Technical Report

On the Mineral Resources and Reserves of the Touro Copper Project

Located in Galicia, Spain

Prepared For

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Glossary of Geological Terms

Albite – The pure sodium-feldspar mineral; can be used as a glaze in ceramics.

Almandine –Mineral of the garnet group, it occurs in medium-grade metamorphic rock and felsic igneous rocks; used as a gemstone and an abrasive.

Amphibole – A group of ferromagnesian silicate minerals that occur as major or minor constituents in a wide variety of rocks. Amphiboles are common minerals in many types of igneous rocks and are important rock-forming minerals in many types of metamorphic rocks. They are particularly abundant in rocks of basaltic composition at most grades of metamorphism. Are divided into four main groups according to the chemistry.

Amphibolite – A class of metamorphic rock with one of the amphibole minerals as the dominant constituent. The features of the original rock are commonly obliterated, thus it is difficult and sometimes impossible to determine the premetamorphic rock.

Amphibolite facies – Metamorphic facies. It's a facies of medium pressure and average to high temperature. It is named after amphiboles that form under such circumstances.

Anticline – It's a type of fold that is convex-up, an arch-like shape and has its oldest beds at its core. The rock layers which form the anticline become progressively older toward the center of the fold

Antiform – is used to describe any fold that is convex up. It is the relative ages of the rock strata that distinguish anticlines from antiforms.

Aureole – A zone surrounding something. A circular or crescentic distribution pattern about the source or origin of a mineral, ore, mineral association, or petrographic feature.

Axial plane – A more or less planar surface that intersects a fold in such a manner that the limbs of the fold are symmetrically arranged with reference to it.

Back-arc basin – Back-arc basins are geologic basins, submarine features associated with island arcs and subduction zones. They are found at some convergent plate boundaries, presently concentrated in the western Pacific Ocean. Back-arc basins are typically very long and narrow.

Basalt – A general term for dark-colored mafic igneous rocks, commonly extrusive but locally intrusive (e.g., as dikes), composed chiefly of calcic plagioclase and clinopyroxene; the fine-grained equivalent of gabbro.

Biotite – Mineral of the mica group; a common rock-forming mineral in crystalline rocks, either as an original crystal in igneous rocks or as a metamorphic product in gneisses and schists; a detrital constituent of sedimentary rocks.

Cataclasis – Rock deformation accomplished by fracture and rotation of mineral grains or aggregates without chemical reconstitution.

Chalcopyrite – Mineral consisting of a sulfide of copper and iron. It is the most important source of copper.

Chlorite – Mineral which is associated with and resemble micas, they may also be considered as clay minerals when very fine grained. Chlorites are widely distributed, esp. in low-grade metamorphic rocks, or as alteration products of ferromagnesian minerals.

Clastic – Consisting of fragments of minerals, rocks, or organic structures that have been moved individually from their places of origin.

Clinozoisite – Mineral of the Epidote Group (calcium aluminum iron sorosilicate mineral).

Cummingtonite – Mineral of the amphibole group.

Fabric – The complete spatial and geometrical configuration of all those components (crystals, particles, cement) that make up a rock. It covers such terms as texture, structure, and preferred orientation.

Feeder zone – Stockwork or stringer zone in VMS deposits.

Felsic – Relating to or denoting a group of light-colored minerals including feldspar, feldspathoids, quartz, and muscovite.

Flysch sequence – It's a sequence of sedimentary rocks that are deposited in a deep marine facies in the foreland basin of a developing orogen. Flysch is typically deposited during an early stage of the orogenesis. It is called a syn-orogenic sediment (deposited contemporaneously with mountain building).

Fold – A geological fold occurs when one or a stack of originally flat and planar surfaces are bent or curved as a result of permanent deformation. Folds in rocks vary in size from microscopic crinkles to mountain-sized folds. Types of folds include intra-foliar, isoclinal, recumbent, open and tight.

Foliation – Refers to repetitive layering in metamorphic rocks. It's any penetrative planar fabric present in metamorphic rocks. It is caused by shearing forces or differential pressure.

Galena – It's the natural mineral form of lead sulfide. It is the most important ore of lead and an important source of silver.

Garnet (Garnet Group) – A group pf silicate minerals.

Garnetite – A metamorphic rock consisting of an aggregate of interlocking garnet grains, composed of more than 75% vol. garnet.

Gedrite – Mineral of the amphibole group.

Gneiss – A metamorphic rock with a banded or foliated structure, typically coarse-grained and consisting mainly of feldspar, quartz, and mica.

Graben – An elongate, relatively depressed crustal unit or block that is bounded by parallel faults on its long sides.

Greeshist facies – Metamorphic facies. It's at medium pressure and temperature. The facies is named for the typical schistose texture of the rocks and green color of the minerals chlorite, epidote, and actinolite.

Greywackes – It's dark coarse-grained sandstone containing more than 15 percent clay.

Halo – see: aureole.

Hinge – The hinge of a fold is where the flanks join together, is the point of minimum radius of curvature (maximum curvature) for a fold.

Hornblende – Mineral of the amphibole group.

Hydrothermal – Relating to or denoting the action of heated water in the earth's crust.

Iberian Massif – It's the core of the Iberian Peninsula consisting of a Hercynian cratonic block. The Variscan or Hercynian orogeny is a geologic mountain-building event caused by a Late Paleozoic continental collision between Euramerica (Laurussia) and Gondwana to form the supercontinent of Pangaea.

Kyanite – Aluminum silicate mineral, used in heat-resistant ceramics. A common rock-forming mineral in schist and gneiss.

Limb – The limbs are the flanks of a fold; the sides of the fold that dip away from the hinge.

Mafic – Relating to, denoting, or containing a group of dark-colored, mainly ferromagnesian minerals such as pyroxene and olivine.

Metabasite – A metamorphosed mafic rock that has lost all traces of its original texture and mineralogy owing to complete recrystallization.

Metagreywacke – A metamorphosed greywacke.

Meta-igneous – A metamorphic rock, the rock was first an igneous rock.

Metamorphic facies – It's a set of metamorphic mineral assemblages that were formed under similar pressure and temperature conditions of metamorphism.

Metamorphism – Process by which the rocks are converted to a new set of minerals with little or no change in bulk composition effected by temperature and pressure.

Metapelite – Metamorphosed mudstones and siltstones.

Metasediment – A type of metamorphic rock, the rock was first formed through the deposition and solidification of sediment.

Metasomatism – Process by which the bulk chemical composition of a rock is changed from some previous state by the introduction of components from an external source. It involves the import and export of chemical components through the agency of a chemically active fluid.

MORB – A mid-ocean ridge basalt. A mid-ocean ridge is a continuous, seismic, median mountain range extending through the North and South Atlantic Oceans, the Indian Ocean, and the South Pacific Ocean. It is a broad, fractured swell with a central rift valley and usually extremely rugged topography. According to the hypothesis of sea-floor spreading, the mid-ocean ridge is the source of new crustal material.

Muscovite – A silver-gray form of mica, a common rock-forming mineral in silicic plutonic rocks, mica schists, gneisses, and commercially in pegmatites; also a hydrothermal and weathering product of feldspar and in detrital sediments.

Mylonitic – Relating to or of the nature of mylonite.

Mylonite – A fine-grained metamorphic rock, typically banded, resulting from the grinding or crushing of other rocks. Produced by the extreme granulation and shearing of rocks that have been pulverized and rolled during overthrusting or intense dynamic metamorphism. Mylonite may also be described as a microbreccia with flow texture

Nappe – A sheetlike, allochthonous rock unit that has moved sideways over neighboring strata as a result of an overthrust or folding on a predominantly horizontal surface.

Nematoblastic –It's a metamorphic texture in which prismatic minerals such as sillimanite or amphiboles are orientated to produce a linear fabric.

Oligoclase – A feldspar mineral common in siliceous igneous rocks, consisting of a sodium-rich plagioclase.

Ophiolite – A group of mafic and ultramafic igneous rocks ranging from spilite and basalt to gabbro and peridotite, including rocks rich in serpentine, chlorite, epidote, and albite derived from them by later metamorphism, believed to have been formed from the submarine eruption of oceanic crustal and upper mantle material.

Paragneiss – A gneiss formed by the metamorphism of a sedimentary rock.

Pelite – A fine-grained sedimentary rock; an aluminous sediment.

Porphyroblast – It is a large mineral crystal in a metamorphic rock which has grown within the finer-grained groundmass.

Pressure/Strain shadows – The area in a metamorphic rock which is protected from deformation by the presence of a relatively rigid porphyroblast.

Prograde metamorphism – Metamorphic change resulting from an increase in temperature or pressure or both.

Protolith – The parent rock from which a given metamorphic rock developed.

Pyrite – Mineral consisting of iron disulfide.

Pyrrhotite – Mineral consisting of iron sulfide.

Retrogressive metamorphism – Metamorphic change resulting from a decrease in temperature or pressure.

Schist – A medium-grained strongly-foliated rock that can be readily split into flakes or slabs due to the well-developed preferred orientation of the majority of the minerals present.

Schistosity – The foliation in schist or other coarse-grained, crystalline rock due to the parallel, planar arrangement of mineral grains of the platy, prismatic, or ellipsoidal types, usually mica.

Shear zone – A tabular zone of rock that has been crushed and brecciated by many parallel fractures due to shear strain. Such an area is often mineralized by ore-forming solutions.

Siliciclastic – Relating to or denoting clastic rocks consisting largely of silica or silicates.

Sill – A tabular sheet of igneous rock intruded between and parallel with the existing strata.

Sphalerite – A mineral consisting of zinc sulfide.

Staurolite – Silicate mineral of aluminum and iron. A common accessory in medium-grade regional metamorphic rocks.

Stockwork – It's a complex 3D network of structurally controlled or randomly oriented veins. They are common in many ore deposit types. They are also referred to as stringer zones.

Subvolcanic – Pertaining to an igneous intrusion, or to the rock of that intrusion, whose depth is intermediate between that of deep plutonic and the surface.

Synsedimentary – That forms within a sediment during sedimentation.

Syn-tectonic – A geologic process or event occurring during any kind of tectonic activity, or of a rock or feature so formed.

Thrust – An overriding movement of one crustal unit over another.

Ultramafic – Igneous rocks that contain more than 90 vol-% mafic minerals.

Variscan Belt (of Europe) – A series of mountain ranges that developed during a span of time extending from 370 million to 290 million years ago (Variscan-Hercynian Orogeny). The Variscan orogenic belt extends in western Europe for more than 3,000 km from Portugal, Ireland, and England in the west through Spain, France, and Germany to the Czech Republic.

VMS – Volcanogenic massive sulfide deposits. They are a type of metal sulfide ore deposit, mainly copper-zinc which are associated with and created by volcanic-associated hydrothermal events in submarine environments. They are predominantly stratiform accumulations of sulfide minerals that precipitate from hydrothermal fluids on or below the seafloor in a wide range of ancient and modern geological settings.

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1 EXECUTIVE SUMMARY

1.1 Project Overview and Introduction

Atalaya Mining Plc. is a European mining and development company producing copper concentrate from the Riotinto deposit in southern Spain. Ore Reserve Engineering (ORE) has prepared the Technical Report for the Touro Project (the Project), a brownfield copper project located in the A Coruña province of the Galicia region, in northwest Spain (see Figure 1.1). The report was prepared using the headings of and guidance set out in, NI43-101F1.

1.2 Property Description and Location

The Project site is located in the A Coruña province of the Galicia Autonomous Region in north-western Spain. Santiago de Compostela is the nearest major regional center to the Project site with a metropolitan population of approximately 180,000. The project site is located approximately 17 km east of Santiago de Compostela and approximately 7 km east of the Santiago de Compostela international airport. The site is serviced by existing bitumen sealed roads approaching the site from various directions.

Approximate coordinates of the site in the European ETRS 89 survey grid are Northing 554,054 m and Easting 4,747,955 m (Universal Transverse Mercator [UTM] Zone 29N).

The Project consists of a series of deposits of which 4 have partially been mined previously (Arinteiro, Vieiro, Bama and Brandelos) and 2 that have not been mined previously (Monte de las Minas and Arca). The Project site comprises an influence area of approximately 1,060 ha. Current mineral rights include approximately 15,300 ha of contiguous ground (including a mining concession).

Figure 1.1– Touro Project Location (Atalaya 2017)

The Project is located in a well-developed and accessible area. The local climate is transitional between an Atlantic climate and a mild form of Continental climate. The Project Site is located at an average elevation of approximately 315 meters above sea level (masl), with an elevation of 157 masl at the lowest and 445 masl at the highest point.

The Project is located in an area with a well-developed infrastructure network of roads, rail system, airport and ports including A Coruña, El Ferrol, and Vilagarcia de Arousa 45 km south of Santiago de Compostela.

1.3 Touro Copper Project Area

Atalaya has entered into a purchase option agreement to acquire a majority interest in the Project. The option agreement gave Atalaya an exclusive option to purchase an 80% interest in the Project.

Following the acquisition of the initial 10% of Cobre San Rafael S.L. share capital, the Agreement includes the following four phases:

- Phase 1 – Atalaya paid €0.5 million to secure the exclusivity agreement and will continue to fund up to maximum of €5 million to get the project through the permitting and financing stages

- Phase 2 – When permits are granted, Atalaya will pay €2 million to earn-in an additional 30% interest in the project (cumulative 40%)

- Phase 3 – Once development capital is in place and construction is underway, Atalaya will pay €5 million to earn-in an additional 30% interest in the project (cumulative 70%)

- Phase 4 – Once commercial production is declared, Atalaya will purchase an additional 10% interest in the project (80% cumulative) in return for a 0.75% Net Smelter Return (NSR) royalty, with a buyback option.

The Agreement has been structured so that the various phases and payments will only occur once the project is de-risked, permitted and in operation.

Mineral resources identified to date are located in the San Rafael Mining Concession. The claims are map-staked using digital maps, and the claims do not overlap each other. The main requirements to keep the claims in good standing are:

- to comply with the approved work plans;
- to comply with the payments; and
- to apply for the extension/renewals on time.

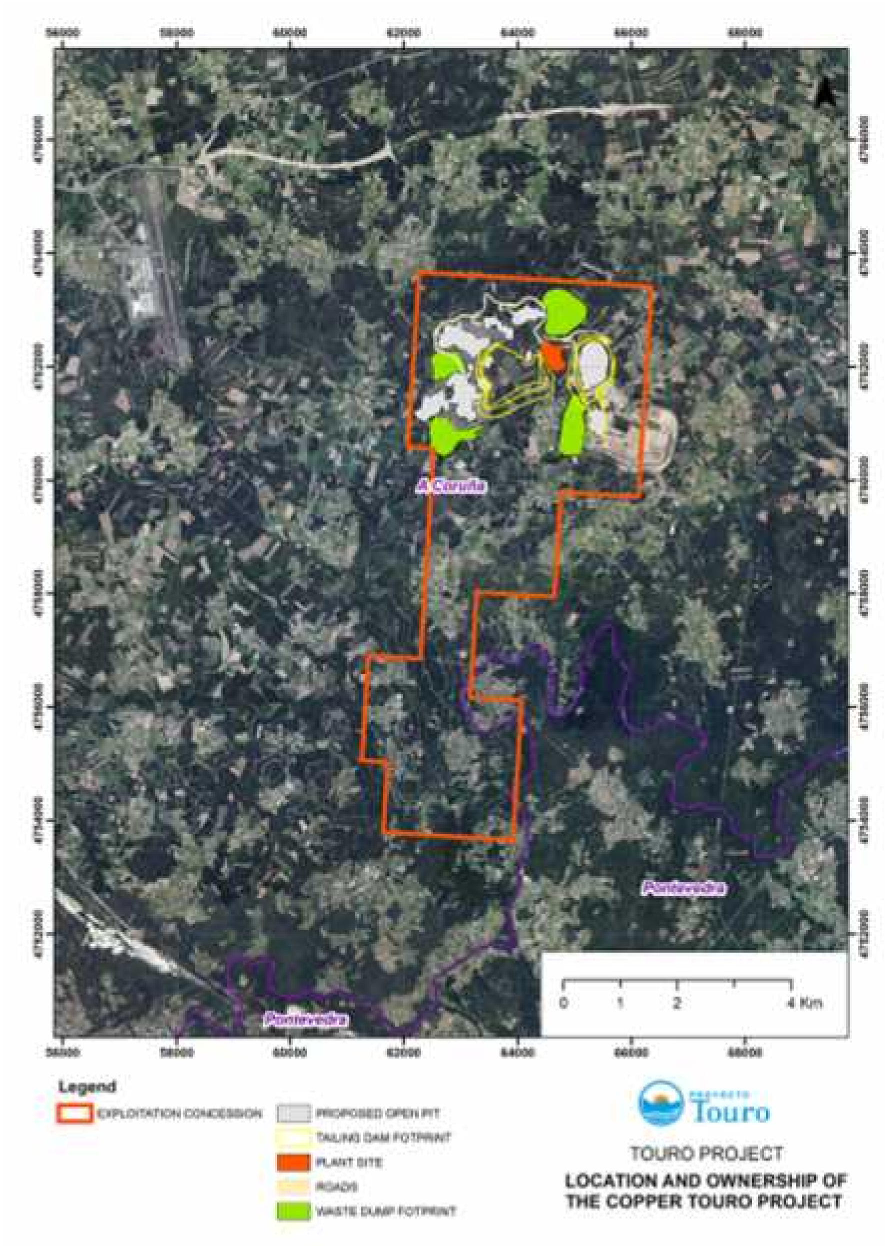


Figure 1.2 – Location and ownership of the Touro Copper Project (Atalaya 2017)

1.4 History

Historically, the Project was explored and operated between 1970 and 1986 by Rio Tinto Patiño (RTP). RTP commenced commercial production in 1973 and closed the operation in 1986 due to low copper prices. The deposits that were mined by RTP included Vieiro, Arinteiro, Bama and Brandelos (Figure 1.3). Following the mine closure, RTP sold the San Rafael Exploitation Concession to Explotaciones Gallegas S.L. who has kept the mining permits in place.

During the 13 years of mine operations, 21 Mt of ore at an average grade of 0.61% Cu was extracted from four primary open pits; Arinteiro, Vieiro, Bama, and Brandelos.

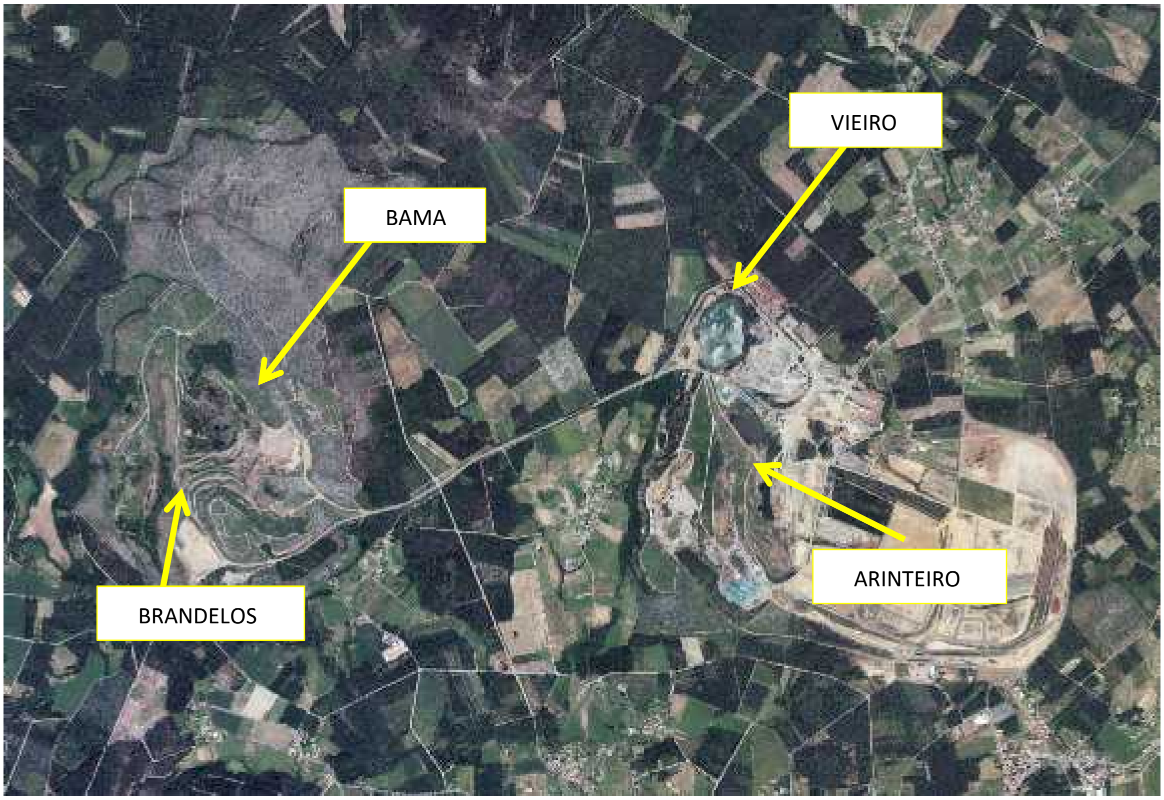


Figure 1.3 – Historical Pit Locations (Atalaya 2017)

The ore zone consists of one or two mineralized horizons of disseminated mineralization closely associated with coarse-grained garnet amphibolite that is poor in calcium. Garnet content is up the 90% of the volume of the rock (garnetite). The garnet amphibolite grades into non-mineralized normal amphibolite with decreasing garnet content.

The mineralization is represented in order of abundance by pyrrhotite, chalcopyrite and minor pyrite and sphalerite. Williams (1983) described evidences of pre- and syn-deformation in garnets and sulphides (pyrrhotite and chalcopyrite).

The sulphides in the amphibolite are mainly disseminated and aligned with foliation planes, are either interstitial to amphibole crystals, infill the cracks of garnet porphyroblasts together with quartz, concentrated in pressure/strain shadows, or form veinlets parallel to the rock schistosity, with quartz, chlorite, and carbonates.

Weathering effects are limited to very near surface, not more than a few meters deep. Disseminated-style mineralization is most common, but there are significant zones of massive to semi-massive sulphides (5 to 10 m thick) particularly at Arca and Monte de las Minas.

- disseminated sulphides occur in different forms: along foliation and schistofsite-cisaillement (S-C) planes, on pressure shadows, extensional joints and late remobilizations;

- grain size is relatively coarse, mineralogy simple and sulphides relatively free of impurities; and
- higher grade zones are not always related to massive/semi-massive sulphides.

Copper mineralization (pyrrhotite<chalcopyrite<pyrite<sphalerite) is usually associated with chlorite alteration. More distal alteration types include epidote, carbonates, sericite, and chlorite.

1.5 Geology and Reserves

The Project is located in the NW of the Iberian Massif, a sector of the Variscan Belt of Europe, and includes the Spanish regions of Galicia and the Cantabrian Mountains, as well as northern Portugal. The northwestern Iberian basement consists of plutonic and metamorphic rocks and a clear separation can be established between autochthonous and allochthonous terranes. The autochthon consists of a thick metasedimentary sequence whereas the allochthon consists of the remnants of a huge and structurally complex nappe pile preserved in the core of late Variscan synforms. Both are separated by a thrust sheet, several kilometers thick, consisting of metasediments and volcanic rocks (Martinez Catalan, 2007). There are three allochthonous complexes in Galicia (Cabo Ortegal, Ordenes, and Malpica-Tui), and two in northern Portugal (Bragança and Morais).

The Project area is located in the High-Pressure/High-Temperature Upper Units of the SW sector of the Ordenes Complex, as shown in Figure 1.4Figure 1.4. Within the Ordenes Complex, copper mineralization is associated with the Arinteiro Unit which contains the metabasites hosting the Touro group and Fornás-Mañoca orebodies.

The composition and grain size of the metabasites (amphibolites) is very variable but mostly contains amphiboles (hornblende), garnet (almandine), pyroxene, quartz, and biotite.

The metabasites are interbedded with metasediments (paragneisses that were pelites and greywackes in origin) of the O Pino Unit. The composition of these metasediments is predominantly kyanite-staurolite-garnet-two mica schist and quartz-plagioclase-biotite-garnet gneiss (Castiñeiras, 2005). Of minor occurrence, ultramafic rocks and thin bands of quartz are in places interlayered with the metabasites.

According to Serrati et al (2002), the Ordenes Complex hosts two different types of sulfide mineralization, both closely concordant with the schistosity. One type is represented by the Arinteiro and Bama orebodies, with pyrrhotite and chalcopyrite disseminated in a coarsely grained garnet amphibolite. The other type, which occurs a few kilometers westwards at Fornás and Mañoca, are massive sulfides composed by massive pyrrhotite, chalcopyrite and scarce pyrite and sphalerite.

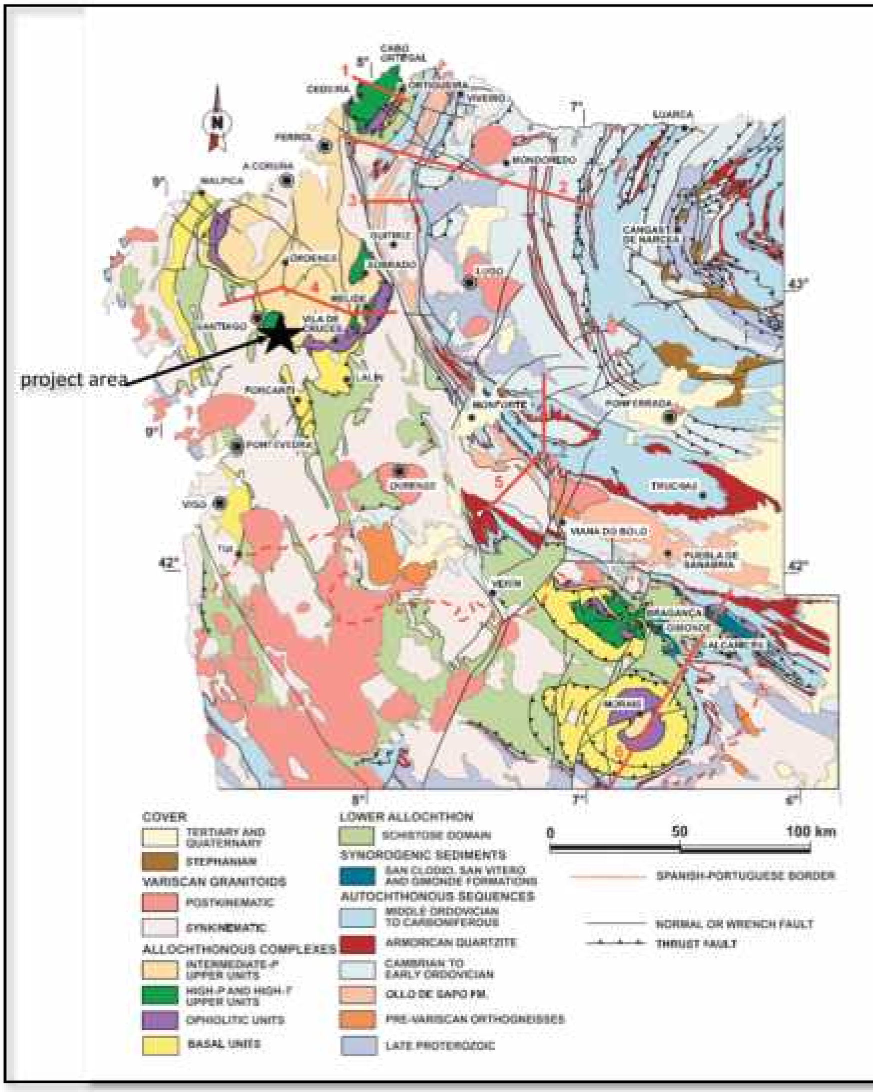


Figure 1.4 - Geological sketch map of northwestern Iberia, showing the allochthonous complexes and their units. (Source: Martinez Catalan 2007)

1.6 Deposit Types

Copper mining in the Project area started in the 1970's with Riotinto Patino (RTP) in the Arinteiro deposit. Several other deposits were discovered afterwards and mining continued in the 1980's on the Vieiro, Bama and Brandelos deposits. Further exploration by RTP during the 1980's and by Lundin Mining in 2012 extended the mineralization towards the north to the Arca and Monte das Minas deposits that remain unmined. Atalaya Mining further subdivided each deposit in different ore zones (Table 1.1 and Figure 1.5).

Within each of these deposits, the style of the mineralization is similar, occurring as elongated ore zone-lenses.

The local geology of the project has been reviewed by the Exploration Department of Atalaya Mining in 2017 based on field mapping observations and a set of geological sections produced from the drilling data. In total, 81 east-west cross sections, approximately each 100m apart and 4 north-south cross sections have been completed. An updated geological map based on the interpreted cross sections is presented in Figure 1.6.

Table 1.1 - Ore zones within each of deposits, Touro Project (provided by Atalaya Mining)

DEPOSITS	ORE ZONES
Arinteiro	Arinteiro
Vieiro	Vieiro, Middle Vieiro, Upper Vieiro and Vieiro NW
Brandelos	Brandelos and Brandelos N.
Bama	Bama, Lower Bama and Bama E.
Arca	Arca, Arca S, Lower Arca E and Upper Arca E.
Monte das Minas	Monteminas, Lower Monteminas, Monteminas S, Upper Monteminas W and Monteminas N

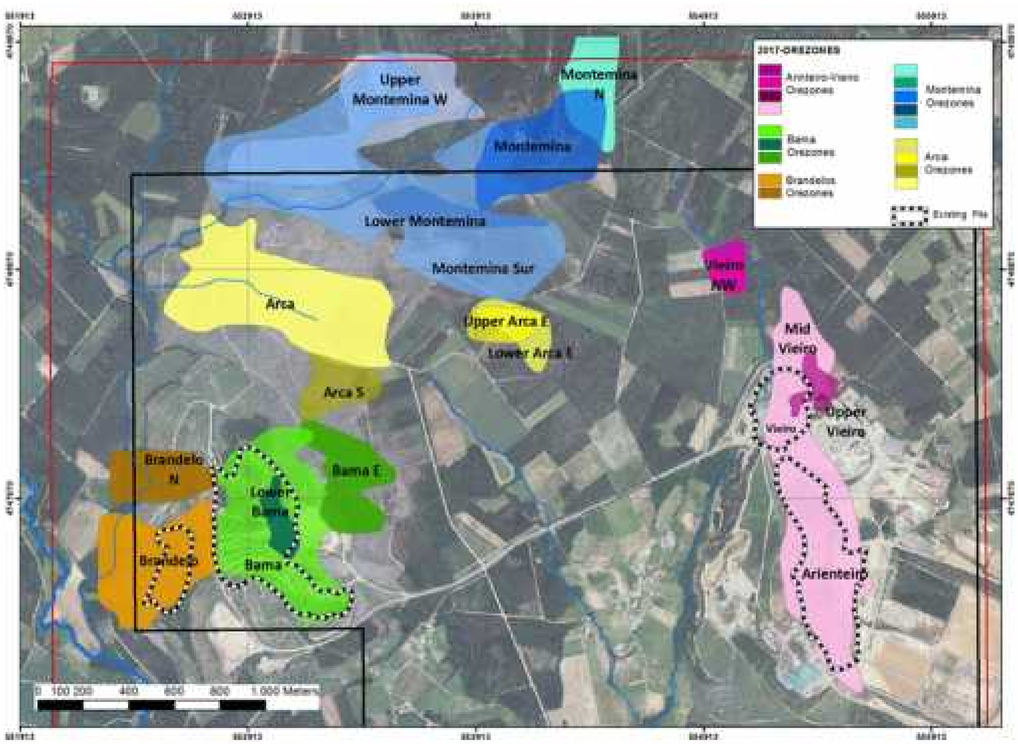


Figure 1.5 – Location of the different ore zones in the deposit, Touro Project (Atalaya 2017)

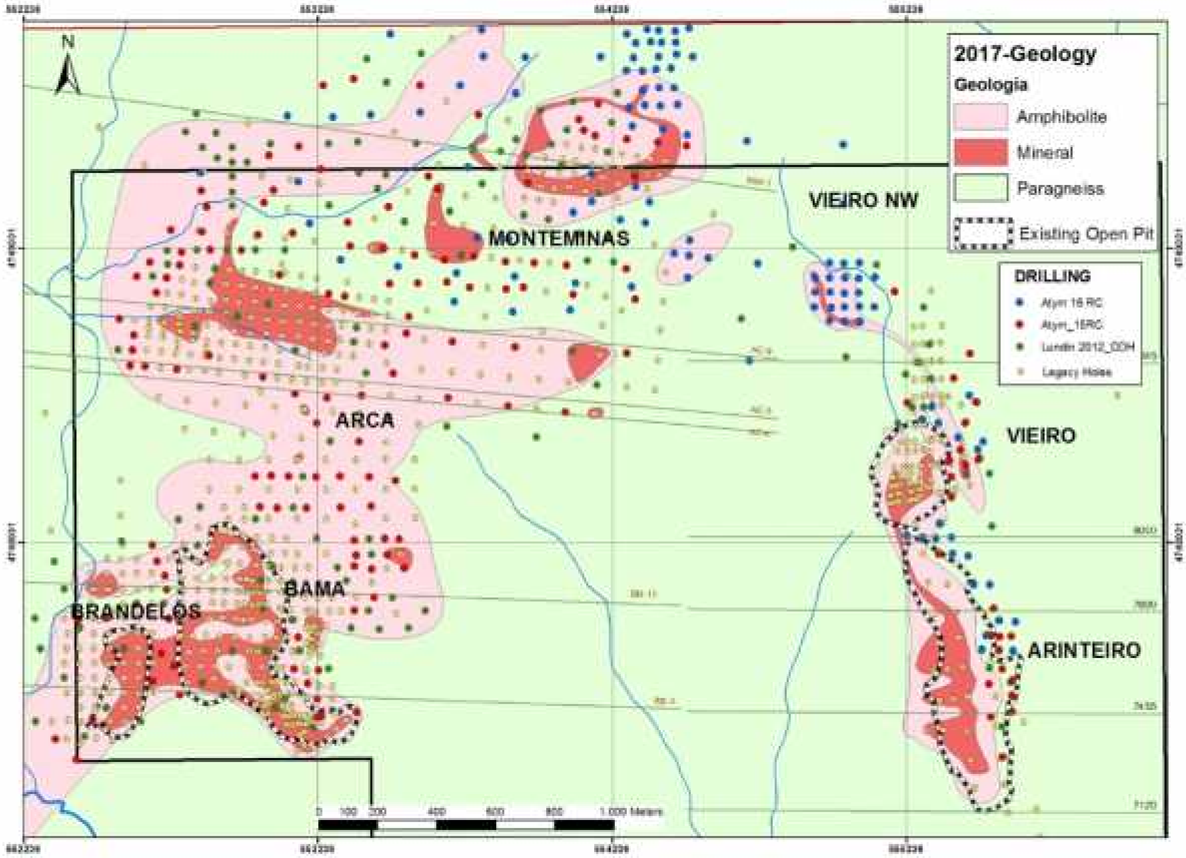


Figure 1.6 – Updated Geological map (Atalaya 2017)

The deposits of Arinteiro, Bama, Fornás and Manoca have been characterized as metamorphosed ophiolitic VMS deposits (Badham and Williams 1981; Williams 1983) and classified as Cu or Cu–Zn type (Castroviejo, 2001). More recent studies suggest an origin related with syn-metamorphic metasomatism channeled through shear zones (Castiñeiras et al., 2002; Gomez Barreiro, 2007).

In Atalaya's opinion, the deposit can be classed as Mafic Siliciclastic type (Besshi-type) Volcanogenic Massive Sulfide (VMS) deposit, according with the lithological classification of Shanks and Thurston and equivalent to the pelitic–mafic VMS deposits of Galley.

1.7 Drilling and Exploration

The Project went through several stages of geological exploration undertaken by different mining companies for a period of over 45 years.

Exploration started in the region in the 1960s by Rio Tinto (RTP and RTM) with activities in San Rafael and other mining permits in the vicinity of the current Touro Project (Maldonado, 2012). RTM undertook a comprehensive exploration program that included:

- Geological mapping (regional and local scale).
- Geophysical exploration: airborne and ground electromagnetic and magnetometer surveys (EM electromagnetic Gun survey, ground Turam method).
- Soil geochemistry programs.
- Shallow exploration by trenching and Voletrac drilling.

- Diamond core drilling.

Lundin Mining completed a due-diligence program in 2012. Exploration was carried out in both the current project area (San Rafael) and in Fuente Rosas to the west of the project area, including geological mapping and diamond core drilling. Outside the current project and Fuente Rosas, Lundin Mining compiled available exploration information produced by Rio Tinto between 1965 and 1986 (Maldonado, C., 2012).

Since 2015, Atalaya Mining has been focused on San Rafael mining permit. Exploration includes legacy data gathering, integration of legacy spatial data into Geographic Information Systems (GIS), geological mapping of the San Rafael permit, diamond core and reverse circulation drilling (15RC and 16RC drilling programs) and soil geochemistry over limited geographic areas.

Atalaya Mining began its drilling program in Touro in November 2015, completing the first stage in May 2016 (15RC program).

The program was initially designed as a fully reverse circulation (RC) drilling campaign with two RC drilling rigs on site (Atlas Copco Mustang 5F4 and 5P4). In November 2015, a diamond drilling (Spidrill 160D) rig joined the program to improve drilling capabilities and efficiency in the deepest holes and to increase recovery in the presence of water. Therefore, switching from reverse circulation to diamond drilling was necessary for some of the holes. In February 2016, a third RC drilling rig was also mobilized in order to speed up the program.

RC drilling has been the main drilling method employed by Atalaya Mining since 2015. RC was generally carried out using Atlas Copco Mustang 5F4 and 5P4 drill rigs using bits ranging in diameter from 14.0 to 14.6 cm (5 ½ to 5 ¾ inches). RC drilling was performed dry to ensure proper movement of drill cuttings through the drill stem, cyclone and sampling equipment, and for dust control. Drilling campaigns 15RC and 16RC were completed by drilling contractor SPI. When the RC drilling encountered groundwater in the deeper holes, the method was changed to core drilling. Atalaya core holes are also drilled by SPI using a Spidrill drill rig and HQ (63.5 mm) size core.

A total of 152 holes were completed of which 124 were RC holes, 24 were combined RC/DD, and 3 were DD holes. In total, 14,278 meters of drilling were completed, of which 12,230 meters were RC and 2,048 meters were DD. The average bore hole depth was 94 meters.

Most of the completed holes were infill holes (to gain confidence in the ore zone continuity) as well as step-out holes (to expand the extension of the orebody). Additionally, five "twin-holes" were drilled, consisting of two DD holes to check RC grades and three twin RC holes to validate historical drill hole grades. In addition, PVC casing was installed in twenty of the completed holes for water level monitoring purposes.

The second campaign of the Atalaya Mining drilling program began in October 2016 (16RC program) and ended in April 2017. The purpose of this program was also infill and step-out drilling. A total of 120 drill holes were completed with 93 RC holes, 1 DD hole, and 26 combined RC/DD holes, totalling 10,838.5 meters of RC and 2,443.3 meters of DD. Average drill hole depth was 95 meters.

1.8 Mineral Resource

The resource model was created as a three-dimensional block model using Datamine Studio 3 software. The model block size is 10x10x10 meters, which is consistent with the proposed mining bench height and the estimated selective mining unit. The horizontal extent of the model is defined to cover the entire resource and ownership area, plus sufficient space outside the deposits to cover the ultimate pit. Resource model size and location parameters are shown in Table 1.2.

Table 1.2 - Resource Model Size and Location Parameters

	Minimum (ETRS meters)	Maximum (ETRS meters)	Cell Size (meters)	Number Cells	Model Size (meters)
Easting (X)	551,700	556,100	10	440	4,400
Northing (Y)	4,746,600	4,749,950	10	335	3,350
Elevation(Z)	-100	500	10	60	600

Key items included in the block model are copper grade, copper grade zone, density, trend-indexed Z coordinates, oxidation codes, resource classification codes, and codes indicating whether a block is mined out, backfill, or rock. Copper grade was estimated using inverse-distance-power estimation.

The copper resource was summarized using a Lerches-Grossman pit shell that was run using a copper price of $3.20/lb Cu and all resources including inferred resources. All other slope and economic parameters are the same as those used for design of the open pit for reserve estimation. The resulting pit shell is considered to have reasonable prospects for economic extraction, assuming that the inferred resource is converted to measured and indicated by drilling, and that the copper price returns to previous levels that were substantially above $3.20/lb Cu. The resource estimate is summarized in Table 1.3.

Table 1.3 - Touro Project - Resource Summary-Constrained by the $3.20/lb Cu Pit

	>= 8.14 NSR $/t (Internal Cutoff)				>= 9.71 NSR $/t (Breakeven Cutoff)			
Resource Class	ktonnes	NSR $/t	Cu%	RCu%	ktonnes	NSR $/t	Cu%	RCu%
Measured	69,258	22.55	0.42	0.37	67,886	22.82	0.42	0.37
Indicated	60,592	19.24	0.36	0.31	59,188	19.49	0.37	0.32
Measured + Indicated	129,850	21.00	0.39	0.34	127,074	21.27	0.40	0.35
Inferred	46,521	19.33	0.37	0.32	45,822	19.48	0.37	0.32

1.9 Mineral Reserves & Mining

Mining at Touro will use conventional, open pit methods working from 10-m high bench faces. Atalaya Mining anticipates using contractors for all mining work, including drilling and blasting, at the project site. Contractors' small- to medium-scale mining equipment will likely include: rock drills capable of drilling 102- to 127-mm-diameter blastholes, hydraulic excavators and/or front-end loaders with bucket capacities of 6-13 m^3, off-highway trucks with 55- to 91-t payload capacities, and suitably sized support equipment. Total mining personnel, contractor plus Atalaya Mining, is estimated at 101-111 for the peak total material production period of Years 1-4.

Seven open pits, based on a Cu price of $2.60/lb, will be developed around the arc of mineralization at Touro: Arca, Arinteiro-Vieiro, Bama, Brandelos, and three separate pits in the Monte de las Minas area. Five of these pits were subdivided into two pushbacks (phases) each, with the intent of improving mill head grades in the first five years. A total of 12 mining phases were designed from which a mine production schedule was estimated using variable net smelter return (NSR) cutoff grades ranging from $10.00/t to $14.00/t through Year 5 and $8.14/t (the internal cutoff) thereafter. Typically, 4-5 phases will be worked at some point in each year to manage ore blending, sinking rates, and stripping ratios; only 2-3 phases will be active at any given time. The schedule targets an annual copper production rate of approximately 30,000 t after recoveries. Following an initial ramp-up and debugging period in Year 1, ore processing rates will average about 5.8 Mt/a through Year 5, then increase in stages to offset declining head grades until reaching a maximum throughput rate of 10.0 Mt/a in Year 9 and thereafter. About 3.0 Mt of low grade ore above an NSR cutoff of $9.00/t will be stockpiled through Year 5 and then reclaimed as mill feed in Years 11 and 12. The resulting mine production schedule is presented in Table 1.4.

Table 1.4 – Mine Production Schedule

Time	Cutoff	Direct Mine to Plant			To ROM Stockpile			To LG Stkpl >= 9.00 NSR			From LG Stkpl to Plant			Total Plant Feed			Waste	Total	Strip
Period	NSR $/t	Ktonnes	NSR $/t	Cu%	Ktonnes	NSR $/t	Cu%	Ktonnes	NSR $/t	Cu%	Ktonnes	NSR $/t	Cu%	Ktonnes	NSR $/t	Cu%	Ktonnes	Ktonnes	Ratio
PP	10.00				449	21.23	0.49	12	9.63	0.24							35,201	35,662	999.99
1*	10.00	5,100	24.13	0.55				184	9.52	0.23				5,100	24.13	0.55	29,646	34,930	5.81
2	10.00	6,000	23.08	0.53				312	9.46	0.23				6,000	23.08	0.53	28,618	34,930	4.77
3	14.00	6,000	23.04	0.53				1,488	11.50	0.27				6,000	23.04	0.53	23,762	31,250	3.96
4	12.00	5,700	25.60	0.58				852	10.60	0.25				5,700	25.60	0.58	8,448	15,000	1.48
5	10.00	5,500	27.29	0.61				172	9.55	0.24				5,500	27.29	0.61	9,128	14,800	1.66
6	8.14	7,200	20.34	0.48										7,200	20.34	0.48	9,300	16,500	1.29
7	8.14	8,500	16.99	0.40										8,500	16.99	0.40	13,850	22,350	1.63
8	8.14	9,000	16.27	0.39										9,000	16.27	0.39	13,850	22,850	1.54
9	8.14	10,000	14.10	0.34										10,000	14.10	0.34	13,850	23,850	1.39
10	8.14	10,000	14.43	0.35										10,000	14.43	0.35	15,700	25,700	1.57
11	8.14	8,797	14.28	0.34							1,203	10.79	0.26	10,000	13.86	0.33	15,646	25,646	1.56
12	8.14	5,725	16.15	0.39							1,817	10.79	0.26	7,542	14.86	0.35	4,180	11,722	0.55
13	8.14	364	14.66	0.35										364	14.66	0.35	150	514	0.41
Total		**87,886**	**18.68**	**0.44**	**449**	**21.23**	**0.49**	**3,020**	**10.79**	**0.26**	**3,020**	**10.79**	**0.26**	**90,906**	**18.42**	**0.43**	**221,329**	**315,704**	**2.43**

Notes: NSR values are in US dollars base on a Cu price of $2.60/lb.
* Includes ROM stockpile reclamation in Y1 direct mine to plant.

Table 1.5 summarizes the mineral reserve estimates by classification at a Cu price of $2.60/lb using the cutoff grades by time period listed above.

Table 1.5 – Mineral Reserve Estimates by Classification

	Mineral Reserves		
Classification	**kt**	**Cu (%)**	**RCu (%)**
Proven	56,769	0.44	0.39
Probable	34,137	0.41	0.36
Total	**90,906**	**0.43**	**0.38**

Total proven and probable mineral reserves at $2.60/lb Cu price are estimated at nearly 91 Mt grading 0.43% Cu. Contained copper is estimated at nearly 392,000 tonnes. Waste rock tonnages are projected at about 221 Mt, resulting in an average stripping ratio of 2.43. Mine life is projected at just over 12 years, excluding the preproduction stripping period. All of the mineral reserves are contained within the estimates of mineral resources.

The effective date of the mineral reserve estimate is 1 September 2017.

1.10 Mineral Processing and Recovery Methods

A rigorous testwork program was completed at SGS Australia in Perth, Western Australia to provide inputs to the Touro Technical Report. Eighty-eight samples were selected from drill core produced by the 2012 campaign undertaken by Lundin Mining for use in the testwork program. From the 88 samples, 12 composites were generated based on identified ore types. A high-level sample summary is detailed in Table 1.6.

Table 1.6 - Sample Summary

Composite	No. Samples	Orebody	Sample Grades			
		Avg Cu (%)	Avg Cu (%)	Avg S (%)	S:Cu	Mass (kg)
Arca A	4	-	0.35	6.65	19.0	75.8
Arca D	2	-	0.52	4.50	8.65	55.4
Arca Main	9	0.36	0.41	10.3	25.1	232
Arinteiro	6	0.54	0.50	4.47	8.94	287
Bama	14	0.37	0.42	3.48	8.29	347
Brandelos	10	0.37	0.40	4.66	11.7	308
Monte Minas Garnetite	8	0.51	0.48	8.51	17.7	209
Monte Minas Paragneiss	5	-	0.51	7.43	14.6	128
Monte Minas Upper	6	-	0.47	5.67	12.1	149
Vieiro	8	0.59	0.60	4.80	8.00	325
Vieiro High Grade	4	-	1.27	5.63	4.43	152
Vieiro Hardest Sample	1	-	0.02	1.10	55.0	42.3
Not Composited	11	-	-	-	-	258
Total	**88**	**0.46**	**0.50**	**5.60**	**11.3**	**2,567**

Flowsheet development was based on the testwork reported by SGS to produce a concentrate grade of 27% Cu at 90% recovery. Locked cycle tests completed during the latest testwork program produced concentrates that averaged 29.1% Cu at 87.0% recovery indicating that this is achievable. Refer to Chapter 13 for details and results of the testwork program.

The proposed process flowsheet uses a conventional SAG mill - ball mill (SAB) grinding circuit followed by a copper flotation recovery circuit. The initial concentrator includes:

- Primary crushing.
- Primary and secondary grinding.
- Rougher flotation.
- Regrinding.
- Three stages of cleaner flotation.
- Concentrate thickening and filtration.

The concentrator and associated service facilities will process run-of-mine (ROM) ore as delivered to the ROM pad to produce a dewatered copper concentrate and tailings slurry. The preliminary mining schedule is outlined as follows:

- In the first five years of operation, the peak annual ore movement to the plant is 5.9 million tonnes, which occurs in year 2.

- In the following two years of operation, the peak annual ore movement to the plant is 6.7 million tonnes, which occurs in year 6.
- From year 8, the annual ore movement to the plant increases to 9.3 million tonnes.
- Over the life of mine, the peak annual ore movement to the plant is 10 million tonnes, which occurs in year 9.

To avoid three successive plant upgrades, a simplified approach has been taken with only two design points and upgrades considered for the process plant:

- Initial Phase 1 plant throughput of 6.0 Mt/y.
- Phase 2 upgrade to increase plant throughput of 10 Mt/y prior to year 8.

The Phase 1 average head grade is 0.53% Cu with a peak of 0.57% Cu. The Phase 2 average head grade is 0.35% Cu with a peak of 0.38% Cu. The overall result is consistent concentrate and copper production for both phases.

1.11 Infrastructure

The Project is supported by an adequate network of public roads that will also be used to transport concentrate in single trailer trucks to the existing port of Vilagarcia de Arousa. The Vilagarcia port also has adequate existing bulk materials storage, handling and ship loading facilities. Upgrading of these off-site roads will not be required to support the Project

The concentrator location has been based on an assessment of the open pit distances and geographical features. A preliminary location situated approximately 500 m to the north-west of the current Vieiro and Arinteiro deposits has been used for the study. This is adjacent to the proposed Tailings Management Area (TMA) and the pond from which plant water will be drawn. It is also conveniently located to connect to the existing 220 kV power line to the south of the mine which will supply power to the new facilities.

The Project will utilize in-pit tailings disposal for a substantial part of the mine life due to the multiple pit mine operating plan. During the initial years of operation, tailings will be stored in a surface Tailings Management Facility (TMF). After year 8, tailings will be stored in the exhausted Vieiro and Arinteiro open pits.

Tailings production is estimated at 91 million tonnes during the 13 years of operation of the concentrator. The tailings will have an initial solids content of 35% and are expected to be potentially acid generating (PAG). The surface TMF will be plastic lined and have a capacity for 44 Mt of tailings and the Vieiro-Arinteiro TMF will store 47 Mt of tailings.

A tailings thickening system will be implemented downstream of the concentrator to produce a total of 91 Mt of thickened tailings with a final 67% solids content. This will ensure physical and chemical stability of tailings while reducing the size of the dam wall required for the surface TMF. This technology will also achieve greater densities and deposition slope angles, and thus increase the storage capacity while reducing the footprint and the quantiy of fill required for dam construction. Furthermore it will mitigate seepage and can help to control acid generation as a result of a lower hydraulic conductivity and oxygen transmissivity.

The nominal power demand for the operation is approximately 25 MW. Power for the Project will be supplied from the Portodemouros electrical substation, owned by UNÍON FENOSA electric distribution company via the nearby 66 kV power lines. A 66 kV to 6.3 kV transformer and associated switch gear will be located in a substation at the tie-in point from the main 66 kV line. A new 12 km 6.3 kV overhead power line will be constructed from the Municipality of Vila de Cruces (A Coruña) to the Project site.

All water recovered through the various drainage and seepage control systems is piped to the recovered water pond. This pond is located near the process plant where the water will primarily be pumped to the plant for reuse or treated until it complies with the required environmental levels for discharge.

Similarly, the water recovered from the plant tailings storage operational lagoons will be recovered by barge pumps that will deliver the water to the recovered water pool to be re-used as process water.

An intercept trench has been designed that will collect all of the Project run-off water. The trench starts at the head of the PAG waste rock facility and runs north to south along the slope situated to the east of the Vieiro and Arinteiro deposit. The discharge is downstream from the old plant tailings basin.

Potable water for the Project will be supplied from the local water distribution pipe network. The connection point is 1.5 km from the site. Potable water will be stored in a 100 m^3 tank and distributed to the respective users via the potable water pumps and piping network. The estimated potable water consumption will be 270 litres per person per day, which includes drinking water and sanitary use. Safety showers will also be supplied with potable water.

A 1,013 m^2 mine workshop has been designed for mine equipment fleet maintenance, and is equipped with one office. It will be able to accommodate the larger mine haul trucks as well as the support equipment. It will be equipped with a bridge crane as well as auxiliary machinery owned by the mine contractor (truck crane, forklift and lifting platform) in order to perform maintenance. In addition, the system will have a truck wash area with a physical-chemical treatment plant.

1.12 Market Studies and Contracts

Atalaya has been actively marketing the copper concentrate product from the Riotinto Mine to global consumers since 2016. The copper concentrate to be produced at Touro is expected to have high grade copper content and to attract premiums in international markets due to its very low deleterious elements.

Copper is an internationally traded commodity and prices are set through trading on the major metals exchanges: the London Metal Exchange (LME), the New York Commodity Exchange (COMEX) and the Shanghai Futures Exchange (SHFE). Copper prices on these exchanges generally reflect the worldwide balance of copper supply and demand, but are also influenced significantly by investment flows and currency exchange rates.

Copper concentrate, one of the many copper products, is generally sold through long- term contracts to smelters and refineries on a competitive basis. Atalaya expects that terms contained within any sales contract that could be entered into would be typical of, and consistent with, standard industry.

1.13 Environmental Studies, Permitting, and Social or Community Impact

The Touro copper deposit was explored and mined extensively between 1970 and 1986 by Rio Tinto Patiño, (RTP). Rio Tinto Patiño started commercial production in 1973 and ceased operations in 1986 due to the fall in the price of copper.

Most of the historical mining activity occurred near four pits. The Arinteiro and Vieiro pits, the old plant tailings deposit and Arinteiro waste rock facilities are located to the East of the site; both the tailings deposit and the waste rack facilities are currently partially reclaimed. The Bama and Brandelos pits and the future Monte de las Minas and Arca pits are located in the Western sector of the site. Another old waste rack facility, also reclaimed, is located to the south of Brandelos.

Following the cessation of mining, the Arinteiro pit has become partially flooded. The settling ponds and waste dumps are subject to on-going reclamation work. The tailings settling ponds from the processing plant, which has been silting for years, were restored using local soil stockpiles. The southern area has become an industrial park which is already reclaimed and ready to begin operations.

The Bama and Brandelos pit waste dumps are in the reclamation process, although in different phases. There is already a waste dump in this area where the reclamation work has led to well-developed and consolidated vegetation.

In Spain, there are typically three different types of mining permits and concessions:

- Exploration permits (Art. 40.2 Mining Law) granted for a period of 1 year may be extended for a maximum of one more year.
- Investigation or Research permits (Art. 45 Mining Law) granted for the period requested, which may not be more than 3 years and may be extended twice for a further 3 years.
- Operating concessions (Art. 62 Mining Law) also referred to as a Mining Permit, granted for a 30-year period, and may be extended for equal periods up to a maximum of 90 years.

In general, the Exploration and Investigation Permits or the Operating Concession does not grant the surface rights. These must be purchased or leased from the surface rights owner. The Spanish and Galician mining laws state that the owner of an investigation permit must comply with the approved annual work plans. For this reason, the law determines that if a friendly agreement is not reached with a landowner, it is obligatory to initiate a temporary occupation process.

For an Operating Concession, the same principle is applicable, where agreements with local surface right owners is required but a forced expropriation process can be resorted to, if necessary. There are no royalties on the property.

At the national level, the environment is administered by the Ministerio de Agricultura, Alimentación y Medio Ambiente. The Ministerio de Agricultura, Alimentación y Medio Ambiente is responsible for setting policy and enacting into legislation EU policy. At the regional level, the environment is administered by the Consejería de Medio Ambiente y Ordenación del Territorio. The Consejería de Medio Ambiente y Ordenación del Territorio is responsible for ensuring that national policy is implemented and also has auditing responsibilities. Additionally, the Consejería de Medio Ambiente y Ordenación del Territorio has authority to issue environmental permits.

A series of steps will be undertaken during the development phase to prepare the ground and build the facilities and infrastructure needed for the Project. These will include: clearing and grubbing vegetation, removal and storage of topsoil, construction of stockpiles, ground conditioning and excavation, paving, construction of entrance routes, fencing, a pipe system, pumps, etc.

Other activities will include: ore will be mined, transported, and processed at the plant. Mine waste will be stored both on surface waste dumps and back filled into pits. The treatment process will generate plant tailings that will be pumped and deposited in surface and in-pit storage facilities. Other impact-generating actions will include vehicle transit, the operation of the rest of the facilities associated with the process plant, fresh water pumping, etc.

Figure 1.7 shows the planned facilities for the Project as per early 2017 design. The San Rafael mining concession boundary is shown in yellow around the perimeter of the project area.

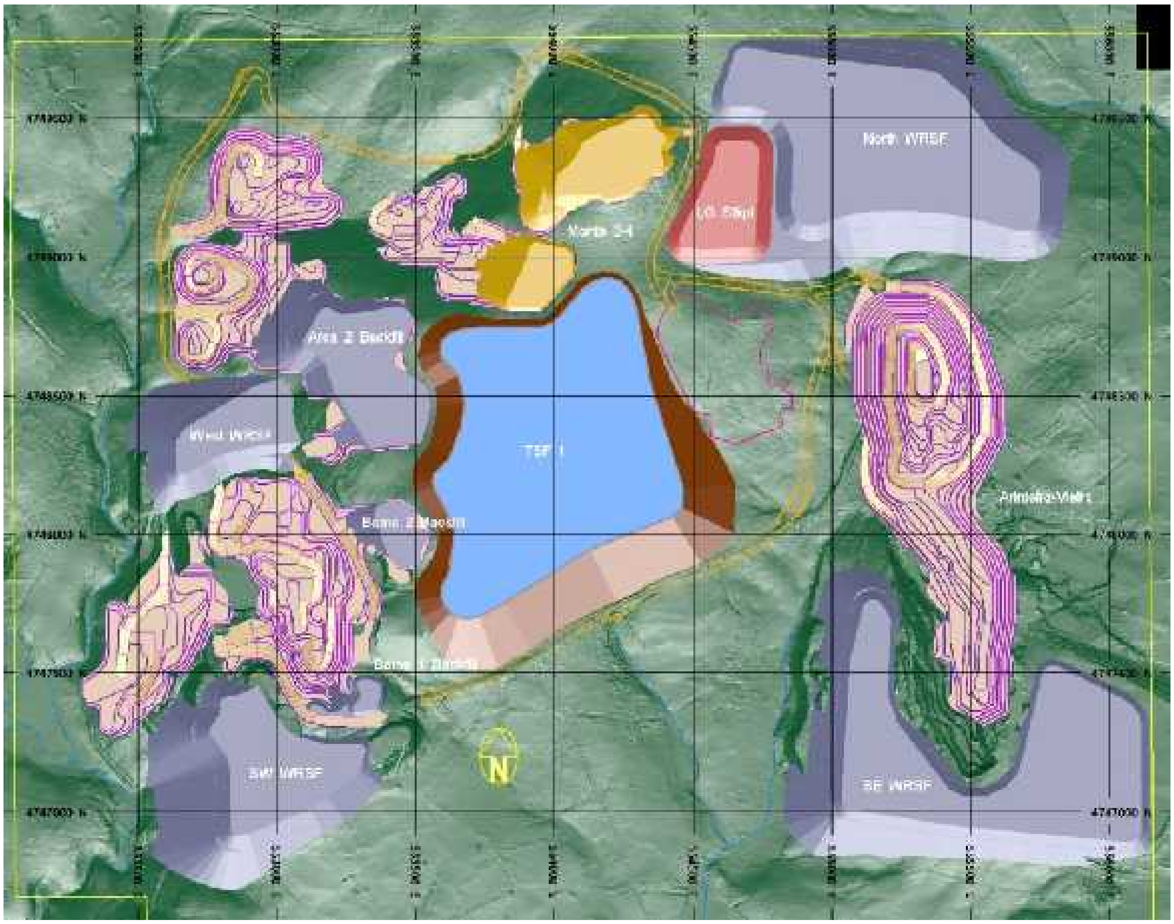


Figure 1.7 – Touro Project Facilities (Rose 2017)

Once the mine operation has ended and all of the ore processing completed or at the end of a mining phase when possible, the mine facility will be closed and the buildings and structures will be dismantled. The mine waste and tailing storage facilities will be reclaimed and the disturbed land will be re-contoured. Top soil will be added to the reclaimed areas and seeded.

The main environmental impacts will be identified and analysed, and mitigation measures will be developed. These data are reported in the environmental impact statement.

Final restoration is an integral part of the Touro Project. Both the operating and final restoration plans (FRP) have been developed to make them compatible with each other. The final reclamation can be implemented and completed in parallel after the cessation of mining in each area and as soon as possible after cessation of processing operations. The objectives of Atalaya's FRP are to:

- Protect the environment,
- Minimize any long term negative environmental impacts of the project,
- Guarantee the chemical stability of water re-use,
- Ensure that the physical stability of any soils is maintained,
- Recover any soils that will be disturbed during mining operations and reuse them appropriately,
- Recover the natural vegetation in a manner that is compatible with the surrounding habitat,
- Reduce the impact to external areas by dust or other emissions,
- Minimize social impacts as a result of the mine closure at the end of its life.

In accordance with current applicable legislation, Atalaya has submitted for approval, an FRP as part of the project approval process. After approval of the DIA, the FRP, with any amendments brought about as a result of the review, would be subsequently approved along with final bonding amounts to the Consejeria de Economia, Empleo, and Industria of Galicia Government.

Occupational risk prevention, as an activity that is performed within the company, is being included in the general management system, which includes all of the activities as well as all of the hierarchical levels for the implementation and application of an occupational risk prevention plan.

An Occupational Risk Prevention Plan will be established as a tool through which the company's prevention activities will be included in the general management system. The necessary resources to perform the prevention activities are organized through its own prevention service. The internal prevention service is a specific organizational unit and only the members thereof decide upon the activities and how they are integrated within the entire organization.

The necessary resources to perform safety activities are organized as per company criteria through its own safety area. The internal safety service is a specific organizational unit which decides the safety activities to be developed and the meanings to integrate them within the entire organization.

Atalaya Mining promotes the establishment of extensive communication channels and actively seeks opportunities for dialogue with its stakeholders, in order to ensure its business objectives are in line with societal needs and expectations. The company aims to be transparent by providing relevant and accurate information on its activities, fostering constructive dialogue, and encouraging continuous improvement.

Since the Project began, the company has fostered a direct relationship and proactive line of communication with the groups, entities, government authorities, institutions, press and public in general that are interested in its operations with an open-door policy with a view to being transparent about its activities.

The Company has been effectively using all available channels to communicate new developments and explain its ideas using internal resources (website, social media, newsletters, e-mailing etc.) as well as the press (press releases, interviews, participation in special editions, press visits, etc.).

1.14 Capital and Operating Costs

The capital and operating costs, expressed in 2017 US dollars, and the following tables, were extracted from the financial analysis prepared by Atalaya and are discussed further in Section 21.

The ore reserve discussed in Chapter 15 is estimated at 90.91 M tonnes of ore averaging 0.43% Cu. Production over the life of mine is summarized in Table 1.7. Although silver was not included in the reserve, it was calculated from the concentrate assays and is shown as a credit in the copper concentrate.

Table 1.7 – Life of Mine Production (total)

Waste	221.33	M tonnes
Ore	90.91	M tonnes
Grade Cu	0.431	%
Contained Metal in concentrate, Cu	346.82	k tonnes
Payable Metal, Cu	340.74	k tonnes
Payable Metal, Ag	925.0	k ounces

1.15 Life of Mine Capital Costs

Life of mine capital costs detailed in Table 1.8 below include both the initial development capital of $164.91M and a capital expansion in year 8 of $30.91M. The capital expansion in Year 8 is required to increase throughput capacity up to 10 Mtpa for treatment of lower grade ore to maintain copper production rates from Year 8 onwards. Sustaining capital averages $3.7M per annum with a total expenditure of $55.3M over the life of mine. The total estimated capital expenditure over the life of mine is $259.54M.

Table 1.8 – Capital Expenditure

Area	Development Capex USD	Expansion Capex USD	Total LOM Capex USD
Mining	$3.88 M	$0.00 M	$3.88 M
Ore Processing	$61.91 M	$18.73 M	$80.64 M
Tailings & Waste	$20.15 M	$0.00 M	$20.15 M
On Site Infrastructure	$13.36 M	$0.00 M	$13.36 M
Indirects	$20.89 M	$4.94 M	$25.83 M
Owner's Costs	$19.42 M	$2.90 M	$22.32 M
Miscellaneous	$25.30 M	$3.62 M	$28.92 M
Sustaining Capex			$55.22 M
Closure Capex			$9.22 M
Total LOM Capex	**$164.91 M**	**$30.19 M**	**$259.54 M**

1.16 Life of Mine Operating Costs

The life of mine operating costs are based on a combination of estimated costs and actual operating costs obtained from the Atalaya's existing Riotinto operations located in the south of Spain. Both fixed and variable costs have been estimated for the life of mine and are summarized in Table 1.9 below;

Table 1.9 – Life of Mine Operating Costs

Description	USD	$/tonne ore	$/lb Cu
Exploration	$13.49 M	0.15	0.02
Mining Costs	$541.37 M	5.96	0.71
<u>Processing Costs</u>			
Labor	$69.79 M	0.77	0.09
Power	$194.30 M	2.14	0.25
Maintenance Materials	$38.58 M	0.42	0.05
Reagents	$24.54 M	0.27	0.03
Consumables	$102.72 M	1.13	0.13

Miscellaneous	$35.71 M	0.39	0.05
TSF Management	$11.62 M	0.13	0.02
Water Treatments	$36.01 M	0.40	0.05
G&A	$72.85 M	0.80	0.10
Total Site Operating Costs	**$1,141.00 M**	**12.55**	**1.49**
Total Off-Site Operating Costs	**$230.65 M**	**2.54**	**0.30**
Total Operating Costs	**$1,371.65 M**	**15.09**	**1.79**
C1 Cash Costs (net silver credits)			**1.73**
AISC (net silver credits)			**1.85**

1.17 Economic Analysis

Atalaya has developed a financial model for the Touro Project that incorporates the updated reserve and resources. On the basis of the latest update of that model, the summary financial forecast for the project is shown in Table 1.10 below. The assumptions for price and financial factors utilized in the financial model and resultant forecasts are as follows:

- All amounts are in constant 2018 US dollars (US$).
- Amounts in Euros (€) were converted to US$ at an average life of mine exchange rate of €1.00:US$1.15
- Copper production is sold at average life of mine copper price of US$3.00/lb.
- Corporate tax rate is 25%.

Table 1.10 – Summary of Key Economic Results

Parameter	Units	Value
Total Cu Production	tonnes Cu in concentrate	346,818
Payable Cu Production	tonnes Cu in concentrate	340,741
Mine Life	Years	12
Operating Cash Cost	US$/lb	1.73
NPV after tax @ 8 %	US$M	179.9
IRR	%	20.5
Copper price	US$/lb	3.00

This financial forecast shows that after tax, capital expenditures, and closure costs, the project will generate unlevered total free cash flow of $489.3M which results in an NPV of $179.9M at an 8% discount rate and an IRR of 20.5%. The overall project cash costs (C1), net of silver credits is US$1.73 per pound of copper increasing to US$1.85 per pound of copper, net of silver credits, adjusting for the sustaining costs (AISC).

1.18 Conclusions and Recommendations

Atalaya is currently exploring and developing the Touro copper Project. The Project consists of a series of deposits of which 4 have partially been mined previously (Arinteiro, Vieiro, Bama and Brandelos) and 2 that have not been mined previously (Monte de las Minas and Arca). The Project site comprises an area of influence of approximately 1,060 ha. Current mineral rights include approximately 15,300 ha of contiguous ground (including a mining concession).

Total proven and probable mineral reserves at $2.60/lb. Cu price are estimated at nearly 91 Mt grading 0.43% Cu. Contained copper is estimated at just over 391,000 tonnes.

The results of this prefeasibility study support a recommendation to advance the project to the next level of evaluation – a definitive feasibility study. The following actions are recommended to be completed by the early stages of the feasibility study:

The main conclusion of this report is that the Touro project is profitable under the evaluation scenario used here. Higher commodity prices could increase mineral reserves through both lower cutoff grades and potentially larger pit limits.

Sensitivity analyses on the project NPV were performed using 5% increments up to ± 20% on copper pricing, capital costs, operating cost inputs (mining, processing, offsite costs and G&A respectively), and exchange rate for the Euro:US dollar. As expected, the copper price variation has the greatest impact on the project NPV both positive and negative. However, the project NPV remains positive in almost all scenarios regardless of the decreased copper price or increase in capital and operating costs and exchange rate differential.

The following are the main recommendations:

During 2017, Atalaya has completed an exploration drilling program targeting areas of inferred mineral resources with the intent of upgrading classifications and/or expanding known mineralized areas. Atalaya plans an additional 2,000 m of RC and 1000 m of diamond drilling for 2018. This should be sufficient to bring the drill hole database to the feasibility level.

The resource model should be updated to include the additional drilling that was completed during 2017. This update should include estimation of sulfur and iron grades, using cokriging with copper in those areas with missing S and Fe assay. Incorporate a model for non-acid-generating/potentially-acid generation material based on geochemical testing that is currently in progress.

Identify possible additional waste rock storage sites if new areas are found that could be mined and thus require more waste rock stripping. Condemnation drilling should be conducted to confirm each site's suitability for waste rock storage.

Complete studies to characterize NAG/PAG waste rock so that quantities of each type can be estimated and appropriate plans developed. Such studies are currently in progress.

Complete hydrology studies, including groundwater inflows to planned open pits, with the intent of developing a comprehensive site water balance. Water quality from pit inflows should be evaluated for suitability for plant make-up water or for potential discharge, and the study should include recommendations for any treatment that may be required. The study would be useful for determining dewatering requirements for open pit development and pit slope depressurization.

Perform a geotechnical analysis of the existing open pit designs to check stability using preliminary results from the above hydrology/groundwater study. Identify areas of concern and new slope angle recommendations, if needed.

Tailings are expected to be potentially acid generating and will be disposed in a HDPE plastic lined impoundment. A geochemical testing program should be conducted to determine the oxidation rates of tailings and the metal leaching potential. Depending on the results obtained, the inclusion of wet or dry covers, or any other suitable methodology, to avoid oxidation may be considered. These studies are currently in progress.

The requirement of the proposed buttress and the foundation preparation at the southern sector of the surface plastic lined TMF's dam should be verified based on an additional geotechnical study to better characterize the foundations particularly the low-strength sand lens. The geotechnical field campaign should include CPTu, geophysics survey methods and the undisturbed soil sampling when feasible.

A better understanding of the rheology, particle size distribution, mineralogy and dewatering properties of the tailings, obtained by laboratory testwork, will refine the design criteria of the Tailings Processing Plant. These studies are currently in progress.

Once this information is produced in the next phases of the project, the thickener unit area can be optimized to obtain the thickened tailings solids content and yield stress required for the tailings management strategy as previously discussed. This laboratory data will also aid in the proper sizing of pumps, pipelines and the flocculant plant.

A comminution trade-off study was completed based on an initial processing rate of 5 Mt/y increasing to 8 Mt/y over the life of mine. It is recommended that a further trade-off study be conducted in the next phase to consider 6 Mt/y and 10 Mt/y. It is unlikely the comminution circuit will change but this should be confirmed prior to commencing the Definitive Feasibility Study (DFS).

It is recommended that a detailed geotechnical investigation be conducted by a suitably qualified geotechnical consultant on the proposed plant site location and borrow pits during the DFS with a combination of test pits, boreholes, and testwork to define in-situ ground conditions and engineering design parameters for earthworks and civils.

It is recommended that ground survey in the proposed plant and non-process infrastructure areas be checked against contour data to validate same. Depending on the results of the confirmatory survey further survey may be required over these areas to develop detailed contour data for the DFS design.

Continue instilling a culture of safety and safe practices both at work and home. Make environmental compliance equal to safety and production.

2 INTRODUCTION AND TERMS OF REFERENCE

The Touro Project (the Project) is a brownfields copper project located 17km from the city of Santiago de Compostela in the A Coruña province of the Galicia region of northwest Spain (see Figure 2.1). The project operated during the period 1973 to 1986 by Rio Tinto Patiño (RTP). The operation was shut down in 1986 due to low copper prices.

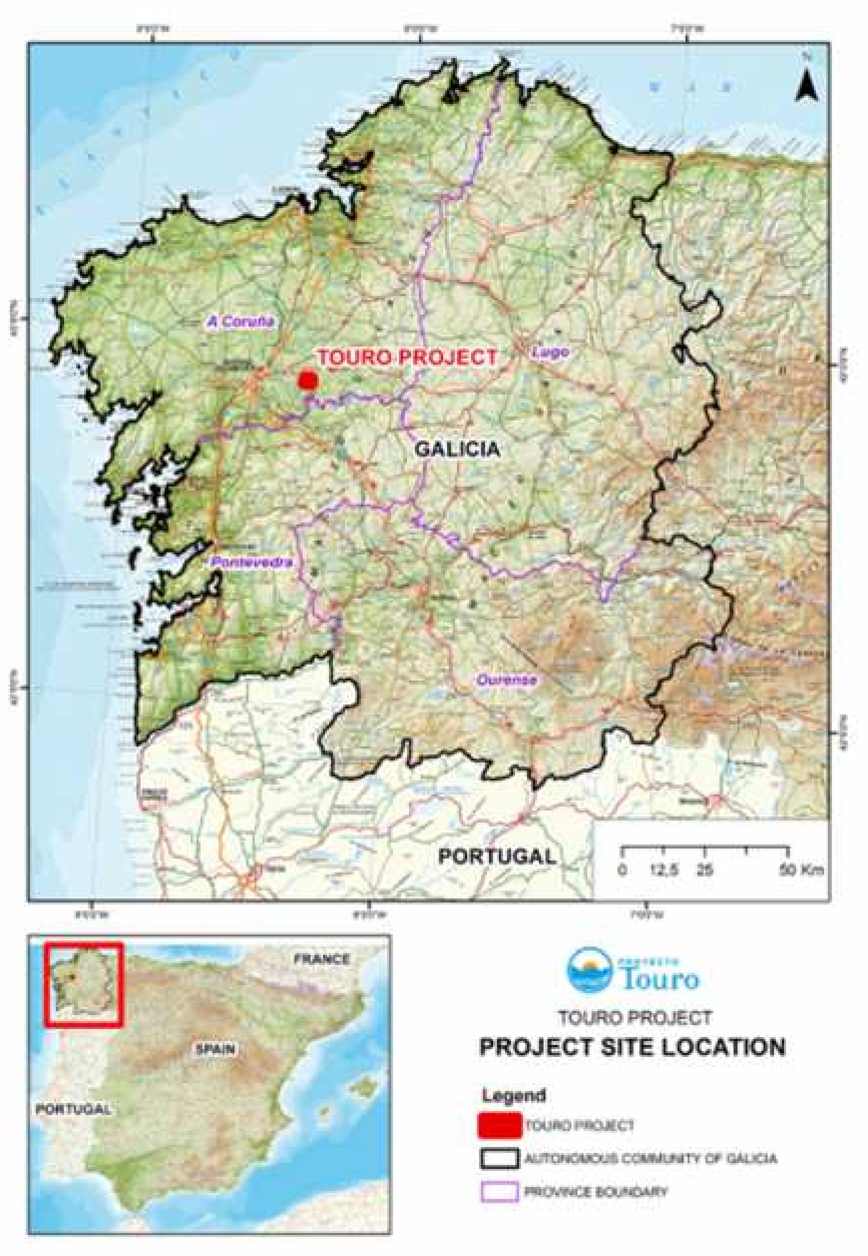


Figure 2.1 – Touro Project Location (Atalaya 2017)

2.1 Background Information and Terms of Reference

Ore Reserves Engineering was contacted by Atalaya Mining in 2016 and was requested to prepare a resource estimate for Atalaya's Touro Copper Project located in Galicia, Spain. This resource estimate has been incorporated into this Preliminary Feasibility level Technical Report and has been prepared in accordance with NI 43-101 guidelines.

Atalaya Mining Plc (Atalaya) is a base metals mining company that owns and operates the Riotinto Project in southern Spain producing copper concentrate. Atalaya has entered into a purchase option agreement to acquire a majority interest in the Touro Project. This exclusive option allows Atalaya to purchase an 80% interest in the Project.

Pursuant to accomplishing the above tasks, Mr. Alan C. Noble of Ore Reserves Engineering (ORE), Mr. William Rose of WLR Consulting along with Mr. Jaye T. Pickarts traveled to Spain in November 2016 and conducted a site visit over a period of one day. During the site visit, the following personal inspections were conducted:

Mr. Noble:

1) Reviewed the overall project status and history with project personnel.
2) Reviewed the geologic interpretation with project geologic personnel.
3) Reviewed drilling methods and exploration with project geologic personnel.
4) Visited the core-logging facility, inspected a representative selection of core, reviewed geologic logging procedures.
5) Discussed current methods for resource estimation with mine technical staff.
6) Visited the assay lab and reviewed sample preparation and assaying procedures.

Mr. Rose:

1) Reviewed the overall project status and history with project personnel.
2) Reviewed project infrastructure.
3) Visited the core-logging facility and inspected a representative selection of core.
4) Visited the open-pit mine areas, potential WRSF sites, and the primary crusher area.
5) Reviewed existing mine development plans.

Mr. Pickarts:

1) Reviewed the overall project status and history with project personnel.
2) Reviewed project infrastructure.
3) Reviewed marketing studies and contracts.
4) Reviewed environmental permitting and compliance procedures.
5) Reviewed project safety procedures.

In addition, Minnovo Pty Ltd. (Minnovo) was retained by Atalaya to assist in the preparation of the report. Minnovo's principal scope is in the area of metallurgy, process plant and infrastructure. Matt Langridge and John Fleay visited the site for two days from 7-8 September 2015. During the site visit, the following personal inspections were conducted:

Mr. Langridge:

1) Reviewed the overall project status and history with project personnel.
2) Inspected the overall project site.
3) Inspected remnants of original Touro process plant.
4) Inspected aggregate quarrying and crushing operations located on the site.

5) Visited potential project infrastructure (office buildings, laboratory, water supply infrastructure, power supply).
6) Reviewed potential sites for locating plant and tailings dam.
7) Reviewed existing access roads.
8) Discussed hydrology and geotechnical aspects of project site with project personnel.

Mr. Fleay:

1) Reviewed the overall project status and history with project personnel.
2) Inspected the overall project site.
3) Inspected remnants of original Touro process plant.
4) Inspected aggregate crushing operations.
5) Visited office buildings, laboratory, water supply infrastructure, power supply infrastructure.
6) Inspected potential sites for locating plant and tailings dam.
7) Inspected existing power lines and substations.
8) Reviewed geology and drill core and selected samples for metallurgical testwork.

Golder and Associates (Golder) was retained by Atalaya to assist in the preparation of the report. Golder's scope was limited to the design and operations management of the tailing management facility. Mr. Alistair Cadden provided the direction and oversite. Specifically:

Mr. Cadden:

1) Reviewed the tailings management facilities design.
2) Reviewed the tailings parameters and design criteria.
3) Reviewed the tailings deposition plans.
4) Reviewed the tailings processing plant design.
5) Reviewed the closure measures proposed for the tailings management facilities.
6) Reviewed the recommendations for the next Project Study Phase.

Atalaya have managed the site activities including additional drilling, geotechnical, hydrology and environmental investigations. ORE has prepared this Technical Report based on these inputs.

All of the above listed professionals are independent Qualified Persons according to the definitions of NI 43-101 and have conducted this work as independent consulting engineers.

The overall objective of this Technical Report is to further define the project and optimise its value. This Technical Report incorporates new in-fill and extension resource drilling, together with new metallurgical testwork, plant design and a staged project development. In addition, further work has been undertaken in the areas of geotechnical, geohydrology, hydrology and environmental studies, allowing for definitive designs and cost estimates.

The scope of work for the project included:

1) Preparation of a resource estimate for copper.
2) Preparation of open pit mine designs, including phasing, main haul road plans, annual production schedule, minerals reserve estimates, conceptual waste rock storage facility plans, haulage profile measurements for preliminary contract mining bids, and mine operating cost estimates from a review of submitted bids.
3) Review metallurgical testwork
4) Preparation of the plant facility and plant operations design.
5) Review of environmental, safety, marketing, and costs.

6) Preparation of a NI 43-101 compliant report to document the above.

2.2 Sources of Information and Data

Electronic data files containing geologic interpretations, drill hole data, surface topography, and plant flowsheets were provided by project technical staff. Additional data sources include other technical reports and independent resource estimation reports.

2.3 Definitions and Units of Measure

Units of measure in this report are SI Units including meters, kilometers, kilograms, metric tonnes, liters, etc., unless explicitly stated. Currency units are in U.S. dollars, and copper prices are in $US/pound copper (454 grams).

This report has been prepared in accordance with Form 43-101F1 Technical Report and the CIM Definition Standards for Mineral Resources and Mineral Reserves adopted by the CIM Council in April, 2011.

3 RELIANCE ON OTHER EXPERTS

The authors used their experience to determine if the information provided was suitable for inclusion in this Technical Report. Except where noted, the author has relied upon the information provided by Atalaya as being accurate, reliable, and suitable for use in the report. This Report includes technical information, which required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

4 PROPERTY DESCRIPTION AND LOCATION

The Project is located in the A Coruña province of the Galicia region of northwest Spain. The geographical location of the Project is presented in Figure 4.1. The existing and new pit outlines are shown superimposed on an aerial photograph of the area in Figure 4.2. The mineral tenements associated with the Project are shown in Figure 4.3 and listed in Table 4.2 below. All pits, plant, tailings and mine waste for the Project are contained within the Exploitation Concession lease boundary.

Santiago de Compostela is the nearest major regional centre to the Project site with a metropolitan population of approximately 180,000. The project site is located approximately 17 km east of Santiago de Compostela and approximately 7 km east of the Santiago de Compostela international airport. The site is serviced by existing bitumen sealed roads approaching the site from various directions.

Approximate coordinates of the site in the European ETRS 89 survey grid are Northing 554,054 m and Easting 4,747,955 m (Universal Transverse Mercator [UTM] Zone 29N).

Figure 4.1 – Touro Project Location (Atalaya 2017)

Galicia has its own regional government, Xunta de Galicia, a collective entity with executive and administrative power. Several permits for the Project will be issued by the regional government, as outlined in Section 20.4 (Permitting).

Galicia also has its own mining law launched on May 2008 (Ley de Ordenación de la Minera de Galicia) which complements the Spanish mining law. Modifications to the Spanish laws are very minimal and refer mainly to application costs, maintenance costs, minimum investments applicable in certain circumstances, bank guarantees requested in certain cases. An overview of the main permits required for the Project is provided in Section 20.5

4.1 Touro Copper Project Area

Historically, the Project was explored and exploited between 1970 and 1986 by Rio Tinto Patiño (RTP). RTP commenced commercial production in 1973 and closed the operation in 1986 due to low copper prices. The deposits that were partially mined by RTP included Vieiro, Arinteiro, Bama and Brandelos. Following the mine closure, RTP sold the San Rafael Exploitation Concession to Explotaciones Gallegas S.L.

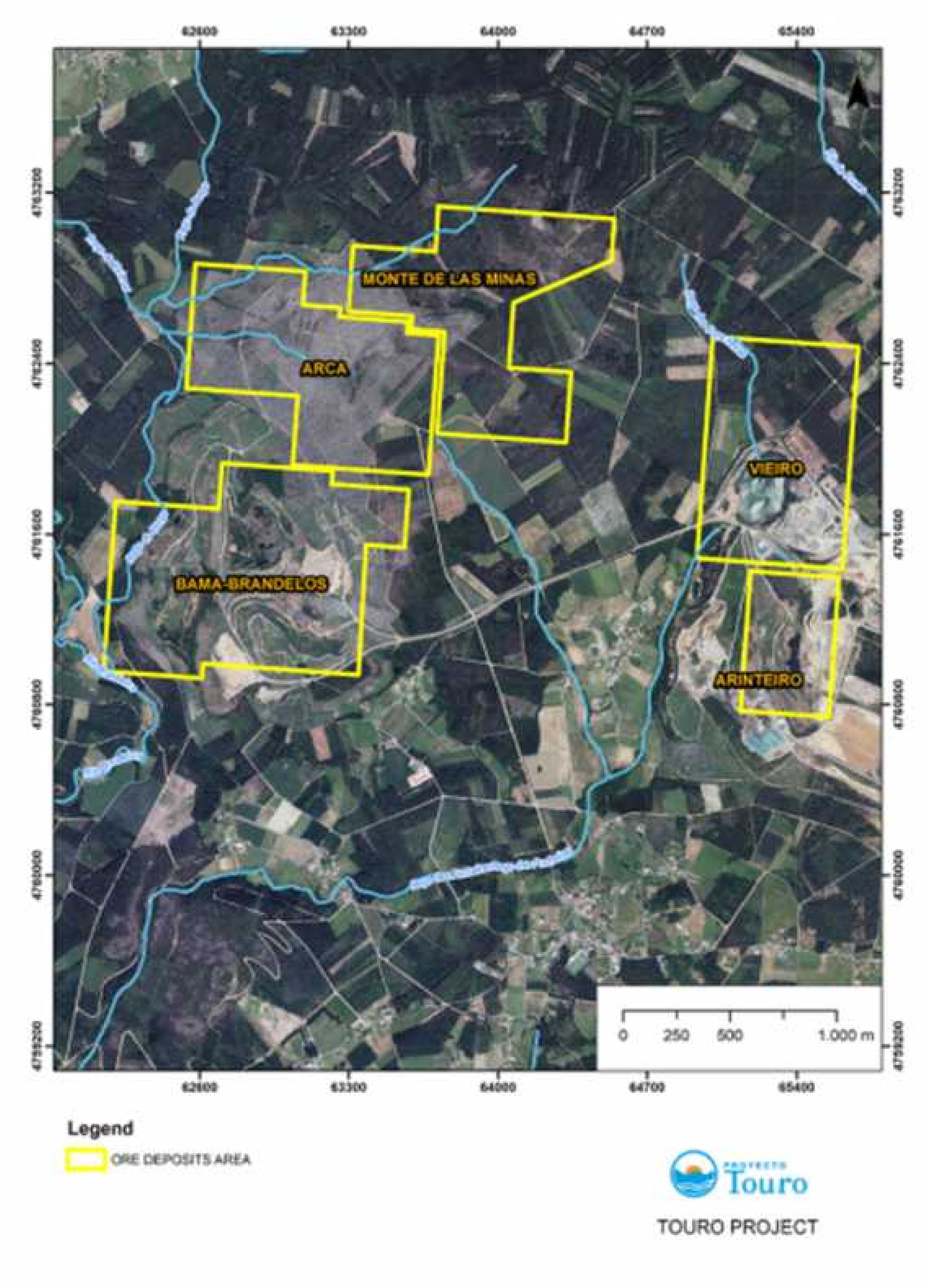


Figure 4.2 – Historical Mining Activity (Atalaya 2017)

4.2 Land Position at Touro

Atalaya has entered into a purchase option agreement to acquire a majority interest in the Project. The option agreement gave Atalaya an exclusive option to purchase an 80% interest in the Project.

Following the acquisition of the initial 10% of Cobre San Rafael S.L. share capital, the Agreement includes the following four phases:

- Phase 1 – Atalaya paid €0.5 million to secure the exclusivity agreement and will continue to fund up to maximum of €5 million to get the project through the permitting and financing stages.

- Phase 2 – When permits are granted, Atalaya will pay €2 million to earn-in an additional 30% interest in the project (cumulative 40%)

- Phase 3 – Once development capital is in place and construction is underway, Atalaya will pay €5 million to earn-in an additional 30% interest in the project (cumulative 70%)

- Phase 4 – Once commercial production is declared, Atalaya will purchase an additional 10% interest in the project (80% cumulative) in return for a 0.75% Net Smelter Return (NSR) royalty, with a buyback option.

The Agreement has been structured so that the various phases and payments will only occur once the project is de-risked, permitted and in operation.

Figure 4-3 and Table 4-1 provide information on existing mineral rights. An Operating Concession is available for San Rafael.

Mineral resources identified to date are located in the San Rafael Mining Concession. The claims are map-staked using digital maps, and the claims do not overlap each other. The main requirements to keep the claims in good standing are:

- to comply with the approved work plans;
- to comply with the payments; and
- to apply for the extension/renewals on time.

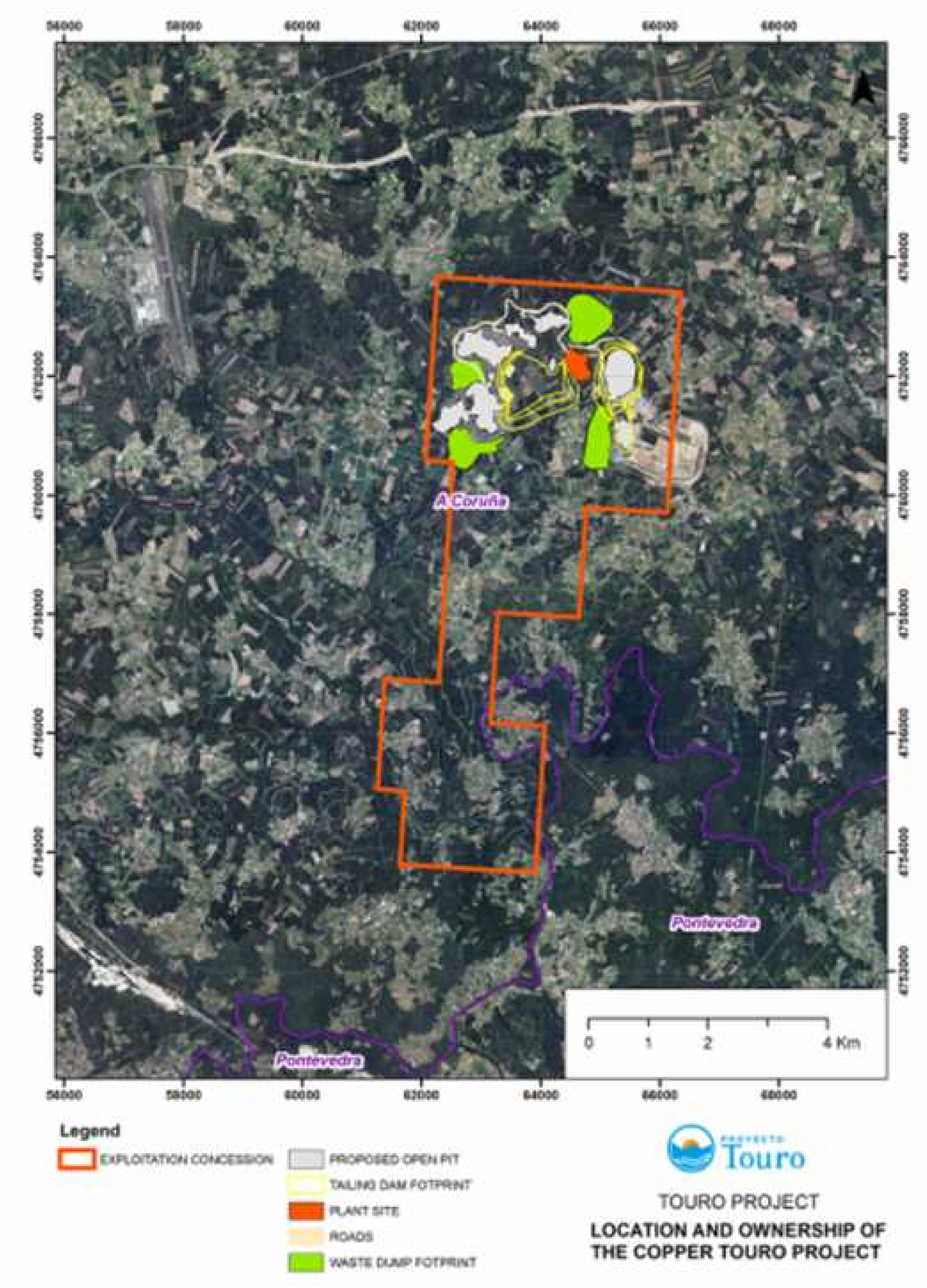


Figure 4.3 – Location and ownership of the Touro Copper Project (Atalaya 2017)

Two additional ore deposits, Monte de las Minas and Arca have not previously been mined. Figure 4-4 shows an overview of the future pit outlines and waste rock storage facilities. The Project Site has an area of influence of approximately 1,060 ha, while the total mineral rights (investigation and exploitation concessions) cover an area of approximately 2,744 ha.

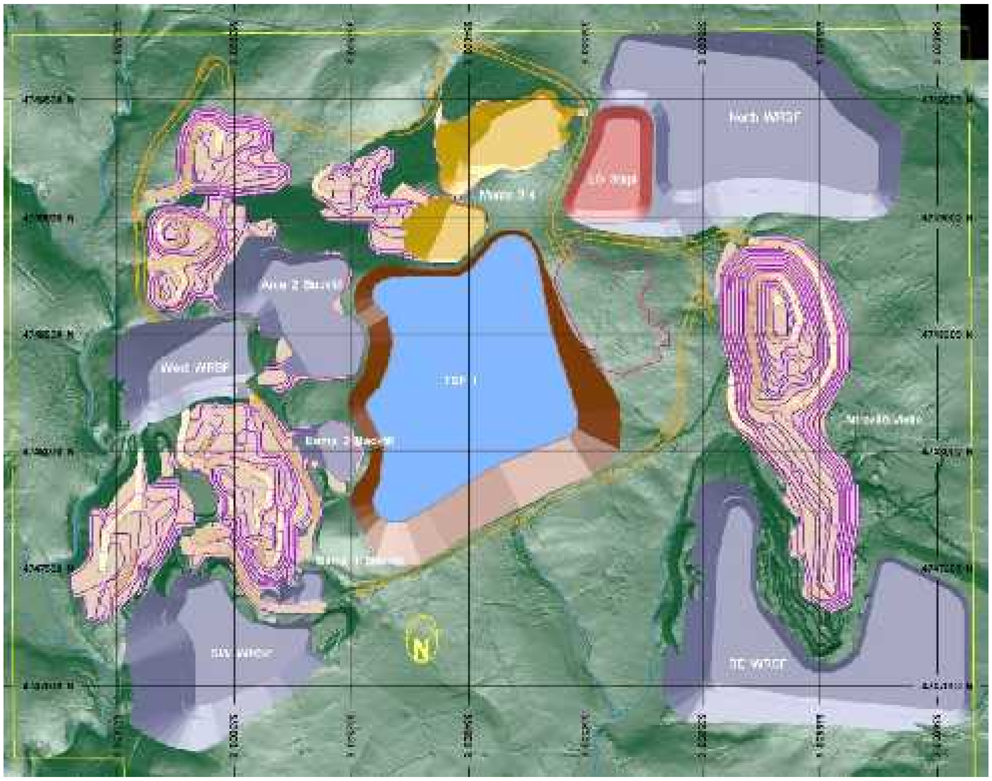


Figure 4.4 – Future Mining Pits and Waste Rock Storage Facilities (WLR, 2017)

Table 4.1 – Mineral Rights of Explotaciones Gallegas de Cobre S.L.

Name	No.	First Owner	Type	Area (mining square)	Area (ha)	Granted	Expiry Date	Previous Owner Transfer
San Rafael	CO/02946	Cobre San Rafael, S.L. 09/02/2017	EC	98	2,744	GRANT OF THE EC 19/06/1958 CONSOLIDATION OF THE MINING RIGHT 18/04/1978 GRANT OF THE EC DEMASÍAS 18/10/2011	18/04/2068	Explotaciones Gallegas, S.L.,

4.3 Surface Rights

Explotaciones Gallegas S.L. owns 350 hectares of land in the San Rafael EC.

Land acquisitions will need to be conducted to obtain surface rights required for the future mining operation. This will be achieved through negotiation or, as a last resort, through expropriation according to the legislation.

4.4 Permits

Table 4-2 lists the main permits required for the ongoing exploration activities (mainly drilling). All exploration activities have been conducted in accordance with jurisdictional requirements.

Table 4.2 – Permits Required for Exploration Activities

Permit	Relevant Regulatory Agency
Approval of the annual work plan	La Coruña Mines Department
Water use	Aguas de Galicia
Surface rights	Obtained from local landowners

4.5 Environmental Liabilities

There are potential existing environmental liabilities at the site originating from the previous mining of the deposits during 1973-1986 by RTP. Historical works consist of numerous waste-rock dumps, the existing Tailings Management Facility (TMF) and 4 abandoned open pits. Acid Rock Drainage (ARD) is currently the main environmental concern at the site. These old workings have been partly restored. The restoration, using “tecnosol” - mainly compost soils with high alkalinity - has had an impressive effect on re-vegetation of the area.

Information on environmental permitting, related studies, and other environmental relevant information on the existing and future Project is provided in Section 20.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Project is located in the A Coruña province of the Galicia region of northwest Spain, approximately 610 km northwest of Madrid. Santiago de Compostela is the nearest major regional centre to the Project site with a metropolitan population of approximately 180,000. The project site is located approximately 17 km east of Santiago de Compostela and approximately 7 km east of the Santiago de Compostela international airport. The site is serviced by existing bitumen sealed roads approaching the site from various directions.

5.1 Accessibility

The Project is located in an easily accessible area, close to the city of Santiago de Compostela. It is serviced by a good road network, and has an International airport nearby.

Table 5.1 - Distances of the Project to Adjacent Cities and Communities (Atalaya 2017)

Name of the City/Community/Place	Distance from the Project Site
Madrid	610 km
Santiago de Compostela	16.6 km
Santiago Airport	6.6 km
Touro	3.1 km
Arzua	15.2 km
O Pino	6.0 km
O Pedrouzo	2.5 km
Salceda	7.4 km

Santiago de Compostela has an International airport located between the city and the Project site and accessed by the A-54 road.

Starting from Santiago de Compostela, road access to the Project is via the A-54 road going east for approximately 15km (past the airport) before turning onto the N-547 for approximately 8km passing the community of O Pedrouzo, then turning south to the Project site via CP-0605 for approximately 2km and DP-6602 for approximately 3km. Overall driving distance is approximately 28km with a driving time of approximately 40 minutes.

For concentrate transport from the future mine operations, the existing road system will be used. The ports of A Coruña, El Ferrol, and Vilagarcia de Arousa are located some 45-54km by road south-west Santiago de Compostela, connected by the AP-9 and N-640 main roads. This port was used by RTP during former mining operations. Copper concentrate will be transported from the Project to the port of Vilagarcia de Arousa, as one of the available options, by road using covered truck trailers.

Figure 5.1 – Site Location and Access (Atalaya 2017)

5.2 Climate and Physiography

The climate in the area near Santiago de Compostela can be defined as a mild Atlantic climate with a continental tendency. The rainfall regime varies substantially with intense rain in the autumn, winter and part of the spring and low rainfall in the summer. The rainfall module ranges between 900 mm and 3200 mm in general. The annual average rainfall module is 1789.6 mm for a 48-year thermopluviometric series.

The average monthly temperature ranges from 5ºC to 22ºC with an annual average of 12.6ºC.

Table 5.2 provides a summary of Santiago de Compostela's climate from the airport weather station records.

Table 5.2 – Climate of Santiago de Compostela Airport, 1971 – 2000 (Agencia Estatal de Meteorología, 2012)

Climate Data	Value
Annual average of daily maximum temperature	17.6 °C
Annual average of daily minimum temperature	8.3 °C
Annual average temperature	13.0 °C

Average annual precipitation	1,786 mm
Days per year with precipitation	139.5 days
Average annual hours of sunshine	1,998 hours

5.2.1 Local Resources and Infrastructure

As detailed in Section 5.1, the Project is located close to Santiago de Compostela. The region has all the necessary resources including medical services, supplies, fuel, electrical and housing are available locally.

The workforce for the Project is expected to be largely available in the area. The present Project owner, Explotaciones Gallegas S. L., was recently the main employer in the area (200 workers).

5.2.2 Physiography

The Project Site is located at an average elevation of approximately 315 metres above sea level (masl), with an elevation of 157 masl at the lowest point and 445 masl at the highest point. An archaeological site (pre-Roman ditch) located at the San Sebastian Hill south of the Project area has a maximum elevation of approximately 340 masl. Figure 5-2 provides a Project view. The vegetation in the area is dominated by forest and forest plantations (eucalyptus) as well as open areas (grazing land). Further information on vegetation, habitats and the environmental conditions is provided in Section 20.

Figure 5.2 - Project View from Fuente Rosas, from South to North (Atalaya 2017)

6 HISTORY

The project was originally owned and operated by Rio Tinto Patiño (RTP). Exploration started in 1970 followed by mining and processing operations producing copper concentrate from 1973 until the mine was closed in 1986 due to a decline in the copper price.

After closure, RTP sold the San Rafael Operating Concession to Explotaciones Gallegas who has kept the mining permits in place.

6.1 Mining

During the 13 years of mine operations, 21 Mt of ore at an average grade of 0.61% Cu was extracted from four primary open pits; Arinteiro, Vieiro, Bama, and Brandelos, as shown in Figures 6.1 through Figure 6.3.

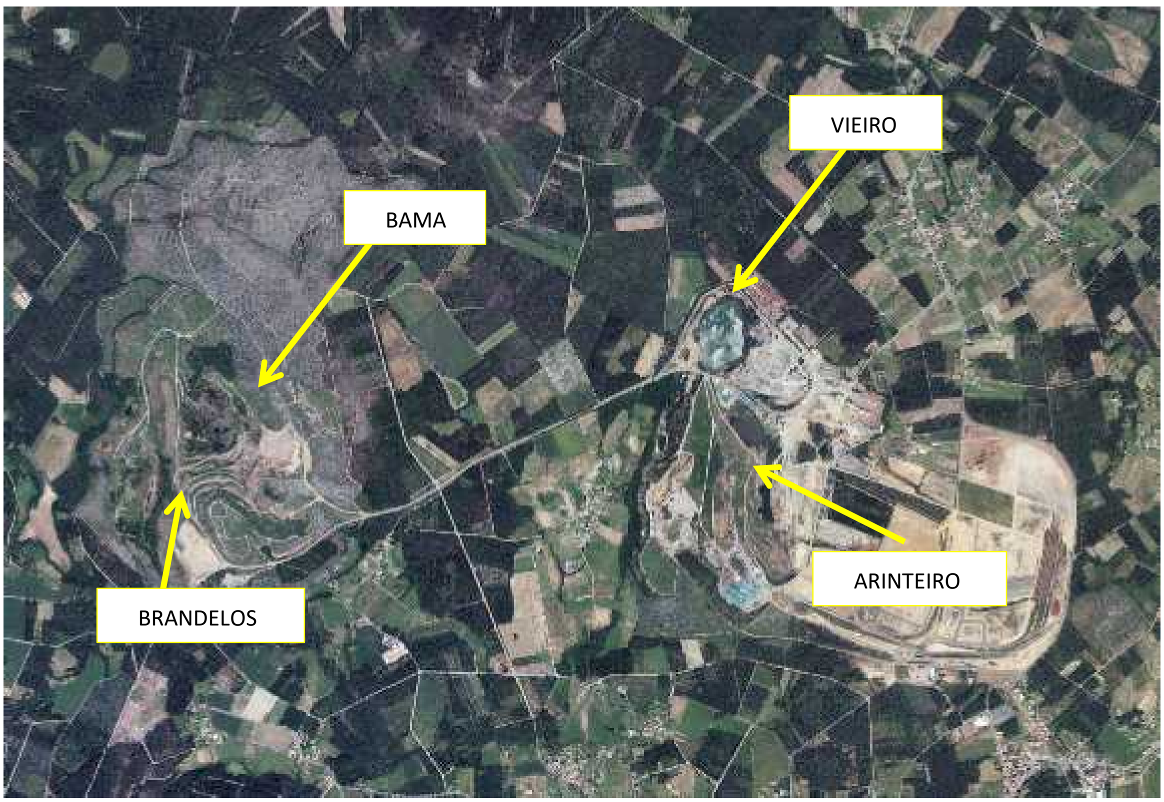


Figure 6.1 – Historical Pit Locations (Atalaya 2017)

Figure 6.2 – Vieiro and Arinteiro Pits in 1986 (Atalaya 2017)

Figure 6.3 – Bama and Brandelos Pits in 1986 (Atalaya 2017)

Based on historic production records, shown in Table 6.1, the total material movement (waste + ore) out of the four pits was 59 Mt with an average strip ratio of 1.70:1 tonnes to tonnes. The average grade of the ore was 0.61% Cu.

Figure 6.4 - Historic Waste Rock Storage Facility for the Vieiro and Arinteiro Pits in 1986 (Atalaya 2017)

Waste rock storage facilities were located close to the pits, as shown in Figure 6.4. Approximately 38 Mt of mine waste rock was deposited in an area of 175 ha by the end of mine operations.

Table 6.1 – Historic Mine Production (Atalaya 2017)

YEAR	Ore (Mt)	Cu grade (%)	Waste (Mt)	Total (Mt)	Ratio
1974			2.136	2.136	
1975	773	0.76	2.584	3.357	3.34
1976	1.592	0.75	3.822	5.414	2.40
1977	1.650	0.78	3.044	4.694	1.84
1978	1.672	0.73	2.766	4.438	1.65
1979	1.705	0.67	3.103	4.808	1.82
1980	1.839	0.68	3.800	5.639	2.07
1981	1.881	0.60	3.711	5.592	1.97
1982	1.878	0.59	2.446	4.324	1.30
1983	1.923	0.56	2.726	4.649	1.42
1984	2.144	0.47	2.888	5.032	1.35
1985	2.310	0.46	2.755	5.065	1.11
1986	1.631	0.42	2.114	3.745	1.30
1987	108	0.49	190	298	1.76
Total/Average	21.106	0.61	38.085	59.191	1.70

6.2 Milling

The process plant was designed and constructed between 1971 and 1973 with a treatment capacity of 1.3 Mt/year. In 1985, the capacity was increased to 2.3 Mt/year.

Processing included the following operating units:

- Primary crushing - 500 t/h, expanded to 700 t/h
- Screening, and secondary and tertiary crushing - 325 t /h
- Milling - 170 t/h then expanded to 290 t/h
- Flotation
- Thickening, Filtration and Concentrate Drying

Historic milling records, shown in Table 6.2, indicate that 490 kt of copper concentrate at a grade of 22.3%, containing 110 kt of copper was produced over 13 years. The average copper recovery was approximately 86%.

Table 6.2 – Historic Milling Production

YEAR	TREATMENT PLANT			
	Processed (kt)	Concentrate (kt)	Cu grade (%)	Cu Recovery (%)
1975	714	24.8	17.5	81.38
1976	1,579	51.0	20.3	87.41
1977	1,640	48.9	23.6	90.00
1978	1,696	45.7	24.1	88.69
1979	1,765	45.0	22.9	87.15
1980	1,798	48.3	22.0	87.10
1981	1,898	43.4	22.5	86.16
1982	1,914	41.5	23.6	86.52
1983	1,915	39.1	23.7	85.81
1984	2,150	37.0	22.9	83.29
1985	2,322	39.9	21.3	79.11
1986	1,591	23.4	20.6	82.11
1987	113	2.1	0.50	80.80
Total/Average	21,115	490.1	22.3	86.00

Tailings were deposited in two tailing storage facilities, Presa Vieja and Presa Nueva (shown in Figure 6.5), and contained approximately 28 Mt of tails. Both storage facilities have been closed and reclaimed.

Figure 6.5 – Presa Nueva Tailing Storage Facility in 1986 (Atalaya 2017)

7 GEOLOGICAL SETTING AND MINERALIZATION

This section was initially compiled by the Atalaya Mining technical staff and has been updated by Monica Barrero Bouza with Alan Noble, Qualified Persons for the purpose of NI 43-101, Standards of Disclosure for Mineral Projects.

The geology of northwestern Iberia has been extensively reviewed since the 1970's by authors like Den Tex and Floor (1971) and Arps (1977), and more recently by Martinez Catalan (2007), and references therein.

This summary of the local project geology is based largely on the works of Williams (1983), Gomez Barreiro (2007) and on the investigations conducted by the several mining companies which worked in the project over the last 45 years.

7.1 Regional Geology

The Project is located in the NW of the Iberian Massif, a sector of the Variscan Belt of Europe, and includes the Spanish regions of Galicia and the Cantabrian Mountains, as well as northern Portugal. The northwestern Iberian basement consists of plutonic and metamorphic rocks and a clear separation can be established between autochthonous and allochthonous terranes. The autochthon consists of a thick metasedimentary sequence whereas the allochthon consists of the remnants of a huge and structurally complex nappe pile preserved in the core of late Variscan synforms. Both are separated by a thrust sheet, several kilometers thick, consisting of metasediments and volcanic rocks (Martinez Catalan, 2007). There are three allochthonous complexes in Galicia (Cabo Ortegal, Ordenes, and Malpica-Tui), and two in northern Portugal (Bragança and Morais).

The Project area is located in the High-Pressure/High-Temperature Upper Units of the SW sector of the Ordenes Complex, as shown in Figure 7.1. Within the Ordenes Complex, copper mineralization is associated with the Arinteiro Unit which contains the metabasites that host the Touro group and Fornás-Mañoca orebodies.

The composition and grain size of the metabasites (amphibolites) are very variable but mostly contain amphiboles (hornblende), garnet (almandine), pyroxene, quartz, and biotite.

The metabasites are interbedded with metasediments (paragneisses that were pelites and greywackes in origin) of the O Pino Unit. The composition of these metasediments is predominantly kyanite-staurolite-garnet-two mica schist and quartz-plagioclase-biotite-garnet gneiss (Castiñeiras, 2005). Of minor occurrence, ultramafic rocks and thin bands of quartz are in places interlayered with the metabasites.

According to Serrati et al (2002), the Ordenes Complex hosts two different types of sulfide mineralization, both closely concordant with the schistosity. One type is represented by the Arinteiro and Bama orebodies, with pyrrhotite and chalcopyrite disseminated in a coarsely grained garnet amphibolite. The other type, which occurs a few kilometers westwards at Fornás and Mañoca, are massive sulfides composed by massive pyrrhotite, chalcopyrite and scarce pyrite and sphalerite.

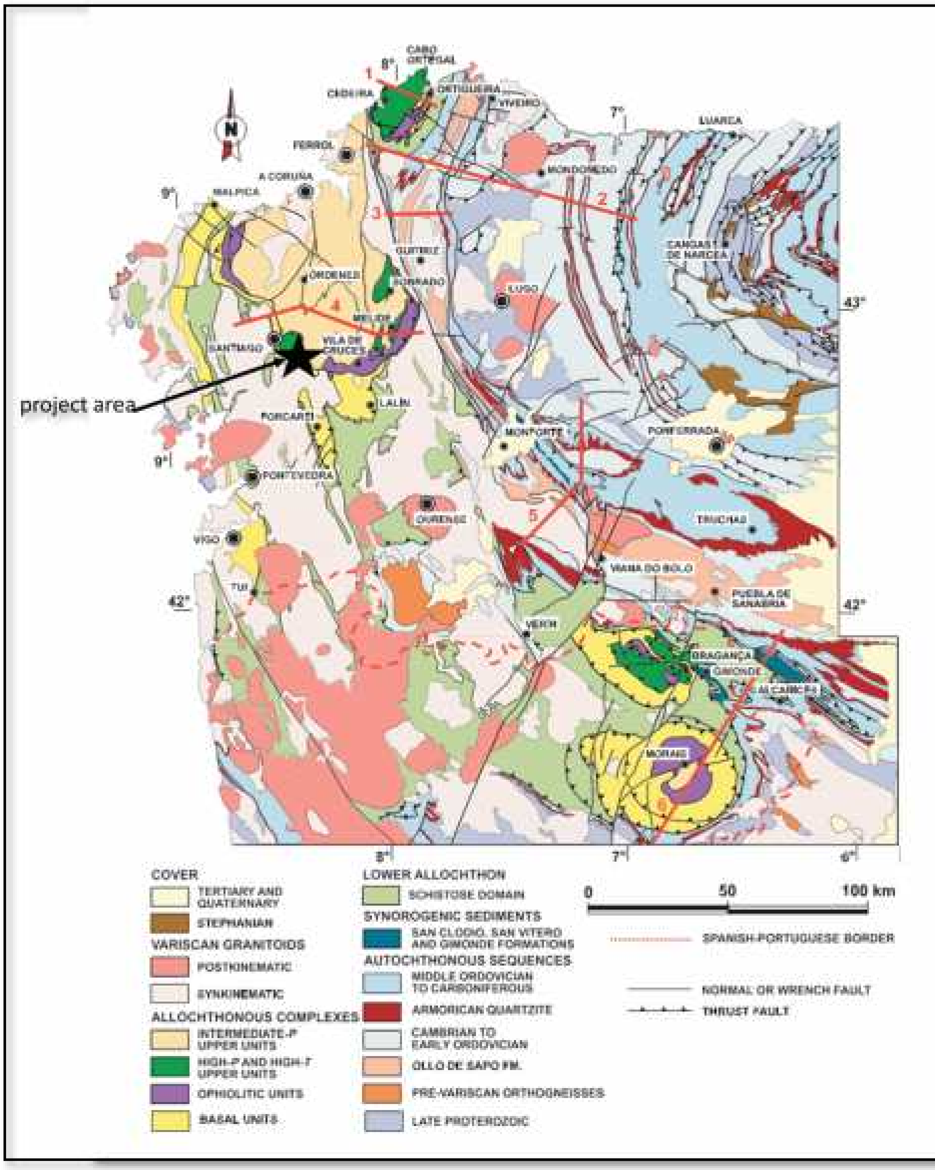


Figure 7.1 - Geological sketch map of northwestern Iberia, showing the allochthonous complexes and their units. (Source: Martinez Catalan (2007))

7.2 Local Geology of the Touro Deposit

The Touro deposit is hosted in a continuous, highly deformed and metamorphosed horizon of metabasite and paragneiss of the Arinteiro Unit (Ordenes Complex). This horizon outcrops along the flanks and hinge zone of a broad north plunging antiformal structure for at least 8km along strike and up to 2km across strike. It extends beyond the project area and is open to depth.

7.2.1 Stratigraphy

The local stratigraphy is represented by the Arinteiro and O Pino Units, both belonging to the Ordenes Complex, as shown in Figure 7.2 and Figure 7.3.

The Arinteiro Unit is composed mainly by metabasites with minor lenses of ultramafic rocks and paragneisses (Gomez Barreiro, 2007). The mineralogy, grain size, and homogeneity of the metabasites are variable, most of the rock types are hornblende-plagioclase rocks with or without garnet (almandine), clinopyroxene (salite), and quartz. The rocks have undergone varying degrees of cataclasis commonly associated with retrogressive greenschist facies assemblages. The metabasites are strongly deformed showing tectonic banding, intrafoliar folds and oriented nematoblastic texture. Very locally, relict igneous textures have been described supporting the theory of a gabbroic protolith (Castiñeiras et al., 2002; Gomez Barreiro, 2007).

The metasediments of the O Pino Unit lie above and below the metabasites that have been interpreted as representing a flysh sequence (Gomez Barreiro, 2007). This unit is composed mainly of paragneisses with predominant greywacke composition. Minor pelitic metasediments, biotitic shists (sometimes rich in graphite), and amphibolite gneisses are also described in this unit.

According to Atalaya Mining, there is a main horizon of paragneiss in the project area with an estimated thickness of around 400m that hosts metabasites that are represented mainly by amphibolites interbedded within the metasediments. The metabasites appear to represent a single level event of volcanic or subvolcanic origin. The thickness of the amphibolite horizon based on the drilling data of Atalaya Mining, varies from a few meters to up to 90 m.

Rock weathering is limited to a few meters below the surface.

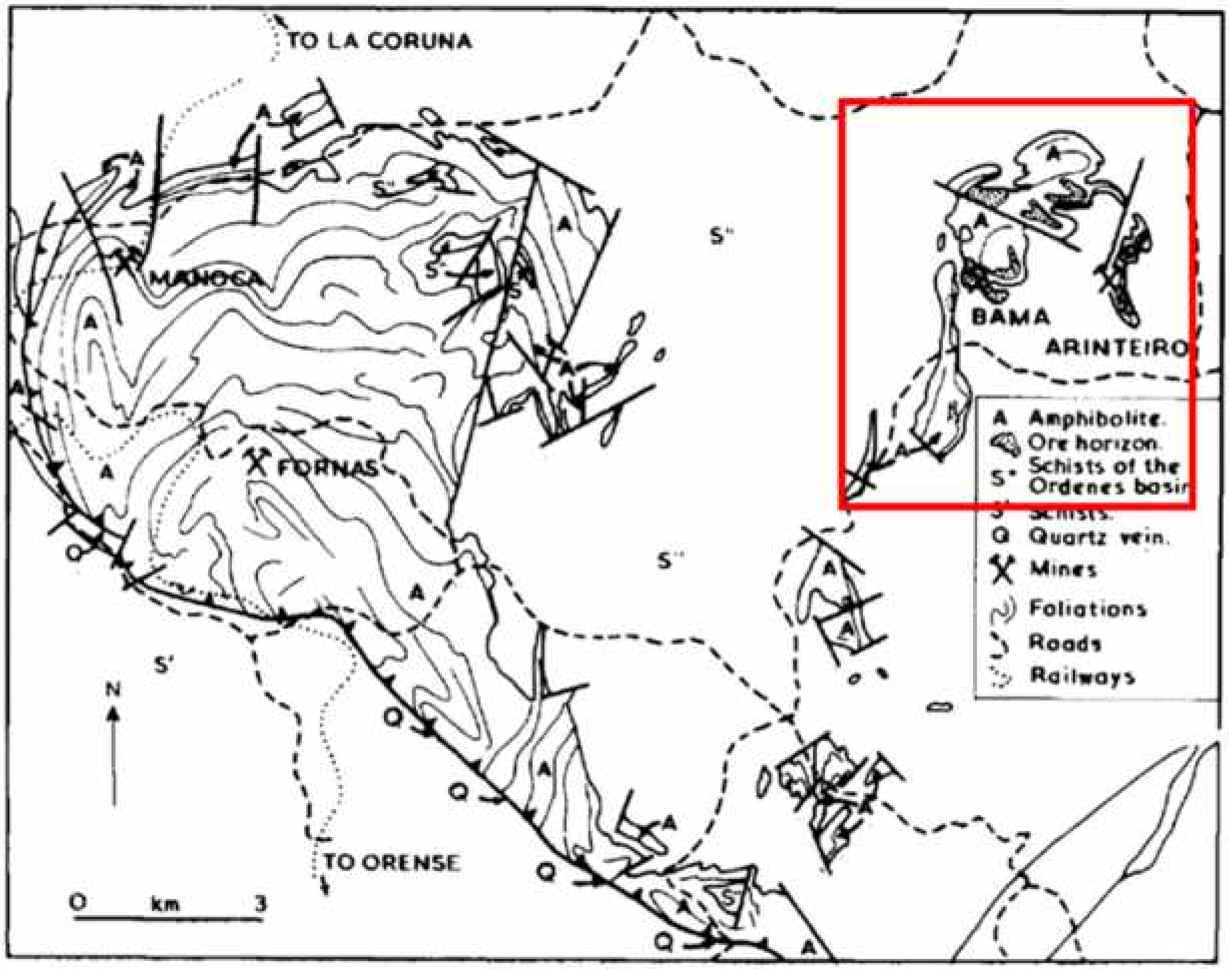


Figure 7.2 - Geological map of the Santiago area amphibolites showing the Arinteiro-Bama deposit (Source: Badham& Williams, 1981)

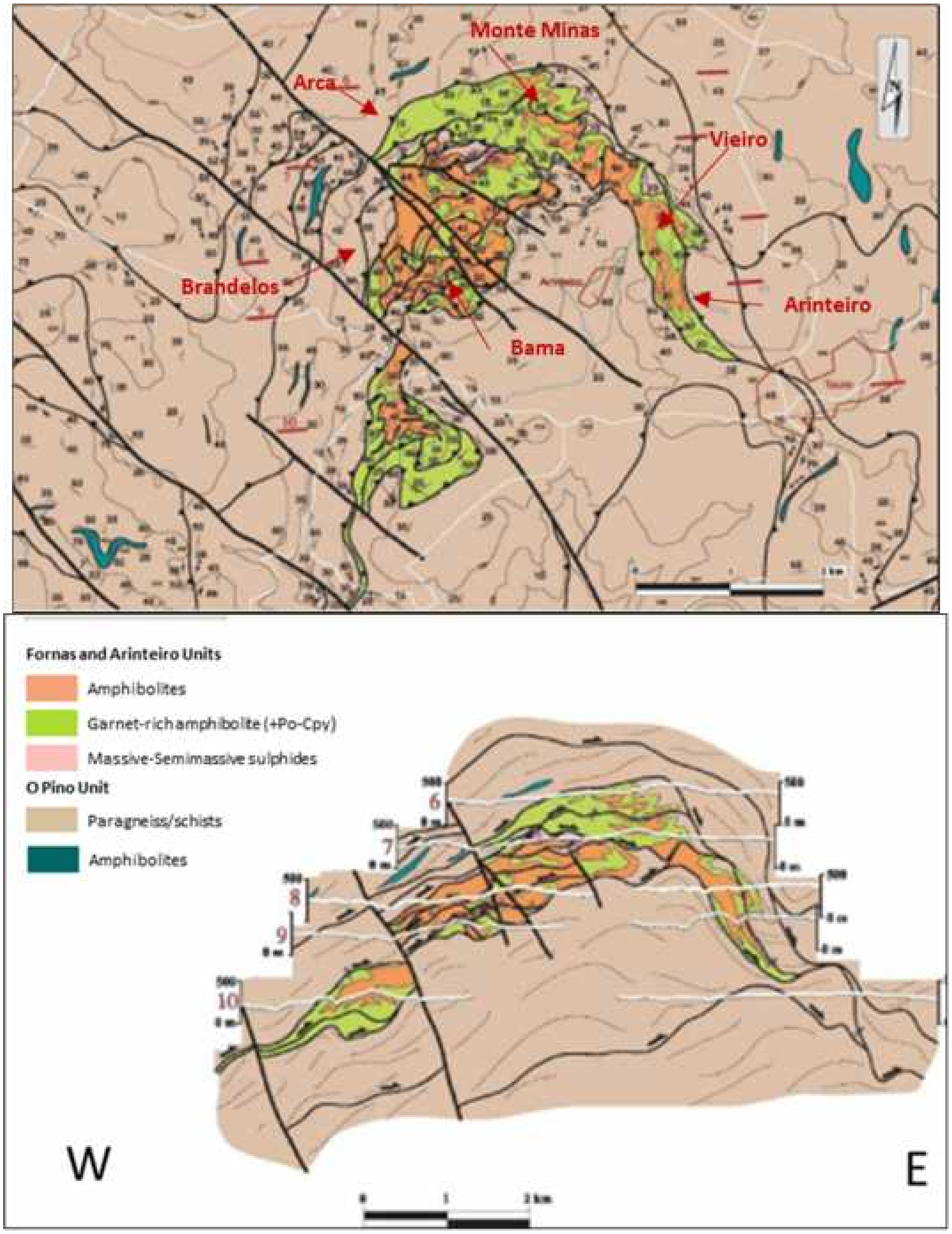


Figure 7.3 - Geological map and cross section of Touro Project area (Gómez Barreriro, 2007)

7.2.2 Mineralization

The ore zone consists of one or two mineralized horizons of disseminated mineralization closely associated with coarse grained garnet amphibolite that is poor in calcium. Garnet content is up the 90% of the volume of the rock (garnetite). The garnet amphibolite grades into non-mineralized normal amphibolite with decreasing garnet content.

The whole amphibolite horizon is surrounded by metasediments, mainly paragneisses that are normally barren or weakly mineralized although mineralized paragneiss has been described in Arca and Monte das Minas in the lower paragneiss.

In general, the ore bodies are tabular, shallow dipping, very consistent and continuous in terms of geometry but with variable thickness. The mineralization hosting amphibolitic metabasites are composed by gedrite, almandine and minor staurolite, and albite-oligoclase.

The mineralization is represented in order of abundance by pyrrhotite, chalcopyrite and minor pyrite and sphalerite. Williams (1983) described evidences of pre- and syn-deformation in garnets and sulphides (pyrrhotite and chalcopyrite).

The sulphides in the amphibolite are mainly disseminated and aligned with foliation planes, are either interstitial to amphibole crystals, infill the cracks of garnet porphyroblasts together with quartz, concentrated in pressure/strain shadows, or form veinlets parallel to the rock schistosity, with quartz, chlorite, and carbonates.

The semi-massive sulphide horizon is represented by rounded fragments of the host rocks within a pyrrhotite matrix with lesser amounts of chalcopyrite and minor pyrite and sphalerite. Massive sulphide orebodies in shear zones are also described in Fornás and Mañoca, further west of Touro. The zinc contents increase significantly in this type of mineralization compared with the disseminated ore type.

7.2.3 Structure and Metamorphism

The structure and metamorphism of the project area has been studied mainly by Zuuren (1969) and Williams (1983) and later by other authors like Castiñeiras (2002). A summary of the tectonic and metamorphic episodes described by Williams (1983) are presented in Table *7.1*.

In the project area, the whole sequence of metabasites and metasediments occur as a gently flattened, open antiformal fold that plunges slightly to the north (D3 fold according to Badham & Williams, 1981). Although the rocks are highly affected by the Hercynian deformation and metamorphism, the overall folding structure is very flat with a wide, partially eroded, hinge zone with approximately 1km width and limbs dipping around 30º.

The axial plane of the fold has a roughly north-south trend and divides the area into three zones, where up to five different deposits have been described: Arinteiro and Vieiro on the east limb, Bama and Brandelos on the west limb and Arca and Monte das Minas on the hinge zone.

According to the Atalaya Mining interpretation of geological vertical cross sections, the occurrence of synsedimentary extensional faults is postulated as the cause of significant changes in the thickness of the metabasites horizons in some areas of the project.

The rocks underwent two periods of amphibolite grade metamorphism (Table *7.1*). During M1 garnet crystallized in the metabasites and kyanite in metasediments reflecting moderately high confining pressures. M2 is defined as encompassing a period of recrystallization which occurred in association with D2 structures, but late D2 structures are associated with green schist facies retrogression, indicating that the M2 was a metamorphic event with both prograde and retrograde phases.

Williams (1983) described a diversity of mineral assemblages in the ore horizon, the predominant one is almandine + gedrite + biotite + staurolite ± quartz ± cummingtonite ± oligoclase. Most sulfide mineral in the ore horizon is in secondary concentrations as quartz-pyrrhotite-sphalerite-chalcopyrite. The ore horizon was profoundly affected by cataclasis related to the relatively late deformational events. This resulted in substantial local redistribution of the most mobile phases, notably quartz and sulfides.

Paragneisses are composed of plagioclase, quartz, muscovite, biotite, chlorite, garnet, and kyanite (± staurolite, rutile, and graphite) with a distinctive metamorphic assemblage of garnet + staurolite + kyanite. They show penetrative planar-linear fabrics, sometimes clearly due to shear deformation.

Table 7.1 - A Summary of the tectonic and metamorphic episodes of the Santiago Unit (modified after Zuuren,1969), Williams (1983)

Tectonic and Metamorphic Episodes	Comments	Related fabric
D1-M1	Style masked by subsequent events; expressed as intra-foliar folds	Enhanced original inhomogeneity and producing metamorphic layering
D2-M2	Recumbent, isoclinal to tight folds verge east, late stage brittle structures involved translation	Early-stage recrystallized hornblende producing a linear mineral fabric
D3	Generally upright tight folds with north-south striking axial planes, sub-horizontal plunge north	Epidote clinozoisite banding approximately parallel to F3 axial planes, crosscuts earlier fabrics
D4	Only locally expressed as open folds having sub-vertical axial planes, attitude variable related to thrusting	Mylonitic metabasites and mechanical sorting of minerals. Also, cataclastic fabrics in syn-tectonic minor granite intrusions

7.2.4 Mineralization Controls

According to Atalaya Mining, the main control of the mineralization is the presence of amphibolite, usually associated with garnets and chlorite alteration. It is noted, however, that mineralization may occur outside the garnet amphibolite, and that the garnet amphibolite may also be barren of mineralization.

7.3 Deposits

Copper mining in the project area started in the 1970's with Riotinto Patino (RTP) in the Arinteiro deposit. Several other deposits were discovered afterwards and mining continued in the 1980's on the Vieiro, Bama and Brandelos deposits. Further exploration by RTP during the 1980's and by Lundin Mining in 2012 extended the mineralization towards the north to the Arca and Monte das Minas deposits that remain unmined. Atalaya Mining further subdivided each deposit in different ore zones (Table 7.2 and Figure 7.4).

Within each of these deposits, the style of the mineralization is similar, occurring as elongated ore zone-lenses.

The local geology of the project has been reviewed by the Exploration Department of Atalaya Mining in 2017 based on field mapping observations and a set of geological sections produced from the drilling data. In total, 81 east-west cross sections, approximately each 100m apart and 4 north-south cross sections have been completed. An updated geological map based on the interpreted cross sections is presented in Figure 7.5.

Table 7.2 - Ore zones within each of deposits, Touro Project (provided by Atalaya Mining)

DEPOSITS	ORE ZONES
Arinteiro	Arinteiro
Vieiro	Vieiro, Middle Vieiro, Upper Vieiro and Vieiro NW
Brandelos	Brandelos and Brandelos N.
Bama	Bama, Lower Bama and Bama E.
Arca	Arca, Arca S, Lower Arca E and Upper Arca E.
Monte das Minas	Monteminas, Lower Monteminas, Monteminas S, Upper Monteminas W and Monteminas N

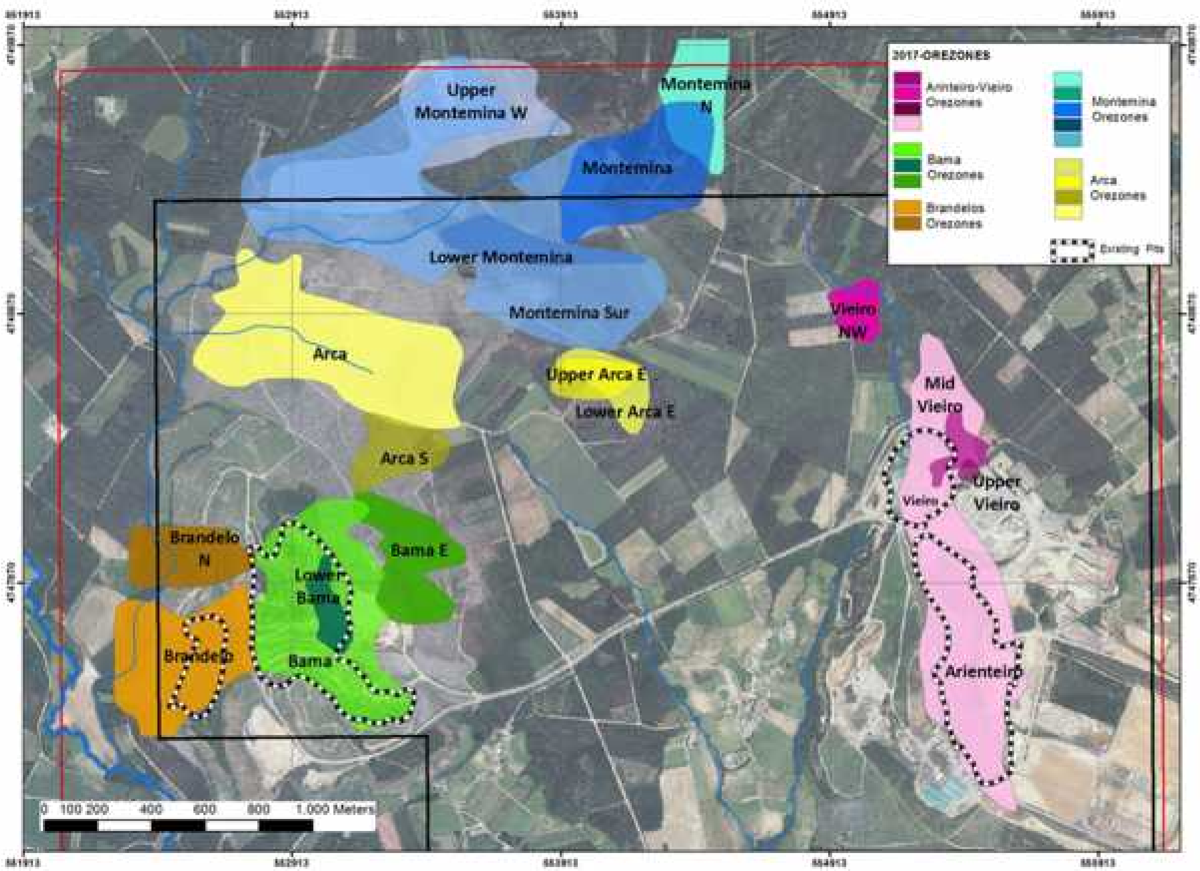


Figure 7.4 – Location of the different ore zones in the deposits, Touro Project (Atalaya 2017)

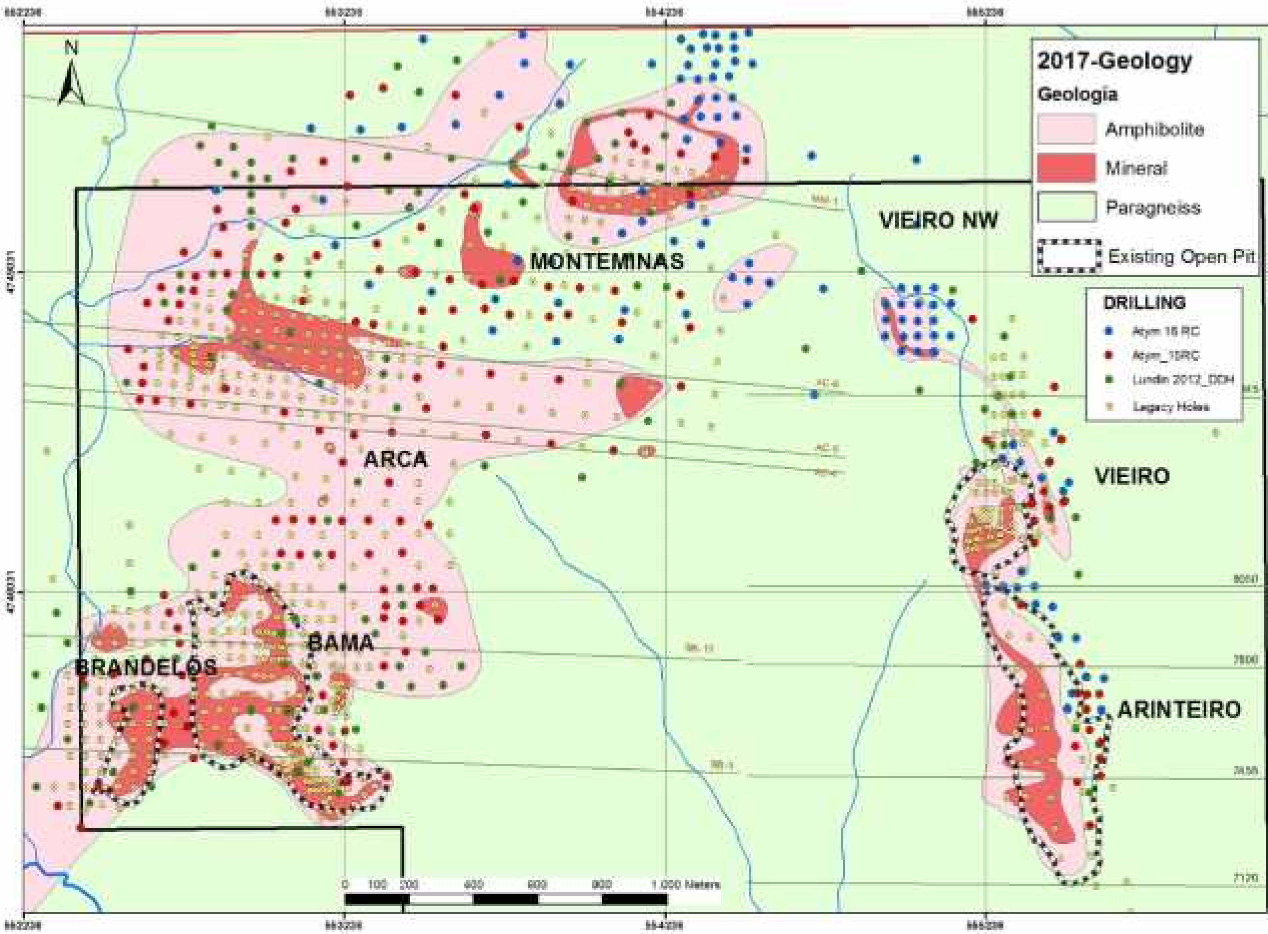


Figure 7.5 Updated Geological map (Atalaya 2017)

7.3.1.1 Arinteiro

The mineralization at Arinteiro is hosted in a single ore zone of garnet amphibolite of roughly N-S to N20E orientation and dipping around 25° to the East. The mineralized garnet amphibolite is flanked by paragneiss in both hanging wall and footwall.

Average thickness of the ore zone ranges from 20 meters up to a maximum of 56 meters (mineralized intersection in drill hole IAR11) and it appears to end at depth towards the east. Average grade is 0.62% Cu. Sulfide mineral assemblage is represented by pyrrhotite, chalcopyrite and minor pyrite (Figure 7.6).

The Arinterio deposit is aligned with and extends to the Vieiro deposit to the north along a N170 trend. In Atalaya's opinion, both deposits are thought to be probably formed from the same "feeder" structure, which could be located between the two deposits.

Part of the Arinteiro orebody was mined by open pit methods during the 1970's and 1980's, the total production reported by Rio Tinto Minera in 1985[1] was of 7,98 Mt at 0.73% Cu with a waste:ore ratio of 1.9.

Figure 7.6 - Mineralized specimens of Arinteiro garnet amphibolite, hole IAR14 (Atalaya 2017)

7.3.1.2 Vieiro

Vieiro is located immediately north of Arinteiro deposit. The mineralization is also hosted in garnet amphibolite that occurs as interdigitated lenses within the paragneisses (Figure 7.7), therefore making the geometry of the orebody more complex than Arinteiro. This complexity is responsible for significant variations in thickness of the ore zone. The general dip of the ore zone is 25º to the east, but on the northern side of the deposit the structure plunges around 15º to the north.

Maximum ore zone thickness intercepted by Atalaya Mining drilling is 56 meters at 0.60% Cu with the best grade intersection of 16 meters at 1.46% Cu.

Part of the orebody was also mined by open pit during the 1970's and 1980's, the total production reported by Rio Tinto Minera in 1985 was of 1 Mt at 0.70% Cu with a waste:ore ratio of 4.2.

[1] The source of this information was provided as part of the scanned legacy documents from RTP/RTM.

Figure 7.7 – Mineralized specimens of Vieiro garnet amphibolite, hole IVR12 (Atalaya 2017)

7.3.1.3 Bama

Bama is located on the hinge zone and western limb of the anticline. Mineralization is also hosted in garnet-rich amphibolite and flanked by paragneisses. It consists of disseminated pyrrhotite and chalcopyrite. Thickness of the mineralized ore zone varies from 15 meters up to 60 meters.

In Atalaya's opinion, the main Bama ore zone has a "graben-like" geometry that is thought to be associated to synsedimentary faults. The shape of the orebody is gently adapted from the margin of the deposit to the center, where the orebody has a greater thickness.

The deposit was mined by open-pit mining in the 1980's, the cumulative production reported by Rio Tinto Minera in September 1985 was of 9,23 Mt at 0.53% Cu with a waste:ore ratio of 1.4, mining in Bama ceased in 1986.

7.3.1.4 Brandelos

Brandelos is located in the western limb of the anticline and dips gently to the west about 10º to 20º. The ore zone is hosted in garnet amphibolite and flanked above and below by paragneisses. The ore zone forms a tabular lens with thickness varying between 15 to 50 meters and average grade of 0.40% Cu. Brandelos is the western extension of the Bama ore zone and remains open to depth down-dip to the west.

The deposit was mined by open-pit mining in the 1980's, the cumulative production reported by Rio Tinto Minera in September 1985 was of 77,705 tonnes at 0.43% Cu with a waste:ore ratio of 10, mining in Brandelos ceased in 1986.

7.3.1.5 Arca

Arca is located in the west-central part of the Project area and connects with Monte das Minas to the north.

Drilling has identified the occurrence of several mineralized lenses, of which the biggest one is located on the western part of Arca. The ore zones are sub-horizontal in the central area, and dip 20º-30º to west in the western side. Most of the lenses are garnet amphibolite with thickness varying from 10 meters up to 100 meters with an average grade of 0.42% Cu (Figure 7.8). The thickest drill hole intersection of Atalaya Mining drilling is 112 meters at 0.46% Cu. The lower mineralized horizon is represented by breccia textured semi-massive sulphides hosted in paragneiss.

According to Atalaya Mining, the projection of the thickness and grades of the ore zones suggest two main directions related to the spatial distribution of the orebody: N30E and East-West. These directions could be related to the location of the feeder zones associated with the deposits of Arca and Monte das Minas.

This deposit has never been mined.

Figure 7.8 – Garnet amphibolite in Arca, hole IAC30 (Atalaya 2017)

7.3.1.6 Monte das Minas

Located in the northern zone of the project area, mostly in the hinge zone of the antiform, this deposit is the extension of the Arca ore zone.

The deposit consists of up to 5 lenses of disseminated sulfides, the upper ones hosted in garnet amphibolite and the lower one in paragneiss (Figure 7.9). The lenses are sub-horizontal and occur at shallow depths, ranging in width between 10 meters and 60 meters. Average grade is 0.45% Cu, although there are some high-grade zones with up to 1% Cu.

Monte das Minas deposit has never been mined.

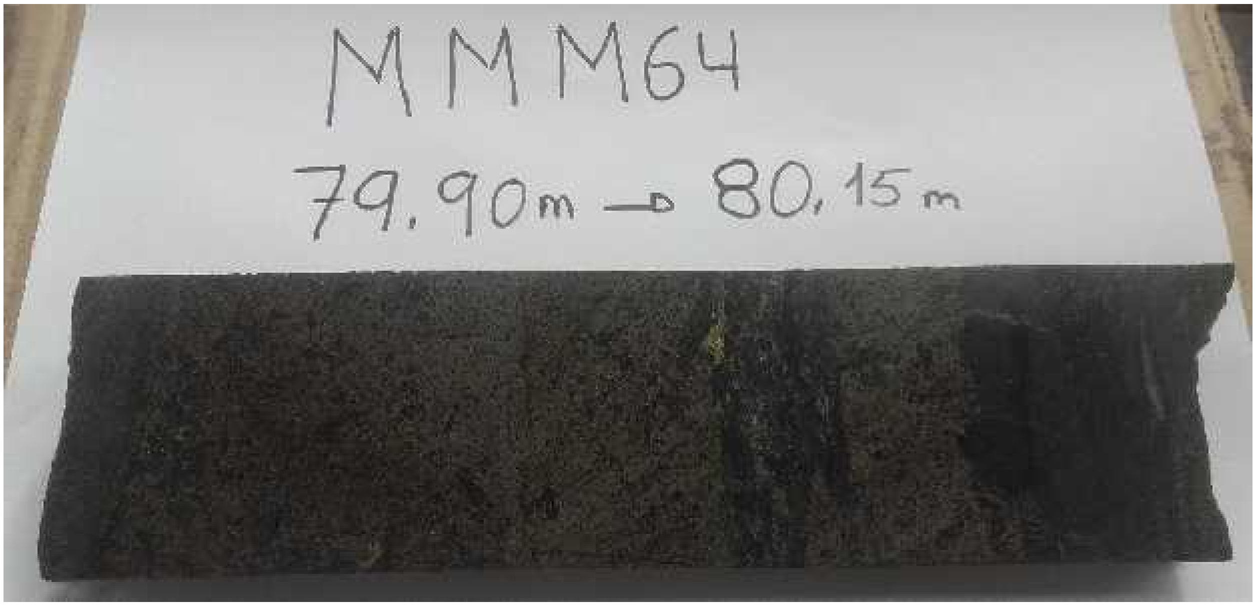


Figure 7.9 – Paragneiss with sulphide mineralization in Monte Minas hole MMM64 (Atalaya 2017)

8 DEPOSIT TYPES

This section was initially compiled by the Atalaya Mining technical staff and was updated and reviewed by Monica Barrero Bouza with Alan Noble, both of whom are Qualified Persons for the purpose of NI 43-101, Standards of Disclosure for Mineral Projects.

The deposits of Arinteiro, Bama, Fornás and Manoca have been characterized as metamorphosed ophiolitic VMS deposits (Badham and Williams 1981; Williams 1983) and classified as Cu or Cu–Zn type (Castroviejo, 2001). More recent studies suggest an origin related with syn-metamorphic metasomatism channeled through shear zones (Castiñeiras et al., 2002; Gomez Barreiro, 2007).

The compiled observations of Lundin Mining in 2012 are in line with the more recent studies, suggesting a controlled shear zone hosted deposit type similar to the Lumwana deposit in Zambia.

In Atalaya's opinion, the deposit can be classed as Mafic Siliciclastic type (Besshi-type) Volcanogenic Massive Sulfide (VMS) deposit, according with the lithological classification of Shanks and Thurston in Figure *8.1* and equivalent to the pelitic–mafic VMS deposits of Galley in Figure 8.2.

The siliciclastic-mafic-type VMS deposits (Besshi type) are well described in the work of Shanks and Thurston (2012), they occur in mature, oceanic, back-arc successions in which thick marine sequences of clastic sedimentary rocks and intercalated mafic, and occasionally ultramafic, rocks are present in sub-equal or pelite-dominated proportions. The mafic component is largely composed of volcanics with MORB-like affinities.

Besshi-type deposits are conformable, stratiform, blanket-like sheets of massive pyrrhotite and/or pyrite with variable contents of chalcopyrite, minor sphalerite and rare galena, being all lead-poor mineral assemblages in general. Copper is the principal economic metal, and there is subordinate zinc, cobalt, silver and/or gold.

The ores occur in submarine mafic volcanic rocks and associated marine sedimentary rocks including metagraywacke, quartzite, and metapelite that are mostly deep-water facies. The mafic volcanic rocks are volumetrically subordinate to sedimentary rocks. Felsic meta-igneous rocks are rare or absent. Sheet-like deposits are characterized by high aspect ratios in which the lengths of sulfide zones exceed thicknesses by an order of magnitude or more.

The ore is characterized by massive sulphide zones, ranging into semi-massive to increasingly sparse sulfide disseminated. This gradation may represent the vertical transition with an underlying stockwork related with the feeder zone of the hydrothermal system.

According to Atalaya Mining, the amphibolites represent sub-volcanic sheet-like sills of basalts intruded into a siliciclastic sequence represented by the paragneisses. The basaltic rocks that host the mineralization had undergone intense chloritization associated with hydrothermal processes. Mineralization is mainly associated with the garnet amphibolites, occasionally in the western and northern deposits mineralization is also hosted in paragneisses.

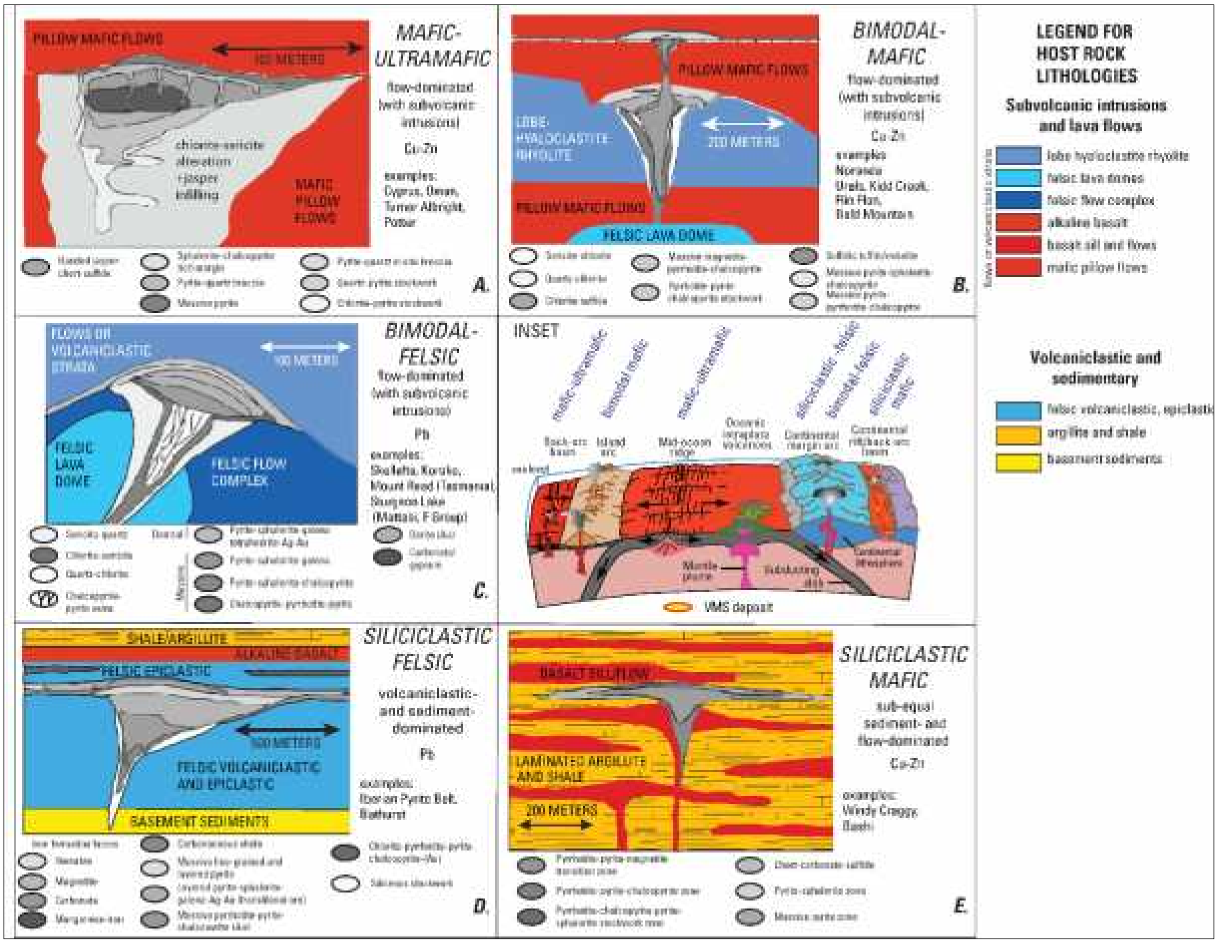


Figure 8.1 - Representation of the lithological classification of VMS deposits (Shanks and Thurston, 2012)

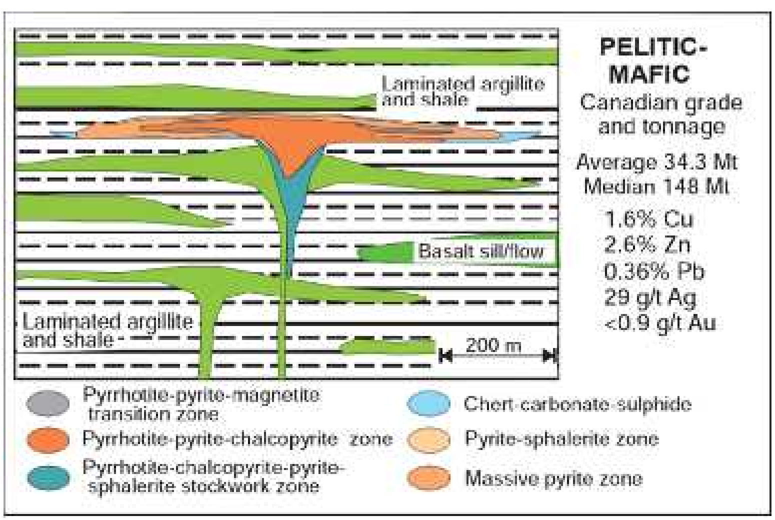


Figure 8.2 - VMS Besshi Type Deposits (Galley 2007)

In the authors' opinion, the Touro deposit is heavily metamorphosed and deformed, but metamorphosed hydrothermal alteration footprints are still identifiable. It seems obvious that the metabasite/amphibolite unit was originally an igneous rock of mafic composition, but whether this was originally a basalt or a gabbro sill is still unclear.

Although the garnet-amphibolite and garnetite can be the metamorphic analogous of the chloritic alteration aureole of VMS deposits, in Touro the hydrothermal alteration halo is confined only to the core and lower part of the metabasites.

It cannot be established with certainty whether the nature of the mineralization is syngenetic or epigenetic, since deformation, remobilization, and recrystallization are prevalent throughout the deposit. The relation of the disseminated mineralization in the garnet amphibolites and the semi-massive sulphide breccia hosted mineralization in the underlying paragneisses has not yet been determined.

It is the view of the authors of this report that while Touro deposit shares some similarities with Besshi-type VMS deposits (e.g. tabular shape, lead-poor sulphide associations, mafic rocks subordinate to sedimentary rocks), the genetic model needs better and more precise definition. While the uncertainty about the genetic model does not negatively affect the reliability of the resource estimate, a more comprehensive understanding would be valuable for continuing geologic investigations.

9 EXPLORATION

This section was compiled by the Atalaya Mining technical staff and reviewed by Monica Barrero Bouza and Alan Noble, both of whom are Qualified Persons for the purpose of NI 43-101, Standards of Disclosure for Mineral Projects.

9.1 Summary

The Project went through several stages of geological exploration undertaken by different mining companies for a period of over 45 years.

Exploration started in the region in the 1960s by Rio Tinto (RTP and RTM) with activities in San Rafael and other mining permits in the vicinity of the current Touro Project (Maldonado, 2012). RTM undertook a comprehensive exploration program that included:

- Geological mapping (regional and local scale).
- Geophysical exploration: airborne and ground electromagnetic and magnetometer surveys (EM electromagnetic Gun survey, ground Turam method).
- Soil geochemistry programs.
- Shallow exploration by trenching and Voletrac drilling.
- Diamond core drilling.

Lundin Mining completed a due-diligence program in 2012. Exploration was carried out in both the current project area (San Rafael) and in Fuente Rosas to the west of the project area, including geological mapping and diamond core drilling. Outside the current project and Fuente Rosas, Lundin Mining compiled available information from the exploration by Rio Tinto between 1965 and 1986 (Maldonado, C., 2012).

Since 2015, Atalaya Mining has been focused on San Rafael mining permit. Exploration carried out includes legacy data gathering, integration of legacy spatial data in Geographic Information Systems (GIS), geological mapping of the San Rafael permit, diamond core and reverse circulation drilling (15RC and 16RC drilling programs) and soil geochemistry over limited geographic areas.

9.2 Data Gathering and GIS Data Integration

In 2015 Atalaya Mining started gathering legacy data from the time the Touro Project was owned by RTP/RTM with the aim of validating and verifying the legacy drill hole database and improving and broadening knowledge about the deposit and all the exploration work that had been done since the early stages of the project.

The extensive record of legacy documentation that remains from the investigations and exploration of RTP/RTM was obtained from different sources (Lundin Mining, Geotrex, Explotaciones Gallegas, historical file system from RTP) and in different formats.

The available legacy spatial data related to geological and structural data, mineral occurrence and exploration documentation including maps and sections were integrated into a Geographic Information Systems (GIS) database in the appropriate Reference System (ETRS 89).

9.3 Geological Mapping

Several detailed geological maps of the project area are found in the studies published by IGME (1975, 1977), and the more recent work of Gómez Barreiro (2007).

Various geological maps and many cross sections from RTP/RTM at different scales and with a differing degree of detail were provided for review, evidencing the thorough work undertaken during the exploration works.

In 2012 Lundin Mining produced an outcrop map of the project area and Fuente Rosas with detailed mapping of the main mine works and other areas where outcrops exist. In addition, they updated the interpretation of the deposits in geological cross sections, primarily based on the geological map of Gómez Barreiro (2007), (Figure 9.1).

Atalaya Mining has developed a new geological map that is based mainly in the interpretation of vertical sections and geological mapping is restricted to the northern area of the project (Figure 9.2).

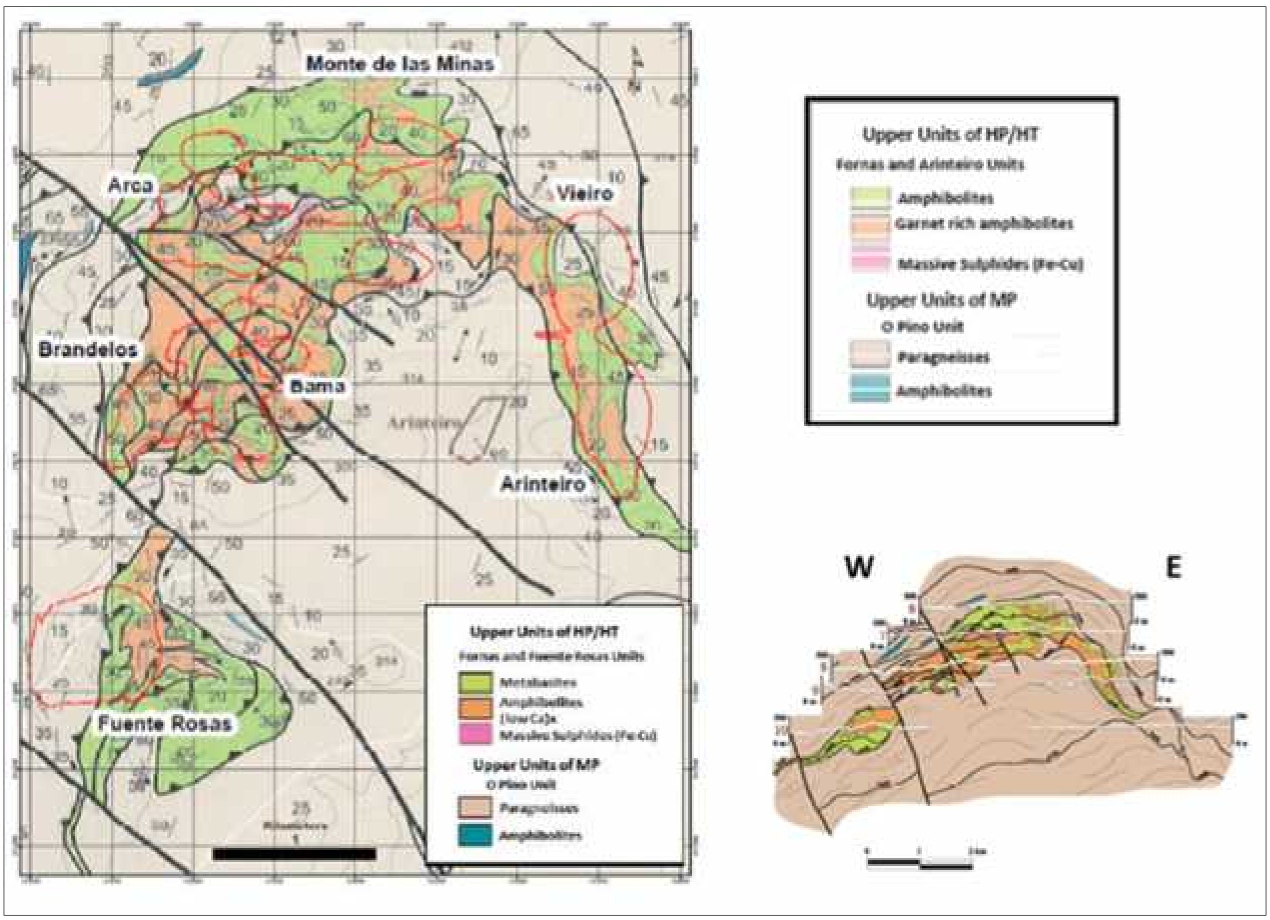


Figure 9.1 - Geological Map, (Gómez Barreiro, 2007))

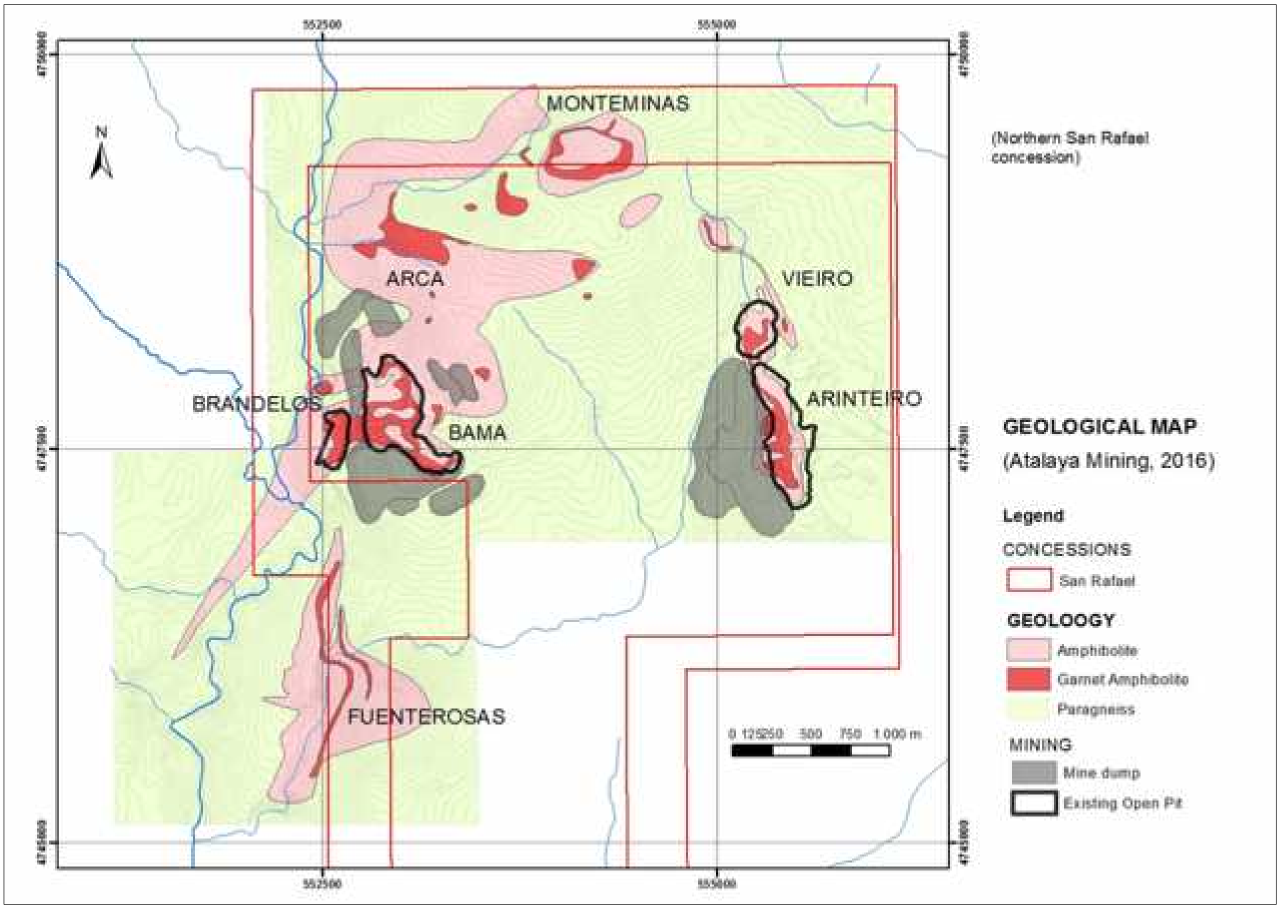


Figure 9.2 - Touro Project Geological map, (Atalaya 2016)

9.4 Geophysics

Airborne and ground electromagnetic and magnetometer surveys (e.g. EM electromagnetic Gun survey, ground Turam method, Jalander Magnetometer) were performed by RTP/RTM in the late 1960's to early 1980's. Atalaya Mining has retained part of this data in the form of hard copy maps, though the data hasn't been re-processed.

The most relevant geophysical work in the region is an airborne electromagnetic and magnetometer survey completed by RTP in 1968. The survey was performed by helicopter at an altitude of 60m. The survey line spacing was approximately ¼ mile and covered an east-west elongated rectangle area of approximately 42 km by 26km. The resulted map shows a series of contours representing outlines of the main EM anomalies with magnetic correlation.

In 2012, Lundin Mining did not do any geophysical surveys over the project area, though routine geophysical measurement on core was performed, including:

- Routine measurements of magnetic susceptibility (and conductivity) with a GDD Inc. MMP-EM2S instrument. There are records every 1.1m (average measurement interval) in 119 of the 169 holes,

- Measurements of resistivity and chargeability with a core IP Tester (GDD Inc. model TDLV), on 86 representative samples from the different lithological types from all the sectors of the project.

Data has been accessed but there is no indication that Atalaya Mining has re-processed this data.

Atalaya Mining has not conducted any geophysical surveys to this date over the project area, though the existing data has been compiled for future evaluations.

9.5 Geochemical Sampling

Exploratory geochemical soil sampling has been conducted in the area since the beginning of the investigations by RTP/RTM in the 1960's. Currently available data is limited, consisting of several contoured soil geochemical anomaly maps and descriptions of the geochemical sampling grids that are found in a very few documents that have been reviewed (e.g. internal RTP reports, IGME, 1977). There is evidence of this type of investigations in some areas of the San Rafael mining concession. Investigations in San Rafael showed that, after discarding contaminated areas, the values below 1000ppm Cu were schist or leached areas, values between 1000 and 2000ppm corresponded to fine grain amphibolites and values above 2000ppm were located on ore outcrops.

Lundin Mining compiled and evaluated the available legacy data but did not perform any geochemical surveys.

Atalaya Mining completed a soil geochemistry survey in Arca and Monteminas where the mineralization occurs at shallow depths, with the aim of testing the effectiveness of the technique for definition of new target areas rather than an exploratory survey (Figure *11.4*). A total of 138 samples were collected using a soil auger sampler at depths of approximately 1m. The samples were sieved to produce two fractions of >180µ and <180µ sizes and the two samples assayed by an XRF spectrometer afterward. The results clearly indicate that the copper anomalies are better defined by the fine fraction of the soil sample.

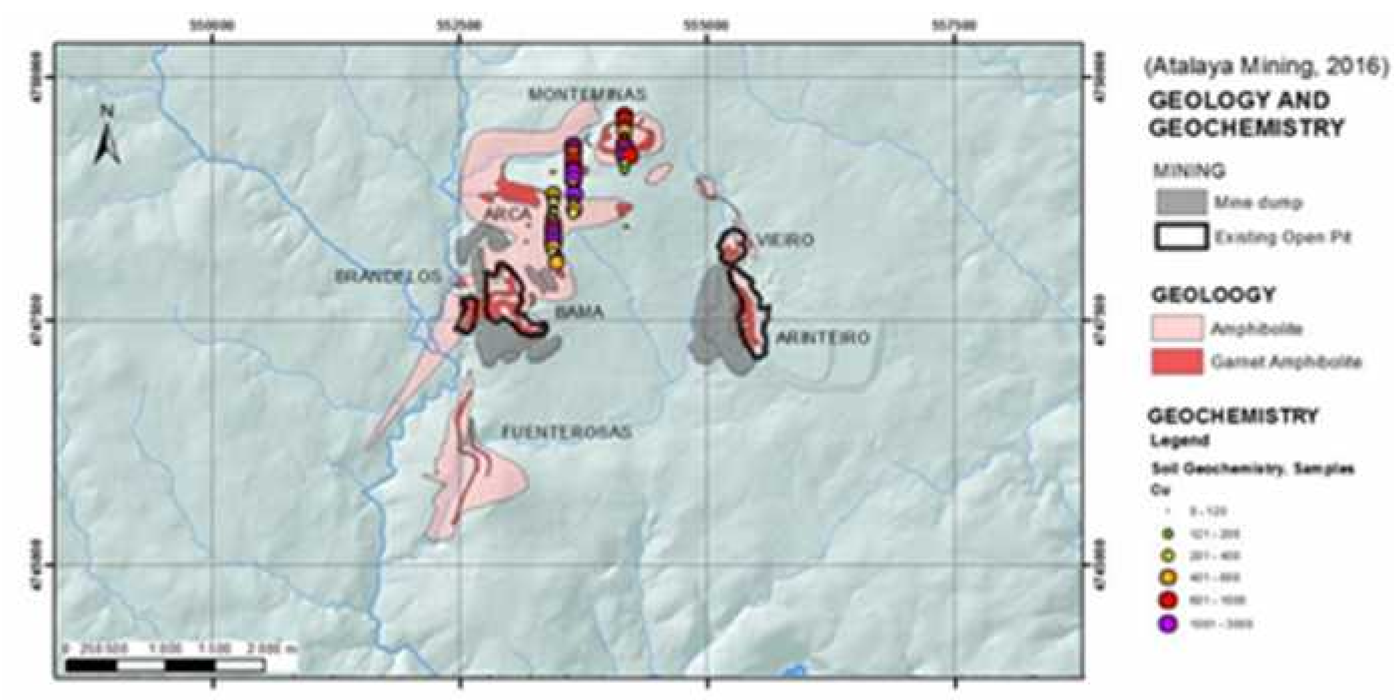


Figure 9.3 - Geochemical sampling and anomaly map, (Atalaya 2016)

9.6 Petrology, Mineralogy, and Research Reports

There is a complete research study of the Arinteiro and Fornás deposits published by IGME (1977) that includes a complete petrographic characterization of the lithology and mineralization through the analysis of polished and thin sections, and geochemical rock analysis (XRF and X-ray diffraction) of samples collected in the Arinteiro and Fornás pits.

Juan Gómez Barreiro (Geology Department of Salamanca University) completed a petrographic study on behalf of Lundin Mining in 2012. The study includes petrographic descriptions and analysis of thin and polish sections of 47 core samples from San Rafael and Fuente Rosas.

Atalaya Mining has not completed any petrological or mineralogical study to date.

9.7 Metallurgical Studies

There are numerous legacy documents available related to metallurgical studies of San Rafael, Fuente Rosas, and Fornás deposits. Lundin Mining started a process of compilation of this type of legacy documents which has been taken up by Atalaya Mining and it is still ongoing.

Wardell Armstrong International (WAI, 2012), on behalf of Lundin Mining, conducted a mineralogical analysis and preliminary laboratory flotation test work on core samples of Bama, Arinteiro, Arca and Fuente Rosas.

A detailed test work program was completed between Q4 2015 and Q2 2016 by Atalaya Mining to confirm the process flowsheet and assess the geo-metallurgical response of Touro ores.

The overview and detailed discussion of the results and interpretations are presented in Section 13.

9.8 Conclusions

The exploration programs of RTP/RTM in the project area are considered adequate and sound. The exploration procedures adopted are considered in line with current industry practices and appropriate to support resource estimation.

The compilation of the legacy exploration data is of great value and could be used as the basis for further exploratory programs in the area.

10 DRILLING

This section was initially compiled by the Atalaya Mining technical staff and was reviewed and updated by Monica Barrero Bouza with Alan Noble, both of whom are Qualified Persons for the purpose of NI 43-101, Standards of Disclosure for Mineral Projects.

10.1 Summary of Drilling

The Project went through several stages of geological exploration by different mining companies over a period of 45 years. A summary of the historical and recent drilling is presented in Table 10.1.

The majority of the exploration and mining activities, during the 1970s and 1980s, were conducted by Rio Tinto Patiño (RTP) and Rio Tinto Minera (RTM) on the San Rafael exploitation or operating concessions.

Lundin Mining also carried out exploration activities in the San Rafael operating concession and in the Fuente Rosas investigation permit (to the west of San Rafael), whereas Atalaya Mining is focused only on San Rafael operating concession. The drilling operators were Geonor and CGS (Compañia General de Sondeos).

Legacy drilling by RTP/RTM and Lundin Mining drilling were completed using diamond coring tools whereas Atalaya Mining mainly used reverse circulation methods in some cases combined with diamond drilling.

In the current project (San Rafael), drill core is available for re-logging only for the Lundin and Atalaya drilling programs.

The main objectives of Atalaya Mining drilling programs are to confirm or gain confidence and expand the historical resource of the project through confirmation, infill, and exploration drilling (step-out). The main drilling operator for Atalaya Mining is SPI (Sondeos y Perforaciones Industriales del Bierzo).

Table 10.1 - Summary of drilling at the Touro Project (from 1970s to present)

		Drilling (meters)			Drill Holes			
Company	Date	DD	RC	Total	DD	RC	RCDD	Total
RTP-RTM	1970s-1980s	59,876.34	-	59,876.34	668	-	-	668
Lundin Mining	Jan-Aug 2012	20,282.85	-	20,282.85	169	-	-	169
ATM-15RC	Nov-15 to May-16	2,027.15	12,249.95	14,277.1	3	124	25	152
ATM-16RC	Oct-16 to Jun-17	1,443.3	10,838.5	12,281.8	1	93	26	120

10.2 Atalaya Drilling

Atalaya Mining began its drilling program in Touro in November 2015, completing the first stage in May 2016 (15RC program).

The program was initially designed as a fully reverse circulation (RC) drilling campaign with two RC drilling rigs on site (Atlas Copco Mustang 5F4 and 5P4). In November 2015, a diamond drilling (Spidrill 160D) rig joined the program to improve drilling capabilities and efficiency in the deepest holes and to increase recovery in the presence of water. Therefore, switching from reverse circulation to diamond drilling was necessary for some of the holes. In February 2016, a third RC drilling rig was also mobilized in order to speed up the program.

RC drilling has been the main drilling method employed by Atalaya Mining since 2015. RC was generally carried out using Atlas Copco Mustang 5F4 and 5P4 drill rigs using bits ranging in diameter from 14.0 to 14.6 cm (5 ½ to 5 ¾ inches). RC drilling was performed dry to ensure proper movement of drill cuttings through the drill stem, cyclone and sampling equipment, and for dust control. Drilling campaigns 15RC and 16RC were completed by drilling contractor SPI. When the RC drilling encountered groundwater in the deeper holes, the method was changed to core drilling. Atalaya core holes are also drilled by SPI using a Spidrill drill rig and HQ (63.5 mm) size core.

A total of 152 holes were completed of which 124 were RC holes, 24 were combined RC/DD, and 3 were DD holes. In total, 14,278 meters of drilling were completed, of which 12,230 meters were RC and 2,048 meters were DD. The average bore hole depth was 94 meters.

Most of the completed holes were infill holes (to gain confidence in the ore zone continuity) as well as step-out holes (to expand the extension of the orebody). Additionally, five "twin-holes" were drilled, consisting of two DD holes to check RC grades and three twin RC holes to validate historical drill hole grades. In addition, PVC casing was installed in twenty of the completed holes for water level monitoring purposes.

The second campaign of the Atalaya Mining drilling program began in October 2016 (16RC program) and ended in April 2017. The purpose of this program was also infill and step-out drilling. A total of 120 drill holes were completed with 93 RC holes, 1 DD hole, and 26 combined RC/DD holes, totalling 10,838.5 meters of RC and 2,443.3 meters of DD. Average drill hole depth was 95 meters.

10.3 Geological Logging

A copy of the drill hole logs for 54 holes (S-1 to S-94) dated 1967 and 1968 of the Arinteiro and Vieiro deposits (Figure 10.1) was provided for review and validated with the scanned legacy information and the digital drill hole database. The information includes a handwritten record of hole sample intervals, core recovery, description of lithology, mineralization and structure, and %Cu and %S assayed values of the original and duplicate samples. No legacy core is available for re-logging.

Of the 169 Lundin Mining core holes, 142 were drilled within the limits of the current project. All holes were geologically logged on paper forms and 116 of the 142 holes were also geotechnically logged. The geological logs (Figure 10.2) were scanned and provided for review in electronic format where both alpha and numeric codes were assigned to each lithology. The lithology code description was also provided. A check of the geological intervals of 16 of the Lundin's holes (11% of the total holes) against the drilling database found less than 1% of errors in the depth and lithology codes of the lithological intervals.

Atalaya Mining used two paper logging forms (Figure 10.3 and Figure 10.4) to record RC drilling and diamond drilling data. RC chips samples are logged in an appropriate form sheet that includes identification of lithology, alteration, mineralization and other notable characteristics. Every sampled interval is stored in the chip trays and photographed; the picture clearly shows the sample numbers and interval of sampling. Diamond drill core is logged using a more detailed geological form that includes lithology, stratigraphy, detailed structure, alteration, and mineralization. Additionally, the main geotechnical features such as recovery and RQD are recorded in a separate paper form.

The geological logs of 91 holes of a total of 152 drilled in the 15RC program were supplied for review. A check of 11% of the holes were against the scanned logs (19% of the logs) found about 7% errors in depths and lithology codes of the lithological intervals (typing errors).

There were no geological descriptions or logging available for the drill holes of the 16RC drilling program at the time of writing this report.

Drilling data was entered daily in a spreadsheet that included: drill program (planned holes), location, drill progress (holes completed), type of drilling, coordinates, dip, azimuth, sampling interval, and other relevant information.

The lithology codes used for logging at Touro are presented in Table 10.2.

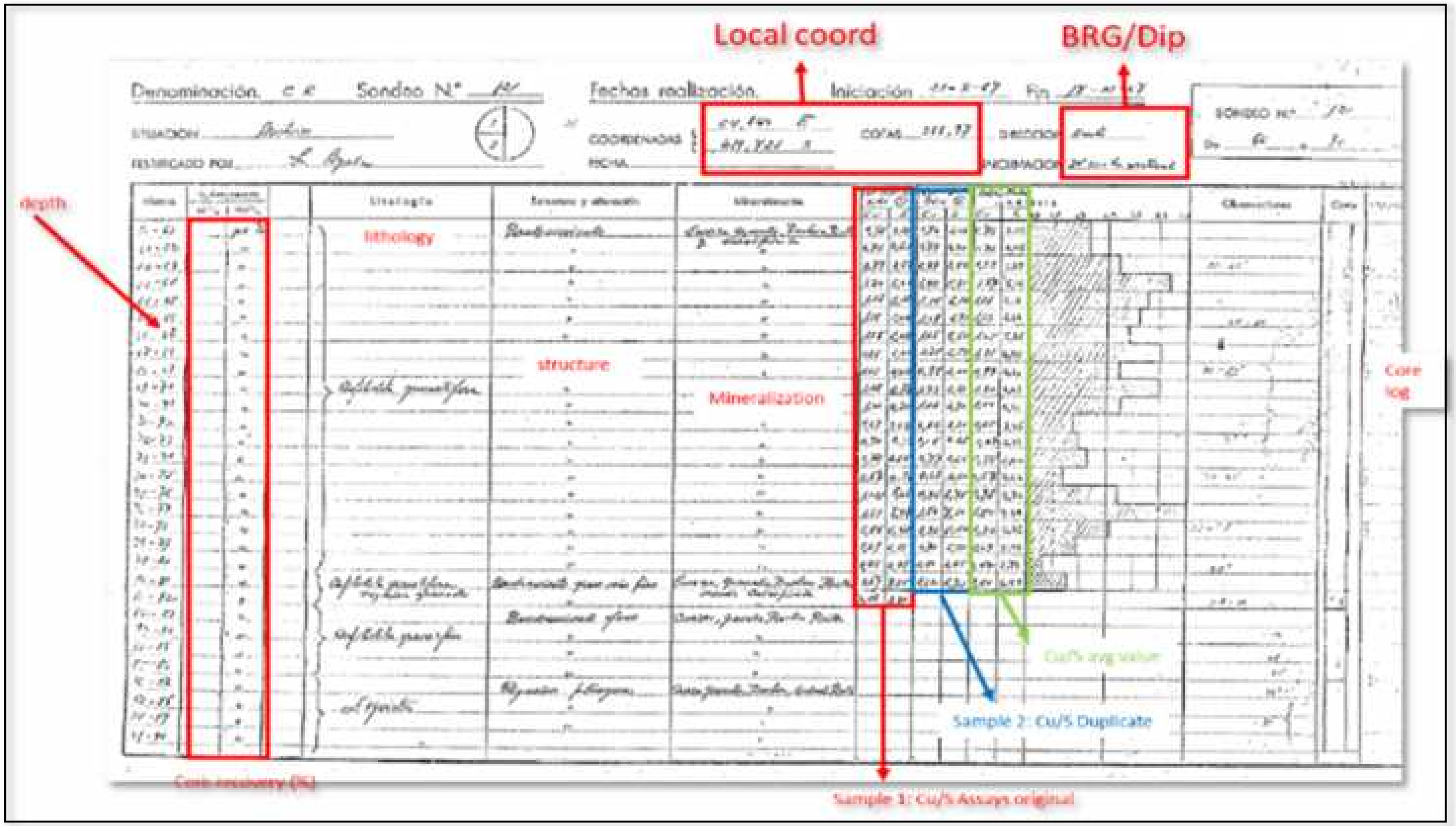


Figure 10.1 - Legacy Geologic Log - Drill hole CR12 dated in 1967 (Barerro 2017)

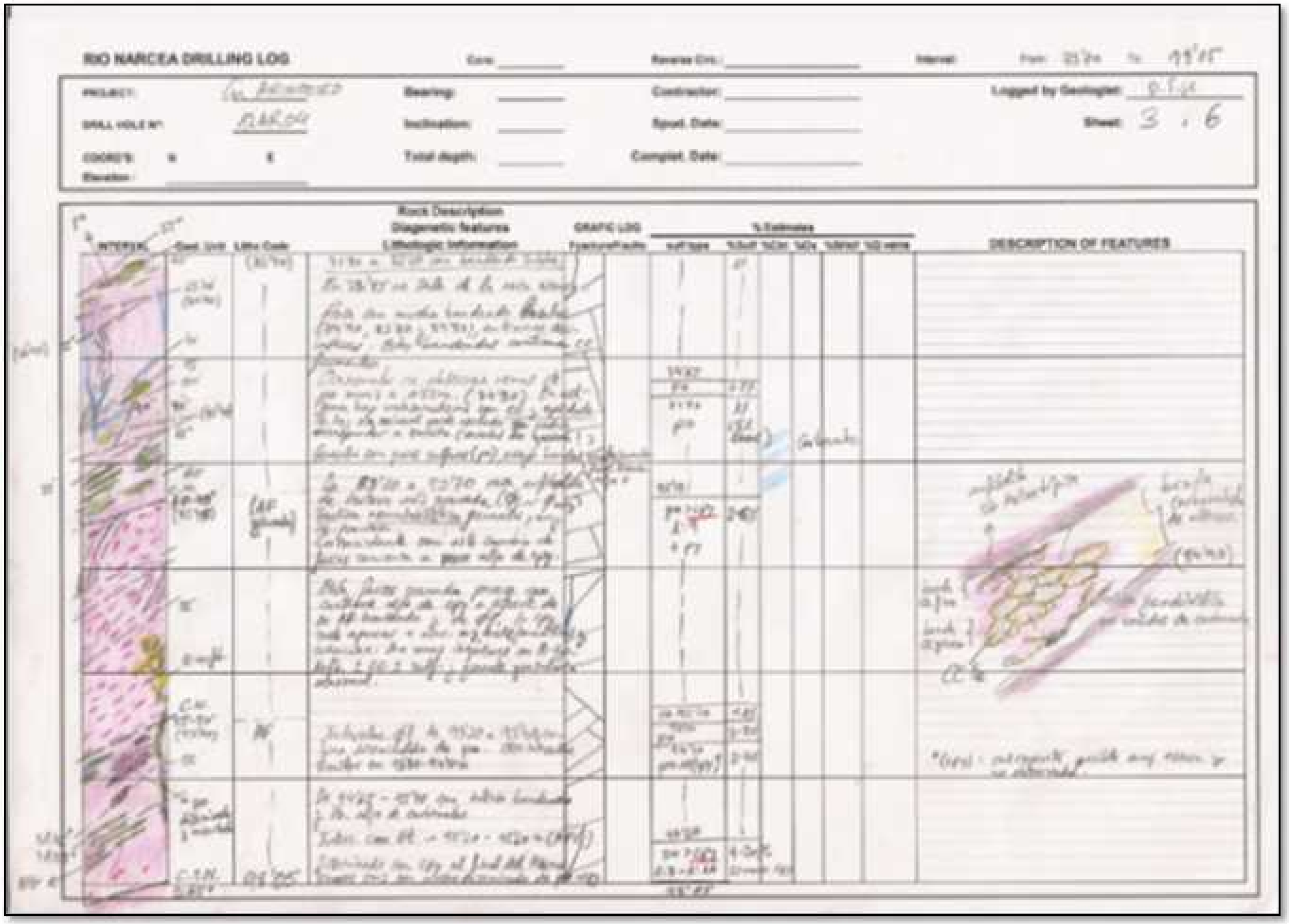
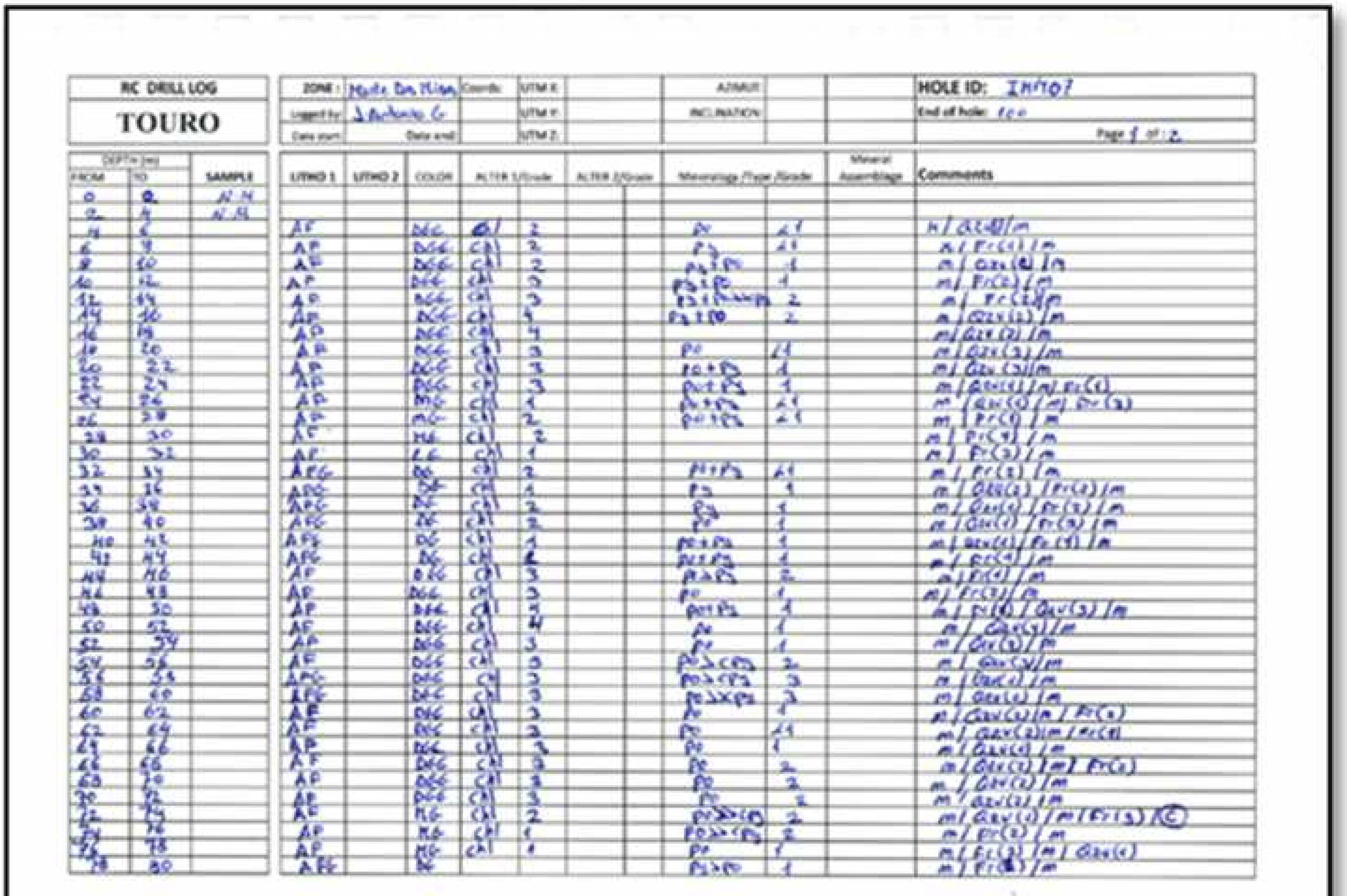


Figure 10.2 - Example of Lundin Mining drill log - hole 12AR04 (Lundin 2012)


RC DRILL LOG

TOURO

ZONE: | Coords: | UTM X: | AZIMUT: | HOLE ID: IMM07
Logged by: | UTM Y: | INCLINATION: | End of hole:
Date start: | Date end: | UTM Z: | Page 1 of 2

DEPTH (m) FROM | TO | SAMPLE | LITHO 1 | LITHO 2 | COLOR | ALTER 1/Grade | ALTER 2/Grade | Mineralogy /Type /Grade | Mineral Assemblage | Comments

Figure 10.3 - Example of an Atalaya RC drill hole log – hole IMM07 (Atalaya 2015)

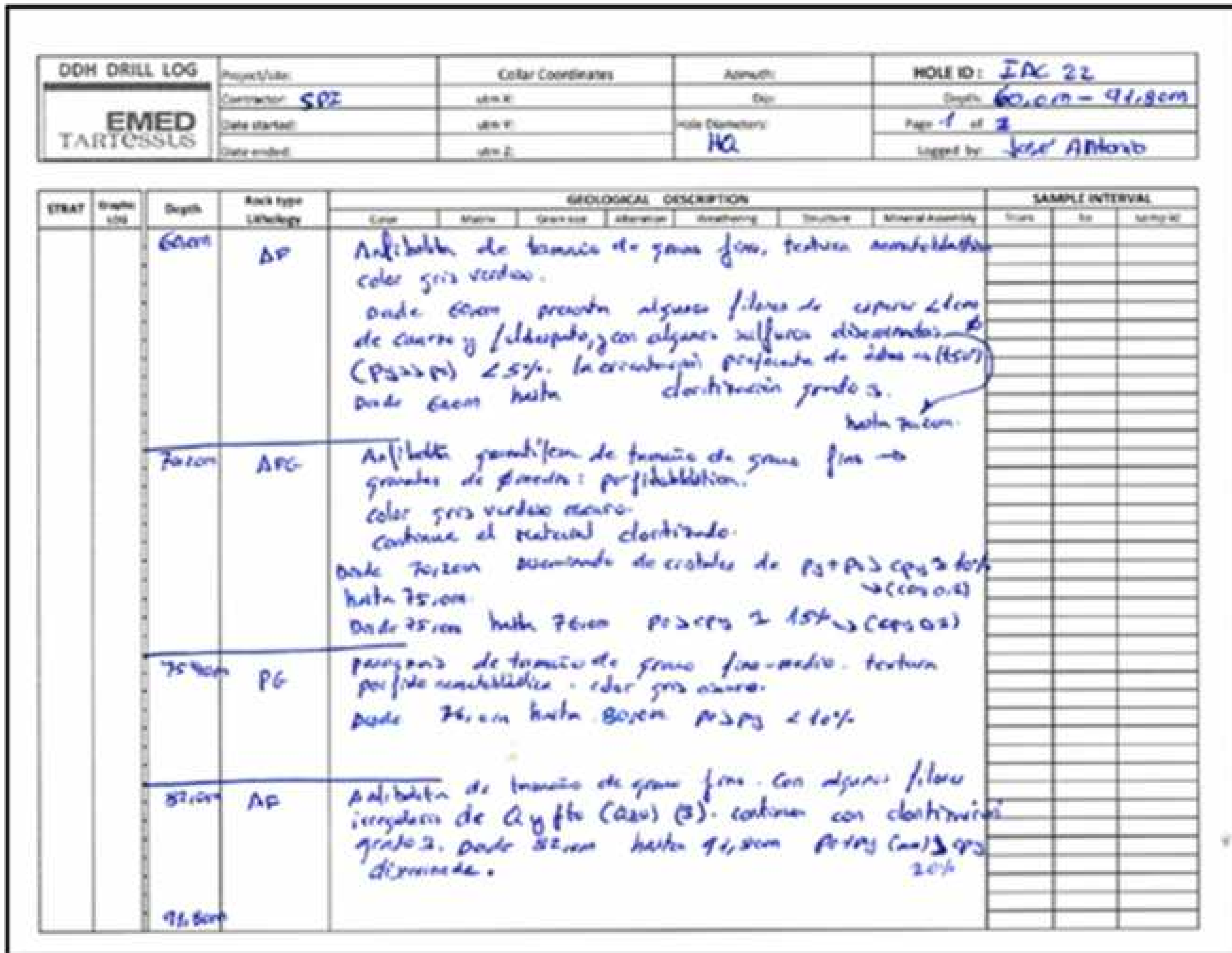


Figure 10.4 - Example of an Atalaya diamond drill hole log – hole IAC22 (Atalaya 2015)

Table 10.2 - Lithology and mineralization codes used at Touro (Atalaya Mining, 15RC and 16RC)

TOURO PROJECT: Lithology and Mineralization codes for logging					
Lithology Code	**Description**	**Alteration Code**			
		Type	**Description**	**Intensity**	**Description**
REC	Soil, Overburden, waste dump				
PG	Paragneiss	WTH	Weathered	1	Very weak
AF	Amfibolite	OX	Oxidation	2	Weak
AFG	Garnet-Amphibolite (with sulphides)	AG	Argilitic	3	Medium
PGS	Paragneiss with sulphides	CHL	Chloritic	4	Strong
ZF	Fault Zone	CLY	Clay alteration	5	Intense
Q	Quartz	SER	Sericitic		
CAL	Carbonate, calcite	FDP	Feldspatic		

Mineralization				**Mineral Assemblage**	
Type	**Description**	**Intensity**	**Description**	py	pyrite
DIS	Diseminated	1	Very weak	cpy	chalcopyrite
MS	Massive sulfides	2	Weak	mal	malachite
		3	Medium	po	pyrrhotite
		4	Strong	bo	bornite
		5	Massive sulfides	sp	sphalerite

10.4 Recovery

The core recovery is not included in the legacy digital database, even though information on sample recovery was found in scanned laboratory certificates and in some of the drill hole logs provided for some of the Arinteiro and Vieiro holes. These data have not been incorporated into the current database.

All the drill holes drilled by Lundin Mining have records of core recovery (samples and non-sampled core intervals) in the database and have an average core recovery of about 97% for the 142 holes reviewed.

For the Atalaya Mining holes, the sample recovery for the 2016 RC samples is based on the total weight of the recovered coarse sample along with a density estimate using the copper and iron grades of the sample.

The total weight of the recovered sample is recorded at the drill site as the sum of weights of the 1/8 fraction of the total sample for assaying and the other 7/8 fraction as sample reject.

The 1/8 fraction of the total sample is reweighed after drying out. The wet:dry ratio is computed and applied to the weight of the wet 7/8 fraction to calculate the dry-weight of the total recovered sample.

The theoretical weight of the sample is calculated by multiplying the volume of the sample (diameter by sample length) by the density of the rock type obtained by a density formula.

The recovery of the sample is calculated as the ratio of the total dry weight and the theoretical weight of the sample, expressed in percent.

The weighing of the sample bags is monitored as a routine control by the Atalaya Mining preparation laboratory officer at project facilities.

The following density formulas are used in the Touro Project depending on the lithology and grade of the RC sample:

Amphibolite:

With Fe Assay:	Density = 2.781 + 0.0235 x %Fe
Only Cu Assay:	Density = 2.984 + 0.3748 x %Cu
No Assay:	Density = 2.95

Paragneiss:

With Fe Assay:	Density = 2.585 + 0.0299 x %Fe
Only Cu Assay:	Density = 2.728 + 1.0348 x %Cu
No Assay:	Density = 2.74

Other (Oxidized Rock, Fault Zone, Quartz):

With Cu & Fe Assay:	Density = 2.227 + 0.0319 x %Fe
Only Cu Assay:	Density = 2.295 + 1.6344 x %Cu
No Assay:	Density = 2.30

The two most important factors in estimating density for the Touro project are rock type (amphibolite or paragneiss) and Fe grade. When there is no Fe assay, the Cu assay is used, but the result is slightly less accurate. The density equations were developed using least-squares regression and the method of Thomas A. Jones (1979), using the analysis of 1,235 samples with density measurements (1,196 of the samples are from Lundin testing and 39 samples are from Atalaya geotechnical drilling).

The estimated recovery average of the 16RC drilling program is 88.2%, however, if those samples with depth (TO) less than or equal to 10 meters are excluded, the average recovery is 92%. Recovery for the near-surface samples is 52% and is a combination of poor ground conditions and difficulty of estimating density in oxidized and transition material. Recovery greater than 100% is observed in 20% of the intervals. About half of the +100% recoveries may be attributed to error in estimation of density, which has a relative standard deviation of 5%. The remainder may be attributed to down-hole contamination and/or incomplete cleaning at the end of the hole and is not at a level that should impact resource estimation.

Core recovery of the Atalaya DD holes is recorded during the geotechnical logging a TCR or total core recovery of the core interval. The overall average core recovery of the Atalaya holes is 95.92%.

10.5 Collar Surveys

Collar coordinates of the legacy holes were provided in a digital database by Atalaya Mining originally in European Datum (ED) 1950 UTM (ED50) Zone 29N format. These collar coordinates were validated with the data files provided by E. J. Sides[2] dated in the 1980's when the project was in operation. The collar coordinates in UTM ED50 format of 604 holes of the database have been fully validated.

During December 2016 and the first half of 2017, interviews were conducted between Barrero and several people who worked with RTP/RTM during the 1970s and 1980s. The interviewees talked about the surveying procedures starting with the staking out of the collar of the planned holes by the surveyor and the location of stone landmarks to mark collar locations of the holes once drilling was completed. A very few legacy holes were found by Atalaya Mining and the collars were re-surveyed (Figure 10.4).

The coordinates found in most of the legacy scanned documents are in a local reference system, the transformation from local to UTM coordinates is not known to date. The use of geo-referenced legacy maps in local coordinates to validate drill hole collar locations is extremely difficult due to the degree of distortion found on the ancient maps.

The collars of the Lundin Mining drilling program were all surveyed by Exploraciones Gallegas with a global positioning system, GPS instrument and in areas of poor GPS signal, the holes were re-surveyed by total station. The collar of each hole was capped, sealed with a concrete, and a metallic cap with the ID of the hole installed. Photos of the collar locations of all the holes were provided for inspection (Figure 10.4).

The original collar coordinates were supplied to Atalaya Mining in European Datum (ED) 1950 UTM (ED50) Zone 29N format and subsequently transformed in UTM European Terrestrial Reference System 1989 (ETRS 89). Additionally, Atalaya Mining re-surveyed 15 Lundin holes finding 100% correlation on the collar coordinates.

The transformation procedure of the reference system ED 50 to ETRS 89 implemented by Atalaya Mining was externally audited and verified by the surveyor consultants IPH[3]. The IPH Surveyor report, dated in 2015, included the validation of UTM ED 50 to ETRS 89 transformation, re-surveying of 15 Lundin holes and 2 legacy holes was provided by Atalaya for review.

All the collars of Atalaya holes were surveyed by external contractors and provided in UTM ETRS 89 Zone 29. The collar coordinates of the 15RC and 16RC were audited and verified by IPH Consultants[4] in March 2017, using a GPS Leica GNSS GS14 instrument. The collar of each hole was capped, sealed

[2] Edmund J. Sides (EurGeol PGeo, Mineral Resource Consultant, member of the PERC committee) worked fourteen years with the Rio Tinto Group in several countries including Spain as part of Rio Tinto Technical Services based in the UK. His experience in the company relates with the development and implementation of mineral resource systems at several mines (e.g. Cerro Colorado and Arinteiro), including training and mentoring of database technicians and resource geologists. More specifically, he did the resource estimation of the Bama and Brandelos deposits prior to mining and geological mapping of these areas for geological validation purposes.

[3] IPH, 2015. Verificación de coordenadas campañas históricas de sondeos en Touro (Galicia), Emed Tartessus Report.

[4] IPH, 2015. Auditoría de coordenadas de sondeos en el sistema oficial UTM ETRS89 huso 29 y trazabilidad de cartografía en proyecto minero de Touro, 01 / Marzo / 2017. Report.

with a concrete, and a metallic cap with the ID of the hole installed. Photos of the collar locations of all the holes were provided for inspection (Figure 10.5).

All the collar coordinates hereinafter are expressed in UTM European Terrestrial Reference System 1989, (ETRS 89, Z29).

Figure 10.5 - Drill hole collar photos of legacy, Lundin, and Atalaya holes (Barerro 2017)

10.6 Downhole Surveys

There is no digital record of downhole surveys for the legacy holes. Evidence was found in scanned legacy documents that downhole surveys were performed using "etch testing" or acid etch test tube with HF acid in 25 vertical holes of Vieiro, though these data have not been incorporated into the database. The testimonies of the interviewees mentioned that Jesus Ayala, the Chief Geologist at that time and the field assistants closely monitored the drilling progress and that, once the drilling was completed, the hole length was systematically checked by the field assistants with a measuring tape.

Downhole survey data exist for the Lundin Mining campaign. However, only 50 holes of a total of 169 were surveyed (44 of the 142 holes in San Rafael concession), because most of the holes were vertical and shallow, representing a 30% survey rate. The surveys were completed by a contractor (Geonor) with a Reflex Gyro Smart gyroscopic downhole survey tool.

Survey quality reports were provided for some of the holes showing that the quality of a few of them is not good and should be checked. No down-hole and up-the-hole surveys were performed on any of the surveyed holes.

Downhole surveys exist for 17 holes of the 15RC program. The survey was performed by SPI in April 2016[5] with a Reflex EZ-TRAC equipped with PVC casing for groundwater monitoring purposes. Survey readings consist of a single shot at the end of the hole and only for the hole dip. The Average dip deviation of the survey data is less than one degree for a mean depth of 80m confirming a minimum deviation from vertical. For this reason, Atalaya decided not to proceed with downhole surveys on the holes drilled during the 16RC drilling campaign.

10.7 Geotechnical and Hydrological Drilling

There are no records of geotechnical logging of the legacy holes.

Lundin Mining did geotechnical logging of 133 holes (116 belonging to the current project). The digital data files of the recorded parameters were provided for inspection.

Evidence was found of the existence of 33 Lundin holes where PVC casing was installed for water level monitoring and the water level readings provided, although the current existence of those holes was not verified.

Atalaya Mining carried out a geotechnical and hydrogeological drilling program, completing eight geotechnical holes and four piezometers, totaling 1,565.85 meters of DD drilling between September and October 2015.

The average depth of the drilled holes was 180m for the geotechnical holes and 33m for the piezometers. Since some of these holes also intersected mineralization, the drilled core was logged and assayed to incorporate the results into the drilling database.

The geotechnical holes and piezometers were carried out and logged by an external consultancy company (Terratec) and Lefranc, with Lugeon tests performed. The final report of Terratec was provided for inspection.

[5] Report: "RESULTADOS DE LAS MEDIDAS DE DESVIACIÓN, REFLEX EZ-TRAC, TOURO (A CORUÑA)-15RC", SPI, Abril 2016.

10.8 Drilling Summary

Location of holes drilled by Atalaya Mining in 15RC and 16RC programs are shown with respect to legacy and Lundin drilling in Figure 10.6. Representative cross-sections across all the deposits are presented in Figure 10.7, Figure 10.8, Figure 10.9, Figure 10.10, and Figure 10.11. Since most of the holes are vertical and the dip of the ore zones is shallow, it is not considered necessary to provide the true thickness of the mineralization. There are no known drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results.

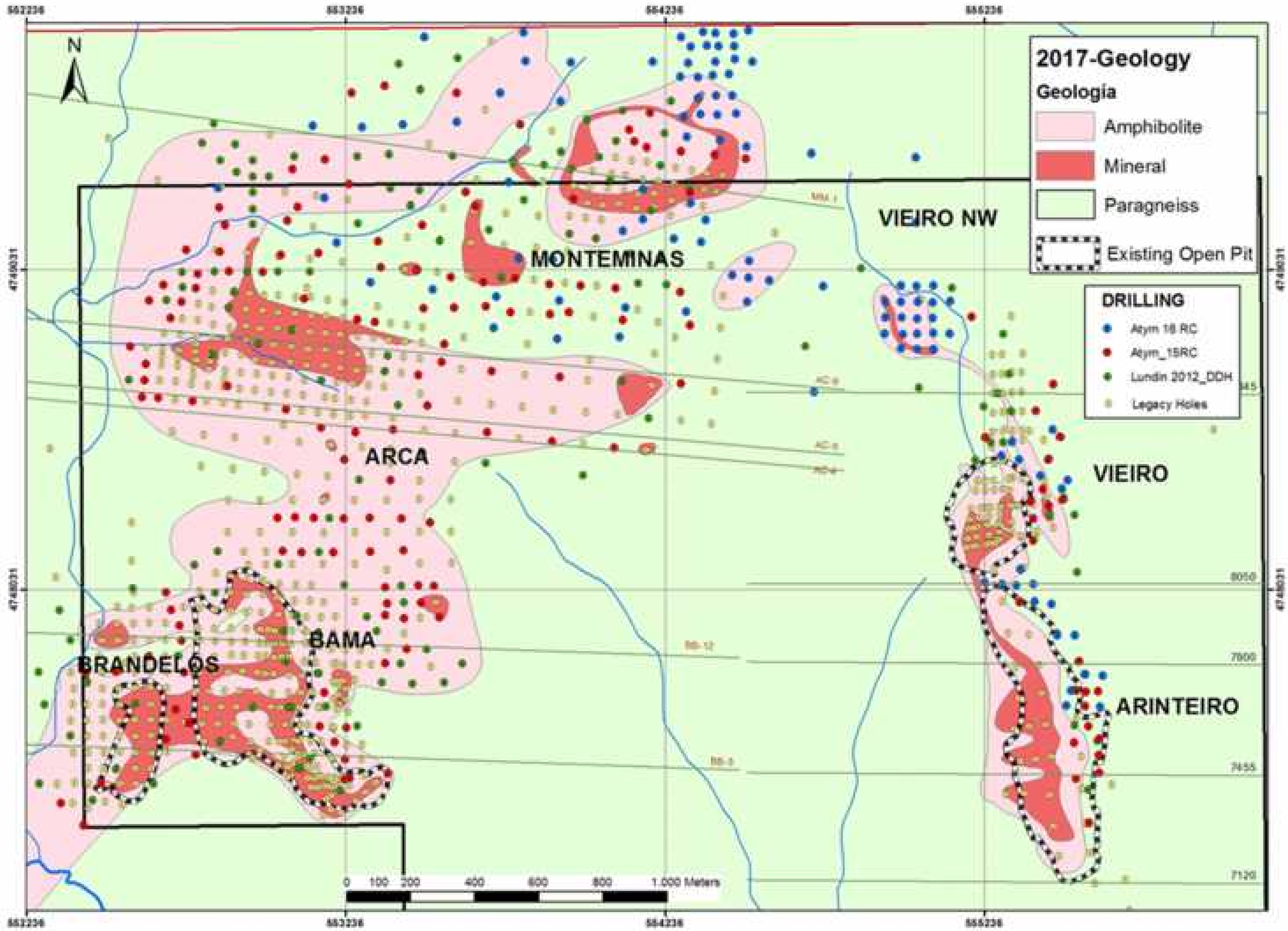


Figure 10.6 - Geological map with the location of the different drilling programs (provided by Atalaya Mining).

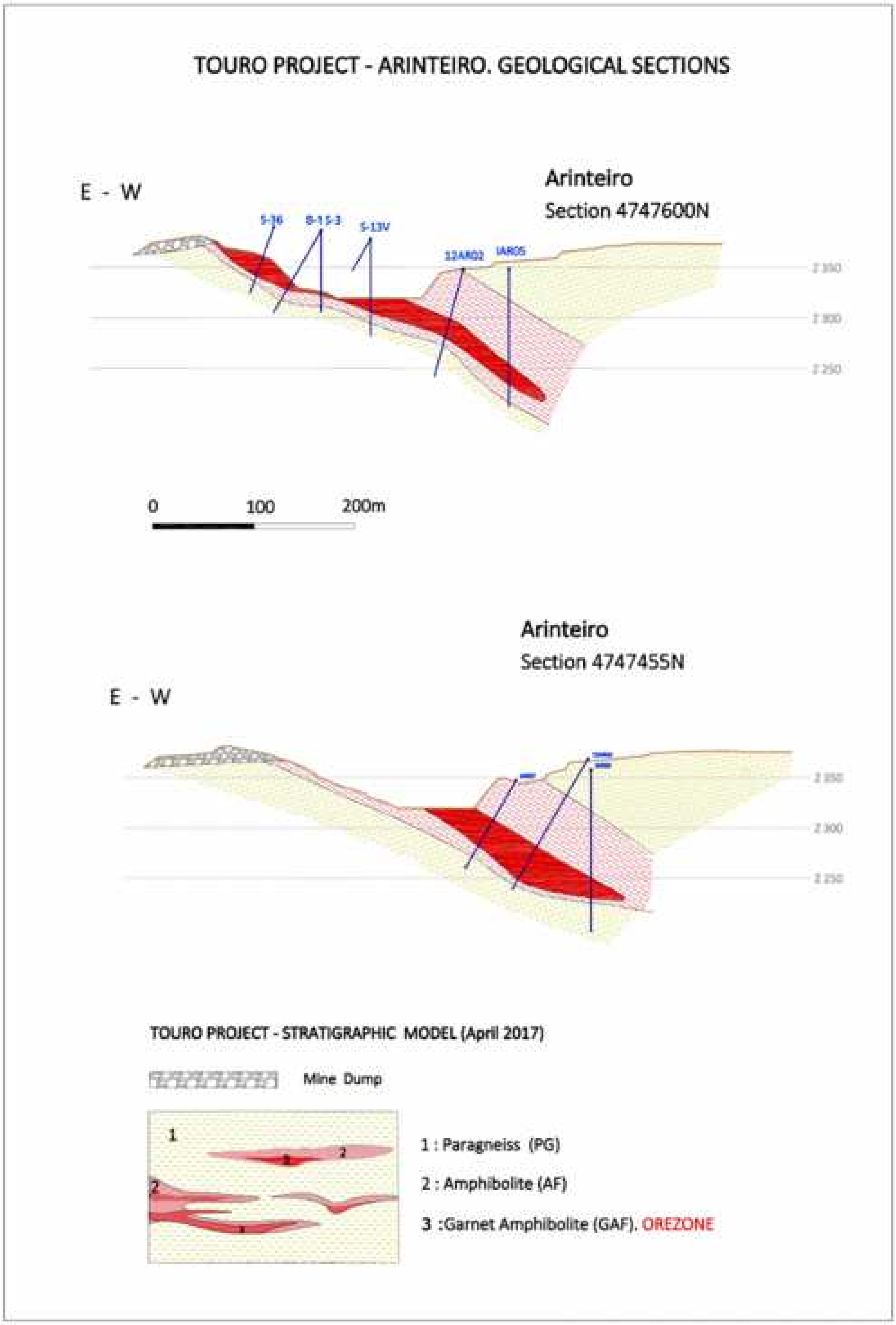


Figure 10.7 - Geological cross section across the Arinteiro deposit (provided by Atalaya Mining).

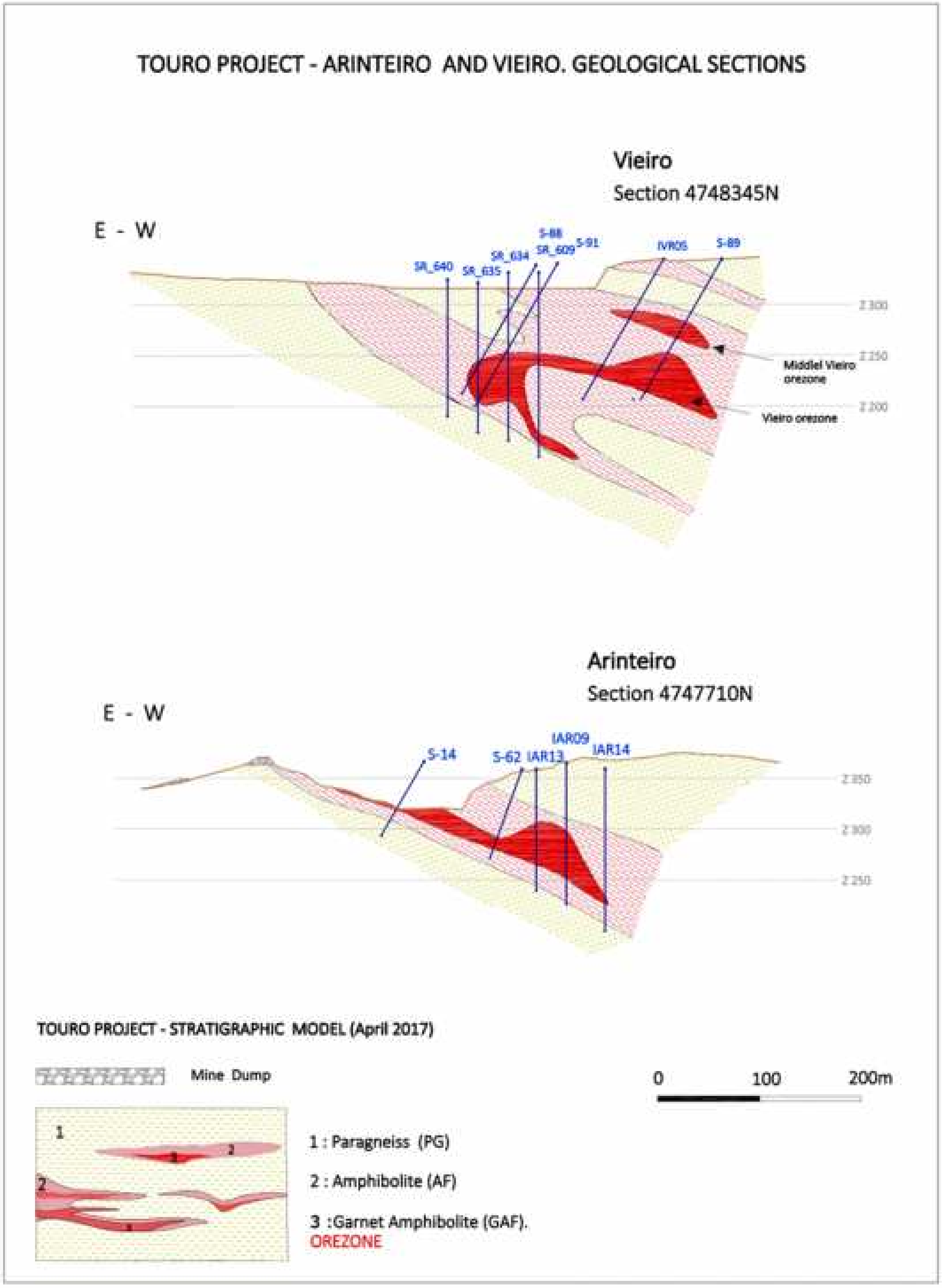


Figure 10.8 - Geological cross section across the Arinteiro and Vieiro deposits (provided by Atalaya Mining).

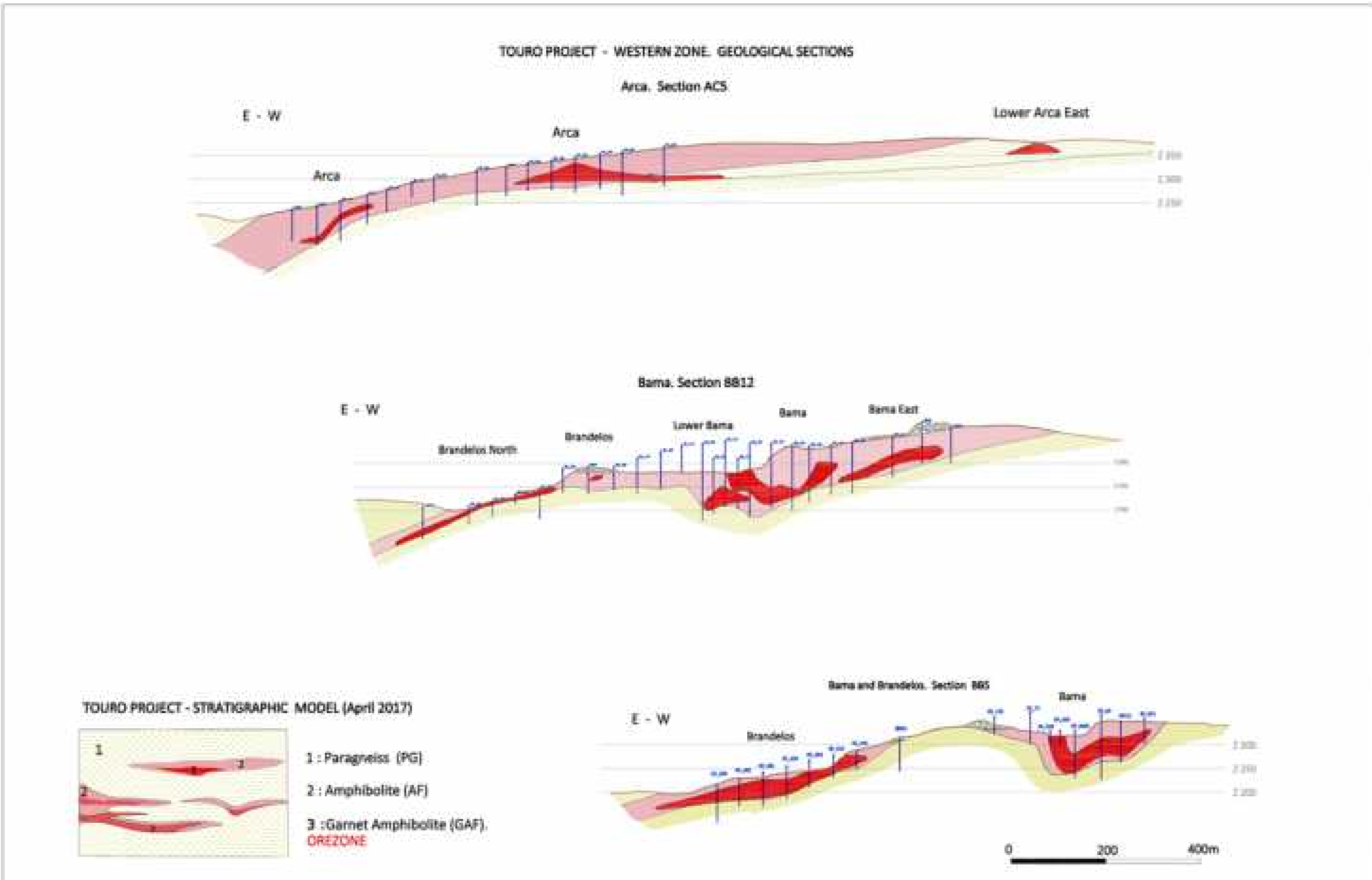


Figure 10.9 - Geological cross section across the Arca, Bama, and Brandelos deposits (provided by Atalaya Mining)

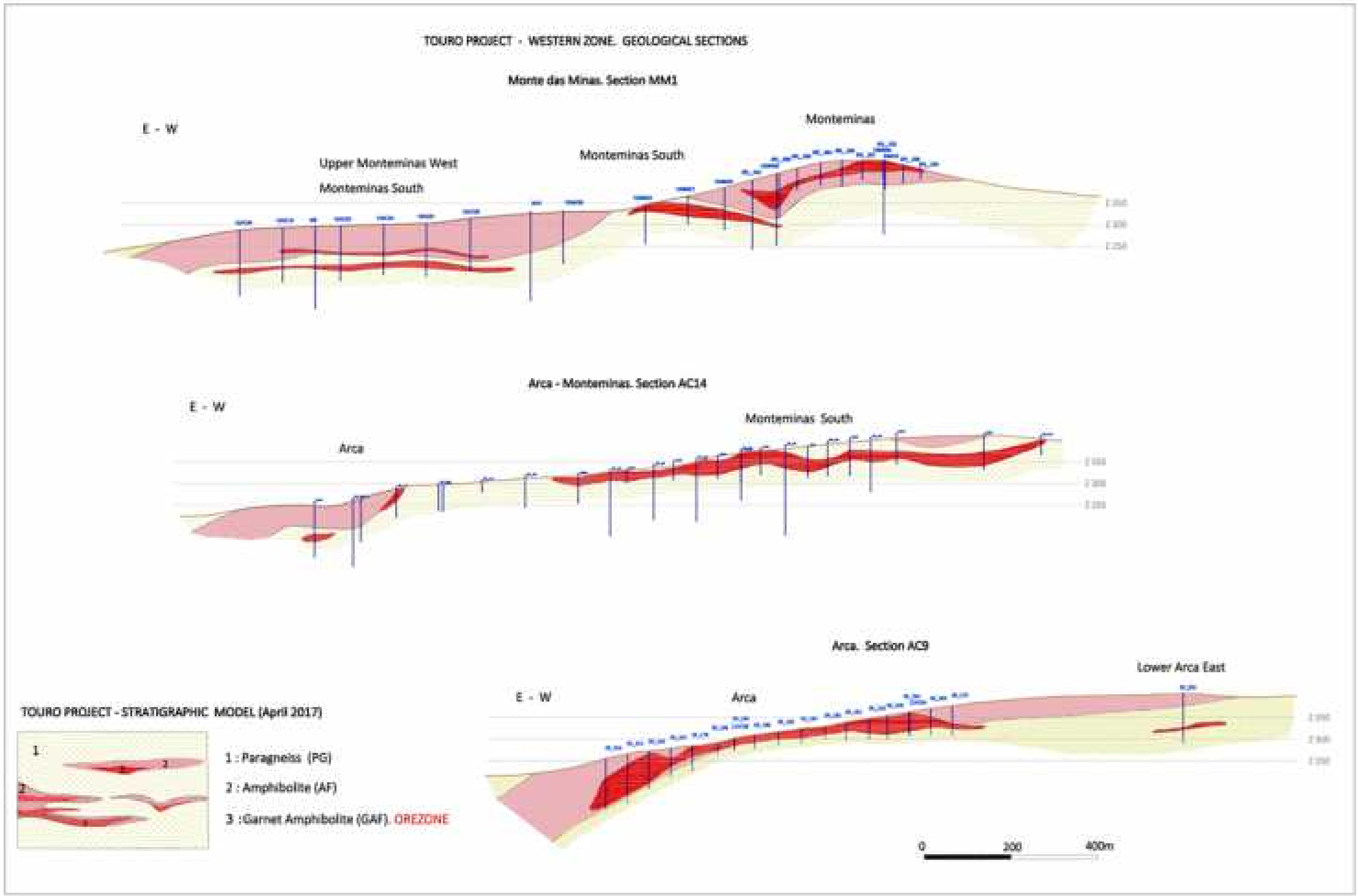


Figure 10.10- Geological cross section across the Arca and Monte das Minas deposits (provided by Atalaya Mining)

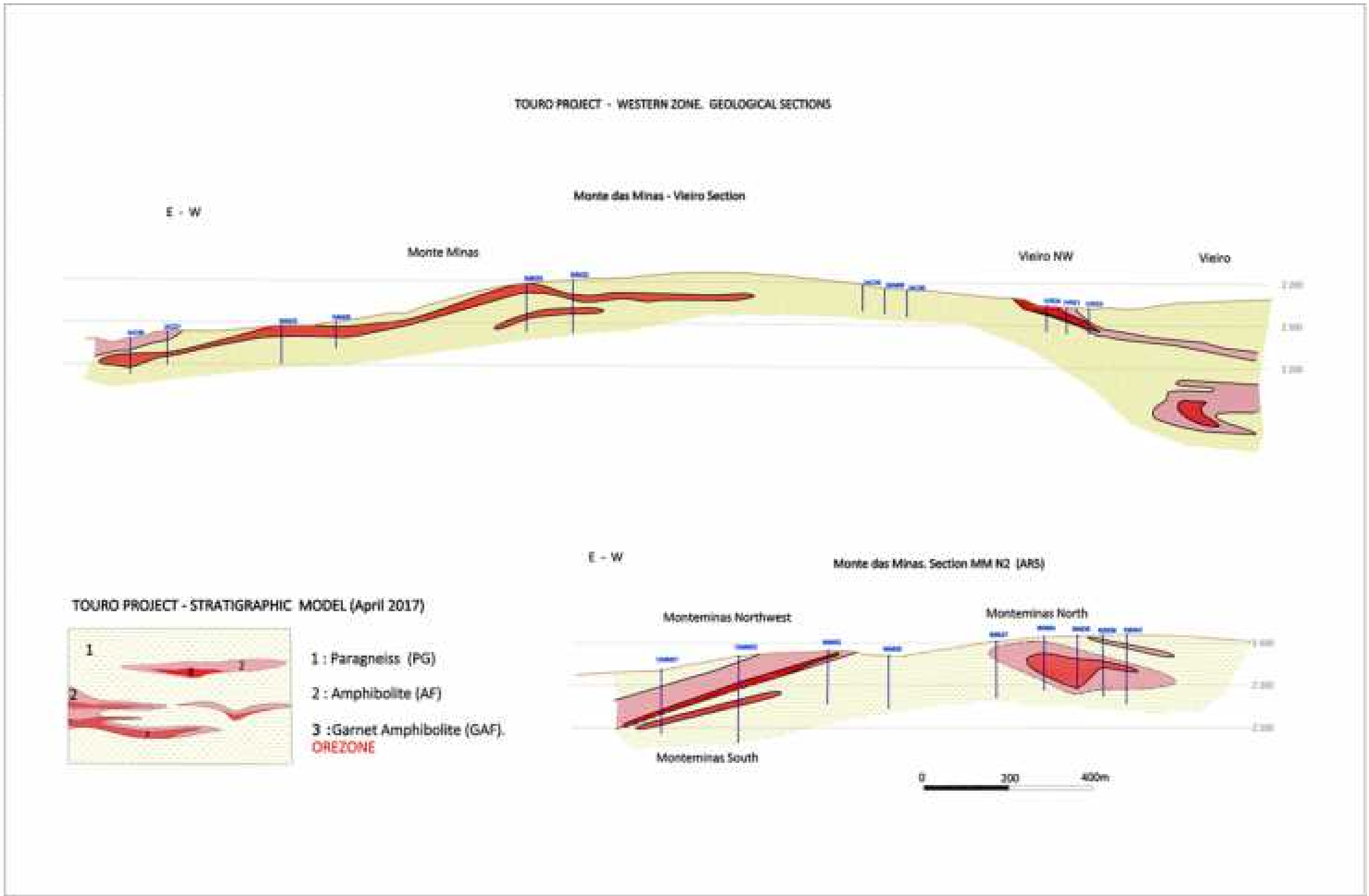


Figure 10.11 - Geological cross section across the Monte das Minas and Vieiro deposits (provided by Atalaya Mining)

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

This section was compiled by the Atalaya Mining technical staff and reviewed by Monica Barrero Bouza and Alan Noble, both of whom are Qualified Persons for the purpose of NI 43-101, Standards of Disclosure for Mineral Projects.

11.1 Summary

The Project drill hole database includes sample data from reverse circulation and core drilling. Drilling was performed by a variety of operators during at least three drilling campaigns over a 45-year period.

Little documentation is available on diamond core drilling and sampling procedures employed by RTP/RTM during the 1970s and 1980s, however, drilling was conducted by a company experienced in exploration and production and is considered reliable. Rio Tinto had its own company standards at that time that were followed throughout exploration projects and operations in Spain and elsewhere. The legacy documentation confirms that the procedures were above the industry standards of that time and close to the current standards.

Procedures employed by Lundin Mining during the diamond drilling program in 2012 are documented and registered. Sample preparation and storage was performed to industry standards and that the analytical data was adequately precise and accurate to support resource estimation.

Atalaya Mining has employed a mix of RC drilling and minor core drilling at Touro for the 15RC and 16RC programs. Drilling, sampling and assaying procedures employed by Atalaya are described and documented in this report and are considered to have been performed to industry standards and are sufficient to support resource estimation.

11.2 Sample Handling and Preparation

11.2.1 Reverse Circulation

RC drilling has been the main drilling method employed by Atalaya Mining since 2015. RC was generally carried out with Atlas Copco Mustang 5F4 and 5P4 drill rigs using bits ranging in diameter from 14.0 to 14.6 cm (5 ½ to 5 ¾ inches). RC drilling was performed dry to ensure proper movement of drill cuttings through the drill stem, cyclone and sampling equipment, and for dust control. RC drilling of 15RC and 16RC was completed by the contractor SPI.

The RC sample interval for resource drilling was selected by Atalaya Mining before starting the 15RC program by a Sample Preparation test[6], designed to determine the minimum representative volume of sample for adequate sample preparation and analytical procedures. Based on the conclusions of the tests, a sample interval of 2 meters was established.

Most drill holes in the western and northern areas (Bama, Brandelos, Arca and Monte das Minas) are fully sampled throughout the drilled interval. The sampled interval for holes in the Arinteiro and Vieiro was determined before drilling so that only the known mineralized horizons were sampled and assayed.

[6] Report "Touro 15RC. Preliminary Sample Preparation Test"

Samples were collected at 2m intervals by trained field assistants who were monitored and supervised by an onsite geologist. Wet samples were prevented using air booster or an auxiliary compressor that maintained dry conditions in the hole. Samples with excess water were not accepted and, when groundwater made it difficult to maintain dry samples, RC drilling was stopped, and drilling was continued using diamond coring methods.

Cuttings from each interval are passed through a cyclone into a riffle splitter where the bulk sample is reduced to a 1/8 split, the other 7/8 of the sample is weighed at the drill site. The 1/8 split is saved and transported to the sample preparation facility, for weighing and drying. The samples are collected in large bags suspended from the discharge tube of the splitter and tagged with the sample ID. The average final sample weight is 10 kg.

A representative split from the discharge material is placed into a plastic RC chip tray for geological logging. The chip tray is marked with the drill hole name and down-hole interval.

Atalaya employs a bar coding system for sample labeling (Figure 11.1).

IAR01052T	Normal RC sample	Example:
IAR01052AT	QC samples	IBA09066T
IAR01052BT		
IAR01052DT		

I	Code used to identify the sample is an RC sample
AR	Code to identify the zone (AR: Arinteiro; VR: Vieiro; MM: Monte das Minas; AC: Arca; BA: Bama; BR: Brandelos
1	Is the number of hole
52	Sample ID (corresponding to the depth of the sample)
A/B/D	QC sample: A: Standards; B: Blank; D: Duplicate
T	Touro project, which informs the lab of the preparation and analytical method requested

Figure 11.1 - Atalaya bar coding system for sample labeling[7] (Atalaya 2017).

RC samples are taken to the secure logging facility by ATM personnel, weighed, if wet, and dried and reweighed. After drying, the sample is reduced again using a riffle splitter to sample size of 800g. The resulting samples are sealed in bags, control samples are inserted and the batch is tagged for shipment to the Atalaya Lab located in Rio Tinto (Huelva).

[7] 051115_Etiquetado y QC_Touro.pdf (Atalaya 2017)

The sample collection and preparation process is shown in photographs in Figure 11.2 and as a flowchart Figure *11.3*. It is fully described in an internal company document[8]. Samples are shipped to the Atalaya Lab facility in Riotinto (Huelva, Spain) by a well-known Spanish courier service, MRW.

During 16RC campaign, additional testing was conducted to evaluate the effect of dust losses on sample copper grade from the sample-collection cyclone that is routed to a dust-collector filter. This testing verified that the copper grade of the dust sample is higher than the coarse sample that is collected at the cyclone, and that a slight negative bias is introduced to the RC copper grades by dust losses. A summary of the results of this testing is provided in Chapter 12-Data Verification.

[8] Atalaya Mining Analytical Methods, A10-2015 v.01 r.03: "Sample Preparation Method for Mine, Geology and Exploration".

Picture 1 RC drilling on site

Picture 2 Reception of RC samples

Picture 3 Drying ovens

Picture 4 sample drying

Picture 5 Riffle splitter

Picture 6 RC chip tray

Picture 7 RC logging on site

Picture 8 RC rejects storage on site

Figure 11.2 - RC sample collection and preparation process (Atalaya 2017)

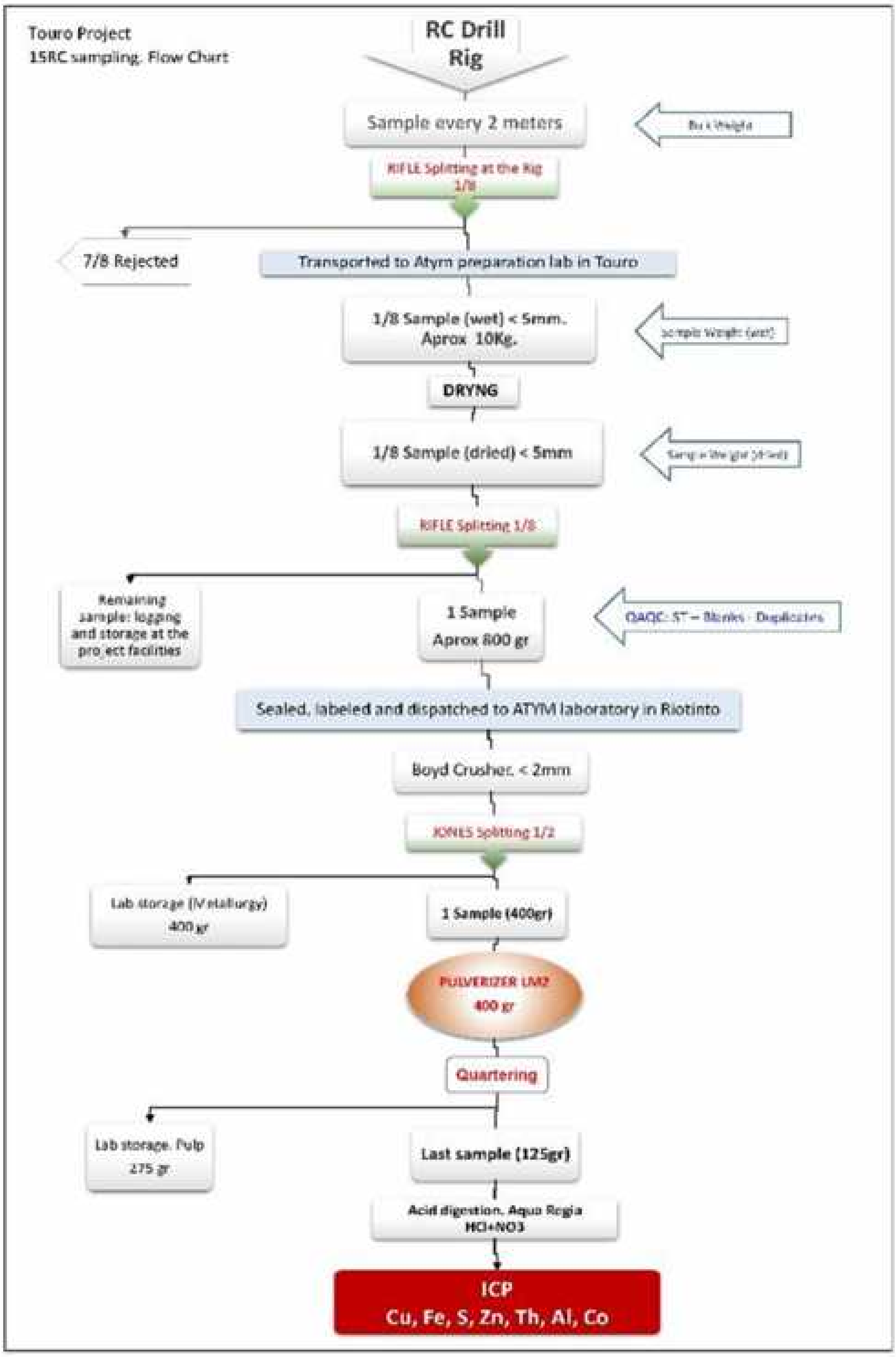


Figure 11.3 - Updated sample preparation flowchart (Atalaya 2017)

11.2.2 Diamond Drilling

As reported previously, little documentation exists regarding sample handling protocols at the legacy drill sites, and the only available information is based on testimonies of people that worked for RTP/RTM at that time. Limited information is still available regarding sample handling between the drill site and assay laboratory for RTP/RTM drilling programs.

There is no specific documentation about sample preparation methods during RTP/RTM programs. The information of this regard was extracted from the report "PROYECTO DE EXPLOTACION DEL YACIMIENTO DE BAMA, Rio Tinto Patiño, 1979" for the first stages of investigation of the Bama deposit. No information regarding sampling methods and sample preparation protocols used for Arinteiro, Brandelos, Arca and Monte Minas was available at the time of the writing. In the above-mentioned report, the sample preparation procedures used during the two stages of the investigation of the Bama deposit are explained. Sample preparation flowcharts for core samples of BX diameter are shown in Figure 11.4.

Legacy core drilling was conducted using BX and AQ core sizes and a typical sample length of 1 meter.

The procedures are considered to exceed industry standards of that time, even though the insertion of control samples was not systematic and neither blanks nor standards were used as control samples.

Lundin Mining used diamond drilling at Touro, typically HQ (63.5 mm) and sometimes NQ sizes (47.6 mm) as reduced diameter in the deeper holes. The core boxes photos are available for all the Lundin holes. The typical sample interval is 2m, though smaller and greater intervals were also used.

Logging and sampling of core was performed onsite and core was split using a diamond saw. A half split was used for assay purposes and the other half was saved in the box. The core archive is still available at the Touro facilities, although Lundin Mining and Atalaya have consumed a small part of the core for metallurgical test work.

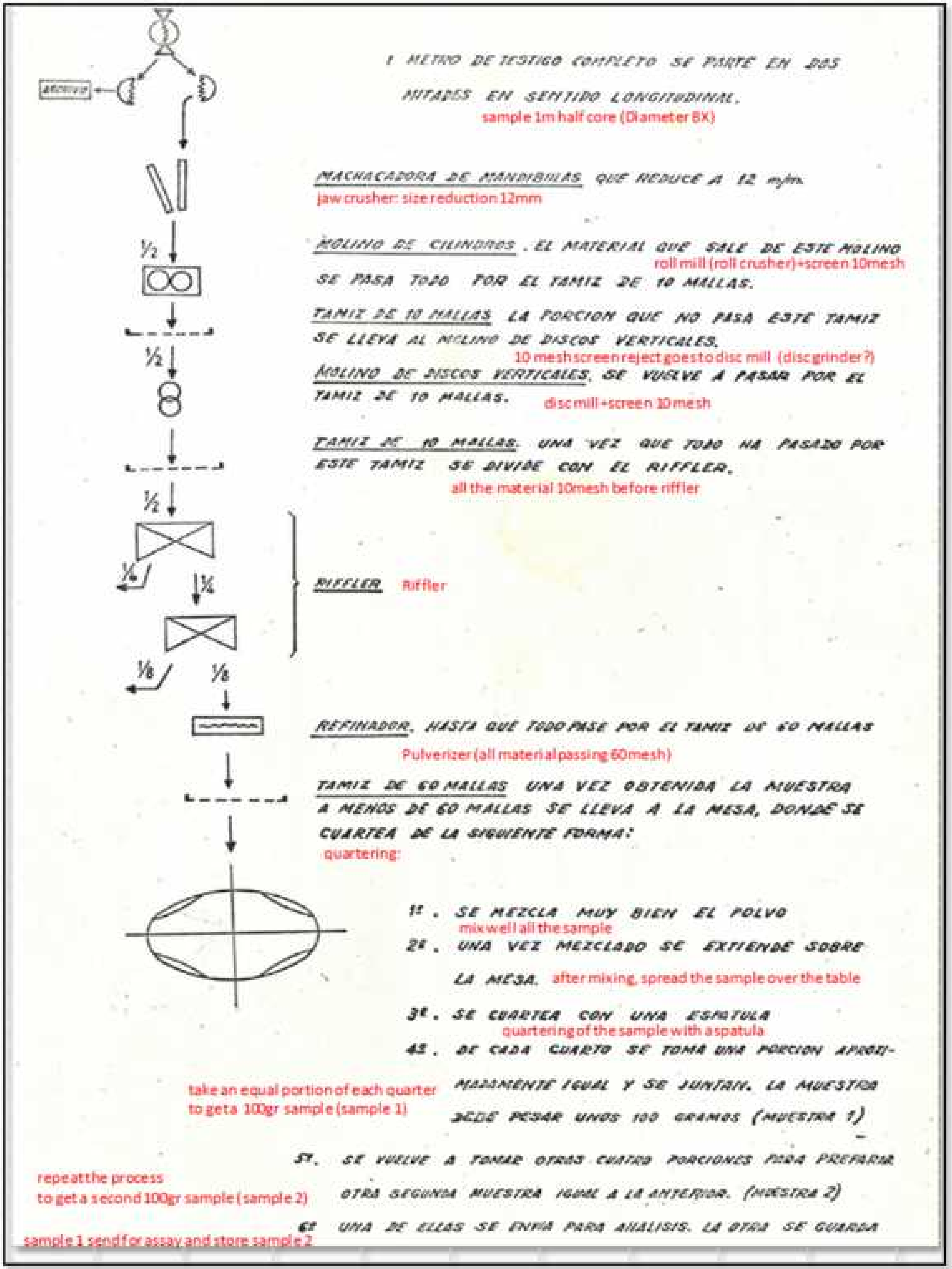


Figure 11.4 - Sample preparation flowchart (BX core diameter), RTP (1979)

The half core for assaying was bagged, tagged, and shipped to the ALS laboratory in Seville (Spain) where sample preparation was performed. The sample preparation protocol was extracted from the sample submittal forms to ALS, as follows:

- Measure and record received Sample weight
- Sample Login- Received w/o bar code
- Pulp Login-Received w/op bar code
- Crushing QC Test
- Pulverizing QC Test
- Fine crushing to -70% <2mm
- Split sample - Boyd Rotary Splitter
- Pulverize split to 85% <75µm
- Pulp and rejects returned

Atalaya core holes are completed using HQ (63.5 mm) size core from the ground surface or to continue drilling of RC holes when drilling encountered groundwater in the deeper holes. Diamond drilling was completed by contactor SPI using a Spidrill drill rig.

During core drilling, the core is placed in the core boxes by the driller's assistant. Wooden markers are placed and the drilling depth marked for each core run. Each core box is marked with the hole ID and the depth of the core interval. The core boxes are then picked up from the rig daily and transported to the ATM logging facility by two field assistants. The core is then washed and photographed.

A geologist inspects, logs the core, and selects appropriate intervals for sampling. A detailed sample list with the control samples that must be inserted is prepared prior to sampling. The preferred sample interval is 2m. The same sample labeling procedure is used for core samples as is used for RC samples (Figure 11.1).

The core is always split using a diamond saw. A half split is used for assay purposes and the other half is saved in the box for the core archive (Figure 11.5). The sample split for assay is tagged, sealed in a bag, and shipped to Atalaya Laboratory located in Riotinto (Huelva).

The sample is dried and crushed to 70% < 2mm in a Boyd crusher at the lab and, at this point, the core samples follow the same preparation protocol[9] than for RC samples explained in the previous section (Figure 11.3).

The half split that is not sampled remains in Touro core shed storage.

[9]Atalaya Mining Analytical Methods, A10-2015 v.01 r.03: "Sample Preparation Method for Mine, Geology and Exploration".

Diamond core box before splitting

Core splitter

Core archive

Figure 11.5 - Core photo record, sampling and storage (Atalaya Mining, 2017)

11.3 Density Determination

There are no density entries in the legacy drill-hole database.

Lundin Mining used the water immersion method for specific gravity determination of core where the density equals the core dry weight in air divided by the core wet weight in air minus the weight of the non-coated core submerged in water. The core length selected for density determinations has an average length of 20cm and HQ diameter.

A total of 948 data corresponding to density determinations of a selected core of 21 different lithologies in the 142 diamond drill holes drilled in 2012 (covering the areas of Arinteiro, Arca, Bama, Brandelos, Monte Minas and Vieiro), was provided as part of the Lundin Mining database.

Atalaya Mining also uses the water immersion method for specific gravity determination in core. A total of 43 data corresponding to density determinations of a selected core of Arca, Vieiro and Monte das Minas were conducted on geotechnical drilling.

It is noted that both Lundin and Atalaya density measurements were completed using raw samples, rather than first coating the samples with wax or sealing the samples in plastic covering. Where there is significant porosity in the samples, measuring density without coating or sealing the samples results in density measurements that are biased high. While porosity is minimal in the mineral zones at Touro, future density measurements should consider whether coating or sealing is justified.

A statistical analysis of the density data was performed by ORE in November 2015 to examine the relationships between rock type, copper grade, and iron grade in more detail. It was determined that the two most important factors in estimating density for the Touro project are rock type (amphibolite or paragneiss) and Fe grade. Where there is no Fe assay, the Cu assay may be used, but the result will be slightly less accurate. These results are discussed in detail in Chapter 14.

Additional testing is recommended to refine the density relationships for oxidized rocks if they are intersected during the current and future drilling campaigns.

11.4 Sample Storage and Security

There's no information on sample storage or sample security or core from the legacy holes.

Core boxes of Lundin program are adequately stored in the Touro Project facilities together with sample pulps and rejects returned from ALS stored in proper bins.

Security of Atalaya samples is based on procedures designed by the company and relies on the principle that sample collection and transport, sample handling, and sample preparation are completed by company personnel using company vehicles. Sample shipment to Atalaya Lab in Huelva is performed by a well-known courier service. Sample submittal forms are sent to the Atalaya Lab by e-mail and shipments are subsequently checked once the samples arrive to the lab.

With respect to security, there is no reason to believe that sample tampering has occurred in any of the historical or current drilling programs. Mining at Touro has demonstrated the presence of copper in the ground and copper sulphides are observed in core, pits and outcrops at the site. The analytical techniques used are industry standards for analysis of copper and are acceptable.

11.5 Analyses

11.5.1 Analytical Methods

No information is available on the analytical methods used for the legacy data.

Scanned handwritten laboratory sheets (Figure 11.6) were provided as part of the legacy information hard data. Part of the data is from the RTM Santiago Laboratory; most of the sheets are dated in 1982 and signed by the Laboratory Manager. There are some grade control assays and Au and Ag from Bama holes, but the majority of the samples are from Vieiro holes and were submitted by Jesus Ayala (Chief Geologist), the assay results are Cu values in percent and sample preparation details are also included at the top of the sheets.

The rest of the laboratory sheets come from RTP-Exploración Minera and the assayed holes are from the Bama area. There is no reference on the location of this laboratory and the sheets are not signed or dated. The information includes sample interval, assay results in %Cu, %S, and % core recovery. The laboratory sheets include 3018 samples with %Cu assays corresponding to 45 holes from Bama and Vieiro.

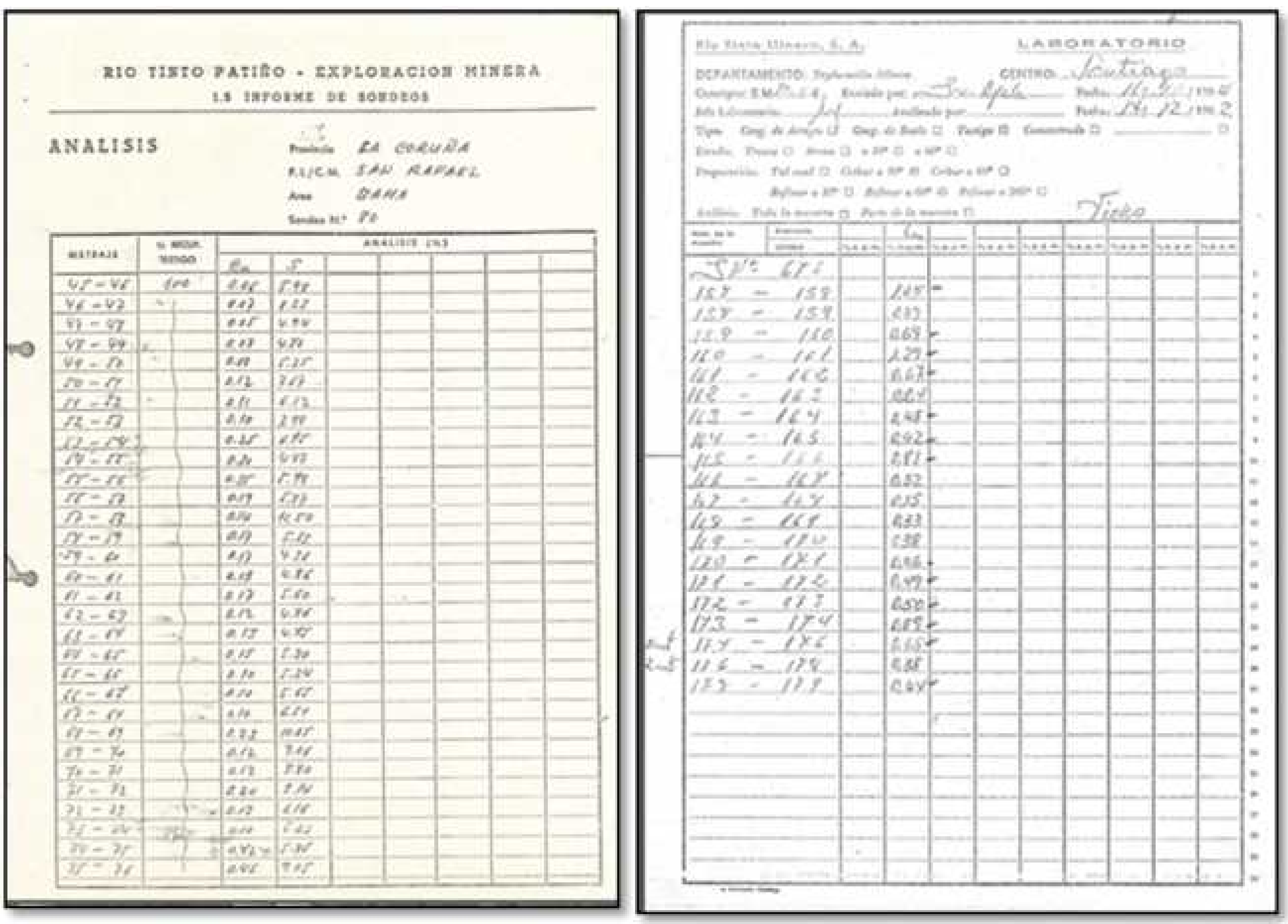


Figure 11.6 - Legacy laboratory sheets from RTP/RTM.

There is evidence that, not systematically, but with a relatively high frequency, duplicate samples were sent to either the Cerro Colorado Laboratory (Riotinto Mine in Huelva), or more frequently, to the Huelva Laboratory located at the Huelva smelter, or abroad.

Lundin Mining used ALS Laboratories as the primary assay laboratory (ALS Laboratory Group SL, Seville, Spain). ALS Sample Submittal Forms, Work order confirmations, Certificate Invoices and ALS Certificate of analysis in Excel and protected pdf format including ALS QC data were provided as part of the Lundin Mining Documentation.

ALS Inspection has the QMs framework either Certified to ISO 9001:2008 or Accredited to ISO 17025:2005 in all of its locations.

The following analytical work was requested to ALS:

- Au-AA223 Au 30g FA-AA finish, if Au>10.0ppm then run method Au-GRA21.
- ME-MS61 48 element four acid ICP-MS (Ag, Al, As, Ba, Be, Bi, Ca, Cd, Ce, Co, Cr, Cs, Cu, Fe, Ga, Ge, Hf, In, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, Rb, Re, S, Sb, Sc, Se, Sn, Sr, Ta, Te, Th, Ti, Tl, U, V, W, Y, Zn and Zr).
- ME-OG62 Ore-grade Elements-15 elements four Acid ICP-AES (Ag, As, Bi, Cd, Co, Cr, Cu, Fe, Mg, Mn, Mo, Ni, Pb, S and Zn, Sb might be added).
- Cu-OG62 Ore Grade Cu-Four Acid (instrument variable)

Atalaya Mining used the Atalaya Mining Laboratory located at the Riotinto Mine in Huelva as the primary laboratory for the Touro Project using aqua regia digestion and ICP including the elements: Cu %, Fe %, S %, Zn %, Th ppm, Al %, and Co ppm. Atalaya Mining Laboratory is not a certified laboratory.

The detailed description of the analytical method currently used was provided within the document “Atalaya Mining Analytical Methods A11-2015 v.01r.04: Chemical Attack Methods for Mining, Geology and Exploration Samples”.

The Atalaya Mining Laboratory has its own QA-QC system even though this information has not been accessed.

A test program designed by Noble was conducted by Atalaya in August 2017 to provide additional verification of both the Lundin copper assays and copper assays from the Atalaya Laboratory. The results of this testing are provided in Chapter 12-Data Verification.

External duplicates were submitted by Atalaya to the certified laboratories listed below:

Alex Stewart Assayers Iberica (Bilbao, Spain), aqua regia digestion and ICP including the elements:
Cu %, Fe %, S %, Zn %, Th ppm, Al %, and Co ppm.

AGQ Labs (Sevilla, Spain): PE 000 aqua regia digestion and ICP, including the elements Al %, S %, Ba mg/kg, Co mg/kg, Cu %, Fe %, Th mg/kg and Zn %.

OMAC Laboratories Ltd trading as ALS Loughrea (Galway, Ireland): ME-ICP41 aqua regia disgestion and ICP/AES, including the elements Cu ppm, Fe %, S %, Zn ppm, Th ppm, Al %, and Co ppm.

Alex Stewart laboratory is Certified to UNE-EN ISO 9001-2008, AGQ is Certified to UNE-EN ISO 9001-2008 and Accredited to ISO/IEC 17025:2005, and OMAC is Accredited to ISO/IEC 17025:2005.

11.6 Quality Assurance and Quality Control

Even though there is little information available regarding QAQC procedures during RTM/RTP drilling programs, duplicates of pulp and coarse samples were normal Quality Control practice. The existing records of duplicate samples of Bama and Arinteiro represent 5% of the total number of legacy samples.

Assay QA-QC data of Lundin Mining drilling program includes standards, blanks and pulp duplicates inserted into the sample stream and QAQC data of the ALS laboratory. Seven Geostats Pty copper and gold standards were used with an average frequency of insertion of one in 25. There is no information about reference values of the blank samples, there is an average frequency of insertion of at least one blank sample per batch. Pulp duplicates performed on ALS returned pulps started once the rejects and pulps were received from ALS. There is no evidence of second laboratory to check ALS results.

Atalaya Mining assay QAQC data includes standards, blanks, and both internal and external duplicates. Four copper standards prepared by Ore Research & Exploration (OREAS) are inserted with a frequency of one in 20. More recently, Atalaya has discontinued use of the OREAS standards and is purchasing a new set of samples from Geostats Pty. Blank samples are inserted at a rate of one in 20 samples and one internal pulp duplicate at the end of each hole. Reference values for the blank material are provided by the in-house laboratory. External duplicates were sent to Alex Stewart and AGQ Labs.

Additional discussion about the results of the QC of the legacy, Lundin, and Atalaya drilling data is provided in Chapter 12.

12 DATA VERIFICATION

Alan Noble and Monica Barrero Bouza, both Qualified Persons for the purpose of NI 43-101, Standards of Disclosure for Mineral Projects, reviewed and observed various data collection procedures and are of the opinion that they meet current industry standards and requirements. The Atalaya technical staff are very competent and consistently follow the procedures and protocols necessary to ensure that the data being collected is of the highest quality.

12.1 Legacy Data

The legacy drill hole database was provided in electronic format and validated against an extensive record of legacy documentation (data files, maps, plans and sections, reports, laboratory certificates, geological logs…etc.), that remains from previous operations and exploration.

Even though the Quality Control (QC) of the geological data from the time RTP/RTM was operating the Touro Project is available only partially, the procedures appear to have exceeded the industry standards of that time and are close to current industry standards.

The legacy data has been compared to the Lundin and Atalaya assays to determine whether there is any apparent bias in the legacy assays.

The first test was a "twinned hole study", in which 33 drill holes from the legacy period were compared with closely located holes that were drilled by Lundin and Atalaya. Using a 0.0% Cu cutoff grade, the relative difference between the legacy holes and twins was only 1.4%, which is not significant on either a practical or statistical basis. Using a 0.2% Cu cutoff, it was observed that the twin holes had about 5% greater thickness above cutoff and were about 5% lower grade than the legacy holes. Analysis of the grade difference shows that where both the legacy and twin hole are above cutoff, the relative grade difference is less than 3%. A pooled t-test of the paired and unpaired differences shows that the 5% difference in grade is not statistically significant.

The second test for the legacy data is to compare three the overlapping nearest-neighbor (NN) block models that were estimated using only legacy, Lundin, or Atalaya drill holes. The resulting models were compared where the distance between the block center and the nearest drill hole was less than 35m. This comparison showed that the Atalaya and legacy copper grades were almost identical, but that the Lundin copper grades were 9% higher than the legacy grades and 15% higher than the Atalaya grades.

Based on the twin study and the NN study, it is concluded that the legacy copper assays are generally similar to the Lundin and Atalaya assays and that there is no need to modify or otherwise discount legacy copper grade.

12.2 Lundin Data

The drill hole database and a significant amount of additional and complementary information has been made available for review from the Lundin Mining drilling program. The drill hole database is considered adequate and within industry standards.

In addition to geologic logging, sampling and assaying, Lundin conducted 948 density tests using the water immersion method and 20cm intervals of core. Density formulas were developed from these tests to estimate density based on rock type (Amphibolite, Paragneiss, Other), iron grade, and copper grade.

A minor issue with the core samples is that the samples were not sealed before water immersion, and the measured densities will be overstated if there is high porosity in the samples. Although the Touro samples tend to have very low porosity, sealing the core in plastic envelopes is recommended for future density measurements. Caliper measurement of core diameter in several locations could also serve to measure core volume for density measurement. It is also recommended that the exact sample that is used for density measurement be prepared and assayed in its entirety. Wax coating is not recommended, since it would interfere with assaying of the density sample.

The Lundin Quality Assurance (QA) and QC data included standards, blank assays, and duplicate assays. The standards assays were statistically different from the certified values for three of the five standards, but the relative differences were less than 3.5% and do not indicate a significant problem. Blank control assays average 35 ppm Cu (0.0035% Cu) and are typical of blank sample assays. All Lundin assays were completed by a single laboratory, ALS, and there are no check lab duplicates. Re-assaying of a set of 110 samples over a broad range of copper grade from the Lundin drilling was recommended to provide additional confirmation of the Lundin data and is in progress at the time of this report. This program has been completed and no issues were found with the Lundin assays. The results of this study are discussed in Section 12.4.

12.3 Atalaya Data

The geological data of ATM drilling campaigns is not complete, logging of the 15-RC stage appears to be acceptable but there is no logging available of the 16-RC drilling. Geological logging should be completed to support resource estimation, particularly because the density uses lithology as one input to the formulas.

QA-QC data for the Atalaya data include assaying of standard reference samples, blanks, and duplicate assays. Duplicate assays include both internal duplicates and external duplicates at two outside laboratories.

The results of the standard reference samples show that the Atalaya assays are statistically different from the certified reference values for assays for three of the four samples. Inspection of the raw data used for certification of the reference samples suggests that the reference samples may have not be homogeneous that could render these results questionable. Blank control assays average 0.004% Cu and are typical of blank sample assays.

Internal duplicate assays at the Atalaya lab correspond closely to the original assays, indicating good repeatability within the lab, although several large outliers, possibly mislabeled samples, are observed. External duplicates at Alex Stewart laboratories in Bilbao, Spain confirm the Atalaya assays are within about 1% of the Alex Stewart assays. External duplicates at the AGQ laboratory in Sevilla, Spain average about 4% higher than the Atalaya assays. Several aspects of the AGQ results indicate that the AGQ assays may not be comparable to the Atalaya assays, and AGQ is not recommended at this stage as an external check lab.

The sample preparation and analytical methods used by the Atalaya Mining Laboratory are considered adequate, although the following is recommended:

- Provide clear identification of the date of receipt of the samples, sample preparation, and the assayed values delivery of the batch in each certificate. In addition, the signature of the laboratory manager and the internal QAQC of the laboratory performed should be provided.
- Prepare coarse or preparation duplicates after the first crushing or splitting step for core or use of drill rig duplicates in the case of reverse circulation drilling.
- Check samples or pulp duplicates should be resubmitted to a second and external certified laboratory. A frequency of 1 in 30 is considered appropriate.
- Provide a more detailed characterization of the blanks used in terms mean value and standard deviation or confidence interval. The use of coarse and pulp blanks is also recommended.

Even though most of the holes drilled by Atalaya are vertical and no major deviations are expected, there are a significant number of holes with considerable length: down-hole surveys should be performed in future drilling programs.

During the site visit to Touro it was observed that dust from the RC drilling was being lost at the sample-collection cyclone and was being collected in a dust-collector filter. Since there was some evidence that copper grade might concentrate in fines, a protocol was developed to test whether dust loss introduces a bias in copper grade. The results of this study showed that an average of 8% of the sample was lost to dust and that the grades of all metallic elements and aluminum are significantly higher in the dust sample compared to the coarse sample. The dust test shows that copper grade is biased 5% low for all samples, but only 2.7% low for samples above 0.2% Cu. It is concluded that dust losses in RC drilling introduce a slight conservative bias, but that the bias is not significant for resource estimation.

12.4 External Laboratory Check Assay Program

A test program designed by Noble was conducted by Atalaya in August 2017 to provide additional verification of both the Lundin copper assays and copper assays from the Atalaya Lab. The procedure for this test was as follows:

1. A set of 110 samples was selected from the Touro archive of minus 5mm fine-crushed rejects prepared by Lundin, as follows:
    a) The samples were selected from the Lundin minus 5mm fine-crushed rejects. Samples were selected to cover all parts of the project area.
    b) The samples were selected to cover the entire range of copper grades with 20 samples each in the following grade ranges: 0 to 0.1% Cu, 0.1% to 0.2% Cu, 0.2 to 0.3% Cu, 0.3 to 0.5% Cu, 0.5 to 1% Cu, plus 10 samples above 1% Cu. (110 samples).

   c) The Atalaya sample prep lab prepared 4 pulp samples for each of the selected samples.
   d) For every 5 samples above, a sample from the OREAS standards was inserted (22 additional samples).
   e) One blank sample in was inserted for every 10 samples (11 additional samples).
   f) The samples were randomized by the Atalaya Geology Department and numbered so that the standards are blind to all assay labs including the Atalaya Riotinto lab.
2. Samples were submitted for assay to the following laboratories:
   a) Atalaya Riotinto
   b) Alex Stewart Iberica
   c) ALS Omac in Ireland
   d) Keep the extra sample for future use.

The comparison of the Lundin copper assays to the three comparison labs, summarized in Table 12.1, shows that there is no statistically significant difference between the Lundin copper assays and any of the comparison labs. Those Lundin assays completed with the standard assay procedure (less than 1% Cu) are virtually indistinguishable from the other three labs. Over-limit samples greater than 1% Cu (10,000 ppm) have slightly larger differences, but those differences are not statistically significant and the average of the comparison labs is less than 0.2% different from the Lundin average assay.

In addition to the verification of the Lundin copper assays, this test provides a round-robin comparison of the Atalaya lab with assays from Alex Stewart and OMAC, both of whom are highly respected, reliable laboratories. These results, summarized in Table 12.2, indicate that the average of all Atalaya copper assays is about 1.5% higher than the other two labs. Those differences are statistically significant for the Atalaya-OMAC pairing and marginally significant for the Atalaya-Alex Stewart pairing. Those samples below 1% Cu are virtually identical for the three labs and most of the difference is in the over-limit samples above 1% copper, which are assayed with a different procedure.

The Atalaya over-limit assays are nearly 4% higher than the other labs, however, and the difference is strongly statistically significant. It is notable that in the previous outside lab tests discussed in Section 12.3, the Atalaya over-limit assays were generally lower than those for Alex Stewart and AGQ. Further investigation of the over limit copper assays is recommended to resolve this issues with the over-limit copper assays. Because the Atalaya over-limit assays are only 1% of the all assays above 0.2% Cu, however, even if the 4% bias in over limit samples is correct, it would not materially affect the resource estimate.

In addition to the assays of the Lundin samples from Touro, 21 of the OREAS standard reference samples and 10 blank samples were included in the test protocol. These results, summarized in Table 12.3, show that Atalaya, OMAC and Alex Stewart all have reasonable close assays with respect to each other, but only the OMAC assays are consistent with the stated reference values. Considering that the three labs compare well with each other, these results further suggest that the OREAS standards are not reliable for QAQC purposes.

Although reference assays are not available for the blanks, all assays are very low grade and consistent among the three comparison laboratories.

Table 12.1 - External Lab Verification of Lundin Assays

Lab 1	Lab 2	Count	Average Lab 1	Average Lab 2	Diff (1-2)	Std Dev Diff	Relative Diff	t-dist[1]
All Samples								
Lundin	Atalaya	110	0.420	0.424	-0.0039	0.056	-0.9%	0.467
Lundin	Alex Stewart	110	0.420	0.419	0.0010	0.054	0.2%	0.851
Lundin	OMAC	110	0.420	0.416	0.0035	0.057	0.8%	0.521
Samples less than 1% Cu								
Lab 1	Lab 2	Count	Average Lab 1	Average Lab 2	Diff (1-2)	Std Dev Diff	Relative Diff	t-dist[1]
Lundin	Atalaya	100	0.303	0.303	-0.0003	0.036	-0.1%	0.929
Lundin	Alex Stewart	100	0.303	0.303	0.0001	0.034	0.0%	0.980
Lundin	OMAC	100	0.303	0.303	0.0002	0.032	0.1%	0.949
Over-limit Samples (greater than or equal to 1% Cu)								
Lab 1	Lab 2	Count	Average Lab 1	Average Lab 2	Diff (1-2)	Std Dev Diff	Relative Diff	t-dist[1]
Lundin	Atalaya	10	1.587	1.627	-0.0396	0.148	-2.5%	0.422
Lundin	Alex Stewart	10	1.587	1.577	0.0097	0.148	0.6%	0.841
Lundin	OMAC	10	1.587	1.551	0.0365	0.164	2.3%	0.501

[1]A t-test value less than 0.05 indicates that the difference between labs is statistically significant from zero.

Table 12.2 - External Lab Verification of Atalaya Assays

Lab 1	Lab 2	Count	Average Lab 1	Average Lab 2	Diff (1-2)	Std Dev Diff	Relative Diff	t-dist[1]
All Samples								
Atalaya	Alex Stewart	110	0.424	0.419	0.0049	0.026	1.2%	0.055
Atalaya	OMAC	110	0.424	0.416	0.0074	0.038	1.8%	0.043
Alex Stewart	OMAC	110	0.419	0.416	0.0025	0.025	0.6%	0.284
Samples less than 1% Cu								
Lab 1	Lab 2	Count	Average Lab 1	Average Lab 2	Diff (1-2)	Std Dev Diff	Relative Diff	t-dist[1]
Atalaya	Alex Stewart	100	0.303	0.303	0.0004	0.020	0.1%	0.837
Atalaya	OMAC	100	0.303	0.303	0.0005	0.026	0.2%	0.837
Alex Stewart	OMAC	100	0.303	0.303	0.0001	0.020	0.0%	0.952
Over-limit Samples (greater than or equal to 1% Cu)								
Lab 1	Lab 2	Count	Average Lab 1	Average Lab 2	Diff (1-2)	Std Dev Diff	Relative Diff	t-dist[1]
Atalaya	Alex Stewart	10	1.627	1.577	0.0493	0.040	3.1%	0.004
Atalaya	OMAC	10	1.627	1.551	0.0761	0.066	4.8%	0.006
Alex Stewart	OMAC	10	1.577	1.551	0.0268	0.050	1.7%	0.128

[1]A t-test value less than 0.05 indicates that the difference between labs is statistically significant from zero.

Table 12.3 - Results of Blanks and Standard Reference Samples

Standard	Lab	Count	Lab Average	Lab Std.Dev	Num. Ref Assays	%Cu Ref Value	Std.Dev Ref	Ave Diff. (Lab-Ref)	Relative Diff	Std Dev of Ave Diff	t-test[1] of Diff
blank	Atalaya	11	0.003	0.0035	No Reference Assay for Blanks						
	Alex Stewart	11	0.002	0.0035							
	OMAC	11	0.001	0.0035							
	Average	33	0.002	0.0019							
Oreas-922	Atalaya	5	0.212	0.005	107	0.212	0.009	0.000	-0.1%	0.002	0.959
	Alex Stewart	5	0.213	0.008	107	0.212	0.009	0.001	0.7%	0.004	0.730
	OMAC	5	0.219	0.005	107	0.212	0.009	0.007	3.4%	0.002	0.160
	Average	15	0.215	0.0068	107	0.212	0.009	0.003	1.3%	0.002	0.160
Oreas-923	Atalaya	6	0.433	0.017	112	0.423	0.024	0.010	2.4%	0.007	0.215
	Alex Stewart	6	0.437	0.015	112	0.423	0.024	0.014	3.2%	0.007	0.088
	OMAC	6	0.431	0.010	112	0.423	0.024	0.008	1.9%	0.005	0.128
	Average	18	0.434	0.0138	112	0.423	0.024	0.011	2.5%	0.004	0.011
Oreas-926	Atalaya	6	0.799	0.009	113	0.813	0.031	-0.014	-1.8%	0.005	0.011
	Alex Stewart	6	0.804	0.008	113	0.813	0.031	-0.009	-1.1%	0.004	0.052
	OMAC	6	0.815	0.007	113	0.813	0.031	0.002	0.2%	0.004	0.689
	Average	18	0.806	0.0104	113	0.813	0.031	-0.007	-0.9%	0.004	0.058
Oreas-927	Atalaya	4	1.070	0.049	101	1.077	0.030	-0.007	-0.6%	0.025	0.805
	Alex Stewart	4	1.061	0.028	101	1.077	0.030	-0.016	-1.5%	0.014	0.347
	OMAC	4	1.071	0.016	101	1.077	0.030	-0.006	-0.5%	0.009	0.548
	Average	12	1.068	0.0310	101	1.077	0.030	-0.009	-0.9%	0.009	0.336

[1]A t-test value less than 0.05 indicates that the difference between labs is statistically significant from zero.

13 MINERAL PROCESSING AND TESTING

13.1 Testwork Executive Summary

A rigorous testwork program was completed at SGS Australia in Perth, Western Australia to provide inputs to the Touro Technical Report. Eighty-eight samples were selected from drill core produced by the 2012 campaign undertaken by Lundin Mining for use in the testwork program. From the 88 samples, 12 composites were generated based on identified ore types. A high-level sample summary is detailed in Table 13.1.

Table 13.1 - Sample Summary

Composite	No. Samples	Orebody	Sample Grades			
		Avg Cu (%)	Avg Cu (%)	Avg S (%)	S:Cu	Mass (kg)
Arca A	4	-	0.35	6.65	19.0	75.8
Arca D	2	-	0.52	4.50	8.65	55.4
Arca Main	9	0.36	0.41	10.3	25.1	232
Arinteiro	6	0.54	0.50	4.47	8.94	287
Bama	14	0.37	0.42	3.48	8.29	347
Brandelos	10	0.37	0.40	4.66	11.7	308
Monte Minas Garnetite	8	0.51	0.48	8.51	17.7	209
Monte Minas Paragneiss	5	-	0.51	7.43	14.6	128
Monte Minas Upper	6	-	0.47	5.67	12.1	149
Vieiro	8	0.59	0.60	4.80	8.00	325
Vieiro High Grade	4	-	1.27	5.63	4.43	152
Vieiro Hardest Sample	1	-	0.02	1.10	55.0	42.3
Not Composited	11	-	-	-	-	258
Total	**88**	**0.46**	**0.50**	**5.60**	**11.3**	**2,567**

Comminution testwork was completed to confirm design parameters used in the crushing and grinding circuits. Results from the tests are summarized in Table 13.2.

Table 13.2 - Comminution Testwork Summary

Parameter	Unit	Minimum	Average	Maximum	Design
SMC DWi	kWh/m^3	5.72	8.18	9.90	**9.43**
SMC Axb	-	31.5	40.4	55.3	**34.0**
Ai	g	0.15	0.19	0.24	**0.19**
RWi	kWh/t	15.9	17.4	19.1	**18.4**
BWi	kWh/t	14.5	15.5	17.0	**16.1**

Flotation grind optimization tests were completed to enable a simple primary grind size economic evaluation to be performed. Tests were carried out at grind size P_{80} of 212, 150, 125, 106 and 75 µm. The analysis determined that a grind size P_{80} of 125 µm was the most economic.

Flotation tests were completed on the composites to determine the optimum reagent addition scheme and optimum regrind size. From this work, an optimized batch rougher/3 stage cleaner test was determined. The optimized batch test conditions were:

- Primary grind size P_{80} of 125 µm.
- Rougher laboratory flotation residence time of 10 minutes and rougher flotation reagent scheme of 15 g/t DSP 009 and 10 g/t PAX.
- Regrind size P_{80} of 20 µm.
- Cleaner 1, 2, and 3 laboratory residence times of 10, 7 and 4 minutes, respectively and cleaner reagent scheme of 15, 2.5 and 2.5 g/t DSP 009, respectively.

Batch tests were completed for the composites and two low grade samples (12BR10-PER-46 and 12BR08-PER-49) using the above conditions. Locked cycle tests were also performed on the composites to provide an estimation of plant performance in terms of middlings recovery. The locked cycle flowsheet used is detailed in Figure 13.21. The results from these tests are shown in Table 13.3.

Table 13.3 - Batch Cleaner Test Results

Sample	Head Grade %		Batch Cleaner Test							Locked Cycle	
			Cu Recovery %				Conc Grade % Cu				
	Cu	S	Rghr	Clnr 1	Clnr 2	Clnr 3	Clnr 1	Clnr 2	Clnr 3	%Rec	% Cu
Arca Main	0.41	10.3	86.4	85.4	83.6	78.1	16.4	23.3	28.7	87.0	23.6
Vieiro	0.60	4.80	92.9	92.3	91.0	87.8	22.5	29.3	32.4	92.5	30.7
Arinteiro	0.50	4.47	89.7	88.7	87.8	83.8	21.1	28.2	31.8	89.2	31.4
Bama	0.42	3.48	89.0	87.8	86.2	81.6	20.6	27.4	31.5	86.7	30.9
Brandelos	0.40	4.66	85.2	83.7	82.6	78.2	22.3	29.4	32.3	85.0	30.9
MM Garnetite	0.48	8.51	88.2	86.7	84.9	79.0	23.9	30.0	32.7	89.0	29.5
Viero High Grade	1.27	5.63	96.0	95.7	95.4	94.5	26.2	29.8	32.0	-	-
MM Upper	0.47	5.67	88.4	87.5	86.2	80.6	22.0	27.9	31.1	87.8	28.1
12BR10-PER-46	0.28	3.28	84.5	82.4	79.7	75.3	19.5	26.8	30.6	82.9	30.7
12BR08-PER-49	0.34	7.94	86.2	83.9	80.9	77.8	22.8	29.6	32.1	83.7	29.5
Average	**0.52**	**5.87**	**88.7**	**87.4**	**85.8**	**81.7**	**21.7**	**28.2**	**31.5**	**87.9**	**29.5**

Results from the locked cycle tests were compared with the corresponding batch tests and the following conclusions drawn:

- The locked cycle tests provided significantly higher recoveries to the final concentrate (cleaner 3) than the batch tests, demonstrating that the majority of the copper that reports to the batch test middlings can be recovered to final concentrate without significant dilution of the concentrate grade.
- The locked cycle test recovery is similar to the batch cleaner 1 recovery, at 0.54% higher on average.
- The locked cycle test concentrate grade is similar to the batch cleaner 2 concentrate grade, at 1.34% higher on average.

The above relationships were used to calculate locked cycle equivalent results for 74 variability sample batch testwork results. These data were then used to determine concentrate grade and recovery models from geo-chemical data available in the mine block model.

Evaluation of the variability concentrate grade data concluded:

- The concentrate grade is moderately correlated with S:Cu ratio and weakly correlated with copper head grade.
- Apart from ore classified as oxide, all other ore types behave similarly.

From this, a single equation based on copper head grade and S:Cu ratio provided the best prediction of concentrate grade (up to a maximum of 33% Cu) for all ore types except oxide:

$$C \quad G = 30.1 + 4.93 \times H \quad G - 0.29 \times \left(\frac{S}{C}\right)$$

Evaluation of the variability recovery data concluded:

- Copper recovery is strongly correlated with copper head grade.
- Vieiro ore consistently provided better recovery than the other ore types.
- Transitional ore provided a slightly lower recovery than primary ore.
- Oxide ore provided very low recovery and given the low concentrate grades achieved with the oxide samples, oxide ore has been classified as waste.
- Considering the typical scatter with this type of testwork, all other primary ore provided similar results.

From this evaluation, three separate recovery models were determined:

- **Transitional Ore:** Fixed tail grade of 0.094% Cu based on the average of the 6 transitional ore samples.
- **Vieiro Primary Ore:** Logarithmic model using copper head grade based on the 11 Vieiro primary ore samples.
- **Other Primary Ore:** Logarithmic model using copper grade based on the 55 other primary variability samples.

The two primary recovery models were then calibrated against the composite sample locked cycle test results. This process ensures that the overall recovery estimates hold true to the measured composite performance, which best emulate the performance of each ore type, while up-holding the recovery performance trend with head grade. An additional 0.8% recovery is added to the variability recovery formula to account for the average difference between variability samples and composites.

The final calibrated equations for determining the plant recovery for each ore domain are:

$$V \quad P \quad O \quad R = 5.09 \times \ln(H) + 94.9$$

$$A \quad O\,he \quad P \quad O \quad R = 7.196 \times \ln(H) + 92.9$$

$$Tr \quad O \quad R = \frac{100 \times C \times (H - 0.094)}{H \times (C - 0.094)}$$

Where:

- CG = Concentrate Grade (%Cu).
- HG = Head Grade (%Cu).

13.2 Testwork and Flowsheet Development

Minnovo reviewed the testwork report "*Lundin Mining – Santiago Copper project, Preliminary Testwork on Samples from a Spanish Copper Project*" dated June 2012, provided by Wardell Armstrong International (WAI) and also reviewed the preliminary flowsheet developed by AMEC for the October 2012 PEA. The flowsheet used for this report is similar to the preliminary flowsheet developed by AMEC in the previous PEA. The flowsheet and equipment selection have been refined by Minnovo based on recent experience at Atalaya's Riotinto operation and optimized based on preliminary economic evaluation of the WAI testwork results to reduce the capital cost of the process plant. A new testwork program has been conducted to support the optimized design and Technical Report.

13.3 Previous Metallurgical Studies

Table 13.4 provides an overview of existing metallurgical studies available for the Project.

Table 13.4 – Existing Metallurgical Studies and Documents

Date	Title	Author
1969	Informe Flotación Fornas	Rio Tinto Minera S.A., Jefe Mineralurgia, J.M. Nieto, November 1969
1969	Proyecto Santiago, Anexos a los Informes de Concentración por Flotación de los Minerals de Arinteiro (Octubre 1969) y Fornás (Noviembre 1969)	Laboratorio Preparacion Minerales Rio Tinto Patiño, S.A., Rio Tinto (prov. Huelva), December 1969
1970	Proyecto Santiago, Informe de Concentración por Flotación Minerales de Fornas y Arinteiro (Santiago de Compostela)	Laboratorio Preparacion Minerales Rio Tinto Patiño, S.A., Rio Tinto (prov. Huelva), April 1970
1970	Algunas sugerencias al proyecto de Santiago actual con 30% de aumento de tonelaje sobre la capacidad original (véase carta de McKee del 25 de Octubre de 1972)	Rio Tinto Patiño, S.A , 10 November 1970
1973	Copies of Progress Reports No.2 on an investigation of the recovery of copper on an ore sample from the Bama deposit, submitted on behalf of Rio Tinto	Lakefield Research of Canada Limited, Lakefield Ontario, Canada, A.G. Scobie, 23 April 1973
1973	Progress Report No.3 on an investigation of the recovery of copper on an ore sample from the Bama deposit, submitted on behalf of Rio Tinto	Lakefield Research of Canada Limited, Lakefield Ontario, Canada, 28 September 1973
1973	Comentarios Lakefield	Not available.
1974	Memorandum de Orden Interno: Informe Tratmiento por Flotación de Mineral Arinteiro Sur (Proyecto Santiago)	Rio Tinto Patiño, S.A , J.M. Nieto, 20 July 1974
1974	Lakefield Comments On Bama Ore	Lakefield Research of Canada Limited, Lakefield Ontario, Canada, Fernando Herranz, January 1974
1974	Fuente Rosas - Muestra 519 – Estado Actual del Estudio Asunto Mineral Cobre de Santiago	Sociedad Minera y Metalurgica de Peñarroya – España, Centro de Estudios y Analysis Mineralurgicos, August 1974

Date	Title	Author
1978	Proyecto de Explotacion del Yacimiento de Bama, Santiago de Compostela	Rio Tinto Minera S.A., February 1978
1979	Proyecto de Explotacion del Yacimiento de Bama, Santiago de Compostela	Rio Tinto Minera S.A., January 1979
1985	Analisis de las Producciones Acturales del Concentrador de Santiago	Rio Tinto Minera S.A., 1 July 1985
Not available	Biolixiviation des Mineraux Pauvres de Cuivre de Santiago de Compostelle, Note de Synthèse, Y.M. le Nindre, J.F. Sureau, M. Leleu	Ministère de L'Industrie et de la Recherche, Bureau de Recherches Géologiques et Minières, Service Géologiques National, BRGM, Département labaratoires
Not available	Biolixiviation des Mineraux Pauvres de Cuivre de Santiago de Compostelle, Rapport Préliminaire, Y.M. le Nindre, J.F. Sureau, M. Leleu	Ministère de L'Industrie et de la Recherche, Bureau de Recherches Géologiques et Minières, Service Géologiques National, BRGM, Département labaratoires
Not available	Investigación realizada sobre la recuperación de granates a partir del estéril de la planta de flotación de cobre de Arinteiro	Rio Tinto Minera S.A., Carracedo Villamor, H.-M.
Not available	Investigación realizada sobre la recuperación de granates a partir del estéril de la planta de flotación de cobre de Arinteiro.	Rio Tinto Minera S.A., Carracedo Villamor, H.-M.
June 2012	Preliminary Testwork On Samples from the Santiago Copper Project – Lundin Mining	Wardell Armstrong International (UK)
June 2016	Comminution And Flotation Testwork On Copper Samples From The Touro Project	SGS Australia

13.3.1 Wardell Armstrong Testwork Summary

In 2012, Wardell Armstrong International (WAI) completed preliminary laboratory testwork on samples of mineralization for the Touro Copper Project.

Four samples of split drill core were received at the WAI laboratory from the Bama, Arinteiro, Arca, and Fuente Rosas deposits. The samples graded from 0.51% Cu (Arca) to 0.87% Cu (Fuente Rosas) with an average of 0.69% Cu. Note that the Fuenta Rosas deposit is not part of the current evaluation and the results are only included for completeness.

Bond Ball Mill Work Index (BWi) tests were completed on each ore type, with the results summarized in Table 13.5.

Table 13.5 - WAI Comminution Test Results

Sample	Ball Mill Bond Work Index (kWh/t)
Arca	16.9
Arinteiro	18.7
Bama	16.8
Fuente Rosas	17.7
Average	17.5

Rougher flotation testwork was completed on each ore type. Variables tested were grind size, reagent type and dosage rate and pulp pH.

Table 13.6 summarizes the optimum rougher flotation parameters determined for each ore type, as well as for the ARACBA composite (equal parts of the Arinteiro, Arca and Bama ore types).

Table 13.6 - WAI Rougher Flotation Test Results

Sample	Grind Size P_{80} (µm)	pH	Time (min)	Collector	Dosage (g/t)	Cu Grade (% Cu)	Cu Rec. (%)
Bama	75	11	15	PAX	50	9.7	95.7
Arinteiro	75	11	15	C4132	25	9.3	95.0
Arca	75	11	15	C4132	25	7.1	95.0
Fuente Rosas	75	11	15	C4132	25	9.1	95.8
ARACBA	75	11	15	C4132 + PAX	20 + 5	5.7	96.3

Cleaner tests were completed on the Fuente Rosas ore type and the ARACBA composite. Table 13.7 summarizes the optimum cleaner parameters that were determined.

Table 13.7 - WAI Cleaner Flotation Test Results

Sample	Grind Size P_{80} (µm)	Cleaning Stages	Time (mins)	Collector	Dosage (g/t)	Cu Grade (% Cu)	Cu Rec. (%)
Fuente Rosas	21	1	8	C4132	10	20.5	93.0
Fuente Rosas	35	3	10 / 7 / 4	C4132	5	26.5	87.5
ARACBA	22	1	10	C4132	5	18.2	94.3
ARACBA	35	3	10 / 7 / 4	C4132	5	26.3	90.4

Locked cycle tests were completed on each ore type and the ARACBA composite. Table 13.8 summarizes the optimum locked cycle parameters determined.

Table 13.8 - WAI Locked Cycle Flotation Test Results

Sample	Grind size P_{80} (µm)	Regrind Size P_{80} (µm)	Cleaner Stages	Time (mins)	Collector	Dosage (g/t)	Cu Grade (% Cu)	Cu Rec. (%)
Bama	75	24	3	10 / 7 / 4	PAX	5	26.2	88.9
Arinteiro	75	20	3	10 / 7 / 4	C4132	5	27.6	89.6
Arca	75	23	3	10 / 7 / 4	C4132	5	27.2	91.6
Fuente Rosas	75	20	3	10 / 7 / 4	C4132	10	29.8	93.0
ARACBA	75	23	3	10 / 7 / 4	C4132	5	26.1	92.9

The WAI testwork focussed on a primary grind size P_{80} of 75 µm to maximise recovery. An economic evaluation of the testwork results indicated that a coarser primary grind size with a P_{80} greater than 106 µm will improve overall project economics, despite a slight reduction in recovery.

The outcomes of the WAI testwork provided input into the 2015 testwork program in terms of residence time, reagent selection, primary grind size and regrind size.

13.4 Metallurgical Testwork

The latest testwork program was developed by Minnovo to supplement the data from the WAI program and provide input to the technical report. Samples were selected from the core drilled in the 2012 Lundin drilling campaign.

The testwork program was completed between December 2015 and June 2016 at SGS Australia in Perth, Western Australia.

The testwork program included:

- Mineralogical analysis.
- Comminution tests.
- Flotation tests.
- Materials handling tests.

13.4.1 Samples

Eighty-eight samples of approximately 25 kg each were selected from the remaining core from the 2012 drilling campaign. A key criterion for sample selection was that the sample contained at least a 10 meter continuous interval of core, including typical levels of internal waste. This was conducted to replicate a full bench height, typical of the feed that a concentrator would receive.

Individual samples were selected so that a range of head grades, orebody locations, depths, lithologies and oxidation states could be tested. Samples were selected so that the minimum and maximum head grade range expected for each orebody was represented. The minimum head grade expected for most orebodies was 0.20% Cu, (material containing less than 0.20% Cu was considered waste) and the maximum was typically around 1.0% Cu.

The individual samples were selected so that when all samples from a particular orebody were composited, the composite would have a head grade representative of that orebody. As the majority of flotation testwork was completed on orebody composites, sample selection was critical to ensure the composite head grade was representative. Careful consideration was made when selecting the individual core samples for each orebody and several iterations were required before the core sample selections were finalised. A summary of the orebody head grade and orebody composite head grade is detailed in Table 13.9.

The map detailing the individual orebodies and the locations of all holes drilled during the 2012 campaign was examined. Individual drillholes were chosen so that the extents of the orebodies were represented in the material selected for the testwork program. For the drillholes chosen for an orebody, depth intervals were selected so that the depth variation was represented as far as possible.

Major lithologies that were present in most orebodies included garnetite, amphibolite and paragneiss. Samples were selected from each orebody so that all lithologies were represented.

Wherever possible, drillhole intervals that included variations in oxidation were selected. This was not always possible as the samples that exhibited the greatest oxidation were typically samples close to the surface and were considered waste.

An example of the core is shown in Figure 13.1. The Metallurgical drillholes are shown in Figure 13.2.

Figure 13.1 - Typical Drill Core Photo (Photo taken by T. van Bockxmeer, Minovo 2016)

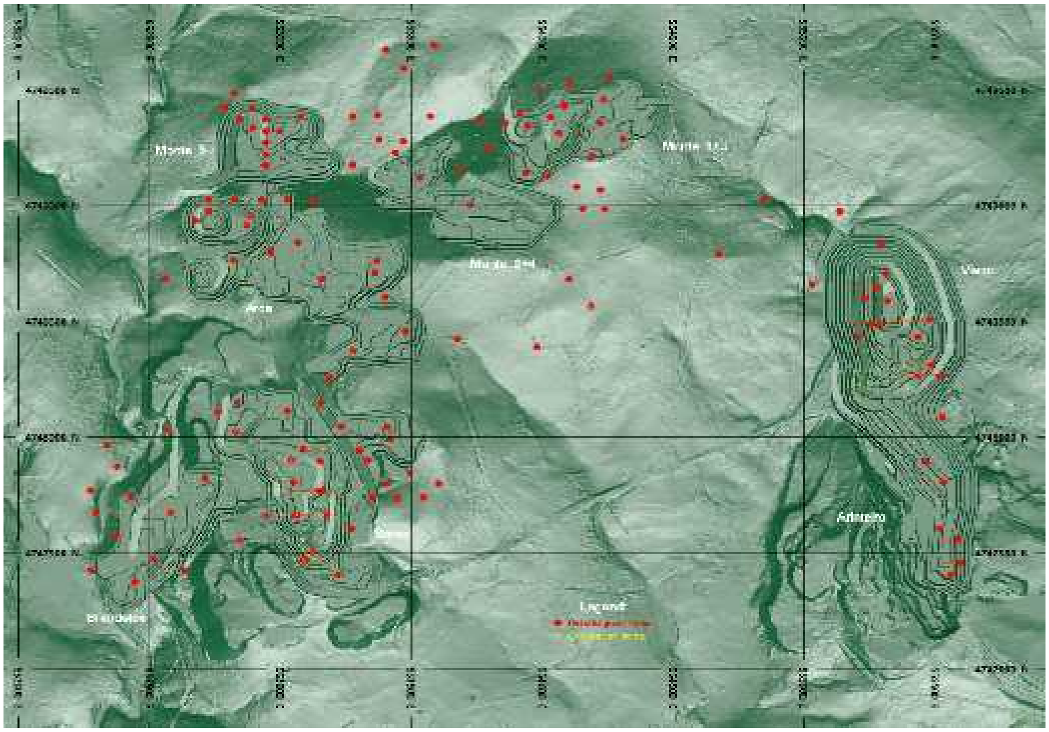


Figure 13.2 – Metallurgical drill Hole Locations (Atalaya 2016)

13.4.2 Ore Types

The following ore types were defined for the testwork program, principally based on each ore deposit in the mine schedule:

- Arca
- Arca A
- Arca D
- Vieiro
- Arinteiro
- Bama
- Brandelos
- Monte Minas Garnetite
- Monte Minas Paragneiss
- Monte Minas Upper
- Monte Minas
- Vieiro High Grade

Samples that exhibited oxidation in terms of physical appearance, cyanide soluble copper (CN_{Cu}) assays above 7% and acid soluble copper assays above 2% were defined as transitional samples. Transitional samples were separated from non-oxidised or 'primary ore' samples in the analysis of grade and recovery.

Oxide samples (as identified by Lundin Mining during the 2012 drilling campaign) were evaluated separately and eventually removed from the data set on the basis of very poor results achieved on the two samples tested. Though not included in model development, these samples are included on relevant plots to demonstrate how poorly the oxide material performed relative to primary and transitional ores.

Composite samples for the major orebodies (Arca, Vieiro, Arinteiro, Bama and Brandelos) were generated from the individual samples. Composites were also generated for the Arca A and Arca D orebodies. The Monte de la Minas samples were separated into Monte Minas Upper, Monte Minas Garnetite and Monte Minas Paragneiss composites.

The composites were prepared so that the grades matched closely with the average ore type grade while maximising the spatial distribution. A summary of the composites produced from the samples selected is detailed in Table 13.9.

Table 13.9 – Testwork Sample High Level Summary

Composite	No. Samples	Orebody	Samples			
		Avg Cu (%)	Avg Cu (%)	Avg S (%)	S:Cu	Mass (kg)
Arca A	4	-	0.35	6.65	19.0	75.8
Arca D	2	-	0.52	4.50	8.65	55.4
Arca Main	9	0.36	0.41	10.3	25.1	232
Arinteiro	6	0.54	0.50	4.47	8.94	287
Bama	14	0.37	0.42	3.48	8.29	347
Brandelos	10	0.37	0.40	4.66	11.7	308
Monte Minas Garnetite	8	0.51	0.48	8.51	17.7	209
Monte Minas Paragneiss	5	-	0.51	7.43	14.6	128
Monte Minas Upper	6	-	0.47	5.67	12.1	149
Vieiro	8	0.59	0.60	4.80	8.00	325
Vieiro High Grade	4	-	1.27	5.63	4.43	152
Vieiro Hardest Sample	1	-	0.02	1.10	55.0	42.3
Not Composited	11	-	-	-	-	258
Total	**88**	**0.46**	**0.50**	**5.60**	**11.3**	**2,567**

13.4.3 Mineralogical Analysis

13.4.3.1 Head Analysis

A sample was taken from each of the 12 composites for head analysis with the results shown in Table 13.10. In addition, a subsample was taken for 74 of the 88 variability samples for assay. (A number of samples were retained as drill core in case additional comminution testwork was required).

Table 13.10 - Composite Head Assay Results

Assay (ppm*)	Composite								
	Arca Main	Vieiro	Arinteiro	Bama	Brandelos	MM Garnetite	Arca A	MM Upper	Vieiro HG
Cu_{TOT}, %	0.36	0.47	0.43	0.34	0.35	0.44	0.26	0.42	1.10
SUL_{Cu}	20	50	30	40	40	20	20	100	30
CN_{Cu}	240	730	330	210	240	250	180	330	550
Au	0.02	0.04	0.05	0.03	0.03	0.04	0.01	0.03	0.07
Ag	3.60	1.30	0.70	0.70	1.00	2.80	2.10	1.70	1.50
Pb	39	27	13	32	32	80	38	18	18
Zn	4.290	392	220	280	594	2,810	2,290	2,060	266
As	8	36	3	5	<3	4	<3	6	7
Ni	102	118	128	96	129	83	66	122	143
Fe, %	22.3	16.8	14.1	16.1	17.1	19.5	18.2	15.3	16.3
Co	152	170	128	116	110	135	113	121	296
S, %	10.3	4.80	4.47	3.48	4.66	8.51	6.65	5.67	5.63

* Unless otherwise noted

A comparison of the measured (assay) head grade vs. the calculated head grade from the batch flotation tests is detailed in Figure 13.3. The difference between the measured and calculated values averages 4%, though some samples exhibited differences of up to 14%, indicating that there may have been errors in assaying.

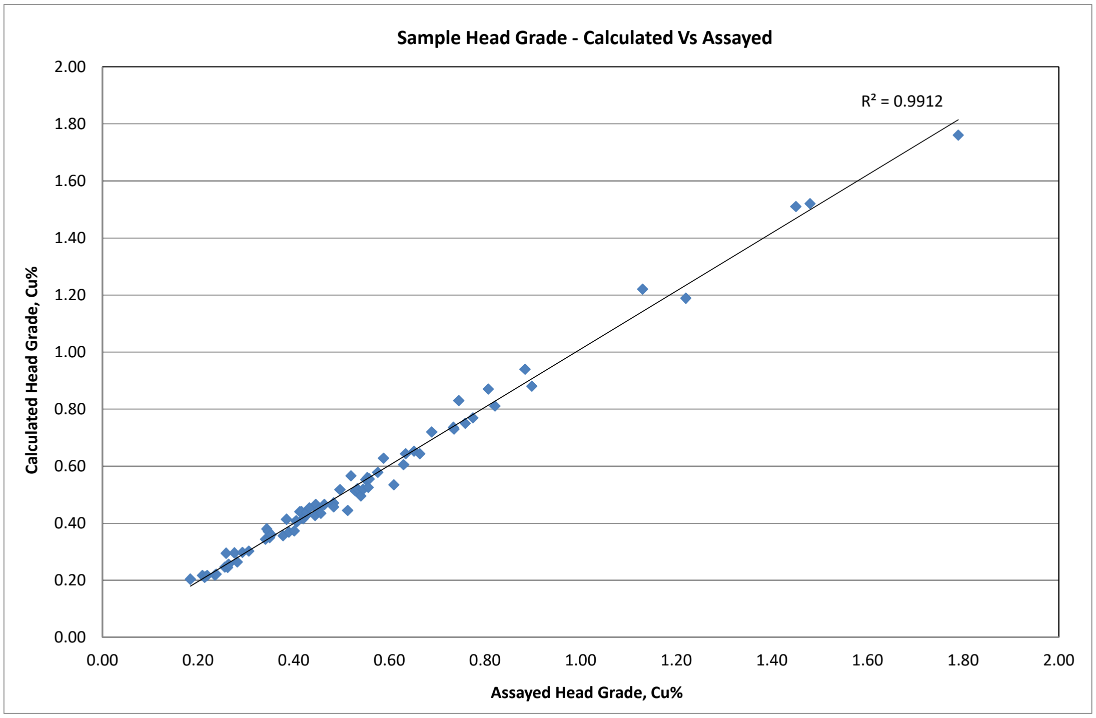


Figure 13.3 – Graph of Variability Sample Head Grade, Calculated vs. Assay (Touro testwork analysis, Minnovo 2016)

13.4.3.2 Mineralogical Analysis

Optical mineralogical analysis was completed by SGS on samples of the Arca and Vieiro composites, prepared to the nominal primary grind size of P_{80} 125µm. QEMSCAN PMA (particle mineralogical analysis) and SMS (specific mineral search) measurements were conducted on prepared mounts from which the following data was obtained:

- Mineral list.
- Bulk modal mineralogy.
- Estimated grain and particle sizes.
- Cu and Zn deportment.
- Liberation and associations of chalcopyrite, sphalerite and pyrrhotite.
- Mineralogically limiting grade-recovery curves for Cu.
- Particle mineral maps of selected fractions.

Both tested composites have a similar mineralogy, albeit with minerals in different proportions. The sulfides consist mainly of pyrrhotite, chalcopyrite, sphalerite with lesser pyrite and trace amounts of cobaltite and arsenopyrite. In the Arca composite, the sulfides comprise just over 25% of the sample, with pyrrhotite making up the bulk of the sulfides. Chalcopyrite contents are 1.4% and sphalerite 0.7%. In the Vieiro composite, sulfides comprise only 12% of the sample, with pyrrhotite making up 6.6%. Chalcopyrite contents are 3.3% higher in this composite.

Silicates consist mainly of quartz, chlorite, garnet and amphibole with lesser feldspars, micas, epidote and staurolite and andalusite/kyanite/sillimanite. Staurolite is noteworthy in that it contains quantities of zinc. The Vieiro composite contains more silicates than the Arca composite because it contained fewer sulfides. The Vieiro composite contains proportionally higher levels of garnet and quartz.

Other minerals include carbonates, rutile, ilmenite, iron oxides/hydroxides and the usual accessory phases; apatite, titanate and zircon. A zinc-bearing spinel was also noted.

In both composites, the non-sulphide gangue is the coarsest material followed by pyrrhotite and pyrite. Chalcopyrite and sphalerite have similar size profiles.

All copper in both composites deports to chalcopyrite. Most of the zinc deports to sphalerite but there are trace-to-minor quantities in staurolite and in Zn-spinel. In the Vieiro composite, about 3.3% of the zinc is in staurolite and about 0.8% in Zn-spinel. The Vieiro composite has only trace quantities of sphalerite (0.06% overall) whereas the Arca composite has 0.7% sphalerite.

In the Arca composite, around 55% of the chalcopyrite is liberated; locked grains are associated mostly with pyrrhotite, non-sulphide gangue and in multiphase particles. In the Vieiro composite, around 66% of the chalcopyrite is liberated; locked grains occur mostly with non-sulphide gangue. The lower amount of locking with pyrrhotite reflects the lower concentration of pyrrhotite in this composite.

Both the Arca and Vieiro composites show that about 25% of the locked chalcopyrite is associated with the non-sulphide gangue, indicating that overall recovery may increase with finer primary grinding prior to rougher flotation. Testwork completed to assess the rougher flotation recovery at various grind sizes tends to support this assumption.

In the Arca composite, around 43% of the sphalerite is liberated. Most of the locking occurs with non-sulphide gangue (34%) and pyrrhotite (14%). There is a much lower liberation of sphalerite in the Vieiro composite. This composite contains only trace amounts of sphalerite. Most of the locking is with non-sulphide gangue (64%).

Pyrrhotite is well liberated in both composites (79% in both). Most locking occurs in non-sulphide gangue (13 to 14%) with some locking with chalcopyrite (6% for Arca and 3% for Vieiro).

Mineralogical limiting grade-recovery curves for chalcopyrite were calculated using the liberation data as shown in Figure 13.4. This shows the Vierio composite having a higher grade-recovery (greater liberation) than Arca. The mineralogical limiting Cu grade and recovery curves closely align with experimental data generated for rougher flotation completed under semi-optimised conditions. The curves demonstrate relatively high copper recovery of approximately 84% and 78% at 27% concentrate grade for Vierio and Arca respectively at the primary grind size, and that a regrind will be required to reach acceptable recovery.

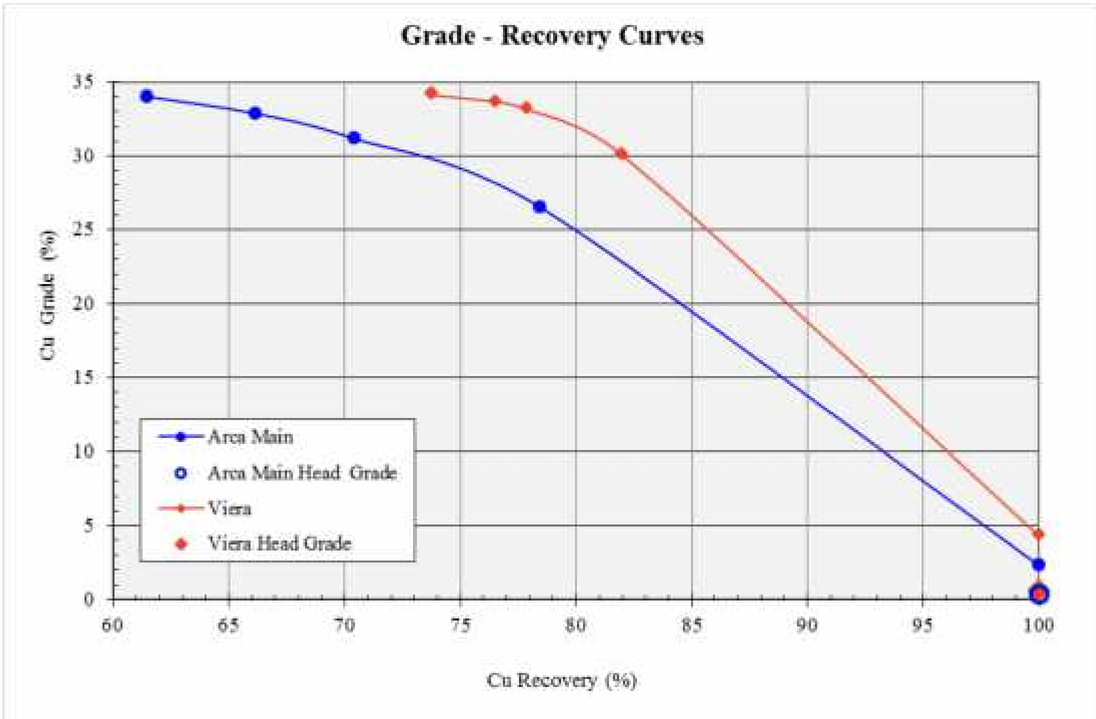


Figure 13.4 – Mineralogical Limiting Grade Recovery Curves (Touro testwork, SGS 2016)

13.4.4 Comminution Testwork

13.4.4.1 SMC Test

An SMC Test was completed on each of the 12 composites. The SMC Test was developed by Steve Morrell of SMCT Testing. The parameter determined by the test, Axb, is a measure of ore competency. The results are summarized in Table 13.11. The samples are of moderate competency with an average Axb of 40.4 and a 25th percentile value of 34.0.

Note, the Axb value is inversely proportional to ore competency and as such, the 25th percentile (not 75th percentile) is used for design i.e. lower Axb signifies a more competent ore.

Table 13.11 - SMC Test Results

Composite	Axb
Arca Main	34.0
Vieiro	47.2
Arinteiro	34.5
Bama	34.0
Brandelos	36.7
Monte Minas Garnetite	45.7
Arca A	48.0
Arca D	37.2
Monte Minas Upper	31.5
Monte Minas Paragneiss	47.6
Vieiro High Grade	55.3
Vieiro Hardest Sample	32.8
Minimum	**31.5**
Maximum	**55.3**
Average	**40.4**
25^{th} Percentile	**34.0**
Design	**34.0**

13.4.4.2 Bond Rod Mill Work Index

Bond Rod Mill Work (RWi) tests were completed on the 6 main composites with a closing screen of 1.18 mm. The results are summarized in Table 13.12.

The samples have a moderate hardness with an average RWi of 17.4 kWh/t. It was observed that the variation of ore hardness from the different deposits was low, with a range of 15.9 - 19.1 kWh/t. The 75^{th} percentile value is 18.4 kWh/t.

Table 13.12 - Bond Rod Mill Work Index Results

Composite	Bond Rod Mill Work Index, (kWh/t)
Arca Main	16.3
Vieiro	15.9
Arinteiro	18.0
Bama	19.1
Brandelos	18.6
Monte Minas Garnetite	16.4
Minimum	**15.9**
Maximum	**19.1**
Average	**17.4**
75th Percentile	**18.4**
Design	**18.4**

13.4.4.3 Bond Ball Mill Work Index

Bond Ball Mill Work (BWi) tests were completed on each of the 12 composites with a 150 µm closing screen resulting in P_{80}'s of approximately 120 µm. The results are summarized in Table 13.13 below.

The samples have a moderate hardness with an average BWi of 15.5 kWh/t. It was observed that the variation of ore hardness from the different deposits was low, with a range of 14.5 to 17.0 kWh/t. The BWi design point is the 75th percentile value of all bond ball mill work index test results.

While these results are slightly lower than those obtained by WAI, this is consistent with the different closing screen sizes used for the tests (150 µm for SGS and 100 µm for WAI).

Table 13.13 - Bond Ball Mill Work Index Results

Composite	Bond Ball Mill Work Index, (kWh/t)
Arca Main	14.9
Vieiro	14.7
Arinteiro	16.2
Bama	16.1
Brandelos	15.9
Monte Minas Garnetite	14.5
Arca A	15.2
Arca D	16.2
Monte Minas Upper	17.0
Monte Minas Paragneiss	14.7
Vieiro High Grade	15.7
Vieiro Hardest Sample	15.5
Minimum	**14.5**
Maximum	**17.0**
Average	**15.5**
75^{th} Percentile	**16.1**
Design	**16.5**

13.4.4.4 Abrasion Index

Abrasion Index (Ai) tests were completed on each of the 6 main composites. The results are summarized in Table 13.14 below. The samples show a moderate abrasion with an average Ai of 0.19 g

Table 13.14 - Abrasion Index Results

Composite	Abrasion Index, (g)
Arca Main	0.15
Vieiro	0.24
Arinteiro	0.24
Bama	0.15
Brandelos	0.18
Monte Minas Garnetite	0.20
Minimum	**0.15**
Maximum	**0.24**
Average	**0.19**
75th Percentile	**0.23**
Design	**0.19**

13.4.5 Grind Calibration Testwork

Grind calibration tests were completed on the major composites to determine the laboratory grind time required to achieve a nominal product with a given F_{80} particle size. A plot of Bond Ball Mill Work Index against grind time (Figure 13.5) shows a distinct trend for all composites, and there is little variance among the results for the nine composites tested.

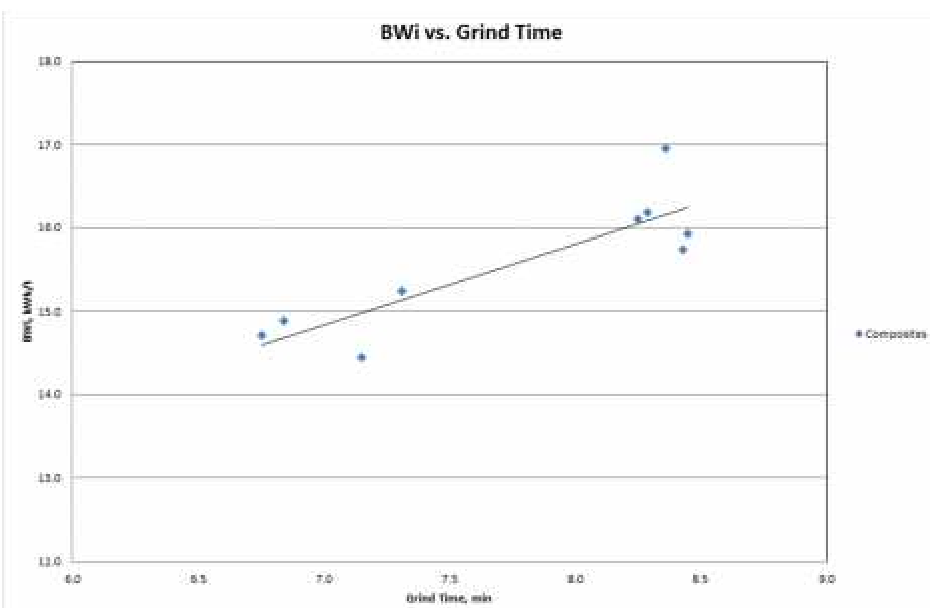


Figure 13.5 - Composite Bond Ball Mill Work Index vs. Grind Time (Touro testwork analysis, Minnovo 2016)

13.4.6 Materials Handling Testwork

Material handling testwork was completed at Bulk Materials Engineering Australia (BMEA) at the University of Wollongong, NSW Australia. The testwork is summarized below.

Material from Composite 2 (Vieiro) was used for the tests and included:

- Worst case moisture determination/handle-ability.
- Low consolidation testing under instantaneous conditions.
- Low consolidation testing with undisturbed storage time.
- High consolidation testing under instantaneous conditions.
- High consolidation testing with undisturbed storage time.
- Static angle of repose.
- Conveyor surcharge angle.

Flow property tests were completed on material to determine the moisture content range for the maximum bulk strength (i.e. the worst handle-ability), which was determined to be 6.2% moisture. The material at 6.2% moisture displayed moderate cohesive strength. In this condition the material is expected to exhibit satisfactory flow properties when handled on conveyors, in chutes, in bins, stockpiles, and on feeders.

From the flow property tests, preliminary design parameters for bins and hoppers were determined and the following recommendations were made:

- For mass-flow bins, wedge or plane-flow hoppers should be considered wherever possible.
- Conical or axisymmetrical hoppers should be avoided due to the steep angles required.
- Expanded-flow bins can provide a compromise between mass-flow and funnel-flow (e.g. reliable flow in the mass-flow hopper and complete clean-out of the funnel-flow bin taking critical rat-hole diameters into consideration). The funnel-flow section of the expanded-flow bin is not subjected to the relatively high wear rates that occur in mass-flow bins/hoppers.
- Arcoplate 1600 with a polished finish should be considered for a mass-flow hopper liner. Duaplate D80 and Bisplate 400 displayed high wall friction and required steeper walls for mass-flow.
- Plots of critical rat-hole diameter as a function of effective head of solids were generated. These plots indicate a high likelihood for the material to form large, stable rat-holes. As a consequence of this, care is needed when designing funnel-flow bins and hoppers, and also stockpiles.

13.4.7 Flotation Testwork

13.4.7.1 Methodology

A series of flotation tests were performed on the major composites in order to determine the optimum conditions for maximising grade and recovery of the copper minerals. All laboratory flotation tests were completed using an FLS flotation machine. Grinding was completed at nominally 60% w/w solids with lime added to the mill to maintain the required pH.

All rougher tests were completed using a 5.0 L cell, with water added to the mill discharge slurry to achieve a pulp density of 33% w/w solids. The impeller speed used was 850 rpm and air was added to the cell at the rate of approximately 20 L/min.

Cleaner tests were completed using a 0.5 L cell, with water added to cell to achieve the required pulp level. The pulp densities are dictated by the concentrate mass from the preceding stage. The typical densities achieved were:

- Cleaner 1 15 - 20 % w/w solids.
- Cleaner–scavenger 8 – 12 % w/w solids.
- Cleaner 2 6 – 10 % w/w solids.
- Cleaner 3 5 – 8 % w/w solids.

The impeller speed used was 1,200 rpm and air was added to the cell at the rate of between 1 to 3 L/min as required.

Locked cycle tests were completed on the Arca, Vieiro, Arinteiro, Bama, Brandelos, Monte Minas Garnetite and Monte Minas Upper composites and two low grade samples (2BR10-PER-46 and 12BR08-PER-49). The tests were completed with the objective of recycling the tailings streams from the second and third cleaner stages, thereby generating a final concentrate of grade and recovery representative of an operating plant.

Flotation tests using site water were scheduled, though not completed. It was considered that the logistics of having site water shipped to Perth, Western Australia outweighed the value of completing the tests.

Subsequent feedback from Atalaya's Riotinto project has indicated that site water quality may become an issue due to acidification of the process water by active sulfides. Additional flotation testwork may be required to confirm if this is an issue for Touro.

13.4.7.2 Sighter Tests

A series of 4 sighter tests were carried out on the Arca composite at conditions identified by the earlier WAI testwork, though at the coarser primary grind size P_{80} of 150 µm. The conditions tested and results achieved are summarized in Table 13.15.

Orica DSP 009 collector was used in place of Flomin C4132 as it was unavailable at the time of testing. Orica DSP 009 and Flomin C4132 are chemically equivalent (isopropyl ethyl thionocarbamate).

Once Flomin C4132 was available, tests were completed (TOR062 and TOR063) to compare the performance of the two collectors. DSP 009 was shown to provide improved recovery at the same concentrate grade using the same flotation parameters and the decision was made to continue using DSP 009 rather than change to Flomin C4132.

Table 13.15 - Arca Sighter Tests

Test	Grind Size P_{80} (µm)	pH	Rghr Residence Time (min)	Cleaning Stages	Residence Time (min)	Collector	Dose Rate (g/t)	Recovery at 27% Cu
TOR001	150	11	15	1	15	PAX	55	-
TOR002	150	11	10	3	10/7/4	DSP 009	35	75.5
TOR003	150	11	10	3	10/7/4	DSP 009 + PAX	20 / 10	79.5
TOR004	150	11	10	3	10/7/4	DSP 009 + PAX	22 / 12.5	75.0

Results of these four tests are detailed in Figure 13.6, in terms of copper grade and recovery. The curve from FCT5 from the WAI testwork has been included for comparison. Test FCT5 was conducted with a different collector regime and at a finer grind size P_{80} 75 µm.

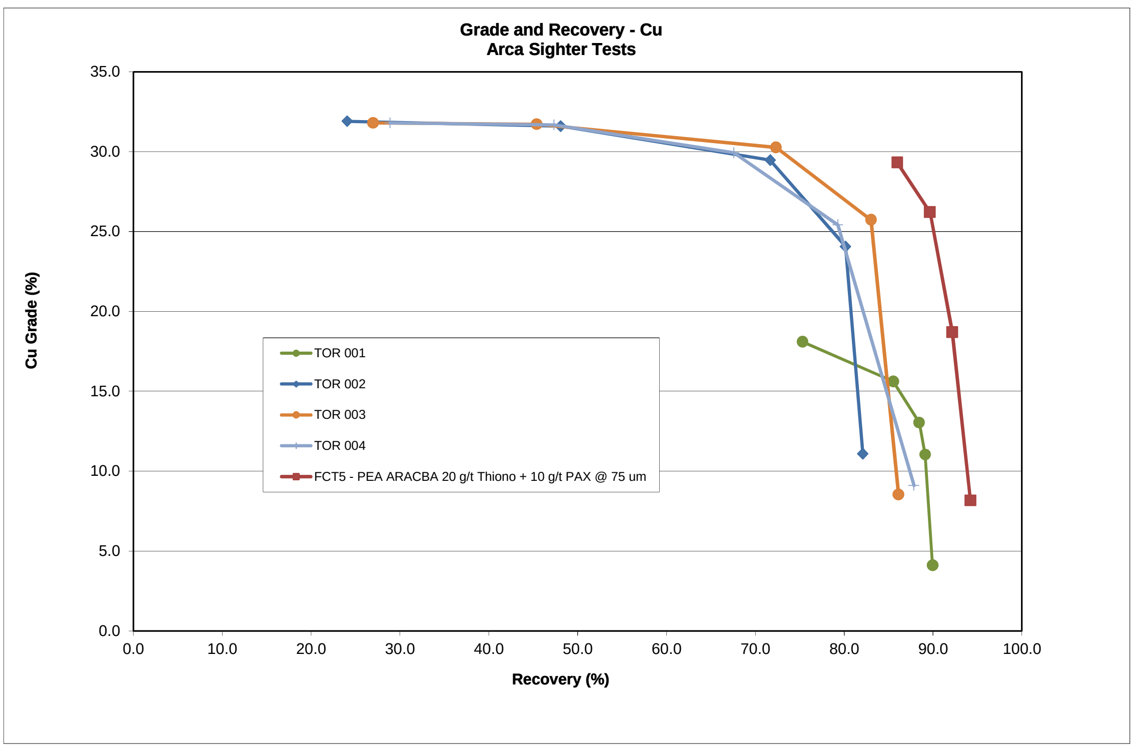


Figure 13.6 - Arca Sighter Cleaner Test Grade vs. Recovery Curves (Touro testwork analysis, Minnovo 2016)

The results of the sighter tests are considered to be good given that the tests were open circuit with no recycle of the tailings streams. The recoveries achieved are lower than those seen in the WAI testwork which was expected given the coarser primary grind size.

13.4.7.3 Rougher Grind Optimization Tests

Rougher flotation tests were completed on the main composites at the following conditions:

- Primary grind size varied between a P_{80} of 212 µm and a P_{80} of 75 µm.
- pH 11.
- Rougher residence time of 14 minutes.
- DSP 009 and PAX collectors added at 17.5 g/t and 12.5 g/t respectively.
- Frother added as required.

The results achieved are summarized in Table 13.16.

The Touro ores are fast-floating, achieving an average rougher concentrate grade of 12.6% Cu with an average Cu recovery of 90.0% after 6 minutes residence time at grind P_{80} of 125 µm. The average concentrate achieved after 10 minutes was 10.8% Cu with a recovery of 91.3%.

The results show that recovery increases with decreasing grind size as expected due to improved liberation. It was observed for some ore types that the grade-recovery curves at P_{80} of 125 and 106 µm were very similar, indicating that the majority of the recovered copper minerals were liberated at 125 µm. That this was not evident in all ore types indicates that liberation size varies between the ore types.

Table 13.16 - Rougher Grind Optimization Test Results

Composite	Test	Grind Size P_{80} (µm)	Grade (%Cu)	Recovery (%)	Grade at 10 min (%Cu)	Recovery at 10 min (%)
Arca	TOR005	212	8.19	86.8	8.91	85.2
	TOR006	150	8.73	88.8	9.49	87.7
	TOR007	106	8.73	91.7	9.46	91.0
	TOR008	75	9.18	93.2	9.90	92.7
	TOR017	125	8.30	90.9	9.05	89.9
Vieiro	TOR018	212	10.0	89.8	11.3	88.7
	TOR019	150	10.3	93.3	11.7	92.5
	TOR020	125	11.7	93.5	13.2	92.8
	TOR021	106	12.3	94.5	13.8	93.9
	TOR022	75	14.7	94.3	16.3	93.9
Arinteiro	TOR036	150	9.09	89.2	10.4	88.1
	TOR037	125	9.56	91.6	11.0	90.9

Composite	Test	Grind Size P_{80} (µm)	Grade (%Cu)	Recovery (%)	Grade at 10 min (%Cu)	Recovery at 10 min (%)
	TOR038	106	9.48	92.7	11.0	92.7
Bama	TOR039	150	9.53	88.4	10.9	87.5
	TOR040	125	9.43	90.9	10.8	90.2
	TOR041	106	10.6	91.6	12.0	90.9
Brandelos	TOR042	150	8.12	84.8	9.21	83.7
	TOR043	125	7.15	89.9	8.40	89.1
	TOR044	106	8.47	90.5	9.68	89.8
Monte Minas Garnetite	TOR046	150	9.86	90.1	10.9	89.3
	TOR047	125	9.85	91.8	10.9	91.3
	TOR048	106	10.4	93.2	11.6	92.7
Arca A	TOR049	150	4.70	90.9	5.14	90.0
	TOR050	125	4.62	92.7	5.10	92.0
	TOR051	106	4.41	93.4	4.91	92.7
Monte Minas Upper	TOR054	150	8.40	88.6	9.62	87.8
	TOR055	125	8.95	89.9	10.2	89.3
	TOR056	106	9.29	91.1	10.6	90.6
Vieiro High Grade	TOR057	150	14.9	95.8	16.5	95.3
	TOR058	125	16.8	96.7	18.7	96.3
	TOR059	106	18.1	96.9	19.7	96.5

13.4.7.4 Cleaner Tests

Cleaner tests were completed on the main composites at the following conditions:

- Primary grind P_{80} of 150 µm.
- pH 11.
- Rougher residence time of 10 minutes.
- DSP 009 and PAX collectors added to the rougher stage at 15 g/t and 10 g/t respectively.
- Regrind P_{80} of approximately 20 µm.
- 3-stage cleaning at 10 / 7 / 4 minutes residence time.
- DSP 009 collector added to cleaner 1 and cleaner 3 stages at 7 g/t and 2 g/t respectively.
- PAX added to the first cleaner stage at 2.5 g/t.
- Frother added as required.

The results from the tests are summarized in Table 13.17 and Figure 13.7. Note that the actual regrind P_{80} achieved in these tests varied between 17 and 21 µm, due to a fixed regrind time and variable rougher concentrate mass.

All tests showed acceptable concentrate grades, in excess of 27% Cu, with the exception of Arca A. Observations made during testing indicated that the Arca A rougher concentrate appeared to have talc scum on top of chalcopyrite and that the froth appeared greyer in colour compared to the other composites.

Table 13.17 - Cleaner Tests at Target Regrind P_{80} of 20 µm

Composite	Test	Grade (Cu%)	Recovery (%)	Recovery at 27% Cu	Actual regrind P_{80} (µm)
Arca	TOR004	32.4	47.5	76.8	17.1
Vieiro	TOR009	32.9	61.3	88.9	16.6
Arinteiro	TOR010	34.0	71.0	85.9	16.6
Bama	TOR011	30.5	73.4	80.6	nd
Brandelos	TOR012	28.8	70.0	76.0	19.8
Monte Minas Garnetite	TOR013	32.0	77.6	83.2	nd
Arca A	TOR014	26.5	67.4	-	17.0
Monte Minas Upper	TOR015	28.4	69.3	72.2	19.3
Vieiro High Grade	TOR016	35.8	56.6	93.0	20.7

nd = not determined

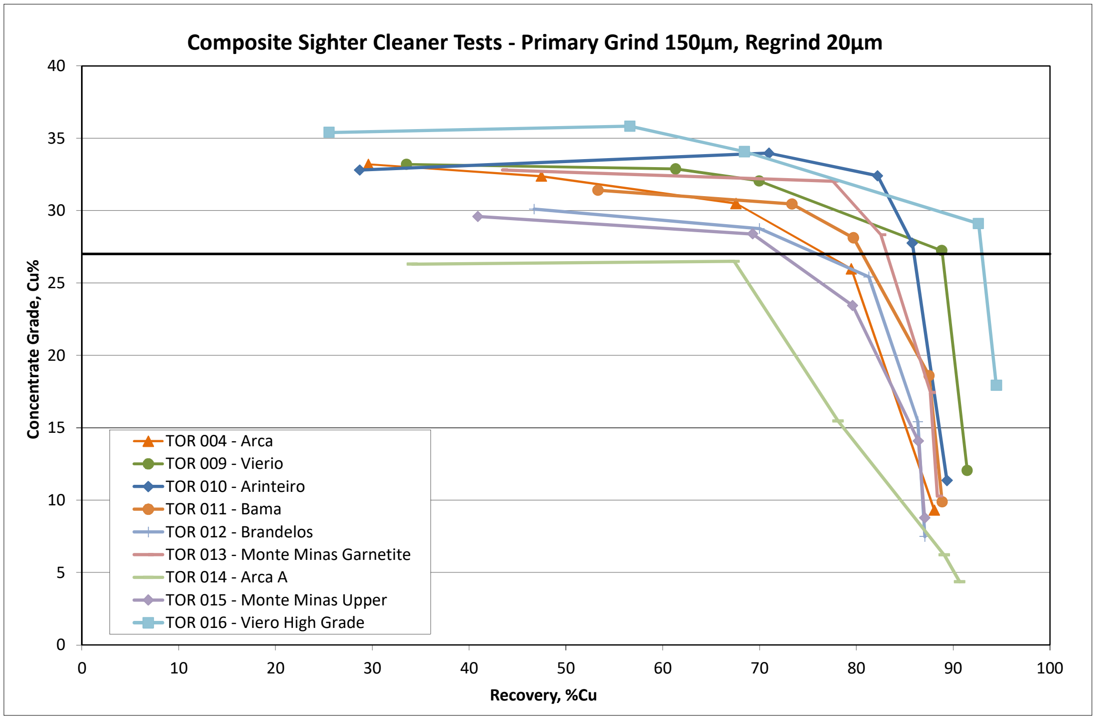


Figure 13.7 – Composite Cleaner Tests at Primary Grind of 150 µm and Regrind of 20 µm (Touro testwork analysis, Minnovo 2016)

A second group of cleaner tests were completed on the main composites at the following conditions:

- Primary grind P_{80} of 106 µm.
- pH of 11.
- Rougher residence time 10 of minutes.
- DSP 009 and PAX added to the rougher stage at 15 g/t and 10 g/t respectively.
- Regrind P_{80} of approximately 15 µm.
- 3-stage cleaning at 10 / 7 / 4 minutes residence time.
- DSP 009 added to cleaning 1 and cleaning 3 stages at 7 g/t and 2 g/t respectively.
- PAX added to the first cleaning stage at 2.5 g/t.
- Frother added as required.

These cleaner tests (TOR023 – TOR030) yielded grade-recovery curves that were fairly flat in nature, and were not consistent with the earlier results. From observations during the completion of the tests, it was evident that the cell used for the first cleaner in these tests was too small (0.5 L) and did not allow for sufficient froth to be removed as shown in Figure 13.8.

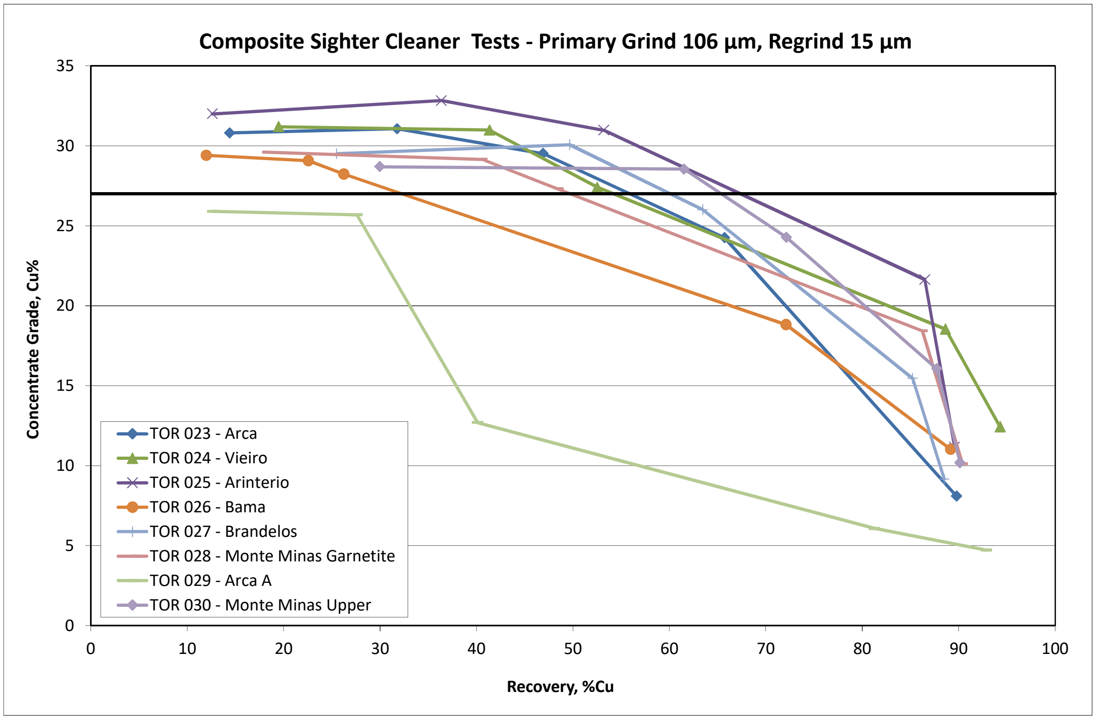


Figure 13.8 – Composite Cleaner Tests at 106 µm and 15 µm (Touro testwork analysis, Minnovo 2016)

A final group of cleaner tests were completed on the main composites (TOR071 – TOR079) using the larger cleaner 1 cell and the following conditions:

- Primary grind P_{80} of 125 µm.
- pH of 11.
- Rougher residence time of 10 minutes.
- DSP 009 and PAX added to the rougher stage at 15 g/t and 10 g/t respectively.
- Regrind P_{80} of approximately 15 µm.
- 3-stage cleaning at 10 / 7 / 4 minutes residence time.
- DSP 009 collector added to cleaning stages at 15 / 2.5 / 2.5 g/t respectively.
- Frother added as required.

The results from these tests are summarized in Table 13.18 and Figure 13.9. These cleaning tests were successful with greatly improved grades and recoveries achieved.

Arca A again showed lower final concentrate grade than the other samples, but still had good recovery.

Table 13.18 - Cleaner Tests at Target 15 µm

Composite	Test	Grade (Cu%)	Recovery (%)	Recovery at 27% Cu	Actual regrind P_{80}
Arca	TOR071	28.7	78.1	80.0	19.6
Vieiro	TOR072	32.4	87.8	91.5	16.5
Arinteiro	TOR073	31.8	83.8	88.0	18.9
Bama	TOR074	31.5	81.6	86.0	17.4
Brandelos	TOR075	32.3	78.2	83.0	19.0
Monte Minas Garnetite	TOR076	32.7	79.0	86.0	20.6
Monte Minas Upper	TOR077	31.1	80.6	86.0	nd
Arca A	TOR078	23.4	87.2	-	18.2
Vieiro High Grade	TOR079	32.0	94.5	95.5	nd

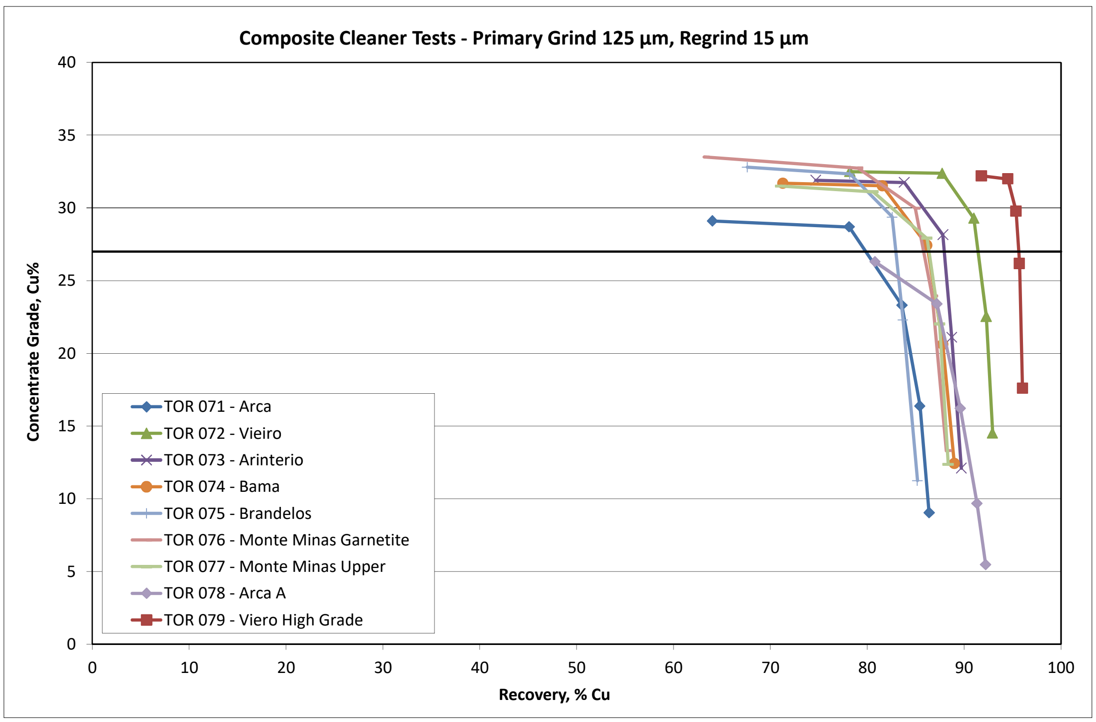


Figure 13.9 - Composite Cleaner Tests at 125 µm and 15 µm (Touro testwork analysis, Minnovo 2016)

These tests resulted in increased recoveries relative to the tests completed at the initial coarser grind conditions of P_{80} 150µm primary grind and target P_{80} 20µm regrind size. In some cases the cleaner 3 concentrate grade achieved was lower than the tests completed at 20 µm, though the overall copper yield was higher. However, it should be noted that there was no significant change in the actual regrind size achieved in the two sets of tests. Therefore the change is attributed primarily to the change in primary grind size.

13.4.7.5 Regrind Optimization

Tests were completed on the Arca composite with a primary grind P_{80} of 125 µm and regrind sizes (P_{80}) of:

- 68 µm (no regrind) - test TOR032.
- 40 µm - test TOR045.
- 30 µm - test TOR33.
- 20 µm - test TOR034.
- 10 µm - test TOR035.

Concentrate grade generally increased with decreasing regrind size, while recovery improved from a P80 of 68 µm down to 30 µm, as shown by Figure 13.10. Below this grind size, issues were encountered during the tests which impacted on recovery. This was attributed to:

- Insufficient reagent addition for the increased slurry surface area produced by the fine regrind size.
- Acid formation during regrind due to oxidation reactions that occurred when 'new' copper mineral surfaces were generated.

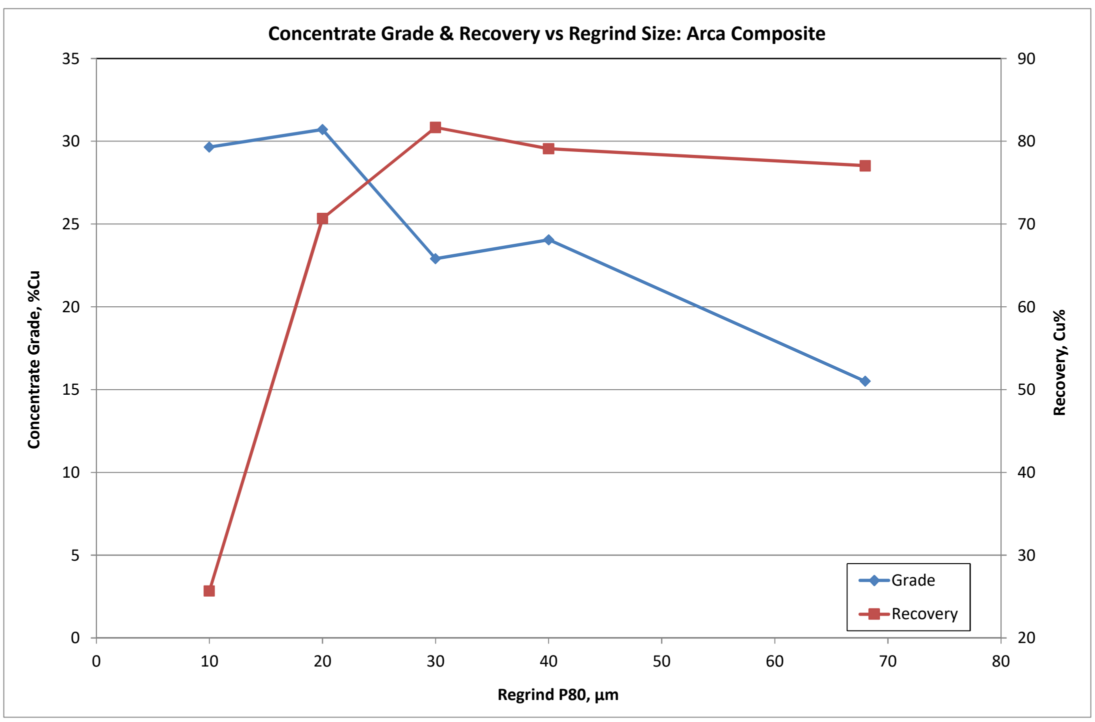


Figure 13.10 – Effect of Regrind Size (Touro testwork analysis, Minnovo 2016)

Further regrind tests were completed on Arca at regrind P_{80} between 20 µm and 11 µm (tests TOR060, 63, 64, 68). During these tests, reagent addition was adjusted to ensure that sufficient dosage occurred and lime was added to the regrind mill to limit the extent of oxidation reactions. Results are shown in Figure 13.11.

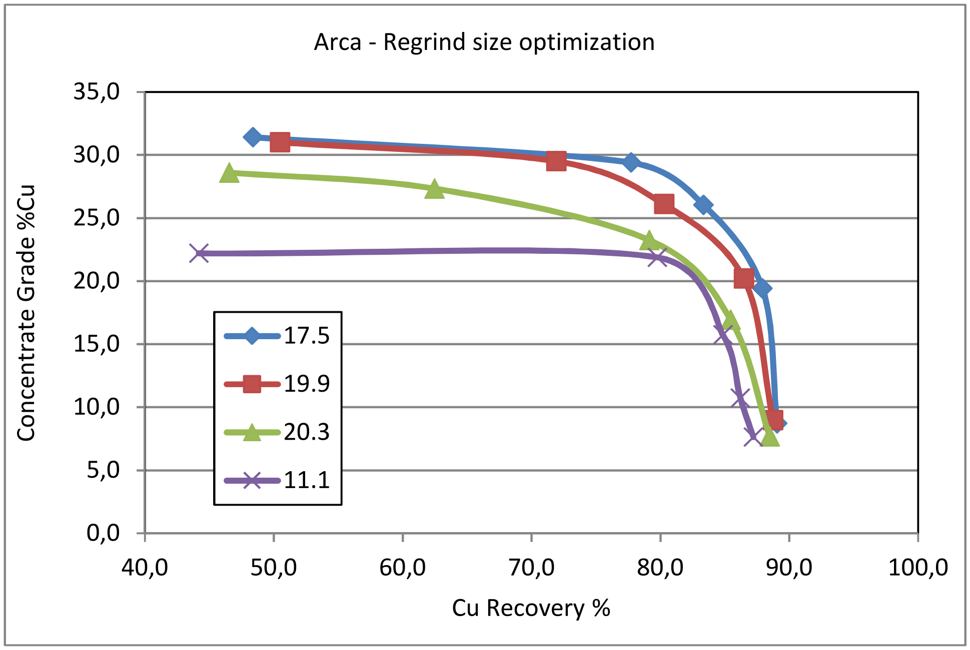


Figure 13.11 – Arca Regrind Optimization Tests (Touro testwork analysis, Minnovo 2016)

Based on these results, and the results presented in the Cleaner Tests section above, the regrind target was set at 20µm.

The outcome of the regrind optimization cleaner testwork was a standard batch test that was developed at a primary grind size P_{80} of 125 µm and regrind size of 20 µm. These conditions were tested on each of the composites as reported above in Figure 13.9.and all the variability samples, reported in the following section.

13.4.7.6 Variability Testwork

Tests were completed on 83 of the 88 variability samples selected for the testwork program. Samples that were not tested were reserved as drill-core should additional testwork be required.

Tests completed included a head assay with copper speciation and a 'standard' batch flotation test developed during the flotation composite work. The standard batch test for each variability sample was completed at the following conditions:

- Primary grind P_{80} of 125 µm.
- pH of 11.
- Rougher residence time of 10 minutes.

- DSP 009 and PAX added to the rougher stage at 15 g/t and 10 g/t respectively.
- 4 minute regrind time for a regrind P_{80} of approximately 20µm.
- 3-stage cleaning at 10 /7 / 4 minutes residence time.
- DSP 009 added to cleaning stages at 15 / 2.5 / 2.5 g/t respectively.
- Frother added as required.

Results of the sample head assays are summarized in Table 13.19 and results of the batch flotation tests are summarized in Table 13.20.

Table 13.19 - Variability Results – Head Assay

Sample ID	Ore Type	Cu_{TOT} (%)	Cu_{CNSOL} (%)	$Cu_{ACIDSOL}$ (%)	S (%)	S:Cu
12AC16-PER-1	Arca A	Sample Not Tested				
12AC11-PER-2	Arca	0.22	0.01	0	6.51	30.3
12AC28-PER-3	Arca	0.24	0.02	0	6.73	28.6
12AC27-PER-4	Arca	0.28	0.02	0	7.74	27.4
12AC33-PER-5	Arca	Sample Not Tested				
12AC19-PER-6	Arca A	Sample Not Tested				
12AC14-PER-7	Arca A	Sample Not Tested				
12AC12-PER-8	Arca	0.31	0.01	0	8.69	28.4
12AC07-PER-9	Arca	Sample Not Tested				
12AC06-PER-10	Arca	0.35	0.02	0.01	2.98	8.54
12AC13-PER-11	Arca D	0.46	0.04	0.02	10.7	23.4
12AC17-PER-12	Arca A	Sample Not Tested				
12AC08-PER-13	Arca	0.53	0.06	0.03	1.05	1.98
12AC01-PER-14	Arca D	0.55	0.04	0	9.31	17.1
12AC04-PER-15	Arca D	0.53	0.02	0	3.76	7.05
12AC23-PER-16	Arca	0.66	0.03	0	18.9	28.5
12AC29-PER-17	Arca	0.73	0.05	0	25.7	35.0
12AC14-PER-18	Arca	0.74	0.05	0	23.3	31.7
12AC05-PER-19	Arca	1.22	0.19	0.17	4.22	3.46
12AR03-PER-20	Arinteiro	0.43	0.03	0	2.91	6.83
12AR02-PER-21	Arinteiro	0.42	0.04	0	2.96	7.05
12AR04-PER-22	Arinteiro	Sample Not Tested				
12AR01-PER-23	Arinteiro	0.38	0.04	0	4.05	10.7

Sample ID	Ore Type	Cu_{TOT} (%)	Cu_{CNSOL} (%)	$Cu_{ACIDSOL}$ (%)	S (%)	S:Cu
12AR05-PER-24	Arinteiro	0.65	0.03	0.01	4.34	6.66
12AR06-PER-25	Arinteiro	0.90	0.03	0.01	4.76	5.30
12BA22-PER-26	Bama	0.21	0.01	0.01	2.07	9.67
12BA26-PER-27	Bama	0.26	0.01	0	4.00	15.6
12BA30-PER-28	Bama	0.26	0.01	0	3.70	14.1
12BA32-PER-29	Bama	0.26	0.02	0.01	4.93	18.7
12BA19-PER-30	Bama	0.26	0.01	0	2.20	8.40
12BA27-PER-31	Bama	Sample Not Tested				
12BA18-PER-32	Bama	Sample Not Tested – Quarantined Sample				
12BA21-PER-33	Bama	0.41	0.02	0.01	2.66	6.57
12BA07-PER-34	Bama	0.39	0.03	0.01	4.18	10.9
12BA15-PER-35	Bama	0.43	0.02	0.01	2.11	4.95
12BA02-PER-36	Bama	0.40	0.02	0.01	3.02	7.53
12BA08-PER-37	Bama	0.50	0.03	0.01	3.16	6.36
12BA01-PER-38	Bama	0.41	0.03	0.01	3.13	7.69
12BA12-PER-39	Bama	Sample Not Tested				
12BA11-PER-41	Bama	0.56	0.03	0.01	3.03	5.43
12BA13-PER-42	Bama	0.75	0.03	0.02	4.19	5.62
12BA05-PER-43	Bama	1.45	0.04	0.02	5.19	3.90
12BR13-PER-44	Brandelos	0.18	0.01	0	4.45	24.2
12BR07-PER-45	Brandelos	0.21	0.01	0	3.58	17.1
12BR10-PER-46	Brandelos	0.28	0.01	0	3.28	11.9
12BR01-PER-47	Brandelos	0.35	0.10	0.08	1.12	3.23
12BR03-PER-48	Brandelos	Sample Not Tested				
12BR08-PER-49	Brandelos	0.34	0.01	0	7.94	23.1
12BR06-PER-50	Brandelos	0.41	0.02	0.01	3.15	7.63
12BR11-PER-51	Brandelos	0.43	0.02	0	4.68	10.8
12BR04-PER-52	Brandelos	0.45	0.02	0.01	3.40	7.62
12BR09-PER-53	Brandelos	0.55	0.03	0	4.19	7.60
12BR12-PER-54	Brandelos	0.58	0.02	0	5.48	9.51

Sample ID	Ore Type	Cu_{TOT} (%)	Cu_{CNSOL} (%)	$Cu_{ACIDSOL}$ (%)	S (%)	S:Cu
12BR05-PER-55	Brandelos	0.56	0.02	0.01	3.70	6.65
12MM14-PER-56	MM Garnetite	0.24	0.01	0	6.01	25.3
12MM23-PER-57	MM Garnetite	0.22	0.01	0	6.69	30.5
12MM08-PER-58	MM Upper	0.29	0.02	0	2.90	9.90
12MM09-PER-59	MM Paragneiss	0.35	0.01	0	9.23	26.4
12MM32-PER-60	Monte Minas	0.34	0.01	0	9.64	28.3
12MM05-PER-61	MM Upper	0.35	0.01	0	2.21	6.26
12MM22-PER-62	MM Paragneiss	0.51	0.02	0	8.89	17.3
12MM01-PER-63	MM Garnetite	0.48	0.01	0	12.9	26.7
12MM34-PER-64	MM Upper	0.48	0.02	0.01	14.3	29.5
12MM32-PER-65	Monte Minas	0.39	0.03	0.06	1.10	2.82
12MM17-PER-66	MM Upper	0.45	0.01	0	2.21	4.90
12MM38-PER-67	MM Garnetite	Sample Not Tested				
12MM01-PER-68	MM Garnetite	0.45	0.02	0	3.67	8.25
12MM04-PER-69	MM Upper	0.54	0.02	0	3.03	5.60
12MM24-PER-70	MM Paragneiss	0.61	0.01	0	16.9	27.7
12MM21-PER-71	MM Paragneiss	0.63	0.02	0	12.5	19.8
12MM12-PER-72	MM Paragneiss	0.55	0.02	0	13.1	23.6
12MM10-PER-73	MM Garnetite	0.78	0.02	0	13.5	17.4
12MM04-PER-74	MM Upper	0.63	0.03	0	7.03	11.1
12MM02-PER-75	MM Garnetite	0.76	0.03	0	2.53	3.33
12MM09-PER-76	MM Garnetite	0.88	0.03	0.01	2.94	3.33
12VR05-PER-77	Vieiro Hardest	Sample Not Tested				
12VR03-PER-78	Vieiro	0.26	0.01	0.01	1.85	7.14
12VR09-PER-79	Vieiro	0.42	0.02	0.01	4.09	9.83
12VR10-PER-80	Vieiro	0.46	0.02	0.01	5.64	12.2
12VR02-PER-81	Vieiro	0.52	0.03	0	4.13	7.94
12VR05-PER-082	Vieiro	Sample Not Tested				
12VR17-PER-083	Vieiro	0.59	0.03	0	4.42	7.52
12VR06-PER-084	Vieiro	0.69	0.02	0.01	4.17	6.05

Sample ID	Ore Type	Cu_{TOT} (%)	Cu_{CNSOL} (%)	$Cu_{ACIDSOL}$ (%)	S (%)	S:Cu
12VR08-PER-085	Vieiro	0.82	0.02	0.01	3.87	4.71
12VR10-PER-086	Vieiro High Grade	0.81	0.03	0.01	4.58	5.68
12VR05-PER-087	Vieiro High Grade	1.13	0.05	0	6.96	6.16
12VR07-PER-088	Vieiro High Grade	1.48	0.06	0	5.30	3.58
12VR02-PER-089	Vieiro High Grade	1.79	0.07	0	5.49	3.07

Table 13.20 - Variability Results – Flotation Tests

Sample ID	Ore Type	Grade (%Cu)	Recovery (%)	Rec. at 20% Cu (%)	Rec. at 27% Cu (%)
12AC16-PER-1	Arca A	Sample Not Tested			
12AC11-PER-2	Arca	24.3	74.4	79.0	0
12AC28-PER-3	Arca	29.5	47.3	72.5	61.0
12AC27-PER-4	Arca	25.6	75.9	81	0
12AC33-PER-5	Arca	Sample Not Tested			
12AC19-PER-6	Arca A	Sample Not Tested			
12AC14-PER-7	Arca A	Sample Not Tested			
12AC12-PER-8	Arca	31.0	60.1	80	73.0
12AC07-PER-9	Arca	Sample Not Tested			
12AC06-PER-10	Arca	30.7	75.1	79.5	78.0
12AC13-PER-11	Arca D	20.4	81.9	82	0
12AC17-PER-12	Arca A	Sample Not Tested			
12AC08-PER-13	Arca	31.4	85.6	88.5	87.5
12AC01-PER-14	Arca D	25.8	81.6	83.5	82.0
12AC04-PER-15	Arca D	31.2	85.7	89.5	88.5
12AC23-PER-16	Arca	29.2	89.3	94.0	91.0
12AC29-PER-17	Arca	19.7	90.0	89.5	0
12AC14-PER-18	Arca	26.3	86.9	89.0	84.0
12AC05-PER-19	Arca	33.4	81.9	84.5	83.5
12AR03-PER-20	Arinteiro	31.6	84.5	87.0	86.0

Sample ID	Ore Type	Grade (%Cu)	Recovery (%)	Rec. at 20% Cu (%)	Rec. at 27% Cu (%)
12AR02-PER-21	Arinteiro	32.5	85.5	88.0	87.5
12AR04-PER-22	Arinteiro	Sample Not Tested			
12AR01-PER-23	Arinteiro	29.2	67.1	81	73.5
12AR05-PER-24	Arinteiro	32.7	83.6	87.5	86.0
12AR06-PER-25	Arinteiro	32.2	86.5	90.0	88.5
12BA22-PER-26	Bama	29.7	75.8	80.5	78.0
12BA26-PER-27	Bama	29.7	73.4	84.0	78.5
12BA30-PER-28	Bama	30.7	65.4	83.0	78.0
12BA32-PER-29	Bama	23.8	72.5	74.5	0
12BA19-PER-30	Bama	31.3	58.1	83.0	0
12BA27-PER-31	Bama	Sample Not Tested			
12BA18-PER-32	Bama	Sample Not Tested			
12BA21-PER-33	Bama	33.5	80.0	88.5	87.0
12BA07-PER-34	Bama	33.6	75.2	84.5	83.5
12BA15-PER-35	Bama	33.5	75.9	82.5	81.5
12BA02-PER-36	Bama	31.9	70.8	82.0	78.0
12BA08-PER-37	Bama	32.6	82.7	87	86
12BA01-PER-38	Bama	26.3	82.2	84.5	0
12BA12-PER-39	Bama	Sample Not Tested			
12BA11-PER-41	Bama	32.5	84.3	89.0	87.5
12BA13-PER-42	Bama	32.2	81.5	89.0	87.5
12BA05-PER-43	Bama	35.1	91.4	93.0	92.0
12BR13-PER-44	Brandelos	30.2	73.1	80.5	79.0
12BR07-PER-45	Brandelos	26.7	74.4	76.5	73.5
12BR10-PER-46	Brandelos	30.6	75.3	82	80.0
12BR01-PER-47	Brandelos	16.3	28.6	20	0
12BR03-PER-48	Brandelos	Sample Not Tested			
12BR08-PER-49	Brandelos	32.1	77.8	84.5	82.0
12BR06-PER-50	Brandelos	30.5	80.5	84	83.0

Sample ID	Ore Type	Grade (%Cu)	Recovery (%)	Rec. at 20% Cu (%)	Rec. at 27% Cu (%)
12BR11-PER-51	Brandelos	31.3	80.6	86	84.5
12BR04-PER-52	Brandelos	32.1	79.4	82.5	81.5
12BR09-PER-53	Brandelos	31.1	85.4	89.0	88.0
12BR12-PER-54	Brandelos	31.7	78.7	86.0	84.0
12BR05-PER-55	Brandelos	30.1	83	87.5	86.0
12MM14-PER-56	MM Garnetite	26.9	74.1	81.0	74.0
12MM23-PER-57	MM Garnetite	27.8	75.5	82.0	76.5
12MM08-PER-58	MM Upper	29.8	79.5	84.5	82
12MM09-PER-59	MM Paragneiss	25.8	86.4	88.5	81.5
12MM32-PER-60	Monte Minas	30.9	85.0	90.0	88.5
12MM05-PER-61	MM Upper	31.1	80.4	89.5	85.5
12MM22-PER-62	MM Paragneiss	31	69.6	87	83.5
12MM01-PER-63	MM Garnetite	27.9	85.7	87.0	86.0
12MM34-PER-64	MM Upper	22.4	77.1	78.0	0
12MM32-PER-65	Monte Minas	15.7	11.8	0	0
12MM17-PER-66	MM Upper	32.2	84.8	87.0	86.5
12MM38-PER-67	MM Garnetite	Sample Not Tested			
12MM01-PER-68	MM Garnetite	30.9	74.3	84.0	80.5
12MM04-PER-69	MM Upper	31.4	84.5	88.0	87
12MM24-PER-70	MM Paragneiss	21.8	88.0	89.0	0
12MM21-PER-71	MM Paragneiss	28.6	87.4	92.0	89.5
12MM12-PER-72	MM Paragneiss	29.3	83.6	85.5	84.5
12MM10-PER-73	MM Garnetite	29.5	87.7	92.0	89.5
12MM04-PER-74	MM Upper	32.3	79.6	86.5	84.5
12MM02-PER-75	MM Garnetite	32.6	82.9	89	87.5
12MM09-PER-76	MM Garnetite	33.1	89.4	92	91.5
12VR05-PER-77	Vieiro Hardest	Sample Not Tested			
12VR03-PER-78	Vieiro	30.8	82	85.5	84
12VR09-PER-79	Vieiro	31	87.3	90	89

Sample ID	Ore Type	Grade (%Cu)	Recovery (%)	Rec. at 20% Cu (%)	Rec. at 27% Cu (%)
12VR10-PER-80	Vieiro	33.1	87.3	91	90
12VR02-PER-81	Vieiro	31.3	81.6	89	86.5
12VR05-PER-082	Vieiro	Sample Not Tested			
12VR17-PER-083	Vieiro	31.5	88	91	90
12VR06-PER-084	Vieiro	33.3	86.2	93.5	92
12VR08-PER-085	Vieiro	31.3	92.1	95	94
12VR10-PER-086	Vieiro High Grade	32.7	90.9	92.5	92
12VR05-PER-087	Vieiro High Grade	32.7	70.8	95.5	94.5
12VR07-PER-088	Vieiro High Grade	33.1	92.8	95	94.5
12VR02-PER-089	Vieiro High Grade	32.1	93.8	97	96.5

A number of batch tests yielded results that were considered inconsistent and were repeated. The results were considered inconsistent for a variety of reasons including:

- Lower than expected rougher concentrate grade and recovery.
- Higher than expected rougher tailings grade and recovery.
- Lower cleaner 3 concentrate grade and recovery than expected.
- Cleaner 3 concentrate not achieving 20% Cu and/or 27% Cu.

Repeat test results are denoted in the table below in **bold**; the results from the original tests are included for comparison. In all cases the tests were repeated with rougher reagent dosage rates between 1.5 and 3 times those used in the original tests.

Table 13.21 - Variability Results – Repeat Tests

Sample ID	Ore Type	Grade (%Cu)	Recovery (%)	Rec. at 20% Cu (%)	Rec. at 27% Cu (%)
12AC13-PER-11	Arca D	18.9	81.6	0	0
		20.4	**81.9**	**82**	**0**
12AC01-PER-14	Arca D	27.0	77.8	80.0	78
		25.8	**81.6**	**83.5**	**82**
12AC29-PER-17	Arca	19.7	90.0	89.5	0
		22.8	**79.2**	**80**	**0**
12AC14-PER-18	Arca	25.0	83.4	86.0	0

Sample ID	Ore Type	Grade (%Cu)	Recovery (%)	Rec. at 20% Cu (%)	Rec. at 27% Cu (%)
		26.3	**86.9**	**89.0**	**84.0**
12AC05-PER-19	Arca	33.4	81.9	84.5	83.5
		32.3	**78.8**	**83.5**	**82.0**
12BR01-PER-47	Brandelos	16.3	28.6	20	0
		11.5	**30.4**	**0**	**0**
12MM34-PER-64	MM Upper	22.4	77.1	78.0	0
		22.7	**75.1**	**77.5**	**0**
12MM32-PER-65	Monte Minas	15.7	11.8	0	0
		16.6	**9.8**	**0**	**0**
12MM24-PER-70	MM Paragneiss	21.8	88.0	89.0	0
		23.8	**82.7**	**83.7**	**0**
12MM12-PER-72	MM Paragneiss	29.9	81.0	84.0	82.0
		29.3	**83.6**	**85.5**	**84.5**

From Table 13.21 it can be seen that a number of repeat tests yielded positive results in terms of higher grade and recoveries. Contrary to this, a number of repeat tests gave results of lower grade and recoveries.

As the regrind time for the batch tests was fixed for all tests, the regrind size achieved for each test varied as a result of variation in rougher concentrate mass pull and ore hardness. A plot of regrind size achieved vs. rougher concentrate mass pull is detailed in Figure 13.12. As expected, there is a general trend of increasing regrind size as mass pull increases. The study design point is included on the graph for reference.

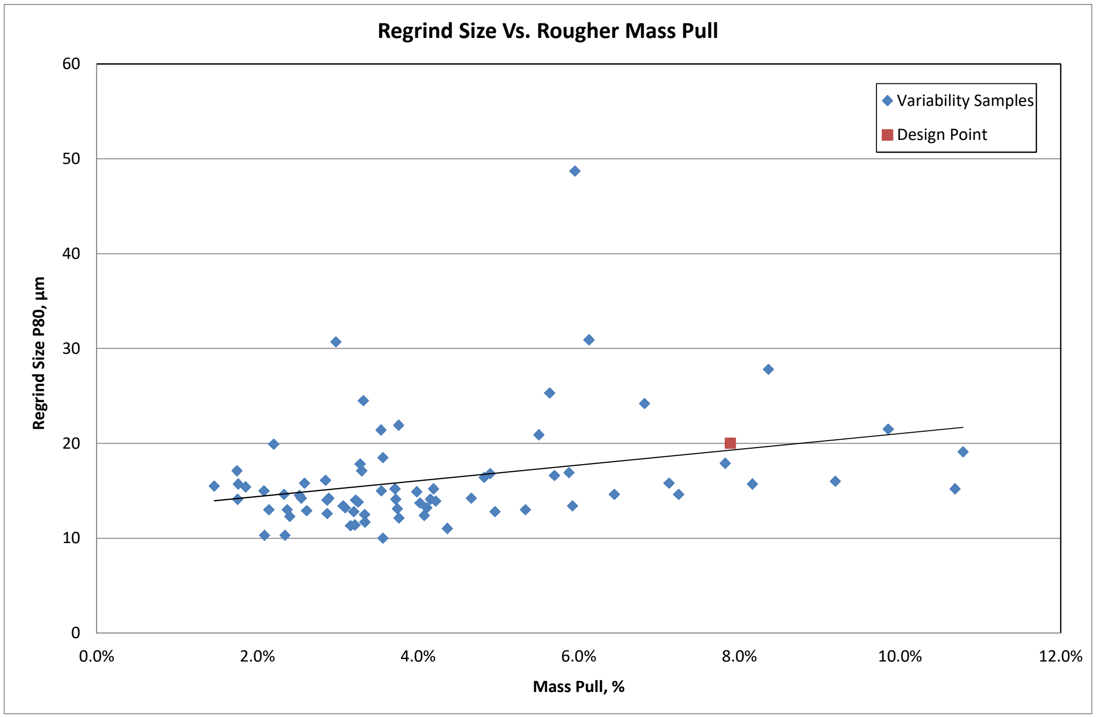


Figure 13.12 - Variability Sample Regrind Size vs. Mass Pull (Touro testwork analysis, Minnovo 2016)

A plot of regrind size achieved vs. sample head grade detailed in Figure 13.13. There is a slight trend of increasing regrind size with increasing head grade. The study design point is included for reference.

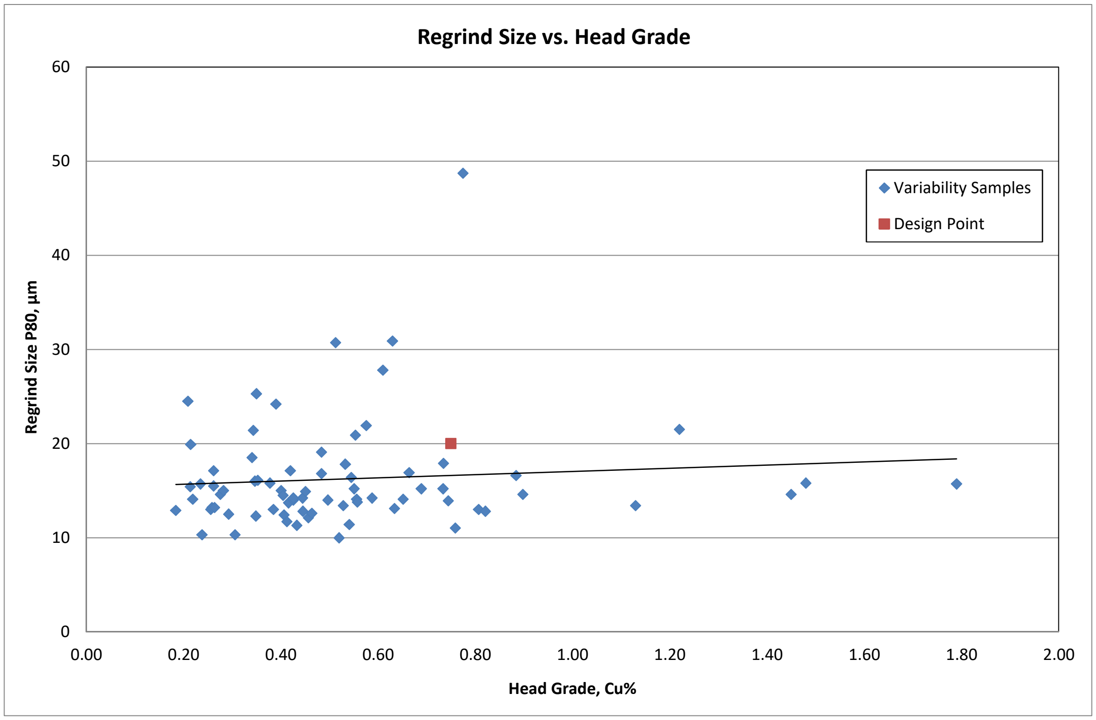


Figure 13.13 - Variability Sample Regrind Size vs. Head Grade (Touro testwork analysis, Minnovo 2016)

As shown by Figure 13.14 through Figure 13.20, the variability samples tested gave results that formed an envelope around the comparable composite result in terms of grade and recovery. This is expected as the composite sample tested was of a head grade representative of that orebody, while variability samples covered a range of head grades.

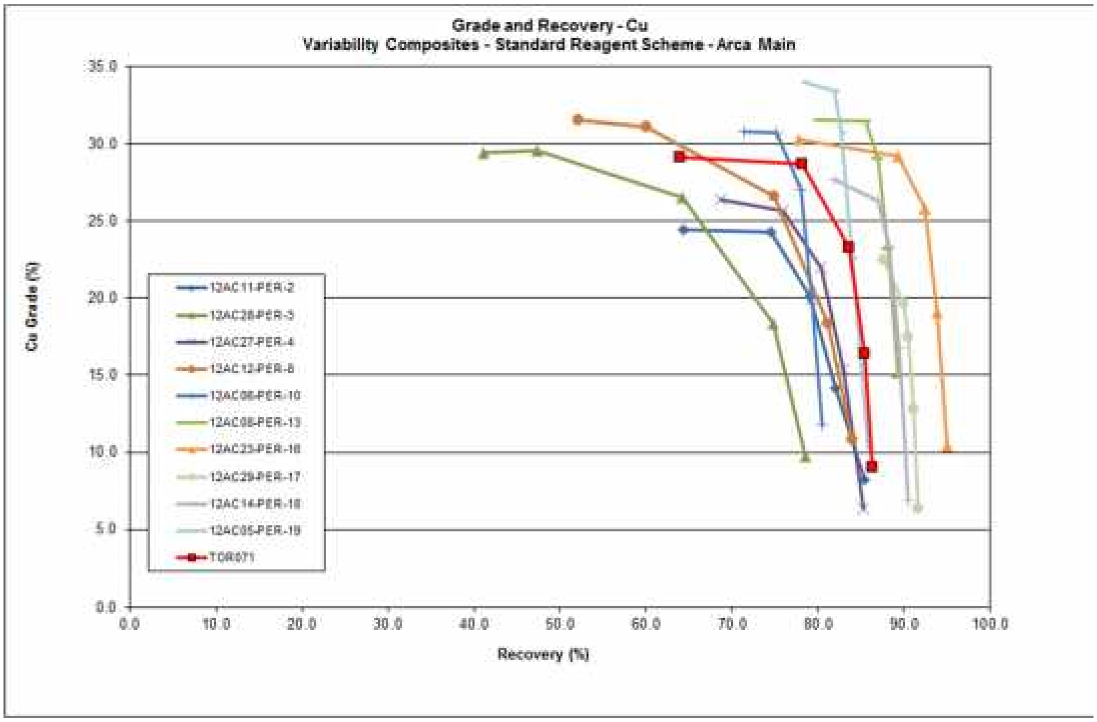


Figure 13.14 – Arca Variability Samples Grade vs. Recovery Curves (Touro testwork analysis, Minnovo 2016)

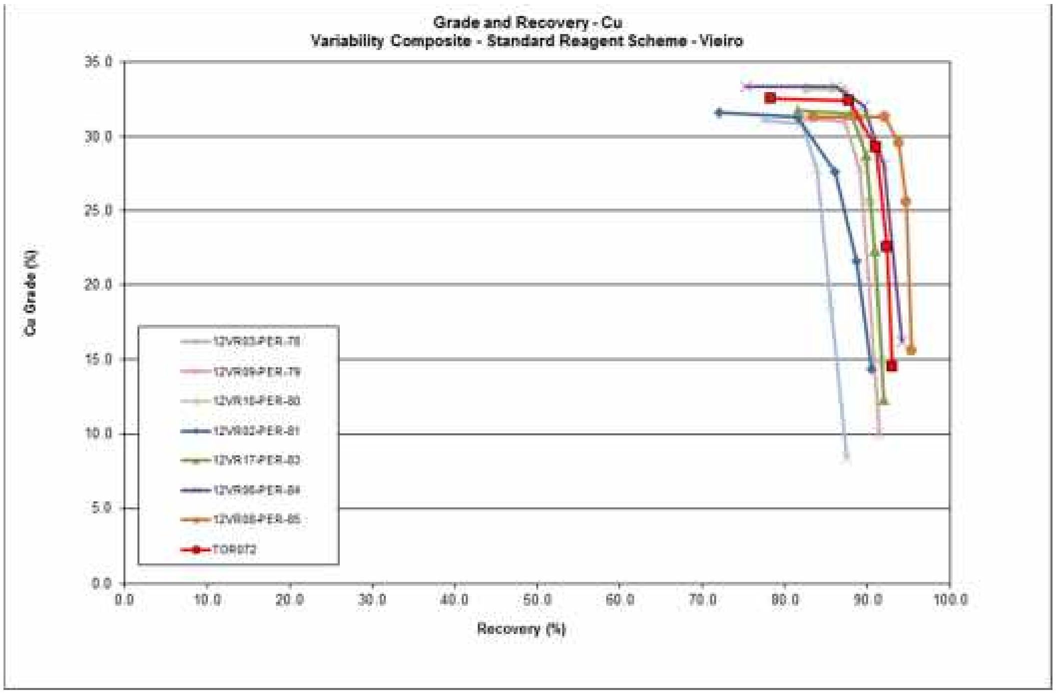


Figure 13.15 – Vieiro Variability Samples Grade vs. Recovery Curves (Touro testwork analysis, Minnovo 2016)

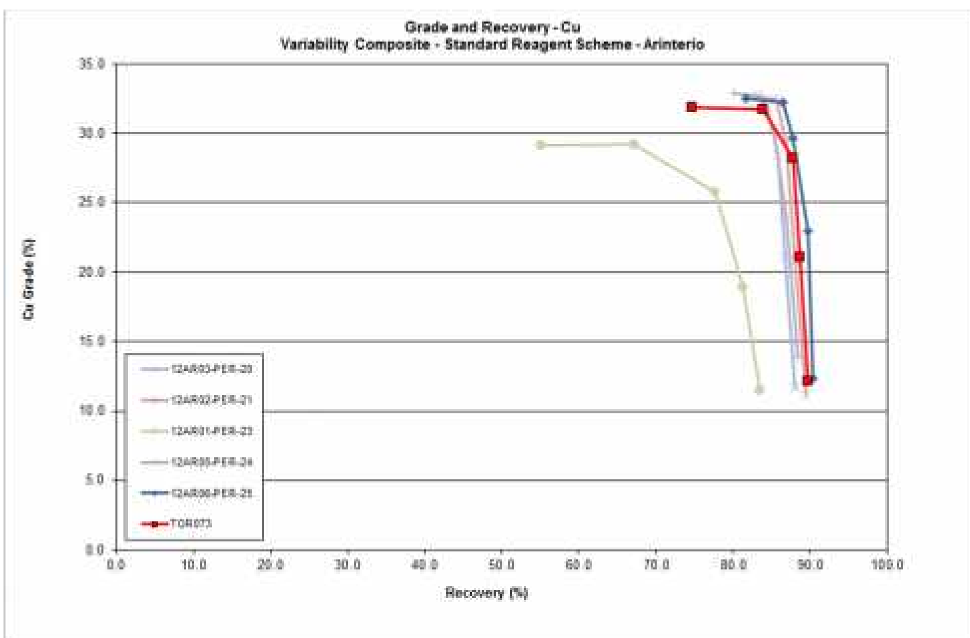


Figure 13.16 - Arinteiro Variability Samples Grade vs. Recovery Curves (Touro testwork analysis, Minnovo 2016)

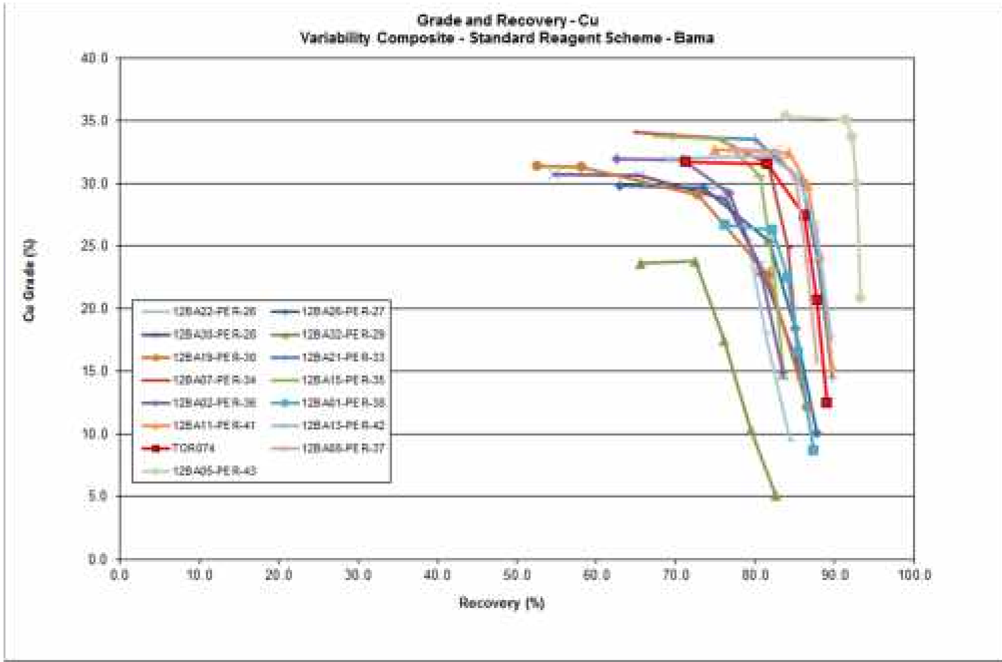


Figure 13.17 – Bama Variability Samples Grade vs. Recovery Curves (Touro testwork analysis, Minnovo 2016)

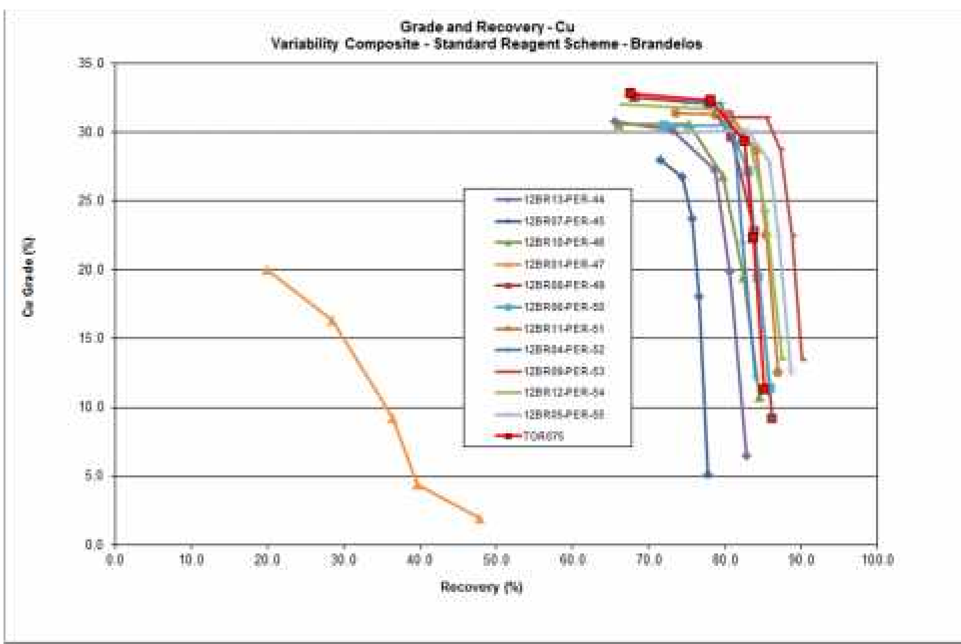


Figure 13.18 - Brandelos Variability Samples Grade vs. Recovery Curves (Touro testwork analysis, Minnovo 2016)

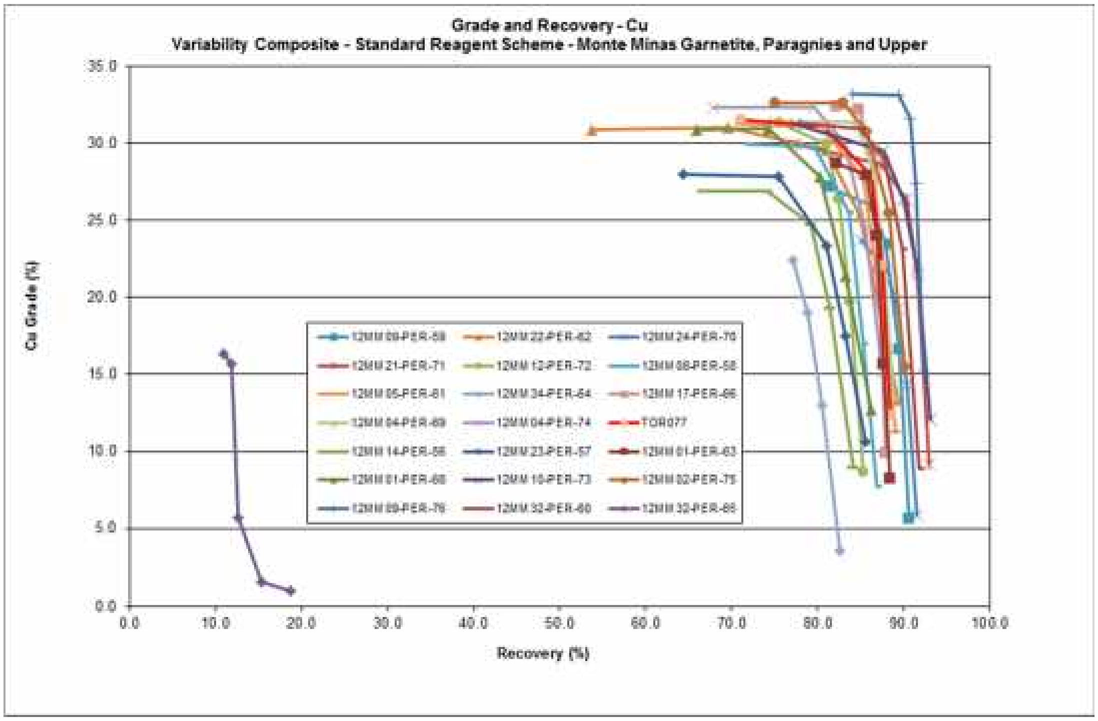


Figure 13.19 – Monte Minas Variability Samples Grade vs. Recovery Curves (Touro testwork analysis, Minnovo 2016)

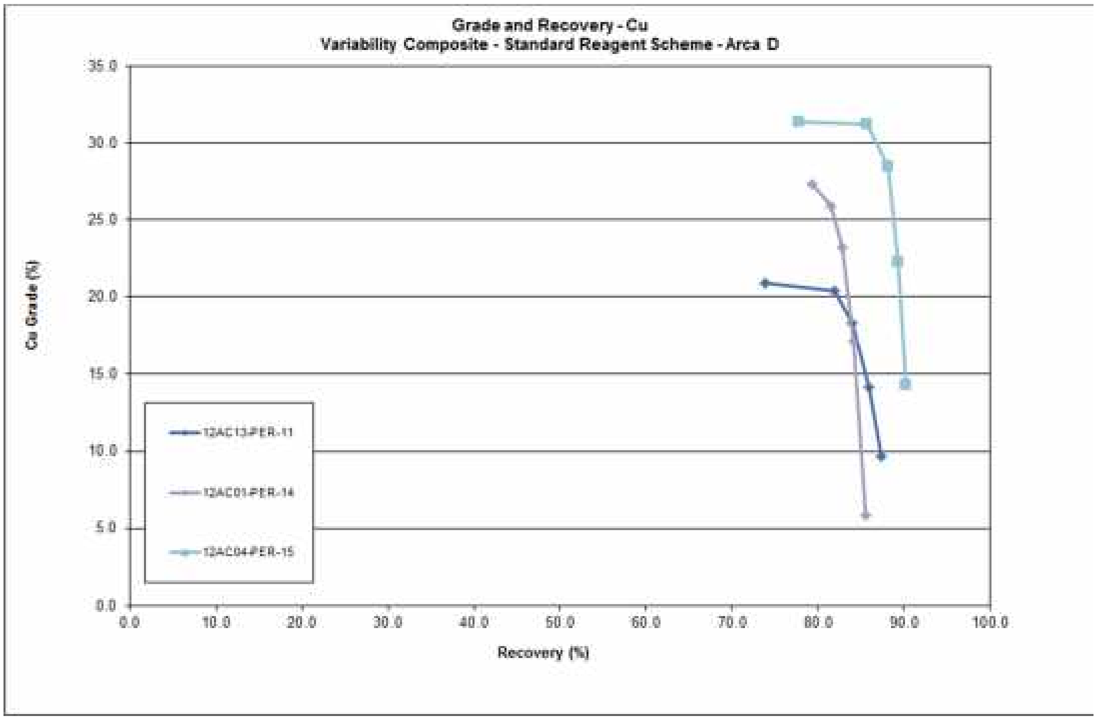


Figure 13.20 – Arca D Variability Samples Grade vs. Recovery Curves (Touro testwork analysis, Minnovo 2016)

In general, the variability testwork program was considered to be a success and did not identify any major issues. Testing of two oxide samples (12BR01-PER-47 and 12MM32-PER-65) indicated that the oxide material should be considered waste, as these gave poor concentrate grades and recoveries. All other variability samples tested (including oxidised sulphide samples) yielded grade-recovery curves that were consistent with the optimised composite cleaner tests.

13.4.7.7 Locked Cycle Tests

Locked cycle tests were completed on the Arca Main, Vieiro, Arinteiro, Bama, Brandelos, Monte Minas Garnetite and Monte Minas Upper composites, and two low-grade variability samples (12BR10-PER-46 and 12BR08-PER-49) using the flowsheet outlined in Figure 13.21.
The following conditions were used for each test:

- Primary grind P_{80} of 125 µm.
- pH of 11 for all flotation stages.
- Rougher flotation with regrinding of rougher concentrate to a P_{80} of 15 µm and rejection of rougher tailings.
- Three stages of cleaning flotation.
- Scavenging of the cleaner 1 tailings with recycle of the concentrate to cleaner 1 and rejection of cleaner-scavenger tailings.
- Recycle of cleaner 2 and 3 tailings to cleaner 1 and cleaner 2 stages respectively.
- Rougher residence time of 10 minutes.

- DSP 009 and PAX added to the rougher stage at 15 g/t and 10 g/t respectively.
- Rejection of rougher tails.
- 4 minute regrind time providing an average concentrate regrind P_{80} of 18 µm.
- Cleaner 1 residence time of 7.5 minutes.
- DSP 009 added to the first cleaner stage at 12.5 g/t.
- Cleaner-scavenger residence time of 7.5 minutes.
- DSP 009 added to the cleaner-scavenger stage at 2.5 g/t.
- Cleaner 2 residence time of 7 minutes.
- DSP 009 added to the second cleaner stage at 2.5 g/t.
- Cleaner 3 residence time 4 minutes.
- DSP 009 added to the third cleaner stage at 2.5 g/t.
- Frother added as required.

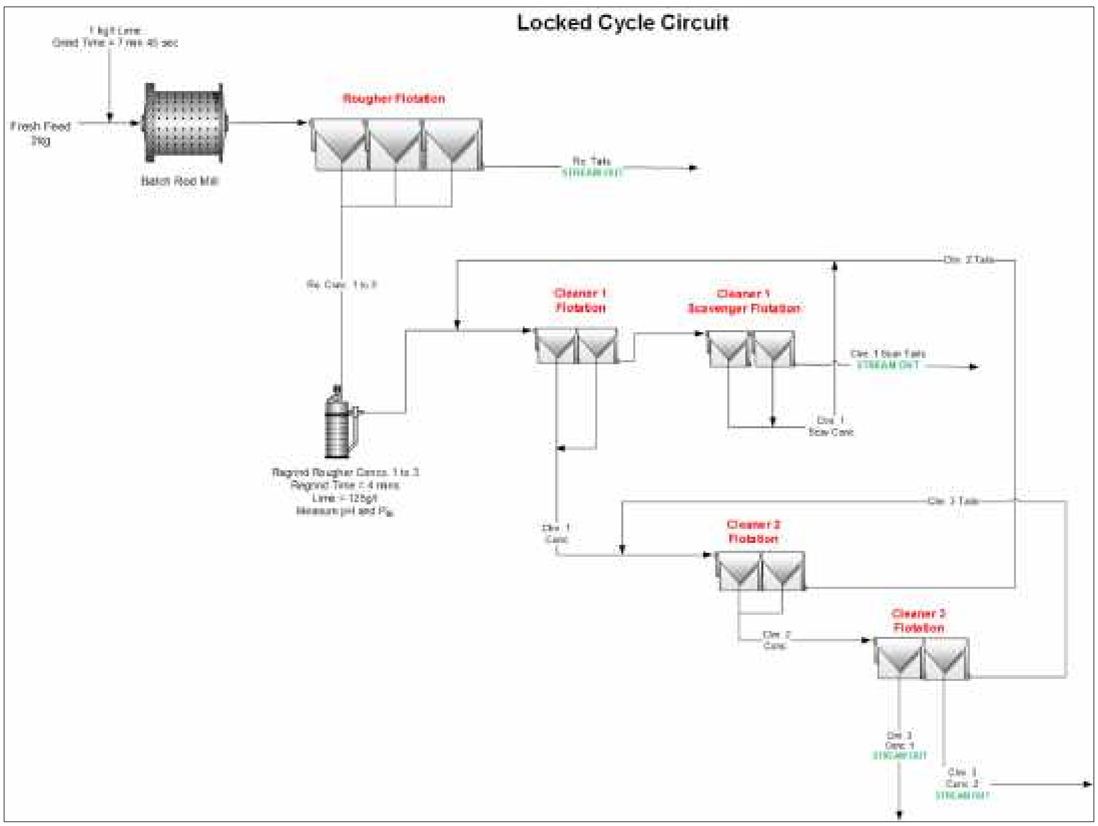


Figure 13.21 – Locked Cycle Circuit Diagram (Touro testwork analysis, Minnovo 2016)

The results of the locked cycle tests are summarized in Table 13.22. The results for the cleaner 1 concentrate recovery and cleaner 2 concentrate grade from the relevant batch tests are included for comparison.

Table 13.22 - Locked Cycle Tests

4	Locked Cycle Tests			Batch Tests	
	Head Grade (Cu%)	Conc. Grade (Cu%)	Recovery (%)	Clnr 2 Grade (Cu%)	Clnr 1 Rec (%Cu)
Arca	0.41	23.6	87.0	23.3	85.4
Vieiro	0.60	30.7	92.5	29.3	92.3
Arinteiro	0.50	31.4	89.2	28.2	88.7
Bama	0.42	30.9	86.7	27.4	87.8
Brandelos	0.40	30.9	85.0	29.4	83.7
Monte Minas Garnetite	0.48	29.5	89.0	30.0	86.7
Monte Minas Upper	0.47	28.1	87.8	27.9	87.5
12BR10-PER-46	0.28	30.7	82.9	26.8	82.4
12BR08-PER-49	0.34	29.5	83.7	29.6	83.9
Average	**0.43**	**29.5**	**87.1**	**28.0**	**86.5**

The locked cycle tests provided significantly higher recoveries at a final grade than the batch tests, demonstrating that the majority (95%) of the copper that reports to the batch test middlings can be recovered to final concentrate.

13.4.7.8 Detailed Concentrate Assays

Table 13.23 Locked Cycle Results – Concentrate Assay

Ore Type	Cu_{TOTAL} (%)	$Cu_{ACIDSOL}$ (ppm)	Cu_{CNSOL} (ppm)	Au (ppm)	Ag (ppm)	Fe (%)	S_{TOTAL} (%)
Arca	24.7	2540	5850	0.14	83	28.3	32.6
Vieiro	31.6	3410	7270	0.7	43.3	31.7	34.2
Arinteiro	30.7	2330	5980	2.06	26.8	31.7	33.6
Bama	31.1	2770	6740	0.87	31.6	30.4	34.3
Brandelos	32.1	2320	6620	0.56	37.1	29.9	33.7
MM Garnetite	29.7	2090	5890	0.7	90.5	29.2	33.2
MM Upper	27.9	3240	5260	1.5	67.3	22.5	32.3
PER-46	29.8	2760	6320	0.51	69.9	23.2	32.3
PER-49	29.6	3080	5380	0.32	77.3	23.1	34.2

13.5 Recovery Evaluation

Recovery and concentrate grade models were developed based on the results on the latest testwork program and are summarized in the following section.

Locked cycle results are generally assumed to predict plant recovery. Results from the locked cycle tests were compared with the corresponding batch tests (refer Table 13.24) and the following conclusions drawn:

- The locked cycle tests provided significantly higher recoveries to the final concentrate (Cleaner 3) than the batch tests, demonstrating that the majority of the copper that reports to the batch test middlings can be recovered to final concentrate without significant dilution of the concentrate grade.
- The locked cycle test recovery is similar to the batch cleaner 1 recovery, at 0.54% higher on average.
- The locked cycle test concentrate grade is similar to the batch cleaner 2 concentrate grade, at 1.34% higher on average.

The above relationships were used to estimate locked cycle results for 74 variability sample batch testwork results, which were then used to determine concentrate grade and recovery models from geo-chemical data available in the mine block model.

Table 13.24 - Composite Batch vs Locked Cycle Test Results

Sample	Head Grade %		Batch Cleaner Test							Locked Cycle	
			Cu Recovery %				Concentrate Grade % Cu				
	Cu	S	Rghr	Clnr 1	Clnr 2	Clnr 3	Clnr 1	Clnr 2	Clnr 3	%Rec	% Cu
Arca	0.41	10.3	86.4	85.4	83.6	78.1	16.4	23.3	28.7	87.0	23.6
Vieiro	0.60	4.80	92.9	92.3	91.0	87.8	22.5	29.3	32.4	92.5	30.7
Arinteiro	0.50	4.47	89.7	88.7	87.8	83.8	21.1	28.2	31.8	89.2	31.4
Bama	0.42	3.48	89.0	87.8	86.2	81.6	20.6	27.4	31.5	86.7	30.9
Brandelos	0.40	4.66	85.2	83.7	82.6	78.2	22.3	29.4	32.3	85.0	30.9
MM Garnetite	0.48	8.51	88.2	86.7	84.9	79.0	23.9	30.0	32.7	89.0	29.5
Viero High Grade	1.27	5.63	96.0	95.7	95.4	94.5	26.2	29.8	32.0	-	-
MM Upper	0.47	5.67	88.4	87.5	86.2	80.6	22.0	27.9	31.1	87.8	28.1
12BR10-PER-46	0.28	3.28	84.5	82.4	79.7	75.3	19.5	26.8	30.6	82.9	30.7
12BR08-PER-49	0.34	7.94	86.2	83.9	80.9	77.8	22.8	29.6	32.1	83.7	29.5
Average	**0.52**	**5.87**	**88.7**	**87.4**	**85.8**	**81.7**	**21.7**	**28.2**	**31.5**	**87.9**	**29.5**

13.5.1 Concentrate Grade Model

Evaluation of the variability data concluded:

- The concentrate grade is moderately correlated with S:Cu ratio and weakly correlated with copper head grade.
- Apart from ore classified as oxide, all other ore types behave similarly.

A single equation based on copper head grade and S:Cu ratio provided the best prediction of concentrate grade (up to a cap of 33% Cu) for all ore types except oxide:

$$C \qquad G \quad = 30.1 + 4.93 \times H \quad G \quad - 0.29 \times \left(\frac{S}{C}\right)$$

This is represented graphically inFigure *13.22* Figure 13.22

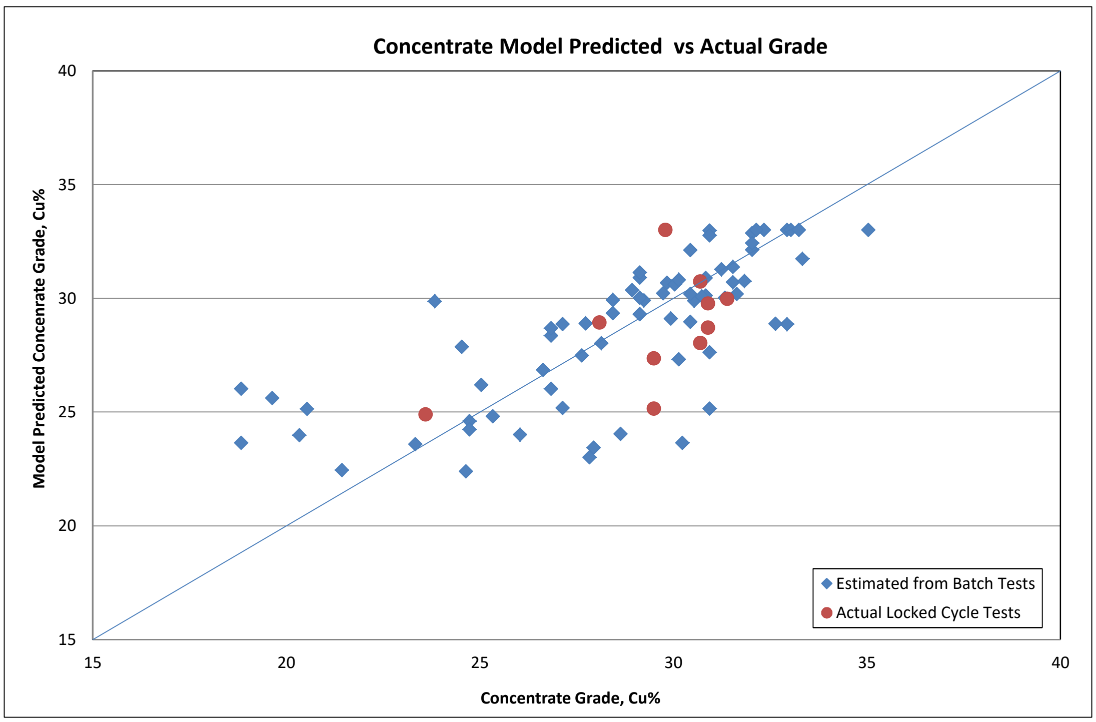


Figure 13.22 - Model Predicted Grade vs Actual Grade (Touro testwork analysis, Minnovo 2016)

13.5.2 Recovery Models

Figure 13.23 shows the complete data set of estimated locked cycle recovery verses head grade. It shows the higher recovery of the Vierio deposit compared to the other primary ore types.

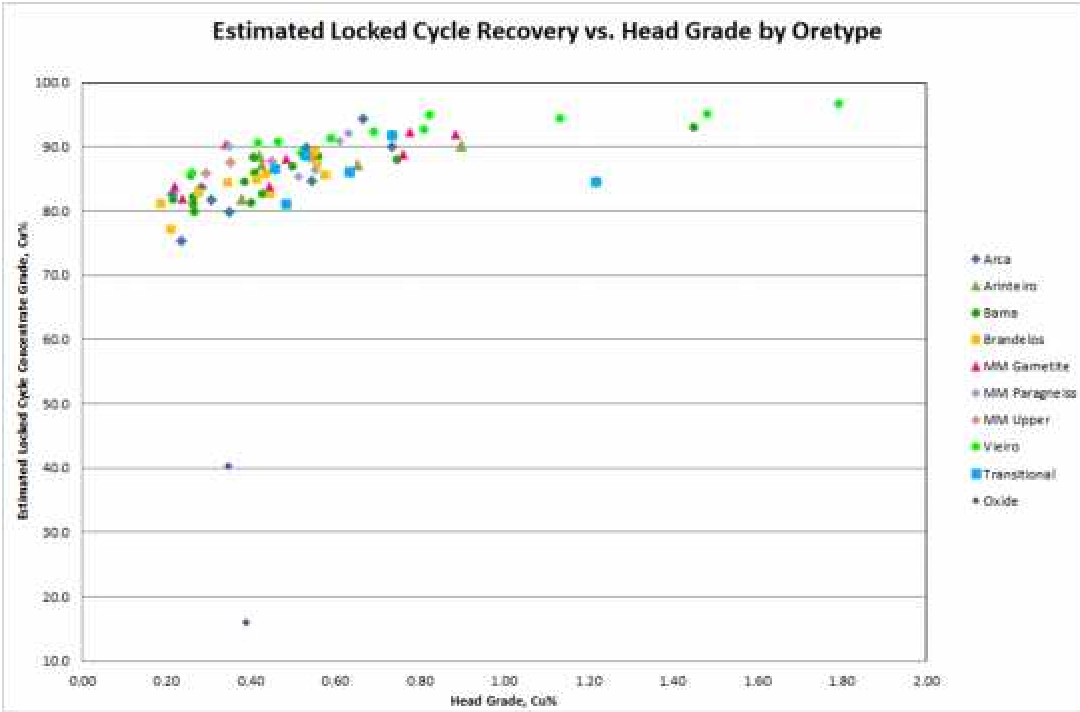


Figure 13.23 – Estimated Locked Cycle Recovery vs Head Grade (Touro testwork analysis, Minnovo 2016)

Evaluation of the variability data concluded:

- Copper recovery is strongly correlated with copper head grade.
- Vieiro ore consistently provided better recovery than the other ore types.
- Transitional ore provided a slightly lower recovery than primary ore. Recovery is affected by the amount of oxidation, as measured by either acid or cyanide soluble copper content. This data is not available in the block model so can't be used as a predictor. Given transitional ore is only a small portion of the overall reserve, the results for the 6 transitional samples have been averaged to provide a separate recovery estimate for this ore type, using a fixed tail grade recovery model.
- Oxide ore provided very low recovery and given the low concentrate grades achieved with the oxide samples, oxide ore has been classified as waste.
- Considering the typical scatter with this type of testwork, all other primary ore provided similar results.

Three separate recovery models were determined based on this evaluation:

- **Oxide:** Considered waste as recoveries are very low and a saleable concentrate grade can't be achieved based on the 2 oxide tests conducted.

- **Transitional Ore:** Fixed tail grade of 0.094% Cu based on the average of the 6 transitional ore samples.
- **Vieiro:** Logarithmic model using copper head grade based on the 11 Vieiro primary ore samples.
- **Other Primary Ore:** Logarithmic model using copper head grade based on the 55 other primary variability samples.

These models are shown together with the testwork recovery data in Figure 13.24 below.

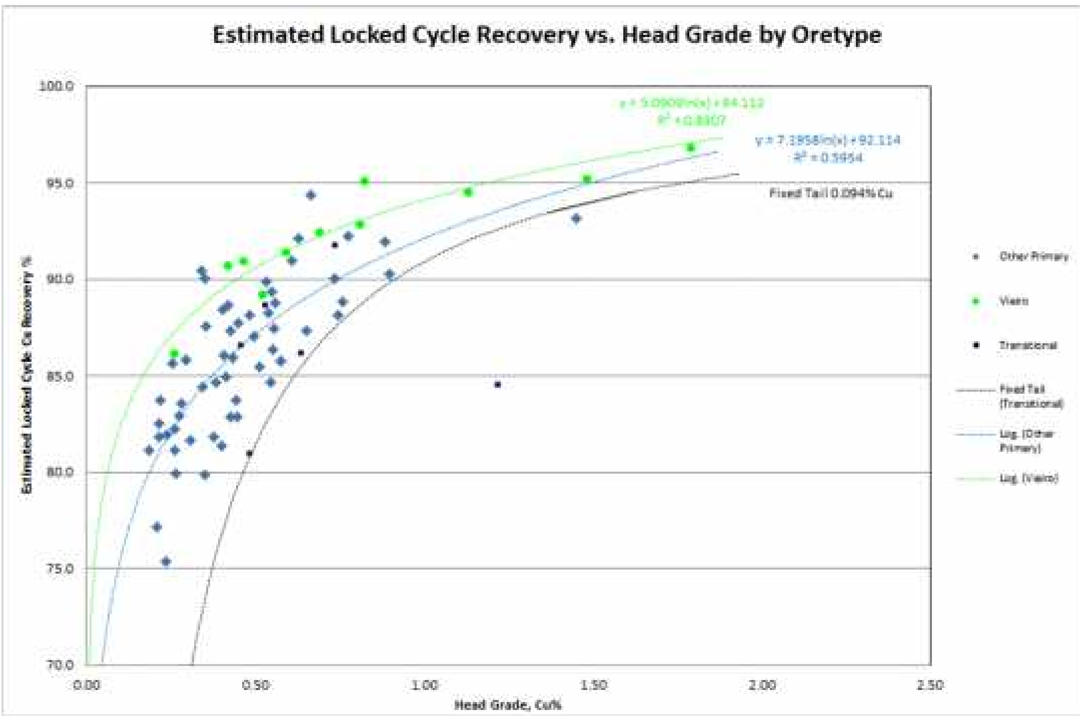


Figure 13.24 - Recovery Models (Touro testwork analysis, Minnovo 2016)

The recovery estimates derived from the variability sample test results have been validated by back calculation of the recovery, via the recovery formula, for each ore deposit composite. This is then compared back to the actual composite LC result. The validation data are shown in Table 13.25.

Table 13.25 - Locked Cycle vs. Model Predicted Recovery

Ore Type	Locked Cycle Recovery (%)	Model Predicted Recovery (%)	Delta (%)	Corrected Model Recovery (%)	Delta (%)
Arca	87.0	85.7	-1.3	86.5	-0.5
Vieiro	92.5	91.5	-1.0	92.3	-0.2
Arinteiro	89.2	87.1	-2.1	87.9	-1.3
Bama	86.7	85.9	-0.8	86.7	0.0
Brandelos	85.0	85.5	0.5	86.3	1.3
MM Garnetite	89.0	86.8	-2.2	87.6	-1.4
Viero High Grade	95.7	95.3	-0.4	96.1	0.4
MM Upper	87.8	86.7	-1.1	87.5	-0.3
12BR10-PER-46	82.9	82.8	-0.1	83.6	0.7
12BR08-PER-49	83.7	84.4	0.7	85.2	1.5
Cu weighted Average	**89.6**	**88.8**	**-0.8**	**89.6**	**0.0**

Table 13.25 shows that the original recovery equations have an average under-call of 0.8%, and the offset is relatively evenly distributed across all ore types. Therefore, when the original equations are corrected, by adding 0.8 to the constant in the formula, this difference is reduced to zero. This represents the most accurate estimate for recovery, being true to the composite locked cycle test results and the overall copper weighted deposit average, while providing correction for the relevant ore variability parameters (head grade and S:Cu ratio).

The final corrected equations for determining the plant recovery for each ore domain are:

$$V \quad P \quad O \quad R = 5.09\, x \ln(H\ ) + 94.9$$

$$A \quad O\ he \quad P \quad O \quad R = 7.196 \times \ln(H\ ) + 92.9$$

$$T \quad O \quad R = \frac{100 \times C \quad \times (H \quad - 0.094)}{H \quad \times (C \quad - 0.094)}$$

Where:

- CG = Concentrate Grade (Cu%).
- HG = Head Grade (Cu%).

The results of these models are shown in Figure 13.25, where the tight fit of predicted and actual recovery for the locked cycle composite tests can be seen.

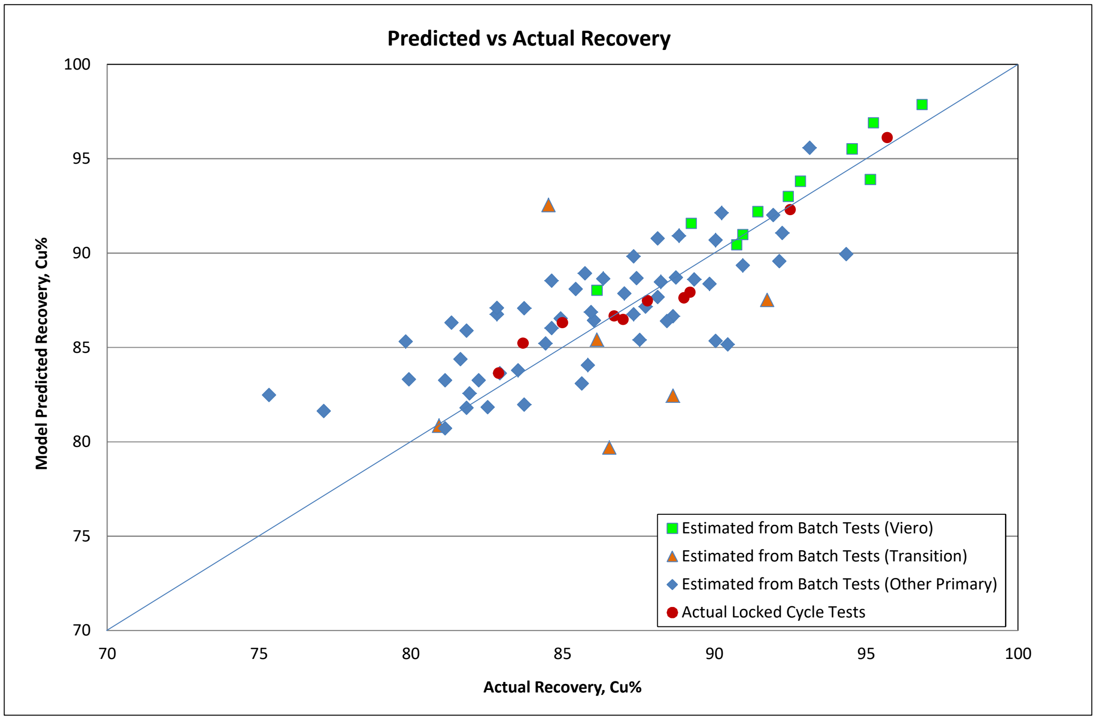


Figure 13.25 – Model Predicted Grade vs Actual Recovery (Touro testwork analysis, Minnovo 2016)

13.6 Deleterious Elements

The concentrate assays for the locked cycle tests reported low levels of deleterious elements such as arsenic, bismuth, lead and tellurium. The average level of deleterious and penalty elements is summarized in Table 13.26 with the associated penalty limits and maximum limits for a US smelter. The average levels of gold and silver recovered to final concentrate were 93 ppm and 54.2 g/t respectively, of which only silver provides minimal economic value.

The only elements that may lead to penalty issues with smelters are aluminium (Al), chlorine (Cl), antimony (Sb) and zinc (Zn). Typical penalty rates for these elements are US$ 0.50 for each 0.1% of Al and US$ 0.50 for each 0.1% of zinc. Antimony and chloride are undesirable at any level and specific penalties may apply depending on the smelter.

Table 13.26 - Penalty Elements – Concentrate Assay

Assay	Concentrate Average (ppm)	Penalty Limit (ppm)	Maximum Limit (ppm)
Al	**4,311**	**1,000**	**50,000**
As	15.3	1,000	2,000
Ba	9	5,000	10,000
Be	<0.1		<10
Bi	4.8	1,000	4,000
Cd	45.44	1,000	4,000
Cl	**71.7**		**<10**
Co	105		5,000
Cr	30.5	1,000	30,000
F	96.7	1,000	5,000
Hg	0.27	10	10
Mg	2,901		100,000
Mn	244		20,000
Na	560		50,000
Ni	113	1,000	20,000
P	33		30,000
Pb	157	1,000	10,000
Sb	**3.34**		**<10**
Se	111		1,000
Sn	8.02	1,000	30,000
Te	6.10		100
Tl	0.38		100
Zn	**16,892**	**1,000**	**40,000**

13.7 Preliminary AMD Testwork

13.7.1 Background

Preliminary Acid Metalliferous Drainage (AMD) refers to the outflow of acidic water from mines, waste rock storage facilities, and Tailings Storage Facilities (TSF). Although a host of chemical processes contribute to AMD, pyrite oxidation is by far the greatest contributor; this is relevant to the Touro deposits given the relative abundance of pyrite in the ore and waste. The general equation for this process is:

$$2\,F\ \ S_2(s) + 7O_2(g) + 2H_2O\,(l) = 2Fe^{2+}(a\ \ ) + 4SO_4^{2-}(a\ \ ) + 4H^+(a\ \ )$$

13.7.2 Testwork

The material tested was rougher flotation tailings from tests on the Arca and Vieiro composites. The preliminary tests completed included:

- Total sulphur measurement and Maximum Potential Acidity (MPA).
- HCL Extractable S, Ca and Mg.
- Acid Neutralising Capacity or Neutralisation Potential (ANC/NP).
- Net Acid Generation Potential (NAGP).
- Single Addition Net Acid Generation (NAG).

The results of these preliminary tests indicated that both samples are both NAGP positive, and that neutralisation of outflow water from pits, waste rock storage facilities, and the TSF would be required. More detailed testwork was completed by AGQ in Spain.

14 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

This resource estimate was prepared by Alan C. Noble, P.E. of Ore Reserves Engineering, Lakewood, CO, USA. Mr. Noble is a qualified person for resource estimation based on having received a B.S. Degree in Mining Engineering from the Colorado School of Mines, registration as a Professional Engineer in the State of Colorado, USA, and over 47 years of experience with resource estimation on over 156 mineral deposits throughout the world. Mr. Noble is independent of Atalaya Mining and Proyecto Touro using all the tests of NI 43-101.

The compilation and verification of the geological and drilling data for resource estimation was prepared by Monica Barrero Bouza, EuroGeol. Ms. Barrero Bouza is a qualified person based on having received a BS Degree in Geology from the University of Oviedo (Spain), registered member of the Official Association of Professional Geologist of Spain (ICOG) and registered Eurogeologist, and 20 years of diverse experience in the geology of precious and base metal projects. Ms. Barrero Bouza is independent of Atalaya Mining and Proyecto Touro using all the tests of NI 43-101.

14.1 Resource Block Model

The resource model was created as a three-dimensional block model using Datamine Studio 3 software. The model block size is 10x10x10 meters, which is consistent with the proposed mining bench height and the estimated selective mining unit. The horizontal extent of the model is defined to cover the entire resource and ownership area, plus sufficient space outside the deposits to cover the ultimate pit. Resource model size and location parameters are shown in Table 14.1.

Table 14.1 - Resource Model Size and Location Parameters

	Minimum (ETRS meters)	Maximum (ETRS meters)	Cell Size (meters)	Number Cells	Model Size (meters)
Easting (X)	551,700	556,100	10	440	4,400
Northing (Y)	4,746,600	4,749,950	10	335	3,350
Elevation(Z)	-100	500	10	60	600

Key items included in the block model are copper grade, copper grade zone, density, trend-indexed Z coordinates, oxidation codes, resource classification codes, and codes indicating whether a block is mined out, backfill, or rock. Copper grade was estimated using inverse-distance-power estimation.

14.2 Drill Hole Sample Database

14.2.1 Database Content

The drill-hole data was provided by Atalaya and Touro Project engineering and geology personnel in Excel and ASCII formatted files containing assays, collar locations, down-hole surveys, and geologic logging for all drilling in the resource area. Drilling is a combination of three periods of drilling: Legacy diamond drill data from the 1970’s and 1980’s by Rio Tinto Minera SA and Rio Tinto Patiño; diamond drill data from 2012 by Lundin Mining, and reverse circulation and diamond drilling by Atalaya in 2015 and 2016.

These data were reviewed extensively by Monica Barrero Bouza and Alan Noble to confirm that they were suitable for resource estimation. A remaining issue with the assay data, particularly the legacy data, is that long intervals of core were not sampled or assayed when the core was recognized visually as being devoid of mineralization. Those intervals were set to a low default value (0.01101) that is representative of unmineralized core and is also easily recognized in the data. Also, the legacy data was not assayed for iron and sulfur, so those elements could not be reliably modeled.

After data review and during interpretation of the drilling, a total of 90 drill holes with 7,272.8m length were removed because they were insufficiently assayed or there were questions about collar location.

Drilling used for this resource estimate is summarized by company in Table 14.2 and by deposit area in Table 14.3. Drill hole collar locations are shown in **¡Error! No se encuentra el origen de la referencia.** along with the resource model limits, the ownership boundary, and deposit boundaries.

Table 14.2 - Summary of Drilling by Company

Company	Number Holes	Hole Length (m)			Average Length (m)
		Total	Diamond Drill	Reverse Circulation	
Legacy (RTP-RTM)	587	53,882.74	53,882.92	-	91.8
Lundin	139	15,583.60	15,583.60	-	112.1
Atalaya - Total	228	23,122.10	3,935.10	19,187.00	101.4
DD	3	380.00	380.00	-	126.7
RC	177	15,841.00	-	15,841.00	89.5
RC+DD	41	5,804.40	2,458.40	3,346.00	141.6
Geotech	7	1,096.70	1,096.70	-	156.7
Total Drilling	954	92,588.44	73,401.62	19,187.00	97.1

Table 14.3 - Summary of Drilling by Deposit Area

Deposit Area	Number Holes	Hole Length (m)			Average Length (m)
		Total	DD	RC	
Arca	267	19,914.42	14,207.42	5,707.00	74.6
Bama	231	20,848.46	18,920.64	1,928.00	90.3
Brandelos	98	6,168.22	5,194.22	974.00	62.9
Monteminas	173	18,618.58	13,050.58	5,568.00	107.6
Arinteiro	64	6,987.80	4,130.80	2,857.00	109.2
Vieiro	121	20,050.96	17,897.96	2,153.00	165.7
Total	954	92,588.44	73,401.62	19,187.00	97.1

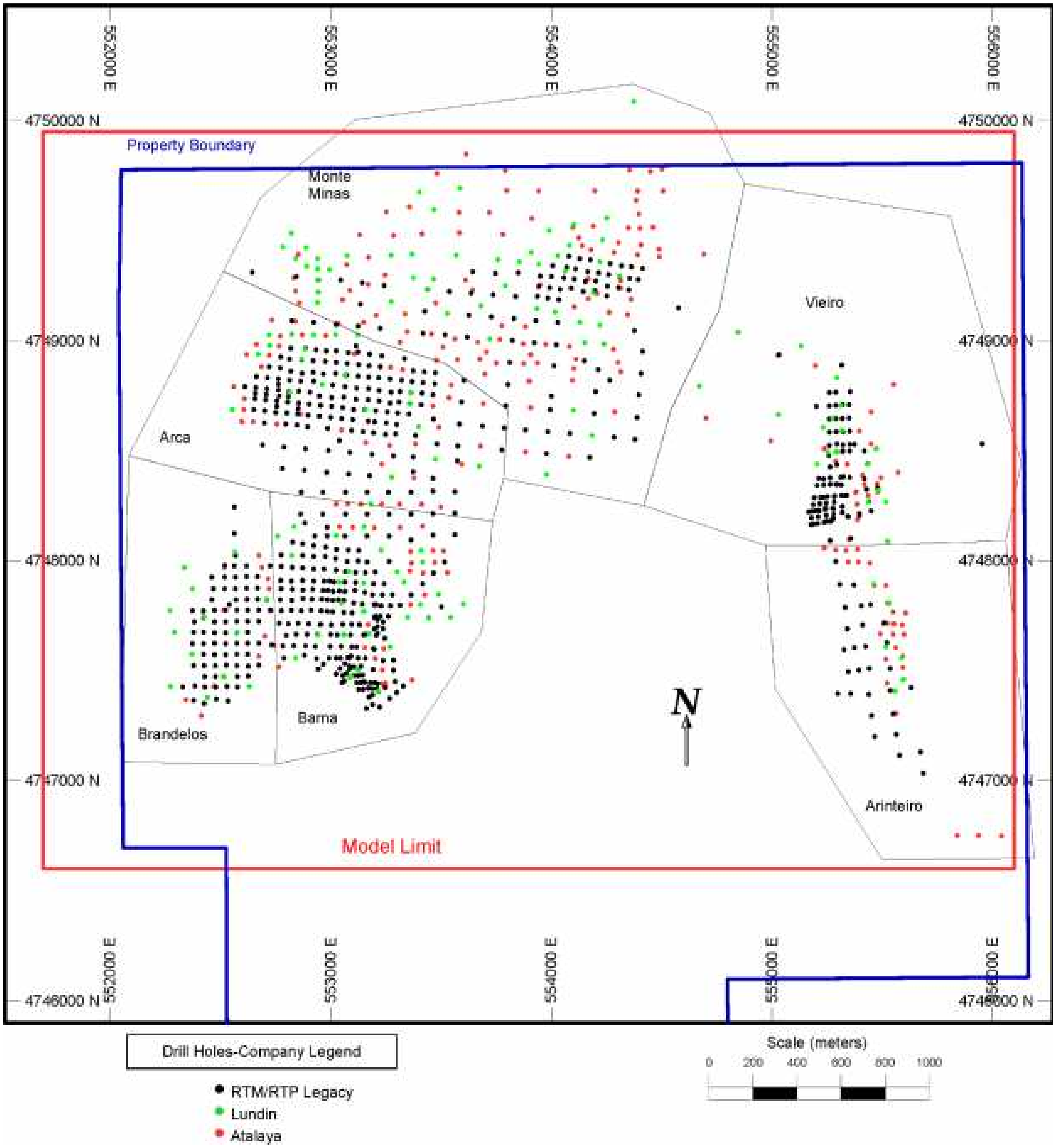


Figure 14.1 - Plan Map Showing Drill Collar Locations, Resource Model Limit, Property Boundary, and Deposit Areas (Noble 2017)

14.3 Bulk Density

14.3.1 Density Studies

Bulk density is estimated using a formula correlating density and either iron or copper grade within generalized rock type group. The Lundin density data were joined to the assay and lithologic data for regression analysis. This study demonstrated that the generalized rock type (Amphibolite, Paragneiss, or Other) along with iron grade provided usable formulas for estimating density. When an iron assay was not available, copper grade was used to estimate density.

The regression equation densities were reduced by 2% for fresh rock and by 10% for oxidized and other rocks to compensate for fracturing and other void space. The resulting density formulas are summarized Table 14.4. The correlations between density and iron/copper are shown in **¡Error! No se encuentra el origen de la referencia.** through **¡Error! No se encuentra el origen de la referencia.**.

Table 14.4 - Density Formulas by Lithologic Group

Amphibolite:		
	With Fe Assay:	Density = 2.781 + 0.0235 x %Fe
	Only Cu Assay:	Density = 2.984 + 0.3748 x %Cu
	No Assay	Density = 2.95
Paragneiss:		
	With Fe Assay:	Density = 2.585 + 0.0299 x %Fe
	Only Cu Assay:	Density = 2.728 + 1.0348 x %Cu
	No Assay	Density = 2.74
Other (Oxidized Rock, Fault Zone, Quartz):		
	With Cu & Fe Assay:	Density = 2.227 + 0.0319 x %Fe
	Only Cu Assay:	Density = 2.295 + 1.6344 x %Cu
	No Assay	Density = 2.30

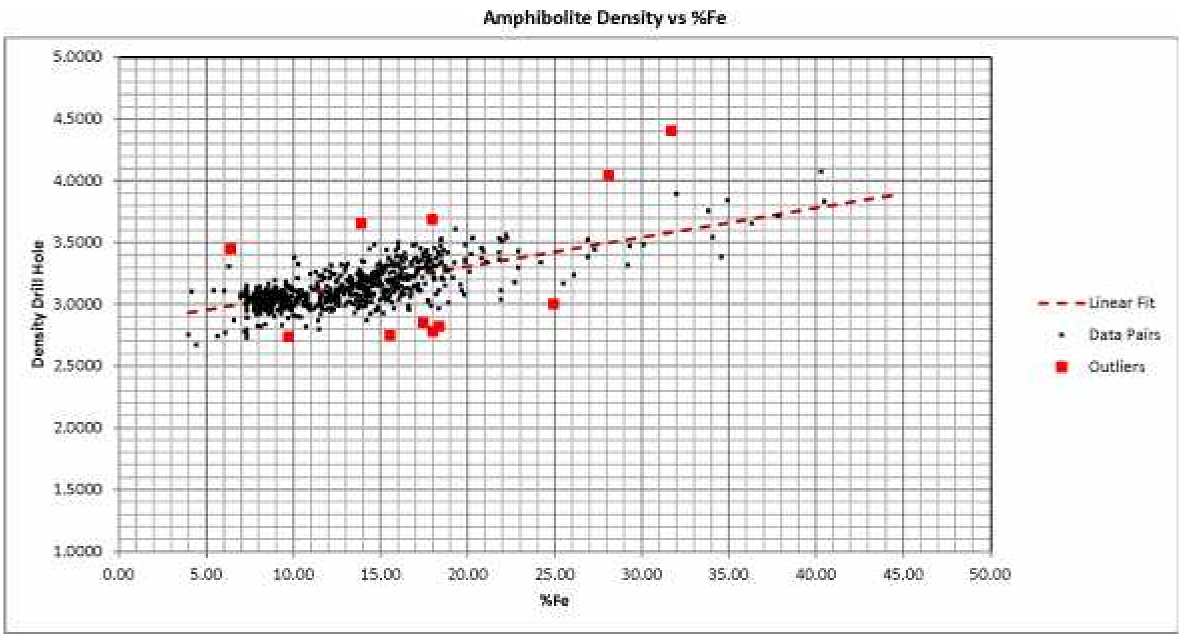


Figure 14.2 - Correlation between Density and Iron for Amphibolite (Noble 2017)

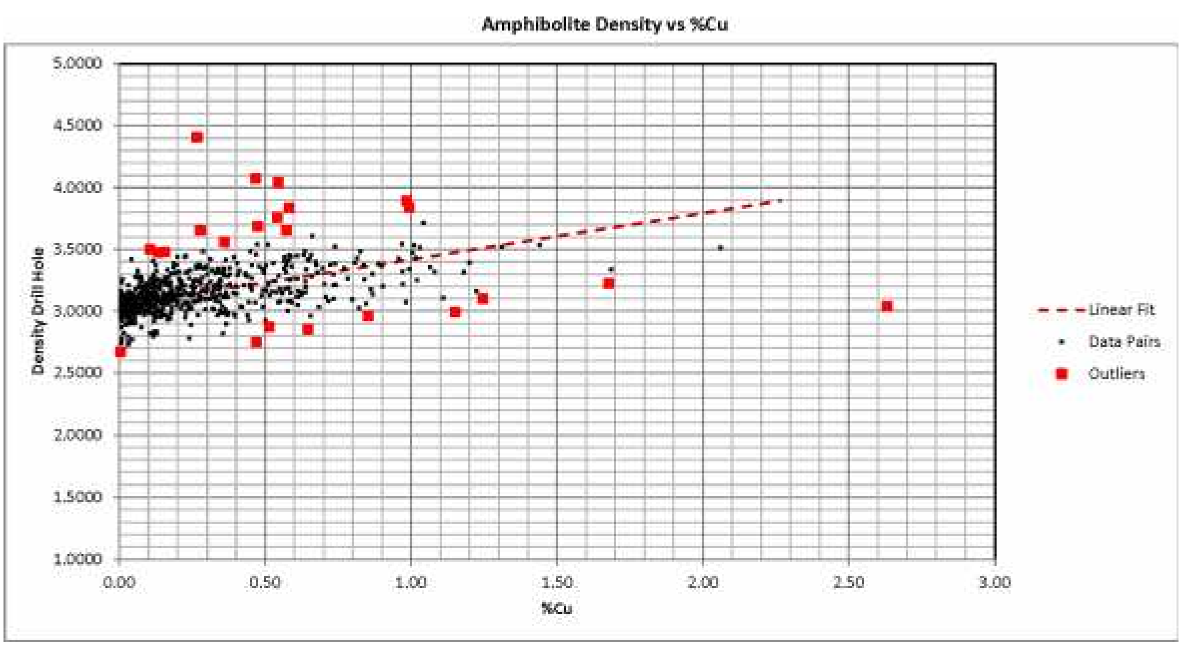


Figure 14.3 - Correlation between Density and Copper for Amphibolites (Noble 2017)

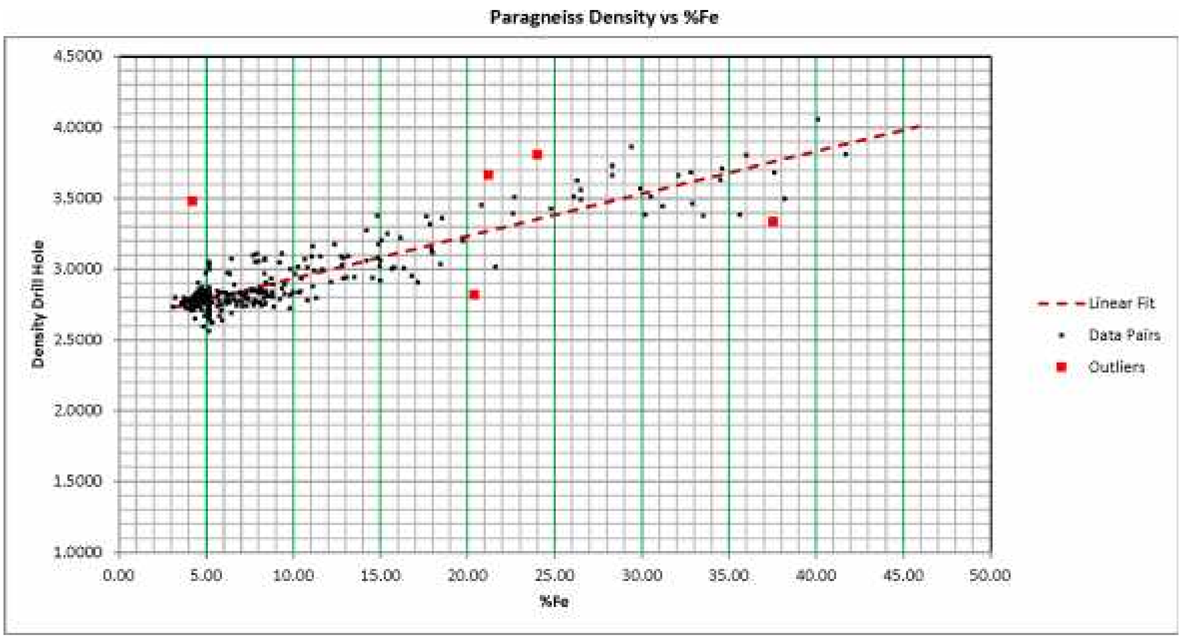


Figure 14.4 - Correlation between Density and Iron for Paragneiss (Noble 2017)

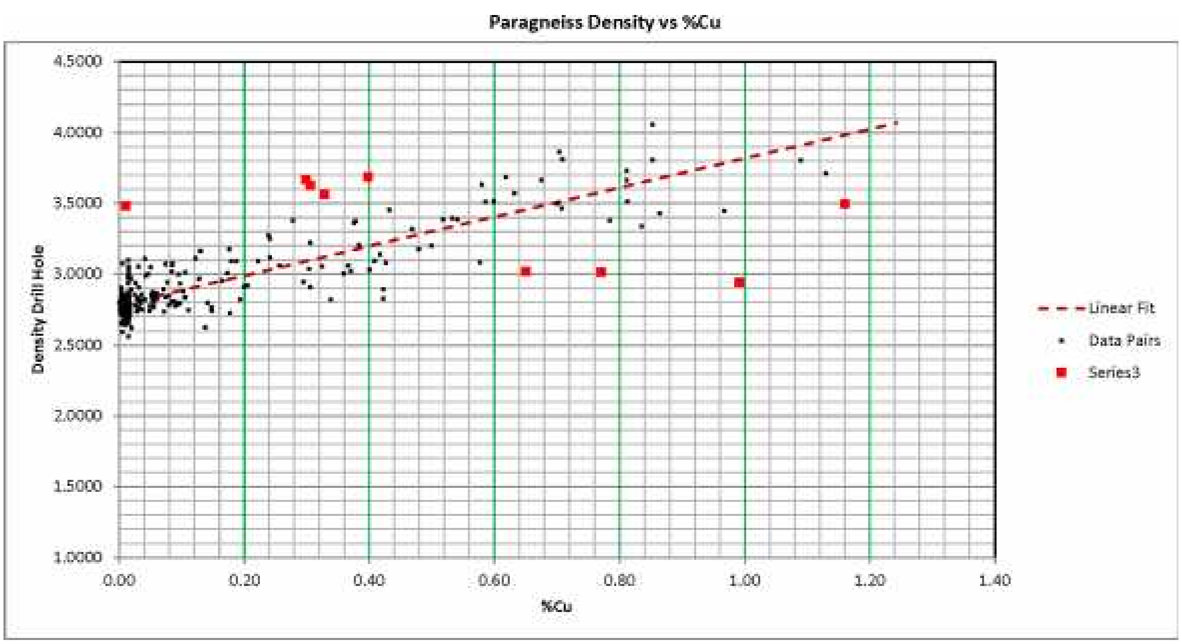


Figure 14.5 - Correlation between Density and Copper for Paragneiss (Noble 2017)

14.4 Topographic Model

Topographic contour data were provided by Atalaya as an AutoCAD dxf file containing 1-m contour lines. This topographic data is based on a Lidar survey dated 2011 by the Instituto Geográfico Nacional (IGN*)*, or National Geographic Institute of Spain. In addition to the topographic contours, contours based on bathymetric measurements were provided by Atalaya for the bottom of the Vieiro and Arinteiro pits, which are currently underwater.

Triangulated digital terrain models (DTMs) were prepared from the original data as follows:

1. The topographic contours were edited slightly to remove internal edge effects along the edges of smaller areas that were combined to create the full topography map.
2. The lake bottom contours were spliced into the base topographic contours.
3. The topographic data were triangulated using the Datamine DTM creation tool.
4. The resulting DTM was cropped to the resource model limits.

A pre-mine topographic DTM was created as follows:

1. The topographic contours from historical maps in the Bama+Brandelos and Vieiro +Arinteiro areas were aligned to the current topography as best possible using the historical maps, which were not in UTM coordinates.
2. The pre-mining contours were triangulated and spliced into the current topography DTM.
3. It is noted that the pre-mining topography is approximate, but is only used to define fill volumes in waste dump areas and the oxidation volume, which was approximated as 10m below the pre-mining DTM.

During topographic modeling and interpretation of the resource, it was determined that the collar elevations of the legacy data were approximately 2m higher than the current surveying DATUM and 2 meters was subtracted from the elevations of the legacy drill holes.

14.5 Resource Modeling Considerations

The Touro deposits and data have many significant characteristics that govern the resource modelling process, as follows:

1. The mineralized zones are highly strata-bound lenses draped around a north-plunging anticlinal structure.
2. The mineralized zones on the west side of the anticline (Brandelos, Bama, Arca, and Monte Minas) are generally near surface and dip subparallel to surface topography at about 10 to 20 degrees to the west. The thickness of the west-side zones averages between 10 and 40 meters. Mineralization on the west side is generally continuous from south to north at a low-grade cutoff of 0.05% Cu, but forms pods of higher-grade mineralization corresponding to the named deposits.

   Although the west mineralization is very continuous on an overall basis, with lateral extents as much as one kilometer, there are local undulations that require extensive modelling of the local trend to maintain proper correlation between drill holes. Local variability appears to be more pronounced along strike than along dip.

3. The mineralized zones on the east side of the anticline (Arinteiro and Vieiro) dip about 20 to 30 degrees to the east and are continuous from south to north. The southern part of the zone, Arinteiro, is a single zone with a thickness of 20 to 40 meters. Moving north to Vieiro, at about 4748200N, the style of mineralization changes and multiple zones are observed that sometimes stack to a vertical thickness of 80 meters. Except for the difference in the style of mineralization, Arinteiro and Vieiro are a single deposit.

   Although there are still some local undulations in the eastern mineralized zones, these zones appear to be much more continuous than the western zones.

4. Lithology is considered by project geologists to be a strong control for copper mineralization with garnet amphibolite being the highest grade, followed by amphibolite and paragneiss. Statistical analysis shows that this is true in the overall sense, but that in detail, all three lithologies are composed of three copper grade populations, including barren (0.01 %Cu), low-grade (0.1% Cu), and high-grade (0.5% Cu), and grade zoning is required for proper modeling of the deposits.

5. There is a very abrupt change in grade from barren rock to mineralized rock and it is not unusual to have a 0.2 %Cu change in copper grade over a few meters. Since the Touro rocks are very hard and competent, it will be impossible to segregate ore and waste along the contacts, and dilution will be significant along contacts.

14.6 Compositing

Based on the tabular, stratigraphically controlled nature of the Touro mineralized zones, one compositing option is to composite with “SEAM” compositing, where the composites are optimized to define the “ore zone” subject to a defined cutoff grade and minimum thickness and dilution criteria. There are several problems with this style of compositing in that it is based on a set cutoff, and analysis of the resource at multiple cutoffs is extremely difficult. In addition, mining dilution, which will vary depending on the dip and thickness of the zone is difficult to add to the estimate.

To address the problem of dilution more simply, two options were considered. One method is down-hole, fixed length compositing where drill holes are composited to fixed lengths starting from the top of the drill hole. Another method is bench compositing, where drill holes are composited to the block model mining benches. Bench compositing was selected for this estimate using the Datamine bench compositing routine COMPBE with a minimum composite length of 5 m and a maximum composite length of 15 m. Compositing intervals start at the toe of each mining bench and end at the crest. Composites with less than 5 meters of assays were discarded.

Calculated composite dilution for 10-meter bench composites is compared with the raw assays and seam composites in Table 14.5. These calculations show that with seam compositing, which represents the minimum dilution with a minimum minable thickness of 10 meters, 8.5% of ore-grade assays are diluted below cutoff and are lost. In addition, 14% dilution is added to bring the mineable thickness up to 10 meters.
With bench compositing, slightly less than 1% of the original ore-grade assays are discarded from bench composites because the 5-meter minimum assay length is not met. In addition, 15% of ore-grade assays

are lost because the diluted composite grade is below cutoff. The remaining ore-grade composites are diluted by nearly 40% with low-grade mineralization. Comparing the bench composites to the seam composites, the bench composites have 10.7% greater dilution and 7.2% lower grade. It is believed that bench compositing introduces sufficient dilution to account for mining dilution on the contacts of the mineralized zones.

Table 14.5 - Composite Dilution with a 0.2% Cu Cutoff Grade

Optimized Seam Composites with 10m Minimum Length					
Category	Length (meters)	Comp. (%Cu)	Assay (%Cu)	%Dilution/Loss (length)	%Dilution/Loss (Copper)
Waste	68928.37	0.0334	0.0276		
Ore Lost to Dilution	1764.24	0.0727	0.2968	-8.5%	-4.9%
Dilution in ORE	3076.59	0.3537	0.1193	14.0%	3.5%
Undiluted ORE	18891.92	0.4983	0.5364		
Total Seam Composite ORE	21968.51	0.4780			
10m Bench Composites					
Waste Lost to Min Length	2058.17	-	0.0185		
Ore Lost to Min Length	140.31	-	0.4002	-0.7%	-0.5%
Waste	63031.91	0.0352	0.0270		
Ore Lost to Dilution	3106.21	0.1440	0.3080	-15.0%	-9.0%
Dilution in ORE	6914.88	0.3220	0.0773	28.4%	5.2%
Undiluted ORE	17409.65	0.4918	0.5540		
Total BENCH Composite ORE	24324.53	0.4436			
Difference vs Seam	10.7%	-7.2%			

14.7 Trend Model

The Touro mineralization is confined to strata-bound lenses in an overall unit consisting of paragneiss and included lenses of amphibolite. The overall unit has been a gently folded into a north-plunging anticlinal structure. The individual deposits lie around the flanks of the anticline and are weakly continuous around the entire structure. There is sometimes complex local folding and it was determined that a trend surface is required to define the short-range continuity of mineralization between drill holes.

The trend surface model was constructed as follows:

1. The project area was divided into a west zone containing the Bama, Brandelos, Arca, and Monte Minas deposits and an east zone containing the Vieiro and Arinteiro deposits. It is possible that the mineral hosting horizon is continuous from west to east, but there is currently a gap in the drilling that separates west and east.
2. The footwall of the mineral host horizon was digitized in cross-section view based on drill hole geology and copper grades as a guide.

3. The digitized footwall strings were linked to form a wireframe model defining the footwall of the deposits in three dimensions.
4. Trend strings were digitized in cross-section view to define the trend of mineralization through adjacent drill holes. In general, the mineralization trend is subparallel to the footwall.

Cross-sections showing drill-hole composites, the footwall of the host unit, and the mineralization trend are shown in **¡Error! No se encuentra el origen de la referencia.** through **¡Error! No se encuentra el origen de la referencia.**, a contour map of the trend surface is shown in Figure 14.18. Bama and Brandelos mineralization, as shown in **¡Error! No se encuentra el origen de la referencia.** to **¡Error! No se encuentra el origen de la referencia.** is tabular and very continuous in Brandelos (left side of the sections), but is more complex at Bama, which has a downward-cupped shape. The mineral trend on the south of Bama appears to dip down along the base of the host unit. To the north, Bama mineralization is stacked vertically and the trend flattens and cuts across the middle of the zone. Between Bama and Arca the mineral trend is continuous, but mineralization is mostly low-grade material between 0.1 and 0.2% Cu, as shown in **¡Error! No se encuentra el origen de la referencia.**. Moving north into Arca, mineralization is generally confined to a single well-defined zone, as shown in **¡Error! No se encuentra el origen de la referencia.**. Further north, the Arca zone weakens and the Monteminas zone develops to the east, as shown in **¡Error! No se encuentra el origen de la referencia.**.

Mineralization in the Arinteiro area is generally confined to a single, well-defined zone that weakens at depth, as shown in **¡Error! No se encuentra el origen de la referencia.** and **¡Error! No se encuentra el origen de la referencia.**. Mineralization is continuous into Vieiro, but the style of mineralization changes and the trends are weaker and mineralization stacks vertically, as shown in **¡Error! No se encuentra el origen de la referencia.**.

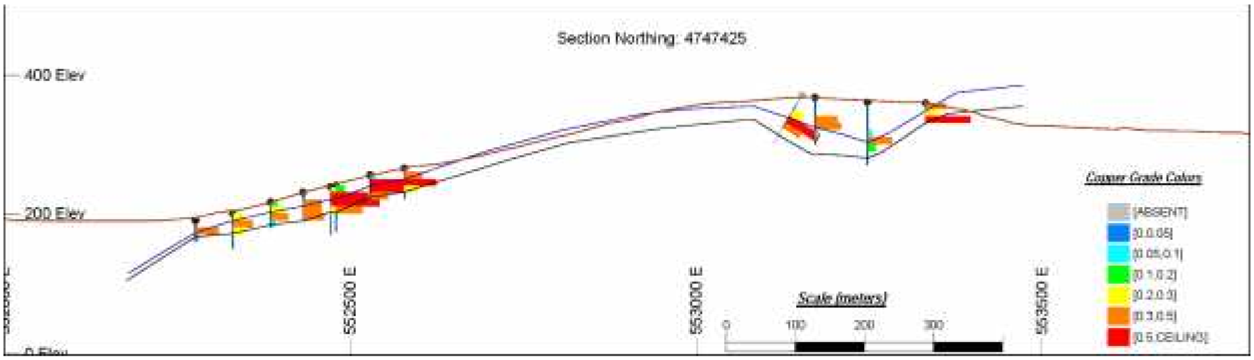


Figure 14.6 - Section Through North-Central Brandelos and Bama (Noble 2017)

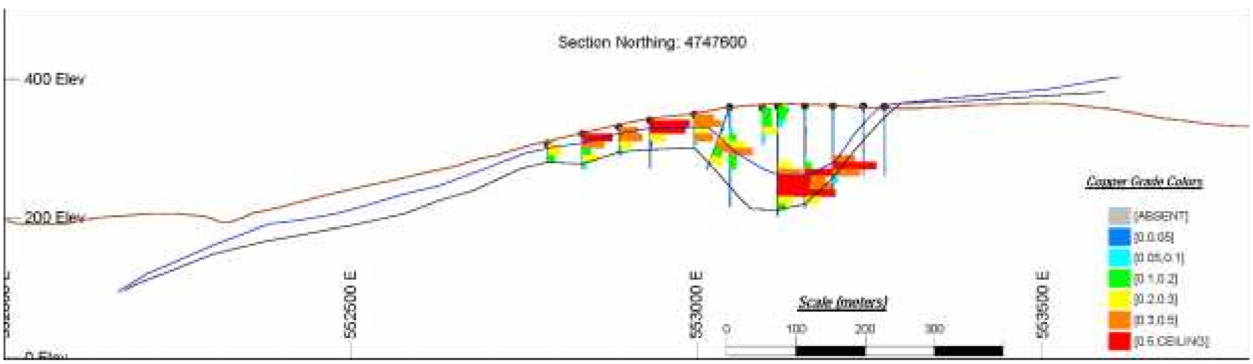


Figure 14.7 - Section Through South-Central Brandelos and Bama (Noble 2017)

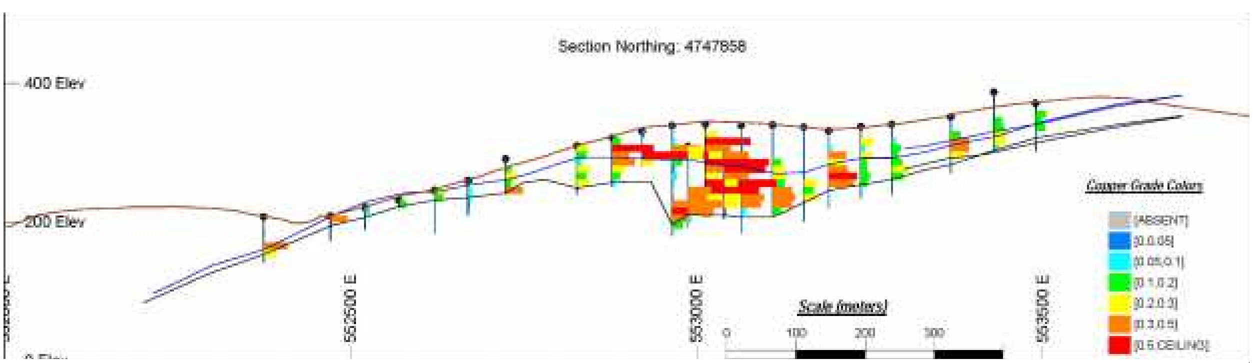


Figure 14.8 - *Section Through South Brandelos and Bama (Noble2017)*

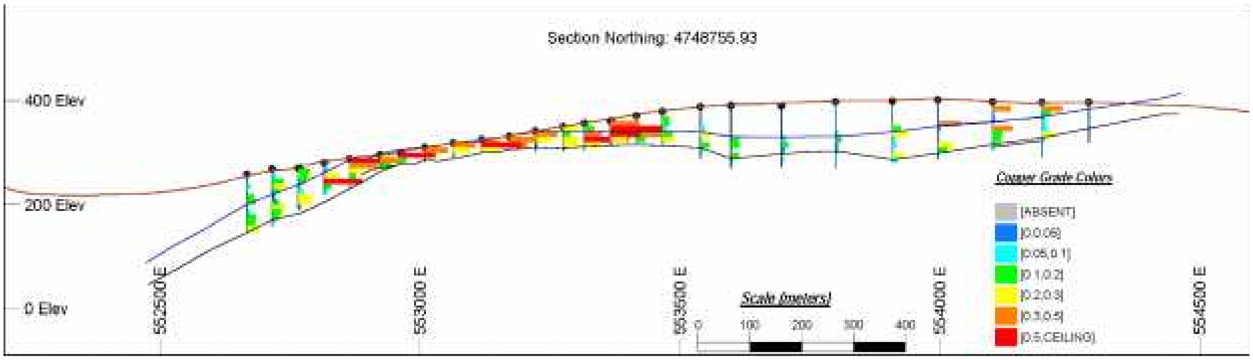


Figure 14.9 - Section Between Bama and Arca (Noble 2017)

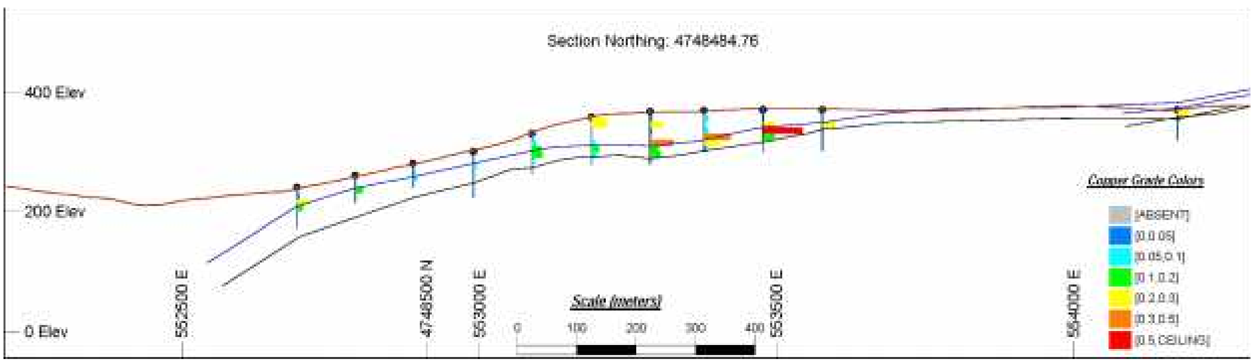


Figure 14.10 - Section Through Monteminas (Noble 2017)

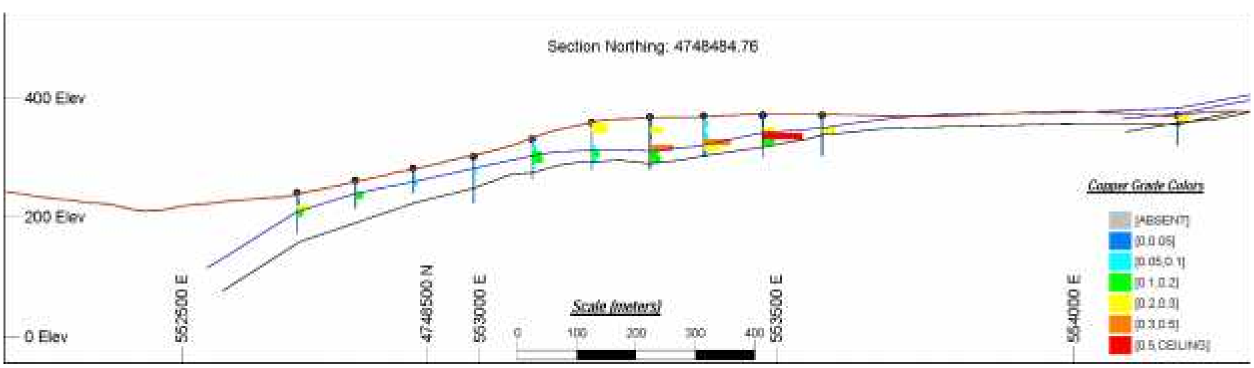


Figure 14.11 - Section Through Arca (Noble 2017)

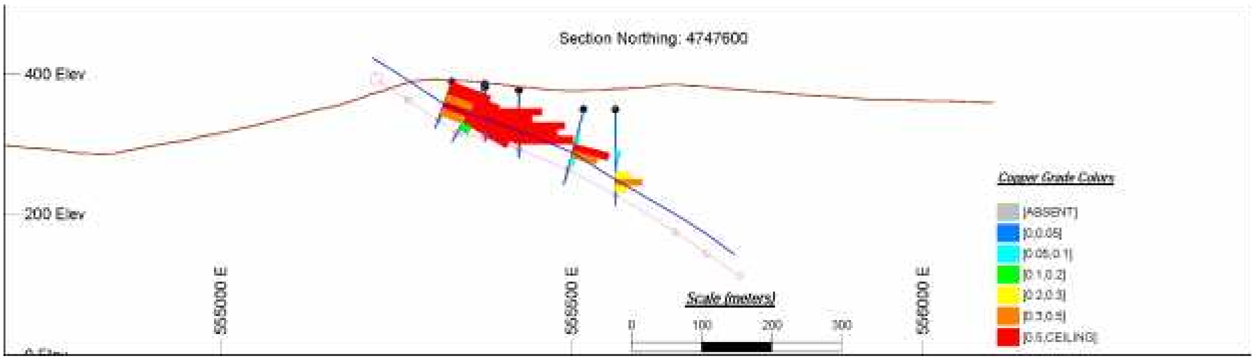


Figure 14.12 - Section Through Central Arinteiro (Noble 2017)

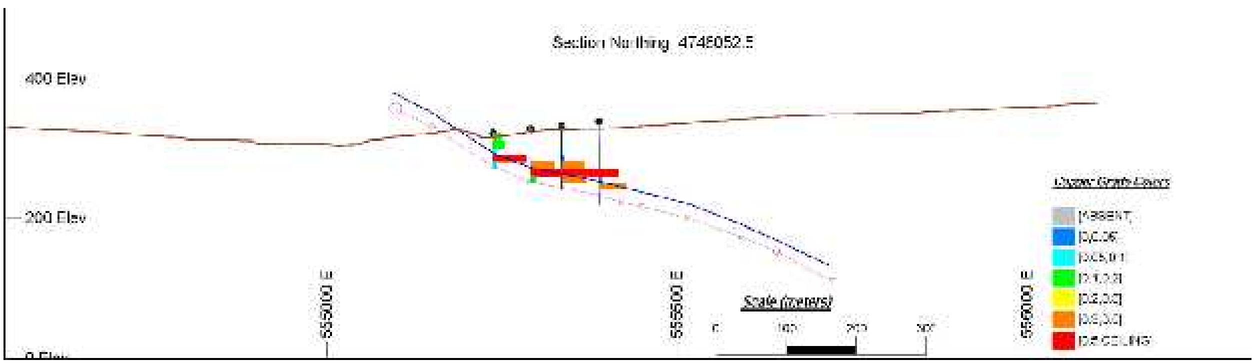


Figure 14.13 - Section Through North Arinteiro/South Vieiro (Noble 2017)

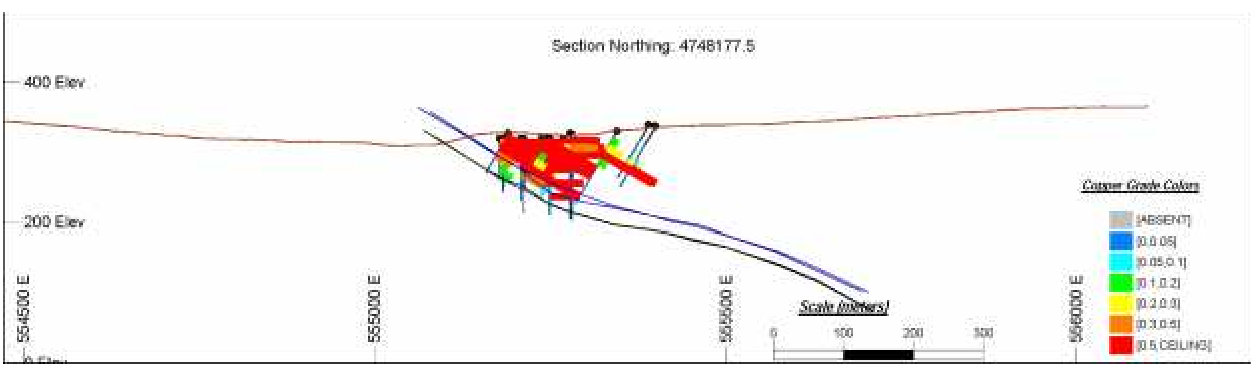


Figure 14.14 - Section through South Vieiro (Noble 2017)

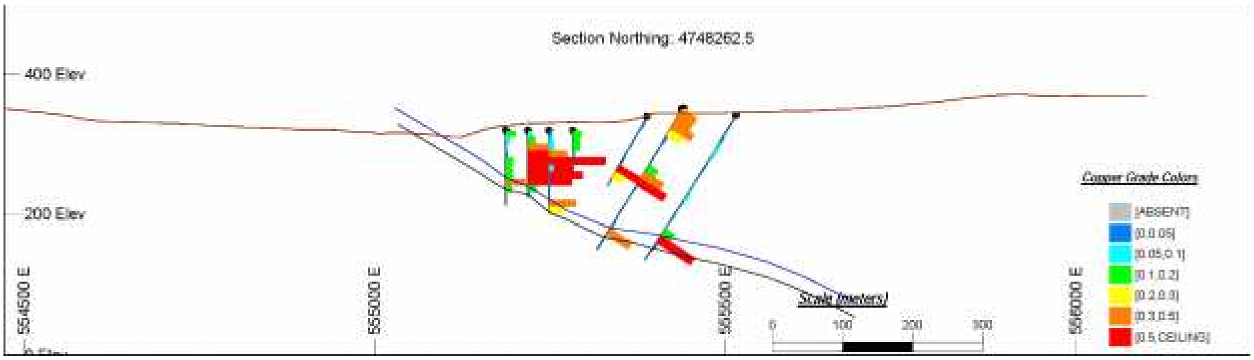


Figure 14.15 - Section Through Central Vieiro (Noble 2017)

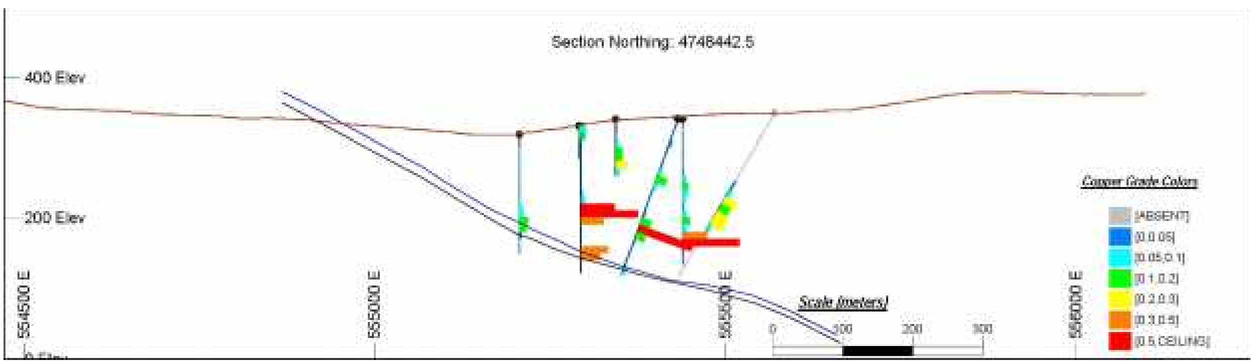


Figure 14.16 - Section Through North-Central Vieiro (Noble 2017)

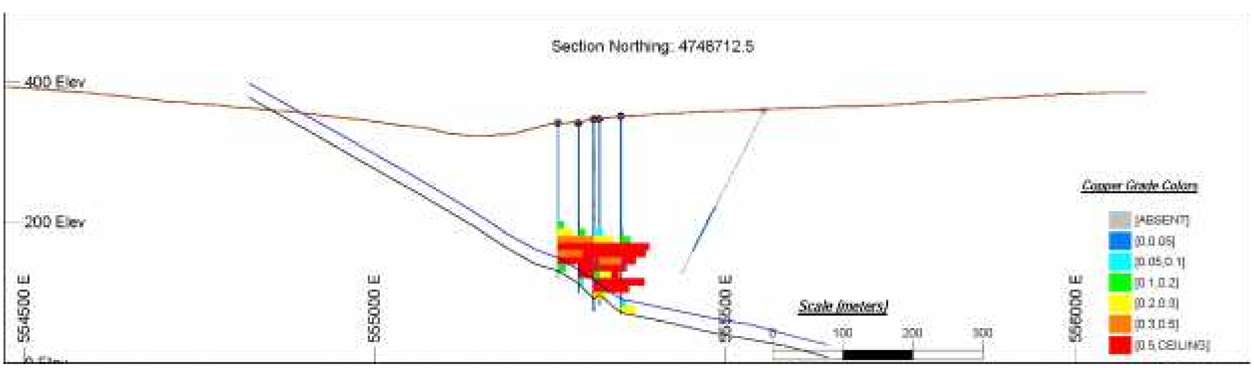


Figure 14.17 - Section through North Vieiro (Noble 2017)

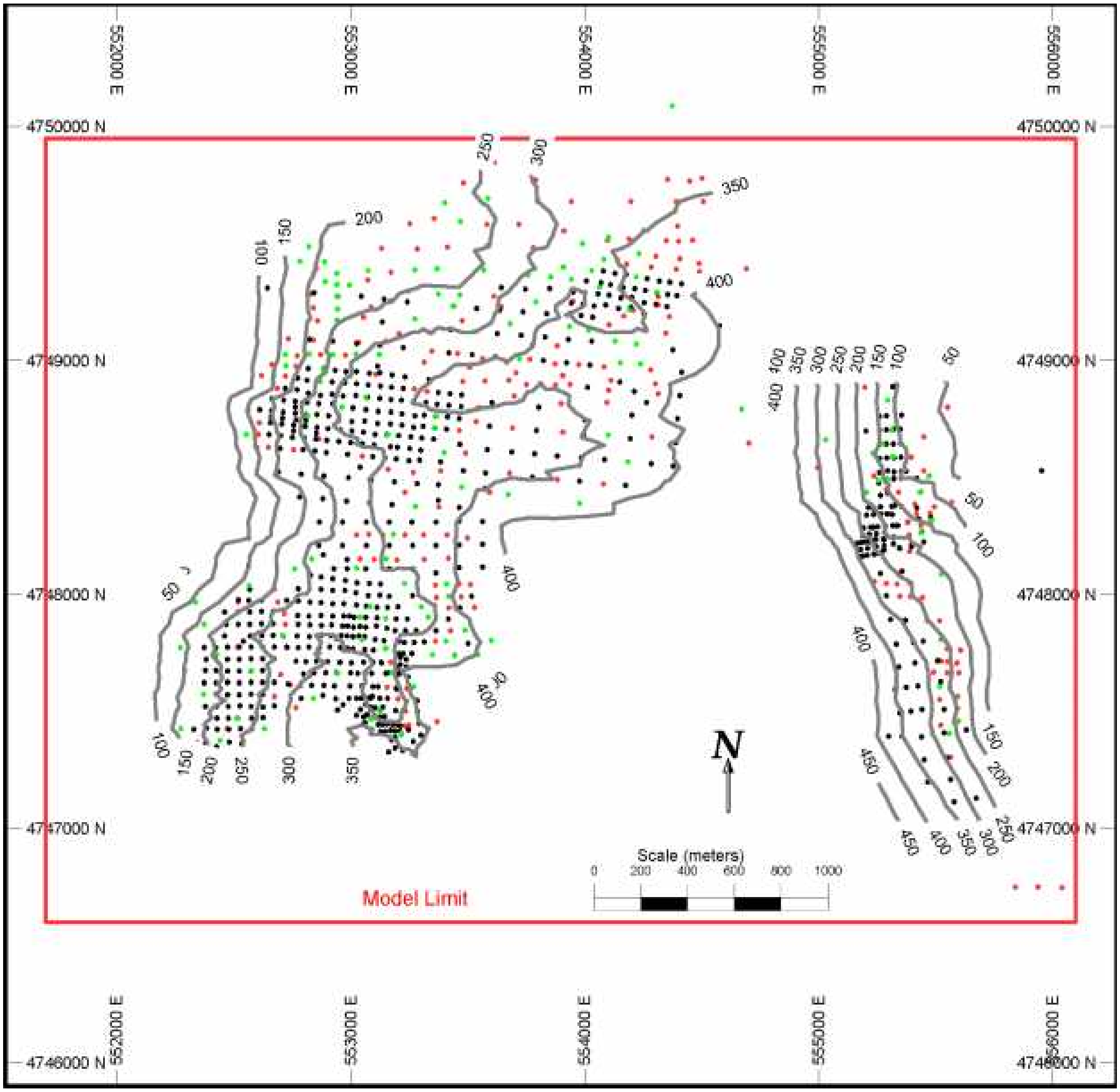


Figure 14.18 - Contour Map of the Mineralization Trend Surface (Noble 2017)

The contour map of the trend surface in Figure 14.18 shows that the east-side mineralization is very consistent, both along strike and down dip. West side mineralization, however, is much more consistent down dip and is more irregular along strike.

14.7.1 Trend-adjusted Coordinates

Trend-adjusted coordinates were computed for composites and block centers, as follows:

1. The elevation of the trend model DTM was determined at the XY point to be adjusted.
2. The zone elevation was computed as: ZELEV = ZTRUE – ZDTM, where ZTRUE is the original elevation and ZDTM is the elevation of the trend model DTM.
3. A second value "ZTREND" was computed by rounding the zone elevation to the 10-meters with an offset of 5-meters. The purpose of the rounded ZTREND elevation was to provide a nominal, "mid-bench", elevation for the nearest neighbor and grid spacing models. The rounded elevation was required to prevent edge-effect issues from the Datamine search ellipse because those models were estimated within 10-meter thick bands.

14.7.2 Base Block Model

An "empty" block model was created that contained the block model template for grade estimation as follows:

1. Block models were defined by creating blocks below the pre-mining topo surface and below the current topography surface.
2. The bottom of the oxide zone was defined as a nominal 10 meters below the pre-mining topography. Blocks above the bottom of the oxide zone and below the top of the pre-mining surface were flagged as oxide.
3. The pre-mining and current topography were combined and codes for MINED and FILL blocks were defined as follows:
    a. If the "CURRENT" block was undefined and the "PREMINE" block was defined, the block was flagged as "MINED".
    b. If the "CURRENT" block was defined and the "PREMINE" block was undefined, the block was flagged as "FILL".
4. The mineral host-zone block model was defined by adjusting the host-zone footwall wireframes downward 10 meters and creating blocks above the adjusted wireframes. The 10-meter adjustment ensures that no blocks are lost to round-off errors near the footwall. Codes for the EWZONE (East and West Areas) were encoded at this step.
5. The host-zone block model and the OXIDE-FILL-MINED block models were combined and clipped so that all blocks are inside the property boundary.
6. Trend-adjusted Z-coordinates were computed for all blocks as discussed previously.

14.8 Copper Grade-Zone Models

14.8.1 Copper Grade Distributions

Copper grades were declustered for analysis of the grade distribution using a nearest-neighbor block model with the bench-composited copper grades and the trend-surface. The resulting distribution of copper grade, presented as lognormal cumulative probability plots and lognormal histogram plots in Figure 14.19, indicates that copper grade is composed of several lognormal populations.

A set of four sub-populations were fitted to the raw data using least square fitting on the histogram data. The resulting distribution fit included the following component populations.

1. A spike at 0.01% Cu, which corresponds to the default value used for low-grade samples that were not assayed.
2. A nominally barren population averaging 0.015% Cu. Together the default values and the barren population are about 42% of the total population.
3. A low-grade distribution averaging 0.135% Cu with 29% of the total population.
4. A high-grade distribution that averages over 0.386% Cu with 29% of the total population.

As shown in the histogram plots, there is significant overlap between the low-grade and the high-grade distributions, and it is very difficult to differentiate between the two based on grade alone. The probability that a sample is in the high-grade distribution was computed from the population models and is displayed in Figure 14.20. The cross-over point where a composite has greater than a 50% point of being in the high-grade population is approximately 0.15% Cu, but over the range between 0.10% Cu to 0.25% Cu a composite has a significant probability of being part of either the high-grade or low-grade population.

Probability Fit Copper - NN Model Measured&Indicated

	Population	Default	Barren	Low Grade	High Grade	Total
Fraction		25.6%	16.8%	29.0%	28.6%	100.0%
Mean	0.154855	0.011	0.015	0.135	0.386	0.155
Beta		0.024	0.639	0.888	0.606	
Median		0.0112	0.0122	0.0910	0.3211	
Log Mean		-4.4889	-4.4045	-2.3965	-1.1359	

Sum of Errors	SSQ	Weight		
%CFREQ	0.000451041	0		
%HISTO	0.00013709	1		
Z-Fit	0.28596349	0	Hist/Z-fit	0.000478
FIT	0.00013709			

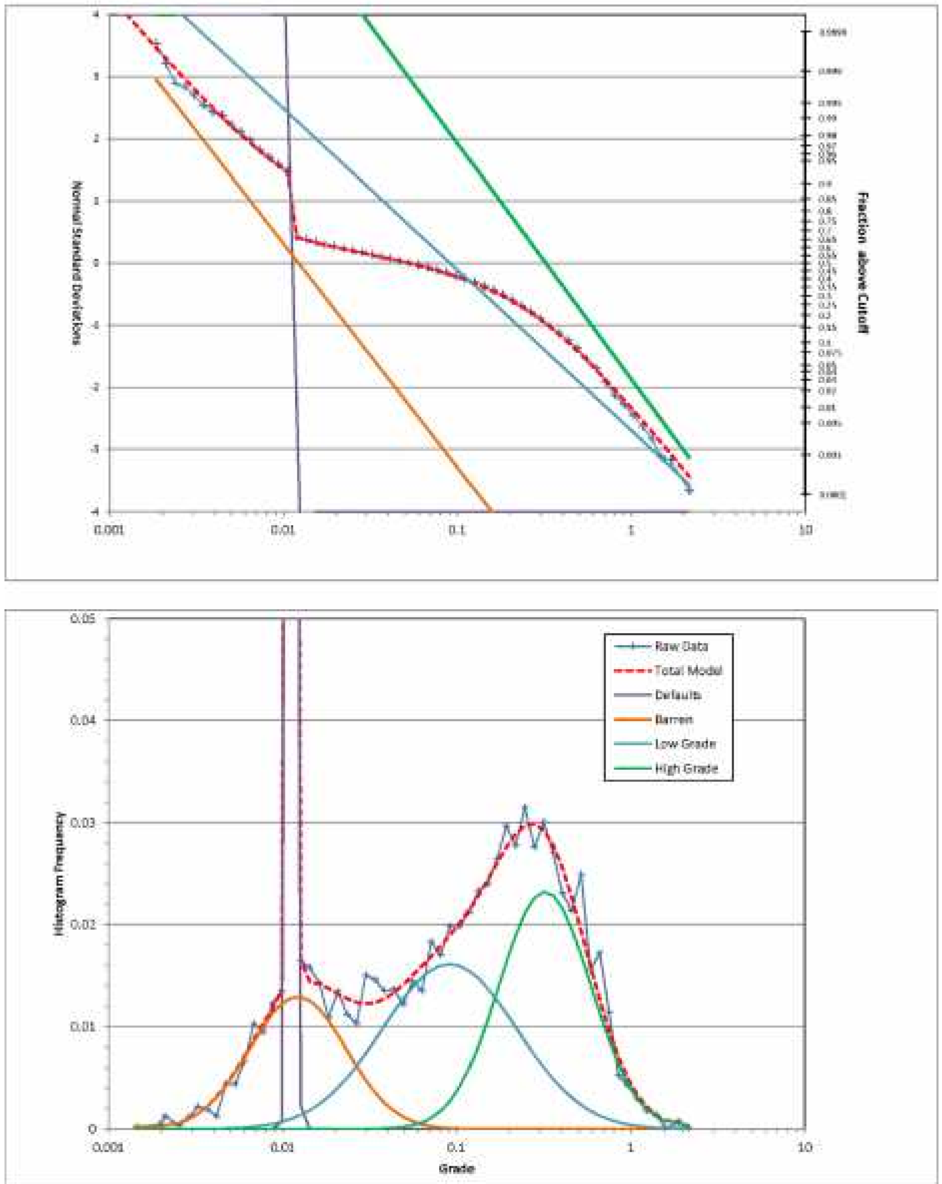


Figure 14.19 - Copper Grade Distributions (Noble 2017)

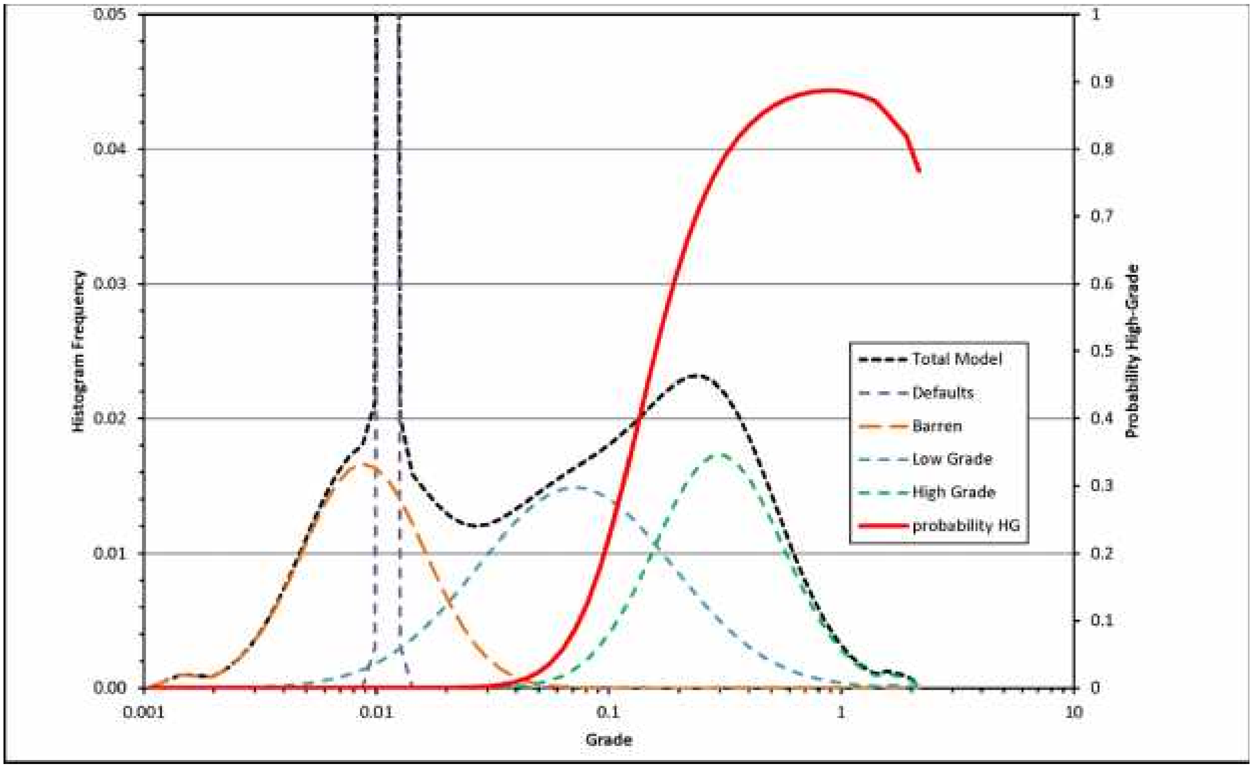


Figure 14.20 - Probability that a Composite is in the High-Grade Population (Noble 2017)

14.9 Variograms

Variograms were computed for the East and West Areas using composites from the combined low-grade and high-grade copper grade zones. Sage2001 variography software was used for variogram computation with the covariance computation option and the trend-adjusted Z-coordinate. The resulting variograms are summarized in Table 14.6 and are shown graphically in Figure 14.21 and Figure 14.22. The variograms show that continuity is significantly better in the East area than in the West area, probably because of the simpler geometry in the East. The best continuity is roughly parallel to the strike of the mineralized zones.

Table 14.6 - Summary of Copper Variogram Models – Low Grade + High Grade Combined

	Sage Variograms – Scaled to Sill =1.0									
			Exponential Structure 1				Exponential Structure 2			
Area	Rotation	Nugget	Range (X')	Range (Y')	Range (Z')	Sill	Range (X')	Range (Y')	Range (Z')	Sill
East	4°	0.20	30	98	20	0.61	80	17.0	17.5	0.19
West	28°	0.50	28.3	65.9	20.1	0.50				

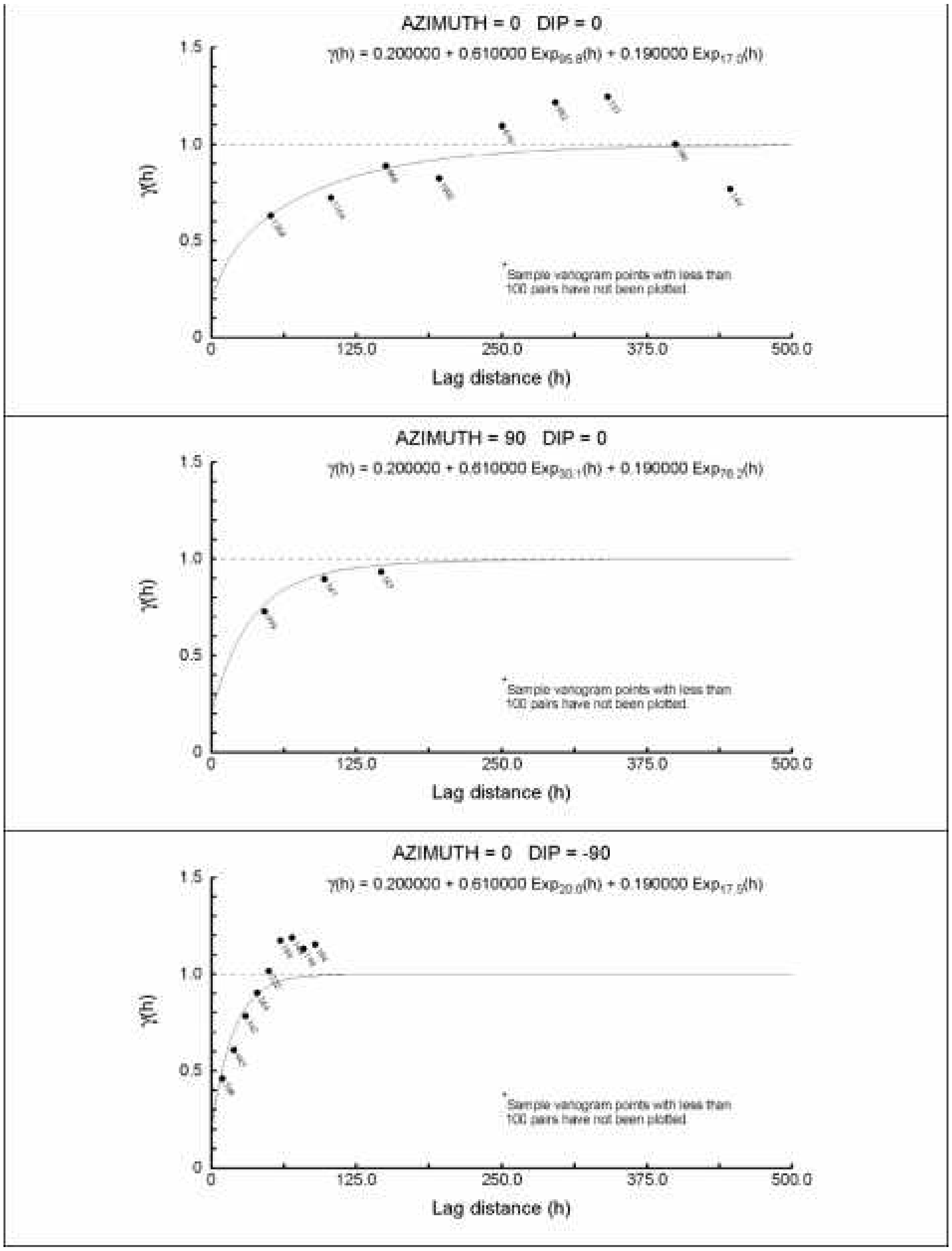


Figure 14.21 - East Area Copper Variograms Using Trend-Flattened Coordinates (Noble 2017)

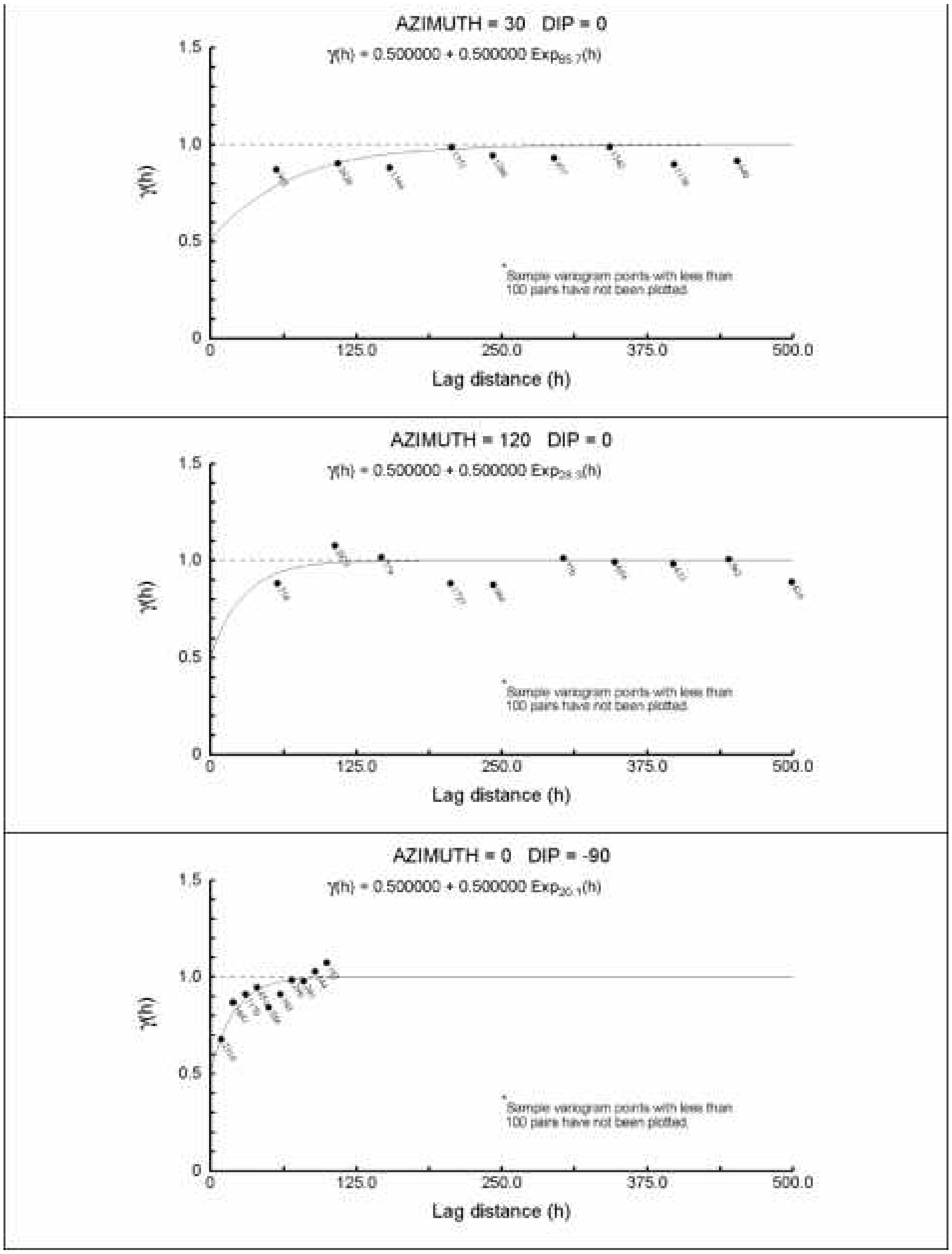


Figure 14.22 - West Area Copper Variograms Using Trend-Flattened Coordinates (Noble 2017)

14.10 Grade Models

Grade models were estimated for copper using the trend-adjusted coordinate space with both nearest-neighbor (NN) and inverse-distance-power (IDP) interpolation. Barren, low-grade, and high-grade zones were used to control interpolation.

14.10.1 Copper Grade Zone Models

The large overlap between the low-grade and high-grade populations is problematic for resource estimation, since there is no easy way to differentiate the populations. Accordingly, a simple strategy was developed to provide grade-zoning control as follows:

1. A simple nearest-neighbor (NN) model was created for copper grade, and grade zones were assigned using the grade-range parameters in Table 14.7. Search parameters for the NN model are documented in Table 14.8.
2. Interpolation was done using composites with overlapping grade ranges. The overlapping grade ranges are required to prevent polygonal edge effects on the boundaries of the grade zones. Composite grade-range parameters were optimized during grade estimation.

Table 14.7 - Grade-Zone Parameters for Block Model and Composites

Zone Code	Description	Block Model Grade Ranges		Composite Grade Ranges	
		Lower	Upper	Lower	Upper
1	Barren	0.00	0.04	0.00	0.05
2	Low-Grade	0.04	0.15	0.02	0.20
3	High Grade	0.15	Maximum	0.11	Maximum

Table 14.8 - Grade-Zone NN Search Ellipses

Zone	X’ (m)	Y’ (m)	Z’ (m)
All Zones	100	100	5
Search parameters are relative to trend-flattened coordinates rounded to nearest 10 meters, starting at 5 meters			

14.10.2 Search Ellipse Parameters

Search ellipse parameters were developed for each zone based on the variograms and a general assessment of the continuity of grades. All search ellipses are relative to the trend-adjusted coordinate system. The grid flag uses rounded "Z-trend" coordinates so that the search is confined to a single 10-meter thick trend band. The Datamine search expansion feature was used to expand an initial search ellipse until the desired number of composites were located inside the search ellipse. The primary objective of the search ellipse expansion was to keep the search as localized as possible, subject to finding sufficient samples for reliable estimation. The final search ellipse expansion was set to provide estimates in areas with widely spaced drilling. Search ellipse parameters are summarized in Table 14.9.

Table 14.9 - Search Ellipse Parameters

			Search Radius (m)			Number Composites		Max
Case	Rotation (degrees)	Expansion Volume	X'	Y'	Z'	Min	Max	Compos Per Hole
Grid Flag	0	1	90	90	0.5	8	12	1
		2	135	135	0.75	8	12	
		3	270	270	1.5	1	12	
West Area – Cu	27	1	75	110	10	9	12	3
		2	112.5	165	15	9	12	
		3	225	330	30	1	12	
East Area – Cu	2	1	75	110	10	6	9	1
		2	112.5	165	15	6	9	
		3	225	330	30	1	9	
The grid FLAG uses rounded trend coordinates for the Z-coordinates. Cu grade estimation uses unrounded trend coordinates.								

14.10.3 Copper Grade Estimation

Copper grade estimation was done inverse-distance-power (IDP) with grade zone control. IDP estimation was done using the trend-adjusted coordinate system with estimation parameters tailored to each zone. The IDP model was optimized relative to the NN model as follows:

1. The average copper grade of the IDP model should be as close as possible to the average grade of the NN model to ensure that the overall estimates are unbiased.
2. The variance of the models should be reduced by an appropriate amount, as determined from the variograms.

The optimized IDP powers are shown in Table 14.10. The comparison between IDP and NN Estimates in Table 14.11 demonstrates that the IDP estimates are unbiased relative to the NN estimates and that a level of variance reduction, or smoothing, has been introduced that is consistent with the target block variances predicted by the variograms.

Table 14.10 - IDP Estimation Powers by Grade Zone

Case	Rotation	Anisotropy Distances			Power
		X'	Y'	Z'	
West Barren	27	50	100	32	2.0
West Low Grade	27	50	100	32	2.0
West High Grade	27	50	100	32	2.5
East Barren	2	45	100	25	2.3
East Low Grade	2	45	100	25	2.3
East High Grade	1	45	100	25	3.2

Table 14.11 - Comparison of IDP vs NN Estimates for Measured and Indicated Blocks

East-West Area	Grade Zone	Number Blocks	IDP Blocks		NN Blocks		Ratio Average IDP/NN	Ratio Relative Variance IDP/NN	Target Variance Ratio
			Average	Relative Variance	Average	Relative Variance			
West	Low Grade	28298	0.094	0.057	0.091	0.124	1.037	0.458	0.435
West	High Grade	50223	0.344	0.121	0.349	0.253	0.987	0.480	0.435
West	Low + High	78521	0.254	0.369	0.256	0.541	0.993	0.681	
East	Low Grade	5760	0.088	0.082	0.086	0.118	1.027	0.695	0.691
East	High Grade	12889	0.520	0.273	0.518	0.399	1.003	0.684	0.691
East	Low + High	18649	0.386	0.609	0.385	0.772	1.004	0.789	
East + West	Low + High	97170	0.279	0.505	0.280	0.675	0.996	0.747	

14.11 Resource Classification

Classification of resources into measured, indicated, and inferred resource classes is based on drill-hole spacing and the number of drill holes selected for estimation. Drill-hole spacing is measured based on the kriging variance from a point-kriging estimate using a "FLAG" variable that is set to 1.0 for composites with copper values and "absent" for composites with insufficient sampling to make a composite. A linear, zero-nugget variogram with a slope of 0.5 is used for this kriging run. The kriging variance for block at the center of a 4-point, square drill-hole pattern is approximately equal to 28% of the drill-hole spacing. If the block is outside the drilling pattern (extrapolated), the kriging variance is equal to the distance to the nearest drill hole. The resource classification parameters are summarized in Table 14.12.

Table 14.12 - Resource Classification Parameters

Resource Class	Drill Hole Spacing (m)	Search Samples
Measured	<=60 m	SVOL<=1
Indicated	>60 m to <100 m	SVOL<=2
Inferred	>100 m to <= 150 m	
Unclassified	>150 m or no estimate	

14.12 Resource Summary

The copper resource was summarized using a Lerches-Grossman pit shell that was run using a copper price of $3.20/lb Cu and all resources including inferred resources. All other slope and economic parameters are the same as those used for design of the open pit for reserve estimation. The resulting pit shell is considered to have reasonable prospects for economic extraction, assuming that the inferred resource is converted to measured and indicated by drilling and that the copper price returns to previous levels that were substantially above $3.20/lb Cu. The resource estimate is summarized in Table 14.13.

Table 14.13 - Touro Project - Resource Summary-Constrained by the $3.20/lb Cu Pit

	>= 8.14 NSR $/t (Internal Cutoff)				>= 9.71 NSR $/t (Breakeven Cutoff)			
Resource Class	Ktonnes	NSR $/t	Cu%	RCu%	Ktonnes	NSR $/t	Cu%	RCu%
Measured	69,258	22.55	0.42	0.37	67,886	22.82	0.42	0.37
Indicated	60,592	19.24	0.36	0.31	59,188	19.49	0.37	0.32
Measured + Indicated	129,850	21.00	0.39	0.34	127,074	21.27	0.40	0.35
Inferred	46,521	19.33	0.37	0.32	45,822	19.48	0.37	0.32

14.13 Discussion of Factors Affecting Resources

There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that would materially affect the resource estimate. Copper price is currently rising from historically low levels, however, and the return of higher copper prices is needed to achieve the full resource potential.

15 MINERAL RESERVE ESTIMATES

The Touro project mineral reserve estimates are based on open pit development of the Arinteiro, Vieiro, Arca, Bama, Brandelos, and Monte de las Minas deposit areas and are derived from the same 27 March 2017 deposit block model that was used to estimate mineral resources as described in Section 14. Portions of the Arinteiro, Vieiro, Bama, and Brandelos deposits were mined by Rio Tinto Patiño using open pit methods during the 1970s and 1980s. Copper mining and processing operations ceased in 1986 due to low metal prices. Subsequent activity on the project site has consisted of an aggregate operation and other industrial mineral activities operated by the private company Explotaciones Gallegas. The deposit model is based on current topography and accounts for this previous mining activity.

A range of economic pit shells were generated using the Lerchs-Grossmann (LG) algorithm to help define the likely extents of the open pit and the most favorable development sequence. A pit shell based on a copper price of $2.60/lb was then used to guide the design of the open pits. The mine design work and the level of accuracy of the mineral reserve estimates are consistent with industry practices for a preliminary feasibility study.

15.1 Definitions

Canadian National Instrument 43-101 (NI 43-101) references the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves. The mineral reserve estimates reported in this section follow the CIM Definition Standards. The following definition is from those standards:

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Mineral reserve is subdivided to indicate the degree of certainty that can be attached to the estimate. For mineral reserve, the following definitions are from the CIM Definition Standards and are applicable to this report:

A "Proven Mineral Reserve‟ is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

A "Probable Mineral Reserve‟ is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

In this study, mineral reserve is defined as the measured and indicated mineral resource that would be extracted by the mine design and which can then be processed at a profit. All measured resources meeting that standard are herein classified as proven mineral reserves, while all indicated resources meeting that standard are classified as probable mineral reserves.

15.2 Reserve Estimation Parameters

15.2.1 Metallurgical Recoveries

The projected plant recoveries are discussed in Section 13.5 and are summarized below for the reader's convenience:

$$V \quad P \quad S \quad O \quad R = 5.09\, x \ln(H\ ) + 94.9$$

$$A \quad O\ he \quad P \quad S \quad G \quad R = 7.196 \times \ln(H\ ) + 92.9$$

$$T \quad O \quad R = \frac{100 \times C \quad \times (H \quad - 0.094)}{H \quad \times (C \quad - 0.094)}$$

Where:

- CG = Concentrate Grade (Cu%)
- HG = Head Grade (Cu%)

Recoverable Cu grades were computed by block and stored in the deposit model. Primary sulfide recoveries average about 88% for direct plant feed and 86% for low grade stockpile material. Similarly, transitional ore recoveries average about 78% for direct plant feed and 73% for low grade stockpile material. No recoveries from oxide or fill materials are considered in this study.

15.2.2 Ore Definition Parameters

Table 15.1 lists the base economic parameters used in the pit limit analyses and cutoff grade calculations. All costs are expressed in 2017 U.S. dollars. Conversions from Euro currency are based on an exchange rate of US$1.07 per €1.00. Mining costs were estimated from contract mining rates quoted to Atalaya Mining and approximate haulage distance measurements for ore and waste from each deposit area. Atalaya Mining provided estimates of ore processing and general/administration costs. Freight, smelting and refining (FSR) charges were calculated from the average concentrate grades, concentrate treatment costs of $95/dmt, freight costs of $40/dmt, 8% moisture, and Cu refining costs of $0.095/lb.

Table 15.1 – *Ore Definition Parameters*

Parameter	Deposit					
	Vieiro	Arinteiro	Arca	Monte	Bama	Brandelos
Cu price, US$/lb Cu	2.60	2.60	2.60	2.60	2.60	2.60
Primary ore Cu recovery, %	91%	87%	87%	87%	87%	87%
Transitional ore Cu recovery, %	78%	78%	78%	78%	78%	78%
Primary ore concentrate grade, % Cu	29.5	29.5	29.5	29.5	29.5	29.5
Transition ore concentrate grade, % Cu	25.0	25.0	25.0	25.0	25.0	25.0
FSR charge for primary ore, US$/lb Cu payable	0.32	0.32	0.32	0.32	0.32	0.32
FSR charge for transition ore, US$/lb Cu payable	0.36	0.36	0.36	0.36	0.36	0.36
Cu payable, %	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%
Ore mining, US$/t mined	1.49	1.45	1.62	1.41	1.72	1.75
Waste mining, US$/t mined	1.54	1.53	1.64	1.50	1.64	1.61
Ore processing, US$/t ore	6.89	6.89	6.89	6.89	6.89	6.89
General admin & land rehab, US$/t ore	1.25	1.25	1.25	1.25	1.25	1.25
Internal cutoff (ICOG), NSR US$/t	8.09	8.06	8.12	8.05	8.22	8.28
Breakeven cutoff (BECOG), NSR US$/t	9.63	9.59	9.76	9.55	9.86	9.89
Primary Sulfide Ore:						
Cu recovery at ICOG, %	87%	81%	81%	81%	82%	82%
Cu recovery at BECOG, %	87%	83%	83%	83%	83%	83%
Internal cutoff, Cu %	0.19	0.20	0.21	0.20	0.21	0.21
Breakeven cutoff, Cu %	0.23	0.24	0.24	0.24	0.25	0.25
Internal cutoff, RCu %	0.17	0.17	0.17	0.17	0.17	0.17
Breakeven cutoff, RCu %	0.20	0.20	0.20	0.20	0.20	0.20
Transitional Ore:						
Cu recovery at ICOG, %	64%	64%	65%	64%	65%	65%
Cu recovery at BECOG, %	68%	68%	69%	68%	69%	69%
Internal cutoff, Cu %	0.26	0.26	0.26	0.26	0.27	0.27
Breakeven cutoff, Cu %	0.29	0.29	0.30	0.29	0.30	0.30
Internal cutoff, RCu %	0.17	0.17	0.17	0.17	0.17	0.17
Breakeven cutoff, RCu %	0.20	0.20	0.20	0.20	0.21	0.21
Low Grade Ore Stockpile:						
LG stockpile rehandling cost, US$/t	0.83	0.83	0.83	0.83	0.83	0.83
Model Cu recovery at LG stockpile cutoff	77.9%	73.4%	73.5%	73.4%	73.5%	73.6%
Stockpile reclaim losses	7%	7%	7%	7%	7%	7%
LG stkpl cutoff, block model NSR US$/t	8.92	8.89	8.95	8.88	9.05	9.11
LG stockpile cutoff, Cu %	0.25	0.27	0.27	0.27	0.27	0.27
LG stockpile cutoff, RCu %	0.20	0.20	0.20	0.20	0.20	0.20

NSR refers to Net Smelter Return. The internal cutoff grades include differential ore mining (i.e., ore mining less waste mining costs, which can result in a credit), plus ore processing, plant maintenance, and general/administration costs. The breakeven cutoff grades include the full ore mining cost, plus ore processing, plant maintenance, and general/administration costs. Net revenue is computed by subtracting the FSR cost from the Cu price and then applying the metallurgical recovery and Cu payable factors to the result. No royalty or severance obligations are anticipated for the Touro project.

15.2.3 Overall Slope Angles

Atalaya Mining provided slope angle design criteria that was, in part, derived from a geotechnical study completed by Terratec in November 2016. Inter-ramp angles (IRAs) varied between 50° to 60°, depending on rock wall orientation. A fixed 45° overall slope angle (OSA) was used in the LG pit limit evaluations for all deposit areas, accounting for haulage ramps and providing some conservatism for the steeper slope areas.

15.2.4 Bulk Densities

Each block in the deposit model was assigned a density corresponding to the rock type. Fill material was assigned a density of 2.00 t/m^3 and oxides received a value of 2.30 t/m^3. Densities average 2.82 t/m^3 for transitional material and 2.83 t/m^3 for primary sulfides. Overall, densities average 2.73 t/m^3 within the planned open pits.

15.2.5 Dilution and Ore Loss

The deposit model was constructed to include the effects of mining selectivity and dilution. No additional provisions outside of the block model have been made for mining dilution and ore loss for this preliminary feasibility analysis.

15.3 Economic Pit Limit Analyses

The Lerchs-Grossmann (LG) algorithm was used to analyze economic pit limits based on the recoveries and other parameters discussed in the previous sections. In all cases, only mineral resources classified as measured and indicated (M&I) were considered as potential ore; all inferred mineral resources were treated as waste.

The sensitivities of economic pit limits to copper price were evaluated for a range of Cu prices from $1.00/lb to $4.00/lb in $0.25/lb increments. A run was also made at the projected long-term Cu price of $2.60/lb. No time value of money optimization analyses were performed for this level of study. Recoveries, payables, FSR costs, and operating costs are as described in Sections 15.2.1 and 15.2.2. Table 15.2 summarizes the measured and indicated mineral resources contained within the pit shells generated by the LG price sensitivity evaluations above an average $8.14/t NSR cutoff. Figure 15.1 illustrates the corresponding tonnage and grade curves.

Table 15.2– Lerchs-Grossmann Cu Price Sensitivity Analyses

Cu Price	Contained M+I Resources >= $8.14/t NSR				Waste	Total	Strip
US$/lb	kt	NSR $/t	Cu%	RCu%	kt	kt	Ratio
1.00	663	11.96	0.888	0.822	702	1,365	1.06
1.25	3,334	12.87	0.712	0.648	3,447	6,781	1.03
1.50	11,360	13.64	0.604	0.542	14,858	26,218	1.31
1.75	23,219	14.60	0.538	0.479	32,432	55,651	1.40
2.00	49,150	16.39	0.513	0.458	119,332	168,482	2.43
2.25	71,058	17.11	0.470	0.416	159,957	231,015	2.25
2.50	90,424	17.99	0.440	0.387	194,497	284,921	2.15
2.60	95,715	18.42	0.431	0.379	201,909	297,624	2.11
2.75	103,185	19.15	0.421	0.370	216,796	319,981	2.10
3.00	116,685	20.23	0.405	0.354	246,294	362,979	2.11
3.25	126,124	21.51	0.395	0.345	269,626	395,750	2.14
3.50	133,485	22.82	0.387	0.337	287,174	420,659	2.15
3.75	139,228	24.24	0.381	0.332	306,936	446,164	2.20
4.00	145,227	25.52	0.374	0.326	323,858	469,085	2.23

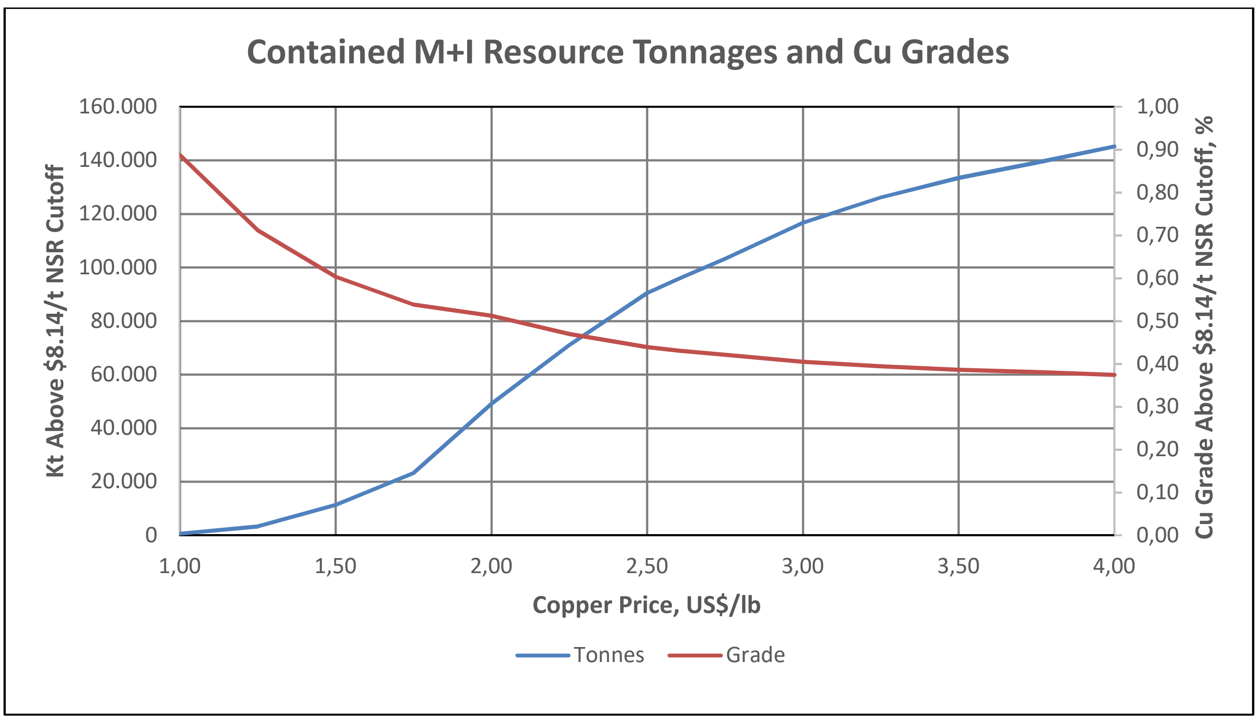


Figure 15.1- LG Price Sensitivity Tonnage and Cu Grade Curves (WLRC 2017)

15.4 Open Pit Designs

The ultimate pits and internal phases for the Touro project were designed to accommodate contractors' small- to medium-scale mining equipment operating on 10-m bench intervals. This equipment includes rock drills capable of drilling blastholes of up to 127 mm in diameter, hydraulic excavators and/or front-end loaders with bucket capacities of about 6-13 m^3, off-highway trucks with up to 91-t payload capacities, and appropriately sized support equipment.

The $2.60/lb Cu LG shell was used as the basis for the ultimate pit designs. Seven pits were indicated: one for the Arca deposit area, one combined pit for Arinteiro and Vieiro, one for Bama, one for Brandelos, and three pits within the Monte de las Minas area. In the interest of improving mill head grades in the first five years of operation, the $1.75 and $2.00/lb Cu LG shells were used to guide the design of internal phases within six of the pits. A total of 13 phase designs were developed.

15.4.1 Design Parameters

Pit walls were smoothed from guiding LG pit shells. The smoothing minimizes or eliminates, where possible, noses and notches that could affect slope stability. Internal haulage ramps were included to allow for truck access to working faces on each level. The basic parameters used in the design of the internal mining phases and ultimate pit are summarized in Table 15.3.

Table 15.3 – Basic Pit Design Parameters

Parameter	Unit	Value
Bench height	m	10
Haul road width, 2-way (including ditch & safety berm)	m	26
Haul road width, 1-way (including ditch & safety berm)	m	19
Internal ramp gradient	%	10
Minimum pushback width	m	40

Short-term haulage roads near pit bottoms were occasionally steepened to gradients of up to 12%. Additionally, six-meter-deep "good-bye" cuts were sometimes used to extract small ore zones on or near pit bottoms to reduce ramp construction and waste stripping.

Table 15.4 lists the slope design parameters for each deposit area. IRA refers to inter-ramp angle, BFA refers to bench face angle, CBI refers to catch bench interval (vertical), and CBW refers to catch bench width. The pit/phase designs were based the Atalaya Mining slope angle design parameters referenced in Section 15.2.3, except that the 60° IRA slopes were reduced to 55° to provide adequate catch bench widths for rock fall safety reasons.

Table 15.4 – Pit Slope Design Parameters

Deposit Area	Pit Walls	Angles in Degrees		CBI	CBW
		IRA	BFA	m	m
Arinteiro	S, SW, W (az 180-270)	50	68	20	8.70
Arinteiro	Other (az 270-180)	55	75	20	8.65
Vieiro	SW, W, NW (az 200-320)	50	68	20	8.70
Vieiro	Other (az 320-200)	55	75	20	8.65
Arca	E, SE, S (az 90-180)	50	68	20	8.70
Arca	Other (az 180-90)	55	75	20	8.65
Monte de las Minas	S, SW, W (az 180-270)	50	68	20	8.70
Monte de las Minas	Other (az 270-180)	50	68	20	8.70
Bama & Brandelos	All	55	75	20	8.65

15.4.2 Initial Mining Phases

The six initial mining phases: Arinteiro-Vieiro, Brandelos 1, Arca 1, Monte 1, Monte 2 and Bama 1, and major haul roads are shown in Figure 15.2. Old waste rock storage areas are visible in the green topographic surface around the Arinteiro-Vieiro, Bama, and Brandelos pits. The plant site is outlined in red. The coordinate grids are on 500 m intervals. The internal phases targeted LG pit shells generated using Cu prices of $1.75 to $2.00/lb.

The Arinteiro-Vieiro deposits will be mined in one phase to the ultimate extents and has a maximum depth of about 260 m and a pit bottom elevation of 100 m. The next deepest initial phase is Bama 1 at about 130 m from crest to a pit bottom elevation of 220 m. The remaining internal phases are relatively shallow at less than 60-70 m below the topographic surface.

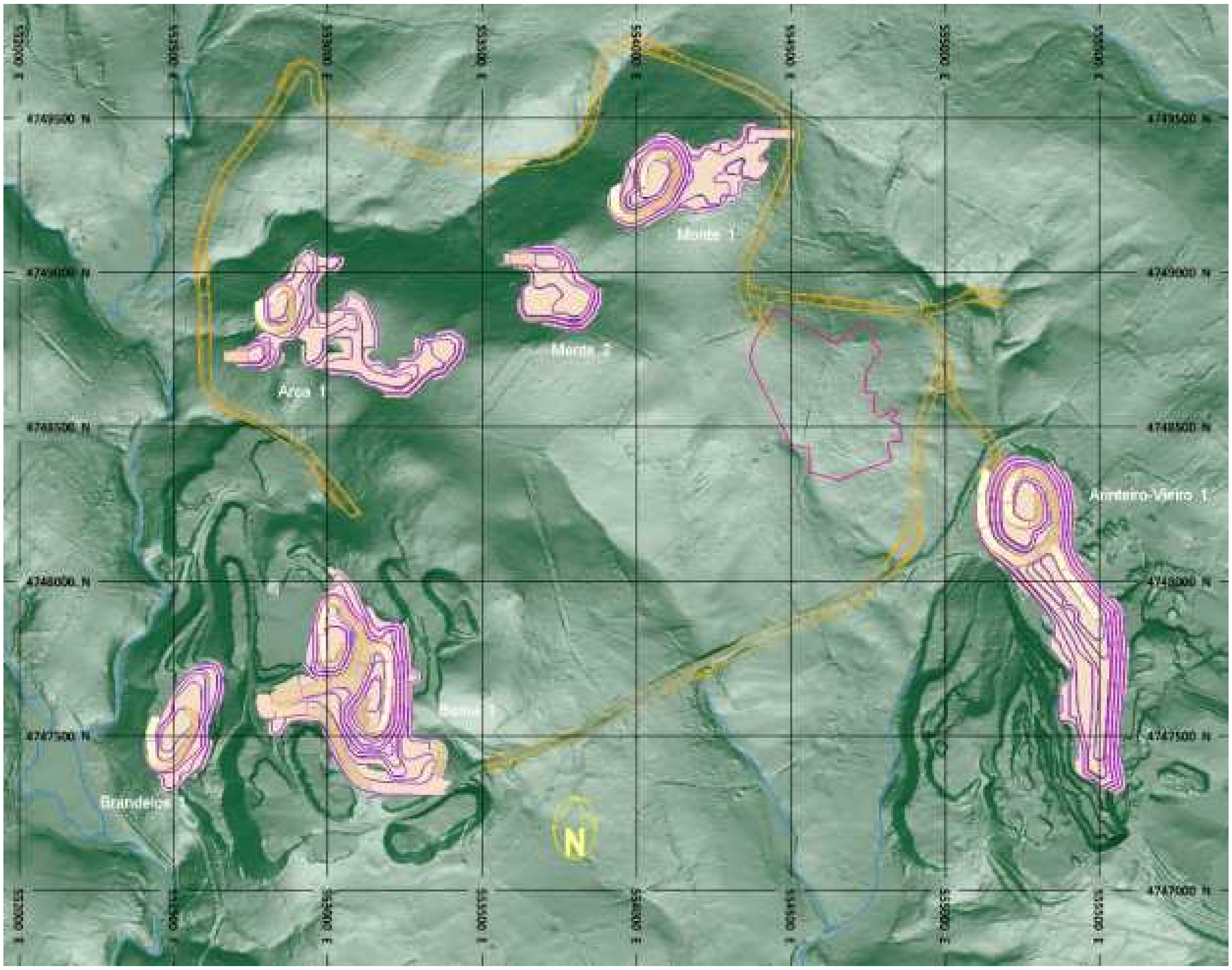


Figure 15.2– Initial Mining Phases (WLRC 2017)

15.4.3 Ultimate Pits

The ultimate pit limits are illustrated in Figure 15.3. The remaining mining pushbacks expanding from the initial phases are: Brandelos 2, Monte 3, Arca 2, Bama 2, and Monte 4. Monte 5 is a new pit that will be developed late in the project's life. The final phases were fit to LG shells generated by a $2.60/lb Cu price. As before, the plant site is outlined in red and the grids are on 500 m intervals.

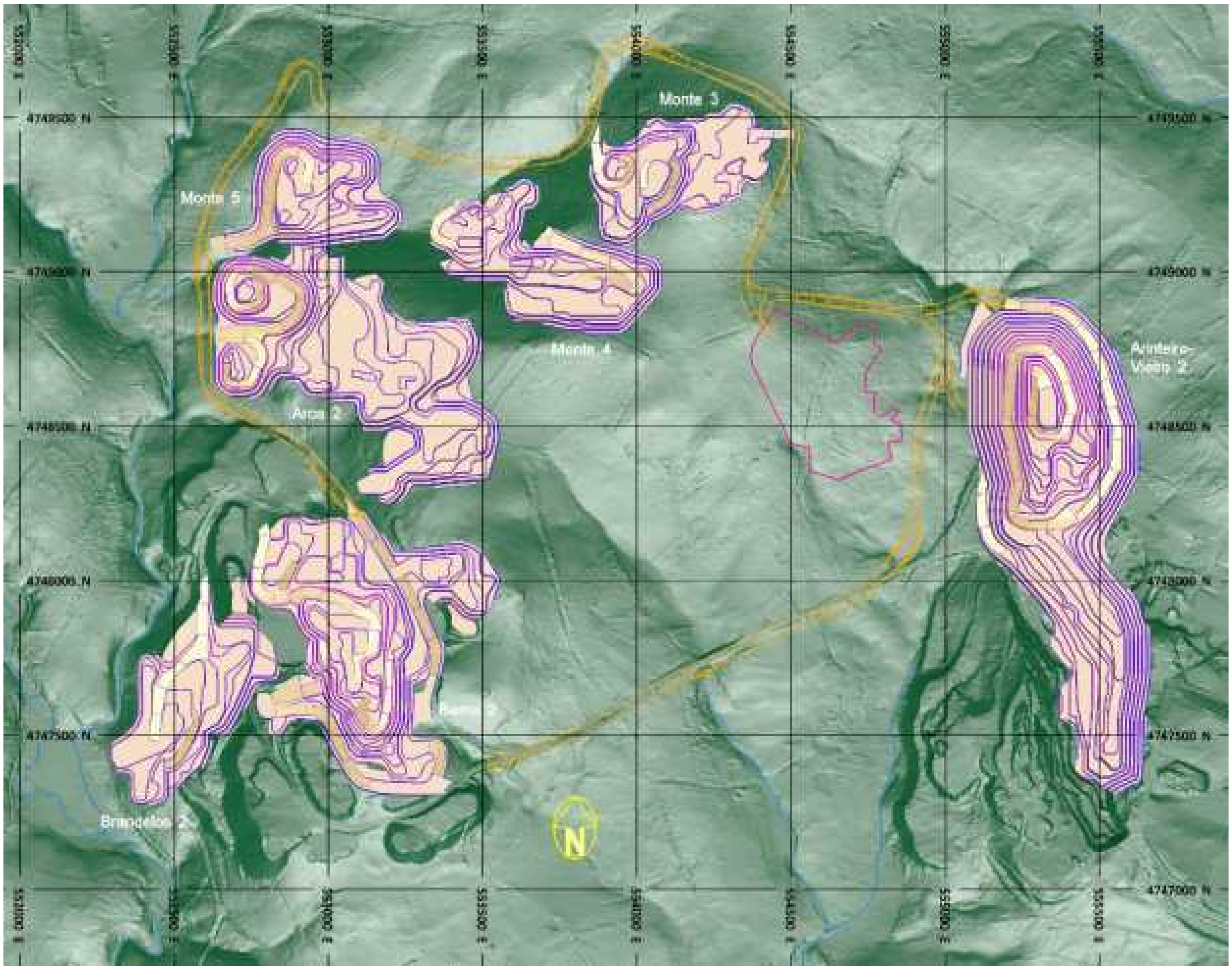


Figure 15.3 – Ultimate Pit Limits (WLRC 2017)

The deepest pit will be Arinteiro-Vieiro at 260 m from crest to pit bottom and is about 1.6 km in length. The Bama pit has a maximum depth of approximately 190 m. The bottoms of the remaining pits are less than 100-120 m below the topographic surface.

15.5 Mineral Reserve Statement

The estimated mineral reserves for the Touro project are fully contained within the ultimate pit limits discussed in the previous section. A mine production schedule, described further in Section 16, was developed using 12 mining phases from seven pits and targeting annual copper production of approximately 30,000 tonnes. This schedule served as the basis for the mineral reserve estimates.

15.5.1 Cutoff Grades

Variable NSR cutoff grades were used during the first five years of operation to improve project returns and balance mill feed and waste stripping requirements. A $10.00/t NSR cutoff was used for preproduction stripping, Years 1-2, and Year 5. NSR cutoffs for Years 3 and 4 were $14.00/t and $12.00/t, respectively, and were lowered to an average internal cutoff of $8.14/t after Year 5. The elevated cutoffs were essentially applied to most of the initial mining phases (Arinteiro-Vieiro, Arca 1, Brandelos 1, Monte 1, and Monte 2) and the upper portions of Bama 1. An average low-grade stockpile cutoff of $9.00/t NSR was applied to material below the mill cutoff during the first five years. The low-grade material would be reclaimed as mill feed near the end of mining operations in Years 11 and 12.

15.5.2 Mineral Reserve Estimate

Mineral reserves are the measured and indicated mineral resources that could be profitably extracted by the mine plan and processed by the plant, estimates for which are derived from the mine production schedule. All measured mineral resources meeting that standard are classified as proven mineral reserves, while all indicated mineral resources meeting that standard are classified as probable mineral reserves. All inferred mineral resources and unclassified material are treated as waste.

Tables 15.5 presents the estimates of proven and probable mineral reserves by mining phase taken from the mine production schedule. The mining phases are listed in order of development. “Monte” refers to the Monte de las Minas deposit area. NSR values are based on a copper price of $2.60/lb.

Table 15.5– Proven and Probable Mineral Reserve Estimates by Mining Phase

Phase	Total Proven Reserves				Total Probable Reserves				Total Proven+Probable Reserves			
	Kt	NSR $/t	Cu%	RCu%	Kt	NSR $/t	Cu%	RCu%	Kt	NSR $/t	Cu%	RCu%
Arinteiro-Vieiro	15,047	25.62	0.57	0.53	8,426	24.60	0.56	0.51	23,473	25.25	0.57	0.52
Brandelos 1	3,341	18.81	0.44	0.39	190	20.65	0.49	0.43	3,531	18.91	0.45	0.39
Arca 1	3,483	19.55	0.46	0.40	361	17.85	0.43	0.37	3,844	19.39	0.46	0.40
Monte 1	3,119	22.08	0.51	0.45	955	20.37	0.48	0.42	4,074	21.68	0.51	0.45
Monte 2	32	19.27	0.45	0.40	1,627	19.86	0.47	0.41	1,658	19.85	0.47	0.41
Bama 1	9,933	18.23	0.43	0.37	803	18.89	0.45	0.39	10,736	18.28	0.43	0.38
Brandelos 2	3,623	13.52	0.33	0.28	2,544	13.27	0.32	0.27	6,167	13.42	0.33	0.28
Monte 3	654	16.99	0.40	0.35	2,006	13.77	0.33	0.28	2,661	14.56	0.35	0.30
Arca 2	9,428	14.26	0.34	0.29	5,149	13.62	0.33	0.28	14,577	14.03	0.34	0.29
Bama 2	7,649	14.81	0.36	0.30	2,454	13.69	0.33	0.28	10,102	14.54	0.35	0.30
Monte 4	15	11.94	0.29	0.25	4,729	13.64	0.33	0.28	4,743	13.64	0.33	0.28
Monte 5	447	17.80	0.42	0.37	4,893	15.82	0.38	0.33	5,340	15.99	0.38	0.33
Total	**56,769**	**19.08**	**0.44**	**0.39**	**34,137**	**17.33**	**0.41**	**0.36**	**90,906**	**18.42**	**0.43**	**0.38**

Note: Mineral reserve estimates are based on variable NSR cutoff grades used in the mine productions schedule (see Section 16). NSR cutoffs were $10.00/t for preproduction stripping through Year 2, $14.00/t in Year 3, $12.00/t in Year 4, and $10.00/t in Year 5 and then lowered to the internal $8.14/t NSR cutoff for all remaining years.

Table 15.6 summarizes the estimates of combined proven and probable mineral reserves, along with waste rock, total material tonnages, and strip ratios by mining phase. This table does not include the rehandling of 449 kt of Run-Of-Mine (ROM) or 3,020 kt of low grade ore stockpiles that are included in the mine production schedule described in Section 16.

Table 15.6 – Estimates of Proven+Probable Mineral Reserves with Waste Rock by Mining Phase

Phase	Proven+Probable Reserves				Waste	Total	Strip
	kt	NSR $/t	Cu%	RCu%	kt	kt	Ratio
Arinteiro-Vieiro	23,473	25.25	0.57	0.52	118,951	142,424	5.07
Brandelos 1	3,531	18.91	0.45	0.39	1,192	4,723	0.34
Arca 1	3,844	19.39	0.46	0.40	3,748	7,591	0.98
Monte 1	4,074	21.68	0.51	0.45	6,444	10,518	1.58
Monte 2	1,658	19.85	0.47	0.41	2,819	4,477	1.70
Bama 1	10,736	18.28	0.43	0.38	13,192	23,927	1.23
Brandelos 2	6,167	13.42	0.33	0.28	6,157	12,324	1.00
Monte 3	2,661	14.56	0.35	0.30	5,215	7,876	1.96
Arca 2	14,577	14.03	0.34	0.29	22,214	36,791	1.52
Bama 2	10,102	14.54	0.35	0.30	17,488	27,590	1.73
Monte 4	4,743	13.64	0.33	0.28	8,396	13,139	1.77
Monte 5	5,340	15.99	0.38	0.33	15,515	20,855	2.91
Total	90,906	18.42	0.43	0.38	221,329	312,235	2.43

Note: Mineral reserve estimates are based on variable NSR cutoff grades used in the mine productions schedule (see Section 16). NSR cutoffs were $10.00/t for preproduction stripping through Year 2, $14.00/t in Year 3, $12.00/t in Year 4, and $10.00/t in Year 5 and then lowered to the internal $8.14/t NSR cutoff for all remaining years.

Total proven and probable mineral reserves are estimated at 90.9 Mt grading 0.43% Cu contained and recoverable copper are estimated at approximately 392,000 t and 345,000 t, respectively. Waste rock is estimated at a little over 221 Mt, resulting in an average stripping ratio of 2.43. All mineral reserves reported in this study are contained within the mineral resources estimated in Section 14.

Compared to the design basis LG shell at a copper price of $2.60/lb, ore-grade material tonnages contained within the mining phases are about 5% lower but have nearly identical average grades. Total material tonnages are about 5% higher than the LG shell, largely due to pit wall smoothing and internal ramp design. The lower amounts of ore-grade material are mostly explained by a 50-m buffer imposed on the west side of Brandelos 2 to avoid disturbance of a nearby stream (the LG pit limit analyses were unconstrained).

It should be noted that the mining phases contain approximately 2.8 Mt of inferred mineral resources grading 0.40% Cu and 0.35% RCu, mostly in Arinteiro-Vieiro, Monte 4, and Monte 5. Inferred mineral resources have been treated as waste in this study. *Mineral resources that are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a great amount of uncertainty as to their existence and as to whether they can be mined legally or economically. There is no certainty that the any of the estimated inferred mineral resources will be converted to measured or indicated mineral resources.*

The mineral reserve estimates are effective as of 1 September 2017.

15.5.3 Sensitivity of Reserves to Other Factors

Potential sensitivities of mineral reserves to variations in copper price are indicated by the pit limit evaluations described in Section 15.3.

Previous mining has shown that some of the waste rock contains potentially-acid-generating (PAG) material. Presently, the criteria required to quantify PAG and non-acid-generating (NAG) waste rock has not been established. Potential impacts would mostly be in the location and design of waste rock storage facilities, but sequencing of waste rock placement into available facilities could affect the mine production schedule and, consequently, some mineral reserves. The magnitude of such impacts are unknown at this time.

No other mining, metallurgical, infrastructure, or permitting factors are presently known that may materially affect the mineral reserve estimate.

16 MINING METHODS

Open pit copper mining operations have been conducted within the Touro project site during the 1970s and 1980s before being shut down in 1986 due to low metal prices. A privately-held company, Explotaciones Gallegas, has conducted limited aggregate and other industrial mineral operations in the area since then, but not copper extraction. Old pit walls are presently standing well and vehicular access exists into the pit areas. Some pit dewatering will be required prior to new development work, particularly in the Vieiro pit.

Redevelopment of copper mining operations will use conventional, open pit methods, mining from benches on 10-m vertical intervals. Atalaya Mining has a history of using contractors to execute its mining plans on other projects and anticipates doing so for Touro development. Contractors' small-to medium-scale mining equipment will be used, including: rock drills capable of drilling 102- to 127-mm-diameter blastholes, hydraulic excavators and/or front-end loaders with bucket capacities of approximately 6-13 m^3, off-highway trucks with 55- to 91-t payload capacities, and suitably sized support equipment.

16.1 Mining Pits/Phases

Seven pits were developed using Lerchs-Grossmann (LG) pit shells based on a copper price of $2.60/lb, the final pushbacks for which are: Arinteiro-Vieiro, Arca 2, Bama 2, Brandelos 2, Monte 3, Monte 4, and Monte 5. Arinteiro-Vieiro has high Cu grades along with a high stripping ratio, and is the primary development target in the early years of operation. Five initial phases were designed to improve mill head grades during the first 6-7 years of operation, targeting smaller pit extents identified by $1.75 and $2.00/lb Cu LG shells. These phases are identified as: Arca 1, Bama 1, Brandelos 1, Monte 1, and Monte 2. "Monte" refers to the Monte de las Minas deposit area.

Details of the economic and pit design parameters are discussed in Sections 15.2 and 15.3. For the reader's convenience, the basic pit and slope design parameters are reproduced in Tables 16.1 and 16.2, respectively. Estimates of proven plus probable mineral reserves and waste rock are summarized by phase (in order of development) in Table 16.3. Figures 16.1 and 16.2 illustrate, respectively, the initial and final mining phase plans, main haul roads, and the plant area (outlined in red).

Table 16.1 – Basic Pit Design Parameters

Parameter	Unit	Value
Bench height	m	10
Haul road width, 2-way (including ditch & safety berm)	m	26
Haul road width, 1-way (including ditch & safety berm)	m	19
Internal ramp gradient	%	10
Minimum pushback width	m	40

Table 16.2 – Pit Slope Design Parameters

Deposit Area	Pit Walls	Angles in Degrees IRA	BFA	CBI m	CBW m
Arinteiro	S, SW, W (az 180-270)	50	68	20	8.70
Arinteiro	Other (az 270-180)	55	75	20	8.65
Vieiro	SW, W, NW (az 200-320)	50	68	20	8.70
Vieiro	Other (az 320-200)	55	75	20	8.65
Arca	E, SE, S (az 90-180)	50	68	20	8.70
Arca	Other (az 180-90)	55	75	20	8.65
Monte de las Minas	S, SW, W (az 180-270)	50	68	20	8.70
Monte de las Minas	Other (az 270-180)	50	68	20	8.70
Bama & Brandelos	All	55	75	20	8.65

Note: CBI refers to catch bench interval (vertical), CBW to catch bench width, IRA to inter-ramp angle, and BFA to bench face angle.

Table 16.3 – Estimates of Proven+Probable Mineral Reserves with Waste Rock by Mining Phase

Phase	Proven+Probable Reserves kt	NSR $/t	Cu%	RCu%	Waste kt	Total kt	Strip Ratio
Arinteiro-Vieiro	23,473	25.25	0.57	0.52	118,951	142,424	5.07
Brandelos 1	3,531	18.91	0.45	0.39	1,192	4,723	0.34
Arca 1	3,844	19.39	0.46	0.40	3,748	7,591	0.98
Monte 1	4,074	21.68	0.51	0.45	6,444	10,518	1.58
Monte 2	1,658	19.85	0.47	0.41	2,819	4,477	1.70
Bama 1	10,736	18.28	0.43	0.38	13,192	23,927	1.23
Brandelos 2	6,167	13.42	0.33	0.28	6,157	12,324	1.00
Monte 3	2,661	14.56	0.35	0.30	5,215	7,876	1.96
Arca 2	14,577	14.03	0.34	0.29	22,214	36,791	1.52
Bama 2	10,102	14.54	0.35	0.30	17,488	27,590	1.73
Monte 4	4,743	13.64	0.33	0.28	8,396	13,139	1.77
Monte 5	5,340	15.99	0.38	0.33	15,515	20,855	2.91
Total	90,906	18.42	0.43	0.38	221,329	312,235	2.43

Note: Mineral reserve estimates are based on variable NSR cutoff grades used in the mine productions schedule (see Section 16.2). NSR cutoffs were $10.00/t for preproduction stripping through Year 2, $14.00/t in Year 3, $12.00/t in Year 4, $10.00/t in Year 5 and then lowered to the internal $8.14/t NSR cutoff for all remaining years.

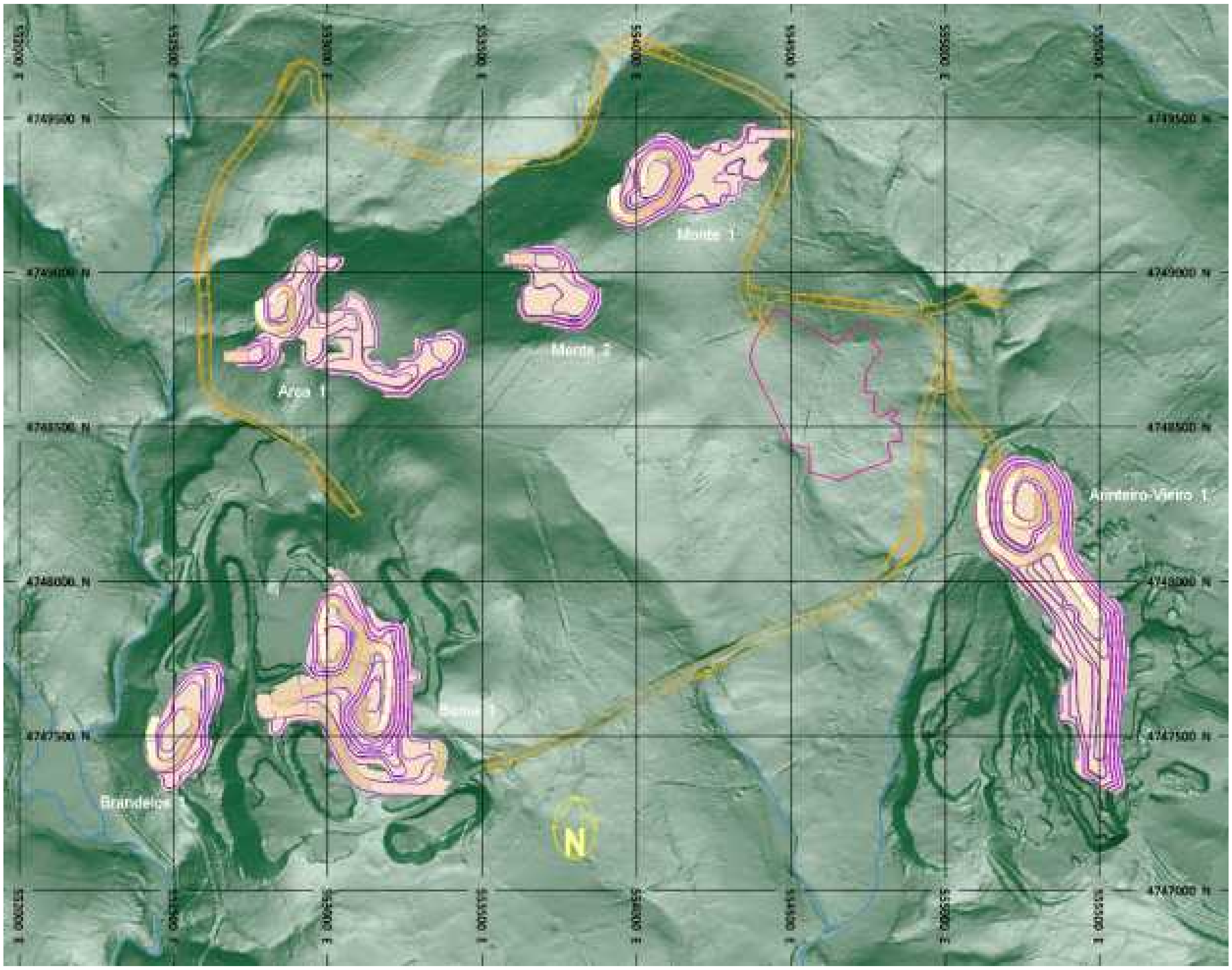


Figure 16.1 – Initial Mining Phases (WLRC 2017)

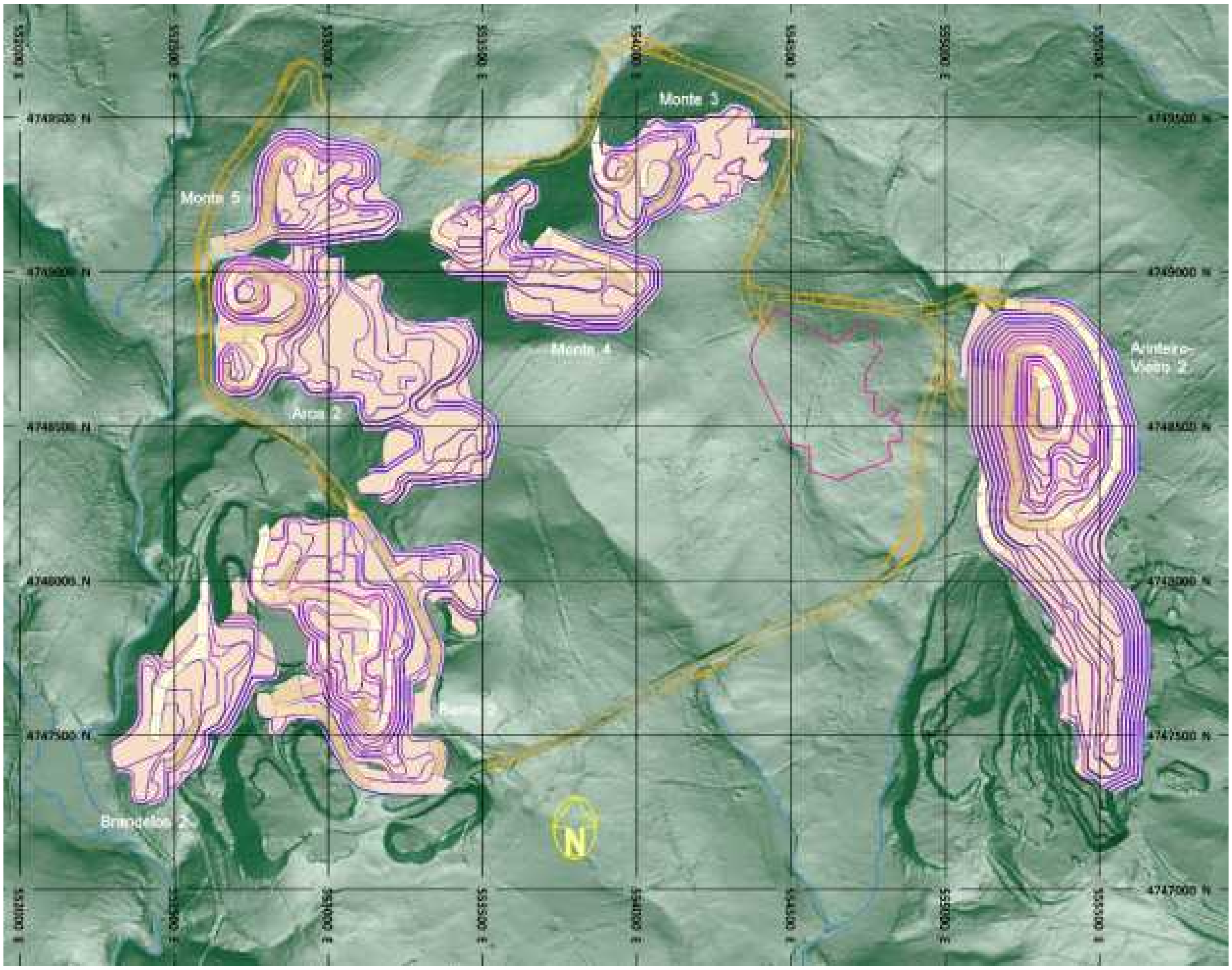


Figure 16.2 – Ultimate Pit Limits (WLRC 2017)

16.2 Mine Production Schedule

A proprietary open pit mining simulation program was used to generate production schedules to meet mill feed targets based on user specified cutoff grades and the mining phases described in the previous section. Mill feed targets and cutoff grades can both be varied by time period. The program, through additional user controls, computes advanced stripping requirements to ensure sufficient ore exposure throughout the schedule. As the deposit block model was developed to incorporate the effects of mining dilution and ore loss, no additional adjustments were made in the generation of the mine production schedule.

16.2.1 Production Scheduling Parameters

The production schedule targets an annual recoverable copper production rate of about 30,000 t. Net smelter return (NSR) cutoff grades were elevated in the first five years of operation to improve the present value of the project. NSR cutoffs were $10.00/t for preproduction stripping through Year 2, $14.00/t in Year 3, $12.00/t in Year 4, $10.00/t in Year 5 and then lowered to the average internal $8.14/t NSR cutoff for all remaining years. The internal cutoff is consistent with industry practices for reserve definition as the mining cost is excluded from its calculation. (The economic pit limit analysis, which includes all operating and off-site costs, determines which material must be mined. The internal cutoff then maximizes cash flow within the pit by treating as ore all measured and indicated mineral resources that cover everything but the mining cost.)

Mill feed targets during the first five years were limited to a maximum of 6.0 Mt/a, with an 85% adjustment factor applied in Year 1 for ramp-up and debugging that yields 5.0 Mt. After an expansion in milling capacity, ore processing would gradually increase to a maximum rate of 10.0 Mt/a while not significantly exceeding the target of 30,000 t/a recoverable Cu production. Through Year 5, low grade reserves below the variable plant feed NSR cutoffs but above $9.00/t (which includes an allowance for rehandling costs), would be stockpiled until fed to the plant in Years 11 and 12.

Pit bottom sinking rates were limited to about four benches per year. Typically, mining activity would occur in 4-5 phases during each year to smooth fluctuations in head grade and stripping rates. Two to three phases would be active at any given time.

16.2.2 Mine Production Schedule Summary

Preproduction stripping, totaling about 35.7 Mt, would commence about 18-24 months prior to mill startup depending on the details of sequencing waste rock placement in the initial tailings dam. Only the Arinteiro-Vieiro would be active during this period.

Peak stripping rates would extend through the latter part of year 3. Mining from the Arinteiro-Vieiro pit comprises just over 90% of the total material movement through the end of Year 3 and 85% through the end of year 5. Over 3.0 Mt of low grade ore would be stockpiled from preproduction stripping through Year 5. Total daily material movement during Years 1-3 would average nearly 97,000 t, assuming a mine operating schedule of 360 days per year.

Stripping rates fall off during years 4-6, but would increase afterwards to open up Arca 2, Bama 2, and the remaining Monte phases. Ore mining and processing rates would increase after Year 5 to compensate for lower head grades. Arinteiro-Vieiro would be mined out in the third quarter of Year 6. Stockpiled low grade ore would be fed to the plant in Years 11 and 12 to augment ore production from the remaining mining phases – primarily Arca 2, Monte 4, and Monte 5. Total daily material movement during Years 7-11 would range between 62,000 and 71,000 t/d (at 360 operating days per year).

Table 16.4 summarizes the mine production schedule and stockpile rehandling. The mine life is estimated at just over 12 years, excluding the preproduction stripping period.

Table 16.4 – Mine Production Schedule

Time	Cutoff	Direct Mine to Plant >= Cutoff				To ROM Stockpile >= Cutoff				To LG Stockpile >= 9.00 NSR $/t				From LG Stkpl to Plant >= 9.00 NSR				Total Plant Feed				Tonnes of Copper		Waste	Total	Strip
Period	NSR $/t	ktonnes	NSR $/t	Cu%	RCu%	ktonnes	NSR $/t	Cu%	RCu%	ktonnes	NSR $/t	Cu%	RCu%	ktonnes	NSR $/t	Cu%	RCu%	ktonnes	NSR $/t	Cu%	RCu%	Contained	Recoverable	ktonnes	ktonnes	Ratio
PP	10.00					449	21.23	0.49	0.44	12	9.63	0.24	0.20											35,201	35,662	999.99
1*	10.00	5,100	24.13	0.55	0.50					184	9.52	0.23	0.20					5,100	24.13	0.55	0.50	28,203	25,347	29,646	34,930	5.81
2	10.00	6,000	23.08	0.53	0.48					312	9.46	0.23	0.20					6,000	23.08	0.53	0.48	31,620	28,500	28,618	34,930	4.77
3	14.00	6,000	23.04	0.53	0.47					1,488	11.50	0.27	0.24					6,000	23.04	0.53	0.47	31,740	28,440	23,762	31,250	3.96
4	12.00	5,700	25.60	0.58	0.53					852	10.60	0.25	0.22					5,700	25.60	0.58	0.53	32,832	30,039	8,448	15,000	1.48
5	10.00	5,500	27.29	0.61	0.56					172	9.55	0.24	0.20					5,500	27.29	0.61	0.56	33,660	30,910	9,128	14,800	1.66
6	8.14	7,200	20.34	0.48	0.42													7,200	20.34	0.48	0.42	34,200	30,168	9,300	16,500	1.29
7	8.14	8,500	16.99	0.40	0.35													8,500	16.99	0.40	0.35	34,255	29,750	13,850	22,350	1.63
8	8.14	9,000	16.27	0.39	0.34													9,000	16.27	0.39	0.34	34,920	30,150	13,850	22,850	1.54
9	8.14	10,000	14.10	0.34	0.29													10,000	14.10	0.34	0.29	34,100	29,000	13,850	23,850	1.39
10	8.14	10,000	14.43	0.35	0.30													10,000	14.43	0.35	0.30	34,800	29,700	15,700	25,700	1.57
11	8.14	8,797	14.28	0.34	0.29									1,203	10.79	0.26	0.22	10,000	13.86	0.33	0.29	33,358	28,538	15,646	25,646	1.56
12	8.14	5,725	16.15	0.39	0.33									1,817	10.79	0.26	0.22	7,542	14.86	0.35	0.31	26,718	23,047	4,180	11,722	0.55
13	8.14	364	14.66	0.35	0.30													364	14.66	0.35	0.30	1,286	1,100	150	514	0.41
Total		**87,886**	**18.68**	**0.44**	**0.38**	**449**	**21.23**	**0.49**	**0.44**	**3,020**	**10.79**	**0.26**	**0.22**	**3,020**	**10.79**	**0.26**	**0.22**	**90,906**	**18.42**	**0.43**	**0.38**	**391,691**	**344,689**	**221,329**	**315,704**	**2.43**

Notes: NSR values are in US dollars base on a Cu price of $2.60/lb.

* Includes ROM stockpile reclamation in Y1 direct mine to plant.

16.3 Waste Rock Storage Facilities

Potential ex-pit waste rock storage facility (WRSF) sites are located to the east of phase Monte 3, southeast and southwest of the Arinteiro pit, north of the Bama and Brandelos pits, and south of the Bama and Brandelos pits. Some portions of the Bama 1, Bama 2, and Arca 2 phases would be available in the later years of mining for backfilling with waste rock. Additionally, some waste rock in the first several years would be directed to constructing a tailings dam for the first tailings storage facility (TSF).

16.3.1 WRSF Design Parameters

Ex-pit WRSFs will be constructed from the bottom levels upward in 10- to 20-m-high lifts. Drainage systems will be constructed in the foundations of the ex-pit WRSF areas prior to waste rock placement. Drainage from the WRSFs will be channelled to sedimentation and water treatment ponds.

At this stage of study (prefeasibility), the WRSF designs are conceptual only and do not show catch benches or internal ramps. The intent is to show likely sites, estimate potential storage capacities and provide preliminary waste rock destinations for haulage profile measurements. More detailed designs must await the establishment of criteria to differentiate potentially acid generating (PAG) from non-acid generating (NAG) waste rock, which may have differing design requirements for storage facilities. The design parameters of the conceptual WRSFs are summarized in Table 16.5.

Since the completion of mine planning work, Golder Associates have developed new waste rock storage plans that incorporate more environmental constraints and accommodate their tailings storage plans.

Table 16.5 – Conceptual WRSF Design Parameters

Parameter	Unit	Value
Net swell factors	%	25 - 30
WRSF/TDam densities	t/m^3	2.18 – 2.26
Lift height (WRSFs)	m	10-20
Angle of repose	degrees	37
Overall slope angle	degrees	28.7

The overall slope angle for WRSFs is based on a 5-m catch bench on 20-m vertical intervals and incorporates one 30-m-wide road on the face of a 120-m-high WRSF.

16.3.2 Ultimate WRSF Plans

Figure 16.3 shows the locations of the WRSFs with respect to the open pits, ore processing plant area, low grade ore stockpile, and the main haulage roads. The San Rafael mining concession boundary is shown in yellow around the perimeter of the project. The grid lines are on 500-m intervals. WRSFs required for the mine production schedule are shown in blue-grey and potential additional backfill areas (which become available in Years 11-12) are shown in dark yellow.

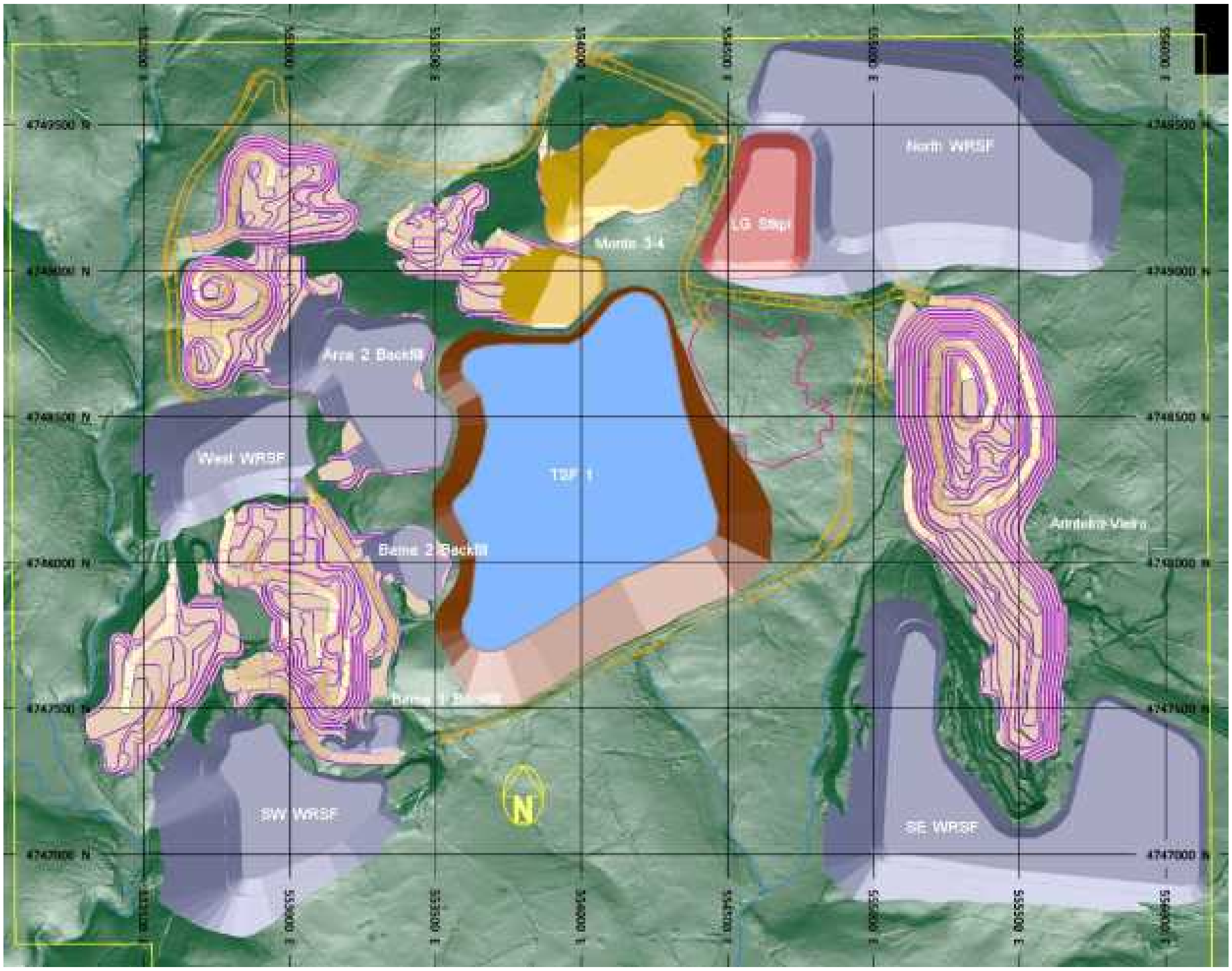


Figure 16.3 – Waste Rock Storage Facility Locations (WLRC 2017)

TSF 1 would be active through Year 7 in the production schedule, after which tailings would be placed in the mined-out Arinteiro open pit. Tailings backfills in open pits will require placement of suitable liner foundation material, which is not presently included in the mine production schedule.

16.3.3 WRSF Capacity Estimate

Table 16.6 summarizes the estimated WRSF and low-grade ore stockpile capacities by location. Total storage capacity for the WRSFs, excluding the Monte 3-4 backfill, is estimated at 233 Mt. The mine production schedule requires storage of approximately 221 Mt. If need be, additional storage capacity is available by constructing more lifts atop the North, SE, and SW WRSFs and/or developing the Monte 3-4 backfills (24 Mt).

Table 16.6 – WRSF and Low-Grade Ore Stockpile Capacities

Facility	Crest Elev m	Volume m^3 x 1000	Density t/m^3	Tonnage x 1000
Low-Grade Ore Stockpile	430	3,450	2.18	7,500
TSF 1 Embankment	419	20,800	2.26	47,000
WRSF North	430	36,100	2.18	78,700
WRSF Southeast (SE)	390	19,800	2.18	43,200
WRSF Southwest (SW)	360	11,900	2.18	25,900
WRSF West	310	5,400	2.18	11,800
Backfill Arca 2	370	10,000	2.18	21,800
Backfill Bama 1	350	400	2.18	900
Backfill Bama 2	380	1,500	2.18	3,300
Totals (excluding LG stockpile)		**105,900**		**232,600**

16.4 Main Haul Roads

Two main haul roads will be constructed during the preproduction stripping period. The East haul road will connect the Arinteiro-Vieiro pits with the primary crusher, LG ore stockpile, and the North WRSF. The South haul road will provide haulage access to the TSF-1 dam and later to the Bama and Brandelos pit. The West haul road will be constructed in Year 1 and will connect the Acra, Brandelos, and Monte pits to the primary crusher and LG ore stockpile area.

The haul roads will be 26 m wide, including safety berms and ditches, and have a maximum gradient of 10%. The central portion of the South haul road will be widened to 35 m, where a public roadway will be separated from mine traffic by a large safety berm and fence. Table 16.7 summarizes the estimated cut and fill quantities for each haul road. These quantities are in addition to the mine production schedule.

Table 16.7 – Estimated Haul Road Earthworks

Haul Road	Cuts m^3	Fills m^3	Length km	Area ha
East	23,000	76,000	1.7	5.0
West	187,000	229,000	4.7	15.6
South	54,000	93,000	2.2	7.8
Totals	**264,000**	**398,000**	**8.6**	**28.4**

16.5 Mining Equipment

Peak mining rates during Years 1-3 are projected at approximately 35 Mt/a, which equates to about 97,000 t/d if 360 working days per year are scheduled or just over 134,000 t/d if only 260 days per year are worked by the mining contractors.

The mining contractors will provide all the primary and auxiliary equipment fleets to meet the mine production schedule, build and maintain all roads, suppress dust from haul roads and muck piles, construct and maintain all WRSFs, perform equipment repairs and maintenance, and conduct all other activities normally associated with mine operations. Contractor equipment fleets and manning levels will be adjusted as necessary to meet mine production targets.

Tables 16.8 and 16.9 summarize the mining contractors' current primary and auxiliary mining equipment fleets, respectively (commercial brands for reference only).

Table 16.8 – Primary Mining Fleet (Y1-Y3)

Equipment	Capacity	Quantity
Drills:		
Sandvik DP 1500i class	114-127 mm diameter	3
Sandvik DP 1100i class	102 mm diameter	2
Hydraulic Excavators:		
180- to 200-t (e.g., Komatsu 2000)	10-12 m^3	2
110- to 120-t (e.g., Komatsu 1250)	6-8 m^3	2
Front-End Loader:		
512 kW (e.g., Caterpillar 990)	8 m^3	1
Haul Trucks:		
780 kW (e.g., Komatsu HD785)	91 t	21-27

Table 16.9 – Auxiliary Mining Fleet (Y1-Y3)

Equipment	Quantity
Wheel dozer, Caterpillar 824-class	1
Track dozer, Caterpillar D8T-class	2-3
Hydraulic backhoe, 50- to 60-t, 2.5-3 m^3	1
Articulated truck, 36- to 40-t	2-3
Motor grader, Caterpillar 14-class	2
Vibratory compactor, 30-t	1
Water truck, 30,000- to 50,000-liter	2

The contractor's ancillary equipment will include hydraulic cranes, maintenance trucks, equipment transporters (i.e., tractor and low-boy trailer), forklifts, portable welders, portable light plants, personnel transport vans, 4x4 pickup trucks, and radio communication systems. Atalaya Mining will provide fuel and lubricant storage tanks and a maintenance shop with warehouse, change rooms, and staff offices.

16.6 Mining Personnel

Mining contractor personnel will be devoted to supervision and craft labor – i.e., mine operations and equipment maintenance. The work of Atalaya Mining's mine department employees will be limited to contract management, safety and supervision tasks, and most technical services (engineering, geology, etc.). Table 16.10 summarizes the estimated levels of mining-related personnel.

Table 16.10 – Mining-Related Personnel (Y1-Y3)

Worker Type	Quantity
Contractor supervision & support staff	8
Contractor craft labor	78-88
Subtotal contractor personnel	86-96
Owner (Atalaya Mining) personnel	15
Total mining personnel	**101-111**

17 RECOVERY METHODS

17.1 Flowsheet Development

As outlined in Chapter 13.2, flowsheet development was based on the testwork reported by Wardell Armstrong International (WAI) to produce a concentrate grade of 27% Cu at 90% recovery. Locked cycle tests completed during the latest testwork program produced concentrates that averaged 29.1% Cu at 87.0% recovery indicating that this is achievable. Refer to Chapter 13 for details and results of the testwork program.

The proposed process flowsheet uses a conventional SAG mill - ball mill (SAB) grinding circuit followed by a copper flotation recovery circuit. The initial concentrator includes:

- Primary crushing.
- Primary and secondary grinding.
- Rougher flotation.
- Regrinding.
- Three stages of cleaner flotation.
- Concentrate thickening and filtration.

The concentrator and associated service facilities will process run-of-mine (ROM) ore as delivered to the ROM pad to produce a dewatered copper concentrate and tailings slurry. The preliminary mining schedule is outlined as follows:

- In the first year of operation, annual ore movement to the plant ramps up to 5.1 million tonnes.
- In the following two years of operation, the peak annual ore movement to the plant is 6.0 million tonnes.
- From year 8 on, the annual ore movement to the plant ramps up again to 10.0 million tonnes.
- Over the life of mine, the peak annual ore movement to the plant is 10 million tonnes.

To avoid three successive plant upgrades, a simplified approach has been taken with only two design points and upgrades considered for the process plant:

- Initial Phase 1 plant throughput of 6.0 Mt/y.
- Phase 2 upgrade to increase plant throughput of 10 Mt/y prior to year 8.

The Phase 1 average head grade is 0.53% Cu with a peak of 0.57% Cu. The Phase 2 average head grade is 0.35% Cu with a peak of 0.38% Cu. The overall result is consistent concentrate and copper production for both phases. A simplified process flow diagram (Figure 17.10 shows the 6.0 Mt/y Phase 1 in black and the 10 Mt/y Phase 2 additions in blue).

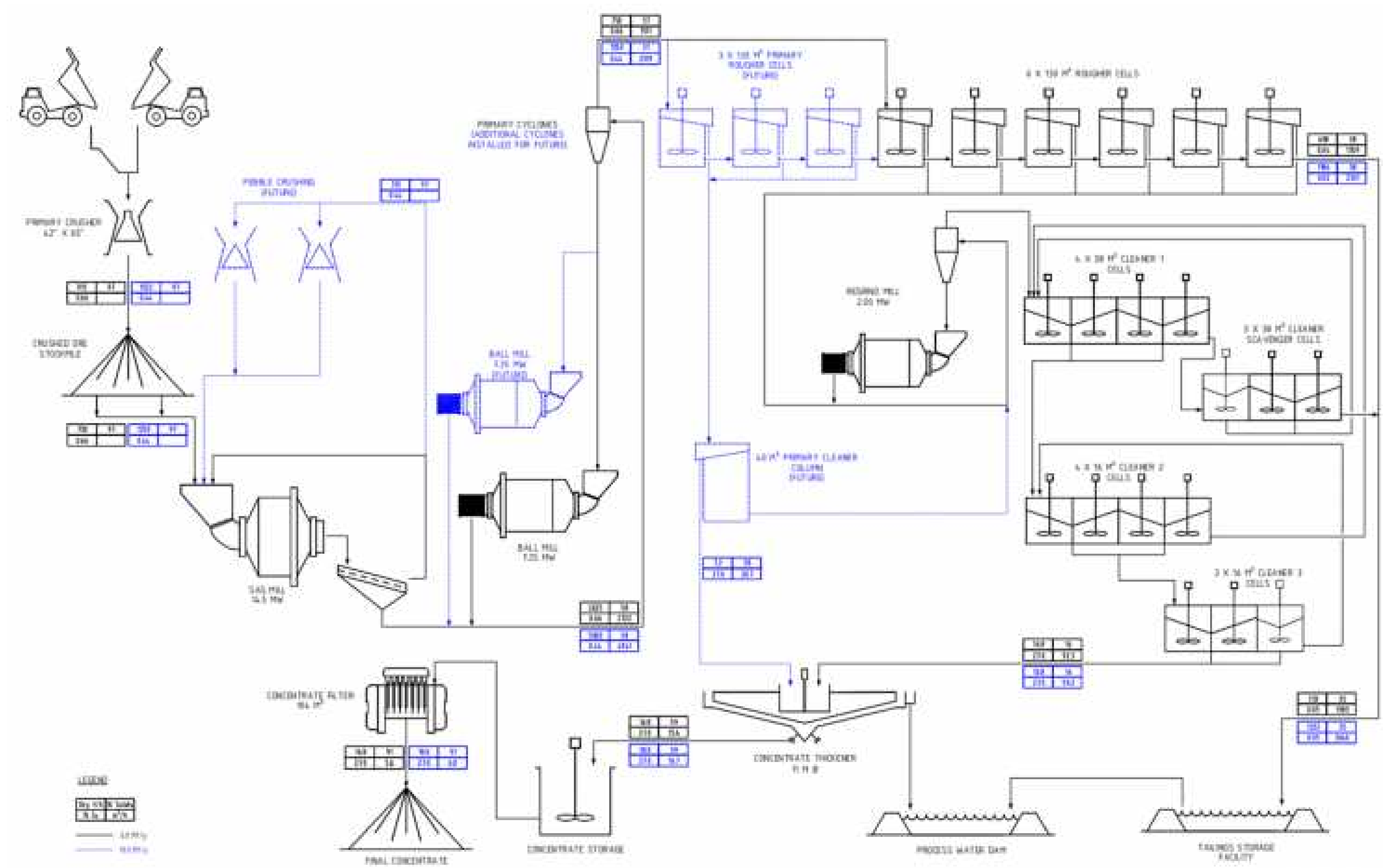


Figure 17.1 – Simplified Process Flow Diagram (Minnovo 2016)

17.2 Plant Design Basics

Ore for the concentrator is sourced from a number of open pits. The Project life of mine (LOM) is currently 13 years and could be extended if additional ore resources are identified. The production schedule envisages a pre-production ramp-up of the process facility in the first eight months followed by 13 years of mining.

In Phase 1, the plant is designed to process up to 6.0 million dry metric tonnes per year of copper sulfide ore at a design plant feed grade of 0.66% Cu, with 3% (nominal) moisture content. The designed copper concentrate production is 134,742 tonnes per year based on a concentrate grade of 27% Cu and 92.5% overall copper recovery. A small amount of gold and silver is also recovered in the copper concentrate, but as noted in Chapter 13, recoveries are considered to be at or lower than the minimum level for smelter credits.

In Phase 2, the plant throughput capacity is increased to 10 Mt/y at a reduced design plant feed grade of 0.44% Cu at 89% copper recovery, for 144,048 t/y of concentrate.

Based on plant utilization of 91.3%, the design plant throughput is at 16,438 t/d (750 t/h) in Phase 1 and 27,397 t/d (1,250 t/h) in Phase 2. A summary of the key process design criteria is shown in Table 17.1.

Table 17.1 – Key Process Design Criteria

Parameter	Unit	Design	
		6.0 Mt/y	10 Mt/y
Annual feed rate	dry t/y	6 000 000	10 000 000
Crushing availability	%	75	75
Crushing operating hours	h/y	6 570	6 570
Concentrator availability	%	91.3	91.3
Concentrator operating hours	h/y	8 000	8 000
Filtration availability	%	80	80
Filtration operating hours	h/y	7 008	7 008
Ore design grade	% Cu	0.66	0.44
Concentrate grade	% Cu	27.0	27.0
Cu recovery	%	92.5[10]	89.0
Mass recovery (% new feed)	%	2.2	1.4
Concentrate production	t/y (dry)	134 742	144 048

The values in Table 17.1are taken from the Process Design Criteria (PDC), developed as part of the overall plant design, and which give the input values used in the mass and metallurgical balances.

The concentrator plant is designed as a greenfield, stand-alone facility. The previous operation on site closed in 1986 and most of the processing equipment has since been removed. No re-use of the previous plant is envisaged. The mechanical equipment list includes estimated dimensions, installed power and consumed power. Equipment design flows are nominal flows with a margin for surge conditions.

All slurry pumps are designed with duty-standby pairs, fitted with variable frequency drives (VFD). Slurry pumps will be a mixture of hard metal pumps for large particle size duties and rubber-lined for the smaller particle size duties.

17.2.1 Concentrator Location

The concentrator location has been based on an assessment of the open pit distances and geographical features. A preliminary location situated approximately 500 m to the west of the current Vieiro and Arinteiro deposits has been used for the study. This is adjacent to the proposed Tailings Management Area (TMA) and the pond from which plant water will be drawn. It is also conveniently located to connect to the existing 220 kV power line to the south of the mine which will supply power to the new facilities.

[10] The recovery at the design head grade is used to estimate the maximum likely concentrate production on high grade ore for sizing of the concentrate handling equipment. It is not an estimate of average recovery.

17.3 Equipment Sizing

17.3.1 Primary Crushing and Coarse Ore Stockpile

The primary crushing circuit has been designed to meet the Phase 2 production capacity at a utilization of 75%. This equates to a nominal average capacity of 1,522 t/h. A 42 x 65 inch gyratory crusher has been selected for the duty, with a conservative maximum capacity of 1,600 t/h. This crusher will accept a top size of 1,000 mm, and produce a product size P_{80} of approximately 125 mm at a closed side setting of 150 mm.

The ROM bin feeding the primary crusher is designed for direct truck dump from two dump points and is designed with a capacity of 200 t. The discharge feeder and stockpile feed conveyors are designed for 2,000 t/h providing peak surge capacity.

Primary crushed ore is stored on an open, conical stockpile of 63 m base diameter and 43º angle of repose. The live capacity is 18,313 t, providing 20 and 12 hour milling capacity buffer for Phases 1 and 2 respectively. Stockpile reclaim is via two gravity reclaim apron feeders of 1,250 t/h capacity each.

17.3.2 Grinding Circuit

17.3.2.1 Primary Grind Size Economic Analysis

Simple economic analysis was completed to determine the optimum primary grind size, based on the recoveries from the rougher flotation grind optimisation tests.

The analysis described below uses a plant throughput of 5 Mt/y which has since changed to 6 Mt/y. Despite the change, the outcome is still considered valid and therefore the calculations have not been repeated.

The economic evaluation was based on the following assumptions:

- Throughput: 625 t/h, equivalent to 5 Mt/y.
- SAG-ball (SAB) circuit.
- Head grade: Relevant composite head assay.
- Primary grind sizes (P_{80}) of 150, 125 and 106 µm.
- Power cost: US$ 0.09 /kWh.
- The ore type work index (refer table below)
- Axb: 34.0.
- Abrasion index: 0.19 g.
- Copper price: US$ 2.75 /lb at 75% payability.
- Grinding media cost: US$ 1,000/t.
- Bare mill cost: US$ 0.7 M/MW.
- Installed mill cost factor: 2.3.

Rougher flotation recovery decreased significantly for the majority of the ores at the coarser P_{80} of 150 µm. As such, the economic comparison was focused on grind sizes of 106 µm and 125 µm as summarised in Table 17.2.

Table 17.2 – Primary Grind Size Economic Analysis

Composite	Work Index kWh/t	Revenue (US$ M)			Capital Cost (US$ M)			Payback (years)
		125 µm	106 µm	Δ	125 µm	106 µm	Δ	
Arca	14.9	74.6	74.8	+0.2	11.9	14.2	2.3	11.2
Vieiro	14.7	117.5	118.2	+0.7	11.7	14.0	2.3	3.3
Arinteiro	16.2	93.3	93.8	+0.5	12.9	15.4	2.5	5.3
Bama	16.1	76.1	75.9	-0.2	12.8	15.3	2.5	-
Brandelos	15.9	71.2	70.9	-0.3	12.7	15.1	2.4	-
Monte Minas Garnetite	14.5	90.6	91.4	0.8	11.5	13.8	2.3	3.1
Arca A	15.2	63.8	63.5	-0.3	12.1	14.5	2.4	-
Monte Minas Upper	17.0	84.9	85.3	0.4	13.5	16.2	2.7	5.8
Vieiro High Grade	15.7	268.3	268.0	-0.3	12.5	14.9	2.4	-

Payback was determined based on improved revenue versus the additional capital cost. No consideration was made for net present value (NPV) in the simple economic analysis.

Where the revenue achieved at the finer grind size, P_{80} 106 µm, is less than the revenue at P_{80} 125 µm, the payback cannot be achieved due to the higher capital cost required to achieve the P_{80} 106 µm grind size.

For the ore types where there is an increase in revenue at P_{80} 106 µm, the payback ranges from 3.1 to 11.2 years, which are considered uneconomic for a project with capital cost constraints. Based on this, a primary grind P_{80} of 125 µm has been selected for the study.

17.3.2.2 Grinding Circuit Configuration and Selection

The Project requires a low capital cost grinding circuit that is readily expandable to match the phased expansion plan. Based on case studies conducted on other projects with similar ore properties, Semi-autogenous grinding (SAG) typically represents the lowest capital cost and lowest operating cost options for projects in the 6 Mt/y throughput range with these ore properties, and can be easily expanded to 10 Mt/y.

While other options such as multi stage crushing and high pressure grinding rolls (HPGR) could be applicable, capital costs for these options are likely to be significantly higher than SAG milling, and given that the capital cost is a key criterion for the project, these options were rejected. In addition, since the ore is consistently hard and abrasive, Autogenous (AG) milling with pebble crushing was also considered but was deemed high in capital cost and was not considered further.

A preliminary analysis of two comminution circuit options was selected for more detailed assessment:

1. SAG-Ball Mill circuit (SAB) at 5.0 Mt/y expandable to 8 Mt/y with the addition of pebble crushing and a second ball mill (SABBC).
2. AG-Ball mill-Pebble Crush circuit (ABC) at 5.0 Mt/y expandable to 8 Mt/y with addition of steel in the SAG mill and a second ball mill (SABBC).

The comminution circuit options described above used a plant throughput of 5 Mt/y expandable to 8 Mt/y. This has since changed to 6 Mt/y expandable to 10 Mt/y. Despite the change, the outcome is still considered valid and the calculations have not been repeated.

A trade-off study has been completed to determine the most suitable comminution circuit design.

17.3.2.3 Grinding Circuit Design

This section details the design and equipment sizing for the 6.0 Mt/y SAB circuit, expanding to a SABC circuit at 10 Mt/y. Both the 6.0 Mt/y and 10 Mt/y designs are presented, with key design values presented in Table 17.3. Minnovo uses power-based grinding models built up over 20 years from numerous operating plants, pilot plants and laboratory data. The design methodology is discussed below.

Table 17.3 – Mill Design Parameters

Parameter	Units	6.0 Mt/y	10 Mt/y
Ore Characteristics			
Axb (25th percentile)		34.0	
RWI (75th percentile)	kWh/t	18.4	
BWI (75th percentile)	kWh/t	16.5	
SG (average)	t/m^3	3.20	
Circuit Input Data			
F_{80}	mm	128	128
P_{80}	µm	125	125
Throughput rate	t/h	750	1250
Circuit type		SAB	SABC
Power Requirement			
Circuit efficiency		1.45	1.38
Base specific power	kWh/t	14.8	14.80
Total specific power	kWh/t	21.86	19.89
SAG mill specific power	kWh/t	11.77	9.78
Ball mill specific power	kWh/t	10.08	10.11
Pebble crusher specific power	kWh/t	n/a	0.23
SAG Mill Sizing			
SAG mill pinion power	kW	10 085	12 219
SAG installed power (twin pinion)	kW	14 500	14 500
SAG shell size (IS, diameter x EGL)	m	10.36 x 6.64	10.36 x 6.64
SAG charge, design, ball / total	%	5 / 30	12 / 26
SAG charge, maximum, ball / total	%	15 / 35	15 / 35
Ball Mill Sizing			
Ball mill pinion power	kW	6 354	12 709
Ball mill installed power	kW	7 250	2 off, 7 250
Ball mill shell size (IS, diameter x EGL)	m	6.25 x 9.8	2 off, 6.25 x 9.8
Ball charge, design	%	30	30
Ball charge, maximum	%	35	35

Pebble Crusher Sizing			
Pebble crusher installed power	kW	n/a	2 x 200
Pebble crusher consumed power	kW	n/a	230

The 75th percentile values (25th percentile for Axb, since low Axb values equate to harder materials) have been selected for design purposes for both phases (refer Chapter 13.4.4). Comparison of the early scheduled deposits to the later schedule deposits showed no significant variance in ore hardness between the two phases of operation.

The primary mill feed size is estimated from a Minnovo model that predicts the crusher product size based on the crusher gap setting and the ore SMC drop weight index.

The base specific power is calculated from the Bond work index values on the basis of ore being reduced from feed size to product size. The required overall specific power is calculated from the base specific power and the circuit efficiency. However, Minnovo's efficiency factor only applies down to 150 µm, below which the uncorrected Bond value is used. This gives the total circuit power requirement.

The SAG mill power component of the total power is determined based on the Axb value. The remaining circuit power is the ball mill power requirement. Pebble crusher power is calculated separately using the standard Bond calculation, and is based on the pebble extraction rate and a crushing work index value 50% greater than that for the ore to allow for the increased hardness of pebbles.

SAG mill size selection is based on the 10 Mt/y requirement for 12,219 kW pinion power draw. A 10.36 m diameter and 6.64 m effective grinding length (EGL) mill is selected with 14,500 kW installed power. This design provides for 18% contingency for charge fluctuations, increased ball loadings for harder ore at 10 Mt/y and significantly more at 6.0 Mt/y. In Phase 2 the grates are pebble ported to allow the extraction of approximately 25% pebble recirculating load.

Ball mill size selection is based on the 6.0 Mt/y requirements, which is duplicated with a second identical mill to meet the 10 Mt/y requirements. A 6.25 m diameter and 9.8 m EGL overflow ball mill is selected with 7,250 kW motor power providing design contingency of 14%. The design incorporates partial return of cyclone underflow to the SAG mill feed in order to assist in circuit power balancing. At 10 Mt/y with two identical mills there is 14% contingency.

The SAG mill is specified as a twin pinion drive powered by two 7,250 kW motors, common with the ball mill motor for critical spares reduction.

Atalaya has two Symons cone crushers available for use on the Touro project. Two 5½ foot Symons pebble crushers with 200 kW motors have been selected at the 10 Mt/y throughput. The Symons crusher are substantially more robust and therefore better suited to the arduous pebble crushing duty verses the more modern higher capacity, lighter weight cone crushers. The crushers are converted from spring to hydraulic tramp relief to provide greater protection from tramp steel grinding media. One crusher is operated continuously choke fed, while the second operates intermittently choke fed to regulate the feed bin level. This operating strategy minimises SAG mill load disturbance, maximizes pebble crusher size reduction, and minimizes pebble crusher wear costs.

SAG mill screen and ball mill trommel undersize slurry is classified in a cluster of 660 mm diameter hydrocyclones to produce the flotation circuit feed with P_{80} 125 µm at 37% solids. The cyclone underflow recirculates to the ball mill. As mentioned above, a portion of cyclone underflow is redirected to the SAG mill in Phase 1 to balance the circuit. Phase 1 has 10 cyclones installed with nominally 7 operating. This increases to 14 installed and 11 operating in Phase 2. It should be noted that Phase 1 has a 14 distributor cyclone cluster such that it does not require upgrading in Phase 2.

17.3.3 Flotation Circuit Design and Sizing

17.3.3.1 Rougher Flotation Residence Time Economic Analysis

The analysis described below uses a plant throughput of 5 Mt/y and 100 m³ rougher float cells, which have since changed to 6 Mt/y and 130 m³ rougher float cells. Despite these changes, the outcome is still considered valid and therefore the calculations have not been repeated at the current conditions.

A simple economic analysis was completed to determine the optimum rougher residence time, based on the recoveries from the rougher grind optimisation tests. The economic evaluation was based on the following inputs:

- Throughput: 625 t/h equivalent to 5 Mt/y.
- Head grade: Relevant composite head assay.
- Recovery of rougher concentrate to final concentrate: 75%.
- Power cost: US$ 0.09 /kWh.
- Cell size: 100 m³.
- Cell power: 110 kW.
- Cell power draw: 1 kW/m³.
- Maintenance cost: 4% of installed capital cost.
- Copper price: US$ 2.75/lb at 75% payability.
- Cell capital cost: US$ 0.26 M.
- Installed cell factor: 2.5.

The results of the analysis are detailed by Table 17.4 and Figure 17.2. The flotation volume required to achieve the equivalent of 6 minutes residence laboratory residency time was considered the base case. The incremental additional volume required to achieve the equivalent of 10 minutes and 14 minutes laboratory residence time was evaluated.

Table 17.4 – Rougher Flotation Residence Time Economic Analysis

Composite	Recovery (%) / (Δ%)			Δ Capital Cost (US$ M)		Payback (years)	
	6 Mins	10 Mins	14 Mins	10 Mins	14 Mins	10 Mins	14 Mins
Arca	88.0	+1.46	+0.73	+1.3	+1.3	1.1	2.7
Vieiro	91.2	+1.15	+0.52	+1.3	+1.3	1.0	2.5
Arinteiro	89.7	+0.90	+0.59	+1.3	+1.3	1.6	2.8
Bama	89.0	+0.83	+0.52	+1.3	+1.3	2.2	4.1
Brandelos	87.7	+1.04	+0.59	+1.3	+1.3	1.7	3. 9
Monte Minas Garnetite	90.2	+0.77	+0.44	+1.3	+1.3	2.0	4.7
Arca A	90.7	+1.01	+0.54	+1.3	+1.3	2.2	5. 8
Monte Minas Upper	88.1	+0.93	+0.48	+1.3	+1.3	1.6	4.2
Vieiro High Grade	95.6	+1.55	+0.85	+1.3	+1.3	1.0	2.0

The base case of 6 minutes laboratory equivalent residence time requires 4 x 100 m^3 cells, with the increase in residence time from 6 to 10 minutes requiring an additional 2 x 100 m^3 cells in all cases. To increase the laboratory equivalent residence time from 10 to 14 minutes requires a further 2 x 100 m^3 cells.

Increasing the laboratory equivalent residence time from 6 to 10 minutes results in an average simple payback period of 1.7 years which is considered economic. Increasing the time even further from 10 to 14 minutes, results in an average simple payback period of 3.8 years which is considered too long for a project with capital cost constraints.

Therefore, 10 minutes laboratory equivalent rougher flotation residence time has been selected for the study.

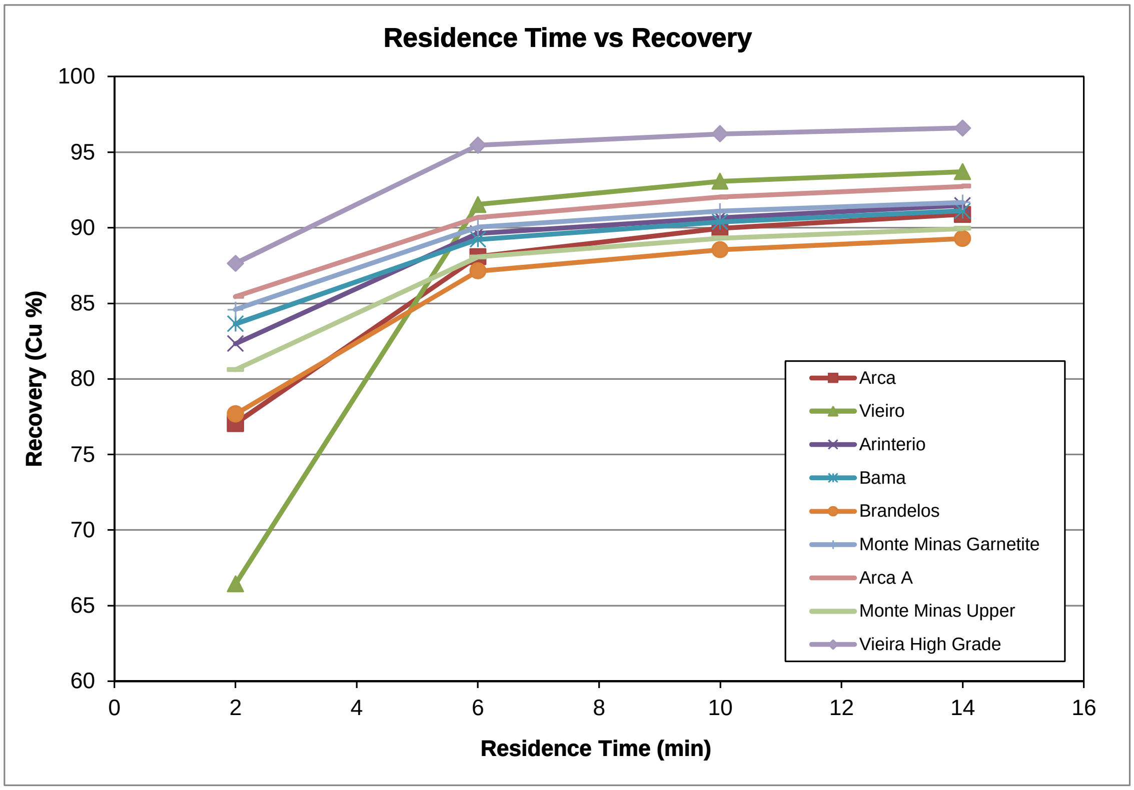


Figure 17.2 - All Composites Recovery vs. Residence Time (Touro testwork analysis, Minnovo 2016)

17.3.4 Flotation Circuit Design Basis

Sizing of the flotation circuit is based on a number of considerations including the following:

- Flotation testwork and in particular the locked cycle test results,
- Adoption of realistic plant scale stage recoveries as compared with laboratory scale test results, cognisant of plant issues such as short circuiting in short bank stages,
- Appropriate scale up of laboratory flotation times to plant flotation times,
- An effective flotation volume of 85%, accounting for the volume loss to mechanism, aeration, froth and launder volumes,
- Mass balance flowrates across the circuit, which incorporates stage recoveries, concentrate grades and flotation densities throughout the circuit,
- A design head grade of 0.66% Cu (Phase 1) and 0.44% Cu (Phase 2), based on a margin of 15% above the highest yearly average, to ensure the circuit is sized sufficiently to cope with shorter term peaks in feed grade,
- Cell volume calculated from the average of feed and tailings flowrates,
- Cell selections ensuring maximum lip loading and surface area carry rates do not exceed 1.5t/m.h or 1.5 t/m^2.h, respectively.
- The 6.0 Mt/y design is first discussed, followed by the expansion design to 10 Mt/y.

17.3.5 6.0 Mt/y Flotation Circuit Sizing

Table 17.5 presents a comparison between the locked cycle laboratory test conditions and results compared with that of the design. Significant points are discussed below.

Table 17.5 – 6.0 Mt/y Flotation Circuit Design

Design Parameter		Rougher	Cleaner 1	Clnr-scav	Cleaner 2	Cleaner 3
Retention time (minutes)	Actual lab LC tests	10	7.5	7.5	7	4
	Design laboratory	10	7.5	7.5	10	10
	Plant scale-up	2.5				
	Plant design	25	18.75	18.75	25	25
	Effective cell volume	85%				
Feed density (% solids)	Average LC tests	33	17	10	8	6
	Plant design	37	19.9	19.8	17.5	15.4
Stage recovery (% Cu)	Ave LC tests	88.5	97.7	39.7	97.3	96.9
	Plant design	95	96.1	50	80	75
Concentrate grade (% Cu)	Average LC tests	9.5	19.2	3.6	26.3	29.5
	Plant design	9.0	19.0	1.5	24	27
Cell selection	Cell size (m^3)	130	38	38	16	16
	Cell number	6	4	3	4	3
	Actual plant RT (minutes)	27.7	30.2	34.0	24.9	24.8

Laboratory locked cycle test results are good for interpreting overall circuit performance, however, they can significantly underestimate the recirculating loads (and therefore overestimate the stage recoveries) compared with full scale plant performance. This is due to a number of differences including short circuiting in full scale plants and significant changes in volume, surface area, lip length ratios, aeration and so on. Consequently the full scale design is based on adjustments to some of the laboratory values, based on Minnovo's experience including benchmarking against similar operating plants.

The rougher residence time was selected as 10 minutes lab equivalent, as discussed above (Rougher Flotation Residence Time Economic Analysis). A plant retention time scale-up factor of 2.5 is used throughout the design, as is an effective cell volume of 85%. The flow rate is calculated at the design cyclone overflow density of 37% solids, which is similar to the laboratory density. Six 130 m^3 tank cells are selected giving an actual plant residence time of 27.7 minutes.

For design purposes, the rougher copper recovery is higher than the test average. This is in line with the high design feed grade and ensures that downstream equipment is adequately sized.

The Cleaner 1 and Cleaner-scavenger cells are sized on the laboratory residence times, 2.5 scale-up factor, and 85% effective volume. The feed density is designed at 20% based on the design regrind cyclone overflow density, similar to the testwork value of 17%. The final Cleaner 1 selection of four 38 m^3 tank cells (2+2 configuration) provides 30 minutes actual residence time compared with the design requirement of 18.75. Similarly the Cleaner-scavenger final selection provides 34 minutes residence time compared with the 18.75 minutes design value. This provides contingency for greater circulating loads, higher throughput or lower density operation if required. The design case increases the Cleaner-scavenger stage recovery and reduces its concentrate grade compared to the locked cycle test results. This takes into account plant operation with higher residence time and larger circulating loads.

Both second and third stage cleaners are operated at higher density than the testwork. This reflects more typical copper flotation cleaner circuit operations and the fact that the laboratory is constrained by cell size and concentrate production.

The plant design increases the residence time to 10 minutes laboratory equivalent each, and reduces the stage recoveries from about 97% to more typical plant values of 80% and 75% for Cleaner 2 and 3, respectively. This ensures the design mass balance has a realistic recirculating load for cell sizing. Final cell selections provide close to 25 minutes residence time each, compared to the design required 25 minutes. Concentrate grades are in line with the testwork upgrade ratios, and adjusted to give the design target 27% final concentrate grade.

17.3.6 Regrind Mill Sizing

The design regrind product size is selected at P_{80} 20 µm, as discussed in Chapter 13.4.7. The key regrind mill design parameters are shown in Table 17.6 and discussed below.

Table 17.6 – Regrind Mill Design

Parameter	Units	6.0 Mt/y	10 Mt/y
Feed rate	t/h	51.9	44.7
F_{80}	µm	125	
P_{80}	µm	20	
Operating work index	kWh/t	20.6	
Specific power	kWh/t	27.7	
Pinion power	kW	1,436	1,237
Installed power	kW	2 000	
Mill size (inside diam x EGL)	m	4.27 x 6.83	
Ball charge (nom / max)	%	25 / 35	21 / 35

Minnovo uses a regrind mill sizing method based on benchmarking of other copper sulfide flotation concentrator regrind mills. This method indicates that for a regrind size of P_{80} 20 µm, the regrind mill operating work index is 1.25 times the feed ore ball mill work index. The regrind mill power and size are calculated based on this operating work index.

The design for this project has used the primary grind size as the regrind mill F_{80}. In practice, the actual rougher concentrate will be at least one root 2 screen series finer. This provides conservatism in the design. In addition, the Phase 1 design has a high design feed rate, due to its high design copper feed grade. This allows for finer regrinding if required during periods of average feed grade.

17.3.7 10 Mt/y Flotation Circuit Sizing

The 10 Mt/y flotation circuit design incorporates the following additional flotation capacity:

- Rougher flotation capacity is increased to retain the 6.0 Mt/y residence time by the installation of new cells ahead of the 6.0 Mt/y roughers, known as primary roughers, and
- A single stage of column flotation (the primary cleaner) to clean the primary rougher concentrate to final concentrate grade, and
- A reduction in the design copper grade to 0.44% Cu. This value is still 15% greater than the highest yearly average grade of 0.38%.

This expansion design, together with the reduced feed grade in Phase 2 allows the rest of the circuit to remain unchanged, thus providing an expansion circuit with minimal circuit disruption and tie-ins. Table 17.7 provides the key parameters for the Phase 2 flotation circuit, again compared back to the laboratory locked cycle test averages.

Table 17.7 – 10 Mt/y Flotation Circuit Design

Design Parameter		Primary Rougher	Rougher	Primary Cleaner	Cleaner 1	Cleaner-scavenger	Cleaner 2	Cleaner 3
Retention time (minutes)	Actual lab LC tests	10		n.a.	7.5	7.5	7	4
	Design lab	3.5	6.5	7.5	7.5	7.5	10	10
	Plant scale-up	2.5		4.0	2.5			
	Plant design	8.8	16.3	30	18.8	18.8	25	25
	Effective cell volume	85%						
Feed density (% solids)	Average LC tests	33		n.a.	17	10	8	6
	Plant design	37	37	27.5	19.9	19.8	17.5	15.4
Stage recovery (% Cu)	Ave LC tests	88.5		n.a.	97.7	39.7	97.3	96.9
	Plant design	50	90	71.2	85.6	50	80	75
Concentrate grade (% Cu)	Average LC tests	9.5		n.a.	19.2	3.6	26.3	29.5
	Plant design	14.0	5.5	27.0	19.0	1.5	24	27
Cell selection	Cell size (m^3)	130	130	40	38	38	16	16
	Cell number	3	6	1	4	3	4	3
	Actual plant RT (min)	8.0	16.5	43.2	22.9	23.3	38.9	38.7

The 10 Mt/y design is based on removing fast floating liberated chalcopyrite from the initial stages of roughing, and cleaning this portion of concentrate to final concentrate grade without regrind. This design is supported by the typical flotation response observed in the rougher testwork and is further supported by the mineralogical limiting grade recovery relationships shown in Chapter 13.4.3, where high grade liberated particles float fastest in the rougher bank. Figure 13.3 shows that more than 75% of the copper is mineralogically recoverable to a 27% concentrate grade without regrind.

The primary rougher is sized at 8 minutes actual residence time. This requires three additional 130 m^3 tank cells, installed ahead of the existing six cells which become the secondary roughers. Grade and recovery are estimated at 14.0% Cu and 50% respectively. These estimates are based on timed rougher concentrate grade recovery profile. The remaining rougher makes up the overall rougher residence time. Its grade and recovery are adjusted such that the overall rougher concentrate grade and recovery are unchanged (9.0% Cu and 95% recovery).

A column cell is selected for the primary cleaner duty taking advantage of the additional froth cleaning available in a column. A laboratory residence time of 7.5 minutes is used, the same as Cleaner 1. However, a more conservative scale up factor of 4.0 is used to size the column. A 2.5 m diameter by 8.0 m high forced aeration column was selected for the duty.

The primary cleaner tail and Secondary rougher concentrate combine as feed to the regrind and cleaning circuits which remain unchanged from the 6.0 Mt/y design. As shown in Tables 17.8 and 17.9, the reduction in feed grade and early removal of fast floating liberated material from the circuit result in a similar mass flow as in the 6.0 Mt/y Phase.

17.3.8 Concentrate Thickening and Filtration

The final concentrate is dewatered by thickening and filtration to a moisture suitable for bulk sea freight.

An 11 m diameter high-rate thickener has been selected based on standard industry values for flotation concentrates of 0.20 t/h/m^2 and the concentrate production from the higher 10 Mt/y throughput. The concentrate is thickened to nominally 59% w/w solids and pumped to an agitated concentrate filter feed tank that provides 12 hours storage capacity. The thickener overflow is pumped to the process water dam. Flocculant is used to aid in settling and is added at 40 g/t.

Concentrate thickener underflow is further dewatered using a vertical plate pressure filter. Filter cake at nominally 9% moisture is produced, with filtrate returned to the concentrate thickener. The filters have been sized based on filtration rates achieved at the Riotinto facility, with 18 plates selected for 97 m^2 filtration area. The filter area is increased to 104 m^2 with the addition of two more plates to accommodate the additional concentrate production at 10 Mt/y.

17.3.9 Flotation Tailings

Tailings from the rougher and cleaner scavenger flotation cells are pumped to the Tailings Storage Facility. The final settled tailings solids concentration is estimated to be 75% w/w, but will need to be verified by testwork. Tailings rheology will also need to be confirmed by testwork in order to confirm the size of the tailings pumps and pipeline. Water released by the settling tailings, as well as rain water collected in the TSF, will be decanted from the surface of the dam and pumped to the process water dam for use in the concentrator.

17.4 Process Plant Description

17.4.1 Primary Crushing

ROM ore is transported from the open pits to the primary crusher by mine haul trucks, which dump directly into a 200 t ROM bin positioned above the primary crusher, from either of two dump points. The bin has sufficient capacity for two truckloads. The ROM bin is equipped with a fixed rock breaker to reduce oversize material and remove potential blockages.

The ore is crushed by a 42 inch primary gyratory crusher with a 375 kW motor. Crushed product with a nominal P_{80} of 128 mm is removed from the crusher discharge chamber via a 1,350 mm wide belt feeder. Crushed ore transfers to the 1,200 mm wide crushed ore stockpile feed conveyor onto the crushed ore stockpile.

Spray nozzles are used around the ROM bin and transfer points to reduce fugitive dust. The crushed ore stockpile provides 18,313 t of live storage with two reclaim apron feeders located in a tunnel under the stockpile. Each feeder is capable of providing the nominal SAG mill feed rate of 1,250 t/h that is required for the 10 Mt/y throughput case.

17.4.2 Grinding Circuit 6.0 Mt/y

Coarse ore reclaimed from the crushed ore stockpile is conveyed to the SAG mill feed chute by the 1,200 mm wide SAG mill feed conveyor at a nominal rate of 750 t/h. Dilution water is added to the SAG mill at the feed chute such that the mill discharge solids concentration is a nominal 72% w/w. Lime is added to the SAG mill feed chute.

17.4.2.1 SAG Mill

The nominal SAG mill dimensions are 10.36 m diameter and effective grinding length (EGL) of 6.64 m. The SAG mill is trunnion mounted and is powered by twin pinions with each pinion driven by a 7.25 MW motor through a gearbox for a total mill power of 14.5 MW. Each motor is a hyper-synchronous SER drive which allows operation between 60 and 80% of the mill critical speed. The nominal mill speed is 75% of critical speed.

The SAG mill is fitted with steel liners and uses 125 mm forged steel grinding media. The grinding media is added to the SAG mill feed conveyor via a ball charger. The SAG mill discharge is equipped with a trommel screen with 12 mm apertures.

17.4.2.2 Scats Recycle

Material passing through the SAG mill discharge trommel reports to the ball mill discharge hopper. Trommel oversize flows by gravity to the scats recycle conveyors (three in series). The first of these conveyors is fitted with a self-cleaning magnet to facilitate removal of tramp metal, which will primarily consist of undersize (worn) grinding media.

Scats are discharged onto the SAG mill feed conveyor, at an estimated nominal scats rate of 15% of new SAG mill feed.

17.4.2.3 Ball Mill

Ground ore passes through the SAG mill discharge trommel to the mill discharge hopper. Slurry from the hopper is pumped by duty/standby slurry pumps to a cluster of eight 660 mm diameter cyclones. The cyclone cluster classifies the slurry to give an overflow of P_{80} of 125 µm at a density of 37% w/w solids. The cyclone underflow is discharged at nominally 75% w/w solids and flows by gravity to the ball mill feed chute, with up to 25% of the underflow able to be directed to the SAG mill.

The ball mill is a trunnion mounted 6.25 m inside shell diameter and EGL of 9.8 m, powered by a single pinion drive with a 7.25 MW fixed speed motor through a gearbox. The motor is started by a liquid resistance starter (LRS).

The ball mill is rubber lined and uses 60 mm steel grinding media. Each mill is fitted with a trommel screen with 8 mm slots to facilitate scats removal.

17.4.3 Grinding Circuit 10 Mt/y

The SAG mill discharge grates are modified to include pebble ports to facilitate pebble extraction at approximately 25% of new feed rate. Two 5½ foot 200 kW Symons cone crushers are installed to reduce the pebble size to a P_{80} of nominally 12 mm. The sizing of the crushers is such that one runs constantly choke fed, while the other is choke fed intermittently to control the surge bin level between high and low levels.

Scats recycle conveyor 2 is extended to provide additional elevation to feed a new pebble crusher feed bin. This bin has two discharge belt feeders, one for each pebble crusher. A new scats recycle conveyor 3 is constructed to transfer pebble crusher product to the SAG mill feed conveyor.

The belt magnet is relocated towards the head end of scats recycle conveyor 2 with a ball return chute bypassing the pebble crushers. A metal detector is installed downstream of the magnet which activates a flop gate to divert tramp steel around the pebble crushers.

A second identical ball mill and four additional 660 mm cyclones (installed in the existing cyclone cluster) are installed for the 10 Mt/y throughput. Cyclone underflow is split equally between the two ball mills using the Phase 1 SAG mill cyclone underflow distribution box.

17.4.4 Flotation Circuit 6.0 Mt/y

17.4.4.1 Rougher Flotation

Ball mill cyclone overflow flows by gravity to the first of six 130 m³ rougher tank cells. Collectors (PAX and thionocarbamate) and frother are added to the first cell. Copper concentrate is collected from each cell and laundered to a common concentrate hopper and pumped via the duty/standby rougher flotation concentrate pumps to the regrind mill discharge hopper. The tailings from the rougher cells discharge into the tailings hopper from where they are pumped to the Tailings Storage Facility (TSF) by duty/standby pumps.

All flotation cells are equipped with agitators and forced air dispersion systems.

17.4.4.2 Regrind Circuit

Rougher flotation concentrate reports to the regrind mill discharge hopper and is classified in a cluster of eight 250 mm diameter cyclones. Cyclone overflow at nominal P_{80} of 20 µm and solids concentration of 20% w/w solids flows to the cleaner conditioning tank. The cyclone underflow at nominal 65% w/w solids flows by gravity to the Regrind ball mill feed chute.

The regrind ball mill dimensions are 4.2 m inside shell diameter and EGL of 7.4 m. The mill is trunnion mounted and powered by a single pinion drive with a 2 MW motor through a gearbox. The motor is fixed speed at 75% of the mill critical speed, and started by a liquid resistance starter (LRS). The ball mill is rubber lined and uses 25 mm high chrome steel grinding media. The mill is fitted with a trommel screen with 1 mm aperture to facilitate scats removal.

17.4.4.3 Cleaner Flotation

The cleaner flotation circuit consists of three stages of closed circuit cleaning, with cleaner scavenging on the first stage cleaner tailings.

17.4.4.4 Cleaner 1 Flotation

Regrind cyclone overflow, cleaner 2 tailings and cleaner-scavenger concentrate are combined in the Cleaner conditioning tank, where collectors (PAX and thionocarbamate) and frother are added. The conditioning tank has 5 minutes residence time.

The cleaner 1 flotation bank consists of four 38 m³ cells in a 2+2 configuration. Concentrate is laundered to the cleaner 1 concentrate hopper and pumped by duty/standby pump pair to the cleaner 2 cells. Cleaner 1 tailings flow directly to the cleaner-scavenger cells.

17.4.4.5 Cleaner-Scavenger Flotation

The cleaner-scavenger bank consists of three, 38 m³ cells. Collectors (PAX and thionocarbamate) and frother are added to the feed box. Concentrate is laundered to the cleaner-scavenger concentrate hopper and pumped by a duty/standby pump pair to the cleaner conditioning tank. Tailings discharge into the cleaner-scavenger tailings hopper from where they are pumped to the rougher tailings hopper by a duty/standby pump pair.

17.4.4.6 Cleaner 2 Flotation

Cleaner 1 concentrate is re-cleaned in the cleaner 2 flotation bank, consisting of four 16 m³ cells in a 2+2 configuration. Collectors (PAX and thionocarbamate) and frother are added to the feed box. Concentrate is laundered to the cleaner 2 concentrate hopper and then pumped to the cleaner 3 cells by a duty/standby pump pair. Tailings flow by gravity to the cleaner conditioning tank.

17.4.4.7 Cleaner 3 Flotation

Cleaner 2 concentrate is re-cleaned in the cleaner 3 flotation bank, consisting of three 16 m³ cells in a single bank. Collectors (PAX and thionocarbamate) and frother are added to the feed box. Cleaner 3 concentrate is laundered to the cleaner 3 concentrate hopper and pumped by duty/standby pump pair to the concentrate thickener feed box. Tailings flow by gravity to the cleaner 2 bank.

17.4.4.8 Samplers and On-Stream Analysis

Plant metallurgical samplers are provided to ensure accurate sampling of main process streams for metallurgical accounting. The following streams are equipped with automated metallurgical samplers:

- Primary ball mill cyclone overflow (metallurgical sampler).
- Final concentrate (metallurgical sampler).
- Final tailings (metallurgical sampler).
- An on-stream X-ray fluorescence (XRF) analysis system (OSA) is provided to monitor the grades of the main flotation circuit streams to provide real time data for plant operation.
- Rougher flotation concentrate.
- Rougher flotation tailings.
- Regrind ball mill cyclone overflow.
- Cleaner 1 flotation concentrate.
- Cleaner-scavenger flotation concentrate.
- Cleaner-scavenger flotation tailings.
- Cleaner 2 flotation concentrate.
- Cleaner 2 flotation tailings.
- Cleaner 3 flotation concentrate.

OSA samples are provided by a combination of pressure pipe and gravity samples and peristaltic hose pumps. Sample returns are split between three return hopper based on grade, and returned to the circuit via three peristaltic hose pumps.

17.4.5 Flotation Circuit 10 Mt/y

The rougher circuit is increased by three additional 130 m³ primary rougher cells installed ahead of the existing rougher cells. The concentrate from these cells is laundered to the primary rougher concentrate hopper and pumped by a duty/standby pump pair to the new primary cleaner flotation column feed box. Tailings from the new primary rougher cells will flow by gravity to the first of the existing rougher cells.

The new primary cleaner flotation columns clean the primary rougher concentrate to produce the final concentrate grade. Collectors (PAX and thionocarbamate) and frother are added to the column feed box. The column is equipped with a forced air dispersion system. The concentrate flows by gravity to the primary cleaner concentrate hopper and is pumped by duty/standby pumps to the final concentrate metallurgical sampler. The tailings from the column cell discharge to the primary cleaner tailings hopper and are pumped by duty/standby pumps to the regrind mill discharge hopper for regrinding.

The primary cleaner concentrate and combined final concentrate (primary cleaner and cleaner 3 concentrates) are added to the OSA system to give a total of 12 streams being measured.

Primary cleaner concentrate and cleaner 3 concentrate report to the concentrate thickener feed box. The concentrate thickener underflow is dewatered in a vertical plate pressure filter to produce filter cake for transport to further processing. The concentrate thickener has been sized to for peak concentrate production at 10 Mt/y throughput. The filter will require additional plates to be fitted for future production increases.

The combined rougher and cleaner-scavenger tailings are pumped to the tailings storage facility.

17.4.6 Concentrate Dewatering

Final concentrate is pumped to the concentrate thickener feed box where it flows by gravity to the thickener feed well. Flocculant is dosed to the thickener by a helical rotor pump. Thickener overflow flows by gravity to the thickener overflow tank where it is pumped by a transfer pump to the process water pond. Thickener underflow is pumped by duty/stand-by peristaltic hose pumps to the 200 m³ concentrate filter feed tank providing 12 hours surge capacity between the thickener and the filter.

The thickened concentrate is dewatered in a vertical plate pressure filter operating on an automated batch cycle. The filter is fed from duty/stand-by positive displacement filter feed pumps that pump from the concentrate storage tank to the filter. Filtrate is returned to the process water pond via the filtrate tank and filtrate transfer pump. The filter has a dedicated air compressor and air receiver for air drying. Filtered concentrate is discharged into a concrete bunker from where it is transferred by FEL to the 3 000 tonne capacity concentrate storage shed.

A concentrate load out facility consisting of weigh bridge and truck wash down facility is provided for concentrate transport to port.

17.4.7 Reagents

Flotation reagents are all stored in the same reagent storage area, except lime, which is stored in a dedicated lime silo. All reagents other than Potassium amyl xanthate (PAX) are mixed in a common bunded mixing area. PAX has a dedicated, bunded mixing area and sump pump due to its hazardous area classification. The nominal flotation reagent consumption rates are indicated in Table 17.8.

Table 17.8 – Flotation Reagent Consumption Rates

Reagent	Dose Rate (g/t)	Consumption @ 6 Mt/y (t/y)	Consumption @ 10 Mt/y (t/y)
pH Modifier (lime)	2 000	12 000	20 000
Collector 1 (PAX)	30	180	300
Collector 2 (Flomin C4132 / Orica DSP009)	40	240	400
Frother (Dowfroth DF250)	40	240	400

With the exception of lime, which will be delivered in a bulk road tanker, all other reagents will be delivered in bulk bags or IBCs.

All reagents (PAX, thionocarbamate and frother) are dosed to the flotation circuit by individual duty pumps to the following locations:

- Rougher flotation cell 1.
- Rougher flotation cell 5.
- Cleaner conditioning tank / Cleaner 1 flotation cell 1.
- Cleaner 2 flotation cell 1.
- Cleaner 3 flotation cell 1.
- Cleaner-scavenger flotation cell 1.

PAX is supplied in 850 kg bulk bags and mixed at 20% w/v in a 4 m^3 mix tank. Mixed PAX is transferred to a 4 m^3 PAX dosing tank. PAX is dosed by six metering pumps (5 duty, 1 stand-by) to the flotation circuit dose points.

Thionocarbamate is supplied in IBCs. An IBC support frame is provided to support two IBCs (duty/stand-by). The IBCs are connected to a dosing header supplying the metering pumps. Thionocarbamate is dosed by six metering pumps (5 duty, 1 standby) to the flotation circuit dose points.

Frother is supplied in IBCs. An IBC support frame is provided to support two IBCs (duty/stand-by). The IBCs are connected to a dosing header supplying the metering pumps. Frother is dosed by six metering pumps (5 duty, 1 standby) to the flotation circuit dose points.

Flocculant consumption of 40 g/t of thickener feed has been assumed for design purposes. To cater for process fluctuations, the flocculant dosing pumps have been sized to allow for twice the required dose rate of flocculant.

Flocculant is supplied in 25 kg bags. It is batch mixed and stored in a vendor supplied flocculant mixing package, complete with two dose pumps (duty/standby) to supply flocculant to the concentrate thickener.

Hydrated lime is supplied in a bulk road tanker and discharged to a 350 tonne lime silo. Lime is drawn from the silo by screw feeder into the lime mixer where water is added to produce 20% w/w lime slurry, which is stored in a 150 m³ storage tank. Lime is delivered to the milling circuit by duty/standby pumps in a ring main arrangement, using air-actuated on/off pinch valves to control the lime addition rate based on the measured pH.

17.4.8 Miscellaneous

A number of additional items are included in the plant equipment list and capital cost estimate. These are:

- Overhead cranes, monorails and hoists provided at various areas to assist in plant installation and maintenance.
- Chutes for connecting conveyors and at the feed to the SAG and ball mills.
- Sump pumps for removal of spills and wash-down water.

Provision has also been made for an equipped metallurgical laboratory located adjacent to the plant.

17.4.9 Services

17.4.9.1 Air

Low pressure compressed air is supplied to the flotation cells by a duty-standby pair of centrifugal multistage blowers, to supply a nominal 19,087 m^3/h at 50 kPa in Phase 1. This is increased to 3 blowers in Phase 2, with 2 operating and 1 standby, providing a nominal 23,472 m³/h at 50 kPa. The primary cleaner flotation column requires air at medium pressure and is to be supplied from the plant air system.

Plant and instrument air is supplied to by two rotary screw compressors with integral dryers in a duty/standby configuration. The additional load placed on the plant air system by the primary cleaner flotation column will require an additional air receiver and associated instrumentation. Filtration (cake blow) air is supplied by a dedicated rotary screw compressor with integral dryer.

17.4.9.2 Water

Fresh water from either the mine area recovered water pond (refer to Chapter 18.4.1.2) or the river pumps will supply a 700 m³ capacity raw water tank. The raw water tank will overflow into a 30,000 m³ process water pond, providing process water make-up. Decant water from the tailings dam, thickener overflow, and filtrate are recycled to the process water pond.

Raw water is used to supply reagent makeup, pump gland seal water, cleaner flotation launder sprays and, after suitable treatment, potable water.

It has been assumed that water reclaimed from the TMA is of sufficient quality that only mechanical filtration is required to remove any oversized debris. A water clarifier has not been included in the equipment list.

Process water is distributed by 3 duty and 1 standby pumps to the process plant with the main addition points being the SAG mill feed, mill discharge hopper, regrind mill discharge hopper and flotation circuit launder water addition points.

Raw water will be passed through filters to produce gland water of acceptable quality. This water will be distributed to low pressure and high pressure gland water circuits with individual take-offs supplying gland water to the slurry pumps.

Potable water will be supplied by local services. Potable water will be stored in a 100 m³ storage tank and distributed to users via the potable water pumps and piping network. A design figure of 270 litres per person per day has been used for potable water consumption which includes drinking water and sanitary use. Safety showers will also be supplied with potable water.

Cooling water is required for the SAG mill and ball mill lube oil systems. A once through flow of fresh water has been assumed for these circuits and will be discharged in the TMF.

A 700 m³ Fire water tank is sized to provide 5 hours fire water supply. The Fire water tank is fitted with one electric and one diesel fire pump, together with one electric fire water jockey pump. A fire water main distributes fire water throughout the plant.

17.4.10 Control Strategy

The general control strategy for the concentrator is:

- Integrated control via a Process Control System (PCS) for plant areas where equipment requires remote start/shutdown, sequencing or process interlocking.
- Use of individual Programmable Logic Controllers (PLCs) for sections of plant or pieces of equipment that are supplied as complete vendor packages. These will typically communicate alarms and status information to the plant PCS for recording and monitoring purposes.
- Monitoring of all required operating conditions on the PCS and recording of selected information for data logging and/or trending.
- Control loops are managed by the PCS except where a vendor PLC directly controls a vendor package.
- Hard-wired interlocks are used for personnel safety.

17.4.11 Electrical Load Requirements

The total Phase 1 process plant installed power is 32.4 MW (based on 100% motor capacity and including stand-by equipment). The estimated Phase 1 consumed motor power (operating load) is 21.8 MW per hour.

For Phase 2, an additional 8.5 MW of equipment will be installed, equating to a total Phase 2 process plant installed power of 40.9 MW.

The consumed load is the actual power required when the plant is operating at nominal capacity. The consumed load is derived from the connected load using typical load factors. For large equipment such as grinding mills, predicted power draw has been used based on comminution calculations, allowing for typical motor and drive efficiencies.

For process equipment such as pumps and agitators, a 75% load factor has been assumed. In the next phase of the project, due consideration will be given to the power supply system for 6.0 Mt/y versus increased load required for the 10 Mt/y option and whether the transmission lines, transformers, switchyards and substations should be designed and installed up front to cater for the potential future increased load.

18 PROJECT INFRASTRUCTURE

18.1 Tailings Management Facilities

This Section is extracted from the Golder Associates report dated December 2017.

18.1.1 Introduction

During the initial years of operation, tailings will be stored in a surface Tailings Management Facility (TMF) situated adjacent and to the west of the process plant. After year 8, tailings will be stored in the exhausted Vieiro and Arinteiro open pits.

Tailings production is estimated at 91 million tonnes during the 13 years of operation of the concentrator. The tailings will have an initial solids content of 35% and are expected to be potentially acid generating (PAG). The surface TMF will be plastic lined and have a capacity for 44 Mt of tailings and the Vieiro-Arinteiro TMF will store 47 Mt of tailings.

A tailings thickening system will be implemented downstream of the concentrator to produce a total of 91 Mt of thickened tailings with a final 67% solids content. This will ensure physical and chemical stability of tailings while reducing the size of the dam wall required for the surface TMF. This technology will also achieve greater densities and deposition slope angles, and thus increase the storage capacity while reducing the footprint and the quantiy of fill required for dam construction. Furthermore it will mitigate seepage and can help to control acid generation as a result of a lower hydraulic conductivity and oxygen transmissivity.

Figure 18.1 shows an overview of the proposed tailings management facilities.

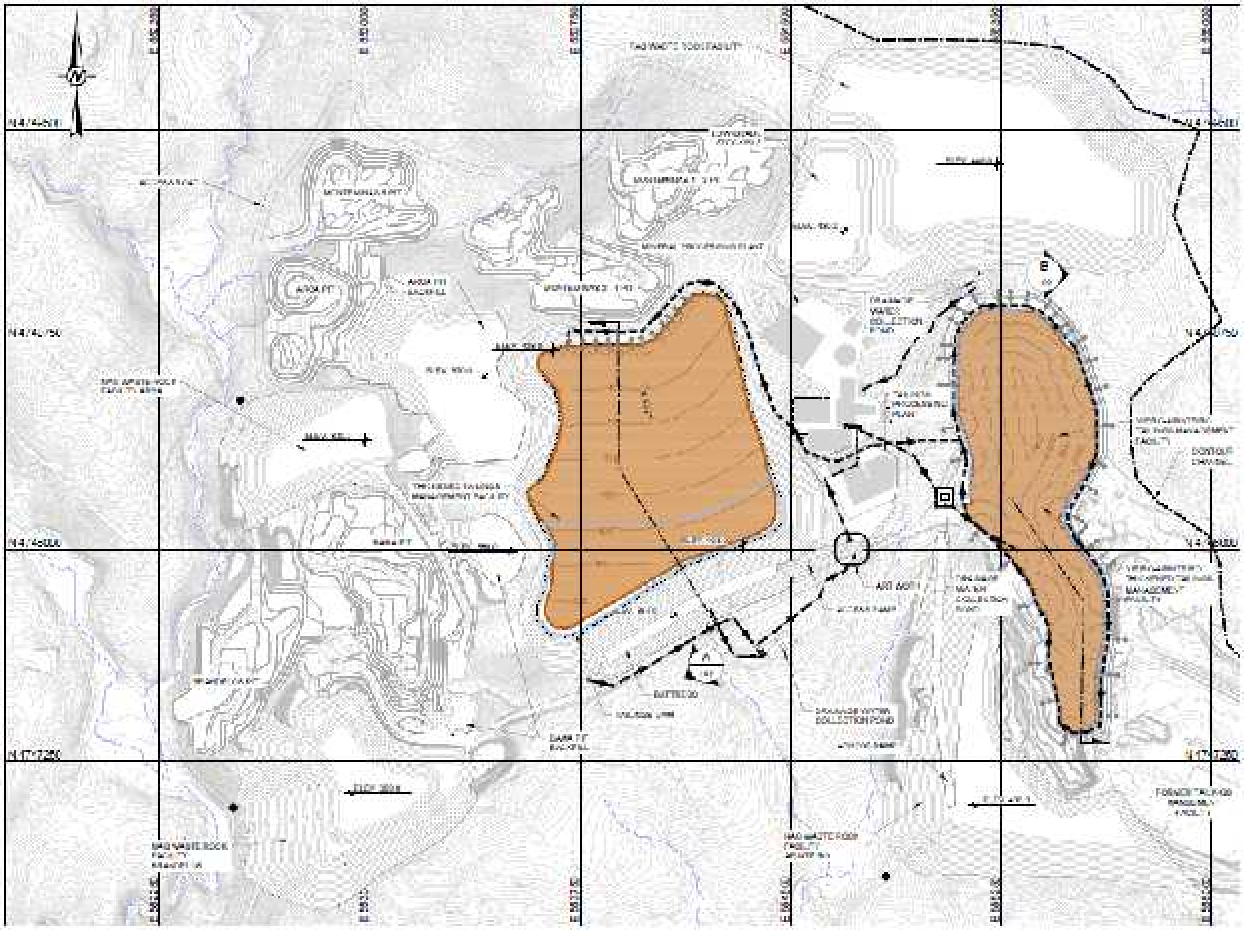


Figure 18.1– Tailings Management Facilities General Arrangement Plan (Golder 2017)

18.1.2 Tailings Storage Site Selection

18.1.2.1 Design Criteria of the Tailings Facilities

The Arinteiro and Vieiro pits were selected as the main plant tailings storage as they offer good storage capacities for most of the production in comparison to the rest of the pits. In addition, both pits are located near the treatment plant and will be mined out by year 8 which means they can be used until the end of the project life.

The parameters and criteria used in the design of the tailings management facilities are provided in the following tables.

Table 18.1 – Tailings Production

Description	Units	Value	Source/Comments
Tailings thickening technology	-	Thickened Tailings (Deep Cone Thickener, DCT)	Atalaya Mining/Golder
Years of operation of Thickened Tailings Plant	Years	Yr 1 to Yr 13	Atalaya Mining/Golder
Nominal Production ofThickened Tailings	tpd	See Table 2.	Atalaya Mining/Golder.
Design Capacity: Surface Thickened TSF Vieiro and Arinteiro Thickened TSF	Mt	 44 47	Atalaya Mining/Golder

Table 18.2 – Nominal Tailings Production

Year	Total Tailings Production (Source: Mine plan s50)	
	Mtpa	**tpd**
1	5.1	13,973
2	6	16,438
3	6	16,438
4	5.7	15,616
5	5.5	15,068
6	7.2	19,726
7	8.5	23,288
8	9	24,658
9	10	27,397
10	10	27,397
11	10	27,397
12	7.5	20,663
13	0.4	998
Total	**90.9**	**-**

Table 18.3 – ThickenedTailings Properties and Design Parameters

Description	Units	Value	Source/Comments
Specific Gravity of Tailings Solids	-	3.2	Minnovo
Average Dry Density	t/m^3	1.8	Adopted. Golder
Tailings Solid Content:	%	67	Adopted. Golder
Beach slope for deposition modelling	%	4	Adopted. Golder

Table 18.4 – Thickened Tailings Management Facility Parameters and Design Criteria

Description	Units	Value	Source/Comments
Impermeabilization system: Surface Thickened TMF Vieiro and Arinteiro Thickened TMF	Basin and upstream face lined with HDPE geomembrane		Adopted. Atalaya Mining/Golder.
Design Water Volume Storage: Surface Thickened TMF Vieiro and Arinteiro Thickened TMF	Mm^3	1.0 1.0	Adopted. Golder. (6 times the 24 h maximum precipitation[1], considering a TSF basin of approximately 700.000 m^2)
Surface Thickened TMF Dam: Dam Crest Width Upstream Slope Downstream Slope Operational/Hydraulic Freeboard Construction Material	 m H:V H:V m -	 10 1.5:1 2:1 2 Waste Rock-NAG	Adopted. Golder
Vieiro and Arinteiro Thickened TMF Pit Slope Configuration [2]: Temporary berm width (for construction) Final berm width Bench height Operational/Hydraulic Freeboard Construction Material	 m m m m -	 4 2 8 2 Waste Rock-PAG	Adopted. Golder
Seismic Basic Acceleration (a_b)	g	<0.04	NCSE-02/ Seismic Risk Map. Ministerio de Fomento. Touro Project Area.
Seismic Acceleration Calculation (a_c)	g	0.054	Calculated. Golder, according to NCSE-02 [3]
Seismic Coefficients: Horizontal (kh) Vertical (kv)	-	 0.027 0.013	Calculated. Golder, according to NCSE-02 [4]
Minimum Factors of Safety: Static Case Pseudo Static	-	 1.4 1.2	ITC 08.02.01

[1] 24 h maximum precipitation registered value = 218 mm. AEMET. Santiago de Compostela Airport. Data Serie 1920-2017.

[2] Minimum slope for in pit benches to be defined by the Stability Analysis

[3] $a_c = S \cdot \rho \cdot a_b$. Where:

$\rho = 1,3$ ([illegible] S[illegible])

$S = \frac{C}{1,2}$, [$C = 1,3$ (g[illegible] t[illegible] II.V[illegible] f[illegible] r[illegible])]

[4] $k_h = 0,5 \cdot a_c$ and $k_v = 0,5 \cdot k_h$

18.1.3 Surface Thickened Tailings Management Facility

18.1.3.1 Site Geotechnical Characterization

The surface TMF will be located on the right slope of the valley formed by the Rego Das Pucheiras river and will occupy a small valley created by a minor tributary to the river.

A geotechnical investigation of the surface TMF site was conducted in 2015 by Terratec. A total of 9 boreholes and 9 test pits were conducted, as shown in Figure *18.2*. Three boreholes were completed with slotted piping for piezometer installation. In two of these boreholes, the water table remained constant at a depth of approximately 10 m, while the third detected a confined aquifer at a depth of 25 m with water flowing to the surface with a constant flow rate over time.

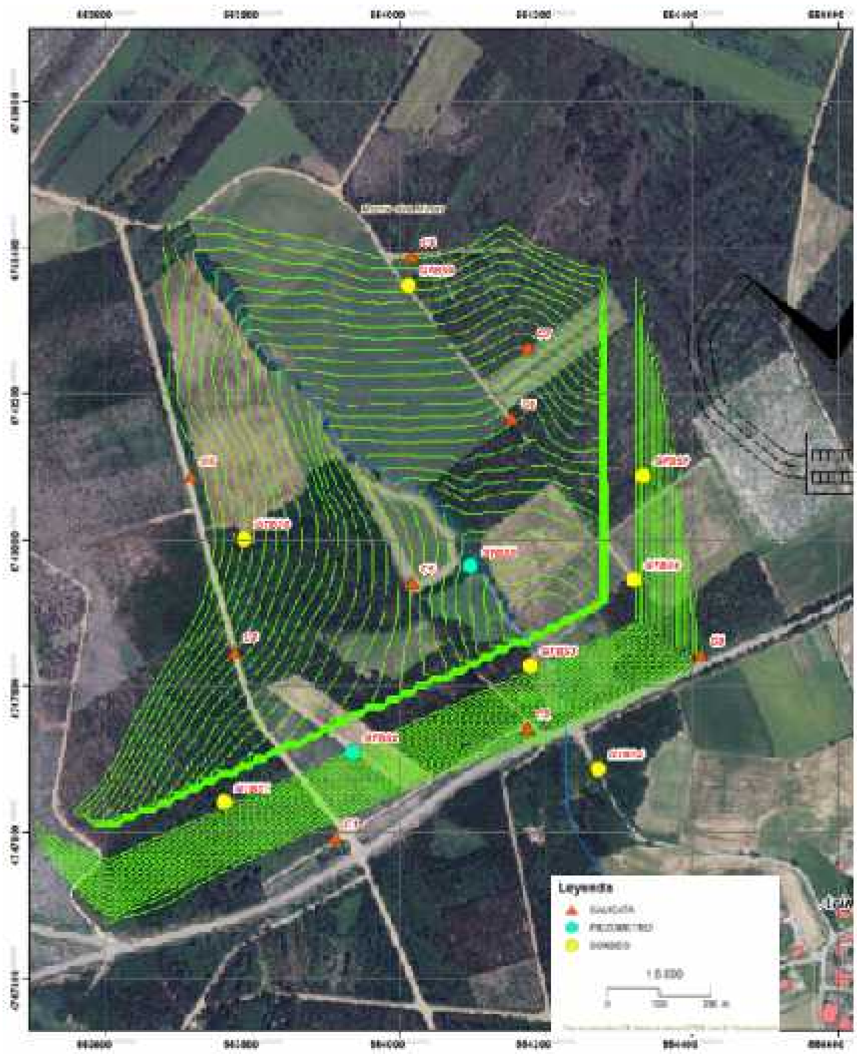


Figure 18.2 - Geotechnical investigation campaign. Borehole and Test Pit Locations. (Terratec 2016)

Based on site reconnaissance and laboratory tests, three general lithologies were identified in the TMF site:

- Eluvium Deposits – Residual materials derived by weathering of the bedrock. The eluvium thickness ranges from 2 m at the top of the valley to 25 m at the lower slope and bottom of the valley. It has a relative density of low to medium under the water table based on SPT 'N' values, with mean N_{SPT} values at 20. It is classified predominantly as silty sand with occasional gravel, type SM, according to the USCS classification, and to a lesser extent as low-plasticity silts or silty gravels with sand (ML to GM). The Eluvium maintains a rock structure from a shallow depth (category V completely decomposed, according to Geoguide 3 Hong Kong 1988).

- Oxidized Bedrock – Located immediately below eluvial sandy deposits between 4 and 36 m below surface, extending on average to a depth of 30m. It forms a compact layer that gradually increases in shear strength with depth. It is moderately to slightly fractured, with fractures mostly associated to bedding. Oxidation starts in fractures and progresses until reaching fresh bedrock (category II to IV highly decomposed to slightly decomposed, according to Geoguide 3 Hong Kong 1988).
- Fresh Paragneiss – Competent rock with limited fracturing, low secondary permeability, and planar, unfilled joints mostly due to bedding. At the center of the wall of the TMF fresh rock occurs approximately below 35 m in depth (categoy I to II fresh to slightly decomposed according to Geoguide 3 Hong Kong 1988).

18.1.3.2 Tailings deposition model

A tailings deposition model for the TMF has been developed using a 4% deposition slope. The tailings distribution system will consist of multiple discharge points at the head of the TMF (north area). Due to the basin's topography, during the first years of operation, tailings are expected to drain over natural creeks from the discharge points and be deposited in the lower areas of the impoundment with a <4% slope (estimated at 1.5% for modelling purposes).

Operations in the impoundment will start in year 1 and last until late in year 7 of the LOM. The final storage capacity will be 44 Mt (24.4 Mm^3 assuming a dry density for deposition of 1.8 t/m^3). Given the high level of tailings thickening (Cw=67%), water could build up as a result of direct precipitation during operation. Such water buildup will be pumped using mobile equipment installed at the dam's crest.

A maximum volume of 1 Mm^3 of storm water has been estimated, which is 6 times the volume associated to the maximum precipitation recorded in 24 hours (218 mm, according to the 1920-2017 hydrometric series, Compostela Airport Station, AEMET) for an area of 700,000 m^2. For hydraulic safety purposes, the TMF will have enough capacity to store the maximum volume of storm water without the need for any pumping system to operate.

It will be necessary to develop a site specific hydrological study and water balance model during the early years of operation (i.e. before end of year 4) to support and permit any adjustments of these water management criteria. Preliminary tailings deposition modelling indicates that a 2 m freeboard is sufficient to store the run-off for a design storm event. Table 18.5 provides a summary of data from the deposition model.

Table 18.5 - Tailings deposition model for the surface TMF. Summary results

Time	Accumulated Tailings Production		Elevations (m a.s.l)			Dam requirements	
						South Sector	
Year of operation	Mt	Mm³	Tailings (north sector)	Tailings (south sector)	Pond	Elevation (m a.s.l.)	Height (m)
1	5.1	2.8	363	358	364	366	54
2	11.1	6.2	376	369	374	376	64
3	17.1	9.5	385	377	382	384	72
4	22.8	12.7	393	384	388	390	78
5	28.3	15.7	405	389	393	395	83
6	35.5	19.7	418	389	395	397	85
7	44	24.4	426	395	401	403	91

Table 18.6 lists the resulting factors of safety for the static and pseudo-static case. In further project study phases, a geotechnical investigation campaign focused on a more comprehensive understanding of the behavior of dam foundation materials is required, in particular, a more precise definition of the shear strength parameters of the eluvial materials in the foundation and the extent of the lens, as well as a verification of the buttress requirement.

Table 18.6 – Stability Analysis. Results

Case	Factor of Safety Required	Factor of Safety Obtained. Circular Failure	Factor of Safety Obtained. Block Failure
Static	1,4	1.4	1.8
Pseudo-Static	1,2	1.3	1.7

18.1.3.3 Surface TMF Design

The TMF will feature a rockfill dam built using NAG waste rock material. To avoid the need for large fill volumes and excessive freeboards, the elevation of the dam crest will vary according to the profile of the modeled tailings. In the south, the dam reaches a crest elevation of 403 m a.s.l. and a maximum height of 91 m measured from the crest to the downstream toe; in the north (head of the TMF), it reaches a crest elevation of 428 m a.s.l. and a maximum height of 30 m. The required fill volume will be approximately 15 Mm^3.

The final crest width will be 10 m and the final dam length will be approximately 3,600 m. The upstream and downstream slopes will be 1.5H:1V and 2H:1V, respectively. A 45-meter wide buttress has been included in the southern sector of the dam to ensure an acceptable factor of safety. The final crest elevation will be of 354 m a.s.l. and the downstream slope will be of 2H:1V.

The foundation preparation will consist of the removal of the low strength aluvial sands located below the downstream slope of the dam to a maximum depth of 8 m. The upstream face of the dam will be lined with a 1.5 mm thick HDPE geomembrane over a 200 g/m^2 geotextile. The liner will be supported by two, 2-meter thick layers of filter and transition material to prevent migration of fines from the tailings to the waste rock. Both the geomembrane and the geotextile will extend over the surface of the entire impoundment basin to form a low permeability barrier to contain the tailings. Figure 18.3 shows a tailings dam cross section at the south sector.

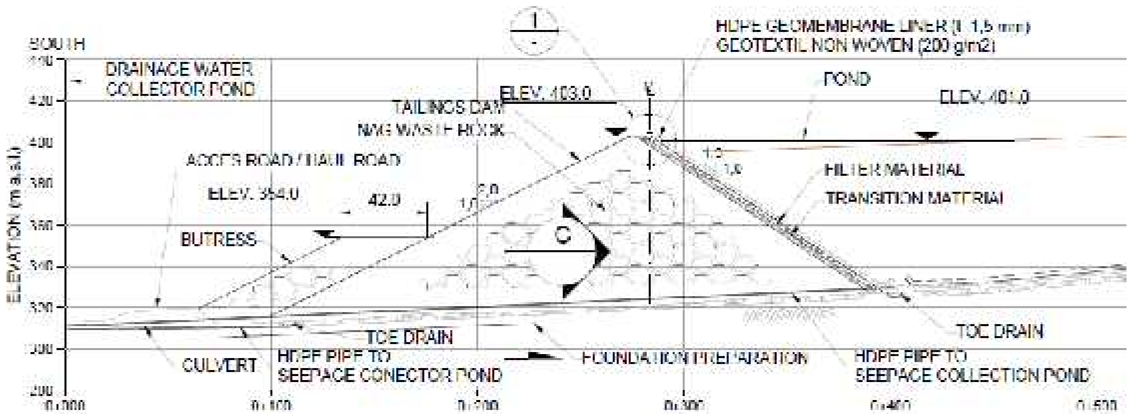


Figure 18.3 - Tailings Dam Cross Section (Golder 2017)

The dam will have a drainage system consisting of two toe drains and collection drains discharging through an HDPE pipeline into the drainage water collection pond, downstream and on the south side of the dam. An underdrain has been designed at the bottom of the impoundment, under the HDPE geomembrane. These drains will connect to the collection drain located under the dam to discharge into the drainage collection pond as well.

In addition, a drainage system has been designed over the HDPE geomembrane to promote tailings consolidation. These drains will discharge into a collection drain located at the upstream foot and at the south side of the dam and will convey drainage into the seepage collection pond using a pipeline through the dam's body.

The drainage collection pond will be sized to collect the drainage and seepage flows from the impoundment and the 24 h storm event run off that will be pumped from the TMF. The pond will have two cells that will be excavated in the ground with a 2H:1V slope. It will be lined with a smooth 1.5 mm HDPE geomembrane over a 200 g/m^2 geotextile. The dam will be instrumented to monitor its performance and structural integrity during operation for deformation and piezometric levels, including:

- Topographical survey monuments every 50 m along the dam's crest for the different phases;

- Piezters along the base of the different phases of the dam and downstream of the TMF to monitor water at the dam's foundation and in drains; and
- Inclinometers will be installed in the dam's crown to record displacements or settlements once construction is complete.

18.1.3.4 Construction

A starter dam to store the tailings production of a 1.5-year operation will be built. To achieve the final configuration of the dam, two 16-meter raises (measured from the dam elevation at its south side) will be built.

Table 18.7 lists elevations, operation dates for each raise and fill volumes associated to each growth phase. Construction dates have been defined according to the growth of tailings in the facility following the deposition model for the project. It is assumed that an adequate supply of competent rock fill will be available for the construction of the dam walls based on the mine plan.

Table 18.7 - Surface TMF Construction Stages.

Stage of Construction	Dam Elevation (m a.s.l.)	Requiered Fill Volume (Mm^3)	Years of TMF Operation according to LOM	Years for construction according to LOM
Stage 1	371	2.7	1 - 2	0
Stage 2	387	4.3	2 - 4	1
Stage 3	403	8.2	4 - 7	3

Installation of the liner system over the impoundment will take place in two phases concurrently with the two first dam construction phases. Drainage pipelines will be laid out according to the proposed dam growth. The drainage water collection pond will be built during the first dam and drainage system construction phase. The dam will be built using NAG waste rock material from the open pits. Waste rock will be laid out and compacted using bulldozer in 1-meter thick layers. Filter, transition, and drain materials will be placed in up to 0.5-meter layers, laid out and compacted.

18.1.3.5 Operation

Tailings discharges into the impoundment will take place from the northest side of the facility using a distribution system with multiple spigots. Tailings will be deposited in thin layers to ensure the design beach slope, geotechncial characteristics and geochemical stabilty are achieved.

During the initial deposition phases, discharges will be supported on natural ground and the construction of a small berm for surface grading may be required. As the tailings are deposited into the impoundment it will be necessary to raise the dam wall in the area of the discharge points. The discharge points themselves will be supported on the wall as it is raised.

Water buildup in the impoundment will be pumped using mobile equipment and discharged by a pipeline into the drainage water collection pond. The drainage water collection pond will have sufficient capacity to receive the water evacuated in a 24h period from the TMF. The pond comprises two cells with equal volume to allow maintenance works to be able to operate either independently.

18.1.4 Vieiro and Arinteiro Thickened Tailings Management Facility

18.1.4.1 Tailings deposition model

A thickened tailings deposition model for the Vieiro and Arinteiro open pits was developed. Perimeter tailings discharges will take place around the Vieiro and Arinteiro pits with tailings flowing into the Vieiro open pit first and, as they increase in elevation, will move towards the Arinteiro open pit.

To maximize the storage capacity of the TMF and minimize the requirement of the lining system, tailings will be discharged above the tailings surface by pipe supports (i.e trestles) to fill the low elevation areas. To simulate this situation, tailings were modeled using a 1% slope.

Tailings deposition in the open pits will start in year 8 and last until late in year 13 of the LOM; the final storage capacity will be 47 Mt (26 Mm^3 assuming a dry density for deposition of 1.8 t/m^3). Any surface water present in the facility is expected to come mainly from direct precipitation, runoff and pit walls seepage. Any water buildup will be pumped using mobile equipment from the west slope of the Arinteiro area.

Based on these criteria, the entire surface of the open pits that would be occupied by tailings or contact water will be lined if needed. This will be verified with hydrochemical models in future study stages of the project. To determine the elevation of the liner, the same water storage criterion used for the surface TMF has been used, assuming a maximum volume of 1 Mm^3 of storm water, which is 6 times the volume associated to the maximum precipitation recorded in 24 h (218 mm, according to the 1920-2017 hydrometric series, Compostela Airport Station, AEMET) for an area of 700,000 m^2.

A site-specific hydrological study and water balance model will have to be developed for the TMF to support verification and adjustment of the above water management criteria. Table 18.8 provides a summary of data from the deposition model. The elevation of the liner system in the open pits enables storm water storage with a 2 meter freeboard.

Table 18.8 – Tailings Deposition Model for the Vieiro and Arinteiro TMF. Summary results

Time	Accumulated Tailings Production		Elevations (m a.s.l)		
Year LOM	Mt	Mm3	Tailings	Pond	Low Permeability Liner System
8	9.0	5.0	224	230	232
9	19.0	10.6	260	258	262
10	29.0	16.1	274	276	278
11	39.0	21.7	289	291	293
12	46.5	25.9	299	300	302
13	46.9	26.1	300	301	303

18.1.4.2 Geotechnical analyses

The slope stability study for the open pits has been analyzed by Terratec to provide recommendations on mine design parameters. For the purpose of the tailings deposition study, the rock mass has been assumed stable under strength parameters and for the geometries analysed by Terratec.

As part of the design for the Vieiro-Arinteiro TMF, a stability analysis for the proposed fills inside the pits has been conducted to determine the minimal slope inclination for the new benches, using the Limit Equilibrium Theory according to the Morgenstern-Price Method. The results are shown in Table 18.9.

Table 18.9 - Stability Analysis. Results

Case	Factor of Safety Required	Factor of Safety Obtained
Static	1.4	1.4
Pseudo-Static	1.2	1.3

18.1.5 Vieiro and Arinteiro TMF Design

The Vieiro and Arinteiro open pits will be lined with a 1.5 mm thick HDPE textured underside geomembrane over a 200 g/m^2 geotextile. Alternative design options will be studied during future stages of the project. To facilitate the installation of the liner, the internal configuration of the open pits will have to be corrected. The bottom of the Vieiro open pit will be backfilled and nominally compacted, to avoid settling, in 1m maximum lifts using NAG waste rock up to an elevation of 160 m a.s.l. The pit walls will be filled to reduce the steepness of the slopes, reduce bench heights, and to provide a smoother geometry. The fill will also consist of PAG waste rock, placed in benches of up to 8 meters; berms will have a minimum width of 4 meters and a final width of 2 meters. The minimum slope inclination will be 1,2H:1V. The final elevation of the fill and liner systems will be 303 m a.s.l. and the required volume of material will be around 6 Mm^3.

As a transition material between the waste rock and the liner system, two 1-meter thick layers of filter and transition material, designed to meet standard filter and internal stability criteria, will be placed to prevent migration of fines from the tailings to the waste rock.

Figure 18.4 shows a cross section of the Vieiro and Arinteiro TMF and Figure 18.5 shows the detail of the proposed new benches.

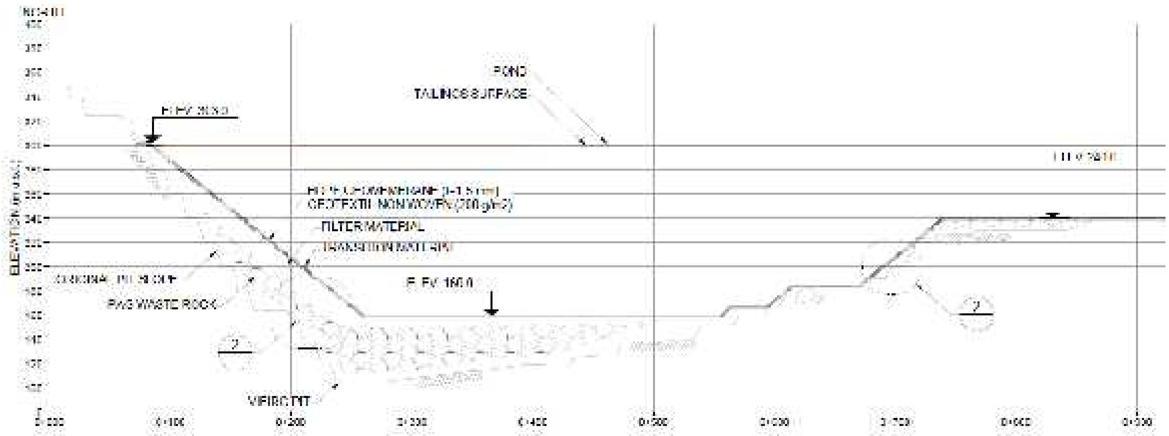


Figure 18.4 - Vieiro Cross Section (Golder 2017)

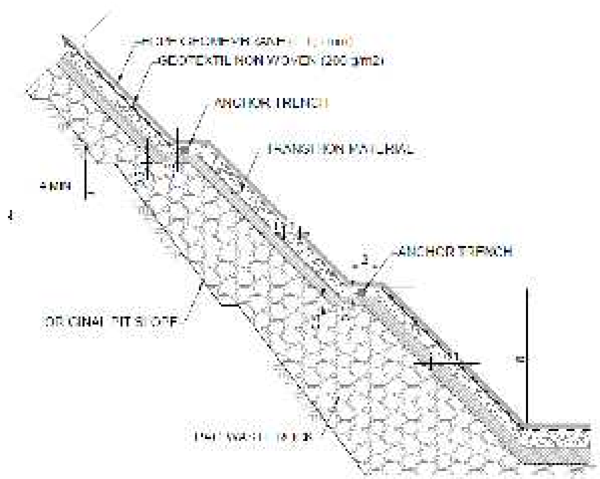


Figure 18.5 - Bench Detail

A battery of pumping wells has been proposed to avoid overpressure in the geomembrane. These will be arranged in the perimeter of benches, with a 400-meter spacing. These wells will maintain the water table below the elevation of the liner or deposited tailings. Water will be pumped into the TMF and managed from the TMF pond.

The water collection pond will store water from the bench depressuring system and precipitation runoff. The pond will feature a cell excavated in the ground with a 2H:1V slope. It will be lined with a smooth 1.5 mm HDPE geomembrane over a 200 g/m^2 geotextile.

To monitor the performance and structural integrity of the TMF during operation for deformation and piezometric levels, the following will be installed:

- Topographical monuments every 50 m along bench crests, and
- Piezometers installed in wells mirroring pumping wells to monitor water in open pit benches.

18.1.5.1 Construction

Construction of the liner system will be ongoing during operation of the TMF. A number of required benches will be built each year according to the rate of rise of the tailings inside the impoundment. Table 18.10 lists elevations, operation dates for each raise and fill volumes associated with each expansion. Construction dates have been defined based on the results of the deposition model for the project. The fill volume for Phase 1 includes filling the bottom of the Vieiro pit up to an elevation of 160 m a.s.l.

Table 18.10 – Surface TMF Construction Stages

Stage of Construction	Final Elevation of the low permeability barrier system (m a.s.l.)	N° of Benches to be constructed	Required Fill Volume (Mm^3)	Year of Operation of the stage according to LOM	Year for Construction according to LOM
Stage 1	232	9	4.14	8	7
Stage 2	262	4	1.24	9	8
Stage 3	278	2	0.47	10	9
Stage 4	293	2	0.34	11	10
Stage 5[1]	303	1	0.05	12-13	11

[1] This stage considers the construction of 1 bench of 10 m height and will operate during years 12 and 13.

Pumping wells will be drilled before placement of fill or installation of geomembranes. Well completions will be raised with the fill material as required. The water collection pond will be available before the pumping wells start operating, in order to store pumped water.

Bench fill will be constructed using PAG waste rock from open pits being mined. Waste rock will be spread and compacted in 1-meter thick layers. Filter, transition, and drain materials will be placed in up to 0.5-meter layers, spread and nominally compacted.

18.1.5.2 Operation

Tailings discharges in the impoundment will take place around the perimeter of Vieiro and Arinteiro pits. The filling sequence will be scheduled according to the construction of internal fills.

During operations, pipe supports (i.e trestles) would be used to discharge tailings towards the central sector of the deposit to fill low elevation areas and so to achieve the maximum storage capacity with minimum liner requirements.

Water from bench depressuring will be conveyed into the impoundment to the operational pond. Water buildup in the impoundment will be recycled to the plant using pumping equipment; water will be discharged by a pipeline into the drainage water collection pond. The drainage water collection pond will have sufficient capacity to store the water that could be accumulated in a 24 h period.

18.1.6 Tailings Processing Plant

In order to maximize the capacities of the TMF and the decommissioned pits for the discharge of the mill tailings, the tailings management strategy requires installation of a tailings dewatering system. This system uses a tailings processing plant to produce thickened tailings to meet the tailings deposition strategy.

18.1.6.1 Design Criteria

The following assumptions and exclusions were used to define the design criteria for the tailings processing plant:

- The tailings production tonnages throughout the life of the mine have been divided in two phases in order to facilitate the plant design and stage the procurement of equipment.
- A design factor of 20% over the nominal rate has been selected for thickener sizing purposes. It is understood that this may change in later phases of the project; however, this factor has been selected in order to be conservative during the equipment sizing process and it will be confirmed with detailed tailings laboratory testing.
- The following design parameters have been estimated based on benchmarking, conversations with dewatering equipment vendors, and Golder's expert opinion and are subject to testwork confirmation:
  - Thickener unit rate;
  - Thickener underflow solids content; and
  - Flocculant dose.

Based on assumed properties of the tailings rheology and its particle size distribution and, therefore the potential high thickener drive/rake requirements, three thickeners of similar diameters have been selected to dewater the maximum tailings tonnages during the LOM. Due to the tailings management system requirements, deep cone thickeners (DCT) have been selected for the study. Since rake torque demands can increase exponentially as the thickener diameter increases, two thickeners, instead of one larger unit, have been selected to process the tailings flowrate during the initial years. Consequently, the cost of the units also increases in a non-linear way between smaller and larger diameter units.

A trade-off study between a larger diameter DCT versus two smaller units, backed up by laboratory testing, for the initial years of the LOM, can potentially offer a less expensive solution for the tailings processing plant.

18.1.6.2 Process Description

The tailings processing plant will be located next to and south east of the concentrator. The tailings production at a nominal throughput of 822 t/h and 1,142 t/h has been estimated for years 1 to 6 and years 7 to 13, respectively.

During the initial 6 years of production, the tailings processing plant will be composed of two, 25 m diameter DCT for production of thickened tailings at, or close to, paste consistency (to be confirmed by laboratory testing). These DCTs will come with shear thinning systems, formed by high flow centrifugal recirculation pumps and ancillary equipment, for the shearing of the thickener underflow to facilitate the thickened tailings transportation.

Tailings from the concentrator will be fed to the thickener feed box. This box will then split the mill tailings stream into two similar streams to feed each of the two DCTs. Flocculant will be prepared in the flocculant preparation system at rate of 30 grams of polymer per tonne of tailings that will be pumped into each of the DCTs' feed well and/or feed lines. Each thickener will process approximately 411 t/h of tailings and produce an underflow of about 67% solids content by weight. The underflow of both DCTs will be pumped into an agitated underflow tank. From this tank the thickened tailings will then be pumped to the surface TMF.

During year 6, an additional DCT with the same design capacity, will be installed to handle the increased tailings throughput from years 7 - 13. The additional tailings production will be split in the thickener feed box to feed the 3 DCTs with similar streams. Flocculant will be fed into each of the 3 DCTs at a rate of 30 grams per tonne of tailings. Each thickener will process approximately 381 t/h of tailings and produce an underflow of about 67% solids content by weight. The underflow of the 3 DCTs will be pumped into an agitated underflow tank. From this tank the thickened tailings will initially be pumped to the surface TMF during year 7 and, starting at year 8, will be pumped to the Vieiro-Arinteiro Pit.

The thickeners overflow is collected in an overflow tank and then distributed throughout the tailings plant as required. Any excess water will be pumped to the concentrator for re-use as process water.

18.1.6.3 Thickened Tailings Distribution

The distribution of the thickened tailings starts at the discharge of the thickeners underflow tank. From years 1 to 7, all of the thickened tailings will be pumped 1.5 km by 4 horizontal centrifugal pumps connected in series through two parallel 6" diameter carbon steel pipelines.

After year 8, a new distribution system will be required to transport the thickened tailings to the Vieiro-Arinteiro Pit. This new system will use 4 horizontal centrifugal pumps, which will pump the thickened tailings 2.1 km through three 10" diameter carbon steel pipelines. Two of these lines, each one connected to a centrifugal pump, are designed to transport up to 100% of the thickened tailings production (50% per line) towards the northern west, north and northern east portion of the pit. The third 10" line will be connected to two centrifugal pumps in series and is designed to transport up to 50% of the thickened tailings to the southern west, south and southern east portion of the pit.

This pipeline configuration will allow flexibility and a consistent discharge of tailings around the perimeter of the pit to maintain the deposition strategy. The pipeline arrangement as designed, will always discharge and distribute 50% of the thickened tailings production along the north side of the pit while the distribution of the other 50% of the thickened tailings can be alternated between the north and the south end of the pit.

18.1.6.4 Reclaim Water from the Tailings Disposal Facilities

The water reclaim system designed for the Surface TMF and Vieiro-Arinteiro Pit will use mobile diesel horizontal centrifugal pumps. These pumps will be installed inside the impoundments to pump the water over the dam to a water collection pond. Collection wells will be installed with vertical turbine pumps to pump the reclaim water through a 24" diameter HDPE pipeline to the process water tank at the concentrator.

18.1.6.5 Reclaim Water from the Tailings Processing Plant

The water reclaim system for the tailings processing plant will use 2 horizontal centrifugal pumps (1 operating and 1 stand by) that will pump the reclaim water through a 24" diameter HDPE pipeline to the process water tank at the concentrator.

18.2 Water Management

18.2.1 Description of the Water Management Facilities

18.2.1.1 Seepage Recovery Ponds

All of the pond designs utilized the same design criteria. This consists of a single cell, excavated in the ground with slopes of 2H:1V. They feature a waterproofing system comprised of a smooth 1.5 mm thick HDPE geomembrane placed over a 200 g/m^2 geotextile. A leak detection system consisting of a grooved pipe located between the excavation slope and waterproofing system will be installed as a control measure.

18.2.1.2 Project Water Management

All water recovered through the various drainage and seepage control systems is piped to the recovered water pond. This pond is located near the process plant where the water will primarily be pumped to the plant for reuse or treated until it complies with the required environmental levels for discharge.

An intercept trench has been designed that will collect all Project run-off water. The trench starts at the head of the PAG waste rock facility and runs north to south along the slope situated to the east of the Vieiro and Arinteiro deposit. The discharge is downstream from the old plant tailings basin.

The trench is approximately 3.6 km long with a trapezoid section and lateral slopes of 1H:1V, coated in riprap material.

18.2.1.3 Water Supply from Rivers

In the early stage of the project (construction and early operation) before the tailings return water is established, the operational water requirement will be met by utilizing local rivers. Two water supply facilities are provided to extract water from the Brandelos and Lañas rivers. Each water pumping station consists of two, 200 m^3/h capacity pumps where water is pumped via HDPE lines to the plant process water pond. Power is distributed to the pump stations by overhead line.

18.2.1.4 Water Treatment and River Discharge

Since the project area has a positive water balance, the excess water will have to be managed and integrated within the mining complex. The project will try to maintain a closed system and re-use the excess water.

Some of this excess water will require treatment, therefore a water treatment plant (PTA) will be constructed to manage a water volume of approximately 2.25 Hm^3/year that will be re-used in the process plant. The sources of this excess water are from the pit drainage, raw water tank surplus, contact water from the PAG waste dump, and the contact water from the MTP. The remaining waters are considered unaffected and will be returned to their natural discharge areas or used as fresh water for the process plant as required.

The mining operation water balance predicts that the surplus treated water will be discharged into the local river system for the first 3 years of operation. After that, all of the water treated in the PTA will be reused in the process plant and will not be discharged into the local fluvial system. There are many alternative discharge points that are currently being studied.

The water treatment process will include several stages of metal precipitation to produce hydroxides or sulfates. This is achieved by raising the pH and adding calcium cations in the form of lime (calcium hydroxide) with a final thickening step to generate a sludge, which is then pumped to a reservoir. The lime provides a dual function where it contributes calcium cations and neutralizes the acid, generating a sufficiently basic environment to precipitate the metals in the form of hydroxides.

In order to save lime consumption, the pH of the PTA discharge water will meet the minimum process requirements of 8.3-8.5 and then will be increased to pH 10.5-11 in the grinding circuit to meet the flotation requirements.

Mixing the PTA sludge with tailings should not impact the quality of the return water to the process plant. The high pH of the float tail together with the residual unreacted lime will aid in pH control in the tailing impoundment and will improve compaction characteristics.

The treatment stages include:

- Aeration: to oxidize Fe_2+ to Fe_3+
- Addition of lime to reach pH above 8, which will precipitate the metals in the form of hydroxides.
- Thicken to separate solids, consisting mainly of a chemical precipitate of hydroxides, heavy metal sulfates, gypsum, pseudo-jarosites, etc.

The main advantages of this process are:

- The high quality of the effluent that is produced.
- The process is easy to automate.
- Extensively tested and proven technology.
- Plants contain simple equipment, which reduces the need for large inventory of spare parts.
- Lower neutralization costs than conventional lime treatments.

The treatment sequence includes aeration, lime addition, and thickening. Figure 18.6 shows schematically the treatment sequence:

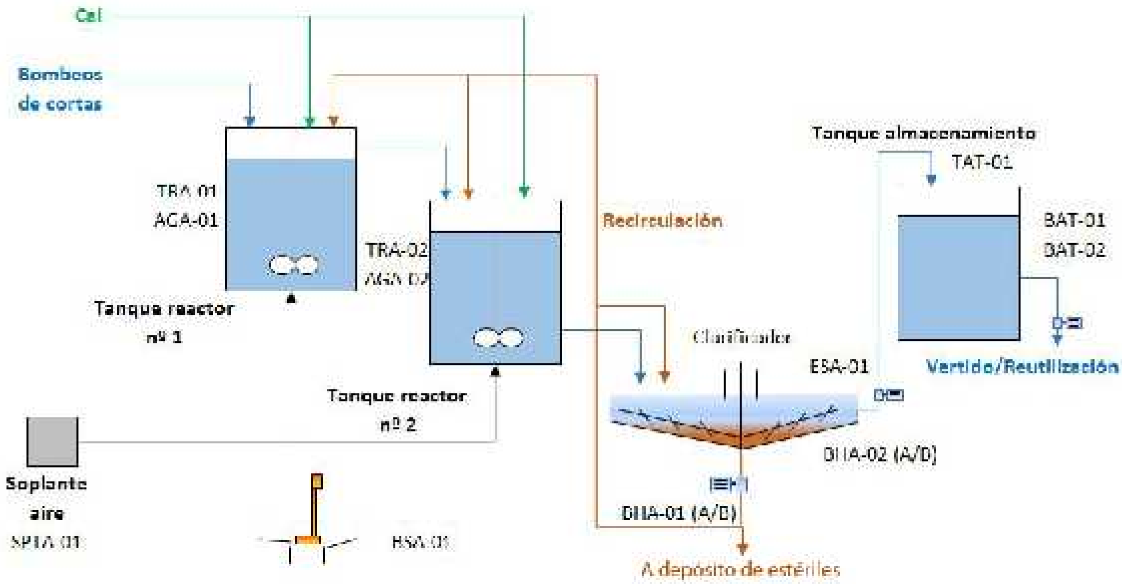


Figure 18.6 – Water Treatment Plant Process (Atalaya 2017)

The PTA facilities will be located within the Touro mining complex, to the east of the PTM facilities. The main components of the plant are listed below:

- Main reactors
- Clarification and pumping of sludge
- Treated water storage tank
- Low pressure air blower and installation
- Water pumps
- Reagents
- Pipe lines
- Motor Control Center (MCC)

<u>Sludge Management</u>
The PTA discharged sludge will require monitoring and analysis to quantify the impact to the float tailings facility. The production of sludge will be maintained between 8 and 10 g/l of treated water in order to maintain the designed water quality. The design parameters are:

- Annual volume of water to be treated – 1.77 Hm^3
- Sludge production – 9.0 kg/m^3
- Sludge production per hour 2,068 kg/h
- Annual sludge production 15,930 t/year

18.2.1.5 Potable Water Supply

Potable water for the Project will be supplied from the local water distribution pipe network. The connection point is 1.5 km from the site. Potable water will be stored in a 100 m^3 tank and distributed to the respective users via the potable water pumps and piping network. The estimated potable water consumption will be 270 litres per person per day, which includes drinking water and sanitary use. Safety showers will also be supplied with potable water.

18.2.1.6 Fire Water

A 700 m^3 Fire Water Tank will be sized to provide 5 h of fire water supply. The Fire Water Tank is fitted with one electric and one diesel pump, together with one electric fire water jockey pump. A fire water main will distribute fire water throughout the plant.

18.2.2 Water Management - Operation

18.2.2.1 Seepage Collection Ponds

The water recovered by the seepage detection systems will be monitored and any tear or hole in the coating geomembrane will be repaired (to prevent infiltration). In addition, any sediments or waste collected in the ponds will be periodically removed.

18.2.2.2 Intercept Trenches

Under normal operations, the intercept trenches will only require maintenance and routine inspections to check the stability of the lateral slopes and the rock-fill coating conditions, and that there are no flow obstructions.

18.2.2.3 Water Supply

The operation of the water supply pumps and pipeline will be monitored regularly and checked for possible pipe leaks.

18.2.2.4 Water Treatment Plant

The water treatment plant will be operated as required to maintain conservative water storage levels. River flows and discharge water quantity and quality will be monitored to ensure license compliance.

18.2.3 Monitoring and Inspection

A formal monitoring and inspection program will be undertaken for all aspects of the mine water management system to check for operational safety and design compliance.

Table 18.11 details the inspection requirements for Tailings Storage water recovery while Table 18.12 details12 details the inspection requirement for the seepage ponds.

Table 18.11 - Tailings Dam Pond Water Recovery Inspection

Failure or deviation mode			Causes			
Description	Detection	Inspection frequency	Description	Parameter	Control	Observation frequency
Reduction of recovered water	Inspection	Daily	A deeper point of the lagoon far from the water recirculation system	Lagoon location	Visual / bathymetry	Weekly
				Geometric lagoon parameters		
			Generation of preferential infiltration line	Preferential routes	Patrolling	Daily
			Low availability of the pumping system	Availability	Visual	Daily
				Line competence		

Table 18.12 - Inspection of seepage collection ponds

Failure or deviation mode			Causes			
Description	Detection	Inspection frequency	Description	Parameter	Control	Observation frequency
Inner wall erosion	Visual	Daily	Dragged sediments or sand from the wall or contaminated filters	Murky water	Visual	Daily
			Filter failure or destruction	Murky water	Visual	Daily

18.3 Power System

18.3.1 Power Supply

The nominal power demand for the operation is approximately 40.9 MW. Power for the Project will be supplied from Portodemouros electrical substation, owned by UNION FENOSA electric distribution company via the nearby 66 kV power lines. A 66 kV to 6.3 kV transformer and associated switch gear will be located in a substation at the tie-in point from the main 66 kV line. A new 12 km 6.3 kV overhead power line will be constructed from the Municipality of Vila de Cruces (A Coruña) to the Project site. At present, three alternatives are being studied regarding the layout of the power line:

- Alternative 1. New 66 kV exclusive use line approximately 14 km long
- Alternative 2. New aerial-underground mixed exclusive use line approximately 14 km long
- Alternative 3. Use of a 20 kV line currently approved for construction owned by UNION FENOSA Electric Distribution Company for shared use from the point of supply (Portodemouros E.S.) to an area near the treatment plant. The route from the final point to the plant site will be completed with a new exclusive-use 66 kV line. This alternative would require an increase in line voltage at the plant.

The public administration will decide which of the three proposed alternatives for the layout of the electric power line is the most appropriate, including, if necessary, any conditions or modifications considered for the construction.

Power Distribution

Power is supplied to the process plant main 6.3 kV/400 V HV switch gear and distributed to the plant motor control centers (MCC) where the voltage is stepped down by transformers.

Other electrical loads shall be supplied from pole-mounted transformers including water supply MCC, recovered water pumps MCC, and decant pumps MCC.

18.3.2 Emergency Power Supply

A 700 kVA diesel powered emergency genset is provided to supply power in the event of main supply failure. The emergency genset is situated in a dedicated building (refer Section 18.10.8) and will supply power to the following items:

- Fire pump (a diesel driven fire water pump is also provided)
- Emergency lighting circuit
- Filter feed tank agitator
- Potable water pumps for safety showers
- 110 V High Voltage Rectifiers

The process control systems (where required) and the plant computers will include uninterrupted power systems (UPS) supported by batteries.

18.4 Site Development, Roads and Port Infrastructure

The Project is supported by an adequate network of public roads that will also be used to transport concentrate in single trailer trucks to the existing port of Vilagarcia de Arousa. The Vilagarcia port also has adequate existing bulk materials storage, handling and ship loading facilities (Figure 18.7). A Coruña, Ferrol and Marin are also alternative ports in the area. Upgrading of these off-site roads will not be required to support the Project.

Figure 18.7 – Project Access (Atalaya 2017)

A total of 5 km of access roads are constructed outside of the main mine and process plant site. These roads are two lane 7.5 m width sealed all weather roads. Two accesses roads are planned to the main facilities i.e. East access and South access.

The East access connects the main facilities with the existing public road DP-6602. It has a total length of 2.48 km. The South access connects the main facilities with the existing public highway AC-240. It has a total length of 3.47 km. The accesses will run primarily along existing roads, however some new construction may be required.

Figure 18.8 shows the two access routes in the Project. In either case, whether the access is an existing route or a new route, the roads will be constructed with a width of 8 m and drainage ditches 50 and 70 cm wide to ensure vehicle maneuverability. All roads will be constructed with 7 cm of asphalt (MBC) on a 35 cm sub-base.

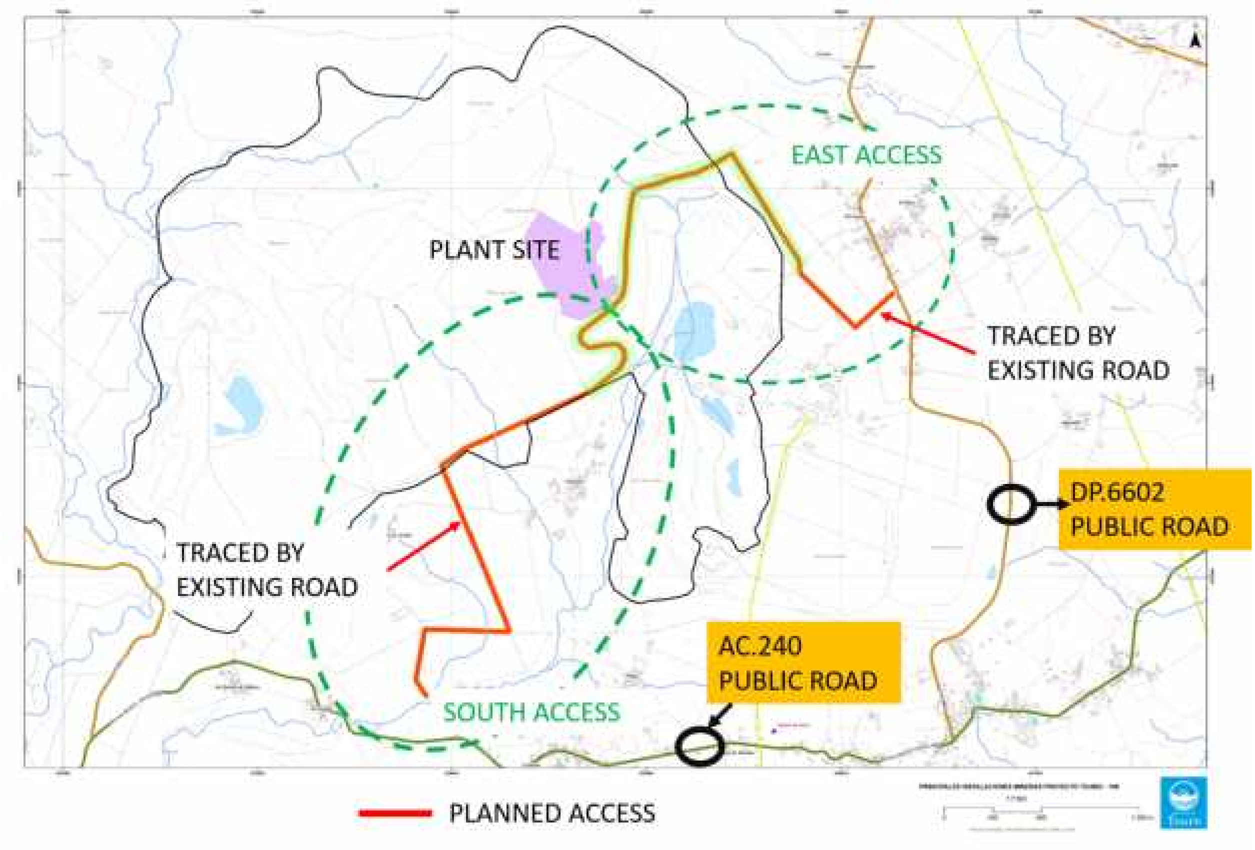


Figure 18.8 – Access Routes (Atalaya 2017)

In addition to the new access roads, approximately 1,500 m of on-site access roads have been designed to provide easy access between the various facilities, including the main mine office, workshop and process plant complex. In general road design will have an 8 m sealed width and 30 m turning radii on intersections to accommodate heavy construction and operations traffic. The design also includes a sealed car parking area at the offices and workshops.

18.5 Waste Water Treatment

All installations where waste water is generated will have a septic tank sanitation system. In addition, the main administrative and operations offices, laboratory, canteen, etc. will have a sanitation network and a septic tank where the waste water will be treated.

Waste water management will be conducted by an external manager authorized by Environmental Public Administration who will periodically clean all septic tanks in the installations.

18.6 Diesel Storage Facility

Diesel fuel will be stored in three fuel tanks (two 40 m^3 tanks for Diesel B and one 20 m^3 tank for Diesel A) located adjacent to the mine contractor workshop. The fuel storage will be constructed on a sloped concrete slab using external double-walled tanks with an immediate leak detection system. Since the tanks are double walled, an external containment system will not be required.

The Diesel B tanks will be provided with a piping network for fuelling as well as adequate electrical installation and structural, electrical, fire and environmental safety measures. The tanks will be located at a distance of more than 10 m from any building and more than 50 m from both public roads and high-power lines.

Fuel will be supplied by tank trucks with direct connection to each tank. The storage of large quantities of diesel will not be required due to close proximity of several diesel suppliers that can provide daily deliveries. The fuel tanks will contain a three-day supply which is sufficient for weekend consumption. Additional storage capacity will be added if the supply becomes insufficient to support mine operations.
The facility will include drive over concrete bunded slabs and refuelling arms for both the mining vehicles and support truck fleet (e.g. lube and refuelling vehicles). The refuelling arms will deliver diesel fuel at a rate of 700 l/min and 1, 000 l/min respectively and supplied from the diesel storage tanks. The fuel delivery pump stations will be controlled via swipe cards with each card having defined user accounts so that fuel consumption can be monitored. The facility will also have a low flow pump station for light vehicle refuelling.

18.7 Laboratory

A 435 m^2 on-site laboratory has been designed that will provide assay services for both the mine grade control samples and the process plant metallurgical control samples. The laboratory will consist of a sample preparation section, a wet chemical digest section, and an instrument section including AAS and ICP-OES. The laboratory will have a daily capacity of approximately 100 samples.

18.8 Mine Truck Shop and Warehouse

18.8.1 Mine Truck Shop

A 1,013 m^2 mine workshop has been designed for mine equipment fleet maintenance and is equipped with one office. It will be able to accommodate the larger mine haul trucks as well as the support equipment. It will be equipped with a bridge crane as well as auxiliary machinery owned by the mine contractor (truck crane, forklift and lifting platform) for maintenance purposes. In addition, the system will have a truck wash area with a physical-chemical treatment plant (described below).

The mine equipment fleet has been described in detail in Chapter 16 and shown in Table 18.13 and 18.14.

Table 18.13 - Mining Mobile Equipment Fleet

Equipment	Capacity	Quantity
Drills:		
Sandvik DP 1500i class	114-127 mm diameter	3
Sandvik DP 1100i class	102 mm diameter	2
Hydraulic Excavators:		
180- to 200-t (e.g., Komatsu 2000)	10-12 m^3	2
110- to 120-t (e.g., Komatsu 1250)	6-8 m^3	2
Front-End Loader:		
512 kW (e.g., Caterpillar 990)	8 m^3	1
Haul Trucks:		
780 kW (e.g., Komatsu HD785)	91 t	21-27

Table 18.14 – Auxiliary Mining Fleet (Y1-Y3)

Equipment	Quantity
Wheel dozer, Caterpillar 824-class	1
Track dozer, Caterpillar D8T-class	2-3
Hydraulic backhoe, 50- to 60-t, 2.5-3 m^3	1
Articulated truck, 36- to 40-t	2-3
Motor grader, Caterpillar 14-class	2
Vibratory compactor, 30-t	1
Water truck, 30,000- to 50,000-liter	2

The workshop building will contain a shift briefing area as well as a tool, hose, and consumable store. Translucent sheeting will be provided throughout the upper roof area and ridge ventilation will be installed.

It is anticipated that an overhead gantry crane will service the entire workshop. Lubricants will be circulated within the workshop and waste oil and waste coolant will be removed.

A parking area adjacent to the workshop will provide parking for 25 heavy vehicles before and after servicing or during downtime.

A compressor and air receiver will be located adjacent to the workshop and will provide compressed air to the workshop, bulk lube storage and tire change areas.

Wash-down and run-off water from the workshop will be captured and drained into a sludge pit and sump complete with a sediment trap. The dirty water will be pumped to the process plant tailing thickener for recycling after passing through an oil-water separation system.

18.8.2 Bulk Lubricant and Hydrocarbon Storage Facility

Hydraulic oil, engine oil and waste oil will be stored in 1,000-liter Intermediate Bulk Containers (IBC) inside the workshop. The IBC containers will be removed weekly by a contractor.

A containment area will be constructed to store up to 10 IBCs of hydraulic oil, 10 IBCs of motor oil, 3 IBCs of antifreeze, and 10 IBCs transmission fluid, which is equivalent to a one-week supply. An additional containment area will be constructed for the storage of special lubricants and greases.

18.8.3 Contract Tire Service

Tire maintenance and service will be performed by an outside contractor and will be the responsibility of the mine contractor. The tire supplier will have all the necessary tools and equipment to service the mine equipment. The tire service will be performed in an enclosed area outside of the mine workshop. A tire storage area will be provided that will contain up to 10 spare tires.

18.8.4 Heavy Vehicle Washdown

A heavy vehicle wash-down facility will be provided in an area adjacent to the heavy vehicle workshop. The facility will consist of a single drive through armoured concrete bunded slab, two wash-down platforms with high pressure water cannons, two off water hose reels and one off detergent hose reels. Wash-down water from the facility will be captured and drained into an adjacent sludge pit and sump complete with sediment trap. The wash-down water will be pumped to the process plant tailing thickener for recycling.

18.8.5 Light Vehicle Washdown

A wash-down facility will be required for light vehicle wash-down. The light vehicle supplier will provide this and also be responsible for maintenance services. The mine contractor will only provide a space for the supplier and will impose water management requirements for washing of vehicles. The light vehicle supplier will be responsible for obtaining the necessary permits and licenses.

18.8.6 Plant Workshop and Warehouse

A 529 m^2 plant workshop has been designed for plant spares storage and maintenance. This building has 3 offices for the maintenance supervisor, warehousing, and purchasing functions.

18.8.7 Light Vehicle Workshop

Light vehicle maintenance facilities will be provided by the supplier.

18.8.8 Reagents Store

A 467 m^2 reagents indoor storage and floor area has been designed for the storage of PAX, thionocarbamate, frother, and flocculant.

18.8.9 Auxiliary Buildings

The following auxiliary buildings have also been designed:

- Canteen – This will consist of a 163.85 m^2 dining room within a closed building. The appropriate support appliances will also be installed.
- Core storage sheds - A warehouse will be located near the core shed.
- Outdoor store - The warehouse will also include a 175 m^2 outdoor laydown area.
- Plant control room - This is located near the main concentrator building and will receive information from all the systems, including feed from the video cameras and data from the continuous analyzer. It will have no assigned staff and will house the analog-to-digital converters and the computer terminals. This control room will be a 110 m^2, stand-alone, covered building distributed over two floors. The motor control centers will be contained on the first floor.
- Crusher control room – this will consist of a 66 m^2, self-contained building
- Emergency generator building – The emergency generator will be installed in a 175 m^2 building and will provide back-up power in case of substation failure.
- Gate house – A 35 m^2 dock type control house will be constructed. An access control barrier will also be installed.

18.8.10 Explosives Compound, Powder Magazine

There will be no facilities for the storage of explosive material. The drilling contractor will be responsible for suppling explosives from a sub-supplier. An explosives supplier will transport the exact amount of material necessary to the facility on a daily basis or as needed for a blast. The management of the explosives will be carried out in accordance with current legislation.

18.9 Communications and Information Technology

The communications and information technology requirements for the project include:

- IP PBX system providing site voice and data provided by Telefonica
- PA system for fixed plant site communications (15 stations)
- RF system covering both mine and plant operations, including 100 W repeater for site wide coverage, 1 desk-top and 30 hand held portable radios

- Data center hardware
- Desktop computers (43) and Advanced Workstations (3)
- Software, including warehouse – administration software package.

18.10 Asset Protection

Asset protection systems for the site are provided in various forms as below:

18.10.1 Video surveillance

A seven-camera video surveillance system is provided to monitor the site. The specific areas monitored will include:

- Main access gate
- Fuel storage
- Main office buildings
- Concentrate handling

18.10.2 Access and Time Control System

A swipe card security access system incorporating 5 standard access control terminals and 2 checkpoint barriers with a capacity for 150 user swipe cards will be installed. This system will log and track site access of all employees and contractors working on site.

18.10.3 Site Fencing

A perimeter fence will be installed to restrict access to authorized personnel. Additional fencing will be installed around the plant site areas.

18.10.4 Gatehouse

Site access is controlled at the entrance gate house by boom gate for vehicle access and turn-style for pedestrian access. The gate house includes both video surveillance and swipe card access monitoring.

18.10.5 Fire Protection

A fire detection and warning system will be provided in all switch rooms, plant control room, offices, laboratory, plant workshop/warehouse, and the mine workshop. This system will include fire indicator panels (FIP), smoke and thermal detection equipment, associated cabling and multi point aspirated smoke detection systems.

The plant control room will be a permanently manned location and will provide after-hours monitoring of FIPs. All FIPs on site will be interfaced to the master FIP in the plant control room. Electrical shut down is specifically excluded in the event of fire alarm, as this is not required by standards or codes. False alarms will cause unacceptable disruptions to operations and will not improve safety (water will not be used for electrical fires and faulty electrical equipment will be shut down by electrical protection). However, when hydraulic power packs are protected by foam systems, electrical supply to the particular area will be interrupted on receipt of a signal from the foam system controller associated with that particular system.

The site is serviced by a fire water main fed by both electric and diesel-powered fire pumps. The site will have hydrants and/or fire reels at the appropriate locations.

A HAZOP program will be implemented during the detailed design phase to ensure that the fire systems are comprehensive, functional, and meet statutory regulations and insurance requirements.

18.10.5.1 Emergency Response and Medical Facilities

- Emergency Response - One of the meeting rooms of the General Offices module will host the Emergency Control Center. The meeting room will be equipped with a telephone and a station with an emergency channel. In addition, an emergency management plan (Self-protection plan, emergency procedures, plans, etc.) will be available.

- First Aid Room - The first aid area will be located in the main office. This facility will consist of a toilet with emergency shower, emergency eye wash, nursing care office, first aid room equipped with first aid supplies, sterilization unit, refrigerator for medicines that require cold storage, and storage room for supplies and medication.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

Atalaya has been actively marketing the copper concentrate product from the Riotinto Mine to global consumers since 2016.

Copper is an internationally traded commodity and prices are set through trading on the major metals exchanges: the London Metal Exchange (LME), the New York Commodity Exchange (COMEX) and the Shanghai Futures Exchange (SHFE). Copper prices on these exchanges generally reflect the worldwide balance of copper supply and demand, but are also influenced significantly by investment flows and currency exchange rates.

Copper concentrate, one of the many copper products, is generally sold through long-term contracts to smelters and refineries on a competitive basis. Atalaya expects that terms contained within any sales contract that could be entered into would be typical of, and consistent with, standard industry contracts.

19.2 Supply and Demand

An increase in the demand for copper is expected in 2018, due to increased demand China and from supply constraints, including mine disruptions in Chile, Indonesia, and Peru. Copper prices increased 11 percent in Q1 2017 due primarily to these disruptions in production. Pricing will ease somewhat once the disruptions have ended.

Copper pricing is forecast to increase because of stronger global demand, slower ramp-up of new capacity, tighter environmental constraints, and policy action that limits exports. Downside risks include slower demand from China and higher-than-expected production, including the restarting of idled capacity. (World Bank Commodity Outlook Q1 2017)

The copper price used for this study including underlying assumptions is discussed in Section 23.

19.3 Sales of Concentrates

The typical copper concentrate specification is shown below in Table 19.1. This specification is based on the assay results obtained from concentrates produced during the testwork in 2016.

Table 19.1 – Copper concentrate typical assay

Element	Unit	Range measured in assays	Average of assays
Cu	%	24.7 – 32.1	29.7
Pb	%	0.0 – 0.05	0.02
Zn	%	1.0 – 4.27	2.57
S	%	31.3 – 33.6	32.5
Fe	%	22.5 – 31.7	27.8
As	ppm	5 – 71	15.3

Sb	ppm	1.3 – 6.4	3.3
Bi	ppm	2.1 – 6.8	4.8
Se	ppm	53 – 167	111
Hg	ppm	0.10 – 0.40	0.27
Al	%	0.27 – 0.75	0.43
Co	ppm	66 – 151	105
Au	ppm	0.14 – 2.06	0.81
Ag	ppm	26.8 – 90.5	58.5

The copper concentrate is expected to have high grade copper content and to attract premiums in international markets due to its very low deleterious elements. The concentrates from the mine have traditionally been delivered to the Atlantic Copper smelter in Huelva and other smelters within Europe.

19.4 Ocean Shipping and Ports

The commercial due diligence identified a number of ports suitable for the ocean shipping of copper concentrates from the Touro Project. These included:

- Villagarcia de Arousa, Spain;
- Marin, Spain;
- Ferrol, Spain;
- La Coruña, Spain;
- Vigo, Spain; and
- Setúbal, Portugal.

The ports of Vigo and La Coruña were immediately rejected based on environmental reasons and lack of current bulk handling facilities.

The ports of La Coruña, Marin, Ferrol and Villagarcia were analyzed in more detail and all were identified as suitable for bulk cargo exports with good road access available. Villagarcia was initially selected as the base case port as it lies close to the Project site, at approximately 80 km distance, has land available for bulk storage warehousing, and was used historically by Rio Tinto Patiño for copper concentrate shipping.

The port of Setúbal in Portugal was only identified as a more distant shipping port on the basis that concentrates are already handled through this port. The possibility of physically blending concentrates prior to sale could be considered for economic reasons with this port. The port has existing bulk cargo handling facilities, ship loaders and spare warehouse facilities.

An assessment of likely copper smelter destinations was undertaken as part of the commercial due diligence, and was largely based on Atalaya's current marketing experience. The concentrate is clean and readily marketable and it is believed that the majority of the concentrate could be placed to the various European copper smelters located in Finland, Sweden, Germany, Bulgaria and Spain. Based on a blend of these destinations, shipping quotations indicated average rates of $25.20/wmt.

19.5 Inland Logistics

The commercial due diligence identified both rail and road haulage options to the shipping ports. Numerous rail heads were identified near the Project site that would serve the potential ports identified above. However, all the rail heads still required truck transport of concentrate to them from the Project site and the costs associated with transferring the concentrate from road to rail were felt to be such that they did not compensate the savings made by the relatively short rail route to the ports. Trucking directly from the Project site to the port was therefore the selected transport base case.

Many trucking companies transporting cargo already exist along the national highways in the area, and competitive tendering of the concentrate haulage will be possible. The road network from the Project site to the selected ports is generally of a high standard. Further investigation is required to identify the optimum road route from the Project site to the main national A54 highway to avoid local villages and towns.

Indicative pricing has been obtained for road haulage of concentrates from the mine site to Villagarcia at $15.20/wmt.

19.6 Contracts

There are currently no contracts in place between Atalaya and other entities regarding the Touro Project.

20 ENVIRONMENTAL STUDIES, PERMITTING, SOCIAL AND COMMUNITY IMPACTS

20.1 Environmental Status & Legacy

The Company is developing the Touro copper project located in the province of A Coruña, Spain. The mining area is comprised of the *San Rafael + Demasía* mining concessions, occupying part of the municipalities of O Pino and Touro and is located within 15 km of the city of Santiago de Compostela, which is the capital of the autonomous community of Galicia, in the Northwest of Spain. The territory covered by these two municipalities has a surface area of 247.4 km^2. Approximately 8,533 people live within this area and they will have the greatest impact from the mine operations.

The mining concession consists of approximately 1,827 ha. The exploration license was requested in July 1946 and awarded in June 1958 to Mr. Rafael Sáenz-Díez García for the development of an iron pyrite deposit. Subsequently, in October 1974 the license was modified to allow the mining of copper pyrite. This concession was extended in April 1978 for 90 years (until 2068). Subsequent extensions to the exploration boundary were granted in 2010 and 2011 which make up the current Touro concession boundary.

The Touro copper deposit was explored and mined extensively between 1973 and 1986 by Rio Tinto Patiño, (RTP). Rio Tinto Patiño started commercial production in 1973 and ceased operations in 1986 due to the fall in the price of copper. After closure, RTP sold the San Rafael Operating Concession to Explotaciones Gallegas.

Most of the historical mining activity occurred near four pits. The Arinteiro and Vieiro pits, the old plant tailings deposit, and the Arinteiro waste rock facilities are located to the East of the site; both the tailings deposit and the waste rock facilities have been partially reclaimed. The Bama and Brandelos pits, and the future Monte de las Minas and Arca pits are located in the Western sector of the site. Another old waste rock facility, also reclaimed, is located to the south of Brandelos.

Figure 20.1 shows the location of the mining concession (outlined in red) and the historical pits (outlined in black).

Figure 20.1 - Touro Project Site (Atalaya 2017)

The historically mined areas have varying amounts of disturbance. Since the amphibolite mineralization has relatively high pyritic sulfur content and the soils have poor organic matter, revegetation is sparse in some areas. This has led to the implementation of a system to create soils suitable for revegetation. It involves the use of Tecnosols to allow adequate rooting of colonizing species that give structure to the soil and foster acceptable successional dynamics.

The use of Tecnosols has also improved the quality of run-off, upwelling, and streams in the area. This has resulted in consolidation of the vegetation established on the surfaces affected by the mining operations as well as in surrounding areas. It has also had positive effects on the brooks in the area and on the evolution of the artificial lagoons which have produced real wetlands where the flora and fauna are similar to natural settings.

The Bama pit wetlands are a clear example of the effectiveness of the Tecnosols in regenerating degraded spaces. The already consolidated vegetation is typical of the local wetlands. There is a well-developed plant community with common reed (Phragmites australis) and broadleaf cattail (Typha latifolia) widely represented; there are also tree species like alder trees, willow trees and ash trees. The fauna populations have prospered in this restored environment.

The Vieiro pit has been flooded since operations ceased. It has become a lagoon with deep water where the surrounding vegetation shows very good development. There is a certain variety of tree species, particularly, the Galician pine, the eucalyptus and the alder tree, among others.

Following the caseation of mining, the Arinteiro pit has become partially flooded. In this case, there is a shallow water surface with mostly grass species and the presence of Phragmites australis, otherwise known as common reed. The tailings settling ponds and waste dumps are subject to reclamation work. Both topsoil and Tecnosols have been used for years with positive results.

The settling ponds from the processing plant, which have been silting for years, were restored using local soil stockpiles. Currently, these surfaces show different types of vegetation including coniferous species (Pinus pinaster) and well-developed undergrowth.

The southern area has become an industrial park which is ready to begin operations. The settling pond situated to the east of the mine was reclaimed and will now use Tecnosols. The first pioneer species have appeared and the results are expected to be satisfactory.

The Bama and Brandelos pit waste dumps are in the reclamation process, although in different phases. There is already a waste dump in this area where the reclamation work has led to well-developed and consolidated vegetation, especially the presence of Pinus pinaster.

The reclamation of the waste dumps in this area of the mine is in full operation. It is different from previous phases in that "next generation" Tecnosols are being made specifically for this area.

20.2 Applicable Legislation

In Spain, there are typically three different types of mining permits and concessions:

- Exploration permits (Art. 40.2 Mining Law) granted for a period of 1 year which may be extended for a maximum of one more year.
- Investigation or Research permits (Art. 45 Mining Law) granted for the period requested, which may not be more than 3 years, but may be extended twice for a further 3 years.
- Operating concessions (Art. 62 Mining Law) also referred to as a Mining Permit, granted for a 30-year period, and may be extended for equal periods up to a maximum of 90 years.

In general, the Exploration and Investigation Permits or the Operating Concession does not grant the surface rights. These must be purchased or leased from the surface rights owner. The Spanish and Galician mining laws state that the owner of an investigation permit must comply with the approved annual work plans. For this reason, the law determines that if a friendly agreement is not reached with a landowner, it is obligatory to initiate a temporary occupation process.

For an Operating Concession, the same principle is applicable, where agreements with local surface right owners is required but a forced expropriation process can be resorted to, if necessary. There are no royalties on the property.

At the national level, the environment is administered by the Ministerio de Agricultura, Alimentación y Medio Ambiente. The Ministerio de Agricultura, Alimentación y Medio Ambiente is responsible for setting policy and enacting into legislation EU policy. At the regional level, the environment is administered by the Consejería de Medio Ambiente y Ordenación del Territorio. The Consejería de Medio Ambiente y Ordenación del Territorio is responsible for ensuring that national policy is implemented and also has auditing responsibilities. Additionally, the Consejería de Medio Ambiente y Ordenación del Territorio has authority to issue environmental permits.

There are both generic and specific (Water, Air Quality, Biodiversity, Protected Spaces, Forests, Wastes & Dangerous Substances and, Restoration) national and regional environmental legislation applicable to the Touro Project during development, operations, final restoration, and post closure.

A list of the general environmental laws is shown below. There are also many federal and regional laws regulating water and air quality, biodiversity, protected lands, forests, wastes and dangerous substances, and restoration.

Generic Legislation

- Law 21/2013, December 9, Environmental Impact Assessment.
- Royal Decree 975/2009, June 12, Management of waste from the extractive industries and the protection and rehabilitation of the area affected by mining activities.
- Royal Decree 777/2012, May 4, modification to Royal decree 975/2009.
- Law 3/2008, May 23, of Mining Management in Galicia.
- Law 26/2007, October 23, Environmental responsibility.
- Law 27/2006, 18 July, which regulates the rights of access to information, public participation and access to justice in environmental matters.

20.3 Environmental & Cultural Approvals

In order to resume work in the concession, according to article 111 of Royal Decree 863/1985, of 2 April, which approves the general basic standards of mining safety regulation, the project will need to complete an environmental impact assessment (EIS). In addition, the Touro Project will need to receive the following approvals:

- Declaration of Environmental Impact,
- Approval of the Restoration Plan,
- Waters approval,
- Cultural approvals.

20.4 Declaration of Environmental Impact (DIA)

The DIA is the main environmental process and approval that has to be completed prior to the start of the mining, processing, and waste deposal activities.

Applicable legislation with regards to the DIA is as follows:

- Law 21/2013, December 9, Environmental Impact Assessment.

Atalaya has already submitted the following documentation and applications:

- An Environmental Impact Study, with all mitigation and prevention measures.
- A Restoration Plan, including management plans and final closure of waste storage facilities.
- Other documentation submitted:
    - Reports from each of the municipalities affected by Touro Project confirming that the Project is compatible with their respective planning programs.
    - Information for authorization to produce non-mining hazardous wastes, oil, tires etc.

20.5 Potential Environmental and Social Impacts

The different phases of the mining project will have an impact on the environment.

A series of steps will be undertaken during the development phase to prepare the ground and build the facilities and infrastructure needed for the Project. These will include: clearing and grubbing vegetation, the removal and storage of topsoil, the construction of stockpiles, ground conditioning and excavation, paving, the construction of entrance routes, fencing, a pipe system, pumps, etc.

Other activities will include: the mining of ore that will be transported and processed at the plant. Mine waste will be stored both on surface waste dumps and back filled into pits. The treatment process will generate plant tailings that will be pumped and deposited in the surface and in-pit storage facilities. Other impact-generating actions will include vehicle transit and the operation of the facilities associated with the process plant, fresh water pumping, etc.

Figure 20.2 shows the planned facilities for the Project. The San Rafael mining concession boundary is shown in yellow around the perimeter of the project area.

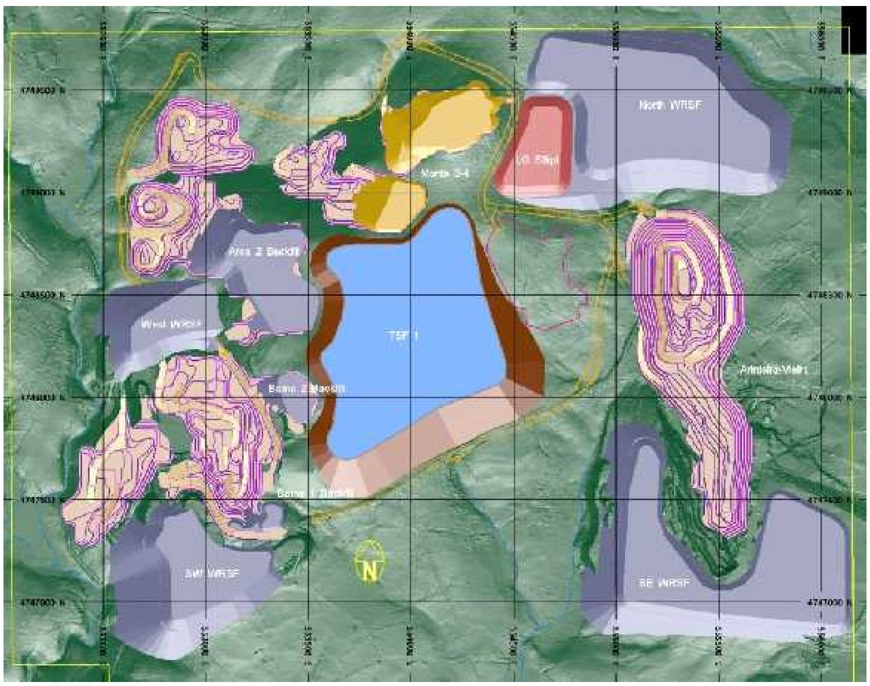


Figure 20.2 – Touro Project Facilities (Rose 2017)

Once the mine operation has ended and all of the ore processing completed or at the end of a mining phase when possible, the mine facility will be closed and the buildings and structures will be dismantled. The mine waste and tailing storage facilities will be reclaimed and the disturbed land will be re-contoured. Top soil will be added to the reclaimed areas and seeded.

The main environmental impacts will be identified and analysed, and mitigation measures will be developed. These data are reported in the environmental impact statement. The primary impacts are:

20.5.1 Air Quality and Noise

The Project will impact air quality and noise in the area but will be reduced during closure. The primary impact will be caused by dust, vehicle emissions, and odors, and the visual impacts will be light and noise**.** The Project has developed a series of mitigation measures to reduce dust emissions such as compacting; maintenance and watering of trails and roads; limiting speed limits for machinery; wheel cleaning;

consistent preventive maintenance of machinery and systems; installing dust collection devices on some equipment; modifying blasting designs, etc.

Both the air quality and noise impacts will potentially have the greatest effect on nearby population centers. Therefore, the company will set up monitoring devices in the area of influence as required by law.

Finally and in order to establish the effect of blasting on the urban areas and nearby homes, blasting will be designed with conditioning factors so as to avoid any risk to people and property.

20.5.2 Impact on the Soil

The main impacts to soils are its' removal and stockpiling, which occurs during pre-stripping and grubbing. The impact of the project on the soil may involve a loss of resources, quality alterations, a risk of pollution, increased erosion, etc. The Company will implement protective and corrective measures to minimize the impact to soils. This is will include reducing the surface area of the affected soils, and stockpiling the removed soil so that it can be re-used for reclamation.

There is also a risk of accidental contamination from the operation of machinery or inadequate waste management. The mitigation measure will be to immediately apply corrective measures to minimize the impact especially if there is a risk to surface or ground water.

20.5.3 Impact on the Geomorphology

All mining projects pose a direct impact to the geomorphology given that the slopes are modified, which creates elevations and depressions on the ground associated with the construction of stockpiles, tailings deposits, mine pits, water basins, etc.

Portions of the mine area have already been disturbed and altered from a geomorphological perspective by the historical mine pits and waste rock storage facilities. The development of the project would increase the impact on the geomorphology during the active mining phase. However, backfilling the pits with mine tailings would aid in returning the initial physiographic characteristics.

20.5.4 Impact on Vegetation

The development of the Project requires the removal of some vegetation in the mining areas or where the support facilities are to be located. This will include the temporary loss of affected habitats which will be progressively restored, to the extent possible, during reclamation.

Several Habitats of European Interest are affected by the Project, two of which are Priority Listed:

- 4020 Temperate Atlantic wet heaths with *Erica ciliaris* and *Erica tetralix*
- 91E0 Alluvial forests with *Alnus glutinosa* and *Fraxinus excelsior*

Therefore, a more in-depth study will be performed to confirm whether these listed species are actually in the areas affected by the project and what the level of conservation is required. If their existence is confirmed, the Environmental Impact Assessment will provide protective, corrective, and compensatory measures for their conservation, as necessary.

20.5.5 Impact on the Fauna

The main direct impact on the fauna is the temporary elimination of surface habitats. The removal of vegetation would displace the associated fauna and it would be placed in nearby areas with similar characteristics.

Prior to the removal of any vegetation, the area will be examined for any nests or areas of animal concentration that would be eliminated. If any protected species are found, a relocation plan will be implemented that will place the individuals in similar habitats with a high hosting capacity.

20.5.6 Impact on the Landscape

Part of the project surface area has previously been altered due to the historic mine operations and shows deep anthropic transformation with respect to the original landscape. Therefore, the current project design and future reclamation will improve the landscape characteristics.

The rest of the Project area will suffer a loss of quality in the landscape due to the construction of the buildings and infrastructure, the development of mine pits, and the creation of the auxiliary tailings deposit, etc.

However, the project provides for the progressive filling of the pits with mine tailings to recover the characteristic contours of the original landscape. This is one of the main landscape impact correction measures. The Project will conduct a Landscape Impact and Integration Study pursuant to Law 7/2008, of 7 July, on the Protection of the Landscape in Galicia.

20.5.7 Impact on Water

The main impacts on surface water will be that the streams and creeks that run through the area will require diversions in order to maintain the structural and hydrological integrity.

The project will have an integrated water management system comprised of perimeter channelling to prevent the non-contact run-off water from entering the project area. All of the run-off water will be collected by an interceptor trench that starts at the head of the PAG waste heap and runs north to south along the east slope of the Vieiro and Arinteiro deposits. The trench will discharge downstream from the old plant tailings basin. The trench will be approximately 3.6 km long with a trapezoidal section and lateral slopes of 1H:1V and coated in riprap material.

Another inner channelling system will transport contact water with a high risk of contamination to corresponding water basins for treatment at the water treatment plant (WTP). The water management system will allow the treated water to be re-used at the process plant.

Proper management of contact water is the main protection measure against the risk of contamination to surface water, groundwater and the soil. To prevent and mitigate the generation of acid drainage, the project mine Waste Management Plan provides for the following actions:

- Underwater waste deposit, whenever possible;
- Waterproofing all waste facilities, as necessary, to prevent possible infiltration to the groundwater;
- Implementation of a water management system with drainage systems, seepage collection ponds, run-off water intercept trenches, and water collection trenches on the waste dumps;
- The progressive closure of the facilities (waste dumps) to encapsulate the waste.

Tailings production is estimated at 91 million tonnes during the 13 years of operation of the concentrator. The tailings will have a final solids content of 67% and are expected to be potentially acid generating (PAG). The surface TMF will have a capacity for 44 Mt of tailings and the Vieiro-Arinteiro TMF will store 47 Mt of tailings.

A tailings thickening system will be implemented downstream of the concentrator to produce a total of 91 Mt of thickened tailings with a final 67% solids content. This will ensure physical and chemical stability of tailings while reducing the size of the embankment wall required for the surface plastic lined TMF. This technology will also achieve greater densities and deposition slope angles, and thus increase the storage capacity while reducing the footprint and the quantiy of fill required for dam construction. Furthermore it will mitigate seepage and can help to control acid generation as a result of a lower hydraulic conductivity and oxygen transmissivity.

All water recovered through the various drainage and seepage control systems is piped to the recovered water pond. This pond is located near the process plant where the water may either be pumped to the plant or treated until it complies with the required environmental levels for discharge.

Seepage detection systems will be installed in lined ponds and will be monitored for tears or holes. Any damage will be immediately repaired to prevent seepage. In addition, the status and operability of the pond pumping systems will be monitored.

A sampling system will be established to control and classify the quality of the surface and groundwater upstream and downstream from the mine.

20.5.8 Impact on the Socioeconomic Environment

The Project will provide many socio-economic benefits for the local and regional communities such as the creation of direct and indirect jobs, licenses, fees and taxes.

The study area has had a progressively decreasing population that is ageing and unemployment levels are increasing. The Project will provide a business opportunity for the region of Galicia due to the generation of direct and indirect jobs and the increased economic benefits in the area. Thus, it will help stabilize the current progressive depopulation which is evident in many Galician towns. Most of the mine employment will come from the local and regional communities.

20.5.9 Social Acceptability

The investments made by the Project and the hiring of personnel will produce benefits for the regional economy as well as create direct and indirect jobs. This is one of the main factors for the social acceptability of the project.

However, the project may also have a negative impact due to environmental disturbances such as dust emissions, noise, vibrations, and increased traffic on the roads, all of which will affect the local communities. Therefore, the Company will provide the most appropriate mitigation measures to minimize these impacts. For example, sampling systems will be installed to monitor the air quality, measure the sound levels from noise sources and sensitive points (nearby homes, estates, etc.), and blasting will be designed to control vibration. These controls will determine whether the measures are adequate to eliminate risks and minimize the disturbance.

20.5.10 Infrastructures and Traffic

The project does not directly impact local roads or other infrastructures. However, the project will affect several roads that run through the area, mainly the primary access roads that connect to the mine site.

The Project will cause increased traffic in the area, which will be particularly heavy during the pre-operational and operational phases. This heavy traffic will also increase the dust emissions and wear on the road surfaces. The Company will employ protective measures to prevent dust and materials from being deposited on the road surfaces by washing wheels and underneath the vehicles. In addition, all loads on trucks will be covered. Road surface deterioration will be monitored and repairs made accordingly.

All traffic regulations on signage and road safety for vehicles exiting the area and entering the roadways will be observed in order to guarantee road safety.

20.5.11 Camino de Santiago

The Camino of Santiago runs through the municipality of O Pino for 18 km. It crosses several parishes, one of which is Arca.

This stretch of the route is characterized by the presence of significant settings and topography with elevation ranging from 250 m to 400 m high. It passes by the Santa Irene Chapel in the parish of Arca where one of the pilgrimage hostels is located.

Decree 227/2011, of the Regional Ministry of Culture and Tourism defines the main Camino de Santiago and establishes the route layout with limits on the historical territory. It also defines the second area of influence, known as the buffer zone.

As shown in Figure 20.3, the Project does not directly affect the Santiago pilgrimage route nor the buffer zone.

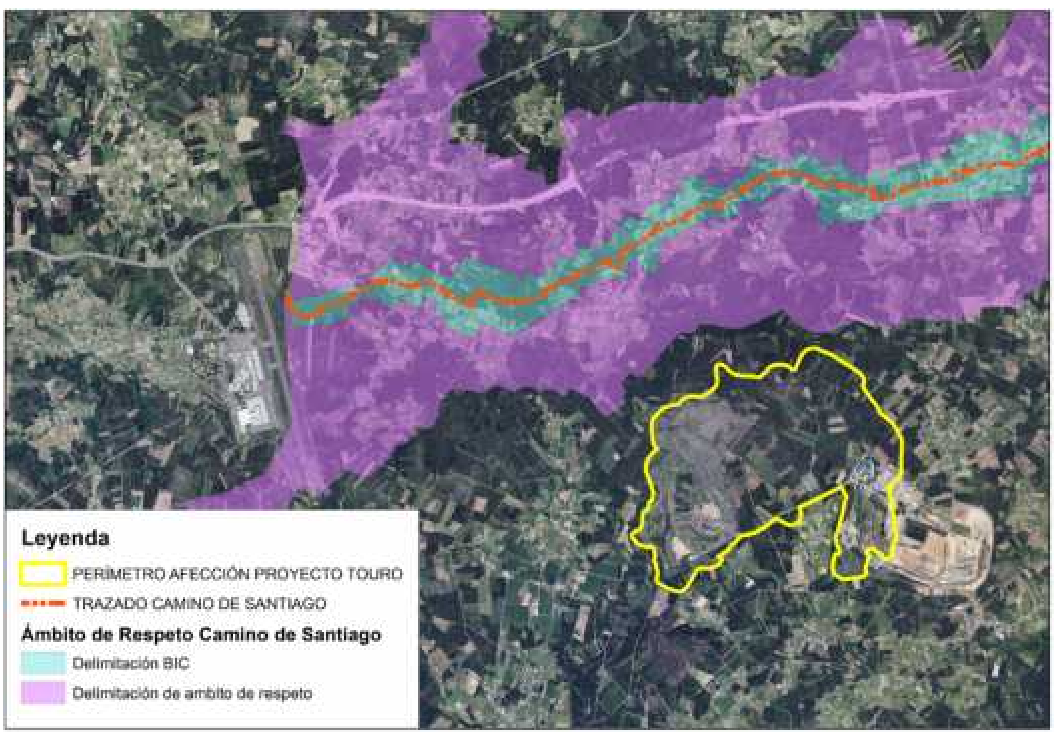


Figure 20.3 - Camino de Santiago (Atalaya 2017)

20.5.12 Archaeological Heritage

The Project could impact archaeology site number 18: Castro da Copa (GA 15085002). Therefore a detailed investigation of the site will be required to quantify the impact and the protective or corrective measures to be implemented. Comprehensive management plans for the protection of cultural heritage on and around the site will be implemented. Any activities affecting items listed in the Heritage Register require detailed documentation and prior authorization from the Department of Culture and Heritage.

20.5.13 Water Users

A detailed study will be conducted to investigate local sources of potable water. This will also include an inventory of local water consumption in order to quantify any impact by the Project. The appropriate mitigation measures will be employed so that local potable water consumers are not affected by the Project.

20.6 Monitoring

Atalaya has developed a comprehensive monitoring program involving a combination of routine visual observations, physical inspections, sampling and analyses of air and water quality, and measurements of noise and vibration. The environmental staff has the responsibility of providing continuous observation and compliance of environmental regulations. The entire mine workforce has a shared responsibility for environmental compliance and undergoes environmental training. A sampling program will be established to assess baseline conditions and monitor seepage from the tailings dam and existing waste dumps. The monitoring program will continually be updated to comply with any regulatory requirements and address operational changes.

20.7 Waste Rock Storage Facilities

The waste rock storage facilities (WRSFs) are described in detail in Chapter 16. In addition, there is old waste-rock material deposited in several areas inside and outside the old workings from the 1970s and 1980s.

Prospective ex-pit waste rock storage facility (WRSF) sites are located to the east of Monte 3, southeast and southwest of the Arinteiro pit, north of the Bama and Brandelos pits, and south of the Bama and Brandelos pits. Some portions of the Bama 1, Bama 2, and Arca 2 phases would be available in the later years of mining for backfilling with waste rock. Additionally, some waste rock in the first several years would be used for constructing the dam for the first tailings storage facility (TSF).

Figure 20.4 shows the locations of the WRSFs with respect to the open pits, ore processing plant area, low grade ore stockpile, and the main haulage roads. WRSFs required for the mine production schedule are shown in blue-grey and additional backfill areas (which become available in Years 11-12) are shown in dark yellow. TSF 1 would be active through Year 7 in the production schedule, after which tailings would be placed in the mined-out Arinteiro-Vieiro open pit. Tailings backfills in open pits may require placement of suitable liner foundation material, which is not presently included in the mine production schedule.

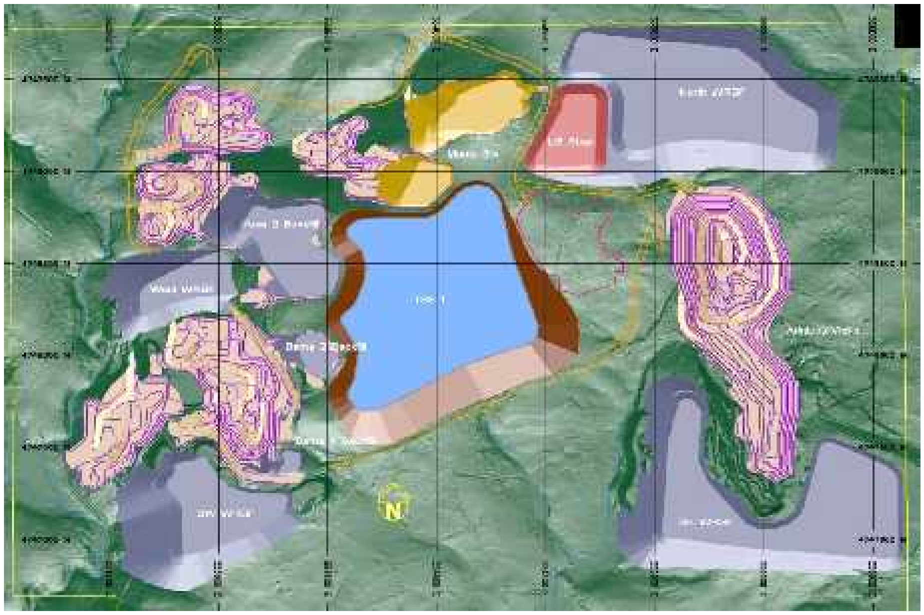


Figure 20.4 - Waste Rock Storage Facility Locations (Rose 2017)

20.8 Tailings Management Facility

The Tailings Management Facility (TMF), as previously described in Chapter 18, will utilize in-pit tailings disposal for a substantial part of the mine life due to the multiple pit operating plan. During the initial years of operation, tailings will be stored in a surface Tailings Management Facility (TMF). After year 8, tailings will be stored in the exhausted Vieiro and Arinteiro open pits.

20.9 Other Wastes

The Atalaya Environmental Management System incorporates procedures and facilities for collecting, segregating, handling, and disposing or recycling all industrial and domestic waste materials. The system includes non-hazardous waste such as paper, glass, aluminum, timber, and other construction materials. Specific bermed areas have been designated for the storage of recyclables. Procedures are also in place for tires, scrap metal and electrical equipment. There are also procedures for hazardous materials such as oils and grease, laboratory reagents and solvents.

20.10 Final Restoration Plan (FRP)

Final restoration is an integral part of the Touro Project. Both the operating and final restoration plans have been developed such that they are compatible with each other. The final reclamation can be implemented and completed in parallel after the cessation of mining in each area and as soon as possible after cessation of processing operations. The objectives of Atalaya's FRP are to:

- Protect the environment,
- Minimize any long term negative environmental impacts of the project,
- Guarantee the chemical stability of water re-uses,
- Ensure that the physical stability of any soils is maintained,
- Recover any soils that will be disturbed during mining operations and reuse them appropriately,
- Recover the natural vegetation in a manner that is compatible with the surrounding habitat,
- Reduce the impact to external areas by dust or other emissions,
- Minimize social impacts as a result of the mine closure at the end of its life.

In accordance with current applicable legislation, Atalaya will submit an FRP as part of the project approval process. After approval of the DIA, the FRP, with any amendments, would be subsequently approved along with final bonding amounts to the Consejeria de Economia, Empleo, and Industria of Galicia Government.

During operations, Atalaya will also submit annual environmental and rehabilitation plans as part of the annual compulsory plans (Plan de Labores). These plans outline the intended environmental monitoring program, completed rehabilitation, on-going rehabilitation of non-operational areas, and details of areas to be disturbed during the forthcoming year. The plans are subject to approval by the Consejeria de Economia, Empleo, and Industria of Galicia Government. Bonding requirements and reimbursement of bonds for satisfactorily completed restoration are also calculated annually.

One year prior to the completion of mining or tailings disposal, Atalaya will submit, for approval, an application for the authorization to abandon the mine, waste dumps, TSF, and site infrastructure. Currently these applications must be made to the Consejeria de Economia, Empleo and Industria of Galicia Government for approval.

20.11 Closure and Environmental Restoration Plan

The main restoration and closure activities are:

- local topographic conditioning,
- backfilling the mine pit with tailings,
- slope drainage,
- closure of tailings deposits via encapsulation,
- facility dismantling (treatment plant and auxiliary facilities),
- waste removal and management,
- addition of topsoil where required,
- preparatory soil work (harrowing or subsoiling with land compacting, fertilisation, etc.),
- grass sowing,
- slope hydroseeding with herbaceous and bushy species,
- planting of tree and/or bushy species.

20.11.1 Closing the Facilities

Closure plans will be completed pursuant to the provisions of Spanish Royal Decree 975/2009 on mine industry waste management and the protection and rehabilitation of spaces affected by mining activities in accordance with article 34.

A description of the closure work for the different Project facilities has been included in this section.

20.11.1.1 Back-filling mine pits

To the extent possible, PAG and NAG mine tailings will be used to back-fill the pits such that the final contour is similar to that of the surrounding area, with gentle slopes and rounded shapes to integrate into the environment, facilitate water evaporation, and proper revegetation.

20.11.1.2 Plant tailings deposit

The following measures are proposed for closure of the tailings management facilities:

Operation of the surface TMF will cease in year 7. The entire tailings surface will be covered with PAG waste rock and compacted granular material up to the elevation of the dam crest. Cover material will be placed forming a minimum 1% slope to help drain direct precipitation water and minimize seepage. The drainage water collection ponds will remain in operation and collected water will be pumped to the water treatment plant located in the concentrator until the seepage flows decrease.

The Vieiro-Arinteiro TMF will cease operation in year 13. Tailings will be at a lower elevation than the minimum exit elevation in the open pits; water will build up as a result of precipitation and runoff inside the open pits. A three-layer cover has been proposed using PAG waste rock, low permeability material, and compacted granular material. The cover will be built up to the maximum elevation for the installation of the liner system, if required, and placed to maintain a 1% slope. The pits are expected be naturally flooded.

As part of the closure measures, dismantling, de-energizing, decommissioning, and removal of remaining facilities, including the tailings processing plant will be undertaken.

The water treatment plant located in the concentrator will remain operative during closure to receive water from the TMFs.

20.11.1.3 PAG Waste Rock Storage Facility (WRF);

The PAG waste rock storage facility (WRF) will be encapsulated on the surface during the operation and will be constructed using a modular layout with the side slopes constructed to their final contour during operations. This will allow the use of organic matter that was removed during site preparation to be re-applied and seeded as the slopes are completed.

The slopes are designed to maintain their physical stability over the long-term and will be inspected periodically to ensure the integrity.

The PAG WRF shall be progressively closed to manage acid drainage and metal leaching. A low permeability cover will be installed to prevent oxidation. The drainage water collection system will remain active during closure and the collected water in the seepage collection pond shall be pumped to the water treatment plant.

20.11.1.4 Facility dismantling

The process plant and associated facilities will be dismantled at closure. All steel structures, concrete, pipes, fixed and portable equipment, metal coatings, etc. will be removed. The surface structures and bearing structures shall be demolished to the ground level, leaving only the foundations at the site.

Any ponds not used during post-closure will be dismantled and filled. Electrical equipment, cables, and line posts will also be removed.

20.11.1.5 Access closure

The main access road to the mine facilities will be left intact to allow access for monitoring personnel during the post-closure period. All other accesses to the mine and waste facilities will be closed.

20.11.1.6 Surface grading

After dismantling and removing all of the structures, the surface will be re-graded to return the area to conditions compatible with the natural landscape. The remaining excavations will be re-graded or filled with local material. Any remaining roads that are not being used for post closure monitoring will be closed and re-graded.

20.11.1.7 Water management

The intercept trench that was installed to capture non-contact run-off will not be modified or reclaimed at closure and will continue to divert water away from the Project site. Waterproof covers will prevent meteoric waters from contacting the waste material. The reclaimed surfaces of these tailings deposits and WRF will have engineered channels to manage run-off. The monitoring wells will be maintained at closure to verify the performance of the facility encapsulation.

20.11.1.8 Environmental restoration actions

The various restoration tasks for the closed and prepared surfaces are as follows:

- Topsoil addition and spreading. The thickness of the topsoil layer will depend on the area where it will be applied and the amount of revegetation, but typically varies between 30-50 cm thick.
- Preparatory soil work. The purpose of this work is to improve the textural and structural characteristics of the soil by increasing the water infiltration capacity and aeration, and facilitating the mechanical penetration of the plant roots. This work consists of ripping the land and making modifications to provide adequate organic matter and nutrients in the soil.
- Sowing. The purpose of this sowing is to plant an herbaceous cover on the land where formal planting will be completed at a later date. There are areas where this type of cover will only be applied to pasture land due to its structural characteristics and to recover habitat.
- Hydrosown land will be used on slopes to stabilise the soil and foster revegetation, and preventing soil erosion.
- Fields. Plant cover similar to the pre-existing vegetation will be chosen for the revegetation of the surfaces affected by the project.
- Provide maintenance of re-vegetated areas until they are self-sustaining.

20.11.2 Maintenance and Control after Closure

Post closure general maintenance and monitoring activities include:

- The intercept trench will be maintained to remove any material that obstructs the flow and repair the sections that may have fallen or suffered other effects.
- An annual inspection of the side slopes to ensure there are no major deformations and/or cracks or erosion gullies that may compromise the long-term stability. Additional inspections may be warranted following extraordinary meteoric or seismic events in the area.
- Monitoring of the groundwater quality downstream from the facilities.
- Monitoring and controlling revegetation through:
    - Work inspections and records,
    - Surveillance of the efficiency of the sowing, hydrosowing, and revegetation,
    - Maintenance of irrigation systems,
    - Herbaceous competition control,
    - Replanting and repeat hydrosowing.

20.12 Health and Safety

Occupational risk prevention, as an activity that is performed within the company, is included in the general management system, which includes all activities and application of an occupational risk prevention plan.

An Occupational Risk Prevention Plan will be established as a tool through which the company's prevention activities will be included in the general management system. The necessary resources to perform the prevention activities are organized through its own prevention service. The internal prevention service is a specific organizational unit that determine the activities and how they are integrated within the entire organization.

For purposes of determining the necessary capacities and skills to evaluate the risks and perform prevention activities, there are three specialty areas or prevention disciplines within the prevention service (workplace safety, industrial hygiene, and applied ergonomics and psycho-sociology). They are implemented by experts with the appropriate skills for the required tasks, with an external occupational medicine service under contract.

Prevention services offer guidance and support based on the types of risk, implementation, and application of a prevention plan. The service also evaluates the risk factors that may affect the workers' safety and health; plan prevention activities and determine the priorities when adopting prevention measures and monitoring their efficiency; employee information and training; provision of first aid and emergency plans; and the surveillance of employee health as related to the risks deriving from their jobs.

The Safety and Health coordinator supervises different activities at the worksite, in particular when those activities may create risks classified as serious or very serious or when activities are performed at the worksite that are incompatible with each other due to implications for workers' health and safety. In addition, necessary measures are adopted so that only company and authorized personnel may access the facilities.

The objective is to establish an integrated system based on OSHA 18001, which validates the management system.

20.13 Public Relations

The Company promotes the establishment of extensive communication channels and actively seeks opportunities for dialogue with its stakeholders to ensure that its business objectives are in line with the needs of society and societal expectations. The company aims to be transparent by providing relevant and accurate information on its activities, fostering constructive dialogue and encouraging continuous improvement.

Since the Project began, the Company has fostered a direct relationship and proactive line of communication with the groups, entities, government authorities, institutions, press and public in general that is interested in its operations. The Company has an open-door policy with a view to being transparent about its activities on the ground.

Members of the Company have also participated in internal, public sector, technical and general events when there is an opportunity to communicate its values and explain its operations and activities. Moreover, the Company is a member of different business and social organizations with which it shares goals and which are used as a platform for its business and communication policies.

Finally, the company has been effectively using all channels inherent to its corporate communication to transmit new developments and explain its ideas using internal resources (website, social media, newsletters, e-mailing etc.) as well as the press (press releases, interviews, participation in special editions, press visits, etc.).

To this end, the hope is that this policy continues to be successful in earning a positive reputation for Atalaya Mining as an excellent and trustworthy mine operator that is integrated within its environment.

This is based on maintaining excellent relations with the media and institutions which lead public opinion through transparency and proactivity, on the one hand; and, on the other, the availability of information and opening of direct communication channels with any member of the public through the extensive circulation of communication materials issued.

Finally, the Company has implemented social responsibility programs through its foundation in order to cover the company objectives beyond the business, leading to a positive reputation for the company.

21 CAPITAL AND OPERATING COSTS

The capital and operating costs provided in the following tables are reported on a 100% project basis and were extracted from the financial analysis prepared by Atalaya which is referenced in Chapter 22. All Euro-based costs have been converted to 2017 US dollars using an average life-of-mine exchange rate of €1:$1.15. Quantities and values are presented in both metric and U.S. customary units as specified. No escalation has been applied to capital or operating costs. All costs have not been adjusted for inflation.

21.1 Assumptions

The parameters used in the analysis are shown in Table 21.1. These parameters are based upon current market conditions, vendor quotes, design criteria developed by Atalaya personnel, and benchmarks against similar existing projects.

Table 21.1 – Assumptions

Description	Parameter	Unit
General Assumptions		
Mine Life	12	years
Operating Days	365	days/year
Production (Years 1-5)	5-6M	tonnes/year
Production (Years 6-8)	7-9M	tonnes/year
Production (Years 9-12)	10M	tonnes/year
Market Assumptions		
Cu price	$3.00	per pound, average life of mine
Ag price	$19.00	per ounce, average life of mine
Concentrate Production (Dry)		
Weight, total life of mine	1,189	kt
Cu Grade	29.7	%, average life of mine
Ag Grade	54.2	g/t, average life of mine
Treatment Charge		
Cu	$95	per t wet concentrate
Refinery Charge		
Cu	$0.10	per pound payable
Ag	$0.40	per ounce payable
Smelter Losses	0.0	%
Freight	$32.30	per t wet concentrate
Penalties		Standard penalty charges apply
Financial Assumptions		
Discount Rate	8	%
Corporate Tax, Net Deduction	25	% of profits
Technical Assumptions		
Diesel Fuel	0.85	$/liter
Power Cost	0.0782	$/kWh
Recovery		
Cu	88.5	%

The revenues from the sale of a copper concentrate containing silver credits are based on an average life-of-mine copper price of $3.00 per pound of contained copper and $19.00 per ounce of contained silver. After deducting refining and treatment charges, penalties and freight and other smelter deductions, the project will realize a net smelter return of approximately 89.8% of the gross concentrate value or $1,716 per tonne of dry concentrate.

21.2 Life of Mine Production

The ore reserve discussed in Chapter 15 is estimated at 90.91 M tonnes of ore averaging 0.43% Cu. Production over the life of mine is summarized in Table 21.2. Although silver was not included in the reserve, it was calculated from the concentrate assays and is shown as a credit in the copper concentrate.

Table 21.2 – Life of Mine Production (total)

Waste	221.33	M tonnes
Ore	90.91	M tonnes
Grade Cu	0.431	%
Contained Metal in concentrate, Cu	346.82	k tonnes
Payable Metal, Cu	340.74	k tonnes
Payable Metal, Ag	925.0	k ounces

21.3 Life of Mine Capital Costs

Life of mine capital costs detailed in Table 21.3 below include both the initial development capital of $164.91M and a capital expansion in year 8 of $30.91M. The capital expansion in Year 8 is required to increase throughput capacity up to 10 Mtpa for treatment of lower grade ore to maintain copper production rates from Year 8 onwards. Sustaining capital averages $3.7M per annum with a total expenditure of $55.3M over the life of mine. The total estimated capital expenditure over the life of mine is $259.54M.

Table 21.3 – Life of Mine Capital Costs

Area	Development Capex USD	Expansion Capex USD	Total LOM Capex USD
Mining	$3.88 M	$0.00 M	$3.88 M
Ore Processing	$61.91 M	$18.73 M	$80.64 M
Tailings & Waste	$20.15 M	$0.00 M	$20.15 M
On Site Infrastructure	$13.36 M	$0.00 M	$13.36 M
Indirects	$20.89 M	$4.94 M	$25.83 M
Owner's Costs	$19.42 M	$2.90 M	$22.32 M
Miscellaneous	$25.30 M	$3.62 M	$28.92 M
Sustaining Capex			$55.22 M
Closure Capex			$9.22 M
Total LOM Capex	**$164.91 M**	**$30.19 M**	**$259.54 M**

21.4 Life of Mine Operating Costs

The life of mine operating costs are based on a combination of estimated costs and actual operating costs obtained from the Atalaya's existing Riotinto operations located in the south of Spain. Both fixed and variable costs have been estimated for the life of mine and are summarized in Table 21.4 below;

Table 21.3 - Life of Mine Operating Costs

Description	**USD**	**$/tonne ore**	**$/lb Cu**
Exploration	$13.49 M	0.15	0.02
Mining Costs	$541.37 M	5.96	0.71
Processing Costs			
Labor	$69.79 M	0.77	0.09
Power	$194.30 M	2.14	0.25
Maintenance Materials	$38.58 M	0.42	0.05
Reagents	$24.54 M	0.27	0.03
Consumables	$102.72 M	1.13	0.13
Miscellaneous	$35.71 M	0.39	0.05
TSF Management	$11.62 M	0.13	0.02
Water Treatments	$36.01 M	0.40	0.05
G&A	$72.85 M	0.80	0.10
Total Site Operating Costs	**$1,141.00 M**	**12.55**	**1.49**
Total Off-Site Operating Costs	**$230.65 M**	**2.54**	**0.30**
Total Operating Costs	**$1,371.65 M**	**15.09**	**1.79**
C1 Cash Costs (net silver credits)			**1.73**
AISC (net silver credits)			**1.85**

Mining costs, inclusive of those capitalized, are equivalent to an average unit cost of USD$5.96 per tonne of ore. The average site operating cost is $12.55 per tonne of ore. Silver by-product credits assume 934,000 oz. are sold at $19.00/oz. over the life of mine. Site Operating Costs equivalent average $1.52 per pound of copper sold with off-site operating costs adding $0.31 per lb. copper sold for a total of $1.73 per lb. copper sold net of silver credits.

21.5 Taxes and Royalties

21.5.1 Royalties

There are no payable royalties applied to this project.

21.5.2 Taxes

Regular tax is computed by subtracting all allowable operating expenses, overhead, depreciation, amortization and depletion from current year revenues to arrive at the taxable income. The tax rate is then determined from the published Spanish progressive tax schedule. An operating loss may be used to offset taxable income, thereby reducing taxes owed.

The general rate of corporate tax in Spain was reduced from 30% to 28% in 2015 and further reduced to 25% in 2016. Tax losses are allowed to be carried forward; up to 60% of previous year's losses could be offset against the current tax year taxable profit.

Specifically, the mining industry in Spain has certain tax benefits such as freedom of amortization and depletion factor. The depletion factor is a tax figure, established in Spain with the aim of promoting geological research and mining of non-renewable resources. By means of this tax, companies have the ability to deduct from their tax base an amount which contributes to a fund which subsequently performs new exploration or research works in order to permit the continuity of the mining activity.

22 ECONOMIC ANALYSIS

Atalaya has developed a financial model for the Touro Project that incorporates the updated minable open-pit reserve. On the basis of the latest update of that model, the summary financial forecast for the project is shown in Table 22.1 below. The assumptions for price and financial factors utilized in the financial model and resultant forecasts are as follows:

- All amounts are in constant 2018 US dollars (US$).
- Amounts in Euros (€) were converted to US$ at an average life of mine exchange rate of €1.00:US$1.15
- Copper production is sold at an average life-of-mine copper price of US$3.00/lb.
- Silver production is sold at an average life-of-mine silver price of US$19/oz.
- Corporate tax rate is 25%.

This financial forecast shows that after tax, capital expenditures, and closure costs, the project will generate unlevered total free cash flow of $489.3M which results in an NPV of $179.9M at an 8% discount rate and an IRR of 20.5%. The overall project cash costs (C1), net of silver credits is US$1.73 per pound of copper increasing to US$1.85 per pound of copper, net of silver credits, adjusting for the sustaining costs (AISC).

Table 22.1 – Cash Flow Forecast

			Year -2	Year -2	Year -2	Year -2	Year -1	Year -1	Year -1	Year -1	Year 1	Year 1	Year 1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Year Ended December 31		**Total / Avg.**	**Q1 2019A**	**Q2 2019A**	**Q3 2019A**	**Q4 2019A**	**Q1 2020A**	**Q2 2020A**	**Q3 2020A**	**Q4 2020A**	**Q1 2021A**	**Q2 2021A**	**Q3 2021A**	**Q4 2021A**	**2022E**	**2023E**	**2024E**	**2025E**	**2026E**	**2027E**	**2028E**	**2029E**	**2030E**	**2031E**	**2032E**	**2033E**	**2034E**	**2035E**	**2036E**
ECONOMIC PARAMETERS																													
Copper	(US$/lb)		$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00
Silver	(US$/oz)		$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00
EUR:USD Exchange Rate (USD/1EUR)	(EUR:USD)		$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15
PRODUCTION SCHEDULE																													
MINING																													
Ore Mined	(KTonne)	90,906	--	--	--	--	--	--	--	--	1,275	1,275	1,275	1,275	6,000	6,000	5,700	5,500	7,200	8,500	9,000	10,000	10,000	10,000	7,542	364	--	--	--
Waste Mined	(KTonne)	221,329	--	--	--	--	8,800	8,800	8,800	8,800	7,412	7,412	7,412	7,412	28,618	23,762	8,448	9,128	9,300	13,850	13,850	13,850	15,700	15,646	4,180	150	--	--	--
Total Material Mined	(KTonne)	312,235	--	--	--	--	8,800	8,800	8,800	8,800	8,687	8,687	8,687	8,687	34,618	29,762	14,148	14,628	16,500	22,350	22,850	23,850	25,700	25,646	11,722	514	--	--	--
Strip Ratio (Waste:Ore)	(ratio)	2.4	--	--	--	--	999.99	999.99	999.99	999.99	5.81	5.81	5.81	5.81	4.77	3.96	1.48	1.66	1.29	1.63	1.54	1.39	1.57	1.56	0.42	(0.07)			
ROM Grade Copper	(%)	0.43%	--	--	--	--	--	--	--	--	0.55	0.55	0.55	0.55	0.53	0.53	0.58	0.61	0.48	0.40	0.39	0.34	0.35	0.33	0.35	0.35	0.00	0.00	0.00
PROCESSING																													
Ore to Mill	(KTonne)	90,906	--	--	--	--	--	--	--	--	1,275	1,275	1,275	1,275	6,000	6,000	5,700	5,500	7,200	8,500	9,000	10,000	10,000	10,000	10,000	- 2,094.00	--	--	--
Recoverable Mill Grade Copper	(%)	0.38%	--	--	--	--	--	--	--	--	0.50	0.50	0.50	0.50	0.48	0.47	0.53	0.56	0.42	0.35	0.34	0.29	0.30	0.29	0.31	0.30	0.00	0.00	0.00
Copper Recovery	(%)	88.6%	--	--	--	--	--	--	--	--	90.9%	90.9%	90.9%	90.9%	90.6%	88.7%	91.4%	91.8%	87.5%	87.5%	87.2%	85.3%	85.7%	87.9%	88.6%	85.7%	--	--	--
Copper Recovered	(Tonne)	346,818.0	--	--	--	--	--	--	--	--	6,375.0	6,375.0	6,375.0	6,375.0	28,800.0	28,200.0	30,210.0	30,800.0	30,240.0	29,750.0	30,600.0	29,000.0	30,000.0	29,000.0	31,000.0	(6,282.0)	--	--	--
COPPER CONCENTRATE PRODUCTION																													
Recovery to Copper Concentrate																													
Contained Copper in Concentrate	(Tonne)	346,818.0	--	--	--	--	--	--	--	--	6,375.0	6,375.0	6,375.0	6,375.0	28,800.0	28,200.0	30,210.0	30,800.0	30,240.0	29,750.0	30,600.0	29,000.0	30,000.0	29,000.0	31,000.0	(6,282.0)	--	--	--
Concentrate Production																													
Copper Concentrate (Dry)	(Tonne)	1,188,888.9	--	--	--	--	--	--	--	--	21,464.6	21,464.6	21,464.6	21,464.6	96,969.7	94,949.5	101,717.2	103,703.7	101,818.2	100,168.4	103,030.3	97,643.1	101,010.1	97,643.1	104,377.1	--	--	--	--
Moisture Content	(%)	9.0%	--	--	--	--	--	--	--	--	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	9.0%	--	--	--	--
Copper Concentrate (Wet)	(Tonne)	1,306,471.3	--	--	--	--	--	--	--	--	23,587.5	23,587.5	23,587.5	23,587.5	106,560.1	104,340.1	111,777.1	113,960.1	111,888.1	110,075.1	113,220.1	107,300.1	111,000.1	107,300.1	114,700.1	--	--	--	--
Copper Payable	(Tonne)	340,741.5	--	--	--	--	--	--	--	--	6,151.9	6,151.9	6,151.9	6,151.9	27,792.0	27,213.0	29,152.7	29,722.0	29,181.6	28,708.8	29,529.0	27,985.0	28,950.0	27,985.0	29,915.0	--	--	--	--
Silver Payable	(koz)	925.0	--	--	--	--	--	--	--	--	16.7	16.7	16.7	16.7	75.4	73.9	79.1	80.7	79.2	77.9	80.2	76.0	78.6	76.0	81.2	--	--	--	--
NET REVENUE																													
Gross Revenue	(US$ mm)	$2,271.2	--	--	--	--	--	--	--	--	$41.0	$41.0	$41.0	$41.0	$185.2	$181.4	$194.3	$198.1	$194.5	$191.4	$196.8	$186.5	$193.0	$186.5	$199.4	--	--	--	--
Offsite Costs	(US$ mm)	($230.7)	--	--	--	--	--	--	--	--	($4.2)	($4.2)	($4.2)	($4.2)	($18.8)	($18.4)	($19.7)	($20.1)	($19.8)	($19.4)	($20.0)	($18.9)	($19.6)	($18.9)	($20.2)	--	--	--	--
Penalties	(US$ mm)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Net Smelter Return	(US$ mm)	$2,040.5	--	--	--	--	--	--	--	--	$36.8	$36.8	$36.8	$36.8	$166.4	$163.0	$174.6	$178.0	$174.8	$171.9	$176.8	$167.6	$173.4	$167.6	$179.1	--	--	--	--
OPERATING COSTS																													
Total Operating Costs																													
Site Operating Costs	(US$ mm)	$1,141.0	--	--	--	--	$14.5	$14.5	$14.5	$14.5	$23.9	$23.9	$23.9	$23.9	$99.7	$92.1	$65.0	$64.5	$77.9	$97.3	$96.7	$104.6	$106.3	$105.7	$81.9	($4.3)	--	--	--
Off-Site Costs	(US$ mm)	$230.7	--	--	--	--	--	--	--	--	$4.2	$4.2	$4.2	$4.2	$18.8	$18.4	$19.7	$20.1	$19.8	$19.4	$20.0	$18.9	$19.6	$18.9	$20.2	--	--	--	--
Total Operating Costs	(US$ mm)	$1,371.7	--	--	--	--	$14.5	$14.5	$14.5	$14.5	$28.0	$28.0	$28.0	$28.0	$118.5	$110.5	$84.8	$84.6	$97.7	$116.7	$116.7	$123.6	$125.9	$124.6	$102.1	($4.3)	--	--	--
C1 Cash Costs (net silver credits)	(US$ / lb Cu)	$1.73	--	--	--	--	--	--	--	--	$2.04	$2.04	$2.04	$2.04	$1.91	$1.82	$1.30	$1.27	$1.49	$1.82	$1.77	$1.98	$1.95	$2.00	$1.53	--	--	--	--
AISC (net silver credits)	(US$ / lb Cu)	$1.85	--	--	--	--	--	--	--	--	$2.24	$2.24	$2.24	$2.24	$1.97	$2.07	$1.35	$1.43	$1.59	$2.07	$1.88	$2.09	$2.02	$2.05	$1.57	--	--	--	--
INCOME STATEMENT																													
Income Statement																													
Net Smelter Return	(EUR mm)	1,774.4	--	--	--	--	--	--	--	--	32.0	32.0	32.0	32.0	144.7	141.7	151.8	154.8	152.0	149.5	153.8	145.7	150.8	145.7	155.8	--	--	--	--
Cash Operating Costs	(EUR mm)	(992.2)	--	--	--	--	(12.6)	(12.6)	(12.6)	(12.6)	(20.8)	(20.8)	(20.8)	(20.8)	(86.7)	(80.1)	(56.5)	(56.1)	(67.8)	(84.6)	(84.1)	(91.0)	(92.4)	(91.9)	(71.2)	3.8	--	--	--
EBITDA	(EUR mm)	782.2	--	--	--	--	(12.6)	(12.6)	(12.6)	(12.6)	11.3	11.3	11.3	11.3	58.0	61.7	95.3	98.7	84.2	64.9	69.6	54.7	58.3	53.8	84.6	3.8	--	--	--
Corporate Tax	(US$ mm)	($165.7)	--	--	--	--	--	--	--	--	($0.6)	($0.6)	($0.6)	($0.6)	($3.8)	($12.9)	($23.3)	($24.0)	($22.1)	($14.5)	($9.7)	($10.8)	($11.7)	($10.2)	($19.1)	($0.9)	($0.1)	--	--
Reclamation	(US$ mm)	$9.9	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	$1.54	$3.39	--	--	$0.36	$5.93	($1.30)	--
Capital Expenditures	(US$ mm)	($254.4)	($14.3)	($21.5)	($21.5)	($21.5)	($21.5)	($21.5)	($21.5)	($21.5)	($2.2)	($2.2)	($2.2)	($2.2)	($0.8)	($13.1)	($0.2)	($7.7)	($3.6)	($13.6)	($34.6)	($4.5)	($1.9)	($0.7)	($0.0)	($0.1)	($0.0)	($0.0)	--
Changes in Working Capital	(US$ mm)	$0.0	--	--	--	--	$1.6	--	--	--	($0.7)	--	--	--	$2.3	($0.7)	($3.5)	($0.2)	$1.6	$2.3	($0.3)	$1.3	($0.1)	$0.2	($3.1)	($0.6)	--	--	--
Unlevered FCF	(US$ mm)	$489.3	($14.3)	($21.5)	($21.5)	($21.5)	($34.4)	($36.0)	($36.0)	($36.0)	$9.5	$10.1	$10.2	$10.2	$64.4	$44.2	$82.6	$81.5	$72.8	$48.8	$35.5	$50.5	$56.8	$51.2	$75.0	$3.0	$5.9	($1.3)	--
PV Factor			0.981	0.962	0.944	0.926	0.908	0.891	0.874	0.857	0.841	0.825	0.809	0.794	0.735	0.680	0.630	0.583	0.540	0.500	0.463	0.429	0.397	0.368	0.340	0.315	0.292	0.270	0.250
NPV (8%)	(US$ mm)	**US$179.9**																											
IRR	(%)	**20.5%**																											

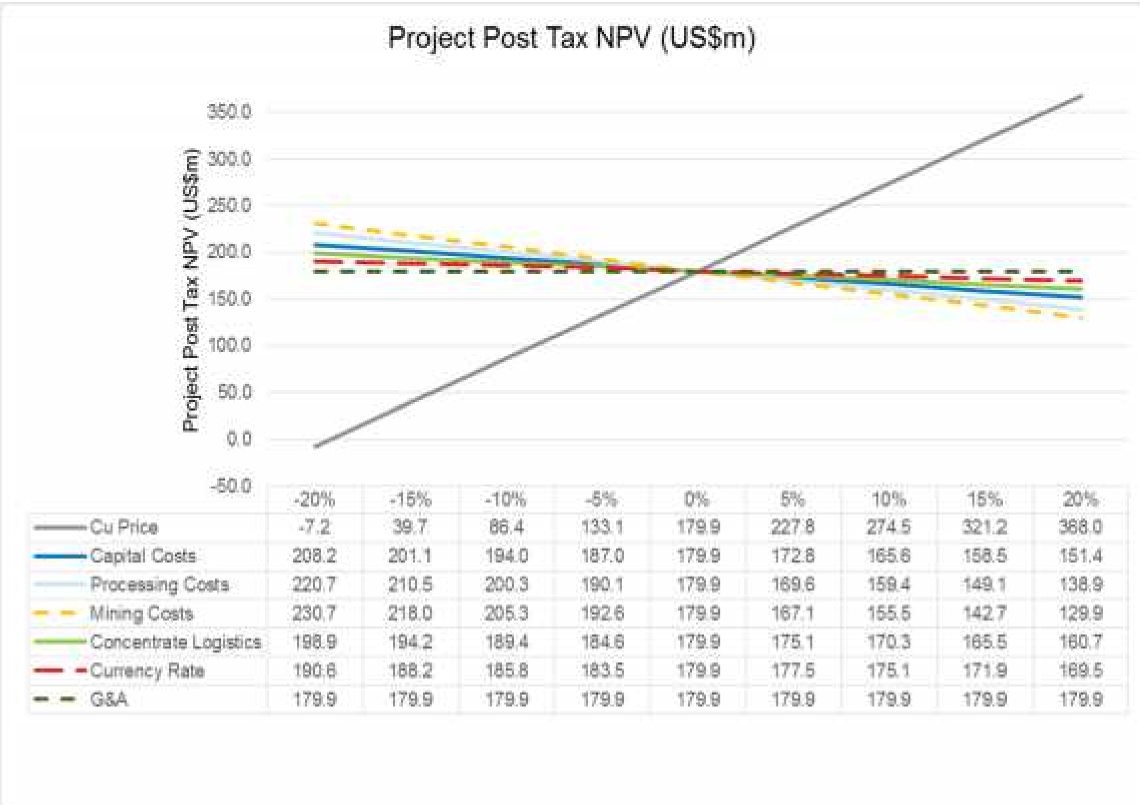
22.1 Forecast Results and Sensitivities

The unlevered free net cash flow of $489.3M generated over the life of the Project as indicated in the cashflow model results in an NPV of $179.9M at a discount rate of 8% and an IRR of 20.5%. The Project's key economic performance parameters are presented below in Table 22.2.

Table 22.2 - Key Performance Parameters

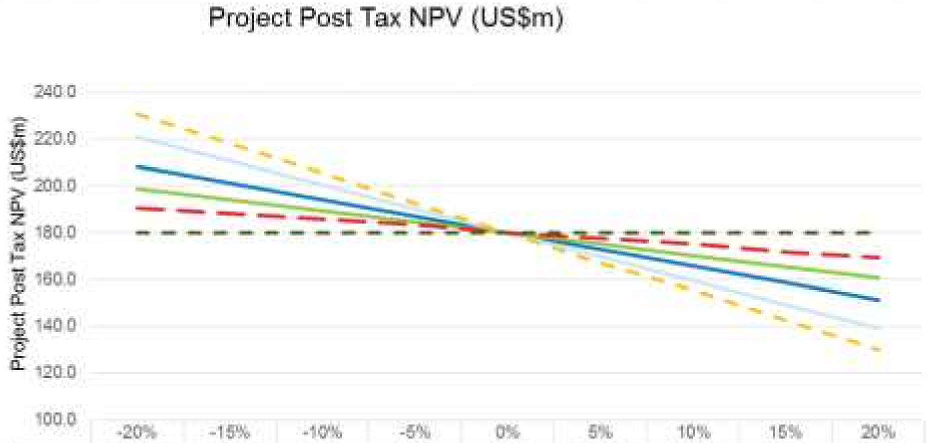
Parameter	Units	Value
Total Cu Production	tonnes Cu in concentrate	346,818
Payable Cu Production	tonnes Cu in concentrate	340,741
Mine Life	Years	12
Operating Cash Cost	US$/lb	1.73
NPV after tax @ 8 %	US$M	179.9
IRR	%	20.5
Copper price	US$/lb	3.00

Sensitivity analyses on the project NPV were performed using 5% increments up to ± 20% on copper pricing, capital costs, operating cost inputs (mining, processing, offsite costs and G&A respectively), and exchange rate for the Euro:US dollar. As expected, the copper price variation has the greatest impact on the project NPV both positive and negative. However, the project NPV remains positive in almost all scenarios regardless of the decreased copper price or increase in capital and operating costs and exchange rate differential (see Figure 22.2).



	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Cu Price	-7.2	39.7	86.4	133.1	179.9	227.8	274.5	321.2	368.0
Capital Costs	208.2	201.1	194.0	187.0	179.9	172.8	165.6	158.5	151.4
Processing Costs	220.7	210.5	200.3	190.1	179.9	169.6	159.4	149.1	138.9
Mining Costs	230.7	218.0	205.3	192.6	179.9	167.1	155.5	142.7	129.9
Concentrate Logistics	198.9	194.2	189.4	184.6	179.9	175.1	170.3	165.5	160.7
Currency Rate	190.6	188.2	185.8	183.5	179.9	177.5	175.1	171.9	169.5
G&A	179.9	179.9	179.9	179.9	179.9	179.9	179.9	179.9	179.9

Figure 22.1 - Sensitivity Analyses (Atalaya 2018)



	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Capital Costs	208.2	201.1	194.0	187.0	179.9	172.8	165.6	158.5	151.4
Processing Costs	220.7	210.5	200.3	190.1	179.9	169.6	159.4	149.1	138.9
Mining Costs	230.7	218.0	205.3	192.6	179.9	167.1	155.5	142.7	129.9
Concentrate Logistics	198.9	194.2	189.4	184.6	179.9	175.1	170.3	165.5	160.7
Currency Rate	190.6	188.2	185.8	183.5	179.9	177.5	175.1	171.9	169.5
G&A	179.9	179.9	179.9	179.9	179.9	179.9	179.9	179.9	179.9

Figure 22.2 – Sensitivity Analysis (Excluding Copper Price) (Atalaya 2018)

23 ADJACENT PROPERTIES

Proyecto Touro is located in the "San Rafael" concession, which is an exploitation permit (CE) 100% owned by Cobre San Rafael S. L. (CSR), a wholly owned subsidiary of Explotaciones Gallegas S. L. (EG). In February 2017, Atalaya exercised an option to acquire a 10% interest in Proyecto Touro located in northwest Spain. The acquisition of the project is based on a staged earn-in agreement increasing from 10% up to an 80% interest once commercial production is achieved.

In addition to the CE San Rafael, where the main orebody is located, Atalaya Mining has acquisition options over a series of exploration permits surrounding Proyecto Touro. These options come from two agreements: Fuente Rosas Concessions (FR) and New CSR applications.

The current permit situation is presented in the table below:

Table 23.1 – Current Permitting

Owner	Permits	Type	Number	Size (mining squares)	Size (ha)	Situation
CSR	SAN RAFAEL	CE	2946	96	2744	Granted
	TOURO	PI	7167	44	1365	New application
	LAVACOLLA	PI	7179	29	811	New application
	A MUIÑA	PI	7178	123	3443	New application
FR	FUENTE ROSAS	PI	7115	64	1792	Granted
	FUENTE ROSAS OESTE	PI	7119	33	924	Granted
	BOIQUIXON	PI	7168	275	7703	New application
	GAMAS	PI	7144	2	56	New application
	GAMAS OESTE	PI	7169	3	84	New application
	FORNAS	PI	7116	12	224	New application

23.1 Cobre San Rafael Concessions (CSR)

In 2017, CSR applied for three new concessions near the San Rafael concession and in the same geological unit and setting. These are Touro, Lavacolla and A Muiña. The total surface area of the four CSR concessions is 84.2 km².

23.2 Fuente Rosas Concessions (FR)

Recently in 2017, Atalaya has signed a new option to acquire 100% of the shares of Explotaciones Gallegas del Cobre (EGC), a privately held company that controls 100% of the exploration concessions immediately surrounding Proyecto Touro.

The FR concessions includes two exploration permits (PI) and four new permit applications. The concessions cover a surface area of 122.7 km², and include some well documented mineralized copper occurrences. (See Figure 24.1).

The financial terms of the deal are subject to a confidentiality agreement and will be announced in due course if the option is exercised. If the acquisition is finally agreed to, the conditions will be similar to the payment terms (based on US cents per pound of reserves) of the current earn-in agreement at Proyecto Touro. The option period is for 2.5 years and approximately 75% of the considerations are conditional on receiving all the operating permits.

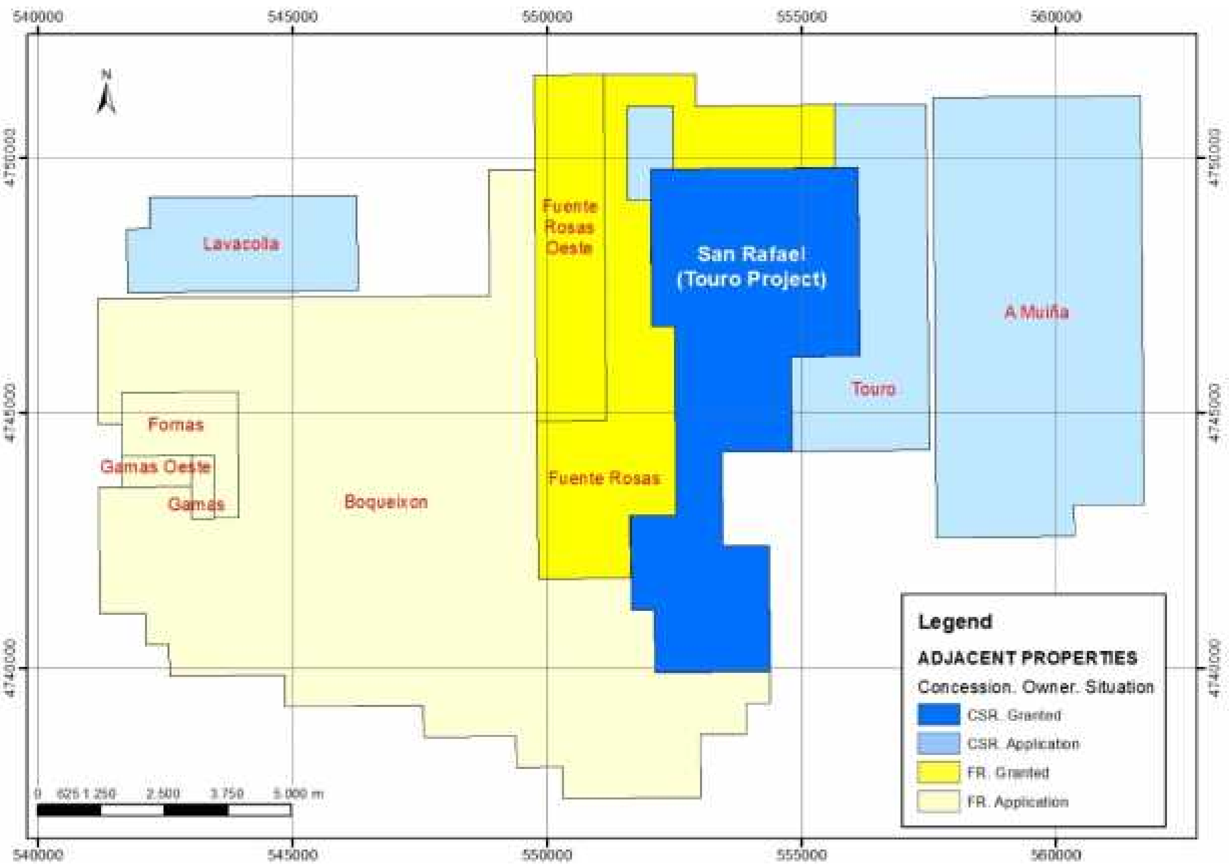


Figure 23.1 – Concession Map (Atalaya 2017)

24 OTHER RELEVANT DATA AND INFORMATION

Atalaya Mining plc (AIM:ATYM, TSX:AYM) announced its unaudited quarterly and interim results for the three and six months to June 30 2017, together with the unaudited, condensed interim consolidated financial statements on September 7, 2017.

The complete unaudited, condensed half yearly financial statements are available under the Company's profile on SEDAR at www.sedar.com and on the Company's website at www.atalayamining.com.

In February 2017, the Company announced the exercise of an option to acquire an initial 10% stake in Proyecto Touro. The agreement is based on a staged earn-in process to acquire up to 80% of the project. The Company has also signed an option agreement to acquire exploration concessions that cover 122.7 km^2 immediately surrounding Proyecto Touro, where mineralized copper occurrences are documented. Permitting of Proyecto Touro is progressing according to schedule.

25 INTERPRETATIONS AND CONCLUSIONS

25.1 Resource Estimation

The resource estimate was prepared using a block model and inverse-distance-power methods to estimate copper grades from the drill hole copper assays. The ore-grade mineralized zones tend to be relatively thin and tabular with significant lateral continuity that is disrupted locally by minor folding and faulting. A trend-surface method was used to allow the estimation to follow the trends despite the minor local fluctuations in the geometry of the mineralization.

Measured and indicated resources have been estimated at nearly 130 Mt grading 0.39% Cu with an additional inferred resource 47 Mt grading 0.37% Cu. The mineral resource is based on a Lerches-Grossman pit shell that was run using a copper price of $3.20/lb. Cu and all resources including inferred resources. All other slope and economic parameters are the same as those used for design of the open pit for reserve estimation. The resulting pit shell is considered to have reasonable prospects for economic extraction, assuming that the inferred resource is converted to measured and indicated by drilling and that the copper price returns to previous levels that were substantially above $3.20/lb. Cu.

25.1.1 Resource Risks and Opportunities

As with all mineral deposits, there is some risk that the resource estimate will not meet expectations due to unknown geologic factors, assaying errors, and limitations of the estimation methods. In addition, if production mining is unable to control dilution and accurately define the limits of ore, ore grades and tonnages may fall short even though the in-situ resource has been estimated accurately. These risks may be controlled through continued re-evaluation and improvement of the resource data and deposit model and stringent grade-control procedures during mining.

Atalaya has continued drilling during 2017 and 2018 and there is significant, completed drilling that has not been incorporated into the resource model. It is expected that this additional drilling will allow conversion of some of the inferred resource to measured and indicated resource. In addition, Atalaya continues to explore adjacent properties and has budgeted for a small additional drilling program in 2018.

25.2 Mining

Mineral reserves have been estimated at nearly 91 Mt grading 0.43% Cu with a stripping ratio of 2.43 (tonnes waste per tonne of ore). This estimate is based on a production schedule derived from designed mining phases based on a Cu price of US$2.60/lb. and a declining cutoff grade strategy to improve project returns. Variable net smelter return (NSR) cutoffs ranging between US$10.00/t and US$14.00/t would be used for the first five years of operation, which would then decline to an internal NSR cutoff of US$8.14/t for the remainder of the mine's life – presently estimated at just over 12 years, excluding an 18 to 24 month preproduction stripping period. Ore processing operations would start at a nominal rate of 6.0 Mtpa, or nearly 16,700 t/d through Year 5. Plant feed rates would then increase to offset declining head grades until reaching a maximum of 10 M t/a, or about 27,800 t/d, in Year 9.

Approximately 62% of the estimated mineral reserves are classified as proven and the remaining 38% as probable mineral reserves.

25.2.1 Risks

As with nearly all metal mining projects, an analysis of economic pit limits indicates potential Touro Project reserve sensitivity to Cu prices. A 13% decline in Cu price from $2.60/lb. to $2.25/lb. would reduce potentially economic measured and indicated resource tonnages by nearly 26% (see Section 15.3).

The economic pit limits and, consequently, mining phase designs were based on ore definition parameters that slightly underestimated the mining costs. Higher operating costs can reduce mineral reserves. This is not however, a significant concern as a conservative Cu price of $2.60/lb. was used instead of higher recent prices of over $3.00/lb. The new knowledge of expected haulage profiles gained from this study will improve future estimation of cost parameters for economic pit analyses.

There are presently no definitive criteria for separating non-acid-generating (NAG) from potentially-acid-generating (PAG) waste rock. Prior mining has shown that significant amounts of PAG material exist in the project site. Atalaya Mining is presently conducting studies to establish criteria for the identification and treatment of PAG waste rock. Such treatment, including appropriate containment facilities, could add significant capital and operating costs to the project's development, and thereby may adversely impact project returns and mineral reserves. The magnitude of such impacts are unknown at this time.

25.2.2 Opportunities

Higher Cu prices could increase mineral reserves through potentially larger economic pit limits and lower cutoff grades. An LG analysis showed that a 15% increase in Cu price from $2.60/lb. to $3.00/lb. could expand potentially economic resources above an internal cutoff by nearly 22%.

In-fill drilling could convert inferred mineral resources to higher classifications. The current pit designs contain nearly 2.8 Mt of inferred mineral resources grading 0.40% Cu. An LG analysis at $2.60/lb. Cu indicates the potential for another 26 Mt of economic mineral resources grading 0.41% Cu if projected inferred mineral resources could be upgraded in classification. It should be remembered, however, that inferred mineral resources are too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that inferred mineral resources will be upgraded to a higher classification.

No resources have been included from Fuente Rosas area where Lundin Mining historically drilled and a mineralized zone is known to exist.

25.3 Processing

The process plant flowsheet was selected based on results from a rigorous testwork program completed at SGS in Perth, Western Australia. The proposed flowsheet will produce a final concentrate of 27% Cu at 90% recovery. The flowsheet will include a conventional SAG mill – ball mill grinding circuit followed by a copper flotation recovery circuit. The 6 Mt/y concentrator includes:

- Primary crushing.
- Primary and secondary grinding.
- Rougher flotation.
- Regrinding.
- Three stages of cleaner flotation.
- Concentrate thickening and filtration.

Design of the concentrator considered the ability to upgrade the circuit in order to process ore at higher throughput rates to maintain a constant production profile. Additional grinding, pebble crushing, and flotation capacity are required to increase the throughput rate from 6 to 10 Mt/y.

The testwork program was considered to be of a DFS level and as such minimal testwork is recommended in the DFS phase. Comminution testwork was completed to confirm parameters for use in the sizing and selection of the crushing and grinding circuits. Flotation grind optimization tests were conducted in order to complete a high-level grind size economic evaluation. The optimum primary grind size was determined to have a P_{80} of 125 µm.

Batch flotation tests were completed to determine the optimal reagent addition scheme and regrind size. The results from the batch tests enabled the development of standard flotation test conditions, used in flotation locked cycle testwork. The results of the locked cycle tests provided an estimation of the concentrate grade and recovery that can be expected from the flotation circuit.

Results from the flotation testwork were also used to determine predictive grade and recovery models. The grade model considered head grade and S:Cu ratio with the following relationship:

$$C \quad G = 30.1 + 4.93 \times H \quad G - 0.29 \times (\frac{S}{C})$$

Three separate recovery models were developed, based on trends observed between different ore types:

$$V \quad P \quad G \quad R = 5.09 \times \ln(H \quad) + 94.9$$

$$A \quad O\ he \quad P \quad G \quad R = 7.196 \times \ln(H \quad) + 92.9$$

$$T \quad G \quad R = \frac{100 \times C \quad \times (H \quad - 0.094)}{H \quad \times (C \quad - 0.094)}$$

Where:

- CG = Concentrate Grade (%Cu).
- HG = Head Grade (%Cu).

Based on the proposed production schedule and the information available to date, it is possible to safely store the tailings produced from the metallurgical process. In order to satisfy the production requirements, both surface disposal and in-pit disposal is required. It is considered that DCTs shall be used from the start of operations, with a filter plant potentially coming on line in year 6.

A total of 44 Mt of tailings will be stored in the surface HDPE plastic lined TMF and 47 Mt in the Viero-Arinteiro Pit. Drainage, seepage and precipitation water will be reclaimed from the facilities and returned to the mineral process plant.

At closure, the facilities shall be capped and drainage and stored water from rainfall shall be reclaimed and treated in the water treatment plant. Tailings processing plant structures will be decommissioned.

26 RECOMMENDATIONS

Atalaya is currently exploring and developing the Touro copper Project. The Project consists of a series of deposits of which 4 have been partially mined previously (Arinteiro, Vieiro, Bama and Brandelos) and 2 that have not been mined previously (Monte de las Minas and Arca). The Project site comprises an area of influence of approximately 1,060 ha. Current mineral rights include approximately 15,300 ha of contiguous ground (including a mining concession).

Total proven and probable mineral reserves at $2.60/lb. Cu price are estimated at nearly 91 Mt grading 0.43% Cu. Contained copper is estimated at just over 391,000 tonnes.

The results of this prefeasibility study support a recommendation to advance the project to the next level of evaluation – a definitive feasibility study. The following actions are recommended to be completed by the early stages of the feasibility study:

- During 2017, Atalaya has completed an exploration drilling program targeting areas of inferred mineral resources with the intent of upgrading classifications and/or expanding known mineralized areas. Atalaya plans an additional 2,000 m of RC and 1000 m of diamond drilling for 2018. This should be sufficient to bring the drill hole database to the feasibility level. The estimated cost for the 2018 drilling is €357,550.

- The resource model should be updated to include the additional drilling that was completed during 2017. This update should include estimation of sulfur and iron grades, using cokriging with copper in those areas with missing S and Fe assay. Incorporate a model for non-acid-generating/potentially-acid generation material based on geochemical testing that is currently in progress. Cost of this work is estimated as $70,000.

- Identify possible additional waste rock storage sites if new areas are found that could be mined and thus require more waste rock stripping. Condemnation drilling should be conducted to confirm each site's suitability for waste rock storage. The cost for this work is estimated at $210,000.

- Complete studies to characterize NAG/PAG waste rock so that quantities of each type can be estimated and appropriate plans developed. Such studies are currently in progress. The cost for this work is estimated at $46,000.

- Complete hydrology studies, including groundwater inflows to planned open pits, with the intent of developing a comprehensive site water balance. Water quality from pit inflows should be evaluated for suitability for plant make-up water or for potential discharge, and the study should include recommendations for any treatment that may be required. The study would be useful for determining dewatering requirements for open pit development and pit slope depressurization. The cost of this work is estimated at $32,000.

- Perform a geotechnical analysis of the existing open pit designs to check stability using preliminary results from the above hydrology/groundwater study. Identify areas of concern and new slope angle recommendations, if needed. The cost for this work is estimated at $11,000.

The proposed design of the tailings management facilities has been developed based on several assumptions taken on the site, tailings and waste rock characteristics and its expected performance. The main aspects and potential impacts on the design that need to be verified in further project study phases are summarized below:

- The design of the TMFs is highly sensitive to variations on the dry densities assumed for the deposition of thickened tailings. If the dry density achieved during operations is lower than 1.5 t/m^3, the required storage capacity may be increased up to a 20% of the current storage design.

  A single deposition slope of 4% has been adopted for this study. No concavity or variability on the beach profile has been considered and should be evaluated in next study phases. While in Vieiro and Arinteiro, the main impact is the potential requirement of the lining system, in the surface TMF, fluctuations on the deposition slopes should impact directly on the enbankment height requirement. To promote higher slopes, a multiple discharge system has been included.

  To better determine the geotechnical properties of tailings, a specific laboratory testing program should be performed in the next study phase. Stability studies on tailings beaches are recommended once the final design slopes have been defined.

  Even the design parameters are confirmed by laboratory testing, during operations, strong efforts and controls should be implemented on the deposition activities to promote consolidation of tailings and enhance the stability of the beaches.

- Tailings are expected to be potentially acid generating and so will be disposed in a plastic lined impoundment. A geochemical testing program should be conducted to determine the oxidation rates of tailings and the metal leaching potential. Depending on the results obtained, and the inclusion of wet or dry covers, or any other suitable methodology, to avoid oxidation may be considered. These studies are currently in progress.

- For construction activities, the availability, quality and appropriate segregation by geochemical properties of the waste rock, has been assumed. Especially during the construction of the first stage of the surface TMF embankment. In the unlikely event that there is not enough good quality rock available, the final dam wall slopes may be flattened and the geotechnical performance of the fill material should be evaluated.

  If NAG/PAG waste rock segregation isn't feasible during mining operations, the surface TMF dam should be constructed with PAG waste rock and the dam design should include measures to ensure both physical and chemical stability, such as zoned saturation of the walls.

- For the Vieiro and Arinteiro pit backfill, the oxidation rate of the PAG waste rock and the natural water level raising has been assumed to enable the proposed construction sequence. Kinetic tests and additional hydrogeological studies should be conducted to determine the feasibility of the proposed solution. If this assumption is not confirmed, other construction material should be evaluated.

  As part of the same assumption, a site-specific hydrogeological model for open pits is recommended to support a better definition of the water table's behavior and to verify that the proposed raises in the TMF are feasible.

- The requirement of the proposed buttress and the foundation preparation at the southern sector of the surface TMF's should be verified based on an additional geotechnical study to better characterize the foundations and particular the low-strength sand lens. The geotechnical field campaign should include CPTu, geophysics survey methods and the undisturbed soil sampling when feasible.

- A better understanding of the rheology, particle size distribution, mineralogy and dewatering properties of the tailings, obtained by laboratory testwork, will refine the design criteria of the Tailings Processing Plant. These studies are currently in progress.

- Once this information is produced in the next phases of the project, the thickener unit area can be optimized to obtain the thickened tailings solids content and yield stress required for the tailings management strategy as previously discussed. This laboratory data will also aid in the proper sizing of pumps, pipelines and the flocculant plant.

Based on the production schedule and the information available to date, 4 Waste Rock Facilities are proposed. According to a preliminary classification of the geochemical characteristics of the waste rock, the NAG waste rock shall be stored in 3 WRF and the PAG waste Rock shall be stored in a one single WRF. Bama and Arca pits will be used for NAG waste rock backfilling.

The PAG WRF will have an underdrain system to collect potential seepage. A progressive closure cover shall be placed during operations to to avoid oxidation. Drainage water will be collected in a pond located downstream of the WRF and conveyed to the water treatment plant located in the process plant.

The WRFs have been designed based on the following assumptions that shall be verified in further project study phases.

- WRFs foundations have been assumed competent. Specific geotechical studies shall be performed to characterize foundations. If any fatal flaw is identified, new locations shall be proposed;

- WRFs geometries have been proposed considering a typical minimum friction angle of 40° for the waste rock. A geotechnical characterizartion of the waste rock should be conducted to establish the streght parameters of the waste rock. If minor angles result, the WRFs design should consider flatter slopes which could increase the footprint, resulting in potential interferences with pit layouts and other infrastructure.

- NAG and PAG production has been defined based on the geochemical characterization report [Ref. 1]. NAG and PAG segregation has been assumed feasible during mining operations. If this couldn´t be achieved, all WRFs would be classified as PAG and their designs shall include water management measures and progressive closure covers.

- Tailings and waste rock codisposal could be anlysed in further project study phase as a measure to avoid oxidation and control acid drainage and metal leaching.

The following testwork is recommended in the next study phase for process plant and tailings design:
- Concentrate thickening.
- Concentrate filtration.
- Concentrate handling.
- Tailings thickening.
- Tailings dam testwork.

A preliminary economic analysis for the optimal comminution circuit grind size was completed from results of the metallurgical testwork program. The analysis was completed for a throughput rate of 5 Mt/y and has not been re-evaluated at 6 Mt/y. A re-evaluation should occur at 6 Mt/y during the next study phase.

A comminution trade-off study was completed based on an initial processing rate of 5 Mt/y increasing to 8 Mt/y over the life of mine. It is recommended that a further trade-off study be conducted in the next phase to consider 6 Mt/y and 10 Mt/y. It is unlikely the comminution circuit will change but this should be confirmed prior to commencing the Definitive Feasibility Study (DFS).

It is recommended that a detailed geotechnical investigation be conducted by a suitably qualified geotechnical consultant on the proposed plant site location and borrow pits during the DFS with a combination of test pits, boreholes, and testwork to define in-situ ground conditions and engineering design parameters for earthworks and civils.

It is recommended that ground survey in the proposed plant and non-process infrastructure areas be checked against contour data to validate same. Depending on the results of the confirmatory survey further survey may be required over these areas to develop detailed contour data for the DFS design.

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28 QUALIFIED PERSONS

The Consultants preparing this Technical Report are specialists in the fields of geology, mineral resource and reserve estimation and classification, environmental engineering, permitting, metallurgical testing, mineral processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants employed in the preparation of this report has any beneficial interest in Atalaya. The results of this Technical Report are not dependent on any prior agreements regarding the conclusions that are reached. There are also no undisclosed agreements concerning any future business between the Consultants and Atalaya.

The following Consultants, by virtue of their education, experience and professional associations, are considered Qualified Persons (QP) as defined by the NI 43-101 standards and are members in good standing of the appropriate professional institutions.

CERTIFICATE OF AUTHOR

Alan C Noble, P.E.
Ore Reserves Engineering
12254 Applewood Knolls Drive
Lakewood, Colorado 80215

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Email: a.noble@comcast.net

I, Alan C Noble, do hereby certify that:

I am a self-employed Mining Engineer working as Ore Reserves Engineering at 12254 Applewood Knolls Drive, Lakewood, Colorado 80215 and have carried out this assignment as overall author/reviewer.

1. This certificate applies to the Technical Report titled "Technical Report on the Mineral Resources and Reserves of the Touro Copper Project, located in the La Coruña province of the Galicia Autonomous Region in north-western Spain (the "Technical Report"), and dated April, 2018 for Atalaya Mining Plc.
2. I graduated from the Colorado School of Mines in Golden, Colorado with a Bachelor of Science Degree in Mineral Engineering in 1970.
3. I am a Registered Professional Engineer in the State of Colorado, USA, PE26122
4. I have practiced my profession as a mining engineer continuously since 1970, for a total of 47 years. During that time, I worked on mineral resource estimates and mine planning for over 158 mineral deposits.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, registration as a professional engineer, and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for the overall review of the Technical Report. I prepared the mineral resource estimate that is the subject of Chapter 14. In addition, I contributed to Chapters 7 through 12.
7. I visited the property on 7 November 2016.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I am independent of the issuer, Atalaya Mining Plc., applying all of the tests of Section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1, and Chapters 7 through 16 of the Technical Report have been prepared in compliance with the instrument and form.
11. At the effective date of the Technical Report, to the best of my information, knowledge and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

I consent of the filing of the Technical Report with any Canadian stock exchange and consent other securities regulatory authority and any publication by them for regulatory purposes of the technical report.

Dated the 19 day of March 2018.

"Signed and Sealed, Alan C. Noble, P.E."

Alan C. Noble, P.E. 26122

CERTIFICATE OF QUALIFIED PERSON

WILLIAM L. ROSE, P.E.

I, William L. Rose, P.E., do hereby certify that:

1. I am the Principal Mining Engineer of:

   WLR Consulting, Inc.
   9386 West Iowa Avenue
   Lakewood, Colorado 80232-6441, USA

   This certificate applies to the Canadian National Instrument 43-101 Technical Report titled "Technical Report on the Mineral Resources and Reserves of the Touro Copper Project, located in the La Coruña province of the Galicia Autonomous Region in north-western Spain (the "Technical Report"), and dated March, 2018 for Atalaya Mining Plc.

2. I graduated from the Colorado School of Mines with a Bachelor of Science degree in Mining Engineering in 1977.

3. I am a:
   - Registered Professional Engineer in the State of Colorado (No. 19296),
   - Registered Professional Engineer in the State of Arizona (No. 15055), and
   - Registered Member of the Society for Mining, Metallurgy and Exploration, Inc. (No. 2762350RM), all in good standing.

4. I have practiced my profession as a mining engineer continuously for 40 years since my graduation from college. I have been involved in open pit mine operations in both management and engineering positions, and have extensive experience in mine design and planning. I have conducted estimations of mineral resources and reserves, mine production schedules, equipment and workforce requirements, and capital and operating costs for numerous projects in North, Central and South America, Europe, Africa and Asia.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 15 and 16, and relevant portions of Sections 1, 25 and 26 of the Technical Report.

7. I have personally inspected the subject property on 7 November 2016.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. As of the effective date of this certificate, to the best of my knowledge, information and belief, Sections 15, 16 and relevant portions of Sections 1, 25 and 26 of the Technical Report contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. I am independent of the issuer, Atalaya RioTinto Minera, S.L.U., applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the public filing of this Technical Report.

Dated this 19 day of March, 2018.

/Original signature and seal on file/

"Signed and Sealed"

____

William L. Rose, P.E. 19296

CERTIFICATE OF QUALIFIED PERSON

Jaye T Pickarts, P.E.
9792 West Unser Avenue
Littleton, Colorado 80128

Telephone: +1 303 570 3370
Email: jtpick2@msn.com

I, Jaye T Pickarts, do hereby certify that:

I am a self-employed Metallurgical and Environmental Engineer, 9792 West Unser Avenue, Littleton, Colorado 80128

12. This certificate applies to the Technical Report titled "Technical Report on the Mineral Resources and Reserves of the Touro Copper Project, located in La Coruña province of the Galicia Autonomous Region in north-western Spain (the "Technical Report"), dated March, 2018 for Atalaya Mining Plc.
13. I graduated from the Montana College of Mineral Science and Technology, Butte, Montana, with a Bachelor of Science Degree in Mineral Processing Engineering in 1982.
14. I am a Licensed Professional Engineer in the State of Colorado, USA, PE37268, State of Wyoming, USA, PE13891 and the State of Nevada, USA, PE020893. In addition, I am a Registered Member of the Society for Mining, Metallurgy, and Exploration (SME) No. 2543360 and a Qualified Person member of the Mining and Metallurgical Society of America (MMSA).
15. I have practiced my profession continuously since 1982, and have been involved in mineral processing, and metallurgical and environmental engineering for a total of 34 years.
16. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, registration as a professional engineer, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
17. I am responsible for the preparation of Chapters 1, 2, 3, 4, 5, 6, 19, 20, 24, 25, 26, and 27 of the Technical Report.
18. I have visited the property on 7 November, 2016.
19. I have not had prior involvement with the property that is the subject of the Technical Report.
20. I am independent of the issuer as described in Section 1.5 of NI 43-101.
21. I have read NI 43-101 and Form 43-101F1, and Chapters 1, 2, 3, 4, 5, 6, 19, 20, 24, 25, 26, and 27 of the Technical Report have been prepared in compliance with the instrument and form.
22. At the effective date of the Technical Report, to the best of my information, knowledge and belief, Chapters 1, 2, 3, 4, 5, 6, 19, 20, 24, 25, 26, and 27 of the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 19 day of March 2018.

"Signed and Sealed, Jaye T. Pickarts, P.E."

Jaye T. Pickarts, P.E. 37268/13891/020893

CERTIFICATE OF QUALIFIED PERSON

Monica Barrero Bouza, EuroGeol.
Consulting Geologist
Rio San Pedro, 7, 4ºD
Oviedo, Asturias 33001, Spain

Telephone: +34 616 56 57 52
Email: monicabbouza@gmail.com

I, Monica Barrero Bouza, do hereby certify that:

I am a self-employed Geologist working as independent consulting geologist at Rio San Pedro 7, 4ºD, Oviedo, 33001 (Asturias, Spain) and have carried out this assignment as author/reviewer.

23. This certificate applies to the Technical Report titled "Technical Report on the Mineral Resources and Reserves of the Touro Copper Project, located in the La Coruña province of the Galicia Autonomous Region in north-western Spain (the "Technical Report"), and dated March, 2018 for Atalaya Mining Plc.
24. I graduated from the Department of Geology of the University of Oviedo (Spain) with a Bachelor of Science Degree in Geology in 1996.
25. I am a registered Eurogeologist and registered member of the Official Association of Professional Geologist of Spain (ICOG), EuroGeol 1328.
26. I have practiced my profession as a geologist continuously since 1997, for a total of 20 years. My relevant experience includes exploration and mining geology, hydrogeology, rock mechanics and ground instrumentation. I have been involved in several scoping studies and pre-feasibility studies. I have participated in projects in precious and base metals.
27. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, registration as a Eurogeologist, and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
28. I am responsible for compiling and verifying the drilling and geological data for resource estimation, Chapters 7 through and 12 of the Technical Report.
29. I last visited the property that is the subject of this Technical Report in September 2012.
30. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was as an employee of Lundin Mining during the due-diligence program carried out in 2012.
31. I am independent of the issuer, Atalaya Mining Plc., applying all of the tests of Section 1.5 of NI 43-101.
32. I have read NI 43-101 and Form 43-101F1, and Chapters 7 through 12 of the Technical Report have been prepared in compliance with the instrument and form.
33. At the effective date of the Technical Report, to the best of my information, knowledge and belief, Chapters 7 through 12 of the Technical Report contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

I consent of the filing of the Technical Report with any Canadian stock exchange and consent other securities regulatory authority and any publication by them for regulatory purposes of the technical report.

Dated the 19 day of March 2018.

"Signed and Sealed, Monica Barrero Bouza, EuroGeol."

Monica Barrero Bouza, EuroGeol 1328

CERTIFICATE OF QUALIFIED PERSON

John Darryl Fleay
Level 1, 632-634 Newcastle Street
Leederville, Perth, Western Australia
6007

Telephone: +61 430 166 522
Email: john.fleah@minnovo.com.au

I, John Darryl Fleay, do hereby certify that:

I am employed as Process Manager at Minnovo Pty Ltd, Level 1, 632-634 Newcastle Street, Leederville, Perth, Western Australia, 6007

1. This certificate applies to the Technical Report titled "Technical Report on the Mineral Resources and Reserves of the Touro Copper Project, located in La Coruna province of the Galicia Autonomous Region in north-western Spain (the "Technical Report"), dated March, 2018 for Atalaya Mining Pic.
2. I graduated from Curtin University (Western Australian School of Mines), Kalgoorlie, Western Australia, with a Bachelor of Engineering Degree in Minerals Engineering in 1986.
3. I am a Fellow of Australian Institute of Minerals and Mining (FAusiMM(CP)) – Member Number 320872.
4. I have practiced my profession continuously since 1992, and have been involved in design, construction, commissioning and operation of mineral processing projects for a total of 25 years.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Chapters 13 and 17 of the Technical Report.
7. I have visited the property 7th and 8th September 2015.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I am independent of the issuer as described in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1, and Chapters 13 and 17 of the Technical Report have been prepared in compliance with the instrument and form.
11. At the effective date of the Technical Report, to the best of my information, knowledge and belief, Chapters 13 and 17 of the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 10th day of March 2018.

Signed and Sealed,

John Darryl Fleay, FAusiMM(CP) # 320872.

CERTIFICATE OF QUALIFIED PERSON

Matthew James Langridge
Level 1, 632-634 Newcastle Street
Leederville, Perth, Western Australia
6007

Telephone: +61 400 202 793
Email: matt.langridge@minnovo.com.au

I, Matthew James Langridge, do hereby certify that:

I am employed as Manager Projects at Minnovo Pty Ltd, Level 1, 632-634 Newcastle Street, Leederville, Perth, Western Australia, 6007

1. This certificate applies to the Technical Report titled "Technical Report on the Mineral Resources and Reserves of the Touro Copper Project, located in La Coruña province of the Galicia Autonomous Region in north-western Spain (the "Technical Report"), dated March, 2018 for Atalaya Mining Pic.
2. I graduated from Curtin University, Perth, Western Australia, with a Bachelor of Science Degree in Civil Engineering in 1986.
3. I am a Fellow of Australian Institute of Minerals and Mining (FAusiMM(CP)) – Member Number 323727.
4. I have practiced my profession continuously since 1987,and have been involved in design and construction of mineral processing projects for a total of 30 years.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Chapters 18 (excluding section 18.1), 21 and 22 of the Technical Report.
7. I have visited the property 7th and 8th September 2015.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I am independent of the issuer as described in Section1.5 of NI 43-101.
10. I have read NI43-101 and Form 43-101F1, and Chapters 18 (excluding section 18.1), 21 and 22 of the Technical Report have been prepared in compliance with the instrument and form.
11. At the effective date of the Technical Report, to the best of my information, knowledge and belief, Chapters 18 (excluding section 18.1), 21 and 22 of the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 10th day of March 2018.

Signed and Sealed,

Matthew J Langridge, FAusiMM(CP) #323727

CERTIFICATE OF QUALIFIED PERSON

Alistair Cadden, C.Eng, MIMMM, B. Eng (Hons), MSc DIC
181, Magdalena, Piso 3
Las Condes, Santiago, Chile

Telephone: +56 22 616 2000
Email: acadden@golder.com

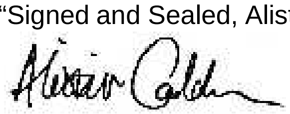
I, Alistair Cadden, do hereby certify that:

I am a consulting civil engineer, employed full time by Golder Associates, S.A, at the address set out above.

1. This certificate applies to the Technical Report titled "Technical Report on the Mineral Resources and Reserves of the Touro Copper Project, located in La Coruña Province of the Galicia region, Spain (the "Technical Report"), dated March 2018 for Atalaya Mining Plc.
2. I graduated from the University of Leeds, UK, with a Bachelor of Science Degree in Civil Engineering in 1987. I hold a Master of Science Degree in Soil Mechanics from the University of London UK (1992), and a Diploma of Imperial College
3. I am a Charted Engineer in the UK, (Engineering Council UK Registration number 569976), and a corporate member of the Institute of Materials, Minerals and Mining (membership number 450990)
4. I have practiced my profession continuously since 1987, and have been involved in civil engineering, mining engineering and mine waste management for a total of 30 years.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, registration as a professional engineer, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Chapters 18.1, 20.8 and 20.11.1.2 of the Technical Report, and portions of Chapter 26 in relation to the Tailings Management Facilities and Tailings Processing Plant.
7. I have not visited the property.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I am independent of the issuer as described in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1, and Chapters 18.1, 20.8 and 20.11.1.2 of the Technical Report, and portions of Chapter 26 in relation to the Tailings Management Facilities and Tailings Processing Plant have been prepared in compliance with the instrument and form.
11. At the effective date of the Technical Report, to the best of my information, knowledge and belief, Chapters 18.1, 20.8 and 20.11.1.2 of the Technical Report, and portions of Chapter 26 in relation to the Tailings Management Facilities and Tailings Processing Plant, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 19th day of February, 2018.

"Signed and Sealed, Alistair Cadden, C.Eng MIMMM, B.Eng (Hons), MSc DIC"



DIC Principal, Golder Associates, S.A