FORTUNA
SILVER MINES INC.

# Fortuna Silver Mines Inc.: Lindero Property, Salta Province, Argentina

## Technical Report
## Effective Date: October 31, 2017


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# Date and Signature Page

**Technical Report**

**Fortuna Silver Mines Inc.: Lindero Property, Salta Province, Argentina**

**Effective date of this report is October 31, 2017**

**Issued by:**

**Fortuna Silver Mines Inc.**

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| Eric Chapman | 31st October 2017 |
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| Edwin Gutierrez | 31st October 2017 |
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# 1 Summary

## 1.1 Introduction

This Technical Report (the Report) has been prepared by Fortuna Silver Mines Inc. (Fortuna) in accordance with the disclosure requirements of Canadian National Instrument 43-101 (NI 43-101) to disclose recent technical and scientific information in respect to the Lindero gold Project (the Project) including:

- Exploration and infill drilling activities conducted since February 23, 2016 (effective date of previous Technical Report)
- Mineral Resource and Mineral Reserve estimates as of September 9, 2017 for the Lindero Deposit that incorporate all relevant information
- Metallurgical testwork conducted since February 23, 2016
- Geotechnical evaluations
- Updated life-of-mine (LOM), capital and operational expenditures, and economic analysis based on the above
- Updating of basic engineering to reflect changes from the optimization in mineral process design, project layout, and economic parameters in Argentina since February 23, 2016

This report supersedes the previous Technical Report filed on March 2, 2016 (KCA, 2016) by Goldrock Mines Corporation (Goldrock).

## 1.2 Property description, location and ownership

The Project is in the Argentine puna, a cool, arid zone with a minimum elevation of approximately 3,500 to 4,000 m. The climate is generally dry and windy; it can be cold and snowy during storms.

The Lindero Project is located 260 km due west of Salta, Argentina, the main service center of the region, at latitude 25° 05' south and longitude 67° 47' west. Drive time from Salta to the Project is approximately 7 to 7.5 hours, over a road distance of 420 km. The nearest town to the Lindero Project is Tolar Grande (population 250) located 75 km to the northeast.

Access to the Project is via National Route 51, which passes through the towns of San Antonio de Los Cobres and Olacapato; and Provincial Route 27, via Pocitos and Tolar Grande.

The Lindero Project contains two known porphyry gold-copper deposits. The Lindero Deposit is the focus of the Feasibility Study and this report; whereas the Arizaro Deposit, located 3.2 km southeast of the Lindero Deposit, is described only in terms of exploration conducted to date.

The mineral tenement holdings cover 3,500 ha, and comprise 35 pertenencias, each of 100 ha, which are constrained by Gauss Kruger Posgar co-ordinates generated by survey.

Tenure is held in the name of Mansfield Minera S.A. (Mansfield), an indirectly wholly-owned subsidiary of Fortuna. There is no expiry date on the pertenencias, providing Mansfield meets expenditure and environmental requirements, and pays the appropriate annual mining fees.

A 3 % provincial royalty "boca mina" is payable on revenue after deduction of direct processing, commercial, general and administrative costs. There are no royalties payable to any other third party.

Surface rights are owned by the provincial state (Propiedad Fiscal) of Salta. There are no reservations, restrictions, rights-of-way or easements on the Project to any third-party. Mansfield holds a registered camp concession, and a granted and surveyed access right-of-way. Water permits and rights of access to the Project are guaranteed through water and access licenses granted by the Mining Court of Salta.

Surface rights for construction of a mining operation and plant have not been granted from the Provincial authorities. Development of such infrastructure will require additional negotiation and potentially, supporting studies. Mansfield do not foresee any issues with obtaining the necessary permits for construction.

## 1.3 History

Gold–copper mineralization associated with potassic alteration was first discovered at Lindero by Goldrock geologists in November 1999, and led to claim staking.

The area was explored using reconnaissance and detailed geological mapping, soil geochemistry (talus fines), trench sampling and mapping during 2000 and early 2001. As a result of this work, mineralization at what is now the Lindero Deposit was identified in September 2000.

From April 2002 to March 2003, Rio Tinto had an option on the property with Goldrock, during which time additional exploration including drilling and metallurgical testwork was conducted. An inhouse preliminary Mineral Resource estimate for the Lindero Deposit was performed. As the tonnage and grade estimate did not meet Rio Tinto's corporate targets, the option was not exercised.

Goldrock resumed as project operator, and between 2005 and 2013 completed additional exploration and drilling. Based on this, a Pre-Feasibility Study for the Lindero Deposit was completed by AMEC in 2010, assuming a production throughput of 30,000 tonnes of ore per day (AMEC Americas Ltd., 2010a; 2010b). In 2012, Goldrock commissioned Kappes, Cassiday & Associates (KCA) to complete a Feasibility Study using a reduced throughput of 18,750 tpd.

In 2015, Goldrock commissioned KCA to work with local engineering firms in advancing the engineering design for the Project to a basic engineering level, and update the 2013 Feasibility Study. A new Feasibility Study incorporating these design changes, additional metallurgical testwork, and updated costs and gold price assumptions was filed by KCA in 2016 (KCA, 2016a).

In July 2016, Fortuna completed the acquisition of all issued and outstanding shares of Goldrock, making Mansfield a wholly-owned subsidiary of Fortuna. Upon completion of the transaction, Fortuna continued to advance the optimization of the 2016 Feasibility Study through additional drilling as well as conducting tradeoff metallurgical tests and detailed engineering revisions with the objective of reaching a construction decision for the Lindero Project (Fortuna, 2017).

## 1.4 Geology and mineralization

In the Central Andes, the altiplano or puna is a high plateau of more subdued relief between the Eastern Cordillera, a rugged region usually rising to between 3 km and 4.5 km, and the Western Cordillera, which is a high spine of mountains that may reach as much as 5 km in height. The Arizaro Volcanic Complex consists of two superimposed concentric volcanic centers, the Arizaro and the Lindero cones, located in the Archibarca volcanic arc at the southern margin of the Salar de Arizaro basin. Basement rocks crop out to the north of the Lindero Deposit, and consist of coarse-grained Ordovician granites uncomformably overlain by Early Tertiary red bed sandstones. The Lindero–Arizaro complex, a series of diorite to monzonite porphyritic stocks, intrudes these units.

Mineralized zones at the Lindero Deposit form a semi-circular shape about 600 m in diameter which extends to a depth of 600 m, consisting of four different zones at the surface. The distribution of gold–copper mineralization at Lindero shows a strong relationship to lithology, stockwork veinlets, and alteration assemblages. Gold values average 0.70 g/t Au and copper values are typically about 0.11 % Cu. Higher grades of gold–copper (approximately 1 g/t Au and 0.1 % Cu) are commonly associated with sigmoidal quartz, quartz–magnetite–sulfide, biotite-magnetite–chalcopyrite, magnetite–chalcopyrite and quartz–limonite–hematite stockworks that are strongly associated with K-feldspar alteration. This association is very common in the east zone of the deposit, where the highest gold grades occur. At other locations where one or more stockwork types are missing or the intensity of fracturing is lower, mineralization tends to be weaker and the grades of gold tends to be lower (approximately 0.4 g/t Au).

Gold mineralization at Lindero is characterized by native, free-milling gold associated with chalcopyrite and/or magnetite grains with rare interstitial quartz.

The weathered oxidation zone at Lindero is generally poorly developed and averages 44 m in thickness.

The Arizaro volcanic center is characterized by fine- to medium-grained hornblende diorite to monzonite porphyritic stocks. The Arizaro Deposit is dominated by a main, moderately to strongly mineralized intrusive unit that crops out in the central part of the prospect area. It consists of fine hornblende porphyritic diorite intruded by several stocks, dikes, igneous-cemented breccias and hydrothermal breccias. Smaller stocks are exposed in a few areas. Dikes of andesitic and dacitic composition are generally distributed radially to the main intrusive unit.

Several alteration assemblages are noted in the Arizaro Deposit area. Alteration patterns are semi-concentric and asymmetric, with a core of moderate to strong potassic alteration including zones of K-feldspar-rich magnetite–silica alteration. An incomplete rim of chloritic alteration is developed outboard of the potassic alteration. In the southeast part of the deposit, intermediate argillic alteration has formed and overprints potassic alteration. Sericitic and very weak argillic alteration (hydrolytic alteration) has developed in the volcanic tuffs. To the south and west of the deposit, chloritic alteration passes directly to propylitic alteration. An actinolite–magnetite alteration assemblage forms in the eastern part of the deposit area.

Arizaro gold–copper mineralization is hosted in one body which has a semi-oval shape at the surface. In the center there is a high-grade body with a semi-ellipsoidal form, extending north-south for 480 m and about 50 m wide. The Arizaro Deposit has mineralization styles with copper–gold grades that are strongly correlated with different alteration assemblages. Mineralization is mainly associated with potassic alteration. This occurs generally in multi-directional veins, vein stockworks and disseminations. In some areas, the vein density is high, forming vein stockworks in the intrusive rocks. These vein stockworks are limited to magnetite–biotite veinlets, quartz–magnetite–chalcopyrite veinlets, late magnetite breccias and in late-stage mineralization events, anhydrite–sulfide veinlets. Chalcopyrite and bornite are the main copper minerals. Coarse gold was observed and confirmed with X-ray diffraction analysis in the University of Neuquen, Argentina, laboratory.

Lindero and Arizaro are examples of gold-rich porphyry copper deposits as described by Sillitoe (2000). More specifically, they show affinities with the porphyry gold deposit model (Rytuba and Cox, 1991; also termed dioritic porphyry gold deposits by Seedorff et al., 2005). These are exemplified by the Refugio, Cerro Casale, Marte, and Lobo gold deposits of the Miocene-age Maricunga belt, Chile, approximately 200 km south of Lindero. Vila and Sillitoe (1991) and Muntean and Einaudi (2000, 2001) described those deposits in detail.

The deposits of the Project area are considered to be examples of porphyry-style deposits, in particular gold-rich porphyries based on the following:

- High level (epizonal) stock emplacement levels in magmatic arc
- High-level stocks and related dikes intrude their coeval and cogenetic volcanic piles. Intrusions range from fine through coarse-grained, equigranular to coarsely porphyritic
- Mineralization in or adjoining porphyritic intrusions of quartz diorite/monzonite composition
- Mineralization is spatially, temporally, and genetically associated with hydrothermal alteration of the intrusive bodies and host rocks
- Gold–copper mineralization formed during intrusion of multiple phases of similar composition intrusive rocks
- Large zones of quartz veining, stockwork mineralization, and disseminated pyrite
- Tenor of gold and copper grades, i.e., large tonnage but low grade

At the Lindero Deposit, native gold and electrum are finely disseminated in subparallel to stockwork quartz + sulfide ± magnetite ± anhydrite veins and in some cases in matrices of hydrothermal breccias. Magnetite is common to abundant in mineralized zones. These mineralized stockworks and potassic alteration are interpreted to have formed as the result of degassing of the early intrusive bodies. Fluid pressures during degassing triggered fracturing of the intrusions and wall rock, allowing gold-rich fluids to circulate and precipitate, forming a gold–copper orebody. Later intrusions resulted in weak to moderate gold–copper mineralization forming mostly along and immediately fringing these intrusive contacts. Finally, post mineralized intrusives were overprinted onto the north and west of the deposit.

Understanding of the geological setting and model concept of the Lindero and Arizaro is adequate to provide guidance for exploration and development of the deposits.

## 1.5 Exploration, drilling and sampling

The Lindero Deposit was discovered in late 2000. Several exploration programs have been conducted by Rio Tinto, Goldrock and Fortuna on the Lindero Property:

- Goldrock campaign: August 2000 to October 2001, which included geologic mapping, soil sampling, and trench sampling
- Rio Tinto Campaign: May 2002 to February 2003, which included road sampling, geophysics (43 km of ground magnetics and 11 km of induced polarization (IP)), and drilling (10 holes for a total of 3,279 m)
- Goldrock campaign: October 2005 to January 2008, which included geologic mapping and modeling, trenching, and a significant drilling program (106 holes for a total of 30,024 m)
- Goldrock campaign: September 2008 and August 2010 to November 2010, which consisted of additional drilling (23 holes) for the Pre-Feasibility Study
- Fortuna campaign: September 2016 to December 2016 consisting of 8 holes for metallurgical samples, 2 holes for geologic interpretation and 2 twin holes

Drilling completed at the Lindero Property comprises 151 diamond drill holes totaling 42,598 m at the Lindero Deposit, as well as 29 diamond drill holes totaling 8,855 m at the Arizaro Deposit. Mineral Resources are only estimated at the Lindero Deposit. Ground conditions were good, and core recovery was generally above 90 %. Drill hole collars were marked with PVC pipes introduced in the hole at surface and then cemented. All holes drilled since 2005 as well as the 10 holes drilled during the 2002 campaign were surveyed by Servicios Topograficos with a differential GPS. Coordinates are projected on the WGS 84 Datum ellipsoid and calibrated according to the position of Geodetic point IGM N° PR-02-015, located a few kilometers from the Project. The results are available in geographic co-ordinates and in metric co-ordinates (UTM and Gauss Kruger), using the WGS 84 datum.

During Rio Tinto's exploration drilling campaign in 2002, undertaken by Connors Drilling, no downhole surveys were completed despite the fact that many of the holes extended beyond 300 m in depth. Holes drilled during the first Goldrock campaign were not originally downhole surveyed either. In June 2006 GEC-Geophysical Exploration & Consulting S.A. (GEC) was contracted by Goldrock to perform borehole surveying services with a Reflex Maxibor II System 3™ Probe (Maxibor™), which is not affected by magnetism. In 2008, Goldrock detected that the Maxibor™ surveys showed an unacceptably large deviation in the drill holes and a decision was made to re-survey all holes that showed a deviation of more than 5 %. Comprobe Chile Ltd. (Comprobe) was contracted to re-survey the holes considered by Goldrock as having incorrect downhole deviations. A surface-recording gyroscopic instrument was used, and orientation and dip parameters were recorded every 10 m. For the 2016 drilling campaign, Fortuna retained the services of Construccion & Mineria S.A., based out of Mendoza, Argentina, to complete downhole surveys for each hole upon completion. Downhole surveys were conducted using Reflex™ gyroscopic equipment with readings taken at 5-m intervals.

All core was logged for geology and geotechnical characteristics. All logging was digital, and was incorporated daily into the Maxwell DataShed™ database system. Data were recorded initially with Excel™ templates, and later with Maxwell LogChief™ application using essentially the same structure. Separate pages were designed to capture metadata, lithology, alteration, veins, sulfide–oxide zones, sulfide–oxide surfaces, minerals (sulfides, oxides, and limonite), sulfates, structures (contacts, fractures, veins, and faults with attitudes to core axis), magnetic susceptibility, and special data (samples collected for geochemistry, thin section examinations, the core library, skeleton core, etc.). Intensity of alteration phases was recorded using a numeric 1 to 4 scale (weak, moderate, strong, complete); abundance of veins and most other minerals were estimated in volume percent.

The Lindero Deposit is a gold-rich porphyry with low-grade mineralization permeating throughout the deposit, making the calculation of true thickness impossible as no definitive across strike direction exists. The mineralization appears to be annular in shape at surface due to the intrusion of barren to low-grade intrusive rocks into the core of the system, but this circular shape is not representative of true thickness.

Core samples are marked and collected on 2 m intervals that honor lithological boundaries. Samples weigh between 4 and 8 kg depending on core diameter and recovery. Channel samples were collected using a rock saw to cut a 2 x 3 cm channel in exposed bedrock in trenches and road cuts. The material was removed from the channel with a chisel. Sample preparation for most samples consisted of crushing to 70 % passing 10 mesh and pulverization to 95 % passing 150 mesh. Density samples are routinely collected by Mansfield from drill core on approximate 10-m intervals. Samples consist of pieces of core approximately 7 cm in length and weighing between 93 g and 408 g.

All samples collected by Mansfield were assayed for gold using a 30 g fire assay–atomic absorption (FA-AA) finish and a second aliquot was selected for copper analysis using aqua regia digestion and AA analyses. For the drill samples only, a full suite of trace elements was analyzed using an aqua regia digestion followed by inductively-coupled plasma (ICP) analysis. Assay results and certificates were reported electronically by e-mail.

Fortuna samples were sent to the ALS Global sample preparation facility in Mendoza, Argentina. Following drying at 55°C, the samples were weighed and the entire sample crushed using a two-stage method, first with a jaw crusher to 1 cm, and then by cone crusher to 70 % passing 10 mesh. The entire crushed sample was then pulverized to a minimum of 95 % passing 80 mesh. Pulverized samples were then split using a riffle splitter to generate a 300 g subsample that was pulverized to 95 % passing 150 mesh. This subsample was then split again using a riffle splitter to generate three 100 g samples.

All samples were sent to accredited laboratories independent of Mansfield, Rio Tinto, and Fortuna.

Implementation of a quality assurance/quality control (QAQC) program is current industry best practice and involves establishing appropriate procedures and the routine insertion of standard reference material (SRMs), blanks, and duplicates to monitor the sampling, sample preparation and analytical process. Fortuna implemented a full QAQC program to monitor the sampling, sample preparation and analytical process for the 2016 drilling campaign in accordance with its companywide procedures. The program involved the routine insertion of SRMs, blanks, and duplicates. Evaluation of the QAQC data indicate that the data are sufficiently accurate and precise to support Mineral Resource estimation.

## 1.6 Data verification

In 2009 an independent audit of the information used for the estimation of Mineral Resources and Mineral Reserves at the time was conducted by AMEC, and summarized in the KCA (2016a) Technical Report. The work included independent audits of the database, collar and downhole surveys, drill logs, assays, bulk density measurements, core recovery, and QAQC results.

The 2009 audit concluded that the data verification programs undertaken on the data collected from the Lindero Deposit up to 2009 supported the geologic interpretations, and the analytical and database quality, and therefore the data could support Mineral Resource and Mineral Reserve estimation.

Fortuna reviewed the work performed by AMEC and concurs with their opinion. Fortuna has conducted additional audits and verification of historical information used in prior Mineral Resource and Mineral Reserve estimates as well as verifying new data generated during the 2016 drilling campaign to support assumptions for a construction decision and the Mineral Resource and Mineral Reserve estimates reported in Section 14 and Section 15 of this Report. The verification process focused on the database; collars and downhole surveys; lithologic logs; assays; metallurgical results; and geotechnical parameters. Fortuna checked all collar and downhole survey information for each campaign against source documentation and completed a hand-held GPS survey of randomly selected drill hole collars. The results showed a good agreement with locations in the database. In August 2016, Fortuna initiated a comprehensive program of relogging to verify the original lithologic descriptions.

Fortuna contracted Call & Nicholas Inc. (CNI) to validate all geotechnical data, data collection methods, slope stability analysis methods, and slope angle recommendations presented previously by other consultants to determine feasibility-level slope angle recommendations for design of the planned Lindero final pit.

The QP is of the opinion that the data verification programs performed on the data collected from the Project are adequate to support the geological interpretations, the analytical and database quality, and Mineral Resource estimation at the Lindero Project.

## 1.7 Mineral processing and metallurgical testing

The Lindero Project has an extensive body of metallurgical investigation comprising several phases of testwork as indicated in the KCA (2016a) Technical Report, and summarized in Section 13 of this Report. In general, the testwork was done to industry standards. However, some leach conditions set for the testwork made interpretation difficult. Reinterpretation of the raw test data provided the basis for advancing the metallurgical knowledge base for Fortuna.

Since September 2016, Fortuna has performed complementary metallurgical testwork in the areas of comminution, heap permeability and cement agglomeration, gold extraction in column tests, and copper removal with sulfidization-acidification-recycle-thickening (SART) technology with the purpose of confirming and optimizing process design criteria.

Table 1.1 shows key gold extraction results for 10-m columns from laboratory testwork, carried out in the first semester of 2017, on material cured in a cyanide solution and agglomerated. A 4 % deduction (absolute) has been used in the design to allow for the differences between laboratory and expected operational results.

**Table 1.1 Key gold extraction results for 10-m columns**

| Met Type | Met Type Description | Met Type as Percentage of Reserve | Gold Extraction | |
|---|---|---|---|---|
| | | | Laboratory (%) | Field (%) |
| 1 | Fresh Intrusive | 63 | 79.4 | 75.4 |
| 2 | Oxide Porphyry | 20 | 82.2 | 78.2 |
| 3 | Fresh porphyry | 9 | 82.5 | 78.5 |
| 4 | Sediments | 8 | 72.5 | 68.5 |
| | | **Weighted average** | **79.7** | **75.7** |

Optimization of the process design has confirmed the benefit of the use of a high-pressure-grinding-roll (HPGR), the inclusion of cyanide cure of ore, and copper removal/cyanide recovery with a SART plant. Results indicate that these components allow for improved gold leaching kinetics and effective extraction of copper from the pregnant solution.

Ore will be crushed at a nominal rate of 18,750 tpd using a three-stage crushing system including a HPGR in the tertiary stage. A final crush size of $P_{80}$ 6.0 mm is projected. The crushed product will be agglomerated and cured with a cyanide solution and then conveyed to the leach pad. A mobile conveying and stacking system will be used to stack ore in 10-m-high lifts. The life-of-mine (LOM) leach pad area is projected at 105 ha with a maximum height of 110 m. Leaching will be carried out in two stages with a first stage of 30 days and a second stage of 60 days.

The gold pregnant solution will be pumped at a rate of 400 $m^3$/hr to a SART plant, where copper in solution will be precipitated to maintain copper levels below 400 ppm in the solution. The Project contemplates an expansion of the pregnant solution flow rate from 400 $m^3$/hr to 600 $m^3$/hr in year four with the objective of reducing gold ounce inventory in the heap at the end of mining.

Following the SART plant, the pregnant solution will go to an adsorption, desorption, recovery (ADR) plant and then to electrowinning and refining where gold will be poured in doré bars. LOM recovery is estimated at 75 %.

It is the opinion of the QP that the Lindero samples tested represent the orebody with respect to grade and metallurgical response. The differences between metallurgical lithologies are minimal with regard to extraction. Cyanide consumptions are higher with the more oxidized Met 2 samples as would be expected. Minimal metallurgical differences were expected after review of the historical work.

Physical differences appear to have greater impact on the processing of the Lindero met types. Of significant importance is the ability of the agglomerated ore to support the planned heap height.

No significant deleterious materials such as mercury or clays were noted in the samples tested.

A high level of metallurgical and process risk mitigation is incorporated in the process design with HPGR crushing, agglomeration and the SART plant. With these installations any expected short-term variation in ore composition (i.e. elevated soluble copper content) or physical properties (i.e. elevated gypsum levels or increased ore hardness at depth) can be accommodated in the normal course of operations.

## 1.8 Mineral Resources and Mineral Reserves

Mineral Resources have only been estimated for the Lindero Deposit.

Mineral Resource estimation involved the use of drill hole and channel sample data in conjunction with surface mapping to construct three-dimensional (3-D) wireframes to define individual lithologic structures and oxide–mixed–sulfide horizons. Drill hole samples were selected inside these wireframes, coded, composited and grade top cuts applied if applicable. Boundaries were treated as either soft, firm or hard with statistical and geostatistical analysis conducted on composites identified in individual lithologic units. Gold and copper grades were estimated into a geological block model consisting of 10 m x 10 m x 4 m selective mining units (SMUs). Grades were estimated using dynamic anisotropy by ordinary kriging (OK) and constrained within an ultimate pit shell based on estimated metal prices, costs, geotechnical constraints, and metallurgical recoveries to fulfill the expectation of reasonable prospects of eventual economic extraction. Estimated grades were validated globally, locally, and visually prior to tabulation of the Mineral Resources.

Mineral Reserves are exclusive of Mineral Resources and Mineral Reserve estimates have considered only Measured and Indicated Mineral Resources as only these categories can be considered Mineral Reserves (CIM, 2014). Subject to the application of modifying factors, Measured Resources may become Proven Reserves and Indicated Resources may become Probable Reserves.

Mineral Reserves and Mineral Resources exclusive of Mineral Reserves as of September 9, 2017 are reported in Table 1.2 and Table 1.3 respectively.

**Table 1.2 Mineral Reserves as of September 9, 2017**

| Classification | Tonnes (000) | Au (g/t) | Cu (%) | Contained Metal |
|---|---|---|---|---|
| | | | | Au (koz) |
| Proven | 26,009 | 0.74 | 0.11 | 618 |
| Probable | 62,263 | 0.57 | 0.11 | 1,131 |
| **Proven + Probable** | **88,272** | **0.62** | **0.11** | **1,749** |

**Table 1.3 Mineral Resources as of September 9, 2017**

| Classification | Tonnes (000) | Au (g/t) | Cu (%) | Contained Metal |
|---|---|---|---|---|
| | | | | Au (koz) |
| Measured | 610 | 0.24 | 0.06 | 5 |
| Indicated | 11,897 | 0.24 | 0.07 | 92 |
| **Measured + Indicated** | **12,507** | **0.24** | **0.07** | **97** |
| **Inferred** | **5,700** | **0.36** | **0.10** | **65** |

Notes:

- Mineral Reserves and Mineral Resources are as defined by CIM Definition Standards on Mineral Resources and Mineral Reserves
- Mineral Resources are exclusive of Mineral Reserves
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
- There are no known legal, political, environmental, or other risks that could materially affect the potential development of the Mineral Resources or Mineral Reserves at Lindero
- Mineral Resources and Mineral Reserves are estimated and reported as of September 9, 2017

- Eric Chapman, P.Geo. (APEGBC #36328) is the Qualified Person for resources and Edwin Gutierrez (SME Registered Member #4119110RM) is the Qualified Person for reserves, both being employees of Fortuna Silver Mines Inc.
- Mineral Reserves for Lindero are reported based on open pit mining within designed pit shells based on variable gold cut-off grades and gold recoveries by metallurgical type. Met type 1 cut-off 0.27 g/t Au, recovery 75.4 %; Met type 2 cut-off 0.26 g/t Au, recovery 78.2 %; Met type 3 cut-off 0.26 g/t Au, recovery 78.5 %; and Met type 4 cut-off 0.30 g/t Au, recovery 61.7 %. The cut-off grades and pit designs are considered appropriate for long-term gold prices of US$ 1,250/oz. Assumptions used in the pit design are the same as those for the resources
- Lindero Mineral Resources are reported within a conceptual pit shell above a 0.2 g/t Au cut-off grade using a long-term gold price of US$ 1,250/oz, mining costs at US$ 1.67 per tonne of material, with total processing and process G&A costs of $7.84 per tonne of mineralized material and an average process recovery of 75 %. The refinery costs net of pay factor were estimated to be US$ 6.90 per ounce gold. Slope angles are based on 3 sectors (39°, 42°, and 47°) consistent with geotechnical consultant recommendations
- Totals may not add due to rounding

Mineral Reserves are estimated at 88.3 Mt as of September 9, 2017 which is sufficient for a thirteen-year LOM considering 350 days in the year for production and a capacity rate of 18,750 tpd. Expectation based on an optimized production schedule is for an annual average production of 129,000 troy ounces of gold.

Proven and Probable Mineral Reserves are estimated to contain 1.75 Moz gold, reflecting a 12 % decrease in contained gold ounces relative to the October, 2015 Mineral Reserve estimate. Variations are the result of:

- A smaller ultimate pit shell based on updated metal prices, mining costs, and metallurgical recoveries resulting in a decrease in the Measured and Indicated Mineral Resources
- 2016 drilling which upgraded 12 Mt to Indicated Mineral Resources with a loss of that amount of Inferred Mineral Resources
- Adjustments to the geological interpretation and estimation methodology

## 1.9 Mining methods

Lindero will be an owner-operated conventional open pit mining operation with a nominal rate of 18,750 tpd of ore and a life of pit operations of 13 years using existing reserves. The ratio of waste to ore over the LOM is 1.2 to 1. The key mining fleet equipment will be initially composed of six 91 tonne (100-ton) trucks and two 17 cubic yard wheel loaders.

In the initial two years, the operation will benefit from mining the higher-grade, outcropping portion of the deposit, with an average head grade of 0.90 g/t Au, and a low strip ratio of 0.77 to 1. For the initial four years, the average head grade is projected at 0.77 g/t Au, and a strip ratio of 1 to 1.

Mining costs benefit from short haul distances from the pit to the primary crusher and waste dumps. Maximum distances are in the range of 2 km. The LOM direct mining cost is estimated at US$ 1.1 per tonne moved.

The QP is of the opinion that:

- The mining method being used is appropriate for the deposit being mined

- The open pit, stockpile, waste dump designs, and equipment fleet selection are appropriate to reach production targets
- The mine plan is based on successful mining philosophy and planning, and presents low risk
- Inferred Mineral Resources are not included in the mine plan and are considered as waste
- The mobile equipment fleet presented is based on simulations and bench marks to similar operations achieving similar production targets
- All mine infrastructure and supporting facilities meet the needs of the current mine plan and production rate
- Major planned maintenance of the main equipment, such as loaders and trucks, have been covered in sustaining capital by purchasing additional equipment that can replace any possible lost production hours and not impact production targets
- The ancillary equipment appears to be undersized, especially dozers, but this would be covered by renting additional equipment as necessary

## 1.10 Recovery methods

Most of the major process concepts presented in the 2016 Technical Report such as: high pressure grind roll (HPGR)-crushing, cyanide heap leaching and carbon adsorption recovery, remain unchanged for this update. Additional physical and metallurgical understanding, developed by the testwork conducted by Fortuna in 2016 and 2017, resulted in modifications in the approach to these major process concepts for the Lindero Project as follows.

- A concentrated cyanide cure was added to shorten the leach cycle and increase extraction
- Agglomeration with cement was added to support a 110-m-high heap with the HPGR-crushed ore
- Conveyor stacking was included from startup
- Two-stage leaching was included to increase preg grades and reduce overall flowrate to the ADR plant
- A SART plant was included to control the copper in solution
- Leach solution flow will be increased 150 % in Year 4 to reduce in-heap gold inventory

Unit operations for the Lindero process were selected based on the physical and metallurgical needs of the Lindero ore to achieve maximum extraction of gold. No novel or untried technology will be employed in the process.

## 1.11 Project infrastructure

The QP is confident that all mine and process infrastructure and supporting facilities have been included in the general layout to ensure that they meet the needs of the mine plan and production rate and notes that:

- The Project will have good year-round access with significant road improvements planned for stretches of road between Tolar Grande and the Fortuna camp
- The Project site infrastructure has a compact layout footprint of approximately 60 ha
- Power will be generated on-site by a contractor through an 8 MW capacity diesel oil plant
- Electrical power will be generated on site under a contract power supply arrangement with a local company who specializes in such services
- Total water requirements are 97.7 $m^3$/hr and will be primarily sourced from two existing wells located 13 km southeast of the Project site, along with an additional well to be drilled as part of construction activities
- Most of the process buildings for the Lindero Project have been primarily designed as steel frame buildings with modular thermo-acoustic panels; in general, these are pre-engineered and pre-fabricated steel buildings which include all structural members, exterior doors and windows, roofs, insulation, interior and exterior wall panels and all connectors required to erect and assemble the buildings on-site
- A permanent accommodation camp for 320 beds will be built for the LOM operation. For the construction period, temporary accommodations will be implemented to accommodate the peak of construction manpower estimated at 600 people

## 1.12 Market studies and contracts

No market studies are currently relevant as the Lindero Project will produce a readily-saleable commodity in the form of doré.

As of the effective date of this report Fortuna has not entered into any material contracts required for the development of the Lindero Project including mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements.

The gold price used for the base case cash flow analysis is $1,250/oz. Sensitivities with variable price projections have also been considered. The Lindero Project, like most gold projects, is highly sensitive to changes in the gold price.

The Lindero mine product will be doré bars containing an estimated gold content averaging 84 % for the Project life. Overall gold extraction in respect to ore placed on the heap leach is estimated to be approximately 75 %.

The QP has reviewed the information provided by Fortuna on marketing, contracts, metal price projections and exchange rate forecasts and notes that the information provided is consistent with the source documents used, and that the information is consistent with what is publicly available regarding industry norms. The information can be used in mine planning and economic analyses for the Lindero Project in the context of this Technical Report.

## 1.13 Environmental studies and permitting

In November 2010, Mansfield submitted an Environmental Impact Assessment (EIA) for the Lindero Project, and in November 2011 received approval through the issue of the Declaración de Impacto Ambiental (DIA). Approval of the EIA represents formal approval for mine construction, allowing excavation to proceed. Environmental law requires that the EIA be updated biannually with the current report submitted in December 2015 and an updated report planned for submission in March 2018.

Mansfield received a mine permit to build a heap-leach gold mine at up to 30,000 tpd as detailed in the Pre-Feasibility Study (AMEC, 2010b).

The Salta Provincial authorities have approved the building and electrical permits that Mansfield requires to commence construction at Lindero. Electrical, structural, building and seismic plans have been reviewed and approved by COPAIPA (Dec 2013), the professional engineering institution that overlooks all construction in Salta Province. Mansfield is planning to submit additional information to COPAIPA in 2017 to obtain the permits for construction of the agglomeration and SART plants that have been added to the process design. Mansfield does not foresee any issues in obtaining the necessary permits to complete construction and commence operation at Lindero.

In addition, a formal public declaration of support for the Lindero development has been issued by the provincial government, recognizing Lindero as the priority development project for the Salta Province.

Environmental risks during the closure stage will be reduced by remediation and monitoring work. At the closure stage, soil will be contoured by heavy machinery to minimize the long-term impact of mining activity, and return the topology of the land to resemble prior conditions. However, the movement of soil, and thus the risk, will be significantly less than in the mining operations stage.

One social-environmental risk will be the completion of contracts of employment directly, or indirectly, through contractors, and the surrounding communities. It will be imperative to implement measures to mitigate this impact during the whole period of mine operation.

A significant environmental risk will also be present during the closure of facilities, which will cause significant production of non-hazardous industrial waste and hazardous products from the movement of heavy machinery. It will be essential to establish clear environmental policies with the contractors during this process.

It is the opinion of the QPs that the appropriate environmental, social and community impact studies have been conducted to date at Lindero. Mansfield have maintained all necessary environmental permits that are the prerequisites for the granting of construction permits that will need to be obtained upon completion of detailed engineering designs for the Project infrastructure.

## 1.14 Capital and operating costs

Capital and operating costs for the Lindero Project were estimated by Fortuna with the assistance of Elbow Creek, Allard Engineering Services, and Saxum Engineered Solutions (Saxum), a local engineering firm. These costs are based on the design outlined in this Report, and are considered to have an accuracy of +/-15 %. All costs are in second and third quarter 2017 US dollars (US$). No escalation factors have been applied to any costs, present or future capital. The total mine capital cost is estimated to be US$ 282 million.

Expansion (future) capital for the Project includes the Phase 2 leach pad construction in Year 3, and expansion of the ADR plant and solutions handling in the leach pad area in Year 3. The total future capital is estimated at US$ 113 million.

Closure and reclamation costs are estimated at US$ 35 million, incurred in Year 13 through Year 17.

The total LOM operating cost for the Lindero Project is US$ 10.32 per tonne of ore processed.

Costs were estimated primarily by Fortuna for mine pre-production and mine equipment costs. Saxum provided cost estimates for major and secondary equipment, buildings, infrastructure and major contracts. All equipment and material requirements are based on the design information described in this Report. Capital cost estimates have been made primarily using budgetary supplier quotes for all major and most minor equipment items, and major construction contract unit rates. Where supplier quotes were not available for minor items, a reasonable cost estimate was made based on supplier quotes in Saxum's project files. All capital cost estimates are based on the purchase of equipment quoted new from the manufacturer, or estimated to be fabricated new.

## 1.15 Economic analysis

The results of the economic analysis discussed in the Report represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Such uncertainties and factors include, among others, changes in general economic conditions and financial markets; changes in prices for gold and other metals; technological and operational hazards during the development of the project; risks inherent in mineral exploration; uncertainties inherent in the estimation of mineral reserves, mineral resources, and metal recoveries; the timing and availability of financing; governmental and other approvals; political unrest or instability; labor relations issues; as well as those factors discussed under "Risk Factors" in Fortuna's most recent Annual Information Form. Although Fortuna has attempted to identify important factors that could cause actual actions, events or results to differ materially from those described in the Report, there may be other factors that cause actions, events or results to differ from those anticipated, estimated or intended.

The Lindero project economics were evaluated using a discounted cash flow (DCF) method, which estimates the net present value (NPV) of future cash flow streams. The final economic model was developed by Fortuna using the following assumptions:

- Period of analysis of 16 years (includes one year of pre-production and investment), 13 years of production, and two years for closure and reclamation

- Gold price of US$ 1,250/oz
- Processing rate of 18,750 tpd ore
- Metallurgical recovery of 75 %
- Initial capital and operating costs as developed in Section 16.5 and 21 of this report
- Closure capital costs as outlined in Section 20

The Lindero Project shows an NPV of US$ 130 million after tax using a discount rate of 5 %, with an internal rate of return (IRR) of 18 %, and a payback period of 3.6 years, based on the LOM production plan, assumed metal prices, and integrated leaching treatment of gold and copper.

NPV and IRR display the greatest sensitivity to gold metal prices and metallurgical recoveries according to the sensitivity analysis.

The QP considers the financial model to be a reasonable estimate of the economic situation at Lindero and based on the assumptions in this Report, the Lindero Project shows a positive DCF over the LOM and supports the Mineral Reserve estimate. The mine plan is achievable under the set of assumptions and parameters presented.

## 1.16 Other relevant data and information

Goldrock commissioned Vector Argentina SA (Ausenco; 2009a, b) and Conhidro (2013) to conduct a hydrologic study of the Project area, during the detailing of the environment base line map and EIA study. As part of the study, the Rio Grande hydrologic basin was defined through the evaluation of various field parameters and review of satellite images. The basin was determined to be 1,687 $km^2$ in size. Exploration for groundwater resources was undertaken, and successfully identified possible sources.

A number of geotechnical studies were performed at Lindero and reviewed by CNI. Those studies form the basis for the pit slope estimates used in the mining model. Included in the studies were geotechnical surveys for heap leach and waste dumps. These studies are considered by the QP to be consistent with industry practices and adequate to support mine design.

## 1.17 Conclusions, risks, and opportunities

This Report represents the most accurate interpretation of the Mineral Reserve and Mineral Resource available as of the effective date of this report. The conversion of Mineral Resources to Mineral Reserves was undertaken using industry-recognized methods, and estimated operational costs, capital costs, and plant performance data. Thus, it is considered to be representative of future operational conditions. This Report has been prepared with the latest information regarding environmental and closure cost requirements.

A number of opportunities and risks were identified by the QPs during the evaluation of the Lindero Project.

Opportunities include:

- Once mining commences there is an opportunity to collect additional geotechnical data from the open pit that could support an increase in final pit slope angles, potentially decreasing stripping ratios and/or increasing Mineral Reserves
- The Arizaro porphyry system is not included in the current mine plan. However, it represents upside opportunity for the Project if a satellite operation can be developed on the deposit
- Infill drilling could support the conversion of Inferred Resources to Measured or Indicated Resources and, with the appropriate studies, to Mineral Reserves. This represents additional upside potential for the planned operation
- The Lindero porphyry gold system remains open at depth below the pit shell constrained reported reserves and resources. An area of interest has been identified by Fortuna during the drilling campaign carried out in 2016 with drill hole LDH-126 encountering 0.97 g/t Au over a 38 m interval (refer to discussion in Section 10). This is supported by historical drilling from 2007 including drill hole LDH-86 averaging 1.06 g/t Au over a 52 m interval which bottomed in mineralization. These intercepts warrant follow-up drill testing
- There are a number of local exploration targets within the concession boundary, that with further work, represent upside opportunity to identify mineralization that can potentially add to the resource base
- If historical samples are assayed for cyanide-soluble copper, there is an opportunity to construct a metallurgical model and incorporate this into the scheduling and process design. This would support optimization of blending strategies and better understanding of recoverable copper as a by-product from the SART plant. Improved copper recoveries could have a minor positive impact on the mine economics
- Performance of the equipment can be tracked with the implementation of a fleet management system to record the main key performance indicators (KPI's) which will provide an opportunity to improve utilization and time loss productivity
- Once mining commences there is an opportunity to conduct additional blasting fragmentation analysis so as to improve mining productivity and optimize mining costs

Risks include:

- Local behavior of cyanide-soluble copper is not fully understood, and cannot be modeled due to a lack of assays from historical core. Levels of soluble copper could be higher than anticipated in certain areas of the deposit requiring adjustments to mine plans and schedules to reduce the impact in the plant. The introduction of a SART plant has greatly reduced the potential impact of soluble copper at the Project
- Delaying the acquisition of fleet equipment could cause delays in the execution of certain activities. It is therefore imperative that a clear schedule of lead times is established, and equipment purchased in a timely manner to ensure on time delivery

- Fortuna calculates that two loaders are needed from Year 3 onwards, but simulations indicate that three may be required in Year 2. Once mining commences and data on loader productivity is collected, a new fleet simulation should be performed to confirm if a third loader is required in Year 2 and if so how this will affect sustaining capital expenditure
- There is a risk that two dozing machines in the original capital estimate are insufficient. Fortuna plans to mitigate this risk by renting additional ancillary equipment as required
- There is a risk that haul truck tire life of 8,500 hours is higher than can be achieved at the operation, which could lead to marginally higher operating costs than anticipated

## 1.18 Recommendations

Recommendations for the next phase of work have been broken into those related to ongoing exploration activities and those related to additional technical studies. Recommended work programs are independent of each other and can be conducted concurrently unless otherwise stated and include:

- Continued work at Arizaro that focuses on the controls of lithology, structure, and alteration on mineralization so as to determine the suitability of material as a potential feed for the Lindero plant and to support the estimation of Mineral Resources. It is recommended that a 2,000-m reverse circulation (RC) drill program (approximately 100 holes at a 75 m spacing) is conducted at a cost of approximately US$ 500,000
- An infill drill program involving the drilling of approximately 3,000-m of RC drill holes is recommended to improve the geological understanding of material planned for extraction in Years 1 and 2 of the mine. The cost of such a program is estimated at approximately US$ 750,000
- Exploration work to date on the Lindero concession has been focused on outcropping porphyry mineralization. It is recommended that the Company evaluate the property for mineralization beyond the two known porphyry systems at Lindero and Arizaro. For example, alteration zones and silica structures located within the concession, 2.5 km due south of the Lindero Project site, remain open for evaluation. Exploration work would primarily involve mapping and carry no additional cost to the Project
- It is recommended that a drill hole spacing study be conducted to establish the density of sampling that is required to reduce the grade variability to acceptable levels for specified extraction time frames in respect to infill and blast control drilling. This will be used to support the estimated meters of infill drilling. The study can be conducted either inhouse (at no cost) or by external consultants, at an estimated cost of US$ 25,000
- Additional analysis is recommended into the mine operating and ore control process, in particular, the usage of optimum dig lines for open pit grade control, with the objective of minimizing ore loss and maximizing profit. The cost of licenses and implementing such software is estimated at US$ 276,000

- A fleet management system should be considered for KPI purposes, which will provide an opportunity to improve utilization and time loss productivity. The cost of licenses and implementing such software is estimated at US$ 1.5 million
- The cement in each lift on the heap will cure for several months before another lift is placed. It may be several years before any block of agglomerated ore receives 110 m of loading. It is recommended that a long-term stacking test be conducted to see if ageing will improve the ability of the ore to support the 110-m height with less cement. The estimated cost of the testwork is US$ 20,000
- The high static holdup (adsorbed moisture) in the heap makes the secondary leach at 6 $l/hr/m^2$ inefficient when the heap height increases. There is a possibility that a surface tension modifier may reduce the amount of adsorbed moisture in the heap reducing the inventory. The estimated cost of the testwork is US$ 20,000

# 2 Introduction

## 2.1 Report purpose

This Technical Report (the Report) has been prepared by Fortuna Silver Mines Inc. (Fortuna) in accordance with the disclosure requirements of Canadian National Instrument 43-101 (NI 43-101).

Information contained within this report has been reproduced and updated where applicable from the previous Technical Report written by Kappes, Cassiday & Associates (KCA, 2016).

The Lindero Project contains two known porphyry gold-copper deposits. The Lindero Deposit is the focus of the Feasibility Study and this report; whereas the Arizaro Deposit, located 3.2 km southeast of the Lindero Deposit, is described only in terms of exploration conducted to date.

The Lindero Project is 100 % owned by Fortuna and is located approximately 420 km by road from Salta with Tolar Grande being the nearest town (population 250), located 75 km to the northeast. The mineral rights of the Lindero Property are held by Mansfield Minera S.A. (Mansfield) an Argentine subsidiary indirectly 100 % owned by Fortuna. Mansfield and the Lindero Project were purchased by Fortuna as a component of purchasing Goldrock Mines Corp. (Goldrock) in July 2016.

Fortuna is based in Vancouver, British Columbia with management offices in Lima, Peru and is listed on the Toronto (TSX:FVI), Frankfurt (FSE:F4S), and New York (NYSE:Fortuna) stock exchanges. Fortuna also owns Compañía Minera Bateas S.A.C. which operates the Caylloma polymetallic mine located in southern Peru, as well as Compañía Minera Cuzcatlan S.A.C. which owns the San Jose Mine in Oaxaca, Mexico.

The primary purpose of this Report is to describe:

- Exploration and infill drilling activities conducted since February 23, 2016 (effective date of previous Technical Report)
- Mineral Resource and Mineral Reserve estimates as of September 9, 2017 taking into account all new relevant information
- Metallurgical testwork conducted since February 23, 2016
- Geotechnical evaluations
- Updated life-of-mine (LOM), capital and operational expenditures, and economic analysis based on the above
- Updating of basic engineering to reflect changes from the optimization in mineral process design, project layout, and economic parameters in Argentina since February 23, 2016

## 2.2 Previous technical reports

The September 9, 2017 Mineral Resource and Mineral Reserve estimates supersede the Mineral Resource and Mineral Reserve estimates reported as of February 23, 2016 (KCA, 2016), and filed by Goldrock at www.sedar.com on March 2, 2016. All previously published Technical Reports on the Lindero Project are as follows (listed in reverse chronological order):

- Kappes, Cassiday & Associates (KCA), 2016. Technical Report update on the Lindero heap leach project, Salta Province, Argentina, prepared by Kappes, Cassiday & Associates for Goldrock Mines Corporation, 23 February 2016
- KCA, 2013. Technical Report on the Lindero Heap Leach Project, Salta Province, Argentina: Prepared by Kappes, Cassiday and Associates for Goldrock Mines Inc., effective date 01 May 2013
- AMEC Americas Ltd, 2010. Technical Report on the Lindero Project, Salta Province Argentina: Project No. 162667, Effective Date 02 March 2010
- Godoy and Palmer, 2008. Technical Report on the Lindero Project, Salta Province, Argentina: Technical report prepared by Golder Associates Argentina SA for Mansfield Minerals Inc., effective date 8 August 2008
- Fuchter and Rennie, 2003. Report on the Lindero Project, Salta Province, Argentina: Technical Report prepared by Roscoe Postle Associates for Mansfield Minerals Inc., effective date 16 October 2003

## 2.3 Scope of personal inspection

Mr. Eric Chapman has been employed as Fortuna's Vice President of Technical Services since January 2017 and prior to that as Mineral Resource Manager for Fortuna since May 2011. He has visited the property on several occasions, the most recent being on March 21, 2017. During his site visits Mr. Chapman has reviewed data collection, drill core, storage facilities, database integrity, procedures, and geological model construction. Discussions on geology and mineralization were held with Mansfield personnel, and field site inspections were performed including a review of surface geology of the Lindero and Arizaro Deposits, and inspection of operating drill machines. He worked with site geological personnel reviewing aspects of data storage (database) and analytical quality control.

Mr. Edwin Gutierrez has been the Manager of Technical Services for Fortuna since July 2015, and has also conducted several visits to the property. During these visits Mr. Gutierrez reviewed geotechnical observations, proposed infrastructure designs, mining methods, road access, and discussed environmental, social, permitting, operating and capital expenditure requirements with Mansfield personnel.

## 2.4 QP responsibilities

Field data were compiled and validated by Mansfield and Fortuna staff. Geological description of the samples, geological interpretations and three-dimensional (3-D) wireframes of the veins were completed by Mansfield under the supervision of a specialist porphyry consultant and reviewed by Fortuna personnel. The August 2017 Mineral Resource estimate was completed by Fortuna's Qualified Person (QP) for Mineral Resources, Mr. Eric Chapman, a Professional Geologist (P.Geo.).

The Mineral Reserve estimate was completed by Fortuna's QP for Mineral Reserves, Mr. Edwin Gutierrez, Registered Member of the Society for Mining, Metallurgy and Exploration (RM SME) and peer reviewed by AGP Mining Consultants Inc.

Metallurgical testwork was organized and supervised by Geoff Allard of Allard Engineering Services LLC, a Professional Engineer (P.E.) of Arizona and a QP.

Water balance, leach pad, and pond construction design work was organized and supervised by Denys Parra Murrugarra, RM SME and QP.

The authors of this Technical Report are QPs as defined by NI 43-101. QP responsibilities for the preparation of the different sections of this Report are shown in Table 2.1.

**Table 2.1 QP responsibilities**

| Author | Responsible for sections |
|---|---|
| Eric Chapman | 1. Summary; 2. Introduction; 3. Reliance on Other Experts; 4. Property Description and Location; 5. Accessibility, Climate, Local Resources, Infrastructure and Physiography; 6. History; 7. Geological Setting and Mineralization; 8. Deposit Types; 9. Exploration; 10. Drilling; 11. Sample Preparation, Analyses and Security; 12. Data Verification; 14. Mineral Resource Estimates; 23. Adjacent Properties; 25. Interpretation and Conclusions; 26. Recommendations; 27. References |
| Edwin Gutierrez | 1. Summary; 15. Mineral Reserve Estimates; 16. Mining Methods; 18. Project Infrastructure; 19. Market Studies and Contracts; 20. Environmental Studies, Permitting and Social or Community Impact; 21. Capital and Operating Costs; 22. Economic Analysis; 24. Other Relevant Data and Information; 25. Interpretation and Conclusions; 26. Recommendations; 27. References |
| Geoff Allard | 1.Summary; 13. Mineral Processing and Metallurgical Testing; 17. Recovery Methods (other than those listed below); 25. Interpretation and Conclusions; 26. Recommendations; 27. References |
| Denys Parra Murrugarra | 17.1.8. Leach pad; 17.1.9. Solution collection system; 17.1.10. Overliner; 17.1.11. Ponds; 17.1.24. Water balance |

## 2.5 Effective dates

The report has a number of effective dates, as follows:

- 8 May 2017; date of database cut-off for assays used in the Mineral Resource estimate for the Lindero Deposit
- 22 August 2017: date of the Mineral Resource estimate for the Lindero Deposit
- 9 September 2017: date of the Mineral Reserve estimate for the Lindero Deposit
- 9 September 2017: date of the financial analysis for the Lindero Deposit

The overall effective date of the Report, based on the supply of the financial model and finalization of the Lindero Mineral Reserves and results is 9 September 2017.

## 2.6 Information sources and references

The main information sources referenced in this Report are listed below. For a full list of all information sources referenced in this report the reader is referred to Section 27.

- Ausenco Vector, 2010. Environmental Impact Assessment, "Informe de Impacto Ambiental, Capitulo 2", October 2010
- Call & Nicholas (CNI), 2017. Geotechnical Review and Feasibility-level Slope Angles for the Lindero Open-pit mine, unpublished report prepared by Call & Nicholas Inc. for Fortuna Silver Mines Inc., May 2017
- KCA, 2016. Technical Report update on the Lindero heap leach project, Salta Province, Argentina, prepared by Kappes, Cassiday & Associates for Goldrock Mines Corporation, 16 February 2016


Some of the more commonly used acronyms used in the Report are detailed in Table 2.2.

**Table 2.2 Acronyms**

| Acronym | Description | Acronym | Description |
|---|---|---|---|
| ADR | adsorption, desorption, recovery | LOM | life-of-mine |
| AR$ | Argentine Pesos | m | meter |
| Ag | silver | mm | millimeter |
| Au | gold | Ma | millions of years |
| CIC | carbon in column | masl | meters above sea level |
| cm | centimeters | Moz | million troy ounces |
| COG | cut-off grade | MW | megawatt |
| Cu | copper | NaOH | sodium cyanide |
| CV | coefficient of variation | N | Newton |
| DC | direct current | NI | National Instrument |
| DCF | discounted cash flow | NPV | net present value |
| EBITDA | earnings before interest, tax, depreciation and amortization | oz | troy ounce |
| FY | fiscal year | oz/t | troy ounce per metric tonne |
| g | gram | ppm | parts per million |
| gpl | grams per liter | QAQC | quality assurance/quality control |
| g/t | grams per metric tonne | QQ | quantile-quantile |
| ha | hectare | RO | reverse osmosis |
| HPGR | high pressure grinding roll | ROM | run of mill |
| hr | hour | RQD | rock quality designation |
| IRR | internal rate of return | SART | sulfidization-acidification-recycle-thickening |
| kg | kilogram | SMU | selective mining unit |
| km | kilometer | t | metric tonne |
| kg/t | kilogram per metric tonne | $t/m^3$ | metric tonnes per cubic meter |
| kPa | kilopascal | tpd | metric tonnes per day |
| kV | kilovolt | V | volts |
| kVA | kilovolt ampere | yr | year |
| kW | Kilowatt(s) | µm | micrometer |
| kWh/t | kilowatt hours per metric tonne | $/t | United States dollar per metric tonne |
| l | liter | $/g | US dollar per gram |

# 3 Reliance on Other Experts

The QPs have not independently reviewed ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from Fortuna and legal experts retained by Fortuna for this information through the following document:

- Marvel, O'Farrell & Mairal, 2016: Legal Due Diligence Report of Mansfield Minera S.A. – Lindero Project. Internal report prepared for Fortuna Silver Mines Inc., 26 May 2016

This information is used in Section 4 of the Report. The information is also used in support of the Mineral Resource estimate in Section 14, the Mineral Reserve estimate in Section 15, and the financial analysis in Section 22.

# 4 Property Description and Location

The Lindero Project is located 260 km due west of Salta City, Argentina. Drive time from Salta to the Project is approximately 7 hours, over a road distance of 420 km. The nearest town to the Lindero Project is Tolar Grande (population 250), located 75 km to the northeast. Access to the Project is via National Route 51, which passes through the towns of San Antonio de Los Cobres and Olacapato; and Provincial Route 27, via Pocitos and Tolar Grande. The Lindero Project is located at (Zone 19J) 7,226,220N and 2,623,090E based on the Posgar datum WGS 84 coordinate system used for legal land tenure in Argentina (Figure 4.1).

**Figure 4.1 Map showing the location of the Lindero Project**

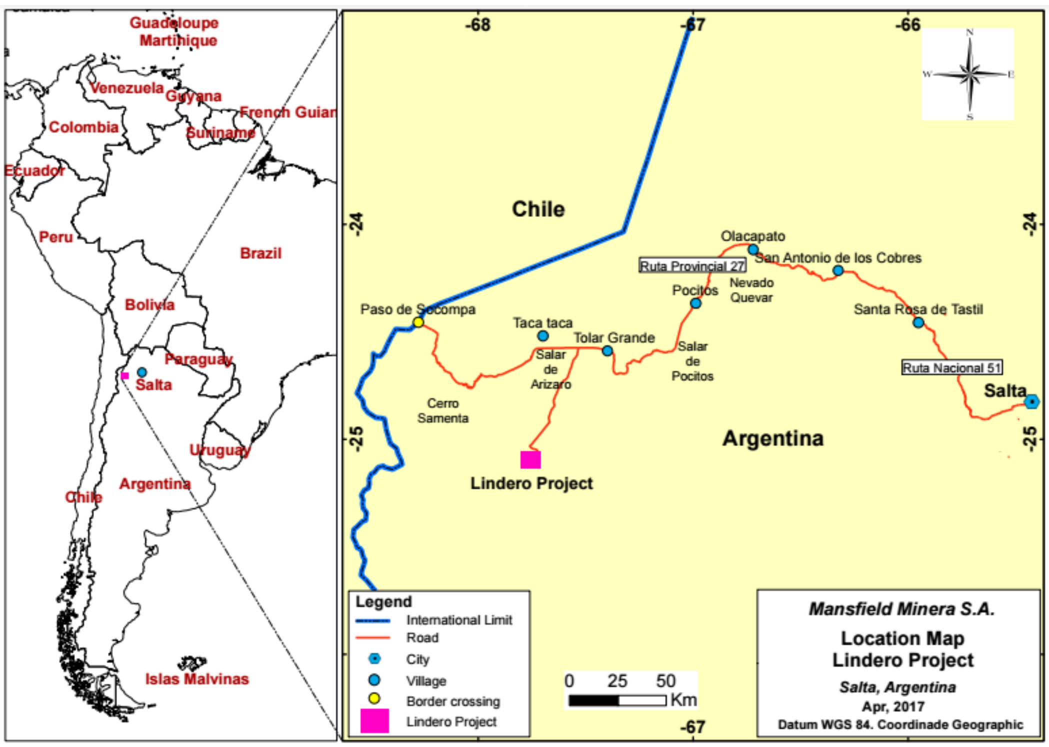



Figure prepared by Mansfield, 2017

## 4.1 Property and title in Argentina

### 4.1.1 Mineral title administration

Information in this section is taken from Godoy and Palmer (2008) and validated as still relevant.

The Argentine Mining Code, which dates back to 1886, is the legislation that deals with mining in the country. Special regimes exist for hydrocarbons and nuclear minerals. In the case of most minerals, the Mining Code dictates that the owner of the surface is not the owner of the mineral rights; these are held by the Provincial State. The State is also bound by the Code to grant to whoever discovers a new mine the rights to obtain a "mining concession".

Owners must comply with three conditions; payment of an annual fee, investment of a minimum amount of capital, and the carrying out of a reasonable level of exploration or works. Failure to do so could lead to forfeiture of the property back to the State.

The administrative organization for mining-specific regulation is the Federal Ministry of Planning, Public Works and Investment which has a Mining Department headed by the Secretary of Mines. The Argentine Mining Law is a federally drafted law implemented through bi-lateral accords with the Provinces that have jurisdiction over mineral rights.

In 1993, Argentina implemented a new Mining Investment Law (No 24,196), a Mining Reorganization Law (No. 24,224), a Mining Modernization Law (No 24,498), a Mining Federal Agreement (No. 24,228), and a Financing and Devolution of IVA Law (No. 24,402). Amendments were also made to update the Mining Law (Decree 456/97). These amendments offered attractive economic incentives for exploration and mining to foreigners, and include both financial and tax guarantees. This group of laws also creates the basis for federal-provincial harmonization of mining rules such as import duty exemptions, unrestricted repatriation of capital and profits, and a 3 % cap on Provincial royalties.

In 2001, Law 25.429 "Update of the Mining Investment Law" was passed and in March 2004 approval was reached for a key provision of the Law allowing refund of the IVA (or value added tax) for exploration related expenses incurred by companies registered under the Mining Investment Law.

In 1995, Law N° 24.585 Environmental Protection (Mining Code) was passed and provides regulation for operations and environmental reporting at the exploration and exploitation levels.

In summary, the major changes to the mining code encompass:

- Exploration areas have been increased to a maximum of 200,000 ha per company and per Province
- Exclusive aerial prospecting areas of 20,000 km² are also permitted
- A guarantee of tax stability for 30 years
- Expenditures made in prospecting, exploring and construction of mining installations are tax deductible and value added taxes are recoverable
- Imports of capital goods, equipment and raw material are exempt from import duties
- Royalties will not exceed 3 % of the ex-mine value of the extracted mineral
- Environmental funds to correct damage are required and are deductible from income taxes; a National system of permanent mining environmental monitoring is set up. Implementation at the provincial level has been variable
- Systemization and digital conversion of mining property registers has been implemented to varying degrees of success in each Province and the definition by geographic co-ordinates now establishes mining rights

### 4.1.2 Mineral title types

A Cateo (exploration right) is an area of land staked during the early stage of exploration. In Argentina, this is called the "Prospecting Stage". Cateos may be contiguous or separate and are subject to certain restrictions on size. A Cateo is sub-divided into 500 ha units with a defined exploration term determined by the cumulative number of units comprised. The maximum possible term is 1,100 days for the maximum lease size of 10,000 ha commencing from the grant date. Prior to its expiry, the holder of a Cateo may apply at any time for conversion to one or more 'Manifestación de Descubrimiento' (application period for a mining lease), which when granted, becomes a 'Mina' (mining lease) right within the perimeter of the Cateo up to its full area. Minas can also be established as the result of a discovery in open ground. A mining lease is subdivided into a minimum of two pertenencias, which are generally 6 ha for small deposits and 100 ha for larger, disseminated deposits.

To apply for a Manifestación, the applicant must present a representative sample of the outcrop as the discovery and indicate its coordinates and the surrounding area to be covered by the title. After about a six-month period the Manifestación will be registered and converted to a 'Mina' or mining lease. Conversions and applications are administratively dependent and not date-dependent and are therefore not automatic. Processing times from one provincial jurisdiction to another may vary.

The application authority for the Lindero Project is the Salta Mining Secretary of Salta and the mining concession is granted by the Salta Mining Court house.

### 4.1.3 Surface rights

Access over surface property rights in Argentina for mining projects are granted by the mining authority for the Province (in Salta this is a Mining Judge), who are required to communicate with the surface owners and ensure that they cooperate with the activities of the exploration/mining companies. Notice can be difficult due to delayed filing of personal property title changes and registry as well as limited staffing and mobility of the relevant authorities. All mining and water rights must also be publicly announced by specific news agencies.

The Argentine legal and constitutional system grants mining properties all the guarantees conferred on property rights, which are absolute, exclusive and perpetual. Mining property may be freely transferred and purchased by foreign companies.

### 4.1.4 Environmental regulations

Mining in Argentina is governed by both federal legislation and provincial laws and decrees. The permitting process, as well as regulatory activities during development and operations, is managed by the Province. A mine is put into operation in two phases, starting with an Environmental Impact Assessment, and then a sector permitting phase. Approval of the Environmental Impact Assessment allows mining development to proceed, subject to obtaining sector permits for specific project facilities.

Law No. 24,585, which came into effect in the 1995, incorporated a complementary title in the Mining Code relevant to the Environmental Protection for Mining Activity regulations. It also recognized complementary regulations approved by the Federal Mining Council (COFEMIN), completing the principles sustained in Article 41 of the National

Constitution by setting a legal regime whose premise is to preserve the right of all inhabitants to enjoy a healthy environment through the balanced development of economic activities and processes that support them.

Law No. 24,585, established in the Federal Mining Council, details the necessary instruments for environmental administration of mining activities including; the "Environmental Impact Assessment" (EIA), whose presentation is compulsory for the mining activity holders before the beginning of operations; and the Statement of Environmental Impact, a statement issued by the relevant authority as an approval to the corresponding EIA.

Consequently, current environment management with regard to mining is based on the following legal regime:

- National Constitution
- Title XIII Section Second of the Mining Council
- Supplementary regulations and minimum requirements
- Provincial decrees that established the application authority of the Title XXI Section Second of the Mining Council
- Implementation of Provincial Decrees of the Supplementary Regulation and Resolutions of institutional character and of administrative internal procedure that complete environmental mining management (Law No.7,070 for Salta Province)

Law No. 24,585 covers the following activities:

- Prospecting, exploration, exploitation, development, preparation, extraction, and storage of mineral substances
- Crushing, milling, extraction, pelletization, sintering, briquetting, primary manufacturing, calcination, melting, and refining; stone sawing, faceting, cutting, and polishing processes; any other process derived from new technologies; and the management and disposal of any kind of waste

The regulations also take into account centralized and decentralized entities, and the national, provincial, and municipal companies that develop such activities.

Argentine Law No. 24,585 establishes that anyone performing mining activities is responsible for all environmental damage produced as a result of the non-fulfillment of the regulations. It is irrelevant whether the damage is caused directly by them or indirectly by persons under their control, or whether it is caused by the risk or vice inherent in the activity. Likewise, the holder of a mining right is jointly responsible for the damage caused by persons involved in mine exploitation.

Under the Environment Mining Management Procedure, prior to the commencement of mining activities that are subject to regulation, companies must prepare and have an environmental impact review (EIR) approved. The application authority evaluates and approves the EIR by means of a Statement of Environmental Impact for each of the effective implementation stages. The EIR must be renewed and lodged every two years.

The Statement of Environmental Impact (detailing actual site activity) must be presented to the authorities twice a year, and include submission of a report that contains results of any environment protection actions that were executed, and detailed information on the new environmental events that have occurred. Mansfield has submitted this report every six months since 2012. Under Law No. 25,675, industrial activities must obtain special insurance covering environmental risk. Fortuna has included this insurance in the financial analysis.

## 4.2 Tenure

The Lindero Property was initially staked under a manifestation-of-discovery and was subsequently covered by legally-surveyed pertenencias, which were approved by, and registered in the Mining Court of Salta (File # 16.835) under the name of Onix Mine on March 30, 2001. The Lindero–Arizaro Property/Onix Mine was subsequently registered as a gold–copper mine in the name of Mansfield Minera S.A., an indirectly wholly-owned subsidiary of Fortuna.

The mineral tenement holdings cover 3,500 ha (Figure 4.2), and comprise 35 pertenencias, each of 100 ha. The holdings are constrained by the Gauss Kruger Posgar coordinates listed in Table 4.1.

**Table 4.1 Tenure boundary coordinates**

| Point | Easting | Northing |
|---|---|---|
| A | 2622009.93 | 7227690.07 |
| B | 2628009.90 | 7227690.07 |
| C | 2628009.90 | 7221690.11 |
| D | 2622509.93 | 7221690.11 |
| E | 2622509.93 | 7223690.09 |
| F | 2622009.93 | 7223690.09 |

**Figure 4.2 Location of the Lindero Project mining concession**

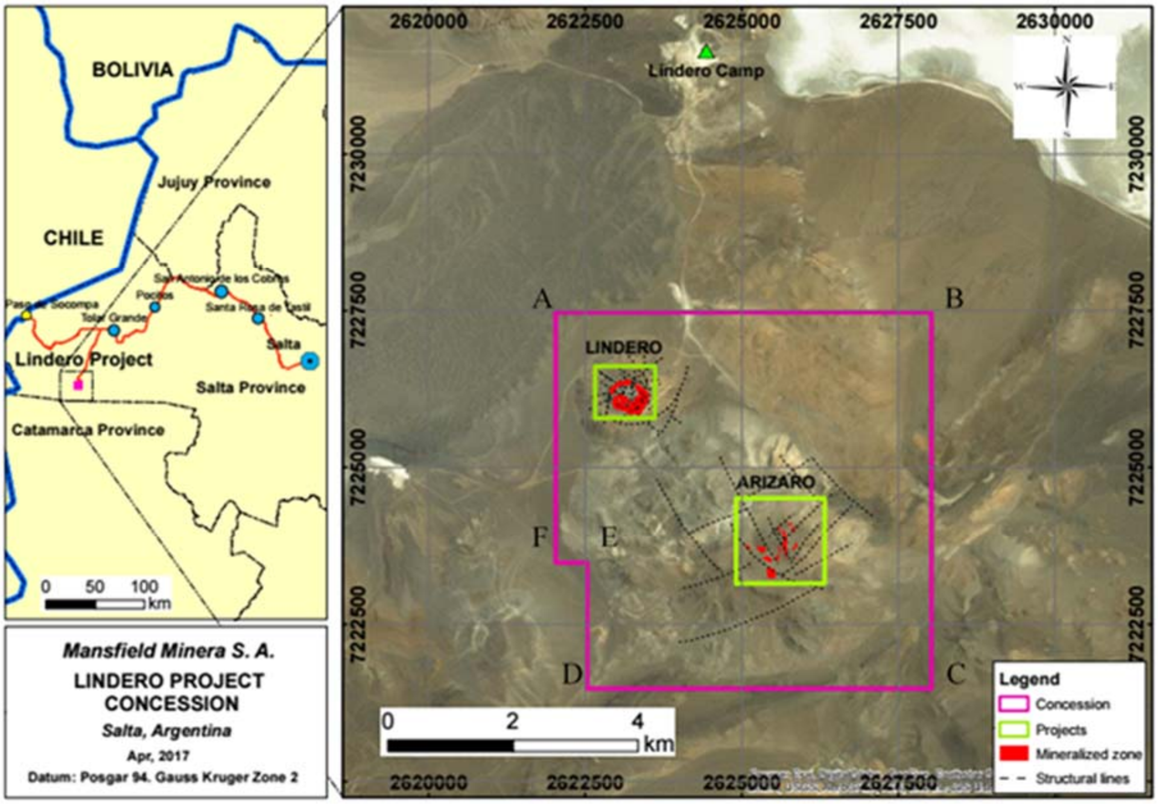



Figure prepared by Mansfield, 2017

The mining area boundaries have been appropriately surveyed to meet jurisdictional requirements. There is no expiry date on the pertenencias, providing Fortuna meets expenditure and environmental requirements, and pays the appropriate annual mining fees. The expenditure and environmental requirements have been met and the annual mining fees are AR$ 112,000 (approximately US$ 6,600 – September 2017). The mining fee is paid a year in advance in two parts, the first instalment on June 30 with the second paid on December 31. Such payments have been made as required, and as at the effective date of this Report, the tenements are in good standing.

## 4.3 Surface rights

Access over surface property rights in Argentina is obtained through the Mining Authority, who are required to communicate with the surface owners and ensure that they cooperate with the activities of the exploration/mining companies. Notice can be difficult due to delayed filing of personal property title changes and registry, as well as limited staffing and mobility of the relevant authorities.

Private property rights are secure rights in Argentina, and the likelihood of expropriation is considered low. The Argentine legal and constitutional system grants mining properties all the guarantees conferred on property rights, which are absolute, exclusive and perpetual. Mining property may be freely transferred and purchased by foreign companies.

Surface rights for the Lindero Project are owned by the provincial state (Propiedad Fiscal) of Salta. There are no reservations, restrictions, rights-of-way or easements on the Project to any third-party. Mansfield has a registered camp concession license, filed in the Mining Court of Salta under File 17,206, which covers the use of land at the Arita and Lindero camps for construction and operation of camp facilities and permits the company to use the camp area.

There are two camps at the Project, located less than 500 m apart. The camp area is situated off the Lindero–Arizaro Property/Onix Mine area, as shown in Figure 4.2. Under File No. 18,387, Mansfield has a right-of-way already granted and surveyed. Mansfield also has the following surface rights: File 19.200 area for water rights, at the water well location; File 21.995 water pipeline surface rights, for the transportation of water for production; File 22.093 plant and process areas; File 22.094 waste dumps; File 22.095 open pit; and File 22.096 camp and facilities area.

The easements for the plant, waste dump, open pit and other facilities, while not required as surface rights, are granted by the underlying mining tenements (the pertenencias) which were applied for, as the law does allow Mansfield to have double tenement holdings via easement over the area of the mine and process plant.

Mansfield is planning to submit additional information to COPAIPA in 2017 to obtain the permits for construction of the agglomeration and SART plants that have been added to the process design. Mansfield does not foresee any issues in obtaining the necessary permits to complete construction and commence operation at Lindero

## 4.4 Royalties

A 3 % provincial royalty "boca mina" is payable on revenue after deduction of direct processing, commercial and general and administrative costs. There are no royalties payable to any other third party.

## 4.5 Environmental aspects

Mansfield is in compliance with Environmental Regulations and Standards set in Argentine Law and has complied with all laws, regulations, norms and standards at every stage of the Project.

Since the discovery of gold mineralization at the Project in 2000, Mansfield has presented more than 20 environmental reports describing various activities such as extraction of samples at initial stages, soil sampling, a program of geophysical surveys, and details of access roads, drilling programs, camp installation, and runways. These reports consist of a brief description of the environmental baseline, the Project, environmental impact, and ways to prevent and mitigate that impact. On many occasions, the Government of Salta Province has inspected the various activities.

In December 2007, Mansfield presented an extensive environmental baseline report (EBL), completed by Vector Argentina, to the Secretariat of Mining for Salta Province. The report included sections on geology, geomorphology, hydrology, sociology, archaeology, local flora and fauna, soil types, and climate and air quality. The EBL was accepted by the Mining Judge of Salta after being examined by environmental technicians of the Secretariat of Mining and the Provincial Secretariat of Environment.

In November of 2010, Mansfield submitted the environmental impact assessment (EIA) report (Ausenco Vector, 2010) for the Lindero Project, and in November 2011, received approval of this EIA through the issue of the Declaración de Impacto Ambiental (DIA).

In December 2015, Mansfield filed the biannual updated EIA report on the Lindero Project as established by environmental law. The next update and submission of this major report is due by March 2018.

Mansfield has no knowledge of any further environmental liabilities related to any of the other concessions connected with the Project.

A summary of the major environmental related activities is detailed in Section 20.

## 4.6 Permits

Specific approvals and permits are required for many aspects of the Project and are detailed in full in Section 20.

## 4.7 Comment on Section 4

In the opinion of the QPs:

- Fortuna was provided with a legal opinion that verified at the time of completion that the mining tenure held by Mansfield for the Project is valid and that Fortuna has a legal right to mine the deposit
- Minimum work requirements under the tenure grant have been met
- Annual land usage and environmental compliance reports have been lodged
- Additional permits are required for Project development and Fortuna has a well-defined legal path forward for permitting
- Fortuna has no knowledge of any additional environmental liabilities related to any of the other concessions connected with the property
- Surface rights are held by the Salta Province. Surface rights for construction of a mining operation and plant have not been granted from the Provincial authorities. Development of such infrastructure will require additional negotiation and potentially, supporting studies

Fortuna advised that to the extent known, there are no other significant factors and risks that may affect access, title or right or ability to perform work on the Project. The information discussed in this section supports the declaration of Mineral Resources. Mineral Reserves and the development of a mine plan with an accompanying financial analysis.

# 5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

## 5.1 Access

Access to the Project from Salta is by paved roads for about 100 km and by all-weather dirt roads for about 350 km along National Highway 51 and Provincial Highway 27. The access route is from Salta to San Antonio de Los Cobres on National Highway 51 (140 km), continuing to the small town of Tolar Grande on Provincial Highway 27 (210 km) and then traveling southward along an undesignated route across the salar to the camp at Mina Arita (80 km). Access to the Project from the camp is via dirt roads for approximately 5 km. Drive time from Salta to the Property is approximately 7 to 7.5 hours.

Several bulldozer roads were constructed on the Project to provide access for drilling and trenching. Other areas within the Project are accessible by four wheel-drive (4 WD) truck or all-terrain vehicle (ATV).

Site access will be primarily by existing national and provincial roads that will be used during construction and operation. These roads are mainly consolidated dirt roads, some of which will require improvements to accommodate transport of special oversized or heavy parts during construction. Limited site improvements for roads will be required.

An international railway line joins the city of Salta with Tolar Grande and the deep-sea port of Puerto Anganos on the Chilean coast. This line is currently operational, and it is expected that the growth of mining activity in the puna could make it economically viable. A new transfer area on the outskirts of Tolar Grande could be built in the future to make use of this area if the appropriate studies indicate this is viable.

The Project is located 60 km from the Salta–Antofagasta railway line, 130 km from the 345 kV Salta–Escondida power transmission line, and 160 km from the Puna gas pipeline.

## 5.2 Climate

The Project is in the Argentine puna, a cool, arid zone with a minimum elevation of approximately 3,500 to 4,000 m. The climate is generally dry and windy; it can be cold and snowy during storms.

In 2007, Goldrock installed a weather station in the Project area. Based on records from the station, the Project conditions include:

- Average annual rainfall of about 37 mm, distributed irregularly throughout the year. The most severe rainstorms registered were in January 2008, during which 82 mm fell in two days. The 100 year 24-hour rainfall event is 127 mm. Since the Goldrock weather station only has the last five to six years of weather data, the heap leach and site water balance were based on historical rainfall data from the Salar de Pocitos, Unquillal and Hombre Muerto weather stations. Based on these three stations that average rainfall was approximately 45 mm
- The annual average temperature is 3.9°C. The warmest months are January, with an average temperature of 8.0°C, and February, with an average temperature of 8.4°C. The coldest month is July, with a monthly average temperature of -2.6°C.

The maximum average high for a month is 30°C and the minimum average low for a month is -10°C

- The annual average relative humidity is 33 %, with a monthly average maximum of 45 % in July, and an average minimum of 27 % from November to February
- The annual average barometric pressure is 75 kPa
- Annual pan evaporation is about 2,700 to 2,800 mm

It is expected that any future mining operations will be able to be conducted year-round.

## 5.3 Topography, elevation and vegetation

The puna area is characterized by north–south-trending mountain ranges rising to over 6,000 m and intermontane basins with attendant salt lakes (salars) with a base level of 3,500 m. Elevations on the property range from 3,700 m to 3,990 m above sea level. The Lindero Deposit is located on the upper section of a conical hill which emerges from an extensive flat area.

The almost total lack of rainfall in the puna determines a vegetational floor that corresponds to the "Province of puna" shrub steppe, an herbaceous steppe with vegetal associations consisting of añagua, lejía y tola, añagua y rica-rica, iros, muña-muña, viravira, and chachacoma, among others. There are many areas with no vegetation whatsoever.

The fauna corresponds to the Andean Patagonian sub-region, Andean district, represented mainly by camelids (llamas, vicunas, and guanacos). Donkeys have been incorporated into the landscape for the last 50 years and compete for pasture with other herbivores.

The Lindero Project falls within the 1,687 $km^2$ Rio Grande watershed. This watershed takes Salar de Arizaro as the local base level. The watershed has little runoff; the majority of the irregularities in the field consist of ephemeral water courses caused by sporadic rainfall. The extremely arid conditions of the region, combined with scarce to null vegetation, allows the little water that does fall to filter rapidly to the subsoil with limited erosion.

## 5.4 Infrastructure

The nearest town to the Lindero Project is Tolar Grande (population 250). Existing Project infrastructure comprises a climate station and two camps that can accommodate approximately 100 people; the camps are located adjacent to the Project tenure within a registered camp concession license (refer to Section 4). The camp has mobile satellite phones, satellite internet, basic link unit (BLU) radios for long range communication and ultra-high frequency (UHF) radios for local connections. Currently, power requirements are supplied by a diesel generator.

Staffing requirements for the Project will primarily come from Argentina at large. Argentina has sufficient experienced and skilled professionals to run the proposed Lindero operation. Qualified engineers and geologists have been involved in several gold mines in operation in the country since 1994 as well as in Chile and Bolivia. Regionally, finding the right people is more difficult, as Salta Province does not have any metallic mines operating, but some local mining engineers have experience running borax mines.

However, in neighboring Provinces, including Jujuy, where Mina Pirquitas and Mina Aguilar are situated, and Catamarca, where Bajo La Alumbrera is situated, qualified professionals, operators, and laborers may be recruited. Personnel may also be sourced in Salta City.

A main camp will be constructed for the Lindero Project and will be used for both construction and permanent operations housing. A permanent accommodation camp for 320 beds will be built for the LOM operation. For the construction period, temporary accommodations will be implemented to accommodate the peak of construction manpower estimated at 600 people. The on-site camp complex will include a dining and recreation hall.

Other planned on-site Project buildings include:

- Administration building
- Process shop
- Warehouse
- Truck shop
- ADR plant building
- SART filter building
- Refinery building
- Cyanide mix building
- Reagents storage building
- Assay and metallurgical laboratory
- Guard shack

Power will be generated on-site by a contractor through an 8 MW capacity diesel oil plant.

Process water requirements are 105 $m^3$/hr and will be sourced from two existing wells located 13 km southeast of the Project site. An additional well is required and will be drilled as part of construction activities.

Waste will be disposed of in compliance with local regulations. Hazardous waste, such as oil filters, used oil, and the like will be transported to Salta and disposed of appropriately. Mansfield is registered at the local authority as a hazardous waste producer.

Additional details on the Project infrastructure are provided in Section 18.

## 5.5 Sufficiency of surface rights

The Project site infrastructure has a compact layout footprint of approximately 60 ha, located around the proposed open pit, see Section 18 for further details. The projected open pit and supporting infrastructure (other than the water borefield and camps) is located well within the area of surface rights and 3,500 ha mineral tenement owned by Mansfield.

## 5.6 Comment on Section 5

In the opinion of the QPs, the existing and planned infrastructure, availability of staff, the existing and planned power, water, and communications facilities, the methods whereby goods are or will be transported to the Project site, and any planned modifications or supporting studies are well-established, or the requirements to establish such, are well understood by Fortuna, and can support the declaration of Mineral Resources and Mineral Reserves and the proposed mine plan.

# 6 History

## 6.1 Ownership and exploration history

Gold–copper mineralization associated with potassic alteration was first discovered by Goldrock geologists at the Arizaro prospect in November 1999, and led to claim staking. The discovery was the result of a regional exploration program undertaken by Goldrock with the participation of Teck Corporation. The area had no previous history of exploration or metals mining, although Goldrock had previously discovered the Rio Grande property about 12 km to the northwest. The Arizaro prospect and surrounding area were explored using reconnaissance and detailed geological mapping, soil geochemistry (talus fines), and trench sampling and mapping during 2000 and early 2001. As a result of this work, the mineralization at what is now the Lindero Deposit was identified in September 2000 (Kesting and Huidobro, 2001).

From April 2002 to March 2003, Rio Tinto had an option on the Property with Goldrock, during which time geological mapping, ground magnetics and induced polarization (IP) geophysical surveying, road-cut and trench sampling, drilling of 10 core holes (3,279 m) at Lindero and two core holes at Arizaro (629 m), metallurgical testwork, and an inhouse preliminary Mineral Resource estimate for the Lindero Deposit were performed (Ruiz et al, 2003). As the tonnage and grade estimate did not meet Rio Tinto's corporate targets, the option was not exercised.

Goldrock resumed as project operator, and between 2005 and 2009 completed additional trenches, metallurgical testwork, geological mapping and a core-relog program, and generation of a 3-D geologic model for the Lindero Deposit. From 2005 to 2010, Goldrock drilled 129 drill holes (34,857 m) at the Lindero Deposit. From 2010 to January 2013, Goldrock drilled 27 holes (8,223 m) at the Arizaro Deposit.

A Pre-Feasibility Study for the Lindero Deposit was completed by AMEC in 2010, assuming a production throughput of 30,000 tpd (AMEC Americas Ltd., 2010a; 2010b). In 2012 and 2013, Goldrock commissioned KCA to complete a Feasibility Study (the 2013 Feasibility Study) using a reduced throughput of 18,750 tpd (KCA, 2013).

In 2015, Goldrock commissioned KCA to work with local engineering firms in advancing the engineering design for the Project to a basic engineering level, and update the 2013 Feasibility Study incorporating some design changes requested by Goldrock. A new Feasibility Study (the 2016 Feasibility Study) incorporating these design changes, additional metallurgical testwork, and updated costs and gold price assumptions was prepared by KCA in 2016 (KCA, 2016a).

In July 2016, Fortuna completed the acquisition of all issued and outstanding shares of Goldrock, making it an indirectly wholly-owned subsidiary of Fortuna. Upon completion of the transaction, Fortuna continued to advance the optimization of the 2016 Feasibility Study through the drilling of 12 drill holes (4,461 m) as well as conducting tradeoff metallurgical tests and detailed engineering revisions with the objective of reaching a construction decision for the Lindero Project by the end of the second half of 2017 (Fortuna, 2017).

## 6.2 Prior Mineral Resources and Mineral Reserve estimates

Mineral Resources were first estimated for the Lindero Deposit during 2003 (Fuchter and Rennie, 2003) and updated in 2008 (Godoy and Palmer, 2008), and again in late 2009 (AMEC Americas Limited, 2010a, 2010b).

Mineral Resources and Reserves for the Lindero Deposit were updated again in 2013 (KCA, 2013). Mineral Resources and Mineral Reserves were further revised in 2015 by MDA and DKT using updated operating costs and gold price assumptions and reported in the 2016 Feasibility Study (KCA, 2016).

All of the above are regarded as prior estimates. Current Mineral Resources and Mineral Reserves are reported in this Technical Report.

# 7 Geological Setting and Mineralization

The geology of both the Lindero Deposit and Arizaro Deposit are discussed in this section. The Arizaro Deposit was discussed in detail by KCA (2016a), and that information has been summarized in this section of the Report. Limited additional exploration has occurred on the deposit since that report. Fortuna has focused on the Lindero Deposit and deferred work on the less explored Arizaro Deposit.

## 7.1 Regional geology

The Andean volcanic arcs are concentrated along the north-trending axis of the high Andes puna region, and along several northwest-trending structural transverse zones. The western part of the Salta Province is underlain by Paleogene and Neogene continental volcanic arcs and related sedimentary rock of the Andean cycle (Figure 7.1). Sedimentary rocks are deposited in large back-arc continental basins similar to the Siete Curvas basin, a portion of which is active and includes the Salar de Arizaro basin.

The Siete Curvas basin is a 100 km by 130 km extensional pull-apart basin filled with continental rocks including immature red beds, volcanic rocks, and evaporites. The active part of this basin is known as the Salar de Arizaro basin. South of the Salar de Arizaro basin, basement exposures are characterized by high- to medium-grade metamorphic rocks of Proterozoic age. To the east of the basin, Cambrian–Ordovician intrusive rocks and associated platform-shelf clastic sedimentary rocks with submarine volcanic facies crop out. The western part of the Salar de Arizaro basin is underlain by Cambrian to Ordovician and Permian to Jurassic intrusive rocks, and by volcanic rocks and porphyries of Eocene–Oligocene age. All these units are covered by Pliocene volcanic rocks of the Andean arc.

The Siete Curvas basin is bounded to the north and south by northwest transverse volcanic arcs of the Andean cycle. The Lindero and Arizaro Deposits are located in the southern Archibarca volcanic belt, (Figure 7.1), which is characterized by adjacent or superimposed stratovolcano complexes commonly manifested by eroded volcanic cones. Rocks exposed in these belts include andesite and dacite porphyries and coeval volcanic and volcaniclastic rocks.

The Siete Curvas basin is structurally bounded by large regional structures: to the north by the Calama–Olacapato–El Toro Transverse Structure, and to the south by the Archibarca Transverse Structure (Figure 7.1). The transverse zones are interpreted to be surface expressions of ancient deep crustal trans-lithospheric structures, which were initially related to the opening of the proto-Atlantic Ocean in the Cretaceous, and have been periodically reactivated (Richards, 2000). The East Fissure fault zone and the Pocitos linear zone bound the basin to the west and east, respectively (Figure 7.1). These regional north–south-trending structures are interpreted to represent suture zones of accreted terranes, similar to the West Fissure fault zone of northern Chile (Richards, 2000). Presently the trans-lithospheric structures mark the transition from flat-slab to steep-slab subduction off the west coast of South America.

**Figure 7.1 Regional geology map of the Lindero Project**

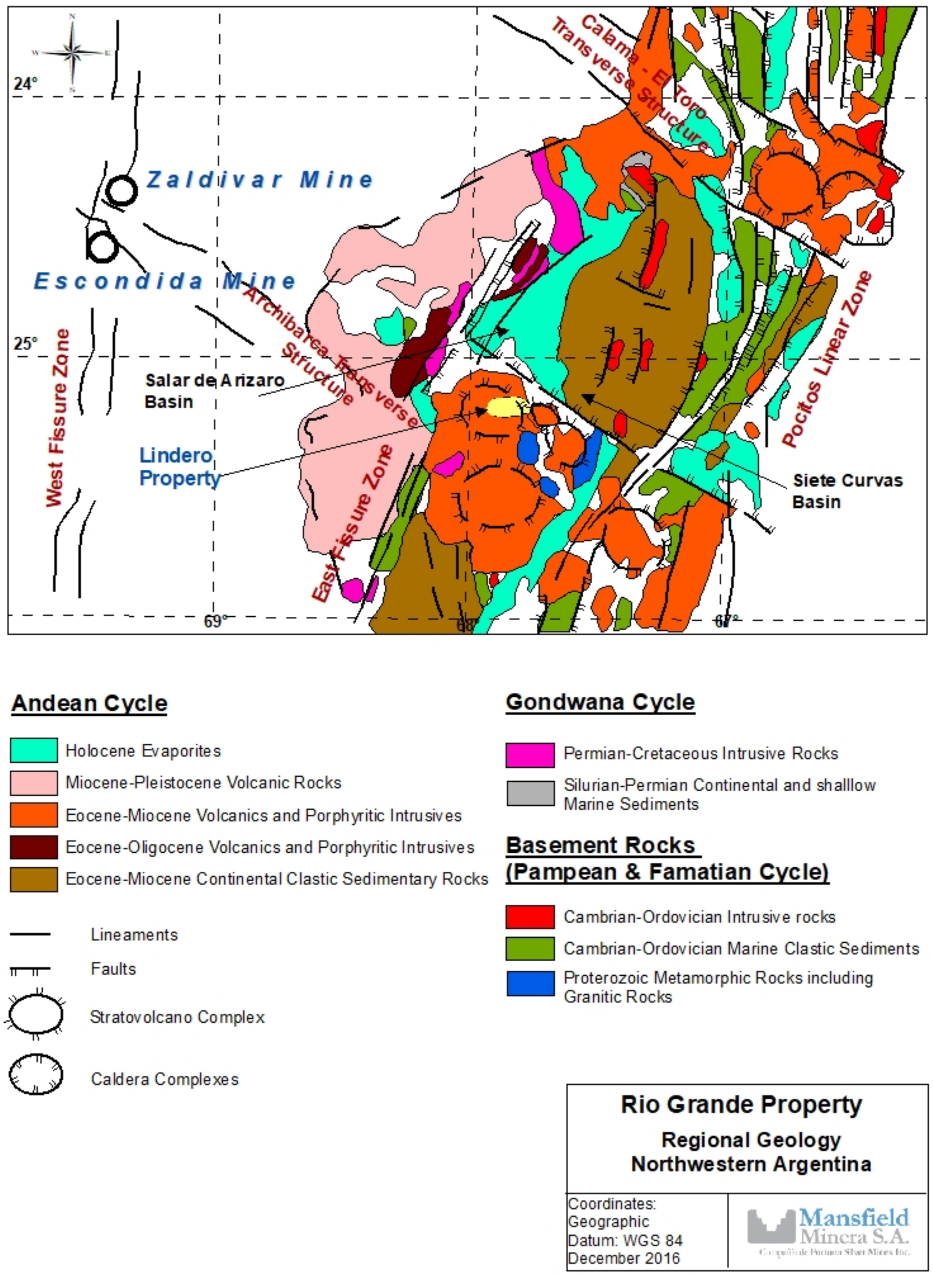



Figure prepared by Mansfield 2001, modified in 2016

## 7.2 Local geology

The Lindero and adjacent Arizaro Deposits are located in the Archibarca volcanic arc at the southern margin of the Salar de Arizaro basin (Figure 7.1). The deposits are part of the Arizaro volcanic-intrusive complex, which consists of diorite to monzonite porphyritic stocks, dacitic plugs, and numerous andesitic and dacitic radial dykes (Figure 7.2). Ordovician granite basement rocks crop out to the north and east of the

Lindero Deposit. These are unconformably overlain by Paleogene red-bed sedimentary rocks that dip 10 to 30 degrees southwesterly.

This sequence is intruded by Miocene porphyries of the Lindero–Arizaro complex that host gold mineralization. The Lindero igneous center consists mostly of shallow intrusive rocks. The Arizaro system 3.2 km to the southeast also includes volcaniclastic mass-flows, tuffs, and fine-grained lava flows. Dating by $^{40}Ar/^{39}Ar$ methods indicates the igneous rocks range from 16.75 to 15.21 Ma at Lindero, and 16.35 to 15.66 Ma at Arizaro (Dow, 2002). All units are overlain by Pleistocene basaltic flows and volcaniclastic rocks from the Archibarca volcanic center.

**Figure 7.2 Local geology of the Lindero Project**

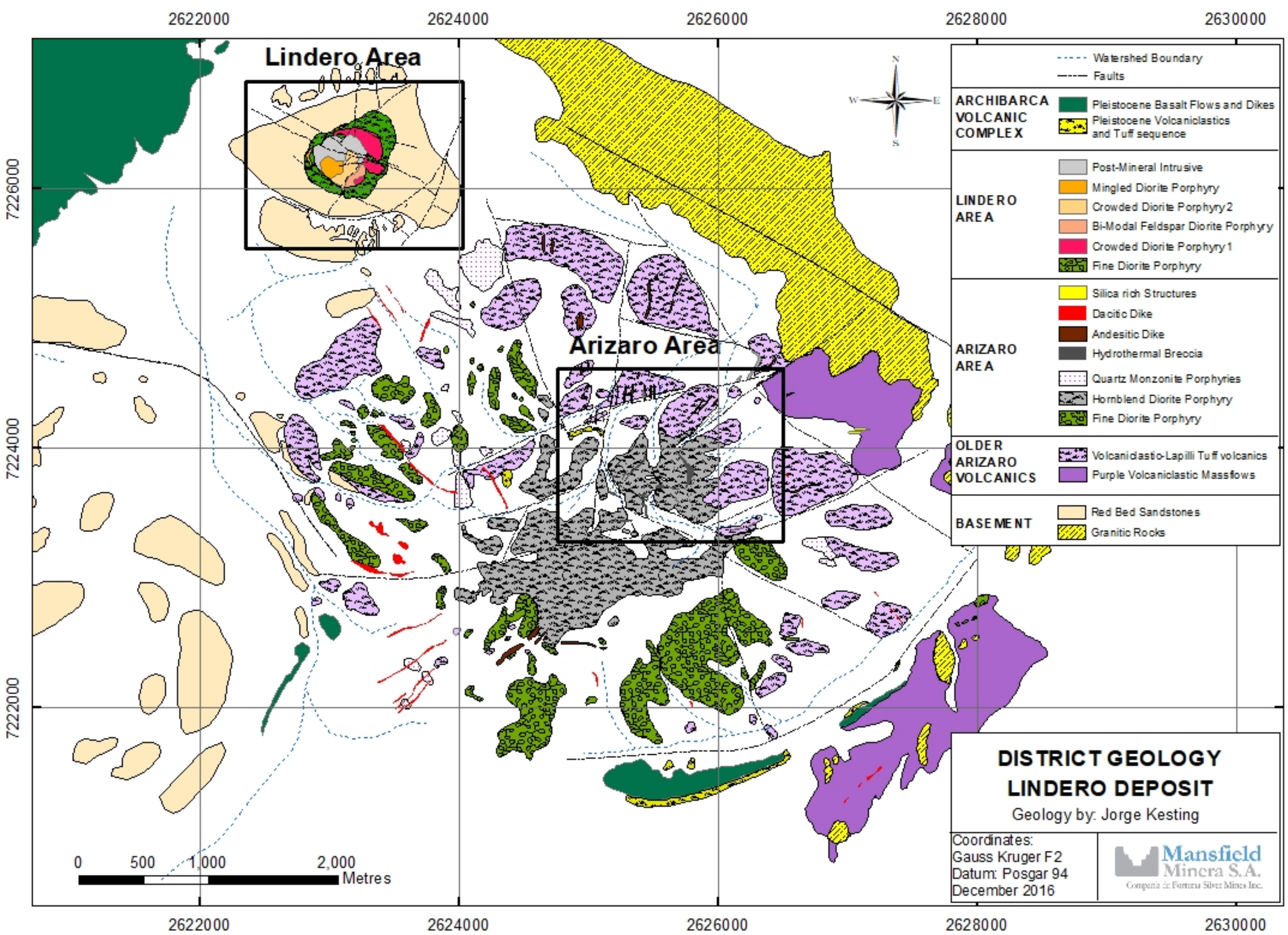



Figure prepared by Mansfield 2001, modified in 2016

## 7.3 Lindero Deposit

### 7.3.1 Lithology

Gold mineralization at Lindero is hosted in a multiphase Miocene intrusive complex. The complex is elongate to the northeast, 750 by 600 m at the surface, and cuts recrystallized

Paleogene sandstones, siltstones and mudstones (Figure 7.3). Most drill holes near the border of the complex pass into the red-bed sequence at depth, which suggests an upwardly-flaring shape (Figure 7.4).

**Figure 7.3 Geology of the Lindero Deposit (lithology codes detailed in text)**

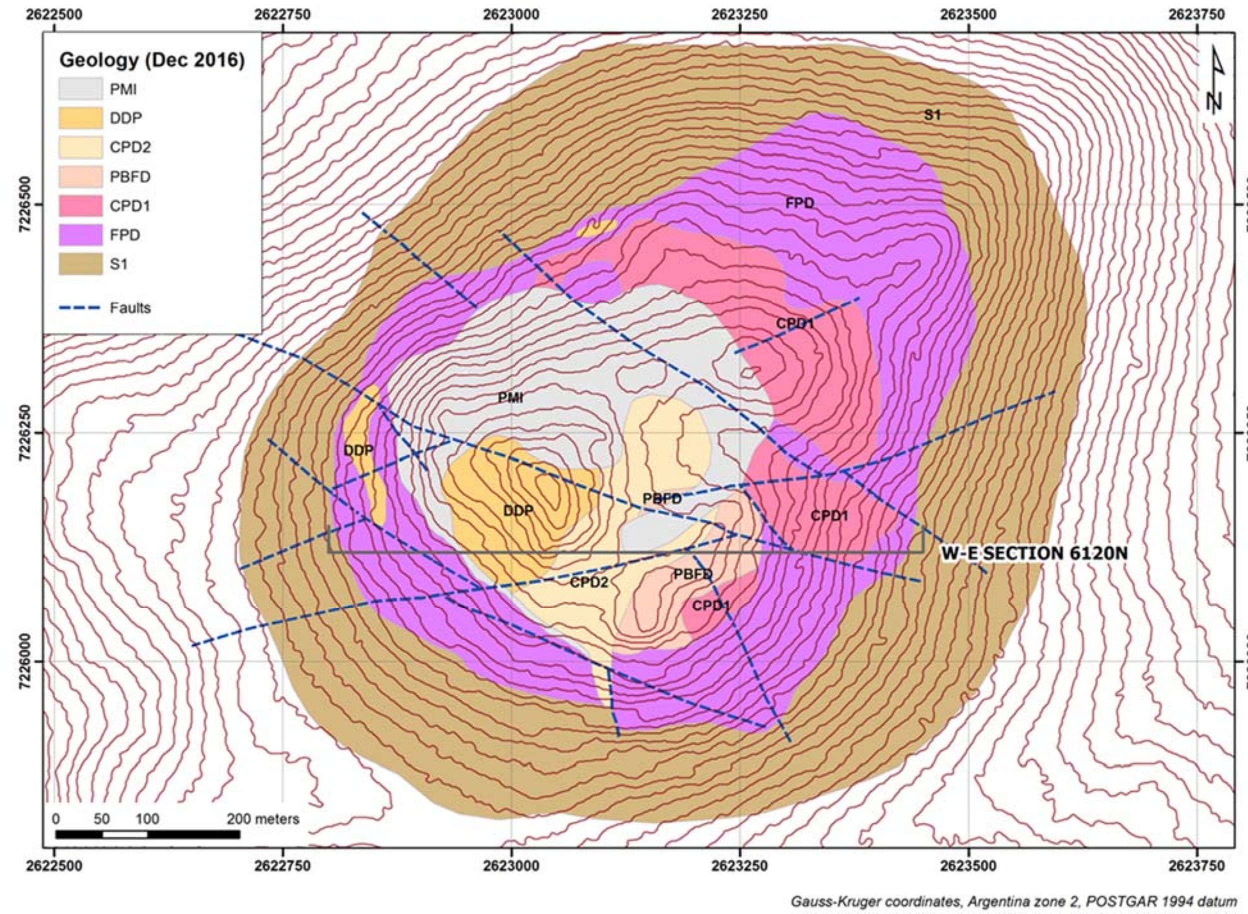



Figure prepared by Mansfield, 2017

Rock units recognized at Lindero are briefly described below, in order from oldest to youngest.

- **Ordovician granite (rock code OG)** is distinguished from the Miocene intrusive rocks by its medium- to coarse-grained equi-granular texture consisting of quartz, plagioclase and K-feldspar. These granites are well exposed east of Lindero (Figure 7.2), but in the deposit area occur only as clasts in S1, FPD and rarely CPD1
- **Tertiary sedimentary rocks (S1)** are a sequence of well bedded arkose, greywacke, and subordinate mudstone and siltstone. These rocks form resistant hornfels within several hundred meters of the Lindero complex, in which the characteristic regional red color turns green, tan, or brown
- **Granodiorite quartz porphyry (GQP)** contains 5 to 10 % conspicuous quartz eyes, with plagioclase and lesser K-feldspar phenocrysts, in a microcrystalline,

quartz-rich groundmass. It is presently known only in a single dike cutting S1 in hole CON-02. It is inferred to be relatively early because it has not been observed cutting any phases of the Lindero complex

- **Fine diorite porphyry (FPD)** forms most of the peripheral portions of the Lindero complex. Relatively sparse phenocrysts, totaling 25 to 40 % of the rock, are mostly plagioclase that is rounded to irregular rather than tabular in shape. Lesser stubby hornblende prisms are also present. The microcrystalline to fine-grained groundmass consists mostly of plagioclase, with minor quartz, K-feldspar, and biotite. FPD probably consists of multiple sub-phases, including a marginal (early?) phase with variable textures and common intrusive breccias, and a later sub-phase that contains minor CPD1 clasts
- **Crowded diorite porphyry 1 (CPD1)** contains approximately 50 % total phenocrysts, mostly of tabular plagioclase, with subordinate slender hornblende and ~2 % pinhead-sized quartz eyes. The aphanitic to microcrystalline groundmass contains 5 to 15 % quartz, which is distinctly coarser than other groundmass material and can be recognized with a hand lens
- **Bimodal feldspar diorite porphyry (PBFD)** is a crowded porphyry similar to CPD1. It is distinguished by a distinct coarse (2–5 mm) population of plagioclase phenocrysts caused by a tendency of the plagioclase to cluster in glomerocrystic aggregates. The glomerocrystic texture is discernible with a hand lens
- **Granitic dykes (GDK)** contain 30 to 45 % phenocrysts of tabular plagioclase, slender hornblende, and minor biotite in aphanitic to microcrystalline groundmasses. These dikes are probably equivalent to CPD1 and/or CPD2
- **Crowded diorite porphyry 2 (CPD2)** forms much of the south-central part of the complex. It averages ~55 % phenocrysts of dominantly tabular plagioclase; slender, commonly lineated hornblende; with minor biotite and quartz. The groundmass is aphanitic (<0.03 mm) and contains primary K-feldspar. CPD2 is very weakly veined and altered; biotitic alteration of hornblende is normally absent
- **Mingled diorite porphyry (DDP)** is a hybrid, variably-textured rock, containing enclaves compositionally similar to the rock matrix but which vary in texture. Plagioclase, hornblende, and minor biotite and quartz phenocrysts total 35 to 45 % of the rock. CPD2 clasts are characteristic, and biotitic alteration is normally absent
- **Post-mineral intrusive (PMI)** forms most of the north-central part of the complex. Plagioclase, hornblende and locally biotite phenocrysts totaling 35 to 45 % of the rock are set in an aphanitic (<0.005 mm) groundmass. The unit is very weakly veined and altered, with biotitic alteration normally absent. The contact zones are characterized by common CPD2 clasts and conspicuously broken plagioclase phenocrysts, which are probably due to explosive venting of this unit. A legacy rock unit, ADK (andesitic dikes), has been reinterpreted as the chilled margin facies of PMI

Magmatic–hydrothermal breccias (BxH) form relatively narrow bodies, typically less than 10 m in drilled thickness. They consist of angular to sub-rounded clasts of one or more rock types in matrices of hydrothermal material and rarely rock flour. Two styles of breccias are distinguished based on nature of the matrix material. In the quartz–sulfide type (BxH-qs), the matrix consists of quartz, gypsum and/or anhydrite, and sulfides. These are commonly strongly mineralized with chalcopyrite and gold. They cut S1, FPD, and CPD1, but none of the younger units. Breccias of the magnetite-type (BxH-mt), in contrast, have matrices of magnetite ± chlorite. The breccias are of variable ages, being most common cutting FPD and CPD1, but also cutting PBFD, CPD2, DDP and PMI. Those breccias cutting FPD and CPD1 are commonly well mineralized, whereas the latter breccias cutting the PBFD, CPD2, DDP and PMI group are typically barren.

**Figure 7.4 Westeast cross section through the Lindero Deposit (lithology codes detailed in text)**

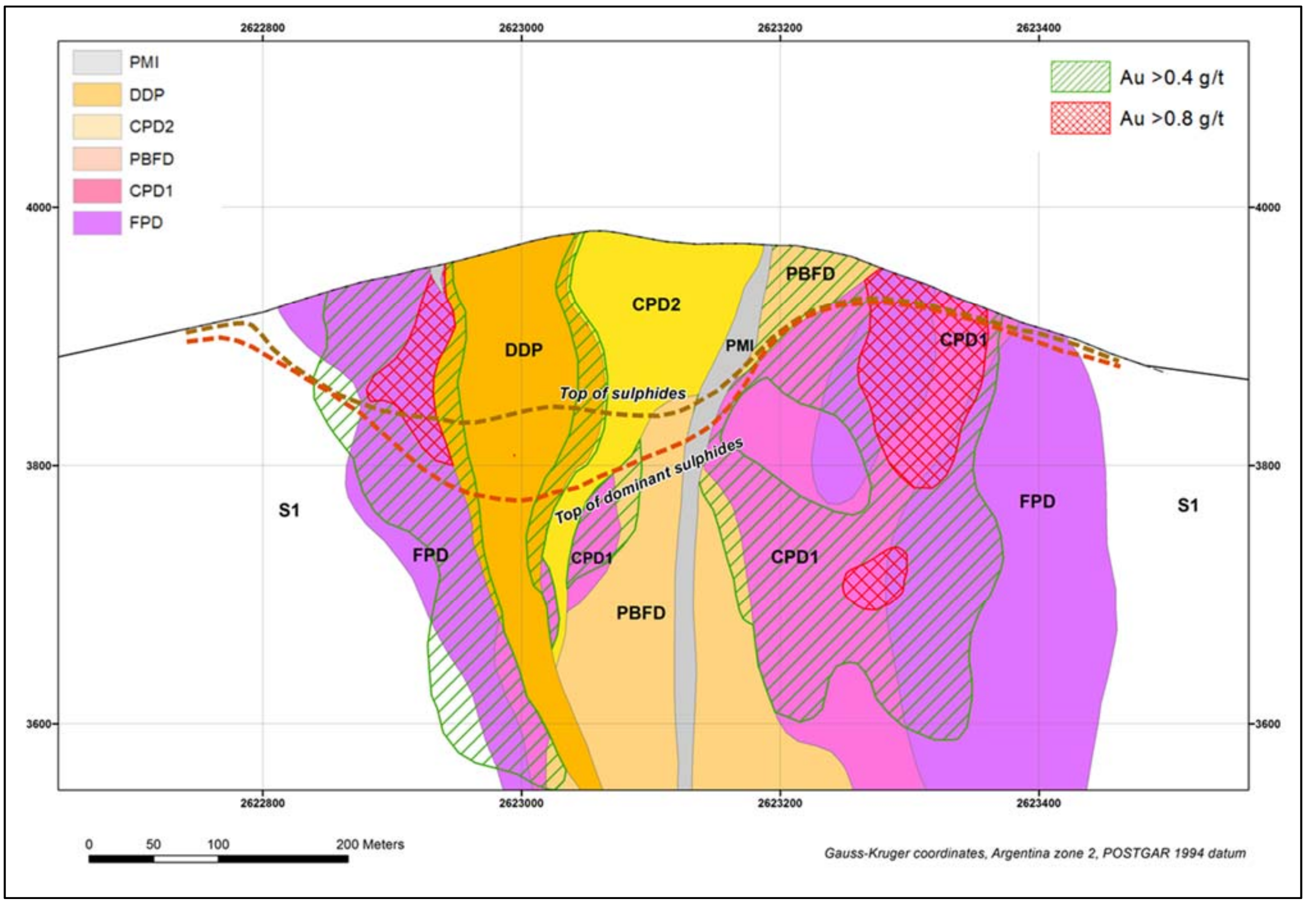



Figure prepared by Mansfield, 2017

The complex consists of six major porphyry phases that form broadly concentric bodies, and several phases present as volumetrically minor dikes. In addition, two styles of magmatic–hydrothermal breccias are recognized. Previous reports (e.g., KCA, 2016) described several types of igneous–matrix breccias, but these have been reclassified based on the rock type forming the breccia matrix.

As the rocks of the complex are mineralogically similar, they are classified based primarily on textural variations (especially phenocryst abundance), age relationships, and their relative intensity of alteration and mineralization. The sequence of igneous phases is interpreted from crosscutting relationships at contacts, which are tracked using a matrix.

Minor reversed relationships between FPD and CPD1 are observed, and suggest that these units overlapped in time.

Historically, Mansfield and Goldrock applied the field term "diorite" to almost all the rocks. Recent petrographic work, however, indicates that two compositional groups of rocks are present (Riedell, 2016a). The early porphyry phases that ring the complex are mostly quartz diorites with 5 to 15 % groundmass quartz. Younger phases in the core of the complex contain primary K-feldspar in their groundmasses, and are monzodiorites to quartz monzodiorites. Younger phases (especially the post-mineral intrusive or PMI unit) show increasing evidence of venting, such as fracturing of phenocrysts and included pyroclastic surge(?) horizons. The Goldrock rock names and codes, although not strictly correct, have been retained in this Report until more compositional data are available.

Hornblende from the relatively unaltered central intrusive rocks of the deposit yielded an $^{40}Ar/^{39}Ar$ age of 16.75±0.35 Ma, whereas biotite from altered rocks yielded an age of 15.21±0.11 Ma (Dow, 2002).

### 7.3.2 Alteration

Alteration assemblages at Lindero display a broadly concentric pattern centered on the core of the Lindero complex. The central core, represented by PMI and CPD2 porphyries, is generally unaltered with local weak carbonate–chlorite–epidote alteration. Surrounding the weakly altered core is a zone of moderate to strong potassic alteration hosted in FPD, CPD1 and PBFD porphyries. Structurally-controlled hydrolytic (dominantly sericite) altered zones overprint potassic alteration, especially at shallow levels. The surrounding sedimentary sequence is affected by early contact metamorphism, by peripheral propylitization and/or weak epidote alteration, and by later argillic alteration. For descriptive purposes, alteration styles are divided into those considered early and higher temperature (potassic, propylitic, and hornfels), versus those forming later and at probably lower temperatures (chloritic, sericitic, argillic).

**Early-stage alteration**

Moderate to strong potassic (biotite–K-feldspar–magnetite) alteration is well developed throughout the better-mineralized parts of the Lindero system. It is defined primarily by partial to complete replacement of magmatic hornblende by aggregates of brown "shreddy" biotite. Chlorite commonly overprints hydrothermal biotite; the presence of earlier hydrothermal biotite is indicated because the chlorite inherits the shreddy habit. Biotite is less strongly replaced than hornblende, and plagioclase is generally unaffected. White hydrothermal K-feldspar is widespread within the broader zone of biotite-dominated potassic alteration. It occurs as rims on plagioclase phenocrysts, as halos around early veins and veinlets, and as patchy replacements of groundmass plagioclase. Almost all of the potassic-altered rocks are cut by quartz–sulfide and magnetite ± sulfide veins and veinlets, as described below. Intensity of veins is typically less than 10 vol %, but zones of very strong quartz stockwork to pervasive silicic alteration occur locally.

Contact metamorphism dominates early alteration in S1 wall rocks within several hundred meters of the Lindero complex. The resulting styles of hornfelses are dependent on primary lithology. Sandstones and siltstones form pale green quartz–biotite(?) ± sericite hornfels. Mudstones characteristically form tan to chocolate-brown quartz–biotite ± K-feldspar hornfels. With increasing intensity of thermal effects, grain size of biotite increases, and K-feldspar becomes more abundant. Peripheral to hornfels zones, the red-bed sedimentary rocks are affected by propylitic alteration, ranging from weak epidote veining to strong epidote–calcite replacement. This alteration is not well understood because it occurs outside of the drilled area.

**Late-stage alteration**

Intermediate- to late-stage alteration includes the following:

- Chlorite replacement of primary mafic minerals and hydrothermal biotite, which is common throughout the potassic zone. There is some suggestion that the intensity of chloritic overprint increases with depth
- Sericitic alteration forms restricted zones overprinting potassic alteration, typically along late veins and structures
- Relatively strong clay–sericite alteration affected the S1 sedimentary rocks outside the zone of strongest contact metamorphism.

It is noteworthy that sericitic alteration is rather restricted, and advanced argillic alteration completely absent, in the preserved and explored parts of the Lindero system.

### 7.3.3 Structural setting

The Lindero Deposit lies within the northwest–southeast-trending (120°) Archibarca transverse structural zone, which forms the dominant set of structures regionally and has been interpreted as controlling large-scale igneous volcanic activity and regional fluid outflow. The combination and intersection of the structures at Lindero are mainly responsible for the emplacement of the porphyries. Three principal sets of structures are recognized from surface mapping and drilling; in the order of importance these fault sets strike 110–130°, 55–75°, and 20–35°.

The most important northwest–southeast fault set comprises transtensional faults that have produced uplift of the central block relative to blocks to the north and south. The second set of structures, also transtensional, underlie the valley between the Lindero and the Arizaro intrusive complexes and controlled emplacement of andesitic and dacitic dikes. The third set of structures apparently controlled the emplacement of the porphyries, and may have provided conduits for mineralizing fluids.

Structures observed in drill core are normally unhealed and consist of fault breccia, zones of crushed rock, and minor gouge. True thicknesses of discrete faults range from centimeters to several meters; crushed zones range up to tens of meters in thickness. Attitudes relative to core axes suggest most faults dip within 30 degrees of vertical. Slickenlines are commonly observed on gypsum veins within fault zones, and most suggest that latest movement ranged from horizontal to oblique.

Very few structures can be reliably interpreted between drill holes in cross section. This partially reflects uncertainty as to the true orientation of structures given the lack of oriented drill core. In addition, cross-section interpretation suggests that later intrusions may have occupied and obliterated earlier structures especially in the deeper parts of the system, as in the subparallel dike-like roots of CPD2 and DDP (refer to Figure 7.4).

### 7.3.4 Mineralization

Gold mineralization in the Lindero Deposit is hosted mostly in four different bodies that form an annular shape at the surface (Figure 7.5). The annular shape is related to the distribution of the earlier, mineralized rock units comprising FPD, CPD1, and PBFD lithologies with the CPD2, DDP, and PMI porphyries in the center of the complex forming the barren to low-grade core of the system.

The east/northeast body is the largest mineralized zone, extending north-south for 450 m. It is at least 190 m wide and extends for at least 300 m vertically, dipping steeply towards the center of the complex. This body has the highest gold grades, averaging approximately 1.0 g/t Au. The Southwest body forms an elliptical shape that is 270 m long, up to 100 m wide and extends for at least 400 m vertically. Grades average approximately 0.8 g/t Au.

The northwest and west bodies are narrower, but thicken with depth, reaching a maximum width of 100 m. Both lie mostly beneath the PMI intrusion. Several other narrow mineralized bodies occur at depth, where bodies of well mineralized CPD1 and PBFD became rafted within the central, late- to post-mineral rock units.

**Figure 7.5 Plan view of mineralized bodies of the Lindero Deposit**

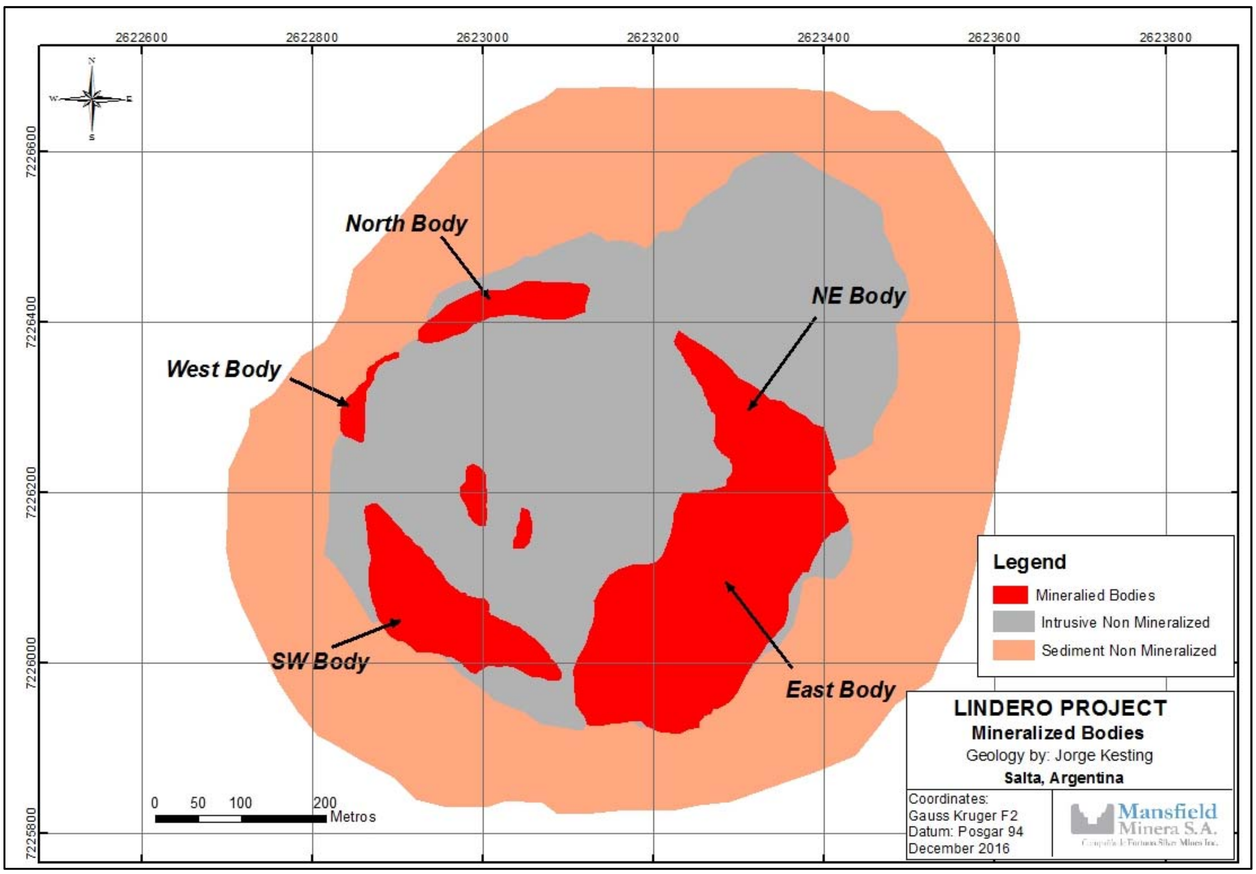



Figure prepared by Mansfield, 2016

### 7.3.5 Hypogene mineralogy

Gold–copper mineralization at Lindero is characterized by simple primary mineralogy, mostly subequal quantities of chalcopyrite and pyrite. Bornite and molybdenite also occur but are much less common. Traces of covellite, galena and sphalerite are observed locally. Grades average 0.70 g/t Au and 0.11 % Cu. Free gold occurs in grains typically 20–30 µm to a maximum of 70 µm, associated with chalcopyrite and/or magnetite grains with rare interstitial quartz. Gold–copper mineralization is controlled primarily by quartz–sulfide and magnetite–sulfide veins as described in the next sub-section. Lesser mineralization occurs as disseminated grains and clusters of chalcopyrite ± magnetite at mafic mineral sites. Gold and copper grades decrease outwards due to an increase in the ratio of pyrite relative to chalcopyrite.

### 7.3.6 Vein types

Most veins are grouped according to the classification scheme of Gustafson and Hunt (1975) and Seedorff et al. (2005). The following sequence of vein types are recognized, from oldest to youngest, based on crosscutting relations. The majority of gold–copper mineralization is associated with the A-vein and magnetite–sulfide types, which are strongly developed in the potassically altered rocks that host the ore zones.

- Quartz–K-feldspar vein-dikes are thin, wispy to planar veinlets of aplitic quartz and K-feldspar without sulfides. They are uncommon, and to date have been noted only in CPD1
- Barren magnetite films are hairline (<1-2 mm thick) veinlets of magnetite without sulfides that commonly form fine sheeted sets
- EB veins are thin veinlets of green or brown biotite and rarely magnetite, that are typically quartz- and sulfide-poor. They have halos of albite or white K-feldspar
- A-veins are early veinlets composed mostly of granular or sugary quartz (50 to 95 %) and disseminated sulfides. They may also contain magnetite, K-feldspar, and/or anhydrite. They commonly show halos of white K-feldspar, but may lack alteration halos altogether. A-veins are commonly thin, 'wispy' or sigmoidal (S-shaped), discontinuous, and of short strike-length. Later A-veins become thicker and more planar. A-veins may appear banded (like many B-veins) where they are re-opened by central sulfide films or late gypsum veins. Their orientation is generally steep
- Magnetite–sulfide veins differ from the barren magnetite films in being distinctly younger and containing significant quantities of copper sulfides. They range from thin hairlines to planar veins locally >10 mm thick. Some contain accessory clinopyroxene; these commonly contain bornite. Magnetite–-sulfide veins both predate and postdate A-veins, suggesting the two vein types overlapped in time
- B-veins are planar veins comprising mostly quartz growing perpendicular to the vein walls (approaching cockscomb or "dog's-tooth" texture). They are generally banded due to centerlines or borders of sulfides, oxides, carbonates, and/or other minerals. They normally lack alteration envelopes. Relative to other porphyry systems, B-veins are very rare at Lindero
- D-veins are planar sulfide-rich (commonly pyrite) veins with minor quartz, and with characteristic sericitic halos. In other systems they are relatively continuous, typically steep, and form radial and/or concentric patterns. They are rare in the

explored rock at Lindero, but may have been more common at higher levels that are now eroded

- Pyrite veins are thin pyrite-only veinlets, similar to D-veins but without the sericitic halo
- Anhydrite/gypsum–sulfide veins are relatively late veinlets and veins dominated by gypsum and/or anhydrite with sulfides. They lack quartz and show no alteration halos
- Gypsum veins are irregular to planar veinlets to veins of gypsum without anhydrite or quartz. They display a fibrous structure approximately perpendicular to vein walls. They normally lack sulfides, but commonly re-open A veins and appear rimmed by sulfides inherited from the older vein style. They are irregularly oriented, forming most of the low-angle veins present at Lindero. They are the latest vein type, related to recent surficial events as discussed below

### 7.3.7 Relationship between intrusive phases and mineralization

The timing and genetic relationships between intrusive phases and alteration-mineralization have been inferred from two main types of evidence. Most important are paragenetic relations among alteration-mineralization and intrusive events, especially crosscutting relationships between veins and intrusive contacts, which are carefully tracked using a matrix. These are supplemented by spatial relationships in plan and section, especially the correlation of mineralized zones with specific intrusive contacts.

Several lines of evidence indicate that the majority of gold-copper at Lindero was introduced with the CPD1 porphyry:

- Gold–copper mineralized zones commonly straddle the contact of CPD1 with FPD, especially on the east and north sides of the complex
- When the original extent of CPD1 is restored by removal of younger intrusive phases in cross section (Section 7.3.9) the distribution of present-day gold zones is nearly symmetrical around the inferred original extent of this unit. (The east and southeast sides of the deposit are better mineralized largely due to the absence of late- to post-mineral units in this area)
- Logging across a number of CPD1–FPD contacts shows that the intensity of gold–copper bearing veins is similar in both units, with little or no change in gold grades at the contact (although a minor increase from CPD1 to FPD is observed in some holes). This suggests that the majority of mineralization in both units developed concurrently, following intrusion of CPD1
- Disseminated chalcopyrite is most commonly observed in CPD1
- No zones of significant mineralization have been found along contacts of FPD with its S1 country rocks. This suggests that FPD acted mainly as a passive host rock, and is well mineralized due to its proximity to the syn-mineral intrusion CPD1

Nonetheless, evidence suggests that FPD introduced a minor amount of mineralization. Some CPD1 contacts truncate mineralized A veins, which requires a pre-CPD1 source intrusion, likely a sub-phase of FPD. Secondly, many mineralized veins in FPD are thicker and richer than those in CPD1. It is not certain if this reflects different physical properties of the two rocks (i.e., CPD1 at sub-magmatic temperatures intruding and fracturing brittle FPD), or if a minor period of mineralization affected FPD but not CPD1.

Mineralized magnetite–sulfide and A veins cut most younger intrusive phases, although vein abundance is generally lower than in CPD1 and FPD. This requires at least one mineralizing intrusion younger than CPD1. Some mineralized zones in section appear to follow the contacts of PBFD and DDP with older rocks, while the centers of both units tend to carry low grades. Both are interpreted to be sources of gold–copper mineralization, but are less important than CPD1. There is no evidence for any contribution of gold or copper from CPD2 or PMI.

In summary, the principal mineral-related intrusion at Lindero is the CPD1 porphyry, with lesser contributions of gold and copper from FPD, PBFD, and DDP. FPD is the most important ore host because it forms the wall rocks to the syn-mineral CPD1. The importance of FPD as a host may also reflect a physical receptivity to gold mineralization, the influence of its own mineralized fluids, and/or peripheral effects of gold and copper introduced by the PBFD or DDP intrusive events.

### 7.3.8 Oxidation and supergene effects

Near-surface oxidation zonation at Lindero has been documented through systematic logging of sulfide and sulfate surfaces according to protocols in widespread use in porphyry copper deposits in northern Chile and elsewhere. The following surfaces are logged in each hole, and are used to define the oxide/sulfide and sulfate zones shown in Table 7.1.

- Top of sulfides (TS): First downhole appearance of sulfides
- Top of dominant sulfides (TDS): Primary (and/or supergene) sulfides become dominant; equivalent to the base of oxidation (except for deeper oxidation along fracture zones, crushed zones, and faults)
- Top of gypsum (TGyp): First appearance of consistent gypsum veining
- Top of anhydrite (TAnh): First appearance of anhydrite in veins

As shown in Table 7.1, the TS and TDS surfaces delineate the oxide, mixed, and sulfide zones. In the oxide and mixed zones, primary chalcopyrite ± bornite is mostly oxidized *in situ* to neotocite (a brownish-black copper-bearing manganese-iron oxide mineraloid; Anderson, 1982) and chrysocolla. Rare secondary chalcocite coats fractures near the TDS surface in some holes. Pyrite is oxidized to goethite and minor jarosite. The depth of oxidation varies considerably across the system, imparting a concave-up shape to the TS and TDS surfaces (Figure 7.4). Oxidation is thickest in the younger rock units in the center of the complex. The TS surface ranges from <10 m to ~150 m deep. Partial oxidation as defined by the TDS surface ranges from <20 m thick on the margins of the deposit to over 200 m thick in some central drill holes.

**Table 7.1 Oxide/sulfide and sulfate zoning**

| OXIDE / SULFIDE ZONES | SURFACES | SULFATE ZONES | ROCK COMPETENCY |
|---|---|---|---|
| OXIDE | | SULFATE-FREE ZONE | Fair to poor |
| | ------- Top of Gypsum ------ | GYPSUM ZONE | Good to excellent |
| | ----- Top of Sulfides ------ | | |
| MIXED | Top of Dominant Sulfides | | |
| SULFIDE | ------ Top of Anhydrite ----- | | |
| | | GYPSUM-ANHYDRITE ZONE | Fair to good |

Sulfates are likewise zoned relative to the present surface. TGyp commonly occurs near the TS surface, but may also be above or below it. TAnh typically lies well below TDS, and is sub-horizontal rather than bowl-shaped. Based on patterns in other deposits, it is hypothesized that the primary sulfate phase was all anhydrite, occurring as an accessory in early A-veins as well as in late anhydrite-sulfide veins. Circulation of ground water up to the present time resulted in hydration of anhydrite to gypsum, which graded upwards from partial (gypsum-anhydrite zone; typically, 0.2 to 0.5 % anhydrite and 1 to 2 % gypsum) to complete (gypsum zone; averaging 2 to 5 % gypsum). Above the gypsum zone, all sulfate is dissolved, producing vuggy cavities in A-veins that were formerly filled by anhydrite. An anhydrite-only zone without gypsum is hypothesized at depth, but has not been penetrated by drilling to date.

As noted in Table 7.1, sulfate zoning has important implications regarding rock competency. The hydration reaction anhydrite to gypsum involves a volume increase, and so gypsum re-opened existing fractures such as A-veins, and created new gypsum-only veins. Within the gypsum zone these fractures were healed, but in the surficial sulfate-free zone they became open fractures. Relatively common near-surface zones of strong fracturing resulted from this process. Rock permeability is hypothesized to decrease significantly from the sulfate-free to the gypsum zone, and to increase slightly into anhydrite-bearing rock at greater depth.

### 7.3.9 Evolution of the system

Figure 7.6 (panels A through H) illustrates the interpreted evolution of the Lindero complex and deposit along present-day west–east section 6120N (Figure 7.4). These interpretations have been compiled by removing rock units sequentially from youngest to oldest, and inferring earlier patterns of mineralization. Geology has been projected above the present surface based on relationships observed within the preserved rocks (such as textural evidence for venting); assumed symmetry of mineralized zones; and patterns observed in the upper levels of other, better preserved porphyry systems.

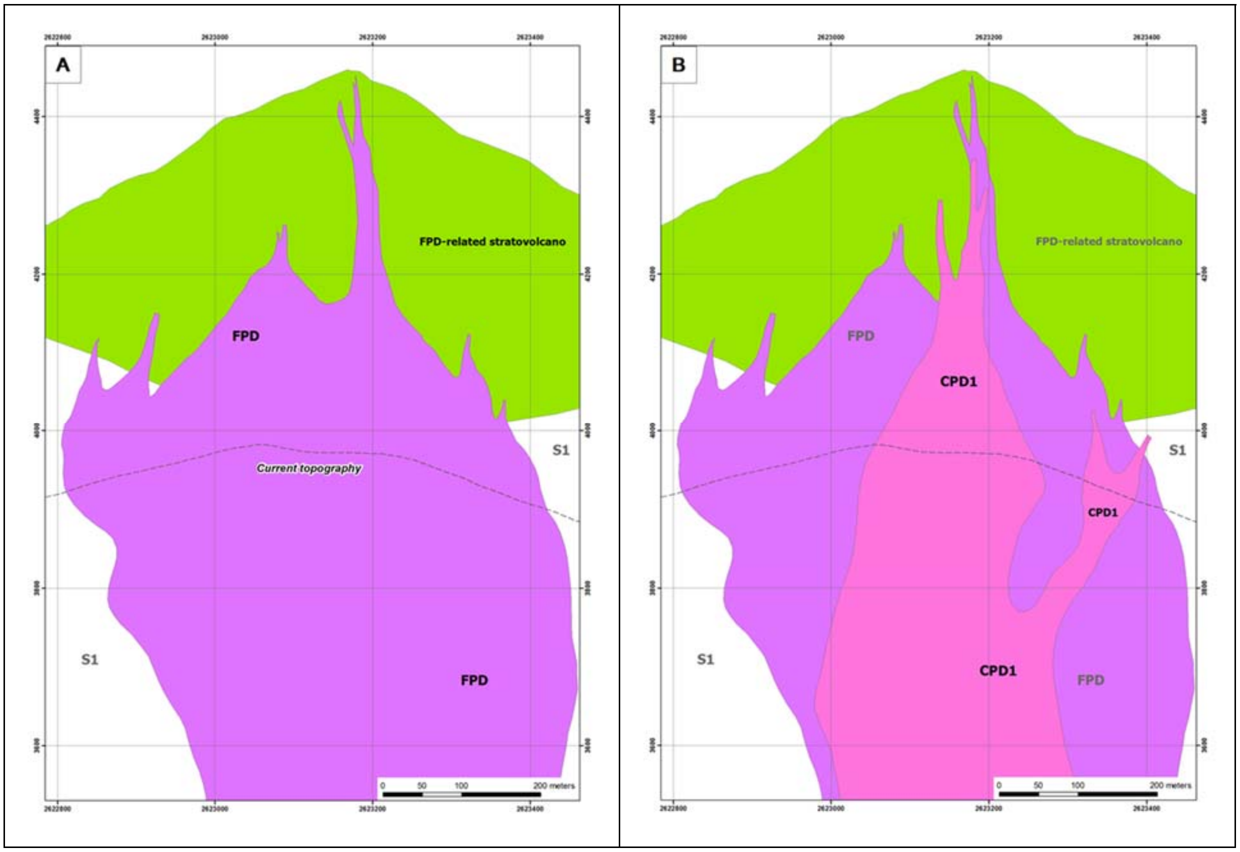
- **Panel A**: Several batches of FPD magma intruded into S1 sedimentary rocks approximately 17 million years ago, coalescing to form a stock 600–750 m across.

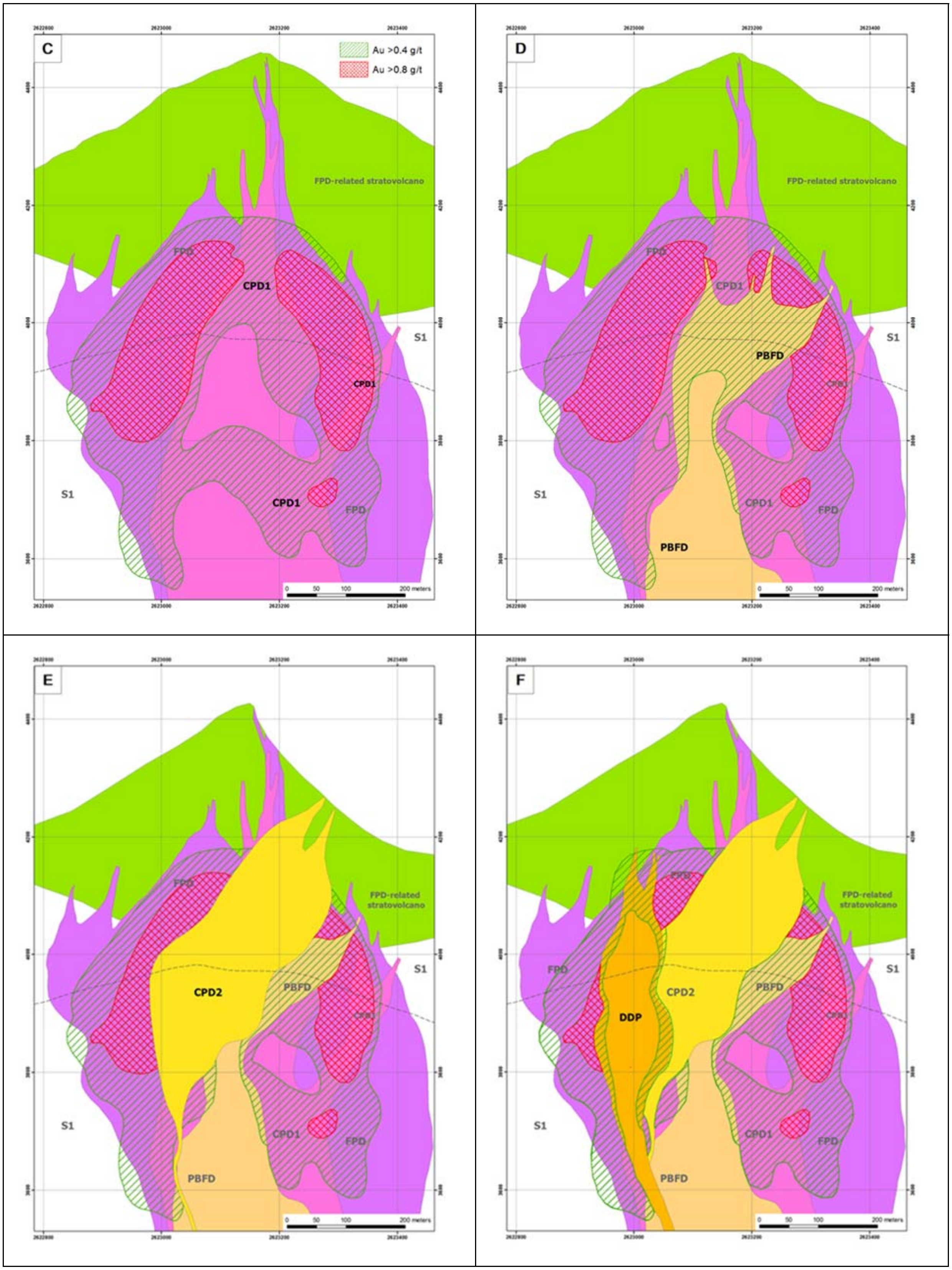
It is hypothesized that one or more of these magmas erupted to form an overlying stratovolcano or dome, although there is no textural evidence for eruption, and any related volcanic rocks have been eroded. Most contact metamorphism of the red-bed sequence occurred at this time. Minor gold–copper mineralization formed in and possibly around FPD, but this is not shown as it cannot be discriminated from CPD1-related mineralization

- **Panel B**: A stock, dikes and small masses of CPD1 porphyry intruded into the FPD stock. Reconstruction of the original extent of CPD1 from blocks rafted within younger units suggest CPD1 was emplaced nearly concentrically within FPD. Its present-day concentration on the east and north sides of the complex is preserved, reflecting the absence of younger units in those areas
- **Panel C**: The major stage of gold–copper mineralization immediately followed CPD1 intrusion. Potentially economic grade (>0.4 g/t) gold formed over a volume 500–550 m across and at least 500–600 m vertically. Gold–copper mineralization affected much of the CPD1 stock and extended up to 200 m into FPD and S1 wall rocks. Higher grade (>0.8 g/t) gold zones are concentrated along CPD1 contacts but are best developed in FPD. Gold–copper mineralization was accompanied by widespread development of potassic alteration with common to abundant A-type and magnetite–sulfide veins. The abundance of thin, wispy A veinlets and truly disseminated chalcopyrite in CPD1 suggest ductile and not brittle behavior, as compared with FPD-hosted veins that are coarser and more planar. The overall shape of potentially-economic grade material appears to have formed two inverted cup-shaped mineralized shells flanking CPD1, separated by a lower-grade gap
- **Panel D**: A narrow stock of PBFD intruded approximately along the axis of the earlier FPD and CPD1 intrusions. Weak to moderate gold–copper mineralization formed mostly along and immediately fringing PBFD contacts, leaving a low-grade core in the center of the PBFD body
- **Panel E**: CPD2 was emplaced slightly to the west of the axis of the FPD, CPD1 and PBFD intrusions. While this magma appears compositionally and texturally "fertile", it produced essentially no gold or copper mineralization and is poorly veined and altered. A likely explanation is that the CPD2 magma vented, with catastrophic loss of metal-bearing volatiles into the atmosphere. Substantial erosion of the volcanic edifice may have resulted from sector collapse accompanying the eruption(s)
- **Panel F**: A small stock of DDP intruded in the west–central part of the complex. Weak to moderate gold–copper mineralization formed along its contacts
- **Panel G**: After a period of significant erosion of the overlying volcano, PMI intruded along dikes in the north, west and east parts of the complex. This magma clearly erupted as evidenced by intensely-fractured plagioclase phenocrysts and local pyroclastic horizons. The dikes flared upward and coalesced into a series of flow-domes. PMI produced no gold or copper and remained very weakly altered. Based on the range of radiometric dates, this latest igneous activity was probably complete by 15 Ma
- **Panel H**: The area was gradually eroded to current topography, exposing ore-grade gold in many areas, but also resulting in loss of considerable mineralized

material. Oxidation of sulfides formed a goethite-neotocite-chrysocolla cap, with the oxide and mixed zones thickening from <20 m in the fringes of the system to >200 m in the center. Essentially no supergene-enriched mineralization developed, reflecting the low pyrite content of the ores that precluded significant leaching of copper (Anderson, 1982). Anhydrite in veins was partially to completely hydrated to gypsum, and dissolved completely from near-surface rocks

**Figure 7.6 Evolution of the Lindero Deposit**




C
Au >0.4 g/t
Au >0.8 g/t
FPD-related stratovolcano
FPD
CPD1
CPD1
S1
S1
CPD1
FPD
0 50 100 200 meters
D
FPD-related stratovolcano
FPD
CPD1
S1
PBFD
S1
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0 50 100 200 meters
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FPD
FPD-related stratovolcano
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CPD1
FPD
PBFD
0 50 100 200 meters
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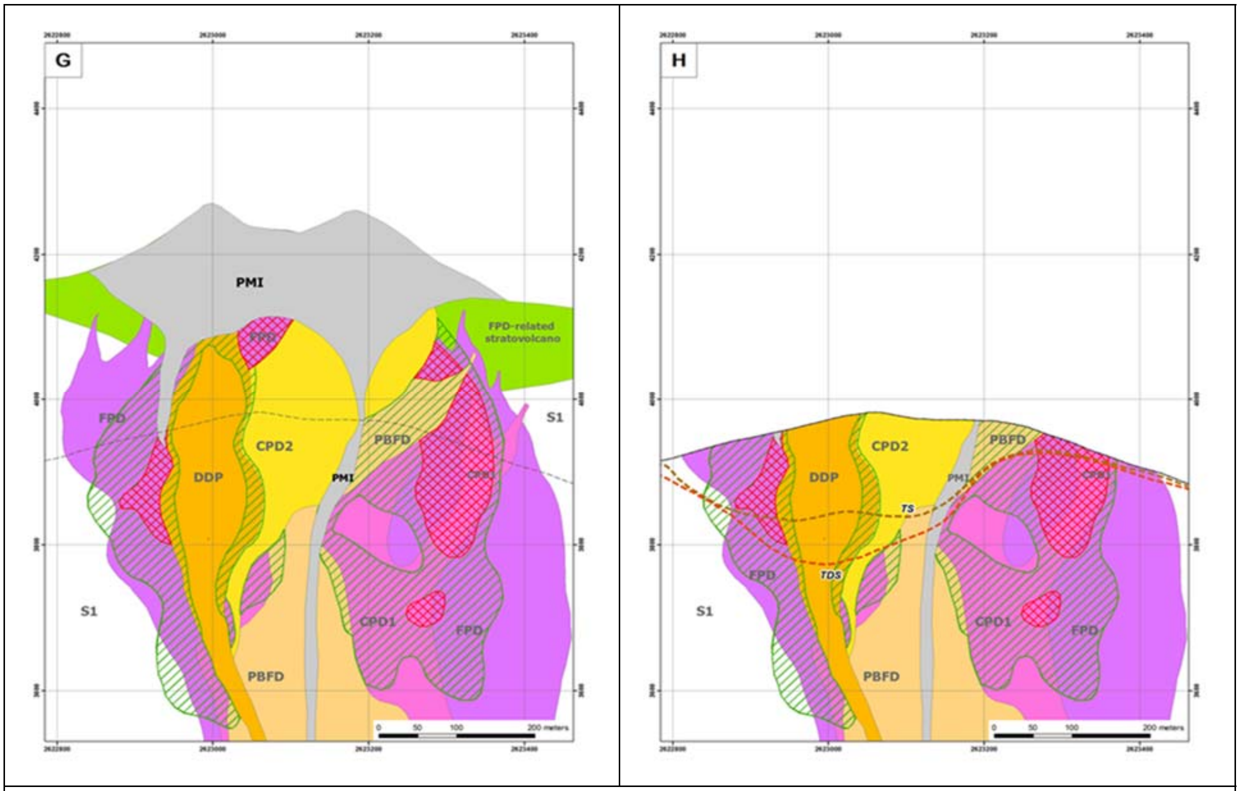



Figures prepared by Riedell and Mansfield, 2017

## 7.4 Arizaro Deposit

### 7.4.1 Local geology

The Arizaro Deposit is dominated by one main moderately to strongly mineralized intrusive unit that crops out in the central part of the prospect area (Figure 7.7). It consists of fine hornblende porphyritic diorite intruded by several stocks, dikes, igneous-cemented breccias and hydrothermal breccias. Smaller stocks are exposed in a few areas. Dikes of andesitic and tonalite composition are generally distributed radially to the main intrusive.

Relatively unaltered intrusive rocks from the Arizaro Deposit have yielded an $^{40}Ar/^{39}Ar$ age of 16.35+0.35 Ma (Dow, 2002). Different types of igneous-cemented breccias have been recognized:

- Magnetite-rich, igneous-cemented breccia with strong gold-copper mineralization
- Biotite-rich, igneous-cemented breccia with gold-copper mineralization
- Polymictic, igneous-cemented breccias

At the margins of the intrusion to the north and east crop out two sequences of volcaniclastic mass-flows, crystalline lapilli tuffs, fine crystalline tuffs and fine-grained lava flows. The volcaniclastic rocks in contact with the intrusive body have undergone strong contact metamorphism. The volcaniclastic rocks are believed to represent the remnants of a volcanic edifice.

**Figure 7.7 Arizaro geological map**

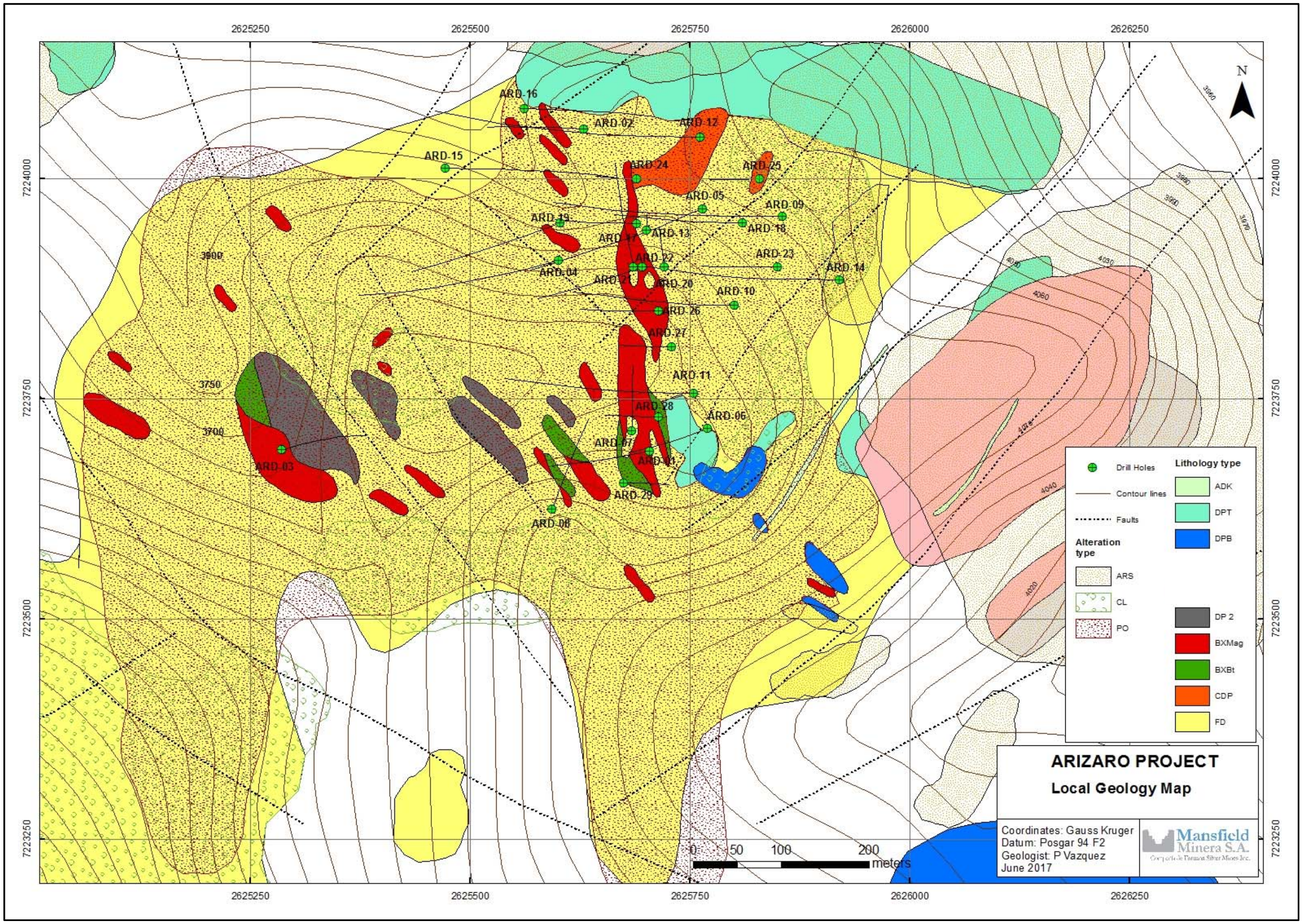



Figure prepared by Mansfield, 2017

### 7.4.2 Lithology

Several intrusive lithologies, extrusive rocks, and breccias have been recognized based on textural and mineralogical characteristics from field observations and descriptions of drill core (Fig. 7.7). Listed in order from oldest to youngest, the units include:

- **Fine-grained Diorite (FD):** dominated by equigranular, fine-grained (2mm) plagioclase and quartz with up to 20 % fine-grained hornblende. Interpreted to be the earliest intrusive rock and pre-mineral
- **Crowded Dioritic Porphyry (CDP):** Contains 30 to 40 % tabular plagioclase, 10 % euhedral hornblende in a fine-grained groundmass of plagioclase and quartz. Considered to be the earliest and strongest mineralizing intrusive. Outcrops in only a small 40 x 100 m area in the northeast limit of the porphyry system but is more ubiquitous in drill core
- **Biotite-matrix Breccia (BxBt):** a magmatic-hydrothermal breccia consisting of a matrix and veinlets of fine-grained, secondary biotite with some magnetite. Considered to be intra-mineral
- **Magnetite Breccia (BxMag):** a magmatic-hydrothermal breccia consisting of a dominant magnetite and lesser quartz, K-feldspar and secondary biotite matrix. As the biotite breccia above, considered intra-mineral
- **Diorite Porphyry 1 (DP1):** seen only in core, contains 20 % sub-rounded plagioclase set in a very-fine-grained groundmass of quartz and plagioclase and <15 % hornblende
- **Diorite Porphyry 2 (DP2):** porphyry with 25 to 30 % tabular, fine-grained plagioclase and 5 % elongate hornblende in a very-fine-grained groundmass of plagioclase and quartz. First of the intrusive units considered as late-mineral
- **Pebble Dike (PD):** seen only in core and limited to only a few drill holes. Made up of 70 % rock flour and fractured, rounded to subrounded clasts of Ordovician granite (basement rock) and rhyolite of uncertain origin
- **Bimodal Porphyritic Diorite (DPB):** contains 20 % plagioclase phenocrysts both subrounded and tabular, 10 % elongated hornblende in a very-fine-grained groundmass of dominant plagioclase and minor quartz
- **Late Porphyritic Diorite (DPT):** intrusive-matrix breccia with 30 % plagioclase and minor hornblende with a groundmass partially replaced by secondary K-feldspar. Locally contains subrounded clasts of basement rock
- **Andesite Dikes (ADK):** consists of 10 % plagioclase crystals and 2 to 5 % hornblende disseminated in aphanitic, green to maroon, groundmass. Considered post-mineral
- **Tonalite Dikes:** seen only in core, coarse-grained quartz and plagioclase and minor hornblende, being post-mineral

**Volcanic rocks**

The central porphyritic intrusive units are ringed by a volcanic sequence which is a remnant of a volcanic edifice. Exposures to the north, northeast and east of the deposit area show textural, lithological and structural differences between different volcanic units. The volcanic sequence consists of volcaniclastic deposits, lapilli crystal-lithic tuffs, fine-grained crystal-lithic tuffs and fine-grained ash. These volcanic rocks represent mass flow deposits – lahars, ash-flows and ash-fall deposits related to a volcanic center. The Central Arizaro porphyry intrudes the volcanic center, which has a metamorphic contact with the volcaniclastic rocks. Numerous dikes radiate from the volcanic center cutting the volcanic sequence. The bedding planes of the volcaniclastic rocks and tuffs located north of the complex dip away from the volcanic center. This unit is commonly overprinted by intermediate argillic alteration, sericite alteration and advanced argillic alteration, commonly next to the main mineralized areas.

Interpretation of the field relationships between the intrusives and volcanic rocks suggests that the volcanic rocks are coeval with the Central Arizaro porphyry and that the Arizaro intrusive units were emplaced at shallow depths.

### 7.4.3 Alteration

Several alteration assemblages are noted in the deposit area. Some assemblages have an unusual alteration mineralogy for porphyry gold–copper deposits and both their spatial and temporal relations are complicated. Field mapping and drill core observations have been used to establish preliminary alteration assemblages and their spatial relationships. The alteration map is shown in Figure 7.7.

Alteration patterns are semi-concentric and are asymmetric, with a core of moderate to strong potassic alteration including zones of K-feldspar-rich magnetite–silica alteration. An incomplete rim of chloritic alteration is developed outboard of the potassic alteration. In the southeast part of the deposit, intermediate argillic alteration has formed and overprints potassic alteration. Sericitic and very weak argillic alteration (hydrolytic alteration) has developed in the volcanic tuffs. To the south and west of the deposit, chloritic alteration passes directly to propylitic alteration. An actinolite-magnetite alteration assemblage forms in the eastern part of the prospect; an altered sample yielded an $^{40}Ar/^{39}Ar$ age of 15.66+0.53 Ma (Dow, 2002).

- PO: Strong K-feldspar magnetite-rich alteration (PO) resembles a "pseudo" breccia texture and includes monomictic clasts with a dioritic composition and magnetite–anhydrite–chalcopyrite cement. The diorite clasts are rounded, K-feldspar–biotite-altered and locally cement-supported. This alteration occurs throughout the central part of the deposit area and follows a structural lineament. It is associated with the highest gold values

- K-feldspar–biotite–magnetite alteration (PO) is characterized by strong, disseminated biotite replacing mafic minerals, with 4 % to 5 % disseminated magnetite. Biotite–magnetite–chalcopyrite veinlets are present with K-feldspar vein haloes. Gold–copper mineralization is hosted in veinlets and disseminations in the rock. The outer zone of this assemblage is characterized by a lack of K-feldspar veinlets, less intense biotite-magnetite stockworks and lesser amounts of disseminated magnetite. Generally, this style of alteration forms an oval–elliptical body in the central and western parts of the deposit. Biotite from a thoroughly biotite-altered rock produced an $^{40}Ar/^{39}Ar$ age of 15.46+0.11 Ma (Dow, 2002)

- K-feldspar-biotite-quartz veinlets (PO) are characterized by strong K-feldspar–biotite–quartz veining. K-feldspar is present as selvedges produced by the vein-forming fluids. This alteration assemblage is only developed in a small area

- Chlorite clots (CL) form an unusual alteration assemblage represented by clots of chlorite developed pervasively throughout the rock, generally occurring at the edges of the K-feldspar–biotite alteration. It is associated with medium-grade gold–copper mineralization

- Green biotite stockwork alteration (CL) is an intense green biotite alteration in a stockwork of veinlets that affects the FED diorite intrusion. The alteration produces a greenish color to the rock. This alteration assemblage is considered to be a post-mineralization alteration; no mineralization is associated with this alteration assemblage
- Propylitic alteration (CL) is noted mainly to the southwest of the deposit area. This alteration assemblage is characterized by epidote, quartz, magnetite, chlorite and carbonates as veinlets and disseminations. The chlorite-dominated part of the assemblage occurs as a rim around weak potassic alteration. It consists of chlorite replacing mafic minerals (actinolite) with 1 % to 2 % disseminated magnetite. It is common to observe the chlorite replacing actinolite, which is in turn replaced by biotite near the potassic core. The epidote-dominated portion of the alteration assemblage occurs outboard of the chlorite-dominated propylitic assemblage and developed in the outer margins of the hydrothermal system. It represents the cooler, more distal portion of the alteration
- Argillic alteration (ARS) is characterized by bleaching of the volcanic tuffs. White clays and very rare, patchy silica are present. The silica likely formed by devitrification of volcanic glass within the tuffs. Destruction of the feldspars, formation of jarosite and limonite and replacement of the matrix by clays and silica are characteristics of this alteration assemblage
- Supergene alteration includes the oxidation of copper minerals, mainly chalcopyrite to copper-bearing limonite and/or chrysocolla and the replacement of magnetite by hematite (martitized). This alteration is widespread over the system and is an effect of surficial weathering and oxidation

### 7.4.4 Mineralization

Gold–copper mineralization is hosted in one body which has a semi-oval shape at the surface. In the center, there is a high-grade body with a semi-ellipsoidal form, extending north–south for 480 m that is about 50 m wide, averaging approximately 0.7 g/t Au at the surface.

The Arizaro Deposit has styles of mineralization with copper–gold grades which are strongly correlated with different alteration assemblages. Mineralization is mainly associated with potassic alteration. This occurs generally in multi-directional veins, vein stockworks and disseminations. In some areas, the vein density is high, forming vein stockworks in the intrusive rocks. These vein stockworks are limited to magnetite–biotite veinlets, quartz–magnetite–chalcopyrite veinlets, late magnetite breccias, and in late-stage mineralization events, anhydrite–sulfide veinlets.

Chalcopyrite and bornite are the main copper minerals. Gold is mainly associated with chalcopyrite, quartz, and anhydrite veinlets. Magnetite is common as massive replacements of the matrix in breccias, in veinlets and as disseminations. Dow (2002) reported the presence of elevated light rare earth elements (LREE) concentrations (La–Ce–Nd) hosted in monazite and allanite, and the presence of sub-micrometer-sized palladium intergrown with free gold in biotite-rich alteration assemblages.

Molybdenite is sporadically present and is associated with anhydrite–chalcopyrite veinlets. The presence of pyrite is limited to the distal margins of the system.

The copper oxide minerals found in the deposit include chrysocolla and brochantite. These occur as fracture-fill, fine veinlets with quartz ± sulfides and replacing feldspar crystals. The iron oxide minerals present are limonite, hematite, and very sporadic jarosite along fractures.

Coarse gold was observed and confirmed with X-ray diffraction analysis at the University of Neuquen, Argentina, laboratory, from drill hole ARD 14 (154.5 m) and was also identified macroscopically in an anhydrite–chalcopyrite–molybdenite vein.

Gold grades average 0.6 g/t Au within the high-grade central core, and 0.2 g/t Au within the margins of the deposit. Copper grades are more consistent across the deposit averaging 0.15 % Cu. The high relative gold to copper ratios suggests higher gold mobilization in the hydrothermal fluids with respect to copper, and may be interpreted as representing the higher levels of metal precipitation from gold-rich, copper-poor, hydrothermal solutions.

## 7.5 Comment on Section 7

In the opinion of the QPs, knowledge of the Lindero Deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation.

Deposits and prospects such as Arizaro are at an earlier stage of exploration, and the controls of lithology, structure, and alteration on mineralization are currently insufficiently understood to support estimation of Mineral Resources. A detailed description of the Arizaro Deposit, from which the information in this subsection is summarized, can be found in KCA (2016a).

# 8 Deposit Types

## 8.1 Mineral deposit type

Early mapping at Lindero and Arizaro led Dow (2001, 2002) to interpret the mineralization as representative of iron oxide–copper–gold deposits. However, subsequent investigations by Mansfield, Goldrock, and Fortuna led to the alternative interpretation that these are gold-rich porphyry copper deposits as described by Sillitoe (2000). More specifically, they show affinities with the porphyry gold deposit model (Rytuba and Cox, 1991; also termed dioritic porphyry gold deposits by Seedorff et al., 2005). These are exemplified by the Refugio, Cerro Casale, Marte, and Lobo gold deposits of the Miocene-age Maricunga belt, Chile, approximately 200 km south of Lindero. Vila and Sillitoe (1991) and Muntean and Einaudi (2000, 2001) described those deposits in detail. Seedorff et al. (2005) noted that porphyry gold systems are transitional into porphyry copper–(gold, molybdenum) deposits associated with tonalitic or granodioritic intrusions (e.g., Los Pelambres, Chile; Batu Hijau, Indonesia).

The general characteristics of porphyry gold deposits are summarized as follows (from Rytuba and Cox, 1991; Vila and Sillitoe, 1991, Muntean and Einaudi, 2000, 2001; and Seedorff et al., 2005) and in Figure 8.1. Specifics applicable to the Maricunga belt are included for reference.

**Figure 8.1 Porphyry gold (copper) model**

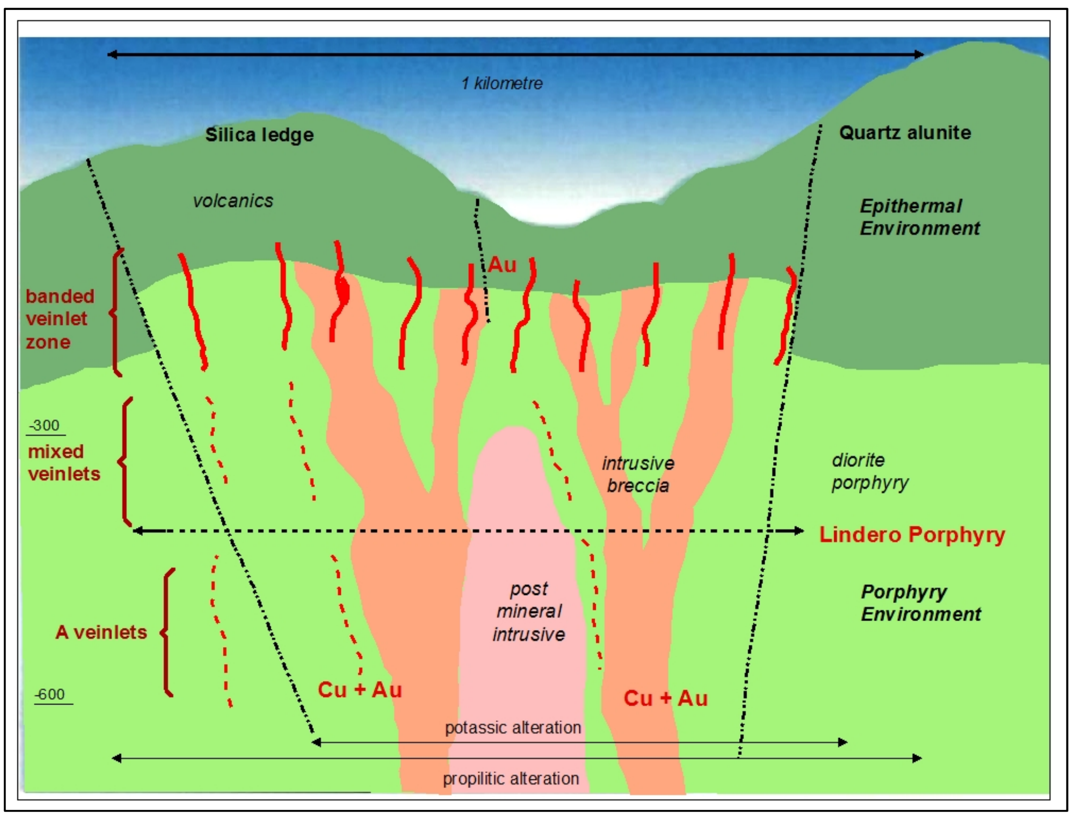



Figure from Belanger (2006) modified by Mansfield (2017)

### 8.1.1 Tectonic setting and age

Porphyry gold deposits form in volcanic arc systems developed along continental margins or island arcs. Deposits are localized by high-level multiphase porphyry stocks emplaced into the cores or flanks of andesitic to dacitic stratovolcanoes, probably associated with local extensional environments related to regional strike-slip faulting.

The vast majority of deposits are Miocene and younger.

### 8.1.2 Host rocks

Gold occurs in phenocryst-rich ("crowded") porphyries of diorite, quartz diorite or tonalite composition and in andesite to dacite flows, tuffs, and breccias of the stratovolcano edifice. Associated rocks may include dacitic domes and flows and intermediate ash flow tuffs. The mineral-related porphyries are described in detail at Refugio, Chile, where they are quartz diorite porphyries with subequal amounts of phenocrysts (plagioclase > hornblende > biotite) and microcrystalline groundmass (Muntean and Einaudi, 2000).

### 8.1.3 Ore and gangue mineralogy and texture

Native gold and electrum are finely disseminated in subparallel to stockwork quartz + sulfide ± magnetite ± anhydrite veins and in some cases in matrices of hydrothermal breccias. Magnetite is common to abundant in mineralized zones, averaging 2 to 10 vol % in the Maricunga deposits – significantly more than in gold-deficient porphyry systems. Oxidized mineralization contains goethite, jarosite, hematite, and in some cases copper oxides. Sulfide mineralization contains pyrite (as much as 8 to 10 vol %), chalcopyrite, and minor molybdenite. The deposits of the Maricunga belt exhibit distinctive gold-mineralized banded quartz veinlets, with dark inclusion- and magnetite-rich bands along their walls, occurring mostly higher in the system than A-type granular quartz–sulfide veins. Banded quartz veins are not generally observed in other types of porphyry deposits.

### 8.1.4 Wall rock alteration

Gold mineralization is associated with potassic alteration with abundant hydrothermal biotite, and/or by overprinted sericitic and sericite–chlorite–clay alteration. Potassic alteration is commonly overlain and overprinted by pyrite- and alunite-rich advanced argillic and vuggy silica alteration. Upper levels of systems may contain enargite-rich high-sulfidation gold–copper deposits.

### 8.1.5 Structural setting

The porphyry intrusions and related deposits are localized where regional fault zones intersect stratovolcanoes. Fault offset during porphyry crystallization results in sheeted and stockwork fracturing that controls mineralized veins and veinlets. Summit craters and small calderas are common in the volcanic edifices.

### 8.1.6 Typical size and dimensions

Tonnages mostly range from 40 to 250 Mt, with typical grades of 0.7-1.5 g/t Au and 0.05-0.2 % Cu. Mineralized bodies are typically steep tabular zones, 50 to 150 m wide, 500 to 600 m long, with vertical extents of 250 to 500 m. Alteration zones around the mineralization are typically 1 by 2 km in extent.

### 8.1.7 Weathering effects

Oxidation of sulfide minerals is critical in making low-grade mineralization amenable to cyanide heap leaching. Supergene copper enrichment is only incipiently developed.

### 8.1.8 Geochemical and geophysical signature

Porphyry zone deposits display deposit- to district-scale anomalies in gold, arsenic, mercury, copper, molybdenum, and lead. The overlying advanced argillic zone is also anomalous in antimony, bismuth, and tellurium.

Due to the high magnetite content, orebodies tend to exhibit strong positive magnetic anomalies.

### 8.1.9 Mineralization controls and exploration guides

The principal guide to mineralization is a sizable zone of magnetite- and biotite-rich potassic alteration and A veining developed in and surrounding a hypabyssal crowded porphyry of quartz dioritic to tonalitic composition. At higher levels, zones of advanced argillic (e.g., quartz-alunite "ledges") and vuggy silica alteration with high-sulfidation gold-copper and/or hot-spring mercury-sulfur mineralization may indicate the presence of porphyry gold mineralization at depth. Relict A-type veins persist into advanced argillic zones and provide an important vector to underlying porphyry mineralization.

### 8.1.10 Fluid inclusions

Fluid inclusions in the Maricunga belt deposits indicate that 'A veins' formed at temperatures as high as 700ºC and depths of 0.8 to 1.6 km. Higher-level banded quartz veins formed at maximum temperatures of 350ºC at depths of 0.2 to 1.0 km (Muntean and Einaudi, 2000, 2001). Porphyry gold deposits are interpreted to form at shallower paleodepths than other porphyry deposits.

## 8.2 Comparison with Maricunga belt deposits

The Lindero gold–copper system shares many attributes with the deposits of the Maricunga belt. The Miocene age, tectonic setting, mineralization-related quartz diorite porphyries, and important mineralization-stage biotitic alteration are similar to the Maricunga deposits. The resource tonnage and copper grades at Lindero are similar to many of the Maricunga deposits, whereas gold grades at Maricunga are generally higher than at Lindero, averaging slightly over 1.0 g/t Au. Lindero is a magnetite-rich system, averaging 2.4 vol %, which is at the low end of the range of Maricunga deposits (2 to 10 %).

Lindero differs from the Maricunga deposits, however, in its limited sericitic alteration and absence of advanced argillic alteration. Magnetite-dominant veins at Lindero are distinct from those in the Maricunga deposits; and the characteristic banded quartz veins of Maricunga are absent. While constraints as to depth of formation are not available at Lindero, these contrasts may simply reflect a deeper level of emplacement or erosion compared with the Maricunga belt deposits.

## 8.3 Comment on Section 8

The deposits of the Project area are considered to be examples of porphyry-style deposits, in particular gold-rich porphyries based on the following:

- High-level (epizonal) stock emplacement levels in magmatic arc
- High-level stocks and related dikes intrude their coeval and cogenetic volcanic piles. Intrusions range from fine through coarse-grained, equigranular to coarsely porphyritic
- Mineralization in or adjoining porphyritic intrusions of quartz diorite/monzonite composition
- Mineralization is spatially, temporally, and genetically associated with hydrothermal alteration of the intrusive bodies and host rocks
- Gold–copper mineralization formed during intrusion of multiple phases of similar composition intrusives rocks
- Large zones of quartz veining, stockwork mineralization, and disseminated pyrite
- Tenor of gold and copper grades, i.e., large tonnage but low grade

Understanding of the geological setting and model concept of the Lindero and Arizaro deposits is adequate to provide guidance for exploration and development.

# 9 Exploration

The Lindero Deposit was discovered in late 2000 as a result of a regional program of exploration undertaken by Goldrock (Kesting and Huidobro, 2001). Several exploration programs have been conducted at the Lindero Deposit by Rio Tinto, Goldrock, and Fortuna as follows:

- Goldrock campaign – August 2000 to October 2001
- Rio Tinto campaign – May 2002 to February 2003
- Goldrock campaign – October 2005 to January 2008
- Goldrock campaign – August 2010 to November 2010
- Fortuna campaign – September 2016 to August 2017

A summary of the work conducted during these work programs is detailed in Table 9.1.

**Table 9.1 Summary of exploration programs at Lindero**

| Year | Type of work | Company |
|---|---|---|
| 2000 | Geology and alteration mapping at 1:20,000 & 1:10,000 & 1:5,000 scale | Goldrock |
| 2000 | Soil sampling grid of 100 m x 50 m | Goldrock |
| 2001 | Trenches totaling 1,752 m | Goldrock |
| 2002 | Road sampling totaling 3,500 m | Rio Tinto |
| 2002 | Geophysics – 43 km of ground magnetics and 11 km of IP | Rio Tinto |
| 2002 | Drilling totaling 3,279 m (10 holes) | Rio Tinto |
| 2004 | Metallurgical testwork on 30 samples | Goldrock |
| 2005 | Trenches totaling 1,264 m (16 trenches) | Goldrock |
| 2005-2006 | Drilling totaling 2,609 m (11 holes) | Goldrock |
| 2006 | Geological mapping at 1:5,000 scale and 3D modelling | Goldrock |
| 2006 | Trenches totaling 332 m (4 trenches) | Goldrock |
| 2007 | Metallurgical testwork on bulk samples totaling 2,200 kg | Goldrock |
| 2006-2008 | Drilling totaling 28,768 m (100 holes) | Goldrock |
| 2010 | Drilling totaling 3,480 m (18 holes) | Goldrock |
| 2016 | Metallurgical testwork on 4 x 750 kg of historical core | Fortuna |
| 2016 | Metallurgical testwork on 2 x 300 kg of historical core | Fortuna |
| 2016 | Drilling totaling 4,461 m (12 holes) | Fortuna |
| 2017 | Metallurgical testwork on 2 x 600 kg of fresh core | Fortuna |

## 9.1 Goldrock campaign (2000 to 2001)

The 2000 to 2001 campaign conducted by Goldrock comprised geological mapping, grid sampling, and trenching.

### 9.1.1 Geological mapping

Regional mapping, completed by December 2000, was initially carried out at scales of 1:20,000 and 1:10,000 accompanied by rock chip sampling, and was followed by detailed geological and alteration mapping of the Lindero Deposit at 1:5,000 scale (Dow, 2001).

### 9.1.2 Soil (talus fines) geochemistry

Sampling was completed on a 100 m x 50 m grid and consisted of 15 north–south lines of approximately 1,000 m in length and spaced 100 m apart with stations/pickets located every 50 m along the line. The lines were surveyed using a hand-held global positioning system (GPS) unit. A total of 304 talus fines samples were collected over an area 1,400 m by 1,200 m. Samples were collected from 20 cm-deep holes dug at each station. About 300–400 g of talus fines was sieved to 95 % passing 80 mesh. Samples were then bagged, numbered, and submitted for analysis to Acme Analytical Laboratories (Argentina) S.A. (ACME) in Mendoza, Argentina. The results indicated a gold-in-talus-fines anomaly (>200 ppb Au) that covered an area of 300 m x 500 m in the southeast part of the grid, approximately coincident with the CPD1–FPD porphyry. Copper results indicated a circular donut-shaped anomaly (>250 ppm Cu) with a radius of about 200 m. centered on the intrusive complex and largely coincident with the CPD1–FPD porphyry and potassic alteration zones (Kesting and Huidobro, 2001).

### 9.1.3 Trenching

Ten trenches, totaling 1,752 m, were excavated on the Lindero Deposit (Kesting and Huidobro, 2001). An access road exposed an additional 102 m of potassic-altered porphyry (trenches LC (N) and LC (S). The objectives of the trenching program were to better define the rock types and alteration assemblages, to expose gold–copper mineralization for better characterization, and to assess the gold and copper content of the mineralized zones. The work was carried out with a hydraulic excavator supplied by a local contractor. The trenches were sampled at 2 m and 5 m intervals except for 42 m where the alluvial cover was too deep (Table 9.2). Samples were collected by hand on a continuous chip basis or from channels cut by a diamond saw.

**Table 9.2 Trench and road cut samples taken in 2001 (see Figure 9.1 for locations)**

| Trench | Length (m) | Sample Interval | Sample Type | No. of Samples | Results of interest |
|---|---|---|---|---|---|
| LD250N | 225 | 5 | chip | 45 | 110 m averaging 404 ppb Au & 0.13 % Cu |
| LD50N | 326 | 5&2 | chip | 132 | 74 m averaging 1,472 ppb Au & 0.12 % Cu |
| LDBL | 136 | 2 | chip | 68 | 42 m averaging 1,029 ppb Au |
| LD50S | 185 | 5 | chip | 37 | 50 m averaging 593 ppb Au & 0.22 % Cu |
| LD200S | 116 | 2 | chip | 58 | 52 m averaging 436 ppb Au & 0.12 % Cu |
| LD300E | 156 | 2 | chip | 78 | 110 m averaging 1,077 ppb Au & 0.11 % Cu |
| LD200E | 208 | 2 | chip | 104 | 34 m averaging 1,644 ppb Au & 0.13 % Cu |
| LD50W | 160 | 2 | chip | 80 | 38 m averaging 911 ppb Au & 0.21 % Cu |
| LD150W | 80 | 5 | chip | 16 | 80 m averaging 442 ppb Au & 0.18 % Cu |
| LD200 | 160 | 5 | chip | 32 | 80 m averaging 442 ppb Au & 0.11 % Cu |
| LC (N) | 14 | 2 | chip | 7 | - |
| LC (S) | 88 | 2 | chip | 44 | - |

Samples were sent to ACME in Mendoza where gold grades were analyzed by fire assay with an atomic absorption finish (FA/AA) and copper was analyzed by AA. The data were presented on maps at a scale of 1:1,000 and in MapInfo™ format. Certain sections of continuous chip sampling were checked by analyzing diamond saw-cut channel samples taken from the same intervals.

Goldrock did not have a formal quality assurance/quality control (QAQC) program with well-defined and documented protocols and procedures in place for the trenching program. No estimate for analytical precision or accuracy was undertaken. Instead, Goldrock relied on

ACME's internal checks in determining the reliability of the analyses. It should be noted that these trench samples have not been used in the estimation of Mineral Resources as detailed in this Report.

A combination of trench mapping and sampling results indicated a semi-annular zone around the PMI–CPD2 porphyries with highest grades confined to the southeast corner of the mineralized zone. This zone measured some 550 m x 100 m, although there was evidence from the road-cutting to the south that the mineralization could be extended at surface for another 200 m of strike. In addition, a smaller zone of lower-grade mineralization was recognized from the semi-annular zone towards the center of the deposit area; this inner zone had dimensions of approximately 400 m x 70 m.

## 9.2 Rio Tinto campaign (2002 to 2003)

The second exploration campaign was carried out by Rio Tinto after the signing of an option agreement in April 2002. The Rio Tinto program (Ruiz et al, 2003) was conducted between May 2002 and February 2003 and consisted of:

- Geological mapping
- Geophysics (ground magnetics and induced polarization surveys)
- Road cut sampling
- Drilling (detailed in Section 10.2)
- Metallurgical testwork (detailed in Section 13)

In January 2003, Rio Tinto completed a 3-D block model and a preliminary "mineral inventory" for the Lindero prospect based on the above work.

### 9.2.1 Geological mapping

Rio Tinto re-mapped the Lindero area in September 2002. Additional data from exposures on new access roads constructed by Rio Tinto, as well as from drill core recovered from the Rio Tinto drilling program, were included in the geological studies. While addition of these newly-acquired data helped refine the deposit geology, no significant modifications to the geology as mapped by Goldrock were introduced. Geological data were presented on a map at a scale of 1:2,500.

Rio Tinto identified additional structural details for the deposit. Two main stockwork orientations s were defined: a northwest orientation, dominant in the western and southwestern part of the intrusive complex; and, a northeast vein orientation dominant in the eastern part of the grid.

Five main mineralizing events were recognized on the basis of crosscutting vein relationships. The veins, from earliest to latest, were interpreted by Rio Tinto to be:

- First-stage, narrow (hairline) magnetite veins (1–5 g/t Au)
- A-type quartz veins with minor sulfides (<1 g/t Au; ±Cu)
- Second stage hairline magnetite veins (>1 g/t Au)
- Coarse magnetite veins with chalcopyrite (Cu)

- Quartz veins with pyrite (weak Au)

### 9.2.2 Geophysics

A total of 43 km of ground magnetic data and 11 km of gradient-array IP data were collected from the grid over the Lindero Deposit. The magnetic data were collected with a line spacing of 100 m with stations spaced every 10 m along the lines, whereas the IP/resistivity survey data were collected with a line spacing of 200 m with stations spaced every 50 m along the lines.

Roscoe Postle Associates (RPA) reviewed the work (RPA, 2003) and in their opinion, both surveys were appropriate for disseminated mineralization; however, the magnetic spacing resulted in a large spatial sampling bias, which precluded effective contouring (Fuchter and Rennie, 2003). The IP survey spacing was deemed appropriate for identification of large-scale anomalies.

The surveys were undertaken with common commercial geophysical instruments (GSM GEM-19™ magnetometers, Elrec-6™ receiver and VIP-3000™ transmitter) by Quantec's local subsidiary based in Mendoza. Basic processing was undertaken by Rio Tinto using Geosoft™ software, and maps were produced in MapInfo™ format. RPA questioned some of Rio Tinto's interpretations (RPA, 2003).

There is no direct IP/resistivity response from the known gold mineralization; however, the IP data form a semi-concentric ring around the mineralization and probably represents pyrite in the propylitic alteration zone, although this has not been adequately tested by drilling. There is no direct magnetic response associated with the mineralization, despite the fact that drill results (e.g., LID-04) report strong magnetic intersections. RPA's opinion was that the magnetic interpretation was incomplete and that further magnetic surveying was required (RPA, 2003).

### 9.2.3 Road-cut sampling

A total of 374 of the chip samples collected from 3.5 km of new roads constructed on the Lindero Deposit in 2002 were used by Rio Tinto in construction their block model. Continuous chip sampling was conducted over 3 to 5 m intervals, and submitted for analysis to the ALS Chemex laboratory in Mendoza. The sample preparation was performed in Mendoza, and a sub-sample of 100 g together with blanks, duplicates, and certified standards was shipped to ALS Chemex in Vancouver for Au + 35 element analyses (Au+35T package). Gold was assayed by FA/AA and the 35 elements by inductively coupled plasma (ICP).

RPA concluded that QAQC procedures were well documented and appropriate (Fuchter and Rennie, 2003). They reported that the additional road-cut sampling data confirmed and extended the known intersections of surface mineralization previously outlined in the trenching undertaken by Goldrock.

### 9.2.4 Drilling and core logging

A total of 3,279 m of core drilling in 10 holes was completed at the Lindero Deposit from May to December 2002 by Rio Tinto. The drilling campaign is described in Section 10.1 of this Report.

## 9.3 Goldrock campaign (2005 to 2008)

The next exploration campaign was conducted by Goldrock with as described in detail in the AMEC (2010b) Technical Report and consisted of:

- Geological mapping
- Trenching
- Digital satellite topographic survey
- Drilling (described in Section 10.2)
- Road-cut sampling
- Metallurgical testwork (detailed in Section 13)

### 9.3.1 Geological mapping

In 2007, previous drilling and trenching information was compiled in order to develop a geologic model for Lindero at a scale of 1:1,000. All drill holes were relogged. Trenches and roads were mapped in detail. A 1:2,000 topographic map was built, based on a satellite image AMEC (2010b).

### 9.3.2 Trenching

In 2005, Goldrock completed a trenching program which was designed to define surface mineralization (AMEC, 2010b). Sixteen trenches totaling 1,234 m were excavated. Trenches were channel sampled every 2 m with a rock saw, and a total of 498 samples were collected.

Four additional trenches (332 m) were excavated in 2007 (NTR-01 to NTR-04) on the west zone of the Lindero Deposit, in order to characterize the nature of surface mineralization in that area. A total of 159 samples were taken.

A summary of trench locations and soil samples collected on the property over the successive exploration surveys is presented in Figure 9.1.

**Figure 9.1 Trench locations and soil grid layout**

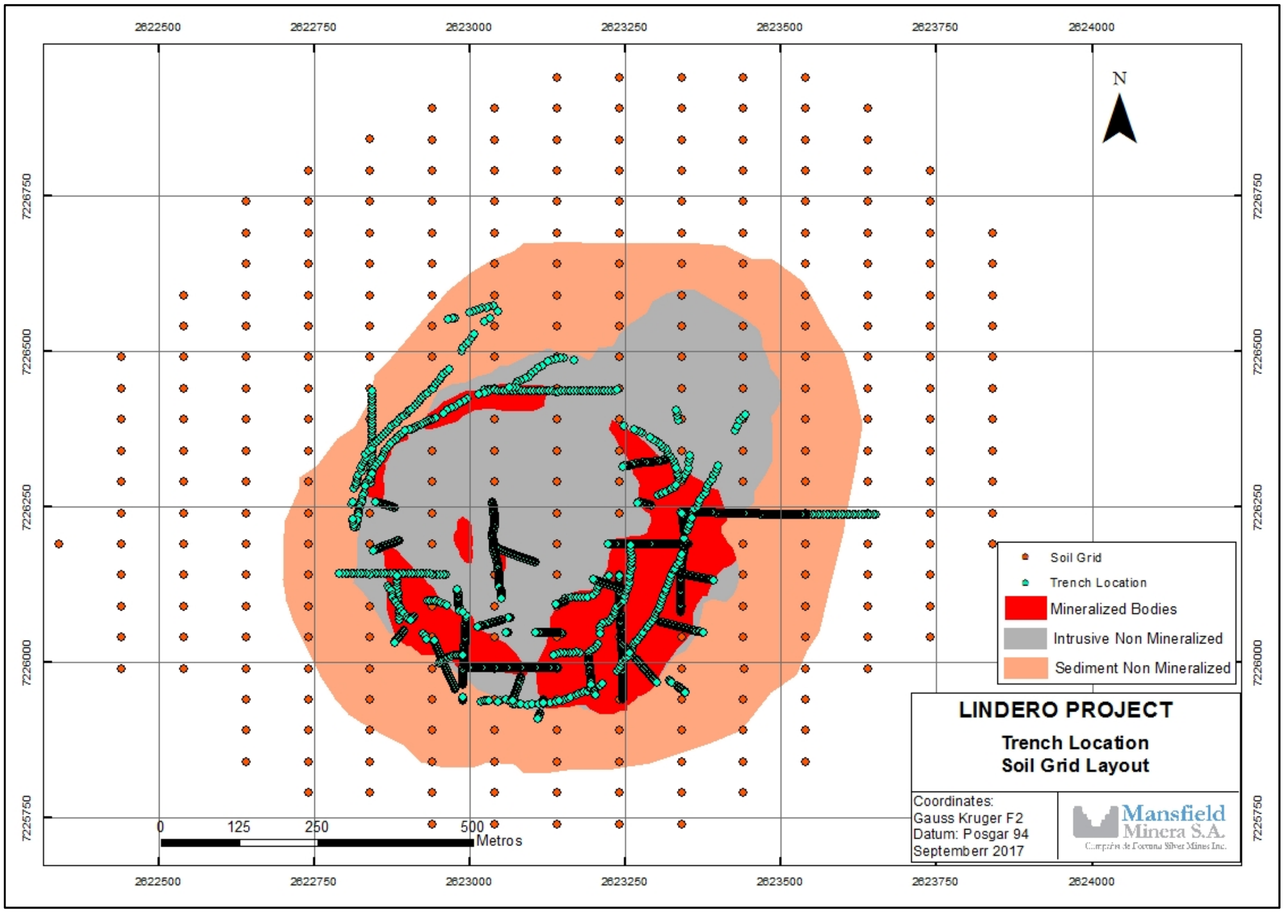



Figure prepared by Mansfield, 2017

### 9.3.3 Topographic survey

The Lindero topographic data were obtained from a QuickBird™, natural color, 64 cm pixel satellite image which covered 64 km². Fixed control points on the ground were used to rectify the image to the proper co-ordinate system. The QuickBird™ image was combined with a stereo Ikonos™ image, which covered 100 km², to generate a 1-m resolution digital elevation model (DEM). These two products were used to construct an ortho-rectification of the image, and Infosat™ using the PCI Geomatica Orthoengine™ software produced a topographic map with 5 m contour levels.

## 9.4 Goldrock campaign (2010)

Between August and November 2010, six holes were drilled for geotechnical studies (AMEC, 2011). Five condemnation holes were drilled in strategic areas where important infrastructure such as leach pads and waste rock piles were planned. An additional seven geotechnical holes were drilled in areas of planned foundation construction. A total of 3,480.5 m was drilled in 18 holes. The locations and lengths of these drill holes are shown in Table 9.3.

**Table 9.3 Drill hole locations for 2010 drill campaign**

| Hole I.D. | Easting (m) | Northing (m) | Elevation (m) | Azimuth (°) | Dip (°) | Depth (m) | Purpose |
|---|---|---|---|---|---|---|---|
| LGT-06 | 2622846 | 7226034 | 3919.6 | 225 | 75 | 350 | Geotechnical |
| LGT-07 | 2622854 | 7226358 | 3908.3 | 315 | 75 | 350 | |
| LGT-08 | 2623088 | 7226474 | 3919.7 | 360 | 75 | 350 | |
| LGT-09 | 2623389 | 7226340 | 3940.4 | 060 | 75 | 350 | |
| LGT-10 | 2623053 | 7225937 | 3942.2 | 180 | 75 | 350 | |
| LGT-11 | 2623368 | 7226050 | 3909.8 | 125 | 75 | 350 | |
| CON-01 | 2623939 | 7225801 | 3828.7 | 000 | 60 | 200 | Condemnation |
| CON-02 | 2623540 | 7226013 | 3854.7 | 250 | 60 | 300 | |
| CON-03 | 2623019 | 7225780 | 3893.7 | 290 | 60 | 200 | |
| CON-04 | 2624147 | 7226481 | 3757.0 | 270 | 60 | 235.5 | |
| CON-05 | 2622638 | 7226896 | 3755.1 | 140 | 60 | 200 | |
| LP | 2624301 | 7226805 | 3732.0 | 000 | 90 | 35 | Foundations |
| ADR | 2623800 | 7227021 | 3736.4 | 000 | 90 | 35 | |
| PWH | 2623991 | 7226913 | 3727.4 | 000 | 90 | 35 | |
| NHPGR | 2623659 | 7225861 | 3835.6 | 000 | 90 | 35 | |
| HPGR | 2623659 | 7225879 | 3831.1 | 000 | 90 | 35 | |
| LIM | 2623641 | 7225961 | 3828.4 | 000 | 90 | 35 | |
| STO | 2623721 | 7225949 | 3817.6 | 000 | 90 | 35 | |

The geotechnical holes were planned by AMEC Earth & Environmental Americas (AMEC, 2011). These holes were part of geotechnical studies conducted by AMEC to complete open pit geotechnical design. The scope of work undertaken included:

- Selection of drill hole locations
- Geotechnical logging and core orientation
- Selection of representative samples for unconfined compression testing and direct shear testing
- Completion of point load tests, core photography, and laboratory testing on selected samples

The geotechnical holes were oriented with a Reflex Act™ instrument and downhole deviations were measured with a Reflex™ gyroscopic instrument.

The condemnation holes were drilled to demonstrate that the areas selected for leach pads, waste rock piles and other facilities on surface did not have underlying mineralization. Holes were logged and sampled with the same methodology as the geotechnical drill holes. The condemnation holes were down-hole surveyed.

Samples from the geotechnical and condemnation drill holes were collected every two meters and analyzed by Alex Stewart in Mendoza. A total of 1,881 samples were assayed by ICP analysis for 39 elements, and were analyzed for gold by FA (30 g charge). Goldrock inserted QAQC samples with the batches submitted to the analytical laboratory.

The assay results from the geotechnical holes showed weak mineralization. The assay results from the condemnation holes did not show any anomalous gold or copper values. The holes drilled for foundation design were not assayed.

## 9.5 Fortuna campaign (2016 to 2017)

The most recent exploration campaign was conducted by Mansfield under the supervision of Fortuna. Tasks that have been completed since August 2016 include:

- Geological mapping and relogging
- Investigation into soluble copper
- Heterogeneity testwork
- Geotechnical investigations
- Drilling (described in Section 10.3)
- Metallurgical testwork (detailed in Section 13)

### 9.5.1 Geological mapping and relogging

Geological mapping of the entire surface area immediately southeast of Lindero up to and including the historic drilling at Arizaro was conducted in 2017. In addition, all historical drill core, for both the Lindero and Arizaro Deposits, was relogged under the supervision of porphyry specialist, Brock Riedell.

### 9.5.2 Soluble copper investigation

As a component of the metallurgical testwork conducted by Fortuna for the Lindero Project, 1,536 samples were taken from eight drill holes spatially distributed around the deposit. Those samples are considered by Fortuna to be representative of the style of mineralization. All samples were submitted to ALS Global for assaying which included the evaluation of soluble copper via cyanide leach with an atomic absorption finish. Assays were entered into the Lindero database and cross-referenced using their intervals against the corresponding lithology description.

The assay results for the gold, total copper, and soluble copper, as well as the calculated soluble copper as a percentage of total copper (Sol Cu: Tot Cu (%)) were analyzed in order to better understand the behavior of soluble copper at Lindero.

The level of soluble copper in the mineralized lithologic units ranges from 6.5 ppm to 2,530 ppm, depending on the copper minerals present. Soluble copper levels as a percentage of total copper appear to be consistent averaging between 9.3 % and 11.2 % in the major mineralized oxide and sulfide domains. Statistical analysis and scatter plots (Figure 9.2) demonstrate there is no significant correlation between total copper and cyanide-soluble copper.

**Figure 9.2 Scatter plot of total copper : cyanide soluble copper**

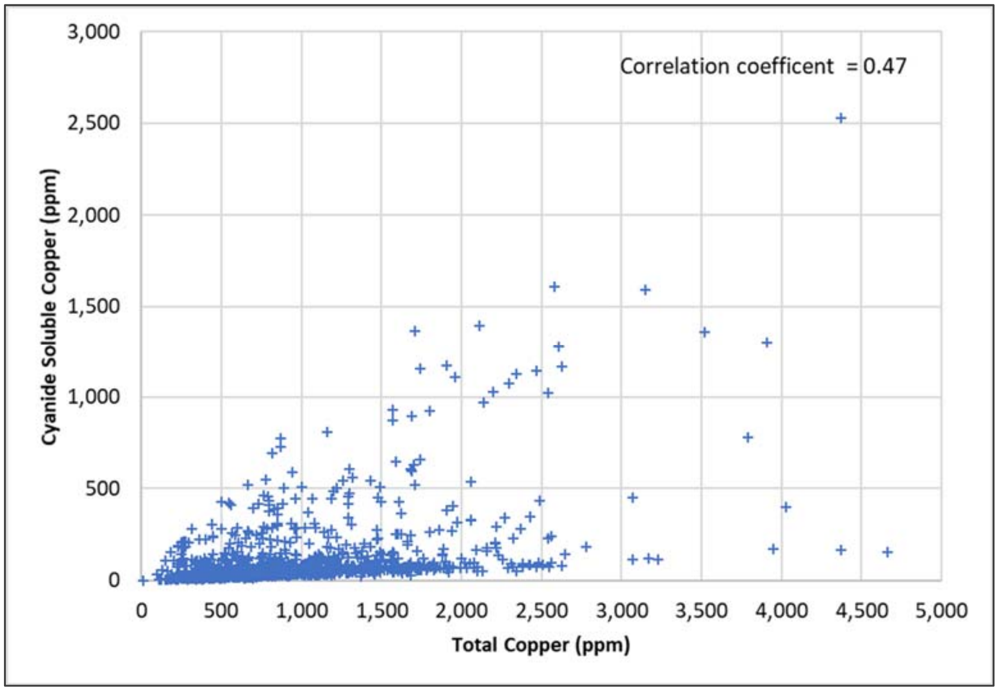



Figure prepared by Fortuna Silver Mines Inc., 2017

It would be logical to assume the sulfide domain would have a lower average soluble copper to total copper percentage, being dominated by low-solubility chalcopyrite (~6 %), but the occasional presence of highly-soluble bornite (~70 %) in the sulfide domains, albeit in small concentrations, results in a significant tail to the distribution. This is in contrast to the oxide domains that contain moderately soluble copper oxide species such as neotocite and chrysocolla (~15%), albeit in higher proportions. Subsequently, the soluble copper as a percentage of total copper in FPD/CPD1 sulfide averages 10.4 %, ranging between 2 % and 90 %; whereas in FPD/CPD1 oxide it averages 11.2 %, ranging between 2 % and 51 %.

Fortuna also commenced an investigation into the soluble copper at Arizaro with the submission of 321 samples from historic core pulps sent to ALS Global for assaying with cyanide leach and an AA finish. Complete evaluation of the analytical information is underway. The archived pulp samples selected are considered to be representative of the oxide, mixed and sulfide mineralization, and the data will be used as a precursor to designing possible future metallurgical testing at Arizaro.

### 9.5.3 Heterogeneity test

As part of the validation process, Fortuna opted to conduct a heterogeneity test on the most abundant ore type to determine the sampling characteristics of both gold and copper. The study uses the principles of the "Theory of Sampling" to determine the characteristics of the metals of interest.

Fortuna collected 87 half core samples totaling 492 kg from a variety of drill holes to represent the most commonly occurring ore type at the Lindero Deposit and transported this material to the Campos, Menichetti & Suarez Spa (CMS) Asociados laboratory in Santiago, Chile who conducted the heterogeneity testwork (CMS, 2016a, 2016b). The sample preparation report and all results were provided to Francis Pitard Sampling Consultants (FPSC) for statistical analysis. A report detailing the results and recommendations from the heterogeneity test was provided to Fortuna (FPSC, 2016), with a summary of the main points from that report detailed below:

- The heterogeneity of the tested ore is reasonably small, which is beneficial and leads to sampling and subsampling protocols that are relatively easy to implement
- The heterogeneity of gold is substantially larger than the heterogeneity of copper, clearly suggesting that a substantial amount of gold is not associated with copper minerals. This is a warning for possible segregation problems in sampling and subsampling protocols
- The experimental values obtained for the sampling constant are unusually low for copper, and very unusual for gold, which is beneficial for obtaining representative samples
- Extraction errors and preparation errors taking place during the implementation of sampling protocols are not likely to introduce large sampling biases for copper and gold: This is a tremendous asset for the Lindero Project

## 9.6 Geotechnical studies

A series of geotechnical studies have been conducted on the Lindero Deposit during the Goldrock campaigns by various consulting firms including Golder (2008), Seegmiller (2009), AMEC (2011), and Seegmiller (2013).

Fortuna contracted CNI to perform a review of all available geotechnical information (CNI, 2016a and 2017).

Recommendations from the review work led to a site visit conducted from 09 to 12 August 2016, by Mr. Robert Pratt, P.E., Senior Engineer, and Mr. Francisco Sanz, Geotechnical Geologist, both of CNI, along with Mr. Edwin Gutierrez of Fortuna to the Lindero Property. During this site visit, the following tasks were completed (CNI, 2016a):

- Review of general geology and rock quality for various alteration types
- Cell mapping of road cuts to collect rock fabric data
- Sampling for laboratory testing
- Review, validation, and correction of AMEC's oriented core
- Measurement of spacing of gypsum-filled bedding joints (hole LGT-08)

The site visit was conducted in conjunction with a full review by CNI of previous geotechnical pit slope design work.

A second site visit was conducted by Mr. Francisco Sanz of CNI from 15 to 22 October 2016 (CNI, 2016b and 2017). The purpose of this visit was to collect data to address the limitations of the available geotechnical information noted during the review and August visit. The following were completed during the October site visit:

- A core drilling program was in progress at the time of the site visit, which provided an opportunity to collect geotechnical data. The drilling program consisted of core holes that were drilled primarily for metallurgical testing purposes. Geologists contracted by Fortuna were trained in geotechnical core logging

- Three trenches were excavated for investigation and sampling of materials comprising the foundation of planned waste stockpiles
- A database of all previous geotechnical data including geomechanical drilling data, oriented core data, and laboratory testing data was compiled by CNI and provided to Fortuna. QAQC checks were conducted on these data prior to providing the data
- Fault projections developed by CNI were reviewed with Fortuna geologists. The conclusion of this work was that it is difficult to accurately project the faults into the subsurface because of good quality rock recorded in drill holes. The faults may be partially sealed with gypsum at depth. The projected faults are valid, given the available information. The position of the faults must be reviewed with surface mapping, when bench faces become available

A final report detailing the findings of the above work program and recommendations in respect to pit slope angles and geotechnical considerations for construction purposes was provided to Fortuna by CNI in May 2017 (CNI, 2017), and considered during the infrastructure and pit design processes. These studies are considered by the QP to be consistent with industry practices and adequate to support mine design.

## 9.7 Hydrogeological studies

Hydrogeological studies were conducted by Vector Argentina on behalf of Mansfield in 2009 to locate adequate water sources for the Lindero Project. Results were presented by Vector in two reports (Vector, 2009a, 2009b).

Several exploratory wells were drilled and pumping tests conducted in the Project area, notably in the Lindero, Arita, and Chachas sub-basins (Andina, 2011a-b, Conhidro 2013, Hidrotec 2012a-e).

During construction, Project water will be supplied by three water wells located on-site near the proposed man camp. These wells combined can provide approximately 50 to 55 $m^3$/hr on a short-term basis, sufficient for water needs during construction and sufficient for supplying water to the potable water treatment plant and for other small miscellaneous needs during operations. However, these wells cannot provide enough water on a continuous basis for the full operational make-up requirement.

Process water requirements are 97.7 $m^3$/hr and will be sourced from two existing wells located 13 km southeast of the Project site. An additional well is required and will be drilled as part of construction activities.

## 9.8 Petrography

A number of petrographic studies have been conducted on the Lindero Deposit, including one prepared for Rio Tinto by Dr. M.F. Marquez Zavalia (2002); and three prepared for Mansfield of which two were authored by Dr. L.T. Larson of LTL Petrographics (2004 and 2006), and the other conducted by Maria de Belen Palacio (2007). A summary of the findings follows:

- Base metal minerals were reasonably consistent with a few minor variations, and included magnetite, hematite, pyrite, and chalcopyrite with bornite, chalcocite, covellite, as well as goethite, cuprite, and malachite in relatively shallow and oxidized samples

- The copper mineralization is primarily chalcopyrite, which is largely refractory to cyanide solutions, but the presence of cyanide-soluble copper minerals was also identified in both the oxide (chrysocolla) and sulfide (bornite) domains
- Gold grains appeared in two dimensions to be largely encapsulated or partially encapsulated in chalcopyrite. Three out of four of the gold grains were only partially encapsulated; there was either a micro-fracture or a micro-inclusion at or near the boundary of the gold grain with a larger chalcopyrite grain
- Potassium alteration is predominant in its feldspathic phase with intensity from light to very strong, being in the latter case a penetrative (pervasive) alteration with almost total rock replacement. A biotitic phase is also present although generally mild to moderate in intensity. Mild to moderate propylitic alteration (calcite + chlorite) is also present in most samples
- Of the total of 37 samples examined by Dr. Larson, 19 % contained microscopically-visible gold. Of 22 samples from the higher-grade portions of the orebody, 27 % contained microscopically-visible gold; 83 % of this subgroup contained gold associated with chalcopyrite

The above observations were further supported by additional petrographic analysis conducted by Riedell (2016b) on behalf of Fortuna to investigate soluble copper behavior on samples obtained during the 2016 drill program.

## 9.9 Exploration potential

### 9.9.1 Lindero Deposit

The Lindero porphyry gold system remains open at depth below the pit shell used to constrain the Mineral Resources. An area of interest was identified by Fortuna during the drilling campaign in 2016 with drill hole LDH-126 encountering 0.97 g/t Au over a 38 m interval (from 444 m to 482 m down-the-hole). This is supported by historical drilling from 2007 including drill hole LDH-86 that averaged 1.06 g/t Au over a 52 m interval (from 400 m to 452 m down-the-hole), and which bottomed in mineralization. This is an area of high-grade gold mineralization worthy of additional exploration at depth.

### 9.9.2 Arizaro Deposit and other prospects

The Arizaro Project Deposit is located within the Lindero mining concession, 3.2 km southeast of the Lindero Project. Historic work has included regional and local mapping, trenching with surface rock-chip sampling (27 trenches, 6,568 total m, 495 samples) and drilling (8,854 m, 29 core holes) completed over four campaigns (2002 and 2010–2013). Table 9.4 provides an example of the types of intercepts recorded from the Arizaro Deposit. Drill results include hole ARD-24 intersecting 160 meters averaging 0.64 g/t Au, including 104 meters averaging 0.82 g/t Au.

Table 9.4 Arizaro composite intercepts of interest

| Drill hole | Easting | Northing | Elevation | Azimuth (°) | Dip (°) | From (m) | To (m) | Interval (m) | Au (g/t) | Cu (%) |
|---|---|---|---|---|---|---|---|---|---|---|
| ARD-01 | 2625701.40 | 7223689.90 | 4147.00 | 260 | 70 | 3.05 | 82 | 78.95 | 0.56 | 0.17 |
| ARD-02 | 2625630.00 | 7224058.50 | 4104.00 | 270 | 70 | 60 | 90 | 30 | 0.45 | 0.27 |
| | | | | | | 190 | 242 | 52 | 0.28 | 0.15 |
| ARD-03 | 2625284.66 | 7223693.79 | 4145.33 | 073 | 70 | 10 | 34 | 24 | 0.70 | 0.36 |
| ARD-04 | 2625600.79 | 7223907.67 | 4121.25 | 255 | 61 | 8 | 188 | 180 | 0.43 | 0.17 |
| ARD-05 | 2625735.64 | 7223995.56 | 4097.77 | 250 | 62 | 84 | 242 | 158 | 0.50 | 0.16 |
| ARD-06 | 2625771.57 | 7223715.96 | 4140.14 | 250 | 70 | 224 | 242 | 18 | 0.35 | 0.09 |
| ARD-07 | 2625688.12 | 7223718.35 | 4149.56 | 355 | 69 | 2 | 148 | 146 | 0.29 | 0.17 |
| ARD-08 | 2625601.28 | 7223626.50 | 4127.15 | 020 | 70 | No intervals of significance | | | | |
| ARD-09 | 2625855.97 | 7223957.64 | 4096.18 | 270 | 61 | 50 | 218 | 168 | 0.34 | 0.16 |
| ARD-10 | 2625801.42 | 7223857.33 | 4118.42 | 270 | 61 | 1 | 40 | 39 | 0.46 | 0.15 |
| | | | | | | 176 | 240 | 64 | 0.28 | 0.14 |
| ARD-11 | 2625754.78 | 7223756.72 | 4138.85 | 270 | 61 | 156 | 208 | 52 | 0.29 | 0.09 |
| ARD-12 | 2625761.88 | 7224047.82 | 4090.69 | 270 | 56 | 180 | 294 | 114 | 0.39 | 0.19 |
| ARD-13 | 2625700.13 | 7223942.90 | 4110.00 | 235 | 86 | 12 | 180 | 168 | 0.61 | 0.22 |
| including | | | | | | 100 | 130 | 30 | 1.81 | 0.40 |
| ARD-14 | 2625920.88 | 7223885.92 | 4094.34 | 270 | 60 | 140 | 214 | 74 | 0.53 | 0.19 |
| | | | | | | 274 | 318 | 44 | 0.29 | 0.19 |
| ARD-15 | 2625471.81 | 7224012.30 | 4092.59 | 090 | 57 | 252 | 384 | 132 | 0.37 | 0.19 |
| ARD-16 | 2625561.60 | 7224079.59 | 4085.50 | 090 | 61 | 24 | 64 | 40 | 0.24 | 0.14 |
| ARD-17 | 2625690.60 | 7223944.96 | 4110.08 | 270 | 60 | 48 | 88 | 40 | 0.66 | 0.18 |
| ARD-18 | 2625807.34 | 7223944.03 | 4106.10 | 270 | 60 | 0 | 246 | 246 | 0.40 | 0.18 |
| ARD-19 | 2625603.21 | 7223941.35 | 4114.34 | 250 | 59 | 126 | 186 | 60 | 0.55 | 0.16 |
| ARD-20 | 2625722.15 | 7223894.30 | 4121.20 | 350 | 81 | 202 | 274 | 72 | 0.32 | 0.21 |
| ARD-21 | 2625686.34 | 7223900.4 | 4121.23 | 348 | 70 | 20 | 359 | 339 | 0.43 | 0.17 |
| including | | | | | | 28 | 122 | 94 | 0.66 | 0.18 |
| ARD-22 | 2625697.06 | 7223898.39 | 4121.17 | 250 | 60 | 4 | 244 | 240 | 0.28 | 0.12 |
| ARD-23 | 2625849.97 | 7223900.35 | 4105.07 | 270 | 60 | 28 | 74 | 46 | 0.36 | 0.20 |
| ARD-24 | 2625689.44 | 7224999.97 | 4102.05 | 270 | 78 | 90 | 250 | 160 | 0.64 | 0.24 |
| including | | | | | | 120 | 224 | 104 | 0.82 | 0.26 |
| ARD-25 | 2625828.12 | 7223999.83 | 4093.44 | 270 | 70 | No intervals of significance | | | | |
| ARD-26 | 2625711.68 | 7223850.91 | 4132.43 | 270 | 59 | 160 | 178 | 18 | 0.47 | 0.17 |
| ARD-27 | 2625726.59 | 7223809.92 | 4142.78 | 270 | 67 | 26 | 136 | 110 | 0.53 | 0.20 |
| ARD-28 | 2625715.16 | 7223728.32 | 4150.54 | 270 | 69 | 6 | 134 | 128 | 0.43 | 0.15 |
| ARD-29 | 2625675.56 | 7223657.31 | 4140.19 | 090 | 70 | 22 | 50 | 28 | 0.44 | 0.22 |

Since August 2016, Fortuna has completed the following work at the Arizaro Deposit aimed at geologic re-interpretation of the porphyry system, and identifying near surface gold resources that could potentially add economic benefit to the Lindero Project:

- Relogging of 8,817 m (29 drill holes) of historic core - similar geology, alteration and mineralization are found at both Lindero and Arizaro
- Remapping (1:2,000 scale) of the entire surface area from immediately southeast of Lindero to and including the zone of historic drilling at Arizaro
- Submission of 321 samples from historic core pulps for cyanide soluble copper analyses with resulting data to be used as a precursor to designing possible future metallurgical testing and subsequent, shallow drilling at Arizaro

Exploration work to date on the Lindero concession has been focused on outcropping porphyry mineralization. Fortuna will evaluate the holding for mineralization beyond the two known porphyry systems at Lindero and Arizaro. For example, alteration zones and silica structures located within the concession, about 2.5 km due south of the Lindero Project site would support additional exploration.

Exploration of these areas, as well as the Arizaro system, will become priorities following commencement of production from the Lindero Deposit.

## 9.10 Comment on Section 9

In the opinion of the QPs:

- the mineralization style and setting of the Lindero Deposit is sufficiently well understood to support Mineral Resource and Mineral Reserve estimation
- Exploration methods are consistent with industry practices and are adequate to support continuing exploration and Mineral Resource estimation
- Exploration results support Fortuna's interpretation of the geological setting and mineralization
- Continuing exploration may identify additional mineralization that could support Mineral Resource estimation

# 10 Drilling

## 10.1 Introduction

A total of 38,137 m of diamond drilling in 139 holes was completed at the Lindero Deposit from April 2002 to November 2010 under the supervision of Goldrock and Rio Tinto. An additional 12 holes totaling 4,461.58 m were drilled under the supervision of Fortuna in 2016, primarily to collect fresh samples for additional metallurgical testwork and to improve the geological interpretation. Ground conditions were good, and core recovery was generally above 90 %. All drilling was conducted by diamond core drilling methods.

**Table 10.1 Drilling by company and period at the Lindero Deposit**

| Company | Period | Drill Holes | Hole Identifier | Diameter | Meters |
|---|---|---|---|---|---|
| Rio Tinto | 2002 | 10 | LID-1 to LID-10 | HQ/NQ | 3,279.12 |
| Goldrock | 2005/06 | 11 | LDH-11 to LDH-21 | HQ/NQ | 2,609.24 |
| | 2006 | 17 | LDH-22 to LDH-38 | HQ/NQ | 5,441.20 |
| | 2006/07 | 48 | LDH-39 to LDH-86 | HQ/NQ | 14,569.90 |
| | 2007/08 | 30 | LDH-87 to LDH-116 | HQ/NQ | 7,404.00 |
| | 2008 | 5 | LGT-1 to LGT-5 | HQ | 1,353.00 |
| | 2010 | 18 | LGT-6 to LGT-11<br>CON-01 to CON-05<br>7 Geotech holes | HQ | 3,480.00 |
| Fortuna | 2016 | 12 | LDH-117 to LDH-128 | HQ/NQ | 4,461.58 |
| Totals | 2002–2017 | 151 | | | 42,598.04 |

A total of 151 diamond core holes totaling 42,598.04 m have been drilled in the Lindero Deposit area (Figure 10.1) with most of the holes being generally orientated perpendicular to the mineralization, forming a radial pattern. The 7 geotechnical holes drilled in 2010 were for foundation design purposes only, each being just 35 m in depth and outside the area of interest. They have not been considered in subsequent sections as part of the Mineral Resource estimation process.

Samples were used not only to determine the distribution of mineralization and alteration in the deposit, but also for lithology, density, metallurgical and geotechnical tests.

All drill holes were drilled using HQ (63.5 mm core diameter) for the first 300 m, and were subsequently reduced to NQ (47.6 mm). This reduction occurred due to the drill rig having insufficient power to drill HQ beyond 300 m. Table 10.2 details the diameter of all core drilled at Lindero.

**Table 10.2 Drilling by core size, Lindero Project**

| Core Size (Diameter) | Meters | Percentage |
|---|---|---|
| HQ (63.5 mm) | 33,327.13 | 78 |
| NQ (47.6 mm) | 8,316.79 | 20 |
| Not recorded | 954.12 | 2 |
| Drill core diameter that was not recorded is related primarily to the condemnation holes drilled during the 2010 campaign in barren areas. Drill core is no longer available for these holes, but core photographs indicate that missing core was HQ in diameter. | | |

**Figure 10.1 Drill hole location map for the Lindero Project**

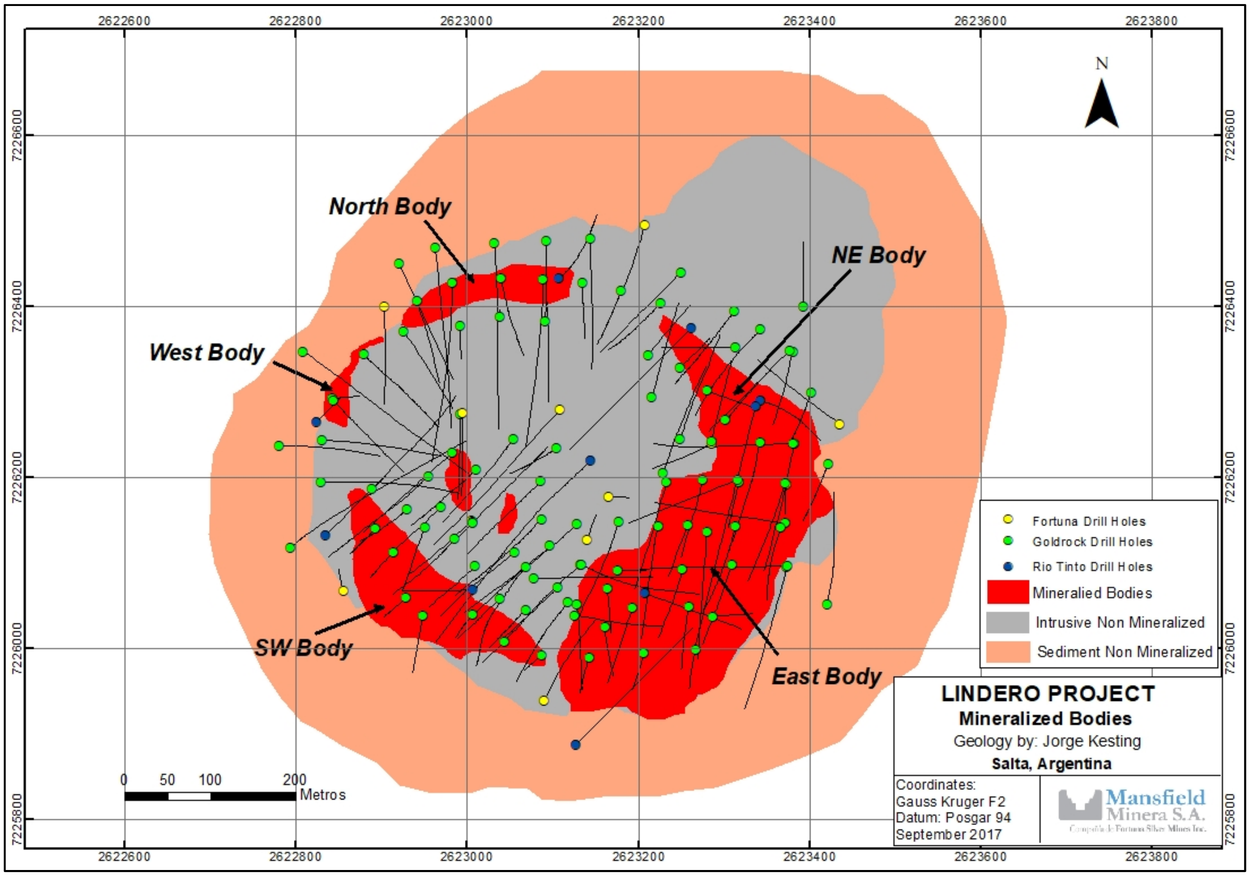



Figure prepared by Mansfield., 2017

## 10.2 Rio Tinto campaign (2002)

Rio Tinto completed a 10-hole core drilling campaign between April and December 2002. These holes were drilled by Connors Drilling from Mendoza, Argentina, using a Longyear 44 wireline drill rig.

Drill holes were cored at HQ size, and the core was logged on site by Rio Tinto geologists. Goldrock personnel helped with the logging and sampling of the core, which was subsequently stored in Goldrock's warehouse in Salta. Core logging was entered directly onto computer with drill logs produced on paper and CD-ROM. Drill sections were produced at a scale of 1:1,000. The holes were labeled LID-1 to LID-10, and totaled 3,279.12 m. The drilling was performed in two phases, with the first six holes testing the highest-grade trench results and the strongest alteration in outcrops, while the final four holes were intended to test the central part of the mineralized porphyry system and to define the extension of the mineralized zone outlined during the previous phase of drilling.

Drilling conditions were good, and core recovery was generally above 90 %, although LID-09 was abandoned in a fault zone (Fuchter and Rennie, 2003).

The drilling outlined gold–copper mineralization that is generally coincident with the CPD1-FPD (Rio Tinto code P2) porphyry and the annular potassic alteration zone, and is largely confined to the eastern, southeastern, and southern parts of the intrusive complex. Two higher-grade (~1 g/t Au) core zones were located within the general zone of mineralization. The Main High-Grade Zone (MVZ) described by Rio Tinto was a semi-annular zone was located in the core of the mineralization along the southeastern and southern part of the complex and appeared to be controlled by the intersection of the west–northwest and north–northeast structural trends. The second, smaller, mineralized body, termed the Parallel High-Grade Zone (PVZ), was parallel to, and located inboard from, the MVZ largely within the DDP and PBFD (Rio Tinto code PA) porphyries in the central part of the intrusive complex.

## 10.3 Goldrock campaign (2005 to 2010)

Goldrock completed a total of six separate drill campaigns on the Lindero Deposit. Drill hole prefixes were changed to LDH for Goldrock drill holes drilled to assess the geology and mineralization from 2005 through January 2008; those drill holes were cored to HQ size, and the core was logged on site by Goldrock geologists.

In the first campaign, between October 2005 and February 2006, Patagonia Drilling Company completed drill holes LDH-11 to LDH-21 for a total of 2,609.24 m of core. Longyear 44 and F-2000 drill rigs were used during this campaign.

Subsequently, four more campaigns were conducted by Falcon Drilling Company (branch Barbados), using a Longyear 38 drill rig. The second campaign started in early 2006, the third between late 2006 and early 2007, the fourth from late 2007 to early 2008, the fifth in September 2008, and the sixth between August and November 2010, (Table 10-2).

In early 2006, holes LDH-22 to LDH-38 were drilled. During this second campaign, a total of 5,441.2 m was drilled, mainly in the eastern and southwestern part of the complex.

The third campaign began in late 2006 and terminated in mid-2007, totaling 48 holes (LDH-39 to hole LDH-86) and 14,569.9 m of core. During this campaign, the northern mineralized zone of the deposit was discovered after holes LDH-48 and LDH-50 intersected significant mineralization.

The fourth campaign from June 2007 to January 2008 comprised 30 holes (LDH-87 to LDH-116) for a total of 7,404.0 m of core.

The fifth campaign in September 2008 consisted of five geotechnical drill holes for a total of 1,353 m of core. The drill holes are located in the red-bed sedimentary rocks surrounding the mineralization. Geotechnical holes were assigned LGT- prefixes.

The sixth drill campaign from August to November 2010 consisted of 18 geotechnical and condemnation holes described in Section 9.4.

## 10.4 Fortuna campaign (2016)

Fortuna conducted an additional drill campaign from September to December 2016, drilling 12 holes totaling 4,461.58 m. The purpose for the drilling campaign was as follows:

- Provide fresh core for metallurgical testwork, with 8 holes (LDH-117 to LDH-124) being drilled around the deposit to provide fresh representative bulk samples of different metallurgical types

- Improve the understanding in areas of geologic uncertainty with 2 holes (LDH-125 and LDH-126) being drilled near the center of the deposit to investigate the contact between mineralized and barren intrusive events
- Confirm previously-drilled assay results and improve the understanding of cyanide-soluble copper, with the drilling of 2 twin holes (LDH-127 and LDH-128) in the east of the deposit following the trace of two previously drilled holes (LDH-25 and LDH-112) from the Goldrock campaigns

The locations and lengths of these drill holes are shown in Table 9.4.

**Table 10.3 Drill hole locations for 2016 drill campaign**

| Hole I.D. | Easting (m) | Northing (m) | Elevation (m) | Azimuth (°) | Dip (°) | Depth (m) | Purpose |
|---|---|---|---|---|---|---|---|
| LDH-117 | 2623089 | 7225940 | 3925.40 | 032 | 75 | 434 | Metallurgy |
| LDH-118 | 2622856 | 7226067 | 3925.95 | 345 | 85 | 296 | |
| LDH-119 | 2622903 | 7226400 | 3903.64 | 180 | 73 | 407 | |
| LDH-120 | 2623207 | 7226495 | 3934.78 | 196 | 75 | 324 | |
| LDH-121 | 2623286 | 7226240 | 3952.99 | 247 | 79 | 419 | |
| LDH-122 | 2623140 | 7226128 | 3967.90 | 024 | 85 | 407 | |
| LDH-123 | 2623004 | 7226150 | 3978.75 | 225 | 76 | 410 | |
| LDH-124 | 2623435 | 7226263 | 3915.90 | 305 | 67 | 317 | |
| LDH-125 | 2623165 | 7226178 | 3964.60 | 090 | 85 | 332 | Geology |
| LDH-126 | 2623108 | 7226280 | 3969.94 | 222 | 65 | 515.58 | |
| LDH-127 | 2622994 | 7226275 | 3972.12 | 180 | 70 | 300 | Twin holes |
| LDH-128 | 2623373 | 7226193 | 3921.52 | 195 | 69 | 300 | |

Falcon Drilling Company conducted the drilling using a DC Falcon HYDX-6 drill rig. Drill holes were cored to HQ or NQ size depending on resistance with depth, and all core was logged on site by Fortuna and Goldrock geologists in conjunction with the relogging of historical core. A geotechnical engineer from CNI (CNI, 2016b; 2017) was contracted to ensure appropriate geotechnical logging was conducted by Mansfield geologists during the 2016 drilling campaign as described below.

## 10.5 Geological and geotechnical logging procedures

### 10.5.1 Mansfield lithological core logging (2005 to 2010)

Mansfield standardized a rock unit classification, logging procedure, and log sheet structure that was used throughout initial logging of all Mansfield holes (LDH-11 through LDH-116, LGT-01 through -LGT-11, and CON-01 through CON-05) as well as relogging of the Rio Tinto holes LID-01 through LID-10. The system used paper forms, and the data were subsequently entered into an Excel™ template. Geologic logging took place after the core was sampled, to take advantage of the flat sawed surface. Rock types and structure were recorded with alphanumeric codes, whereas alteration, veinlets, minerals, and oxidation were recorded by a 1 to 3 scale (weak, moderate, strong). A core library was developed to illustrate all rock and alteration types.

Initial geotechnical logging recorded only recoveries and RQD. From hole LDH-53 onwards, more detailed information was collected, including rock hardness index, rock weathering index, fracture frequency and type, roughness, infilling material and aperture of fractures. Additionally, core samples approximately 10 cm long were collected at 10 m intervals for bulk density measurements.

### 10.5.2 Fortuna relogging and drilling (2016)

A team of seven Fortuna and Mansfield geologists relogged all available core from LID-01 through LID-10, LDH-11 through LGT-116, LGT-01 through LGT-11, and CON-02 between 10 September 2016 and 14 January 2017. In addition, core from infill holes LDH-117 through LDH-126 and twin holes LDH-127 (alongside LDH-112) and LDH-128 (twinning LDH-25) were logged during the same period. The program included approximately 40,000 m of historic and new core. An estimated 10 % of historic core could not be relogged since all material had been consumed for metallurgical studies. For new drilling, both geological and geotechnical logging was conducted on whole core.

The logging program was designed around sets of cross sections oriented south–north, northwest–southeast, southwest–northeast, and west–east to view all holes optimally, given the variety of drill azimuths. Every drill hole was "best-fit" to a particular section. Each geologist was assigned sets of parallel sections, and was responsible for logging of holes assigned to that section, incorporating data from crossing holes logged by others, and ongoing cross-section interpretation of lithology, alteration, oxide/sulfide and sulfate zones. This plan ensured that 3-D interpretation proceeded concurrently with detailed logging.

All logging was digital, and was incorporated daily into the Maxwell DataShed™ database system. Data were recorded initially with Excel™ templates, and later with the Maxwell LogChief™ application using essentially the same structure. Both input methods used pick-lists and data validation rules to ensure consistency between loggers. Separate pages were designed to capture metadata, lithology, alteration, veins, sulfide-oxide zones, sulfide-oxide surfaces, minerals (sulfides, oxides, and limonite), sulfates, structure (contacts, fractures, veins, and faults with attitudes to core axis), magnetic susceptibility, and special data (samples collected for geochemistry, thin section examinations, the core library, skeleton core, etc.). Intensity of alteration phases was recorded using a numeric 1 to 4 scale (weak, moderate, strong, complete); abundance of veins and most other minerals were estimated in volume percent.

A site visit conducted by CNI (2016b) included the training of logging geologists to record appropriate geotechnical information. The geotechnical logging program consisted of collection of the data fields including; recovery; length of broken core; number of whole core pieces; length 2x core diameter (RQD length); length of core longer than 0.3m; longest piece; average hardness (ISRM scale); length of core <= hardness R2; average angle to core axis primary joint set; number of joints in primary joint set; joint separation; joint roughness; joint infilling; and joint weathering. These data fields provide information regarding the degree of fracturing and hardness (strength) of whole core pieces. These data are useful for the assessment of rock quality for analysis of slope stability, prediction of in-situ and run-of-mine fragmentation, and the identification of fault zones, to assist in the geologic interpretation. A tablet-based data entry program was developed by Fortuna using the Maxwell LogChief™ software. Data checks were implemented into this program to prevent entry of erroneous data.

Contract geologists were also trained in sampling of core for geotechnical laboratory strength testing, including sampling of anhydrite fractures and intact sticks of core.

## 10.6 Drill core recovery

Ground conditions are generally good at Lindero, with highly competent rock for the majority of the deposit resulting in 100 % core recovery. However, areas that are highly weathered or affected by faulting can result in lower recovery levels. Core recovery averages 95 % but can vary between 20 % and 100 % (Figure 10.2) in these areas. It should be noted that records of core recovery were unavailable for Rio Tinto holes LID-01 to LID-10 but observations of historical core suggest recovery levels are similar to those observed for other drill programs.

An analysis of core recovery versus gold grade did not identify any significant bias in gold grade that might have occurred due to a loss of material (Figure 10.2).

**Figure 10.2 Histogram showing core recovery**

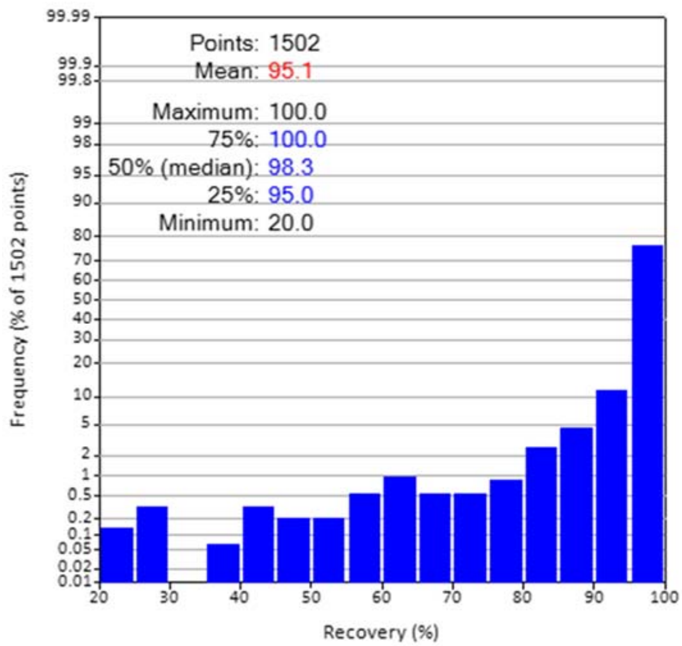



Figure prepared by Fortuna Silver Mines Inc., 2017

## 10.7 Extent of drilling

Drill holes are generally orientated perpendicular to the mineralization forming a radial pattern approximately 600 m across (Figure 10.1). In the eastern portion of the deposit, drill holes are orientated either perpendicular (azimuth 270°) or parallel (azimuth 190°) to the main mineralized body. Dips vary depending on the target and range from -50° to -89°, averaging -70°. The spacing between drill holes is between 40 m to 50 m at surface and tends to increase with depth. Drill hole depths vary according to the area and the purpose for which they were collared. The average depth is 300 m and the deepest hole reached a depth of 576 m.

## 10.8 Drill hole collar surveys

Drill hole collars were marked with PVC pipes introduced in the hole at surface and then cemented. Hole numbers were either sprayed with paint or engraved into the cement blocks.

Metal tags were sometimes used to mark the hole number, depth, orientation and dip. The only hole out of the 116 drilled by Rio Tinto and Goldrock through January 2008 that was not marked properly is LID-04, which is positioned to one side of the main access road to the Project.

During Rio Tinto's 2002 exploration campaign collars were not surveyed. Collars have subsequently been re-surveyed, as discussed below.

All holes drilled since 2005 as well as the 10 holes drilled during the 2002 campaign were surveyed by Servicios Topograficos with a differential GPS. Coordinates are projected on the WGS 84 Datum ellipsoid and calibrated according to the position of Geodetic point IGM N° PR-02-015, located a few kilometers from the Project. The results are available in geographic co-ordinates and in metric co-ordinates (UTM and Gauss Kruger), using the WGS 84 datum.

## 10.9 Downhole surveys

Downhole surveying is an important component of the drilling database and the lack of surveys impacts the reliability of the information collected, and the confidence level of the Mineral Resource estimates derived from the data.

During Rio Tinto's 2002 exploration drilling campaign, no downhole surveys were completed despite the fact that many of the holes extended beyond 300 m in depth.

Holes drilled during the first Goldrock campaign (October 2005 to February 2006) were not originally downhole surveyed. In 2005, Goldrock attempted to survey the holes but only a magnetic instrument was available, and measurements were of very poor quality.

In June 2006, GEC-Geophysical Exploration & Consulting S.A. (GEC) was contracted by Goldrock to perform borehole surveying services with a Reflex Maxibor II System 3™ Probe (Maxibor™), which is not affected by magnetism. This instrument was used from 2006 until early 2008. Downhole surveys were conducted both on the new holes and on the older holes drilled in the previous campaigns. Due to caving of the side-walls in holes LID-04, LID-08, LID-09 and LDH-15, no downhole surveys were performed.

In 2008, Goldrock detected that the Maxibor™ surveys showed an unacceptably large deviation in the drill holes, and a decision was made to re-survey all holes that showed a deviation of more than 5 %.

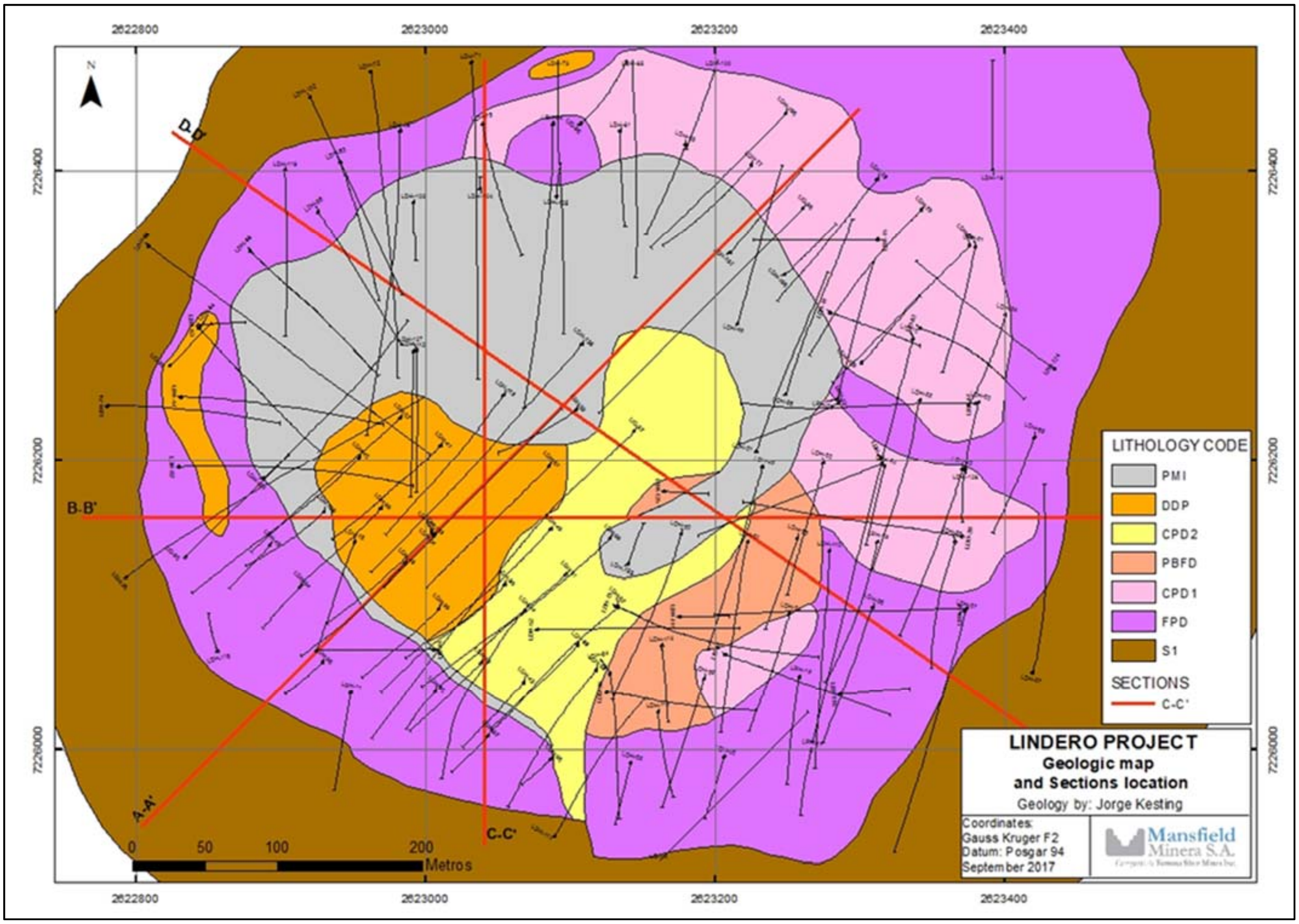
Comprobe Chile Ltd. (Comprobe) was contracted to re-survey the holes considered by Goldrock as having incorrect downhole deviations. A surface-recording gyroscopic instrument was used, and orientation and dip parameters were recorded every 10 m. Eighty percent of the holes were re-surveyed, with most of the holes showing little deviation and the maximum deviation recorded was 8°. This survey meets or exceeds industry standards.

For the 2016 drilling campaign, Fortuna retained the services of Construccion & Mineria S.A., based out of Mendoza, Argentina, to complete downhole surveys for each hole upon completion. Downhole surveys were conducted using Reflex™ gyroscopic equipment with readings taken at 5-m intervals.

## 10.10 Drill sections

Representative drill sections displaying the mineralized interpretation of the Lindero Deposit are displayed in Figures 10.4 to 10.7. A plan view showing the location of the sections is provided in Figure 10.3.

**Figure 10.3 Plan view of Lindero showing location of sectional interpretations**




Figures 10.3 to 10.7 prepared by Riedell and Mansfield, 2017

**Figure 10.4 Section displaying mineralization along section A**

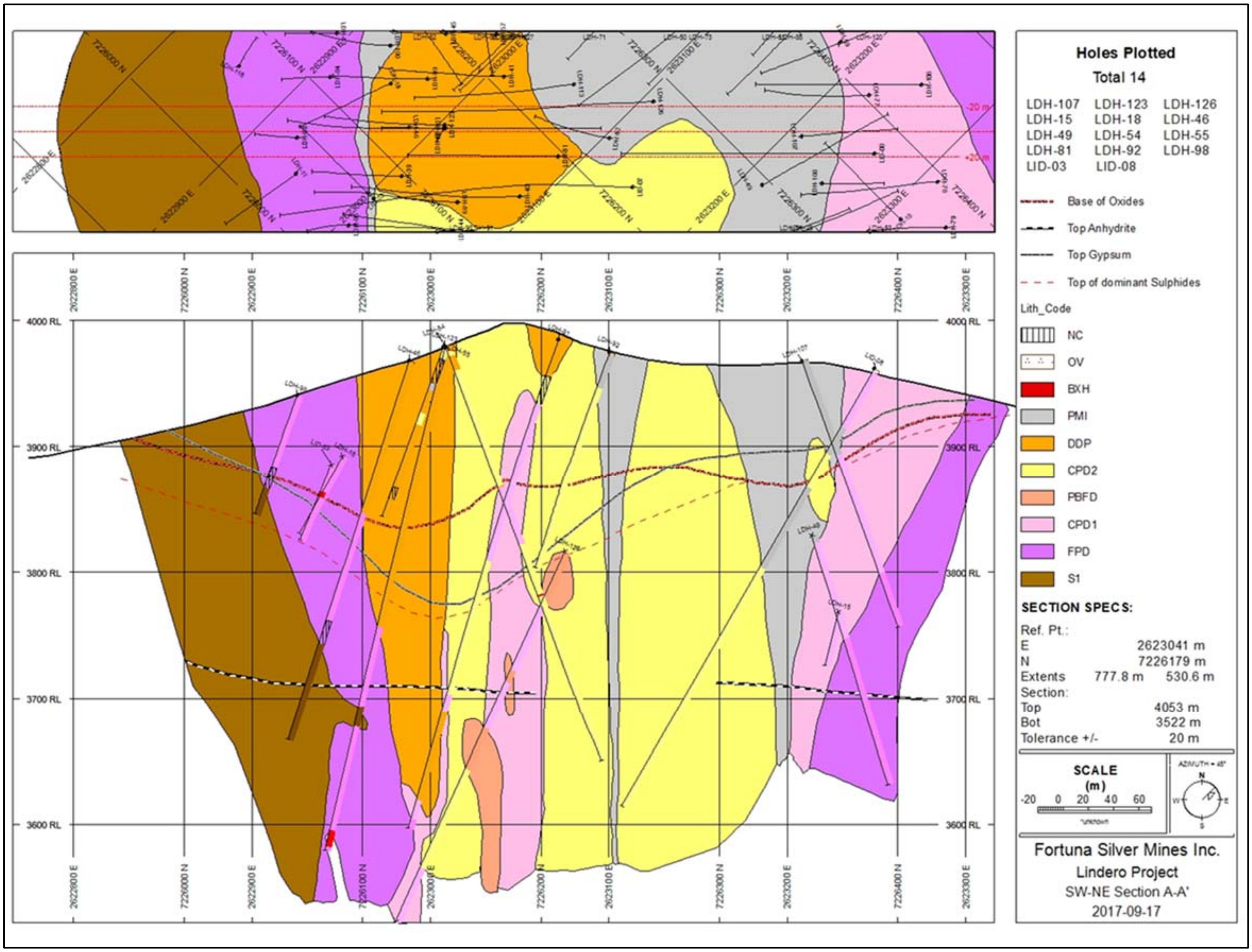


**Figure 10.5 Section displaying mineralization along section B**

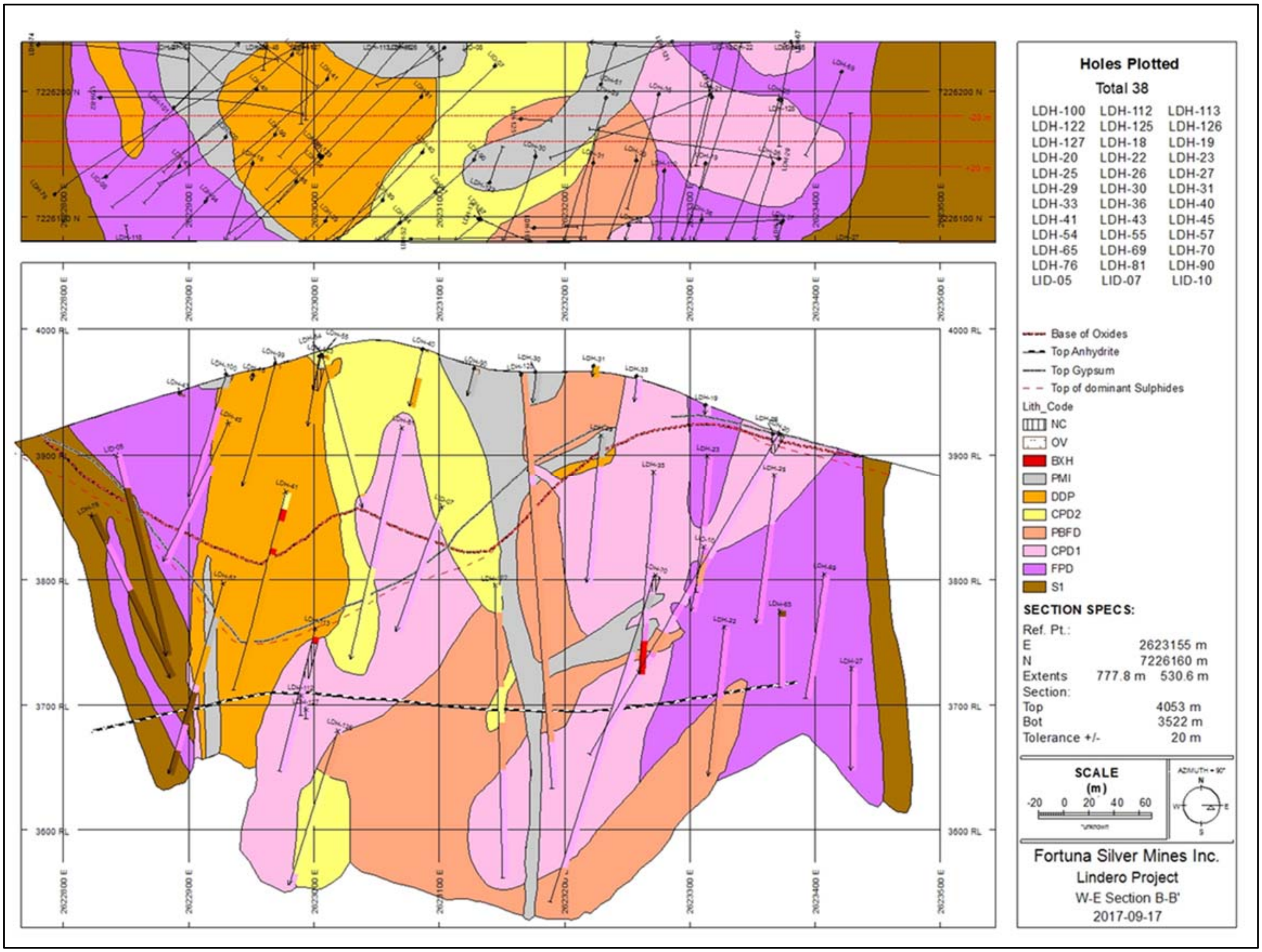


**Figure 10.6 Section displaying mineralization along section C**

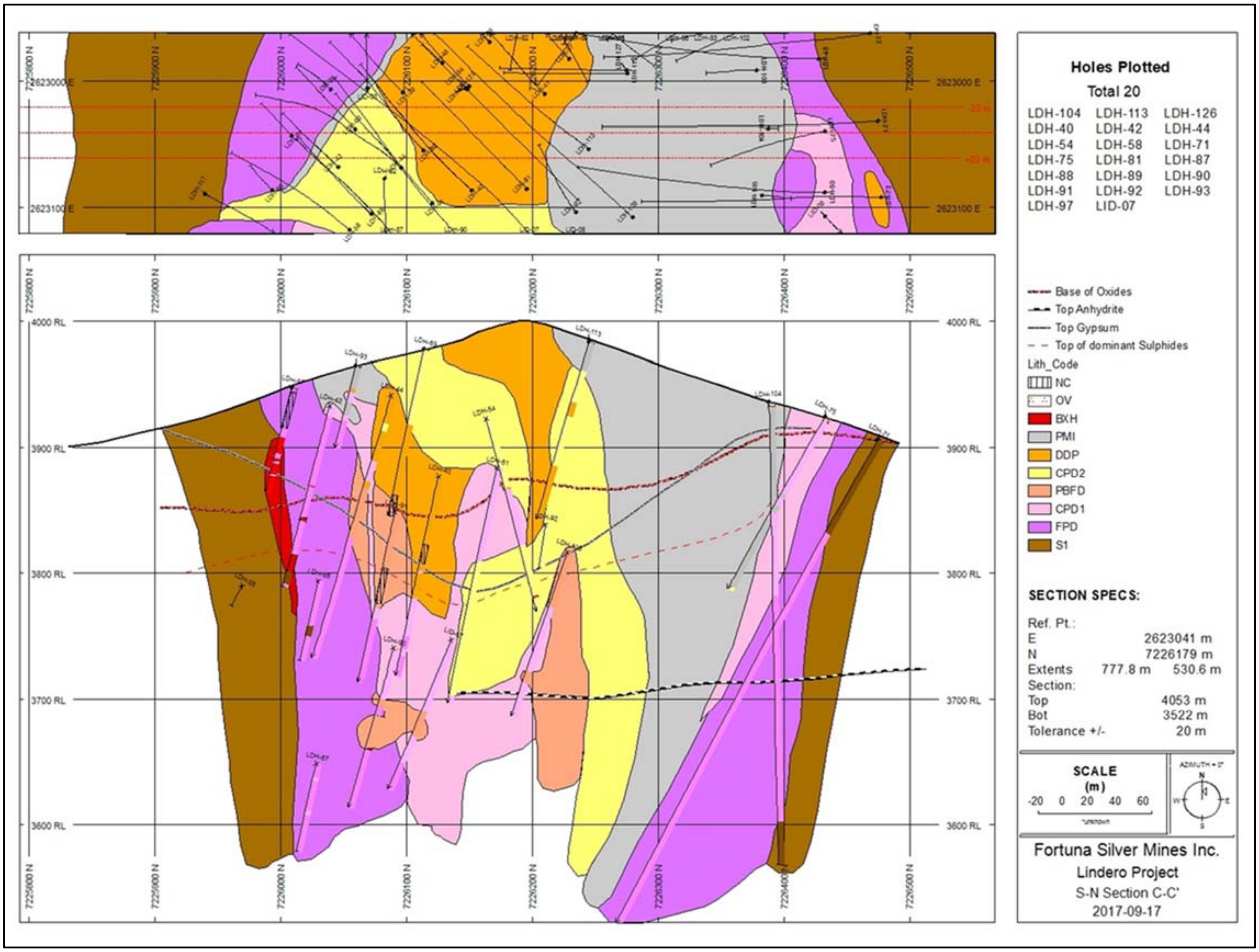
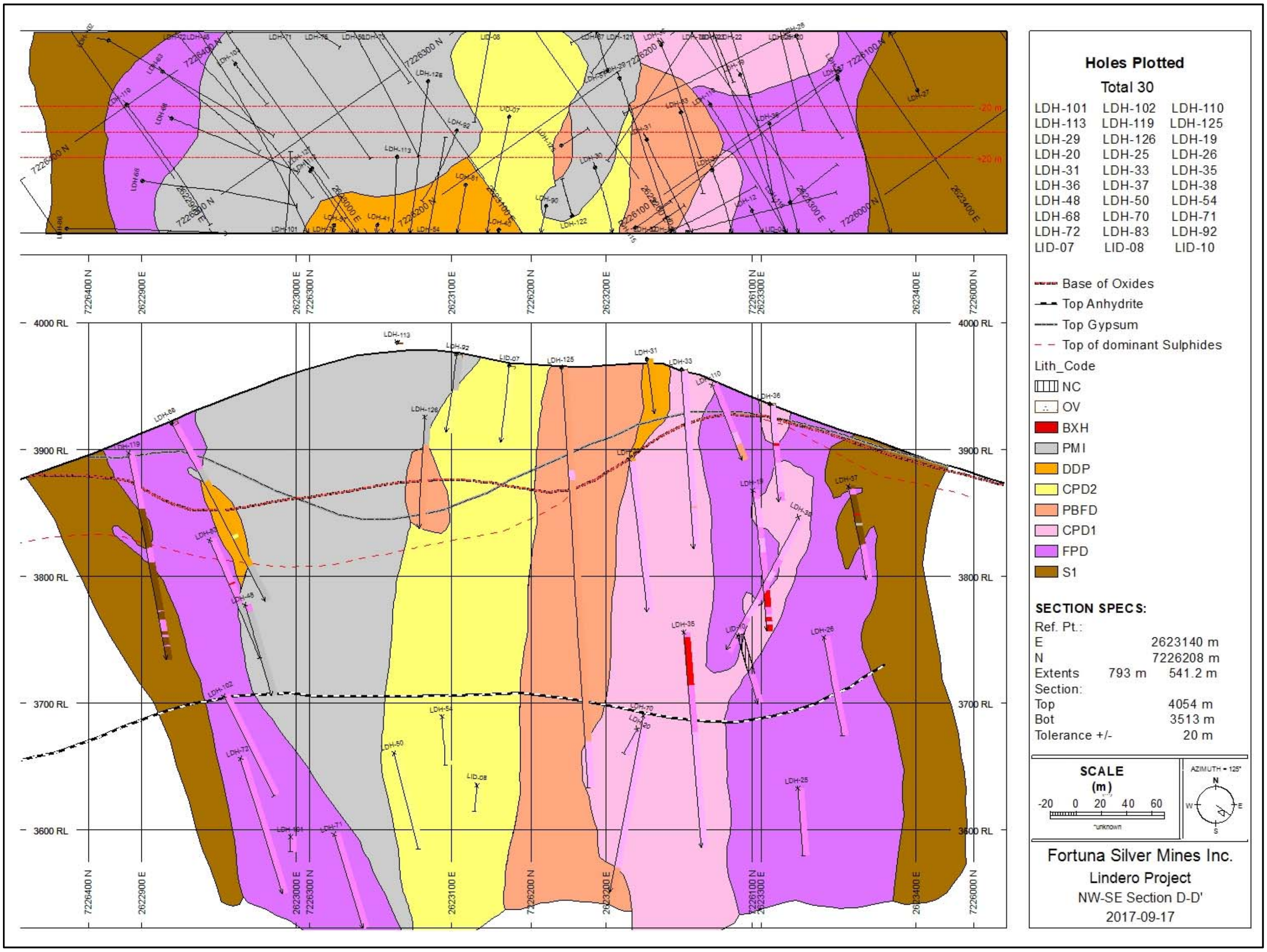


**Figure 10.7 Section displaying mineralization along section D**



## 10.11 Sample length versus true thickness

The Lindero Deposit is a gold-rich porphyry with low-grade mineralization permeating throughout the deposit, making the calculation of true thickness impossible as no definitive across strike direction exists. The mineralization appears annular in shape at surface due to the intrusion of barren to low-grade intrusive units into the core of the system, but this circular shape is not representative of true thickness.

## 10.12 Summary of drill intercepts

Table 10.5 provides a list of the drill hole intercepts at Lindero.

**Table 10.4 Selected Lindero drill hole intercepts**

| Drill hole | Easting | Northing | Elevation | Azimuth (°)** | Dip (°)** | From (m) | To (m) | Interval (m) | Au (g/t) | Cu (%) |
|---|---|---|---|---|---|---|---|---|---|---|
| LID-01 | 2623207.23 | 7226065.06 | 3961.54 | 110 | 70 | 6 | 62 | 56 | 0.43 | - |
| | | | | | | 122 | 316 | 194 | 0.62 | - |
| LID-02 | 2623342.12 | 7226290.70 | 3953.44 | 110 | 71 | 3 | 196 | 193 | 0.76 | - |
| LID-03 | 2623005.61 | 7226068.57 | 3963.59 | 270 | 60 | 3 | 159* | 156 | 0.96 | - |
| LID-04 | 2623126.57 | 7225887.79 | 3907.96 | 45 | 60 | 238 | 266 | 28 | 0.27 | - |
| | | | | | | 340 | 389 | 49 | 0.46 | - |
| LID-05 | 2622833.74 | 7226132.16 | 3931.51 | 45 | 71 | 38 | 300 | 262 | 0.64 | - |
| LID-06 | 2623107.09 | 7226432.50 | 3934.45 | 45 | 70 | 4 | 126 | 122 | 0.37 | - |
| LID-07 | 2623144.23 | 7226219.99 | 3966.10 | 225 | 61 | 152 | 562 | 410 | 0.53 | - |
| LID-08 | 2623261.09 | 7226374.37 | 3962.15 | 225 | 60 | 6 | 54 | 48 | 0.46 | - |
| LID-09 | 2622823.03 | 7226265.48 | 3926.58 | 45 | 60 | 40 | 52 | 12 | 0.36 | - |
| LID-10 | 2623336.93 | 7226283.12 | 3953.74 | 195 | 50 | 3 | 455* | 452 | 0.62 | - |
| LDH-11 | 2622948.17 | 7226038.82 | 3942.20 | 190 | 70 | 3.1 | 61 | 57.9 | 0.63 | 0.14 |
| LDH-12 | 2623258.47 | 7226049.53 | 3945.70 | 190 | 69 | 0 | 207 | 207 | 1.01 | 0.1 |
| LDH-13 | 2623131.42 | 7226098.46 | 3971.38 | 110 | 69 | 12 | 366 | 354 | 0.61 | 0.08 |
| LDH-14 | 2623279.85 | 7226301.46 | 3969.10 | 109 | 80 | 0 | 304 | 304 | 0.64 | 0.1 |
| LDH-15 | 2623312.62 | 7226352.27 | 3961.69 | 270 | 70 | 14 | 250.15* | 236.15 | 0.53 | 0.13 |
| LDH-16 | 2623391.96 | 7226400.46 | 3940.73 | 0 | 60 | No intervals of significance | | | | |
| LDH-17 | 2623266.55 | 7225998.85 | 3929.33 | 192 | 68 | 0 | 100 | 100 | 1.12 | 0.14 |
| LDH-18 | 2622950.84 | 7226142.57 | 3964.16 | 203 | 60 | 24 | 154.95* | 130.95 | 0.84 | 0.19 |
| LDH-19 | 2623312.38 | 7226142.74 | 3939.96 | 190 | 70 | 0 | 300.1* | 300.1 | 1.53 | 0.13 |
| LDH-20 | 2623370.85 | 7226146.58 | 3916.95 | 280 | 59 | 0 | 298.5* | 298.5 | 0.54 | 0.07 |
| LDH-21 | 2623379.99 | 7226347.03 | 3943.21 | 192 | 60 | 72 | 180* | 108 | 0.72 | 0.13 |
| LDH-22 | 2623341.70 | 7226241.30 | 3942.32 | 195 | 70 | 0 | 367.84* | 367.84 | 0.52 | 0.08 |
| LDH-23 | 2623317.58 | 7226195.19 | 3941.81 | 195 | 70 | 2.13 | 161.65* | 159.52 | 0.8 | 0.07 |
| LDH-24 | 2623401.05 | 7226299.97 | 3933.39 | 195 | 70 | 60 | 201.3 | 141.3 | 0.66 | 0.11 |
| LDH-25 | 2623371.07 | 7226193.31 | 3921.84 | 196 | 70 | 3.05 | 297 | 293.95 | 0.65 | 0.1 |
| LDH-26 | 2623366.13 | 7226142.52 | 3916.89 | 195 | 70 | 2 | 258.03 | 256.03 | 0.34 | 0.09 |
| LDH-27 | 2623419.67 | 7226052.50 | 3891.89 | 15 | 60 | 108 | 156 | 48 | 0.29 | 0.07 |
| LDH-28 | 2623248.76 | 7226245.27 | 3954.24 | 17 | 70 | 42 | 373.63 | 331.63 | 0.58 | 0.11 |
| LDH-29 | 2623232.95 | 7226194.88 | 3958.65 | 195 | 70 | 60 | 344 | 284 | 0.56 | 0.07 |
| LDH-30 | 2623176.67 | 7226148.28 | 3966.38 | 197 | 71 | 72 | 110 | 38 | 0.43 | 0.05 |
| | | | | | | 168 | 226 | 58 | 0.69 | 0.1 |
| | | | | | | 330 | 380.94* | 50.94 | 0.6 | 0.16 |
| LDH-31 | 2623222.72 | 7226142.93 | 3971.18 | 195 | 71 | 1.81 | 397.72 | 395.91 | 0.45 | 0.08 |
| LDH-32 | 2623192.43 | 7226048.25 | 3961.38 | 198 | 72 | 30 | 218 | 188 | 0.42 | 0.08 |
| LDH-33 | 2623257.16 | 7226145.05 | 3963.07 | 200 | 75 | 58 | 138 | 80 | 0.51 | 0.09 |
| | | | | | | 196 | 264 | 68 | 0.42 | 0.06 |
| LDH-34 | 2623251.26 | 7226093.45 | 3956.79 | 195 | 75 | 1.83 | 70 | 68.17 | 0.61 | 0.05 |
| | | | | | | 110 | 258 | | 0.39 | 0.06 |
| | | | | | | 282 | 355.93* | 73.93 | 0.47 | 0.1 |

| Drill hole | Easting | Northing | Elevation | Azimuth (°)** | Dip (°)** | From (m) | To (m) | Interval (m) | Au (g/t) | Cu (%) |
|---|---|---|---|---|---|---|---|---|---|---|
| LDH-35 | 2623274.94 | 7226197.57 | 3954.22 | 194 | 75 | 1.83 | 380.64* | 378.81 | 0.47 | 0.06 |
| LDH-36 | 2623309.17 | 7226097.76 | 3935.98 | 203 | 70 | 1.83 | 300.42* | 298.59 | 0.95 | 0.11 |
| LDH-37 | 2623373.18 | 7226095.41 | 3910.64 | 194 | 60 | 128 | 182 | 54 | 0.28 | 0.1 |
| LDH-38 | 2623374.26 | 7226097.14 | 3910.63 | 270 | 58 | 18 | 315.17* | 297.17 | 0.79 | 0.11 |
| LDH-39 | 2623008.88 | 7226096.83 | 3968.48 | 225 | 70 | 62 | 212.39* | 150.39 | 0.65 | 0.16 |
| LDH-40 | 2623086.40 | 7226151.58 | 3984.23 | 225 | 70 | 36 | 186 | 150 | 0.48 | 0.11 |
| | | | | | | 210 | 423.03 | 213.03 | 0.93 | 0.14 |
| LDH-41 | 2623010.29 | 7226209.66 | 3994.73 | 225 | 70 | 40 | 138 | 98 | 1.07 | 0.21 |
| | | | | | | 170 | 298 | 128 | 0.41 | 0.09 |
| LDH-42 | 2623067.94 | 7226045.49 | 3960.60 | 225 | 70 | 14 | 181.47* | 167.47 | 0.76 | 0.17 |
| LDH-43 | 2622892.36 | 7226140.42 | 3949.51 | 224 | 67 | 1.52 | 114.38* | 112.86 | 0.79 | 0.19 |
| LDH-44 | 2623068.45 | 7226095.46 | 3971.93 | 225 | 69 | 86 | 364 | 278 | 0.58 | 0.14 |
| LDH-45 | 2622953.91 | 7226201.38 | 3978.85 | 225 | 60 | 100 | 289.75* | 189.75 | 0.66 | 0.15 |
| LDH-46 | 2622985.48 | 7226128.23 | 3968.40 | 223 | 72 | 94 | 315.67* | 221.67 | 0.62 | 0.18 |
| LDH-47 | 2622830.73 | 7226243.45 | 3931.97 | 90 | 68 | 42 | 358 | 316 | 0.71 | 0.18 |
| LDH-48 | 2622982.34 | 7226427.39 | 3913.64 | 180 | 60 | 1.22 | 166 | 164.78 | 0.76 | 0.15 |
| LDH-49 | 2623215.79 | 7226293.90 | 3962.69 | 15 | 71 | 146 | 349.91* | 203.91 | 0.64 | 0.16 |
| LDH-50 | 2623088.10 | 7226432.12 | 3931.95 | 180 | 59 | 0.61 | 136 | 135.39 | 0.97 | 0.17 |
| LDH-51 | 2623228.75 | 7226205.49 | 3957.54 | 17 | 70 | 208 | 320 | 112 | 0.36 | 0.07 |
| | | | | | | 342 | 359.66* | 17.66 | 0.48 | 0.14 |
| LDH-52 | 2623077.05 | 7226082.31 | 3969.10 | 91 | 70 | 94 | 399.55* | 305.55 | 0.53 | 0.08 |
| LDH-53 | 2623284.79 | 7226242.24 | 3953.53 | 13 | 73 | 1.52 | 300 | 298.48 | 0.69 | 0.09 |
| LDH-54 | 2623006.43 | 7226148.23 | 3979.20 | 45 | 69 | 118 | 220 | 102 | 0.59 | 0.12 |
| LDH-55 | 2623005.55 | 7226146.91 | 3979.30 | 225 | 70 | 2.13 | 88 | 85.87 | 0.34 | 0.08 |
| | | | | | | 118 | 143.35* | 25.35 | 0.33 | 0.09 |
| LDH-56 | 2623124.83 | 7226038.61 | 3971.27 | 93 | 70 | 2.52 | 295.24* | 292.72 | 0.36 | 0.08 |
| LDH-57 | 2622982.23 | 7226229.04 | 3986.55 | 226 | 69 | 270 | 381.55* | 111.55 | 0.52 | 0.11 |
| LDH-58 | 2623117.76 | 7226054.31 | 3971.59 | 220 | 60 | 74 | 166 | 92 | 0.81 | 0.17 |
| LDH-59 | 2623179.60 | 7226417.98 | 3948.28 | 200 | 60 | 10 | 74 | 64 | 0.39 | 0.09 |
| LDH-60 | 2623206.40 | 7225994.53 | 3945.29 | 195 | 70 | 0 | 126 | 126 | 0.89 | 0.12 |
| LDH-61 | 2623134.35 | 7226427.59 | 3942.38 | 180 | 58 | 1.52 | 94 | 92.48 | 0.59 | 0.11 |
| LDH-62 | 2623141.90 | 7225990.47 | 3954.29 | 195 | 69 | 1.52 | 102 | 100.48 | 0.98 | 0.19 |
| LDH-63 | 2622842.50 | 7226292.80 | 3925.20 | 86 | 60 | 2.44 | 65.88 | 63.44 | 0.57 | 0.14 |
| LDH-64 | 2622843.27 | 7226290.65 | 3925.20 | 136 | 62 | 4.57 | 160 | 155.43 | 0.52 | 0.15 |
| LDH-65 | 2623382.40 | 7226239.78 | 3930.49 | 194 | 70 | 1.52 | 232.1 | 230.58 | 0.82 | 0.1 |
| LDH-66 | 2622878.55 | 7226344.21 | 3921.29 | 135 | 68 | 2.44 | 116 | 113.56 | 0.43 | 0.14 |
| | | | | | | 286 | 348.3* | 62.3 | 0.38 | 0.12 |
| LDH-67 | 2623380.27 | 7226239.29 | 3931.09 | 270 | 59 | 0.91 | 118 | 117.09 | 1.34 | 0.12 |
| | | | | | | 152 | 212 | 60 | 0.32 | 0.06 |
| | | | | | | 274 | 341.6 | 67.6 | 0.36 | 0.07 |
| LDH-68 | 2622925.56 | 7226371.35 | 3920.43 | 145 | 60 | No intervals of significance | | | | |
| LDH-69 | 2623420.98 | 7226215.52 | 3911.70 | 200 | 70 | 60 | 182 | 122 | 0.4 | 0.11 |
| LDH-70 | 2623315.87 | 7226197.43 | 3941.55 | 248 | 71 | 2.13 | 400 | 397.87 | 0.45 | 0.07 |
| LDH-71 | 2623031.54 | 7226474.43 | 3906.97 | 175 | 60 | 58 | 290 | 232 | 0.66 | 0.15 |
| LDH-72 | 2622961.84 | 7226468.15 | 3893.33 | 170 | 61 | 34 | 473.05* | 439.05 | 0.48 | 0.12 |
| LDH-73 | 2623091.85 | 7226477.08 | 3919.81 | 180 | 60 | 32 | 234 | 202 | 0.66 | 0.15 |
| LDH-74 | 2622779.90 | 7226237.14 | 3913.91 | 90 | 70 | 220 | 356 | 136 | 0.47 | 0.12 |
| LDH-75 | 2623039.44 | 7226432.37 | 3925.16 | 170 | 60 | 2.44 | 94 | 91.56 | 0.79 | 0.18 |
| LDH-76 | 2622792.75 | 7226118.07 | 3917.14 | 50 | 61 | 86 | 458 | 372 | 0.55 | 0.11 |
| LDH-77 | 2623225.30 | 7226404.13 | 3952.22 | 220 | 61 | 12 | 76 | 64 | 0.61 | 0.12 |
| LDH-78 | 2623312.33 | 7226394.03 | 3950.56 | 222 | 74 | 78 | 408 | 330 | 0.5 | 0.17 |
| LDH-79 | 2623342.68 | 7226372.94 | 3951.22 | 225 | 69 | 72 | 292 | 220 | 0.55 | 0.14 |
| LDH-80 | 2623376.02 | 7226348.19 | 3943.25 | 226 | 69 | 66 | 334 | 56 | 0.37 | 0.12 |
| LDH-81 | 2623085.48 | 7226195.56 | 3984.96 | 225 | 70 | 30 | 184 | 154 | 0.83 | 0.14 |
| | | | | | | 296 | 344 | 48 | 0.45 | 0.08 |
| | | | | | | 390 | 405.65* | 15.65 | 0.98 | 0.14 |
| LDH-82 | 2622829.25 | 7226194.86 | 3932.15 | 88 | 59 | 36 | 124 | 88 | 0.4 | 0.11 |
| | | | | | | 282 | 304 | 22 | 0.42 | 0.08 |

| Drill hole | Easting | Northing | Elevation | Azimuth (°)** | Dip (°)** | From (m) | To (m) | Interval (m) | Au (g/t) | Cu (%) |
|---|---|---|---|---|---|---|---|---|---|---|
| LDH-83 | 2622940.95 | 7226405.85 | 3908.58 | 160 | 60 | 6 | 38 | 32 | 0.55 | 0.11 |
| | | | | | | 112 | 164 | 52 | 0.84 | 0.16 |
| LDH-84 | 2623127.42 | 7226051.95 | 3971.55 | 170 | 74 | 6 | 334 | 328 | 0.57 | 0.11 |
| LDH-85 | 2623143.15 | 7226478.69 | 3934.81 | 180 | 60 | 20 | 222 | 202 | 0.55 | 0.12 |
| LDH-86 | 2622808.25 | 7226347.62 | 3896.09 | 126 | 59 | 118 | 452.01* | 307.01 | 0.71 | 0.14 |
| LDH-87 | 2623132.98 | 7226097.98 | 3971.41 | 228 | 72 | 94 | 344 | 250 | 0.63 | 0.12 |
| | | | | | | 364 | 390 | 26 | 0.44 | 0.13 |
| LDH-88 | 2623105.23 | 7226071.87 | 3969.00 | 220 | 70 | 86 | 251.62* | 165.62 | 0.75 | 0.17 |
| LDH-89 | 2623054.65 | 7226113.44 | 3978.55 | 224 | 72 | 62 | 338 | 276 | 0.64 | 0.14 |
| LDH-90 | 2623127.62 | 7226145.58 | 3969.27 | 226 | 70 | 96 | 160 | 64 | 0.31 | 0.06 |
| | | | | | | 180 | 420.59* | 240.59 | 0.62 | 0.11 |
| LDH-91 | 2623096.69 | 7226120.24 | 3973.48 | 230 | 69 | 34 | 64 | 30 | 0.35 | 0.11 |
| | | | | | | 86 | 361.42* | 275.42 | 0.73 | 0.14 |
| LDH-92 | 2623103.89 | 7226234.57 | 3975.16 | 233 | 69 | 150 | 178 | 28 | 0.29 | 0.1 |
| LDH-93 | 2623038.00 | 7226059.08 | 3965.61 | 225 | 70 | 40 | 202 | 162 | 0.78 | 0.16 |
| LDH-94 | 2622913.22 | 7226112.40 | 3950.09 | 225 | 75 | 0 | 144.87* | 144.87 | 0.59 | 0.15 |
| LDH-95 | 2623006.64 | 7226039.51 | 3957.54 | 225 | 70 | 0 | 100.04* | 100.04 | 0.69 | 0.17 |
| LDH-96 | 2623086.34 | 7225992.85 | 3946.36 | 225 | 69 | 2.17 | 75.03* | 72.86 | 0.56 | 0.12 |
| LDH-97 | 2623042.88 | 7226008.41 | 3948.44 | 226 | 70 | 3.66 | 102.17* | 98.51 | 0.41 | 0.13 |
| LDH-98 | 2622928.56 | 7226059.81 | 3940.76 | 225 | 70 | 0 | 100.04* | 100.04 | 0.64 | 0.14 |
| LDH-99 | 2622968.83 | 7226165.79 | 3973.04 | 226 | 69 | 4 | 72 | 68 | 0.53 | 0.13 |
| | | | | | | 142 | 211.97* | 69.97 | 0.83 | 0.24 |
| LDH-100 | 2622929.32 | 7226163.65 | 3964.78 | 224 | 68 | 30 | 181.47* | 151.47 | 0.82 | 0.2 |
| LDH-101 | 2622887.86 | 7226186.79 | 3954.94 | 45 | 69 | 16 | 62 | 46 | 0.5 | 0.17 |
| | | | | | | 370 | 400.46* | 30.46 | 0.34 | 0.07 |
| LDH-102 | 2622920.13 | 7226450.34 | 3887.07 | 155 | 60 | 126 | 298 | 172 | 0.68 | 0.15 |
| LDH-103 | 2622991.40 | 7226377.91 | 3930.50 | 180 | 85 | 42 | 358 | 316 | 0.72 | 0.13 |
| LDH-104 | 2623037.33 | 7226387.61 | 3936.69 | 0 | 88 | 86 | 367.89* | 281.89 | 0.63 | 0.15 |
| LDH-105 | 2623090.49 | 7226382.43 | 3946.32 | 0 | 87 | 78 | 378 | 300 | 0.83 | 0.17 |
| LDH-106 | 2623248.93 | 7226439.25 | 3943.00 | 225 | 58 | 38 | 186 | 148 | 0.61 | 0.11 |
| LDH-107 | 2623210.60 | 7226343.30 | 3968.04 | 40 | 70 | 80 | 224.33* | 144.33 | 0.44 | 0.13 |
| LDH-108 | 2623248.35 | 7226328.19 | 3972.80 | 46 | 68 | 20 | 129.84* | 109.84 | 0.46 | 0.11 |
| LDH-109 | 2623301.46 | 7226267.29 | 3957.58 | 45 | 69 | 1.21 | 171.9* | 170.69 | 1 | 0.14 |
| LDH-110 | 2623279.37 | 7226136.61 | 3953.44 | 181 | 52 | 3.35 | 239.47* | 236.12 | 0.93 | 0.08 |
| LDH-111 | 2623160.87 | 7226025.49 | 3968.94 | 176 | 68 | 32 | 165* | 133 | 1.09 | 0.14 |
| LDH-112 | 2622991.57 | 7226274.79 | 3972.64 | 181 | 69 | 232 | 299* | 67 | 0.49 | 0.09 |
| LDH-113 | 2623053.75 | 7226244.56 | 3984.30 | 220 | 69 | 128 | 226 | 98 | 0.57 | 0.11 |
| | | | | | | 246 | 359.96* | 113.96 | 0.58 | 0.09 |
| LDH-114 | 2623163.44 | 7226071.06 | 3974.94 | 178 | 80 | 56 | 325.22* | 293.22 | 0.42 | 0.08 |
| LDH-115 | 2623175.60 | 7226091.34 | 3975.21 | 90 | 70 | 1.21 | 99.96* | 98.75 | 0.67 | 0.09 |
| LDH-116 | 2623286.86 | 7226037.80 | 3935.81 | 86 | 70 | 1.52 | 142 | 140.48 | 0.8 | 0.11 |
| LDH-117 | 2623089.31 | 7225939.60 | 3925.40 | 32 | 71 | 6.5 | 50 | 43.5 | 0.35 | 0.14 |
| | | | | | | 95 | 432 | 337 | 0.38 | 0.12 |
| LDH-118 | 2622855.94 | 7226067.35 | 3925.95 | 345 | 85 | No intervals of significance | | | | |
| LDH-119 | 2622902.77 | 7226400.37 | 3903.64 | 180 | 73 | 148 | 346 | 198 | 0.5 | 0.12 |
| | | | | | | 383 | 406 | 23 | 0.37 | 0.13 |
| LDH-120 | 2623206.82 | 7226494.53 | 3934.78 | 196 | 75 | 103 | 159 | 54 | 0.37 | 0.14 |
| | | | | | | 236 | 314 | 78 | 0.38 | 0.12 |
| LDH-121 | 2623285.59 | 7226239.51 | 3952.99 | 247 | 79 | 0 | 160 | 160 | 0.47 | 0.06 |
| | | | | | | 217 | 377 | 160 | 0.52 | 0.08 |
| LDH-122 | 2623140.02 | 7226127.58 | 3967.90 | 24 | 85 | 335 | 387 | 52 | 0.36 | 0.07 |
| LDH-123 | 2623004.17 | 7226149.57 | 3978.75 | 225 | 76 | 0 | 407 | 407 | 0.61 | 0.12 |
| LDH-124 | 2623434.96 | 7226262.78 | 3915.90 | 305 | 67 | 130 | 313 | 183 | 0.34 | 0.09 |
| LDH-125 | 2623165.11 | 7226177.73 | 3964.60 | 90 | 85 | No intervals of significance | | | | |
| LDH-126 | 2623107.87 | 7226279.55 | 3969.94 | 222 | 65 | 215.4 | 321 | 105.6 | 0.56 | 0.13 |
| | | | | | | 444 | 515.58* | 71.58 | 0.69 | 0.08 |
| LDH-127 | 2622993.78 | 7226275.47 | 3972.12 | 180 | 70 | 205 | 300 | 95 | 0.71 | 0.12 |
| LDH-128 | 2623372.97 | 7226192.79 | 3921.52 | 195 | 69 | 3 | 297 | 294 | 0.64 | 0.08 |

| Drill hole | Easting | Northing | Elevation | Azimuth (°)** | Dip (°)** | From (m) | To (m) | Interval (m) | Au (g/t) | Cu (%) |
|---|---|---|---|---|---|---|---|---|---|---|
| CON-01 | 2623939.28 | 7225801.02 | 3828.69 | 0 | 60 | No intervals of significance | | | | |
| CON-02 | 2623540.00 | 7226012.65 | 3854.72 | 250 | 60 | No intervals of significance | | | | |
| CON-03 | 2623019.43 | 7225780.30 | 3893.70 | 290 | 60 | No intervals of significance | | | | |
| CON-04 | 2624147.09 | 7226481.24 | 3756.97 | 270 | 60 | No intervals of significance | | | | |
| CON-05 | 2622638.11 | 7226895.74 | 3755.13 | 140 | 60 | No intervals of significance | | | | |
| LGT-01 | 2623354.03 | 7226431.23 | 3937.02 | 0 | 90 | No intervals of significance | | | | |
| LGT-02 | 2623353.57 | 7225982.54 | 3900.72 | 0 | 90 | 188 | 201.47 | 13.47 | 0.48 | 0.15 |
| LGT-03 | 2622890.77 | 7225972.41 | 3916.46 | 0 | 90 | No intervals of significance | | | | |
| LGT-04 | 2623097.20 | 7226516.22 | 3906.47 | 0 | 90 | 40 | 60 | 20 | 0.26 | 0.13 |
| LGT-05 | 2623151.30 | 7226228.32 | 3965.33 | 0 | 90 | No intervals of significance | | | | |
| LGT-06 | 2622845.72 | 7226034.34 | 3919.62 | 225 | 77 | No intervals of significance | | | | |
| LGT-07 | 2622853.98 | 7226357.89 | 3908.27 | 315 | 75 | 0 | 52 | 52 | 0.27 | - |
| LGT-08 | 2623087.59 | 7226474.29 | 3919.71 | 0 | 75 | No intervals of significance | | | | |
| LGT-09 | 2623388.80 | 7226340.20 | 3940.44 | 60 | 76 | No intervals of significance | | | | |
| LGT-10 | 2623052.53 | 7225936.84 | 3924.17 | 180 | 76 | No intervals of significance | | | | |
| LGT-11 | 2623367.88 | 7226049.68 | 3909.76 | 125 | 75 | No intervals of significance | | | | |
| * Bottomed in mineralization | | | | | | | | | | |
| ** Azimuth and dip values taken at collar location | | | | | | | | | | |

## 10.13 Comment on Section 10

The QP has the following observations and conclusions regarding drilling under Goldrock and Fortuna management in the period 2002–2016:

- Data were collected using industry standard practices
- Drill orientations are appropriate to the orientation of the mineralization
- Core logging meets industry standards for exploration of porphyry-style deposits
- Geotechnical logging is sufficient to support Mineral Resource estimation. The data for the Lindero Deposit have been reviewed by AMEC and CNI with regards to suitability to support detailed mine planning (*it should be noted that additional geotechnical information was collected and reviewed by CNI in 2016 and they also concluded it was suitable for pit design purposes*)
- Collar surveys have been performed using industry-standard instrumentation. Uncertainty in collar locations of Lindero Deposit drill holes, surveyed using compass and tape, have been incorporated into subsequent resource confidence category classification
- Downhole surveys performed during the drill programs have been performed using industry-standard instrumentation. Uncertainties in the downhole locations of the Lindero Deposit drill holes have been incorporated into subsequent resource confidence category classification

# 11 Sample Preparation, Analyses, and Security

Drilling in 2002 was performed by Rio Tinto, and included 10 holes totaling 3,279.58 m of core at Lindero sampled at intervals of 2 m. Sampling was performed by Rio Tinto personnel. In total, 1,628 samples were collected from Lindero.

Goldrock drilled 132 diamond drill holes at Lindero between 2005 and 2010 totaling 34,618.56 m of core. Samples were collected on 2 m intervals, irrespective of lithology, representing approximately 8 kg of rock for HQ core size and 4 kg for NQ core size. All sampling was completed by Goldrock personnel. The majority of these holes (116) were drilled as part of the exploration program between 2005 and 2008, with an additional 5 holes drilled in 2008 and 11 holes drilled in 2010 for geotechnical investigations.

Trenches excavated at Lindero by Goldrock in 2001 were sampled at 2 m and 5 m intervals either by hand on a continuous chip basis or from channels cut by a diamond saw; during the 2005 to 2008 period, trenches were channel sampled with a rock saw every 2 m.

Fortuna drilled 12 diamond drill holes at Lindero between 2016 and 2017 totaling 4,461.58 m of core. Samples were collected on 2 m intervals for all recovered core. Eight holes were drilled to provide fresh core for metallurgical testwork, two holes were drilled to improve understanding in geologically complex areas and two holes were drilled as twins of previously drilled holes to validate reported grades.

## 11.1 Sample preparation prior to dispatch of samples

### 11.1.1 Core samples

Drill core is laid out for sampling and logging at the core logging facility at the camp. Sample intervals are marked on the core and depths recorded on the appropriate box.

A geologist is responsible for determining and marking the drill core intervals to be sampled, selecting them based on geological and structural logging. The first sample of each core hole is marked from the start of core recovery to an acceptable recovery interval (generally 2 m to 4 m from the surface). The remainder of the core is marked at 2 m intervals unless there is a change in the geology. The numbering sequence used allows for the insertion of QAQC samples.

Once the core was marked, geotechnical logging was performed and then all cores were systematically photographed within the box. The core is then halved using a diamond saw. The geologist carefully determines the line of cutting, in such a way that both halves of the core are representative. The core cutting process is performed in a separate building adjacent to the core logging facilities. Water used to cool the saw is not re-circulated but stored in a tank to allow any fines to settle before final disposal. Samples weight between 4 kg and 8 kg depending on the diameter of the core. Field duplicates were prepared from quarter cores. Samples were collected in plastic bags and labelled. The remaining half or quarter cores were stored in the Lindero camp core storage facilities in wooden core boxes. Fortuna's policy is for field duplicates to be represented by half core not quarter core, and this policy will be implemented for future field duplicate sampling.

### 11.1.2 Trench samples

The trenches were excavated, and any loose material cleaned off the bedrock. A rock saw was used to cut a 2 cm to 3 cm wide channel along the exposed bedrock. Samples were approximately 4 kg (based on volume calculations using an assumed bulk density of 2.5 g/cm$^3$).

Field duplicates were collected by taking a second channel sample adjacent to the original sample. Metal tags showing the trench number and sample number were attached to the bedrock exposure.

### 11.1.3 Bulk density determination

Goldrock conducted bulk density measurements on 493 samples. Density samples are routinely collected by Mansfield from drill core on approximate 10-m intervals. Samples consist of pieces of core approximately 7 cm in length and weighing between 93 g and 408 g.

Bulk density measurements were conducted on different intrusive rocks representing mineralized and unmineralized, oxide and sulfide mineralization, and from sedimentary rocks. The original method used was the American Standard Testing Materials (ASTM) Method C97. This method involved weighing a dried sample of core, immersing it in water to fill pore spaces, and then re- weighing the core in both air and water. However, the method can overestimate bulk density when the rock is porous so to confirm the initial results Goldrock re-assessed 840 of the samples using a wax-coating water immersion method (ASTM C914). Goldrock found the initial measurements to be reliable, except for oxidized pieces of rock, which gave different results when compared to the first method.

Fortuna conducted a verification of the density measurements obtained by Goldrock with the re-submission of 30 of the previously-taken density samples. Density tests for these samples were performed at the ALS Global laboratory in Vancouver using the OA-GRA08 methodology. This test consists of coating the core sample in paraffin wax, measuring the sample weight in air then suspending the sample in water and measuring the weight again. The bulk density is calculated using the following equation:

$$\text{Bulk density} = \frac{\Delta}{B - C - [(B - A) / Dwax]}$$

Where

A = weight of sample in air

B = weight of waxed sample in air

C = weight of waxed sample suspended in water

Dwax = density of wax

Results of this analysis are included in Section 12.6 of this Report.

## 11.2 Dispatch of samples, sample preparation, assaying and analytical procedures

### 11.2.1 Sample dispatch

Following the sawing of drill core or the collection of trench samples (described above) samples were placed in polyethylene sample bags with a sample tag detailing a unique sample identifier. The same sample identifier was marked on the outside of the bag and it was sealed with a cable tie. Secured sample bags were then placed in rice sacks labeled with the company name, number of samples contained in the sack and the sample number sequence. The rice sacks with the samples were then sealed with double cable ties and stored in a secure, dry and clean location. The rice sacks were subsequently transported by authorized company personnel to Salta prior to commercial transportation by truck to the selected sample preparation facility.

### 11.2.2 Sample preparation and assaying

**Rio Tinto (2002)**

All exploration core samples were sent to the ALS Chemex sample preparation facility in Mendoza, Argentina. Upon arrival a notification of sample reception was transmitted and the samples entered into the laboratory sample management system. Following drying at 55°C, the samples are weighed and the entire sample (4 to 8 kg) crushed using a two-stage method, first with a jaw crusher to 1 cm, and then by cone crusher so that a minimum of 70 % passed a 10 mesh sieve. The entire crushed sample was then pulverized to a minimum of 95 % passing 80 mesh. The pulverized samples were then split using a riffle splitter to generate a 300 g subsample and further pulverized so that a minimum of 95 % passed 150 mesh. This subsample was then split again using a riffle splitter to generate three 100 g samples. These 100 g pulp samples were shipped by commercial air freight to ALS Chemex's analytical facility in Vancouver, British Columbia for analysis.

In Vancouver, samples were assayed for gold using fire assay with atomic absorption finish (FA-AA) and for 35 elements using inductively coupled plasma (ICP) spectrophotometry. Copper values above 1 % were re-run with atomic absorption. The sample charge for fire assay was 30 g. Assay results were reportedly provided in electronic format and in hard copy, although Mansfield does not have any copies of these original certificates.

Rio Tinto included pulp duplicates, coarse reject duplicates, blanks and standard reference materials (SRMs) in the drill sample submissions to the laboratory to monitor assay accuracy and precision. Samples were sent for check analyses to Alex Stewart laboratory in La Serena or the Bondar Clegg laboratory in Canada. A small number of samples were sent to ACME for metallic screen assays. Coarse reject and pulp duplicates were reportedly taken at an approximate rate of every 30 samples, while standards and blanks were inserted by the laboratory at a rate of 1 in every 20 samples (Fuchter and Rennie, 2013). Analysis of Rio Tinto's QAQC results are described in Section 11.4.

**Goldrock (2005 to 2010)**

Core and trench samples taken by Goldrock between 2005 and 2008 were placed in labeled plastic bags with plastic tags and then packaged into larger rice sacks before being sent to the ACME laboratory (ISO 9001:2000 certified) in Mendoza, Argentina for preparation and then to the Santiago laboratory in Chile for assaying. These included holes LDH-11 to LDH-116 and geotechnical holes LGT-01 to LGT-05. A list of samples and sacks was prepared for each shipment and a copy of the sample submittal form sent for filing to Goldrock's Salta office.

Sample preparation involved drying at 60°C, before the entire sample (4 to 8 kg) was crushed using a jaw crusher so that a minimum of 70 % passed a 10 mesh sieve. The crushed sample was then split using a riffle splitter to generate a 500 g subsample which was homogenized prior to being pulverized so that a minimum of 95 % passed a 200 mesh sieve. Pulverized samples were then split again using a riffle splitter to generate a two 100 g samples and one 300 g sample. The two 100 g pulp samples were shipped by commercial air freight to analytical facilities in Santiago, Chile for analysis. The 300 g sample was retained as a pulp reject by Goldrock at the storage facility at the Lindero Project.

All samples collected by Goldrock were assayed for gold using a 30 g FA-AA finish and a second aliquot was selected for copper analysis using aqua regia digestion and AA analyses. For the drill samples only, a full suite of trace elements was analyzed using an aqua regia digestion followed by ICP analysis. Assay results and certificates were reported electronically by e-mail. ACME used internationally-accepted analytical techniques and SRMs at all levels of the sample preparation and sample assay procedure to monitor quality control. Additionally, the laboratory checked 12 % of all assays with the submission of preparation (coarse reject) and laboratory (pulp) duplicates.

Goldrock included blanks, SRMs, quarter core field duplicates, and check assays in the drill trench sampling submissions to monitor assay accuracy and precision. Batches of samples were sent for duplicate analyses to Alex Stewart laboratory in Mendoza, Argentina or to La Serena, Chile. Additional check assays were sent to ALS Chemex in Mendoza, Argentina.

Core samples for holes drilled in 2010 including geotechnical holes LGT-06 to LGT-11 and condemnation holes CON-01 to CON-05 were prepared in the same manner as described above but were sent to the Alex Stewart laboratory in Mendoza, Argentina albeit with the analysis for gold using a 50 g FA-AA finish. Goldrock also submitted blanks, SRMs, quarter core field duplicates, and check assays with these samples. Batches of samples were sent for duplicate analyses to ALS Chemex in Mendoza, Argentina. Analysis of Goldrock's QAQC results are described in Section 11.4.

**Fortuna (2016)**

All exploration core samples were sent to the ALS Global sample preparation facility in Mendoza, Argentina. Upon arrival, a notification of sample reception was transmitted to Fortuna and the samples entered into the laboratory sample management system. Following drying at 55°C, the samples were weighed and the entire sample (4 to 8 kg) crushed using a two-stage method, first with a jaw crusher to 1 cm, and then by cone crusher so that a minimum of 70 % passed a 10 mesh sieve. The entire crushed sample was then pulverized to a minimum of 95 % passing an 80 mesh sieve. Pulverized samples were then split using a riffle splitter to generate a 300 g subsample and further pulverized so that a minimum of 95 % passed a 150 mesh sieve. This subsample was then split again using a riffle splitter to generate three 100 g samples. These 100 g pulp samples were shipped by commercial air freight to ALS Global's analytical facility in Vancouver, British Columbia for analysis.

In Vancouver, samples were assayed for gold using FA-AA and for 35 elements using a four-acid digestion prior to ICP spectrophotometry. Copper values more than 1 % were re-run with a four-acid digestion prior to a ICP-atomic emission spectroscopy (AES) or AA finish. Soluble copper was assayed using a cyanide leach atomic absorption finish. Assay results were provided in electronic format and in hard copy.

Fortuna submitted field duplicates, reject assays (coarse rejects), duplicate assays (pulps), check assays (pulps sent to an umpire laboratory) blanks and SRMs in the drill sample submissions to the laboratory to monitor assay accuracy and precision. Samples were sent for check analyses to Bureau Veritas laboratory in Vancouver, Canada for assaying. Analysis of Fortuna's QAQC results are described in Section 11.4.

## 11.3 Laboratory accreditation

The ALS Chemex laboratory used by Rio Tinto, Goldrock and Fortuna (renamed to ALS Global) is an independent, privately-owned analytical laboratory group. The Vancouver laboratory holds ISO 17025 accreditation. The Argentine laboratory holds ISO 9001:2000 certification.

ACME, used by both Rio Tinto and Goldrock was an independent, privately-owned American analytical laboratory group, founded in Canada, with two laboratories in North America, and five affiliate laboratories in South America: one each in Argentina, Peru, Ecuador, Brazil and Chile. ACME was acquired by Bureau Veritas in 2012. The Argentine and Chilean laboratories are ISO 9001:2000 certified.

Alex Stewart, used by both Rio Tinto and Goldrock is an independent, privately-owned laboratory, part of the Stewart Group. It is not known what certification the Argentina laboratory held at the time of the analytical work.

Bondar Clegg, used by Rio Tinto, was acquired by ALS Chemex in 2001, and laboratory facilities were merged where there were overlaps. The Chilean laboratory of ALS Chemex is ISO 9001:2000 certified.

The Bureau Veritas laboratory used by Fortuna as an umpire laboratory is an independent, privately-owned analytical laboratory group. The Vancouver laboratory holds ISO 9001:2000, ISO 14001:2004, and OHSAS 18001:2007 certification.

## 11.4 Sample security and chain of custody

Sample collection and transportation of drill core and trench samples is the responsibility of Mansfield's exploration geology department.

Exploration core boxes are sealed and carefully transported to the core logging facilities located adjacent to the exploration camp where there is sufficient room to lay out and examine the core. Once logging and sampling were completed, the core was transferred to a temporary storage facility at the camp. The onsite storage facility is dry and well illuminated. Core is stored chronologically and at the Report effective date was being transferred to metal racks for additional security. The core will be transported to a larger permanent storage facility when construction of the mine camp is completed.

Unfortunately, coarse rejects from the Goldrock drill program were stored outside for several years resulting in the degradation of many of the plastic bags containing the samples. The coarse rejects were assessed in 2017 with many of the samples having to be disposed of due to poor condition. The remaining coarse rejects were transferred for storage in a separate warehouse on site. Pulps from the exploration and infill drill programs are stored in a secure and dry pulp storage facility.

Samples are retained in accordance with the Fortuna corporate sample retention policy. All drill core and coarse rejects and pulps from drill core are stored for the LOM.

## 11.5 Quality control measures

The implementation of a QAQC program is current industry best practice and involves establishing appropriate procedures and the routine insertion of SRMs, blanks, and duplicates to monitor the sampling, sample preparation and analytical process. Analysis of QC data is made to assess the reliability of sample assay data and the confidence in the data used for the estimation.

### 11.5.1 Rio Tinto

QAQC procedures and protocols used by Rio Tinto in the sampling of the first drilling campaign are described in a report compiled by Rio Tinto (Ruiz, et al., 2003) for Goldrock.

The Rio Tinto report states that all samples taken at Lindero, including inserted QC samples, were sent first to the ALS Chemex laboratory in Mendoza for crushing, pulverizing and splitting. For drill holes LID-01 to LID-06 preparation duplicates were not inserted. For holes LID-07 to LID-10, one preparation duplicate (taken after cone crushing to 75 % passing 10 mesh) and one laboratory duplicate (taken from a split at the end of pulverization, 95 % passing 150 mesh) were inserted in every 30 samples with the objective of checking for splitting or laboratory errors. Samples were randomized. Independent checks assays were performed using Alex Stewart in Chile (31), and Bondar Clegg in Canada (112). Table 11.1 details the total number of QC samples submitted as a percentage of core samples during the Rio Tinto campaign.

**Table 11.1 Summary of QC samples – Rio Tinto campaign**

| Sample Type | Number | Insertion Rate |
|---|---|---|
| Core samples | 1,628 | - |
| SRMs | 72 | 4.4 % |
| Blanks | 71 | 4.3 % |
| Preparation Duplicates | 22 | 1.3 % |
| Laboratory Duplicates | 71 | 4.3 % |
| Checks | 143 | 8.8 % |

### 11.5.2 Goldrock

QAQC procedures were implemented throughout the drill programs, and included SRMs, blanks, field duplicates and check samples. Drill holes LDH-11 to LDH-116 were submitted to ACME laboratory in Santiago, Chile during the drilling campaigns between 2005 and 2008. The geotechnical holes LGT-01 to LGT-05 were also sent for assaying to the ACME laboratory in 2008. However, holes LGT-06 to LGT-11 drilled in 2010 were analyzed at the Alex Stewart laboratory in Mendoza, Argentina, along with the condemnation holes CON-01 to CON-05. The condemnation holes have not been considered in the below analysis as they encountered no mineralization. Table 11.2 details the total number of QC samples submitted as a percentage of core samples during all the Goldrock campaigns.

**Table 11.2 Summary of QC samples – Goldrock campaigns**

| Sample Type | Holes LDH-11 to LDH-116 | | Holes LGT-01 to LGT-11 | |
|---|---|---|---|---|
| | Number | Insertion Rate | Number | Insertion Rate |
| Core samples | 14,821 | - | 1,716 | - |
| SRMs | 673 | 4.5 % | 101 | 5.9 % |

| Sample Type | Holes LDH-11 to LDH-116 | | Holes LGT-01 to LGT-11 | |
|---|---|---|---|---|
| | Number | Insertion Rate | Number | Insertion Rate |
| Blanks | 664 | 4.5 % | 101 | 5.9 % |
| Field Duplicates | 669 | 4.5 % | 101 | 5.9 % |
| Check Assay (Alex Stewart) | 1,660 | 11.2 % | - | - |
| Check Assay (ALS Chemex) | 1,660 | 11.2 % | 81 | 4.7 % |

Trench samples from the 2005 to 2006 exploration campaign were submitted along with blanks, field duplicates, and SRMs inserted in each batch of 30 samples. No check assay analyses were performed on the original trench sample assays. Trench samples have not been used in the 2017 resource update and therefore the QC results are not reported here.

### 11.5.3 Fortuna

Fortuna implemented a full QAQC program to monitor the sampling, sample preparation and analytical process. For the 2016 drilling campaign in accordance with its companywide procedures. The program involved the routine insertion of SRMs, blanks, and duplicates. Table 11.3 details the total number of QC samples submitted as a percentage of core samples during the Fortuna campaign.

**Table 11.3 Summary of QC samples – Fortuna campaign**

| Sample Type | Number | Insertion Rate |
|---|---|---|
| Core samples | 2,269 | - |
| SRMs | 126 | 5.6 % |
| Blanks | 130 | 5.7 % |
| Field Duplicates | 124 | 5.5 % |
| Preparation Duplicates | 123 | 5.4 % |
| Laboratory Duplicates | 123 | 5.4 % |
| Duplicate Assay | 55 | 2.4 % |
| Reject Assay | 57 | 2.5 % |
| Check Assay (Bureau Veritas) | 56 | 2.5 % |

### 11.5.4 Standard reference materials

Standard reference materials are samples that are used to measure the accuracy of analytical processes and are composed of material that has been thoroughly analyzed to accurately determine its grade within known error limits. SRMs are inserted by the geologist into the sample stream, and the expected value is concealed from the laboratory, even though the laboratory will inevitably know that the sample is a SRM of some sort. By comparing the results of a laboratory's analysis of a SRM to its certified value, the accuracy of the result is monitored.

SRM results detailed in this Report are presented in a text and tabular form; however, results were assessed using time series graphs to identify trends or biases.

**Rio Tinto (2002 to 2003)**

Rio Tinto used three SRMs named R1, R2, and SOC with recommended values of 0.176 g/t Au, 1.238 g/t Au, and 0.215 g/t Au respectively. No documentation is available describing the origin or certification of the SRMs.

This analysis focuses on the submission of 72 standards with 1,628 samples (submission rate of 1 in 23 samples) between 2002 and 2003 to the ALS Chemex laboratory.

Although the best value and standard deviations for the standards are unknown it is possible to surmise from the reported results that only two results lay significantly outside an expected range representing a pass rate of 97 %. However, the accuracy of these samples cannot be verified and therefore this has been considered during resource classification.

**Goldrock (2005 to 2010)**

Three different groups of SRMs were used for the Lindero Deposit. The first group consisted of three SRMs of different grades which are representative of the mineralization at the Lindero Deposit, and were used during the sampling of holes LDH-11 to LDH-33. These SRMs were certified by ACME in Santiago, Chile.

Sampling of drill holes LDH-34 to LDH-39 used four SRMs obtained from Silex (a subsidiary company of Apex Silver); however, no documentation exists of the best values.

The final group of SRMs was certified by Alex Stewart in 2016 in Mendoza and consisted of four different SRMs that were used during the sampling of drill holes LDH-40 to LDH-116 as well as the geotechnical holes LGT-01 to LGT-11.

A summary of the performance of each SRM is shown in Table 11.4. A fail as described in the tables below is any value being >± 3 standard deviations from the best value.

**Table 11.4 SRM results submitted by Goldrock 2005-2010**

| Standard | No. Submitted | Best Value (g/t) | Mean Value (g/t) | Bias (%) | No. fails | Pass rate (%) |
|---|---|---|---|---|---|---|
| STD 1 ACME | 43 | 0.959 | 0.983 | +2.5 | 0 | 100 |
| STD 2 ACME | 23 | 0.551 | 0.530 | -3.8 | 10 | 57 |
| STD 3 ACME | 52 | 0.343 | 0.357 | +4.1 | 4 | 92 |
| STD 1 Silex | 8 | - | 2.956 | - | - | - |
| STD 2 Silex | 7 | - | 2.197 | - | - | - |
| STD 3 Silex | 6 | - | 0.352 | - | - | - |
| STD 4 Silex | 10 | - | 0.160 | - | - | - |
| STD 1 ASA - 2016 | 162 | 0.351 | 0.344 | -2.0 | 9 | 94 |
| STD 2 ASA - 2016 | 154 | 0.534 | 0.528 | -1.1 | 4 | 97 |
| STD 3 ASA - 2016 | 156 | 1.059 | 1.033 | -2.5 | 7 | 96 |
| STD 4 ASA - 2016 | 154 | 4.770 | 4.713 | -1.2 | 0 | 100 |
| TOTAL | 775 | - | - | - | 34 | 96 |

The results presented in Table 11.4 are a summary of all SRMs submitted with drill core by Goldrock from 2005 to 2010. It should be noted that this information has also been assessed on a campaign by campaign basis as detailed in the 2016 Feasibility Study to ensure reasonable levels of accuracy were achieved after each campaign.

It can be seen from the overall results that the levels of bias are low (less than 5 %) with and a reasonable pass rate above 96 %.

Fortuna concludes that the accuracy of the samples submitted by Goldrock between 2005 and 2010 are within acceptable ranges and suitable for estimation purposes.

**Fortuna (2016)**

During the 2016 Fortuna drill campaign a total of 126 SRMs were inserted into the sample analysis submissions along with 2,269 regular samples, resulting in an overall insertion rate of 1 in 18 samples.

Fortuna used four SRMs (prepared and certified by Alex Stewart in 2007) to monitor the accuracy of results during the 2016 drilling program at Lindero.

For evaluation of the SRMs, Fortuna prepared control charts for each, as well as an accuracy plot. A summary of the performance of each SRM is shown in Table 11.5. A fail as described in the tables below is any value being >± 3 standard deviations from the best value.

**Table 11.5 SRM results submitted to ALS Chemex for the 2016 campaign**

| Standard | No. Submitted | Best Value (g/t) | Mean Value (g/t) | Bias (%) | No. fails | Pass rate (%) |
|---|---|---|---|---|---|---|
| STD 1 ASA | 27 | 0.328 | 0.327 | -0.3 | 5 | 81 |
| STD 2 ASA | 37 | 0.483 | 0.505 | +4.6 | 1 | 97 |
| STD 3 ASA | 36 | 1.205 | 1.244 | +3.2 | 0 | 100 |
| STD 4 ASA | 26 | 3.778 | 3.733 | -1.2 | 0 | 100 |
| TOTAL | 126 | - | - | - | 6 | 95 |

It can be seen from the overall results that the levels of bias are low (less than 5 %) with and a reasonable pass rate above 95 %.

Fortuna concludes that the accuracy of the samples submitted for the 2016 drilling campaign are within acceptable ranges and suitable for estimation purposes.

### 11.5.5 Blank samples

Field blank samples are composed of material that is known to contain grades that are less than the detection limit of the analytical method in use and are inserted by the geologist in the field. Blank sample analysis is a method of determining sample switching and cross-contamination of samples during the sample preparation or analysis processes.

**Rio Tinto (2002 to 2003)**

This analysis focuses on the submission of 84 blanks with 1,628 samples (submission rate of 1 in 20 samples).

Only one sample out of the 84 contained a gold assay greater than five times the analytical detection limit. The ALS Chemex blank analyses for the 2002 to 2003 drilling campaign are considered acceptable.

**Goldrock (2005 to 2010)**

An intrusive granodiorite was used by Goldrock as a source for their blank samples. Rock samples were collected systematically at an intrusive batholith outcrop next to the road to the Lindero Property. The rock samples were broken in a plastic basin to avoid contamination from local soils or pebbles. Only homogeneous pieces of rocks were kept, by removing oxidized surfaces for instance. Finally, blank samples were bagged, labeled, and inserted in the sampling stream. This analysis focuses on the submission of 664 blanks submitted with 14,821 samples (submission rate of 1 in 22 samples) during the 2005 to 2008 drill campaign; and 101 blanks submitted to with 1,716 samples (submission rate of 1 in 17 samples) during the 2005 to 2008 drill campaign.

The ACME blank analyses for the 2005 to 2008 are considered acceptable. Only nine samples out of the 664 submitted contained a gold assay greater than five times the analytical detection limit, a pass rate of greater than 98 %.

The blank analyses for the 2008 to 2010 drilling campaign are also considered acceptable. Only two samples out of 101 samples contained a gold assay greater than five times the analytical detection limit, a pass rate of greater than 98 %.

**Fortuna (2016)**

During the 2016 Fortuna drill campaign a total of 130 blanks were inserted into the sample analysis submissions along with 2,269 regular samples, resulting in an overall insertion rate of 1 in 17 samples.

Fortuna used the same blank material obtained by Goldrock during the 2007 to 2008 drilling program at Lindero.

The blank analyses for the 2016 drilling campaign are considered acceptable. None of the blanks submitted contained a gold assay greater than five times the analytical detection limit, representing a pass rate of 100 %.

### 11.5.6 Duplicate samples

The precision of sampling and analytical results can be measured by re-analyzing the same sample using the same methodology. The variance between the measured results is a measure of their precision. Precision is affected by mineralogical factors such as grain size and distribution and inconsistencies in the sample preparation and analysis processes. There are several different duplicate sample types which can be used to determine the precision for the entire sampling process, sample preparation, and analytical process. A description of the different types of duplicates used by Fortuna is provided in Table 11.6.

Numerous plots and graphs are used to monitor precision and bias levels. A brief description of the plots employed by Fortuna in the analysis of duplicate data, is described below:

- Absolute relative difference (ARD) statistics: relative difference of the paired values divided by their average.
- Scatter plot: assesses the degree of scatter of the duplicate result plotted against the original value, which allows for bias characterization and regression calculations.
- Ranked half absolute relative difference (HARD) of samples plotted against their rank % value.

If both the original and duplicate result returned a value less than ten times the detection limit, the result was disregarded for the ARD analysis due to distortion in the precision levels at very low grades close to the limits at which the instrumentation can measure. These very low values are not seen as material and can distort more meaningful results if they are not removed.

**Table 11.6 Duplicate types used by Fortuna**

| Duplicate | Description |
|---|---|
| Field | Sample generated by another sampling operation at the same collection point. Includes a second channel sample taken parallel to the first or the second half of drill core sample and submitted in the same or separate batch to the same (primary) laboratory. |
| Preparation | Second sample obtained from splitting the coarse crushed rock during sample preparation and submitted in the same batch by the laboratory. |

| Duplicate | Description |
|---|---|
| Laboratory | Second sample obtained from splitting the pulverized material during sample preparation and submitted in the same batch by the laboratory. |
| Reject assay | Second sample obtained from splitting the coarse crushed rock during sample preparation and submitted blind to the same or different laboratory that assayed the original sample. |
| Duplicate assay | Second sample obtained from splitting the pulverized material during sample preparation and submitted blind at a later date to the same laboratory that assayed the original pulp. |
| Check assay | Second sample obtained from the pulverized material during sample preparation and sent to an umpire laboratory for analysis. |

### Rio Tinto (2002 to 2003)

One coarse duplicate (taken after cone crushing to 75 % passing 10 mesh) and one pulp duplicate (taken from a split at the end of pulverization, 95 % passing 150 mesh) were inserted in every 30 samples with the objective of checking for splitting or laboratory errors. Samples were randomized. Independent checks assays were carried out using Alex Stewart in Chile, and Bondar Clegg in Canada. Results for all duplicates submitted are displayed in Table 11.7.

**Table 11.7 Duplicate results for gold - Rio Tinto**

| Type of Duplicate | No. of duplicates analyzed# | HARD* value at 90th percentile |
|---|---|---|
| **Reject assays[1]** | 22 | 9.2 |
| **Duplicate assays[2]** | 71 | 10.3 |
| **Check Assays[3] (Alex Stewart)** | 38 | 21.2 |
| **Check Assays[3] (Bondar Clegg)** | 103 | 9.2 |

*HARD = Half Absolute Relative Difference
# Results below detection limit and with insufficient mass removed from analysis
1. Acceptable HARD value for reject assays is less than 20%
2. Acceptable HARD value for duplicate assays is less than 10%
3. Acceptable HARD value for check assays is less than 10%

Precision for the 22 coarse duplicate samples submitted Rio Tinto are seen as acceptable for gold assays. Fortuna calculated the half absolute relative difference (HARD) of the ALS Chemex laboratory to be ±9.2 % for gold at the 90th percentile. Fortuna considers coarse duplicate assays to be acceptably precise if 90 % of the assays show less than 20 % half absolute relative difference.

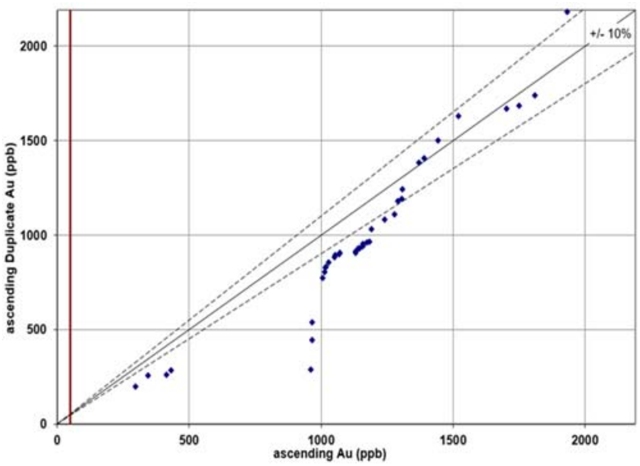
Fortuna calculated the ALS Chemex assay precision for duplicate assays to be ±10.3 % for gold at the 90th percentile. Fortuna considers pulp duplicate assays to be acceptably precise if 90 % of the assays show less than 10 % half absolute relative difference.

Both the coarse reject and pulps independently submitted by Rio Tinto to ALS Chemex display acceptable levels of precision for a gold deposit of this nature.

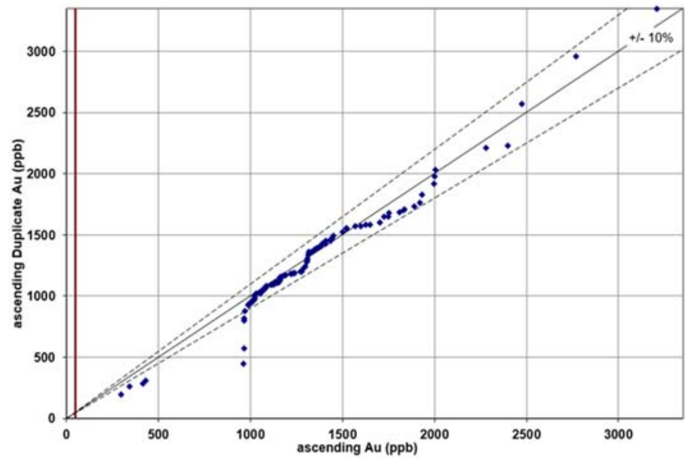
Rio Tinto sent samples for check assays to the Alex Stewart laboratory in La Serena, Chile and Bondar Clegg in Canada. Samples were selected with grades more than 0.90 g/t gold and 500 ppm copper. The results of the analyses show a negative bias in the ALS Chemex assays compared to the results of the other laboratories. This negative bias is at least partially caused by the selection of samples based upon grades, known as a selection bias. The ALS Chemex assays have a fixed lower limit based on the selection, whereas the check assays can return lower grades than this limit.

Fortuna examined the bias between the laboratories by examining quantile-quantile (Q-Q) plots for each of the umpire laboratories compared to the ALS Chemex results for the same samples (Figure 11.1).

**Figure 11.1 Quantile-Quantile Plots for check assays**




Q-Q Plot – Alex Stewart




Q-Q Plot – Bondar Clegg

Figure prepared by Fortuna Silver Mines Inc., 2017

Although it is not possible to definitively identify which laboratory was experiencing problems due to a lack of available supporting QC samples (i.e. standards/blanks) submitted with the check assays, it is believed that the poor results for duplicates submitted to Alex Stewart is likely a reflection of issues at the umpire laboratory rather than at ALS Chemex, especially with significant bias observed for gold assays less than 1,300 ppb Au. Check assays submitted to Bondar Clegg show minor bias, suggesting that the precision levels at ALS Chemex are acceptable.

**Goldrock (2005 to 2010)**

Drill holes LDH-11 to LDH-116 were submitted to ACME laboratory in Santiago, Chile during the drilling campaigns between 2005 and 2008. The geotechnical holes LGT-01 to LGT-05 were also sent for assaying to the ACME laboratory in 2008. However, holes LGT-06 to LGT-11 drilled in 2010 were analyzed at the Alex Stewart laboratory in Mendoza.

For drill holes LDH-11 to LDH-74 a field duplicate was inserted to ACME in each batch of 30 samples. Cross-laboratory check assays were performed on pulp sample material; samples were collected approximately every 10 samples. Two 100 g samples were split from the original pulp sample, with one sent to Alex Stewart in Mendoza, Argentina, and the other to the ALS Chemex in Mendoza (Argentina) and to La Serena (Chile) for assay. Assaying at Alex Stewart and ALS Chemex used a 50 g gold fire assay with an AA finish.

For drill holes LDH-75 to LDH-116, a SRM and field duplicate were inserted to ACME in every batch of 20 samples. Check assays were collected the same way as for holes LDH-11 to LDH-74, with one pulp for every 10 samples, and sent to the same reference laboratories.

For drill holes LDH-75 to LDH-116, field duplicates were collected at a frequency of no less than 15 samples. The half cores were sawed to quarter-core and stored in bags, with different labels to the regular samples, and then inserted in the sampling stream preparation and assaying by ACME laboratories. Check assays were collected as described above, with one pulp for every 10 samples, and sent to the same reference laboratories.

A detailed review of the results drill campaign by drill campaign was conducted by AMEC during the Feasibility Study and reported in the KCA (2016a) Technical Report.

Fortuna conducted a summary analysis of results for all duplicates submitted by Goldrock between 2005 and 2008 (holes LDH-11 to LDH-116) as well as by the individual campaigns described above and for the duplicates submitted for the geotechnical drilled in 2008 and 2010 (LGT-01 to LGT-11). Fortuna concurred with the findings in the Feasibility Study in respect to individual campaigns.

**Drill Campaign 2005 to 2008 (LDH-11 to LDH-116)**

Results for all duplicates submitted between 2005 and 2008 are displayed in Table 11.8.

**Table 11.8 Duplicate results for gold – Goldrock (2005 to 2008)**

| Type of Duplicate | No. of duplicates analyzed# | HARD* value at 90$^{th}$ percentile |
|---|---|---|
| **Field duplicates[1]** | 665 | 19.7 |
| **Check Assays[2] (ALS Chemex)** | 1,601 | 13.5 |
| **Check Assays[2] (Alex Stewart)** | 1,566 | 15.7 |

*HARD = Half Absolute Relative Difference
# Results below detection limit and with insufficient mass removed from analysis
1. Acceptable HARD value for field duplicates is less than 30 %
2. Acceptable HARD value for check assays is less than 10 %

Precision for quarter-core duplicate samples submitted is acceptable for gold assays. Fortuna calculated ACME assay precision to be ±19.7 % for gold at the 90$^{th}$ percentile. Fortuna considers quarter-core duplicate assays to be acceptably precise if 90 % of the assays show less than 30 % half absolute relative difference. The levels of precision do vary between campaigns depending on numbers submitted and grade ranges but overall show an acceptable level of precision.

The Goldrock QAQC program is lacking in pulp duplicate analyses performed at the same laboratory using the same analytical procedures. As a substitute, Fortuna used the check assays to evaluate bias and precision, although this is not ideal as the precision values obtained are a combination of analytical, pulp sub-sampling and between laboratory variances. As such it is to be expected that the precision is somewhat lower than same laboratory duplicates analyzed under the same conditions as the original samples. Additionally, if deviations or bias exists it is impossible to tell which laboratory is at fault.

Fortuna calculated ACME to ALS Chemex assay precision to be ±13.5 % for gold at the 90th percentile. Precision for the ACME to Alex Stewart was found to be ±15.7 %. In addition, Q-Q plots were examined with no significant bias being observed above the line of significance (10 x the detection limit) for either of the umpire laboratories compared to the original assays. Fortuna considers this lack of bias to be acceptable for cross-laboratory pulp duplicates.

No SRMs or blanks were included in the check assay dispatches to control assay accuracy and contamination issues at the secondary check laboratories.

**Drill Campaign 2008 to 2010 (LGT-01 to LGT-11)**

Duplicate results for the geotechnical drill holes drilled between 2008 and 2010 are displayed in Table 11.9.

**Table 11.9 Duplicate results for gold – Goldrock (2008 to 2010)**

| Type of Duplicate | No. of duplicates analyzed# | HARD* value at 90th percentile |
|---|---|---|
| **Field duplicates[1]** | 79 | 22.0 |
| **Check Assays[2] (ALS Chemex)** | 77 | 11.0 |
| **Check Assays[2] (Alex Stewart)** | 71 | 19.3 |

*HARD = Half Absolute Relative Difference
# Results below detection limit and with insufficient mass removed from analysis
1. Acceptable HARD value for field duplicates is less than 30 %
2. Acceptable HARD value for check assays is less than 10 %

Precision for quarter-core duplicate samples submitted is acceptable for gold assays. Fortuna calculated assay precision to be ±22.0 % for gold at the 90th percentile. Fortuna considers quarter-core duplicate assays to be acceptably precise if 90 % of the assays show less than 30 % half absolute relative difference.

As in the prior years, the Goldrock QAQC program did not submit coarse reject or pulp duplicate samples for analyses at the same laboratory using the same analytical procedures. Similar to the results for 2005 to 2008, Fortuna used the check assays to evaluate bias and precision between the primary and umpire laboratories, although this is not ideal as per the reasons described above.

Fortuna calculated primary laboratory to ALS Chemex assay precision to be ±11.0 % for gold at the 90th percentile. Precision for the duplicates submitted to Alex Stewart was found to be ±19.3 %. Fortuna considers pulp duplicate assays to be acceptably precise if 90 % of the assays show less than 10 % half absolute relative difference. In addition, Q-Q plots were examined with no significant bias observed for the pulps sent to ALS Chemex but results from Alex Stewart indicated a low bias in the duplicate results for assays up to 0.12 g/t Au). Although the precision levels at the Alex Stewart laboratory do not achieve the desired levels and bias was observed in the results, it is believed that this is more a reflection of an issue in the umpire laboratory rather than at ACME as the results from the alternative umpire laboratory (ALS Chemex) are reasonable and unbiased. With this in mind, Fortuna considers the level of precision at the primary laboratory (ACME) to be acceptable for resource estimation purposes.

No SRMs or blanks were included in the check assay dispatches to control assay accuracy and contamination issues at the secondary check laboratories.

**Fortuna (2016)**

Drill holes LDH-11 to LDH-116 were submitted to the ALS Global sample preparation facility in Mendoza, Argentina. Pulp samples were shipped by commercial air freight to ALS Global's analytical facility in Vancouver, British Columbia for analysis.

Fortuna submitted field duplicates, reject assays (coarse rejects), duplicate assays (pulps), check assays (pulps sent to an umpire laboratory) with the drill sample submissions to the laboratory to monitor assay precision. Samples were sent for check analyses to Bureau Veritas laboratory in Vancouver, Canada for assaying.

Field, preparation, and laboratory duplicates were submitted at a frequency of no less than 1 in 20 samples. Reject assays, duplicate assays, and check assays were submitted at a frequency of no less than 1 in 40 samples. Duplicate samples were prepared in the same manner as per the Goldrock drilling campaigns, however, standards and blanks were submitted with the duplicates sent to the umpire laboratory and showed reasonable levels of accuracy and no contamination or sampling switching issues. A summary of the duplicate results is displayed in Table 11.10.

**Table 11.10 Duplicate results for gold – Fortuna**

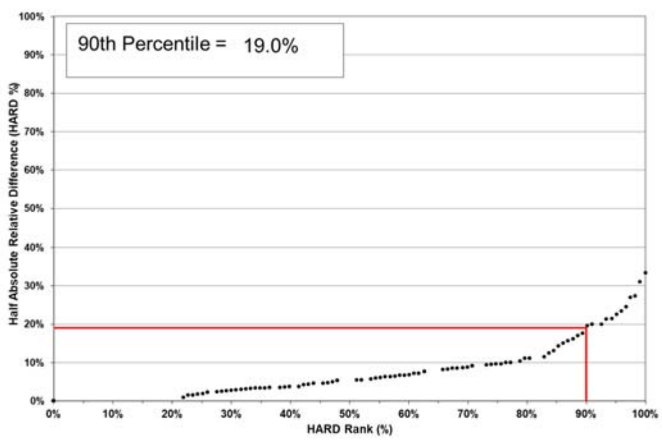
| Type of Duplicate | No. of duplicates analyzed# | HARD* value at 90th percentile |
|---|---|---|
| **Field duplicates[1]** | 124 | 19.0 |
| **Preparation duplicates[2]** | 122 | 7.7 |
| **Laboratory duplicates[3]** | 123 | 6.3 |
| **Reject assays[2]** | 57 | 11.7 |
| **Duplicate assays[3]** | 55 | 9.1 |
| **Check Assays[3] (Bureau Veritas)** | 56 | 10.9 |

*HARD = Half Absolute Relative Difference
# Results below detection limit and with insufficient mass removed from analysis
1. Acceptable HARD value for field duplicates is less than 30 %
2. Acceptable HARD value for preparation duplicates and reject assays is less than 20 %
3. Acceptable HARD value for laboratory duplicates, duplicate assays, and check assays is less than 10 %

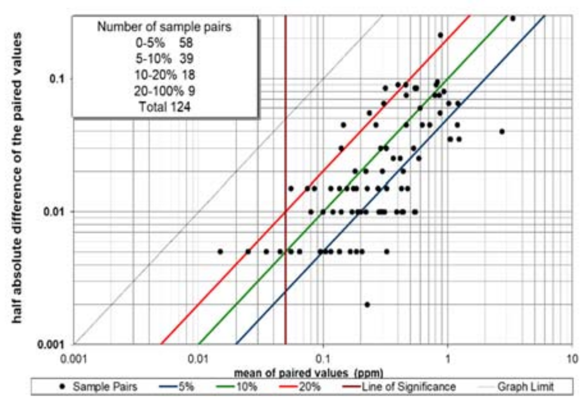
Precision for quarter-core duplicate samples submitted is acceptable for gold assays. Fortuna calculated assay precision to be ±19.0 % for gold at the 90th percentile. Fortuna considers quarter-core duplicate assays to be acceptably precise if 90 % of the assays show less than 30 % half absolute relative difference. Figure 11.2 provides an example of the types of graphs used by Fortuna for assessing duplicate results.

Precision for preparation duplicate samples and reject assays submitted to ALS Global is acceptable for gold assays with precision levels calculated to be ±7.7 % and ±11.3 %, respectively, for gold at the 90th percentile. Fortuna considers coarse reject assays to be acceptably precise if 90 % of the assays show less than 20 % half absolute relative difference.

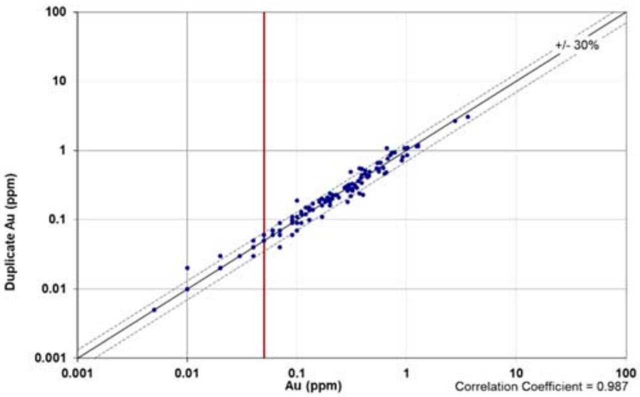
**Figure 11.2 Field duplicate analysis for gold – Fortuna**



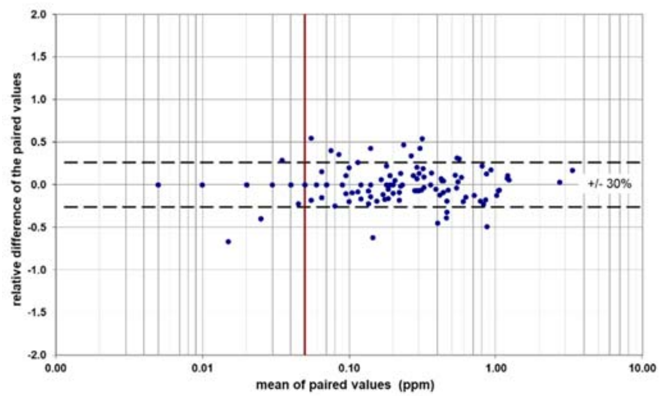

Ranked HARD plot




Precision Pairs Plot




Log scatter plot




Mean of paired relative difference

Figure prepared by Fortuna Silver Mines Inc., 2017

Precision for laboratory duplicate samples and duplicate assays submitted to ALS Global is also acceptable for gold assays with precision levels calculated to be ±6.3 % and ±9.1 %, respectively, for gold at the 90th percentile. Fortuna considers pulp assays to be acceptably precise if 90 % of the assays show less than 10 % half absolute relative difference.

Levels of bias observed through Q-Q plots for the check assays submitted to Bureau Veritas compared to ALS Global were found to be insignificant, further establishing the confidence in the assay results for estimation purposes.

### 11.5.7 Conclusions regarding quality control results

Submission rates and accuracy levels (SRM submission), as well as sample contamination/switching (blank submission) for all laboratories is considered reasonable, with all campaigns showing pass rates of above 95 %. The certificates for SRM's submitted during the Rio Tinto campaign are not available and therefore accuracy levels for these early holes cannot be confirmed and that has been taken into consideration during Mineral Resource confidence category classification.

During both the Rio Tinto and Goldrock drilling campaigns a limited number of duplicate types were submitted meaning that an analysis of each stage of the sample preparation process cannot be assessed. Nevertheless, field duplicates and check assays were submitted regularly, and both report reasonable levels of precision, with the exception of check assays submitted to Alex Stewart during the Rio Tinto drilling campaign. This is believed to represent an issue at the umpire laboratory rather than the primary (ALS Chemex) laboratory as the same pulps submitted to Bondar Clegg reported acceptable precision levels. Fortuna submitted a full suite of duplicate types during the 2016 drilling campaign and reasonable precision levels were observed for all duplicate types. It is common to encounter precision issues in gold deposits due to the highly heterogenous "nugget effect" nature of gold mineralization, however as demonstrated from the heterogeneity test (Section 9.5.3), the mineralization is very homogenous in nature at the Lindero Deposit with good gold grade continuity observed in relation to the intrusive events in the porphyry system.

All drilling data including QC results are stored in Maxwell GeoService's commercial SQL database system, DataShed™, which is stored on the Lindero server. All data must pass a series of validation checks prior to being imported into DataShed™. A dedicated data manager oversees data transfer with QC information exported into an Excel™ format for analysis.

Mansfield are working with Maxwell to allow the automated evaluation of all QC information inside DataShed™ to remove the need for data exportation. Access to DataShed™ is limited to the database manager, being password protected and a backup of the database is taken automatically daily with hard copy discs being taken to Salta for storage weekly.

## 11.6 Comment on Section 11

Implementation of a quality assurance/quality control (QAQC) program is current industry best practice and involves establishing appropriate procedures and the routine insertion of standard reference material (SRMs), blanks, and duplicates to monitor the sampling, sample preparation and analytical process. Fortuna implemented a full QAQC program to monitor the sampling, sample preparation and analytical process for the 2016 drilling campaign in accordance with its companywide procedures. The program involved the routine insertion of SRMs, blanks, and duplicates. Evaluation of the QAQC data indicate that the data are sufficiently accurate and precise to support Mineral Resource estimation.

It is the opinion of the QPs that the sample preparation, security, and analytical procedures for samples taken at Lindero by both Goldrock and Fortuna have been conducted in accordance with acceptable industry standards and that assay results generated following these procedures are adequately precise and accurate for use in Mineral Resource and Mineral Reserve estimation.

# 12 Data Verification

## 12.1 Introduction

In 2009 an independent audit of the information used for the estimation of resources and reserves was conducted by AMEC and summarized in the KCA (2016a) Technical Report. The work included independent audits of the database, collar and downhole surveys, drill logs, assays, bulk density measurements, core recovery, and QAQC results.

The 2009 audit concluded that "*In AMEC's opinion, the data verification programs undertaken on the data collected from the Lindero Deposit up to 2009 support the geological interpretations, and the analytical and database quality, and therefore the data can support mineral resource estimation. Data gathered subsequently have not been independently verified by AMEC.*"

Fortuna has reviewed the work performed by AMEC and concurs with their opinion and has conducted additional audits and verification of historical information used in the most recent resource and reserve estimates as well as verifying new data generated during the 2016 drilling campaign to support assumptions for a construction decision. The verification process has focused on the following items:

- Database
- Collars and down-hole surveys
- Lithological logs
- Assays
- Metallurgical results
- Geotechnical parameters

## 12.2 Database

Goldrock stored the drilling and trenching data in a series of Microsoft Excel™ spreadsheets. This information was audited, and a number of deficiencies were encountered such as the rounding down of assay results and typographic inconsistencies.

Fortuna organized the transfer of all drilling data to Maxwell GeoService's commercial SQL database system, DataShed™, employing a dedicated data manager to oversee the data transfer. All data must pass a series of validation checks prior to being imported into DataShed™.

The database was then reviewed and validated by Mr. Eric Chapman, P. Geo. The data verification procedure involved the following:

- Evaluation of minimum and maximum grade values
- Investigation of minimum and maximum sample lengths
- Randomly selecting assay data from the databases and comparing the stored grades to the original assay certificates
- Assessing for inconsistences in spelling or coding (typographic and case sensitivity errors)

- Ensuring full data entry and that a specific data type (collar, survey, lithology, and assay) is not missing
- Assessing for sample gaps or overlaps

No inconsistencies were discovered.

## 12.3 Collars and downhole surveys

### 12.3.1 Collars

Fortuna checked all collar and downhole survey information for each campaign against source documentation. In addition, Fortuna completed a hand-held GPS survey of randomly selected drill-hole collars. The results showed a good agreement with locations in the database.

Of the 144 holes drilled at Lindero, 128 collars have been surveyed using differential GPS (Geodesico) conducted by an independent surveying company. Of the 16 drill holes that were not GPS surveyed these were related to condemnation and geotechnical drilling conducted in 2010 with only three holes being located in the area of mineralization. Trenches were surveyed using either a hand-held GPS unit or compass and tape measure.

### 12.3.2 Downhole surveys

Of the 144 holes drilled at Lindero, 20 holes do not have validated downhole surveys.

A study of downhole deviations was conducted by AMEC in 2009 to determine whether unsurveyed drill holes or drill holes surveyed using the Maxibor™ instrument should be adjusted.

AMEC used the first and last downhole survey of each drill hole to calculate a total, accumulated azimuth and dip deviation. Vertical deviations in the dip of the drill holes as measured by Maxibor™ and gyroscope surveys are very similar.

Horizontal deviations in the azimuth of the drill holes; however, are different. The gyroscopic survey measurements show that 45 out of 53 drill holes (85 %) deviated in a clockwise direction (i.e. following the rotation of the drill rod) while Maxibor™ measurements show 26 out of 50 drill holes (52 %) deviating in an anti-clockwise direction.

In order to evaluate the risk associated with the Maxibor™ surveys, AMEC constructed a model using gyroscopic downhole surveys with a clockwise azimuth deviation. Each drill hole with a Maxibor™ survey was then evaluated against the model in order to determine the downhole depth at which the horizontal deviation became greater than 10 m in either the easting or northing direction. Results show that a total of 15 drill holes had horizontal deviations of greater than 10 m at downhole depths ranging from 150 m to 490 m.

The depth intervals of those drill holes with excessive horizontal deviations were flagged and were subsequently considered during resource classification.

Fortuna reviewed the analysis and findings from this investigation and are in agreement with the study and therefore have also taken into account the above during resource classification.

The condemnation (5) and geotechnical drill holes (11) drilled in 2010 did not have downhole surveys conducted. These holes were drilled to a maximum depth of 350 m with the condemnation holes drilled at 60° angles and the geotechnical holes drilled vertically.

The 12 holes drilled in 2016 were downhole surveyed using a Reflex Gyro™ survey instrument.

## 12.4 Lithological logs

In August 2016 Fortuna initiated a comprehensive program of relogging to verify the original lithological descriptions. An independent porphyry geology specialist was contracted to lead the relogging program. This involved relogging of holes on a systematic basis along a series of delineated sections to facilitate the generation of geological interpretational cross sections. A core library was created for each of the main lithological units for cross reference purposes and for each hole logged a representative sample was retained for each lithological unit for confirmatory purposes to resolve discrepancies in the sectional interpretations.

The same geologists responsible for relogging were also responsible for logging the 4,461 m of new core drilled in 2016 which was used to confirm geological concepts.

The logging process established cross cutting relationship between intrusive events and vein systems. This assisted in increasing the confidence in the geological interpretation.

Drill core was also reviewed by Mr. Eric Chapman, Vice President of Technical Services for Fortuna, as part of the data verification process.

## 12.5 Assays

### 12.5.1 Rio Tinto

No assay certificates are available from the Rio Tinto drill campaigns. The data are contained in Excel™ spreadsheets. Fortuna has therefore not been able to verify the assay data against source documentation. The lack of assay certificates for the drill holes from the Rio Tinto drilling campaign was considered during resource classification.

### 12.5.2 Goldrock

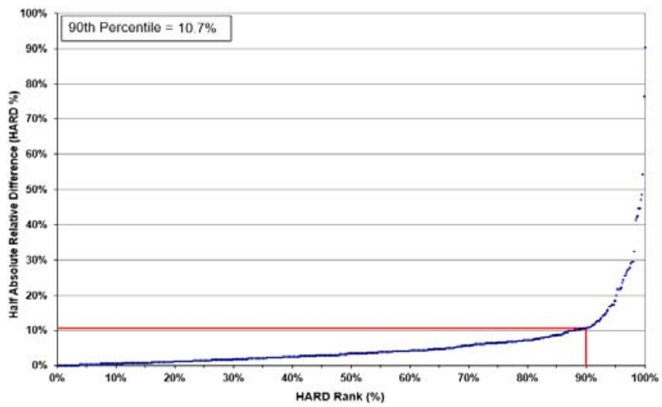
Of the 17,092 assays received by Goldrock during the drilling campaign between 2005 and 2008 original pdf certificates are missing for 5,166 of the samples assayed by ACME in 2007 with only the Excel™ spreadsheets available for review. Fortuna conducted an extensive search which included contacting multiple laboratories and previous consultants in the hope of recovering the missing assay certificates but without success. In order to validate the missing assays, Fortuna randomly selected 520 pulps representing all drill holes, lithologies, and a range of depths and assays of the samples for which certificates are missing (10 % of the total) and resubmitted them for analysis at ALS Global. A statistical evaluation was conducted comparing the original reported values against the resubmitted check assays as shown in Table 12.1 and Figure 12.1.

**Table 12.1 Statistical analysis of original and resubmitted gold assays for samples missing certificates**

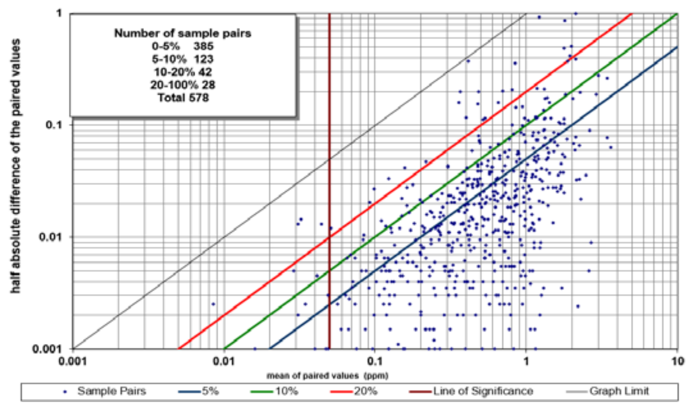
| Parameter | Original | Resubmission |
|---|---|---|
| Minimum (g/t) | 0.011 | 0.006 |
| Lower quartile (g/t) | 0.25 | 0.24 |
| Median (g/t) | 0.53 | 0.51 |

| Parameter | Original | Resubmission |
|---|---|---|
| **Mean (g/t)** | 0.67 | 0.67 |
| **Geometric mean (g/t)** | 0.46 | 0.44 |
| **Log estimate mean (g/t)** | 0.73 | 0.74 |
| **Upper quartile (g/t)** | 0.94 | 0.94 |
| **Maximum (g/t)** | 3.69 | 3.69 |
| **Coefficient of variation** | 0.85 | 0.88 |
| **Standard deviation** | 0.57 | 0.59 |
| **Variance** | 0.33 | 0.35 |
| **Correlation coefficient** | 0.95 | |
| **Log correlation coefficient** | 0.97 | |
| **Half absolute relative difference[1]** | 10.7 | |
| 1. Acceptable HARD value for check assays is less than 10 % | | |

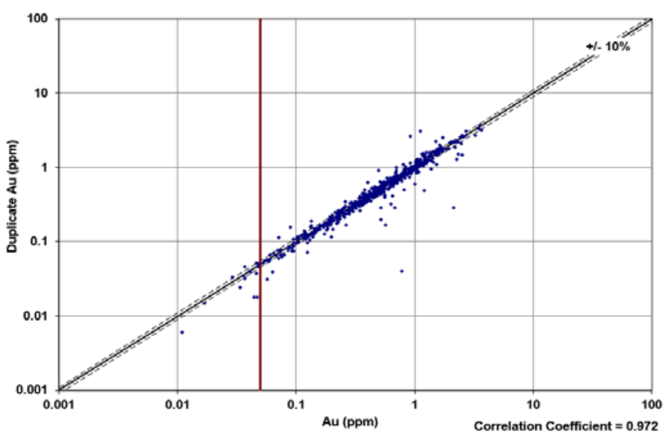
**Figure 12.1 Check assay duplicate analysis for samples missing certificates**



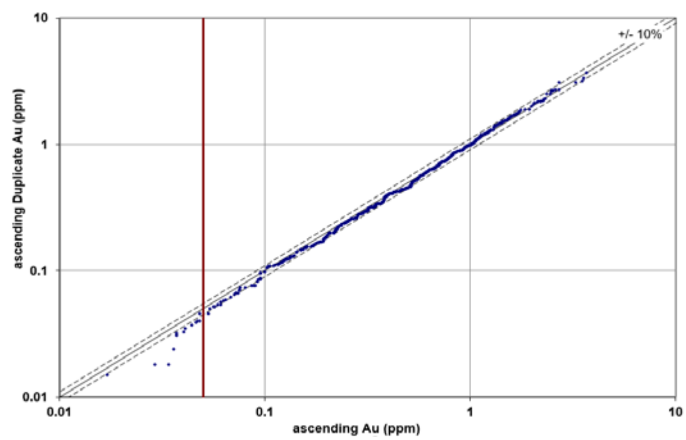

Ranked HARD plot




Precision Pairs Plot




Log scatter plot




Log Q-Q Plot

Figure prepared by Fortuna Silver Mines Inc., 2017

No significant deviation or bias was observed between the randomly selected check assays and the originals beyond that which would be expected when resubmitting pulps to an umpire laboratory. Subsequently, although the pdf certificates of the assays cannot be located, the QP considers the data in the Excel™ spreadsheets to be valid and acceptable for Mineral Resource estimation purposes.

### 12.5.3 Fortuna

A full QAQC program including the insertion of standards, blanks and duplicates was performed to validate the assays reported by ALS Global during the 2016 drilling campaign. A full description of this program and its results is provided in Section 11.5. Fortuna concluded that the reported assays were suitable for resource estimation purposes.

## 12.6 Density measurements

In order to validate the density measurements used for estimating tonnage Fortuna submitted 30 randomly selected density samples to ALS Global for analysis. The laboratory prepared the samples using the OA-GRA08 methodology using a wax coating and comparing the weight in air and weight in water of the selected samples. A comparison of the original and re-submitted density results are displayed in Table 12.2.

**Table 12.2 Comparison of original and re-submitted density measurements**

| Sample ID | Original (g/cm$^3$) | Resubmission (g/cm$^3$) | Percentage Difference |
|---|---|---|---|
| D-52 | 2.67 | 2.69 | 1 |
| D-55 | 2.55 | 2.67 | 5 |
| D-57 | 2.67 | 2.75 | 3 |
| D-60 | 2.63 | 2.65 | 1 |
| D-65 | 2.72 | 2.68 | -1 |
| D-68 | 2.78 | 2.82 | 2 |
| D-72 | 2.75 | 2.83 | 3 |
| D-300 | 2.56 | 2.48 | -3 |
| D-304 | 2.57 | 2.59 | 1 |
| D-318 | 2.59 | 2.55 | -2 |
| D-320 | 2.54 | 2.52 | -1 |
| D-403 | 2.59 | 2.50 | -3 |
| D-409 | 2.70 | 2.59 | -4 |
| D-413 | 2.73 | 2.78 | 2 |
| D-415 | 2.60 | 2.67 | 3 |
| D-444 | 2.66 | 2.69 | 1 |
| D-449 | 2.62 | 2.63 | 0 |
| D-452 | 2.53 | 2.46 | -3 |
| D-457 | 2.63 | 2.69 | 2 |
| D-481 | 2.61 | 2.58 | -1 |
| D-483 | 2.63 | 2.63 | 0 |
| D-488 | 2.61 | 2.62 | 0 |
| D-615 | 2.57 | 2.60 | 1 |
| D-617 | 2.59 | 2.58 | -1 |
| D-725 | 2.64 | 2.64 | 0 |
| D-729 | 2.61 | 2.65 | 2 |
| D-733 | 2.65 | 2.79 | 5 |
| D-734 | 2.63 | 2.57 | -2 |
| D-735 | 2.68 | 2.73 | 2 |
| D-737 | 2.72 | 2.61 | -4 |
| **TOTAL** | **2.63** | **2.64** | **0** |

Although some minor variations are observed, with the original density measurements being biased high to about 2.57 g/cm$^3$, having little bias from 2.57 to 2.68 g/cm$^3$ and biased low above 2.68 g/cm$^3$ when compared to the resubmission, the levels of variation are within expected levels of tolerance and the average density levels being reported show no significant deviation or bias. Fortuna concluded from the above verification that the density measurements used in the 2016 Feasibility Study appear reasonable and have been maintained for the updated resource estimate.

## 12.7 Metallurgical results

An extensive program of metallurgical testwork has been performed in 2016 and 2017 to validate the results and conclusions reported in the KCA (2016a) Feasibility Study. A description of this work is presented in Section 13.

## 12.8 Geotechnical parameters

Fortuna contracted CNI to validate all geotechnical data, data collection methods, slope stability analysis methods, and slope angle recommendations presented previously by other consultants to determine feasibility-level slope angle recommendations for design of the planned Lindero final pit. Based on the review, additional work, including a site visit in August 2016 and slope stability analysis work, were conducted by CNI to determine slope angles for feasibility design purposes.

An initial draft report was completed by CNI in early October 2016 (CNI, 2016). This report included recommendations for trenching to explore the character of materials to comprise the foundation of planned waste stockpiles where there was little drilling information.

The trenching work along with other tasks were completed during a site visit in late October 2016. An updated report was presented to Fortuna in May 2017 (CNI, 2017) which included the results of the trenching work, updated waste dump stability analysis, and study of the impacts of a revised geologic model provided in early 2017 on pit slope stability. The report also presented conclusions and recommendations for the final pit design as well as blasting and slope angle recommendations for the interim pit phases.

All of the above recommendations were taken into account as part of the reserve estimation process conducted by Fortuna as described in Section 15.

## 12.9 Comment on Section 12

In 2009 an independent audit of the information used for the estimation of resources and reserves was conducted by AMEC and summarized in the KCA (2016a) Technical Report. The work included independent audits of the database, collar and downhole surveys, drill logs, assays, bulk density measurements, core recovery, and QAQC results.

The 2009 audit concluded that the data verification programs undertaken on the data collected from the Lindero Deposit up to 2009 support the geological interpretations, and the analytical and database quality, and therefore the data can support mineral resource estimation.

Fortuna has reviewed the work performed by AMEC and concurs with their opinion and has conducted additional audits and verification of historical information used in the most recent resource and reserve estimates as well as verifying new data generated during the 2016 drilling campaign to support assumptions for a construction decision. The verification process focused on the database; collars and downhole surveys; lithological logs; assays; metallurgical results; and geotechnical parameters. Fortuna checked all collar and downhole survey information for each campaign against source documentation and completed a hand-held GPS survey of randomly selected drill-hole collars. The results showed a good agreement with locations in the database. In August 2016 Fortuna initiated a comprehensive program of relogging to verify the original lithological descriptions.

Fortuna contracted CNI to validate all geotechnical data, data collection methods, slope stability analysis methods, and slope angle recommendations presented previously by other consultants to determine feasibility-level slope angle recommendations for design of the planned Lindero final pit.

The QP is of the opinion that the data verification programs performed on the data collected from the Project are adequate to support the geological interpretations, the analytical and database quality, and Mineral Resource estimation at the Lindero Project. This conclusion is based on the following:

- No material sample biases were identified from the QAQC programs. Analytical data that were considered marginal were accounted for in the resource classifications
- Sample data collected adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits
- External reviews of the database were performed in 2003, 2008, and 2009, producing independent assessments of the database quality. No significant problems with the database, sampling protocols, flowsheets, check analysis program, or data storage were noted
- Mansfield compiled and maintains a relational database (DataShed™) for the Lindero Project which contains all collar, assay, density, survey and lithology information as well as all associated QAQC data
- Drill holes lacking surveyed collar coordinates have been resurveyed wherever possible and the original surveyor records stored
- Assays obtained during Goldrock's drilling programs that lacked original assay certificates have been verified by a program of re-assaying of 10 % of the pulps which indicated no significant bias between the original and reassayed results
- Drill data are typically verified prior to Mineral Resource estimation, by running a software program check

# 13 Mineral Processing and Metallurgical Testing

Additional metallurgical testing was completed by Fortuna in 2016 and 2017 to support a proposed processing flowsheet using high pressure grind roll (HPGR) crushing, cyanide cure, agglomeration and heap leaching. Summary values of these investigations are included in Table 13.1 and used in the development of the Project designs and economics.

**Table 13.1 Summary metallurgical values**

| Test | Type | Calculated Head | | | Laboratory Extraction (%) | | | Field Extraction (%) | | | NaCN |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Au (g/t) | Ag (g/t) | Cu (%) | Au | Ag | Cu | Au | Ag | Cu | (kg/t) |
| CL-01 | Met 1 | 0.535 | 0.349 | 0.10 | 79.4 | 71.3 | 11.8 | 75.4 | 71.3 | 11.8 | 0.477 |
| CL-02 | Met 2 | 0.656 | 0.500 | 0.13 | 82.2 | 66.8 | 28.9 | 78.2 | 66.8 | 28.9 | 0.616 |
| CL-03 | Met 3 | 0.423 | 0.751 | 0.09 | 82.5 | 48.1 | 9.8 | 78.5 | 48.1 | 9.8 | 0.441 |
| CL-04 | Met 4 | 0.401 | 0.292 | 0.13 | 72.5 | 65.8 | 8.6 | 68.5 | 65.8 | 8.6 | 0.419 |

In Table 13.1, a 4 % deduct was taken to extrapolate laboratory gold extraction results to actual operations. No other deducts for silver, copper or cyanide consumption are applied.

## 13.1 Prior metallurgical testwork

Extensive metallurgical testing has been conducted on the Lindero Deposit. The bulk of this testing was reported as part of the February 2016 Technical Report (KCA, 2016a), and has been summarized into this section of the Report.

Data extracted from the 2016 Technical Report and the supporting metallurgical laboratory documents (KCA, 2009, 2009a; 2012; 2013; 2014) are included here to show continuity with the current metallurgical work. The selected data included in this section were reviewed and are considered within industry standards for such work.

During review of the existing body of work, an alternative interpretation of some of the data was developed, which led to the current set of testwork. Most notable conclusions of this reinterpretation are the necessity for a SART plant, the stability of a multi-lift HPGR-crushed heap, and the benefit of a cyanide cure step. The reader is referred to the 2016 Technical Report and the supporting laboratory reports for an independent check of this interpretation. However, the successful conclusion of the current test program would appear to be sufficient corroboration of the alternative interpretation of the existing raw data.

### 13.1.1 Samples

Four lithological types were tested and designated as Met 1, Met 2, Met 3, and Met 4. Table 13.2 is partially reproduced from the Table 13-3 of the 2016 Technical Report.

**Table 13.2 Key lithologies for the Lindero Deposit**

| Lithology Code | Description | Mineralogical Grouping | Oxides/Fresh |
|---|---|---|---|
| FPD/CPD1 | Fine Porphyry Diorite | MET 1 | Fresh (sulfide) |
| PBFDox | Bi-Modal Porphyry | MET 1 | Fresh (sulfide) |
| BxMagmt | Magmatic Breccia | MET 1 | Fresh (sulfide) |
| FPDox | Oxide Fine Porphyry Diorite | MET 2 | Oxide (incl. mixed) |
| DDP/DDPox | Mingled Porphyry | MET 3 | Fresh (sulfide) & Oxides |
| SED | Sediments | MET 4 | Fresh (sulfide) & Oxides |
| PMI | Post Mineral Quartz Diorite | | Fresh (sulfide) |

Six phases of testing were conducted between 2004 and 2012.

- Phase I – October 2004: Bottle roll tests and three small columns were conducted on Met 1 and Met 2 samples. This program was noted in the 2016 Technical Report but no summary of results was given
- Phase II – August 2006: 28 bottle roll tests were conducted on a range of crushed sizes and 15 column tests were performed on all four met types
- Phase III – July/September 2007: Seventeen column tests were conducted on hammermill-crushed samples. The samples were reduced to nominal passing 9 mm. The four met types were represented

Four additional column tests were conducted on HPGR-crushed materials reduced to nominal passing 9 mm. These materials were determined to be contaminated, resulting in increased cyanide consumptions to "abnormally high values" although the gold extraction was deemed to be acceptable.

In addition to head assays, whole rock analysis, and multi-element ICP analysis; the crushing work index (Wi), abrasion index (Ai) and the HPGR roll wear rate were determined.

- Phase IV – March 2008: Material crushed by hammermill to nominal passing 6.3 mm was supplied for testing. These samples were not tested due to a boron contamination picked up during crushing
- Phase V – 2009: Met 2 samples were crushed in three different types of comminution equipment: a vertical shaft impactor (REMco), a cone crusher (Metso) and an HPGR (KHD Humboldt Wedag). Each sample was crushed to a nominal passing 6.3 mm. Two column tests were performed on each different crushed material
- "All-OX" Tests (Phase VI) – 2012: These tests were conducted on oxidized material selected for the pad overliner. Bottle rolls and column tests were conducted on this material

### 13.1.2 Gold extraction

The response of the Lindero ore to cyanide column leaching in the existing testwork was reviewed. Where test conditions were maintained throughout the test cycle the leach profiles are typical for cyanide leach extraction tests. The extraction curves for selected tests are presented in Figure 13.1. These curves were derived from the raw data from Phase I and Phase III tests. Of note in Figure 13.2 is the lengthy duration of the leach which is atypical for a gold ore.

The samples in Figure 13.2 were crushed to nominal minus 9 mm for leaching. Leach solution application rate was between 10 $l/hr.m^2$ and 15 $l/hr.m^2$. Concentrations of approximately 0.2 gpl NaCN was used in the leach solutions. Tests were run in columns in closed cycle with activated carbon and ranged from approximately 1 m to 4 m tall.

The numbers in the legend on the graph refer to the laboratory test number. The tests without a Met Type designation are from the Phase 1 columns tests.

**Figure 13.1 Typical cyanide leach response curves**

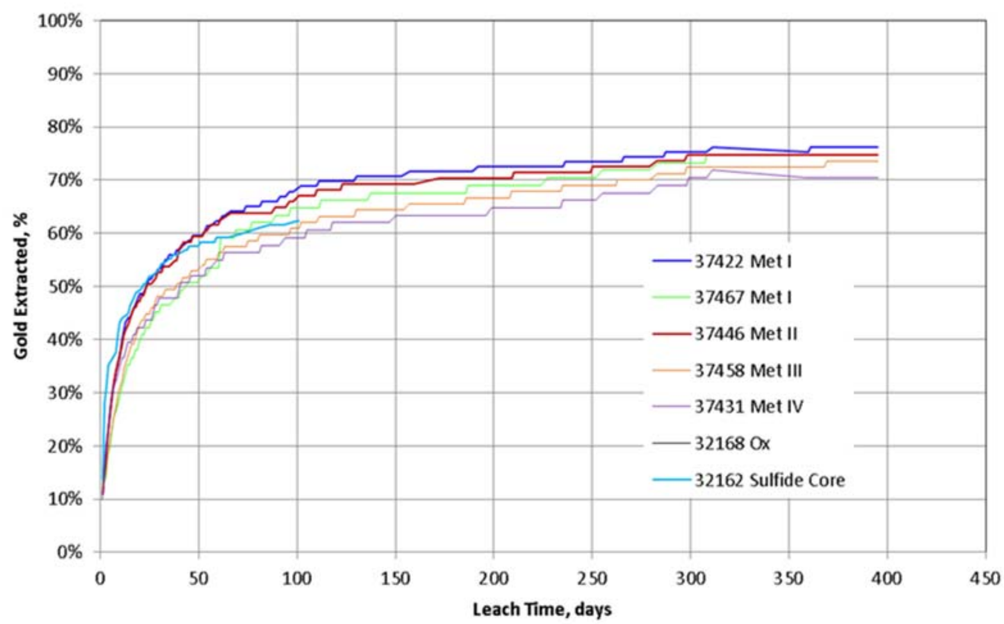



Figure prepared by Allard, 2017

The extractions reported in the 2016 Technical Report were independently reviewed using the Phase III column tests and the results reported in Table 13.3. It is accepted practice to deduct a percentage from the laboratory testwork to accommodate operational inefficiencies. Typically, this deduct value ranges between 2 % and 5 %. The review incorporated a 4 % deduct, the 2016 Technical Report used a variable deduct from 0 % to 5 % depending on ore type with an average deduct of 2.7 %. In any case, the ultimate weighted average field extractions in the review and Technical Report are nearly identical.

**Table 13.3 Reported and reviewed gold extractions from existing testwork**

| Met Type | Review | | 2016 Tech. Report | |
|---|---|---|---|---|
| | Test Au Extraction | Field Au Extraction | Test Au Extraction | Field Au Extraction |
| MET I | 71.2% | 67.2% | 70.0% | 68.0% |
| MET II | 78.6% | 74.6% | 74.0% | 74.0% |
| MET III | 72.3% | 68.3% | 74.0% | 69.0% |
| MET IV | 70.3% | 66.3% | 67.0% | 62.0% |
| Wt'd Avg. | 72.3% | 68.3% | 70.7% | 68.0% |

The column tests analyzed for preparation of Table 13.3 were predominantly Phase III tests. This did not include HPGR-crushed material.

The influence of crushing type on gold extraction was observed with the Phase V (Met II) tests. These tests used a single sample and crushed the material to test-size using three different types of crusher. Gold extraction curves for these tests are reproduced from the 2016 Technical Report and shown in Figure 13.2.

**Figure 13.2 Phase V leach column extraction curves**

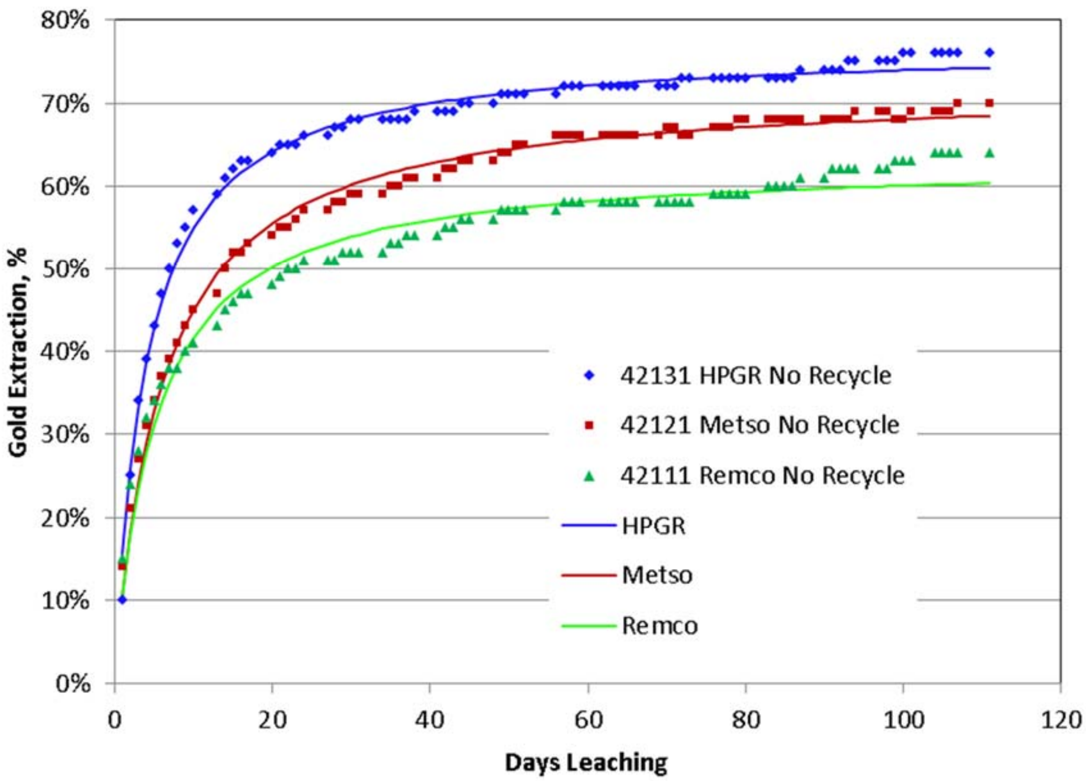



Figure prepared by Allard, 2017

The column charges for the tests shown in Figure 13.2 were initially wet with 0.5 gpl NaCN solution then leached with low concentration cyanide solution (0.1 gpl). After about 40 days of leaching the solution application rate was increased until it was double the rate by day 70. Cyanide concentration was also increased during this period until it was doubled.

Changes in flowrate and cyanide concentration late in the test have little effect on the extraction, which is to be expected. The "pore diffusion controlled" leaching regime which is predominant in the later stages of gold leaching are less affected by flow and reagent concentrations than they are by product concentration gradients. The rationale for this approach was not documented.

Comparing the initial leach rate demonstrated in Figure 13.2 with that shown in Figure 13.1 indicates the rate of extraction for these tests is significantly higher. The other observation from Figure 13.2 is that the high pressure grinding roll (HPGR) appears to achieve higher extraction than the conventional cone crusher or the vertical shaft impactor (VSI).

In order to confirm the recovery improvement of the HPGR the tails screen analysis was plotted (from the lab report raw data) for the three tests as in Figure 13.3. A head screen analysis was not done for these samples, so the differences must be inferred from the tails assays by size. Figure 13.3 shows the tails assay by size for the three tests (solid lines) and the gold distribution (dashed lines).

**Figure 13.3 Tails screen analysis for Phase V column tests**

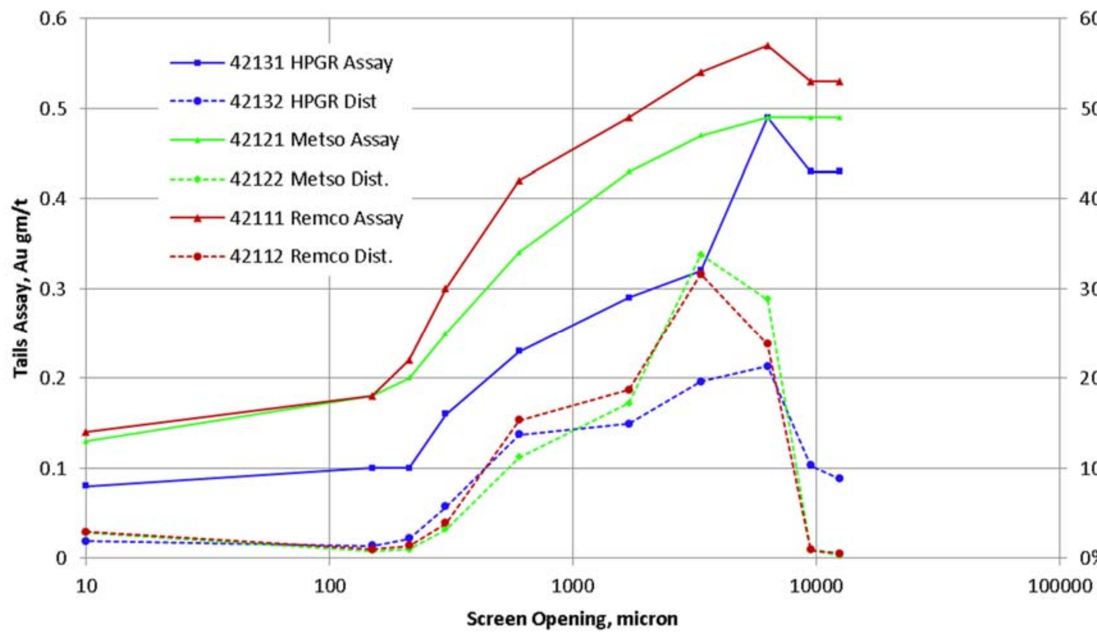



Figure prepared by Allard, 2017

Figure 13.3 demonstrates that the HPGR crushed material has lower tails assays across almost all particle sizes. What is most interesting is that the HPGR crushed material apparently achieves greater extraction even in the finest particle size ranges.

HPGR crushing has been shown to generate micro-fractures in the rocks during crushing. These micro-fractures allow solution to penetrate the rocks more easily and achieve higher extractions. Figure 13.3 may show some evidence of micro-fracturing with the reduced distribution of gold in the coarser fractions.

The crushing action of the HPGR tends to fracture rock along grain boundaries. This action may make for greater liberation of the gold particles, resulting in the increased extraction at both fine and coarse particle sizes.

A few tests were conducted to determine the influence of cyanide concentration on gold extraction. Several bottle roll tests were conducted on splits of the same sample using 0.25 gpl NaCN and 1.0 gpl NaCN respectively. A typical gold extraction response to these conditions is shown in Figure 13.4.

**Figure 13.4 Influence of cyanide concentration on extraction**

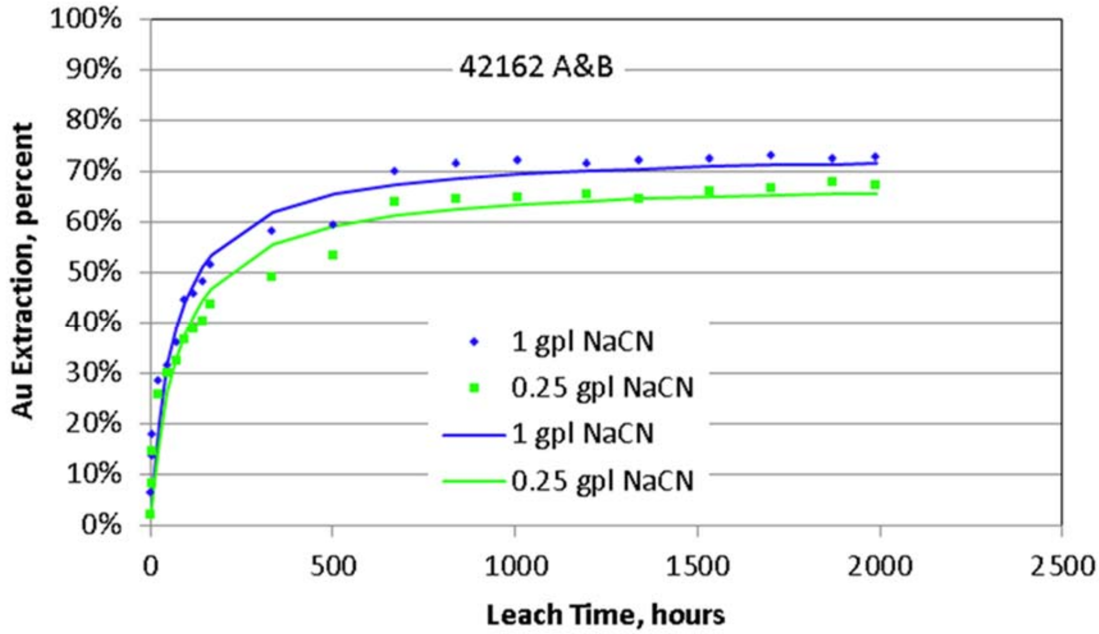



Figure prepared by Allard, 2017

Figure 13.4 shows that an additional 5 % extraction is available with the higher cyanide concentration. The other point to note in Figure 13.4 is that all the additional extraction is obtained in the first quarter of the leach time. After the initial rapid dissolution period the extraction rates are nearly identical as evidenced by the parallel curves. The conclusion is that high cyanide concentrations only have a benefit during the initial "bulk diffusion" rate controlled leach period.

### 13.1.3 Copper/silver extraction

Silver extraction was ignored for most of the historical tests. Silver content of the ore is low, and with the lean cyanide conditions employed little silver dissolved.

Copper in the leach solutions was only sampled periodically. Figure 13.5 shows a typical pattern of copper accumulation for a test where the leach conditions were not changed late during the test. Increasing the solution application rate to the columns late in the leach cycle had the unfortunate side-effect of diluting the copper in solution and resulted in a misinterpretation of the data.

### 13.1.4 Cyanide consumption

It was observed that the cyanide consumption for most of the existing column tests that were analyzed followed a pattern similar to that in Figure 13.6. This is indicative of a leach system that is starved for cyanide. This is the same column depicted in Figure 13.5.

**Figure 13.5 Copper concentration in solution versus time**

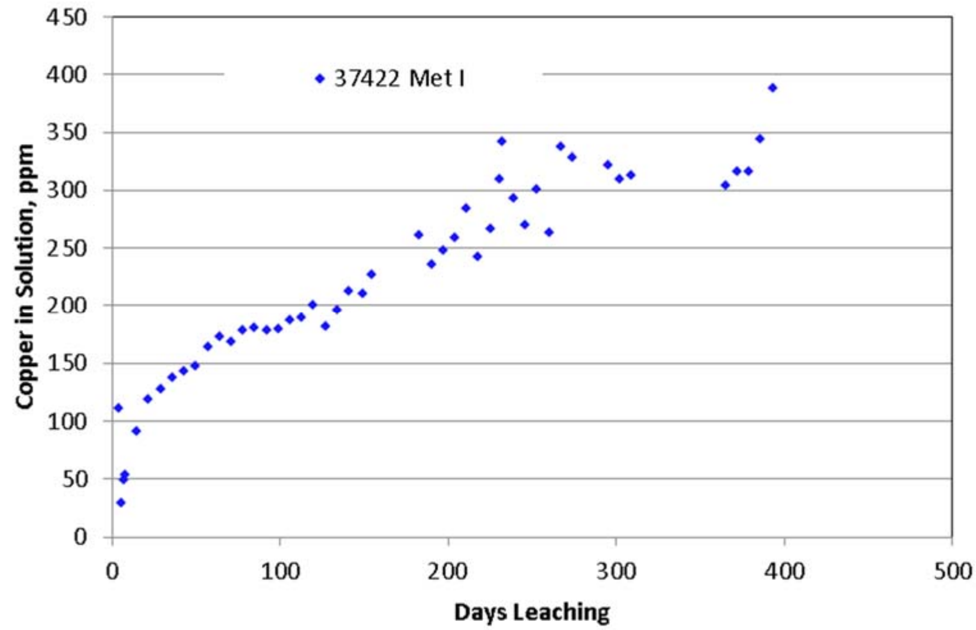



Figure prepared by Allard, 2017

**Figure 13.6 Typical consumption of NaCN versus leach time**

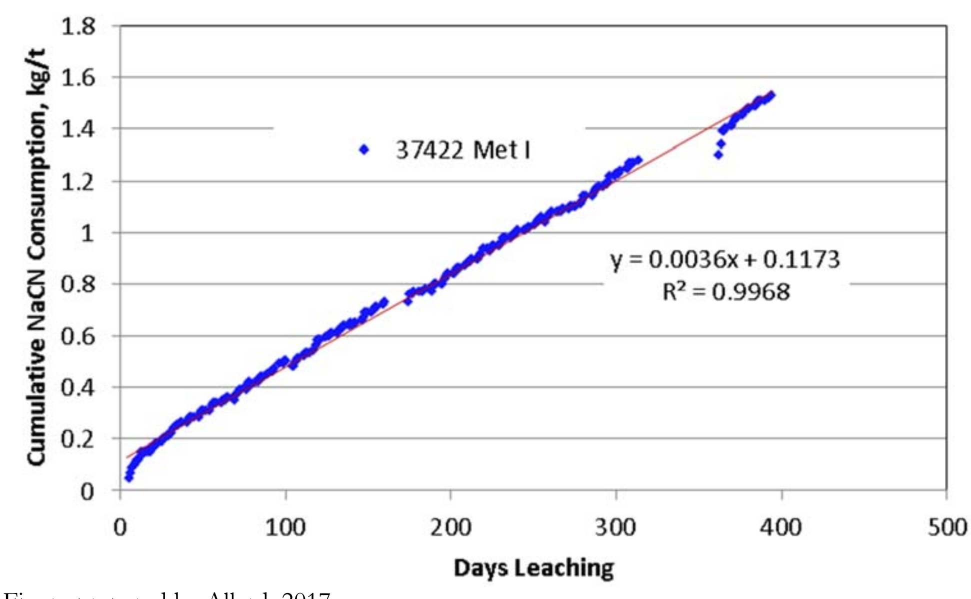



Figure prepared by Allard, 2017

There is a prevailing opinion in the industry that cyanide consumptions from laboratory column tests are much higher than that experienced in the operation on those same ores. This is, for the most part, true. In the case of Lindero ores, this is complicated by the presence of copper in solution. Discussions in the 2016 Technical Report indicate that this consideration will reduce cyanide consumption to 40 % of the test values for operations. This would result in a weighted field consumption of 0.45 kg/t. The cyanide consumptions for Phase III were independently reviewed and the results presented in Table 13.4. The same 40 % reduction factor was applied to both. It is readily apparent that cyanide consumptions are dependent on which data are analyzed.

**Table 13.4 Cyanide consumptions by metallurgical type**

| Met Type | Phase III Review | | 2016 Tech. Report | |
|---|---|---|---|---|
| | Test NaCN (kg/t) | Field NaCN (kg/t) | Test NaCN (kg/t) | Field NaCN (kg/t) |
| MET I | 1.6 | 0.62 | 1.1 | 0.44 |
| MET II | 2.0 | 0.80 | 1.3 | 0.52 |
| MET III | 1.3 | 0.51 | 1.0 | 0.40 |
| MET IV | 1.7 | 0.68 | 1.2 | 0.48 |
| Wt'd Ave. | 1.6 | 0.64 | 1.1 | 0.45 |

The tests depicted in Figure 13.2 show a different cyanide consumption curve than that shown in Figure 13.4. The difference in the shape of the consumption curve is likely due to wetting the ore with 0.5 gpl NaCN during the first days of leaching. The cyanide consumption of the Phase V column tests was complicated by an increase in cyanide concentration late in the leach cycle. Figure 13.7 represents the cumulative cyanide consumption of the Phase V column tests for the period prior to changing the solution concentration.

**Figure 13.7 Phase III - cyanide consumptions by metallurgical type**

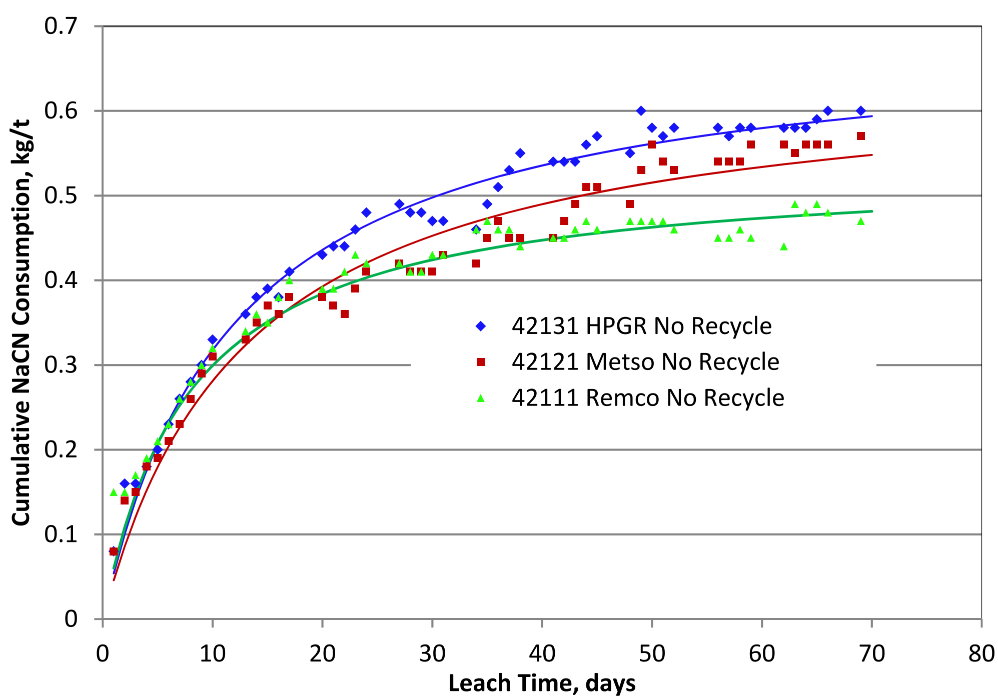



Figure prepared by Allard, 2017

Extrapolating the curve for the HPGR-crushed test to 400 days gives a projected cyanide consumption of 0.675 kg/t. Based on copper extraction results presented in the 2016 Technical Report (Table 13-22) the amount of NaCN complexed with copper would be 0.31 kg/t.

It is readily apparent that cyanide consumption is an important consideration for the Lindero Project.

#### 13.1.5 Additional information

Several investigations were completed to obtain general information for the processing of Lindero ores. These results are taken directly from the 2016 Technical Report and included here for continuity.

Crushing work index (Wi) and abrasion index (Ai) were prepared for the Lindero ore by Hazen Research. The results are shown in Table 13.5. This shows that the Lindero ore is moderately abrasive with an average crushing work index. However, these values must be interpreted with caution since the source material used for these tests was not identified in either the 2016 Feasibility Study or the Hazen report.

**Table 13.5 Abrasion index and crushing work index**

| Test | Abrasion Index | Work Index (kWh/t) |
|---|---|---|
| 1 | 0.4111 | 5.66 |
| 2 | 0.4685 | 5.37 |
| 3 | 0.4846 | 6.93 |
| 4 | 0.5437 | -- |
| 5 | 0.3933 | -- |
| Average | 0.460 | 5.99 |

HPGR wear rates were predicted by KHD Humboldt Wedag (KHD) for the unoxidized material, typical of Met 1 and the oxide material typical of Met 2. The wear rate and roll life is reproduced from the 2016 Technical Study as presented in Table 13.6.

**Table 13.6 HPGR wear rates and roll life**

| Oxidation | Met Type | Specific Gravity | Wear Rate (g/min) | Stud Length | | |
|---|---|---|---|---|---|---|
| | | | | 45 mm | 55 mm | 65 mm |
| Hypogene | Met 1 | 2.69 | 0.0295 | 7,500 hrs | 9,100 hrs | 10,800 hrs |
| Oxide | Met 2 | 2.70 | 0.0147 | 13,300 hrs | 16,200 hrs | 19,000 hrs |
| Oxide | Met 2 | 2.67 | 0.0200 | | | |

HPGR testing by KHD resulted in a recommended specific grinding force of 3.8 N/mm$^2$ to 4.0 N/mm$^2$ for the Lindero ore.

## 13.2 Current metallurgical testwork

Additional testing was completed by Fortuna as part of an investigation to clarify the results of the historical testing. The metallurgical testing reported in this study was supervised by Geoff Allard of Allard Engineering Services LLC of Tucson, Arizona USA.

The metallurgical work conducted by Fortuna is documented in a report "*Lindero Project – Metallurgical Report", 9 September 2017, Allard Engineering Services LLC* (Allard, 2017). This document is referred to as the "Met. Report" in the following discussions.

Physical and metallurgical testing of the Lindero Project samples was conducted using the services of the following independent laboratories:

- Hydrogeosense Inc. (HGS) in Tucson, AZ USA was employed to investigate the physical characteristics of the samples to determine the necessity and extent of agglomeration required to support the design criteria requirement of a 110-meter high heap
- Base Metal Laboratories, Inc. (BML) in Kamloops, BC Canada was employed to determine the metallurgical response of the Lindero samples to high cyanide cure leaching

Test reports for these efforts are included in the metallurgical report and a summary of the investigations is presented below.

### 13.2.1 Samples

Three sets of samples were obtained for this testwork. These samples were collected by Fortuna geologists to represent the four geological types and are identified as Met 1, Met 2; Met 3 and Met 4. These classifications are the same as those presented in the 2016 Technical Report. The samples were as follows:

1. Existing Core: Core drilling at Lindero has been conducted over the course of a decade or more. Core was available from these exercises to prepare a composite of the four met types. These composite samples were crushed with an HPGR at 3.8 $N/mm^2$
2. Tradeoff: This sample was collected by taking low-grade intervals of the existing core to represent Met 1 and Met 2. These were used to determine if the physical characteristics of conventional-crushed (jaw/cone crushing) and lower pressure (3.3 $N/mm^2$) HPGR-crushed material would allow higher heap construction requiring less cement addition. No metallurgical testing was conducted on these samples
3. Fresh Core: These samples were collected in the fall of 2016 from a new drilling program initiated to obtain verification data and metallurgical core. These samples represented Met 1 and Met 4 types. These composite samples were crushed with an HPGR (3.8 $N/mm^2$) and a conventional jaw/cone crusher

An additional Met 1 Fresh Core sample is currently in testing. This sample is being leached under the identical conditions as the Existing Core Met 1 sample (10-m-high, HPGR-crushed, cyanide cure).

Sample locations are identified in Figure 13.8. A summary of the field sampling data is presented in Table 13.7. The weighted average gold and copper grade of the Existing Core and Fresh Core samples are representative of the life of mine average for the various ore types and the style of mineralization of the deposit as a whole.

**Figure 13.8 Sample location plan for 2016-2017 metallurgcal testwork**

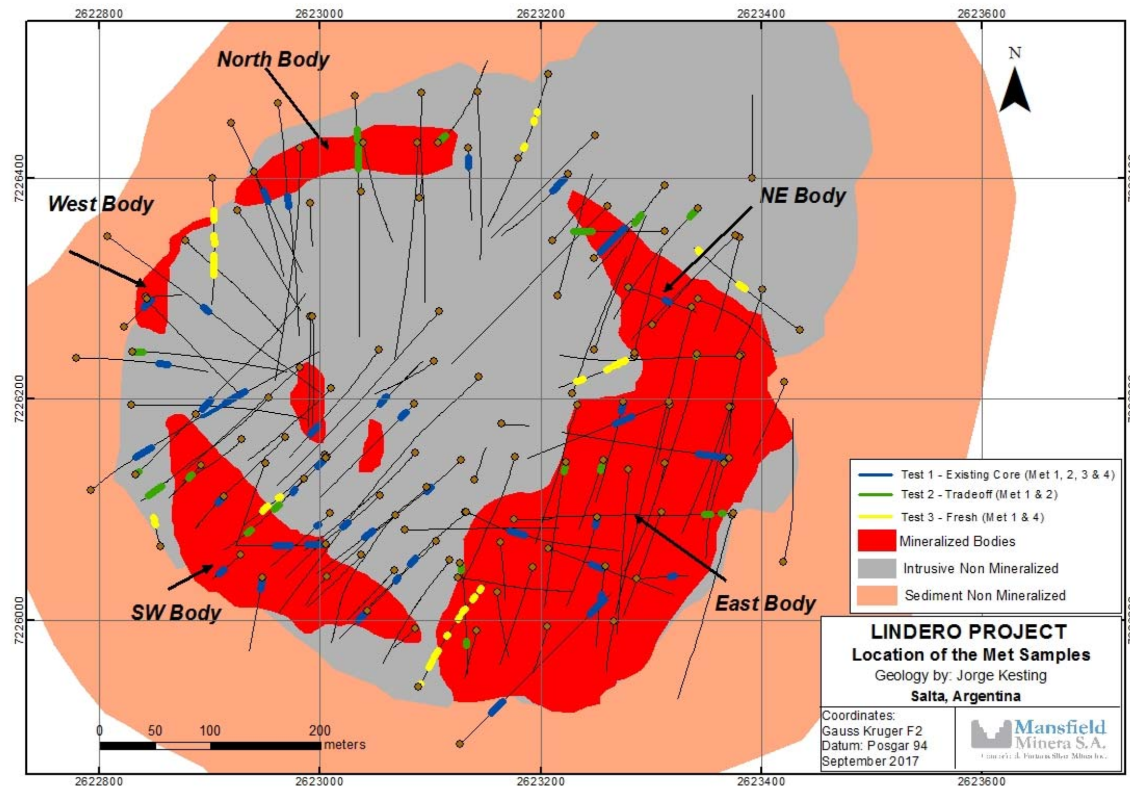



Figure prepared by Mansfield, 2017

**Table 13.7 Characteristics for samples taken in 2016-2017**

| Met Type | Weight (kg) | Meters sampled | No. Drill holes sampled | Sample depth range (m) | Au (g/t) | | | Cu (%) | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Min | Max | Wt'd Avg. | Min | Max | Wt'd Avg. |
| **Existing Core** | | | | | | | | | | |
| 1 | 761.75 | 288 | 10 | 60 - 342 | 0.12 | 1.9 | 0.6 | 0.02 | 0.31 | 0.10 |
| 2 | 786.93 | 306 | 13 | 0 - 52 | 0.14 | 2.4 | 0.73 | 0.02 | 0.40 | 0.13 |
| 3 | 749.87 | 200 | 12 | 10 - 280 | 0.05 | 1.74 | 0.44 | 0.03 | 0.19 | 0.09 |
| 4 | 762.54 | 236 | 11 | 50 - 370 | 0.08 | 1.91 | 0.42 | 0.03 | 0.37 | 0.11 |
| **Tradeoff** | | | | | | | | | | |
| 1 | 318.23 | 194 | 7 | 18 - 276 | 0.11 | 0.82 | 0.34 | 0.02 | 0.27 | 0.10 |
| 2 | 324.79 | 198 | 14 | 0 - 128 | 0.08 | 0.87 | 0.37 | 0.02 | 0.24 | 0.09 |
| **Fresh Core** | | | | | | | | | | |
| 1 | 643.6 | 212 | 6 | 52 - 410 | 0.106 | 3.03 | 0.64 | 0.02 | 0.47 | 0.14 |
| 4 | 609.4 | 182 | 7 | 6.5 - 309 | 0.058 | 1.275 | 0.47 | 0.02 | 0.27 | 0.11 |

### 13.2.2 Sample preparation

Sample preparation was conducted by BML in Kamloops, BC. Where needed, a pilot-scale HPGR crusher at the University of British Columbia was employed. Figure 13.9 shows the particle size distribution (PSD) on a semi-log plot of cumulative percent passing versus screen opening.

**Figure 13.9 Existing Core PSD**

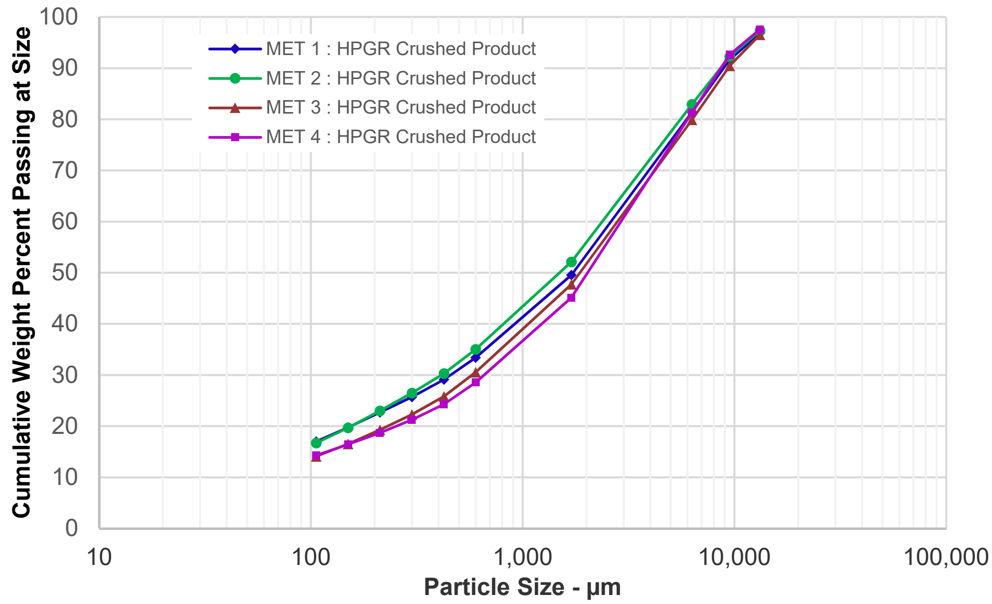



Figure prepared by Allard, 2017

Figure 13.10 shows the PSD for the Tradeoff samples compared to the appropriate Existing Core samples. Figure 13.11 shows the PSD for the Fresh Core sample in comparison to the appropriate Existing Core sample.

**Figure 13.10 Tradeoff sample PSD**

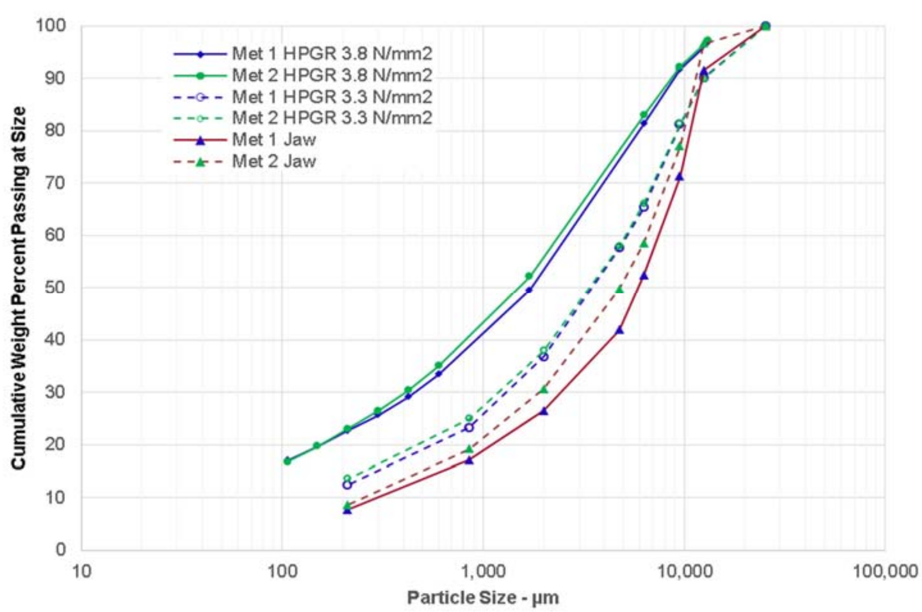



Figure prepared by Allard, 2017

Figure 13.9 shows slight variations in PSD between the various Met types with Met 1 and Met 2 generating slightly more fines than Met 3 and Met 4. Figure 13.10 shows that reducing the pressure of the HPGR creates a coarser product, which is to be expected, and using conventional compression crushing equipment generates even less fines.

Figure 13.11 for Fresh Core shows a similar PSD for the conventionally crushed material as was shown in Figure 13.9. The HPGR crushed material in Figure 13.11 is significantly coarser than the same material in Figure 13.9. This is likely due to the small sample available for crushing in the HPGR at the University of British Columbia.

**Figure 13.11 Fresh Core sample PSD**

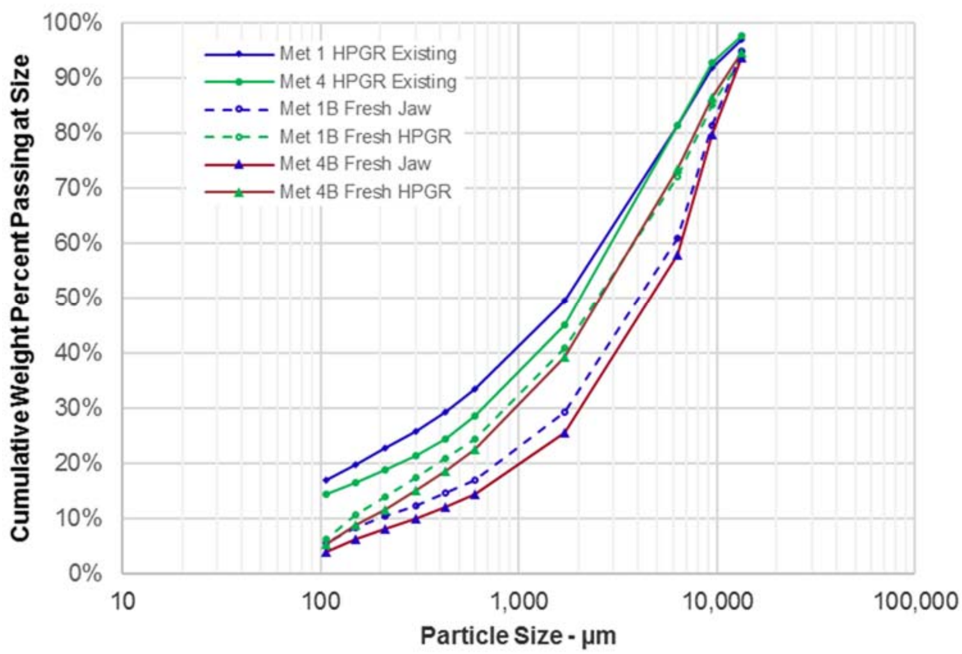



Figure prepared by Allard, 2017

Figure 13.12 shows the distribution of gold, silver and copper by size in the Met 1 HPGR-crushed Existing Core sample. Of particular note is that the gold, silver and copper are distributed almost identically which indicate that the minerals hosting the three metals are intimately associated. This pattern is typical of the materials tested.

**Figure 13.12 Metals distribution by size – Met 1 Existing Core HPGR-crushed**

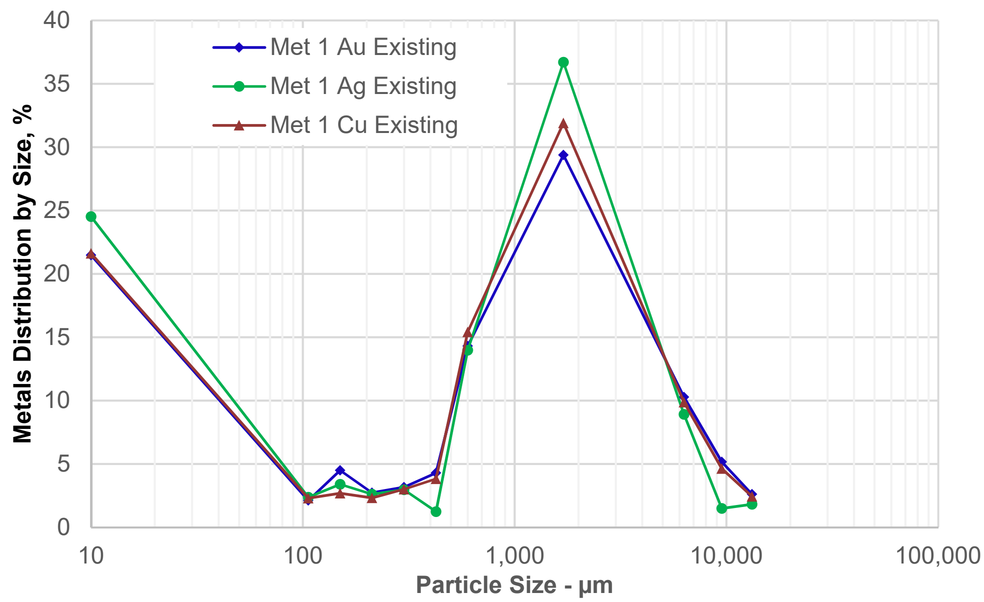



Figure prepared by Allard, 2017

### 13.2.3 Physical testing

Two types of tests were employed by HGS in investigating the Lindero samples. One test is a "stacking test" where the agglomerated material is placed under load and the saturated flow determined. The porosity and bulk density is determined at each loading up to the expected heap height. Seventy-eight stacking tests were conducted on Lindero ore testing various conditions of cement addition and cement type. Most of the work was conducted using Portland Type II cement as a basis, however, locally available cement (IRAM specification CPP40 and CPF40) was also tested. These local cements showed equally good agglomeration properties.

The second test employed by HGS was a "hydrodynamic column" test. In this test, the ore is placed in a column in lifts and each lift is loaded to achieve the same bulk density as experienced in the stacking tests. Two columns per sample were tested representing a 10-m lift and a 100-m loading height. The test columns are suspended on load cells so that the mass of the column is continuously monitored. The columns are leached at various rates up to the design flow of 12 $l/hr/m^2$. Knowing the porosity of the charge and the mass of solution in the charge allows percent saturation and holdup to be calculated. Sixteen hydrodynamic columns were run on the Lindero ore.

Figure 13.13 shows a typical set of stacking test curves. The curves show the residual porosity (void volume) of the Met 1 Existing Core agglomerated at various levels of cement addition versus loading represented as heap height. The porosity is presented on a dry ore basis. Available void volume (in Figure 13.12 is reduced by adsorbed moisture, active holdup at 12 $l/hr/m^2$ and the criteria that 25 % to 30 % of the total void volume must remain (70 % to 75 % saturation limit) during leaching to avoid phreatic head in the heap. As a rule-of-thumb when the total void volume is less than 30 % the void volume is insufficient to maintain heap stability. At a criteria range of 70 % to 75 % this equates to 210 to 225 l/m$^3$ of heap available for absorption (moisture that is held in place by surface tension after wetting) and active holdup. These values will vary for every sample depending on the fines and level of agglomeration.

**Figure 13.13 Typical stacking test curves**

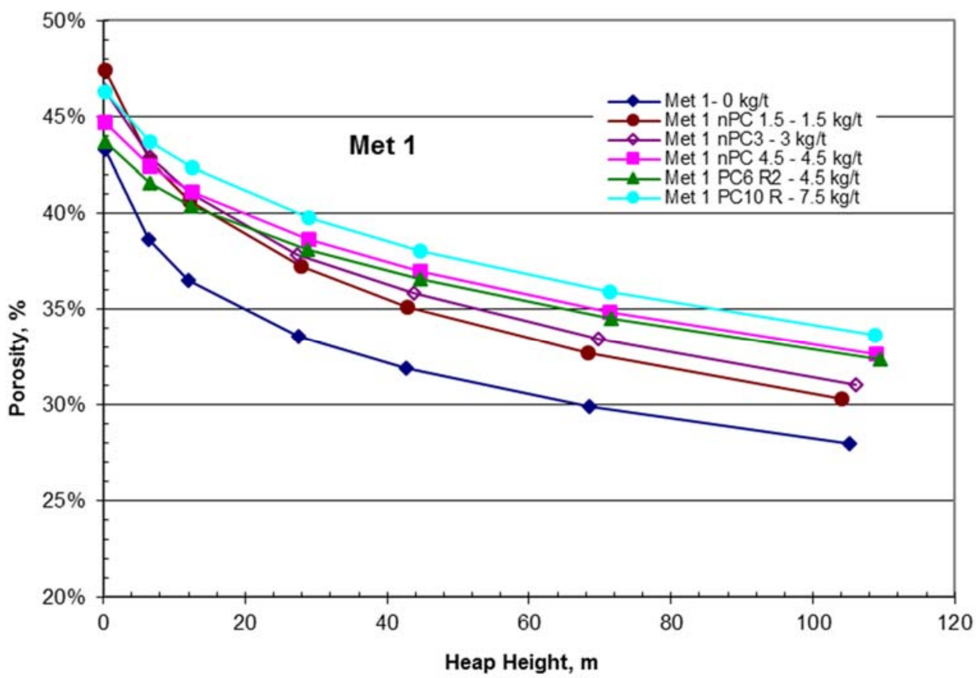



Figure prepared by Allard, 2017

Table 13.8 shows the hydrodynamic column data for the existing core. This is typical of the data generated by this test. As can be noted the volumetric moisture content in l/m$^3$ is very similar to the rule-of-thumb expectations noted above.

**Table 13.8 Typical hydrodynamic column data**

| Property | Met 1 at 12 l/h/m$^2$ | | Met 2 at 12 l/h/m$^2$ | | Met 3 at 12 l/h/m$^2$ | | Met 4 at 12 l/h/m$^2$ | |
|---|---|---|---|---|---|---|---|---|
| | 10 m | 100 m | 10 m | 100 m | 10 m | 100 m | 10 m | 100 m |
| Cement addition (kg/t) | 4.5 | 4.5 | 4.5 | 4.5 | 1.5 | 1.5 | 1.5 | 1.5 |
| Dry bulk density (m$^3$/t) | 1.598 | 1.834 | 1.545 | 1.773 | 1.66 | 1.847 | 1.656 | 1.854 |
| Ore SG | 2.717 | 2.717 | 2.727 | 2.727 | 2.734 | 2.734 | 2.69 | 2.69 |
| Porosity (%) | 42.3 | 33.4 | 44.5 | 35.7 | 40.4 | 33.3 | 39.5 | 32.3 |
| Dynamic holdup (l/m$^3$) | 46 | 64 | 52 | 43 | 43 | 43 | 49 | 48 |
| Static inventory (l/m$^3$) | 173 | 175 | 157 | 204 | 162 | 178 | 143 | 148 |
| Vol. moisture content (l/m$^3$) | 218 | 239 | 209 | 247 | 205 | 222 | 193 | 196 |
| Liquid saturation (%) | 52.6 | 72.5 | 47.5 | 69.9 | 51.6 | 68.0 | 48.4 | 59.8 |

The prime consideration in determining the suitability of conditions for leaching in the proposed multi-lift heap leaching is the retention of porosity under load. Fine crushing and agglomeration increases the volume of solution that is held within the heap after free draining. This moisture reduces the void space available for the active holdup of leaching solutions and generates high saturations (percent of available voids filled by solution). Higher levels of adsorbed moisture will also increase the inventory as the heap height increases.

Analyses of the testwork by HGS in the Met Report indicate that the allowable heap height is dependent on several factors. These factors are:

1. Solution application rate
2. Metallurgical type
3. Crushing equipment type
4. HPGR crushing force
5. Crushed particle size
6. Cement addition

The estimated allowable heights for crushed material are listed in Table 13.9. The number after HPGR in the table represents the specific grinding force in N/mm². The "minimum porosity" is the minimum void volume, as a percentage of the total ore plus voids volume, that will allow a maximum of 70 % saturation at 12 l/hr/m² (and represents a magnitude similar to the 210 to 225 l/m³ above for the Existing Core).

**Table 13.9 Allowable heap heights at 70 % saturation**

| Existing Designation | Crusher Type | Nom. $P_{80\ mm}$ | Minimum Porosity (%) | Cement Addition (kg/t) | | Heap Height (m) | |
|---|---|---|---|---|---|---|---|
| | | | | Min. | Max. | Min. | Max |
| Met 1 | HPGR 3.8 | 6.0 | 23.6 | 0.5 | 6.0 | 49 | 97 |
| Met 2 | HPGR 3.8 | 5.6 | 24.6 | 0.5 | 4.75 | 57 | 98 |
| Met 3 | HPGR 3.8 | 6.3 | 22.4 | 0.25 | 1.5 | 100 | 117 |
| Met 4 | HPGR 3.8 | 6.0 | 19.7 | 0.25 | 0.25 | 135 | 135 |
| **Tradeoff Designation** | **Crusher Type** | **Nom. $P_{80\ mm}$** | **Minimum Porosity (%)** | **Cement Addition (kg/t)** | | **Heap Height (m)** | |
| | | | | **Min.** | **Max.** | **Min.** | **Max** |
| Met 1H | HPGR 3.3 | 9.3 | 18.6 | 0 | 0 | 124 | 124 |
| Met 1J | Jaw/Cone | 10.8 | 16.4 | 0 | 0 | 110+ | 110+ |
| Met 2H | HPGR 3.3 | 9.2 | 18.2 | 0 | 0 | 167 | 167 |
| Met 2J | Jaw/Cone | 10.0 | 14.7 | 0 | 0 | 110+ | 110+ |
| **Fresh Designation** | **Crusher Type** | **Nom. $P_{80\ mm}$** | **Minimum Porosity (%)** | **Cement Addition (kg/t)** | | **Heap Height (m)** | |
| | | | | **Min.** | **Max.** | **Min.** | **Max** |
| Met 1H | HPGR 3.8 | 8.1 | 18.4 | 0 | 1.5 | 112 | 119 |
| Met 1J | Jaw/Cone | 9.3 | N/A | 0 | 0 | 110+ | 110+ |
| Met 4H | HPGR 3.8 | 7.8 | 18.4 | 0 | 1.0 | 116 | 145 |
| Met 4J | Jaw/Cone | 9.6 | N/A | 0 | 0 | 110+ | 110+ |

It is apparent that coarser crushing, whether using conventional equipment or an HPGR operating at a lower pressure, will allow stacking to 110 m high without agglomeration with cement. This is accompanied by a reduction in extraction.

### 13.2.4 Leach testing

A series of bottle roll tests and 32 small column tests were conducted by BML on the existing core to define the leach parameters for the cyanide cure and leach application rate for the 10 m leach columns. Agglomeration conditions were determined from the HGS tests. Conditions for the leach columns were developed as in Table 13.10.

**Table 13.10 Leach column conditions**

| Existing Test | 10-meter Type | Weight (kg) | Lime (kg/t) | Cement (kg/t) | NaCN (kg/t) | Moisture% (w/w) | Cure days | Leach (l/hr/m²) | Leach days | Red. Flow at day No. | pH Target |
|---|---|---|---|---|---|---|---|---|---|---|---|
| CL-01 | Met 1 | 443.4 | 0 | 4.5 | 0.75 | 7.10 | 4 | 12 | 101 | 40 | 10-10.5 |
| CL-02 | Met 2 | 432.8 | 0 | 4.5 | 0.75 | 6.90 | 4 | 12 | 101 | 54 | 10-10.5 |
| CL-03 | Met 3 | 475.9 | 0 | 1.5 | 0.75 | 5.90 | 4 | 12 | 101 | 39 | 10-10.5 |
| CL-04 | Met 4 | 446 | 0 | 1.5 | 0.75 | 5.10 | 4 | 12 | 89 | 39 | 10-10.5 |
| **Fresh Test** | **5-meter Type** | **Weight (kg)** | **Lime (kg/t)** | **Cement (kg/t)** | **NaCN (kg/t)** | **Moisture% (w/w)** | **Cure days** | **Leach (l/hr/m²)** | **Leach days** | **Red. Flow at day No.** | **pH Target** |
| CL-05 | Met 1B Jaw | 222 | 0 | 4.5 | 0.75 | 6.50 | 4 | 12 | 95 | 31 | 10-10.5 |
| CL-06 | Met 1B HPGR | 222 | 0 | 4.5 | 0.75 | 7.10 | 4 | 12 | 80 | 30 | 10-10.5 |
| CL-07 | Met 4B Jaw | 222 | 0 | 1.5 | 0.75 | 5.00 | 4 | 12 | 90 | 30 | 10-10.5 |
| CL-08 | Met 4B HPGR | 222 | 0 | 1.5 | 0.75 | 5.00 | 4 | 12 | 92 | 30 | 10-10.5 |

Column test charges were weighed to ±30 kg and placed in a ¼ $m^3$ cement mixer with the lifters removed. A known amount of Portland Type II cement was added to the charge, the mixer sealed and rotated to mix the components. The mixing required roughly 30 seconds. A measured amount of concentrated cyanide solution was then added, and the contents again mixed. Sufficient tap water was then placed in the mixer to achieve the desired agglomeration moisture and the mixer rotated for 2 to 3 minutes more. During this time, any material adhering to the drum was scraped off and coarse agglomerates migrating to the periphery of the drum were manually reintroduced to the tumbling mass.

Once the agglomeration was complete the charge was tipped out into buckets and lifted to the top of the section of column being loaded. This was repeated until the leach columns were filled.

Each test column was divided into 2.5 m flanged sections to ease loading. A 75 mm tube was lowered into the center of the column and filled with green agglomerates. As more agglomerates were added to the top the tube was lifted to allow the agglomerates in the bottom to discharge gently and uniformly into the column. This was continued until each section was filled and a new section added.

Barren solution was made from tap water, adjusted to pH 10 to 10.5 with lime and made up to 0.25 gpl NaCN in a 5-gallon bucket. A separate bucket was made for each column. The bucket was weighed every morning to determine how much solution was applied to each column in the preceding 24 hours. Pregnant solution was allowed to discharge freely from the bottom of the column and was collected in a bucket. This bucket was weighed each day, sampled and replaced with an empty bucket. Activated carbon was added to the pregnant solution stirred, until the next morning when the carbon was removed, and added to the next day's pregnant solution. The solution after contact with the activated carbon was added to the barren solution bucket and made up for feed to the column. Each bucket was sampled for gold, silver, copper, pH, oxidation-reduction-potential (ORP) and weakly acid dissociable (WAD) cyanide.

Initial solution application rate was set at 12 l/hr/$m^2$ until the bulk of the extraction was achieved, then the application rate was reduced to 6 l/hr/$m^2$ to simulate a secondary leach cycle. The day on which the flow was reduced is noted in Table 13.10.

### 13.2.5 Gold extraction

Figure 13.14 shows that the cyanide cure creates a very rapid initial extraction rate and the bulk of the extractable gold is obtained in the first 20 days of leaching. Met 3 and 4 are almost completely leached by day 40, whereas Met 1 and 2 evidently contained a slower leaching component that continues at a constant, albeit slow, rate for the duration of leaching. It is important to note in Figure 13.13 that the rate of extraction (slope of the curve) for Met 1 and 2, immediately before and after changing to 6 l/hr/m² is unchanged.

The slight increase in extraction at the end of the curves is due to rinsing the column to extract all the gold in the entrained solution.

**Figure 13.14 Existing core 10-m column gold extractions**

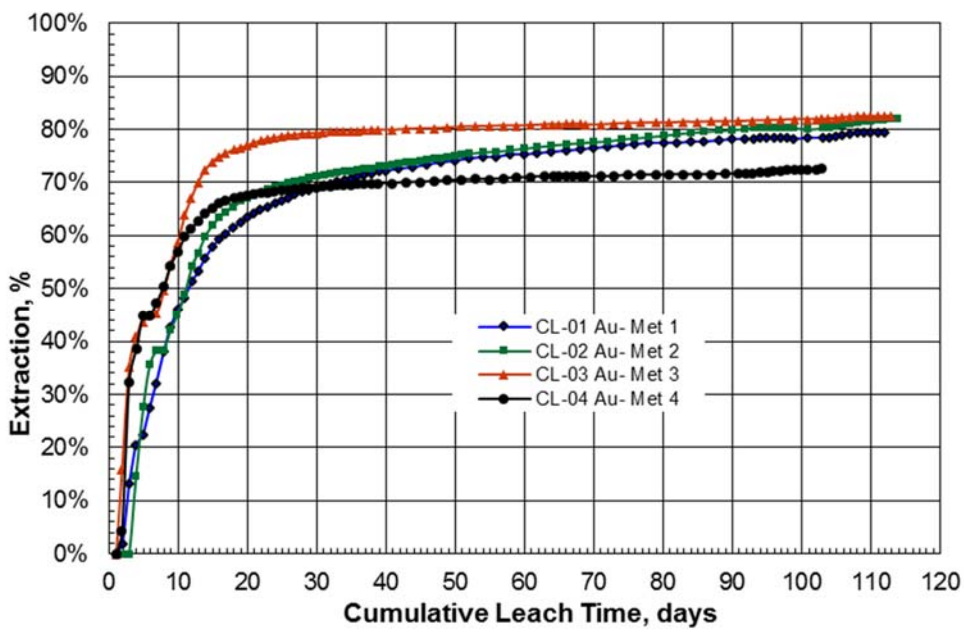



Figure prepared by Allard, 2017

Figure 13.15 shows the gold extraction curves for the Fresh Core medium columns and shows a similar rapid extraction response to the cyanide cure that was observed in the Existing Core tall columns. As with the tall column tests, the bulk of the extractable gold is obtained in the first 20 days of leaching. All the columns show slow leaching after day 20 up to day 60. Around day 30, the application rate was changed to 6 l/hr/m², and it should be observed that the rate of gold extraction was unaffected.

The fresh core 5-m columns were operated in locked cycle with carbon in an identical manner as the existing core tall columns. Carbon was added more frequently to reduce the risk of breakthrough of gold on the carbon. It is apparent that this was inadequate to prevent some disruption in the gold extraction curve for CL-08. As with the earlier columns, some silver and a significant quantity of copper was returned to the barren feed tank.

As is apparent from Figure 13.15, HPGR crushing allows increased extraction of gold from the ore. The Met 1 HPGR test (CL-06) resulted in an increase in gold extraction of 11.3 % over jaw-crushed Met 1. HPGR crushing of Met 4 resulted in a 3.3 % increase in gold extraction over jaw-crushed Met 4.

**Figure 13.15 Fresh Core 5-m column gold extractions**

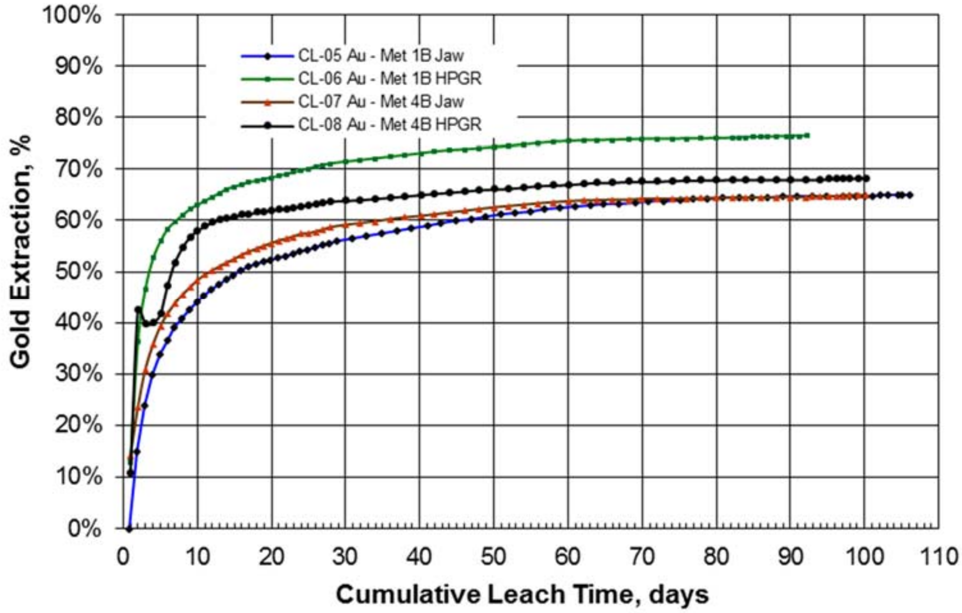



Figure prepared by Allard, 2017

Metals extraction from the major column test programs is summarized in Table 13.11. This data does not include field deduct to adjust from laboratory to field extractions.

**Table 13.11 Extractions from leach columns**

| Existing Test | Type | Head Assay | | | Tails Assay | | | Calculated Head | | | Extraction (%) | | | NaCN (kg/t) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Au (g/t) | Ag (g/t) | Cu (%) | Au (g/t) | Ag (g/t) | Cu (%) | Au (g/t) | Ag (g/t) | Cu (%) | Au | Ag | Cu | |
| CL-01 | Met 1 | 0.512 | 0.267 | 0.10 | 0.110 | 0.100 | 0.08 | 0.535 | 0.349 | 0.10 | 79.4 | 71.3 | 11.8 | 0.477 |
| CL-02 | Met 2 | 0.659 | 0.467 | 0.13 | 0.117 | 0.166 | 0.09 | 0.656 | 0.500 | 0.13 | 82.2 | 66.8 | 28.9 | 0.616 |
| CL-03 | Met 3 | 0.440 | 0.667 | 0.10 | 0.074 | 0.390 | 0.08 | 0.423 | 0.751 | 0.09 | 82.5 | 48.1 | 9.8 | 0.441 |
| CL-04 | Met 4 | 0.371 | 0.200 | 0.11 | 0.110 | 0.100 | 0.12 | 0.401 | 0.292 | 0.13 | 72.5 | 65.8 | 8.6 | 0.419 |
| **Fresh Test** | **Type** | **Head Assay** | | | **Tails Assay** | | | **Calculated Head** | | | **Extraction (%)** | | | **NaCN (kg/t)** |
| | | **Au (g/t)** | **Ag (g/t)** | **Cu (%)** | **Au (g/t)** | **Ag (g/t)** | **Cu (%)** | **Au (g/t)** | **Ag (g/t)** | **Cu (%)** | **Au** | **Ag** | **Cu** | |
| CL-05 | Met 1B Jaw | 0.589 | 0.417 | 0.13 | 0.176 | 0.100 | 0.11 | 0.503 | 0.285 | 0.12 | 65.0 | 64.9 | 8.5 | 0.499 |
| CL-06 | Met 1B HPGR | 0.649 | 0.417 | 0.14 | 0.135 | 0.170 | 0.14 | 0.571 | 0.402 | 0.15 | 76.3 | 57.7 | 9.9 | 0.483 |
| CL-07 | Met 4B Jaw | 0.444 | 0.586 | 0.10 | 0.149 | 0.163 | 0.10 | 0.425 | 0.444 | 0.11 | 65.0 | 63.3 | 9.0 | 0.508 |
| CL-08 | Met 4B HPGR | 0.472 | 0.681 | 0.11 | 0.133 | 0.273 | 0.09 | 0.420 | 0.636 | 0.10 | 68.3 | 57.1 | 8.8 | 0.511 |

Any benefit obtained by coarse crushing to improve the heap stability is offset with a reduction in extraction. Table 13.11 shows that the difference in gold extraction using conventional crushing as opposed to HPGR crushing (CL-01 to CL-05) is as high as 14.4 % for Met 1 and 7.5 % for Met 4 (CL-04 to CL-07).

The reduction between the Existing HPGR-crushed material in Table 13.11 and the same Met type for the Fresh HPGR-crushed material (CL-01 to CL-06 and CL-04 to CL-08) is likely due to reduced grinding efficiency with the smaller sample size.

### 13.2.6 Copper/silver extraction

Silver extraction was monitored throughout the leach tests. Table 13.11 shows that silver is extracted to a similar magnitude as gold. This is likely due to the current test column not being starved for cyanide. Also, the presence of electrum in some samples indicate that gold extraction is somewhat dependent on silver extraction.

Percent copper extraction, for the Existing Core tests, ranges from a low of 8.6 % for Met 4 to a high of 28.9 % for Met 2. However, the copper represented by these extractions are 400 to 700 times higher than the gold extracted. It is apparent copper has a significant role to play in the Lindero Project.

Figure 13.16 shows the copper concentration in the recirculating solution for the 10-m column tests. It is apparent the cyanide cure accelerates the early copper dissolution but the concentration in solution continues to increase over the duration of the leach.

**Figure 13.16 Copper concentration in solution for existing core columns**

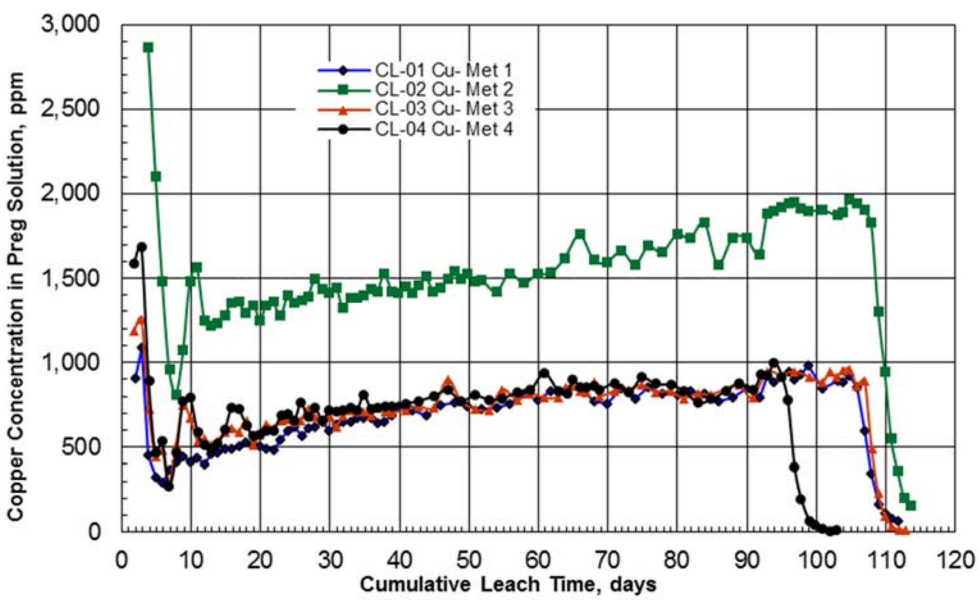



Figure prepared by Allard, 2017

### 13.2.7 Cyanide consumption

Consumption of cyanide is included in Table 13.9. This consumption is based on having a SART plant to recover cyanide as free cyanide for further use in the process. Cyanide consumption for the 5-m columns shows a slight increase that is not unexpected for smaller column tests.

### 13.2.8 Weakly acid dissociable (WAD) cyanide determinations

Pregnant and barren leach solutions generated during laboratory column and bottle-roll tests were analyzed for WAD cyanide. WAD cyanide was the preferred method for determining cyanide, since elevated copper levels in solution render the conventional free cyanide analysis inaccurate (free cyanide by silver nitrate titration). The WAD cyanide analytical method selected for the work was the MP-WAD technique as described by Botz et al. (2013). WAD cyanide QAQC standards containing sodium cyanide and copper cyanide were routinely analyzed during the work.

### 13.2.9 SART laboratory testwork

Batch laboratory SART tests were conducted using samples of barren solution generated from column leach tests for the four Met types. The primary objective of the tests was to generate copper-sulfide solids that could be assayed for metals content. The assayed composition of the solids would be used to negotiate a contract for future sales of the SART copper-sulfide product. The efficiencies of copper and silver recoveries achieved during the tests are summarized in Table 13.12. Copper recoveries ranged from 88 % to 95 %, while silver recoveries ranged from 92 % to 98 %. These results are consistent with expected performance of the SART process.

**Table 13.12 Results from laboratory SART tests**

| Test No. | NaHS (% Stoich.) | pH | Recovery | |
|---|---|---|---|---|
| | | | Copper (%) | Silver (%) |
| MET-1-1 | 121 | 3.80 | 91 | - |
| MET-1-2 | 120 | 3.50 | 89 | 97 |
| MET-2-1 | 120 | 3.90 | 95 | - |
| MET-2-2 | 120 | 3.80 | 90 | 98 |
| MET-3-1 | 120 | 3.70 | 94 | - |
| MET-3-2 | 120 | 3.80 | 88 | 98 |
| MET-4-1 | 120 | 3.80 | 92 | - |
| MET-4-2 | 120 | 3.80 | 89 | 92 |

Copper-sulfide solids generated in the laboratory SART tests were assayed for metals content and the results are summarized in Table 13.13. Since the tests were conducted with barren solution rather than pregnant solution, grades of gold and silver in the solids are relatively low. Higher grades of gold and silver are expected in the actual material and this was taken into account when developing the mass balance for the SART plant. The assayed copper content in the solids ranged from 59 % to 74 %, which is consistent with solids generated from other SART plants.

**Table 13.13 Assays of SART copper–sulfide solids**

| Test No. | Cu (%) | Au (ppm) | Ag (ppm) | Zn (ppm) |
|---|---|---|---|---|
| MET-1 | 65.5 | 0.2 | 188 | 368 |
| MET-2 | 58.8 | 0.1 | 84 | >5,000 |
| MET-3 | 73.6 | 0.1 | 79 | 185 |
| MET-4 | - | 0.2 | 86 | 1,860 |

### 13.2.10 Carbon adsorption

Gold loading on carbon was lower than anticipated during the column tests. Carbon loading equilibrium has a significant impact on the adsorption, desorption, recovery (ADR) plant design and the heap inventory. In order to clarify the loading levels of the leach test solutions, carbon adsorption isotherms were generated at BML. Forty-four tests were conducted to investigate the carbon loading.

Residual leach solutions were processed through a bench-scale SART process to simulate the level of constituents in solution that might be experienced in the field. These solutions were reduced in gold from the locked cycle leach and reduced in silver and copper due to the SART process. These solutions were spiked with gold to roughly 2 ppm and assayed for gold, silver, copper and WAD cyanide.

The carbon and leach solution were combined in a jar which was capped and placed on a rolling table for a fixed amount of time. Contact time is typically set at 24 hours; however, the low gold loading rate prompted some tests to be run for 48 hours.

At the end of the contact time the carbon was filtered out and the filtrate was assayed for gold, silver, copper and WAD cyanide.

Adsorption coefficients of the Freundlich Adsorption Equation were calculated using the regression tools in Excel™. The Freundlich Equation is:

$$\frac{x}{m} = \mathrm{K}c^{1/n}$$

Where:

x is the mass of gold adsorbed on the carbon in grams

m is the mass of carbon in tonnes

K is a constant equivalent to, in this case, the adsorbed gold in g/t of carbon at equilibrium with 1 ppm in solution

c is the concentration of gold in solution in g/t (ppm)

n is a constant relating to the properties of the carbon

It is apparent that the loading is controlled by bulk diffusion limitations inherent in the test procedure. This has been mitigated by mixing the carbon and solutions more vigorously. The increase in gold loading on carbon with extended contact time may be an indication that the influence of bulk diffusion has not been entirely overcome with the increased agitation in the tests and higher agitation tests are in progress.

Testing shows that the loading is insensitive to pH and copper concentrations expected from a SART process. Gold loading on carbon has a slight inverse dependence with free cyanide concentration.

Testing to date results in a set of Freundlich isotherms as follows:

For 24 hr contact time:

$$\frac{x}{m} = 2250 \times c^{\frac{1}{3.46}}$$

For 48 hr contact time:

$$\frac{x}{m} = 3405 \times^{\frac{1}{2.38}}$$

The above equations are adequate for design of a conventional carbon-in-column circuit. It is anticipated the factors that influence a batch bench scale test will not be as prevalent in a continuous fluidized bed carbon adsorption column in the operation. However, carbon column designs will be optimized to maintain elevated interstitial solution velocities.

Additional carbon loading studies are being conducted using site water to introduce site conditions into the leach and carbon adsorption.

## 13.3 Heap leach modeling

The heap leach and associated processes were modeled using METSIM simulation software (version 2015.08) to develop a dynamic mass balance for the life of the operation. The primary objectives were to estimate monthly metal productions for gold, silver and copper and to evaluate the buildup of copper that will occur in leach solution.

For the model, the heap leach was divided into about 1,700 3-D blocks to represent the geometric shape of the heap as it is stacked with ore. The heap blocks were sequentially filled with ore according to the mine plan, and leach extractions for gold, silver and copper were performed according to extraction curves established for each ore type.

Solution transport through the heap lifts was calculated according to the primary and secondary applications of leach solution to ore. The dynamic model tracked inventories in heap pore solution for water, cyanide and metals (Au, Ag, and Cu). Solution exiting the heap was managed for preg-building by routing the highest gold tenors to the pregnant leach solution (PLS) tank, while lower tenor solution was recycled and applied to the heap as intermediate leach solution (ILS).

A daily mass balance was calculated for the heap leach, solution ponds/tanks, the ADR plant and the SART plant. The size of the SART plant was varied in early modeling trials and it was determined the SART plant will be designed to treat 400 $m^3$/hr of pregnant solution. The model was developed according to the configuration of systems shown on the overall process flow diagram as detailed in Figure 17.2 in Section 17.

## 13.4 Conclusions

It is most apparent that the HPGR crusher is of significant benefit to the metallurgical performance of the Lindero ore. HPGR crushing also creates a condition for some of the met types where agglomeration with moderate quantities of cement is required to sustain a 110-meter heap height.

The cyanide cure dramatically increases the rate of extraction of metals from the Lindero ore.

Copper dissolution from the ore is significant and a SART plant will be required.

Carbon loading is lower than might be expected but sufficient to design a conventional carbon plant to recover gold from solution.

On the surface it would seem apparent that a single 10 m column per Met type operated in support of the proposed flowsheet is an insufficient quantity to support a decision to proceed with the Lindero Project. There is significant historical testing that shows that the Lindero ore is amenable to cyanide leaching. The current metallurgical testwork is a continuation of the prior work and, as such, does not stand alone.

## 13.5 Additional work

Some additional work is currently in progress. These tests are:

- A 10 m Met 1 fresh core column is currently under leach. This column is identical to the existing core Met 1 column, CL-01
- Several small columns designed to compare the effects of site water and local cement with the base condition of Portland Type II and tap water
- A leach column to generate preg solution using site water for additional SART and carbon adsorption tests
- Carbon adsorption tests investigating the effects of pH, free cyanide and copper concentration
- Mineral Liberation Analysis (MLA) of leach column tails and a feed concentrate

In addition to the tests in progress a few other avenues of investigation are recommended.

- The cement in each lift on the heap will cure for several months before another lift is placed. It may be several years before any block of agglomerated ore receives 110 m of loading. It is recommended that a long-term stacking test be conducted to see if ageing will improve the ability of the ore to support the 110 m height with less cement
- The high static holdup (adsorbed moisture) in the heap makes the secondary leach at 6 $l/hr.m^2$ inefficient when the heap height increases. There is a possibility that a surface tension modifier may reduce the amount of adsorbed moisture in the heap reducing the inventory

## 13.6 Comments on Section 13

It is the opinion of the QP that the Lindero Project has an extensive body of metallurgical investigation comprising several phases of testwork as indicated in the KCA (2016a) Technical Report. In general, the testwork was done to industry standards. However, some leach conditions set for the testwork made interpretation difficult. Reinterpretation of the raw test data provided the basis for advancing the metallurgical knowledge base for Fortuna.

In the opinion of the QP the 2017 work was consistent with industry practices and provided the basis for optimization of the process design at Lindero.

It is the opinion of the QP that the Lindero samples tested represent the orebody with respect to grade and metallurgical response. The differences between metallurgical lithologies are minimal with regard to extraction. Cyanide consumptions are higher with the more oxidized Met 2 samples as would be expected. Minimal metallurgical differences were expected after review of the historical work as indicated in Figure 13.1.

Physical differences appear to have greater impact on the processing of the Lindero met types. Of significant importance is the ability of the agglomerated ore to support the planned heap height.

No significant deleterious materials such as mercury or clays were noted in the samples tested.

A high level of metallurgical and process risk mitigation is incorporated in the process design with HPGR crushing, agglomeration and the SART plant. With these installations any expected short-term variation in ore composition (i.e. elevated soluble copper content) or physical properties (i.e. elevated gypsum levels or increased ore hardness at depth) can be accommodated in the normal course of operations.

# 14 Mineral Resource Estimates

## 14.1 Introduction

The following chapter describes in detail the Mineral Resource estimation methodology for the Lindero Project. Mineral Resources have only been estimated for the Lindero Deposit.

## 14.2 Disclosure

Mineral Resources were prepared on behalf of Mansfield by Eric Chapman (P.Geo.) a Qualified Person as defined in National Instrument 43-101. Mr. Chapman is an employee of Fortuna.

The estimation methodology follows a similar methodology as that employed in the previous estimate (KCA, 2016a) by DKT Geosolutions Inc. (DKT), and incorporates the recent drill program completed in 2016

### 14.2.1 Known issues that materially affect Mineral Resources

Fortuna does not know of any issues that materially affect the Mineral Resource estimates. These conclusions are based on the following:

**Environmental**

Mansfield is in compliance with Environmental Regulations and Standards set in Argentine Law and has complied with all laws, regulations, norms and standards at every stage of exploration, as detailed in Section 20.

**Permitting**

Mansfield has represented that all necessary permits required to date for mine construction have been obtained and are in good standing.

**Legal**

Mansfield has represented that there are no outstanding legal issues; no legal actions, and/or injunctions pending against the Project.

**Title**

Mansfield has represented that the mineral and surface rights have secure title.

**Taxation**

No known issues other than those discussed in Section 4.

**Socio-economic**

Mansfield has represented that the operation has community support from the local town of Tolar Grande.

**Marketing**

No known issues.

**Political**

Mansfield believes that the current Government and Province are supportive of the Project.

**Other relevant issues**

No known issues.

**Mining**

No known issues.

**Metallurgical**

Mansfield and Fortuna have conducted extensive metallurgical testwork using external consultants to define an appropriate processing circuit based on a heap leach design. This work has been described in Section 13.

**Infrastructure**

No known issues.

## 14.3 Assumptions, methods and parameters

The 2017 Mineral Resource estimates were prepared using the following steps:

- Data validation as performed by Fortuna and detailed in Section 12
- Data preparation including importation to various software packages
- Geological interpretation and modeling of lithologic domains
- Coding of drill hole samples within lithologic domains (channels were not considered in the estimation of grades)
- Sample length compositing of drill hole samples
- Exploratory data analysis of gold and copper
- Analysis of boundary conditions
- Analysis of extreme data values and application of top cuts
- Unfolding of composites by lithological domains based on the morphology of the intrusives
- Generation of indicators to identify high, medium and low gold grade domains for domaining and estimation purposes
- Variogram analysis and modeling of indicator variograms
- Probability assigned constrained kriging using dynamic anisotropy to identify high- and medium-grade domains for block modeling and composites
- Estimation of gold and copper grades by ordinary kriging (OK) and nearest neighbor (NN) interpolation methods, using dynamic anisotropy in selected domains, and assignment of density values
- Validation of estimated gold and copper grades
- Classification of estimates with respect to 2014 CIM guidelines
- Mineral Resource tabulation and reporting

## 14.4 Supplied data, data transformations and data validation

Mansfield information used in the 2017 estimation is sourced from a Maxwell DataShed™ database.

### 14.4.1 Data transformations

All data are stored using the Gauss Kruger coordinate system (WGS 84 datum) commonly employed in Argentina and the same unit convention. Transformations of the supplied drill hole and channel information, including assay grades, were not required.

### 14.4.2 Software

Mineral Resource estimates have relied on several software packages for undertaking modeling, statistical, geostatistical and grade interpolation activities. Wireframe modeling of the mineralized envelopes was performed in Leapfrog Geo™ version 2.2. Data preparation, block modeling, unfolding and OK grade interpolations were performed in Datamine RM™ version 1.1.20.0. Statistical and variographic analysis was performed in Supervisor™ version 8.7.0.

### 14.4.3 Data preparation

Collar, survey, lithology, and assay data exported from the DataShed™ database provided by Mansfield was imported into Datamine™ and used to build 3D representations of the drill holes and trenches, although trenches were used only for the geologic interpretation and not for resource estimation of gold or copper grades. Assay values at the detection limit were adjusted to half the detection limit. Absent assay values were adjusted to a zero grade. A total of 144 surface drill holes totaling 42,359.72 m were used in the Lindero 2017 resource update. The majority of the drill core has been assayed. Table 4.1 details the data by company and sample type with Mansfield being responsible for collecting over 92 % of the drilling data.

**Table 14.1 Data used in 2017 Mineral Resource update**

| Company | Sample Type | Count | Meters | Percent of Total |
|---|---|---|---|---|
| Rio Tinto | Surface diamond drill holes | 10 | 3,279 | 7.74 |
| Goldrock/Mansfield | Surface diamond drill holes | 122 | 34,618 | 81.73 |
| Fortuna/Mansfield | Surface diamond drill holes | 12 | 4,462 | 10.53 |
| **TOTAL** | **n/a** | **144** | **42,359** | **100** |

### 14.4.4 Data validation

An extensive data validation process was conducted by Fortuna's Technical Services department prior to Mineral Resource estimation with a more detailed description of this process provided in Section 12. Validation checks were also performed upon importation into Datamine™ mining software and included searches for overlaps or gaps in sample and geology intervals, inconsistent drill hole identifiers, and missing data. No significant discrepancies were identified.

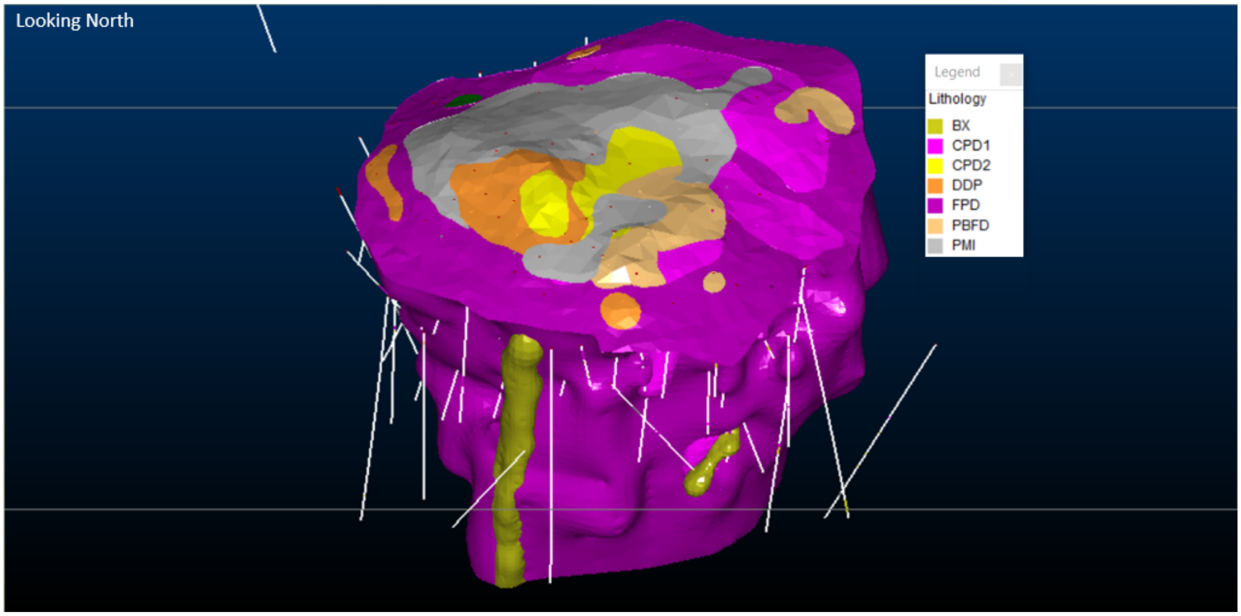
## 14.5 Geological interpretation and domaining

Gold mineralization at Lindero is hosted in a multiphase Miocene intrusive complex. The complex is elongated to the northeast, 750 by 600 m at the surface, and cuts recrystallized Paleogene sandstones, siltstones and mudstones (Figure 7.3). Most drill holes near the border of the complex pass into the red-bed sequence at depth, which suggests an upwardly-flaring shape. Mansfield under the guidance of the porphyry specialist, Brock Riedell, in preparing 3-D mineralized solids using Leapfrog Geo™ to reflect the geological interpretations generated from cross-sectional and plan views. The geological models used to constrain the resource estimation comprised eight lithological models:

- **Fine diorite porphyry (FPD)** forms most of the peripheral portions of the Lindero complex being the youngest intrusive event. Gold mineralization likely introduced into this lithology by the intrusion of the CPD1 unit
- **Crowded diorite porphyry 1 (CPD1)** contains approximately 50 % total phenocrysts, mostly of tabular plagioclase, with subordinate slender hornblende and ~2 % pinhead-sized quartz eyes. The majority of gold-copper at Lindero was introduced with CPD1 porphyry. When the original extent of CPD1 is restored by removal of younger intrusive phases, the distribution of present-day gold zones is nearly symmetrical around the deposit
- **Bimodal feldspar diorite porphyry (PBFD)** is a crowded porphyry similar to CPD1. Represents a small percentage of the intrusive complex and hosts medium-grade gold mineralization
- **Mingled diorite porphyry (DDP)** This unit occurs at the center of the intrusive complex and is associated with CPD2. Hosts low- to medium-grade mineralization in the 0.1 g/t Au to 0.4 g/t Au range
- **Tertiary sedimentary rocks (S1)** are a sequence of well-bedded arkose, greywacke, and subordinate mudstone and siltstone. These rocks form resistant hornfels within several hundred meters of the Lindero complex. Gold mineralization is present in this unit related to the contact with the FPD/CPD1 intrusions
- **Crowded diorite porphyry 2 (CPD2)** forms much of the south–central part of the complex and is generally barren
- **Post-mineral intrusive (PMI)** forms most of the north–central part of the complex, being the youngest intrusive event with no gold mineralization
- **Magmatic-hydrothermal breccias (BX)** form relatively narrow bodies, typically less than 10 m in drilled thickness. Two styles of breccias are distinguished based on the nature of the matrix material. In the quartz–sulfide type (BX-QS), the matrix consists of quartz, gypsum and/or anhydrite, and sulfides with low/moderate gold mineralization that cut S1, FPD, and CPD1, but none of the younger units. Breccias of the magnetite-type (BX-M), in contrast, have matrices of magnetite ± chlorite and are commonly strongly mineralized with chalcopyrite and gold

A 3-D perspective of the wireframes representing the intrusive events present at Lindero is displayed in Figure 14.1.

**Figure 14.1 3-D schematic of Lindero Deposit showing vein lithology wireframes**




Note: S1 lithology not shown as it represents the country rock
Figure prepared by Fortuna, 2017

In addition to the lithology domains, Mansfield constructed surfaces to represent the contact between the oxide and mixed horizon and the contact between the mixed and sulfide horizons to reproduce vertical trends in the gold grades.

**Figure 14.2 3-D schematic of Lindero Deposit showing oxide/sulfide horizons**

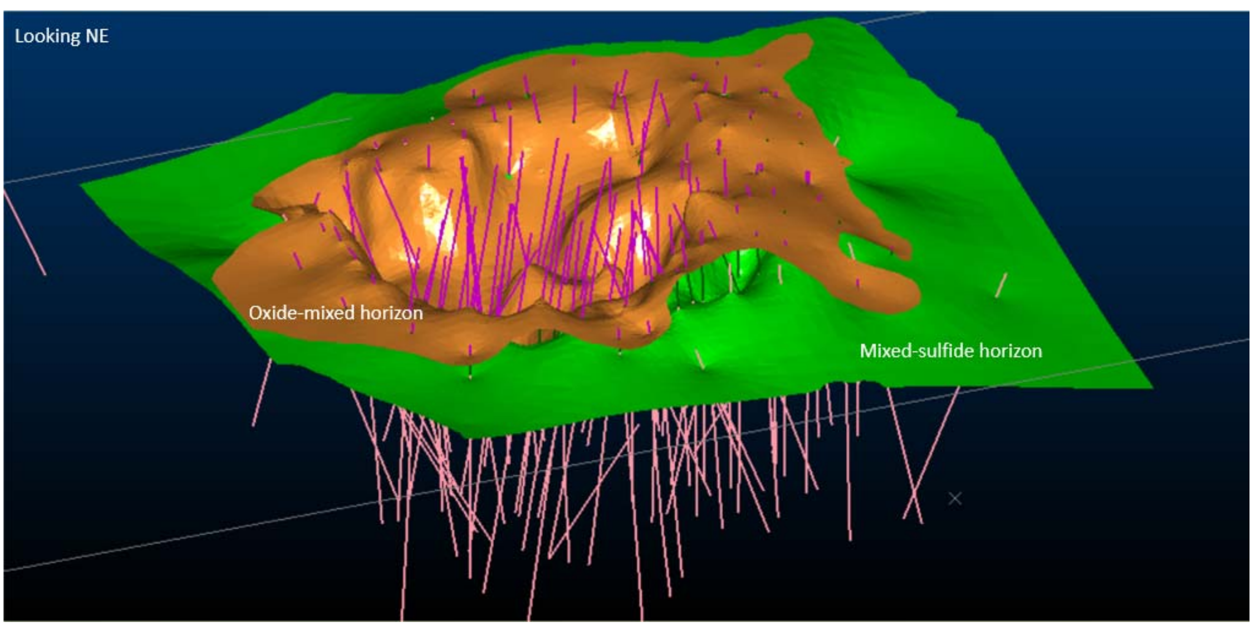



Figure prepared by Fortuna, 2017

Note that the oxide and mixed zones were combined and referred to as the 'oxide' domain for statistical analysis and estimation purposes.

### 14.5.1 Probabilistic grade shells

Fortuna used probability-assigned constrained kriging (PACK) to estimate the location of moderate and high gold grade regions of the deposit. PACK was designed to define economic envelopes around mineralized zones digitally that are difficult to outline and delineate using more traditional and labor-intensive methods such as wireframing. Probabilistic envelopes are first generated using indicators to define the limits of the economic mineralization and then the envelopes are used in the resource estimation to confine the higher-grade assays from smearing into lower-grade zones and restrict lower-grade assays from diluting the higher-grade zones.

PACK models were constructed for all domains except the barren PMI and CPD2 domains as follows:

- Indicator thresholds were selected for samples in all mineralized domains with grades above the threshold set to one and below to zero. Two thresholds were chosen to represent gold grade domains, 0.5 g/t Au for high grades and 0.2 g/t Au for moderate grades. One threshold of 0.1 % Cu was chosen to represent copper grades
- Indicator values for the three chosen thresholds were then unfolded to remove the circular nature of the mineralization and variograms modeled to represent the spatial variability of these indicators
- Indicator values were estimated by OK into a 2 x 2 x 2 m block model using the modeled variograms and associated search neighborhoods that employed dynamic anisotropy (where the search ellipse orientation changes direction according to dip and strike values assigned to the block in relation to its location around the circular mineralized body)
- Upon completion of the estimate, all blocks with a probability value greater than or equal to 0.5 were assigned a code of one and blocks with a probability below 0.5 were assigned a code of zero
- Three wireframes were generated identifying the location of the block codes equal to one for each of the three thresholds (high gold grade domains ≥0.5 g/t Au, moderate gold grade domains ≥0.2 g/t Au, and high copper grade domains ≥0.1 % Cu)
- The high gold grade wireframe (threshold of 0.5 g/t Au) was used to define a high gold grade domain in the strongly mineralized lithologies of FPD, CPD1, and S1
- The moderate gold grade wireframe (threshold 0.2 g/t Au) was used to define the moderate gold grade domain in the weakly mineralized lithologies of PBFD, DDP, and BX-M
- The high copper grade wireframe (threshold 0.1 % Cu) was used to define the high copper grade domain in all mineralized lithologies

The wireframes detailed above defining lithology, sulfide/oxide, and grade were used to define sub-domain codes as detailed in Table 14.2 that were used for controlling the estimation.

**Table 14.2 Domains used in 2017 Mineral Resource update**

| Lithology | Oxide/Sulfide | Grade Domain | Rock Code | Min Code | Domain Code |
|---|---|---|---|---|---|
| FDP | Sulfide | Low | 1 | 1 | 10 |
| | | High | | | 10.5 |
| | Oxide/Mixed | Low | | | 11 |
| | | High | | | 11.5 |
| CPD1 | Sulfide | Low | 2 | | 20 |
| | | High | | | 20.5 |
| | Oxide/Mixed | Low | | | 21 |
| | | High | | | 21.5 |
| S1 | Sulfide | Low | 7 | | 70 |
| | | High | | | 70.5 |
| | Oxide/Mixed | Low | | | 71 |
| | | High | | | 71.5 |
| PBFD | Sulfide | Low | 3 | | 30 |
| | | Moderate | | | 30.2 |
| | Oxide/Mixed | Low | | | 31 |
| | | Moderate | | | 31.2 |
| DDP | Sulfide | Low | 5 | | 50 |
| | | Moderate | | | 50.2 |
| | Oxide/Mixed | Low | | | 51 |
| | | Moderate | | | 51.2 |
| CPD2 | Sulfide | n/a | 4 | 2 | 40 |
| | Oxide | n/a | | | 41 |
| PMI | Sulfide | n/a | 6 | 3 | 60 |
| | Oxide | n/a | | | 61 |
| BX-M | Sulfide | Low | 9 | 4 | 90 |
| | | Moderate | | | 90.2 |
| | Oxide/Mixed | Low | | | 91 |
| | | Moderate | | | 91.2 |
| BX-QS | Sulfide | n/a | 8 | | 80 |
| | Oxide | n/a | | | 81 |

## 14.6 Exploratory data analysis

### 14.6.1 Compositing of assay intervals

Compositing of sample lengths was performed so that the samples used in statistical analysis and estimations have similar support (i.e., length). Mansfield sample drill holes at 2 m interval lengths although this may be shortened at geological contacts. The vast majority of samples (>99 %) were sampled on lengths of 2 m or less.

Based on the average sampling length and the selective mining unit, a 4 m composite was chosen as suitable.

### 14.6.2 Statistical analysis of composites

Exploratory data analysis was performed on composites identified in each geological vein (Table14.3). Statistical and graphical analysis (including histograms, probability plots, scatter plots) were investigated for each domain to assess if stationarity had been achieved.

**Table 14.3 Univariate statistics for gold of undeclustered composites by domain**

| Lithology | Oxide/Sulfide | Grade Domain | Domain Code | Count | Minimum (g/t) | Maximum (g/t) | Mean (g/t) | Standard Deviation | Coefficient of Variation |
|---|---|---|---|---|---|---|---|---|---|
| FDP | Sulfide | Low | 10 | 1,772 | 0.01 | 1.92 | **0.29** | 0.18 | 0.63 |
| | | High | 10.5 | 1,288 | 0.09 | 8.51 | **0.87** | 0.53 | 0.61 |
| | Oxide/Mixed | Low | 11 | 201 | 0.01 | 2.38 | **0.30** | 0.26 | 0.87 |
| | | High | 11.5 | 381 | 0.25 | 2.58 | **0.93** | 0.39 | 0.42 |
| CPD1 | Sulfide | Low | 20 | 1,093 | 0.01 | 2.54 | **0.33** | 0.20 | 0.62 |
| | | High | 20.5 | 697 | 0.08 | 4.69 | **0.90** | 0.52 | 0.58 |
| | Oxide/Mixed | Low | 21 | 218 | 0.02 | 1.99 | **0.30** | 0.20 | 0.68 |
| | | High | 21.5 | 194 | 0.26 | 2.42 | **0.93** | 0.42 | 0.45 |
| S1 | Sulfide | Low | 70 | 1,658 | 0.01 | 2.37 | **0.14** | 0.16 | 1.16 |
| | | High | 70.5 | 167 | 0.25 | 2.41 | **0.76** | 0.32 | 0.42 |
| | Oxide/Mixed | Low | 71 | 136 | 0.01 | 1.07 | **0.18** | 0.17 | 0.95 |
| | | High | 71.5 | 7 | 0.43 | 1.49 | **0.82** | 0.35 | 0.42 |
| PBFD | Sulfide | Low | 30 | 117 | 0.01 | 0.35 | **0.09** | 0.07 | 0.80 |
| | | Moderate | 30.2 | 96 | 0.18 | 2.29 | **0.60** | 0.35 | 0.58 |
| | Oxide/Mixed | Low | 31 | 101 | 0.01 | 0.75 | **0.07** | 0.11 | 1.61 |
| | | Moderate | 31.2 | 177 | 0.13 | 5.22 | **0.69** | 0.66 | 0.95 |
| DDP | Sulfide | Low | 50 | 34 | 0.03 | 0.30 | **0.12** | 0.07 | 0.56 |
| | | Moderate | 50.2 | 30 | 0.14 | 0.60 | **0.36** | 0.15 | 0.41 |
| | Oxide/Mixed | Low | 51 | 239 | 0.01 | 0.54 | **0.12** | 0.07 | 0.61 |
| | | Moderate | 51.2 | 432 | 0.07 | 1.83 | **0.48** | 0.29 | 0.60 |
| BX-M | Sulfide | Low | 90 | 0 | - | - | **-** | - | - |
| | | Moderate | 90.2 | 70 | 0.15 | 3.48 | **0.59** | 0.55 | 0.94 |
| | Oxide/Mixed | Low | 91 | 7 | 0.06 | 0.24 | **0.14** | 0.07 | 0.48 |
| | | Moderate | 91.2 | 4 | 0.30 | 1.27 | **0.80** | 0.37 | 0.47 |
| BX-QS | Sulfide | n/a | 80 | 17 | 0.14 | 0.54 | **0.27** | 0.11 | 0.41 |
| | Oxide | n/a | 81 | 8 | 0.09 | 1.08 | **0.52** | 0.29 | 0.55 |
| CPD2 | Sulfide | n/a | 40 | 287 | 0.00 | 1.56 | **0.10** | 0.15 | 1.47 |
| | Oxide | n/a | 41 | 279 | 0.00 | 0.77 | **0.11** | 0.12 | 1.05 |
| PMI | Sulfide | n/a | 60 | 393 | 0.00 | 2.24 | **0.11** | 0.20 | 1.84 |
| | Oxide | n/a | 61 | 394 | 0.00 | 0.82 | **0.07** | 0.11 | 1.67 |

### 14.6.3 Contact analysis

To determine whether composites should be used across lithological boundaries during gold and copper estimation, Fortuna constructed contact plots for the different combinations of lithological boundaries. Hard contacts (only those composites that lie within each domain are used for estimation in that domain) are generally justified if there is a substantial grade difference between the domains and soft contacts (composites in adjacent domains are included in the estimation) are justified if the grade difference is minor or if the grades at the boundary are nearly identical. Firm boundaries are justified where grades change gradually across the contact. If a hard boundary was imposed where grades tend to change gradually, grades may be overestimated on one side of the boundary and underestimated on the opposite side.

Results from the Lindero contact profiles showed that hard, soft and firm contacts exist. Hard boundaries were defined using the 'MIN' code as detailed in Table 14.2. Firm boundaries were defined using the 'DOMAIN' code where modeling allowed composites on either side of the boundary to be used in estimation of blocks in the first pass but not in subsequent passes.

### 14.6.4 Extreme value treatment

Top cuts of extreme grade values prevent over-estimation in domains due to disproportionately high-grade samples. Whenever the domain contains an extreme grade value, this extreme grade will overly influence the estimated grade.

If the extreme values are supported by surrounding data, are a valid part of the sample population, and are not considered to pose a risk to estimation quality, then they can be left untreated. If the extreme values are considered a valid part of the population but are considered to pose a risk for estimation quality (e.g., because they are poorly supported by neighboring values), they should be top cut. Top cutting is the practice of resetting all values above a certain threshold value to the threshold value.

Fortuna examined the grades of gold and copper by each domain to identify the presence and nature of extreme grade values. This was done by examining the sample histogram, log histogram, log-probability plot, and by examining the spatial location of extreme values. Top cut thresholds were determined by examination of the statistical plots and determination of the effect of top cuts on the mean, variance, and coefficient of variation (CV) of the sample data. Top cut thresholds used for each domain are shown in Table 14.4.

**Table 14.4 Top cut thresholds by domain**

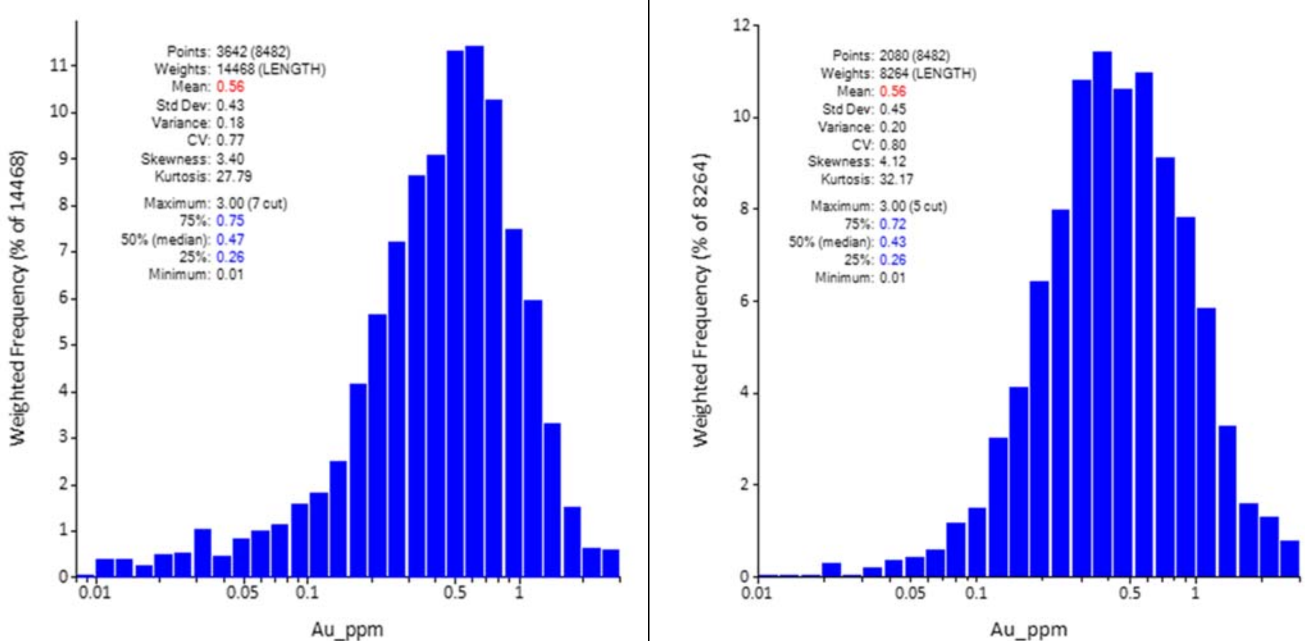
| Lithology | Oxide/Sulfide | Grade Domain | Domain Code | Top cut (g/t) | Mean (g/t) | Top cut Mean (g/t) | Difference (%) |
|---|---|---|---|---|---|---|---|
| FDP | Sulfide | Low | 10 | 1.2 | 0.29 | **0.29** | 0 |
| | | High | 10.5 | 3.0 | 0.87 | **0.86** | -1 |
| | Oxide/Mixed | Low | 11 | 1.2 | 0.30 | **0.30** | -2 |
| | | High | 11.5 | 3.0 | 0.93 | **0.93** | 0 |
| CPD1 | Sulfide | Low | 20 | 1.2 | 0.33 | **0.33** | -1 |
| | | High | 20.5 | 3.0 | 0.90 | **0.87** | -3 |
| | Oxide/Mixed | Low | 21 | 1.2 | 0.30 | **0.29** | -2 |
| | | High | 21.5 | 3.0 | 0.93 | **0.93** | 0 |
| S1 | Sulfide | Low | 70 | 1.1 | 0.14 | **0.14** | -3 |
| | | High | 70.5 | 1.9 | 0.76 | **0.76** | 0 |
| | Oxide/Mixed | Low | 71 | 1.1 | 0.18 | **0.18** | -1 |
| | | High | 71.5 | 1.9 | 0.82 | **0.82** | 0 |
| PBFD | Sulfide | Low | 30 | 0.75 | 0.09 | **0.09** | 0 |
| | | Moderate | 30.2 | 2.0 | 0.60 | **0.60** | -1 |
| | Oxide/Mixed | Low | 31 | 0.75 | 0.07 | **0.07** | -1 |
| | | Moderate | 31.2 | 2.0 | 0.69 | **0.64** | -7 |
| DDP | Sulfide | Low | 50 | 0.75 | 0.12 | **0.12** | 0 |
| | | Moderate | 50.2 | 1.55 | 0.36 | **0.36** | 0 |
| | Oxide/Mixed | Low | 51 | 0.75 | 0.12 | **0.12** | 0 |
| | | Moderate | 51.2 | 1.55 | 0.48 | **0.47** | -1 |
| BX-M | Sulfide | Low | 90 | | - | **-** | - |
| | | Moderate | 90.2 | 1.55 | 0.59 | **0.54** | -10 |
| | Oxide/Mixed | Low | 91 | 0.75 | 0.14 | **0.14** | 0 |
| | | Moderate | 91.2 | 1.55 | 0.80 | **0.80** | 0 |
| BX-QS | Sulfide | n/a | 80 | 0.55 | 0.27 | **0.27** | 0 |
| | Oxide | n/a | 81 | 0.55 | 0.52 | **0.42** | -24 |
| CPD2 | Sulfide | n/a | 40 | 0.4 | 0.10 | **0.09** | -9 |
| | Oxide | n/a | 41 | 0.4 | 0.11 | **0.11** | -3 |
| PMI | Sulfide | n/a | 60 | 0.55 | 0.11 | **0.09** | -15 |
| | Oxide | n/a | 61 | 0.55 | 0.07 | **0.06** | -8 |

The application of the top cuts has not dramatically altered the mean of the sample data in most of the domains except for the Breccia (BX-M and BX-QS) domains. This is because these domains are defined by very few samples with a small number (2 to 3) having extreme values far in excess of any other composites. Once these composites are reset the effect on the mean is dramatic, but likely to be more representative of the domain as a whole.

The gold grade distribution presented by lithology is graphically displayed using histograms in Figure 14.3.

**Figure 14.3 Gold grade distributions by lithology**

**FPD** | **CPD1**

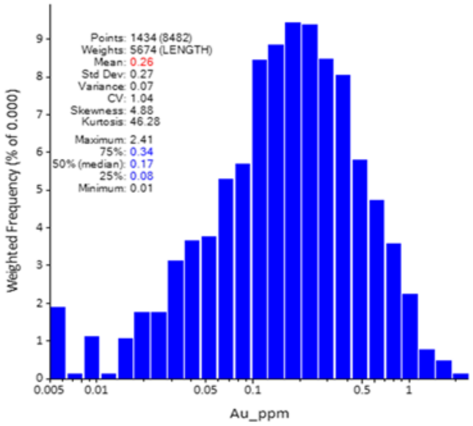



Log Histogram for Au (g/t) | Log Histogram for Au (g/t)

Figure prepared by Fortuna, 2017

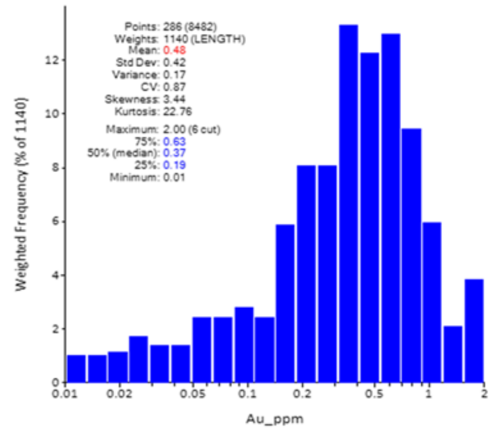
**Figure 14-3 Gold grade distributions by lithology (continued)**

**S1**



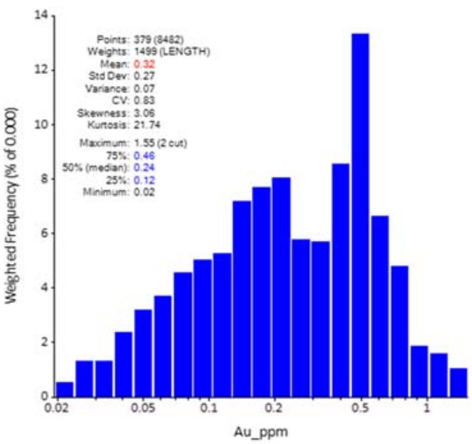

Log Histogram for Au (g/t)

**PBFD**



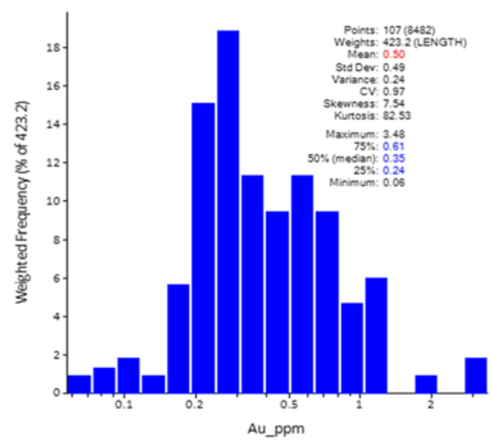

Log Histogram for Au (g/t)

**DDP**



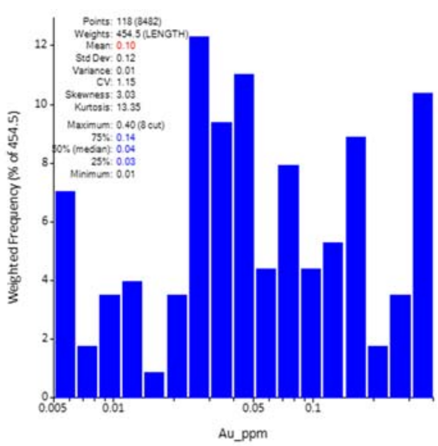

Log Histogram for Au (g/t)

**BX**



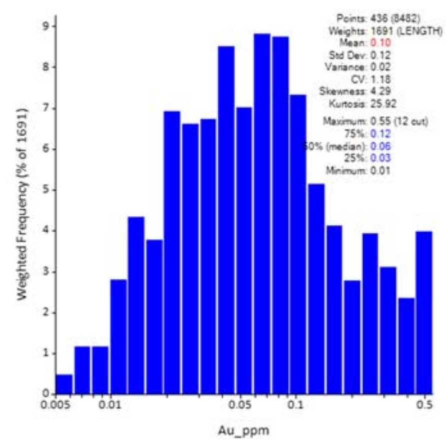

Log Histogram for Au (g/t)

**CPD2**




Log Histogram for Ag (g/t)

**PMI**




Log Histogram for Au (g/t)

### 14.6.5 Grade correlation

It is important that the relationship between major constituents is maintained in each of the domains after estimation. Subsequently the correlation between gold and copper grades has been investigated.

A positive correlation exists between gold and copper composite grades in the primary mineralized domains with a correlation coefficient of 0.71. The correlation statistics are reinforced by examining scatterplots comparing gold and copper grades for all composites.

It is expected that similar correlation coefficients and positive grade relationships are present in the estimates. These correlations have been tested as part of the validation process as described in Section 14.7.

### 14.6.6 Unfolding

The main purpose of unfolding strata is to calculate the stratigraphic distances between points rather than straight line distances which is used for variogram calculation. The nature of the emplacement of multiple intrusive events into the S1 Redbeds at Lindero has resulted in a hollow cylinder nature with the earliest mineralized FPD and CPD1 intrusive units being overprinted with later moderately mineralized (PBFD or DDP) or barren units (CPD2 and PMI). To account for the different orientation of the mineralized material, unfolding has been performed vertically around the body for variogram analysis.

### 14.6.7 Continuity analysis

Continuity analysis refers to the analysis of the spatial correlation between sample pairs to determine the major axis of spatial continuity. The analysis has been performed on the indicator thresholds; for each of the lithologic units; and for the defined mineralized units as described in Section 14.5.

Horizontal, across strike, and down dip continuity maps were examined (and their underlying variograms) for gold and copper to determine the directions of greatest and least continuity.

Continuity maps of the dip plane were examined to ascertain if a plunge was present in any of the domains. An example of the continuity map is displayed in Figure 14.4 (the lower the value the better the continuity with values greater than one representing no continuity). The presence of a distinctive plunge in the grade continuity could not be established for certainty in any of the domains and therefore variograms were modeled along strike (around the body) and down dip (vertical).

**Figure 14.4 Continuity map of normal score Au values for the FDP domain**

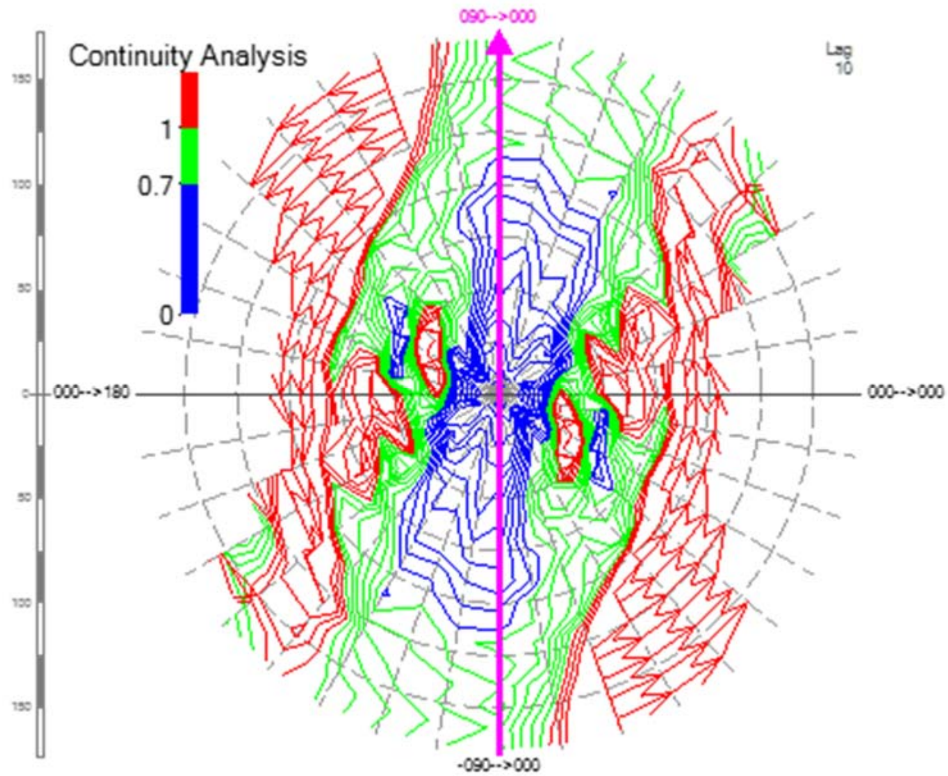



Figure prepared by Fortuna, 2017

### 14.6.8 Variogram modeling

The next step is to model the variograms for the major, semi-major, and minor axes. This exercise creates a mathematical model of the spatial variance that can be used to in ordinary kriging. The most important aspects of the variogram model are the nugget and the short-range characteristics. These aspects have the most influence on the estimation of grades.

The nugget effect is the variance between sample pairs at the same location (zero distance). Nugget effect contains components of inherent variability, sampling error, and analytical error. A high nugget effect implies that there is a high degree of randomness in the sample grades (i.e., samples taken even at the same location can have very different grades). The best technique for determining the nugget effect is to examine the downhole variogram calculated with lags equal to the composite length.

After determining the nugget effect, the next step was to model directional variograms in the three principal directions based on the directions chosen from the continuity maps (Figure 14.5). It was not always possible to produce a variogram for the minor axes, and in these cases the ranges for the minor axes were taken from the downhole variograms.

**Figure 14.5 Modeled variograms for normal score Au grades in the unfolded FPD domain**

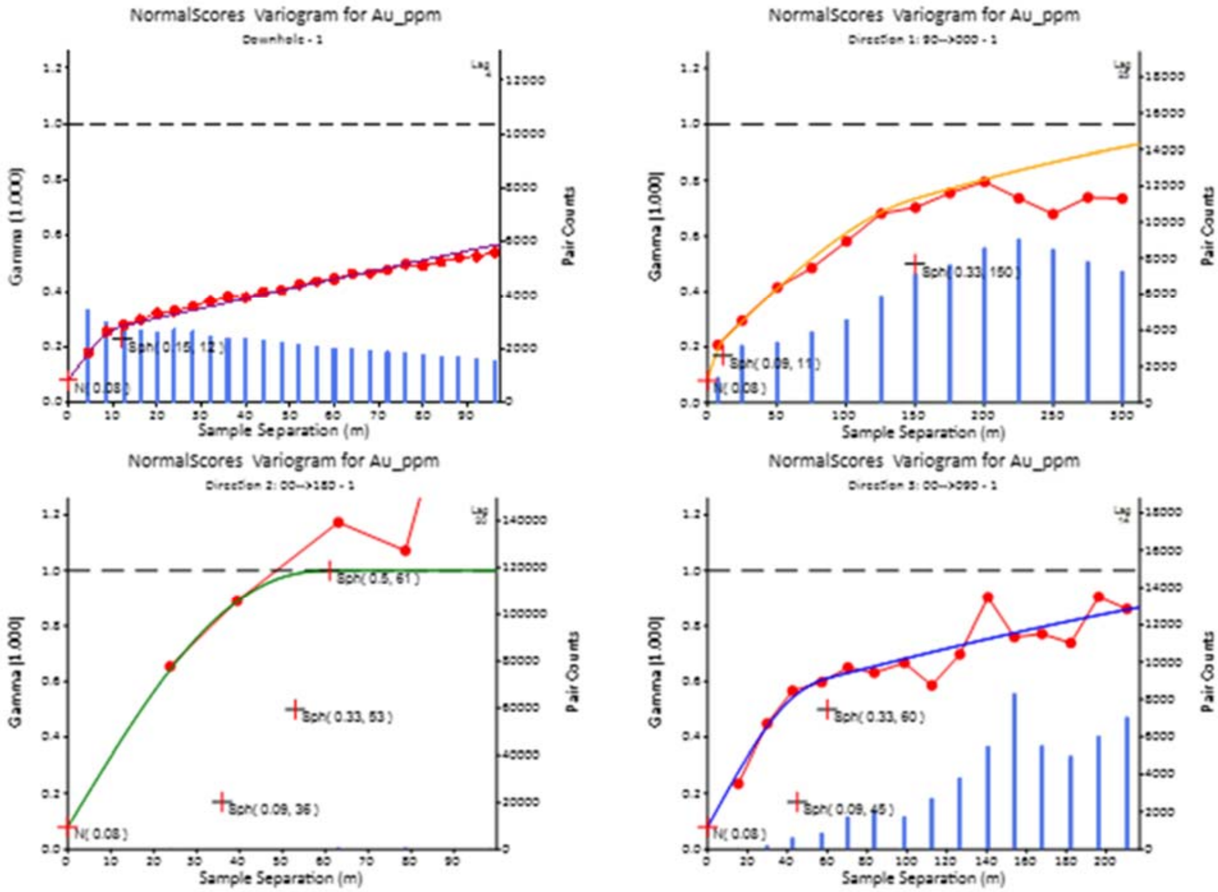


Figure prepared by Fortuna, 2017

Variogram parameters for each lithology are detailed in Table 14.5. Continuity analysis and variogram modelling were conducted in Supervisor™ version 8. It should be noted that the variograms were modelled using Normal Score transformation and then back transformed prior to estimation with the parameters reported in Table 14.5 representing the back-transformed numbers.

**Table 14.5 Variogram model parameters**

| Domain | Metal | Major axis orientation | $C_0$§ | $C_1$§ | Ranges (m)† | $C_2$§ | Ranges (m)† | $C_3$§ | Ranges (m)† |
|---|---|---|---|---|---|---|---|---|---|
| **0.2 g/t indicator** | Au | 90° → 000° | 0.24 | 0.11 | 15,62,30 | 0.18 | 105,66,135 | 0.47 | 569,74,751 |
| **0.5 g/t indicator** | Au | 90° → 000° | 0.31 | 0.11 | 7,37,36 | 0.24 | 40,48,59 | 0.34 | 191,58,156 |
| **0.1 % indicator** | Cu | 90° → 000° | 0.04 | 0.96 | 140,26,47 | - | - | - | - |
| **FPD** | Au | 90° → 000° | 0.09 | 0.10 | 11,36,45 | 0.35 | 150,53,60 | 0.45 | 464,61,400 |
| | Cu | 90° → 000° | 0.09 | 0.17 | 11,63,61 | 0.34 | 223,64,124 | 0.40 | 456,70,420 |
| **CPD1** | Au | 90° → 000° | 0.17 | 0.10 | 11,36,45 | 0.35 | 150,53,60 | 0.45 | 464,61,400 |
| | Cu | 90° → 000° | 0.21 | 0.22 | 8,72,53 | 0.33 | 55,89,109 | 0.24 | 264,90,184 |
| **S1** | Au | 90° → 000° | 0.10 | 0.90 | 230,80,311 | - | - | - | - |
| | Cu | 90° → 000° | 0.16 | 0.16 | 24,25,50 | 0.26 | 268,41,234 | 0.42 | 300,74,345 |

| Domain | Metal | Major axis orientation | $C_0$§ | $C_1$§ | Ranges (m)† | $C_2$§ | Ranges (m)† | $C_3$§ | Ranges (m)† |
|---|---|---|---|---|---|---|---|---|---|
| **PBFD** | Au | 90° → 000° | 0.07 | 0.34 | 98,56,110 | 0.59 | 162,69,275 | - | - |
| | Cu | 90° → 000° | 0.08 | 0.35 | 98,56,110 | 0.57 | 162,69,275 | - | - |
| **DDP** | Au | 90° → 000° | 0.07 | 0.31 | 47,56,27 | 0.62 | 101,69,275 | - | - |
| | Cu | 90° → 000° | 0.07 | 0.32 | 47,56,27 | 0.61 | 63,57,136 | - | - |
| **CPD2** | Au | 90° → 000° | 0.09 | 0.34 | 98,56,22 | 0.56 | 149,69,186 | - | - |
| | Cu | 90° → 000° | 0.09 | 0.34 | 98,56,22 | 0.56 | 149,69,186 | - | - |
| **PMI** | Au | 90° → 000° | 0.16 | 0.37 | 15,73,44 | 0.47 | 133,80,141 | - | - |
| | Cu | 90° → 000° | 0.15 | 0.37 | 44,73,15 | 0.47 | 118,80,133 | - | - |
| **BX-QS** | Au | 90° → 000° | 0.17 | 0.10 | 42,58,11 | 0.73 | 92,65,268 | - | - |
| | Cu | 90° → 000° | 0.21 | 0.27 | 87,35,18 | 0.51 | 129,93,248 | - | - |
| **MIN 1** | Au | 90° → 000° | 0.12 | 0.30 | 80,52,60 | 0.30 | 271,78,139 | 0.28 | 650,79,600 |
| | Cu | 90° → 000° | 0.11 | 0.14 | 11,52,12 | 0.25 | 71,80,69 | 0.49 | 513,87,316 |
| Note: § variances have been normalized to a total of one; † ranges for major, semi-major, and minor axes, respectively; structures are modelled with a spherical model | | | | | | | | | |

### 14.6.9 Opinion on the quality of the modeled variograms

Modeling of variograms can be somewhat of a subjective process depending on the quality of the experimental variograms. Confidence in the modeled variograms for the indicators, FPD, CPD1, and MIN domains is high due to the clearly-defined continuity displayed by the experimental variograms. The confidence is lower for the other domains due to the low composite numbers resulting in poorer experimental variograms. The BX-M domain has insufficient composites to allow modeling of variography and therefore the variogram corresponding to the DDP domain has been applied to the estimation of grades for this domain as they display similar grade characteristics.

### 14.6.10 Block model

The ultimate purpose of the estimation process is to estimate the tonnes and grade of potentially economically recoverable resources in accordance with the proposed mining method to be employed at the Project. The block size for the Lindero model has been set at 10 m x 10 m x 4 m. The block size was chosen such that geological contacts are reasonably well reflected and to support an open pit mining scenario. A bench height of 8 m is expected with the possibility of mining 4 m flitches to improve selectivity in variable grade areas of the deposit.

Block model parameters used for compiling the Lindero model are detailed in Table 14.6.

**Table 14.6 Block model parameters**

| Direction | Model Origin | Block size (m) | No. of blocks |
|---|---|---|---|
| Easting | 2622500 | 8 | 125 |
| Northing | 7225600 | 8 | 120 |
| Elevation | 3450 | 4 | 150 |

The geometry of each of the lithologic units has also been considered in the block modeling process. Blocks are sub-celled to fit the contacts exactly during the modelling process, with the sub-celled model used for estimation before being regularized, with the predominant lithologic code associated with the full block size. In this way, external dilution has been accounted for at the edge of the mineralized units in relation to the block size. Table 14.7 details the codes assigned to the block model for reporting purposes.

**Table 14.7 Block model domain codes**

| Lithology | Oxide/Sulfide | Grade Domain | MIN Code | DOMAIN Code |
|---|---|---|---|---|
| FDP | Sulfide | Low | 1 | 10 |
| | | High | | 10.5 |
| | Oxide/Mixed | Low | | 11 |
| | | High | | 11.5 |
| CPD1 | Sulfide | Low | | 20 |
| | | High | | 20.5 |
| | Oxide/Mixed | Low | | 21 |
| | | High | | 21.5 |
| S1 | Sulfide | Low | | 70 |
| | | High | | 70.5 |
| | Oxide/Mixed | Low | | 71 |
| | | High | | 71.5 |
| PBFD | Sulfide | Low | | 30 |
| | | Moderate | | 30.2 |
| | Oxide/Mixed | Low | | 31 |
| | | Moderate | | 31.2 |
| DDP | Sulfide | Low | | 50 |
| | | Moderate | | 50.2 |
| | Oxide/Mixed | Low | | 51 |
| | | Moderate | | 51.2 |
| CPD2 | Sulfide | n/a | 2 | 40 |
| | Oxide | n/a | | 41 |
| PMI | Sulfide | n/a | 3 | 60 |
| | Oxide | n/a | | 61 |
| BX-M | Sulfide | Low | 4 | 90 |
| | | Moderate | | 90.2 |
| | Oxide/Mixed | Low | | 91 |
| | | Moderate | | 91.2 |
| BX-QS | Sulfide | n/a | | 80 |
| | Oxide | n/a | | 81 |

## 14.7 Grade estimation

Blocks within a mineralized domain were interpolated using composites assigned to the same domain. Fortuna chose to interpolate the grades using OK, as the variability in grade is not problematic. Hard, firm and soft contacts were defined between different domains and sub-domains as described in Section 14.6.3.

The sample data and the blocks were coded using the MIN and DOMAIN codes detailed in Table 14.7. Sample data were composited (Section 14.6.1) and, where necessary, top cut (Section 14.6.4) prior to estimation. Each block was discretized (an array of points to ensure grade variability is represented within the block) using an array of 27 points (3 x 3 x 3) in each block with grades interpolated into full parent cells (Datamine ESTIMA parameter PARENT=1).

A three-pass method was employed to assign estimated grades. The search strategy employed concentric expanding search radii restricted by the block domain code and the applicable variogram. The distances for the three passes were determined as follows:

- Pass 1: a short search radius of derived from twice the distance between drill holes, based on typical drill hole spacing such that at least four drill holes would be found within the search ellipse. The MIN field was used to control the estimation to model hard boundaries between these units
- Pass 2: an intermediate search radius derived by twice the typical drill hole spacing plus 10 % to account for drill pattern irregularities. The DOMAIN field was used to control the estimation to model firm boundaries between these units
- Pass 3: a larger, less restrictive search radius still adhering to the DOMAIN field but of a size that all blocks would have interpolated grades. This distance varied depending on the domain

In the first and second passes, grade interpolation required a minimum of three composites, a maximum of 12 composites, and no more than two composites per drill hole. In the third pass, grade interpolation required a minimum of one composite, a maximum of 12 composites and no more than three composites per drill hole.

### 14.7.1 Dynamic anisotropy

Dynamic anisotropy is a tool in Datamine RM™ software that allows the estimator to account for gradual undulations or changes in orientation of the unit being estimated. It requires a dip and dip direction of the lithologic unit to be assigned to the block model based on the orientation of the hanging wall and/or footwall with these values being used to adjust the orientation of the variogram and search parameters during estimation. The concept of dynamic anisotropy is demonstrated in Figure 14.6.

**Figure 14.6 Diagram demonstrating concept of dynamic anisotropy**

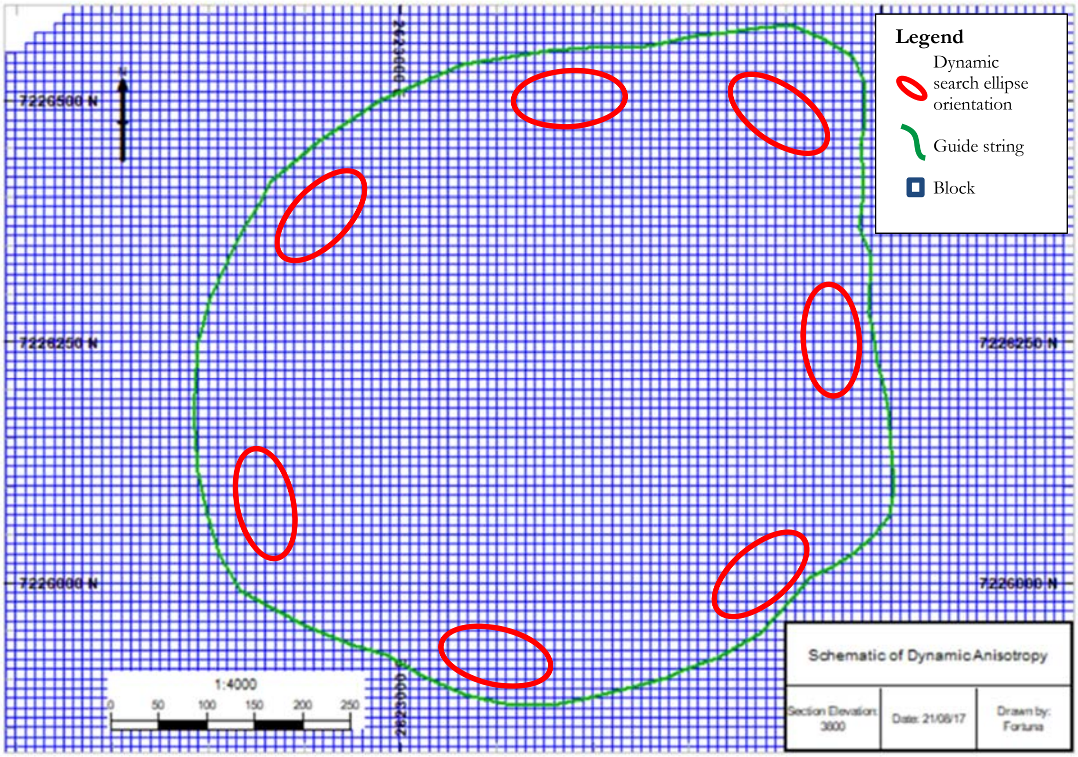



Figure prepared by Fortuna, 2017

By using this methodology, the deposit does not have to be split into separate sectors as was performed in the previous resource estimate, and results in a good representation of the circular nature of the mineralization.

## 14.8 Density

Fortuna used the density values shown in Table 14.8, which were established during the 2016 Pre-Feasibility Study using work conducted by AMEC (2010b). Fortuna submitted randomly selected density samples for reanalysis which confirmed those originally reported (Section 12). Density values were assigned by lithology and oxide/sulfide domain.

**Table 14.8 Density by domain**

| Lithology | Oxide/Sulfide | Density (t/m$^3$) |
|---|---|---|
| **FPD/CPD1** | Sulfide | **2.58** |
| | Oxide | **2.50** |
| **S1** | Sulfide | **2.57** |
| | Oxide | **2.50** |
| **PBFD** | Sulfide | **2.59** |
| | Oxide | **2.51** |
| **DDP** | Sulfide | **2.59** |
| | Oxide | **2.58** |
| **CPD2** | Sulfide | **2.64** |
| | Oxide | **2.56** |

| Lithology | Oxide/Sulfide | Density (t/m$^3$) |
|---|---|---|
| **PMI** | Sulfide | **2.49** |
| | Oxide | **2.51** |
| **BX** | Sulfide | **2.59** |
| | Oxide | **2.44** |

It is recommended that additional density measurements be taken throughout the development stages of the Project to better establish the characteristics of the deposit and potentially allow for the estimation of density values into the block model.

## 14.9 Block model validation

The block model has been assessed using several standard industry techniques to confirm the validity of the estimates.

### 14.9.1 Global bias

Fortuna checked the gold model for global bias by comparing the means of the OK model with means from the NN model by domain. The NN model theoretically produces an unbiased estimate of average value at a zero cut-off grade. A relative percentage value of less than 5 % difference between the means is an acceptable result and indicates good correlation between the two models. All the domain estimates are within the 5 % limit and indicate a good correlation with the exception of the BX domains that show differences of 12 % and 7 %; however, this is due to the small volumes these domains represent that can be affected by isolated grades with the OK evaluation providing a more conservative estimate of the gold grades. Fortuna considers that this is acceptable when the global statistics are evaluated with the combined lithologies displaying an overall difference of less than 0.1 % as shown in Table 14.9.

**Table 14.9 Global bias check by rock type**

| Lithology | Rock Code | NN Mean Gold Grade (g/t) | OK Mean Gold Grade (g/t) | Relative Percent Difference |
|---|---|---|---|---|
| FPD | 1 | 0.545 | 0.545 | **0.02** |
| CPD1 | 2 | 0.554 | 0.555 | **0.21** |
| S1 | 7 | 0.180 | 0.180 | **-0.05** |
| PBFD | 3 | 0.335 | 0.340 | **1.49** |
| DDP | 5 | 0.353 | 0.350 | **-0.69** |
| BX-M | 9 | 0.675 | 0.595 | **-11.90** |
| BX-QS | 8 | 0.394 | 0.367 | **-6.85** |
| CPD2 | 4 | 0.116 | 0.115 | **-0.93** |
| PMI | 6 | 0.075 | 0.076 | **1.81** |
| Global | | 0.354 | 0.353 | **-0.06** |

### 14.9.2 Local bias

**Visual validation**

Fortuna completed a detailed visual validation of the Lindero resource model. Models were checked for proper coding of drill hole intervals and block model cells, in both section and plan. Coding was found to be properly done. Grade interpolation was examined relative to drill hole composite values by inspecting sections and plans. Checks showed good agreement between drill hole composite values and model cell values. An example of these sections is displayed in Figure 14.7 (plan) and Figure 14.8 (west–east) showing gold grades for the estimated block model and composites clipped to within 20 m of the section.

**Figure 14.7 Plan view at 3850 m elevation displaying gold grades of block model and composites**

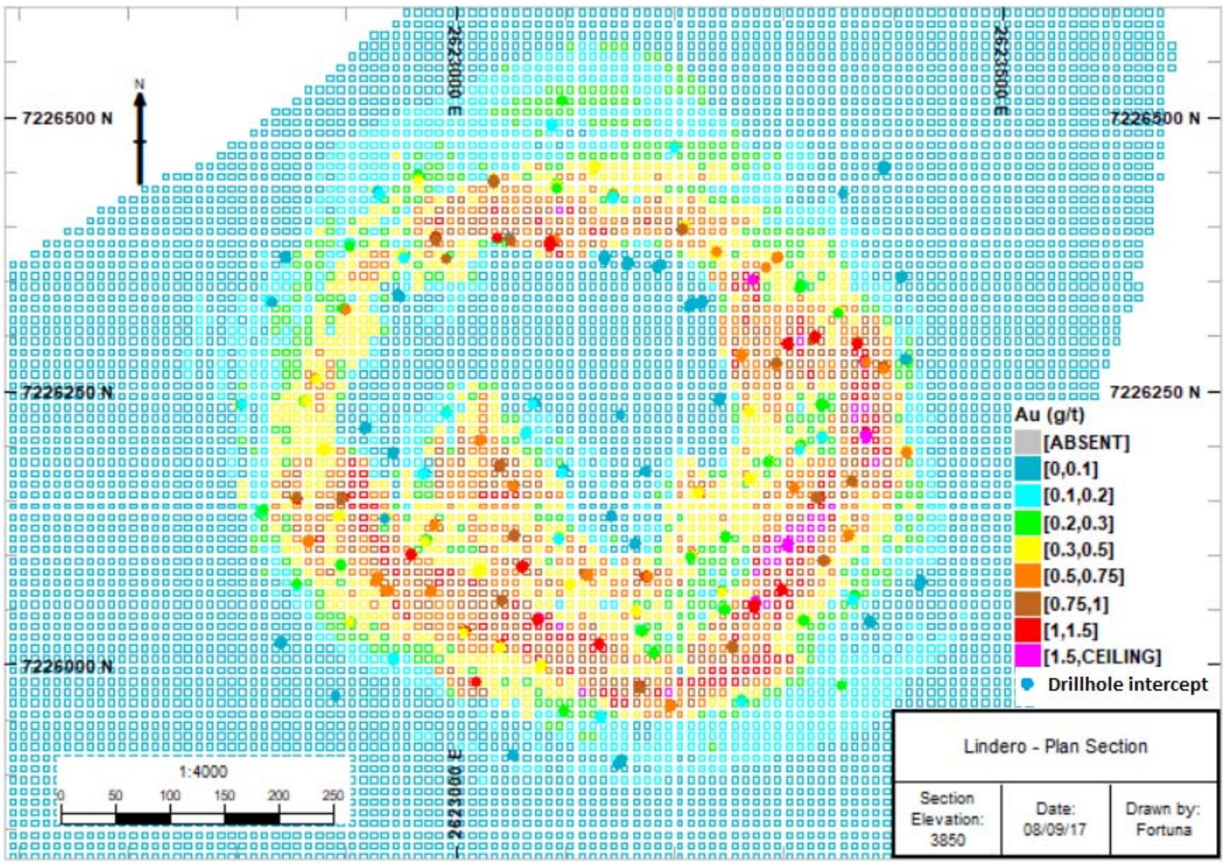



Figure prepared by Fortuna, 2017

**Figure 14.8 West-east sectional view (7226200N) displaying gold grades of block model and composites**

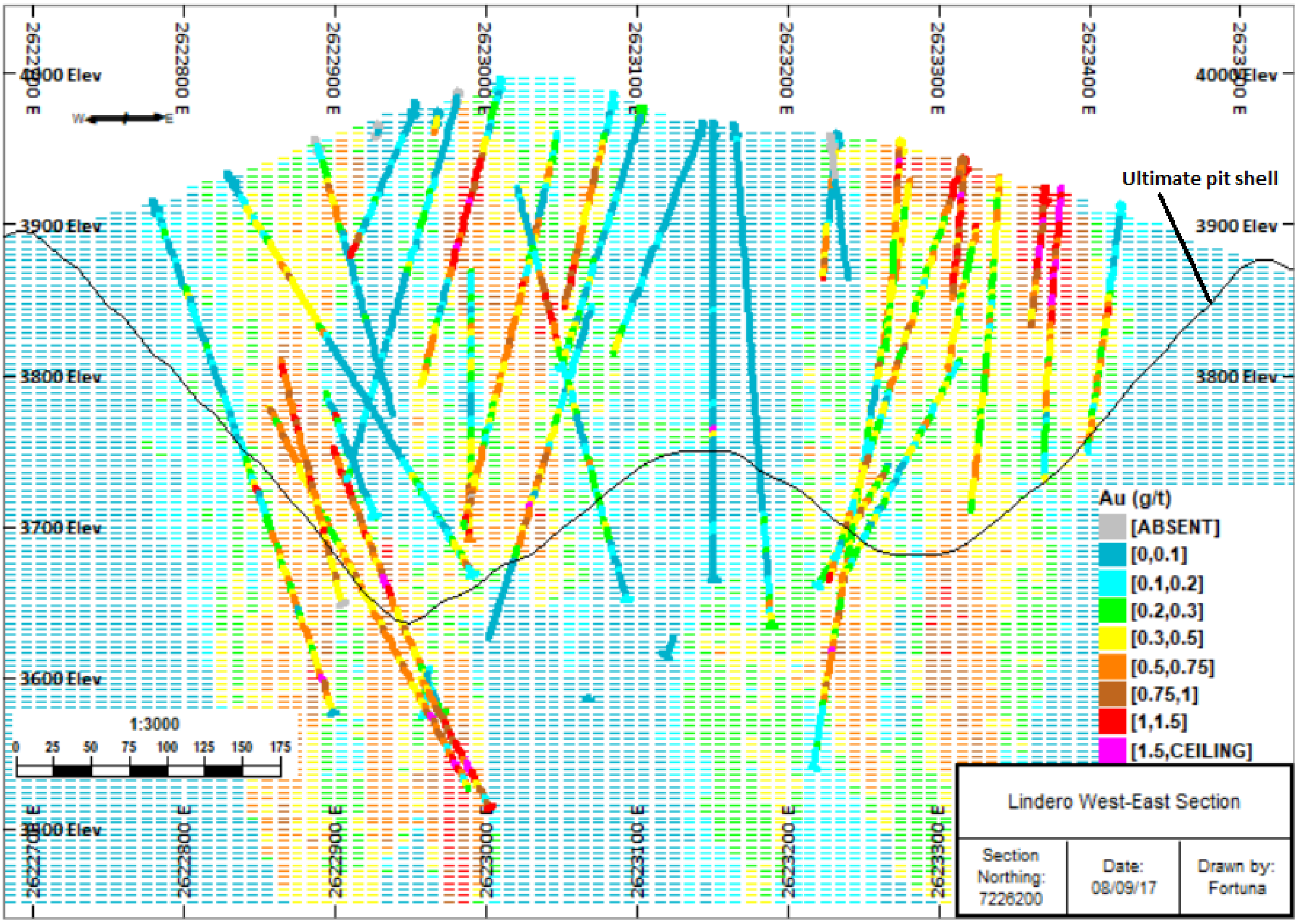



Figure prepared by Fortuna, 2017

### Swath plots

Swath plot validation was performed that allows a visual comparison of local bias between the kriged estimates and declustered composite grades used in the estimate. The process separates the block model into user-defined orthogonal slices (swaths) along easting, northing, and elevation axis and calculates the average grade for each swath. Fortuna reviewed the swath plots and determined that gold grades from kriged block estimates and composites especially in more densely sampled areas with gold and copper grades having similar peaks and valleys and the block grades displaying reasonable levels of smoothing compared to the much smaller volume composites. This level of smoothing was further verified as detailed below. Fortuna concluded that the estimation was locally unbiased.

### Change of support smoothing check

Fortuna used the study conducted by AMEC (KCA, 2016a) to check the level of grade smoothing in the estimation process. All linear interpolation methods cause some degree of smoothing in the final block model grade distribution relative to the composite grade distribution. This is caused in part due to the change of support during grade estimation of blocks from the grades of composite sized samples. An excess of smoothing can result in incorrect production forecasts of tonnage and average grade. Smoothing excess can be detected by comparing the OK grade-tonnage curve against a Discrete Gaussian (Herco)-adjusted NN grade tonnage curve; the Herco-adjusted NN block model grade distribution is used as a proxy for reality.

If the smoothing excess is too severe (more than 10 % relative difference), it can usually be reduced by restricting the number of composites used in estimation or subdividing the deposit into domains with different average grades via deterministic methods, spatial domains or soft–firm–hard (SFH) boundaries on lithology. Conversely, if the degree of smoothing is too low, the number of composites used in estimation and/or the size of the search neighborhood should be increased.

The results of the estimated block model were seen as acceptable with grade–tonnage curves being highly comparable based on a 10 m (easting) x 10 m (northing) x 8 m (elevation) selective mining unit.

## 14.10 Mineral Resource classification

Resource confidence classification considers several aspects affecting confidence in the resource estimation, such as:

- Geological continuity (including geological understanding and complexity)
- Data density and orientation
- Data accuracy and precision
- Grade continuity (including spatial continuity of mineralization)

### 14.10.1 Geological continuity

There is substantial geological information to support a good understanding of the Lindero gold-rich porphyry deposit. Exploration drilling has been conducted on an approximate 40 m to 50 m grid at surface with holes tending to diverge at depth. The relatively simple geology of the Lindero Deposit has meant that drilling conducted to date has allowed a clear definition of the geological continuity of the intrusive events both horizontally and vertically to a high level of confidence. Relogging of all historical core and the 2016 drill campaign have confirmed and further improved the geologic understanding of the porphyry system.

### 14.10.2 Data density and orientation

Drill holes are generally orientated perpendicular to the mineralization forming a radial pattern approximately 600 m in diameter. In the eastern portion, drill holes are orientated either perpendicular (azimuth 270°) or parallel (azimuth 190°) to the main mineralized body. Dips vary depending on the target and range from -50° to -89°, averaging -70°. Drill hole depths vary according to the area and the purpose for which they were collared. The average depth is 300 m and the deepest hole reached a depth of 576 m.

Geological confidence and estimation quality are closely related to data density and this is reflected in the classification.

### 14.10.3 Data accuracy and precision

Classification of Mineral Resource confidence is also influenced by the accuracy and precision of the available data. The accuracy and the precision of the data is determined through QAQC programs and through an analysis of the methods used to measure the data.

All exploration drill core has been sent to certified laboratories, including ACME, Alex Stewart, and most recently ALS Global for sample preparation and analysis.

Quality control results from the different laboratories indicate reasonable levels of accuracy with no material issues of sample switching or contamination. However, certificates for SRM's submitted during the Rio Tinto's campaign are not available and therefore accuracy levels for these early holes cannot be confirmed and that has been taken into consideration during classification.

During both the Rio Tinto and Goldrock drilling campaigns a limited number of duplicate types were submitted meaning that an analysis of each stage of the sample preparation process cannot be assessed. Nevertheless, field duplicates and check assays were submitted regularly, and both show reasonable levels of precision.

Fortuna submitted a full suite of duplicate types during the 2016 drilling campaign and reasonable precision levels were observed for all duplicate types. The mineralization is very homogenous in nature at the Lindero Deposit, as demonstrated by the heterogeneity test (Section 9.5.3), with good gold grade continuity observed in relation to the intrusive events in the porphyry system. The QC results indicate that grades reported from all laboratories are suitable for Mineral Resource estimation.

### 14.10.4 Spatial grade continuity

Spatial grade continuity, as indicated by the variogram, is an important consideration when assigning resource confidence classification. Confidence in the variogram characteristics, such as the nugget variance and ranges, strongly influence estimation quality parameters.

The variogram structures for the mineralized FPD, CPD1, and S1 lithologies are well defined and there is a high level of confidence in the modeled variograms.

The nugget effect and short-range variance characteristics of the variogram are the most important measures of continuity. The nugget variance is low for all modeled lithologies in respect to gold, ranging between 7 % and 17 % of total variance, indicating good grade continuity at short distances. Ranges (the distance at which continuity between sample grades is no longer present) are approximately 150 to 200 m vertically (down dip) and around the intrusive (along strike), while across the intrusive (across strike) is approximately 40 to 50 m for the main mineralized FPD/CPD1 units. These distances are typical for porphyry style mineralization and suggest that the drilling grids suggested below as a result of the AMEC study are reasonable for representative grade estimation.

### 14.10.5 Classification

Geostatistics provides an assortment of tools to establish confidence levels on resource estimates. The simplest of these methods involves evaluation of estimation variances for large blocks. This method was employed by AMEC for classifying resources in 2015 (KCA, 2016a) and gives an estimate of global confidence or confidence over large production areas. The method is not dependent on the local data.

**Inferred drill hole grid spacing**

Mineral Resources were classified as Inferred when a block was located within 70 m of the nearest composite. Drill hole spacing for Inferred Mineral Resources would broadly correspond to blocks with grades extrapolated within 70 m from the nearest drill hole.

**Indicated drill hole grid spacing**

AMEC calculated the confidence limits for determining the appropriate drill grid spacing for Indicated Mineral Resources (KCA, 2016a). AMEC considers that Indicated Mineral Resources should be known within ±15 % with 90 % confidence on an annual basis (production year) and Fortuna agrees with this approach.

A drill grid spacing of 75 m gives 90 % confidence levels in individual domains ranging from ±8 % to ±11 % for an annual product increment and is within the suggested limits of ±15 %. Subsequently, a drill spacing of 90 m was selected to ensure that the continuity of discontinuous high-grade gold zones, along with the extent and shape of the mineralization, is sufficiently delineated to give a reliable estimate of tonnes and grade.

Mineral Resources were classified as Indicated when a block was located within 54 m of the nearest composite and one additional composite from another drill hole was within 90 m. Drill hole spacing for Indicated Mineral Resources broadly correspond to a 75 m x 75 m grid.

Classification was manually adjusted in peripheral areas of the drill grid so that blocks extrapolated greater than 30 m from the closest drilling were classified as Inferred.

**Measured drill hole grid spacing**

AMEC calculated the confidence limits for determining appropriate drill grid spacing for Measured Resources (KCA, 2016a). AMEC considers that Measured Mineral Resources should be known within ±15 % with 90 % confidence on a quarterly basis (production quarter) and Fortuna agrees with this approach.

A drill grid spacing of 37.5 m gives 90 % confidence levels in individual domains ranging from ±8 % to ±11 % for a quarterly product increment.

Mineral Resources were classified as Measured when a block was located within 35 m of the nearest composite and two composites from two additional drill holes was within 45 m. Drill hole spacing for Measured Mineral Resources broadly correspond to a 37.5 m x 37.5 m grid.

### 14.10.6 Comments regarding classification

The Mineral Resource confidence classification of the Lindero Mineral Resource model incorporated the confidence in the drill hole data, the geological interpretation, geological continuity, data density and orientation, spatial grade continuity, and estimation quality. The resource models were coded as Inferred, Indicated, and Measured in accordance with CIM (2014) standards.

Fortuna visually reviewed the continuity of resource blocks with gold grades equal to or greater than the base case cut-off of 0.2 g/t Au in section and plan. The Mineral Resource model showed good grade and geologic continuity in areas of the FPD/CPD1 domains with 37.5 m drill spacing, and adequate continuity for grade interpolation and open-pit mine planning along strike and dip in areas with drill hole spacing at 75 m with a distance of less than 30 m extrapolation away from the drilled area.

Geological and grade continuity in the central and western parts of the intrusive complex is complicated by the occurrence of multiple intrusive events and magmatic breccias which increase the uncertainty in this area. Therefore, blocks within this area were downgraded to the Indicated category.

In addition, areas with the majority of the composites derived from drill holes with uncertain locations or assays with no supporting laboratory certificates were downgraded from the Measured to the Indicated category.

The above criteria ensure a gradation in confidence from Measured to Indicated to Inferred Mineral Resource blocks. It also ensures that blocks considered as Measured are informed from at least three sides, blocks considered as Indicated from at least two sides, and blocks considered as Inferred from at least one side. A plan section displaying the classification at Lindero is provided in Figure 14.9.

**Figure 14.9 Plan section of Lindero displaying Mineral Resource categorization**

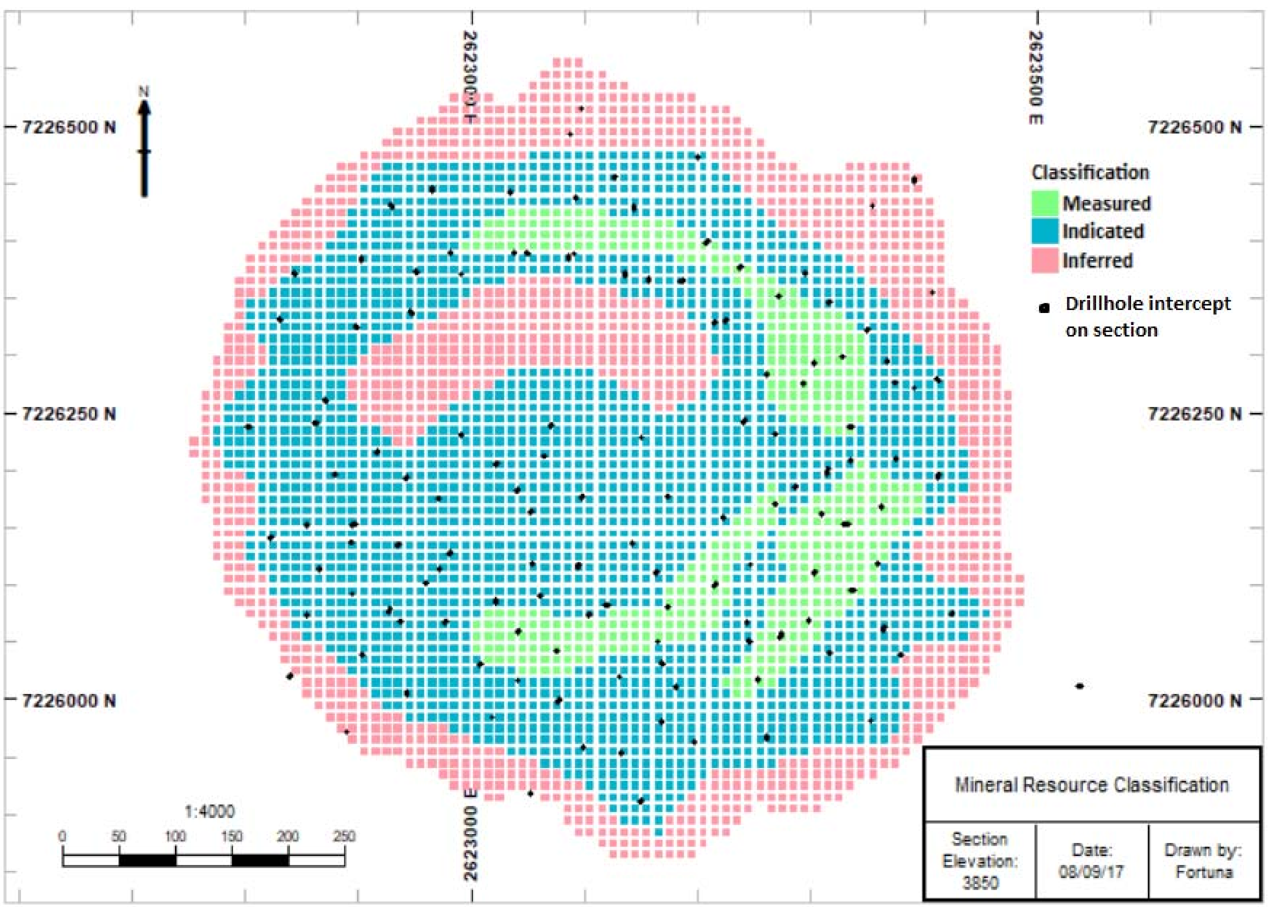



Figure prepared by Fortuna, 2017

## 14.11 Mineral Resource reporting

### 14.11.1 Reasonable prospects for eventual economic extraction

To assess reasonable prospects of economic extraction as required by the definition of Mineral Resource by the CIM, the mineralization was confined within a Lerchs–Grossmann (LG) optimization. Fortuna considered that the mineralized material that displays geological and grade continuity, and which falls within a pit shell constructed using reasonable economic parameters is likely to support economic extraction. A gold price of US$1,250/oz was used to generate the ultimate pit shell along with the processing and mining costs as detailed in Table 14.10.

**Table 14.10 Cost assumptions for ultimate pit design purposes**

| Cost | Price (US$) |
|---|---|
| Mining cost | $1.67/t |
| Ore processing and G&A cost | $7.84/t |
| Treatment and refining cost | $6.90/oz |

An average metallurgical recovery rate of 75 % was used for the evaluation based on the recent metallurgical testwork conducted in 2017.

Slope angles of 39°, 42°, and 47° were assigned to three separate sectors of the ultimate pit, based on the latest geotechnical report completed by CNI (2017).

Fortuna calculated a marginal cut-off grade using gold price, ore-based costs, and metallurgical recovery. The marginal cut-off is based on the generally-accepted practice that a decision is made at the pit rim if mined material above the marginal cut-off grade will lose less money if it is sent to the mill rather than if it is sent to the waste dump. It is considered to be mill feed if it contains a value that is greater than the costs to process it.

Revenue per gram was calculated using the metal price per gram of gold with an increase of 15 % to represent upside long-term metal prices multiplied by the process recovery (75 %). Total mill feed-based costs are shown in Table 14.10.

The cut-off grade was calculated using the following formula:

> Cut-Off Grade = Total Mill Feed Costs / (Revenue per Gram x Metallurgical Recovery)

Fortuna determined a marginal cut-off grade of 0.20 g/t Au for reporting resources. This is the same cut-off as used in the 2016 Feasibility Study, providing continuity.

It is the opinion of the QP that by constraining the Mineral Resources within an open pit shell based on established mining and processing costs; recommended slope angles based on independent geotechnical investigations; metallurgical recoveries from extensive testwork; reasonable long-term metal prices; and the application of a transparent marginal cut-off grade, the Mineral Resources do have 'reasonable prospects for eventual economic extraction'.

### 14.11.2 Mineral Resource statement

Eric Chapman P. Geo. is the QP for the Mineral Resource estimate for the Lindero Deposit. Mineral Resources have an effective date of September 9, 2017. Mineral Resources for the Project are summarized in Table 14.11. Mineral Resources are reported as undiluted and in-situ using a 0.20 g/t Au cut-off grade within the ultimate pit shell. The Measured and Indicated Mineral Resources are exclusive of those Mineral Resources modified to produce the Mineral Reserves through the process described in Section 15.

**Table 14.11 Mineral Resources exclusive of Mineral Reserves reported as of September 9, 2017**

| Classification | Tonnes (000) | Au (g/t) | Cu (%) | Contained Metal |
|---|---|---|---|---|
| | | | | Au (koz) |
| Measured | 610 | 0.24 | 0.06 | 5 |
| Indicated | 11,897 | 0.24 | 0.07 | 92 |
| **Measured + Indicated** | **12,507** | **0.24** | **0.07** | **97** |
| **Inferred** | **5,700** | **0.36** | **0.10** | **65** |

Notes on Mineral Resources:

- Mineral Resources are as defined by CIM Definition Standards on Mineral Resources and Mineral Reserves 2014
- Mineral Resources are reported as of September 9, 2017
- Mr. Eric Chapman P. Geo, a Fortuna employee, is the Qualified Person for the estimate
- Mineral Resources are reported exclusive of Mineral Reserves
- 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, socio-political, marketing, or other relevant issues
- Resources are reported at a 0.2 g/t Au cut-off grade
- Mineral Resources are reported within a conceptual pit shell using a long-term gold price of US$1,250/oz, mining costs at US$1.67 per tonne of material, with total processing and process G&A costs of US$ 7.84 per tonne of ore and an average process recovery of 75 %. The refinery costs net of pay factor were estimated to be US$ 6.90 per ounce gold. Slope angles are based on 3 sectors (39°, 42°, and 47°) consistent with geotechnical consultant recommendations
- Mineral Resource tonnes are rounded to the nearest hundred thousand
- Totals may not add due to rounding

### 14.11.3 Mineral Resources by key geologic attributes

The following section provides a breakdown of the resources based on various key geologic attributes. It important to note that all numbers presented in this section are not additive to the Mineral Resources presented in Table 14.11. A cornerstone of this analysis involves the evaluation of the Mineral Resource inclusive of Mineral Reserves for the Lindero Deposit, as summarized in Table 14.12. Mineral Resources are reported as undiluted and in-situ using a 0.20 g/t Au cut-off grade within the ultimate pit shell.

**Table 14.12 Mineral Resources inclusive of Mineral Reserves reported as of September 9, 2017**

| Category | Tonnes (Mt) | Au (g/t) | Cu (%) | Contained Metal Au (Moz) |
|---|---|---|---|---|
| Measured | 26.9 | 0.73 | 0.11 | 0.63 |
| Indicated | 77.1 | 0.51 | 0.10 | 1.26 |
| Measured + Indicated Resources | 104.0 | 0.56 | 0.10 | 1.88 |
| Inferred Resources | 5.7 | 0.36 | 0.10 | 0.07 |

Notes on Mineral Resources:

- Mineral Resources are as defined by CIM Definition Standards on Mineral Resources and Mineral Reserves 2014
- Mineral Resources are reported as of September 9, 2017
- Mr. Eric Chapman P. Geo, a Fortuna employee, is the Qualified Person for the estimate
- Mineral Resources are reported inclusive of Mineral Reserves
- The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues
- Resources are reported at a 0.2 g/t Au cut-off grade
- Mineral Resources are reported within a conceptual pit shell using a long-term gold price of US$1,250/oz, mining costs at US$1.67 per tonne of material, with total processing and process G&A costs of US$ 7.84 per tonne of ore and an average process recovery of 75 %. The refinery costs net of pay factor were estimated to be US$ 6.90 per ounce gold. Slope angles are based on 3 sectors (39°, 42°, and 47°) consistent with geotechnical consultant recommendations
- Mineral Resource tonnes are rounded to the nearest hundred thousand
- Totals may not add due to rounding

Table 14.13 shows the breakdown of the Mineral Resources reported by oxide or sulfide domain inside the ultimate pit shell above the 0.2 g/t Au cut-off grade. It can be seen from this breakdown that 70 % of the Measured + Indicated Mineral Resources are associated with sulfide material, and 30 % with oxide. Metallurgical studies have shown that both materials will be treated in the same manner, but slightly higher recoveries can be expected from oxide material (Section 13). Higher levels of soluble copper tend to be associated with the oxide material, and this has been accounted for with the inclusion of the SART plant in the Project design.

**Table 14.13 Mineral Resources inclusive of Mineral Reserves by oxide/sulfide domain reported as of September 9, 2017**

| Domain | Category | Tonnes (Mt) | Au (g/t) | Cu (%) | Contained Metal Au (Moz) |
|---|---|---|---|---|---|
| Oxide | Measured | 6.8 | 0.79 | 0.12 | 0.17 |
| | Indicated | 23.8 | 0.51 | 0.10 | 0.39 |
| | Measured + Indicated Resources | 30.6 | 0.57 | 0.10 | 0.56 |
| | Inferred Resources | 1.3 | 0.34 | 0.07 | 0.01 |
| Sulfide | Measured | 20.0 | 0.71 | 0.11 | 0.46 |
| | Indicated | 53.3 | 0.51 | 0.10 | 0.87 |
| | Measured + Indicated Resources | 73.3 | 0.56 | 0.10 | 1.32 |
| | Inferred Resources | 4.4 | 0.36 | 0.10 | 0.05 |

Refer to notes on Mineral Resources below Table 14.12. Mineral Resources in Table 14.13 are not additive to the Mineral Resources reported in Table 14.11 or Table 14.12.

Table 14.14 shows the breakdown of the Mineral Resources reported by lithology inside the ultimate pit shell above the 0.2 g/t Au cut-off grade. It should be noted that 77 % of the

Measured + Indicated Resources are associated with the FPD and CPD1 mineralized units, underlying the importance of these intrusive rocks. It is also important to note that tonnes associated with the barren CPD2 and PMI units are the result of the coding process of whole blocks to a particular rock code. Blocks located at the boundary of two rock types are assigned the code of the predominant lithology. If the majority of the block is in the CPD2 or PMI lithology it is assigned this rock code, even though the lesser percentage if mineralized material within the block may result in the whole block grade being above the cut-off and reported in the resource. In this way dilution is being accounted for at the SMU scale for the resource block model.

**Table 14.14 Mineral Resources inclusive of Mineral Reserves by lithology reported as of September 9, 2017**

| Lithology | Category | Tonnes (Mt) | Au (g/t) | Cu (%) | Contained Metal |
|---|---|---|---|---|---|
| | | | | | Au (Moz) |
| **FPD** | Measured | 11.5 | 0.78 | 0.13 | 0.29 |
| | Indicated | 30.3 | 0.59 | 0.12 | 0.58 |
| | Measured + Indicated | 41.7 | 0.64 | 0.12 | 0.86 |
| | Inferred | 2.4 | 0.39 | 0.11 | 0.03 |
| **CPD1** | Measured | 31.1 | 0.70 | 0.10 | 0.30 |
| | Indicated | 18.0 | 0.49 | 0.09 | 0.29 |
| | Measured + Indicated | 31.2 | 0.58 | 0.09 | 0.58 |
| | Inferred | 0.2 | 0.35 | 0.08 | 0.00 |
| **PBFD** | Measured | 2.0 | 0.63 | 0.09 | 0.04 |
| | Indicated | 3.6 | 0.50 | 0.07 | 0.06 |
| | Measured + Indicated | 5.6 | 0.54 | 0.08 | 0.10 |
| | Inferred | 0.0 | | | 0.00 |
| **DDP** | Measured | 0.1 | 0.75 | 0.13 | 0.00 |
| | Indicated | 9.4 | 0.43 | 0.08 | 0.13 |
| | Measured + Indicated | 9.6 | 0.43 | 0.08 | 0.13 |
| | Inferred | 0.2 | 0.42 | 0.02 | 0.00 |
| **BX-M** | Measured | 0.0 | 0.95 | 0.11 | 0.00 |
| | Indicated | 1.0 | 0.58 | 0.15 | 0.20 |
| | Measured + Indicated | 1.0 | 0.59 | 0.15 | 0.20 |
| | Inferred | 0.2 | 0.70 | 0.13 | 0.00 |
| **BX-QS** | Measured | 0.1 | 0.32 | 0.12 | 0.00 |
| | Indicated | 0.8 | 0.38 | 0.10 | 0.01 |
| | Measured + Indicated | 0.9 | 0.37 | 0.10 | 0.01 |
| | Inferred | 0.0 | | | 0.00 |
| **CPD2** | Measured | 0.0 | | | 0.00 |
| | Indicated | 1.4 | 0.27 | 0.08 | 0.01 |
| | Measured + Indicated | 1.4 | 0.27 | 0.08 | 0.01 |
| | Inferred | 0.0 | | | 0.00 |
| **PMI** | Measured | 0.1 | 0.37 | 0.06 | 0.00 |
| | Indicated | 1.6 | 0.29 | 0.07 | 0.01 |
| | Measured + Indicated | 1.6 | 0.30 | 0.07 | 0.02 |
| | Inferred | 0.4 | 0.27 | 0.07 | 0.00 |

Refer to notes on Mineral Resources below Table 14.12. Mineral Resources in Table 14.14 are not additive to the Mineral Resources reported in Table 14.11, Table 14.12 or Table 14.13.

A sensitivity analysis has been conducted on the resources to evaluate the effect of changing cut-off grades on tonnes and grade with the results reported in Table 14.14. The base case is highlighted.

**Table 14.15 Mineral Resources inclusive of Mineral Reserves as of September 9, 2017 reported at a range of gold cut-off grades**

| Category | Au Cut-off (g/t) | Tonnes (000) | Au (g/t) | Cu (%) | Contained Metal Au (Moz) |
|---|---|---|---|---|---|
| Measured | 0.1 | 27.0 | 0.72 | 0.11 | 0.63 |
| | **0.2** | **26.9** | **0.73** | **0.11** | **0.63** |
| | 0.3 | 25.6 | 0.75 | 0.11 | 0.62 |
| | 0.4 | 22.6 | 0.80 | 0.12 | 0.58 |
| | 0.5 | 19.1 | 0.87 | 0.12 | 0.53 |
| Indicated | 0.1 | 95.0 | 0.44 | 0.09 | 1.34 |
| | **0.2** | **77.1** | **0.51** | **0.10** | **1.26** |
| | 0.3 | **58.8** | **0.59** | **0.11** | **1.11** |
| | 0.4 | 42.6 | 0.68 | 0.12 | 0.93 |
| | 0.5 | 30.4 | 0.77 | 0.13 | 0.75 |
| Measured + Indicated Resources | 0.1 | 122.0 | 0.50 | 0.10 | 1.97 |
| | **0.2** | **104.0** | **0.56** | **0.10** | **1.88** |
| | 0.3 | **84.5** | **0.64** | **0.11** | **1.73** |
| | 0.4 | 65.2 | 0.72 | 0.12 | 1.51 |
| | 0.5 | 49.4 | 0.81 | 0.13 | 1.28 |
| Inferred Resources | 0.1 | 15.4 | 0.22 | 0.07 | 0.11 |
| | **0.2** | **5.7** | **0.36** | **0.10** | **0.07** |
| | 0.3 | **2.7** | **0.48** | **0.11** | **0.04** |
| | 0.4 | 1.5 | 0.59 | 0.13 | 0.03 |
| | 0.5 | 0.9 | 0.70 | 0.13 | 0.02 |

Refer to notes on Mineral Resources below Table 14.12. Mineral Resources in Table 14.15 are not additive to the Mineral Resources reported in Table 14.11, Table 14.12, Table 14.13 or Table 14.14.

### 14.11.4 Comparison to previous estimate

The 2016 Feasibility Study (KCA, 2016a) detailed the previous estimate of Mineral Resources of Lindero as of October 23, 2015. The QPs of this report have not verified the previous estimate and therefore it should not be relied upon.

The primary reasons for the change in estimates are as follows:

- Definition of a smaller ultimate pit shell based on updated metal prices, mining costs and metallurgical recoveries resulting in a decrease in the Measured and Indicated Mineral Resources by 32.5 Mt or 0.45 Moz Au and a decrease in the Inferred Mineral Resource by 34.2 Mt or 0.47 Moz
- The 2016 drilling program led to the upgrading of 12 Mt or 0.17 Moz Au into the Indicated category resulting in the loss of approximately the same quantity from the Inferred Mineral Resource
- Adjustments in the geological interpretation and estimation methodology resulting in the addition of 7.5 Mt or 0.03 Moz Au in the Measured and Indicated Mineral Resources and an increase of 5.6 Mt or 0.07 Moz in the Inferred Mineral Resource

Taking into account all of the above, the contained gold ounces in the Measured and Indicated classification decreased from 2.14 Moz Au to 1.88 Moz Au while contained gold ounces in the Inferred classification decreased from 0.61 Moz to 0.07 Moz Au.

## 14.12 Comment on Section 14

The QPs are of the opinion that the Mineral Resources for the Project, which have been estimated using core drill and channel data, have been performed to industry best practices, and conform to the requirements of CIM (2014). The Mineral Resources are acceptable to support declaration of Mineral Reserves.

Furthermore, it is the opinion of the QPs that by constraining the Mineral Resources within an open pit shell based on established mining and processing costs; recommended slope angles based on independent geotechnical investigations; metallurgical recoveries from extensive testwork; reasonable long-term metal prices; and the application of a transparent marginal cut-off grade, the Mineral Resources have 'reasonable prospects for eventual economic extraction'.

# 15 Mineral Reserve Estimates

This section provides details of the Mineral Reserve estimation methodology performed in September 2017 based on the updated Mineral Resources as well as adjustments made to account for the updated metallurgical testwork, geotechnical evaluations, and operating costs.

Mineral Resources have been reported in three categories, Measured, Indicated, and Inferred. The Mineral Reserve estimate has considered only Measured and Indicated Mineral Resources as these categories have sufficient geological confidence to be considered Mineral Reserves (CIM, 2014). Measured Resources may become Proven Reserves and Indicated Resources may become Probable Reserves.

## 15.1 Mineral Reserve methodology

An economic pit shell generated using cost criteria, metallurgical recoveries, geological and geotechnical considerations guides the final pit design. The economic pit shell used to define the final pit limit was developed using Datamine's NPV Scheduler™ software (NPVS). NPVS uses LG algorithm to define blocks that can be mined at a profit. The Mineral Reserve estimation procedure for the Lindero Deposit is defined as follows:

- Review of Mineral Resources in longitudinal sections and grade–tonnage curves
- Define economic parameters for computing an appropriate cut-off grade to be applied to the open pit optimization such as downstream costs, process and mining costs, haulage incremental cost, metallurgical recoveries by mineralization type, and sustaining capital
- Slope parameters based on geotechnical considerations were applied to the pit optimization and subsequently used to generate overall slope angles for the block model
- Compute the dollar value for each block to define blocks that can be mined at a profit in the LG algorithm
- Use of 'block discounting' to account for time value of money effect, resulting in a slightly smaller pit shell than would have been generated otherwise. A 5 % discount rate and 6 eight-meter benches per period were used when discounting
- Inferred Mineral Resources are considered as waste material in the optimization process
- Perform a LG pit optimization using Datamine's NPV Scheduler™
- Mineral Reserves are reported within an ultimate pit design at variable cut-off grades that are based on the process type, operating costs and metallurgical recovery
- A dilution allowance for the Mineral Reserve estimate is applied using diluted model grades. The diluted model, which was built from the Mineral Resource block model, incorporates dilution and ore loss and eliminates the need for applying additional factors
- Mineral Reserve tabulation and reporting as of September 9, 2017

## 15.2 Mineral Resource handover

The Mineral Resource reported in Table 14.11 contains mineralization that has been classified into the Measured, Indicated and Inferred categories.

Upon receipt of the block model a review was conducted to confirm the Mineral Resource was reported correctly and to validate the various fields in the model.

For estimating Mineral Reserves, only Measured and Indicated Mineral Resources were considered in the open pit optimization process. Inferred Mineral Resources were treated as waste material.

The Mineral Reserve estimation process considered the Mineral Resources above a 0.2 g/t Au cut-off grade.

## 15.3 Key mining parameters

### 15.3.1 Dilution and mining loss

The geologic model was constructed using 10 x 10 x 4 m blocks for the selective mining unit, with the thought that mining in ore would generally be performed on 8 m benches, while allowing flexibility to mine 4m slices in more variable areas. The gold grades in the Mineral Resource block model have been estimated with the appropriate degree of smoothing to account for expected dilution levels at the 10 x 10 x 4 block size and include external dilution at the contacts of ore-waste boundaries at this scale. No additional dilution or ore loss is expected beyond that accounted for in the model therefore eliminating the need for the application of additional mining recovery or dilution factors.

### 15.3.2 Pit slope parameters

Pit slope parameters based on geotechnical considerations were applied to the pit design along with ramps and catch benches, and subsequently used to generate overall slope angles. The overall slope angles used in pit optimization are shown in Table 15.1.

**Table 15.1 Overall slope angles used in the pit optimization process**

| Domain | Domain model Code | Mine Planning Azimuth (°) | | Overall slope angle (°) |
|---|---|---|---|---|
| | | From | To | |
| Red Bed Sediments | 1 | 0 | 120 | 47 |
| | | 120 | 150 | 42 |
| | | 150 | 360 | 47 |
| Intrusive | 2 | 0 | 220 | 47 |
| | | 220 | 260 | 42 |
| | | 260 | 360 | 47 |
| RQD < 40 | 3 | 0 | 360 | 39 |

### 15.3.3 Metallurgical recoveries

Gold recovery is determined by metallurgical type as detailed in Section 13. The expected gold recovery by metallurgical type is shown in Table 15.2.

**Table 15.2 Process recoveries**

| Ore type | unit | Gold % Extracted recovery |
|---|---|---|
| Met 1 | % | 75.4 |
| Met 2 | % | 78.2 |
| Met 3 | % | 78.5 |
| Met 4 | % | 68.5 |

### 15.3.4 LG optimization parameters

The Mineral Reserve Estimate was prepared using the December, 2010 topographic survey conducted by the Mineral Surveying Company (Section 9.3.3) and the parameters detailed in Table 15.3.

**Table 15.3 LG optimization parameters**

| Parameter | Units | Cost / Assumption |
|---|---|---|
| Gold Price | US$/oz | 1,250 |
| Royalty (3 %) | US$/oz | 37.5 |
| Refining & Selling | US$/oz | 6.9 |
| Mining Cost | US$/t mined waste | 1.67 |
| | US$/t mined ore | 1.69 |
| Bench Height | m | 8 |
| Haulage Increment per 8m Bench | US$/t mined | 0.015 |
| SART Plant | US$/t ore | 0.89 |
| G&A | US$/t ore | 1.65 |
| Processing Cost | US$/t ore | 5.3 |
| Recovery Met 1 | % | 75.4 |
| Recovery Met 2 | % | 78.2 |
| Recovery Met 3 | % | 78.5 |
| Recovery Met 4 | % | 68.5 |
| Overall Slope Angle | degrees | by domain |
| Rehandle | US$/t ore | 0.39 |
| **Cut-off Met 1** | **g/t** | **0.27** |
| **Cut-off Met 2** | **g/t** | **0.26** |
| **Cut-off Met 3** | **g/t** | **0.26** |
| **Cut-off Met 4** | **g/t** | **0.30** |
| Discount rate | % | 5 |
| Vertical advance rate | Bench / year | 6 |
| Discount per bench | IR/#bench | 0.83 |
| **Cut-off Met 1 + Rehandle** | **g/t** | **0.29** |
| **Cut-off Met 2 + Rehandle** | **g/t** | **0.27** |
| **Cut-off Met 3 + Rehandle** | **g/t** | **0.27** |
| **Cut-off Met 4 + Rehandle** | **g/t** | **0.31** |

Internal and breakeven cut-off grades were calculated based on the aforementioned economic parameters. The internal cut-off grade assumes an economic pit has been defined, and considers the mining cost as a sunk cost. The breakeven cut-off grade includes mining cost considerations.

## 15.4 Mineral Reserves

Mineral Reserves, as reported in Table 15.4, are based on an internal cut-off grade which assumes that an economic pit has been defined. Mineral Resource exclusive of Mineral Reserves are reported in Table 15.5.

**Table 15.4 Mineral Reserves as of September 9, 2017**

| Classification | Tonnes (000) | Au (g/t) | Cu (%) | Contained Metal |
|---|---|---|---|---|
| | | | | Au (koz) |
| Proven | 26,009 | 0.74 | 0.11 | 618 |
| Probable | 62,263 | 0.57 | 0.11 | 1,131 |
| **Proven + Probable** | **88,272** | **0.62** | **0.11** | **1,749** |

Notes:

- Mineral Reserves are as defined by CIM Definition Standards on Mineral Resources and Mineral Reserves
- Mineral Reserves are reported as of September 9, 2017
- There are no known legal, political, environmental, or other risks that could materially affect the potential development of the Mineral Reserves at Lindero
- Mineral Reserves for Lindero are reported based on open pit mining within designed pit shells, on variable gold internal cut-off grades and gold recoveries by metallurgical type. Met type 1 cut-off 0.27 g/t Au, recovery 75.4 %; Met type 2 cut-off 0.26 g/t Au, recovery 78.2 %; Met type 3 cut-off 0.26 g/t Au, recovery 78.5 %; and Met type 4 cut-off 0.30 g/t Au, recovery 61.7 %
- The cut-off grades and pit designs are considered appropriate for long-term gold prices of US$ 1,250/oz less 3 % gross royalty and refinery costs net of pay factor estimated to be US$ 6.9 per ounce gold
- Mineral Reserves are reported within the final pit design shell using a long-term gold price of US$1,250/oz, mining costs at US$1.67 per tonne of material, with total processing and process G&A costs of US$ 7.84 per tonne of ore and an average process recovery of 75 %. The refinery costs net of pay factor were estimated to be US$ 6.90 per ounce gold. Slope angles are based on 3 sectors (Z1= 47°, Z2= 47°, Z3= 39°) consistent with geotechnical consultant recommendations
- Totals may not add due to rounding

### 15.4.1 Comparison to previous reserve estimate

The 2016 Feasibility Study (KCA, 2016a) detailed the previous estimate of Mineral Reserves of Lindero. The QPs of this report have not verified the previous estimate and therefore it should not be relied upon.

The primary reasons for the change in reserve estimates are the same as those described for the resources in Section 14.11.3.

## 15.5 Comments on Section 15

The QP is of the opinion that the Proven and Probable Mineral Reserve estimate has been undertaken with reasonable care, and has been classified using the 2014 CIM Definition Standards. Mineral Reserves are to be extracted using open pit mining methods.

# 16 Mining Methods

This section summarizes the mine design and planning work completed to support the preparation of the Mineral Reserve statement. Mining method selection is also critical as it impacts dilution, productivity, product consistency, production capacity, and sequence of extraction.

The Lindero Deposit is suitable for open pit mining methods. Gold grade distribution and the results of the mineral processing testing indicate that ore from Lindero can be processed by conventional heap leaching methods.

Hydrological considerations and geotechnical studies are discussed in Section 24. Additional information on geotechnical considerations is provided in Section 9.6.

## 16.1 Material types

Material was divided into ore and waste categories for scheduling purposes. Waste consists of material that is not classified as either Proven or Probable Mineral Reserves. Ore is categorized by grade, resource classification, and metallurgical type.

### 16.1.1 Waste material

Waste material is to be placed in the waste dump immediately to the west of the crusher as a valley fill. Initial waste material will be built up around the crusher to provide a level stockpiling area.

### 16.1.2 Ore material types

Mineral Reserves above cut-off grade and inside of the pit were classified by both grade and metallurgical domain. The ore types consist of low-grade and high-grade, using minimum cut-off grades of 0.26 and 0.40 g/t Au respectively as grade breaks. The minimum cut-off grades were based on cut-off grades determined for each metallurgical type: 0.27 g/t Au, 0.26 g/t Au, 0.26 g/t Au, and 0.30 g/t Au for Met 1, Met 2, Met 3, and Met 4 material respectively. All ore is either fed directly to the crusher, or placed in stockpiles near the crusher.

## 16.2 Mining methods

The Lindero Project has been planned as an open pit truck and wheel loader operation. The truck and wheel loader method provide reasonable cost benefits and selectivity for this type of deposit. Only open pit mining methods are considered for mining at Lindero.

### 16.2.1 Pit designs

Detailed pit design was completed, including an ultimate pit and five internal phased pits. The ultimate pit was designed to allow mining of mill feed material identified by NPVS optimization while providing safe access for people and equipment. Internal pits or phases within the ultimate pits were designed to enhance the Project economics by providing higher-value material to the leach pad earlier in the mine life.

### 16.2.2 Bench height

Pit designs were developed based on 8 m benches for mining. This corresponds to the resource model block heights, and Fortuna believes this to be reasonable with respect to dilution and equipment anticipated to be used in mining.

### 16.2.3 Ultimate pit and pit phase design slopes

Ultimate pit and pit phases slope parameters were provided in a report by CNI (2017). Slope recommendations were given as double and single bench parameters for the corresponding domains as shown in Tables 16.1 for the ultimate pit design and 16.2 for internal phase designs or pushbacks.

Table 16.1 Final pit design slope parameters

| Domain | Domain model Code | Mine Planning Azimuth (°) | | Inter-ramp slope angle (°) | Design Bench Height (m) | Benching Increment | Catch Bench (m) | Bench Face (°) |
|---|---|---|---|---|---|---|---|---|
| | | From | To | | | | | |
| **Red Bed Sediments** | **1** | 0 | 120 | 50 | 16 | Double | 7.7 | 70 |
| | | 120 | 150 | 45 | 16 | Double | 7.5 | 62 |
| | | 150 | 360 | 50 | 16 | Double | 7.7 | 70 |
| **Intrusive** | **2** | 0 | 220 | 50 | 16 | Double | 7.7 | 70 |
| | | 220 | 260 | 45 | 16 | Double | 7.5 | 62 |
| | | 260 | 360 | 50 | 16 | Double | 7.7 | 70 |
| **RQD < 40** | **3** | 0 | 360 | 42 | 8 | Single | 4.4 | 60 |

Table 16.2 Pushback design slope parameters

| Domain | Domain model Code | Mine Planning Azimuth (°) | | Inter-ramp slope angle (°) | Design Bench Height (m) | Benching Increment | Catch Bench (m) | Bench Face (°) |
|---|---|---|---|---|---|---|---|---|
| | | From | To | | | | | |
| **Red Bed Sediments** | **1** | 0 | 120 | 50 | 16 | Double | 7.7 | 70 |
| | | 120 | 150 | 45 | 16 | Double | 7.5 | 62 |
| | | 150 | 360 | 50 | 16 | Double | 7.7 | 70 |
| **Intrusive** | **2** | 0 | 220 | 50 | 16 | Double | 7.7 | 70 |
| | | 220 | 260 | 45 | 16 | Double | 7.5 | 62 |
| | | 260 | 360 | 50 | 16 | Double | 7.7 | 70 |
| **RQD < 40** | **3** | 0 | 360 | 42 | 8 | Single | 4.4 | 60 |

### 16.2.4 Haulage considerations

Ramps were designed to have a maximum centerline gradient of 10 %. In areas where the ramps may curve along the outside of the pit, the pit inside gradient may be as much as 11 % or 12 % for short distances.

Ramp width was determined as a function of the largest truck width to be used in mine planning. Design criteria accounts for 3.5 times the width of the truck for running room in areas using two-way traffic. An additional width is added to the ramp for a single safety berm at least 2/3 of a tire height inside the pit. For roads designed outside of the pit, an additional safety berm is accounted for in the road width.

The proposed mining truck fleet will be made up of 91 tonne (100 ton) capacity units. The phases were designed to have 23.43 m wide ramps where two-way traffic is anticipated.

A vertical section displaying haulage road design parameters is provided in Figure 16.1.

**Figure 16.1 Plan section displaying haul road design parameters**

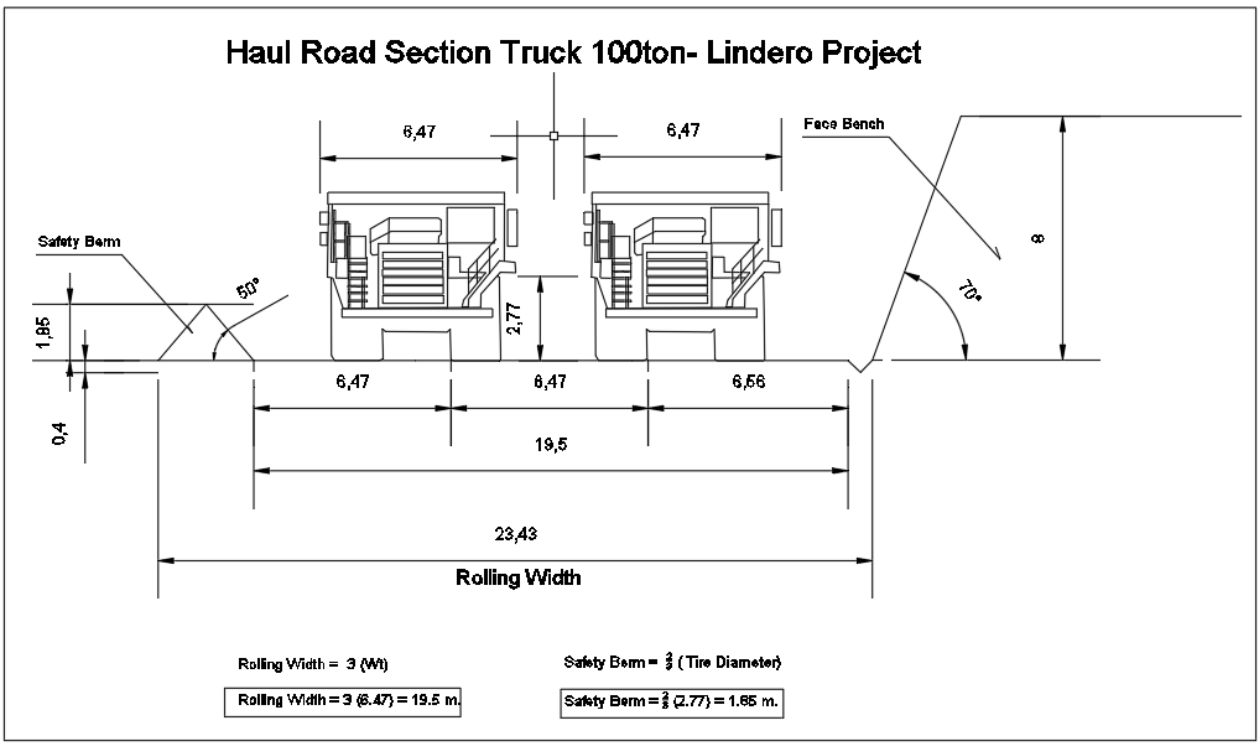



Figure from Thompson, 2015

### 16.2.5 Ultimate pit and pit phasing

The ultimate pit limit uses the pit shell generated by Datamine's NPVS open pit optimization software.

The mine operating costs used for pit optimization include ongoing major mine equipment capital costs. The mine equipment sustaining capital cost estimate is used in the economic model to simulate mine capital expenditures when generating the economic pit shell.

The top-down discount method was used for pit optimization. This is a procedure based on multiplying the block value by a discount factor that is a function of the annual cost of money, an estimate of the average annual vertical advance rate of mining, and the relative depth of the block. This method helps to simulate the actual mine plans that are burdened with up-front stripping costs and aids in the selection of a higher-value pit.

The ultimate pit design is circular with a maximum diameter of 850 m. Figure 16.2 shows a plan view of the ultimate pit design.

**Figure 16.2 Ultimate pit design**

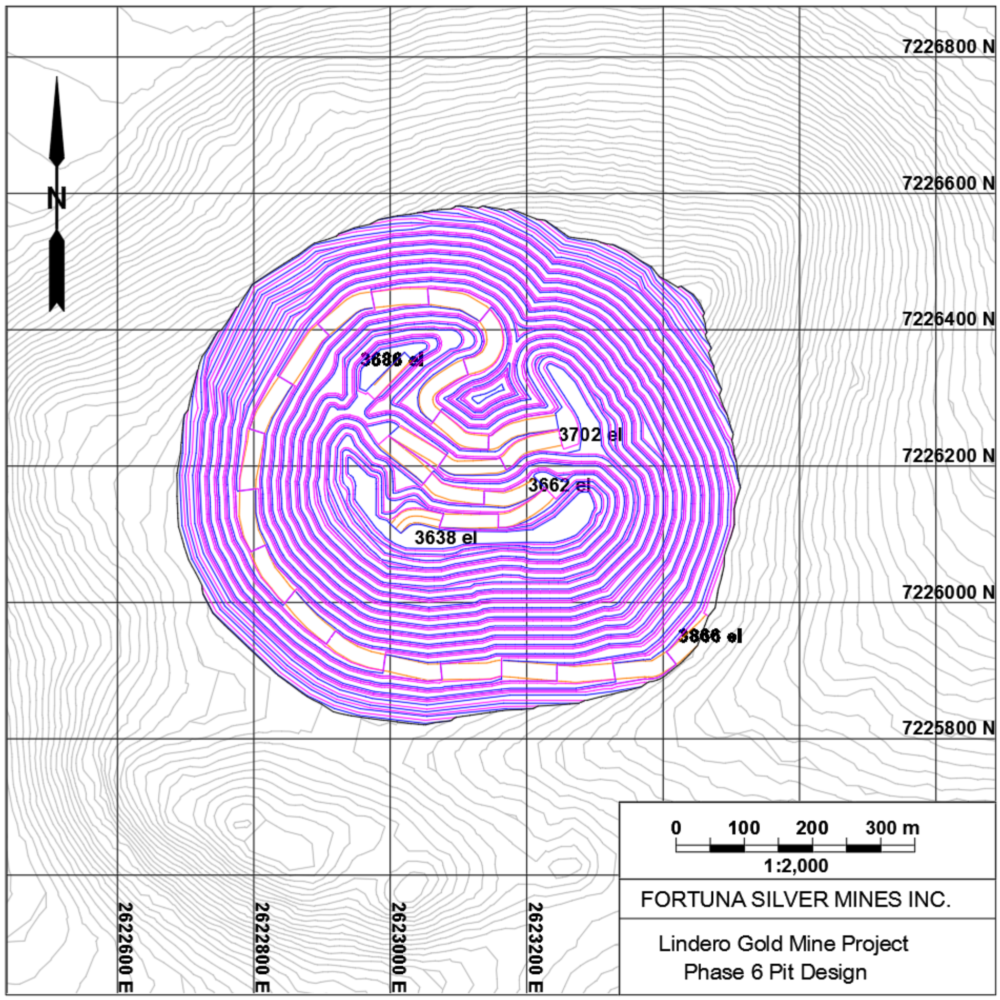



Figure prepared by Fortuna, 2017

The ultimate pit and internal phases were designed to maximize net present value (NPV) and cash flow of the Project. The first three phases determine the payback period, which represents production Year 4. Phases four and five pit walls merge with the final pit wall limit based on CNI's recommendations.

Figures 16.3 to 16.7 show the phase1, 2, 3, 4, and 5 pit designs respectively. Phase 6 is depicted in Figure 16.2 as the ultimate pit. The subset of the Mineral Reserve for each pit phase is shown in Table 16.3.

**Figure 16.3 Phase 1 pit design**

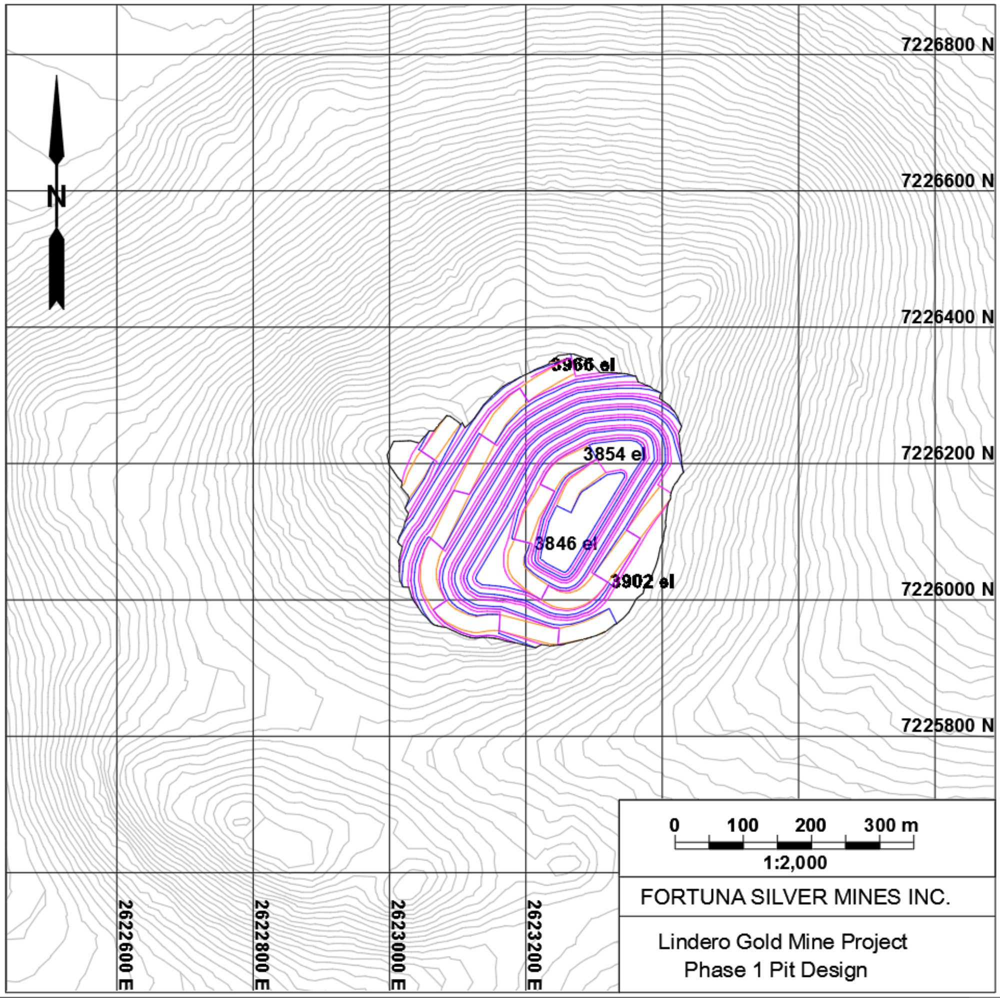



Figure prepared by Fortuna, 2017

**Figure 16.4 Phase 2 pit design**

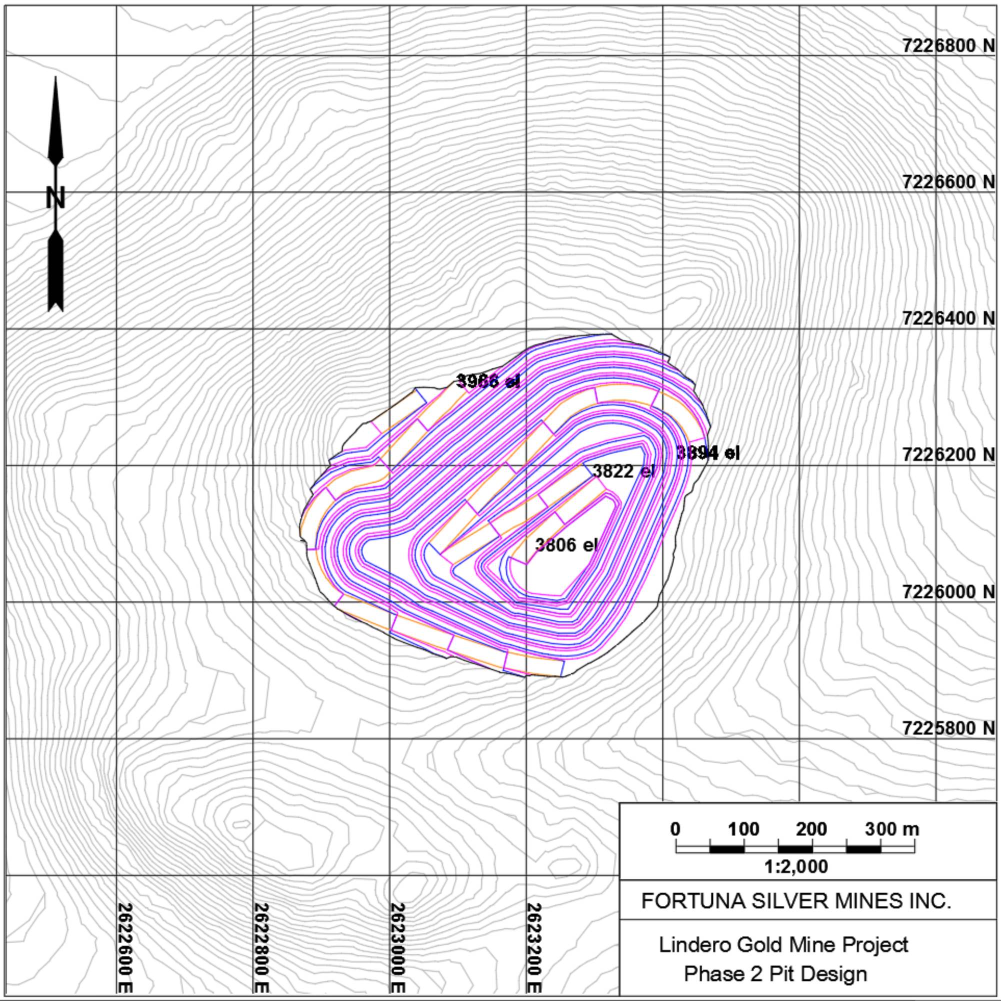



Figure prepared by Fortuna, 2017

**Figure 16.5 Phase 3 pit design**

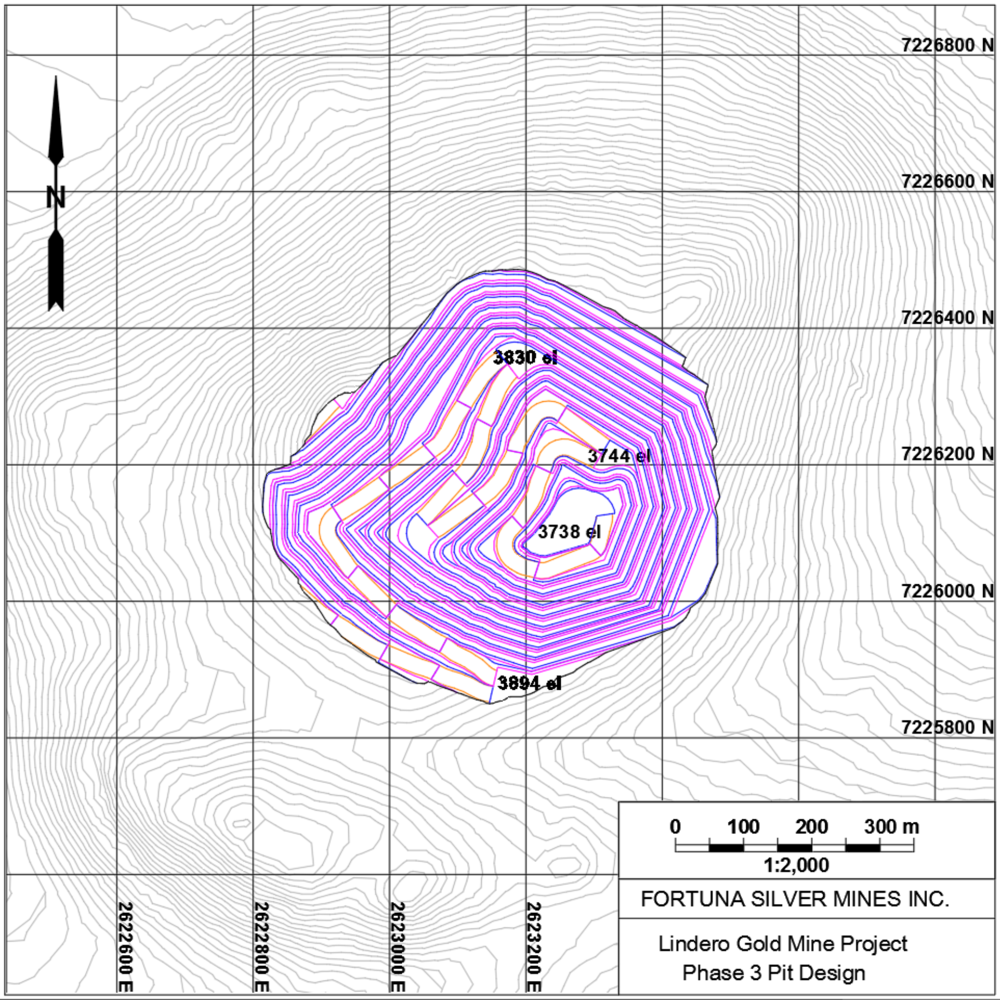



Figure prepared by Fortuna, 2017

**Figure 16.6 Phase 4 pit design**

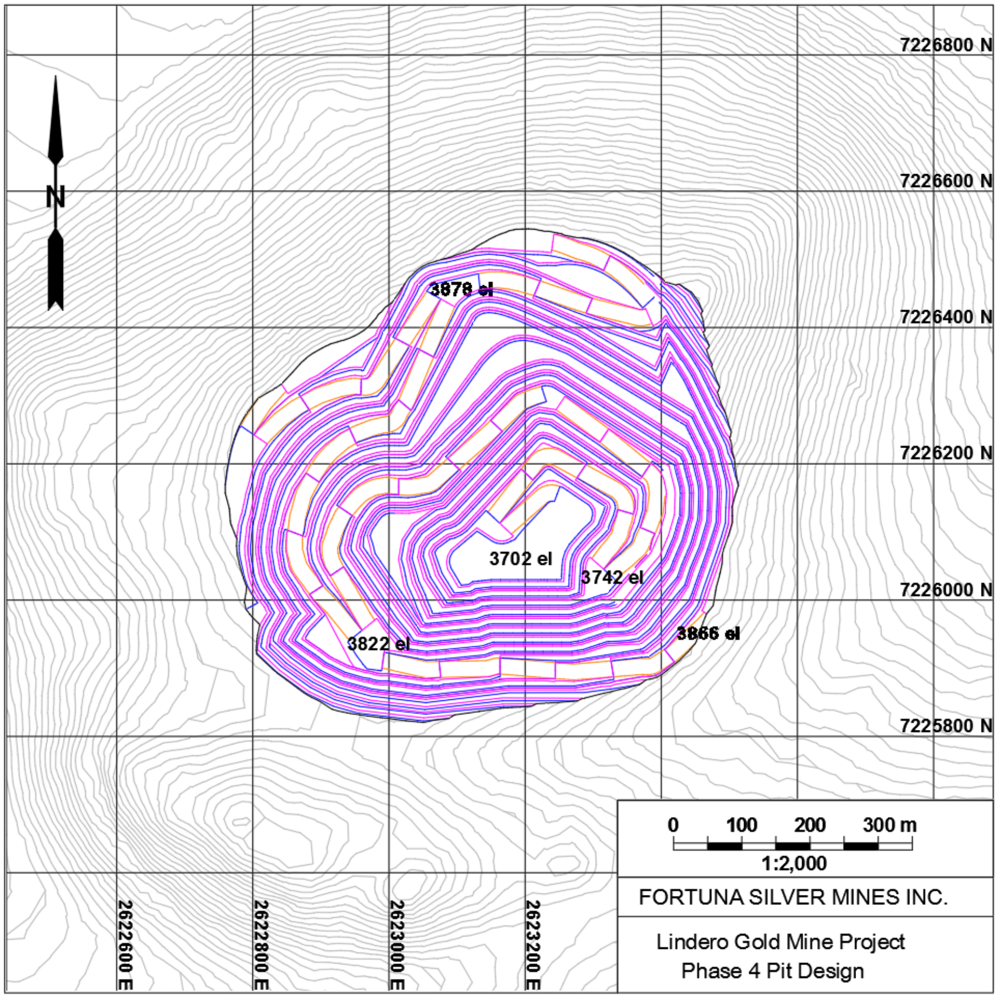



Figure prepared by Fortuna, 2017

**Figure 16.7 Phase 5 pit design**

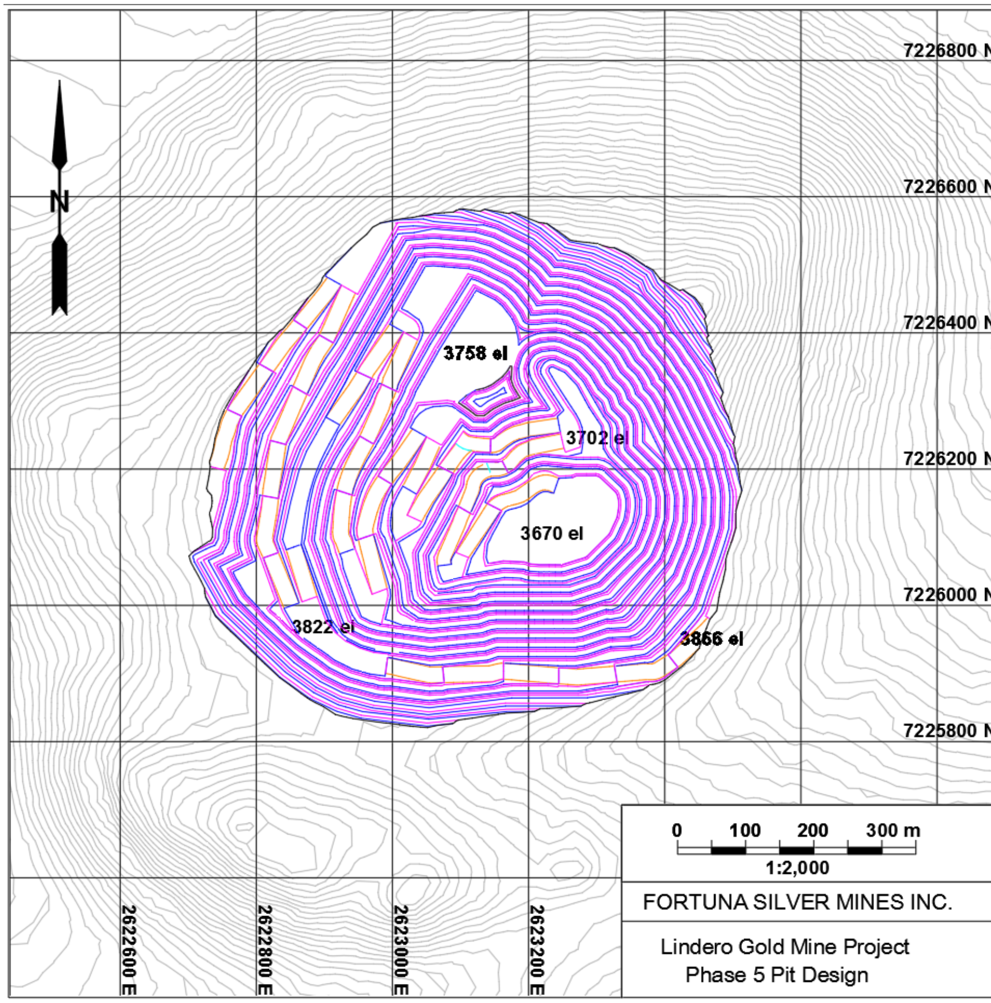



Figure prepared by Fortuna, 2017

**Table 16.3 Mineral Reserves and associated waste by Phase**

| | Proven | | | Probable | | | Total Proven & Probable | | | Waste | Total | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Phase | Tonnes (000's) | Au (g/t) | Au (koz) | Tonnes (000's) | Au (g/t) | Au (koz) | Tonnes (000's) | Au (g/t) | Au (koz) | Tonnes (000's) | Tonnes (000's) | Strip Ratio |
| Phase 1 | 6,940 | 0.84 | 188 | 3,725 | 0.80 | 96 | 10,665 | 0.83 | 284 | 5,141 | 15,806 | 0.5 |
| Phase 2 | 4,786 | 0.78 | 120 | 7,161 | 0.63 | 146 | 11,947 | 0.69 | 265 | 9,681 | 21,628 | 0.8 |
| Phase 3 | 5,026 | 0.68 | 109 | 13,015 | 0.55 | 229 | 18,041 | 0.58 | 338 | 21,436 | 39,477 | 1.2 |
| Phase 4 | 3,495 | 0.70 | 79 | 12,470 | 0.52 | 209 | 15,965 | 0.56 | 287 | 27,056 | 43,021 | 1.7 |
| Phase 5 | 4,081 | 0.66 | 86 | 13,856 | 0.53 | 234 | 17,937 | 0.56 | 321 | 26,560 | 44,497 | 1.5 |
| Phase 6 | 1,681 | 0.68 | 37 | 12,037 | 0.56 | 218 | 13,717 | 0.58 | 254 | 19,925 | 33,642 | 1.5 |
| **Total** | **26,009** | **0.74** | **618** | **62,263** | **0.57** | **1,131** | **88,272** | **0.62** | **1,750** | **109,798** | **198,070** | **1.2** |

## 16.3 Mine waste dump

Waste rock material is to be placed in the waste dump immediately to the west of the crusher as a valley fill. Initial material will be built up around the crusher to provide a level stockpiling area. The total waste dump capacity is approximately 120 Mt.

The waste dump stability analysis was conducted by CNI (2017). The overall slope and single lift heights were analyzed utilizing the SLOPE/W™ slope stability software and Spencer´s method of slices.

The waste dump design consists of two consecutive lifts of 100-m height for facility lift phases 1 and 3, and an additional 50-m height lift for facility lift phase 2 that serves as a buttress for the facility phase 1 lift, resulting in a total facility storage height of 205 m. The slope of the lifts is designed at a 37° angle. A variable offset between the lifts and ramp is included on each lift. The result is a maximum overall angle of 26°. Figure 16.8 shows a plan view of the waste dump location with respect to the final pit. Figure 16.9 shows a cross section of the static analysis of the waste dump resulting in a safety factor of 1.2.

The waste dump footprint and height requirements were revised based on an analysis optimizing haulage from the pit. The waste dump is located within the footprint of previous studies and no modification will be required to the currently granted permits.

**Figure 16.8 Waste dump location plan**

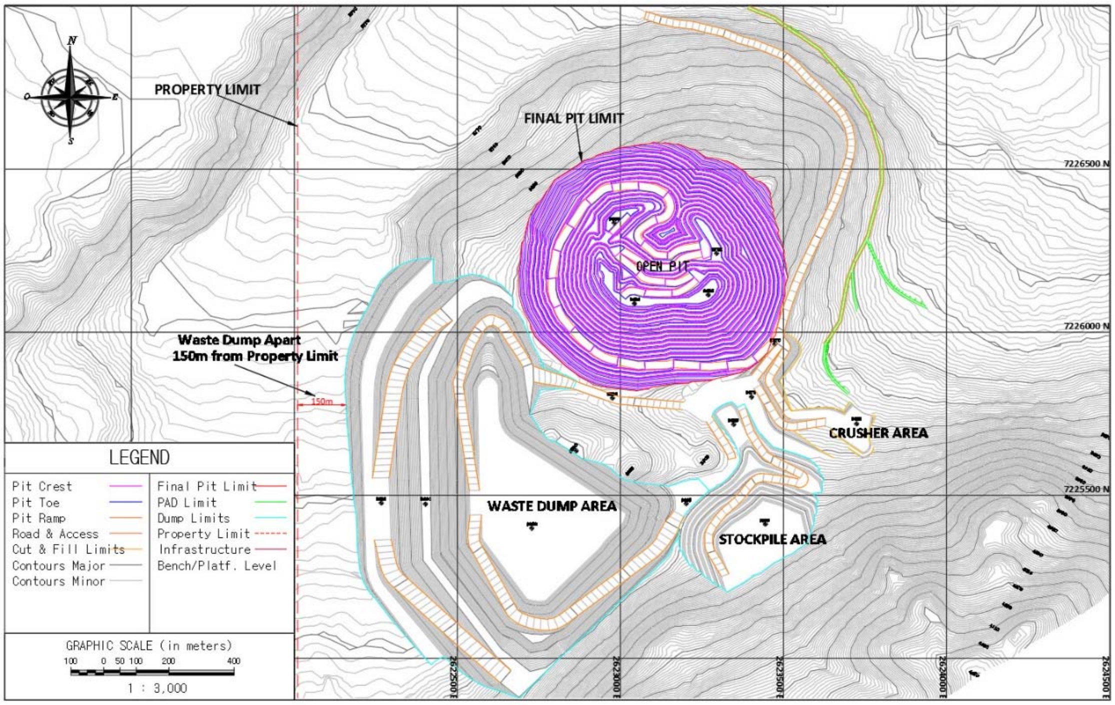



Figure prepared by Fortuna, 2017

**Figure 16.9 Cross section of static analysis results**

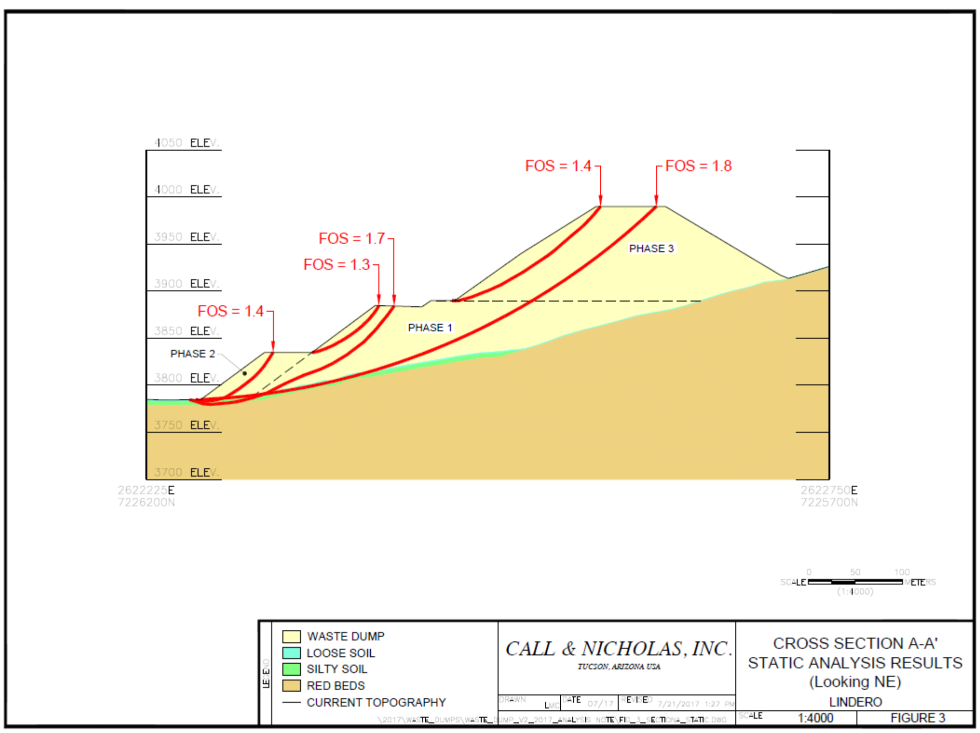



Figure prepared by CNI, 2017

## 16.4 Mine production schedule

Proven and Probable Mineral Reserves were used to schedule the mine production, and Inferred Mineral Resources inside the pit were considered as waste. The final production schedule uses trucks and wheel loaders as required to produce the ore to be fed into the crusher plant, and maintain stripping requirements for each phase.

Table 16.3 shows the planned mine production schedule, including re-handle from stockpile. Ore production will consist of Proven and Probable Mineral Reserves mined from the pit. The material stockpiled will be divided into two classes by grade: high-grade ore is ore equal to or above a 0.4 g/t Au cut-off; low-grade ore is ore equal to or above 0.26 g/t Au but below a 0.4 g/t Au cut-off grade.

The proposed production schedule starts with two months of pre-production, initially mining 821,000 t of ore. This material will be fed to the process plant as part of commissioning. The annual process production rate is projected to be 6,750,000 t, which represents a daily rate of 18,750 t for 360 calendar days.

Mining rates will vary, and ore will be hauled to either the crusher or a nearby stockpile. Stockpiles will be maintained to smooth production to the crusher, as well as to optimize Project economics by allowing higher-grade material to be fed before lower-grade material. All high-grade material will be hauled directly to the crusher.

When possible, material will be fed directly into the crusher to reduce rehandle costs. Table16.3 shows the planned production schedule and stockpile rehandle.

### Table 16.4 Proposed production schedule including stockpile rehandle

| Parameter | Units | FY0 | FY1 | FY2 | FY3 | FY4 | FY5 | FY6 | FY7 | FY8 | FY9 | FY10 | FY11 | FY12 | FY13 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Plant Days | | 60 | 360 | 360 | 360 | 360 | 360 | 360 | 360 | 360 | 360 | 360 | 360 | 360 | 360 | |
| Mine Days | | 60 | 365 | 366 | 365 | 365 | 365 | 366 | 365 | 365 | 365 | 366 | 365 | 365 | 365 | |
| Ore to Crusher | kt | 822 | 6,750 | 6,750 | 6,750 | 6,750 | 6,750 | 6,750 | 6,750 | 6,750 | 6,750 | 6,750 | 6,750 | 6,561 | 6,639 | **88,272** |
| Au | g/t | 0.55 | 0.94 | 0.86 | 0.66 | 0.63 | 0.56 | 0.49 | 0.58 | 0.54 | 0.50 | 0.59 | 0.52 | 0.56 | 0.59 | **0.62** |
| Cu | % | 0.08 | 0.10 | 0.13 | 0.10 | 0.12 | 0.10 | 0.11 | 0.11 | 0.09 | 0.12 | 0.11 | 0.10 | 0.12 | 0.11 | **0.11** |
| Oz Recovery | % | 78.26 | 77.4 | 76.4 | 76.0 | 76.6 | 76.1 | 76.1 | 75.6 | 75.2 | 75.3 | 75.6 | 75.3 | 73.4 | 75.1 | **75.74** |
| Oz Extracted | koz | 11.5 | 157.3 | 143.2 | 108.5 | 104.2 | 91.7 | 81.3 | 94.4 | 87.6 | 82.0 | 96.1 | 85.6 | 87.4 | 96.9 | **1,328** |
| Oz Contained | koz | 14.6 | 203.7 | 187.5 | 142.7 | 136.1 | 120.5 | 106.8 | 125.0 | 116.5 | 108.9 | 127.1 | 113.6 | 119.0 | 129.0 | **1,751** |
| **ROM to Crusher** | **kt** | **822** | **6,750** | **6,448** | **6,631** | **6,168** | **6,034** | **4,413** | **6,750** | **6,749** | **5,268** | **6,662** | **5,771** | **6,250** | **5,753** | **80,468** |
| HG Out Stock | kt | - | - | 302 | 119 | 582 | - | 548 | - | - | - | - | 737 | 142 | 726 | **3,156** |
| Au | g/t | - | - | 0.77 | 0.59 | 0.53 | - | 0.46 | - | - | - | - | 0.47 | 0.56 | 0.71 | **0.57** |
| Cu | % | - | - | 0.09 | 0.08 | 0.08 | - | 0.08 | - | - | - | - | 0.10 | 0.10 | 0.12 | **0.09** |
| HG In Stock | kt | - | 696 | 855 | - | - | - | - | - | - | - | 737 | 142 | 175 | 551 | **3,156** |
| Au | g/t | - | 0.63 | 0.50 | - | - | - | - | - | - | - | 0.47 | 0.56 | 0.96 | 0.63 | **0.57** |
| Cu | % | - | 0.08 | 0.08 | - | - | - | - | - | - | - | 0.10 | 0.10 | 0.17 | 0.10 | **0.09** |
| HG Stock Balance | kt | - | 696 | 1,249 | 1,130 | 548 | 548 | - | - | - | - | 737 | 142 | 175 | - | - |
| Au | g/t | - | 0.63 | 0.51 | 0.50 | 0.46 | 0.46 | - | - | - | - | 0.47 | 0.56 | 0.96 | - | - |
| Cu | % | - | 0.08 | 0.08 | 0.08 | 0.08 | 0.08 | - | - | - | - | 0.10 | 0.10 | 0.17 | - | - |
| LG Out Stock | kt | - | - | - | - | - | 716 | 1,789 | - | 1 | 1,482 | 88 | 242 | 169 | 160 | **4,648** |
| Au | g/t | - | - | - | - | - | 0.40 | 0.33 | - | 0.29 | 0.28 | 0.31 | 0.28 | 0.37 | 0.38 | **0.33** |
| Cu | % | - | - | - | - | - | 0.07 | 0.07 | - | 0.10 | 0.07 | 0.12 | 0.07 | 0.08 | 0.07 | **0.07** |
| LG In Stock | kt | - | 1,249 | 1,706 | 539 | - | 141 | - | 24 | 330 | 92 | 238 | 169 | 160 | - | **4,648** |
| Au | g/t | - | 0.34 | 0.34 | 0.28 | - | 0.28 | - | 0.28 | 0.28 | 0.31 | 0.28 | 0.37 | 0.38 | - | **0.33** |
| Cu | % | - | 0.06 | 0.08 | 0.07 | - | 0.07 | - | 0.08 | 0.06 | 0.12 | 0.07 | 0.08 | 0.07 | - | **0.07** |
| LG Stock Balance | kt | - | 1,249 | 2,955 | 3,493 | 3,493 | 2,918 | 1,129 | 1,153 | 1,482 | 92 | 242 | 169 | 160 | - | - |
| Au | g/t | - | 0.34 | 0.34 | 0.33 | 0.33 | 0.31 | 0.28 | 0.28 | 0.28 | 0.31 | 0.28 | 0.37 | 0.38 | - | - |
| Cu | % | - | 0.06 | 0.07 | 0.07 | 0.07 | 0.07 | 0.07 | 0.07 | 0.07 | 0.12 | 0.07 | 0.08 | 0.07 | - | - |
| **Waste** | **kt** | **1,762** | **6,805** | **7,290** | **8,211** | **10,308** | **10,200** | **10,409** | **8,805** | **8,481** | **10,671** | **9,932** | **7,204** | **8,636** | **1,084** | **109,798** |
| **Total Movement** | **kt** | **2,583** | **15,500** | **16,600** | **15,500** | **17,058** | **17,091** | **17,159** | **15,580** | **15,561** | **17,513** | **17,658** | **14,265** | **15,533** | **8,274** | **205,874** |
| SR: | w/o | 2.14 | 0.78 | 0.81 | 1.15 | 1.67 | 1.65 | 2.36 | 1.30 | 1.20 | 1.99 | 1.30 | 1.18 | 1.31 | 0.17 | **1.24** |
| Throughput Ore | tpd | 13,693 | 18,750 | 18,750 | 18,750 | 18,750 | 18,750 | 18,750 | 18,750 | 18,750 | 18,750 | 18,750 | 18,750 | 18,226 | 18,442 | - |
| Production | tpd | 43,056 | 42,466 | 44,531 | 42,139 | 45,139 | 44,863 | 40,498 | 42,684 | 42,629 | 43,919 | 48,004 | 36,401 | 41,704 | 20,239 | - |
| Movement | tpd | 43,056 | 42,466 | 45,355 | 42,465 | 46,734 | 46,825 | 46,882 | 42,684 | 42,633 | 47,980 | 48,245 | 39,083 | 42,556 | 22,668 | - |

## 16.5 Equipment selection and productivities

The planned Lindero mine will be an open pit mine using conventional haul trucks and wheel loader equipment. The proposed owner-mining will cover the conventional practices of loading, hauling, and drilling. Blasting will be conducted by contractors throughout the LOM.

The initial mining fleet will consist of two 17 $yd^3$ wheel loaders to load 91 tonne haul trucks with a third wheel loader to be in Year 3 for the LOM as part of the mines sustaining capex. A loader cycle time of 39 seconds is used for loading and placing material into a truck. A 95 % bucket fill factor has been assumed. Total productivity is estimated to be 1,600 tonnes per operating hour which includes a 94 % efficiency factor. It has been assumed that 16 operating hours are available per 24-hour day, which allows for startup, breaks, safety, meetings, and shutdown.

Truck productivity for owner-mining operations is based on haul and return speeds for each segment of travel. 3-D line segments were drawn using MineSight™ haulage mine planning software, and the speeds were flagged by using rimpull curves for Komatsu HD785 trucks. The haulage profile represents the travel routes along in-pit ramps and external pit roads. MSO and haulage MineSight™ software was used for scheduling, as well as to report the truck hours required based on profiles and speeds.

Available truck hours were based on 16 operating hours per 24-hour day and were adjusted by truck availability. Truck availability was assumed to start at 87 % for new trucks, and reduced by 1 % per year to a minimum value of 85 %. These factors are considered to be reasonable with respect to the operating conditions at Lindero.

Haul truck and wheel loader equipment fleet estimates for the LOM including mine plan sustaining capex and based on simulation analysis are shown in Figure 16.10.

**Figure 16.10 Fleet equipment for hauling and loading**

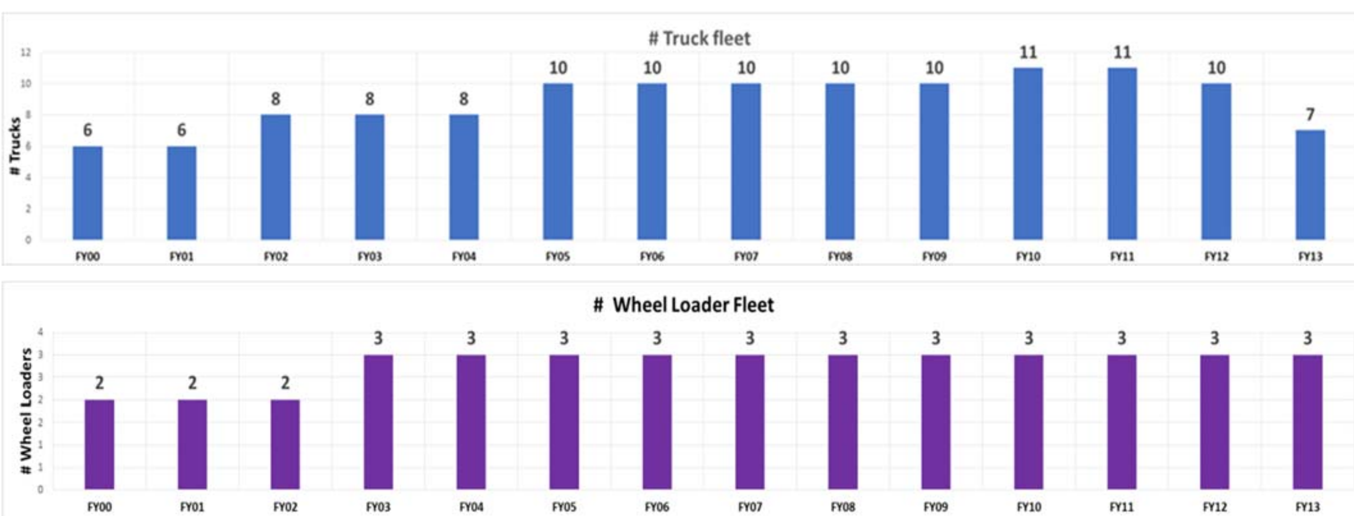



Figure prepared by Fortuna, 2017

## 16.6 Mine personnel

The Lindero mining operation is planned to be owner-operated. using conventional open pit operational practices, with drilling, loading, hauling, support, and administrative functions. The operational practice for blasting will be conducted by contractors. The mine will operate 365 scheduled days per year, 24 hours per day, divided into two 12-hour shifts per day for mine operations and mine maintenance.

The mine organization will include functional groups for mine operations (drilling, blasting, loading and hauling), maintenance (mine and plant), laboratory, logistic, geology and mine planning. In addition, the organization will be staffed to support safety and environmental requirements. Proposed mining-related functional groups are organized under the operation manager. The operation manager is allocated functional groups for mine operations, maintenance and technical services (geology, planning, and laboratory). Among the functional groups responsible for mine operations, drilling and blasting are managed together, as are loading and hauling. The technical services function for the planned operations will include technical engineers in mine planning, surveying, geotechnical engineering and mine operations, as well as geologists for grade control. The Health, Safety and Environmental manager is allocated functional groups for health, safety, and environmental as shown in Figure 16.8.

Total mining manpower is projected to be 340 internal employees and 85 contractor employees.

**Figure 16.11 Owner mining mine organization chart**

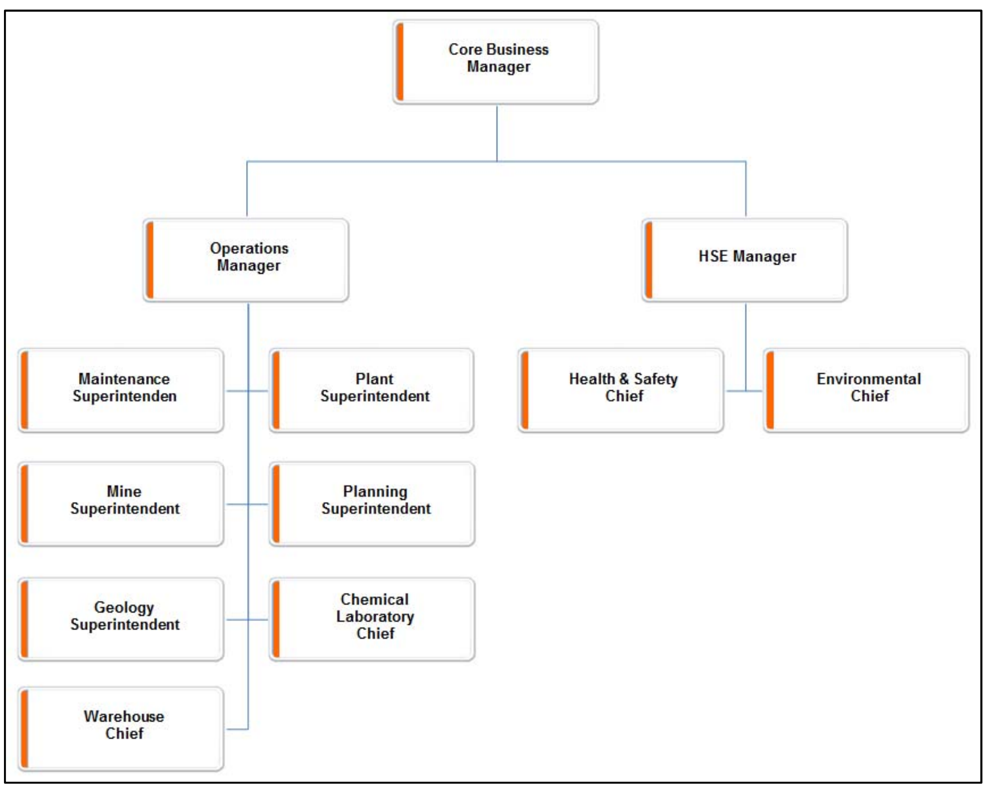



Figure prepared by Fortuna, 2017

## 16.7 Comments on Section 16

The QP is of the opinion that:

- The mining method being used is appropriate for the deposit being mined. The open pit, stockpile, waste dump designs, and equipment fleet selection are appropriate to reach production targets
- The mine plan is based on successful mining philosophy and planning, and presents low risk
- Inferred Resources are not included in the mine plan
- The mobile equipment fleet presented is based on simulations and bench marks of similar operations achieving similar production targets
- All mine infrastructure and supporting facilities meet the needs of the current mine plan and production rate
- Major planned maintenance of the main equipment, such as loaders and trucks, have been cover in sustaining capital by purchasing additional equipment that can replace any possible lost production hours and not impact production targets
- The ancillary equipment appears to be undersized, especially dozers, but this would be covered by renting additional equipment as necessary

# 17 Recovery Methods

## 17.1 Proposed process description

Most of the major process concepts presented in the 2016 Technical Report such as: HPGR-crushing, cyanide heap leaching and carbon adsorption recovery, remain unchanged for this update. Additional physical and metallurgical interpretations, developed by the testwork conducted by Fortuna in 2016 and 2017, resulted in modifications in the approach to these major process concepts for the Lindero Project in this Report as follows:

- A concentrated cyanide cure was added to shorten the leach cycle and increase extraction
- Agglomeration with cement was added to support a 110-m-high heap with the HPGR-crushed ore
- Conveyor stacking was included from start-up
- Two-stage leaching was included to increase pregnant grades and reduce overall flowrate to the ADR plant
- A SART plant was included to control the copper in solution
- Leach solution flow will be increased 150 % in Year 4 to reduce in-heap gold inventory

Unit operations for the Lindero process design were selected based on the physical and metallurgical needs of the Lindero ore to achieve maximum gold extraction. No novel or untried technology will be employed in the process. A general site plan showing the process facilities is included as Figure 17.1.

**Figure 17.1 Simplified site plan showing process-related facilities**

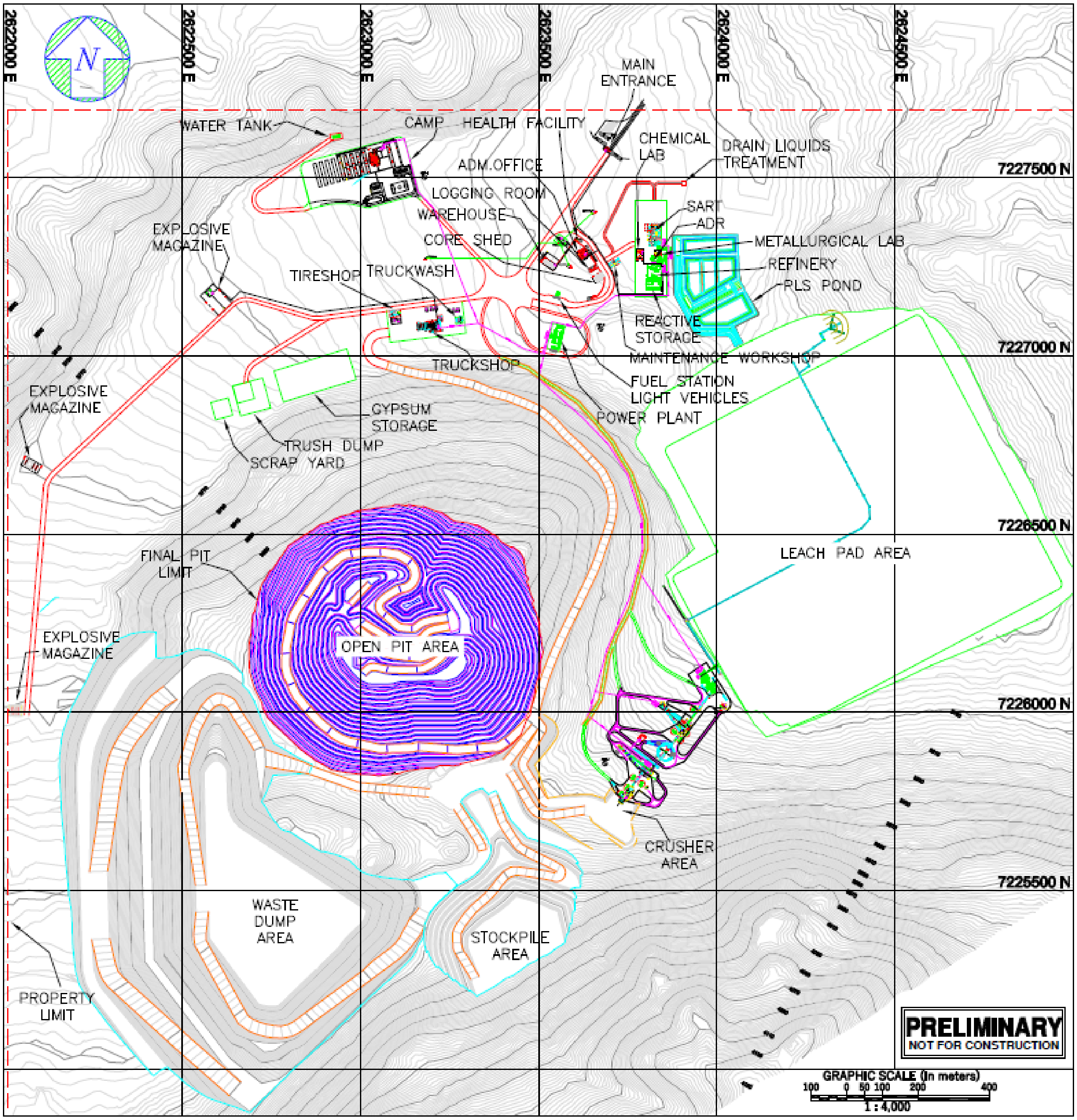



Figure prepared by Saxum, 2017

### 17.1.1 Fundamental design criteria

Basic design is complete for most process areas and includes flowsheets, mass balances, equipment lists, general arrangements, piping line schedules, instrument lists and piping and instrumentation diagrams (P&IDs) developed to a basic engineering level.

Table 17.1 shows the basic production design criteria that apply to the processing facilities. It should be noted that the angle of repose for the heap leach pad is relatively shallow due to the ore agglomerating to relatively spherical pellets and the requirement for a low moisture content in the cyanide cure.

**Table 17.1 Fundamental processing design criteria.**

| | | | |
|---|---|---|---|
| **Production Rate:** | | | |
| | Average, dry tpd | | 18,750 |
| **Ore:** | Gold/After Field Deduct | Au (g/t) | Extraction % |
| | Met 1 | 0.65 | 75.4 |
| | Met 2 | 0.68 | 78.2 |
| | Met 3 | 0.46 | 78.5 |
| | Met 4 | 0.49 | 68.5 |
| **Ore:** | Copper | Cu (%) | Extraction % |
| | Met 1 | 0.11 | 11.8 |
| | Met 2 | 0.12 | 28.9 |
| | Met 3 | 0.10 | 9.8 |
| | Met 4 | 0.11 | 8.6 |
| **Operating Schedule:** | | | |
| | Days per Week: | | 7 |
| | Shifts per Day: | | 2 |
| | Shift Duration, hours: | | 12 |
| | Total operating, days/yr: | | 365 |
| **Heap Configuration:** | | | |
| | Height, m/lift: | | 10 |
| | Number of lifts: | | 11 |
| | Average Angle of Repose, | | 32 |
| | Overall Angle of Repose, X:1: | | 2 |
| | Cell Width, m: | | 60 |
| | Cure time on heap, days: | | 4 |
| | Application rate, l/hr•m$^2$: | Primary: | 12 |
| | | Secondary: | 6 |
| | **Heap Flows Year 0 - 3** | | |
| | Primary Cycle, days: | | 30 |
| | Secondary Cycle, days: | | 60 |
| | Solution Flows, m$^3$/hr: | Primary: | 400 |
| | | Secondary | 400 |
| | **Heap Flows Year 4 - 13** | | |
| | Primary Cycle, days: | | 45 |
| | Secondary Cycle, days: | | 90 |
| | Solution Flows, m$^3$/hr: | Primary: | 600 |
| | | Secondary | 600 |
| **SART** | | | |
| | PLS to SART, m$^3$/hr: | | 400 |

### 17.1.2 Flowsheets

A simplified crushing flowsheet is included as Figure 17.2 and a simplified process flowsheet is shown in Figure 17.3.

**Figure 17.2 Simplified crushing flowsheet**

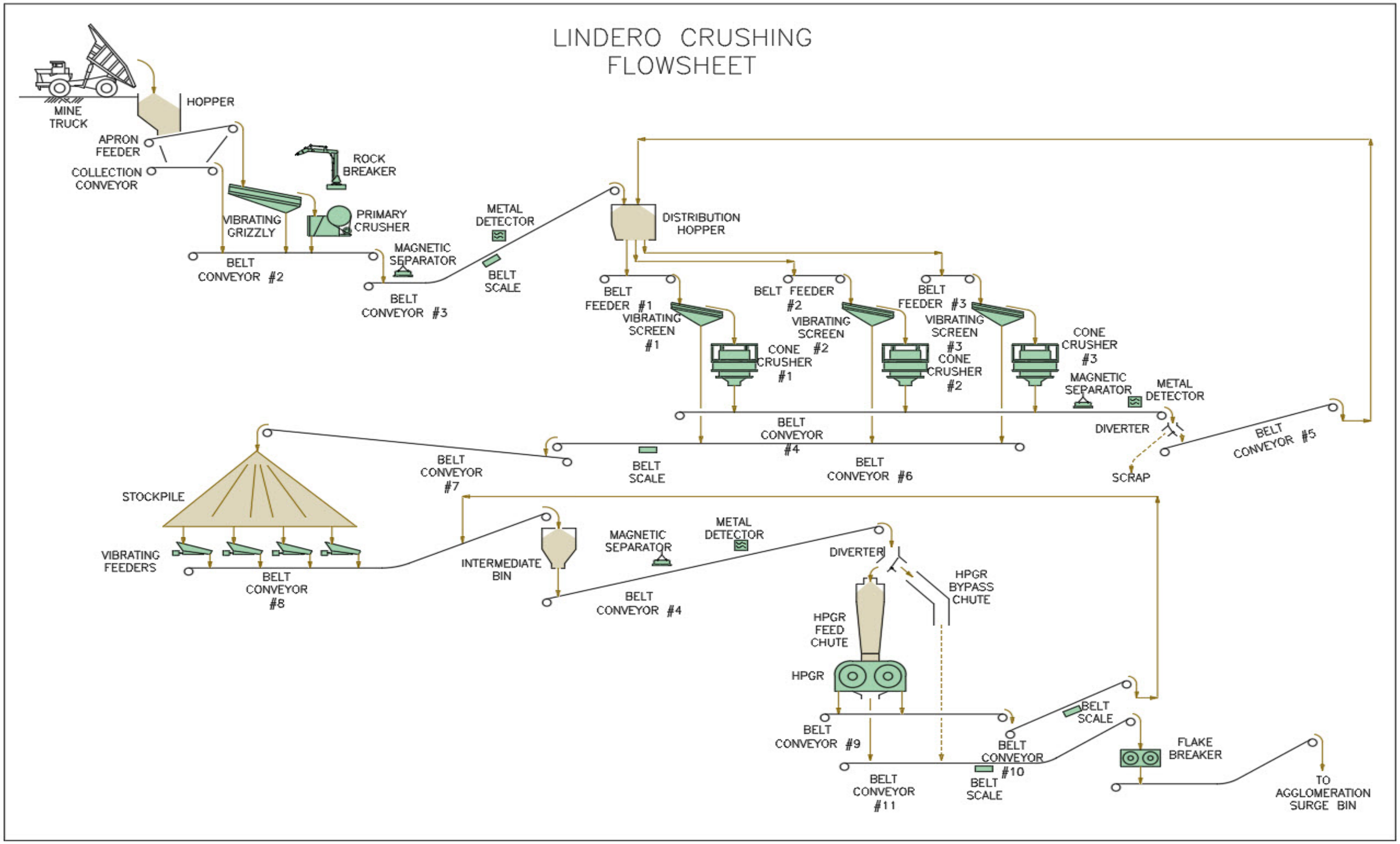



Figure prepared by Saxum, 2017

### Figure 17.3 Simplified overall process flowsheet

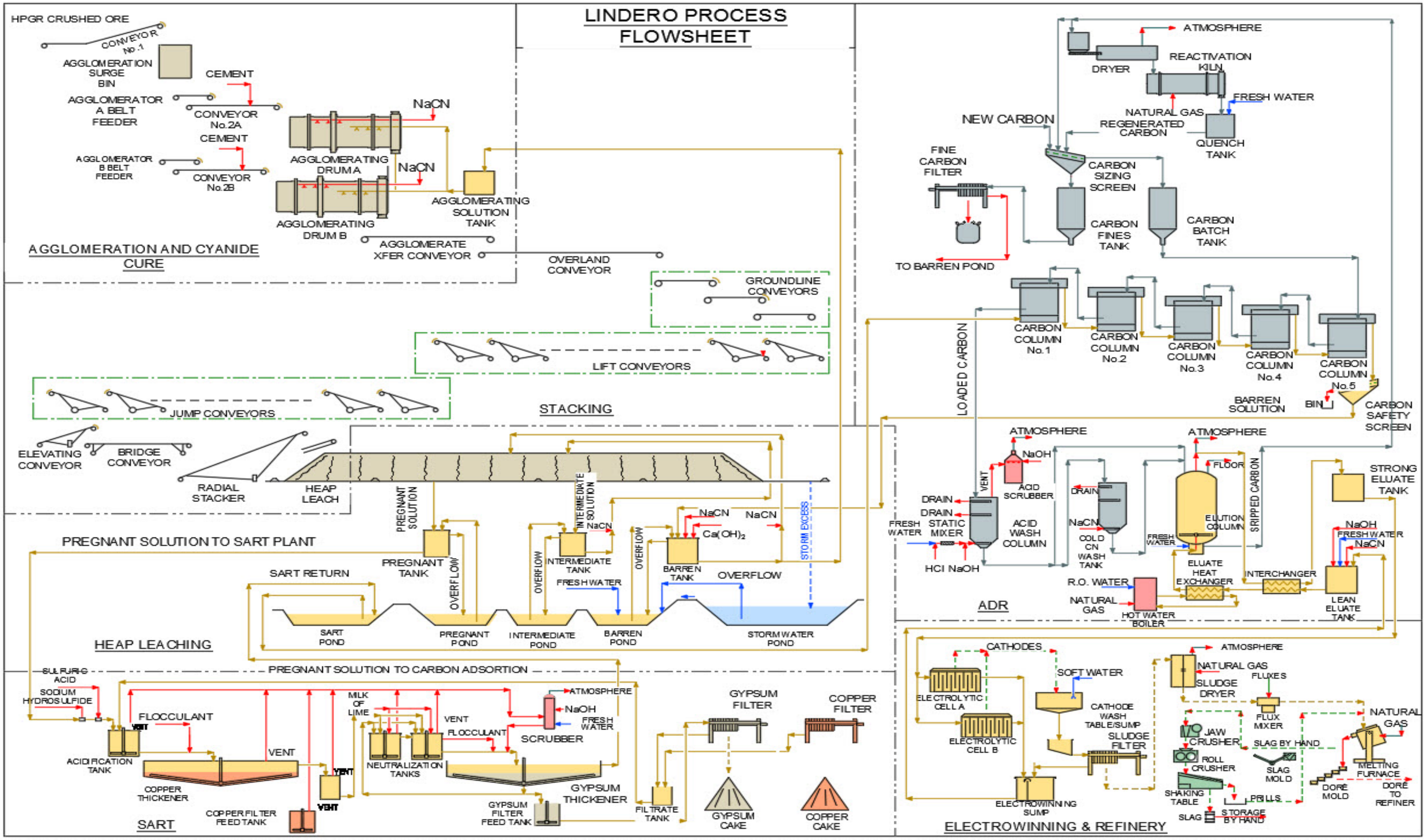



Figure prepared by Saxum, 2017

### 17.1.3 Crushing

Run-of-mill (ROM) ore will be transported from the mine in 91-tonne surface haul trucks and dumped in a 200-tonne capacity ROM feed bin.

Material will be fed from the ROM feed bin to a 1.2 m x 7.9 m apron feeder and then to a grizzly feeder with 150 mm openings. The grizzly oversize will be fed to a 1500 mm x 1300 mm jaw crusher and the grizzly undersize will be recombined with the jaw crusher product on the primary crusher discharge conveyor which will feed a 175-t secondary crushing distribution bin. A tramp metal electromagnet and metal detector will be installed on the primary crusher discharge conveyor to protect the secondary crushers.

Ore from the distribution bin will be fed equally to three 1.8 m x 6.1 m secondary double deck screens by belt feeders. Screen over-size will report to a 300 kW cone crusher. Three screens/cone crushers in parallel are planned. The cones will be run in closed-circuit with the screens.

Stockpiled material will be reclaimed using vibrating feeders underneath the pile, which will feed a conveyor and then into a 140-t tertiary crusher surge bin. Discharge from the surge bin will be transported to the feed chute immediately above the HPGR. A tramp metal electromagnet and metal detector will be installed on the feed belt immediately before the HPGR chute to protect the HPGR from tramp metal.

The HPGR is a Weir-KHD Model RPS 16-170/180 with 1.7 m diameter by 1.8 m wide rolls, and was purchased by Goldrock. The edge material which represents roughly 25 % of the total product will be recycled back to the HPGR feed bin. The final plant product from the HPGR will be nominally 80 % passing 6 mm. The center product will be fed to the flake breaker.

The crushing flowsheet shown in Figure 17.2 is virtually unchanged from that outlined in the 2016 Technical Report. The only substantive changes were to include a surfactant enhanced dust suppression system to minimize water addition, and add a flake breaker to the end of the crushing circuit.

### 17.1.4 Flake breaker

Observation shows that HPGR crushing will generate flakes when treating the Lindero ore. These flakes will be sufficiently durable that they will survive the agglomerating drum and will pass unagglomerated to the heap. In order to break these flakes, a Gundlach continuous tooth roll crusher will be placed in the circuit. The roll crusher will contain two opposing inter-meshed timed rolls with a 25 mm gap. A fixed scalping screen will divert any non-flaked material around the crusher. The product of the de-flaking crusher will report to the agglomeration surge bin.

### 17.1.5 Agglomeration

The agglomeration circuit is designed to mix concentrated cyanide and cement with the HPGR-crushed ore, and adhere the fine particles to the coarse rock by tumbling the mixture in a rotating drum.

The surge bin will contain 1,000 t. This bin is designed to allow smooth flow to the agglomerating drums while isolating the HPGR circuit from the agglomerating circuit. This is needed to allow the frequent shutdown of the stacking system to allow conveyor moves without disrupting the HPGR operation.

Two 3.66 m diameter by 12.2 m long agglomerating drums are required for the operation. Each drum will be fed by a variable speed feeder under the agglomerating surge bin leading to a separate conveyor that will elevate the ore to the drum. Cement will be metered onto the conveyor, controlled proportionally by a belt scale. The conveyor and discharge will be covered to prevent the wind from blowing the cement off the belt.

Two cement silos are included, one for each drum. Each silo will hold 325 t of cement. Cemento Filler (CPF40 per IRAM specification) is available locally, and has been selected for the Project. Bulk cement will be delivered by pneumatic truck. A blower will be provided to unload the trucks.

Discharge of the silo will include a vibrating bin bottom and variable rate feeder. Screw conveyors will transport the material to the belt conveyor where it will be placed on top of the ore. A range of 0.25 to 6.5 kg/t is specified for the cement addition system.

As the dry ore and cement enter the drum, cyanide solution will be sprayed onto the ore. The moisture will combine with the cement and the tumbling action in the drum will blend the cement with the fines and the coarser rocks rolling down the cascading charge will pick up the fines that coat the exterior.

The agglomerating drums will be fitted with an expanded metal liner and a scraper bar. The spray manifold on the discharge end will contain control valves and flowmeters to allow the operator to adjust the addition of cyanide and moisture as the ore mix dictates.

Two solution tanks are included at the agglomerating area for applying the appropriate moisture and cyanide to the agglomerating drums. One tank will hold concentrated cyanide solution and the second will hold barren solution. Solution will be applied to the tumbling ore near the feed end of the drum. A fixed quantity of concentrated cyanide will be added equivalent to a fraction of the expected consumption for the ore type mix, and the quality of agglomerate will be adjusted with barren solution.

Moisture in the HPGR product will be limited to 2 % moisture to allow sufficient solution to be added to achieve the cyanide concentration desired in the cure. Moisture in the agglomerates will be kept at a minimal level so the fines from the uncured agglomerates do not stick to the conveyor belts. Also, since leaching is very rapid, any excess solution discharging from the agglomerates would contain high-grade gold.

The agglomerating drums, solution tanks and conveyors will be in a contained area that will discharge to the heap leach pad. The green agglomerates will overflow the discharge end of the drum onto a conveyor that will transfer the material to the overland conveyor on the leach pad.

### 17.1.6 Stacking

Agglomerated ore stacking will be undertaken using conventional conveyors and a radial stacker. Due to the size of the Lindero heap leach pad, the conveyor stacking system requires a considerable quantity of equipment. Most of the ore will be retreat stacked; however due to the steep side slopes, portions near the edge of each lift will be advance stacked. The entire heap stacking sequence was designed such that the quantity of advance stacking was minimized. Each cell was divided into 60 m x 60 m x 10 m blocks and ore was assigned to each block based on the mine plan. The amount of cement for each block was then calculated for the ore blend and the weight of material in the lifts above each block.

The number of pieces in the conveyor string varies based on the heap height and phase. In addition to the radial stacker and bridge conveyor, different types of conveyors will be required to transport the ore to the heap. Ground line conveyors will be portable 150-m-long conveyors that are used to cover longer distances. Jump conveyors will be 30-m-long conveyors that are used to feed the radial stacker and allow it to move across the surface of the heap by adding or removing conveyors. Some of the jump conveyors will have higher kW motors that will be used to transfer the agglomerates between lifts. The requirement for these conveyors with respect to time is detailed in Table 17.2.

**Table 17.2 Conveyor stacking equipment**

| Conveyor | | | |
|---|---|---|---|
| Cum. Yrs | Ground Line | Jump | Cum. Jump |
| 0.1 | 3 | 15 | 15 |
| 0.5 | 3 | 24 | 24 |
| 1.2 | 3 | 24 | 24 |
| 2.0 | 3 | 28 | 28 |
| 2.8 | 3 | 33 | 33 |
| 3.5 | 3 | 32 | 33 |
| 3.7 | 3 | 20 | 33 |
| 4.1 | 3 | 29 | 33 |
| 5.0 | 3 | 30 | 33 |
| 6.1 | 2 | 47 | 47 |
| 7.8 | 2 | 42 | 47 |
| 9.3 | 2 | 42 | 47 |
| 10.6 | 2 | 46 | 47 |
| 11.7 | 2 | 36 | 47 |
| 12.7 | 2 | 34 | 47 |
| 13.0 | 2 | 33 | 47 |

The jump conveyors will transfer ore to the elevating conveyor which will place the ore on the bridge conveyor.

The bridge conveyor will be self-propelled and will be attached to the radial stacker. The radial stacker will be equipped with luffing and slewing capability. It will also be equipped with an extendable "stinger" to adjust the length, allowing greater placement of ore between moves.

The bridge conveyor and radial stacker will move in retreat as a unit. As the load point on the bridge conveyor reaches the limit of travel, the system will be shut down, a jump conveyor will be removed from the string, and the elevating conveyor will be relocated to continue stacking.

A low ground pressure loader will be used to remove the individual jump conveyors during stacking moves. These conveyors will be placed at an identical position in the next cell so as to have the conveyor string in place when the stacker is relocated to that cell.

### 17.1.7 Heap leaching and solution handling

Once the heap has been stacked and the agglomerates have been allowed to cure for four days, leach solution will be applied to the ore through an array of emitters. Emitters will be placed in 60 m x 60 m panels. Each panel will equate to 43.2 $m^3$/hr of flow in the primary cycle. Additional emitters will be placed to cover the side slopes. The emitters in the main area of the heap will be buried up to the headers to reduce evaporation. The emitters will be recovered prior to placing a new lift on top. Miniature valves will be placed on every other emitter line to allow the flow to be reduced to 6 l/hr/$m^2$ for the secondary cycle.

Effluent solutions will be segregated using berms at the bottom of the heap. Two stages of leaching will be employed.

The first stage of leaching will apply the intermediate pregnant solution to the newest ore at a rate of 12 l/hr/$m^2$. This solution will be applied to the ore for 30 days. The effluent from the first stage leaching will flow to the pregnant tank.

Barren solution from the carbon-in-column (CIC) will be the leach solution for the second stage of leach. This solution will be applied to the ore, after it has received 30 days of leaching in the primary cycle, at a rate of 6 l/hr/$m^2$. The secondary cycle will be maintained for 60 days. Effluent from the heap due to the secondary cycle will be the intermediate solution that will flow to the intermediate tanks and will be reapplied to the heap in the primary cycle.

Solution flow is assumed to be 400 $m^3$/hr for both the primary and secondary cycle. In Year 4 the flowrate is planned to increase to 600 $m^3$/hr. This is required to provide more rinsing of the heap at the higher lifts to reduce inventory.

Effluent from the heap will flow by gravity in pipelines on the down-slope edge of the heap to either the pregnant tank or the intermediate tank. Barren from the CIC circuit will flow to the barren tank. The heap solution tanks will be large vertical carbon steel tanks. These tanks overflow to individual ponds that provide for surge.

A submersible pump will be provided at a low point in each pond to return the solution to the respective tanks.

Pregnant solution from the heap will flow to the pregnant tank. These solutions will be pumped to the SART plant for copper removal.

### 17.1.8 Leach pad

Total ore production is estimated at 88.7 Mt with the designed leach pad capacity being approximately 90.7 Mt, assuming a heap bulk density of 1.60 t/$m^3$ and taking into account a loss of capacity due to ramps, The leach pad will be divided into multiple drainage cells, and will have a lined surface area slightly greater than 101 ha. The leach pad will be constructed in two phases. The Phase 1 pad will comprise about 49 % of the ultimate area. Phase 2 will see the installation of the balance of the lined area. The Phase 2 pad construction will begin in Year 3.5. The pad surface will match the natural topography as much as practicable so as to minimize required earthworks.

A monitoring system will be installed below the heap leach pad consisting of a grid of perforated polyethylene pipes placed in trenches and filled with drainage gravel which will capture any leaks through the leach pad geomembrane liner and report to a monitoring sump downstream of the ponds.

Downslope berms will be 2.0 m high with 1.5:1 (horizontal: vertical) slopes. Exterior berms for the up-slope and sides will be 1.2 m high with a similar profile. Interior berms separating the cells will be 1.5 m high.

The pad liner system will consist of a combination of low permeability soil (natural soil mixed with bentonite) and geosynthetic clay liner (GCL) overlain by a 1.5 mm linear low-density polyethylene (LLDPE) single-side-textured geomembrane liner.

The low permeability soil liner will be used in the toe area for stability purposes, which will consist of in-situ soil mixed with bentonite. In the balance of the leach pad area GCL will be placed over compacted native soil. The LLDPE geomembrane will cover the amended soil and GCL.

### 17.1.9 Solution collection system

Corrugated, perforated 100 mm polyethylene pipes will be placed on top of the geomembrane under the overliner to collect solution. These lateral collection pipes will be placed on nominal 10 m centers and will be connected to the primary collection pipes located along the internal berms at the lower edge of each cell. Primary collection pipes will also be corrugated and perforated polyethylene. The primary pipes will vary from 300 mm to 450 mm in diameter depending on the collection area and slope of the drainage cell.

The primary collection pipe for each cell will transition to a solid pipe at the down slope berm and then pass through the berm to connect to two main collection pipes in the solution collection ditch. The intermediate and pregnant solution will be directed by means of valves, and will flow by gravity to the intermediate or pregnant tank. During a high rainfall event, flow in excess of the capacity of the active main collection pipes will back up at the down-slope heap edge until it reaches the cell overflow point. The overflow will report to the solution collection ditch and thence to the storm water pond.

### 17.1.10 Overliner

A 60 cm thick drainage layer (overliner) of screened rock will be placed on top of the geomembrane and perforated solution collection pipes. This layer will facilitate solution drainage and protect the liner during heap stacking. It is anticipated that the overliner material will be sourced from a borrow pit adjacent to the leach pad and screened to 100 % passing 37.5 mm. This material will be placed in the area that is retreat stacked using over-the-road dump trucks backing onto the pad and using a dozer to spread the material. This material will cover the internal berms to a 60 cm depth to allow equipment to move the stacking equipment from one cell to the next across the berms.

### 17.1.11 Ponds

The process pond system will be divided into three parts. The active solution pond will be divided by two low berms in the bottom to partition it into barren, intermediate and pregnant ponds without significantly reducing the total surge volume. A second pond (storm water pond) will be used for containment of excess solution during a sudden storm event. The third pond will be used for surge and particulate settling between the SART plant and the carbon columns.

Process ponds are sized to contain a working volume of 24 hours at the total heap irrigation flow rate (primary and secondary), plus a drain-down volume equal to 24 hours at the total heap irrigation flow rate. The combined capacity of the process ponds will be 50,000 $m^3$, with the planned capacity of the storm water pond also being 50,000 $m^3$. The combined capacity of these ponds will be sufficient to contain a 100-year 24-hour rainfall event of 56.1 mm.

The ponds will be constructed of a layer of GCL placed on compacted soil, followed by a secondary liner of 1.5 mm high density polyethylene (HDPE). A geonet will be placed on top of the secondary liner followed by a 1.5 mm HDPE primary liner.

Leak prevention of the ponds is via a geonet sandwiched between the two HDPE geomembranes on top of the GCL liner for collecting any leakage through the primary liner. Below the ponds a monitoring system consisting of a grid of perforated polyethylene pipes placed in trenches and filled with drainage gravel will capture any leakage through the secondary liner. This monitoring system will report to a monitoring sump downstream of the ponds.

#### 17.1.12 SART plant

Due to the elevated content of cyanide-soluble copper in the deposit, the SART process was selected to control copper concentrations in the leach solution. Elevated levels of copper in leach solution increase the consumption of NaCN, and under some circumstances can interfere with gold recovery operations. The SART process will remove copper from pregnant solution as copper-sulfide solids; these solids will in turn be sold for the contained metal values. Free cyanide will also be regenerated in the process as a result of copper removal, thereby reducing the overall consumption of NaCN at the site.

The SART process is shown in Figure 17.2. SART feed will be pregnant solution from the heap leach at a design flow rate of 400 $m^3$/hr. Sulfuric acid ($H_2SO_4$) will be injected into the solution to lower the pH to about 4.0, and sodium hydrosulfide (NaHS) will be added to precipitate copper-sulfide solids as synthetic chalcocite ($Cu_2S$).

Copper-sulfide solids will be thickened and then filtered for final dewatering. The final copper-sulfide filter cake will be bagged in 1 t supersacks, and then transported off-site for sale.

Overflow solution from the copper-sulfide thickener will be neutralized to pH 10.5 using lime slurry. Gypsum solids formed during neutralization will be removed from the solution using a thickener and then the solids will be filtered for additional dewatering. The filtered gypsum solids will be permanently disposed on site.

Treated solution from the SART process will have a low copper concentration and high free cyanide concentration, and this solution will overflow into the SART pond, and then be pumped to the carbon columns for gold recovery. A majority of the silver in pregnant solution will be recovered into the copper-sulfide solids as silver sulfide ($Ag_2S$). Sales terms for the copper-sulfide filter cake are anticipated to include payments for contained copper, gold, silver and zinc.

The copper-sulfide section of the SART plant will have solutions at pH 4.0, and at this pH, some degree of hydrogen cyanide (HCN) and hydrogen sulfide ($H_2S$) off-gassing will occur. To prevent releases of these gases, associated equipment will be fully enclosed and ventilated to a high-efficiency gas scrubbing system.

Hazardous gas monitors will be located throughout the SART plant to ensure a safe working environment. Process equipment containing low-pH liquids will be constructed of stainless steel for corrosion resistance. Acid-resistant concrete coatings will be used in areas where sulfuric acid is handled.

Three separate secondary containment areas will be provided for the sodium hydrosulfide area, the sulfuric acid area and for the main SART plant area. The main SART plant area secondary containment will also be configured to overflow into the SART pond and then pumped to the CIC circuit.

### 17.1.13 Carbon-in-column

Gold will be adsorbed from the SART effluent using activated carbon. The columns that contain the carbon will be configured as an up-flow cascading circuit where each column is elevated above the next and solution flow from one column to the next will be assisted by gravity. As the carbon in the first column becomes loaded with metals it will removed from the column using a pump. When the first column is empty the carbon batch in the second column will be transferred to the first column using an educator. This will be repeated until the fifth and last column is empty. The last column will then be filled with a batch of freshly reactivated carbon.

Solution overflowing the last column will then be barren solution, and will pass over a screen to retain any carbon, either fines or from an upset condition. The underflow of the screen will report to the barren tank for return to the heap for the secondary leach.

The CIC circuit is designed for 6 m x 16 m coconut shell carbon.

### 17.1.14 Acid washing

Activated carbon will accumulate carbonate scale during the adsorption cycle. This scale is caused by the calcium in the lime used for pH control and $CO_2$ absorbed from the atmosphere. This scale, typically, does not interfere with adsorption of gold onto the carbon but will extend the elution time and consume more carbon during reactivation. Scale will be removed by soaking the carbon in a hydrochloric (HCl) acid solution.

The acid wash solution will be made up to about 3 % HCl and pumped into the acid wash vessel which will be constructed of fibre-reinforced plastic (FRP). The carbon from the first stage of the CIC will be dewatered and discharge into the acid wash tank. Once the column is full, solution will be circulated through the bed of carbon and additional acid will be injected into the recirculating stream as required.

Once the pH of the circulating stream stabilizes below 1.0, in roughly two hours, the solution will be drained from the carbon and the carbon will be rinsed with a sodium hydroxide solution to neutralize the acid. The neutralization step is required to reduce the evolution of hydrogen cyanide (HCN) and the corrosion in subsequent unit operations. The neutralized carbon will be transferred by pump to the cold cyanide wash circuit.

Hydrogen cyanide monitors will be included in this area to notify the operators if HCN gas is present. The acid wash area will be isolated from the rest of the plant by low containment walls with a separate gravity sump that flows to the barren pond. The floor will be coated with an acid resistant coating. All trench drains/sumps in the ADR will be fitted with carbon retention screens to prevent carbon from reaching the ponds through the plant drains. If circumstances require the discharge of significant quantities of acid solution to the gravity sump a mechanism will exist to add caustic solution to the sump to partially neutralize the acid.

Since HCl is a gas at room temperature, and with the high elevation of the Project site, all the tanks that have acid will be closed-top and will be vented to a caustic scrubber. An induced draft (ID) fan will maintain a slight negative pressure on the tanks to minimize acid vapor from entering the operating area. Overflow of the scrubber sump will report to the acid gravity drain.

### 17.1.15 Cold cyanide wash

Copper adsorbed onto the activated carbon is detrimental to the electrowinning circuit and the subsequent doré bar. In the electrowinning circuit the copper can manifest itself as a rapid dendritic formation which grows rapidly and will short the cathode to the anode. High copper content in the doré is penalized by the refiner.

The adsorption of copper onto carbon is dependent on the number of cyanide molecules associated with the copper ion. Copper cyanide is loaded, in order of preference:

$$Cu(CN)_2^- > Cu(CN)_3^= > Cu(CN)_4^\equiv$$

Soaking the acid washed and neutralized carbon in a sodium cyanide solution will convert some of the highly adsorbed $Cu(CN)_2$ species to the weakly adsorbed $Cu(CN)_4^\equiv$ species.

The arrangement of the cold cyanide wash circuit will be similar to the acid wash circuit but will be constructed of epoxy-lined carbon steel. Dewatered carbon will report to the cold cyanide wash tank where it will be contacted with a recirculating stream of 3 % NaCN solution for two hours. The solution will be discharged to the pregnant pond as part of the feed to the SART plant.

Once the cycle is complete the carbon will be drained and transferred by pump to the elution column.

### 17.1.16 Elution

Conditions that enhance gold adsorption onto carbon are high ionic strength, divalent cations, low temperatures and low pH (within reason). The elution circuit reverses some of these conditions to remove the gold from the carbon. Reverse osmosis (RO) water is used to make the elution solution to give it a lower ionic strength. Acid washing removes some of the divalent cations and the cold cyanide wash converts the adsorbed species to sodium based species. The elution solution is made with NaOH to achieve a high pH and the solution is heated to around 150ºC. As would be expected with the elevated temperature the elution column will be a coded carbon steel, epoxy-lined pressure vessel.

Elution solution will be stored in the lean eluate tank which will be insulated and constructed of epoxy-lined carbon steel. The solution will be pumped through heat exchangers where it will be heated to appropriate temperature and enter the bottom of the elution column. The solution flows upwards and will carry away the desorbed metals.

Solution will exit the top of the column and be cooled to below the flash point. The solution will then flow to the strong eluate tank, which is identical to the lean eluate tank. Strong eluate will be pumped to the electrowinning area. After electrowinning the gold from the rich eluate, the lean eluate will return to the lean eluate tank for recycling to the elution column.

Once the elution cycle is complete, the carbon will be cooled, drained and transferred by pump to either the carbon sizing screen or the reactivation kiln.

### 17.1.17 Electrowinning

Gold will be removed from the strong eluate by electro-plating it on stainless steel mesh. A direct current (DC) electrical current will be passed through the solution and the gold plus any silver or copper will be reduced at the cathode to form a metallic plating. The elevated voltage will generate hydrogen gas at the cathode which will dislodge some of the plated material, which will fall to the bottom of the cell as a metallic sludge.

Two electrowinning cells in parallel are included. Periodically the cathodes will be removed, and the adhering gold sludge washed off. The sludge in the bottom of the cell will be removed, and all the gold sludge will be filtered. The filter cake will be dried and sent to the refinery. No mercury will be present therefore there is no need for retorting.

### 17.1.18 Refinery

In the refinery, the gold sludge filtercake will be mixed with fluxes and smelted in a propane-fired nose-pour crucible furnace. The mixture will be heated until liquid and the impurities are dissolved in the slag. The slag will be poured off, and the liquid metal remaining will be poured into molds where it will be allowed to cool and solidify into a doré bar.

The doré will be cleaned, weighed and shipped offsite for refining into bullion.

Combustion products from the smelting furnace will be vented from the refinery through a baghouse. Any dust in the baghouse will occasionally be returned to the furnace.

Slag handling equipment will be included to recover any prills (droplets of metal that solidify in the slag) that may form. Residual slag will either be stored, or manually added to the crushing circuit for blending with the ore and placed on the heap.

### 17.1.19 Carbon handling

As carbon is processed through each unit operation a small portion will be abraded into fines. These fines will be removed each cycle and replaced with new carbon. In addition, the activity (the ability to adsorb gold to a high level) will diminish over time. The carbon will be thermally reactivated to regain the original activity.

Eluted carbon will be transferred to a hopper that feeds a reactivation kiln. The kiln will be sized to reactivate 100 % of the carbon batch each elution cycle. Operating experience may allow a reduced reactivation frequency.

The wet carbon will be heated to 650ºC in a steam atmosphere in a rotary kiln where the conditions will be such that the carbon activity is restored. The white-hot carbon will be discharged sub-surface into a tank filled with water to cool and quench the carbon. The quenched carbon will be transferred by pump to the carbon sizing screen.

The carbon sizing screen will be a wet, horizontal vibrating screen with a 850 μm aperture where the finer fractions will be discharged to a tank. The coarser fractions will report to a holding tank for return to the CIC circuit.

The holding tank will be an open topped vessel and will be used to measure a unit batch of carbon is transferred to the CIC. As the transfer of the carbon batch from the fifth carbon column is completed, the batch in the holding tank will be transferred with a pump to the fifth column.

Fines from the carbon-sizing screen will be pumped through a plate and frame filter to dewater the fines. When the filter is full, the cake will be blown down with compressed air and the cake will be manually discharged into a supersack and stored.

New coconut shell activated carbon will be delivered in supersacks. New, dry activated carbon has a dry bulk density of 0.48 t/m$^3$ due to the large quantity of pores created in the activation process. Dry carbon needs to soak in water to allow the pores to fill and displace the air in the pores; otherwise the carbon would float in the process. Typically, 24 to 48 hours is sufficient to displace the air. An agitated tank will be provided to soak the fresh carbon. In addition to de-gassing the carbon, the agitator action will abrade the carbon edges, and break up any fragile pieces before they are introduced into the process. Degassed and abraded carbon will be stored under water in this tank, and will be transferred to the carbon sizing screen as necessary to make up the unit batch of carbon.

### 17.1.20 Cyanide preparation

Concentrated cyanide solution will be prepared in a dedicated mix building separated from the rest of the process. Two carbon steel tanks will be provided in the mix building for preparing cyanide solution.

Sodium cyanide is available as briquettes in 1 t bags. One of the tanks will be filled with water and sufficient NaOH is added to elevate the pH. The cyanide bags will be emptied into the tank and the agitator started. Once the briquettes are dissolved the solution will be transferred by pump to the second tank. Cyanide will be mixed to 23 % NaCN by weight in solution.

A portion of the concentrated cyanide solution will be pumped to the storage tank at the agglomeration area.

Hydrogen cyanide monitors will be included in this area to notify the operators if HCN gas is present. The cyanide mix area will be inside a containment wall that will contain the total volume of the two tanks.

### 17.1.21 Reagents

Hydrochloric acid will be used in the carbon acid-wash circuit. HCl is available as a 34 % solution in 1,000-l totes. This solution will be metered directly into the process stream as necessary.

Sodium hydrosulfide (NaHS) will be used in the SART plant to precipitate copper as the sulfide. NaHS will be delivered to site in dry form in one-tonne supersacks and stored indoors until needed in the process. Concentrated NaHS solution will periodically be prepared in the SART plant and added to the process at a rate proportional to the concentration of copper in feed solution.

Sulfuric acid ($H_2SO_4$) will be used in the SART plant to adjust the pH of incoming pregnant leach solution to about 4.0. Sulfuric acid will be delivered to site by bulk tanker truck as concentrated solution (93 % to 96 % strength). A dedicated sulfuric acid storage tank will be located in the SART plant.

Hydrated lime ($Ca(OH)_2$) will be used in the SART plant to neutralize treated solution to a pH of about 10.5. Hydrated lime will be delivered to site by bulk pneumatic truck and unloaded into a dedicated storage silo at the SART plant. As needed, batches of lime slurry will be prepared in the plant. It is expected that a small quantity of hydrated lime slurry will be required to adjust the pH of the fresh water at the start of operations to add cyanide to the initial leach solutions.

Sodium hydroxide (NaOH) will be used in the SART and ADR plants. Sodium hydroxide will be delivered to site in solid flake form in 25-kg bags or 1-t supersacks. A mix tank and storage tank will be used to prepare solution at 25 % strength. The uses of NaOH in the SART plant are in the gas scrubber and for neutralization of copper-sulfide solids ahead of filtration. The uses of NaOH in the ADR plant are for carbon stripping (NaCN/NaOH solution), neutralization of acid wash solution, acid scrubber and pH adjustment of the cyanide mix solution.

Diatomaceous earth (DE) will be used as a pre-coat filtration aid in the SART plant for the copper-sulfide solids. The material will be delivered to site in 25-kg bags or 1-t supersacks. Diatomaceous earth slurry will be prepared as needed for filter cloth pre-coat. A small amount of DE filter aid will be required in the refinery to assist with the dewatering of the gold sludge. This will be prepared manually in the refinery.

Flocculant will be used in the SART plant thickeners to assist with solids settling. Flocculants will be delivered in dry form in 25-kg bags and fed into an automated system for make-up and dosing of flocculant solutions to the SART thickeners.

Antiscalant will be metered into the barren and intermediate solutions that flow to the heap to protect the emitters from calcium-based scale. Antiscalant will also be dosed into neutralized solution produced by the SART plant to reduce the potential for calcium-based scaling to occur in downstream equipment. Antiscalant will be delivered to site in concentrated liquid form in 1,000-liter totes.

### 17.1.22 Utilities

Propane will be used to fire the smelting furnace, elution boiler, carbon reactivation kiln and various drying ovens.

Compressed air will be used in the refinery for air tools, the baghouse and to blow down the gold sludge filter. In the ADR compressed air will be used to blow down the fine carbon filter. Compressed air will also be used at the agglomerating drum for cleaning the bin vents on the storage silos and flake breaker. The SART plant will use compressed air for blow down on the copper and gypsum filters.

Instrument air will be used for refinery pneumatic pumps, operation of the air-over-hydraulic closure on the filters, and in the metallurgical laboratory.

A separate compressed air building is included with the SART and ADR compressors, receivers and instrument air driers. Compressed air at the agglomerating area will be met with a dedicated air compressor-receiver.

Power consumption for the process is discussed in Section 18.

### 17.1.23 Water

Raw water for the Project is available from wells, as discussed in Section 18.3. This water will be used as makeup for the heap leach. Evaporation from leaching operations and adsorption by the ore represent the bulk of the water requirement for the process area.

The water from these sources has a high total dissolved solids (TDS) content and will be unsuitable for certain process requirements. A water softener and RO unit are included to obtain higher quality water for these uses.

Soft water will be required for use in safety showers and eyewashes and cathode washing in the refinery. The safety water system will be fitted with a tepid water heater and thermostatic bypass valves at each safety station that will allow recirculation of cooler water from the stations to return to the soft water tank and allowing tepid water to fill the lines.

Softened water will also feed the RO unit. RO water will be used to make up elution solution, makeup boiler solution as needed and in the metallurgical laboratory.

A small building will be included to house the RO and soft water equipment and storage tanks. The tanks will be constructed of FRP.

### 17.1.24 Water balance

Monthly precipitation data for the Project was obtained from registered data from the: Unidad Lindero (3,926 masl) and Campamento Fénix - Salar de Hombre Muerto (3,990 masl) stations for the period 1992 to 2016, covering a 25-year timeframe. Those data indicate that the average annual precipitation for the Project area is 71.7 mm. Maximum and minimum monthly precipitation values were extracted and monthly averages calculated. These figures are shown in Table 17.3 along with the annual values.

**Table 17.3 Precipitation data**

| Month | Maximum (mm) | Minimum (mm) | Average (mm) |
|---|---|---|---|
| Jan | 108.4 | 0.0 | 26.8 |
| Feb | 73.6 | 0.0 | 20.0 |
| Mar | 104.8 | 0.0 | 9.7 |
| Apr | 11.0 | 0.0 | 1.3 |
| May | 16.2 | 0.0 | 1.4 |
| Jun | 5.0 | 0.0 | 0.9 |
| Jul | 20.0 | 0.0 | 2.1 |
| Aug | 7.0 | 0.0 | 0.8 |
| Sep | 8.0 | 0.0 | 1.2 |
| Oct | 0.0 | 0.0 | 0.0 |
| Nov | 3.0 | 0.0 | 0.2 |
| Dec | 73.0 | 0.0 | 7.4 |
| **Annual** | **176.5** | **8.6** | **71.7** |

Table 17.3 confirms the dry condition of the Project area in which the maximum precipitations registered for a month was 108.4 mm and the maximum precipitation registered for a year was 176.5 mm.

A total of 25 historical series were generated using the 25-year historical record to project forward on a year-by-year basis providing iterations of pad operations assuming the conditions each year from 1992 to 2016, based on the historical records of precipitation and evaporation (monthly) for the Project area. Each historical series includes 156 months of operation of the leach pad and 24 months for the rinsing period. These series were used for the water balance.

To characterize the annual evaporation, the information available from the Campamento Fénix del Salar del Hombre Muerto station was used, due to similar geographic conditions and proximity to the Project area. The estimated average annual evaporation for the Project area is 2,750 mm.

The representative monthly evaporation series for the Project area was obtained based on the completed records of the Socaire station and scaled by an allocation factor of 1.4 in order to obtain the monthly evaporation historical series. Table 17.4 shows the estimated average, maximum and minimum monthly evaporation characteristic for the Project area.

**Table 17.4 Evaporation data for Project area**

| Month | Minimum (mm) | Maximum (mm) | Average (mm) |
|---|---|---|---|
| Jan | 274.0 | 315.7 | 291.2 |
| Feb | 187.3 | 264.7 | 231.1 |
| Mar | 224.3 | 266.7 | 248.5 |
| Apr | 183.7 | 221.1 | 205.3 |
| May | 158.3 | 183.8 | 173.4 |
| Jun | 28.3 | 166.7 | 131.5 |
| Jul | 150.1 | 180.0 | 164.3 |
| Aug | 191.3 | 208.9 | 197.0 |
| Sep | 234.8 | 242.5 | 238.0 |
| Oct | 258.1 | 289.9 | 275.7 |
| Nov | 220.7 | 302.1 | 283.3 |
| Dec | 297.9 | 327.9 | 310.8 |
| **Annual** | **2,520.8** | **2,857.0** | **2,750.0** |

Based on the preceding rainfall data, active water balances were calculated using the requirement for the full processing tonnage of 18,750 tonnes per day. Water balance spreadsheets were prepared for the 25-historical series for Phases 1 and 2 of the heap leach pad. The water balances estimate monthly maximum, average and minimum water requirements.

For all scenarios, it was determined that the Lindero Project will be in a water deficit and makeup water will be required. Makeup water requirements vary minimally between maximum, average, and minimum of the 25-historical series. Table 17.5 shows the results of the minimums, maximums and average fresh water requirement during the operation of the leach pad.

**Table 17.5 Fresh water requirements for the heap leach pad**

| Month | Minimum ($m^3$/hr) | Maximum ($m^3$/hr) | Average ($m^3$/hr) |
|---|---|---|---|
| Jan | 0.0 | 37.3 | 31.9 |
| Feb | 2.6 | 37.2 | 31.7 |
| Mar | 0.0 | 37.1 | 34.6 |
| Apr | 30.8 | 37.1 | 36.7 |
| May | 35.0 | 36.9 | 36.8 |
| Jun | 35.1 | 36.9 | 36.7 |
| Jul | 34.8 | 36.9 | 36.7 |
| Aug | 35.4 | 37.0 | 36.9 |
| Sep | 35.5 | 37.1 | 37.0 |
| Oct | 35.9 | 37.2 | 37.1 |
| Nov | 35.8 | 37.3 | 37.1 |
| Dec | 4.9 | 37.3 | 35.7 |
| **Annual** | **23.1** | **43.3** | **37.1** |

On an annual basis, the water balance should remain relatively constant during the operating life of the heap leach. The minimum, maximum and average values of annual fresh water demand are equal to 23.1 $m^3$/hr, 43.3 $m^3$/hr, and 37.1 $m^3$/hr respectively.

Table 17.6 shows an example of the water balance simulation for the first three years of one of the 25 series.

**Table 17.6 Typical results of the water balance simulation**

**A. Data**

| | | | |
|---|---|---|---|
| Combined Capacity of Ponds: | 80 000 $m^3$ | Incoming Ore moisture (per dry solids) | 6,38 % |
| Capacity of Process Ponds | 50 000 $m^3$ | Final Ore Moisture (per dry solids) | 11,37 % |
| Capacity of Storm Water Pond | 30 000 $m^3$ | Evaporation Factor at Ponds: | 1,00 - |
| Ore Bulk Density: | 1,60 Ton/$m^3$ | Losses by Evaporation | 0,3 % |
| Primary Leach Irrigation Rate | 12,0 lt/h/$m^2$ | Evaporation Factor in Non-activ area | 0,15 - |
| Secondary Leach Irrigation Rate | 6,0 lt/h/$m^2$ | Storm Event (T=100 años) | 56,1 mm |
| Time to restore power | 48 horas | | |
| Evaporation factor in Active Area: | 0,80 - | | |

**B. Water Balance**

| Year | Month | Phase | Monthly Production (tn) | Inputs: Precipitation in Active Area ($m^3$) | Inputs: Precipitation in Non-active Area ($m^3$) | Inputs: Precipitation at Ponds ($m^3$) | Inputs: Incoming ore moisture ($m^3$) | Outputs: Final Ore Moisture ($m^3$) | Outputs: Losses in Irrigation ($m^3$) | Outputs: Evaporation at Ponds ($m^3$) | Outputs: Evaporation in Active Area ($m^3$) | Outputs: Evaporation in Non-active Area ($m^3$) | Balance ($m^3$) | Flow: Water Requirement ($m^3$/h) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 2019 | Abr | Fase 1 | 562 500 | 1093 | 4 132 | 377 | 40 388 | 63 962 | 2 576 | 418 | 1 093 | 4 132 | -26 191 | 36,4 |
| | May | | 581 250 | 99 | 376 | 34 | 41 734 | 66 094 | 1 803 | 371 | 99 | 376 | -26 499 | 35,6 |
| | Jun | | 562 500 | 398 | 1 502 | 137 | 40 388 | 63 962 | 1 744 | 290 | 398 | 1 502 | -25 472 | 35,4 |
| | Jul | | 581 250 | 746 | 2 817 | 257 | 41 734 | 66 094 | 1 803 | 308 | 746 | 2 817 | -26 214 | 35,2 |
| | Ago | | 581 250 | 0 | 0 | 0 | 41 734 | 66 094 | 1 803 | 395 | 0 | - | -26 558 | 35,7 |
| | Sep | | 562 500 | 0 | 0 | 0 | 40 388 | 63 962 | 1 744 | 498 | 0 | - | -25 817 | 35,9 |
| | Oct | | 581 250 | 0 | 0 | 0 | 41 734 | 66 094 | 1 803 | 544 | 0 | - | -26 707 | 35,9 |
| | Nov | | 562 500 | 0 | 0 | 0 | 40 388 | 63 962 | 1 744 | 602 | 0 | - | -25 920 | 36,0 |
| | Dic | | 581 250 | 467 | 1 765 | 161 | 41 734 | 66 094 | 1 803 | 612 | 467 | 1 765 | -26 614 | 35,8 |
| 2020 | Ene | | 581 250 | 10775 | 40 715 | 3718 | 41 734 | 66 094 | 1 803 | 574 | 10 775 | 15 731 | 1 965 | - |
| | Feb | | 543 750 | 577 | 2 178 | 199 | 39 041 | 61 830 | 1 686 | 401 | 577 | 2 178 | -24 676 | 32,6 |
| | Mar | | 581 250 | 119 | 451 | 41 | 41 734 | 66 094 | 1 803 | 515 | 119 | 451 | -26 637 | 35,8 |
| | Abr | | 562 500 | 0 | 0 | 0 | 40 388 | 63 962 | 1 744 | 377 | 0 | - | -25 696 | 35,7 |
| | May | | 581 250 | 199 | 751 | 69 | 41 734 | 66 094 | 1 803 | 378 | 199 | 751 | -26 472 | 35,6 |
| | Jun | | 562 500 | 0 | 0 | 0 | 40 388 | 63 962 | 1 744 | 58 | 0 | - | -25 377 | 35,2 |
| | Jul | | 581 250 | 0 | 0 | 0 | 41 734 | 66 094 | 1 803 | 338 | 0 | - | -26 500 | 35,6 |
| | Ago | | 581 250 | 616 | 2 329 | 213 | 41 734 | 66 094 | 1 803 | 405 | 616 | 2 329 | -26 355 | 35,4 |
| | Sep | | 562 500 | 20 | 75 | 7 | 40 388 | 63 962 | 1 744 | 489 | 20 | 75 | -25 801 | 35,8 |
| | Oct | | 581 250 | 0 | 0 | 0 | 41 734 | 66 094 | 1 803 | 567 | 0 | - | -26 729 | 35,9 |
| | Nov | | 562 500 | 0 | 0 | 0 | 40 388 | 63 962 | 1 744 | 454 | 0 | - | -25 772 | 35,8 |
| | Dic | | 581 250 | 0 | 0 | 0 | 41 734 | 66 094 | 1 803 | 626 | 0 | - | -26 788 | 36,0 |
| 2021 | Ene | | 581 250 | 109 | 413 | 38 | 41 734 | 66 094 | 1 803 | 617 | 109 | 413 | -26 742 | 35,9 |
| | Feb | | 525 000 | 0 | 0 | 0 | 37 695 | 59 698 | 1 628 | 496 | 0 | - | -24 127 | 35,9 |
| | Mar | | 581 250 | 50 | 188 | 17 | 41 734 | 66 094 | 1 803 | 533 | 50 | 188 | -26 678 | 35,9 |
| | Abr | | 562 500 | 0 | 0 | 0 | 40 388 | 63 962 | 1 744 | 454 | 0 | - | -25 773 | 35,8 |
| | May | | 581 250 | 0 | 0 | 0 | 41 734 | 66 094 | 1 803 | 345 | 0 | - | -26 508 | 35,6 |
| | Jun | | 562 500 | 0 | 0 | 0 | 40 388 | 63 962 | 1 744 | 294 | 0 | - | -25 613 | 35,6 |
| | Jul | | 581 250 | 10 | 38 | 3 | 41 734 | 66 094 | 1 803 | 340 | 10 | 38 | -26 499 | 35,6 |
| | Ago | | 581 250 | 10 | 38 | 3 | 41 734 | 66 094 | 1 803 | 393 | 10 | 38 | -26 552 | 35,7 |
| | Sep | | 562 500 | 0 | 0 | 0 | 40 388 | 63 962 | 1 744 | 484 | 0 | - | -25 802 | 35,8 |
| | Oct | | 581 250 | 0 | 0 | 0 | 41 734 | 66 094 | 1 803 | 576 | 0 | - | -26 738 | 35,9 |
| | Nov | | 562 500 | 40 | 150 | 14 | 40 388 | 63 962 | 1 744 | 608 | 40 | 150 | -25 913 | 36,0 |
| | Dic | | 581 250 | 2137 | 8 075 | 737 | 41 734 | 66 094 | 1 803 | 641 | 2 137 | 8 075 | -26 066 | 35,0 |

Table 17.7 summarizes the water requirements for the whole mine site. Average requirements for the heap leach pad as shown in Table 17.6 have been taken into account. As the water balance in the heap leach pad uses agglomerated ore, the water added in the crushing and agglomeration circuits has been included. The figures also include the water losses in the precipitates from the SART plant that are filtered to a 50 % moisture content.

**Table 17.7 Average annual water requirements**

| Description | Average Requirement ($m^3$/hr) |
|---|---|
| Heap leach | 37.1 |
| Crushing, dust control and agglomeration | 40.2 |
| Road dust control | 12.8 |
| SART plant, loss in copper concentrate, and gypsum | 1.3 |
| Other losses at ADR and SART plants | 1.0 |
| Truck shop | 1.0 |
| Camp usage (including offices and warehouses) | 3.3 |
| Laboratories and other ancillary buildings | 1.0 |
| **Total** | **97.7** |

Figure 17.4 shows the water balance block diagram for the mine site, which includes the water requirements shown in Table 17.7.

**Figure 17.4 Water balance diagram**

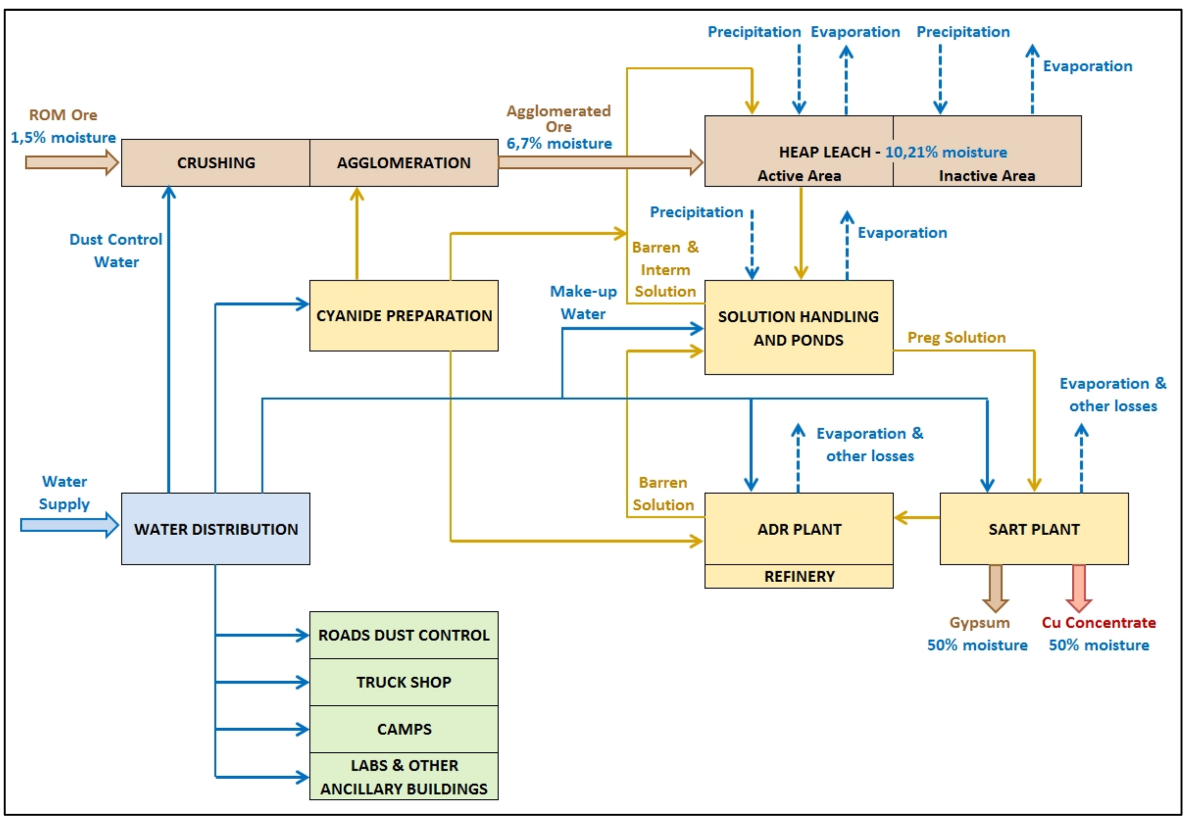



Figure prepared by Anddes, 2017

## 17.2 Requirements for energy, water, and process materials

Annual usage of various ADR consumables was calculated from the mass balance, equipment vendor specifications and the mine plan. The annual ADR consumables are presented in Table 17.8.

**Table 17.8 Annual process plant consumables**

| Consumable | Year | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 |
| Cement Consumption (1,000 kg) | 17,978 | 15,856 | 10,100 | 13,167 | 15,370 | 9,952 | 6,834, | 9,671 | 4,787 | 4,584 | 4,070 | 2,654 | 3,132 | 328 |
| Average Cement Consumption (kg/t) | 2.66 | 2.35 | 1.5 | 1.95 | 2.28 | 1.47 | 1.01 | 1.43 | 0.71 | 0.68 | 0.6 | 0.4 | 0.46 | 0.46 |
| NaCN Consumption (1,000 kg) | 4,460 | 4,012 | 3,846 | 3,983 | 3,741 | 3,714 | 3,650 | 3,528 | 3,588 | 3,446 | 3,460 | 3,320 | 3,397 | 360 |
| Average NaCN Consumption (kg/t) | 0.66 | 0.59 | 0.57 | 0.59 | 0.55 | 0.55 | 0.54 | 0.52 | 0.53 | 0.51 | 0.51 | 0.51 | 0.5 | 0.51 |
| Carbon Loss (1,000 kg) | 90 | 89 | 73 | 154 | 133 | 127 | 135 | 131 | 128 | 130 | 131 | 138 | 129 | 55 |
| Elution Cycles | 134.5 | 132.8 | 109.6 | 229.8 | 198.8 | 189.2 | 201.7 | 195.1 | 191.3 | 193.5 | 195 | 205.4 | 192.3 | 81.7 |
| HCl Consumed (1,000 liters at 35%) | 9.5 | 9.4 | 7.8 | 16.3 | 14.1 | 13.4 | 14.3 | 13.8. | 13.6 | 13.70 | 13.80 | 14.60 | 13.70 | 5.80 |
| NaCN Consumed in Elution (1,000 kg) | 34 | 34 | 28 | 58 | 50 | 48 | 51 | 49 | 48 | 49 | 49 | 52 | 49 | 21 |
| NaOH Consumed (kg) | 299 | 293 | 245 | 488 | 424 | 405 | 429 | 415 | 408 | 412 | 415 | 435 | 409 | 167 |
| Antiscalant Consumed (1,000 liters) | 47 | 47 | 70 | 70 | 70 | 70 | 70 | 70 | 70 | 70 | 70 | 70 | 70 | 45 |
| Silica (1,000 kg) | 0.8 | 0.8 | 6.1 | 7.0 | 5.1 | 4.7 | 5.2 | 4.9 | 4.6 | 4.7 | 4.8 | 5.0 | 4.5 | 1.7 |
| Sodium Carbonate (1,000 kg) | 5.5 | 5.6 | 5.8 | 7.4 | 5.7 | 5.3 | 5.8 | 5.5 | 5.2 | 5.3 | 5.4 | 5.7 | 5.1 | 2.0 |
| Borax 10 mol (1,000 kg) | 2 | 2 | 5.8 | 7.4 | 5.7 | 5.3 | 5.8 | 5.5 | 5.2 | 5.3 | 5.4 | 6 | 5.1 | 2.0 |
| $KNO_3$ (kg) | 94 | 95 | 362 | 464 | 357 | 330 | 360 | 342 | 327 | 332 | 336 | 353 | 320 | 127 |
| Filter aid DE (kg) | 134 | 133 | 110 | 230 | 199 | 189 | 202 | 195 | 191 | 194 | 195 | 205 | 192 | 82 |
| Propane Consumption (1000 M btu) | 29 | 29 | 28 | 33 | 31 | 31 | 32 | 31 | 31 | 31 | 31 | 32 | 31 | 18 |
| Hydrated Lime (kg) | 1,500 | | | | | | | | | | | | | |
| Crucibles (each unit) | 6 | 8 | 10 | 22 | 19 | 18 | 20 | 19 | 19 | 19 | 19 | 20 | 19 | 7 |
| Refinery Supplies (US$1,000) | 97 | 98 | 75 | 96 | 73 | 68 | 74 | 70 | 67 | 68 | 69 | 73 | 66 | 26 |
| NaCl (kg) | 876 | 876 | 876 | 878 | 876 | 876 | 876 | 878 | 876 | 876 | 876 | 878 | 876 | 559 |
| Laboratory Supplies (US$) | 54,750 | 54,750 | 54,750 | 54,750 | 54,750 | 54,750 | 54,750 | 54,750 | 54,750 | 54,750 | 54,750 | 54,750 | 54,750 | 34,950 |

The reagent consumptions for the SART plant were calculated on using average year and are presented in Table 17.9.

**Table 17.9 Unit SART plant consumables**

| Reactant | Consumption |
|---|---|
| Sodium hydrosulfide (tpd) | 3.22 |
| Sulfuric acid (tpd) | 16.04 |
| Hydrated lime (tpd) | 14.51 |
| Sodium hydroxide (tpd) | 0.36 |
| Flocculant (kg per day) | 105 |
| Antiscalant (kg per day) | 75 |
| Diatomaceous earth (kg per day) | 500 |

Propane will be required for the reactivation kiln, elution solution boiler, smelting furnace and various ovens and driers. The propane requirement is included in Table 17.8.

Power consumption is discussed in Section 18.

## 17.3 Other Considerations

Copper crowding on the activated carbon will limit the adsorption gold onto the carbon. The requirement to go to 600 $m^3$/hr in Year 4 is a late refinement to the operating philosophy. It is recognized that bypassing 200 $m^3$/hr around the SART plant will increase the concentration of copper in the feed to the CIC circuit. Carbon-loading testwork is ongoing to define the magnitude of these changes. In any case, a change to the CIC and downstream unit operations will likely be a matter of scale, and is expected to avoid any unconventional processes. An allotment for additional capital cost has been included for these considerations, which is reasonable for the anticipated changes. Process plant reagent consumptions also have been estimated for the maximum anticipated carbon transfer rate. The current carbon loading testwork is being conducted to optimize and reduce these costs.

Variations in the availability of some reagents (bulk dry or solution) will depend on negotiated contracts, which may vary the reagent handling facilities between solution storage or mixing on-site. It is anticipated that either of these scenarios will be selected on the least cost option impact to capital and operating costs. and will not have a negative impact on the Project.

## 17.4 Comment on Section 17

The QP considers process requirements to be well understood, and consistent with similar projects elsewhere. Unit operations for the Lindero process were selected based on the physical and metallurgical needs of the Lindero ore to achieve maximum extraction of gold. No novel or untried technology is employed in the process.

# 18 Project Infrastructure

The infrastructure and services were developed to support the planned Lindero Project operations. A layout plan displaying the location of the major infrastructure components is displayed in Figure 17.1. The Project infrastructure includes the following major areas:

- Access roads, including upgrades to existing roads
- Power connection and supply from diesel generators to site facilities
- Process water supply and distribution, including a fire water system
- Potable water system
- Sewage system
- Project buildings, including:
  - Administration building
  - ADR process area/building
  - SART process area/building
  - Agglomeration process area/building
  - Refinery building
  - Reagent storage
  - Process shop
  - Chemical and metallurgic laboratory buildings
  - Mine truck shop
  - Light vehicles shop
  - Crushing plant shop
  - Guard house
  - Dispatch office
  - Infirmary
  - General warehouse
- Liquefied petroleum gas and diesel fuel storage systems
- Explosives storage
- Personnel camps for construction and operations, including a dining and recreation area, and a laundry building
- Miscellaneous site services such as:
  - Security
  - First aid
  - Communications

- Employee transport
- Solid waste disposal

## 18.1 Access roads and site access

A discussion of Project access routes is provided in Section 5.

Significant road improvements are planned for stretches of road between Tolar Grande and the Fortuna camp. These upgrades are necessary to accommodate construction and mine activities for the Project. A report was prepared by a local engineering firm (Solid CC, 2015), detailing the designs and activities required for the planned upgrades and associated costs were obtained based on these designs by a local contractor.

The general Project location is shown in Figure 4.1.

## 18.2 Power supply

### 18.2.1 Power requirements

The initial peak demand load for the processing and crushing plant, heap leach and ancillary facilities at the Lindero Project site is estimated at 7.5 MW, with an average draw of approximately 6.4 MW. This demand will increase over the life of the Project as the heap height increases and with the pump and conveyor stacking system demands. The maximum estimated peak and average demands are estimated at 7.7 MW and 6.8 MW, respectively. Electrical power will be generated on site at a power generation plant located adjacent to the ADR plant area platform.

An extensive review of power sourcing options was developed during 2017, which indicated that diesel-fueled generation was the most technically viable and efficient option for the Lindero Project. Diesel will be trucked in from Salta.

At the Project, electrical power will be generated on site under a "take-and-pay" contract power supply arrangement with a local company who specializes in such services. The contractor will supply, install, operate, and maintain the power generation plant for the life of the Project (see Section 21.7 for additional comments on the proposed arrangement). The preliminary contract specifies 12 generators at 1,037 kVA each (nameplate before derating) to meet the maximum peak power demands, along with the necessary infrastructure, which will deliver power to a single point outside the plant. From the contractors supply point, power will be distributed on-site at 6.6 kV, 3 phase, 50 Hz and will be stepped down where necessary to 380 V, 3 phase, and 220 V, accordingly. Large motors such as the HPGR motors will operate at 6.6 kV and smaller operating motors will use 380 V.

A detailed study was prepared which compared the power generation plant option described above with an owner-purchased/operated plant, considering capital and operating costs (Saxum, 2017). See Section 21.7 for additional details.

The estimated project electrical power consumption is presented in Table 18.1.

Table 18.1 Power consumption forecast by area

| Area | Total Inst. Power (1000 kW) | Average (kW) (1) | Million kWh/year | kWh/t ore |
|---|---|---|---|---|
| Area 010 – General | 0 | 0 | 0 | 0 |
| Area 020 – Mine | 0 | 0 | 0 | 0 |
| Area 100 – Crushing | 5.28 | 4,040 | 28.34 | 4.198 |
| Area 200 – Agglomeration | 1.08 | 810 | 6.08 | 0.889 |
| Area 300 – Heap Leach/Solution Handling | 2.30 | 1,510 | 12.08 | 1.766 |
| Area 400 – ADR Plant | 0.34 | 270 | 1.11 | 0.163 |
| Area 500 – Reagents | 0.04 | 25 | 0.12 | 0.017 |
| Area 600 – SART Plant | 0.70 | 470 | 3.46 | 0.513 |
| Area 650 – Laboratories | 0.28 | 230 | 0.85 | 0.126 |
| Area 700 – Energy | 0.21 | 190 | 0.83 | 0.123 |
| Area 800 – Fuel Storage and Distribution | 0.03 | 27 | 0.12 | 0.017 |
| Area 090 – Building | 1.04 | 800 | 4.93 | 0.73 |
| Area 010 – Water Storage and Distribution | 0.25 | 100 | 0.12 | 0.06 |
| Area 010 – Waste Management | 0 | 0 | 0 | 0 |
| **Total (2)** | **11.54** | **8,500** | **58.03** | **8.6** |

Notes: 1) Total installed power does not consider stand- by equipment.
2) Power value considers that all equipment is running at the same time.

### 18.2.2 Construction and emergency power

Semi-portable diesel generators (550 kVA nameplate) will be purchased early in the construction phase of the Project to provide power to the Lindero camp. Generators will be provided to power the camp and the laboratory during the late stages of construction, until the main power generation plant is complete and operating.

## 18.3 Water supply

The Lindero Project will require water for the following uses:

- Mining operations for dust control
- Crushing for dust control
- Agglomeration process
- Makeup water for the heap leach pad
- Process plant and laboratory
- Main camp and administration
- Construction activities
- Fire water

The expected maximum monthly (design) water demand for the Project is estimated at 27.2 l/s (97.7 $m^3$/hr) for the full processing tonnage for all site facilities including the heap leach, crusher dust control, road dust suppression, and infrastructure, as determined by the Project water balance (see Section 17.8).

A summary of the predicted maximum monthly Project water demands is shown in Table 18.2.

**Table 18.2 Forecast site water demand (design)**

| Description | Demand ($m^3$/hr) | Demand (l/s) |
|---|---|---|
| Heap leach | 37.1 | 10.3 |
| Crushing, dust control & agglomeration | 40.2 | 11.2 |
| Road dust control | 12.8 | 3.5 |
| SART plant, loss in copper concentrates and gypsum | 1.3 | 0.4 |
| Other demands at ADR and SART plants | 1.0 | 0.3 |
| Truck shop | 1.0 | 0.3 |
| Camp usage (including offices and warehouse) | 3.3 | 0.9 |
| Laboratories and other ancillary buildings | 1.0 | 0.3 |
| **Total Water Required** | **97.7** | **27.2** |

### 18.3.1 Water source – wells

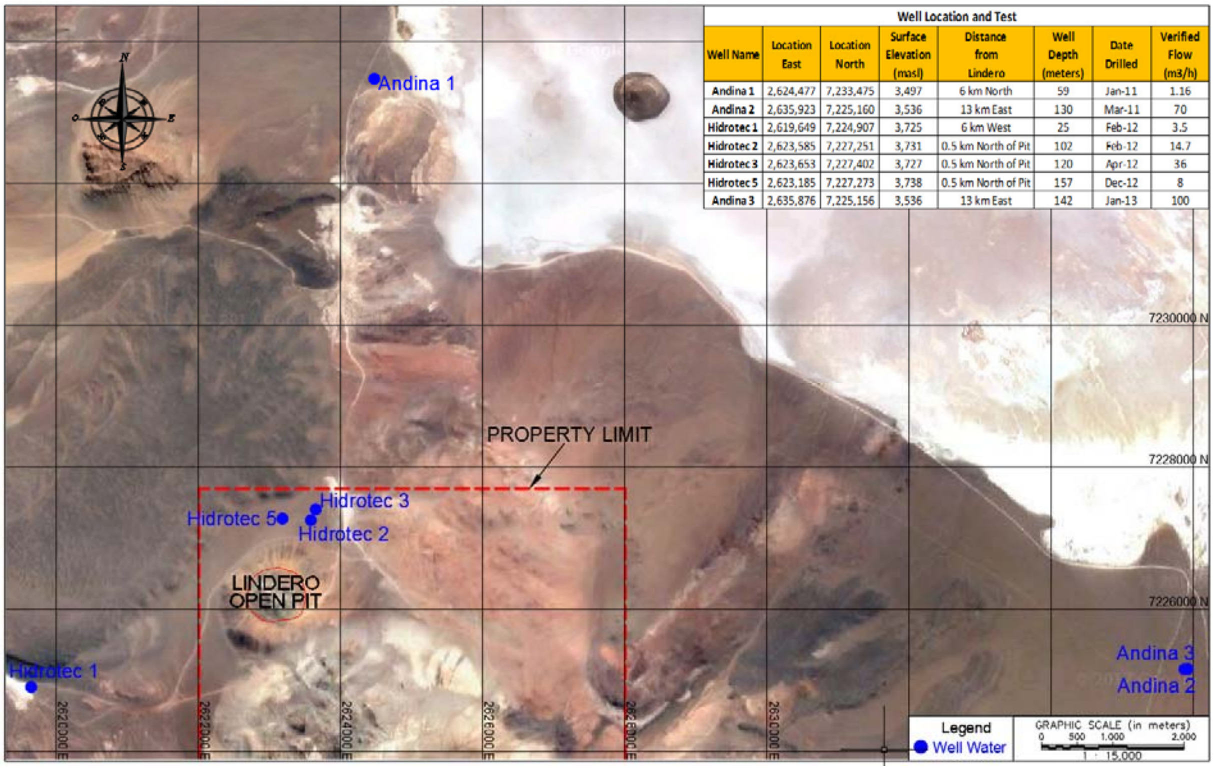
Significant work was completed in locating adequate water sources for the Project. Hydrology and hydrogeology studies were conducted to identify the Rio Grande basin and its several sub-basins, followed by geo-electric profiling to determine drilling targets (Vector, 2009a; Vector, 2009b). These initial exploratory activities are discussed in Section 24.1.

Several exploratory wells were drilled and pumping tests conducted in the Project area, notably in the Lindero, Arita, and Chachas sub-basins (Andina, 201la-b, Conhidro, 2013, Hidrotec 2012a-e). The results of the well pumps are shown in Table 18.3 and the well locations in Figure 18.1.

**Table 18.3 Summary of well locations and tests**

| Well Name | Easting | Northing | Elevation (masl) | Distance from Lindero | Well Depth (m) | Date Drilled | Verified Flow ($m^3$/hr) |
|---|---|---|---|---|---|---|---|
| Andina 1 | 2,624,477 | 7,233,475 | 3,497 | 6 km North | 59 | Jan-11 | 1.16 |
| Andina 2 | 2,635,923 | 7,225,160 | 3,536 | 13 km East | 130 | Mar-11 | 70 |
| Hidrotec 1 | 2,619,649 | 7,224,907 | 3,725 | 6 km West | 25 | Feb-12 | 3.5 |
| Hidrotec 2 | 2,623,585 | 7,227,251 | 3,731 | 0.5 km North of Pit | 102 | Feb-12 | 14.7 |
| Hidrotec 3 | 2,623,653 | 7,227,402 | 3,727 | 0.5 km North of Pit | 120 | Apr-12 | 36 |
| Hidrotec 5 | 2,623,185 | 7,227,273 | 3,738 | 0.5 km North of Pit | 157 | Dec-12 | 8 |
| Andina 3 | 2,635,876 | 7,225,156 | 3,536 | 13 km East | 142 | Jan-13 | 100 |

**Figure 18.1 Well test locations**




Well Location and Test

| Well Name | Location East | Location North | Surface Elevation (masl) | Distance from Lindero | Well Depth (meters) | Date Drilled | Verified Flow (m3/h) |
|---|---|---|---|---|---|---|---|
| Andina 1 | 2,624,477 | 7,233,475 | 3,497 | 6 km North | 59 | Jan-11 | 1.16 |
| Andina 2 | 2,635,923 | 7,225,160 | 3,536 | 13 km East | 130 | Mar-11 | 70 |
| Hidrotec 1 | 2,619,649 | 7,224,907 | 3,725 | 6 km West | 25 | Feb-12 | 3.5 |
| Hidrotec 2 | 2,623,585 | 7,227,251 | 3,731 | 0.5 km North of Pit | 102 | Feb-12 | 14.7 |
| Hidrotec 3 | 2,623,653 | 7,227,402 | 3,727 | 0.5 km North of Pit | 120 | Apr-12 | 36 |
| Hidrotec 5 | 2,623,185 | 7,227,273 | 3,738 | 0.5 km North of Pit | 157 | Dec-12 | 8 |
| Andina 3 | 2,635,876 | 7,225,156 | 3,536 | 13 km East | 142 | Jan-13 | 100 |

Figure from KCA, 2016b

During construction, Project water will be supplied by three water wells located on-site near the proposed main camp (Hidrotec 2, Hidrotec 3, and Hidrotec 5). These wells combined can provide approximately 50 to 55 $m^3/hr$ on a short-term basis, sufficient for water needs during construction, and sufficient for supplying water to the potable water treatment plant and for other small miscellaneous needs during operations. However, these wells cannot provide enough water on a continuous basis for the full operational make-up requirement.

During operations, the majority of the production make-up water requirement will be met using three wells located about 13 km to the east of the site in the Chachas area, carrying water to site via a buried pipeline (Andina 2, Andina 3 and a third well). The off-site well pumps combined are designed to provide up to 140 $m^3/hr$ continuously. The water pipeline system is designed for a flow rate of 140 $m^3/hr$ (39 l/s), which considers flow from both the Chachas area wells operating simultaneously, and the third well still to be defined.

The water pipeline will consist of the well field area and two booster stations located approximately 2.3 km and 4.9 km west of the well field. Power to all three areas will be provided by local diesel generators (one operating and one standby at each location). Each of the booster stations will be equipped with a surge tank of approximately 35 minutes capacity at the nominal flow, and booster pumps.

### 18.3.2 Raw water and fire water distribution

There will be two main reservoir tanks for combined raw water/fire water storage for the Lindero Project, fed by the water pipeline.

This primary tank will have a capacity of approximately 900 $m^3$ and will be located near the surge bin for the agglomeration plant. The tank will supply all road and crusher dust control, crusher cleaning water, fire water for crusher and agglomeration areas; and will gravity feed the secondary tank. The fire water reserve capacity will be 120 $m^3$, based on the fire water site analysis conducted by Saxum.

This secondary tank will have a capacity of approximately 440 $m^3$ and will be located near the process plant area. This tank will supply all processing water for the heap leach build up, and fire water for the entire process plant area, administration building area, truck shop area, camp area and explosives magazine. The fire water reserve capacity will be 240 $m^3$, based on the fire water site analysis conducted by Saxum.

### 18.3.3 Potable water system

A potable water system will be installed to supply potable water to the camp area. The potable water system will consist of a pre-filtration unit coupled to a reverse osmosis unit and associated piping, tankage, and controls. Chemical controls will be included for preservation in the potable water storage tank. The plant is sized based on the estimated construction population and the storage reserve is sized for 24 hours of nominal flow based on the assumed construction population.

### 18.3.4 Sewage treatment system

An anaerobic sewage treatment system will be installed near the man camp to treat waste water generated on-site. Effluents from the treatment plant can be reused as make-up water for the heap leach barren solution tank or reintegrated into the environment via leach fields.

## 18.4 Process area buildings

Most of the process buildings for the Lindero Project were designed as steel frame buildings with modular thermo-acoustic panels. In general, these are pre- engineered and pre-fabricated steel buildings which include all structural members, exterior doors and windows, roofs, insulation, interior and exterior wall panels and all connectors required to erect and assemble the building on-site.

For the refinery, a reinforced concrete block/masonry type building design will be used for security purposes.

### 18.4.1 Administration building

For the administration building, a pre-engineered steel frame and plasterboard type building was designed, which will be mounted on a concrete slab to minimize on-site installation time. This design was selected by Fortuna, and specifications and layouts were developed by Saxum. The administration building has a footprint of about 520 $m^2$ in total, which includes offices, meeting room, bathrooms, IT and server room, and a storage room.

General and administrative staff along with certain key management in both operations and mining will occupy the administration building.

### 18.4.2 Guard house

A guard house will be erected at the mine entrance. The guard house will include an office, a training room, a security inspection area and rest rooms for the security personnel. The entrance will be continuously monitored.

### 18.4.3 Process shop

The process shop will be a 300 m$^2$ pre-engineered steel building and will be located close to the process area. The process shop will have a main work area for repairs and maintenance for equipment for process plant, and will include an office area, toilet facilities, and tool rooms.

### 18.4.4 Warehouse

The warehouse will be a 672 m$^2$ pre-engineered steel building located near the administration building. The warehouse will contain critical spares for the process areas and includes an office area, bathroom and a reception area. Adjacent to the building, another 672 m$^2$ fenced area with a concrete floor will be available to serve as a repository for major spares and equipment.

### 18.4.5 ADR plant building

The ADR plant will be housed in a pre-engineered steel building. The building will accommodate the pressure stripping circuit, acid wash, carbon regeneration, and caustic and cyanide mixing areas. The ADR building will have a footprint of approximately 660 m$^2$. Due to the relatively high profile of the reactivation kiln, the building will be large, approximately 18 m at the roof apex.

The carbon columns will be located outside of the ADR building.

The ADR building will be located in the same general area as the chemical and metallurgic laboratory where chemical analysis of samples from the ADR plant will be performed.

### 18.4.6 SART filter building

The majority of the equipment associated with the SART plant will be located outdoors, with the following exceptions:

1. The two flocculant systems will be located inside a heated enclosure to prevent exposure to wind, rain and for freeze protection. Space will be provided in the enclosure for storage of two stacked pallets of dry flocculant (25 kg bags).
2. The copper and gypsum filters will be located inside a heated filter building, along with the associated hopper, screw feeder, copper bagging system and the gypsum filter cake dumping area. At the floor level of the filter building, space will be available for the storage of about 10 to 20 one-tonne supersacks of copper filter cake. Bagged copper filter cake should be stored to prevent exposure to water and sunlight, and water-tight bags should be used in the system.

### 18.4.7 Refinery building

The refinery will be connected to, and share a wall with, the ADR building. The refinery will be placed inside a fenced area and will be constructed of reinforced concrete block/masonry with a steel roof. A separate secure staging area will be included at the entrance of the building. The interior and exterior of the building will be under surveillance via cameras and closed-circuit television.

### 18.4.8 Cyanide mix building

Cyanide solution is prepared in a separate, limited-access building. This building is located in the fenced enclosure for the cyanide storage and contains an agitated mix tank and a day tank for the cyanide solution. This building is 70 $m^2$ and consists of an upper "mixing level" and a lower "pumping level". The lower portion of the building is supported by a concrete containment wall of sufficient height to contain the total volume of the two tanks. The upper portion is a pre-engineered building. The building is fitted with HCN gas monitors and ventilation.

### 18.4.9 Laboratory

Full-service laboratory facilities will include a chemical laboratory and a metallurgic laboratory. The design of the laboratories includes:

- Pre-engineered steel frame and plasterboard building
- support slabs for equipment
- concrete sidewalks/halls
- a fire assay lab section consisting of:
  - concrete columns
  - masonry type walls
- office sector constructed with modular wall panels
- bathrooms with ceramic tile floor
- wet laboratory areas with ceramic floors
- a sample preparation area
- a fire assay area
- data input, AA, and weighing areas
- a metallurgical laboratory area

The laboratory is sized to process 160 fire-assay samples per day and 260 solution samples per day.

### 18.4.10 Reagents storage area

The reagents storage area will be located outdoors in a fenced in area, with concrete secondary containment. For the caustic storage, a container will be available inside the fenced area. Fluxes for the smelting of doré will be stored inside the refinery. Each containment area will drain to a floor sump equipped with a small sump pump in case of spills or for rain events.

#### 18.4.11 Mine area buildings

Mining operations will be supported using a series of buildings located in an area adjacent to the administration building.

#### 18.4.12 Truck shop

A shop will be erected on a site west of the administration building and is connected to the haul road from the pit and crushing areas. The shop will be approximately 700 $m^2$ in area, and will have two work bays to accommodate the earthmoving equipment anticipated for the Project.

The shop will have concrete floors and metal siding and equipment to facilitate all anticipated repairs on the earthmoving fleet. Areas will be allocated for mechanical, electrical, welding, and maintenance activities. Near the shop building will be an outdoor truck wash area. The wash down drainage from equipment washing will be directed to an oil-water separator. A tire repair area will also be available in an area adjacent to the truck shop building.

The shop will be equipped with a large air compressor/receiver to service the wash down area and other tooling.

Offices for contractors, as well as a break area and first aid area will be provided in the building.

Equipment spares for the mining area will be stored in the warehouse with over size storage in an adjacent fenced storage area.

A complete single bay light vehicles maintenance building will be erected next to the truck shop.

## 18.5 Explosive storage

The bulk explosives facility will be divided into three areas at least 650 m from each other. These three areas will be comprised as follow:

- Low density ammonium nitrate up to 500 t. This area will have a fire fighting system
- Reagent storage with 330 t in each silo. This area will have an office, a container for spare parts and a maintenance area
- Primary explosives and detonators storage. These will be stored in three separate shipping containers

All of the containers will be situated within a secure area, surrounded by fencing, and managed accordingly.

Raw materials, such as ammonium nitrate, fuel oil, and primary explosives used in the explosive manufacturing process, will be brought to site by road and stored at the explosives facility site until required.

## 18.6 Camp

### 18.6.1 Permanent/construction camp

A main camp will be constructed for the Lindero Project and will be used for both construction and permanent operations housing. The camp will be erected to the north of the planned pit and along the northern project boundary. The facility will include modular dormitory-type facilities for workers and for supervisors. The worker/operator quarters will house 320 people and the supervisor quarters will house 32 people for a total of 352 beds. Under normal operations a total of about 460 people are expected to work at the Project site, including processing, mining, and G&A personnel. The two-shift schedule planned during operations will mean that approximately 230 personnel will be on site at any given time.

**Dormitories**

The operator accommodations will consist of five permanent dormitories for a total capacity of 320 persons. Each dormitory will contain 32 rooms, two persons per room, and will have one communal men's and women's bathroom and shower area for each dormitory.

The supervisor's accommodations will consist of one 34-man dormitory. It will contain 34 private rooms with a small private bathroom containing a toilet and shower.

The dormitory buildings are designed as fast-assembly modular-type buildings, which will incorporate the use of pre-cast concrete sleepers, pre-constructed modules, pre-wired lighting, heating, and electrical to minimize onsite installation time.

**Dining facilities**

A kitchen, recreation, and dining facility will be constructed and located adjacent to the staff housing. This facility will be a permanent facility that will continue to be used through operations. The dining area will be a large single room with adjacent bathrooms. The kitchen will be complete with all facilities necessary to prepare meals for the anticipated operations and construction workforce (the kitchen equipment will be provided under the catering contract). The recreation area will include a TV room, internet café, and additional room for table games.

The dining and recreation area will also be constructed of pre-engineered steel frame and plasterboard to minimize site erection time.

**Laundry room**

A 60 $m^2$ laundry room will be included in the camp and will be located next to the dining and recreation building. This building will also contain a separate electrical room for distribution of power to the camp.

### 18.6.2 Temporary expansion

**Construction phase**

Construction activities will require an anticipated 600 contractor personnel working on day shifts. Contractors will be on site for up to 16 months constructing buildings, erecting the crushing plant, water storage and distribution systems, process facility, and preparing and lining the leach pad and ponds. Mining contractors will establish haul roads, erect blasting storage facilities and begin pioneering the mine pit for production.

Fortuna currently owns two camp facilities, the Lindero camp and Regulus camp, both located about 4 km north of the Project site. The Regulus camp is planned to be upgraded to fit about 140 people prior to the main site construction and to utilize this camp for a portion of the construction crew.

The extra accommodation needed during construction shall be provided by the civil contractor (about 140 additional beds).

## 18.7 Diesel fuel delivery and storage systems

Diesel fuel will be delivered to the mine site via contractor owned tanker trucks and stored in 50 $m^3$ tanks on site. Three tanks will store mine vehicle diesel fuel and five tanks will store fuel for light vehicles and for power generation. Each storage tank will be contained in a lined basin to assure no fuel is leaked to the environment. Fuel trucks will be used to deliver fuel to the mine mobile equipment.

A third-party supplier (fuel vendor) will be used to directly manage the fuel in order to minimize fraudulent activity. Numerous fuel vendors in Argentina provide this service.

## 18.8 Site services

### 18.8.1 First aid

Emergency medical staff will be available at an on-site clinic located near the administration building. Medical personnel will be contracted by Fortuna. The medical staff includes one on-site nurse, one assistant/ambulance driver, and one on-call doctor. The contract also includes a 4 x 4 wheel ambulance.

### 18.8.2 Communications

An internet protocol telephone system will be used for off-site communications. The system includes a device for connecting to the public network, voice mail, and direct selection of extension number. The telephone switchboard will share installations with the data network, which will have power and air conditioning back-up support systems. Phones will be installed in all buildings and facilities. Cellular telephone and internet coverage will be available in the camp and mining areas.

Hand-held and base station radios will be provided for survey and ore control personnel, geologists, and operators for on-site communications. Mobile equipment will be equipped with radios.

### 18.8.3 Transportation

Lindero personnel will be transported to the Project from nearby communities and the city of Salta, and will reside in the camp during work periods. It is anticipated that shift change over (including a pre-start safety meeting) for all workers will occur either at the main office or at the truck shop, and as these locations are just a short distance from the camp, employees will walk to these areas.

### 18.8.4 Solid waste disposal

Solid wastes will be disposed of in a manner complying with local regulations. Allowable products will be disposed of in a solid waste landfill constructed on site. Products not allowed to be disposed of in the landfill will be transported to appropriate facilities off site.

### 18.8.5 Site fencing

The Lindero site is very remote and, as such, it is not considered to be necessary to fence the entire Project site. Specific parts of the Project facilities will be fenced, including chain-link security fences around the refinery, the electrical substation, the warehouse yard, and the magazines. Fencing will also be installed off-site around the remote well field and booster stations to secure the well pumps and generators in that area.

## 18.9 Comments on Section 18

The QP is confident that all mine and process infrastructure and supporting facilities have been included in the general layout to ensure that they meet the needs of the proposed mine plan and production rate

# 19 Market Studies and Contracts

## 19.1 Market Studies

No market studies are currently relevant as the Lindero Project will produce a readily-saleable commodity in the form of doré.

## 19.2 Gold pricing

Gold and silver are typically sold through commercial banks and metal dealers. Sales prices are obtained based on world spot or London fixes and are easily transacted. The gold price used for the base case cash flow analysis is US$1,250/oz. Sensitivities with variable price projections are also considered. The sensitivities are provided in Section 22.

The Lindero Project, like most gold projects, is highly sensitive to changes in the gold price. To determine the adequacy of the gold prices used for cash flow analysis, a review was completed of prices from previous years.

A summary of gold prices since 2012 to the present is presented in Table 19.1.

**Table 19.1 Recent metal prices**

| Year | Low | High | Average | 3-Year Trailing Average |
|---|---|---|---|---|
| 2012 | $1,540 | $1,792 | $1,667 | $1,487 |
| 2013 | $1,192 | $1,694 | $1,411 | $1,549 |
| 2014 | $1,142 | $1,385 | $1,266 | $1,449 |
| 2015 | $1,049 | $1,296 | $1,160 | $1,279 |
| 2016 | $1,077 | $1,368 | $1,248 | $1,225 |

## 19.3 Gold price forecast

In May 2017, Fortuna collected metal price forecasts from selected well-known financial institutions. The median long-term price quoted for gold was US$ 1,300/oz. Fortuna elected to use a more conservative price of $1,250/oz for the base case scenario for the estimation of Mineral Resources and Mineral Resources and economic analysis.

## 19.4 Doré production rate forecast

The Lindero mine product will be doré bars containing an estimated gold content averaging 84 % for the Project life. Doré bars will be stored in a secured vault at the plant and transported by a security company armored car from Lindero to a security transport company in Salta and then shipped overseas for refining. The total weekly shipment of doré for the first 13 years of production will average about 2,740 troy oz or 85 kg. The estimated weekly doré production rate declines in Years 14 and 15 to about 50 kg and less frequent or smaller shipments will be made.

As part of the Feasibility Study, a copper by-product resulting from the operation of the SART plant is considered in the form of a copper precipitate, representing between 2 % and 3 % of revenue.

## 19.5 Refining terms

When a shipment of doré is sent to the refiner, the refiner melts it and samples the melt. The refiner can also drill the doré for samples as per the agreed contract. Settlement is decided as per agreements between the refiner and company. If the samples do not agree within set splitting limits, settlement is then based on umpire assays from a mutually agreed upon umpire assayer.

## 19.6 Gold production schedule

Table 19.2 summarizes the estimated production of ounces of gold stacked and recovered by year.

**Table 19.2 Forecast summary of gold produced**

| Year | Gold extracted to doré (oz) |
|---|---:|
| Pre-production | 12,800 |
| 1 | 136,700 |
| 2 | 138,200 |
| 3 | 103,900 |
| 4 | 115,200 |
| 5 | 87,700 |
| 6 | 79,700 |
| 7 | 89,600 |
| 8 | 85,100 |
| 9 | 80,400 |
| 10 | 83,100 |
| 11 | 84,300 |
| 12 | 87,800 |
| 13 | 81,200 |
| 14 | 33,000 |
| 15 | 16,900 |
| **Total** | **1,315,500** |

Overall gold extracted in respect to ore placed on the heap leach is estimated to be approximately 75 %. Silver will also be recovered but is not reported in the Mineral Reserve estimations. Silver production is not used in the cash flow model. It represents a minor upside opportunity for the Project.

## 19.7 Refining and transportation costs

Precious metal refining companies were contacted to obtain budgetary quotes for refining, transportation and insurance. The entire mass of the doré, including silver and copper, incurs a treatment charge. As silver revenue is not included in the mine economics, no silver refining charge is considered.

## 19.8 Contracts

As of the effective date of this report Fortuna has not entered into any material contracts required for the development of the Lindero Project including mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements.

## 19.9 Comments on Section 19

The QP has reviewed the information provided by Fortuna on marketing, contracts, metal price projections and exchange rate forecasts and notes that the information provided is consistent with the source documents used, and that the information is consistent with what is publicly available within the industry norms. The information can be used in mine planning and economic analyses for the Lindero Project in the context of this Technical Report.

# 20 Environmental Studies, Permitting and Social or Community Impact

Most of information included in this section is sourced from the EIA report prepared by Ausenco Vector (2010) and presented to the Argentine government. Where applicable, sections have been updated with the most recent information available as of the effective date of this Report.

## 20.1 Environmental

In November 2011, the Salta Provincial government granted the principal environmental DIA permit, which is the primary mining permit required for Project development, enabling a project operator to start construction and proceed to full mine operating status.

Mansfield had received a mine permit to build a heap-leach gold mine at up to 30,000 tpd as detailed in the Pre-Feasibility Study (AMEC, 2010b).

The 2016 Feasibility Study submitted by KCA (2016a) altered the mining to 18,750 tpd using a variable cut-off grade of approximately 0.3 g/t Au, increasing average grades early in the production schedule and lowering initial capital requirements.

This updated Feasibility Study has maintained the same mining rate but has made adjustments to the process design and subsequent capital and operating expenditure as detailed within this Report. These changes will be reported to the relevant government authorities, and the Secretary of Mining of the Province of Salta. The conditions of the permit stipulate that progress on the Project is to be reported every six months up to the construction phase with the most recent progress report submitted in September 2017.

### 20.1.1 Legal requirements

The environmental legislation relevant to the Lindero Project is as follows:

- National Law No. 24,585, of the Environmental Protection for Mining Activity Act, incorporated into the Mining Code. The scope of this act was adapted by the Salta Province for application to any mining activity within its territory. Salta Province stipulates that the Mining Secretary is the implementing authority. The submission of environmental impact reports and the corresponding monitoring is the responsibility of this government authority

- Provincial Environmental Protection Law No. 7,070. This law declares that it is a matter of Provincial public order that all necessary actions, activities, programs, and projects must be carried out to preserve, protect, defend, improve, and restore the environment and natural resources within a framework of sustainable development. The Provincial government is task to act as above except in matters governed by special laws such as Law No. 24,585

- Civil Code, Art 14 and 240, this new code (2015) falls under the category of "collective rights" as recognized by the National Constitution (also called rights of the 3rd generation, with its peculiarities such as rights that do not belong to an individual, but everyone). Article 14 states, "*the law does not protect abuse of individual rights where they may affect the environment and collective rights in general.*" Article 240 regulates the limits on the exercise of individual property rights including environmental rights

Article 41 of the National Constitution refers to the rights of individuals residing in an area of productive activity to enjoy a healthy environment without compromising future generations. It also indicates that the Argentine State and Provincial governments will issue rules as appropriate to their jurisdictions. Other related legislation includes:

- Mining Code of the Argentine Republic
- Law No. 24,051: Hazardous Waste
- Law No. 24,196: Mining Investments
- Law No. 25,612: Comprehensive Management of Industrial Waste and Service Activities
- Law No. 25,675: General Environment
- Law No. 25,688: Environmental Water Management
- Law No. 25,743: Cultural and Archaeological Heritage
- Law No. 25,831: Free Access to Public Environmental Information
- Law No. 7,017: Water Code of Salta Province
- Provincial Constitution, Articles No. 30, 79, 81, 82
- Provincial Laws No. 7, 107: Protected Areas
- Articles corresponding to bylaws 005/2008: Environmental Code of the Municipality of San Antonio de los Cobres

**Environmental obligations**

According to Provincial Law 7070, Article 41, Decrees 3097/00 and 1587/03, the technical manager of any environmental work must be registered as an Environmental Impact Evaluator in the Ministry of Environment and Sustainable Development of Salta Province. Maintenance of this registration requires biannual re-enrollment and the updating of the technical manager's curriculum vitae. An EIA must be signed as an affidavit, and the person signing the EIA is liable under civil and criminal law for the content of the report. The directors of the company that own the mining concession are also liable. If environmental damage occurs through mining activity, the mining company may be responsible in the following ways:

- Environmental (recovery to the state prior to production)
- Civil (compensation)
- Administrative (fines)
- Criminal (jail)

Where damage is caused by legally-sanctioned persons, the legislation states that authorities and professionals are responsible to the extent of their participation.

**Other obligations**

In November 2011, Mansfield received approval of the EIA on for Lindero Project. The EIA was prepared by Ausenco Vector (2010) and presented to the government. An update was submitted in November 2013 and a further update in December 2015 (which will be current until March 2018 when the next update is required).

The EIA included several points detailing the obligations of Mansfield to the government, and the following are noted as being significant:

- In each of the periodic extensions attached to the Application Authority, or biannual EIA renewals, Mansfield and the Environmental Professionals Assessors must satisfy all requirements of Resolution No. 172/10 of the Ministry of Economic Development and Resolution N° 448/09 of the Mining Secretary
- All future EIA filings must be consistent in the various stages with the Fixed Capital Investment under Articles 217 and 218 of the Mining Code
- Mansfield must satisfy the requirements of Water Code (Law 7017) and Decree N° 2.299 of its own integral regulations and resolutions of the Water Resources Ministry
- Transport, storage, use and disposal of explosives detonators and blasting accessories for use in blasting shall be governed by regulations from RENAR (National Weapons Register). In Argentina, acts with explosives within the civil (mining) area, are governed by the "National Law on Firearms and Explosives No. 20,429, Decree No. 302/83"
- Mansfield must allow free access to the supervisory government officials from the Provincial government including environmental, health and safety officials, and also officials to review production and cost control data for the determination of Provincial mining royalties
- All service companies that work on the Project shall comply with the Environmental Mining Law, and the Occupational Health and Safety Law
- All mining input suppliers (such as lime, bentonite, drilling and other additives) must have an environmental quality certificate demonstrating compliance with the Environmental Mining Law. Mansfield shall endeavor to ensure that all input suppliers comply with the applicable environmental laws
- Any changes to the original project, including new development and evaluations, which have not been reported in updates required by law or the implementing authority, shall be reported immediately to the implementing authority
- Mansfield must provide an environmental procedures manual that describes all security measures and contingency plans for each activity related directly or indirectly to the Project, including for all carriers and drivers specific to the type of load transported
- Prior to construction, Mansfield shall submit the Measures Plan for the control and suppression of fugitive dust to the Environmental Mining Authority (EMA) for approval (according to the environmental and safety point of view), which must include alternative methods to abatement of particulate material generated in mining operations, crushing and transportation

- Any proposed alternative road route must be approved by the competent authorities and incorporated into the monitoring plan
- Prior to the commencement of the initial stage, Mansfield must submit a Safety and Traffic Control Plan (according to the environmental and safety point of view) to the EMA for approval
- Mansfield must instruct and provide training to all personnel and contractors with respect to environmentally responsible behavior (Law 24,585, Article 3)
- Environmental emergency plans shall "be run periodically" through practice trials and shall be evaluated to ensure an excellent response in the case of a real emergency
- Mansfield must prepare an environmental management plan and monitoring manual to be used for many activities including timing, location, frequency, parameters, methods and standards for the variables listed in the EIA and the technical report, and delivered for approval to the relevant authority prior to commencement of the operational phase
- Prior to the commencement of mining and processing, Mansfield shall submit to the EMA an update of the satellite image of the area of influence of the Project. This image will be in multispectral digital format, with 2.4 m resolution similar to that provided in the environmental permit application. This image will be provided before the beginning of the construction of the Project and then annually (mapping the progress of the operation). This image will show the status of environmental impact of the Project
- Mud from treatment plants must be treated by authorized hazardous waste operators and Mansfield shall submit monthly return receipts thereof to the Water Resources Secretary and the Mining Ministry
- Prior to construction, Mansfield must submit the operation and design of the activated carbon furnace to the EMA for approval. The focus will be on emissions generated by this furnace
- Mansfield must minimize irreversible landscape impact during the construction, closure and post closure phases
- Mansfield shall not mix or bury hazardous waste or industrial tires
- All hazardous waste such as oils, hydrocarbons, contaminated soils, etc. shall be evacuated from the Project and disposed of by authorized operators and carriers, such disposal to be recorded in a Registry Book which must be available to the relevant authorities
- Mansfield shall immediately inform the relevant authority of all oil spills and hazardous substance accidents on routes within the Province or on the Project. In all cases, Mansfield shall immediately provide and implement a contingency plan
- The calculation and design of the leach pad must comply with building regulations and standards according to international rules and guidelines, and all available national and provincial precedents, and this work must not commence without the appropriate approval of the relevant authorities

- Necessary measures must be taken to implement and ensure a continuous and permanent quality control during the installation of the membranes, drainage material and waterproof substrate in the leaching system, to ensure compliance with the provisions of the EIA
- Prior to construction, an electronic monitoring plan for leak detection in the leach pad must be issued by the EMA (according to the environmental and safety point of view)
- Mansfield must submit a monthly field survey undertaken to detect leaks in the leaching system, detailing visual inspection and systematic pattern on the slopes of the leach pad bases and sectors topographically lower around the stack, and to detect areas with moisture or water surge that may come from a leak, in addition to the control methods contemplated in the EIA
- Mansfield shall ensure that the theoretical models presented in the EIA are consistent with the methodical observation of the performance of the waste dump and leaching systems. These results shall be reported and submitted quarterly to the EMA
- Mansfield must submit the waste dump design to the EMA for approval, such design to be compliant with international, national and provincial building regulations and standards
- At Mansfield's expense, the EMA may carry out excavations in the waste dump area or any other area affected by the Project to determine if an improper environmental incident has occurred
- All changes in the Project that impact water use or that generate unanticipated effects on the EIA will be reported to the Application Authority
- Mansfield must submit quarterly reports on environmental monitoring hydrochemistry, hydrogeology, climate, flora and fauna monitoring, waste management, security checks, health and hygiene, with details of subcontractors, their actions, impacts and assigned responsibilities in the monitoring process, or collaboration in monitoring plans and controls
- Five years prior to the mine closure, Mansfield must submit a closure and post closure plan, together with its monitoring plans for all environmental components, to the EMA for approval. It must include a traffic and access plan to prevent accidents and to control the process

**Environmental reports**

Since the discovery of gold mineralization at the Project in 2000, Mansfield has provided more than 20 environmental reports describing various activities such as extraction of samples at initial stages, soil sampling, a program of geophysical surveys, and details of access roads, drilling programs, camp installation, and runways. These reports each consist of a brief description of the environmental baseline, the Project, environmental impact, and ways to prevent and mitigate that impact. On many occasions, the Government of Salta Province has inspected the various Project activities.

In December 2007, Mansfield presented an extensive environmental baseline report (EBL), completed by Vector Argentina, to the Secretariat of Mining for Salta Province.

That report included sections on geology, geomorphology, hydrology, sociology, archaeology, local flora and fauna, soil types, and climate and air quality. The EBL was accepted by the Mining Judge of Salta after being examined by environmental technicians of the Secretariat of Mining and the Provincial Secretariat of Environment. There are no known current environmental liabilities for this Project.

In September 2007, Mansfield installed a weather station (Davis Vantage PRO2™) at the site to record temperature, humidity, wind speed and direction, precipitation, atmospheric pressure, solar radiation, and evaporation. All of these parameters are recorded on a daily basis in a database at the camp. The weather station allows the analysis of updated data on a daily basis and analysis of the data across time.

It is important to mention that there is no historical environmental information for the area before 2007, when the environmental baseline was established, and the weather station installed on site. These data do not allow the drawing of detailed models, such as a water balance model. Data for such models must come from weather stations such as from the La Casualidad, Unquillal, and Hombre Muerto stations, and so any Project-specific models developed from data collected from the actual site- must be used with caution due to the limited time period.

It is important to note that Mansfield has filed an advance activities report every six months since 2012 on the Lindero Project, as established by DIA requirements. The most recent report was filed in October 2017.

### 20.1.2 Existing conditions

**Location and access routes**

The Lindero Project is located southwest of the southern edge of Arizaro Salar in the department of Los Andes, about 7 km south of the old onyx mine of Arita in the southwest of Salta Province. The nearest road access lies about 5 km south of the camp area. The Project is located 60 km from the Salta–Antofagasta railway line, 130 km from the 345 kV Salta–Escondida power transmission line, and 160 km from the puna gas pipeline.

**Climatology**

The Project area has a very harsh climate defined as the Andean puna, which is characteristic of heights over 3,500 m. In general terms, this is a region of aridity that displays distinctly continental features.

The climate is dry and cold, with substantial daily temperature variations in both summer and winter. The temperature in winter averages 5°C. In summer, the maximum temperature can be as high as 30°C.

Annual average rainfall does not exceed 50 mm. Rainfall starts in December and ends in March, but precipitation in the form of snow or sleet can still fall in the months of June, July, and August. The lack of moisture produces static and lightning electric charges and discharges, as well as static electricity in devices and appliances.

Winds are usually relatively calm during the night and mornings, but significantly increase from west to east in the afternoon, especially during the winter months, with recorded speeds exceeding 100 km/h.

Air quality is generally good, except when the wind carries dust that may interfere with breathing. Low oxygen levels, and reduced air pressure caused by altitude can result in mountain sickness (soroche), a common illness on the puna.

**Air quality**

Weather conditions in the area contribute to the generation of dust, including ultra-fine dust particles generated by soil erosion. This is especially the case when weather conditions discharge soil layers, causing increased levels of suspended particles in the atmosphere. Removal of material by mining activity will cause an increase in wind erosion and fine material blasting (silt, clay, and sand).

In the EIA report, air quality was analyzed, with the results indicating that for selected pollutants, the concentrations are below the guideline levels established by current legislation. The results also correspond to natural levels at each site, with levels of pollutants from activities carried out in the area at the time of evaluation corresponding to levels of wind-borne particulate matter.

In February 2012, Ausenco Vector submitted an air quality modeling report. This report was carried out to determine the effects of the proposed Project area on the air quality in the immediate environment in which the Project would be located. This report presented an estimation of emissions and modeling results of air quality during the operational (mining) phase of the Project.

The report concluded that there will not be adverse effects on the most important town near the Project (Tolar Grande) nor are adverse effects expected in rural settlements or in Antofallita and Cavi Vega.

**Water quality**

For the EIA studies, physical–chemical analyses were performed at the Environmental Studies Laboratory (National University of Salta) and at Alex Stewart Assayers Argentina S.A. in Mendoza. During the pre-feasibility phase, control analyses were performed at the Induser laboratory in Salta.

The selected parameters for the laboratory analyses corresponded to those established in Annex IV of Law No. 24,585–Environmental Legal Framework for Mining Activities. In the EBL study, the results were compared with the water quality guideline levels established in Annex IV of Law No. 24,585, both for drinking water and protection of water life, and with the values specified in the Argentine Food Code.

The analyzed surface water bodies are mainly used for drinking by native fauna and for general domestic consumption by residents of the area. Water from water bodies assessed in the Project area was used for exploration-related activities and for some domestic uses. For culinary purposes, and human consumption in the camps, offices and work areas, an RO water treatment plant is budgeted to provide potable drinking water to the mine site.

**Hydrology**

According to the EIA, the Rio Grande basin and watershed take the Arizaro Salar as the local base level.

In general, the Rio Grande basin does not have significant superficial run-off, with most of the streams in the area being intermittent, as evidenced by the few drainage lines developed. Only the Rio Grande River can be considered a water source of importance.

This streams flow permanently at its origins but disappears into the salar after a few kilometers. There are no measurements of this watercourse.

The Rio Grande basin has an area of 1,687 km² and consists of numerous sub-basins, including the Rio Grande peninsula, Lindero, Arita, and Chaschas. Most of these sub-basins are tributaries of the Arizaro Salar, with the remaining defined sub-basins, Cori and Emboscadero, forming from centripetal flow. In its lower regions, the Rio Grande basin has numerous flat lowlands with lagoons fed by springs that reach their peak during the winter months.

**Soil science**

According to the soil atlas of the Argentine Republic–Salta Province (1:500,000 scale) developed by the National Institute for Agricultural Technology (INTA) the following taxonomical classifications apply to the study area:

- Order: Entisols
- Suborder: Ortentes
- Major Group: Torriortentes
- Sub-group: Lithic and Typical

**Fauna and flora**

In the Project area, weather conditions substantially affect vegetation growth. The almost total lack of rainfall in the puna determines a vegetation floor that corresponds to the "Province of puna" shrub steppe, an herbaceous steppe. Key flora includes; añagua, lejía y tola, añagua y rica–rica, iros, muña–muña, vira–vira, and chachacoma. There are some areas with no vegetation whatsoever.

The fauna is typical of the Andean Patagonian sub-region, Andean district, and is mainly represented by camelids (the llamas, vicunas, and guanacos). Donkeys have been incorporated into the landscape for the last 50 years and compete for pasture with other herbivores.

Carnivorous animals such as the fox, skunk, and lynx, and rodents such as "the hidden" (a kind of mole), small guinea pig (cuis), mouse, and chinchillas, are present. Bird life can include the Andean ostrich (rhea or small ostrich), hill partridge, ordinary coot, lapwing, and parinas (flamingos). Reptiles are represented by lizards, which are common, and snakes, which are very rare.

**Ecosystem characterization**

The EIA included a detailed description of the eco-regions for the exploration phase by characterizing the eco-regions, the communities, and various agencies involved in the Project area, and their interactions. Ecosystem characterization allows a more complete understanding of the existing interactions between a given project and the environment, and vice versa.

Eco-regions, or biomes present in the Project area are the puna eco-region and the high Andean eco-region. These eco-regions were divided into ecological units summarized by descriptions of the soil, flora, and fauna, and their particularities and interactions (shelters, niches, eco-tones, barriers, and corridors comprising areas of frequent use). The degree of disturbance that these communities are experiencing due to human activity is currently being evaluated.

### 20.1.3 Local ecosystem characterization

To characterize the local ecosystem presented in the EIA, a summary of environmental conditions of the area was formulated, based on the data from the baseline studies conducted for the Project, including flora (Vector, 2007a), fauna (Vector, 2007b), soil (Vector, 2007c), social (Vector, 2007d), and other environmental reports and studies.

### 20.1.4 Identification of protected areas

The protected area nearest to the Lindero Project is the Provincial Fauna Reserve of Los Andes (protected since 1980), located 45 km to the north.

The Provincial System of Protected Areas of Salta (PSPAS) was created in 2000 by Act No. 7,107. Until then the Province of Salta lacked a legal framework for the protection of existing natural areas associated to a production area. The PSPAS comprises the protected areas of the Province, and its goal is to promote the management and effective protection of national parks, reserves, and natural and cultural monuments of the Province.

The Lindero Project area is highlighted by the red circle on Figure 20.1 that shows the location of the Project in relation to the nearest protected areas.

**Figure 20.1 Location of National Parks**

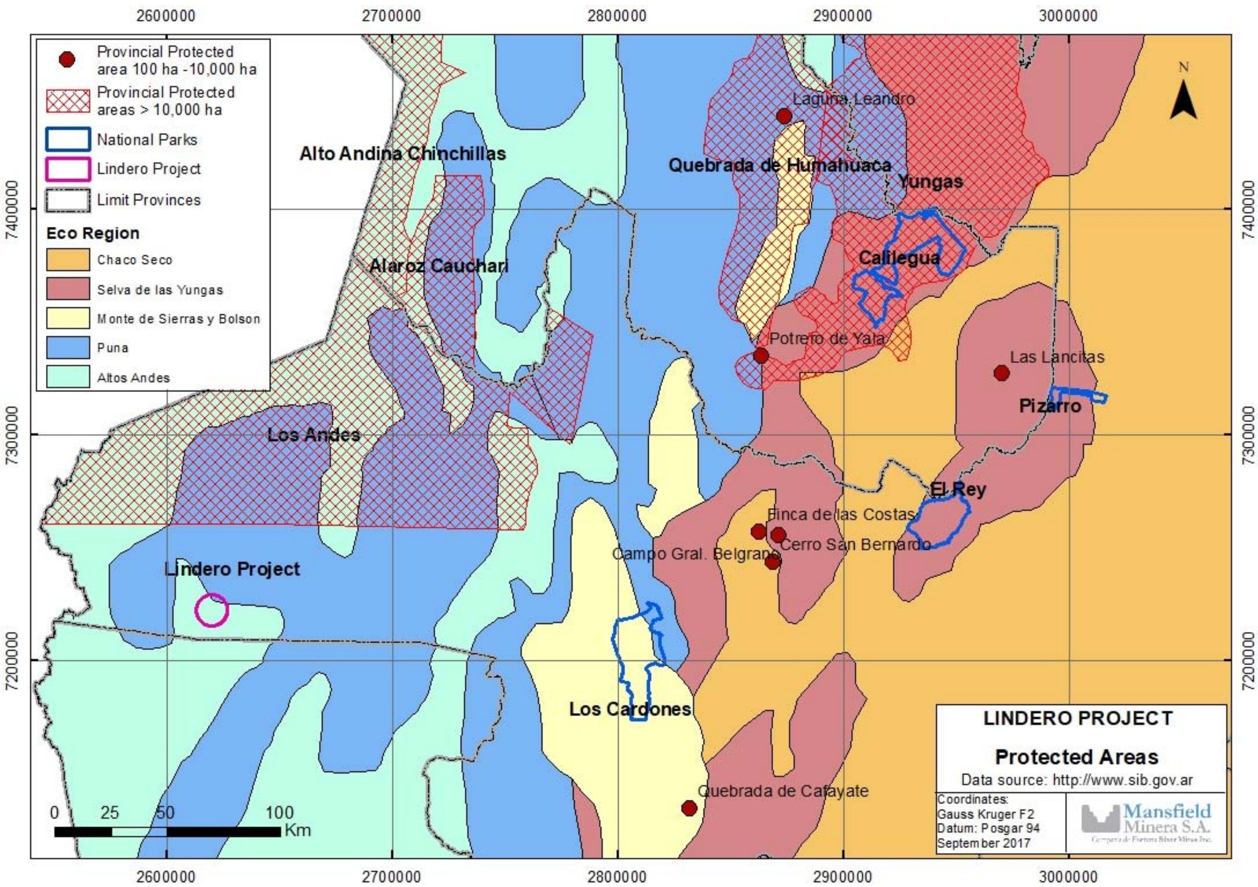



Figure from www.sib.gov.ar and modified by Mansfield, 2017

### 20.1.5 Archaeology

The prospecting undertaken in the region of the Project showed the presence of archaeological points of interest and isolated scatters. These points comprise four main zones, as shown in Figure 20.2.

**Figure 20.2 Location of archaeological sites**

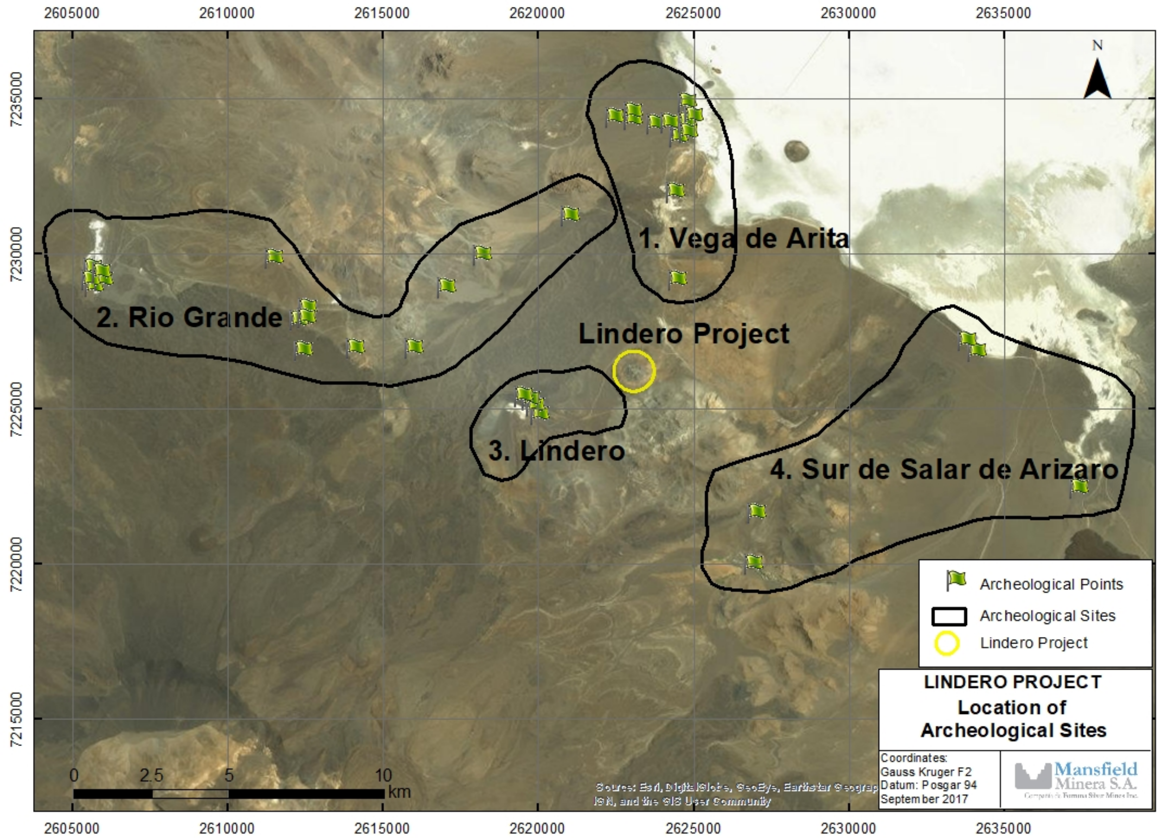



Figure prepared by Vector (2007e) and modified by Mansfield, 2017

The prospecting undertaken in the areas of primary or direct impact of the Lindero Project indicated no archaeological and historical sites were present. However, along the access roads to the Project and in other areas of Project influence, 36 archaeological sites were located. These sites were grouped into four sectors based on the proximity to the Project.

The chronology of the identified archaeological sites situates them at the end of the Pre-ceramic or Paleoindian periods (7,000 BCE onwards), as set out in the EBL (Vector, 2007e). The age of these sites in the Arizaro Salar gives them patrimonial significance.

The key sites are as follows:

- Arita vega and camp sectors
- Arita vega
- The camp area
- Rio Grande sector
- Sites south of Rio Grande Hill
- Lindero sector

- Emboscadero vega
- Arizaro Salar south sector
- Cavi vega
- Outcrop site in Arizaro Salar
- Chaschas vega

### 20.1.6 Environmental risks

**Risk assessment**

Irreversible impacts are those changes to the environment that persist indefinitely because it is not feasible to mitigate or restore the environment to an original or equivalent condition. Such irreversible impacts of the Project will be related to the pit, waste dump, and heap leach pads, as these works will permanently modify the local landscape.

**Risk register**

In each stage of the Project development, an analysis of associated potential environmental risks must be undertaken. Potential risks will be considered at the stages of construction, development, closure, and post-closure of the mine. All of the facilities will be designed to provide safe operating conditions in order to reduce potential impacts on surface waters in the Project area of influence.

**Construction phase**

At the construction stage, the risk arises from the intense activity that the construction of the mine will require. The estimated time of construction is approximately 21 months. During this period, up to 600 people will be housed in the camp and will be involved in the construction of facilities such as the camp, water infrastructure, water pipeline, power plant, roads, process plant, leach pad and crusher.

The greatest risk during construction will arise from the fact that the majority of staff will be under the control of various contractor firms with different environmental management policies. Mansfield must be especially careful to include strict environmental regulations in contracts and enforce those regulations with contractors.

At the construction stage, increases in personnel and equipment transport will be significant, leading to increased environmental risks caused by accidents, release of particulate matter (dust) into the atmosphere, and, from the social point of view, disturbance of the peace in villages and communities.

A social-environmental risk that may also occur arises from the expectations of the people in relation to job creation. While up to 600 people will be hired in the construction stage, many from nearby communities, only 360 jobs will be required during the mining operations stage.

It is estimated that during the construction phase, about 1,000 kg of household waste per day will be produced. This much waste constitutes a serious environmental risk of pollution if measures are not implemented to deal with it appropriately.

The disposal of grey and black wastewater generated by the workforce (camp and offices) will be treated at a water treatment plant located at the accommodation camp and the treated water from this water treatment plant will be used in the heap leach operation, effectively recycling this water.

There will also be significant production of hazardous industrial waste, such as used oil, grease, batteries, waste fuels, and associated materials such as containers. There will also be non-hazardous industrial waste, such as used personal protective equipment, waste timber, geo-membrane fabrics, uncontaminated textiles, bricks, insulating plastics, uncontaminated plastic containers, construction debris, and iron and nonferrous scrap. The creation and possession of these wastes are an environmental risk that must be dealt with.

During the construction stage, approximately 1.0 $Mm^3$ of soil will be moved during leveling and construction of various facilities and foundations. The majority of this area will be the installation of the leach pads and pit waste storage piles. Such soil movement constitutes a risk in itself, but it also damages the almost nonexistent (very thin) layer of soil, with its associated flora and fauna. Even the access roads to these constructions pose significant risk of disrupting the environment, so strict policies must be set forth to minimize their impact.

During the construction phase, there will also be considerable consumption of fuels for transport tasks of all kinds.

**Operation phase**

The topography of the land will be altered by mineral extraction, especially in the area of the pit and by construction of the waste dumps and heap leach area. The modifications made by the waste dumps and the heap leach area do not represent geomorphological risks because they will be installed on sites that have no relevant natural hazards and will be designed to withstand earthquakes and extreme rainfall events.

**Dump or sterile rock deposit**

While it is predicted that the waste dump area will have adequate long-term stability, monitoring of slope stability and deformation of the dump will be carried out.

**Water**

The risk of water pollution will always be an issue for consideration and control. Monitoring and sampling must be constant, and the results reported to the relevant authorities. A comprehensive program of monitoring downstream water quality at several points must be implemented in order to put aside any uncertainty about potential contamination. Normally in this type of mine there are concerns about the risks of acid drainage and cyanide presence from the leach circuit although this is minimized by the lack of precipitation and drainages.

Water extraction from two or three planned wells in the Chachas area, 13 km east of the Project, creates a low risk of reducing the down slope subterranean flow of fresh water to the salar. There is no production activity (farming or residential) in the sub-basin or near the shore of Arizaro Salar from where the water will be taken.

**Air quality**

The operation of the Lindero Project will generate emissions of particulate material (dust) in the mine and plant area, mainly from mining operations such as drilling and blasting, movement of vehicles on dirt roads, loading and unloading of mined materials; and mineral processing operations such as crushing and agglomeration.

A plan for monitoring, control, and mitigation or reduction of suspended particulate material (dust) will be implemented through the installation of various types of dust suppressants and road irrigators, preferably using brackish water as a flocculent of airborne dust. The nearest community is not downwind of the Project.

Gas pollution from vehicles and generators, the gold smelting furnace, the carbon regeneration furnace, and the analytical and metallurgical laboratory, will be carefully controlled and mitigated.

**Soil**

Both the leaching area and process plant were designed as closed circuits and therefore no soil contamination is expected. There is, however, always a potential risk of failure in the leach circuit, in the transfer of solutions to different facilities, from drainage of leaching solutions from pads and ponds, and from heavy precipitation that may cause solution in the circuit to exceed levels. The construction of a contingency (excess) pond will be included to provide protection against such events.

The ADR and SART plants will have a central drainage system to collect any spillage and send it to the containment ponds.

During construction, operation, closure, and post-closure, events or situations of low probability of occurrence may arise involving the contamination of the land, such as an oil spill, chemical spill, or accidental discharge of process solutions. Each of these events will have health and safety action plans.

**Flora and fauna**

During mine operations, flora may be at risk where mining activities are undertaken, such as at waste dump areas in the vicinity of the open pit, and where blasting, processing, crushing, and ore transport activities, and roads occur. All of these areas can cause deposition of particulate matter.

Human activity, particularly household waste, can attract animals to the area, especially foxes. For this reason, the site must be closed to such animals so as to mitigate the risk of accidental poisoning, or other impacts such as waste dispersion. By regular soil covering in the disposal pit, access by animals will be avoided.

**Sites of archaeological interest**

No sites of archaeological significance were found in the area that will be directly affected by mining activity. The nearest sites are the Arita vega and the south sector of the Arizaro Salar. A control plan must be implemented in areas of archaeological interest closer to the Project, including fencing and a staff awareness program.

**Noise**

Due to the location of the Project, the environmental noise impact will be virtually nil for nearby residents and local wildlife. Noise and vibration control will be planned and installed correctly, and then maintained during the operation stage.

**Closure and post closure phase**

Environmental risks during the closure stage will be reduced by remediation and monitoring work. At the closure stage, soil will be contoured by heavy machinery to minimize the long-term impact of mining activity, and return the topology of the land to resemble prior conditions. However, the movement of soil, and thus the risk, will be significantly less than in the mining operations stage.

One social-environmental risk will be the completion of contracts of employment directly, or indirectly, through contractors, and the surrounding communities. It will be imperative to implement measures to mitigate this impact during the whole period of mine operation.

A significant environmental risk will also be present during the closure of facilities, which will cause significant production of non-hazardous industrial waste and hazardous products from the movement of heavy machinery. It will be essential to establish clear environmental policies with the contractors during this process.

The landscape is an important factor to be considered during this stage because the mine will have a definite and obvious impact on the natural landscape. To the extent possible, the impact on landscape must be minimized.

### 20.1.7 Environmental management plan

One of the priorities of Mansfield is the care and protection of the environment. During the exploration phase, an attempt was made to control to the greatest extent possible any potential environmental impacts on the area. The same effort must be made in the next stages (construction and operations) of the Project. Therefore, Mansfield has defined environmental principles that will enable the development of mining operations efficiently from a productivity standpoint and from the environmental point of view:

- Comply with existing environmental laws and regulations
- Establish and maintain an environmental management program to guide operations
- Involve the entire staff of Mansfield and contractors in the Environmental Management Plan (EMP)
- Promote environmental awareness among employees and the communities where operations occur
- Mitigate the potential environmental impacts that do occur and support environmental improvement programs for common benefit

The EMP defines the criteria, the design of specifications, and management practices that will be applied to the Lindero Project in order to mitigate, control, and monitor changes in the baseline conditions during construction, operations, closure, and post-closure of the Project.

Corresponding prevention measures, mitigation of potential environmental impacts, as well as rehabilitation measures, are outlined as appropriate. The phases of construction, operation, and closure of the Project are addressed.

Prevention measures will avoid potential environmental impacts, while mitigation actions are intended to minimize, correct, or compensate for environmental impacts of the Project at different stages. Measures to increase, improve, and enhance the positive environmental impacts caused by the Project will also be implemented.

An environmental contingency plan will be implemented in order to predict potential environmental incidents that were not taken into account during the life of the mine.

**Measuring conditions**

The following criteria were applied when identifying the environmental protection measures to be proposed for the Project:

- A clear and understandable proposal allowing reliable and consistent implementation
- Technically feasible activities that can be reliably implemented in practice
- Proposed measures that are economically appropriate to the scale of the Lindero Project
- Ease of monitoring and control during the different stages of the mine life

The Protection for Mining Activity Plan (PMA) that Mansfield will implement for the exploitation phase of the Lindero Project will follow the considerations established in Law No. 24,585 of the Environmental Protection for Mining Activity Act.

Erosion control works such as trenches and gabions will be built as necessary during any stage of the Project to ensure the physical stability of the facilities against possible flooding caused by rare heavy rains.

The stability of the waste dumps, leach pads, and pit will be monitored to prevent any slippage or collapse. In the bank area of the heap leach, piezometers will be installed to continuously measure pore pressures in order to detect the formation of phreatic levels in the slope that may cause slope failures.

In the construction and closure phases, the heap leach pads will have an appropriate slope. Berms will be built around the pad perimeter to prevent any overflow. Drainpipes will be maintained around the pad perimeter in case of possible storms.

A closed-circuit leach system was designed to minimize fresh water requirements, prevent discharges to the environment, and maximize the reuse of all process solutions. As a control measure, water downstream of the leaching zone, plant, and sterile tank will be monitored regularly to identify any abnormalities in the chemical properties of the groundwater.

Contingency measures will be implemented to mitigate or remedy any impact which may occur. In the leaching circuit, a contingency (excess) pond will be built to absorb solution in the leaching area and other ponds in the event of a malfunction of the leach circuit. As a corrective measure, downstream areas will be designed to recover any leaching solution and return them to the circuit. In the waste dump area, the water and soil will be monitored for acid rock drainage (ARD) using an acid–base-accounting (ABA) testwork. Travertine (abundantly located nearby) can be used for neutralization of acid if required.

Reclamation of the site during the closure and post-closure phases will help to reinforce all other measures taken.

One of the environmental risks previously identified is the impact on the landscape of vehicular traffic, such as new roads, spills, hazardous waste such as fuel and diesel oil, cargoes such as process reagents, and dispersion of these residues in the soil and air. Preventive measures will be implemented to mitigate these risks. Contractors will be required to have a program of security and risk prevention for vehicles, adjusted to Project regulations in regards maintenance of vehicles, traffic speeds, night traffic restrictions, driver-training requirements, and load handling.

As a preventive measure, signs will be installed along the roads in areas that are considered to be dangerous. Communications between vehicles and with the Project control center will be established within the mine. Communications will also be established between the Project and various control points on the access routes, such as at Salta and the mine for monitoring vehicle status, load and source data, and estimated times of arrival. If any environmental accident occurs, a contingency plan will be triggered to avoid a significant environmental impact.

A lesser environmental risk related to vehicle traffic is that of emissions of particulate material (dust) and exhaust gases. As a precautionary measure regarding exhaust gases, no vehicles on the Project can be more than 15 years old, and all must comply with current national legislation with respect to a mandatory technical review. To minimize emissions of particulate matter (dust), besides the restrictions on speed limits, water trucks will wet the roads constantly.

Experience in the puna region has shown that use of brackish water or brine that could be easily be sourced from the Arizaro Salar, can achieve 80 % efficiency in the reduction of particulate matter (dust) emissions into the atmosphere. The conventional method of using fresh water is estimated to function at 60 % efficiency. Therefore, using brackish water will decrease the volume of fresh water used for dust suppression. There will be permanent monitoring of atmospheric particulates (PM10) in the mine area and surrounding communities to establish comparative benchmarks against the environmental baseline. The results of the monitoring program will be communicated to the local population. Company policy and Project development and operation must not only be environmentally and socially sustainable, but the awareness of social and environmental status must be demonstrated to the local community and other stakeholders.

Regulations will require that vehicles be maintained to minimize noise and vibrations. Ad hoc measures will be established requiring drivers and contractors to carry loads well stowed, especially during the mine construction and closure phases.

Emissions of blasting products and gases into the atmosphere will be specific and very limited in duration (time). It is estimated that there will be one blast per day, and at this frequency is not anticipated to have significant effects. As a preventive measure, emissions will be monitored because the excessive release of particulates (dust) into the atmosphere may mean that adjustments in the blasting procedure are necessary to ensure maximum efficiency is achieved with each blast.

The plan is to place restricted area signs and, in some cases, fencing at the closest archaeological sites in order to protect them from any possible impact, especially those that are closest to the access roads. To protect any archeologically valuable sites or material found, a professional archaeologist will be consulted in the planning stage of the Project and during any soil removal in areas considered to be of high archaeological sensitivity.

To mitigate the risk of environmental impact from solid waste and domestic and industrial fluids, a comprehensive plan will be established for managing these wastes, including classification by type, hazard, and recyclability. The waste management plan will be presented as a compulsory preventive measure in the induction report to any person entering the Project area. A part of the property will be designated as a waste storage area and surrounded with a small wire fence or frame, to prevent the entry of persons or animals and to avoid dispersion of the waste by wind.

The entrance to the waste storage area will be controlled by staff dedicated to that purpose; Mansfield's employees will record the quantity, type, origin, and contractor bringing waste for storage. The waste storage area will be waterproofed with geomembrane beds or depositories duly identified for the different types of waste. The master plan for waste management will establish the guidelines for removal, transportation, and final disposal of this waste. It will be required by contract that those contractors who are hazardous-waste generators must be registered in the Province of Salta under current regulations and, upon final payment of the contract, a certificate of final disposal of waste removed from the mine will be required.

As a mitigation measure, practices will be established that decrease the amount of waste in the mine, such as the use of reusable drinking water containers. Grey water from the kitchen and shower, and black water from the toilets, will be dealt with appropriately (treated and used in the leaching process). Waste water treatment will be conducted using a water treatment plant sized for the camp and offices.

Flora and fauna will not be significantly affected because the main activity will take place in areas where there are no significant numbers of plants or animals. However, some consideration has been given to marking areas such as the Emboscadero area where wildlife such as flamingos and vicuñas and micro-organisms are known to occur.

Mansfield will ban hunting, pets, and other activities that could scare away native animals in the mine property and permitted surrounding areas agreed by government officials.

Mansfield will continue with regular information meetings regarding the Project with the surrounding communities. To support positive social impacts at the beginning of construction, Mansfield will conduct a survey to determine the population's knowledge, so as to properly implement training programs on the more common and necessary tasks during the initial stage of construction.

Contractors will be instructed to perform the same training practices in these communities. Meetings will be carried out with the community to create co-operative mining services (i.e. catering and cleaning services). After the permanent mine closure, the camp facilities may provide alternative employment such as tourist accommodations.

While the construction and improvement of roads will cause an environmental impact on the landscape and soil, there will be a positive impact from the social point of view for those people living in isolated places. These inhabitants will have better roads, the flow of tourists in the region may increase, and the local economies may have better opportunities for growth.

To mitigate the strong, negative social impact when a company withdraws from an activity such as a mining operation, it is necessary to implement measures during the operations stage. These measures could include employment training that will be useful to the employees after the closure of the mine.

### 20.1.8 Issues of environmental concern

The following general list specifies the most important issues to be considered from the environmental viewpoint and that must be detailed carefully in the EMP.

- Surveys of residents living in communities near the Project
- Interests of those agencies evaluating the EIAs
- Special requests by the Ministry of Environment and Production of the Province of Salta (DIA, Environmental Impact Declaration)

- Publications by Argentine media
- Mining activity of concern to Argentine citizens
- Statements by NGOs, politicians, and other opinion makers

The key areas of community environmental concern are:

- Use of cyanide processes in the mine: special care will be placed on the prevention and establishment of monitoring controls of this input for transport, use, and final disposal. It is recommended that a clear diffusion of knowledge be undertaken through authoritative communicators during the feasibility process
- Formation of acid rock drainage in the waste: the prevention, mitigation, and restructuring plan must explain in detail how this issue will be handled
- Water use by the process plant and leach pads: publications and comments concerning the excessive use of water and pollution of the same in mining activity are common. It will be necessary to implement strict monitoring of water and make this process a participatory activity in which companies, governments, non-government organizations (NGOs), and the community are involved. It is recommended that water quality data be communicated with higher frequency than that required by the supervisory authority
- The environmental impact on populated areas: the closest community to the Lindero Project is 15 km away in the Cavi vega. There are also more distant communities such as Tolar Grande and Antofallita. A communication plan with these communities that is both diplomatic and discretionary will be required in order to learn, anticipate, and act against false ideas from whatever source about potential environmental pollution. It is important to note that any abnormality occurring in the mine locale, such as the untimely death of a llama, may be blamed on the Project notwithstanding that investigations will later prove to the contrary. It is very important to establish Mansfield as a "good neighbor," such as having a technician make frequent visits to maintain communications and deal with concerns of the communities as they arise. It is best if aggressive security measures, such as strongly worded no-trespassing signs, be avoided, and more informative messages emphasizing safety aspects be conveyed
- Effects of blasting operations on the environment: during the operations stages, detailed information on the rock blasting process should be disseminated to the stakeholders
- Doubts concerning the environmental and social impact of the closure and post-closure phases: the implementation of plans to minimize environmental impacts after the mine closure will be described in more detail during the mine construction and operations stages. Details about the work carried out must be communicated during these stages in order to minimize a traumatic and abrupt change during the closure process. There must be an open and honest communication with stakeholders about the final status of the mine, especially its open pit, waste dumps, and heap leach pads, which will be the largest remnants of the mine on the landscape

### 20.1.9 Operations and management

It has been anticipated that operation and management of the Project will be dynamic.

There will be environmental monitoring as in any mining operation, but a community input monitoring plan will also be implemented regarding the activity of Mansfield. This will allow Mansfield to hear directly from the local residents what they expect from Mansfield on issues related to environment. Responding to these concerns quickly and honestly will show the community that Mansfield takes its concerns seriously.

The goal is to have the local people consider Mansfield as an important part of the local community, with an open-door policy for the population in general.

While there are strict requirements from the competent authority (Secretariat of Mining of the Province) for items such as the EIAs on the Project and the respective bi-annual renewal reports, it would be extremely valuable to also submit to the enforcement authority and the community in general the environmental monitoring report due to be performed internally within Mansfield every six months. This will bring more contact with the government and community.

Environmental remediation costs will also decrease upon closure if all measures are taken to mitigate negative impacts and remediate all possible damage from the beginning of mining activities in the mine, taking into account every consideration that both the Secretariat of Mining and the community may consider.

## 20.2 Socio-economic and cultural aspects

Within Los Andes Department, the municipality of Tolar Grande is the closest and covers the Project area. It is located at 3,500 masl, and is reached by Provincial Route No. 27. The municipality covers 13,785 km² and has a population of approximately 250 inhabitants, most of whom live in the town of Tolar Grande.

The mine will be located in an area where mining activity has historically taken place. For this reason, all of the people interviewed demonstrated knowledge of the mining activity in the area. The vast majority of residents interviewed highlighted mining activity as a job generator in the region.

The environment is a serious issue for the residents. Although they believe that the mining activity does not substantially affect the environment, they refer to certain risks, such as water use, dust pollution, and the effects on the people living in the vicinity of the Project.

As viewed by local residents, the relationship between mining activity and job generation has three main aspects. First, the region has a history of mining activity, so the people are familiar with what it means to the area. Second, several people in Tolar Grande have worked on the exploration phase of the Lindero Project and other mining ventures, so the Project is familiar. Finally, there is the belief that new mines will provide jobs and help avoid the migration of young people to larger towns or cities.

Work in the mines is valued because it offers comparatively better salaries, establishes access to social security, and now, due to better mining technology, is seen as less difficult and labor intensive than in the past. One reason is the pursuit of opportunities that can provide training for future work in mining.

In relation to the economic benefit that the mining activity implies, some respondents recognized that the limited mine-life is a negative aspect. This is exemplified by local experiences with the La Casualidad mine, where the mining boom was followed by total abandonment.

The Lindero Project area is not an area of direct historical value. The socio-cultural study conducted in the EBL indicates a positive trend in terms of mine impact.

### 20.2.1 Community relations

The closest town is Tolar Grande, 75 km to the northeast of the Project. The town local government is represented by the "Intendente" or local mayor, who governs the town and local surroundings, which includes the Project area.

Continuous contact between Mansfield and the community of Tolar Grande dates back to 1994. Mansfield has contributed to the education of the local population, particularly with regards the potential mining activities and the development of the Lindero Project. This contribution has been carried out through technical talks to the community and the organization of visits to the Lindero Project for local authorities, teachers, and students.

Mansfield places a high priority on employment of local workers.

Mining development can have a remarkably positive socio-cultural impact. For the exploration phase, staff was recruited from the communities of Antofalla, Antofallita, Tolar Grande, Olajacal, Pocitos, San Antonio de los Cobres, and Santa Rosa de los Pastos Grandes. These individuals demonstrated a great change of attitude regarding responsibility and predisposition towards private activity (mining). It is worth mentioning that the constant training they received in their work, gave them experience as trained personnel for future mining development work in the area.

Traffic in the area generates an indirect positive impact on the lives of isolated villagers. Communication with friends and family increased due to the visits and packages they received via company vehicles that regularly travel to urban centers.

In order to guarantee the security of the local population and visitors to the region, the exploration companies and the municipality of Tolar Grande have regular meetings where emergency and safety and security measures are discussed, and related operating protocols are agreed on. For example, they have produced a map of the area that shows the location of roads, mining camps, areas covered by cellular phone towers, shelters, and nature reserves.

Mansfield has developed guidelines requesting support and contributions to the community as follows:

- Ensure that requests to Mansfield are made in the presence of two or more people that have no direct relation to each other
- The request should be submitted in writing, describing details, giving the names of the requesters, and mentioning the reason for the request
- Requests should be submitted directly to the responsible authority in the Mansfield to ensure that the resulting support will be viewed and provided by the company, and will not be associated with a particular person

- Support and donations always have to be provided in the presence of more than one person, directly to the requester, and never through intermediaries
- A receipt must be signed to register the donation. In some cases, it is also advisable to request a written note of thanks, for Mansfield's audit purposes
- Ensure that the request relates to a general use or purpose for the entire community and that it provides a lasting benefit (i.e. food, fuel and other consumables are excluded)
- Rotate the benefits as much as possible to avoid favoring the same sectors
- Learn to say NO on occasion and explain the reasons. Do not postpone saying no to a later date
- Never give money; it does not contribute to the goals of CSR and is undoubtedly counterproductive

Mansfield participated in the signing of an agreement between the Salta Mining Chamber, the Salta Mining Secretary, the Salta Employment Secretary and the Salta Small and Medium-sized Enterprises Secretary to support training and capacity building. The courses and conferences offered are supported by other companies and foundations such as the Foundation UOCRA (Unión Obrera de la Construcción de la República Argentina – Construction Workers Union of the Argentine Republic), Finning and Austin Powder, and Manpower. Courses offered include building, construction carpentry, electricity at home, mining safety and security, blasting and heavy equipment maintenance. These were offered in combination with intensive courses on the use of personal protection equipment, looking for work, and other basic aspects of labor relations.

In addition, a formal public declaration of support for the Lindero development has been issued by the Provincial government, recognizing Lindero as the priority development project for the Salta Province.

According to the EIA, a regular social survey between residents of direct influence area (Tolar Grande and neighboring positions) and indirect (involving the populated area located between Campo Quijano and Tolar Grande or any other way used by the company, to construction work or during the operation) shall be completed in order to have a full view of the opinions of those people regarding Project activity. Mansfield is expected to report these results to the competent authority every six months.

Mansfield is required not to modify or alter the social and cultural communities or customs of the area surrounding the Project.

## 20.3 Permitting

Mining in Argentina is governed by both Federal legislation and Provincial laws and decrees. The permitting process, as well as regulatory activities during development and operations, are managed by Salta Province.

A mine is put into operation in two phases, starting with an Environmental Impact Assessment, and then a sector permitting phase. Each of these phases is described further below.

Approval of the Environmental Impact Assessment allows mining development to proceed, subject to obtaining sector permits for specific project facilities.

### 20.3.1 Permit commitments and obligations

**Environmental Impact Assessment**

The Lindero Environmental Impact Assessment (EIA) – Operation Stage (Proyecto Lindero Informe de Impacto Ambiental – Etapa de Explotación), was submitted in October 2015 to the Salta Provincial Mining Authority. Approval of the EIA was granted in January 2016 and represents formal approval for mine construction, allowing excavation to proceed. EIA approval is also a requirement for several of the sector permits that are required for certain installations and workings as described below.

The EIA report describes and assesses associated environmental impacts for exploitation of the Lindero Deposit via open pit and heap leach processing of the ore, in accordance with the Project as described in the 2016 Feasibility Study.

Salta Province supports responsible mining, and the Sub-secretary of the Ministry of Mines is intent on streamlining the EIA review and approval processes. A Multidisciplinary Mine Environmental Evaluating Commission (Comisión Evaluadora Multidisciplinaria Ambiental Minera), has been assembled to revise the EIA process.

The Company's DIA was approved in October 2011. This document is the guiding (primary) operating permit during the life of the Project. It is also a requirement for the granting of most sector permits for the Project. The obtained DIA establishes 109 conditions to put in production the mine, related to social, environmental, health and safety and taxes matters.

**Sector permits**

Specific approvals and permits are required for many aspects of the Project, including:

- Construction permits
- On-site bulk fuel storage
- Domestic and industrial effluents
- Authorization for explosives use and storage
- Water management
- Mine operations
- Communications

The Salta Provincial authorities have approved the building and electrical permits that Mansfield requires to commence construction at Lindero. Electrical, structural, building and seismic plans have been reviewed and approved by COPAIPA (Dec 2013), the professional engineering institution that overlooks all construction in Salta Province. Mansfield is planning to submit additional information to COPAIPA in 2017 to obtain the permits for construction of the agglomeration and SART plants that have been added to the process design.

**Permitting agencies**

Permitting government agencies are listed in Table 20.1, including a summary of the most important required approvals and permits.

### Table 20.1 Required key permits and authorizations

| Area | Key permits and Authorizations | Jurisdiction |
|---|---|---|
| Mining Law 24, 196 | Certificate of Registration under number 242 | Secretaría de Minería de la Nación |
| Import and export | Certificate of Import and Export (done) | Customs |
| Mining Property | Onix mine - File 16,835 granted | Salta Mining Court House |
| Surface Rights | File 17,206 Arita camp rights (granted)<br>File 18,387 right of way and water rights (granted)<br>File 19,200 Water rights (granted)<br>File 21,995 Surface water right and pipeline<br>File 22,096 Camp rights (mine site)<br>File 22,093 Plant and process easement<br>File 22,094 Waste dump easement<br>File 22,095 Open Pit easement | Salta Mining Court House |
| Water Resources | Water Concession for Mining Activity Use<br>File 34-13173/12 Mine site Hole 1<br>File 34-56786/12 Mine site Hole 3<br>File 34-223780/12 Mine site Hole 2<br>File 34-42367/10 Chachas area Hole 1<br>File 34-23834/13 Chachas area hole 2<br>File 34-6251/05 Arita Camp superficial water | Secretaría de Recursos Hídricos de Salta |
| Waste Management | Disposal of Hazardous Waste via Landfill<br>Pathogenic Waste Provincial Registration as Generator, Operator and transporter of Hazardous Wastes (registered, needs upgrade) | Secretaría de Medioambiente y Desarrollo Sustentable.<br>Secretaría de Minería Municipio de Tolar Grande |
| Environmental Management | Environmental Impact Declaration (Approval of Environmental Impact Statement, art.250 Mining Code, art.34 of Provincial Law 7,141)<br>Comply with the DIA requirements | Secretaría de Minería de Salta |
| Camp qualification | Arita Camp municipal qualification (is process of renewal) | Municipio de Tolar Grande |
| Explosives | Registration for Explosives Users (Mansfield Minera S.A.)<br>Registration for Users and Vendors of Explosive Services (vendor)<br>Certification of Powder Magazines (contractor)<br>Storage of Ammonium Nitrate and Controlled Products (blasting contractor and MDR) | Ministerio del Interior Registro Nacional de Armas (RENAR) |
| Contract foreign professionals | Hire foreign professionals | RENURE (Migrations) |
| Fuel | Registration in the Liquid Fuel Dispensing Registry or National Liquid Petroleum Gas Registry (contractor) | Secretaría de Energía |
| Gas compression plant (Salar de Pocitos) | Environmental Impact Declaration | Secretaría de Medioambiente y Desarrollo Sustentable. Secretaría de Energía.<br>Municipio de San Antonio de los Cobres |
| Chemicals | Use of chemical products | SEDRONAR |
| Communications | Use of satellite telephone and internet<br>Use of VHF hand held radios | Private contracts with vendors.<br>Comisión Nacional de Comunicaciones |
| Cultural and Natural Heritage | Notification of Accidental Discovery of Artifacts Potentially Relevant to Cultural or Natural Heritage of the Province.<br>Request for Liberation of an Area (i.e., free of culturally significant artifacts) | Secretaría de Cultura de la Provincia de Salta |
| Health | Authorization for Installation and Operation for Food Preparation and Operation of Dining Area<br>Authorization to Operate Water Potabilization Plant<br>Potable Water Certificate<br>Medical Post, Doctors and Ambulances<br>Emergency Plan for Contingencies Health Security in Mining Activities | Municipio de Tolar Grande/AOMA (Mining Union) |

| Area | Key permits and Authorizations | Jurisdiction |
|---|---|---|
| Transit and transportation | Transit of special machinery and gasoline | National roads: Vialidad Nacional<br>Provincial roads: Vialidad de la Provincia |
| Use of Soil | Quarry Concession in fiscal provincial areas | Juzgado de Minería de la Provincia de Salta |
| Construction | Approval of structural plans<br>Architectural plans<br>Gas mine plans<br>Electrical plans<br>Certificate of Hygiene and Safety standards in the workplace<br>Safety plans<br>Starting civil construction certificate (Municipal tax)<br>Final construction certificate<br>Fire suppression system certificate | COPAIPA Engineer Chamber of Salta<br><br><br><br><br><br>Municipio de Tolar Grande<br><br>Jefatura de Bomberos (Fire Chief) |
| Water Management Works and Structures | Authorization for construction a water management structure<br>Water management works Approval and authorization to operate<br>Environmental impact report for the water pipes and construction | Secretaría de Recursos Hídricos de la Provincia de Salta<br>Secretaría de Minería |
| Mining Operations | Mining producer registration<br>Ore transport guidelines<br>Notice of start-up of mining activity<br>Notice of suspension of operations or abandonment<br>Habilitation plant and facilities | Secretaría de Minería de Salta<br><br><br><br>Municipio de Tolar Grande |

## 20.4 Closure

### 20.4.1 Legal requirements and other obligations

The mine closure plan will consider the requirements stated by:

- National Constitution, Article 41
- Provincial Constitution, Articles 30, 79, 81, 82
- Mining Code of the Argentine Republic
- Mining Investments Law 24.196
- Water Code of Salta Province, Law 7.017
- General Environmental Law 25.675
- National Law of Environmental Protection 24.585
- Provincial Law of Environmental Protection 7.070
- Provincial Law of Protective Areas 7.107
- Environmental Code, Municipality of San Antonio de los Cobres, Bylaw Articles 005/2008
- Cultural and Archaeological Heritage Law 25.743
- Environmental Impact Assessment (EIA) report, provincial resolution of the Ministry of Economic Development, Mining Secretariat 316/2011

### 20.4.2 Closure management

**Closure considerations**

The Lindero Project will be managed to protect human life and the environment to the fullest possible extent. The objective of the closure plan is to ensure that after mining activities cease, the land and communities are left in a state that allows maximum opportunities for the future.

**Closure plan**

The mine is anticipated to have an operational lifespan of approximately 13 years with closure activities anticipated to start in year 14. The final closure plan will be developed and submitted to the mining authority five years before the mine closure (as per conditions of the granted environmental development permit).

Technical reports will project the mine situation at the time of the closure, including the environmental and cumulative effects on the area. The closure plan will incorporate this information to ensure all impacts that can be anticipated are dealt with. The closure plan must ensure site water quality and the long-term stability of the heaps, waste-rock storage areas or other stock pile storage areas with these objectives being achieved through monitoring. It is envisaged that a period of post-reclamation monitoring will be required until it has been satisfactorily demonstrated by the results of site monitoring that reclamation measures have achieved the required outcomes and are self-sustaining.

Post-reclamation tasks will include:

- Repair of internal roads
- Ensuring the chemical and physical stability of the heap leach area
- Ensuring the physical stability of the waste dumps
- Removal of smaller structures built at ground level

The main objectives of the closure plan are to:

- Comply with the regulatory requirements to which Mansfield has committed regarding the final closure of the Project
- Prevent, minimize, and mitigate adverse environmental impacts
- Leave the site in a condition in which environmental and public safety are protected
- Comply with all social obligations so as to retain the confidence of the stakeholders in the mining sector

**Closure plan phases**

The closure plan proposed by Mansfield includes the following phases:

- Progressive closure, which includes those activities that occur during operation of the mine. This phase includes reclamation activities that take place simultaneously with mine operations, such as waste dump and road reclamation
- Final closure, which includes the reclamation of the land not being used for permanent Project work, such as the internal roads, crushing plant, and camp,

and integration of permanent Project work into the landscape. This phase includes the closure of the heap leach, process plant, and pit areas

- Post-closure, which includes monitoring of the physical and geochemical stability of the reclaimed areas following the progressive and final phases of closure. This phase includes monitoring and if necessary, maintenance activities

During the progressive and final phases of closure, it is necessary to consider the dismantling of equipment and facilities, the demolition of smaller structures built at ground level, the stabilization of waste, and the physical and chemical stability of the site.

### 20.4.3 Reclamation and closure of affected areas

**Facilities and infrastructure**

In addition to the ADR, SART plants and laboratory, the Project will include an administrative office, warehouse and maintenance buildings and accommodation buildings with living facilities.

The camp and the other buildings will be built with materials that allow their dismantling and removal from the mine site. Part of the camp might be kept to be used in future mining activities or possible tourism activity, which would be beneficial to the community.

Utility poles, power lines, and any generators and transformers will be removed from the site. Any perimeter fencing will be dismantled and removed.

Dismantling of plant facilities will include clearing all valuable or usable materials out of the facilities, and then dismantling, disassembling, and disposing of the facilities.

The ADR and SART plants and the adjacent laboratory will be removed once they are no longer needed to support residual leaching and drain-down activities, which will occur in later stages of closure in Year 17.

Fuel storage tanks will be emptied, washed, dismantled, and removed from the site.

For all areas, ground and concrete conditions will be evaluated visually and chemically to determine if areas or components with traces of fuel will require special treatment.

**Open Pit**

All equipment will be removed from the pit area. However, because of its size, the mine pit itself cannot be reclaimed to its original status. In the meantime, rock barriers will be erected to keep people away from dangerous areas.

**Heap leach**

The heap leach system will be designed and built so as to minimize the need for long term active maintenance of the site in the post-closure period. However, closing the heap leach area will still require recovery and recycling of heap-leach solution, chemical stabilization of the heap leach area, landscaping of the area to manage run-off, and road removal.

It will also be necessary to maintain the heap leach area with an adequate slope. It will be smoothed and rounded to prevent water from concentrating which will prevent erosion during an unusual precipitation event. Gutters will be left in place to handle storm runoff.

The management of superficial drain-down will be improved during the closure activities through the creation of stable channels with few or no long-term maintenance requirements.

The major activities surrounding closure of the heap are proposed as follows:

- Year 14 – Residual leaching will continue at a reduced NaCN level, to rinse out remaining recoverable gold while preparing the heap for rinsing
- Year 15 - A period will follow active leaching where rinsing of the heap with low level (residual) cyanide solution occurs. No new cyanide will be added to the leach solution during this time. Raw water addition to the circuit will be maximized to dilute cyanide concentration and the leach solution will be applied with sprinklers to maximize evaporation rates and reduce solution inventory. If necessary evaporators will be used to assist in solution reduction
- Year 16 – The heap will be closed in a sequential manner, consisting of shutting off rinse solution, draining of upper lifts, followed by rounding and smoothing of heap benches
- Year 17 – Reclamation of heap and ponds by constructing evapotranspiration basins in existing ponds to collect and evaporate draindown solution from the heap. These basins will also serve to collect and evaporate future design storm run-off from the heap and ponds areas

Cyanide and metals will be reduced in the heap leach effluents until acceptable standards are met. This reduction will be achieved in the above process through;

- Recirculation of the solution
- Natural mechanisms of cyanide degradation such as UV and air destruction
- Formation of chemical complexes (most cyanide complexes are less toxic than the cyanide)
- Precipitation of insoluble cyanide salts
- Natural biological degradation

If the above mechanisms are not satisfactory in reducing cyanide levels, additional processes such as water treatment and chemical destruction processes can be considered during the closure phase. It is currently assumed these processes will not be necessary.

**Waste dump**

One area will be used as waste dump storage, to the southwest of the pit. Waste rock will be sampled, analyzed using acid-generating static tests, and classified before and during mine operations. Final reclamation of the waste dumps will occur in Year 14 once the pit is mined out and there is no need for waste rock disposal.

**Acid drainage**

Two fundamental conditions must be present for the generation of acid drainage: plentiful rainfall and the presence of abundant sulfide-rich minerals in the host rocks. The mine is located in an extremely arid region with an average rainfall of approximately 50 mm per year. The host geology and mineralization contain very little sulfide-rich minerals (i.e. pyrite) making the probability of acid generation unlikely. Additionally, the host rocks contain minerals such as carbonates, epidotes, etc. that are expected to counteract or buffer against acid generation. Finally, the mine and dumps are located in a closed basin with no external drainage.

**Domestic and industrial waste depot**

The mine will follow a very strict policy during its lifetime regarding the management of waste in order to minimize remediation at closure. This policy will classify waste for recycling or final treatment.

Superficial infrastructure related to domestic and industrial waste will be removed, such as the fence around the depot. Industrial and hazardous waste itself will be periodically removed from the mine during operations to the appropriate facilities, such as Agrotecnica Fueguina S.A. Under the waste management plan, non-recyclable domestic waste, which is projected to be mainly organic, will be duly isolated, backfilled, and landscaped. The need for further treatment for all waste will be evaluated by trained personnel as necessary.

Backfilled areas will be contoured to allow superficial water to flow naturally in case of rain and then covered with a 0.5 m layer of surface material from the area to allow natural re-vegetation. Before backfilling, the material coming from the foundation structure demolition, if there is any, will also be deposited in the waste dump to help protect against erosion. An inspection by both the operating and community authorities will be made before the closure of the waste depots.

**Water wells**

Production water wells will be abandoned in accordance with government regulations or transferred to support an approved post-mining land use. Monitoring wells will be abandoned when the government and company decide that they are no longer needed for long-term monitoring purposes.

**Internal and access roads**

All roads within the Project, including remnants of old access roads, will be reclaimed. The use of the property access road will be restricted during and after the closure. A minimum number of internal access roads will be kept open so that monitoring and inspections of specific areas of the property can be made. These inspection areas will include the pit, waste dumps, water wells, and heap leach and surface water areas.

### 20.4.4 Monitoring during closure

The environmental monitoring plan implemented during the operation stage will continue in force during closure activities. The monitoring plan will be updated as the Project approaches the closure stage, with elements added and deleted as appropriate to the conditions.

### 20.4.5 Post-closure monitoring

Post-closure monitoring has as its main objectives:

- Confirming the long-term physical and chemical stability of reclaimed surfaces such as pit, waste dumps, and heap leach areas
- Monitoring underground and surface water flows
- Evaluating the heap drainage of water content
- Evaluating the achievement of water quality standards in the area affected by the Project

- Monitoring flora and fauna in the area affected by the Project

The post-closure monitoring program, including monitoring, will be carried out with the participation of the community of Tolar Grande. This program will continue in force for approximately five years after the final mine closure.

## 20.5 Closure cost basis

Costs of mine closures and reclamation of mine sites vary considerably due to factors such as location, climate, rainfall, environmental vulnerability, age of the mine, mining method, minerals being mined, waste characteristics, and labor costs. Closure cost estimates should be reviewed regularly to reflect changing circumstances and adjusted according to inflation and work requirements, as well as undergo a thorough reassessment on a predetermined cycle to account for changing community standards and expectations.

The total estimated closure cost for the Project is US$ 35 million. Major closure costs are expected to be incurred on Years 14 and 15. The yearly summary of these costs and credits is presented in Table 20.2.

Note that Year 14 operating costs are split partially between production operating costs and closure costs. It is assumed the heap will still be under active leaching for approximately six months in Year 14, so six months of operating costs are included under production operating costs (in Section 19) and the remaining six months are included as closure costs.

Table 20.2 Closure costs (US$)

| Capital costs | Yr 14 | Yr 15 | Yr 16 | Yr 17 | Total |
|---|---|---|---|---|---|
| Closure plan allowance (Regulatory/Approval) | 15,000 | | | | **15,000** |
| Crushing and stockpiling gravel for ponds | 852,000 | | | | **852,000** |
| Crushing plant dismantling & removal | 990,000 | | | | **990,000** |
| Heap smoothing & re-contouring | | | 2,520,000 | | **2,520,000** |
| ADR chemical remediation allowance | | | | 100,000 | **100,000** |
| ADR chemical testing allowance | | | | 75,000 | **75,000** |
| Gravel for fill of pregnant and excess ponds | | | | 520,000 | **520,000** |
| Gravel for drain channel fill | | | | 192,000 | **192,000** |
| Geotextile for channels and ponds | | | | 265,000 | **265,000** |
| Internal access road reclamation allowance | | | | 50,000 | **50,000** |
| ADR plant dismantling and removal | | | | 230,000 | **230,000** |
| Year 15+post closure monitoring | | | | 100,000 | **100,000** |
| **Total capital costs** | **1,857,000** | **0** | **2,520,000** | **1,532,000** | **5,909,000** |
| | | | | | |
| **Operating costs** | **Yr 14** | **Yr 15** | **Yr 16** | **Yr 17** | **Total** |
| Mine | 0 | 0 | 0 | 0 | **0** |
| Plant | 5,531,000 | 5,368,000 | 163,000 | 163,000 | **11,226,000** |
| General services | 1,929,000 | 1,984,000 | 217,000 | 217,000 | **4,348,000** |
| Administrative services mine | 3,180,000 | 3,188,000 | 100,000 | 100,000 | **6,568,000** |
| Distribution and gold refining costs | 508,000 | 201,000 | 0 | 0 | **709,000** |
| G&A | 2,772,000 | 2,772,000 | 220,000 | 220,000 | **5,984,000** |
| Institutional and community relations | 358,000 | 358,000 | 0 | 0 | **716,000** |
| **Total operating costs** | **14,279,000** | **13,871,000** | **701,000** | **701,000** | **29,551,000** |
| | | | | | |
| **Total closure costs** | **16,136,000** | **13,871,000** | **3,221,000** | **2,233,000** | **35,460,000** |

## 20.6 Comment on Section 20

It is the opinion of the QPs that the appropriate environmental, social and community impact studies have been conducted to date at Lindero. Mansfield have maintained all necessary environmental permits that are prerequisite for the granting of construction permits that will need to be obtained upon completion of detailed engineering designs for the project infrastructure.

During the construction and operation, it is recommended that Mansfield:

- Maintain continuous communication with stakeholders
- Conduct air quality sampling in camp locations and operating facilities, as well as on routes near local villages
- Monitor during the stages of construction, operation, and closure so as to remain aware and keep control of any possible impacts caused by the operations at the mine

During the construction, operation, and closure phases it is recommended that Mansfield:

- Give priority to the changes in the region from which the community will benefit, such as road construction
- Conduct flora and fauna monitoring surveys periodically

- Perform soil monitoring at vulnerable points once a year during the lifetime of the Project and at least two years after its closure. These vulnerable points correspond to areas where liquids, effluents, or solid waste disposal will be handled including the waste dump area; workshop areas; sanitary landfill areas; temporary hazardous waste management areas; process plant area; and heap leach area

# 21 Capital and Operating Costs

## 21.1 Introduction

Capital and operating costs for the Lindero Project were estimated by Fortuna with the assistance of Elbow Creek, Allard Engineering Services and a local engineering firm, Saxum. These costs are based on the design outlined in this study and are considered to have an accuracy of ±15 %. All costs correspond to quotes obtained during the second and third quarters 2017 and expressed in US dollars (US$) at an exchange rate of 17.80 Argentine pesos per US dollar. No escalation factors have been applied to any costs, present or future capital. Table 21.1 summarizes the pre-production capital costs for the Lindero Project, including considerations of value-added tax (IVA).

**Table 21.1 Lindero Project pre-production capital cost estimate**

| Plant Total Direct Costs | Pre-production Total US$ (millions) |
|---|---|
| Total direct costs | $168.99 |
| Total direct and indirect before working capital & IVA | $239.03 |
| Pre-production operating expenses | $9.85 |
| Working capital | $4.06 |
| Capital credit from pre-production gold revenue | ($13.70) |
| Total pre-production capital (excluding IVA) | $239.25 |
| IVA | $42.84 |
| **Total pre-production capital (including IVA)** | **$282.08** |

Sustaining capital for the Project over the LOM includes the Phase 2 leach pad construction in Year 3. A non-sustaining capital expenditure item has been considered in the expansion of the ADR plant and solutions handling in the leach pad area in Year 3. The total capital over the LOM is estimated at US$ 113 million.

Closure and reclamation costs are estimated at US$ 35 million, incurred in Year 13 through Year 17.

The total LOM operating cost for the Lindero Project is US$ 10.32 per tonne of ore processed. Table 21.2 summarizes the average operating costs for the Lindero Project.

**Table 21.2 Lindero Project average operating cost projection**

| Description | Cost (US$/t ore) |
|---|---|
| Mine | 2.47 |
| Plant | 5.52 |
| General Services | 1.20 |
| Administrative Services mine | 0.95 |
| Distribution and gold refining costs | 0.17 |
| **Total** | **10.30** |

## 21.2 Capital costs

The required pre-production capital expenditure for the Lindero Project is summarized in Table 21.3. These costs are based on the design outlined in this study and are considered to have an accuracy of ±15 %. The scope of these costs includes all mining equipment, process facilities, and infrastructure for the Project.

The costs presented have been estimated primarily by Fortuna with input from its Technical Services staff on mine pre-production and mine equipment costs. Saxum provided cost estimates for major and secondary equipment, buildings, infrastructure and major contracts. All equipment and material requirements are based on the design information described in previous sections of this study. Capital cost estimates have been made primarily using budgetary supplier quotes for all major and most minor equipment items, and major construction contract unit rates. Where supplier quotes were not available for minor items, a reasonable cost estimate was made based on supplier quotes in Saxum's project files.

All capital cost estimates are based on the purchase of equipment quoted new from the manufacturer, or estimated to be fabricated new.

Table 21.3 present the capital cost summary by area for the initial capital requirements, including the pre-production mining fleet. No escalation factors have been applied to any present or future capital costs. Where prices of equipment were supplied in Argentine pesos, an average exchange rate of 17.80 pesos per US dollar was used.

## 21.3 Mine capital costs

Mine capital costs include mine development, mining and mobile maintenance support equipment, and other mining costs. The mining capital cost estimate is presented in Table 21.4. All values are rounded to millions of US$.

Capital cost estimates have been made based on supplier's and contractor's quotations, equipment purchasing agents, information provided by Fortuna and estimations based on similar projects. The capital cost estimate includes the following.

### 21.3.1 Drilling

Drilling equipment capital cost is estimated at US$ 3.44 million for initial capital and an additional US$2.56 million for sustaining capital.

### 21.3.2 Loading

Loading equipment capital cost is estimated at US$ 2.95 million for wheel loaders which will begin with two-wheel loaders with one available for the mine development.

### 21.3.3 Trucks

Rear dump trucks with 91 tonne capacity will be used. The mine operation will begin with six trucks, some of which will be available for the mine development. Sustaining capital is estimated to be US$ 6.79 million. A total of US$ 14.13 million has been estimated for the purchase of trucks through the mine life.

Table 21.3 Summary of initial capital cost forecast by area

| Plant Total Direct Costs | Supply (Million US$) | Install (Million US$) | Grand Total (Million US$) |
|---|---|---|---|
| Road infrastructure | - | $2.07 | $2.07 |
| Crushing | $14.25 | $11.86 | $26.10 |
| Agglomeration | $4.98 | $4.06 | $9.04 |
| Leach pad | $1.18 | $26.26 | $27.44 |
| Stacking & Solution handling | $10.92 | $6.81 | $17.74 |
| ADR, Electrowinning & refinery | $6.09 | $7.14 | $13.24 |
| SART plant | $9.68 | $11.99 | $21.66 |
| Laboratories | $1.90 | $0.77 | $2.67 |
| Power distribution | $1.62 | $1.05 | $2.67 |
| Ancillary facilities | $3.61 | $11.12 | $14.73 |
| Water supply & distribution | $5.10 | $4.47 | $9.57 |
| **Process plant total direct costs** | **$59.34** | **$87.59** | **$146.93** |
| Mine total direct costs | | | $22.06 |
| Commissioning | | | $1.50 |
| Spare parts | | | $2.00 |
| Contingency | | | $23.53 |
| **Total direct costs** | | | **$196.02** |
| Indirect costs | | | $8.05 |
| EPCM | | | $16.00 |
| Owners Costs | | | $18.96 |
| **Subtotal before working capital & IVA** | | | **$239.03** |
| Pre-production operating expenses | | | $9.85 |
| Working capital | | | $4.06 |
| Capital credit from pre-production gold revenue (net of charges, taxes and royalties) | | | ($13.70) |
| **Total initial capital (excluding IVA)** | | | **$239.25** |
| IVA | | | $42.84 |
| **Total initial capital (including IVA)** | | | **$282.08** |

### 21.3.4 Support equipment

A total of US$ 4.66 million has been estimated for owner-operated support equipment, such as track dozers, rubber tire dozers, graders, water trucks, a backhoe, pit pumps, and a flatbed truck.

### 21.3.5 Blasting equipment

Equipment, bulk ANFO storage, and blasting accessory magazines will be provided by the supplier through a blasting service contract covering the explosives and accessories supply as well the blasthole loading. Site preparations have been included in the capital estimate.

Bulk handling facilities will be provided by the contractor, which includes mobilization and pre-paid demobilization.

### 21.3.6 Other mine capital

Initial other capital is estimated at US$ 1.01 million with another US$ 0.71 million estimated for sustaining capital. Other mine capital includes costs for light vehicles, light towers, mobile radios, water storage, GPS base station and surveying systems as well as other miscellaneous equipment.

All building and infrastructure capital was estimated by Saxum, reviewed by Fortuna and included in the capital cost sections. These items include shop equipment, fuel facilities, and shop and office buildings.

Mine capital costs include pre-stripping, mine equipment, mine maintenance, and other mining costs. Total pre-production mining capital was estimated to be US$ 19.40 million and the total LOM mining capital was estimated to be US$ 60.99 million. The mining capital cost estimate is presented in Table 21.4.

Mining capital costs were estimated by Fortuna with contributions from Saxum. Saxum's contributions include:

- Additional freight charges for transport of equipment from the port of Buenos Aires to the mine site
- Customs charges on imported equipment
- IVA taxes added for goods and services
- Cost for earthworks

**Table 21.4 Mine capital cost forecast (US$ millions)**

| Equipment | FY0 | FY1 | FY2 | FY3 | FY4 | FY5 | FY6 | FY7 | FY8 | FY9 | FY10 | FY11 | FY12 | FY13 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Truck Komatsu HD 785 (Proposed) | 7.33 | | 1.36 | 1.36 | 1.36 | | 1.36 | | | | 1.36 | | | | **14.13** |
| Loader WA900 (Proposed) | 2.95 | | | 1.64 | | | | | | | | | | | **4.59** |
| Drill Rig DML SP@1900(5¨-97/8¨) | 3.44 | | | | 1.72 | | | | | | | | | | **5.17** |
| Drill Rig 5¨ | | | | | 0.84 | | | | | | | | | | **0.84** |
| Dozer D275 (KOM) | 1.39 | | | 0.77 | | | | | | | | | | | **2.16** |
| Water Truck 10,000 gal | 0.43 | | | | | | | | | | | | | | **0.43** |
| Motograder GD705 (KOM) | 0.66 | | | | | | | | | | | | | | **0.66** |
| Rock Breaker-Impact Hammer | 0.02 | | | | | | 0.02 | | | | | | | | **0.05** |
| Excavator PC300LC-8 (KOM) | 0.34 | | | | | | | | | | | | | | **0.34** |
| Lube Truck | 0.40 | | | | | | | | | | | | | | **0.40** |
| Fuel Truck | 0.30 | | | | | | | | | | | | | | **0.30** |
| Mechanics Truck | 0.36 | | | | | | | | | | | | | | **0.36** |
| Forklift | | | 0.03 | | | | | | | | | | | | **0.03** |
| Tire Handler (WA500) | 0.76 | | | | | | | | | | | | | | **0.76** |
| Light Plant | 0.10 | | | | | | | | | | | | | | **0.1** |
| Engineering & Office Equipment | 0.24 | | | | | | | | | | | | | | **0.24** |
| Base Ratio & GPS Stations | 0.17 | | | | | | | | | | | | | | **0.17** |
| Pipeline | | | 0.03 | | | | | | | | | | | | **0.03** |
| Liner | | | 0.03 | | | | | | | | | | | | **0.03** |
| Survey Equipment | 0.13 | | | | | | | | | | | | | | **0.13** |
| Light Vehicle (pick up) | 0.28 | | | | | | | | | | | | | | **0.28** |
| Passenger Van | 0.09 | | | | | | | | | | | | | | **0.09** |
| **Total Capex (M USD)** | **19.40** | | **1.45** | **3.77** | **3.94** | **0.30** | **1.40** | | **0.09** | **0.28** | **1.36** | | | | **31.98** |
| **Total Sustaining CAPEX (M USD)** | | | **0.06** | **3.87** | **1.52** | **1.90** | **3.77** | **3.18** | **2.77** | **3.71** | **3.25** | **2.54** | **1.76** | **0.67** | **29.02** |
| **CAPEX + Sustaining CAPEX (M USD)** | **19.40** | | **1.51** | **7.64** | **5.46** | **2.19** | **5.17** | **3.18** | **2.86** | **3.99** | **4.61** | **2.54** | **1.76** | **0.67** | **60.99** |

## 21.4 Process capital costs

Process costs have been estimated by Fortuna with inputs from Allard Engineering, Elbow Creek and Saxum. Capital cost estimates have been made primarily using budgetary supplier quotes for all major and most minor equipment items. Where supplier quotes were not available for minor items, a reasonable cost estimate was made based on costs from Saxum's project files.

### 21.4.1 Process cost basis

Each area in the process cost buildup, including crushing, agglomerating, stacking, heap leach and solution handling, recovery plant, SART plant, etc. in the capital cost table is separated into the following disciplines, where applicable:

- Civil (including earthworks and concrete)
- Structural steel
- Mechanical
- Piping
- Electrical
- Instrumentation
- Equipment (mechanical, platework, electrical and instruments)

Supply, freight, customs fees and duties, installation costs, spare parts (where applicable), and contingencies are included as part of the direct costs for each discipline. IVA is also included in the capital cost buildup for each discipline. These cost areas are discussed further in the following sub-sections.

Engineering, procurement, and construction management (EPCM) indirect costs are added to the total direct costs.

### 21.4.2 Freight

Where available and applicable, supplier-quoted freight cost estimates for equipment packages are used. Alternatively, freight costs for certain major equipment (e.g. HPGR) have been estimated based on a packing list, applying costs Saxum and Fortuna have obtained from freight forwarders.

### 21.4.3 Duties and customs fees

An import tax is levied on all imported items at 1.0 % of the equipment supply cost. Additional charges of 0.5 % of the equipment supply apply for customs broker fees for each import declaration.

### 21.4.4 Value-added tax (IVA)

IVA is not considered as a capital cost of the project but will represent a cash outlay during the construction phase. The total IVA associated with the initial capital expenditure of

US$ 239 million is estimated to be US$ 43 million for a total funding requirement of US$ 282 million. There are two IVA rates:

- For equipment and machinery, and certain hard assets of the Project such as steel structures and buildings, a rate of 10.5 % is applied
- For freight and all other supplies and services, a rate of 21.0 % applies

IVA will be refunded in full to the Project once certain obligations are met and information supplied to the Argentine taxation authorities.

### 21.4.5 Major earthworks

Major earthwork quantities were estimated based on the preliminary site, facilities, and heap leach pad design. This includes earthworks for providing level areas for the heap pad and ponds, building and process areas, and interconnecting roads. Also included here are detailed earthworks and lining costs for the leach pad and ponds, diversion channels, and retaining walls for the crushing plant.

All major earthworks volumes have been estimated from basic engineering drawings and general arrangements and unit rates have been developed from local contractors with significant experience working at altitude in the Puna.

Anddes Asociados S.A.C. (Anddes) provided earthwork quantities for the heap leach pad and ponds. Saxum developed earthwork quantities for all other areas of the Project site.

### 21.4.6 Civils

Civils include detailed earthworks and concrete. Concrete quantities have been estimated from takeoffs based on quantities from basic engineering design drawings developed by Saxum, previous similar equipment installations, calculations, or from estimates.

Supply and installation rates for concrete are based on quotations from local contractors. The supply rate of concrete includes importation of aggregate from contractor recommended borrow sources, and supply of all necessary equipment (i.e. batch plant) to produce concrete on-site. The installation rate includes all equipment and labor for formwork, placement, and finishing activities.

### 21.4.7 Structural steel

Structural steel requirements for the various areas were developed from takeoff lists from basic engineering design drawings developed by Saxum, similar project installations, and from available general arrangement drawings. Unit costs for steel, including installation labor and equipment requirements, were developed from local contractor quotes.

### 21.4.8 Mechanical equipment

Costs for all major items of new equipment are based on budgetary quotes from vendors. Costs for minor equipment items are based on supplier quotes, or from Saxum's in-house database. Where possible, quotes from Argentine vendors were obtained.

The platework discipline includes the supply and installation of steel tanks, bins, and chutes. All crushing area and stacking area platework was developed from basic engineering drawings provided by Saxum. Costs were developed from material takeoffs from these drawings, along with local contractor bid estimates.

Installation estimates are based on equipment type and cost and include installation labor and equipment usage. Where available, vendor estimated installation costs are used.

In 2013, Goldrock purchased the HPGR and this equipment is currently being stored within Argentina. The cost of the HPGR is considered a sunk cost and is excluded from the capital cost totals. Note that the equipment has not been officially imported (the equipment is currently in storage in a "Zona Franca") and so import tax and IVA are due once imported (included in pre-production capital). Freight costs to site are also included for the HPGR.

### 21.4.9 Piping and valves

Piping, fittings, and valve costs for major areas, including the heap leach irrigation and drainage, process and fire water distribution, and the water pipeline are based on material take offs, and vendor budget quotes for supply of major items. Saxum provided material takeoffs for all water distribution piping. Anddes prepared material takeoffs for heap leach irrigation, as well as for all heap leach pad and pond drainage piping. Allard Engineering and Elbow Creek provided material takeoffs for all process piping for the agglomeration, ADR, refinery, reagent and SART plant. Saxum obtained estimates for all piping based on the developed material lists. Piping installation costs are based on unit rates from a combination of contractor quotes and estimates from similar installations. Saxum applied the unit rate costs to the material take-off quantities.

Piping, fittings, and valve costs for other areas, and miscellaneous small piping (less than 75 mm diameter), have been estimated on a percentage basis of the mechanical equipment costs.

### 21.4.10 Electrical

Site power distribution (medium voltage), low and medium voltage transformers, motor control centers (MCCs), low-voltage distribution panels, and electrical room costs were developed by Saxum based on Saxum's basic engineering design, local vendor costs for electrical equipment and cabling supply, and local contractor costs for installation.

Major electrical equipment selection was based on Saxum's basic engineering development of site distribution and area single-line diagrams, from Saxum's major mechanical equipment list and the general site layout. Preliminary MCC and distribution panel layouts and designs were based on single-lines and local availability of equipment.

### 21.4.11 Instrumentation

Instrumentation costs were developed by Saxum based on Saxum's basic engineering design, local vendor costs for instrumentation equipment, and local contractor costs for installation.

### 21.4.12 Infrastructure capital costs

**Buildings**

Costs for the camp are based on contractor quotes for modular, quick-install type buildings. Freight and install for these modular buildings are estimated from contractor inputs and local freight companies.

The kitchen/dining building, administration building, truck shop, warehouses, process shops, ADR building, SART plant building, refinery and laboratories costs are based on local contractor prices for supply and installation of the building shell, and a combination of contractor prices and Saxum material takeoff and cost estimates for supply and installation of architectural and interior finishes.

**Access roads**

The main site access road from Salta to the Property becomes narrow and windy in the areas between Pocitos, Tolar Grande, and Lindero and will require some upgrading. Road improvements include re-grading some sections of existing road, widening sections, creating embankments and culverts. In some areas the road improvements will require significant cut and fill. The total cost of the improvements along the stretch between Pocitos and the Lindero site is estimated at US$ 1.49 million.

**Power generation plant**

Power for the Lindero Project is to be supplied by diesel generators. The generation plant and all of its ancillary equipment and facilities will be supplied, installed, operated and maintained under a "Take-and-Pay" contract by a local contractor who specializes in energy supply contracts. Therefore, the cost of the generation plant is incurred as an operating cost, described further under the operating cost subsection.

**Raw water supply**

A significant amount of the site water requirement will be met by wells located approximately 13 km to the east of the site. The majority of the water supply capital cost will be in constructing a 200 mm (8 inch) pipeline and two booster pumping stations to transfer water from these wells to the site. Costs for the water pipeline system are based on local contractors who are experienced in designing and constructing pipelines, and these costs are distributed throughout the various cost disciplines.

Additional costs are included for a third well, also located in the Chachas area. These drilling costs were obtained through Fortuna from a local vendor who has provided well drilling services for the existing Lindero wells.

**Camp facilities**

Capital costs for the camp are based on general arrangements developed by Saxum, and local contractor costs for supply and installation. The camp buildings consist of pre-manufactured modular units for five 64-man operator dormitories, and one 32-man supervisor dormitories housing a total of 352 persons, and laundry room facilities.

**Other general infrastructure costs**

Fencing for the reagent storage area, warehouse, explosives magazine, electrical substations, has been estimated from Saxum general arrangements, local vendor quotes for supply, and local contractor installation rates.

- A total of 1,211 m of fencing is estimated at a cost of US$ 95,000
- A cost of US$ 549,000 is included for site communications (internet, phone, radio) based on local vendor quotes
- A cost of US$ 594,000 is included for the sewage treatment facilities based on local vendor supply quotes
- A cost of US$ 142,000 is included for the containerized potable water treatment system based on local vendor supply quotes
- A cost of US$ 1,438,000 is included for complete firefighting system of the entire mining and process plant, and for the buildings

### 21.4.13 Indirect costs

Indirect costs include contractor's costs for items such as temporary construction facilities, camp operation during construction, power genset rental and fuel during construction, construction warehouse and fenced yards, support equipment and security. These costs have been estimated based on contractor/vendor quotations and from Saxum and Fortuna estimates.

### 21.4.14 Spare parts

For certain major equipment such as the HPGR, spare parts costs were obtained directly from vendor quotes. Where quotes for spares were not available, spares were estimated as a percentage of the mechanical equipment costs. For most equipment a percentage of 3 % was applied to the equipment supply cost. Freight and IVA were added to the estimated spare parts supply costs.

### 21.4.15 Engineering, procurement and construction management

The estimated cost for engineering, procurement and construction management (EPCM) for the development, construction, and commissioning was based on estimates from Saxum and local firms.

The EPCM costs cover services and expenses for the following areas:

- Project management
- Process engineering, international procurement assistance, technical oversight of detailed engineering, construction management, and commissioning
- Detailed engineering
- Heap leach pad and ponds
- Crushing plant, site utilities, infrastructure and process plant
- Procurement
- Construction management

### 21.4.16 Contingency

A contingency is applied at a rate of 12 % for all elements (supply and installation costs). In both cases the contingency is applied to the direct costs, and the overall average project cost contingency is US$ 23.5 million.

### 21.4.17 Life-of-mine capital

LOM capital expenditures include additional costs for all plant additions constructed after the initial construction period. The costs for the LOM capital are structured the same way as costs for the pre-production period.

The major expansion projects planned for Lindero in the future are:

- Expansion of the heap leach pad to Phase 2 in Year 3
- Expansion of the ADR plant in Year 3
- Reclamation and closure activities in Years 13 to 17

Table 21.5 summarizes the future capital estimates for the Lindero Project.

**Table 21.5 Life-of-mine capital summary projections**

| Project | Total US$ (million) |
|---|---:|
| Leach pad second phase | $28.50 |
| ADR expansion | $8.00 |
| Other | $0.27 |
| **Process plant total direct costs** | **$36.78** |
| Mine equipment | $41.60 |
| **Total expansion & sustaining capital** | **$78.38** |
| Total closure costs | $35.19 |
| **Total LOM capital** | **$113.57** |

### 21.4.18 Reclamation and closure costs

The plan and design for Project closure is detailed in Section 20.4 and 20.5. Costs were developed according to this plan and the capital costs summarized in the future capital summary shown in Table 21.5.

The major components to the closure capital costs are the earthworks involved in smoothing out the ultimate heap, lining of the channels and ponds with gravel, and demolition and transport of the structures. Operating costs are also associated with the owner's labor involved in closure activities and are briefly discussed with the Project operating costs.

### 21.4.19 Owner’s costs

Owner's costs are included which essentially cover pre-production G&A fixed and variable costs, as well as G&A labor.

Pre-production G&A fixed costs include office operating expenses, legal fees, phones/internet, office supplies, insurance, IT services and computers, travel, community assistance, government affairs and PR, environmental permits and licensure, and property taxes. Pre-production variable costs include camp accommodations and transportation, environmental and waste management, equipment rentals, light vehicle usage, safety supplies, general consulting, employee recruiting and training, medical suppliers, and medical and security contracts/services.

Owner's costs were developed by Fortuna. The total cost is US$ 21 million.

#### 21.4.20 Working capital and pre-production operating expenses

Working capital in general is required to sustain operations before any gold revenue is realized, and is based on a period of three months to cover total operating costs for the mine, process, and G&A.

Pre-production operating expenses includes capitalized operating costs in the pre-production period of three months right before the start of Year 1. This pre- production period consists of a general start-up adaptation to the process. Pre-production operating expenses are approximately US$ 9.8 million.

During the pre-production period, before the start of Year 1, approximately 12,761 ounces of gold is forecast to be produced and sold. Total revenue net of taxes anticipated from gold sales during this period is approximately US$ 13.7 million.

Working capital includes a 30-day reserve (at full production operating costs) for salaries, services and non-imported materials during Month 1 when the Project transitions to becoming cash flow positive. Working capital also allows for a 45-day reserve for imported materials, such as cyanide. Working capital is approximately US$ 4.1 million.

#### 21.4.21 Exclusions

The following capital costs are either excluded or else not considered within Saxum's scope of supply and estimate:

- Finance charges and interest during construction
- Escalation costs
- Currency exchange fluctuations

## 21.5 Operating costs

LOM operating costs for the Lindero Project is based on the information presented in earlier sections of this study. Mine operating costs are estimated to average US$ 2.47/t ore processed. Estimated LOM operating costs for the process is estimated to be US$ 5.52/t ore processed. Mine G&A costs are estimated to be US$ 2.15/t ore processed. Distribution and refining costs are estimated to be US$ 0.17/t ore processed. Total Project average operating cost is estimated to be US$ 10.30/t ore processed and is summarized in Table 21.2 and presented in Section 21.0.

Operating costs for all areas of the Project have been estimated from first principles. Labor costs are estimated using project-specific staffing, salary, wage, and benefit requirements. Unit consumptions of materials, supplies, power, water, and delivered supply costs are also estimated. Table 21.6 summarizes the process operating costs on a yearly schedule.

Table 21.6 Operating cost forecast by year (million US$)

| Parameter | FY1 | FY2 | FY3 | FY4 | FY5 | FY6 | FY7 | FY8 | FY9 | FY10 | FY11 | FY12 | FY13 | Total (Year 1-13) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ore Processed (Mt) | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.56 | 6.64 | **87.45** |
| **Annual Cost (M USD)** | | | | | | | | | | | | | | |
| Plant | 14.44 | 15.74 | 15.36 | 16.34 | 16.31 | 16.83 | 17.24 | 17.93 | 17.26 | 18.60 | 16.78 | 17.57 | 15.23 | **215.63** |
| Plant Supervision | 38.19 | 40.92 | 38.85 | 39.25 | 38.00 | 37.22 | 36.14 | 36.68 | 36.00 | 35.61 | 35.34 | 35.33 | 35.06 | **482.60** |
| Crushing | 7.92 | 7.94 | 8.06 | 8.09 | 8.07 | 8.07 | 8.10 | 8.10 | 8.00 | 7.96 | 8.13 | 8.21 | 8.05 | **104.68** |
| Agglomeration | 6.33 | 6.34 | 6.34 | 6.34 | 6.36 | 6.38 | 6.41 | 6.41 | 6.41 | 6.41 | 6.41 | 6.41 | 6.36 | **82.90** |
| Leach Pad | 1.38 | 1.44 | 1.29 | 1.25 | 1.13 | 1.11 | 1.13 | 1.08 | 1.07 | 1.06 | 1.05 | 1.06 | 1.03 | **15.07** |
| **Total Annual Cost (M USD)** | **68.27** | **72.38** | **69.89** | **71.27** | **69.87** | **69.62** | **69.02** | **70.20** | **68.73** | **69.64** | **67.70** | **68.58** | **65.73** | **900.88** |
| **Ore Processed (USD/t)** | | | | | | | | | | | | | | |
| Plant | 2.14 | 2.33 | 2.28 | 2.42 | 2.42 | 2.49 | 2.55 | 2.66 | 2.56 | 2.76 | 2.49 | 2.68 | 2.29 | **2.47** |
| Plant Supervision | 5.66 | 6.06 | 5.76 | 5.81 | 5.63 | 5.51 | 5.35 | 5.43 | 5.33 | 5.28 | 5.23 | 5.39 | 5.28 | **5.52** |
| Crushing | 1.17 | 1.18 | 1.19 | 1.20 | 1.19 | 1.20 | 1.20 | 1.20 | 1.18 | 1.18 | 1.20 | 1.25 | 1.21 | **1.20** |
| Agglomeration | 0.94 | 0.94 | 0.94 | 0.94 | 0.94 | 0.95 | 0.95 | 0.95 | 0.95 | 0.95 | 0.95 | 0.98 | 0.96 | **0.95** |
| Leach Pad | 0.20 | 0.21 | 0.19 | 0.19 | 0.17 | 0.16 | 0.17 | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | 0.15 | **0.17** |
| **Total Cost (USD/t)** | **10.11** | **10.72** | **10.35** | **10.56** | **10.35** | **10.40** | **10.23** | **10.4** | **10.18** | **10.32** | **10.03** | **10.45** | **9.90** | **10.30** |

### 21.5.1 Basis of operating cost estimate

The operating costs presented are based upon ownership of all project production equipment and site facilities, as well as the owner employing and directing all operating, maintenance, and support personnel, with the exception of the power plant which is contracted for the duration of the Project life.

The operating costs have been estimated and are presented without any added contingency allowances. The mine, processing, support and general and administrative operating costs are considered to have an accuracy range of ± 15 %.

- Operating costs estimates have been based upon information obtained from the following sources
- Technical Services owner mining costs
- Project metallurgical testwork and process engineering
- Budgetary quotations from potential suppliers of project operating and maintenance supplies and materials
- Recent Saxum project file data
- Experience of Saxum staff with other similar operations

Where specific data do not exist, cost allowances have been based upon consumption and operating requirements from other similar properties for which reliable data exists. Freight costs have been estimated from a combination of freight forwarder quotes and similar project files, where delivered prices were not available.

All costs are presented in second and third quarter 2017 US dollars (US$). For labor wages which were supplied in Argentine pesos, an exchange rate of 17.80 pesos per US dollar was used.

It is assumed that the Lindero Project will become eligible for a rebate on the IVA applied to any and all operating costs; therefore, IVA has not been included in any of the operating cost estimates.

### 21.5.2 Labor and wages

Staffing will be by Argentine nationals, wherever practicable. A limited number of key supervisory positions will be held by expatriates, at least for the start-up of the mine.

Labor rates for hourly and staff employees were provided by Fortuna. The average wage burden rate (approximately 31.6 % over gross salary rate) was established to include the requisite statutory Argentine social and medical insurance and pension, and bonus costs.

### 21.5.3 Mine operating costs

Mining operating costs were estimated by Fortuna. It is expected that the bulk of the deposit will be amenable to mining using standard drill, blast, load, and haul open pit mining methods.

Table 21.7 summarizes the annual mine operating costs in terms of US$/tonne mined and also in terms of US$/tonne moved which includes the re-handling of stockpiles. The upper portion of Table 21.7 shows the tonnage mined and the stockpile re-handling.

Costs are provided based exclusively on the mining operation process: supervision, drilling, blasting, loading, hauling, and ancillary services. The total average mining cost is estimated to be US$ 1.09/t mined and US$ 1.05/t moved.

### Table 21.7 Annual mine operating cost forecast

| Parameter | Units | FY1 | FY2 | FY3 | FY4 | FY5 | FY6 | FY7 | FY8 | FY9 | FY10 | FY11 | FY12 | FY13 | Total (Year 1-13) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mining - Ore | kt | 8,695 | 9,008 | 7,170 | 6,168 | 6,175 | 4,413 | 6,774 | 7,079 | 5,360 | 7,637 | 6,082 | 6,586 | 6,304 | **87,451** |
| Mining - Waste | kt | 6,805 | 7,290 | 8,211 | 10,308 | 10,200 | 10,409 | 8,805 | 8,481 | 10,671 | 9,932 | 7,204 | 8,636 | 2,845 | **109,798** |
| **Mined** | kt | 15,500 | 16,298 | 15,381 | 16,476 | 16,375 | 14,822 | 15,580 | 15,560 | 16,081 | 17,570 | 13,286 | 15,222 | 9,149 | **197,249** |
| Mining - Rehandle | kt | | 302 | 119 | 582 | 716 | 2,337 | | 1 | 1,482 | 300 | 979 | 500 | 486 | **7,804** |
| **Mining - Moved** | **kt** | **15,500** | **16,600** | **15,500** | **17,058** | **17,091** | **17,159** | **15,580** | **15,561** | **17,513** | **17,870** | **14,265** | **15,722** | **9,635** | **205,053** |
| **Mining Cost Summary** | | | | | | | | | | | | | | | |
| Mine Supervision | kUS$ | 791 | 791 | 791 | 791 | 791 | 791 | 791 | 791 | 791 | 791 | 791 | 791 | 791 | **10,285** |
| Drill | kUS$ | 2,509 | 2,506 | 2,446 | 2,549 | 2,566 | 2,524 | 2,477 | 2,746 | 2,648 | 3,141 | 2,470 | 2,645 | 2,586 | **33,815** |
| Blast | kUS$ | 5,599 | 5,600 | 5,505 | 5,613 | 5,555 | 5,399 | 5,498 | 5,557 | 5,491 | 5,873 | 5,276 | 5,484 | 4,849 | **71,301** |
| Load | kUS$ | 1,122 | 1,467 | 1,263 | 1,571 | 1,571 | 1,415 | 1,425 | 1,342 | 1,532 | 1,626 | 1,257 | 1,386 | 1,044 | **18,020** |
| Haul | kUS$ | 3,060 | 3,832 | 3,818 | 4,227 | 4,361 | 5,158 | 5,504 | 5,917 | 5,488 | 5,777 | 5,506 | 5,773 | 4,489 | **62,909** |
| Ancillary services | kUS$ | 1,362 | 1,541 | 1,536 | 1,592 | 1,466 | 1,547 | 1,548 | 1,578 | 1,309 | 1,388 | 1,477 | 1,487 | 1,469 | **19,301** |
| **Total Mining Cost** | **kUS$** | **14,444** | **15,738** | **15,360** | **16,344** | **16,310** | **16,835** | **17,244** | **17,931** | **17,258** | **18,597** | **16,777** | **17,566** | **15,227** | **215,632** |
| **Mine Cost per tonne mined** | | | | | | | | | | | | | | | |
| Mine Supervision | US$/t | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.06 | 0.05 | 0.09 | **0.05** |
| Drill | US$/t | 0.16 | 0.15 | 0.16 | 0.15 | 0.16 | 0.17 | 0.16 | 0.18 | 0.17 | 0.18 | 0.19 | 0.17 | 0.28 | **0.17** |
| Blast | US$/t | 0.36 | 0.34 | 0.36 | 0.34 | 0.34 | 0.36 | 0.35 | 0.36 | 0.34 | 0.33 | 0.40 | 0.36 | 0.53 | **0.36** |
| Load | US$/t | 0.07 | 0.09 | 0.08 | 0.10 | 0.10 | 0.10 | 0.09 | 0.09 | 0.10 | 0.09 | 0.09 | 0.09 | 0.11 | **0.09** |
| Haul | US$/t | 0.20 | 0.24 | 0.25 | 0.26 | 0.27 | 0.35 | 0.35 | 0.38 | 0.34 | 0.33 | 0.41 | 0.38 | 0.49 | **0.32** |
| Ancillary services | US$/t | 0.09 | 0.09 | 0.10 | 0.10 | 0.09 | 0.10 | 0.10 | 0.10 | 0.08 | 0.08 | 0.11 | 0.10 | 0.16 | **0.10** |
| **Total Mining Cost mined** | **US$/t** | **0.93** | **0.97** | **1.00** | **0.99** | **1.00** | **1.14** | **1.11** | **1.15** | **1.08** | **1.06** | **1.26** | **1.15** | **1.66** | **1.09** |
| **Mine Cost per tonne moved** | | | | | | | | | | | | | | | |
| Mine Supervision | US$/t | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.04 | 0.06 | 0.05 | 0.08 | **0.05** |
| Drill | US$/t | 0.16 | 0.15 | 0.16 | 0.15 | 0.15 | 0.15 | 0.16 | 0.18 | 0.15 | 0.18 | 0.17 | 0.17 | 0.27 | **0.16** |
| Blast | US$/t | 0.36 | 0.34 | 0.36 | 0.33 | 0.33 | 0.31 | 0.35 | 0.36 | 0.31 | 0.33 | 0.37 | 0.35 | 0.50 | **0.35** |
| Load | US$/t | 0.07 | 0.09 | 0.08 | 0.09 | 0.09 | 0.08 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.11 | **0.09** |
| Haul | US$/t | 0.20 | 0.23 | 0.25 | 0.25 | 0.26 | 0.30 | 0.35 | 0.38 | 0.31 | 0.32 | 0.39 | 0.37 | 0.47 | **0.31** |
| Ancillary services | US$/t | 0.09 | 0.09 | 0.10 | 0.09 | 0.09 | 0.09 | 0.10 | 0.10 | 0.07 | 0.08 | 0.10 | 0.09 | 0.15 | **0.09** |
| **Total Mining Cost moved** | **US$/t** | **0.93** | **0.95** | **0.99** | **0.96** | **0.95** | **0.98** | **1.11** | **1.15** | **0.99** | **1.04** | **1.18** | **1.12** | **1.58** | **1.05** |

### 21.5.4 Owner mining costs

Mining cost estimates have been made based on supplier's and contractor's quotations, equipment purchasing agents, information provided by Fortuna and estimations based on similar projects. Mining costs are built up by cost areas of drilling, blasting, loading, haulage, mine support, maintenance, and mine general services.

**Drilling costs**

The average LOM drilling cost is estimated to be US$ 0.17/t mined. This includes maintenance labor and parts allocated to drill maintenance. Drilling costs are estimated assuming that all mining is done on eight-meter benches. A penetration rate is estimated to be 20.2 meters per hour based on drill specifications and rock strength.

**Blasting costs**

The average LOM blasting cost is estimated to be US$ 0.36/t mined. Blasting costs are based on contract proposals to supply personnel, equipment, bulk explosives and accessories to the site as well as storage and delivery of the products to the blasthole for each blast pattern.

**Loading costs**

The average LOM loading cost is estimated to be US$ 0.09/t moved, which includes the re-handle of ore from stockpiles and maintenance labor and parts allocated to loaders maintenance.

**Haulage costs**

The average LOM haulage cost is estimated to be US$ 0.31/t moved. The cost per tonne moved includes re-handle of stockpiled ore and maintenance labor and parts allocated to truck maintenance.

**Ancillary service costs**

Ancillary services costs include the operation and maintenance of all mine-support equipment for haul roads, stockpiles, waste dumps and mine slopes maintenance. The average LOM support cost is estimated to be US$ 0.09/t moved.

### 21.5.5 Mine personnel and staffing

Mine personnel estimates include operating and mine-staff personnel, for both contract and owner mining. Operating personnel are estimated as the number of people required to operate drills, trucks, loading, and support equipment to achieve the production schedule as well as those persons required for blasting. Mine staff is based on the people required for supervision and support of mine production.

The estimated number of mine personnel required to execute the mine plan is discussed in Section 16, along with the specific personnel costs.

### 21.5.6 Process plant operating costs

Processing plant operating costs have been estimated based upon unit costs and consumption, and, where possible, have been broken down by area: Plant supervision, crushing, agglomeration, leach pad, SART plant, ADR plant, refinery, ancillary services and power.

The average life of mine operating cost for the Lindero Project is US$ 5.52 per tonne of ore. Table 21.8 details the average process operating costs on a yearly schedule.

### Table 21.8 Annual processing plant operating cost forecast

| Parameter | Units | FY1 | FY2 | FY3 | FY4 | FY5 | FY6 | FY7 | FY8 | FY9 | FY10 | FY11 | FY12 | FY13 | Total (Year 1-13) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Consumable | M US$ | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | **7.28** |
| Sodium Cyanide | M US$ | 2.30 | 4.25 | 4.25 | 4.25 | 4.25 | 4.25 | 4.25 | 4.25 | 4.25 | 4.25 | 4.25 | 4.25 | 4.25 | **53.34** |
| Cement | M US$ | 5.02 | 4.92 | 2.96 | 4.75 | 3.56 | 2.80 | 2.26 | 2.45 | 1.81 | 1.56 | 1.37 | 1.32 | 1.30 | **36.09** |
| Sulfuric acid | M US$ | 10.77 | 10.94 | 10.87 | 10.70 | 10.60 | 10.55 | 10.29 | 10.53 | 10.48 | 10.28 | 10.38 | 10.43 | 10.26 | **137.07** |
| Lime | M US$ | 6.15 | 6.54 | 6.54 | 5.31 | 5.31 | 5.31 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | **70.20** |
| Sodium hydrosulfide | M US$ | 1.33 | 1.50 | 1.45 | 1.42 | 1.41 | 1.40 | 1.39 | 1.42 | 1.39 | 1.41 | 1.42 | 1.41 | 1.41 | **18.35** |
| Filter aid | M US$ | 0.18 | 0.23 | 0.21 | 0.20 | 0.19 | 0.18 | 0.18 | 0.19 | 0.18 | 0.18 | 0.18 | 0.19 | 0.18 | **2.47** |
| Fluxes | M US$ | 0.62 | 0.62 | 0.62 | 0.62 | 0.62 | 0.62 | 0.62 | 0.62 | 0.62 | 0.62 | 0.62 | 0.62 | 0.55 | **7.98** |
| Sodium hydroxide | M US$ | 11.25 | 11.35 | 11.40 | 11.45 | 11.50 | 11.54 | 11.59 | 11.64 | 11.69 | 11.74 | 11.55 | 11.55 | 11.55 | **149.81** |
| **Total Process Costs** | **M US$** | **38.19** | **40.92** | **38.85** | **39.25** | **38.00** | **37.22** | **36.14** | **36.68** | **36.00** | **35.61** | **35.34** | **35.33** | **35.06** | **482.60** |
| **Processing Plant** | | | | | | | | | | | | | | | |
| Consumable | US$/t | 0.08 | 0.08 | 0.08 | 0.08 | 0.08 | 0.08 | 0.08 | 0.08 | 0.08 | 0.08 | 0.08 | 0.09 | 0.08 | **0.08** |
| Sodium Cyanide | US$/t | 0.34 | 0.63 | 0.63 | 0.63 | 0.63 | 0.63 | 0.63 | 0.63 | 0.63 | 0.63 | 0.63 | 0.65 | 0.64 | **0.61** |
| Cement | US$/t | 0.74 | 0.73 | 0.44 | 0.70 | 0.53 | 0.41 | 0.33 | 0.36 | 0.27 | 0.23 | 0.20 | 0.20 | 0.20 | **0.41** |
| Sulfuric acid | US$/t | 1.60 | 1.62 | 1.61 | 1.59 | 1.57 | 1.56 | 1.52 | 1.56 | 1.55 | 1.52 | 1.54 | 1.59 | 1.55 | **1.57** |
| Lime | US$/t | 0.91 | 0.97 | 0.97 | 0.79 | 0.79 | 0.79 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.76 | 0.75 | **0.80** |
| Sodium hydrosulfide | US$/t | 0.20 | 0.22 | 0.21 | 0.21 | 0.21 | 0.21 | 0.21 | 0.21 | 0.21 | 0.21 | 0.21 | 0.21 | 0.21 | **0.21** |
| Filter aid | US$/t | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | **0.03** |
| Fluxes | US$/t | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.08 | **0.09** |
| Sodium hydroxide | US$/t | 1.67 | 1.68 | 1.69 | 1.70 | 1.70 | 1.71 | 1.72 | 1.72 | 1.73 | 1.74 | 1.71 | 1.76 | 1.74 | **1.71** |
| **Total Process Costs** | **US$/t** | **5.66** | **6.06** | **5.76** | **5.81** | **5.63** | **5.51** | **5.35** | **5.43** | **5.33** | **5.28** | **5.24** | **5.39** | **5.28** | 5.52 |

### 21.5.7 Power

Electrical power for the process will be produced at site using fuel oil generators, under contract with a local firm who specializes in contract power supply, and will supply and install the power plant and its facilities. The contractor will also supply personnel and materials to operate and maintain the plant. Fortuna is responsible for supplying fuel oil to the plant.

The preliminary power supply contract consists of a fixed monthly charge and a variable charge based on the total power consumed. Power is guaranteed by the contractor based on the power demand specified by the owner, which if not met results in penalties to the contractor; conversely, Fortuna must pay for a certain threshold minimum amount of kWh per year regardless of whether that power is consumed or not. As Fortuna is not responsible for any capital charges up-front in the contract, the contracted rates include the contractor's cost of capital.

Saxum originally prepared a detailed model of the costs for power generation on first principles as part of the 2013 Feasibility Study, considering an owner-supplied power generation plant. This model was updated in 2017 by Saxum and Mansfield to reflect a "leased" power model, with costs to operate and maintain the plant provided by a local contractor who specializes in contract power supply (Saxum, 2017).

The leased power model includes the following considerations:

- Fuel oil consumption for power generation
- The fixed and variable charges for the power generation plant contract, which includes the labor, supplies, and equipment for operation and maintenance of the power generation plant
- Maintenance of the road between Pocitos and the site

The above cost areas are all-inclusive in the power cost applied to Lindero. The power cost assumes a unit price of US$ 0.58 per liter of diesel oil. Argentina provides a reduced diesel oil price for power generation.

The average LOM unit price for power for Lindero on the above considerations is US$ 0.20 per kWh.

Power usage for the Project was derived, starting with the connected-load data from the mechanical equipment list. Equipment power demands under normal operation were assigned and coupled with equipment on-stream times to determine the average annual energy usage and cost. A year-by-year energy consumption estimate was developed from commissioning through the end of the closure period, which was used as the basis for the power cost and for defining the preliminary power supply contract described above. A summary of the average power consumption for Lindero by process area during operations is summarized in Table 21.9. The increase power consumption from Year 3 through Year 13 compared with Year 1 is basically the response to the increased activity on the leach pad.

**Table 21.9 Processing plant power and consumption projection**

| Area | Year 1 | | Year 2 to 13 | |
|---|---|---|---|---|
| | M kWh/year | kWh/t | M kWh/year | kWh/t |
| Crushing | 28.38 | 4.20 | 28.34 | 4.21 |
| Agglomeration | 6.12 | 0.91 | 6.08 | 0.90 |
| Leach Pad | 8.67 | 1.28 | 12.08 | 1.80 |
| ADR Plant | 1.11 | 0.16 | 1.11 | 0.16 |
| Reagents | 0.12 | 0.02 | 0.12 | 0.02 |
| SART plant | 3.46 | 0.51 | 3.46 | 0.52 |
| Laboratories | 0.85 | 0.13 | 0.85 | 0.13 |
| Energy | 0.83 | 0.12 | 0.83 | 0.12 |
| Fuel oil distribution and storage | 0.12 | 0.02 | 0.12 | 0.02 |
| Facilities | 4.93 | 0.73 | 4.93 | 0.73 |
| Water distribution and supply | 0.12 | 0.02 | 0.12 | 0.02 |
| **Total** | **54.71** | **8.11** | **58.03** | **8.63** |

### 21.5.8 Reagents, consumables, and other operating costs

Operating reagent and consumable requirements have been estimated based upon unit costs and consumption, where possible, and have been distributed by area. In the following sections the assumptions and unit costs associated with the development of the operating costs are presented. All freight costs have been included.

**Reagents and consumables**

Reagent consumptions are derived from testwork performed for Lindero and from the process design criteria. Table 21.10 shows the projected consumption of major consumables and Table 21.11 shows the forecast reagent prices used.

Operating costs for these items have been distributed based on tonnage and gold production, or smelting batches, as appropriate.

**Table 21.10 Process consumables forecast**

| Item | Form | Storage Capacity | Annual Consumption | |
|---|---|---|---|---|
| Sodium Cyanide | Container (IBC) - 1 tonne | 342 tonnes | 3,778 | tonnes |
| Cement | Bulk - Truck 20 tonnes | 500 tonnes | 10,129 | tonnes |
| Sulfuric acid | Liquid - Tank 20 tonnes | 148 tonnes | 4,796 | tonnes |
| Lime | Bulk - Truck 27 tonnes | 145 tonnes | 4,339 | tonnes |
| Sodium hydrosulfide | Liquid - Tank 20 tonnes | 63 tonnes | 963 | tonnes |
| Filter aid | Bag - 23 Kg | N/A | 86 | tonnes |
| Fluxes | Bag - 25 Kg | N/A | 31 | tonnes |
| Sodium Hydroxide | Bag - 25 Kg | 27 tonnes | 161 | tonnes |
| Sodium Hydroxide, solution 50% | Liquid - Tank 20 tonnes | 74 tonnes | 108 | tonnes |
| Activated Carbon | Bag - 500 kg | 8 tonnes | 38 | tonnes |
| Antiscalant | Liquid - Bin 1 tonne | 3 tonnes | 27 | tonnes |
| Hydrochloric Acid | Liquid, Tank - 20 tonnes | 7 tonnes | 114 | tonnes |
| Sodium Carbonate | Dry Solid Sacks | 1 tonne | 4 | tonnes |
| Borax 10 mol | Dry Solid Sacks | 1 tonne | 1 | tonnes |
| Sodium chloride (Salt) | Dry Solid Sacks | 1 tonne | 1 | tonnes |
| Silica | Dry Solid Sacks | 1 tonne | 0.4 | tonnes |
| Potassium nitrate | Dry Solid Sacks | 1 tonne | 0.1 | tonnes |
| Gas propane | Bulk Delivery (truck) | N/A | 188 | tonnes |

Table 21.11 Reagent price projection

| Reagent | Unit | Unit Price | Freight |
|---|---|---|---|
| Sodium Cyanide | US$/kg | 1.79 | 0.41 |
| Cement | US$/kg | 0.14 | 0.10 |
| Sulfuric acid | US$/kg | 0.34 | --- |
| Lime | US$/kg | 0.17 | 0.17 |
| Sodium hydrosulfide | US$/kg | 0.69 | 0.11 |
| Filter aid | US$/kg | 2.64 | 0.11 |
| Fluxes | US$/kg | 6.05 | 0.18 |
| Sodium Hydroxide | US$/kg | 1.06 | 0.11 |
| Sodium Hydroxide, solution 50% | US$/kg | 0.40 | 0.14 |
| Activated carbon | US$/kg | 2.12 | 0.28 |
| Antiscalant | US$/kg | 2.70 | 0.18 |
| Hydrochloric Acid | US$/l | 0.13 | 0.11 |
| Crucibles | US$/Unit | 946 | --- |
| Sodium Carbonate | US$/kg | 1.84 | --- |
| Borax 10 mol | US$/kg | 2.75 | --- |
| Sodium chloride (Salt) | US$/kg | 0.22 | 0.37 |
| Silica | US$/kg | 0.44 | --- |
| Potassium nitrate | US$/kg | 2.45 | --- |
| Gas propane | US$/kg | 0.89 | --- |
| Fuel oil | US$/l | 0.71 | --- |

### 21.5.9 Crusher wear parts and maintenance

Crusher liners and other wear parts for the primary (jaw) and secondary (cone) crushing areas were based on a unit cost per tonne of material processed. The unit cost was developed based on similar operations and supplier's information.

### 21.5.10 HPGR roll replacement

Costs for periodic wear replacement of HPGR rolls were based on quotations from Weir. The replacement interval for the tires was determined by special wear testwork performed by Weir on the Lindero ore with an applied safety factor (see Section 13.1.5 for testwork results). Note Weir provided a wear guarantee for the first set of rolls based on the testwork, which pro-rates the cost of the first set of replacement rolls if the promised wear life is not achieved. The first set of rolls purchased with the HPGR contains 55 mm studs, which are expected to provide additional wear life over shorter studs (and the longer studs are backed with a longer wear guarantee).

Costs include charges for the roll change service (estimated by Weir), rental of a 175 t crane, and freight.

### 21.5.11 Leach pad stacking

The crushed and agglomerated mineral is transported towards the leach pad through overland conveyor belts, portable belts (grasshoppers) and distributed with a radial stacker into cells to conform 10 meter-lifts according to a pre-stablished stacking plan to be leached according to an irrigation plan.

Stacking equipment consumables, parts and maintenance were based on similar operations and supplier's information.

### 21.5.12 Heap leach consumables

Heap leach consumables include expenses for broken pipe, fittings and valves and abandoned drip tubing, based on cost per tonne of material processed based on similar operations and supplier's information.

### 21.5.13 ADR plant/refinery

Propane gas is consumed in the process by the carbon regeneration kiln, elution solution boiler, and the smelting furnace in the refinery. Total consumption of propane gas for all equipment in the ADR plant is 13,500 kg monthly, and is based on process calculations and vendor information.

Miscellaneous operating and maintenance supplies are also included based on cost per tonne of material processed from similar operations and supplier's information.

### 21.5.14 Water distribution pipeline (well generators)

Water pumps and facilities at the remote well field and two booster stations located 13 km east of the Project site, will be run using diesel generators (3 operating). Diesel fuel consumption to operate the facilities was estimated based on preliminary loads for the equipment and typical consumption data for the selected gensets. The average diesel consumption for all gensets is approximately 380,000 l per year.

### 21.5.15 Metallurgical laboratory

Mechanical fire assaying and solution assaying of samples from both the mine and processing plant areas will be conducted at the on-site laboratory and will be operated by one laboratory manager and five technicians. The total number of samples processed will vary by year due to the requirements of the mine. An allowance of US$ 55,000 per year is also included for miscellaneous supplies.

### 21.5.16 Fuels

Propane gas will be required for stripping, carbon regeneration, and smelting operations as described above.

Diesel fuel will be required to operate mobile equipment assigned to the processing areas. The cost of diesel fuel is included in the hourly operating cost for these units.

### 21.5.17 Mobile equipment

Numerous pieces of support equipment are required for the processing area of the Project. These include light vehicles, a flatbed truck, forklift, a 50-ton crane and a dozer. The costs to operate and maintain each of these pieces of equipment has been estimated using primarily published information. Otherwise, allowances have been made based upon experience in similar operations.

## 21.6 General and administrative

General service costs include the costs of general management and operation support areas: Operation management, maintenance, geology, planning, safety and environment and chemical laboratory.

Annual general service costs have been estimated for personnel, equipment, consumables and parts were applicable. For maintenance, although most of the cost is distributed to the service, some overhead costs are registered as general services.

These estimations were developed by Fortuna based on its own operating units as well as from other similar operations.

Table 21.12 summarizes the annual general services costs per year as well as per processed tonne. The total average general services cost is estimated to be US$ 1.20/t processed.

Administrative services mine costs include administrative supporting areas in the mine site: Administration, human resources, community relations and logistics.

Annual administrative service mine costs have been estimated for personnel, equipment, consumables and parts were applicable.

These estimations were developed by Fortuna based on its own operating units as well as from other similar operations.

Table 21.13 summarizes the annual administrative service mine costs per year as well as per processed tonne. The total average administrative service mine cost is estimated to be US$ 0.95/t processed.

## 21.7 Comment on Section 21

The QP considers the capital and operating costs estimated for the Lindero Project as reasonable based on industry standard practices.

Costs were estimated primarily by Fortuna for mine pre-production and mine equipment costs. Saxum provided cost estimates for major and secondary equipment, buildings, infrastructure and major contracts. All equipment and material requirements are based on the design information described in previous sections of this study. Capital cost estimates have been made primarily using budgetary supplier quotes for all major and most minor equipment items, and major construction contract unit rates. Where supplier quotes were not available for minor items, a reasonable cost estimate was made based on supplier quotes in Saxum project files. All capital cost estimates are based on the purchase of equipment quoted new from the manufacturer, or estimated to be fabricated new.

Table 21.12 Annual general services cost forecast

| Cost | Units | FY1 | FY2 | FY3 | FY4 | FY5 | FY6 | FY7 | FY8 | FY9 | FY10 | FY11 | FY12 | FY13 | Total (Year 1-13) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ore Processed | **Mt** | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.56 | 6.64 | **87.45** |
| **General Services Costs** | | | | | | | | | | | | | | | |
| Operations Management | **M US$** | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | 0.34 | **4.47** |
| Maintenance | **M US$** | 2.94 | 2.96 | 3.09 | 3.10 | 3.07 | 3.07 | 3.09 | 3.10 | 2.99 | 2.96 | 3.12 | 3.20 | 3.05 | **39.74** |
| Geology | **M US$** | 0.68 | 0.68 | 0.68 | 0.68 | 0.68 | 0.68 | 0.68 | 0.68 | 0.68 | 0.68 | 0.68 | 0.68 | 0.68 | **8.89** |
| Planning | **M US$** | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | **9.68** |
| Safety and Environment | **M US$** | 2.13 | 2.13 | 2.11 | 2.13 | 2.14 | 2.15 | 2.15 | 2.15 | 2.15 | 2.15 | 2.15 | 2.15 | 2.14 | **27.84** |
| Chemical Lab | **M US$** | 1.08 | 1.08 | 1.08 | 1.08 | 1.08 | 1.08 | 1.08 | 1.08 | 1.08 | 1.08 | 1.08 | 1.08 | 1.08 | **14.06** |
| **Total General Services Cost** | **M US$** | **7.92** | **7.94** | **8.06** | **8.09** | **8.07** | **8.07** | **8.10** | **8.10** | **8.00** | **7.96** | **8.13** | **8.21** | **8.05** | **104.68** |
| **General Services Costs processed** | | | | | | | | | | | | | | | |
| Operations Management | **US$/t** | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | **0.05** |
| Maintenance | **US$/t** | 0.44 | 0.44 | 0.46 | 0.46 | 0.45 | 0.45 | 0.46 | 0.46 | 0.44 | 0.44 | 0.46 | 0.47 | 0.45 | **0.45** |
| Geology | **US$/t** | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | **0.10** |
| Planning | **US$/t** | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | **0.11** |
| Safety and Environment | **US$/t** | 0.32 | 0.32 | 0.31 | 0.32 | 0.32 | 0.32 | 0.32 | 0.32 | 0.32 | 0.32 | 0.32 | 0.32 | 0.32 | **0.32** |
| Chemical Lab | **US$/t** | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | 0.16 | **0.16** |
| **Total General Services Cost** | **US$/t** | **1.17** | **1.18** | **1.19** | **1.20** | **1.20** | **1.20** | **1.20** | **1.20** | **1.19** | **1.18** | **1.20** | **1.22** | **1.19** | **1.20** |

Table 21.13 Annual administrative mine cost projection

| Cost | Units | FY1 | FY2 | FY3 | FY4 | FY5 | FY6 | FY7 | FY8 | FY9 | FY10 | FY11 | FY12 | FY13 | Total (Year 1-13) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ore Processed | Mt | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.56 | 6.64 | **87.45** |
| **Administrative Services Mine Costs** | | | | | | | | | | | | | | | |
| Administration | M US$ | 5.44 | 5.44 | 5.44 | 5.44 | 5.47 | 5.49 | 5.51 | 5.51 | 5.51 | 5.51 | 5.51 | 5.51 | 5.47 | **71.28** |
| Human Resources | M US$ | 0.65 | 0.65 | 0.65 | 0.65 | 0.65 | 0.65 | 0.66 | 0.66 | 0.66 | 0.66 | 0.66 | 0.66 | 0.65 | **8.49** |
| Logistics | M US$ | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | 0.24 | **3.13** |
| **Total Adm. Services Mine Cost** | **M US$** | **6.33** | **6.33** | **6.33** | **6.33** | **6.36** | **6.38** | **6.41** | **6.41** | **6.41** | **6.41** | **6.41** | **6.41** | **6.36** | **82.90** |
| **Administrative Services Mine Costs processed** | | | | | | | | | | | | | | | |
| Administration | US$/t | 0.81 | 0.81 | 0.81 | 0.81 | 0.81 | 0.81 | 0.82 | 0.82 | 0.82 | 0.82 | 0.82 | 0.84 | 0.82 | **0.82** |
| Human Resources | US$/t | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | **0.10** |
| Logistics | US$/t | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | **0.04** |
| **Total Adm. Services Mine Cost** | **US$/t** | **0.94** | **0.94** | **0.94** | **0.94** | **0.94** | **0.95** | **0.95** | **0.95** | **0.95** | **0.95** | **0.95** | **0.98** | **0.96** | **0.95** |

# 22 Economic Analysis

## 22.1 Forward looking statement

The results of the economic analysis discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Such uncertainties and factors include, among others, changes in general economic conditions and financial markets; changes in prices for gold and other metals; technological and operational hazards during the development of the project; risks inherent in mineral exploration; uncertainties inherent in the estimation of mineral reserves, mineral resources, and metal recoveries; the timing and availability of financing; governmental and other approvals; political unrest or instability; labor relations issues; as well as those factors discussed under "Risk Factors" in Fortuna's most recent Annual Information Form. Although Fortuna has attempted to identify important factors that could cause actual actions, events or results to differ materially from those described in this section, there may be other factors that cause actions, events or results to differ from those anticipated, estimated or intended.

## 22.2 Summary

The Lindero Project economics were evaluated using a discounted cash flow (DCF) method, which estimates the NPV of future cash flow streams. The final economic model was developed by Fortuna using the following assumptions:

- Period of analysis of 16 years (includes one year of pre-production and investment), 13 years of production, and two years for closure and reclamation
- Gold price of US$ 1,250/oz
- Processing rate of 18,750 tpd ore
- Metallurgical recovery of 75 %
- Initial capital and operating costs as developed in Section 16.6 and 21 of this report
- Closure capital costs as outlined in Section 20

The Project economics based on these criteria from the cash flow model are summarized in Table 22.1.

## 22.3 Methodology

The Lindero Project economics were evaluated using a DCF method that requires annual cash inflows and outflows to be projected, from which the resulting net annual cash flows are estimated and then discounted back to the Project financing date. Considerations for this analysis include the following:

- The cash flow model was prepared by Fortuna

- The period of analysis is 16 years (including one year of pre-production and investment), 13 years of production, and two years for closure and reclamation
- All cash flow amounts are in US dollars (US$). All costs are considered to be third quarter 2017 US dollars. Inflation is not included in this model
- The Argentina peso exchange rate is 17.80 pesos to the US dollar for capital and operating costs
- The internal rate of return (IRR) is calculated as the discount rate that yields a NPV of zero
- The NPV is calculated by discounting the annual cash back to the Year 0 period at the selected discount rates. All annual cash flows are assumed to occur at the end of each respective year. Note that for simplification purposes, the initial capital cost is being projected in Year 0

**Table 22.1 Economic evaluation summary**

| | |
|---|---|
| **Production** | |
| Mine life[1] (years) | 15 |
| Annual ore placed in leach pad (Mt) | 6.75 |
| Strip ratio (waste to ore) | 1.2 |
| Head grade (g/t) | 0.62 |
| Recovery (%) | 75 |
| Gold recovered to doré (Moz) | 1.3 |
| Average annual gold recovered to doré[2] (Koz) | 96 |
| Peak annual gold recovered to doré (Koz) | 138 |
| AISC[3] ($/oz Au) | 802 |
| Initial capital ($ M) | 239 |
| Sustaining capital ($ M) | 105 |
| **Base Case Economics** | |
| Gold price ($) | 1,250 |
| Exchange rate (ARS[4]:USD) | 17.80 |
| After-tax NPV[5] @ 5 % ($ M) | 130 |
| After-tax IRR[6] (%) | 18 |
| Payback period[7] (years) | 3.6 |

Notes:
1. Includes 20 months of heap rinsing of gold inventory
2. Average over years 1 – 13; does not include gold from heap rinsing.
3. All-In Sustaining Cash Cost
4. Argentine Peso
5. Net Present Value; considers initial capital in one single annual period; excludes High-Pressure-Grinding-Roll (HPGR) acquired upon the acquisition of Goldrock Mines Corp.
6. Considers initial capital in one single annual period; excludes HPGR acquired upon the acquisition of Goldrock Mines Corp.
7. Payback based on undiscounted cash flow

- The payback period is the amount of time, in years, required to recover the initial capital cost

- Working capital is included in the model
- Government royalties and bank transaction taxes are included in the model
- Taxes (including depreciation and loss carry forwards) are included
- Salvage value for certain capital items is included
- 100 % equity financing is assumed
- Reclamation and closure costs are included

## 22.4 General assumptions

A summary of the general assumptions for cost inputs, parameters, royalties, and taxes used in the economic analysis are as follows.

- The gold pay factor is 99.94 %. An average refining charge of $0.49 per ounce is included, plus shipping and insurance average costs of $8.30 per ounce
- A gold price of US$1,250/oz is used as the base case commodity price
- Specific annual operating costs are applied to the cash flow model as shown in Table 22.3
- The initial capital costs for Project construction are incurred in the first year of development (Year 0). The expansion and sustaining capital is also included as shown in Table 22.2 and includes:
  - Replacement of mining equipment throughout the Project life
  - Expansions of the heap leach in Year 3
  - Installation of a conveyor stacking system in Year 3
- The HPGR has already been purchased and excluded from the initial capital cost total
- A 3 % royalty is payable on the net smelter return, less all operating costs except mining. This royalty applies to 100 % total sales
- Working capital includes a 30-day reserve (at full production operating costs) for salaries, services and non-imported materials during Month 1 when the Project transitions to becoming cash flow positive. It also considers a 45-day reserve for imported materials, such as cyanide
- It is assumed that working capital will be recovered at the end of the mine life (Year 15) and hence the effective sum of working capital over the LOM is zero
- Closure and reclamation costs of US$ 35.1 million are included, which are assumed to occur in Year 14 and continue through to the end of Year 15. Also, during those two years sales of gold production from heap rinsing is considered
- Salvage value of mining, mechanical, and electrical equipment is credited at 10 % at the end of the Project life in Year 15. This amounts to US$ 6.7 million
- The key tax assumptions are:

- o An accelerated 60-20-20 depreciation schedule (60 % depreciation the first year, 20 % second year, 20 % third year) is applied to all infrastructure capital costs (approximately 52 % of initial capital costs)
- o An accelerated 3-year straight line depreciation (33.3 % for each of the first three years after expenditure) is applied to equipment capital costs (approximately 38 % of initial capital costs)
- o 5 % of the initial capital costs are being depleted for tax purposes in the LOM at a rate equal to processed ore dived by reserve balance at the beginning of the year. This is related to indirect costs of the construction of the project such as catering and energy

**Table 22.2 Capital cost estimtae**

| Capex | Unit | Yr 0 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Yr 10 | Yr 11 | Yr 12 | Yr 13 | Yr 14 | Yr 15 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Initial | M US$ | 239.03 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Sustaining (incl. Closure) | M US$ | - | 1.59 | 0.06 | 36.14 | 5.46 | 2.19 | 5.31 | 3.18 | 2.86 | 3.99 | 4.61 | 2.54 | 1.76 | 0.59 | 16.22 | 18.96 |
| Non-sustaining | M US$ | - | - | - | 8.00 | - | - | - | - | - | - | - | - | - | - | - | - |
| **Total** | **M US$** | **239.03** | **1.59** | **0.06** | **44.14** | **5.46** | **2.19** | **5.31** | **3.18** | **2.86** | **3.99** | **4.61** | **2.54** | **1.76** | **0.59** | **16.22** | **18.96** |

- 5 % of the initial capital costs, in relation to G&A costs during construction, is being carried as a loss in Year 0, accumulating a loss carryforward
- The Argentine income tax rate of 35 % is applied to the estimated taxable income
- Bank transaction taxes are calculated at a rate of 0.6 % of each credit and debit, excluding sales from gold
- A double deduction of exploration costs is being applied in the model for the amount of $6.4 million. The first time in year 1 and the second time evenly spreader over the LOM
- Advance deduction of closure cost is being applied in the model for the amount of $35.1 million. Each year an amount equal or up to 5 % of the cash cost of that year is deducted for tax purposes until the total closure cost is reached

### 22.4.1 Average cash cost

The average cash cost for the life of the mine is calculated by adding all the mining, process, general and administrative services for the site and distribution and refining costs deducted by copper credits and dividing that number by the total ounces payable. The total operating costs net of by-product credits for the Project are US$ 856 million with the total payable gold ounces at 1,301,903 ounces, which results in average cash cost per ounce net of by-product credits of US$ 658.

### 22.4.2 All-in sustaining cash cost

In June 2013, the World Gold Council (WGC), which is the market development organization for the gold industry, published guidance on "all-in sustaining costs" and "all-in cost" metrics, which gold mining companies can use to report their costs as part of their overall reporting disclosure. The World Gold Council has worked closely with its member companies to develop these non-GAAP measures which are intended to provide transparency into the costs associated with producing gold.

The "all-in sustaining cost", which includes G&A costs, sustaining capital costs and royalties, but excluding income taxes, is US$ 802/oz Au.

## 22.5 Financial model and results

A DCF method was used to evaluate the economics of the Lindero Project. The DCF method measures the NPV of future cash flow streams. Table 22.3 presents the cash flows.

The Lindero Project shows an NPV of US$ 130 million after tax using a discount rate of 5 %, with an IRR of 18 %, and a payback period of 3.6 years, based on the LOM production plan, assumed metal prices, and integrated leaching treatment of gold and copper.

## Table 22.3 Cash flow analysis

| Production | Unit | Yr 0 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Yr 10 | Yr 11 | Yr 12 | Yr 13 | Yr 14 | Yr 15 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Processed Ore | Mt | 0.82 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.75 | 6.56 | 6.64 | | |
| Gold head grade | g/t | 0.55 | 0.94 | 0.86 | 0.66 | 0.63 | 0.56 | 0.49 | 0.58 | 0.54 | 0.50 | 0.59 | 0.52 | 0.56 | 0.60 | | |
| Gold recovered to doré | koz | 12.8 | 136.7 | 138.2 | 103.9 | 115.2 | 87.7 | 79.7 | 89.6 | 85.1 | 80.4 | 83.1 | 84.3 | 87.8 | 81.2 | 33.0 | 16.9 |
| Copper recovered to SART | t | | 1,326 | 1,470 | 1,157 | 1,000 | 755 | 717 | 754 | 650 | 633 | 600 | 582 | 586 | 520 | 258 | |
| **Sales** | | | | | | | | | | | | | | | | | |
| Gold | M US$ | | 170.8 | 172.7 | 129.8 | 143.9 | 109.5 | 99.5 | 111.9 | 106.3 | 100.5 | 103.8 | 105.3 | 109.6 | 101.5 | 41.2 | 21.1 |
| Copper | M US$ | | 5.4 | 6.0 | 4.7 | 4.1 | 3.1 | 2.9 | 3.1 | 2.6 | 2.6 | 2.4 | 2.4 | 2.4 | 2.1 | 1.0 | 0.0 |
| **Total** | **M US$** | | **176.2** | **178.7** | **134.5** | **147.9** | **112.6** | **102.4** | **115.0** | **109.0** | **103.1** | **106.2** | **107.6** | **112.0** | **103.6** | **42.3** | **21.1** |
| **Cash Cost and G&A** | | | | | | | | | | | | | | | | | |
| Mine | M US$ | - | 14.44 | 15.74 | 15.36 | 16.34 | 16.31 | 16.83 | 17.24 | 17.93 | 17.26 | 18.60 | 16.78 | 17.57 | 15.23 | - | - |
| Plant | M US$ | - | 38.19 | 40.92 | 38.85 | 39.25 | 38.00 | 37.22 | 36.14 | 36.68 | 36.00 | 35.61 | 35.34 | 35.33 | 35.06 | - | - |
| General services | M US$ | - | 7.92 | 7.94 | 8.06 | 8.09 | 8.07 | 8.07 | 8.10 | 8.10 | 8.00 | 7.96 | 8.13 | 8.21 | 8.05 | - | - |
| Administrative services mine | M US$ | - | 6.33 | 6.34 | 6.34 | 6.34 | 6.36 | 6.38 | 6.41 | 6.41 | 6.41 | 6.41 | 6.41 | 6.41 | 6.36 | - | - |
| Distribution and gold refining costs | M US$ | - | 1.38 | 1.44 | 1.29 | 1.25 | 1.13 | 1.11 | 1.13 | 1.08 | 1.07 | 1.06 | 1.05 | 1.06 | 1.03 | - | - |
| G&A | M US$ | - | 3.46 | 3.46 | 3.50 | 3.46 | 3.46 | 3.46 | 3.46 | 3.46 | 3.46 | 3.46 | 3.46 | 3.46 | 3.46 | - | - |
| **Total** | **M US$** | **-** | **71.73** | **75.83** | **73.39** | **74.73** | **73.33** | **73.07** | **72.48** | **73.66** | **72.19** | **73.09** | **71.16** | **72.03** | **69.19** | **0.00** | **0.00** |
| **CAPEX** | | | | | | | | | | | | | | | | | |
| Initial | M US$ | 239.03 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Sustaining (incl. Closure) | M US$ | - | 1.59 | 0.06 | 36.14 | 5.46 | 2.19 | 5.31 | 3.18 | 2.86 | 3.99 | 4.61 | 2.54 | 1.76 | 0.59 | 16.22 | 18.96 |
| Non-sustaining | M US$ | - | - | - | 8.00 | - | - | - | - | - | - | - | - | - | - | - | - |
| **Total** | **M US$** | **239.03** | **1.59** | **0.06** | **44.14** | **5.46** | **2.19** | **5.31** | **3.18** | **2.86** | **3.99** | **4.61** | **2.54** | **1.76** | **0.59** | **16.22** | **18.96** |
| **EBITDA calculation** | | | | | | | | | | | | | | | | | |
| Sales | M US$ | 0.00 | 176.17 | 178.66 | 134.53 | 147.92 | 112.59 | 102.43 | 114.95 | 108.97 | 103.06 | 106.23 | 107.63 | 112.03 | 103.57 | 42.28 | 21.12 |
| Costs and G&A | M US$ | 0.00 | -71.73 | -75.83 | -73.39 | -74.73 | -73.33 | -73.07 | -72.48 | -73.66 | -72.19 | -73.09 | -71.16 | -72.03 | -69.19 | - | - |
| Royalty | M US$ | 0.00 | -3.71 | -3.70 | -2.44 | -2.84 | -1.83 | -1.55 | -1.96 | -1.77 | -1.61 | -1.73 | -1.77 | -1.90 | -1.65 | -1.27 | -0.63 |
| Bank transactions tax | M US$ | 0.00 | -0.55 | -0.58 | -0.86 | -0.59 | -0.55 | -0.57 | -0.55 | -0.56 | -0.55 | -0.56 | -0.53 | -0.53 | -0.50 | -0.12 | -0.13 |
| **Total** | **M US$** | **0.00** | **100.18** | **98.55** | **57.83** | **69.76** | **36.88** | **27.24** | **39.96** | **32.99** | **28.70** | **30.84** | **34.17** | **37.56** | **32.23** | **40.89** | **20.35** |
| **Free cash flow calculation** | | | | | | | | | | | | | | | | | |
| EBITDA | M US$ | - | 100.18 | 98.55 | 57.83 | 69.76 | 36.88 | 27.24 | 39.96 | 32.99 | 28.70 | 30.84 | 34.17 | 37.56 | 32.23 | 40.89 | 20.35 |
| Income taxes | M US$ | - | 0.00 | -2.07 | 0.00 | -12.97 | -7.11 | -3.48 | -10.77 | -8.58 | -7.03 | -7.92 | -10.08 | -11.37 | -9.76 | -13.74 | -6.75 |
| Initial Capex | M US$ | -239.03 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Sustaining and non-sustaining capex | M US$ | - | -1.59 | -0.06 | -44.14 | -5.46 | -2.19 | -5.31 | -3.18 | -2.86 | -3.99 | -4.61 | -2.54 | -1.76 | -0.59 | -16.22 | -18.96 |
| Pre-production costs | M US$ | -9.85 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Pre-production revenue (net of taxes) | M US$ | 13.70 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Working capital | M US$ | -4.06 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 4.06 |
| Bank transaction taxes for year 0 | M US$ | -2.78 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Salvage value | M US$ | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 6.72 |
| **Total** | **M US$** | **-242.03** | **98.59** | **96.41** | **13.69** | **51.34** | **27.58** | **18.44** | **26.00** | **21.55** | **17.69** | **18.32** | **21.55** | **24.43** | **21.89** | **10.93** | **5.42** |

Total NPV @ 5% = US$ 130 M; IRR = 18 %; Payback Period = 3.6 years

## 22.6 Sensitivity analysis

An after-tax sensitivity analysis on metal price, metal recovery, operating costs and capital costs at ±10 % increments is presented in Table 22.4 and Figure 22.1 for NPV and Table 22.5 and Figure 22.2 for IRR.

**Table 22.4 NPV sensitivities (US$ million)**

| Range | Gold price | Gold recovery | Operating cash cost | Initial capital cost |
|---|---|---|---|---|
| -20% | -33 | -31 | 220 | 164 |
| -10% | 52 | 53 | 175 | 147 |
| **0%** | **130** | **130** | **130** | **130** |
| 10% | 207 | 206 | 85 | 113 |
| 20% | 284 | 282 | 39 | 96 |

**Figure 22.1 NPV sensitivity to significant parameters**

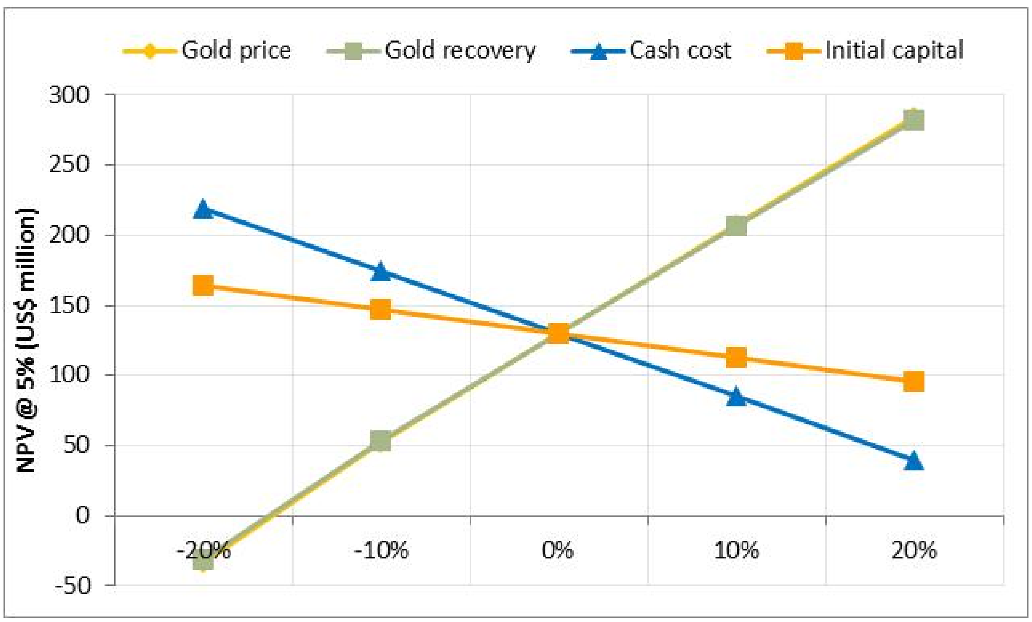
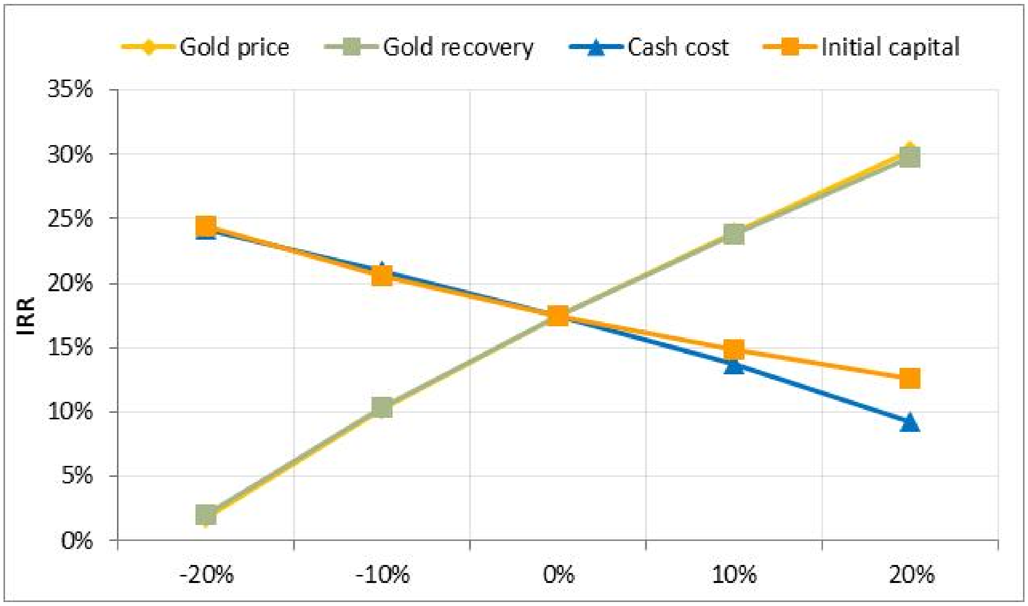



**Table 22.5 IRR sensitivities**

| Range | Gold price | Gold recovery | Cash cost | Initial capital |
|---|---|---|---|---|
| -20% | 2% | 2% | 24% | 24% |
| -10% | 10% | 10% | 21% | 21% |
| **0%** | **18%** | **18%** | **18%** | **18%** |
| 10% | 24% | 24% | 14% | 15% |
| 20% | 30% | 30% | 9% | 13% |

**Figure 22.2 IRR sensitivity to significant parameters**




The analysis indicates that the Project has robust economics across the range of variation in the key parameters examined with the greatest sensitivity being to gold price and recovery.

## 22.7 Comments on Section 22

The Lindero Project shows an NPV of US$ 130 million after tax using a discount rate of 5 %, with an internal rate of return (IRR) of 18 %, and a payback period of 3.6 years, based on the LOM production plan, assumed metal prices, and integrated leaching treatment of gold and copper.

NPV and IRR display the greatest sensitivity to gold metal prices and metallurgical recoveries according to the sensitivity analysis.

The QP considers the financial model to be a reasonable estimate of the economic situation at Lindero and based on the assumptions in this Report, the Lindero Project shows a positive discounted cash flow over the LOM and supports the Mineral Reserve estimate.

# 23 Adjacent Properties

This section is not relevant to this Report.

# 24 Other Relevant Data and Information

## 24.1 Hydrology

The hydrogeology of the area is summarized in this sub-section. Additional information is available in Section 9.7, Section 18.3.1, Section 20.1.2, and in KCA (2016a).

### 24.1.1 General characteristics and climate

The Project, located within the puna region, has a desert/arid climate. Rainfall in the region normally does not exceed 50 mm per year, evapotranspiration values are high and exceed 2,000 mm per year, and the soil retention capacity is low at 50 mm. These characteristics result in a significant deficit of water throughout the year, evidenced by the desert landscape. The only surface runoff is observed during the few precipitation events in summer. Very few permanent streams are observed, and generally streams are temporary and lagoons (vegas) are very small.

The regional geomorphology described in the environmental baseline study and environmental impact assessment study outlines an extensive environment of quaternary and tertiary sedimentary basins and valleys formed by ancient alluvial cones, coalescent alluvial deposits and salt lakes. The ranges of higher altitude formed by volcanic and intrusive rocks are zones where precipitation (mainly snow) is more significant; these ranges recharge the lower alluvial basins.

During exploration of Lindero, numerous wells or pools were dug adjacent to the perimeter of the Salar de Arizaro to supply water for drilling activities and camp usage. These wells and pools provided water of good quality and were recharged at sufficient rates to suggest a sufficient quantity of extractable water resources for the Project.

### 24.1.2 Hydrology and hydrogeology

Goldrock commissioned Vector Argentina SA (Ausenco; 2009a, b) and Conhidro (2013) to conduct a hydrologic study of the Project area, during the detailing of the environment base line map and EIA study. As part of the study, the Rio Grande hydrologic basin was defined through the study of various field parameters and review of satellite images. The basin was determined to be 1,687 km² in size. The basin includes seven hydrologic sub-basins, namely the Rio Grande, Peninsula, Cori, Arita, Chachas, Lindero and Emboscadero. Because of proximity to the Project, only the four sub-basins Arita, Chachas, Lindero and Emboscadero were studied with the purpose of finding water supply for the mining operation. These sub-basins are briefly described below:

- Chachas sub-basin - the most extensive in the area at 808 km², Chachas is characterized by a big wide alluvial valley flanked by rocky outcrops with temporary streams, and some springs (vegas) and temporary snow cover
- Lindero sub-basin – this basin extends for 234.8 km² entirely in the area of the Project. Streams are temporary and weakly developed, and the heads of the fluvial streams are in the Archibarca volcano
- Emboscadero sub-basin – this basin is the smallest of those reviewed, and extends for 22.2 km², and can be considered part of the Lindero sub-basin

- Arita sub-basin – this basin is located farthest south of the studied areas, is about 58.4 km² and is primarily a large flat area adjacent to the salar Arizaro. Some springs were identified, but no streams were recognized in this sub-basin

Based on regional data, the recharge rates of the basins were estimated. Those rates are summarized in Table 24.1. This estimate reveals that the Chachas sub-basin, 13 km to the east of the Project, has the highest aquifer recharge. The Andina 2 and Andina 3 water wells in the Chachas sub-basin have demonstrated a production potential of 70 $m^3$/h and 100 $m^3$/h respectively.

**Table 24.1 Recharge rates of Arita, Chachas, Lindero and Emboscadero sub-basins**

| Sub-Basin | Surface ($km^2$) | Recharge ($Hm^3$/year) |
|---|---|---|
| Chachas | 808.4 | 1.46 |
| Lindero | 234.8 | 0.42 |
| Arita | 58.4 | 0.11 |
| Emboscadero | 22.2 | 0.04 |

### 24.1.3 Geo-electric exploration

A geo-electric exploration program was completed to identify those sedimentary facies that allow the storage and movement of groundwater, to help define the drilling targets. A total of five areas were chosen, where 31 vertical electrical soundings were performed.

The area with the highest hydrogeological interest based on the results of the geoelectric profiles was the Chachas area, where subsequently two wells were drilled with very positive results.

During January 2012, Hidrotec conducted a geoelectrical profile in the sub-basin Lindero-Emboscadero near the Project site, with the focus to assess water resource possibilities close to the site and thus to reduce potential water supply costs. This area was determined to have sufficient potential to provide construction water requirements, while the permanent bore field in the Chachas region is being developed.

## 24.2 Geotechnical studies

This subsection summarizes the geotechnical investigations conducted previously by Goldrock prior to the 2013 Feasibility Study. Additional information can be found in Section 9.6 and KCA (2016a).

These studies were carried out by various consultants and relate to the leach pad, ponds, and open pit areas. The following is a list of geotechnical investigation documents previously developed in the leach pad area.

- The Mines Group Inc.,"G-3 Lindero Test Pits 1-21", 2010
- AMEC, "Plant Site Foundation Geotechnical Report Feasibility Level, Lindero Project", 2011
- Logging form and RQD of drill hole LP

- Logging form and RQD of drill holes CON 1-2-3-4-5
- Sergio R. López - Geologist, "Prospecting by Clays for construction of the leach pad around the Lindero (Gold) project, department Los Andes, Salta, Argentina", 2013
- Faculty of Exact, Physical and Natural Sciences, UNC, "Soils and clays characterization for bed support for the leach pad of Lindero Project in Salta Puna - Salta - Argentina", 2014

In the years 2010 to 2014, as part of the investigations by different consultants involved in the development of the feasibility engineering for the leach pad, 21 pits were executed without sampling, four topsoil samplings were taken and seven geotechnical holes were drilled. In addition, two condemnation drill holes were drilled. For the low permeability soil (clay liner) study, sampling was also conducted.

Several sources of low permeability soil (clay) were investigated in an October 2013 study. The search was concentrated within 25 km and identified several areas of potential clay borrow sources. Clays from the edge of the Salar de Arizaro were considered to be the best quality.

A laboratory test program on low permeability soil samples was completed and subgrade material was extracted to determine the conditions for the formation of a suitable soil liner. The in-situ soils are clayey or silty sand. Permeability tests were performed to find optimum compaction and moisture to attain the low permeability required for heap leach liners.

In 2015, designs of the pad and ponds were updated and geotechnical investigations to support the designs were completed. The geotechnical investigation included the execution of test pits, trial pads with on-site density tests, plate tests and infiltrometry tests; identification of quarries for soil liner, sampling, soil mechanics lab tests for mineral and material to be used as overliner; the available information of previous research has also been assessed. The work also included recommendations for a future geotechnical research campaign and its corresponding field and lab tests.

All historical geotechnical studies were assessed by CNI in 2017 who conducted additional testwork and evaluations to provide updated recommendations and conclusions for this Technical Report as detailed in Section 9.6.

## 24.3 Comments on Section 24

A number of geotechnical studies were performed at Lindero. Those studies form the basis for the pit slope estimates used in the mining model. Included in the studies were geotechnical surveys for heap leach and waste dumps. These studies are considered by the QP to be consistent with industry practices and adequate to support mine design.

# 25 Interpretation and Conclusions

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

## 25.1 Mineral tenure, surface rights, water rights, royalties and agreements

Fortuna was provided with a legal opinion that supports that the mining tenure held by Mansfield for the Project is valid and that Fortuna has a legal right to mine the deposit.

Minimum work requirements under the tenure grant have been met. Annual land usage and environmental compliance reports have been lodged. Additional permits are required for Project development and Fortuna has a well-defined legal path forward for permitting. Fortuna has no knowledge of any additional environmental liabilities related to any of the other concessions connected with the property.

The mineral tenement holdings cover 3,500 ha, and comprise 35 pertenencias, each of 100 ha, which are constrained by Gauss Kruger Posgar co-ordinates generated by survey. Tenure is held in the name of Mansfield Minera S.A. (Mansfield), a wholly-owned subsidiary of Fortuna. There is no expiry date on the pertenencias, providing Fortuna meets expenditure and environmental requirements, and pays the appropriate annual mining fees.

A 3 % provincial royalty "boca mina" is payable on revenue after deduction of direct processing, commercial and general and administrative costs. There are no royalties payable to any other third party.

Current permits have allowed exploration and associated supporting testwork to be conducted under appropriate Provincial and Federal laws.

In November of 2010, Mansfield submitted the EIA report for the Lindero Project, and in November 2011, received approval of this EIA through the issue of the DIA, which represents formal approval for mine construction, allowing excavation to proceed. EIA approval is also a requirement for several of the permits that are required for certain installations and workings. In December 2015, Mansfield filed the biannual updated EIA report on the Lindero Project as established by environmental law. The next update and submission of this major report is due by March 2018.

Surface rights are owned by the provincial state (Propiedad Fiscal) of Salta. There are no reservations, restrictions, rights-of-way or easements on the Project to any third-party. Mansfield holds a registered camp concession, and a granted and surveyed access right-of-way. Water permits and rights of access to the Project are guaranteed through water and access licenses granted by the Mining Court of Salta.

Surface rights for construction of a mining operation and plant have not been granted from the Provincial authorities. Development of such infrastructure will require additional negotiation and potentially, additional supporting studies.

## 25.2 Geology and mineralization

The Arizaro Volcanic Complex consists of two superimposed concentric volcanic centers, the Arizaro and the Lindero cones, located in the Archibarca volcanic arc at the southern margin of the Salar de Arizaro basin. Basement rocks crop out to the north of the Lindero Deposit, and consist of coarse-grained Ordovician granites uncomformably overlain by Early Tertiary red bed sandstones. The Lindero–Arizaro complex, a series of diorite to monzonite porphyritic stocks, intrudes these units.

Mineralized zones at the Lindero Deposit form a semi-circular shape about 600 m in diameter which extends to a depth of 600 m, consisting of four different zones at the surface. The distribution of gold–copper mineralization at Lindero shows a strong relationship to lithology, stockwork veinlets, and alteration assemblages. Gold values average 0.70 g/t Au and copper values are typically about 0.11 % Cu. Higher grades of gold– copper (approximately 1 g/t Au and 0.1 % Cu) are commonly associated with sigmoidal quartz, quartz–magnetite–sulfide, biotite-magnetite–chalcopyrite, magnetite–chalcopyrite and quartz–limonite–hematite stockworks that are strongly associated with K-feldspar alteration. This association is very common in the east zone of the deposit, where the highest gold grades occur. At other locations where one or more stockwork types are missing or the intensity of fracturing is lower, mineralization tends to be weaker and the grades of gold tends to be lower (approximately 0.4 g/t Au).

Gold mineralization at Lindero is characterized by native, free milling gold associated with chalcopyrite and/or magnetite grains with rare interstitial quartz.

The weathered oxidation zone at Lindero is generally poorly developed, and averages 44 m in thickness.

The Arizaro volcanic center is characterized by fine- to medium-grained hornblende diorite to monzonite porphyritic stocks. The Arizaro Deposit is dominated by a main, moderately to strongly mineralized intrusive unit that crops out in the central part of the prospect area. It consists of fine hornblende porphyritic diorite intruded by several stocks, dikes, igneous-cemented breccias and hydrothermal breccias. Smaller stocks are exposed in a few areas. Dikes of andesitic and dacitic composition are generally distributed radially to the main intrusive.

Several alteration assemblages are noted in the Arizaro Deposit area. Alteration patterns are semi-concentric and are asymmetric, with a core of moderate to strong potassic alteration including zones of K-feldspar-rich magnetite-silica alteration. An incomplete rim of chloritic alteration is developed outboard of the potassic alteration. In the southeast part of the deposit, intermediate argillic alteration has formed and overprints potassic alteration. Sericitic and very weak argillic alteration (hydrolytic alteration) has developed in the volcanic tuffs. To the south and west of the deposit, chloritic alteration passes directly to propylitic alteration. An actinolite-magnetite alteration assemblage forms in the eastern part of the prospect.

Arizaro gold–copper mineralization is hosted in one body which has a semi-oval shape at the surface. In the center there is a high-grade body with a semi-ellipsoidal form, extending north-south for 480 m and about 50 m wide. The Arizaro Deposit has styles of mineralization with copper–gold grades that are strongly correlated with different alteration assemblages. Mineralization is mainly associated with potassic alteration. This occurs generally in multi-directional veins, vein stockworks and disseminations. In some areas, the vein density is high, forming vein stockworks in the intrusive rocks. These vein stockworks are limited to magnetite–biotite veinlets, quartz–magnetite–chalcopyrite veinlets, late magnetite breccias, and, in late-stage mineralization events, anhydrite-sulfide veinlets. Chalcopyrite and bornite are the main copper minerals. Coarse gold was observed and confirmed with x-ray diffraction analysis in the University of Neuquen, Argentina, laboratory.

The Lindero Deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation.

Arizaro is at an earlier stage of exploration, and the controls of lithology, structure, and alteration on mineralization are currently insufficiently understood to support estimation of Mineral Resources.

## 25.3 Exploration, drilling and analytical data collection in support of Mineral Resource estimation

Drill holes drilled under Mansfield management in the period 2002 to 2016 have data collected using industry-standard practices. Drill orientations are appropriate to the orientation of the mineralization and core logging meets industry standards for exploration of a porphyry-style deposit.

Geotechnical logging is sufficient to support Mineral Resource estimation with the data for the Lindero Deposit having been reviewed by AMEC in 2010, and CNI in 2017, with regards to suitability to support detailed mine planning.

Collar surveys have been performed using industry-standard instrumentation. Uncertainty in collar locations of Lindero Deposit drill holes, surveyed using compass and tape, have been incorporated into subsequent resource classification.

Downhole surveys performed during the drill programs have been performed using industry-standard instrumentation. Uncertainties in the downhole locations of the Lindero Deposit drill holes have been incorporated into subsequent resource confidence category classification.

All collection, splitting, and bagging of trench and core samples were carried out by Mansfield personnel from 2000 to 2016 with the exception of campaigns conducted by Rio Tinto. No material factors were identified with the drilling programs that could affect Mineral Resource or Mineral Reserve estimation.

Sample preparation and assaying for samples that support Mineral Resource estimation has followed approximately similar procedures for most drill programs since 2000. The preparation and assay procedures are adequate for the type of deposit, and follow industry standard practices.

Sample security procedures met industry standards at the time the samples were collected. Current core and pulp sample storage procedures and storage areas are consistent with industry standards, although the storage of coarse rejects had to be improved as many of these samples were stored outside and subsequently damaged.

In 2009 an independent audit of the information used for the estimation of resources and reserves was conducted by AMEC and summarized in the KCA (2016a) Technical Report. The work included independent audits of the database, collar and downhole surveys, drill logs, assays, bulk density measurements, core recovery, and QAQC results.

The 2009 audit concluded that the data verification programs undertaken on the data collected from the Lindero Deposit up to 2009 support the geological interpretations, and the analytical and database quality, and therefore the data can support Mineral Resource estimation.

Fortuna has reviewed the work performed by AMEC and concurs with their opinion and has conducted additional audits and verification of historical information used in the most recent resource and reserve estimates as well as verifying new data generated during the 2016 drilling campaign to support assumptions for a construction decision and the updated Mineral Resource estimate. The verification process focused on the database; collars and downhole surveys; lithological logs; assays; metallurgical results; and geotechnical parameters. Fortuna checked all collar and downhole survey information for each campaign against source documentation and completed a hand-held GPS survey of randomly selected drill-hole collars. The results showed a good agreement with locations in the database. In August 2016, Fortuna initiated a comprehensive program of relogging to verify the original lithological descriptions.

Fortuna contracted CNI to validate all geotechnical data, data collection methods, slope stability analysis methods, and slope angle recommendations presented previously by other consultants to determine feasibility-level slope angle recommendations for design of the planned Lindero final pit.

The QP is of the opinion that the data verification programs performed on the data collected from the Project are adequate to support the geological interpretations, the analytical and database quality, and Mineral Resource estimation at the Lindero Project. This conclusion is based on the following:

- No material sample biases were identified from the QAQC programs. Analytical data that were considered marginal were accounted for in the resource classifications
- Sample data collected adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits
- External reviews of the database were performed in 2003, 2008, and 2009, producing independent assessments of the database quality. No significant problems with the database, sampling protocols, flowsheets, check analysis program, or data storage were noted
- Mansfield compiled and maintains a relational database (DataShed™) for the Lindero Project which contains all collar, assay, density, survey and lithology information as well as all associated QAQC data
- Drill holes lacking surveyed collar coordinates have been resurveyed wherever possible and the original surveyor records stored
- Assays obtained during Goldrock's drilling programs that lacked original assay certificates have been verified by a program of re-assaying of 10 % of the pulps which indicated no significant bias between the original and reassayed results

Drill data are typically verified prior to Mineral Resource estimation, by running a software program check.

## 25.4 Metallurgical testwork

The Lindero Project has an extensive body of metallurgical investigation comprising several phases of testwork as indicated in the KCA (2016a) Technical Report. In general, the testwork was done to industry standards. However, some leach conditions set for the testwork made interpretation difficult. Reinterpretation of the raw test data provided the basis for advancing the metallurgical knowledge base for Fortuna.

Since September 2016, Fortuna has performed complementary metallurgical testwork in the areas of comminution, heap permeability and cement agglomeration, gold extraction in column tests, and copper removal with SART technology with the purpose of confirming and optimizing process design criteria.

Optimization of the process design has confirmed the benefit of the use of a HPGR, the inclusion of cyanide cure of ore, and copper removal/ cyanide recovery with a SART plant. Results indicate that these components allow for improved gold leaching kinetics and effective extraction of copper from the pregnant solution.

Ore will be crushed at a nominal rate of 18,750 tpd using a three-stage crushing system including a HPGR in the tertiary stage. A final crush size of $P^{80}$ 9.0 mm is projected. The crushed product will be agglomerated and cured with a cyanide solution and then conveyed to the leach pad. A mobile conveying and stacking system will be used to stack ore in 10-m-high lifts. The LOM leach pad area is projected at 105 ha with a maximum height of 100 m. Leaching will be carried in two stages with a first stage of 30 days and a second stage of 60 days.

The gold pregnant solution will be pumped at a rate of 400 $m^3$/hr to a SART plant, where copper content in solution will be precipitated in order to maintain copper levels below 400 ppm in the solution. The Project contemplates an expansion of the pregnant solution flow rate from 400 $m^3$/hr to 600 $m^3$/hr in Year 4 with the objective of reducing gold ounce inventory in the heap at the end of mining.

Following the SART plant, the pregnant solution will go to an ADR plant and then to electrowinning and refining where gold will be poured in doré bars. LOM gold recovery is estimated at 75 %.

The 2017 work was consistent with industry practices and provided the basis for optimization of the process design at Lindero.

It is the opinion of the QP that the Lindero samples tested represent the orebody with respect to grade and metallurgical response. The differences between metallurgical lithologies are minimal with regard to extraction. Cyanide consumptions are higher with the more oxidized Met 2 samples as would be expected. Minimal metallurgical differences were expected after review of the historical work.

Physical differences appear to have greater impact on the processing of the Lindero met types. Of significant importance is the ability of the agglomerated ore to support the planned heap height.

No significant deleterious materials such as mercury or clays were noted in the samples tested.

A high level of metallurgical and process risk mitigation is incorporated in the process design with HPGR crushing, agglomeration and the SART plant. With these installations any expected short-term variation in ore composition (i.e. elevated soluble copper content) or physical properties (i.e. elevated gypsum levels or increased ore hardness at depth) can be accommodated in the normal course of operations.

## 25.5 Mineral Resource estimation

Mineral Resource estimation involved the usage of drill hole and channel sample data in conjunction with surface mapping to construct 3-D wireframes to define individual lithologic structures and oxide/mixed/sulfide horizons. Drill hole samples were selected inside these wireframes, coded, composited and top cuts applied if applicable. Boundaries were treated as either soft, firm or hard with statistical and geostatistical analysis conducted on composites identified in individual lithologic units. Gold and copper grades were estimated into a geological block model consisting of 10 m x 10 m x 4 m SMUs. Primary mineralized units include the FPD and CPD1 lithologies with lower grade mineralization present in the S1, PBFD, and DDP units. Grades were estimated using dynamic anisotropy by ordinary kriging and constrained within an ultimate pit shell based on estimated metal prices, costs, geotechnical constraints, and metallurgical recoveries so as to fulfill the expectation of reasonable prospects for eventual economic extraction. Estimated grades were validated globally, locally, and visually prior to tabulation of the Mineral Resources.

The QP is of the opinion that the Mineral Resources for the Project, which have been estimated using core drill data, have been performed to industry best practices, and conform to the requirements of CIM (2014). The Mineral Resources are acceptable to support declaration of Mineral Reserves.

Furthermore, it is the opinion of the QP that by constraining the Mineral Resources within an open pit shell based on established mining and processing costs; recommended slope angles based on independent geotechnical investigations; metallurgical recoveries from extensive testwork; reasonable long-term metal prices; and the application of a transparent marginal cut-off grade, the Mineral Resources have 'reasonable prospects for eventual economic extraction'.

## 25.6 Mineral Reserve estimation

Mineral Reserve estimates have considered only Measured and Indicated Mineral Resources as only these categories have sufficient geological confidence to be considered Mineral Reserves (CIM, 2014). Subject to the application of certain economic and mining-related qualifying factors, Measured Resources may become Proven Reserves and Indicated Resources may become Probable Reserves.

The Mineral Reserve estimation procedure for the Lindero Deposit is defined as follows:

- Handover and review of Mineral Resources in longitudinal sections and grade–tonnage curves
- Definition of economic parameters for computing an appropriate cut-off grade to be applied to the open pit optimization such as downstream costs, process and mining costs, haulage incremental cost, metallurgical recoveries by mineralization, and sustaining capital
- Slope parameters based on geotechnical considerations were applied to the pit optimization subsequently used to generate overall slope angles and applied to the block model
- Compute the dollar value for each block to define blocks that can be mined at a profit in the LG algorithm

- Use of 'block discounting' to account for time value of money effect, resulting in a slightly smaller pit shell than would have been generated otherwise. A 5 % discount rate and 6 eight-meter benches per period were used for the discounting
- Inferred Resources are considered as waste material in the optimization process
- Performed a LG pit optimization using Datamine's NPV Scheduler™
- Mineral Reserves are reported within an ultimate pit design at variable cut-off grades that are based on the process type, operating costs and metallurgical recovery
- A dilution allowance for the Mineral Reserve estimate is applied using diluted model grades. The diluted model, which was built from the Mineral Resource block model, incorporates dilution and ore loss and eliminates the need for applying additional factors
- Mineral Reserve and Mineral Resources exclusive of Mineral Reserves tabulation and reporting as of September 9, 2017

Mineral Reserves will support a 13-year LOM considering 350 days in the year for production and a capacity rate of 18,750 tpd. The expectation based on an optimized production schedule is for an annual average production of 129,000 troy ounces of gold.

The conversion of Mineral Resources to Mineral Reserves was undertaken using industry-recognized methods, estimated operational costs, capital costs, and plant performance data. Thus, it is considered to be representative of future operational conditions. This Report has been prepared with the latest information regarding environmental and closure cost requirements.

## 25.7 Mine plan

Lindero will be an owner-operated conventional open pit mining operation with a nominal rate of 18,750 tpd of ore and a life of pit operations of 13 years using based on the Mineral Reserve estimate. The ratio of waste to ore over LOM is 1.2 to 1. The key mining fleet equipment will be initially composed of six 91 tonne trucks and two 17 cubic yard wheel loaders.

In the initial two years, the operation will benefit from mining the higher-grade, outcropping portion of the deposit, with an average head grade of 0.90 g/t Au, and a low strip ratio of 0.77 to 1. For the initial four years, the average head grade is projected at 0.77 g/t Au and a strip ratio of 1 to 1.

Mining costs benefit from short haul distances from the pit to the primary crusher and waste dumps. Maximum distances are in the range of 2 km. LOM direct mine cost is estimated at $1.1 per tonne moved.

The QP is of the opinion that:

- The mining method being used is appropriate for the deposit being mined. The open pit, stockpile, waste dump designs, and equipment fleet selection are appropriate to reach production targets
- The mine plan is based on successful mining philosophy and planning, and presents low risk

- Inferred Resources are not included in the mine plan
- The mobile equipment fleet presented is based on simulations and bench marks of similar operations achieving similar production targets
- All mine infrastructure and supporting facilities meet the needs of the current mine plan and production rate
- Major planned maintenance of the main equipment, such as loaders and trucks, have been cover in sustaining capital by purchasing additional equipment that can replace any possible lost production hours and not impact production targets
- The ancillary equipment appears to be undersized, especially dozers, but this would be covered by renting additional equipment as necessary

## 25.8 Recovery

Most of the major process concepts presented in the 2016 Technical Report such as: HPGR-crushing, cyanide heap leaching and carbon adsorption recovery, remain unchanged for this update. Additional physical and metallurgical understanding, developed by the testwork conducted by Fortuna in 2016 and 2017, resulted in modifications in the approach to these major process concepts for the Lindero Project as follows.

- A concentrated cyanide cure was added to shorten the leach cycle and increase extraction
- Agglomeration with cement was added to support a 110-meter-high heap with the HPGR-crushed ore
- Conveyor stacking was included from startup
- Two-stage leaching was included to increase pregnant grades and reduce overall flowrate to the ADR plant
- A SART plant was included to control the copper in solution
- Leach solution flow will be increased 150 % in Year 4 to reduce in-heap gold inventory

Unit operations for the Lindero process were selected based on the physical and metallurgical needs of the Lindero ore to achieve maximum extraction of gold. No novel or untried technology is employed in the process.

## 25.9 Infrastructure

The QP is confident that all mine and process infrastructure and supporting facilities have been included in the general layout to ensure that they meet the needs of the mine plan and production rate and notes that:

- Lindero is located 260 km due west of the city of Salta, the main service center for the Project and the region. Drive time from Salta to the Project is approximately 7.5 hrs over a road distance of 420 km with good year-round access. Significant road improvements are planned for stretches of road between the Tolar Grande and the Fortuna camp

- The Project site infrastructure has a compact layout footprint of approximately 60 ha
- Power will be generated on-site by a contractor through an 8 MW capacity diesel oil plant. Electrical power will be generated on site under a contract power supply arrangement with a local company who specializes in such services
- Total water requirements are 97.7 $m^3$/hr and will be primarily sourced from two existing wells located 13 kilometers southeast from the Project site. An additional well is required and will be drilled as part of construction activities
- Most of the process buildings for the Lindero Project have been primarily designed as steel frame buildings with modular thermo-acoustic panels; in general, these are pre-engineered and pre-fabricated steel buildings which include all structural members, exterior doors and windows, roofs, insulation, interior and exterior wall panels and all connectors required to erect and assemble the building on-site
- A permanent accommodation camp for 320 beds will be built for the LOM operation. For the construction period, temporary accommodations will be implemented to accommodate the peak of construction manpower estimated at 600 people

## 25.10 Markets and contracts

No market studies are currently relevant as the Lindero Project will produce a readily-saleable commodity in the form of doré.

As of the effective date of this report Fortuna has not entered into any material contracts required for the development of the Lindero Project including mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements.

The gold price used for the base case cash flow analysis is $1,250/oz. Sensitivities with variable price projections have also been considered. The Lindero Project, like most gold projects, is highly sensitive to changes in the gold price.

The Lindero mine product will be doré bars containing an estimated gold content averaging 84 % for the Project life. Overall gold extraction in respect to ore placed on the heap leach is estimated to be approximately 75 %.

## 25.11 Environmental, permitting and social considerations

In November 2010, Mansfield submitted an Environmental Impact Assessment (EIA) for the Lindero Project, and in November 2011 received approval through the issue of the Declaración de Impacto Ambiental (DIA). Approval of the EIA represents formal approval for mine construction, allowing excavation to proceed. Environmental law requires that the EIA be updated biannually with the current report submitted in December 2015 and an updated report planned for submission in March 2018.

Mansfield received a mine permit to build a heap-leach gold mine at up to 30,000 tpd as detailed in the Pre-Feasibility Study (AMEC, 2010b).

The Salta Provincial authorities have approved the building and electrical permits that Mansfield requires to commence construction at Lindero. Electrical, structural, building and seismic plans have been reviewed and approved by COPAIPA (Dec 2013), the professional engineering institution that overlooks all construction in Salta Province. Mansfield is planning to submit additional information to COPAIPA in 2017 to obtain the permits for construction of the agglomeration and SART plants that have been added to the process design. Mansfield does not foresee any issues in obtaining the necessary permits to complete construction and commence operation at Lindero.

In addition, a formal public declaration of support for the Lindero development has been issued by the provincial government, recognizing Lindero as the priority development project for the Salta Province.

Environmental risks during the closure stage will be reduced by remediation and monitoring work. At the closure stage, soil will be contoured by heavy machinery to minimize the long-term impact of mining activity, and return the topology of the land to resemble prior conditions. However, the movement of soil, and thus the risk, will be significantly less than in the mining operations stage.

One social-environmental risk will be the completion of contracts of employment directly, or indirectly, through contractors, and the surrounding communities. It will be imperative to implement measures to mitigate this impact during the whole period of mine operation.

A significant environmental risk will also be present during the closure of facilities, which will cause significant production of non-hazardous industrial waste and hazardous products from the movement of heavy machinery. It will be essential to establish clear environmental policies with the contractors during this process.

It is the opinion of the QPs that the appropriate environmental, social and community impact studies have been conducted to date at Lindero. Mansfield have maintained all necessary environmental permits that are the prerequisites for the granting of construction permits that will need to be obtained upon completion of detailed engineering designs for the Project infrastructure.

## 25.12 Capital and operating costs

Capital and operating costs for the Lindero Project were estimated by Fortuna with the assistance of Elbow Creek, Allard Engineering Services and a local engineering firm, Saxum. These costs are based on the design outlined in this study and are considered to have an accuracy of ±15 %. Total mine capital cost is estimated to be US$ 282.08 million using costs are per the second and third quarter 2017. No escalation factors have been applied to any costs, present or future capital.

Expansion (future) capital for the Project includes the Phase 2 leach pad construction in Year 3, expansion of the ADR plant and solutions handling in the leach pad area in Year 3. The total future capital is estimated at US$ 113 million.

Closure and reclamation costs are estimated at US$ 35 million, incurred in Year 13 through Year 17.

The total LOM operating cost for the Lindero Project is US$ 10.32 per tonne of ore processed.

Costs were estimated primarily by Fortuna for mine pre-production and mine equipment costs. Saxum provided cost estimates for major and secondary equipment, buildings, infrastructure and major contracts. All equipment and material requirements are based on the design information described in previous sections of this study. Capital cost estimates have been made primarily using budgetary supplier quotes for all major and most minor equipment items, and major construction contract unit rates. Where supplier quotes were not available for minor items, a reasonable cost estimate was made based on supplier quotes in Saxum project files. All capital cost estimates are based on the purchase of equipment quoted new from the manufacturer, or estimated to be fabricated new.

## 25.13 Economic analysis

The Lindero Project economics were evaluated using a DCF method, which estimates the NPV of future cash flow streams. The final economic model was developed by Fortuna using the following assumptions:

- Period of analysis of 16 years (includes one year of pre-production and investment), 13 years of production, and two years for closure and reclamation
- Gold price of US$ 1,250/oz
- Processing rate of 18,750 tpd ore
- Metallurgical recovery of 75 %
- Initial capital and operating costs as developed in Section 16.6 and 21 of this report
- Closure capital costs as outlined in Section 20

The Lindero Project shows an NPV of US$ 130 million after tax using a discount rate of 5 %, with an internal rate of return (IRR) of 18 %, and a payback period of 3.6 years, based on the LOM production plan, assumed metal prices, and integrated leaching treatment of gold and copper.

NPV and IRR display the greatest sensitivity to gold metal prices and metallurgical recoveries according to the sensitivity analysis.

The QP considers the financial model to be a reasonable estimate of the economic situation at Lindero and based on the assumptions in this Technical Report, the Lindero Project shows a positive discounted cash flow over the life-of-mine and supports the Mineral Reserve estimate.

## 25.14 Risks and opportunities

A number of opportunities and risks were identified by the QPs during the evaluation of the Lindero Project.

Opportunities include:

- Once mining commences there is an opportunity to collect additional geotechnical data from the open pit that could support an increase in final pit slope angles, potentially decreasing stripping ratios and/or increasing Mineral Reserves

- The Arizaro porphyry system is not included in the current mine plan. However, it represents upside opportunity for the Project if a satellite operation can be developed on the deposit
- Infill drilling could support the conversion of Inferred Resources to Measured or Indicated Resources and, with the appropriate studies, to Mineral Reserves. This represents additional upside potential for the planned operation
- The Lindero porphyry gold system remains open at depth below the pit shell constrained reported reserves and resources. An area of interest has been identified by Fortuna during the drilling campaign carried out in 2016 with drill hole LDH-126 encountering 0.97 g/t Au over a 38 m interval (refer to discussion in Section 10). This is supported by historical drilling from 2007 including drill hole LDH-86 averaging 1.06 g/t Au over a 52 m interval which bottomed in mineralization. These intercepts warrant follow-up drill testing
- There are a number of local exploration targets within the concession boundary, that with further work, represent upside opportunity to identify mineralization that can potentially add to the resource base
- If historical samples are assayed for cyanide-soluble copper, there is an opportunity to construct a metallurgical model and incorporate this into the scheduling and process design. This would support optimization of blending strategies and better understanding of recoverable copper as a by-product from the SART plant. Improved copper recoveries could have a minor positive impact on the mine economics
- Performance of the equipment can be tracked with the implementation of a fleet management system to record the main key performance indicators (KPI's) which will provide an opportunity to improve utilization and time loss productivity
- Once mining commences there is an opportunity to conduct additional blasting fragmentation analysis so as to improve mining productivity and optimize mining costs

Risks include:

- Local behavior of cyanide-soluble copper is not fully understood, and cannot be modeled due to a lack of assays from historical core. Levels of soluble copper could be higher than anticipated in certain areas of the deposit requiring adjustments to mine plans and schedules to reduce the impact in the plant. The introduction of a SART plant has greatly reduced the potential impact of soluble copper at the Project
- Delaying the acquisition of fleet equipment could cause delays in the execution of certain activities. It is therefore imperative that a clear schedule of lead times is established, and equipment purchased in a timely manner to ensure on time delivery
- Fortuna calculates that two loaders are needed from Year 3 onwards, but simulations indicate that three may be required in Year 2. Once mining commences and data on loader productivity is collected, a new fleet simulation should be performed to confirm if a third loader is required in Year 2 and if so how this will affect sustaining capital expenditure

- There is a risk that two dozing machines in the original capital estimate are insufficient. Fortuna plans to mitigate this risk by renting additional ancillary equipment as required
- There is a risk that haul truck tire life of 8,500 hours is higher than can be achieved at the operation, which could lead to marginally higher operating costs than anticipated

# 26 Recommendations

The work completed to date has continued to demonstrate that Lindero is a technically and economically viable project with the latest technical studies de risking many aspects of the Project and leading to the positive construction decision (Fortuna, 2017).

Recommendations for the next phase of work have been broken into those related to ongoing exploration activities and those related to additional technical studies. Recommended work programs are independent of each other and can be conducted concurrently unless otherwise stated.

## 26.1 Exploration

### 26.1.1 Arizaro

Continued work at Arizaro that focuses on the controls of lithology, structure, and alteration on mineralization so as to determine the suitability of material as a potential feed for the Lindero plant and to support the estimation of Mineral Resources. It is recommended that a 2,000-m reverse circulation (RC) drill program (approximately 100 holes at a 75 m spacing) is conducted at a cost of approximately US$ 500,000.

### 26.1.2 Lindero

An infill drill program involving the drilling of approximately 3,000-m of RC drill holes is recommended to improve the geological understanding of material planned for extraction in Years 1 and 2 of the mine. The cost of such a program is estimated at approximately US$ 750,000.

### 26.1.3 Other

Exploration work to date on the Lindero concession has been focused on outcropping porphyry mineralization. It is recommended that the Company evaluate the property for mineralization beyond the two known porphyry systems at Lindero and Arizaro. For example, alteration zones and silica structures located within the concession, 2.5 km due south of the Lindero Project site, remain open for evaluation. Exploration work would primarily involve mapping and carry no additional cost to the Project.

## 26.2 Technical

The following technical studies are recommended to improve the understanding of the Lindero Project:

### 26.2.1 Mineral Resources and Reserves

It is recommended that a drill hole spacing study be conducted to establish the density of sampling that is required to reduce the grade variability to acceptable levels for specified extraction time frames in respect to infill and blast control drilling. This will be used to support the estimated meters of infill drilling. The study can be conducted either inhouse (at no cost) or by external consultants, at an estimated cost of US$ 25,000.

### 26.2.2 Mining

Additional analysis is recommended into the mine operating and ore control process, in particular, the usage of optimum dig lines for open pit grade control, with the objective of minimizing ore loss and maximizing profit. The cost of licenses and implementing such software is estimated at US$ 276,000

A fleet management system should be considered for KPI purposes, which will provide an opportunity to improve utilization and time loss productivity. The cost of licenses and implementing such software is estimated at US$ 1.5 million

### 26.2.3 Metallurgical

Additional metallurgical testwork is currently in progress. These tests include:

- A 10-meter Met 1 fresh core column is currently under leach. This column is identical to the existing core Met 1 column, CL-01
- Several small columns designed to compare the effects of site water and local cement with the base condition of Portland Type II and tap water
- A leach column to generate pregnant solution using site water for additional SART and carbon adsorption tests
- Carbon adsorption tests investigating the effects of pH, free cyanide and copper concentration
- Mineral Liberation Analysis (MLA) of leach column tails and a feed concentrate

The cost of the above testwork is estimated at US$ 200,000.

In addition to the tests in progress a few other avenues of investigation are recommended.

- The cement in each lift on the heap will cure for several months before another lift is placed. It may be several years before any block of agglomerated ore receives 110 m of loading. It is recommended that a long-term stacking test be conducted to see if ageing will improve the ability of the ore to support the 110-m height with less cement. The estimated cost of the testwork is US$ 20,000
- The high static holdup (adsorbed moisture) in the heap makes the secondary leach at 6 $l/hr/m^2$ inefficient when the heap height increases. There is a possibility that a surface tension modifier may reduce the amount of adsorbed moisture in the heap reducing the inventory. The estimated cost of the testwork is US$ 20,000

### 26.2.4 Environmental, social and permitting

In addition, during construction it is recommended that Mansfield:

- Submit EIA update reports as required
- Confirm water well flow rates are suitable for mine requirements
- Conduct air quality sampling in camp locations and operating facilities, as well as on routes near local villages

- Monitor during the stages of construction so as to remain aware and keep control of any possible impacts caused by the operations at the mine

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# Certificates

CERTIFICATE of QUALIFIED PERSON

(a) I, Eric N. Chapman, Vice President of Technical Services for Fortuna Silver Mines Inc., 650-200 Burrard St, Vancouver, BC, V6C 3L6 Canada; do hereby certify that:

(b) I am the co-author of the technical report titled Fortuna Silver Mines Inc. Lindero Property, Salta Province, Argentina dated October 31, 2017 (the "Technical Report").

(c) I graduated with a Bachelor of Science (Honors) Degree in Geology from the University of Southampton (UK) in 1996

and a Master of Science (Distinction) Degree in Mining Geology from the Camborne School of Mines (UK) in 2003. I am a Professional Geologist of the Association of Professional Engineers and Geoscientists of the Province of British Columbia (Registration No. 36328) and a Chartered Geologist of the Geological Society of London (Membership No. 1007330). I have been practicing as a geoscientist and preparing resource estimates for approximately fourteen years and have completed more than twenty resource estimates for a variety of deposit types such as epithermal gold/silver veins, porphyry gold deposits, banded iron formations and volcanogenic massive sulfide deposits. I have completed at least eight Mineral Resource estimates for precious metal projects over the past five years.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements of a 'qualified person' for the purposes of the Instrument.

(d) I last visited the property on March 21, 2017;

(e) I am responsible for the preparation of sections 2: Introduction; 3: Reliance on other experts; 4: Property description and location; 5: Accessibility, climate, local resources, infrastructure and physiography; 6: History; 7:Geological setting and mineralization; 8: Deposit types; 9: Exploration; 10: Drilling; 11: Sample preparation, analyses and security; 12: Data verification; 14: Mineral Resource estimates; 23: Adjacent properties; 27: References; and the conclusions derived therefrom in sections 1: Summary; 25: Interpretation and conclusions; and 26: Recommendations.

(f) I am an employee and not independent of the issuer, Fortuna Silver Mines Inc.

(g) I have been an employee of Fortuna since May 2011 and involved with the property that is the subject of the Technical Report since August 2016.

(h) I have read NI 43–101, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

(i) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report that I prepared contain all the scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated at Vancouver, BC, this 31st day of October 2017.

[signed]

Eric N. Chapman, P. Geo., C. Geol. (FGS)

CERTIFICATE of QUALIFIED PERSON

(a) I, Edwin Gutierrez, Technical Services Manager for Fortuna Silver Mines Inc., 650-200 Burrard St, Vancouver, BC, V6C 3L6 Canada; do hereby certify that:

(b) I am the co-author of the technical report titled Fortuna Silver Mines Inc. Lindero Property, Salta Province, Argentina dated October 31, 2017 (the "Technical Report").

(c) I graduated with a Bachelor of Science Degree in Mining from Pontificia Universidad Catolica del Peru, Lima, Peru in 2000. I have a Master of Science Degree in Mining from University of Arizona, USA, granted in 2008. I am a Registered Member of the Society for Mining, Metallurgy and Exploration, Inc. (SME Registered Member Number 4119110RM). I have practiced my profession for 16 years. I have been directly involved in underground and open pit operations, mining consulting, and assisting in the development of mining projects in Peru, Brazil, Chile, Argentina, Ghana, Democratic Republic of Congo, Indonesia, Canada, United States of America, and Mexico.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements of a 'qualified person' for the purposes of the Instrument.

(d) I last visited the property on March 21, 2017;

(e) I am responsible for the preparation of sections 15: Mineral Reserve estimate; 16: Mining Methods; 18: Project Infrastructure; 19: Market studies and contracts; 20: Environmental studies, permitting and social or community impact; 21: Capital and operating costs; 22: Economic analysis; 24: Other relevant information; 27: References; and the conclusions derived therefrom in sections 1: Summary; 25: Interpretation and conclusions; and 26: Recommendations.

(f) I am an employee and not independent of the issuer, Fortuna Silver Mines Inc.

(g) I have been an employee of Fortuna since July 2015 and involved with the property that is the subject of the Technical Report since August 2016.

(h) I have read NI 43–101, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

(i) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report that I prepared contain all the scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated at Vancouver, Canada, this 31st day of October 2017.

[signed]

Edwin Gutierrez, SME Registered Member

CERTIFICATE of QUALIFIED PERSON

(a) I, Geoff Allard, a consulting engineer and owner of Allard Engineering Services LLC, 6080 W Peregrine Way, Tucson, AZ USA; do hereby certify that:

(b) I am the co-author of the technical report titled Fortuna Silver Mines Inc. Lindero Property, Salta Province, Argentina dated October 31, 2017 (the "Technical Report").

(c) I graduated with a Bachelor of Science Degree in Chemical Engineering in 1977 from the University of Nevada, Reno USA. I have a Master of Science Degree in Metallurgical Engineering from the University of Nevada, Reno USA, granted in 1982. I am a registered professional engineer in the State of Arizona (49711), Nevada (8476) and New Mexico (21781). I am a Registered Member of the Society for Mining, Metallurgy and Exploration, Inc. (36750RM). I have practiced my profession continuously for 36 years, the most recent 12 of those years as an independent consultant. My relevant work experience includes the metallurgical investigation, design, construction and/or operation of over 25 heap leach processing operations in North and South America and Africa.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements of a 'qualified person' for the purposes of the Instrument.

(d) I have not visited the property.

(e) I am responsible for the preparation of sections 13: Mineral Processing and Metallurgical Testing and 17: Recovery (with the exception of sections 17.1.8 through 17.1.11 and 17.1.24); and the conclusions derived therefrom in sections 1: Summary; 25: Interpretation and conclusions; and 26: Recommendations.

(f) I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

(g) I have had no prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43–101, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

(i) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report that I prepared contain all the scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated at Tucson, Arizona USA, this 31st day of October 2017.

[signed]

Geoff Allard, PE

CERTIFICATE of QUALIFIED PERSON

(a) I, Denys Parra Murrugarra, General Manager of Anddes Asociados S.A.C., Av. Javier Prado Este Cdra. 48, Edificio Capital Golf, Piso 13, Surco, Lima 33, Peru; do hereby certify that:

(b) I am the co-author of the technical report titled Fortuna Silver Mines Inc. Lindero Property, Salta Province, Argentina dated October 31, 2017 (the "Technical Report").

(c) I graduated with a Bachelor of Science Degree in Civil Engineering in 1989 and Professional Title of Civil Engineer in 1991 from Universidad Nacional de Ingeniería, Lima Peru. I have a Master in Engineering Degree in Geotechnical Engineering from Pontifícia Universidade Católica, Rio de Janerio Brazil, granted in 1996. I am a registered professional engineer in Lima, Peru (CIP 42347). I am a Registered Member of the Society for Mining, Metallurgy and Exploration, Inc. (SME Registered Member Number 04222036). I have practiced my profession continuously for 26 years. My relevant work experience includes analysis and design of heap leach pads, tailings storage facilities, residues storage facilities, process ponds, water dams and other mining facilities in Peru, Chile, Brazil, Argentina, Colombia, México and Bolivia.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements of a 'qualified person' for the purposes of the Instrument.

(d) I have not visited the property.

(e) I am responsible for the preparation of subsections 17.1.8 to 17.1.11 and 17.1.24.

(f) I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

(g) I have had no prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43–101, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

(i) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report that I prepared contain all the scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated at Lima, Peru, this 31st day of October 2017.

[signed]

Denys Parra Murrugarra, SME Registered Member