness)
- Project Closure

The PEP will be executed by a combination of teams:

- Contractor’s team
- Detailed Engineering Team
- Procurement Team
- Construction Management Team
- Owner's teams

24.1.2 Scope of Work

All work acquired for completion of the Pulacayo Project is covered in the Work Breakdown Structure (WBS) shown in the Table 24.1. The WBS contains a complete definition of the project’s scope and forms the basis for planning, execution and control.

Table 24.1: Work Breakdown Structure (WBS – level 1 only)

<table>
<thead>
<tr>
<th>Area</th>
<th>Description</th>
<th>Area</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>0</td>
<td>PROCESS PLANT (General)</td>
<td>3100</td>
<td>Entrance Attendance Office</td>
</tr>
<tr>
<td>100</td>
<td>Crushing Area</td>
<td>3110</td>
<td>Visitors Parking</td>
</tr>
<tr>
<td>200</td>
<td>Milling Circuit 1 Area</td>
<td>3120</td>
<td>Internal Plant Parking</td>
</tr>
<tr>
<td></td>
<td>Milling Circuit 2 Area</td>
<td>3130</td>
<td>Weighbridge</td>
</tr>
<tr>
<td>300</td>
<td>Bulk And Lead Flotation Circuit 1 Area</td>
<td>3150</td>
<td>Central Offices</td>
</tr>
<tr>
<td></td>
<td>Bulk And Lead Flotation Circuit 2 Area</td>
<td>3160</td>
<td>Laydown Area</td>
</tr>
<tr>
<td>400</td>
<td>Zinc Flotation Circuit 1 Area</td>
<td>3180</td>
<td>Workshops</td>
</tr>
<tr>
<td></td>
<td>Zinc Flotation Circuit 2 Area</td>
<td>3190</td>
<td>Warehouse</td>
</tr>
<tr>
<td>500</td>
<td>Separation Solid/Liquid</td>
<td>3200</td>
<td>Oil Storage</td>
</tr>
</tbody>
</table>
### Area Description

<table>
<thead>
<tr>
<th>Area</th>
<th>Description</th>
<th>Area</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>900</td>
<td>Concentrate Filtration Area</td>
<td>3210</td>
<td>Gas Storage</td>
</tr>
<tr>
<td>1000</td>
<td>Reagents Area</td>
<td>3220</td>
<td>Paint Store</td>
</tr>
<tr>
<td>1200</td>
<td>Process Water Area</td>
<td>3230</td>
<td>Nuclear Source Store</td>
</tr>
<tr>
<td>1300</td>
<td>Auxiliary Services</td>
<td>3240</td>
<td>Diesel Storage</td>
</tr>
<tr>
<td>1400</td>
<td>Tailings Disposal Area</td>
<td>3250</td>
<td>Potable Water Tank</td>
</tr>
<tr>
<td>1500</td>
<td>Backfill</td>
<td>3260</td>
<td>Fresh Water Tanks (Storage)</td>
</tr>
<tr>
<td>2000</td>
<td>GENERAL SITE DEVELOPMENT</td>
<td>3270</td>
<td>Main Site Fence</td>
</tr>
<tr>
<td>2010</td>
<td>Main Access Road</td>
<td>3300</td>
<td>Laboratory</td>
</tr>
<tr>
<td>2020</td>
<td>High Tension (HT) Electrical Supply (External)</td>
<td>3310</td>
<td>Laundry</td>
</tr>
<tr>
<td>2030</td>
<td>Main Low Tension (LT) Substation</td>
<td>3320</td>
<td>Changehouse</td>
</tr>
<tr>
<td>2040</td>
<td>Medium Tension (MT) Electrical Supply (external)</td>
<td>4000</td>
<td>MINE COSTS</td>
</tr>
<tr>
<td>2050</td>
<td>Main Water Supply (Yanapollera Reservoir)</td>
<td>4100</td>
<td>Capital Development</td>
</tr>
<tr>
<td>2060</td>
<td>Waste Dump Area</td>
<td>4300</td>
<td>Capital Equipment</td>
</tr>
<tr>
<td>2070</td>
<td>Railways</td>
<td>4400</td>
<td>Auxiliary Services</td>
</tr>
<tr>
<td>2080</td>
<td>Camp</td>
<td>5000</td>
<td>WORKING CAPITAL</td>
</tr>
<tr>
<td>3000</td>
<td>PROCESS PLANT SITE DEVELOPMENT</td>
<td>5100</td>
<td>HT Power Line Package</td>
</tr>
<tr>
<td>3010</td>
<td>Internal Plant Roads</td>
<td>6000</td>
<td>INDIRECT COSTS</td>
</tr>
<tr>
<td>3030</td>
<td>Plant Spillage Containment Dam</td>
<td>6100</td>
<td>Engineering, Procurement &amp; Construction Management (EPCM)</td>
</tr>
<tr>
<td>3040</td>
<td>Electrical Generators</td>
<td>6300</td>
<td>Freight, Taxes, and Insurance</td>
</tr>
<tr>
<td>3050</td>
<td>Septic Tank</td>
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<td>Start-up and Commissioning</td>
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<td>3090</td>
<td>Entrance Induction Office</td>
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### 24.1.3 Project Schedule

#### 24.1.3.1 Critical Success Factors

- Early freezing of the design, and commitment to procure long lead items.
- Rapid and clear communication lines between stakeholders.
- Clearly defined roles and responsibilities for project team members (owner and contractor).
• Definition and implementation of HSE (Health, Safety and Environment) plan prior to construction mobilization.
• The quality and continuity of the project team through the project phases.
• Scope change control.

24.1.3.2 Critical Schedule Activities

• Approval of all required permits.
• All stakeholders should be involved and aligned early (integrated team approach).
• Identify documentation to be created and updated.
• Power line construction
• Long lead items should be ordered early (slurry pumps, hydro cyclones, VFDs, substations, switchgears, switchboards, power transformers, etc.).
• Contracting strategy should be selected to avoid tendering delays.
• Driving design by constructability and operability requirements, to avoid delays and modifications.

24.1.3.3 Project Schedule

See Figure 24.1.

24.1.3.4 Project Milestones

Project milestones are summarized in Table 24.2.

<table>
<thead>
<tr>
<th>Item</th>
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<tr>
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<tr>
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<td>4th Quarter, 2013</td>
<td>2nd Quarter, 2014</td>
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<tr>
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<td>10</td>
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**Figure 24.1: Simplified Project Schedule**
25 INTERPRATION AND CONCLUSIONS

25.1 Project Feasibility

Much progress has been made in the development of the Pulacayo Project since the publishing of the Preliminary Assessment of the Pulacayo Project in 25 June 2010. Additional drilling has allowed for a significant increase of mineral reserves. Mineral reserves have now been established at 19.5 million ounces of silver in concentrate in the Pulacayo deposit. During the course of the Feasibility Study, a detailed analysis of numerous mine plan options was performed by TWP and the most cost effective plan was selected as the mine/waste production schedule for this study.

Extensive metallurgical test work highlighted the complexity of the Project deposit mineralization and resulted in reorienting the flotation step in the process flow sheet from bulk flotation process to conventional differential flotation process.

Furthermore, all environmental aspects and impacts were studied in detail and allowed for the development of a solid environmental management plan for tailings and waste disposal.

The operations were engineered to process ore at a full design rate of 1,000 tons per day through two modular circuits of 500 tons per day. Although, operations were fully permitted to process ore at a rate of 300 tons per day, Apogee Silver is in the process of getting the environmental permit to process ore at full design capacity (EIA extension).

The increase in the level of engineering developed during the course of the Feasibility Study allowed for significant de-risking. Thereafter, the capital cost and operating cost estimates associated with the Project have been developed to a significantly improved degree of accuracy.

The Feasibility Study demonstrates that the project is technically feasible and has a robust economic investment case.

25.1.1 Geology

The Pulacayo deposit is located on the western flank of the Cordillera Oriental, near the Cordillera-Altiplano boundary. The area is underlain by folded sedimentary and igneous rocks of Silurian, Tertiary and Quaternary ages that locally host low sulphidation epithermal polymetallic vein and stock work styles of mineralization.

Polymetallic (Ag-Pb-Zn) vein and disseminated wall rock mineralization at Pulacayo are controlled by east west trending secondary faults. The main mineralized trend was emplaced on the southern side of the Pulacayo dome complex and is best exemplified by the Tajo Vein System (TVS) that is the subject of this Feasibility Study.

The TVS trends east-west, dips 75° to 90° to the south along 2,700 m of defined surface strike length, and is present in mine workings at a depth of 1,000 m below surface. In the upper levels of the deposit, where volcanic strata host mineralization, the TVS consists of a stock work vein system that locally reaches 120 m in mineralized width.
Sedimentary strata host the vein system at depth, where narrower widths of 1 to 3 meters are apparent. Mineralization of economic interest is comprised of sphalerite, galena and tetrahedrite in sulphide-rich veins that are accompanied by locally abundant quartz, barite and pyrite. Disseminated mineralization is preferentially developed around and between veins hosted by andesite.

The Pulacayo deposit is well defined by recent core drilling by Apogee in combination with compiled extents of historic underground workings. Modeling of individual metal distribution and host rock lithology trends has provided sufficient definition of the main vein system to support feasibility level mine planning and geotechnical studies. Further core drilling within the currently defined deposit limits can be used to convert existing Inferred resources to Indicated status and good opportunities exist to extend deposit limits both along strike and at depth.

Further infill drilling is required to improve geological interpretations in order to better define stoping layouts. In order to advance the project to basic/detailed engineering stage, additional in-fill drilling should be completed. The information from this drilling would be used to complete reserve definition, geotechnical studies, and hydrogeologic investigations.

### 25.1.2 Mining

Pulacayo is an existing mine with a long history of development, recently on a small scale. This project plans to increase production significantly and exploit previously unmined material utilizing a combination of two methods. Shrinkage stoping will be used on zero level and longhole open stoping will be used on all levels below zero level. The stope widths vary from 3 m to 6 m, depending on the mining method and width of mineralization.

The mine has an existing tunnel (San Leon Tunnel) located on zero level which starts in the town of Pulacayo and exits on the northern side of the mountain that hosts the mineralized rock. This tunnel will continue to be used but a new decline ramp system will be constructed to allow access with trackless equipment and to exit closer to the mineral processing plant. This decline ramp system will be developed from surface (at an inclination of 8 degrees from the horizontal) using conventional drill and blast techniques.

Conventional drill and blast techniques will also be used to advance existing ore and waste drives, and to develop new shrinkage stopes. Air loaders will be used to clean development faces. The broken rock will be loaded into locomotives and trammed out of the mine. A face jumbo will be used to develop tunnels that are required to access the long hole stopes. These tunnels are the main haulage, crosscuts and ore drives. An LHD will be used to muck out, and the broken rock will be hauled out of the mine using the existing 15 tonne truck.

The development will take place by drilling a planned round of 4.2 m (at a blasthole diameter of 45mm) for the trackless section and 2.4 m (at a blasthole diameter of 38mm) for the conventional section of the mine to achieve an effective face advance of 3.8 m and 2.2 m respectively per blast. The blastholes will be charged with Anfex.
25.1.3 Hydrogeology

An in-depth study on the hydrogeological conditions has not been completed on this project thus far but seeing that this mine is in current operation and was in operation for many years, underground mine water is not expected to pose a risk to future mining operations. It is known that old workings are flooded, but the current level of this water is significantly below the planned lowest level in the new mine. However, this needs to be monitored regularly to see if there is a trend of rising water. If that occurs, a dewatering scheme will have to be implemented.

25.1.4 Paste Backfill

The mine design at present includes 100% cemented paste backfill, which is a conservative approach. Backfill testing has been completed to ascertain a suitable mix design, which includes 6% cement content. In addition, a reinforced plug at the bottom of each backfill section has been designed and will be constructed prior to backfilling taking place.

25.1.5 Metallurgy and Mineral Processing

As the material to be processed are mainly sulphides, a flotation method to improve and optimize the level of lead, zinc and silver recovery was selected. Challenges related to flotation selectivity and low lead concentrate grades were due to the presence of mixed, intertwined galena-sphalerite assemblage. The selectivity issue was overcome by selecting an adequate reagents suite. The consistency of the results obtained at the laboratory level, combined with higher recoveries, led to the selection of the conventional differential flotation. Bulk flotation did not render adequate performance (recovery and grade). Reclaimed process water was pre-treated with activated carbon in order to obtain higher lead concentrate grades as the carbon adsorbs the dissolved copper ions ($Cu^{2+}$) present in the water, which causes pre-activation of sphalerite and pyrite.

The straightforward conventional selective lead-silver flotation (locked cycle tests) show that good lead recoveries of between 88% and 93% can be achieved with a large variation in head grade from 1.5% Pb to 4.3% Pb. The silver recoveries range between 67% and 93.5% with a variation in head grade of between 136 g/t Ag and 375 g/t Ag.

Saleable lead and zinc concentrates can be produced from the Pulacayo ore using conventional mineral processing techniques including crushing, milling and differential flotation.

For tailings disposal the underflow from the tailings paste thickener will be either sent to the TSF or used as paste fill for underground stope support. The paste thickener overflow water will be reclaimed for the plant process water.

Results from metallurgical test work carried out in four stages have been used to guide process plant design. The Feasibility Study process comprises crushing, milling and classification, differential lead and zinc flotation, tailings disposal (tailings storage facility and cemented paste fill plant) and concentrate dewatering. Although a significant amount of test work has been completed, additional variability metallurgical tests for process flow circuit
validation are required. The “name-plate” capacity of the designed processing facility is 1,000 t/d.

25.1.6 Environmental and Permitting

The development of the Pulacayo Project poses certain significant challenges with respect to environmental management, including passive contamination from previous activities. This issue is covered in the environmental Baseline Study. The Environmental Impact Study (EIA) was filed and submitted to government entities for review, categorization and approval in January 2013. This study includes the outcome of a public consultation meeting held in December 2012 with the representatives of communities within the area of influence of the project. The result of this meeting was positive.

The environmental practices described in the current Environmental Impact Study will be implemented in the project, whilst the permit extension process continues. The communities will participate in these activities in order to create awareness. While this in progress, Apogee would continue with further engineering and procurement activities.

Apogee has land tenure, surface rights agreements, permits for water supply and discharge and exploration permits, required to carry out exploration activities including the development of an exploration access drive.

25.1.7 Water Management

The main supply of water in the region is the Yanapollera reservoir. The Yanapollera reservoir stores water from the snowmelt, run-off and ground water springs in the upstream Cosuño nevado (snow-capped mountain peak) and distributes the water to local users. At present, the reservoir has three users: A mining cooperative located in the Pulacayo town, the Pulacayo population and the town of Uyuni. The actual water demand of these users from Yanapollera is still somewhat uncertain, and their water demands are supplemented by ground water and other sources.

The hydrologic model evaluated the likelihood of a water deficit over the Life of Mine. This model included a sensitivity analyses on input parameters, to account for the lack of long-term site data normally recommended for model validation. The results of this work indicate that the Yanapollera reservoir alone should not be relied upon for process and potable water related to mine operation, as this would leave a statistically significant chance of a water deficit during the Life of Mine. This probability can be reduced or eliminated by constructing a water storage reservoir, and/or by supplementing the water supply with ground water sources. Investigation and design of the aforementioned items is currently underway.

The spring and summer months will see demand for water to suppress dust as well as to wash plant and other mine equipment. The bulk of such water should come from clean water run-off captured by drains located around the periphery of site.

25.1.8 Infrastructure

The 115 kV power line is one of the first activities to be developed, as the electrical equipment involved are considered long-lead items. Power line procurement and construction can take as much time as needed for the plant construction itself.
Plant site infrastructure includes a power supply line and substation connecting to the Bolivian power grid.

Auxiliary buildings for administration, mine surface workshop, security facilities will be constructed around the plant site.

25.1.9 Project Capital and Operating Cost Estimates

The pre-production (initial) capital cost for the Project was estimated at USD 45.9M. Sustaining capital required over the life of the operation has been estimated to be in the order of USD 41.1M (12.5 full production years).

The LOM operating cost is estimated at USD 54.9 per ton of ore processed.

25.1.10 Financial Analysis

Based on the technical work carried out for the Feasibility Study and the assumptions made, TWP concludes that:

- Capital costs have an impact on NPV and warrant close control and competitive tendering on key contracts in order to maximize project returns.
- Operating costs are significant in terms of labor, power and fuel and any efforts to reduce these will have a positive benefit to the project economics.
- The financial analysis, using a silver price of USD 28 per ounce indicates a pre-tax Net Present Value ("NPV") (using an 8% discount rate) of USD 126M with a pre-tax internal rate of return ("IRR") of 47.1%. The undiscounted payback period is 4.1 years after first capital expenditure.
- The results of the Feasibility Study indicate that Project economics are positive (based on the financial assumptions used), the Project is both technically and economical viable. Fluctuations in metal price (and to a lesser extent capital and operating costs) can significantly affect the economic viability of the Project. A reduction in the silver price from USD 28 per ounce to USD 20.00 would reduce the after tax IRR from 32.1% to 16.3%. Other than the metal price, the project does not seem to be sensitive to any specific parameter. An increase in silver price from USD 28 per ounce to USD 36 per ounce would increase after tax IRR to 45.3%.
- The project is not sensitive to capital expenditures and operating costs, i.e., 10% increase in the on mine operating cost would reduce the after tax IRR from 32.1% to 29.4%. A 10% increase in capital expenditure would reduce the IRR from 32.1% to 28.9%.
- At this time, given the exploration potential elsewhere on the property, Apogee may want to consider further development of the Pulacayo deposit and evaluate if a joint mining operation would improve project economics.
- Considering that the environmental permitting process for this Project may be more complex and take longer than planned, Apogee is currently proceeding with obtaining permits and authorizations for the 1,000 t/d project so that if and when project economics become more favorable, they can proceed quickly with execution. Delays in permitting would be detrimental to project economics.
• Concentrate marketability and the smelter schedule imposed by the purchaser of the concentrates would have a direct effect on project economics

25.2 Risk and Opportunities

25.2.1 Risks

• Significant reduction in metal prices.
• Paste fill delivery sequencing problems creating surface storage issues inconsistent with operating permits.
• New water users while the mine is in production.
• Insufficient power capacity from the Bolivian national grid.
• Management of ground water inflow to the mine is a significant risk. Water inflow rate and water quality affect the permitting process, water pumping infrastructure and therefore the capital and operating expenditures.
• Availability of skilled labor to develop, operate and maintain the mine. Actual productivity rates may impact on project development and ramp-up.
• Backfilling program falls behind mining operation. Lack of supply of backfill material may result in poor geotechnical conditions.
• Old mine working may result in poor stability.
• Dilution of high-grade mineral with lower grade of lead/zinc mineral. This may be due to old waste fill in stopes, weak country rock and collapse in backfill due to size of excavation.
• Long-term stability and accessibility of vent rises.
• Project management risks related to lack of serviceability and maintenance of equipment due to remoteness of site.
• Not involving representatives of the local communities in the environmental activities. This will have a major repercussion on getting a social license to operate the mine.

25.2.2 Opportunities

• The current resources do not take into account mineral in the Paca deposit. Some drilling has taken place at Paca but this is not at Indicated level, and therefore cannot be included in this study.
• The current reserve takes into account only the sulphide resources to be mined underground. In the Resource Statement, there is significant material that can be economically mined using the open pit method.
• The calculated operating cost of $55/t milled is significantly lower than the NSR cut off of $70/t, which has been applied to the Reserve Estimation. This leaves significant upside to expand the Reserve and mine life.
• Treated ground inflow water may be enough to meet plant and mine water requirement.
• Use of higher capacity underground equipment to increase efficiency and productivity.
• The resource models should be updated with results from the most recent drilling and geotechnical programs, to better understand and consider the mineral inventory available for exploitation. Consideration of the silver values and the potential that it
may provide an additional revenue stream is important because recovery of silver may substantially enhance project value. Ultimately, optimization of all project areas to provide the best combined mine and process components and configurations will reduce cost and enhance operating efficiencies.

- Current arrangements for Project access, communication, power and water supply and labor are sufficient to carry out year-round exploration activities, and with the necessary upgrades, the same means can be reasonably expected to meet needs for Project.
- TWP and Apogee identified a number of potential plant design optimizations that could be investigated to save plant CAPEX and to reduce the complexity of the plant.

### 26 RECOMMENDATIONS

The Feasibility Study indicates a robust Project and TWP recommends that the Project advance to the basic/detailed engineering stage in support of the Construction of a mine and Process facilities.

The TWP recommendations are summarized as follows:

#### 26.1 Geology

- Additional exploration drilling is recommended to improve the confidence in indicated resources, especially in the parts of the deposit that are early in the production schedule.
- Additional exploration drilling is also recommended to assess opportunities for extension of the deposit limits both along strike and down dip.
- The resource model should be updated to include any drilling conducted after the completion of the mineral resource estimates included in this study. As new information regarding the mineral resource and economic parameters, the reserves shall be updated.

#### 26.2 Mining

##### 26.2.1 Geotechnical

- Additional geotechnical study is required to support the basic/detailed engineering. Geotechnical work should include site investigation, trenching, drilling, topography and slope stability modeling for all surface installations including the plant site, campsite, waste dumps and tailings disposal facilities.
- As more underground geotechnical information becomes available, a rock mechanics analysis should be conducted to re-assess the backfill material for mined out stopes, which could reduce backfill costs.
In general, the work described next will help advance the project to the basic/detailed engineering stage: Spatial variations in key geomechanical characteristics, geomechanical drill holes, lab strength testing and numerical modeling.

26.2.2 Hydrogeology

- A field investigation campaign including additional hydrogeology drill holes and piezometer installation.
- Acquire water inflow data during tunnel excavation.
- Compile a detailed numerical hydrogeological modeling of mine inflows.
- Perform in-depth hydrogeological studies to determine the groundwater inflow and quality as well as to define fissure water underground.
- Further study on hydrogeological conditions are needed for estimating water handling and underground pumping requirements.

26.2.3 General

- The mobile capital and operating costs require further refinement with manufacturer specific data for spare parts and major rebuilds.
- From a mine planning standpoint, TWP recommends an in-depth examination of the factors that contribute the most to the cost or mined material head grade.
- Modify the mine plan to include increased resources and identify areas of the mine amenable to lower cost bulk mining methods.
- Prepare a mine dewatering plan to comply with safety plans during operation.

26.3 Metallurgy and Mineral Processing

- Conduct additional variability test work to further optimize the flowsheet and reagent suite to ensure the concentrates specifications are optimized.
- Pilot plant pumping tests for cemented paste fill.
- An opportunity exists to optimize process-related capital and operating costs. These include an alternate crushing design furnished with a scrubber for clays washing and handling instead of having high pressure nozzles for clays washing.
- Further mineralogical analysis (QUEMSCAN) of the deposit to further understand the mineralogy throughout the deposit. Further explore the potential influence of the presence of variable amounts of copper in the ore in flotation lead and zinc selectivity.
- Lead and zinc concentrates filtration tests at the selected mill grind size are required to confirm the filter sizing, the type of filter and attainable cake moisture.

26.4 Infrastructure

- Additional engineering work including updates of mechanical, civil, electrical and instrumentation design are recommended for the basic/detailed engineering phase.
- A detailed Project Execution Plan will be developed during the basic/detailed engineering work.
In order to proceed with detailed engineering, additional geotechnical studies must be carried out at the location of all infrastructure and along the railway line from the mine to plant. The program must include drilling, surveys as well as laboratory soil analysis and soil dynamic stability studies and contour and elevation surveying.

- Conduct geotechnical surveys at the location of the TSF to confirm the slope of the TSF and for the stability of the various catchment ponds
- Optimize the auxiliary buildings layout based on new developments proposed by Apogee and on detailed topography and geotechnical surveys.
- Evaluate an alternate tailings storage technology that may result in reduced capital cost and footprint.
- Review opportunities for early completion of construction to reduce start-up times.
- Further drilling is recommended to expand and delineate the mineral resource on the Pulacayo deposit.

26.5 Financial Analysis

- Further technical-financial analysis to be carried out as the Project progresses to the basic/detailed engineering stage, and to evaluate the impact of any new data.
- Consider financial arrangements targeted to further reduce working capital needs.
- The refinery that will purchase the concentrates should be identified, particularly with a view to identifying and understanding the impact penalty elements may have on the revenue and possibly on the process.

26.6 Environmental and Permitting

- Ore acid generating potential should be re-assessed and additional ABA (acid-base accounting) tests are recommended.
- Conduct air quality dispersion modeling using modeling systems to predict future concentrations at and off the boundary of the Project.
- For reclamation and closure, conduct a complete soil and borrow material inventory focused on moisture retention, erodability and other soil quantities for use in final reclamation and site re-vegetation.

26.7 Water Management

- Continuous on monitoring of precipitation, surface water flow and water quality should be conducted to ensure compliance through the development, operational, reclamation and closure plan stages of the Project.
- The mine dewatering system will require modification and refinement as more data becomes available during advanced exploration and initial construction and operation.
- It is recommended that an in-depth site-wide water balance be prepared.
- A detailed hydro-meteorological study for supporting the final design should be conducted.
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