Cobre Las Cruces: Polymetallic Primary Sulphide Project

Andalucía, Spain

NI 43-101 Technical Report

February 2024

Carmelo Gomez Dominguez BSc Hons (Geology), EurGeol, Group Principal Geologist, Mine and Resources, FQM (Australia) Pty Ltd
Anthony Robert Cameron BEng (Min), Grad Dip Bus, M Comm Law, FAusIMM, Mining Consultant, Cameron Mining Consulting
Robert Stone (QP) BSc (Hons), CEng, ACSM, Group Consulting Metallurgist, FQM (Australia) Pty Ltd

DATE AND SIGNATURE PAGE

This Ni43-101 technical report is an update on the Mineral Resources and Reserves of the Cobre Las Cruces Polymetallic Primary Sulphide project in Andalucía, Spain. The report was completed by the following Qualified Persons on behalf of First Quantum Minerals Ltd.

Qualified Person

Signed by:

Carmelo Gomez Dominguez, BSc Hons (Geology), EurGeol
Group Principal Geologist - Mine and Resources, FQM (Australia) Pty Ltd
Responsible for Items: 7-12 and 14
Date: 20.02.2024

Qualified Person

Signed by:

Anthony Robert Cameron, BEng (Min), Grad Dip Bus, M Comm Law, FAusIMM
Consulting Mining Engineer, Cameron Mining Consulting Ltd
Responsible for Items: 15 and 16 and for those items not covered specifically by the other qualified persons. Date: 20.02.2024

Qualified Person

Signed by:

Robert Stone, BSc Hons (CEng), ACSM
Group Consulting Metallurgist FQM (Australia) Pty Ltd
Responsible for Items: 13 & 17
Date: 20.02.2024

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

ITEM 1 SUMMARY

This Technical Report on the Cobre Las Cruces Polymetallic Primary Massive Sulphide (PMS) Project (the property or Project) has been prepared by Carmelo Gomez Dominguez, Anthony Robert Cameron and Robert Stone for First Quantum Minerals Ltd (FQM, the issuer or the Company).

This report supersedes the NI 43-101 Technical Report titled "Cobre Las Cruces: Polymetallic Primary Sulphide Mineral Resources, Andalusia, Spain, NI 43-101 Technical Report, January 2022" (the 2022 CLC Technical Report) and focusses on the Mineral Resources and Reserves of PMS and associated Stockwork mineralisation.

The effective date for the Mineral Resource and Mineral Reserve estimates is 30th September 2023.

1.1 Property Overview, Location & Ownership

Cobre Las Cruces (CLC) is a wholly owned subsidiary company of First Quantum Minerals Ltd (FQM or the Company). The CLC operation is located 20 kilometres northwest of Seville, the capital city of the Seville Province within the Andalusian region of southern Spain (Figure 1-1). From 2009 to 2021 CLC has been producing cathodes from open pit mined secondary copper ore by means of hydrometallurgical processing.

Figure 1-1 Location of Cobre Las Cruces Operation, Seville, Spain (CLC, 2022)

The operation is situated in the relatively flat Guadalquivir River basin that extends some 420km from the Alcaraz and Segura Mountain systems in the east to the Costa de la Luz on the Atlantic Coast in the west. The

Guadalquivir basin is set between a long range of low mountains to the north (Sierra Morena) and a group of formations making up the Cordilleras Béticas mountain range to the south.

Seville province is semi-arid with a Mediterranean climate having hot, dry summers (May through September) and warm, fairly wet winters. Extended periods of drought are common. Autumn and winter rainstorms are generated from south westerly winds that bring hot humid air into the area. Summer temperatures average 30°C and reach above 40° in July and August. Winter temperatures average 15°C with minimum temperatures down to -5°C in December and January. The mean annual rainfall is 550 millimetres but ranges from 300 to 1,000 millimetres with the bulk of this generally falling within a four-month period in winter from October through to the end of January.

The city of Seville has a large well-educated population of approximately 685,000 people. In addition, the village of Gerena is about 4 km to the northwest and has a population of approximately 7,400 people. The Company's local employment policy had 42% of the 700 strong workforce during the open pit operation sourced from the surrounding four municipalities.

CLC is accessible by well-maintained all-weather paved roads. The N-630 highway is in service to the east side of the property, while the SE-3410 highway crosses the property in the south. Seville province has a good rail network, including a high-speed passenger rail service to Madrid. Seville has an international airport with connections throughout Europe and is 30 minutes by car from the operation.

1.2 Project Background

The Las Cruces deposit was discovered in 1994 by Riomin Exploraciones, S.A. (Riomin), a wholly owned subsidiary of Rio Tinto plc. The discovery was the result of exploration drilling on a gravimetric survey anomaly. Riomin drilled the deposit between 1994 and 1999 and prepared a feasibility study in 1998.

On May 3, 2005, Inmet Mining Corporation (Inmet) announced that it had entered into an agreement with the owners MK Resources and its majority shareholder, Leucadia National Corporation, to acquire a 70 percent interest in CLC. The purchase of CLC by Inmet was completed on August 22, 2005.

In January 2007, SNC-Lavalin was awarded the EPCM contract to construct the plant and associated infrastructure. By the end of 2007, Inmet reported that essentially all engineering and procurement was complete, 51% of construction was complete, and 71% of total physical progress was complete. In addition, 7.9 million bank cubic metres (bcms) of waste was removed from the mine in 2006 and 19.9 million bcms were removed in 2007.

In May 2008 the authority responsible for the dewatering and reinjection system (DRS) permit notified CLC that it had suspended authorization of the DRS for the Project. This suspension prevented CLC from mining in the open pit until the suspension was lifted in April 2009. The first ore was delivered from the mine to the plant on May 26, 2009, and the first copper cathode was produced in June 2009.

In April 2013 FQM acquired 100% ownership of CLC after the successful takeover of Inmet.

1.3 Project Approvals

The site operating entity is Cobre Las Cruces S.A.U. (CLC) of which FQM holds a 100% interest.

Mining Concession No. 7,532 was granted to CLC by the Andalusian Regional Ministry for Employment and Technological Development on August 6, 2003 and expires on August 6, 2033. The original concession permitted the production of copper via open pit mining methods.

A modification of the Concession (7532-A) was granted in June 2021 to permit CLC to produce the four metals (copper, zinc, lead and silver) via underground mining operations as well as construct and operate the new Polymetallic Refinery (PMR) plant and associated infrastructure.

CLC currently owns approximately 1,000 hectares of land for its operations with the mineral and surface rights fully enclosing the deposit. Environmental permits for the PMR Project were granted in June 2021 and the water concession was granted in March 2023. At this point in time, there are no known factors or risks that may affect access, title, or the right or ability of CLC to conduct underground and PMR operations.

1.4 Project Development Status

1.4.1 Production History

Production and exploitation of secondary mineralisation on the Property commenced in early 2009 with mining operations ceasing in 2021 during which 14.42Mt of secondary sulphide Mineral Reserve was mined and processed at an average grade of 5.31% copper.

As of September 2023, there remains 5 million tonnes of Indicated Primary Polymetallic sulphide (PPS) Mineral Resource on stockpile with an average copper grade of 1.19%, zinc grade of 2.21%, lead grade of 1.63% and 29.40 g/t of silver grade. There remains 2.7 million tonnes of gossan with gold grades of 2.58 g/t on separate stockpiles.

Tailings reprocessing commenced in January 2021 and was completed in June 2023 after which the current plant was placed onto care and maintenance. A total of 3.5Mt of plant feed was processed with 23,072 tonnes of copper cathode produced at an average recovery of 73.3%.

Table 1-1 Operations Summary for 2009 to 2023

PROCESSING SUMMARY (LOM)	SECONDARY ORE	TAILINGS
LOM	**2009-2021**	**2021-2023**
Tonnes to plant	14,423,926	3,468,371
Cu % average	5.31%	1.04%
Cu metal feed	766,005	35,918
Average Leaching Cu recovery %	90.7%	73.3%
Cathode tonnes	678,734	23,072

1.4.2 PMR Project Scope

The proposed PMR Project comprises a new dual drift access underground mine producing up to 2.0Mtpa, feeding a polymetallic refinery with a design throughput of up to 2.2 Mtpa. Additional ore feed of 0.2Mtpa will be sourced from existing surface stockpiles to meet plant capacity. Specific components of the Project will include:

- A multi-phased underground mine extending to an ultimate depth of 450 m, producing up to 2.0Mtpa and utilising Drift and Fill (DAF) and Longhole Open Stoping (LHOS) extraction methods.
- Reclamation of an existing surface stockpile containing 5Mt of primary sulphide ore that was mined during the open pit operations.
- Modifications to the existing processing facility and the construction of additional processing facilities approximately 1km to the east.
- A new paste plant located midway between the existing and proposed new process plants, to provide cemented backfill suitable for underground support or filtered tailings suitable for dry stacking.

- Expansion of the existing In-Pit Tailings Storage Facility (IPTF) located within the existing open pit.
- Construction of surface facilities to support underground mining including ventilation fans, decline portals, a fuel storage, explosives facility, mine offices and bathhouses.
- Construction of materials handling infrastructure (Railveyor) including three ore passes, transfer points, two rockbreaker silos, two loading stations underground, and a surface unloading facility.
- Construction of underground support infrastructure including shafts, dewatering pump stations, maintenance workshop, paste backfill delivery system, electrical substations, refuge stations, mine communications, and other ancillary installations.
- Expansion of existing water management infrastructure including treatment plants, dewatering bore and supporting pumping systems.
- Upgrading of existing surface haulage and internal access roads.
- Construction of expanded electrical distribution and communication facilities to new processing facilities and underground mine areas.

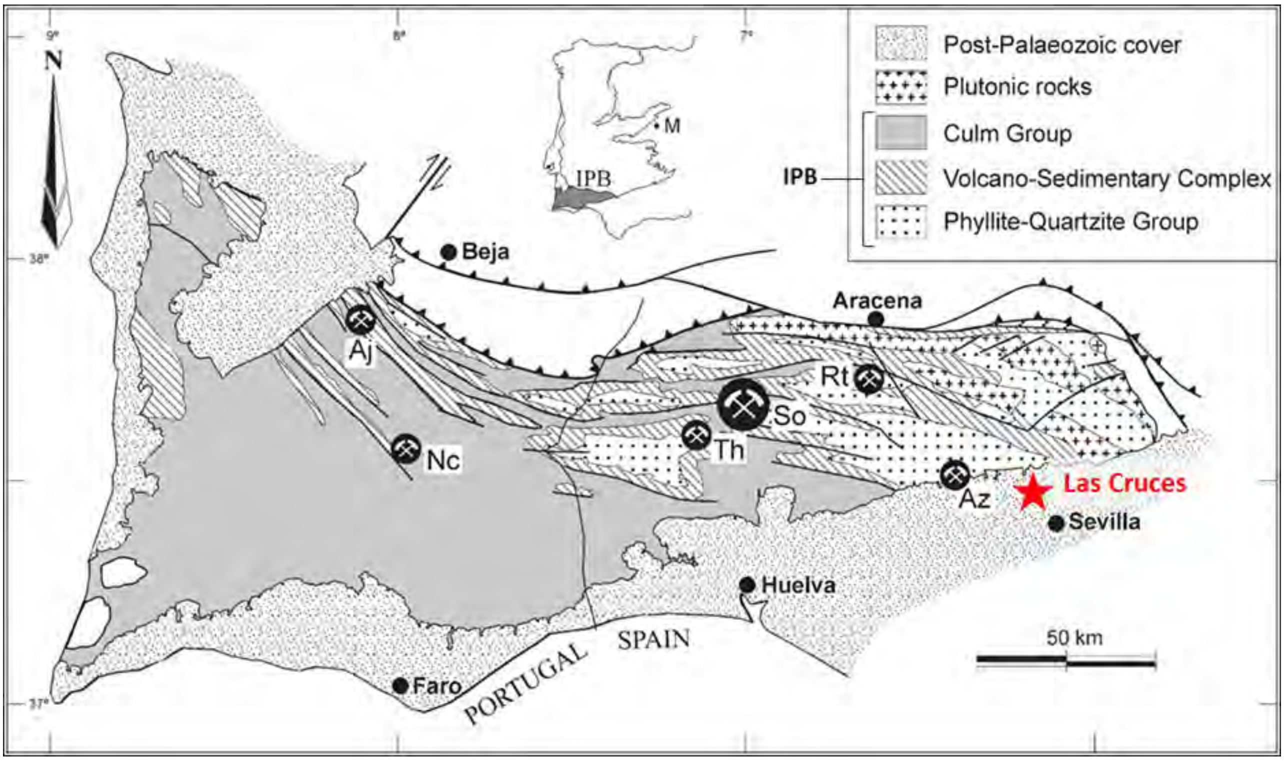
A preliminarily construction schedule has been developed identifying a 6-month pre-project development phase followed by a 25-month construction period commencing from the final investment decision. The process plant schedule has been developed by Lycopodium with the mining schedule developed by the CLC mining team based on mining consultants IGAN production schedules.

1.5 Regional and Local Geology

The CLC Primary Polymetallic Sulphide (PPS) mineralisation is in the eastern portion of the Iberian Pyrite Belt (IBP) geology, well known for its volcanogenic massive sulphide (VHMS) ore deposits. The IPB extends for some 300 kilometres from southern Portugal into southern Spain and can be up to 80 kilometres wide in places. The orebody is one of many (more than 100 deposits) high-grade polymetallic primary massive sulphide deposits located in the IPB.

PPS mineralisation is hosted by volcanic and sedimentary rocks of the late Devonian to early Carboniferous period and were deposited in a narrow, shallow intra-continental submarine setting. Post depositional secondary copper enrichment occurred in the upper part of the deposit, forming massive secondary sulphide mineralisation. Polymetallic Primary massive sulphide and associated semi-massive stockwork deposits are located immediately below the secondary sulphide mineralisation. The CLC deposit is buried under 100 to 150 metres of sandstone and calcareous mudstone (marls) and, as a result, does not outcrop on the surface.

Figure 1-2 **CLC in relation to the prevailing Iberian Pyrite Belt (IPB) Geology (modified from Gonzalez et al, 2006)**



The deposit is a Volcanogenic Massive Sulphide (VMS) deposit which has resulted from precipitation of metals from hydrothermal fluids in a volcanically active sub-marine area. The metals precipitate as sulphides inter-layered with the volcanic sediments. Consequently, the mineralisation is hosted by black shales, sedimentary and felsic volcaniclastic rocks.

The deposit is characterized by a tabular dipping massive sulphide body underlain by a stockwork vein system. The deposit has been folded and faulted, resulting in dislocated mineralised volumes. Hydrothermal alteration is typically chloritic/quartz sericitic and is restricted to the stockwork zones. The CLC deposit shows zonation where copper rich volumes are flanked by zinc and lead mineralisation. Lead, zinc and silver tend to concentrate in unevenly distributed polymetallic zones. Minor gold is noted in both polymetallic and copper rich areas. Pyrite is dispersed throughout.

At CLC, post-deposition tectonism has uplifted the deposit, exposing it to erosion and weathering with partial oxidation of some of the primary sulphides. An uppermost copper-leached gossan resulted with some immobile gold and silver, and overlies the cementation zone of secondary sulphide.

1.6 Metallurgy

Extensive comminution and metallurgical testing have been undertaken both on site and at various reputable laboratory facilities under the guidance of CLC personnel and using representative primary ore samples from both surface and underground.

Concentrate atmospheric leaching for copper and zinc recovery, as well as proof of concept chloride leach testing for lead and silver recovery, was undertaken in 2014. Additional pressure leaching testwork performed by SGS in 2015 was discarded in favour of the atmospheric leaching route, supplemented by successful in-house atmospheric leach testwork performed by the CLC Technology department at the same time.

Having established the flowsheet for the proposed plant expansion, a pilot plant campaign was undertaken between July 2016 and May 2017 by the technology division of the CLC operation at their facilities in Seville. The campaign comprised separate milling and flotation programmes to generate sufficient bulk concentrate for the subsequent hydrometallurgical testwork programme. The pilot plant campaign successfully demonstrated the proposed processing route as economically viable for the treatment of primary CLC sulphide ores.

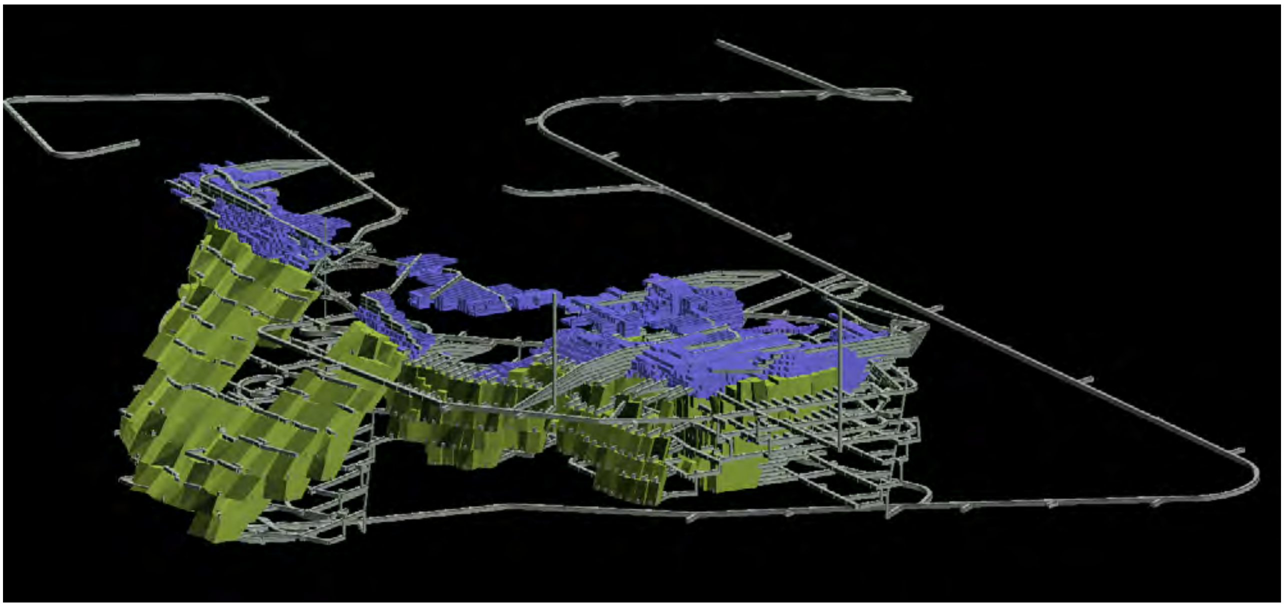
Based on the current mine schedule developed for the Project, the level of testwork is deemed appropriate for the preliminary processing plant design while some more specific testwork may be required to finesse some minor design assumptions.

1.7 Mining

The CLC underground mine plan will produce 2.0Mtpa of polymetallic ore through a combination of Longhole open stoping (LHOS) and Drift and Fill (DAF) extraction methods. The primary means of backfilling will be cemented paste fill, delivered to the underground stope voids through a pipe network, to maximise both mining recovery and productivity.

Figure 1-3 shows where the two mining methods are applied. Blue is DAF, which is in the upper levels of the mine, and green is LHOS.

Figure 1-3 Cobre las Cruces Underground Mining Methods



The mining shapes resulting from the design process are distributed across four discrete mine blocks. The four blocks are Block 1, Block 2, Block 3 and Block 5. The four zones extend over a 260-metre vertical distance, from - 410 mRL to -151 mRL and contain approximately 36.5 Mt of Mineral Resource at an average grade of 1.14% Cu, 1.00% Pb, 2.23% Zn and 26.00 g/t Ag.

The mining method proposed in each of the four main blocks is as follows: -

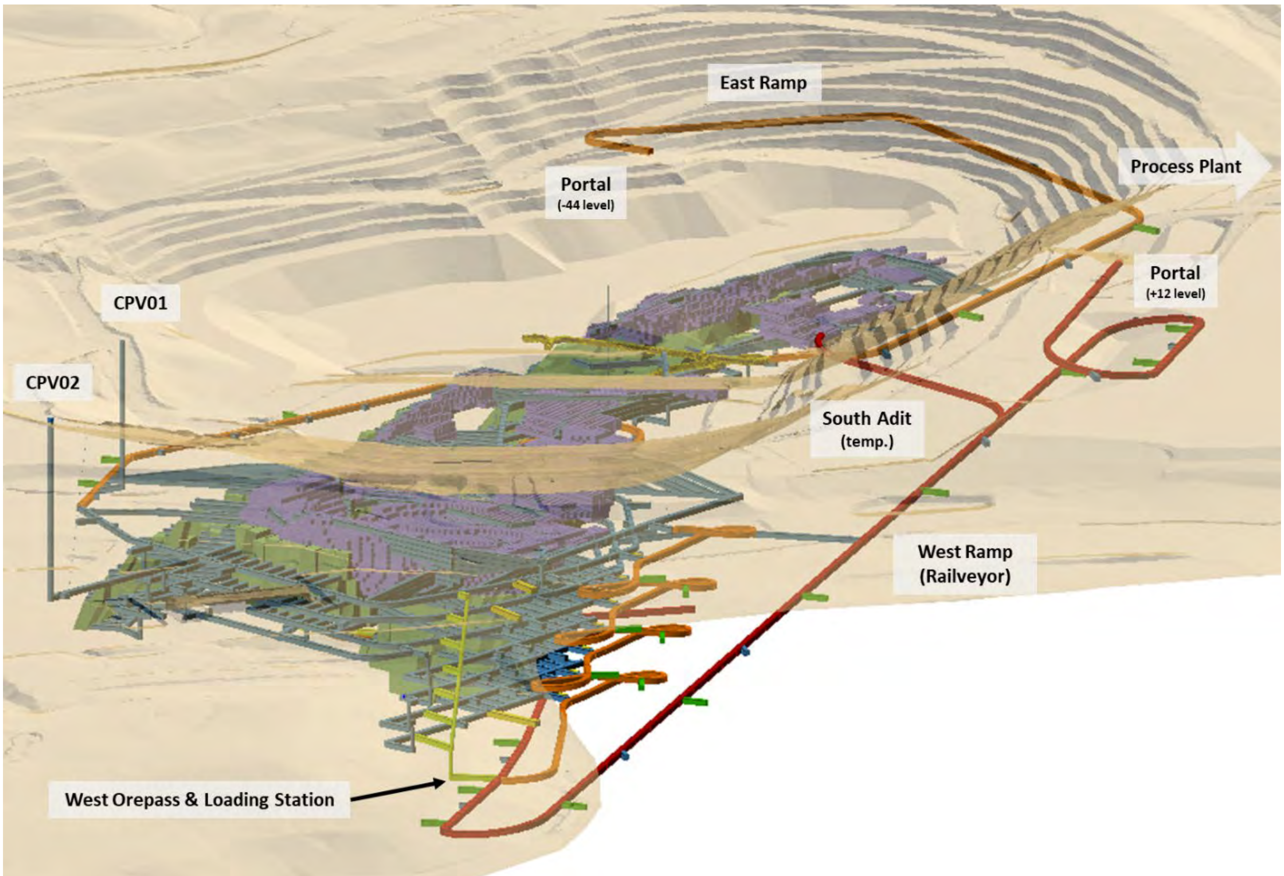
- Block 1 = DAF (upper) and longitudinal LHOS (lower).
- Block 2 = DAF (upper) and transverse LHOS (lower).
- Block 3 = DAF (upper) and transverse LHOS (lower).
- Block 5 = DAF.

Underground access will be via two declines.

1. The East Ramp will decline from a portal located at the -44 level of the existing open pit and will be used for personnel and equipment access.
2. The West Ramp will decline from a portal located on surface outside of the existing open pit and adjacent to the proposed new processing facilities. This ramp has been designed for the installation of an electric powered ore hauling system, the Railveyor, and will also act as an emergency egress route from the mine.

A third internal haulage ramp connects the West zone deep levels and the Railveyor West loading station at the 335 level with the main 185 level. The decline development plan enables the establishment of a primary ventilation circuit with no intermediate ventilation shafts required until the main ventilation infrastructure is developed.

Figure 1-4 Las Cruces Underground Project. View from South-West



1.7.1 Mine Development

A three-year pre-production development program, entailing up to 2,000m per quarter of advance, will be undertaken to both establish the mine infrastructure and provide access to the initial stoping levels. Ongoing development to reach and sustain 2.0Mtpa of mined ore production, will average approximately 7,500m per year during the first three years of production. The required development rate will decrease during years four and five to an average 6,000 m per year. It will then ramp up again once mining of the DAF sub-blocks reaches planned capacity.

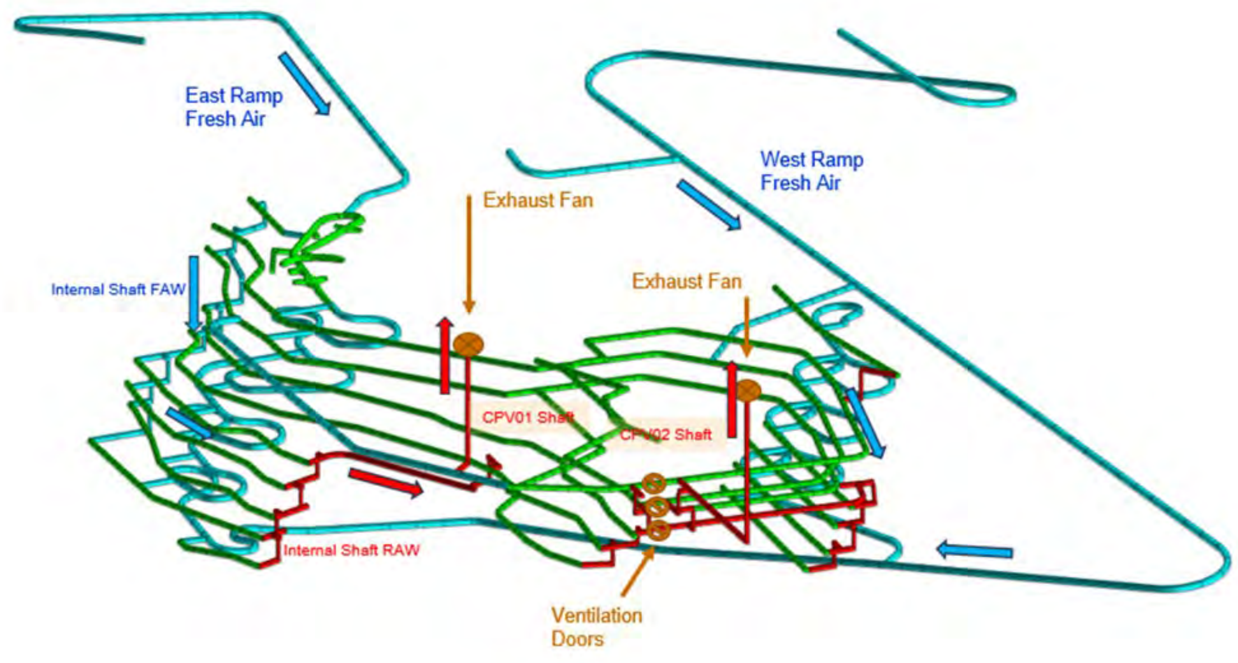
Two temporary access ways will be utilised to accelerate initial underground development by opening up additional available working faces. An exploration decline was developed from within the existing open pit in 2017 and 2018. This decline is being refurbished and will be utilised for early access to the East Ramp and will be integrated into the final East Ramp design. A second temporary portal driven at -110 open pit elevation will provide access to an intermediate point of the West Ramp.

1.7.2 Ventilation

Ventilation will be via a negative pressure circuit with fresh air entering through the declines and distributed to the working levels through dedicated ventilation raises. Return air will be exhausted via a series of raises connecting them to the two main exhaust raises. The primary fans will be located on surface.

The 210RL level crosscut connects the East and West areas and closes the first ventilation circuit at the end of Year 1. In combination with the completion of the CPV01 shaft, it will provide fresh airflow to progress the mine deepening to the East and the development of the Railveyor ramp until the completion of the CPV02 shaft about one year later.

Figure 1-5 Isometric view of the Main Ventilation Network from the NW



All mining areas not supplied by the primary ventilation circuit must be ventilated using auxiliary systems. The most effective method of providing the required airflow to these areas is typically with small-diameter, up to 1,400 mm, axial fans combined with limited-leakage, flexible ducting.

1.7.3 Mobile Equipment

During the pre-production phase approximately 31,000m of lateral development is required to meet the ramp-up production schedule. Development during this period will peak at approximately 2,000m per quarter, averaging approximately 1,900m per quarter.

Table 1-2 shows the supplied equipment for the pre-production development phase.

Table 1-2 UG Equipment Count during Pre-production Phase

DESCRIPTION	TOTAL
Two boom jumbos	5
LHD 12.5 t	7
Bolter	5
Cable Bolter	1
Shotcrete sprayer	5
Transmixer	6
Production driller 100 mm	1
Mobile raiseborer (Easer L)	1
LHD 17 t	5
Haulage Truck, 45t	4
Haulage Truck, 60t	**3**

Table 1-3 summarizes the total peak equipment requirements during the production phase of the UG operation.

Table 1-3 UG Equipment Count during Production Phase

DESCRIPTION	NUMBER
Two boom jumbos	3
LHD 12.5 t	4
Bolter	3
Cable Bolter	1
Shotcrete sprayer	3
Transmixer	4
Production driller 100 mm	3
Mobile raiseborer	1
LHD 17 t	9
Haulage Truck, 65t	**4**

The ancillary equipment listed Table 1-4 will be required to support all activities in the mine.

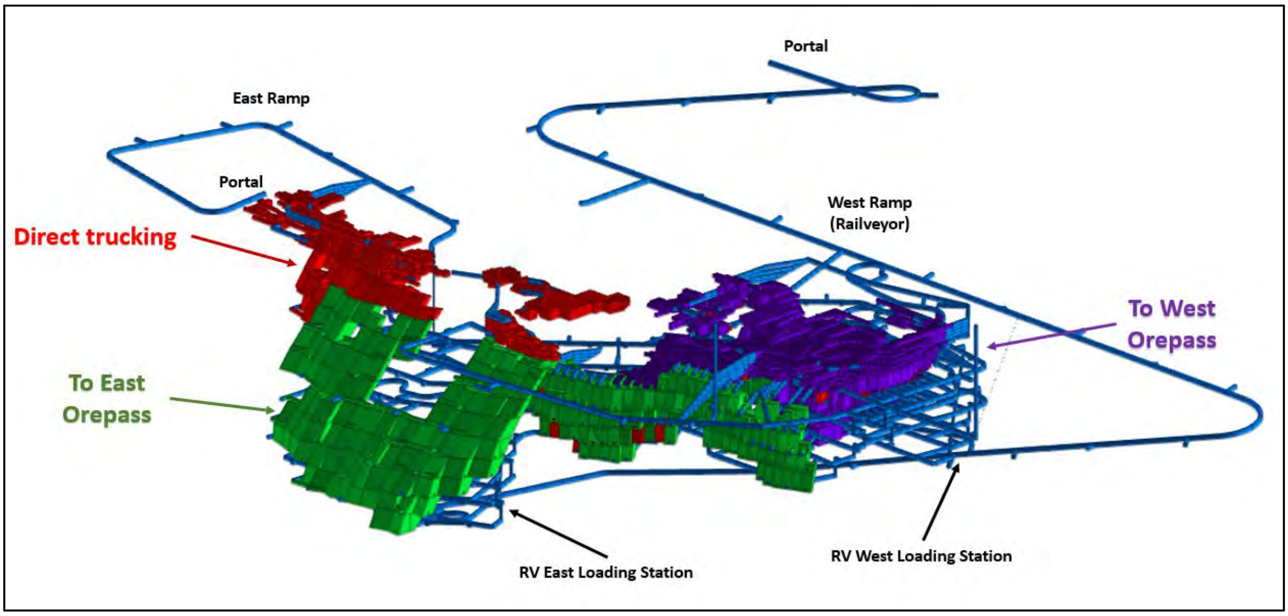
Table 1-4 Ancillary Equipment

DESCRIPTION	TOTAL
Emulsion Loader	5
Scissor Lift	5
Mobile breaker	2
Boom Truck	3
Personnel Carrier	6
UG Forklift	2
Grader	1
Service Truck	5
Pickup Truck	13
Fuel/Lubricant Truck	**3**

1.7.4 Materials Handling

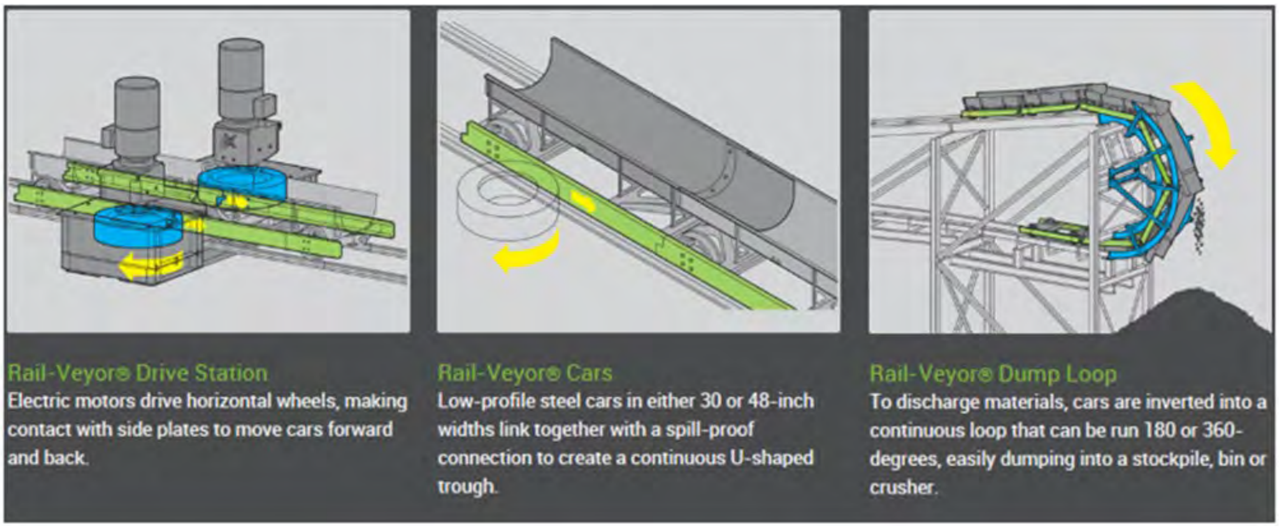
Ore will be brought to the surface via two methods. Ore from DAF mining areas will be hauled by truck via the East Ramp. In the LHOS areas, a fleet of Load-Haul-Dump (LHD) units will take the broken ore to ore passes at different levels which feed one of two rock breaker and ore loading stations of the Railveyor and conveyed to the surface processing facilities.

Figure 1-6 Ore Distribution by Destination. Isometric view – oriented SE



The Railveyor is an innovative material-handling system that combines the flexibility of truck hauling with the energy efficiency of conveyors and rail lines. The principle of the Railveyor system is to fill the cars from one of two loading stations, move them forward using drive stations, then discharge the material from a 180 degrees continuous loop. Figure 1-7 presents an overview of the proposed Railveyor system.

Figure 1-7 Railveyor Equipment Overview

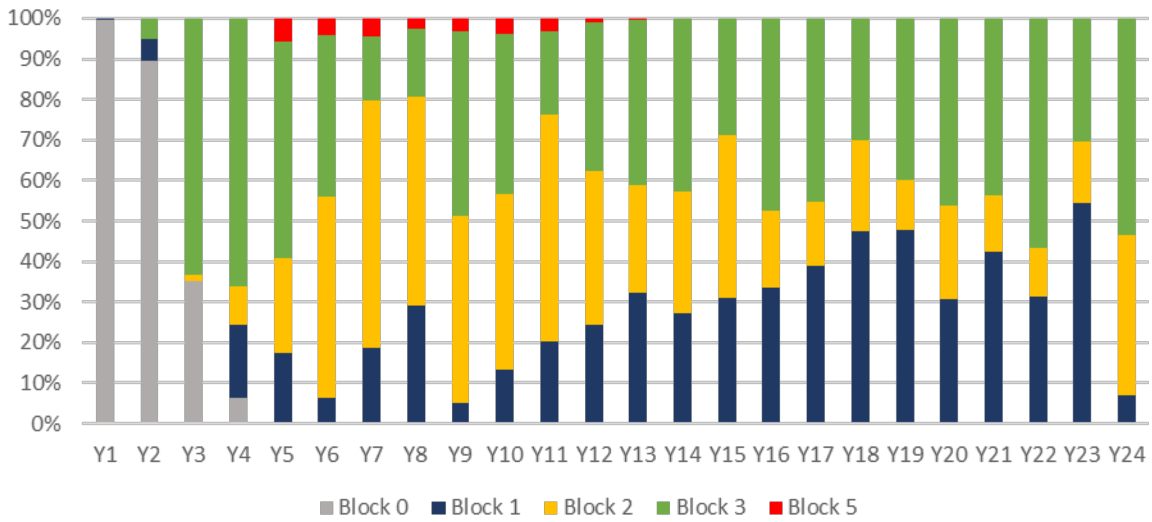


Other major underground infrastructure will include ventilation shafts, dewatering pump stations, a maintenance workshop, the paste backfill delivery system, electrical substations, a fuelling facility, explosives facility, refuge stations, mine communications, and other ancillary installations. The materials handling system will include three ore passes with ten transfer points, two rockbreaker silos and two loading stations

1.7.5 Underground Production Schedule

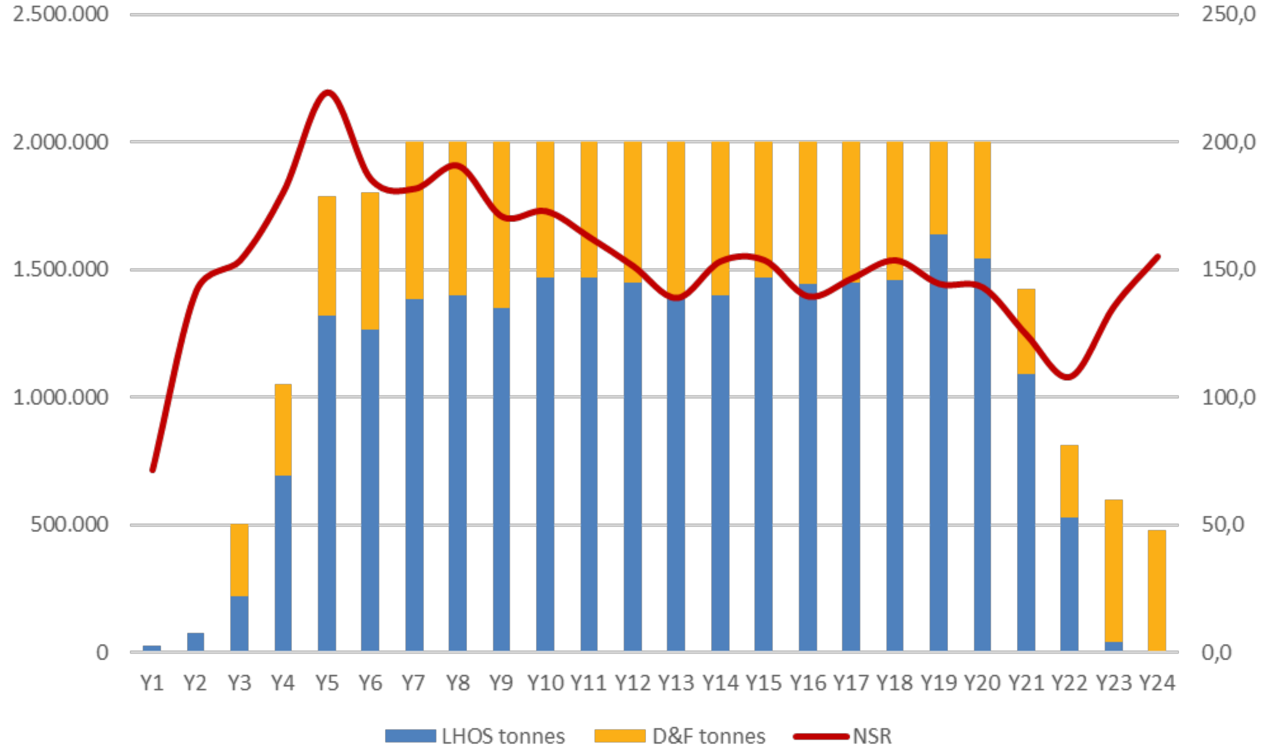
The nominal full 2.0Mtpa production rate is achieved in Year 7 with the bulk of the production sustained by both the PMS and the SMS zones. Figure 1-8 below shows the ore block contribution to LOM production by year. This excludes the scheduling of any Inferred Resources.

Figure 1-8 Contribution of Each Ore Block in the Life of Mine production (excl. Inferred)



The yearly production tonnage and grade is shown in Figure 1-9. The development activity is inclusive of both DAF production and ore development in stoping zones.

Figure 1-9 LOM Production Schedule by Mining Method (excl. Inferred)



The annual tonnages and average annual grades throughout the LOM are shown in Table 1-5 below.

Table 1-5 Mine Production Tonnage and Grades over LOM

YEAR	ORE (KT)	NSR $/t	% CU	% ZN	% PB	PPM AG
1	24	71	0.60	0.70	0.25	10.42
2	75	142	1.01	1.83	0.76	31.45
3	503	154	1.11	2.03	0.92	23.46
4	1,050	181	1.28	2.47	1.10	28.93
5	1,785	220	1.57	2.93	1.37	36.39
6	1,800	185	1.32	2.43	1.20	31.64
7	2,000	182	1.27	2.47	1.13	32.79
8	2,000	191	1.35	2.65	1.17	26.45
9	2,000	171	1.10	2.57	1.11	28.21
10	2,000	173	1.02	2.77	1.32	31.42
11	2,000	163	1.03	2.41	1.12	29.55
12	2,000	151	1.08	2.07	0.88	21.47
13	2,000	139	1.01	1.89	0.74	19.07
14	2,000	153	1.06	2.23	0.88	21.64
15	2,000	154	1.17	1.85	0.89	23.17
16	2,000	140	1.07	1.66	0.75	21.35
17	2,000	147	1.04	1.98	0.86	22.34
18	2,000	154	1.09	2.05	0.95	24.77
19	2,000	144	1.04	1.86	0.82	24.98
20	2,000	143	1.01	1.87	0.87	26.71
21	1,425	124	1.05	1.23	0.59	18.02
22	814	108	0.92	1.03	0.58	14.47
23	597	136	1.03	1.58	0.75	16.55
24	477	155	1.35	1,40	0.61	18.58
Total	**36,546**	**160**	**1.13**	**2.14**	**0.97**	**25.47**

1.8 Metallurgy & Processing

Lycopodium Minerals Pty Ltd (Lycopodium) was engaged by FQM to undertake an Engineering Cost Study (ECS) for the proposed PMR Project. Primary polymetallic sulphides processing treats copper, zinc and lead (bearing silver) primary sulphides to produce the four metals as commercial products, specifically copper cathodes, zinc ingots, lead metal briquettes and silver metal cement. The process design calls for ore to be floated to produce a single bulk concentrate that feeds a hydrometallurgical plant where the four mentioned metals are leached and recovered in sequence. The existing process plant will require modifications, as well as the installation of new comminution flotation circuits and dedicated zinc, lead, and silver refining facilities to achieve this design.

1.8.1 Flowsheet Development

The design basis for the concentrator plant has been developed from the bench and pilot scale testwork described above with appropriate scale-up factors applied. For the hydrometrical plant, the early bench scale work has been used and supplemented by results from the pilot testwork campaigns conducted during 2016 and 2017. Standard chemical relationships have been used in the development of the *Metsim* mass balance and these have been ratified where applicable by the pilot testwork results.

In addition, independent testwork programmes have been conducted by vendors, using representative samples prepared and supplied by CLC, to ratify the sizing and selection of various equipment (e.g. thickeners and filters). The results of these testwork programmes have also been applied to the design criteria developed for the Project.

On the basis of the testwork outcomes, a product P80 of 35µm has been adopted as the target grind size from the milling circuit for subsequent feed to the bulk flotation circuit. Testwork has also demonstrated that a mass pull to concentrate of ~35%w/w may be anticipated and that further grinding of this concentrate to a target P80 of 10µm will be required for optimum downstream leaching performance and metal recovery.

Subsequent to this process design, a capital saving opportunity ($10.2M) was identified by removing the Lead Smelting and Casting circuit and directly selling the lead cement product. Consequently the facilities in this circuit, whilst allowed for in the process design, do not form part of the PMR Project scope and are not planned to be constructed. Payabilities received for lead production and sales have been adjusted down to reflect this change.

Figure 1-10 Overall Block Flow Diagram

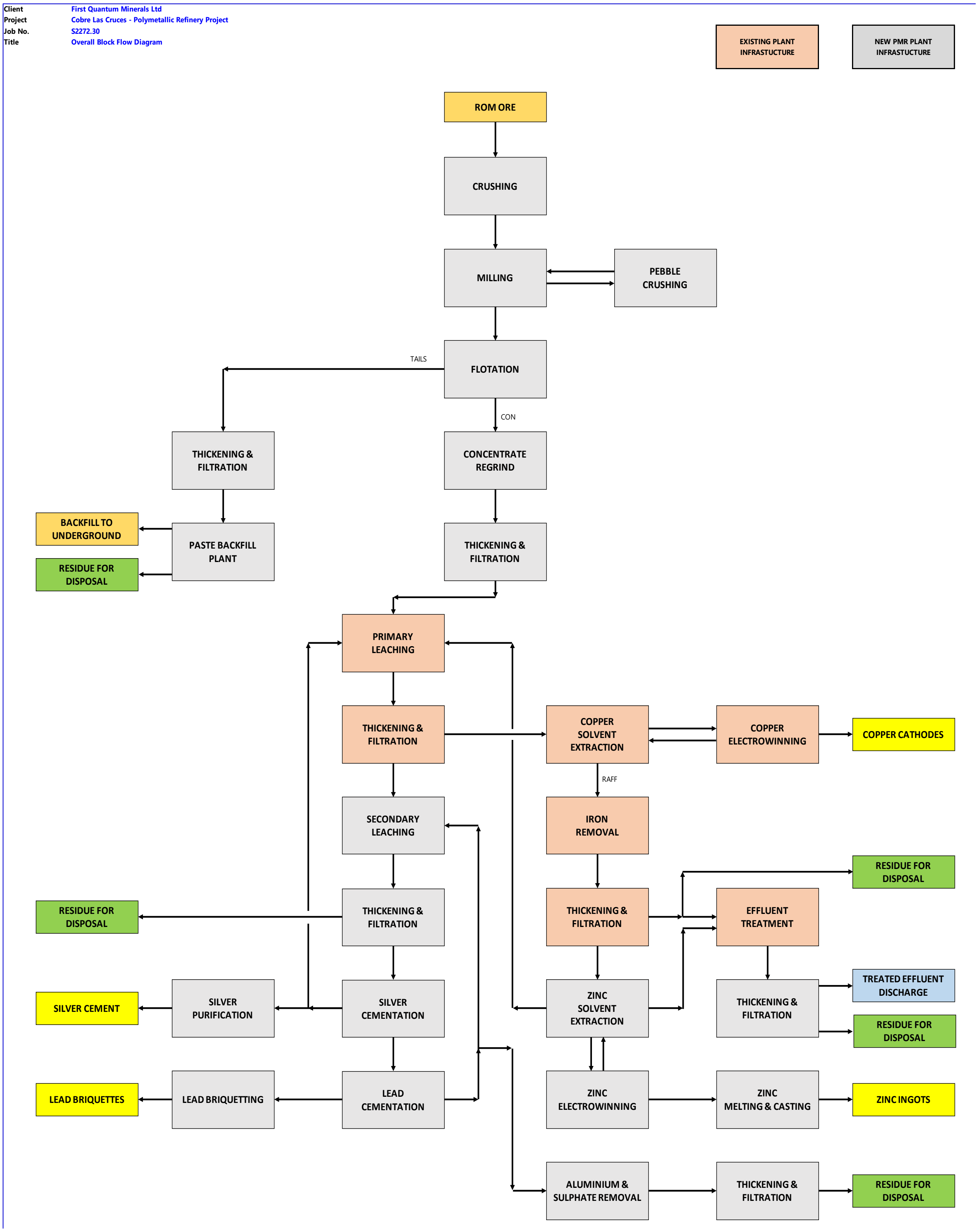
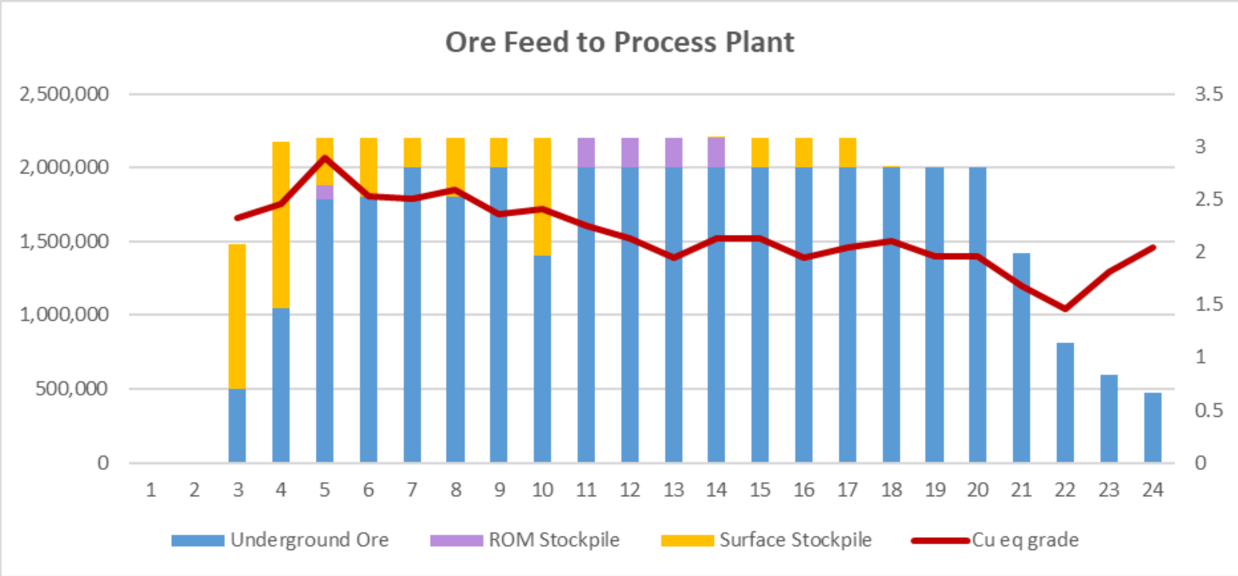


Figure 1-11 Existing and Proposed New Process Plant Layout

1.8.2 Plant Feed Production Schedule

The combined feed to the processing plant will peak at 2.2Mtpa of polymetallic ore through a combination of UG ore feed and stockpile reclaim feed and includes scheduled Inferred resources as shown in Figure 1-12 and Table 1-6.

Figure 1-12 LOM Production Schedule



A total of 36.6Mt of ore will be mined from underground over the LOM. Of this, 35.6 Mt will be fed directly to the Process Plant and another 899 kt will be stockpiled and rehandled. The 5,922 kT from stockpile shown in Table 1-6 consist of the 5.02 Mt from the existing surface stockpile and the 899 kT reclaimed from the ROM stockpile generated during UG mining. Figure 1-12 shows the ore feed by source. As can be seen the existing surface stockpile is used in the early years to supplement underground ore as the mining production ramps up to steady state.

Table 1-6 LOM Production Schedule by ore source

YEAR	Direct feed from UG (KT)	From Stockpile (KT)	% CU	% ZN	% PB	PPM AG	% CU EQ
1	-	-					-
2	-	-					-
3	503	982	1.14	2.21	1.41	27.02	2.32
4	1,050	1,123	1.21	2.39	1.39	28.88	2.46
5	1,785	415	1.56	2.57	1.31	36.04	2.89
6	1,800	400	1.29	2.41	1.28	31.13	2.53
7	2,000	200	1.26	2.45	1.18	32.43	2.50
8	1,800	400	1.31	2.58	1.26	26.88	2.59
9	2,000	200	1.11	2.55	1.16	28.26	2.35
10	1,400	800	1.08	2.59	1.44	30.58	2.41
11	2,000	200	1.03	2.44	1.13	29.72	2.24
12	2,000	200	1.11	2.12	0.91	21.92	2.12
13	2,000	200	1.01	1.97	0.79	20.20	1.94
14	2,000	200	1.06	2.28	0.92	22.53	2.13
15	2,000	200	1.17	1.89	0.96	23.69	2.13
16	2,000	200	1.08	1.72	0.83	22.03	1.95
17	2,000	200	1.05	2.01	0.93	22.93	2.04
18	2,000	2	1.09	2.05	0.95	24.78	2.10
19	1,999	-	1.04	1.86	0.82	24.98	1.96
20	2,000	-	1.01	1.87	0.87	26.71	1.95
21	1,424	-	1.05	1.23	0.59	18.02	1.68
22	814	-	0.92	1.03	0.58	14.47	1.47
23	597	-	1.03	1.58	0.75	16.55	1.81
24	477	-	1.35	1.40	0.61	18.58	2.04
Total	**35,647**	**5,922**	**1.14**	**2.15**	**1.05**	**25.94**	**2.22**

1.9 Existing Infrastructure

CLC is an existing operation that has been open pit mining and processing ore for more than fifteen years. The associated infrastructure as required by these operations is still in place and includes:

- An existing plant which will be fully re-used including copper and zinc leaching, copper recovery by SX and EW, in addition to effluent treatment that includes iron removal sections.
- Access-sealed roads.
- Dumps and stockpiles.
- 220 Kv Power line and substation with capacity
- An incoming water pond from the San Jerónimo sewage plant
- Water treatment plants for plant process water and mine water.
- Facilities for underground water extraction, treatment and injection.

- An in-pit tailings facility
- Surface auxiliary infrastructure including:
  - an electrical sub-station
  - emergency generators
  - maintenance workshops
  - spare parts warehouse
  - truck weigh scale
  - an experimental pilot plant
  - hazardous waste warehouse
  - oxygen plant
  - LNG plants
  - mining workshop
  - site security and access control point
  - a visitors centre and CLC Mining Museum
  - cafeteria
  - laboratory
  - medical centre
  - changing rooms
  - administrative buildings
  - control and communication systems
  - internal roads

1.10 Tailings

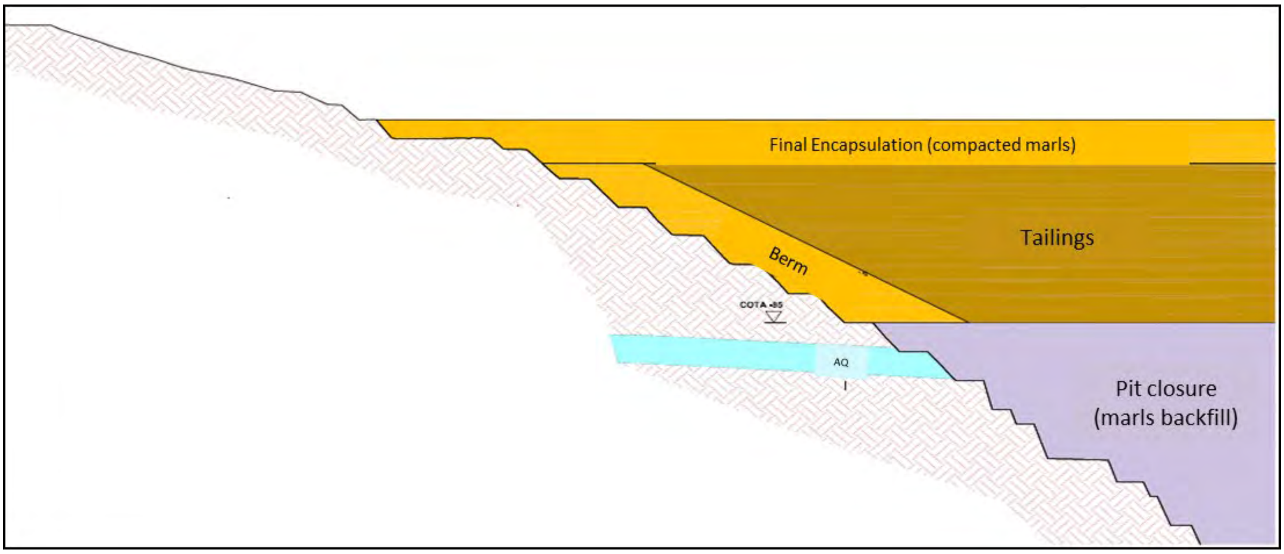
The PMR process plant will generate two tailings streams:

1. 1.4Mtpa flotation tailings, with 63% used as UG mine paste backfill and 37% disposed of in the in-pit tailings facility (IPTF).
2. 0.7Mtpa of leaching tails disposed of in the IPTF.

The current IPTF strategy will be utilized to dispose, store and encapsulate plant tailings. The IPTF has key advantages over the previous surface dry stacking facility, including;

- It is compatible with the pit closure plan and the new facility will be developed over the open pit backfill at the -85 level.
- Reduces the disturbed surface footprint and issues associated with access to land, environmental and permitting versus the surface option.
- IPTF is the lowest CAPEX tailings disposal alternative for this Project.

Figure 1-13 IPTF Conceptual Design (Subterra, 2018)



The total quantity of tailings for in-pit placement in the LOM plan is 33.3Mt dry, which is 94% of the initially approved capacity. The IPTF capacity can be expanded in the future by following a formal permitting procedure with the regional administration. The maximum theoretical capacity of the in-pit storage facility is more than 100Mt dry approximately 300% of total tailings over the planned LOM.

A first cell has been prepared and operated from 2016 to 2023 to encapsulate waste rock and the tailings from the reprocessing project. This cell acts as a real scale trial of the future facility. Continuous geotechnical and hydrogeological monitoring during these years proves that the disposed material is properly encapsulated and validates the in-pit disposal selected for the PMR Project.

1.11 Mineral Resources and Reserves

1.11.1 Mineral Resource Estimate

A Mineral Resource estimate of the CLC PPS mineralisation was updated in September 2021 and is detailed in the 2022 CLC Technical Report. As of the date of this report, there have been limited additions to underlying data and therefore no changes to the previous block model estimates. However, the Mineral Reserve studies detailed herein, use an updated NSR and higher metal prices. The copper equivalent (Cu_{eq}) calculation was updated, and Mineral Resources are now reported at a lower Cu_{eq} cutoff of 0.8%.

The Mineral Resource estimate classification is guided by geological and grade continuity, QAQC, density data, drillhole grid spacing and confidence in the panel grade estimate. Mineral Resources (Table 1-7) are reported in accordance with the guidelines of the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines, CIM November 2019). Minor tonnages of secondary sulphide and gossan gold mineralisation which remain from open pit mining and stockpiles are tabled for completeness (Table 1-8).

There are no known environmental, permitting, legal, title, taxation, socio economic, marketing, or political factors that could affect this.

Table 1-7 CLC Mineral Resource statement as at 30th September 2023.

Mineral Resource statement reported at respective Material type cutoff grades as at 30th September 2023												
Classification	**Tonnes (Mt)**	**Grade**						**Metal**				
		Cu_{eq} (%)	**Cu (%)**	**Au (g/t)**	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Cu (kt)**	**Au (koz)**	**Zn (kt)**	**Pb (kt)**	**Ag (Moz)**
Polymetallic Primary Sulphides (0.8% Cu_{eq} cutoff grade*)												
Measured	20.0	2.62	1.21	-	2.92	1.29	31.7	241		583	257	20.3
Indicated	21.4	1.97	1.13	-	1.65	0.79	23.4	242		353	169	16.1
Measured + Indicated	**41.4**	**2.29**	**1.17**	**-**	**2.26**	**1.03**	**27.4**	**484**		**936**	**427**	**36.5**
Inferred	9.4	1.66	1.08	-	0.94	0.61	26.1	1		1	1	7.9
Secondary Sulphides (1.0% Cu cutoff grade)												
Measured	0.9	-	6.23	-	-	-	-	0.54				
Indicated	0.1	-	2.51	-	-	-	-	0.02				
Measured + Indicated	**0.9**	**-**	**6.01**	**-**	**-**	**-**	**-**	**0.55**				
Inferred	-	-	-	-	-	-	-					
Gossan (1.0 g/t Au cutoff grade)**												
Measured	-	-	-	-	-	-	-					
Indicated	0.01	-	-	1.54	-	0.83	100.5		0.50		0.08	0.03
Measured + Indicated	**0.01**	**-**	**-**	**1.54**	**-**	**0.83**	**100.5**		**0.50**		**0.08**	**0.03**
Inferred	-	-	-	-	-	-	-					
*Resource estimates for the Polymetallic Primary Sulphides are reported on a cutoff grade of **0.8 % copper equivalent (Cu_{eq})**, based upon the following formula which accounts for metal price ($3.77/lb copper, $1.21/lb zinc, $0.94/lb lead and $22.37/lb silver), metallurgical recoveries and amounts payable by the smelter. Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability												
Cu_{eq} = [Cu% + (Zn% x 0.333) + (Pb% x 0.221) + (Ag g/t x 0.005)]												
**** Gossan is only reported within the optimized pitshell of the previous open pit project**												

Table 1-8 CLC Mineral Resource statement of remaining stockpiles as at 30th September 2023.

Classification	**Stockpile**	**Tonnes (Mt)**	**Grade**					**Metal**				
			Cu (%)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)	Cu (kt)	Zn (kt)	Pb (kt)	Au (Moz)	Ag (Moz)
Indicated	Polymetallic Primary Sulphide (PPS)	5.0	1.19	2.21	1.63	-	-	60	111	82		
	Gossan	2.7	-	-	3.30	2.58	82.3			88	0.2	7.1
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability												

Note: The Gossan stockpile is not in Mineral Reserves, nor is Gossan included in the u/g mine plan. Cu_{eq} grade is not reported in the Mineral Reserve statement.

In addition to the tabled Mineral Resource, there is potential for exploration target mineralisation of between 10 and 20 million tonnes of PPS with Cu grades between 0.9 and 1.6%. The targeted mineralisation is located adjacent to existing Mineral Resources and is based on known geological and grade continuity, geophysical data anomalies, diamond core sample assay data and its proximity to existing Mineral Resources. The disclosed tonnes and grades of the target exploration mineralisation potential is conceptual in nature and there is insufficient exploration information to define a Mineral Resource. It is uncertain if further exploration will result in the disclosed exploration target mineralisation potential being defined as a Mineral Resource.

1.11.2 Mineral Reserves Statement

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate followed a conventional approach, commencing with stope optimisation techniques incorporating economic parameters and other Modifying Factors including practical mine design and geotechnical considerations, in addition to mining dilution and recovery adjustments.

The optimisation process incorporated a break-even style calculation, employed to identify the Mineral Resource eligible for conversion to Mineral Reserves. During this process, a Net Smelter Return (NSR) was determined for each block in the Mineral Resource model, utilising consensus metal pricing processing recoveries an operating costs including mining, processing and general and administrative (G&A) costs. The NSR calculation parameters as listed in Table 1-10 defined the Mineral Reserve cutoff grades for each applicable stoping method and ore type.

The mining shapes developed during the optimisation process were then used to develop practical stope designs followed by suitable development designs to provide access to the stopes.

Table 1-9 Element Selling Unit Prices, payable and recoveries

ELEMENT	SELLING PRICE	UNIT	PAYABLE (%)	PMS PROCESS RECOVERY (%)	STW PROCESS RECOVERY (%)
Cu	3.77	$/lb	100	85.12	89
Zn	1.21	$/lb	100	88.41	89
Pb	0.94	$/lb	90	79.38	83
Ag	21.37	$/oz	90	51.88	63

Table 1-10 NSR Calculation – Cost Inputs

Cost Item	Mining Method	
Mining	**LHOS**	**DAF**
Mining Cost USD/t	21.96	28.16
Process USD/t	50.60	
Process (HC4) USD/t	40.44	
G & A USD/t	7.26	3.93

Underground Reserves

The Mineral Reserve estimate for the Las Cruces underground PMR Project has been estimated and classified in accordance with NI 43-101 requirements. The numbers in Table 1-11 and Table 1-12 are presented on a diluted basis for both mass and grades. The effective date of the Mineral Reserve estimate in Table 1-12 is 30th September 2023.

Table 1-11 Cobre las Cruces PMR Mining Inventory by Zone and Classification as at 30th September 2023

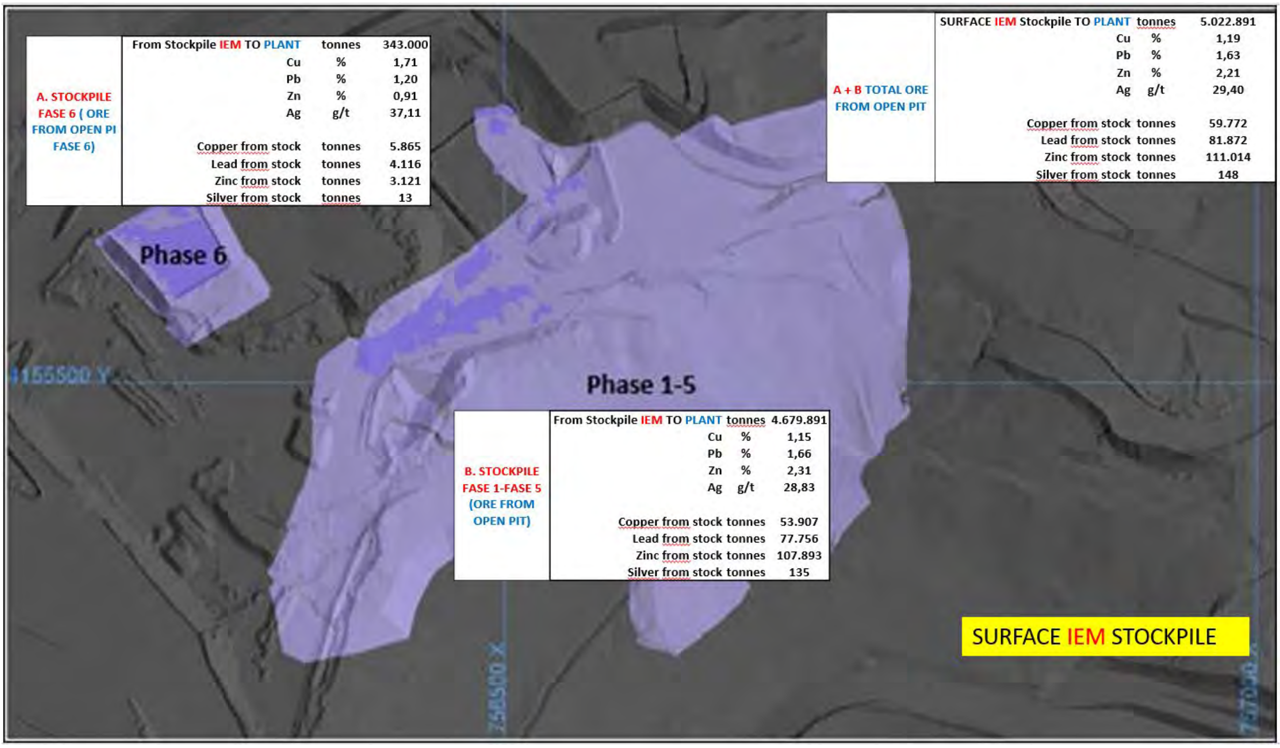
Zone	Classification	Mass (kt)	Cu (%)	Pb (%)	Zn (%)	Ag (ppm)
Block 0	Proven	34.8	0.95	0.26	0.62	9.05
	Probable	44.8	0.67	0.24	0.48	22.30
	Total	**79.6**	**0.79**	**0.25**	**0.54**	**16.51**
Block 1	Proven	2,572.3	1.12	0.70	1.59	22.61
	Probable	7,551.6	0.84	1.09	2.11	31.50
	Total	**10,123.9**	**0.91**	**0.99**	**1.98**	**29.24**
Block 2	Proven	8,888.7	1.02	1.32	3.21	29.92
	Probable	2,772.3	1.44	1.15	2.87	31.98
	Total	**11,661.0**	**1.12**	**1.28**	**3.13**	**30.41**
Block 3	Proven	4,383.6	1.72	1.20	2.56	31.76
	Probable	9,806.5	1.11	0.49	1.00	13.09
	Total	**14,190.1**	**1.30**	**0.71**	**1.48**	**18.86**
Block 5	Proven	89.7	0.70	1.59	1.74	30.32
	Probable	401.7	1.04	0.50	1.08	21.12
	Total	**491.4**	**0.98**	**0.70**	**1.20**	**22.80**
Total	Proven	15,969.1	1.23	1.19	2.76	29.20
	Probable	20,577.0	1.05	0.80	1.66	22.57
	Total	**36,546.1**	**1.13**	**0.97**	**2.14**	**25.47**

Numbers may not add up due to rounding

Surface Primary Sulphide Stockpile

Between 2009 and 2021, approximately 5Mt of primary sulphide was mined from the open pit operations. This material has been managed and stockpiled and now constitutes another source of ore for the PMR Project. Figure 1-14 below shows the location, tonnage, and grades of the primary sulphide ore currently stockpiled at the CLC Mine Rock Storage Facility (MRSF).

Figure 1-14 Existing Primary Sulphide Stockpiles as at 30 2023



Mineral Reserves Statement

The Mineral Reserves estimate for the CLC PMR Project is listed in Table 1-12.

Table 1-12 Mineral Reserve Statement at $3.77/lb Cu, $1.21/lb Zn, $0.94/lb Pb and $21.37/oz Ag

Cobre las Cruces Mineral Reserve Estimate as at 30th September 2023									
		Grade				Contained Metal			
	Mt	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Cu (kt)	Pb (kt)	Zn (kt)	Ag (koz)
Underground (In-situ)									
Proven Reserve	16.0	1.23	1.19	2.76	29.20	196	190	441	150
Probable Reserve	20.6	1.05	0.80	1.66	22.57	216	165	342	149
Total	**36.6**	**1.13**	**0.97**	**2.14**	**25.47**	**413**	**355**	**782**	**299**
Stockpile									
Proven Reserve									
Probable Reserve	5.0	1.19	1.63	2.21	29.40	60	82	111	47
Total	**5.0**	**1.19**	**1.63**	**2.21**	**29.40**	**60**	**82**	**111**	**47**
Combined									
Proven Reserve	16.0	1.23	1.19	2.76	29.20	196	190	441	150
Probable Reserve	25.6	1.08	0.96	1.77	23.85	276	246	453	197
Total	**41.6**	**1.14**	**1.05**	**2.15**	**25.94**	**472**	**437**	**893**	**347**

Numbers may not add up due to rounding

1.12 Capital and Operating Costs Estimates

1.12.1 Capital Costs

The PMR Project will make use of the existing facilities at CLC but will need additional investment to develop and operate an underground mine and polymetallic processing plant. The total initial capital cost estimate for this project is $846M, consisting of two major components, one for the construction of the ore processing plant and the other for costs associated with the development of an underground mine. This includes contingency of 14% or $104M. An additional $104M for sustaining capital and mine closure costs will be incurred after the initial project period.

Table 1-13 PMR Total Capital (including contingency)

COST ITEM	$ Million
Mine	271
Plant	499
Owners and Indirect costs	28
Existing Cu SX/EW plant refurbishment	23
Water facility expansion	21
Other	4
Initial Project Capital	**846**
Sustaining Capital	6
Mine Closure Provision	98
Total Capital	**950**

Mine

Mining costs (Table 1-14) are related to the construction of a new underground mine located underneath the current open pit mine. With a capacity of 2.0Mtpa, total mine capital costs are $271M primarily for the development of underground access to the ore body and construction of mining infrastructure and facilities.

Table 1-14 Mine Capital Cost

COST ITEM	$ Million
Mine Development	99
Railveyor and Ore Handling System	43
Mobile Equipment Fleet	42
Capitalised operating costs	31
UG Main Dewatering Sumps	11
UG Electrical Distribution	8
UG Ventilation	4
Other	8
Contingency	25
Total Mine Capital	**271**

Plant

Capital costs cover the construction of a new process plant with an annual processing capacity of up to 2.2Mtpa of ore and producing a steady state LOM average of ~41,000 tonnes of copper equivalent per annum over the life of mine in the form of four final products; copper, zinc, lead and silver (Table 1-15).

Table 1-15 Plant Capital Cost

COST ITEM	$ Million
Zinc Plants	127
Indirect costs	67
Electrical	48
Lead and Silver Plants	34
Crushing & Grinding	40
Flotation	27
Leaching	24
Iron Removal	21
Paste Plant	15
Water & Reagents	11
General	6
Aluminium Removal	4
Effluent Treatment	2
Contingency	73
Total Plant Capital	**499**

1.13 Operating Cost Estimates

The LOM of the Project is 24 years including initial development. The process plant at 46% is the largest contributor to operating costs followed by the mining operations at 24%, and then maintenance activities at 18% (Table 1-16).

Table 1-16 Steady State Operational Costs (excludes 1 years production ramp up and ramp down)

COST ITEM	LOM Average Annual Cost $ Million	LOM Average Annual Unit Cost $/ore t processed
Plant	90	41.61
Mine	45	21.27
Maintenance	31	14.43
G&A	15	6.93
Water	10	4.62
Royalty	3	1.39
Total	**194**	**90.25**

1.14 Economic Evaluation

The economic analysis in the form of a cash flow model is intended to support the Mineral Reserve estimate, and in order to demonstrate a positive cashflow for mining and processing. Key assumptions for the base case economic analysis are as follows:

- The plant feed tonnes and grade account for mining dilution and mining recovery factors.
- The initial development capital is expended in Years 1 to 6.
- The assumed infrastructure capital spending intensity is 16% in Year 1, 60% in Year 2 and 12% in Year 3.
- Costs associated with the underground decline (i.e., ramp) development, pit bottom development and pre-commercial ore extraction in Years 1 to 2 were assumed to be capitalised.
- Initial mine ore production starts in Year 1 at 0.02Mt, ramping up to 1.8 Mtpa for Years 5 to 6, then to 2.0 Mt from Year 7 onwards.
- Initial plant production starts in Year 3 at 1.5Mtpa ramping up to 2.2Mtpa in Year 4 onwards.
- The first year of processing features down-rated processing recoveries.
- Existing stockpiles of 5,023Mt are available to feed the plant and make up any shortfall between planned direct mine and plant feed.
- The annual operating costs (i.e., processing and G&A unit costs) were not profiled for each year.
- The metal costs were not profiled against varying concentrate grade, payability or other factors.
- Cashflows exclude an export levy, income tax, carbon tax, interest and depreciation.

The undiscounted pre-tax cashflow is $1,801M, with a pre-tax NPV reflecting an 8% discount rate equal to $519M. The pre-tax internal rate of return is 17.4%. A pre-tax NPV reflecting a 10% discount rate equal to $356M demonstrates the economic robustness of the Project. The Project is cashflow positive from Year 4 and payback on the initial development capital is in Year 8.

The valuation excludes various subsidy and financing applications in progress, some of which are in the final stages of approval.

The summary cashflow model for the economic analysis is listed in Table 1-17.

Table 1-17 CLC LOM Cashflow Summary

PHYSICALS	UNITS	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17	Yr18	Yr19	Yr20	Yr21	Yr22	Yr23	Yr24	Yr25
MINING (AFTER MINING DILUTION & RECOVERY)																											
Ore mined direct to plant	Mt	36,447.0	-	-	503.2	1,050.0	1,785.3	1,799.9	1,999.8	1,999.9	1,999.8	2,000.0	2,000.0	1,999.9	1,999.7	1,999.9	1,999.9	1,999.9	2,000.0	1,999.9	1,999.3	2,000.0	1,423.6	813.6	596.6	476.9	-
Ore mined to stockpile	Mt	99.0	23.8	75.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore reclaimed from stockpile	Mt	5,121.9	-	-	981.8	1,122.5	414.7	400.1	200.2	200.0	200.2	200.0	200.0	200.1	200.3	200.1	200.1	200.1	200.0	1.6	-	-	-	-	-	-	-
Waste mined	Mt	3,741.7	729.8	646.0	686.3	383.2	315.7	98.7	72.9	61.9	89.5	78.5	90.4	73.0	77.7	61.9	37.2	18.2	54.0	36.0	42.1	23.9	44.0	12.9	7.8	0.3	-
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)																											
Total direct feed	Mt	41,569.0	-	-	1,485.0	2,172.5	2,200.0	2,200.0	2,200.0	2,199.9	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,001.6	1,999.3	2,000.0	1,423.6	813.6	596.6	476.9	-
	% Cu	1.14%	-	-	1.14%	1.21%	1.56%	1.29%	1.26%	1.31%	1.11%	1.08%	1.03%	1.11%	1.01%	1.06%	1.17%	1.08%	1.05%	1.09%	1.04%	1.01%	1.05%	1.06%	1.03%	1.35%	0.00%
	% Zn	2.15%	-	-	2.21%	2.39%	2.57%	2.41%	2.45%	2.58%	2.55%	2.59%	2.44%	2.12%	1.97%	2.28%	1.89%	1.72%	2.01%	2.05%	1.86%	1.87%	1.23%	1.30%	1.58%	1.40%	0.00%
	% Pb	1.05%	-	-	1.41%	1.39%	1.31%	1.28%	1.18%	1.26%	1.16%	1.44%	1.13%	0.91%	0.79%	0.92%	0.96%	0.83%	0.93%	0.95%	0.82%	0.87%	0.59%	0.64%	0.75%	0.61%	0.00%
	g/t Ag	0.8	-	-	27.0	28.9	36.0	31.1	32.4	26.9	28.3	30.6	29.7	21.9	20.2	22.5	23.7	22.0	22.9	24.8	25.0	26.7	18.0	16.2	16.5	18.6	-
Cu insitu	kt	474.7	-	-	16.91	26.34	34.34	28.42	27.72	28.87	24.36	23.67	22.70	24.34	22.21	23.27	25.76	23.77	23.15	21.74	20.75	20.13	14.98	8.66	6.17	6.42	-
Zn insitu	kt	895.4	-	-	32.83	51.85	56.65	53.03	53.98	56.85	56.09	56.89	53.75	46.66	43.34	50.15	41.58	37.74	44.28	40.99	37.20	37.36	17.52	10.55	9.40	6.70	-
Pb insitu	kt	435.4	-	-	20.93	30.20	28.90	28.18	25.92	27.69	25.47	31.60	24.97	20.01	17.41	20.29	21.17	18.34	20.54	18.96	16.46	17.33	8.41	5.22	4.48	2.92	-
Ag insitu	koz	34,714	-	-	1,290	2,017	2,549	2,202	2,294	1,901	1,999	2,163	2,102	1,550	1,429	1,593	1,676	1,558	1,622	1,594	1,606	1,717	825	423	317	285	-
AVERAGE RECOVERIES																											
Cu	%	85%	-	-	81%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	-
Zn	%	88%	-	-	84%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	-
Pb	%	79%	-	-	64%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	-
Ag	%	52%	-	-	48%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	-
METAL RECOVERED																											
Cu	kt	403.3	-	-	13.7	22.4	29.2	24.2	23.6	24.6	20.7	20.1	19.3	20.7	18.9	19.8	21.9	20.2	19.7	18.5	17.7	17.1	12.8	7.4	5.3	5.5	-
Zn	kt	790.1	-	-	27.6	45.8	50.1	46.9	47.7	50.3	49.6	50.3	47.5	41.2	38.3	44.3	36.8	33.4	39.1	36.2	32.9	33.0	15.5	9.3	8.3	5.9	-
Pb	kt	342.3	-	-	13.3	24.0	22.9	22.4	20.6	22.0	20.2	25.1	19.8	15.9	13.8	16.1	16.8	14.6	16.3	15.0	13.1	13.8	6.7	4.1	3.6	2.3	-
Ag	koz	17,955	-	-	618	1,046	1,322	1,142	1,190	986	1,037	1,122	1,090	804	741	826	869	808	841	827	833	891	428	219	165	148	-

CASHFLOWS	UNITS	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17	Yr18	Yr19	Yr20	Yr21	Yr22	Yr23	Yr24	Yr25
PAYABILITY																											
Cu	%	100%	0%	0%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	0%
Zn	%	100%	0%	0%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	0%
Pb	%	95%	0%	0%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	0%
Ag	%	95%	0%	0%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	0%
Payable Metal Recovered																											
Cu	kt	403.3	-	-	13.7	22.4	29.2	24.2	23.6	24.6	20.7	20.1	19.3	20.7	18.9	19.8	21.9	20.2	19.7	18.5	17.7	17.1	12.8	7.4	5.3	5.5	-
Zn	kt	790.1	-	-	27.6	45.8	50.1	46.9	47.7	50.3	49.6	50.3	47.5	41.2	38.3	44.3	36.8	33.4	39.1	36.2	32.9	33.0	15.5	9.3	8.3	5.9	-
Pb	kt	325.2	-	-	12.6	22.8	21.8	21.3	19.5	20.9	19.2	23.8	18.8	15.1	13.1	15.3	16.0	13.8	15.5	14.3	12.4	13.1	6.3	3.9	3.4	2.2	-
Ag	koz	17,057.3	-	-	587.2	994.1	1,256.2	1,085.0	1,130.2	936.9	985.1	1,065.9	1,035.9	764.0	704.0	785.1	825.7	767.9	799.3	785.6	791.3	846.3	406.4	208.4	156.4	140.4	-
GROSS REVENUE																											
Cu	$/lb	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80
Zn	$/lb	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20
Pb	$/lb	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95
Ag	$/oz	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75
Revenue after payability																											
Copper revenue	$M	$ 3,378.6	$ -	$ -	$ 114.6	$ 187.8	$ 244.9	$ 202.6	$ 197.6	$ 205.9	$ 173.7	$ 168.8	$ 161.9	$ 173.5	$ 158.4	$ 165.9	$ 183.7	$ 169.5	$ 165.1	$ 155.0	$ 148.0	$ 143.5	$ 106.8	$ 61.8	$ 44.0	$ 45.8	$ -
Zinc revenue	$M	$ 2,090.3	$ -	$ -	$ 73.0	$ 121.3	$ 132.5	$ 124.0	$ 126.3	$ 133.0	$ 131.2	$ 133.1	$ 125.7	$ 109.1	$ 101.4	$ 117.3	$ 97.2	$ 88.3	$ 103.6	$ 95.9	$ 87.0	$ 87.4	$ 41.0	$ 24.7	$ 22.0	$ 15.7	$ -
Lead revenue	$M	$ 681.1	$ -	$ -	$ 26.5	$ 47.7	$ 45.6	$ 44.5	$ 40.9	$ 43.7	$ 40.2	$ 49.9	$ 39.4	$ 31.6	$ 27.5	$ 32.1	$ 33.4	$ 29.0	$ 32.4	$ 29.9	$ 26.0	$ 27.4	$ 13.3	$ 8.3	$ 7.1	$ 4.6	$ -
Silver revenue	$M	$ 388.1	$ -	$ -	$ 13.4	$ 22.6	$ 28.6	$ 24.7	$ 25.7	$ 21.3	$ 22.4	$ 24.3	$ 23.6	$ 17.4	$ 16.0	$ 17.9	$ 18.8	$ 17.5	$ 18.2	$ 17.9	$ 18.0	$ 19.3	$ 9.2	$ 4.7	$ 3.6	$ 3.2	$ -
Total revenue	**$M**	**$ 6,538.1**	**$ -**	**$ -**	**$ 227.4**	**$ 379.4**	**$ 451.6**	**$ 395.8**	**$ 390.6**	**$ 403.9**	**$ 367.5**	**$ 376.0**	**$ 350.6**	**$ 331.6**	**$ 303.3**	**$ 333.1**	**$ 333.1**	**$ 304.2**	**$ 319.3**	**$ 298.7**	**$ 279.0**	**$ 277.5**	**$ 170.3**	**$ 99.4**	**$ 76.6**	**$ 69.2**	**$ -**
CAPITAL COSTS																											
Plant	$M	$ 499.4	$ 92.0	$ 379.0	$ 28.4	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Mine	$M	$ 270.9	$ 52.2	$ 70.0	$ 56.5	$ 61.7	$ 26.2	$ 3.7	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owners and Indirect costs	$M	$ 27.9	$ 7.9	$ 19.6	$ 0.4	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Existing plant refurbishment	$M	$ 22.8	$ -	$ 22.8	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Water facility expansion	$M	$ 20.6	$ 3.8	$ 16.5	$ 0.4	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Other	$M	$ 4.4	$ 0.3	$ 3.9	$ 0.1	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Initial Project CAPEX	**$M**	**$ 846.0**	**$ 156.2**	**$ 511.9**	**$ 85.8**	**$ 61.7**	**$ 26.2**	**$ 3.7**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**
Sustaining capex (fleet replacement)	$M	$ 5.9	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1.8	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ -	$ -	$ -	$ -	$ -	$ -	$ -
ARO Closure costs	$M	$ 98.1	$ 8.9	$ 10.5	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 32.3	$ 46.3
Total Project CAPEX	**$M**	**$ 950.0**	**$ 165.1**	**$ 522.4**	**$ 85.8**	**$ 61.7**	**$ 26.2**	**$ 3.7**	**$ -**	**$ 1.8**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ 32.3**	**$ 46.3**
OPERATING COSTS																											
Ore and waste mining	$M	$ 831.1	$ 0.0	$ 0.0	$ 25.3	$ 33.3	$ 42.7	$ 42.8	$ 43.7	$ 45.2	$ 46.5	$ 42.9	$ 44.5	$ 45.4	$ 44.2	$ 43.2	$ 43.0	$ 40.8	$ 44.6	$ 41.6	$ 40.9	$ 41.0	$ 31.3	$ 20.8	$ 15.2	$ 12.2	$ -
Stockpile reclaim and tailings	$M	$ 57.2	$ -	$ -	$ 3.3	$ 4.3	$ 3.2	$ 3.1	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.2	$ 2.2	$ 2.2	$ 1.6	$ 1.7	$ 1.2	$ 1.0	$ -
Processing (inc. maintenance)	$M	$ 2,391.1	$ 8.4	$ 8.4	$ 97.7	$ 124.4	$ 125.6	$ 124.7	$ 124.0	$ 124.9	$ 123.9	$ 126.0	$ 123.5	$ 121.2	$ 119.9	$ 121.5	$ 121.2	$ 119.8	$ 121.1	$ 114.5	$ 113.2	$ 113.5	$ 91.8	$ 54.1	$ 37.0	$ 31.1	$ -
Water management & treatment	$M	$ 243.3	$ 12.5	$ 12.0	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 6.6	$ -
General and adminstration	$M	$ 321.6	$ 6.6	$ 6.6	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 10.9	$ 7.3	$ -
Total operating costs	**$M**	**$ 3,844.3**	**$ 27.4**	**$ 27.0**	**$ 150.9**	**$ 186.7**	**$ 196.1**	**$ 195.3**	**$ 195.1**	**$ 197.6**	**$ 197.8**	**$ 196.4**	**$ 195.4**	**$ 194.0**	**$ 191.5**	**$ 192.2**	**$ 191.6**	**$ 188.0**	**$ 193.1**	**$ 182.9**	**$ 180.9**	**$ 181.4**	**$ 149.3**	**$ 101.2**	**$ 74.4**	**$ 58.1**	**$ -**
METAL COSTS	$M	$ -																									
Metal Premiums	$M	$ -189.9	$ -	$ -	$ -6.6	$ -10.9	$ -12.5	$ -11.3	$ -11.4	$ -11.9	$ -11.4	$ -11.4	$ -10.8	$ -9.9	$ -9.1	$ -10.3	$ -9.2	$ -8.4	$ -9.4	$ -8.7	$ -8.0	$ -8.0	$ -4.3	$ -2.6	$ -1.9	$ -1.9	$ -
TCRCs and Freight	$M	$ 71.4	$ -	$ -	$ 2.6	$ 4.5	$ 4.7	$ 4.4	$ 4.3	$ 4.5	$ 4.3	$ 4.8	$ 4.2	$ 3.6	$ 3.2	$ 3.7	$ 3.4	$ 3.1	$ 3.5	$ 3.2	$ 2.9	$ 2.9	$ 1.4	$ 0.9	$ 0.6	$ 0.7	$ -
Royalites	$M	$ 51.4	$ -	$ -	$ 1.7	$ 2.9	$ 3.7	$ 3.1	$ 3.0	$ 3.1	$ 2.6	$ 2.6	$ 2.5	$ 2.6	$ 2.4	$ 2.5	$ 2.8	$ 2.6	$ 2.5	$ 2.4	$ 2.2	$ 2.2	$ 1.6	$ 0.9	$ 0.7	$ 0.7	$ -
Total Metal Costs	**$M**	**$ -67.2**	**$ -**	**$ -**	**$ -2.2**	**$ -3.6**	**$ -4.1**	**$ -3.8**	**$ -4.1**	**$ -4.3**	**$ -4.4**	**$ -4.1**	**$ -4.2**	**$ -3.7**	**$ -3.5**	**$ -4.1**	**$ -3.0**	**$ -2.8**	**$ -3.4**	**$ -3.1**	**$ -2.9**	**$ -2.8**	**$ -1.2**	**$ -0.8**	**$ -0.6**	**$ -0.6**	**$ -**
MINERAL RESERVE & RESOURCES CASHFLOW (PRE-TAX)																											
Undiscounted	$M	$ 1,800.7	$ -175.5	$ -546.7	$ -39.8	$ 123.3	$ 227.7	$ 205.3	$ 199.9	$ 207.8	$ 176.7	$ 182.5	$ 160.9	$ 142.4	$ 117.1	$ 142.1	$ 144.1	$ 120.8	$ 128.0	$ 119.6	$ 102.5	$ 99.1	$ 29.9	$ -1.0	$ 1.7	$ -20.5	$ -47.2
NPV 8%	$M	$ 518.5																									
NPV 10%	$M	$ 356.1																									
IRR	%	17.4%																									
MINERAL RESERVE & RESOURCES CASHFLOW (POST-TAX)																											
Tax paid	$M	$ 415.1	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 8.4	$ 32.8	$ 41.6	$ 37.9	$ 33.4	$ 27.9	$ 31.5	$ 33.2	$ 28.7	$ 29.1	$ 27.8	$ 24.1	$ 22.7	$ 15.1	$ 9.0	$ 6.3	$ 4.3	$ 1.2
Undiscounted	$M	$ 1,385.7	$ -175.5	$ -546.7	$ -39.8	$ 123.3	$ 227.7	$ 205.3	$ 199.9	$ 199.4	$ 143.9	$ 140.9	$ 123.0	$ 108.9	$ 89.2	$ 110.5	$ 110.9	$ 92.0	$ 98.9	$ 91.7	$ 78.4	$ 76.5	$ 14.8	$ -10.0	$ -4.6	$ -24.9	$ -48.4
NPV 8%	$M	$ 392.0																									
NPV 10%	$M	$ 257.3																									
IRR	%	15.7%																									

1.14.1 Revenue Assumptions

The annual revenues in the cash flow model assume September 2023 broker consensus long term metal prices as per Table 1-18 below. Metal payabilities are based on historical actuals realised by FQM in global markets and have been provided by FQMs internal marketing team.

Table 1-18 Element Selling Unit Prices and Payabilities

ELEMENT	SELLING PRICE	UNIT	PAYABLE (%)
Cu	3.80	$/lb	100
Zn	1.20	$/lb	100
Pb	0.95	$/lb	95
Ag	22.75	$/t oz	95

1.14.2 Processing Recovery Assumptions

Based on the metallurgical testwork, pilot plant results and the mine production schedules, the following average life of mine recoveries are incorporated in the cashflow modelling:

- Copper recovery of 85.12%, producing LME grade “A” copper cathodes
- Zinc recovery of 88.41%, producing Special High Grade (SHG) quality zinc cathodes
- Lead recovery of 79.38%, producing Lead Cement compacted as briquettes
- Silver recovery of 51.87%, with a grade of approximately 26 g/t Ag, producing silver cement

To reflect the expected ramp up of initial metal recovery to the factors above, the following ramp-up factors were also applied to the CLC PMR model during Year 1 as follows:

- Copper recovery of 80.90%
- Zinc recovery of 84.03%
- Lead recovery of 63.53%
- Silver recovery to the copper concentrate of 47.92%,

1.14.3 Sensitivity analysis

Economic model sensitivity analysis was completed on the metal prices, recoveries, as well as capital and operating cost estimates. The results indicate that the Project is robust whilst profitability is most sensitive to variations in metal price, grades, recoveries, operating costs, and capital cost in that order.

Figure 1-15 NPV$_8$ Sensitivity

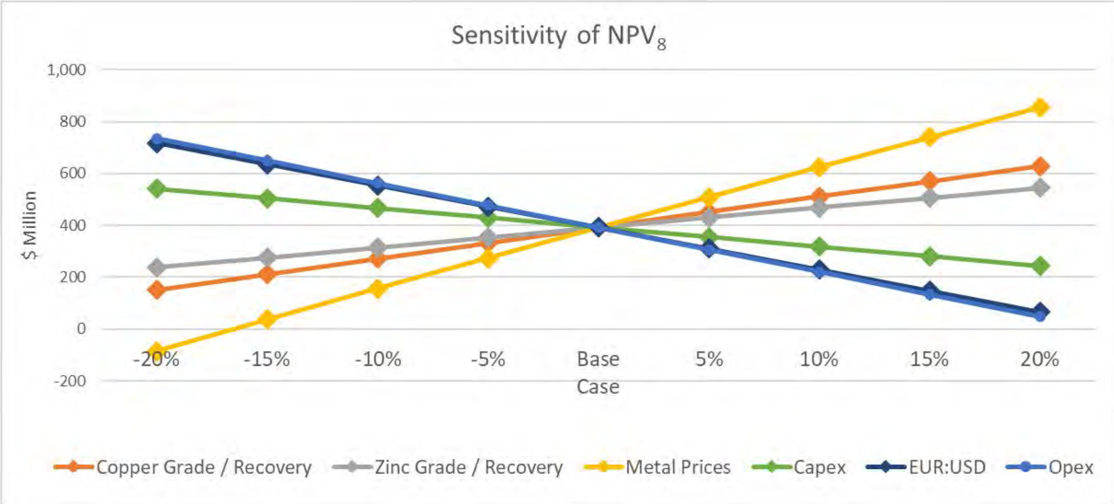


Figure 1-16 NPV$_{10}$ Sensitivity

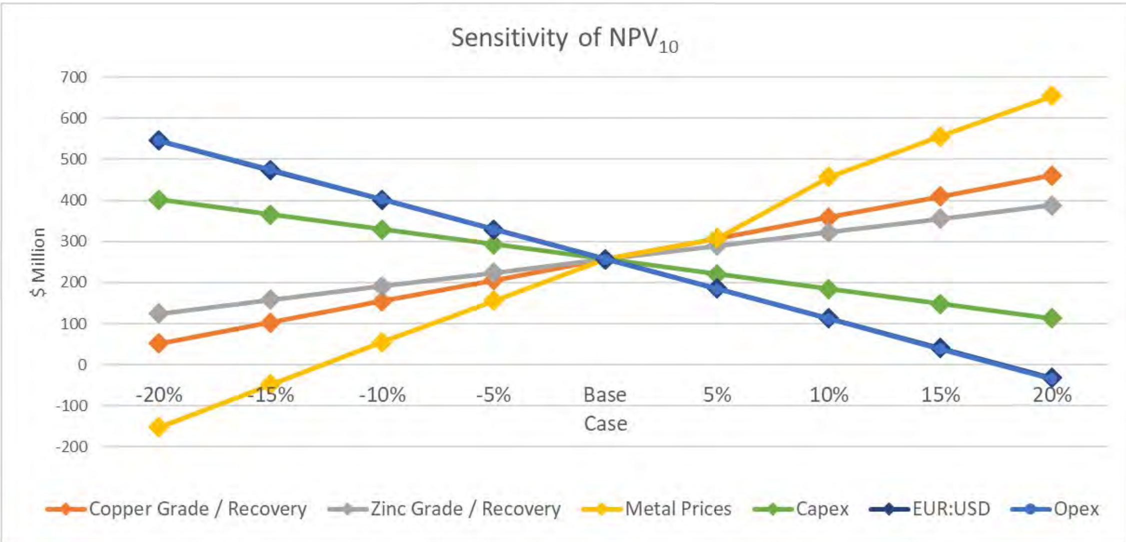


Figure 1-17 Pre-Tax IRR Sensitivity

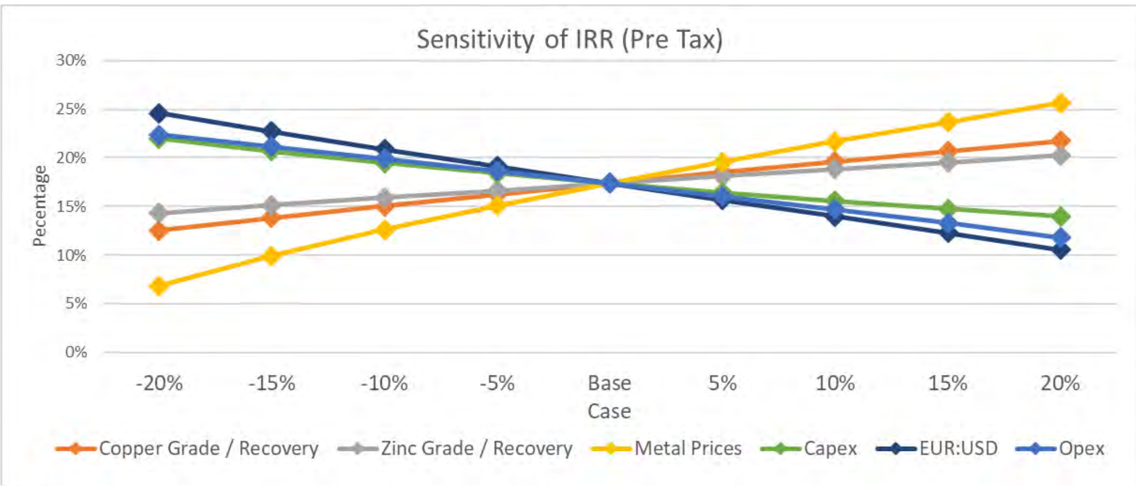
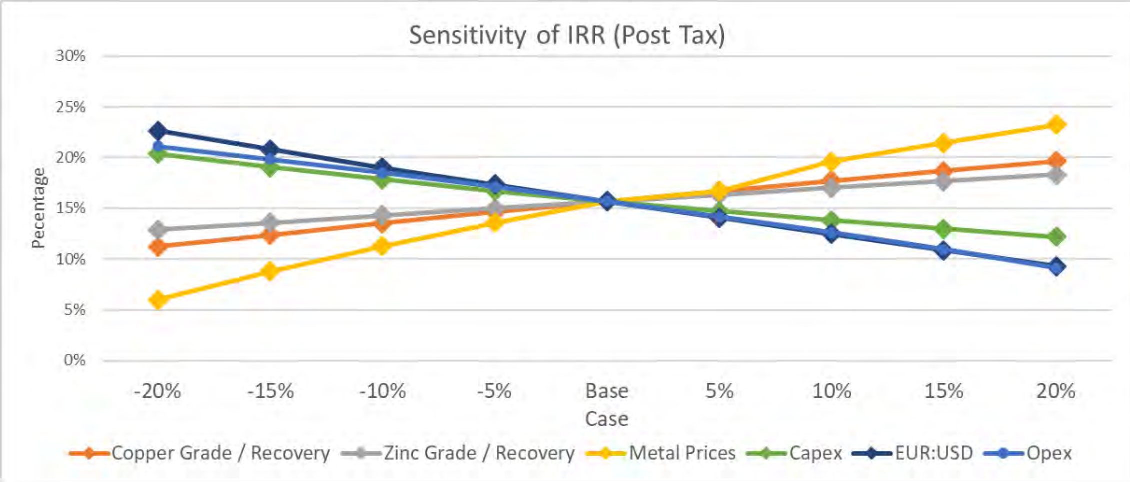


Figure 1-4 Post Tax IRR Sensitivity



1.15 Environmental & Social Compliance

1.15.1 Status of Environmental Approvals

Mining activities in Spain are subject to Spanish national, regional and local environmental laws and regulations which regulate, among other things, air emissions, water discharges, soil contamination, waste management, management of hazardous substances, natural protected sites, protection of natural resources, heritage and endangered species and reclamation. Spain has adopted European Union directives pertaining to environmental matters into its domestic legislation and these rules impose environmental conditions on the management of, among other things, water, waste and air emissions. The valuable experience gained by CLC in environmental management during the operation of the current open pit operations is considered a major strength for the PMR Project.

Under CLC's various licenses, it is required to comply with several environmental commitments including commitments to receiving and observing the terms of several permits. The key permits that CLC have obtained for the PMR Project are;

The Unified Environmental Authorization (AAU)

- The AAU regulates mining activities and auxiliary facilities associated with mining at an environmental level. This authorization includes the original Declaration of Environmental Impact (DEI) of the Las Cruces Secondary Project, which was the formal statement from the regional authority that determined the environmental suitability of Las Cruces. The AAU and the DEI also establish the environmental conditions for the development and operation of Las Cruces regarding protective measures, mitigation and monitoring.

The Integrated Pollution Prevention and Control Authorization (IPPC- AAI)

- The IPPC-AAI regulates the polymetallic refinery in addition to discharge into the Guadalquivir River.

The PMR Water Concession

- The PMR Water concession provides the Project with the necessary water intakes for the development of underground mining and polymetallic refinery operation. In addition, this permit contains the necessary water intake to continue with the artificial recharge of the aquifer using treated and regenerated wastewater. This injection provides an additional guarantee for the preservation of the aquifer located in the area affected by the mine, by maintaining the piezometric levels at the points and values set by the Administration. In order to process this Water Concession application, the Project had to be declared by the Mining Authority as a Project of “Higher Public Interest”. Once this Declaration was obtained, the Guadalquivir Hydrographic Confederation (CHG, the water administration) favourably reported the exceptional authorization regime provided under article 4.7 of the Water Framework Directive for the mine and Polymetallic Refinery Project. This regime is essential to lower the aquifer level and to be able to exploit the mine in optimal safety and operating conditions. This new authorization was granted in March 2023.

The DRS Authorization

- The DRS authorization issued by the CHG, regulates the extraction and re-injection of groundwater surrounding the Las Cruces open pit. The previous DRS Authorization for the Secondary Project has been modified and extended. This new authorization was granted in March 2023 and covers, among other things, the construction of new infrastructure associated with the PMR DRS.

The experience and information of environmental vectors, permits, and community aspects reflect CLC's previous operation and should be considered as a strength and a basis for developing the PMR Project.

1.15.2 Social and Community Related Requirements

CLC maintains good relationships with all four local municipalities (Gerena, Guillena, Salteras and La Algaba) directly affected by the Project. The Company has developed relationships with non-government associations in the area and has meetings with the neighbourhood group, which contains representatives of each of the four municipalities as well as people from nearby properties. These meetings discuss developments, plans and other issues that may be of interest to the group.

In 2010, CLC created the Cobre Las Cruces Foundation (the "Foundation") intended to encourage, promote and develop Environment and Sustainable Development, thereby enabling the Company to meet its objectives regarding social corporate responsibility. Since its start-up, the Foundation has become one of the most active entities of its kind in Andalusia, with more than 500 activities and/or projects performed to date.

The Foundation focuses on two spheres of activity: collaborations with other entities; and its own projects. In the first of these, the Foundation has provided support to other institutions at the regional, provincial and local level to boost education and training, social, charitable work, cultural, sporting and environmental initiatives. Since inception, the foundation has donated more than €10 million to the local economy.

At the time of writing this report there were no material social risks at CLC.

1.15.3 Mine Closure Provisions

A new Restoration Plan was developed and submitted in July 2019. This plan was approved in September 2022. It is based on the PMR Project recently submitted to the authorities and includes a basic Closure Plan for the underground mine project.

The PMR Environmental Restoration Plan for CLC is a continuation of the previous Open Pit Mine Restoration Plan. The current plan incorporates the results of that Restoration Plan plus modifications for project optimizations, updates for the new underground mine project, and incorporation of all permit conditions specified in the DEI, AAU and AAI. The main modifications to the original restoration plan are:

1. A new Tailings Storage Facility (TSF) in the pit. This facility is placed over a partial backfill with marl, which creates a residual void completely lined with marl as a dry seal. The new facility, which includes both plant tailings and waste rocks, is lined to prevent and control the acid drainage. Currently, this facility is already in use. The restoration of the mining pit will be completed with a landfill for inert waste material, mainly from construction and demolition works.
2. The surface areas to be restored have been modified in line with the current underground mining plan.
3. The water supply pond (PSP) will be restored and remain as a wetland since it has now become a lake ecosystem that is home to a multitude of aquatic species of birds and other fauna such as otters.

CLC is not required by law to submit a Construction Project concerning the back-fill of the pit; however, a Performance Project must be registered with the authorities detailing the pit's use as a depository for discarded construction material. This plan was filed in 2012 and has been updated for the PMR Project.

1.16 Risk Analysis and Management

A comprehensive identification of Project risks and the workshopping of management strategies will be completed as the Project engineering phase nears completion. The analysis information below is preliminary and relates to observations that are relevant at this current stage of Project engineering. As conventionally, the identified and grouped Project risks would ultimately be rated in terms of likelihood (or probability) of occurrence, and the consequence to the Project.

1.16.1 Mineral Resource modelling and estimation

Mineralisation volumes

The current extents of mineralised volumes are supported by a reasonable grid of drill data in combination with geophysical information. Previous production experience, during which 5Mt of PPS were mined and stockpiled, is also in support of current 3D modelling of the mineralisation.

The main risk identified in relation to mineralisation volume is the presence of small lenses of barren-silicified shales within the PMS unit. These lenses are sporadic and intersected only by a few drill holes, suggesting that their size is smaller than the drilling grid. Its impact is of small significance to the global resources and the risk has been assessed as low. To mitigate this, close spaced grid drilling is recommended to be conducted ahead of mining.

PMS modelled volume gets thinner towards the North in depth and towards its eastern limit, where it pinches out. Mineralisation in these areas of Inferred resources could potentially be less continuous than interpreted due to its small vertical thickness and overall geological and structural setting. The potential risk to the Mineral Resources Estimate is low, with no impact to Mineral Reserves. Infill drilling in these areas will improve the 3D modelling of the limits of the mineralisation.

Structural model

The Mineral Resource estimate will benefit from continued development of the 3D structural and geological model, improving predicted location and orientation of faults, shear zones, joints, pervasive ductile fabrics and key lithologies. Faults and key structures may have an impact on geology and mineralisation continuity, mine stope designs and mining dilution. The impact of this risk to the Mineral Resource Estimates is considered low to moderate and within the context of the Mineral Resources Estimate classification.

Sulphide and gangue mineralogy

Mineralogical and petrographic knowledge acquired to date is reasonable, however, it is dependent on drill core samples, their location and drilling grid. Although overall mineralogical assemblages are typical of the ones found on most deposits along the Iberian Pyrite Belt, there is a potential risk associated with local variations of these assemblages which could impact the metallurgical performance of the materials. Additional drilling, targeting specific areas and in alignment with metallurgical requirements, is recommended as the Project advances.

Dry bulk density

Dry bulk density shows a significant range of values from non-mineralised to semi-massive sulphides and to massive sulphides, as the presence of sulphides increases. Dry bulk density estimate is included in the Mineral Resources within each mineralisation domain and is considered to capture local changes in a reasonable way.

The most typical bias within the industry on dry bulk density values is related to sample selection towards the more competent parts of the drill core. This type of bias is considered to be at a low level at CLC, as most of the core within the PPS domains is competent, with high core recovery values and excellent RQD index. Nonetheless, any local variance of dry bulk density values not captured by the drilling grid and sample selection could have a potential impact on showing lower or higher density values, which could translate into variances on tonnes and contained metal. The potential risk to the Mineral Resources is assessed as low and is within the context of the classification assignment.

Additional dry bulk density samples and measures are recommended to be taken from future infill drilling programs and from existing core from areas showing significant variability.

Stockwork mineralisation

The mineralisation within the Stockwork domain is present as veins and disseminated forms. The frequency, size and orientation of the ore bearing veins varies, as does grade and contained metal.

Drill spacing has the risk of not providing enough support to the estimates in some areas and mineralisation could have higher variances than predicted. This risk is captured in the mineral resource classification and is considered to be low to moderate within Measured and Indicated, and moderate to high in Inferred materials.

The prediction of size, orientation and grades of these swarms of veins is a challenge and it is recommended to conduct infill diamond drilling in Indicated and Inferred areas in addition to close spaced Ore Control drilling in advance of mining.

1.16.2 Mine planning and Mineral Reserve estimate

Hydrogeology/Mine Dewatering

The hydrological model has been developed using data obtained prior to and during the 12 years of open pit mining. The model was calibrated with both regional groundwater information and piezometric information obtained from existing groundwater control boreholes. In addition, the model was also calibrated with the water balance in the open pit and the existing underground exploration ramp. However, there remains a risk that water inflows encountered during development of the mine exceed modelled outputs and may incur delays and extra costs. The underground dewatering system is designed with 20% spare capacity and 100% pump redundancy at each pump station. The likelihood is possible, whilst the risk has been assessed as low to medium.

Mine geotechnical engineering

Geotechnical inputs are based on data obtained during open pit mining as well as extra data from the exploration decline and geotechnical drilling done for the underground project. The main risk identified geotechnically is how the crown pillar behaves during the partial recovery phase. This is a key part of the Project as it contains almost 30% of the metal contained in the Reserves. Further detailed geotechnical studies are required during the detailed engineering phase in order to mitigate this risk. However, given that the proposed drift and fill mining method includes paste back fill, the likelihood of this risk is considered low.

Faults and Dilution

Faults have been mapped and incorporated into the stope design process; however, it is considered that undetected faults may impact stope hangingwall stability. Longitudinal stopes may have a greater risk of abandonment if the dilution due to instability becomes excessive. Transverse stope design should provide improved control of dilution arising from instability under these geotechnical circumstances.

Ventilation

External review has confirmed that there is a suitable distance between the proposed exhaust and intake sites (>1km separation). A Ventsim model was developed showing airflow quantities (total 270m^3/s flow rate) and velocity rates are in alignment with the peak equipment build up. At this point in time, the only risk related to ventilation that remains unaddressed is in relation to whether the system can be upgraded to cater for future mine extensions.

Development Schedule

The focus of the development schedule is to access the stoping areas and install the Railveyor. Up to 2,000 metres of development per quarter is scheduled over a three-year period leading up to plant commissioning. Delays in the development schedule could cause reduced production rates and increased ore haulage costs in the early years of the schedule. Detailed engineering designs and construction scheduling will be carried out to mitigate a risk which has been assessed as low impact.

Production schedule

Conventional underground mining methods have been selected, making use of equipment suited to the scale of production. As noted above, considerable pre-production mine development is required to access the initial plant feed. To offset this initial hurdle, a sequence of stope development and a production schedule have been devised which seek to mine higher grade ores from Block 2 first. In the initial years, delays, or lower than planned production rates, from the underground mining can be offset by reclaim from the surface stockpile to maintain feed to the process plant. Hence, the potential risk to the Project is considered to be low.

Mine operating costs

The estimated unit mining cost estimates have been built-up from first principles and have been checked against contractor quotes for stoping and development. This work has enabled the calculation of primary mining equipment requirements over the life of the mine. Whilst there is a risk of operating costs being underestimated, the risk and consequences have been assessed to be low.

1.16.3 Metallurgy

Metallurgical Testwork

The testwork performed to date has been undertaken in three phases commencing with conceptual, moving through laboratory and finally into a bespoke pilot plant. The aim of these phases was to confirm and verify the technology and process route. This work was undertaken by a combination of internal and external laboratories.

In the Project phases I and II, laboratory and small-scale pilot plant tests were run to confirm whether the PMS was amenable to the proposed technologies. Tests were successful and resulted in the development of a provisional flowsheet. This led into Phase III of the Project to include in the scope a 1,000kg/hr pilot plant with the target to validate the confirmed technologies under continuous pilot scale trials.

The pilot plant campaign successfully demonstrated both the primary (copper-zinc SICAL) and subsequent secondary (lead-silver brine) leaching technologies as viable processing routes for the treatment of primary CLC sulphide ores. The pilot work also concluded that silver could be successfully recovered from secondary leach solution by cementation. Finally, the use of cementation for lead recovery was subsequently piloted and proved successful. These enhancements were subsequently included in the flowsheet for the PMR.

It is considered that sufficient testwork has been completed to justify a proof of process design concept, capital and operating cost estimates with stated contingency levels, and to advance into detailed design.

Sampling and testing representivity

The early phase testwork was undertaken with a range of samples, mostly composed of drill core used for PMR Mine modelling, with a number of the approximately 114 drill cores selected. These cover 84% of the defined primary deposit. Additionally, two composite samples (laboratory and Pilot Plant) were prepared out of the primary ore stockpiled on surface during the secondary copper ore mining.

To run the pilot plant, a 5,000-tonne sample was prepared by the geology team. Sample was taken from the PMS stockpile, homogenized and characterized. Lead and zinc were upgraded by blending with primary sulphides from the open pit. Note that assays are not exactly the same as for the PMS samples prepared from the drill cores (PMS 4,5 and 6 represent samples used for phases 1 and 2).

Table 1-19 Sampling and testing representivity

		Head grades				Bulk concentrate grades				Sulphate leaching%		chloride leaching%	
	% representativity of deposit	Cu%	Pb%	Zn%	Agppm	Cu%	Pb%	Zn%	Agppm	Cu	Zn	Pb	Ag
PMS4	67	1.05	1.32	2.95	30	3.49	3.45	9.83	86	97.5	94.5	97.60	98.00
PMS5	84	1.16	0.75	1.95	22	4.31	2.58	7.45	67	97.5	95.9	96.46	98.53
PMS6	84	0.82	1.00	2.16	23	2.52	2.41	6.33	53	95.9	97.7	97.56	98.02
Pilot plant	84% from stockpile	1.26	2.7	3.44	68	2.14	4.59	6.26	107	93	94	96.00	97.00
Base case		**1.14**	**1.1**	**2.25**	**27.8**	**2.92**	**2.52**	**6.00**	**58.3**	**94.0**	**94.1**	**98.0**	**98.0**

Whilst the samples prepared for the testwork programs were not entirely representative of the underground orebody, they do demonstrate the robustness of the PMR technology in responding to variations in feed mineralisation and grades.

1.16.4 Processing

The PMR feed is provided from a single underground orebody, with minor blending from existing surface stockpiles. Whilst the stockpiled material may exhibit minor oxidation, the technology selected is unlikely to be affected.

Variability of plant feed

The PMR process plant is designed to process a feed “blend” within an envelope of assays which ensures the capacities of the key areas are not exceeded. While significant “unused” copper capacity exists due to only partial use of the existing facility, zinc, lead and/or silver and limited by design to mine plan grades. Within these grade constraints the process has been proven to possess considerable flexibility.

Equipment sizing

The PMR circuit is designed based on in-house and external engineering understanding of the technology. The Company has significant expertise in crushing, grinding, floatation, filtration and copper leach/SX/EW to have confidence in the design. Zinc design is provided by well-known package suppliers with considerable experience. The silver and lead cementation circuits are bespoke with both having been proven at pilot plant scale.

Design uncertainty

The silver and lead cementation circuits are bespoke and as such may present commissioning challenges. To mitigate these challenges, full-scale equipment is proposed to be piloted prior to the main plant commissioning.

1.16.5 Tailings storage

The continued use of In-pit tailings disposal to dispose, store and encapsulate plant tailings in the In-pit tailings facility (IPTF) is considered a significant advantage of the Project when compared with previous surface dry stacking or conventional tailings disposal methods.

The method has been successfully operated and proven from 2016 to 2023 to encapsulate waste rock and the tailings derived from the reprocessing project. Continuous geotechnical and hydrogeological monitoring during these years has confirmed that the disposed material is properly encapsulated and validates the in-pit disposal selected for the PMR Project.

1.16.6 Water Management

The expansion and continued successful operation of the site water management system to include new underground water sources is critical to the continued mining operations at CLC. The Drainage and Reinjection System (DRS) to maintain aquifer levels and water quality such that they are unaffected as a result of mining operations, is a strict legal permitting requirement and a community priority. This is evidenced by CLC receiving three Notices of Violation (NOV), served in 2014, 2017 and 2019, regarding loss of groundwater and insufficient compensation by the DRS.

After several upgrade projects, the reliability of the neutralization plant has increased markedly and currently the plant produces a very stable high-quality effluent with very few scattered and short periods of non-compliance. The limits for the discharge to the aquifers and to the Guadalquivir River are being consistently met for the current open pit operations. The expertise and experience gained by CLC in operating the water management system during the open pit operations can be considered as a strength and a basis for developing the PMR Project.

1.16.7 Infrastructure

CLC is an existing open pit mining operation and processing operation which has been established for more than fifteen years. The associated infrastructure as required by these operations is still in place and includes sealed roads, power lines and substations, process plant, site offices, workshops, tailings dams, and waste storage facilities.

In general, this infrastructure remains in good working condition and is suitable for reuse to support the PMR Project. Significant additional processing facilities and modifications to existing processing infrastructure is required and is discussed in more detail above.

Relatively modest additional surface infrastructure is required for the mining operations including ventilation fans, site offices, change rooms, light workshops, consumable storages, electrical power extensions and UG utilities distribution systems. This infrastructure is standard to the local mining industry and can be reliably designed, manufactured and constructed by local contractors, with some specialist equipment sourced from within the wider EU.

The two most significant infrastructure additions relate to power supply and underground materials handling with the Railveyor.

Power supply costs

Electrical power costs contribute to ~30% of plant operating costs with large fluctuations in local unit power costs in recent years being attributable to European gas prices. The most critical new infrastructure required is the construction of a new photovoltaic solar farm to provide up to 30% of electrical demand. To minimise execution risk, the plant will be constructed and operated by an external third party similarly to other recently completed projects in the area.

Environmental studies and permitting

All key permits for the PMR Project are in place including the Unified Environmental Authorisation, Integrated Pollution Prevention, and Control Authorization, PMR Water Concession, DRS Authorization, and the modified mining concession. The experience and information of environmental vectors, permits, and community aspects reflect CLC's previous operation and should be considered as a strength and a basis for developing the PMR Project within the requirements of these permits.

A remaining permit required to commence construction is the Construction Permit. A procedural process submitted to the local authorities is expected to be received with 6 months of submission. Whilst procedural in nature, the process and site layout design are required to be developed to an advanced stage and 'locked'. Design changes to these aspects can result in additional engineering costs and time delays in the granting of the Project construction permit.

1.16.8 Cost estimation

The capital cost estimate for the processing plant and related site infrastructure has been comprehensively assessed by Lycopodium including benchmarking to their internal database and specification of an appropriate contingency. Contingency factors of between 15% and 20% have been applied, deriving an overall contingency of 16.9%.

The capital cost estimate for the underground mining and other surface infrastructure has been compiled by the CLC team utilising budget quotes from reputable suppliers. A contingency factor of 10% has been applied after benchmarking overall project contingency against similar recent projects at the PFS and FS stage.

These preliminary capital cost estimates are considered to be suitable for adoption at this stage of Project engineering and evaluation, and will be improved upon as the detailed engineering phase proceeds.

Operating cost unit rates have been calculated utilising a range of sources including existing plant, infrastructure, utilities and administration costs at CLC, in addition to current market rates for consumables. Mining unit rate estimates drew on a 2022 CostMine database, whilst consumable consumption rates in the plant were derived from existing data, pilot plant testing and other metallurgical testwork results. For the mine operations, estimates were completed by mining consultants with extensive knowledge of similar operations. It is generally considered that the North American basis of the CostMine database provides a conservative estimate for the IPB at this stage of project development.

1.17 Conclusions & Recommendations

1.17.1 Mineral Resource Estimate

The CLC PPS Mineral Resource estimate is detailed in the 2022 CLC Technical Report. The Mineral Resource statement has been updated using a lower Cu_{eq} cutoff grade of 0.8% as supported by this Technical Report's Mineral Reserve studies. Consequently, Measured and Indicated Mineral Resources have increased from 36.24 million tonnes to 41.38 million tonnes with the Cu_{eq} grade decreasing from 2.51% to 2.29%. The Mineral Resource classification has considered the relevant factors for reasonable prospects of eventual economic extraction.

The geologically modelled PPS mineralisation and spatially related copper, zinc, lead, and silver grade estimates were supported by high-quality diamond-drilled core samples and detailed 3D geological modelling of the respective domains of mineralisation. Estimates were completed using ordinary kriging, with post processing of parent cells using localised uniform conditioning to ensure that block estimates were relevant to the scale of planned underground mining.

The QP recommends the following for future Mineral Resource estimates;

- Continue to review and update the 3D geology and structural models.
- Complete an infill diamond drill program with two objectives:
  - upgrade Inferred Mineral Resources inside the current underground mining design to potentially increase the Mineral Reserve inventory, and
  - increase Mineral Resources in exploration/extension areas.
- Improve on the host rock model of the PPS (massive) to minimize future dilution and losses.
- Integrate hydrogeological and geotechnical data into the geological model.
- Work on improving the geometallurgical model for improved mine planning definition.
- PPS stockpile volumes and position should be confirmed with updated surveys.
- PPS stockpiles were built from 2011 and may be subject to a degree of oxidation. Stockpile grades and tonnes may benefit from review and an estimate update.

Some of this work is being done inhouse with the remainder outsourced to contractors and consultants. The total cost of this work is estimated to be $2.1M .

1.17.2 Mineral Reserve Estimate

The Las Cruces Mineral Reserve estimate, the subject of this report, has been developed using the Mineral Resource estimate declared in the 2022 CLC Technical Report and which has been validated and updated in this report.

A conventional approach has been adopted in the process of optimising a mine planning model derived from the Mineral Resource model, followed by detailed stope and development design, production scheduling, and Mineral Reserve estimation.

Groundwater inflow has been identified as a mining risk and has been taken into account during the mine design process. The new dewatering strategy based on a combined approach of surface and underground drains is focused on depressing the water level in advance of mine development. A predictive tool has been developed to estimate the type and quantity of water inflow.

A Railveyor hauling system was considered appropriate for the Las Cruces Underground Project in late 2019. The system has developed since then and is now a proven technology with production rates achieved well over CLC targets.

The Mineral Reserve estimate has been developed using a Net Value calculation including expected metallurgical recoveries obtained from the pilot process testing together with other modifying factors that reflect an achievable mining plan and production schedule for the Project, at this stage of evaluation.

Taking into account all the comments above, the technical risk related to mining is considered to be low to medium. Mineral Reserve studies have been finalized up to a feasibility stage. Studies include consideration of technical evaluations associated with mine designs, production schedules and the initial process plant designs and throughput profiles. The QP recommends that: -

- Drilling of additional holes to collect Hydrogeological and Geotechnical data should be undertaken prior to commencement of mining. The data collected should be used to confirm and if necessary, modify design parameters in preparation for engineering design. The drilling would be conducted by drilling contractors with data analysis by hydrological and geotechnical consultants at an estimated cost of $200,000.

- Once mining commences, hydrological and geotechnical reviews should be completed annually to confirm the mine design parameters, update 3D hydrological and geotechnical models, audit the installation of ground support in the active working areas of the mines, and assess the dewatering performance. This work would be conducted by hydrological and geotechnical consultants at an estimated cost of $120,000 per year.
- Develop specific drilling programs to precisely locate ore boundaries and assess ground conditions, for example at the Block 1 (La Manta) hangingwall. The cost of this work is estimated at $360,000 per year.
- Re-estimate Mineral Reserves following updates to Mineral Resource estimates and/or material changes to cost and financial inputs. This work would be completed by CLC personnel at no additional cost.

1.17.3 Metallurgical & Processing

The Polymetallurgical Refinery Project was initiated by CLC in order to process its primary sulphide mineralisation and so extend the life of mine following depletion of the secondary copper ore in 2021. The Project commenced in 2014 by defining the actual technology and Mineral Resources available. The optimum technology route was established after a research period carried out by CLC in conjunction with external organisations, universities and institutions. The proposed process technology was validated in a pilot plant constructed and operated by CLC, treating 5,000 tonnes of primary ore during 2016 to 2017. The “dynamic cementation” process for silver and lead was piloted and validated during 2017-2018. Pilot plant results were used as the basis for the plant conceptual engineering.

Preferred locations have been identified and conceptual designs produced for the Project infrastructure including new hydrometallurgical plant, paste plant, overland piping, expanded water management and treatment equipment and new electrical equipment and power lines. The concepts for these are considered to be suitable for the Project at this stage of the engineering phase.

1.17.4 Cost Estimation and economic outcomes

Mine and process operating costs, plus general and administration costs (G&A) have been estimated from first principles utilising industry databases, supplier quotes and historical operating costs. Metal costs (i.e. including transport and refining charges (TCRCs)) have been advised by the Company’s own metals marketing group. The process plant and related infrastructure capital costs have been estimated by a globally experience metallurgical processing engineering company primarily by the means of consultant and / or vendor estimates. The order of accuracy of the capital cost estimates reflects the adoption of contingency factors of up to 13%.

While budget pricing has been obtained for the majority of the large equipment items, the capital cost estimate for the ECS has also been compiled using pricing information for a number of large vendor packages. In general, there needs to be a greater understanding of the defined scope, battery limits, and service / utility requirements for each of these packages to ensure that all capital costs have been captured. These vendor packages include;

- Zinc Solvent Extraction
- Zinc Electrowinning
- Zinc Melting and Casting
- Paste Plant

An allowance for tie-ins to plant and equipment within the existing plant has been allocated in the capital cost estimate based on a preliminary assessment by CLC. It is recommended that a comprehensive evaluation be performed during the next project phase to ensure that all costs have been captured and to confirm that the proposed location and installation of new plant and equipment within the existing plant site is indeed practical. The cost of this work is estimated at $1M.

ITEM 2 INTRODUCTION

This Technical Report on the Cobre Las Cruces Polymetallic Primary Massive Sulphide (PMS) Project (the property or Project) has been prepared by Carmelo Gomez Dominguez, Anthony Robert Cameron and Robert Stone for First Quantum Minerals Ltd (FQM, the issuer or the Company).

This report supersedes the 2022 CLC Technical Report issued by the company and focusses on the Mineral Resources and Reserves of PMS and associated Stockwork mineralisation.

2.1 Terms of Reference

This Technical Report has been written to comply with the reporting requirements of the Canadian Securities Administrator's National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101 or the Instrument) and with the 'Australasian Code for Reporting of Mineral Resources and Ore Reserves' of December 2012 (the 2012 JORC Code) as produced by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC).

The effective date for the Mineral Resource and Mineral Reserve estimates is 30th September 2023.

2.2 Qualified Persons and Authors

The Mineral Resource estimates were prepared by Mr. Carmelo Gomez Dominguez of FQM who meets the requirements of a Qualified Person according to their Certificate of Qualified Person (QP) attached in Item 28. Contributing authors are Mr Juan Manuel Escobar Torres (CLC Project Geologist) and Mr David Gray (Group Manager, Mine Geology & Resources) both of whom meet the requirements of a QP.

The Mineral Reserve estimates and mining aspects of this report were prepared by Mr. Anthony Cameron of Cameron Mining Consulting who meets the requirements of a QP according to his Certificate of Qualified Persons attached in Item 28. Contributing author is Mr Ivan Carrasco Martiañez (CLC Mining Engineer) who meets the requirements of a QP.

Metallurgical testing, mineral processing and recovery aspects of this report were prepared under the supervision of Mr Robert Stone (a Qualified Person). Mr Stone of FQM meets the requirements of a QP according to his Certificate of Qualified Persons attached in Item 28. Contributing author is Mr Joaquín Gotor Martínez (CLC PMR Technical Manager).

Table 2-1 identifies which items of the Technical Report have been the responsibilities of each person.

Table 2-1 Technical Report Author Details

Name	Position	NI 43-101 Responsibility
Carmelo Gomez Dominguez BSc Hons (Geology), EurGeol	Group Principal Geologist - Mine and Resources FQM (Australia) Pty Ltd	Author and Qualified Person Items 7-12 and 14
Anthony Robert Cameron BEng (Min), Grad Dip Bus, M Comm Law, FAusIMM	Consulting Mining Engineer Cameron Mining Consulting Ltd	Author and Qualified Person Items 1 - 6,15,16,18-26
Robert Stone BSc Hons (CEng), ACSM	Group Consulting Metallurgist FQM (Australia) Pty Ltd	Author and Qualified Person Items 13 and17

2.3 Sources of Information

The sources of information for the geology and Mineral Resource estimate include diamond-drilled core, logging and sample analytical data, in-pit geological mapping and currently relevant information from previous Technical Reports.

Mining, metallurgy, processing and economic sources of information are from the PMR Engineering Study completed in 2023, which was centred on pilot plant testing results and updated underground mine designs.

Other relevant information has been gathered from previous Technical Reports on the property and updated with information provided and translated by senior site personnel.

2.4 Personal Inspections

Mr. Carmelo Gomez Dominguez who is a QP and permanent employee of the Company, has worked at the Cobre Las Cruces operation between 2007 to 2016 as a geologist and has most recently visited the property in September 2023. Mr. Gomez Dominguez was responsible for geology modelling, block model estimates as well as grade control, reconciliation, drilling, logging, sampling and analysis. Most recently, Mr Gomez Dominguez has inspected drill core and reviewed geological data, the geology model interpretations as well as the collection and sample preparation procedures as part of data verification for the CLC Mineral Resource estimate.

Mr. Robert Stone, QP and author, visited the property in April, September and October 2013, November 2014, November 2015 and November 2022. Mr. Stone inspected the process plant and PMR pilot plant, attended technical meetings and reviewed operational performance with site management.

Anthony Cameron, QP and author, visited the property in 2011, 2015, and 2023. During the most recent site visit in June 2023, Mr Cameron visited the portal, the tailings facilities (in-pit and external), the stockpiles, as well as the existing water process facility and the sites being prepared for the new paste fill and process facility.

2.5 Conventions and definitions

Reference in this Technical Report to dollars or $, relates to United States dollars. Copper, lead, and zinc metal production is reported in (metric) tonnes and (imperial) pounds, where the conversion factor is 1 tonne (t) = 2,204.62 pounds (lb). Silver production is reported in (troy) ounces and with an adopted abbreviation of (t)oz.

The conventional chemical abbreviation for copper of Cu is used throughout this report, whilst the abbreviation for lead is Pb, Zinc is Zn, and Silver is Ag. ASCu is used to denote Acid Soluble Copper and TCu is used to denote Total Copper.

Where not explained in the text of this report, specific terms and definitions are as listed in Table 2-1.

Table 2-1 Terms and definitions

Term	Definition	Term	Definition
µm, mm, cm, m, km	microns, millimetres centimetres, metres, kilometres	**Mtpa**	million tonnes per annum
bcm	bank cubic metres	**MW, LG, MG, HG**	mineralised waste, low grade, medium grade, high grade
bn	bornite	**NPV**	net present value
cpy	chalcopyrite	**oz**	ounces
csv	comma separated value	P_{80}	80% passing
g, kg	grams, kilograms	**pH**	potential of hydrogen
g/t, kg/t	grams per tonne, kilograms per tonne	**py**	pyrite
ha	hectares	**Q1, Q2, Q3, Q4**	quarter 1 to 4
IRR	internal rate of return	**t, kt, Mt**	tonnes, thousands of tonnes, millions of tonnes
kWh/t	kilowatt hours per tonne	**tpa**	tonnes per annum
lb	pounds	**tpd**	tonnes per day
LOM	life of mine	**tph**	tonnes per hour
m/s	metres per second	**V, kV**	volts, kilovolts
Ma	mega annum (million years)	**W, MW**	watts, megawatts
masl	metres above sea level	**WGS**	Western Geodetic System
mE, mN	coordinates: metres East, metres North	**L/s**	Litres per second

ITEM 3 RELIANCE ON OTHER EXPERTS

The authors of this Technical Report do not disclaim any responsibility for the content contained herein.

ITEM 4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The CLC operation is located 20 kilometres northwest of Seville, the capital city of the Seville Province within the Andalusian region of southern Spain (Figure 4-1).

Figure 4-1 Location of Cobre Las Cruces Operation, Seville, Spain (CLC, 2022)

The operation is situated in the relatively flat Guadalquivir River basin that extends some 420 km from the Alcaraz and Segura mountain systems in the east to the Costa de la Luz on the Atlantic Coast in the west. The Guadalquivir basin is set between a long range of low mountains to the north (Sierra Morena) and a group of formations making up the Cordilleras Beticas mountain range to the south. The geographic coordinates of the property are 37°19’43”N, 06°06’15”W.

4.2 Tenure

The site operating entity is Cobre Las Cruces S.A.U. (CLC), of which FQM holds a 100% interest.

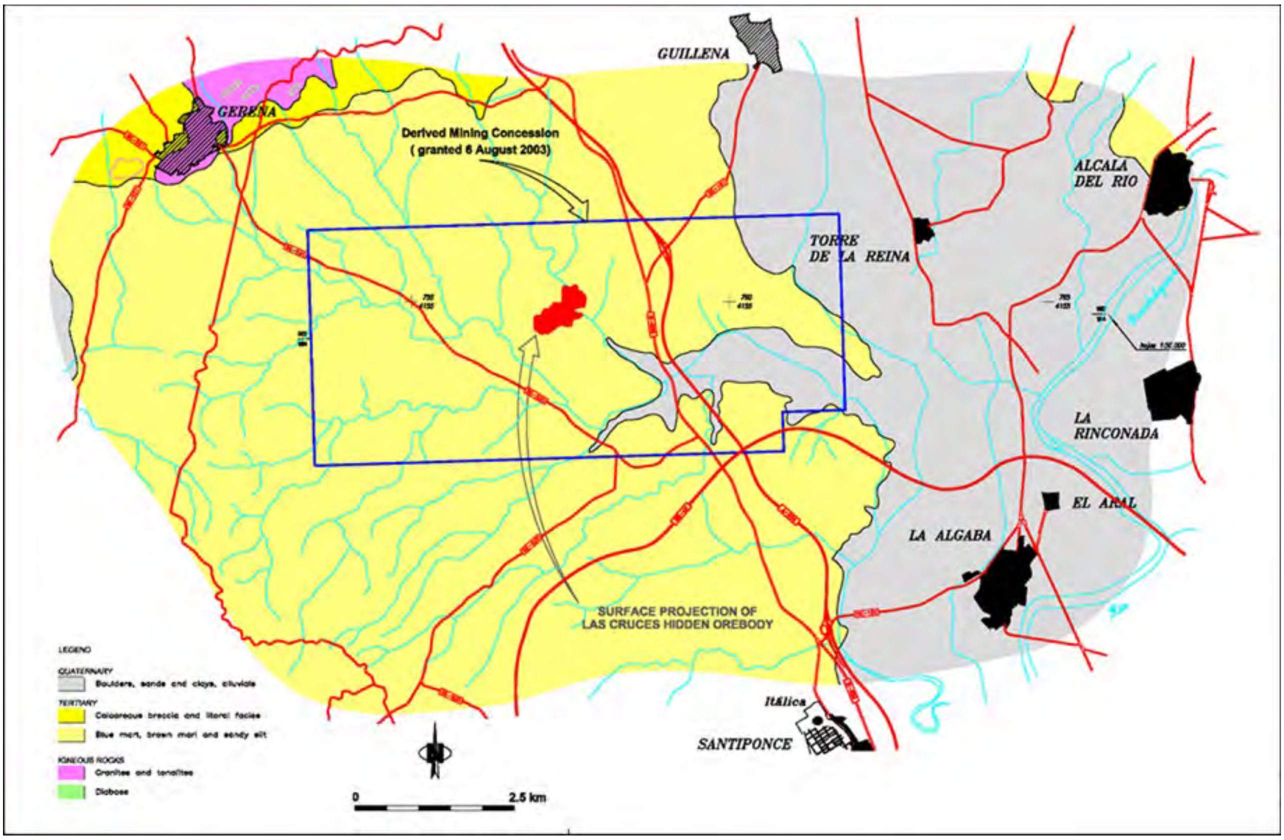
Mining Concession No. 7,532 was granted to CLC by the Andalusian Regional Ministry for Employment and Technological Development on August 6, 2003 and expires on August 6, 2033. The original concession permitted the production of Cu via open pit mining methods.

A modification of the Concession (7532-A) was granted in June 2021 to permit CLC to produce the four metals (Cu, Zn, Pb, Ag), conduct underground mining operations, as well as construct and operate the new PMR plant.

Mining concessions are granted for 30 years with two potential extension periods, each of 30 years, for a total of 90 years. Application for extensions must be made three years before the end of the concession period.

Figure 4-2 shows the mining concession location relative to the Project area.

Figure 4-2 Location of the Cobre Las Cruces Mining Concession (CLC, 2015)



4.3 Rights and Surface Land Ownership

CLC currently owns approximately 1,000 hectares of land for its operations with the mineral and surface rights fully enclosing the deposit. Land ownership within Spain is procedurally clear and is established via survey. Mineral rights and surface land are separately transferable under Spanish law.

The operation also covers some public land, specifically three streams, a livestock trail, and some rural tracks. Some of these were relocated at the request of the authorities.

Rights-of-way for associated infrastructure outside the Project area, such as the water pipeline from San Jeronimo, water wells and pipelines to the site, a 220-kV substation, and two high voltage electrical transmission lines, occupy an area of about 15 hectares.

4.4 Royalties, Payments & Agreements

The property is subject to a royalty of 1.5 percent of the copper metal produced, when copper prices are greater than or equal to $0.80 per pound, and is payable to the International Royalty Corporation as part of the original purchase agreement.

4.5 Environmental Liabilities and Permitting

CLC is located in an agricultural area well away from populated areas. As the open pit operations were developed, rehabilitation liabilities were incurred that were clearly identified in permitting budgets. Permitting included environmental approvals for mining, plant processing, mine waste disposal, as well as land use permits and approvals. Works and activity permits were required from local community councils.

Environmental permits for the PMR Project were granted in June 2021 and the water concession was granted in March 2023. The environmental liabilities incurred in the existing operations carry over to the PMR Project. Further details on current environmental liabilities are provided in Item 20 (Environmental Studies, Permitting and Social or Community Impact)

4.6 Archaeological

All identified archaeological sites were investigated by registered archaeological experts prior to the commencement of open pit operations. Archaeological sites that were not initially identified but subsequently encountered during operations were investigated by an archaeological team retained by CLC. The existence, location and findings were detailed and catalogued before operations were allowed to continue in or near the affected area. Archaeological investigation will continue, using the procedures developed for the open pit operations, during the operation of the PMR Project.

4.7 Potential Access and Exploitation Risks

At this point in time, there are no known factors or risks that may affect access, title, or the right or ability of CLC to continue operations.

ITEM 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

CLC is accessible by well-maintained all-weather paved roads. The N-630 highway is in service to the east side of the property, while the SE-3410 highway crosses the property in the south.

Seville province has a good rail network, including a high-speed passenger rail service to Madrid. Seville has an international airport with connections throughout Europe and is 30 minutes by car from of the operation. Seville is noted for having its own inland port (restricted for large vessels), the only inland port in Spain. Huelva has a shipping port which is 85 km southwest of CLC.

The village of Gerena is located just 4 km to the northwest of CLC.

5.2 Climate

Seville province is semi-arid with a Mediterranean climate having hot, dry summers (May through September) and warm, fairly wet winters. Extended periods of drought are common. Autumn and winter rainstorms are generated from south westerly winds that bring hot humid air into the area. Summer temperatures average 30°C and reach above 40° in July and August. Winter temperatures average 15°C with minimum temperatures down to -5°C in December and January. The mean annual rainfall is 550 millimetres but ranges from 300 to 1,000 millimetres with the bulk of this generally falling within a four-month period in winter from October through to the end of January. The Project will operate continuously through the year.

5.3 Physiography

CLC is located within relatively flat areas of the Guadalquivir River Basin. Before mining, the property elevations ranged from 15 to 45 metres above sea level. Topography surrounding the Guadalquivir River Basin is that of gently rolling hills.

The property is noted for having moderate seismic activity with the potential for a magnitude 5.4 (on the Richter scale) occurrence every 250 years. This data has been defined for the Seville area by the *Instituto Geográfico Nacional* and the *Centro Internacional de Sismología.*

5.4 Vegetation

Prior to acquisition of CLC, the property was used for agriculture. The closure plan states that, where possible, the land will be returned to its former use. Areas not returned to agricultural use will be revegetated with native plant species.

5.5 Local Resources

The city of Seville has a large well-educated population of approximately 685,000 people. In addition, the village of Gerena is about 4 km to the northwest and has a population of approximately 7,400 people. The Company’s local employment policy had 42% of the 492 (2021) strong workforce sourced from the surrounding four municipalities. Where possible, the Company uses locally based contractors and suppliers.

5.6 Infrastructure

Electrical power is supplied to the CLC site via a 4.2 km long dedicated 220 kV branch line. Power is distributed internally through a 20 kV power ring to substations. The primary source of power supply during the PMR Project will be from renewable energies (photovoltaic and biomass).

The PMR Project will process complex polymetallic primary sulphides from a plant throughput of around 2.2 million tonnes per year. High recovery processing will produce final commercial products of copper cathodes, zinc ingots, silver cement and metallic lead briquettes. The existing plant will be supplemented with a new bulk flotation, zinc solvent extraction, zinc electrowinning and a chloride leaching facility, and with a lead and silver recovery plant. The crushing and grinding plant will be new (re-using the existing ball mill and cone crushers). A new pre-reactor for the primary leach stage will also be installed. A paste plant will also be constructed for the underground paste fill operations. Tailings deposition and storage in the existing open pit void will continue. Water will be sourced from underground operations and via the existing 18.6 km pipeline from the San Jerónimo sewage treatment plant located northwest of Seville. This sewage treated water is pumped into a pond (PSP) with a total capacity of 1.3 Mm^3 located within CLC limits.

The infrastructure to manage the underground mine dewatering volumes will entail extraction wells, water treatment plants, and injection wells to return the treated water into the aquifer. Underground water not extracted by means of wells will be pumped via a series of underground pump stations to the surface and treated. The underground water management plan includes a new emergency pond.

Water discharge to the Guadalquivir River will continue through the existing pipeline in accordance with permit requirements.

5.7 Sufficiency of surface rights

As noted in Item 4.3, the Company currently owns around 1,000 hectares of land and holds the mineral and surface rights fully enclosing the operations, including current and proposed infrastructure.

ITEM 6 HISTORY

6.1 Prior Ownership

The Las Cruces deposit was discovered in 1994 by Riomin Exploraciones, S.A. (Riomin), a wholly owned subsidiary of Rio Tinto plc. The discovery was the result of exploration drilling on a gravimetric survey anomaly. Riomin drilled the deposit between 1994 and 1999 and prepared a feasibility study in 1998.

In 1999, MK Resources (formerly MK Gold Company) acquired a 100 percent interest in the project from RTZ and took over Cobre Las Cruces S.A.U. (CLC) as the local Spanish subsidiary company. Leucadia National Corporation, a diversified investment and holding company, had a 72 percent share in MK Resources. In 2001, Bechtel, an international engineering and construction company, completed an independent feasibility study for the secondary copper project. The Bechtel Feasibility Study incorporated results of an environmental impact study completed by *FRASA Ingenieros Consultores*, a team of national and international environmental engineering experts based in Madrid.

Following CLC's 2001 environmental impact study, on May 9, 2002 a favourable Declaration of Environmental Impact was issued, and the Mining Concession was subsequently granted on August 6, 2003.

In November 2003, DMT – Montan Consulting GmbH (DMT-MC) prepared a new independent feasibility study based upon Outokumpu technology, followed by the Feasibility Study Addendum I in May 2004. The DMT-MC Feasibility Study incorporated the requirements from the Declaration of Environmental Impact, the Mining Concession and various water permits into the development plan for the operations. The DMT-MC Feasibility Study and addendum were reviewed by Pincock, Allen and Holt (PAH), an independent mine engineering consulting company.

On May 3, 2005, Inmet Mining Corporation (Inmet) announced that it had entered into an agreement with MK Resources and its majority shareholder, Leucadia National Corporation, to acquire a 70 percent interest in CLC. The purchase of CLC by Inmet was completed on August 22, 2005.

In January 2007, SNC-Lavalin was awarded the EPCM contract to construct the plant and associated infrastructure. By the end of 2007, Inmet reported that essentially all engineering and procurement was complete, 51% of construction was complete, and 71% of total physical progress was complete. In addition, 7.9 million bank cubic metres (bcms) of waste was removed from the mine in 2006 and 19.9 million bcms were removed in 2007.

In May 2008 the authority responsible for the dewatering and reinjection system (DRS) permit notified CLC that it had suspended authorization of the DRS for the Project. This suspension prevented CLC from mining in the open pit until the suspension was lifted in April 2009. The first ore was delivered from the mine to the plant on May 26, 2009, and the first copper cathode was produced in June 2009.

In April 2013 FQM acquired 100% ownership of CLC after the successful takeover of Inmet.

6.2 Previous Mineral Resource and Mineral Reserve Estimates

6.2.1 Mineral Resource

The Mineral Resource estimate disclosed in this Study is unchanged from that documented in the January 2022 CLC Technical Report and remains focused on Primary Polymetallic sulphide mineralisation. The Mineral Resource statement disclosed in this Study uses an updated Cu_{eq} calculation and lower cutoff grade than that adopted for the 2022 CLC Technical Report. In addition, the previously declared secondary stocks of the 2022

CLC Technical Report have now been depleted. No additional drillhole data or geological modelling has been included into the updated Mineral Resource estimate for this Study.

Prior to the January 2022 estimate and 2023 update, a Mineral Resource estimate was completed on 30th June 2015 and is detailed in an NI 43-101 Technical Report titled “Cobre Las Cruces Operation Andalucía, Spain” (the 2015 CLC Technical Report).

The June 2015 Mineral Resource estimate (Table 6-1) was focussed on secondary sulphide mineralisation that had been depleted by open pit mining as at May 2015. The estimate was supported by a closely spaced grid of diamond drill core and blast hole samples. The mineralisation was sub-divided into HC, HC4 and DZ domains. Each domain’s volume was constrained by wireframe shapes derived via interpreted string envelopes on vertical sections spaced at around 25m apart.

Geostatistical and spatial analysis checks supported the use of ordinary kriging as a suitable estimation method. The tonnage estimates were classified according to geological and grade continuity, whilst confidence in the grade estimates was supported by overall data quality and sample assay QAQC.

Hence, it is considered that the June 2015 Mineral Resource estimates were of good confidence and therefore reliably suited for use in mine planning and production scheduling and for conversion into a Mineral Reserve. It is the QP’s assessment that the 2015 estimates of the secondary sulphide mineralisation were representative of the in-situ mineralization as well as the data and information that was available at the time.

In contrast, the June 2015 estimates of Polymetallic Primary Sulphide mineralization were based upon a much wider drill grid of data and were therefore classified as Inferred Mineral Resources. These estimates required additional infill drilling to improve geological understanding and to able to upgrade the Inferred Mineral Resource classification to Measured and Indicated, as included in the 2022 Mineral Resource update to estimates and classification with additional 229 drillholes, with later update on copper equivalent formula on 2023.

The June 2015 and January 2022 Mineral Resource statements are reproduced in Table 6-1 and Table 6-2 below. The 2015 statement was depleted as at the end of May 2015 and the 2022 statement depleted as at end of December 2021. The 2022 statement copper equivalent formulas were updated to reflect the changes in price of the metals, expected metallurgical recoveries and amounts payable by the smelter. The Mineral Resource inventories shown in Tables 61 and 62 are inclusive of the Mineral Reserve inventory. The Mineral Resources that were not in the Mineral Reserves do not have demonstrated economic viability.

Table 6-1 The June 2015 CLC Mineral Resource statement depleted as at end of May 2015

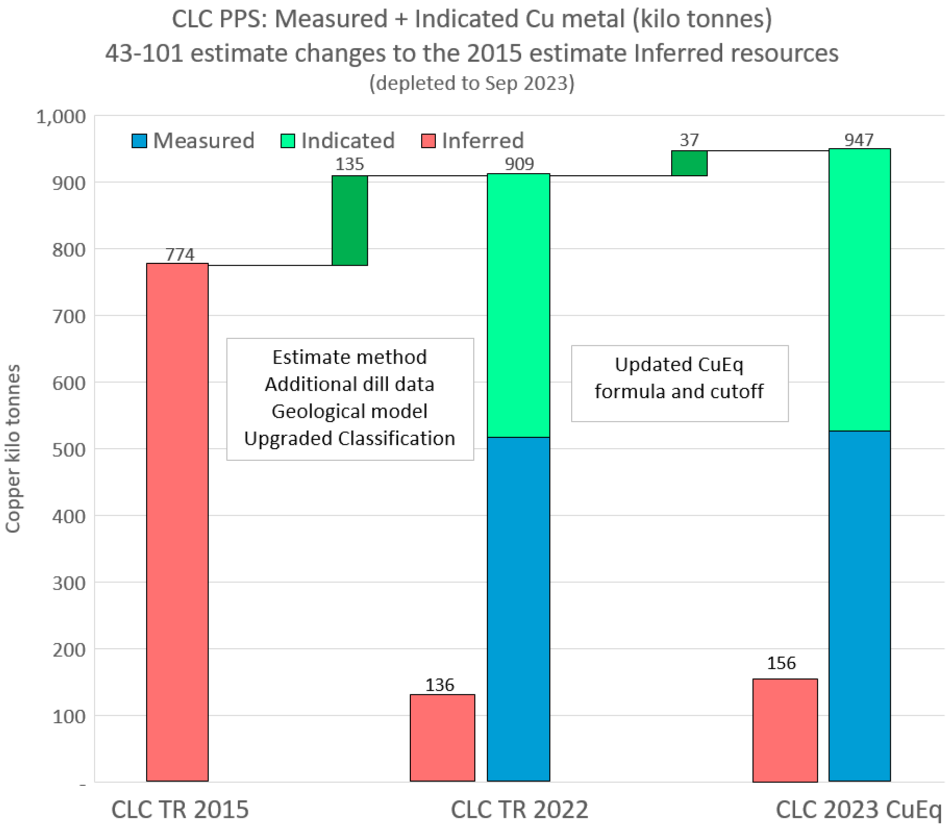
June 2015 Mineral Resource statement reported at respective Material type cutoff grades as at end of May 2015							
Classification	**Tonnes (Mt)**	**CuEq (%)***	**Cu (%)**	**Au (g/t)**	**Pb (%)**	**Ag (g/t)**	**Zn (%)**
Polymetallic Primary Sulphides (1.0% CuEq cutoff grade*)							
Total Inferred	35.77	2.82	1.11	-	1.25	28.83	2.64
Secondary Sulphides (1.0% Cu cutoff grade)							
Total Measured	6.02	-	5.56	-	-	-	-
Total Indicated	1.03	-	5.56	-	-	-	-
Sub Total Measured and Indicated	**7.05**	**-**	**5.56**	**-**	**-**	**-**	**-**
Total Inferred	-	-	-	-	-	-	-
Gossans (1.0 g/t Au cutoff grade)							
Total Indicated	**0.95**	**-**	**-**	**1.74**	**1.80**	**41.16**	**-**
Sub Total Measured and Indicated	**0.95**	**-**	**-**	**1.74**	**1.80**	**41.16**	**-**
*Resource estimates for the Polymetallic Primary Sulphides are reported on a cutoff grade of 1.0 % **copper equivalent (CuEq)**, based upon the following formula which accounts for metal price, metallurgical recoveries and amounts payable by the smelter:							
CuEq = [Tcu% + (Zn% x 0.360) + (Pb% x 0.360) + (Ag g/t x 0.0106)]							
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability							

Table 6-2 The January 2022 CLC Mineral Resource statement depleted as at end of December 2021

Polymetallic Primary Sulphides (PPS)												
Category	**Material Type**	**Density (t/m^3)**	**Tonnes (Mt)**	**CuEq (%)***	**Cu (%)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**Cu metal (kt)**	**Pb metal (kt)**	**Zn metal (kt)**
Measured	PMS	4.19	17.13	2.88	1.27	34.82	1.43	3.29	216.9	244.5	563.5
	SMS	3.43	1.19	1.74	1.37	12.79	0.54	0.55	16.2	6.40	6.5
Total Measured		**4.13**	**18.32**	**2.81**	**1.27**	**33.39**	**1.37**	**3.11**	**233.1**	**250.9**	**569.9**
Indicated	PMS	3.98	7.93	2.74	1.12	37.74	1.52	3.23	88.5	120.6	255.9
	SMS	3.56	8.61	1.78	1.36	13.03	0.37	0.80	117.4	31.60	69.2
	Stockwork	3.07	1.38	1.70	1.19	31.14	0.56	0.67	16.5	7.70	9.2
Total Indicated		**3.69**	**17.92**	**2.20**	**1.24**	**25.36**	**0.89**	**1.87**	**222.4**	**159.9**	**334.3**
Measured + Indicated		**3.90**	**36.24**	**2.51**	**1.26**	**29.42**	**1.13**	**2.50**	**455.4**	**410.8**	**904.3**
Inferred	PMS	3.92	1.98	2.33	0.72	40.89	1.61	3.06	14.3	31.8	60.6
	SMS	3.37	0.53	1.72	1.61	4.61	0.11	0.19	8.6	0.6	1.0
	Stockwork	2.93	4.58	1.77	1.41	27.45	0.42	0.39	64.6	19.2	17.7
Total Inferred		**3.19**	**7.09**	**1.93**	**1.23**	**29.47**	**0.73**	**1.12**	**87.5**	**51.60**	**79.3**
Resource estimates for the Polymetallic Primary Sulphides are reported on a cutoff grade of 1.0 % copper equivalent (CuEq), based upon the following formula which accounts for metal price ($3.37/lb copper, $1.1/lb zinc, $0.91/lb lead and $21.23/lb silver), metallurgical recoveries and amounts payable by the smelter:											
CuEq = [Tcu% + (Zn% x 0.339) + (Pb% x 0.227) + (Ag g/t x 0.005)]											
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability											

The following waterfall chart summarizes the Mineral Resource Changes to the 2015 estimate, with all figures depleted as of September 2023 to allow for better comparisons. Inferred resources have been included in the chart for illustrative purposes only and to provide context to the comparison.

Figure 6-1 **Waterfall chart showing changes to the 2015 estimate. Inferred resources included in the chart for illustrative purposes only**



6.2.2 Mineral Reserve

Mineral Reserve updates from 2009 onwards were due to mining depletion and processing. Mining grade control and reconciliation procedures were used to improve the accuracy of Mineral Resource and Reserve estimates. The open pit amenable secondary copper Mineral Reserve was fully depleted by February 2021.

The previous Mineral Reserve estimates were all in relation to open pit mining production, as follows:

- a 2009 Annual Information Form (Inmet) quoted a Mineral Reserve at the end of 2008 as being 17.6 Mt at 6.2% Cu with 1.1 Mt Cu contained.
- a 2012 Technical Report (FQM), after 3 years of operation, declared a Mineral Reserve as at the end 2011 of 14.7 Mt at 5.5% Cu with 0.8 Mt Cu contained.
- The 2015 CLC Technical Report (FQM), after 6 years of operation, declared a Mineral Reserve, as of May 2015, of 8.3 Mt at 5.05% Cu for 0.43 Mt Cu contained.
- a 2020 Annual Information Form (FQM) quoted a small remnant Mineral Reserve at the end of 2020 as 0.14 Mt at 2.11% Cu with 0.003 Mt Cu contained.

This Technical Report is the first document in which an underground Mineral Reserve estimate is described and stated.

6.3 Production History

Production started early in 2009 and as at the end of September 2023, the following are key milestones:

- 14.42Mt of secondary sulphide Mineral Reserve has been mined and processed at an average grade of 5.31% Cu.
- There is 5Mt of Indicated Primary Polymetallic sulphide (PPS) Mineral Resource on stockpile with an average grade of 1.19% Cu, 2.21% Zn, 1.63% Pb and 29.40 g/t Ag.
- There is 2.7Mt of gossan with a gold grade of 2.58 g/t on separate stockpiles. This material is not intended as feed to the PMR Project.

Table 6-3 Operations Summary 2006 to 2021

CLC Secondary copper ore production by Year from 2009			
Year	**Ore processed (tonnes)**	**Head grade Cu (%)**	**Cathodes (tonnes)**
2009	121,232	6.25	5,527
2010	494,800	6.95	28,453
2011	776,282	6.49	42,141
2012	1,081,769	7.12	67,662
2013	1,252,544	6.26	69,305
2014	1,539,294	5.13	71,092
2015	1,499,931	5.18	70,029
2016	1,538,252	5.19	73,643
2017	1,618,634	5.08	73,664
2018	1,543,856	4.95	70,738
2019	1,353,639	4.14	48,091
2020	1,462,151	4.32	54,352
2021	141,543	2.89	4,037
Total	**14,423,927**	**5.31**	**678,734**

6.3.1 Mining Operations

Mining operations commenced in early 2009 on the open pit secondary sulphide copper operation with total waste of 220.4Mt and ore of 14.4Mt mined from 2009 to 2021 with a strip ratio of 15.3 at an average grade of 5.31% Cu with 766,005t of contained copper.

Table 6-4 Historical Annual Mine Production at CLC

MINING	UNITS		Actual 2006 - 2021
Overburden	Tonnes	Mt	198.36
Rock Waste	Tonnes	Mt	22.11
Total waste	Tonnes	Mt	220.47
Total ore	Tonnes	Mt	14.42
Mined Grade	Cu	%	5.31
Total Mined	Tonnes	Mt	234.89
Strip ratio			15.29

6.3.2 Processing Operations

The process plant for secondary sulphide copper feed started in June 2009. Nominal copper metal feed was around 6,500 tonnes per month which was achieved in the first quarter of 2012 after the initial ramp-up period. LOM cathode production from secondary sulphide reserves was 678,733 tonnes. The average copper leaching recovery over the LOM period was 90.7%.

Tailings reprocessing commenced in January 2021 and was completed in June 2023. A total of 3.47Mt of plant feed was processed with 23,072 tonnes of copper cathode produced at an average recovery of 73.9%.

Table 6-5 Processing Operations Summary for 2009 to September 2023

PROCESSING SUMMARY (LOM)	SECONDARY ORE	TAILINGS
LOM	**2009-2021**	**2021-2023**
Tonnes to plant	14,423,926	3,468,371
Cu % average	5.31%	1.04%
Cu metal feed	766,005	35,918
Average Leaching Cu recovery %	90.7%	73.3%
Cathode tonnes	678,734	23,072

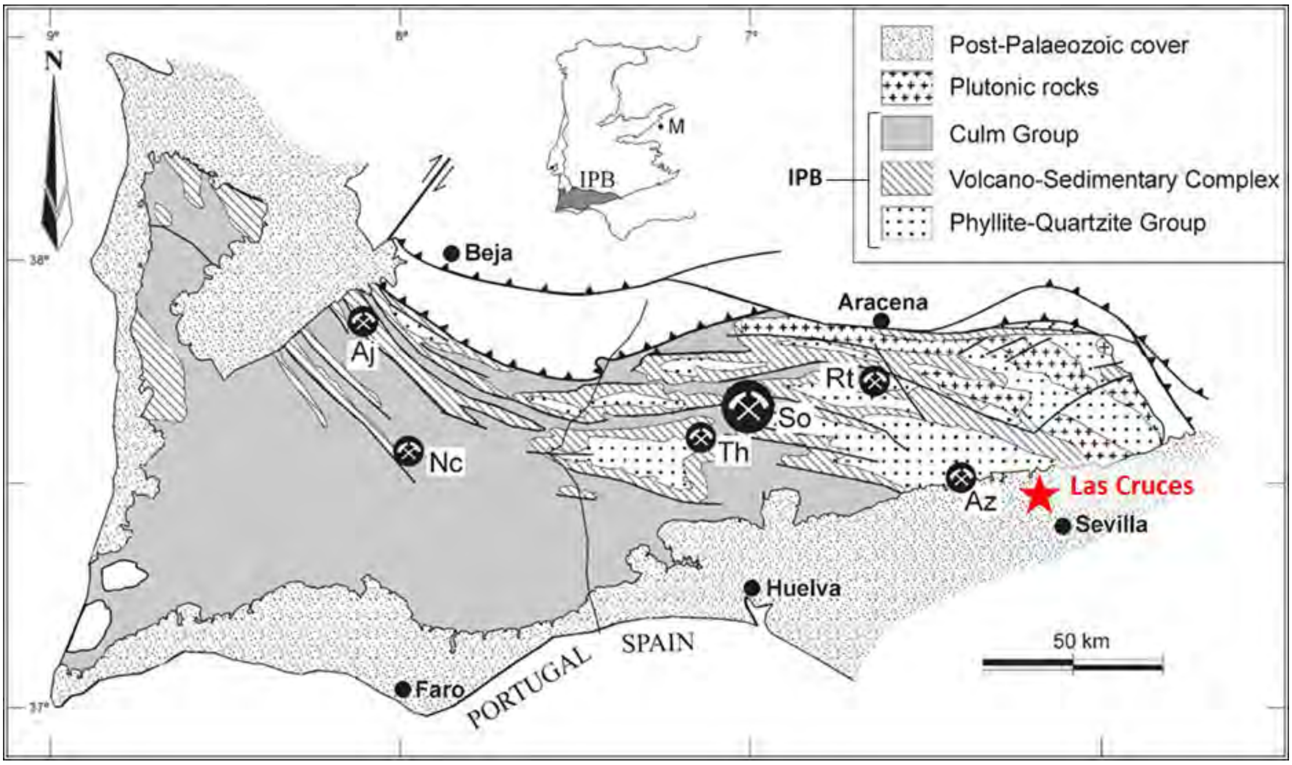
ITEM 7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Iberian Pyrite Belt (IPB) forms the southernmost terrain of the Variscan Belt of Iberia which is one of three key areas making up the South Portuguese Zone. The IPB has a relatively simple geological sequence that includes between 1,000 to 5,000 m of late Palaeozoic rocks. It is around 300 km long and up to 80 km wide, extends from southern Portugal into southern Spain and hosts over 100 volcanogenic massive sulphide (VHMS) deposits. These VHMS sulphide deposits are hosted by volcanic and sedimentary rocks deposited in the late Devonian and early Carboniferous periods, the extents of which are illustrated in Figure 7-1.

The CLC deposit is located near the eastern margin of the IPB and is overlain by the young Neogene-Quaternary (Post-Palaeozoic) Guadalquivir basin sediments.

Figure 7-1 CLC in relation to the prevailing Iberian Pyrite Belt (IPB) Geology (modified from González et all, 2006)



The stratigraphic succession of the IPB consists of three groups: the Phyllite-Quartzite (PQ) Group, the Volcano-Sedimentary (VS) Complex and the Culm Group, a post-volcanic succession (Figure 7-2).

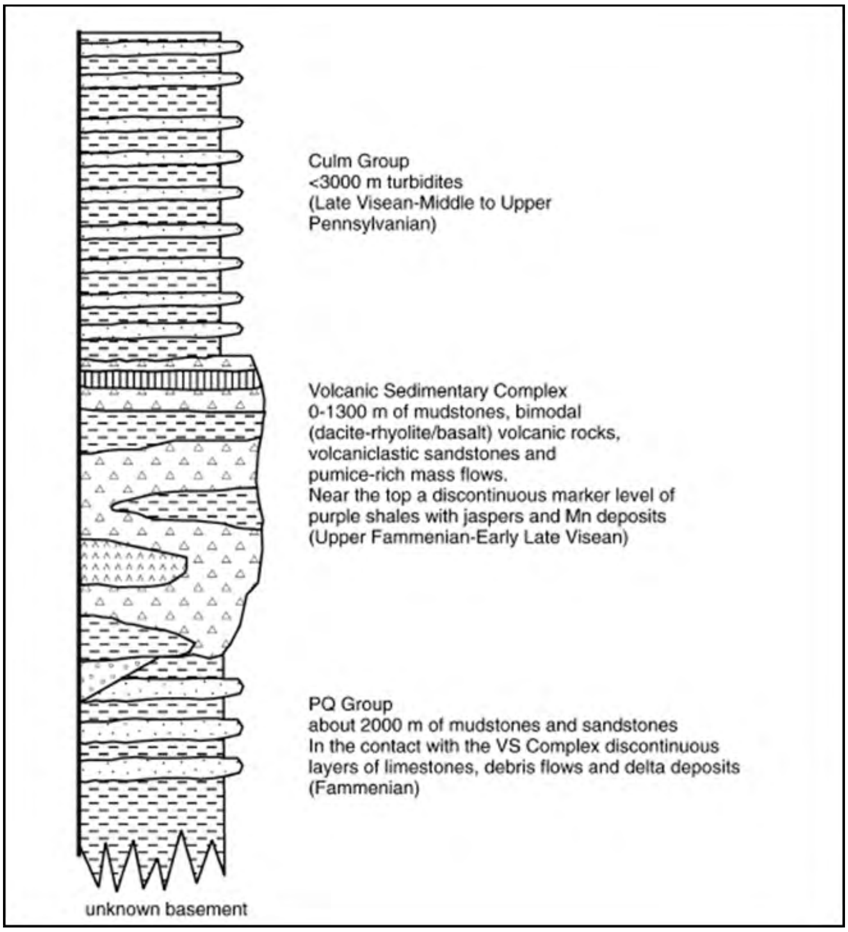
The PQ Group has the oldest rocks and forms the base of the IPB (late Devonian) and consists of a monotonous sequence of alternating mudstone and sandstone, characteristic of stable epicontinental platforms.

The VS Complex overlies the PQ Group and hosts the CLC mineralization, marking the onset of a pull-apart basin with irregular deepening. The VS complex consists of mafic to felsic volcanics interbedded with mudstone and chemical sediments (Late Famennian to Visean of ~350 Ma).

The boundary between the PQ group and the VS Complex is often characterised by upper quartz-arenite layers and the first occurrence of volcanic rocks. In some areas, this transition has reef carbonate lenses, near-shore sandstone bars and sedimentary gravity flows which indicate irregular subsidence and the formation of half graben sub-basins.

The VS Complex is overlain by the Culm Group which is comprised of shale, litharenite and rare conglomerates with turbiditic features. The Culm Group is up to 3,000 m thick (Late Visean to Middle–Upper Pennsylvanian) and represents a synorogenic foreland flysch related to the Variscan collision and tectonic inversion.

Figure 7-2 General Stratigraphic Column of the IPB (Tornos F. et al, 2000)



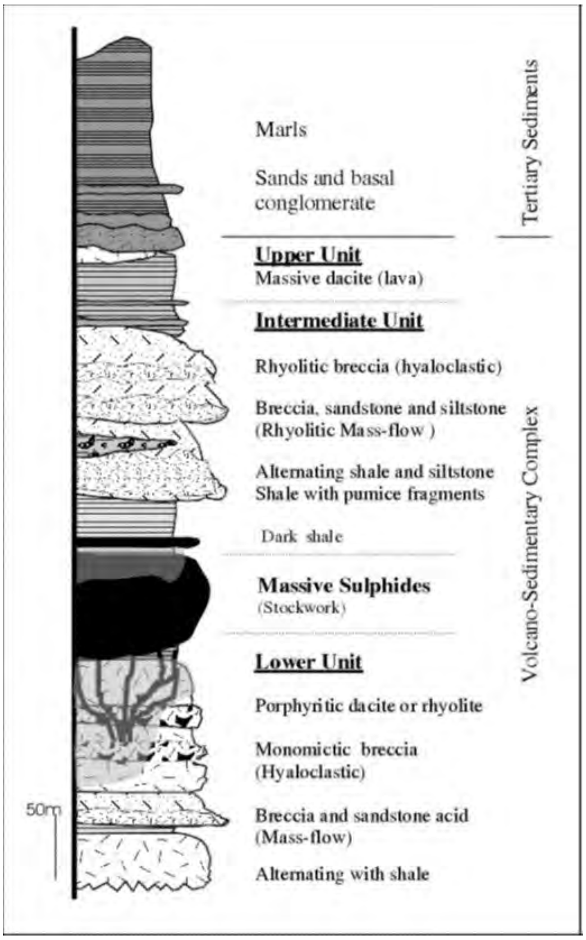
Structure is dominated by southwest to south verging thrust and fold structures (Late Visean–Late Moscovian age of ~345Ma – 315Ma) that formed imbricate fans of about 150 km long and 10 to 15 km thick. The resulting stacked tectonic units are less than two km thick. Recent models emphasize the importance of a long-lasting deformation, with major thrusting synchronous with left lateral wrenching. The location of major structures is influenced by existing earlier extensional or strike-slip faults and major rheologic contrasts between shale and more competent rocks. Some of these faults were re-activated during Late Variscan to Alpine times as strike slip structures coincident with the mafic dike intrusions. The Variscan regional metamorphic grade is very low to low grade, up to lower greenschist facies.

7.2 Local and Property Geology

The CLC deposit is overlain by 150 to 200 m of Cenozoic Guadalquivir Basin sediments which, towards the base, hosts a confined porous aquifer (1 to 3 m thick) known as the Niebla-Posadas Aquifer. The aquifer has a ph of ~ 8.0, low salinity and a temperature of 18-31 °C. The host Paleozoic sequence underlies these basin sediments and has an east-west strike, dipping between 20° to 45° to the north. The Paleozoic rocks also contain pressurised water which is more saline and with temperatures between 22 to 43 °C. No methane has been intersected in either of the aquifers.

The CLC deposit sequence has four volcano-sedimentary units (Figure 7-3). The massive sulphide mineralization is located along the contact between a dacite and black shale with pyrite, pyrrhotite and arsenopyrite. Intense structural deformation is present close to the massive sulphide horizons. The massive sulphide lens is situated in the same stratigraphic interval as the nearby Aznalcollar and Los Frailes deposits.

Figure 7-3 Summary Lithology at CLC



Recent chronostratigraphic (Figure 7-4) and geochemical work (Figure 7-5) on a selection of diamond drill holes has used the texture and structure of volcanic rocks to improve the understanding of local property geology.

Most of the petrology was based upon the downhole XRF chemostratigraphy data which has provided chemical criteria for logging volcanic and sedimentary strata. Additionally, the use of geochronological dating (paleontological and U-Pb radiometric) was used to define the stratigraphic sequences for the lower and upper plate's hangingwall and footwall. Results support thrust faults as a key feature affecting local stratigraphy and geology.

Figure 7-4 Chronostratigraphy of Borehole CR-726 in relation to the updated 3D geology model.

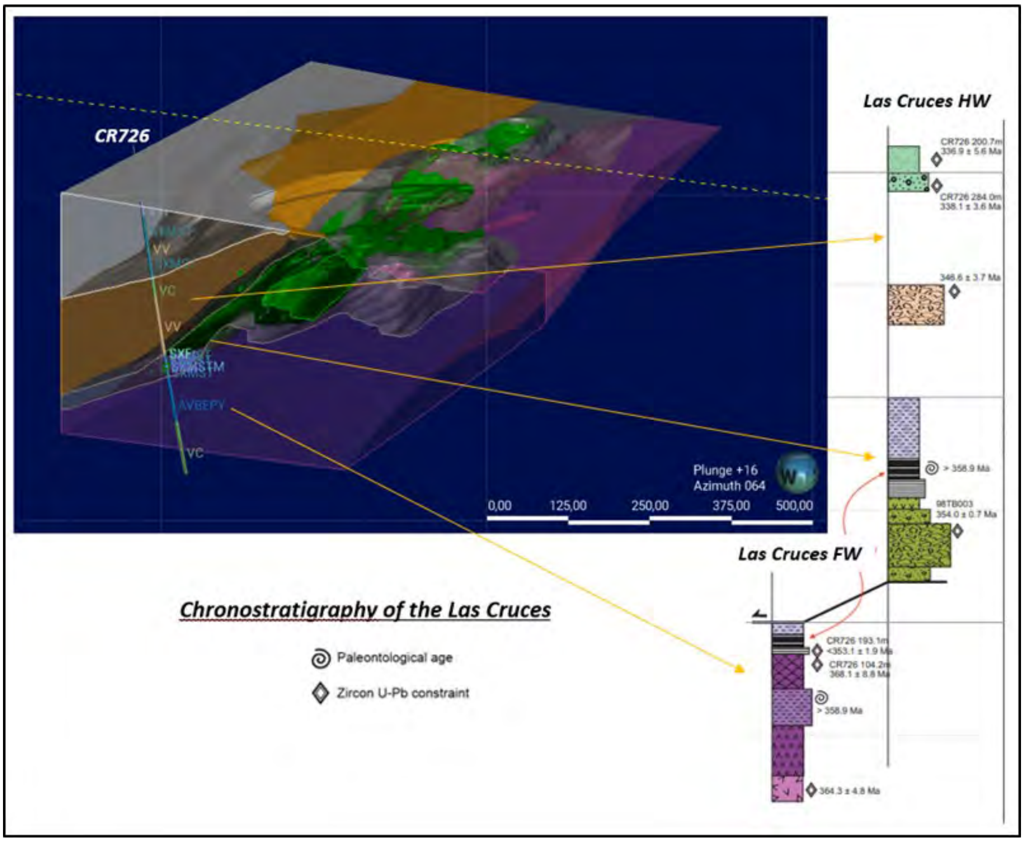
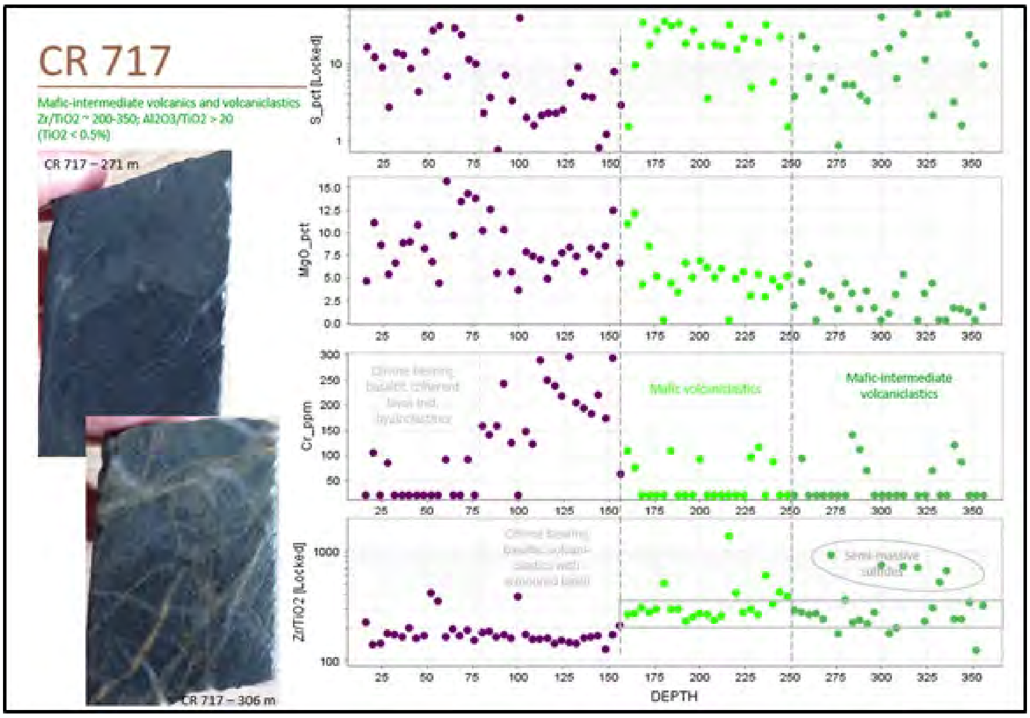


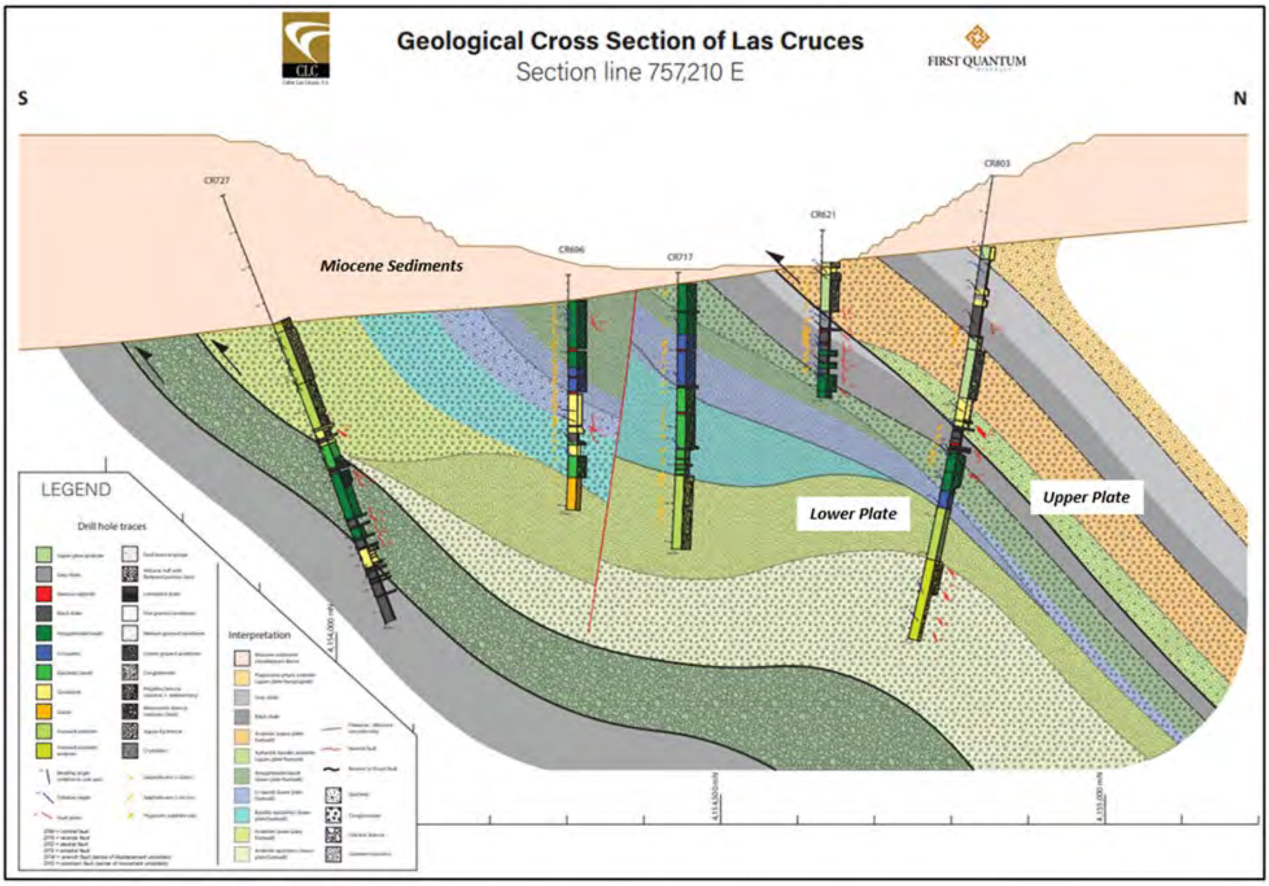
Figure 7-5 Example of downhole XRF Chemostratigraphy in borehole CR-717



Two main thrust fault plates control the local geology of the deposit with the lower plate hosting the massive sulphides mineralization. The hangingwall of the upper plate (Figure 7-6) is comprised of volcanic andesite and epiclastic andesite (plagioclase-phyric) followed by a package of more than 200 m of siltstones. The footwall has a conglomerate textured andesite and an aphanitic basaltic andesite.

The lower plate, from hangingwall to footwall, has a black shale (which hosts inter-layered mineralization) followed by a thick package of variably textured basalt and an underlying coherent andesite with some breccia. To the south, two reverse faults dipping north clearly differentiate between the lower plates andesite and breccia and the grey-black shales.

Figure 7-6 A south to north geological cross section of CLC's local geology.



A result of this work is that there is now greater support for the main thrust fault with improved definition of upper and lower plate stratigraphy. This is key to understanding the controls on mineralization for exploration targeting of other prospective zones. The inter-layered mineralization of the lower plate's black shales has the entire footwall as a potential host to stockwork mineralization.

7.3 Primary Polymetallic Sulphide (PPS) Mineralization

PPS mineralization comprises two mineralization domains, primary massive sulphides and semi-massive primary sulphides or stockwork.

The primary massive sulphide occurs as a single 30 to 130 m thick lens with around 1,000 m of strike and a 500 m down dip length. Primary mineralization underlies secondary mineralization and is open at depth and to the northeast. Mineralization thickens to the west but is truncated by a north-south striking fault. The PPS massive sulphide mineralisation is underlain by a zone of mineralised stockworks.

The primary massive sulphides have an upper Zn-rich zone and a lower Cu-rich zone interbedded with barren massive pyrite. Sulphides are dominated by pyrite with chalcopyrite, sphalerite, and galena. The polymetallic zones contain centimetre-thick bands of alternating pyrite + chalcopyrite and sphalerite + chalcopyrite + galena. Deformation is responsible for the formation of alternating layers of sphalerite-chalcopyrite-galena and pyrite. The sulphides other than pyrite show widespread recrystallization to a granoblastic assemblage hosting abundant minute subhedral pyrite.

The massive sulphides show conspicuous layering with few clear sedimentary features. Features include banding and graded bedding, especially in the interbedded shale and chert. However, the widespread late recrystallization has masked most primary features.

While the uppermost shale-hosted massive sulphides are likely to be exhalative, the lowermost orebody formed mainly by replacement. Evidence of replacement includes the presence of relicts of variably altered dacite within the massive sulphides and a gradual contact with the underlying stockwork mineralised zone. No oxidation has been observed in the hanging wall of the massive sulphides, thus indicating that seafloor oxidation processes were minimal and that the massive sulphides were originally isolated from an oxidising open ocean.

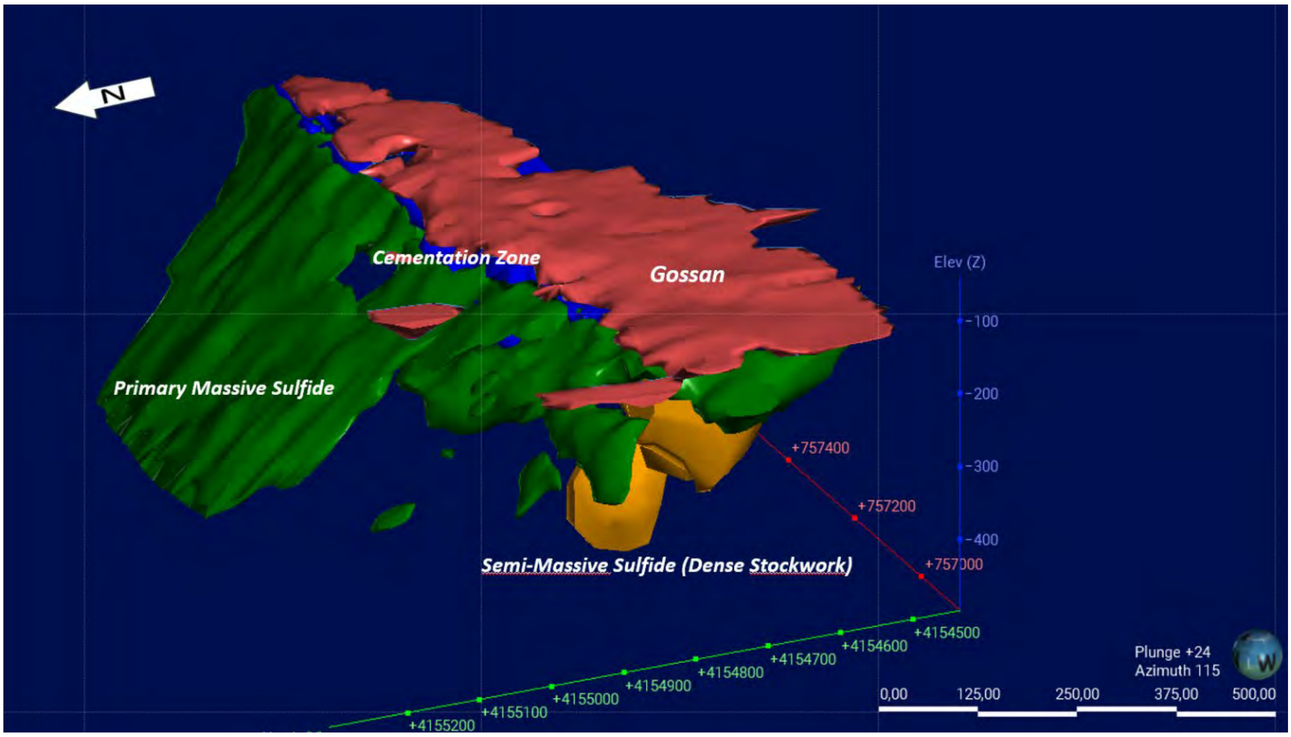
The PPS massive sulphide mineralization is underlain by a mineralised stockwork and related hydrothermal zone which extends some 200 m beneath the massive sulphides and tends to be copper rich with localized zinc. The stockwork is hosted in hyaloclastic dacite and shale having chlorite ± quartz alteration. The stockwork veins are between 0.5 to 2 cm thick and are folded or sub-parallel to the dominant foliation.

ITEM 8 DEPOSIT TYPES

The CLC deposit is a Volcanogenic Massive Sulphide (VMS) deposit which has resulted from precipitation of metals from hydrothermal fluids in a volcanically active sub-marine area. The metals precipitate as sulphides inter-layered with the volcanic sediments. Consequently, CLC's mineralization is hosted by black shales, sedimentary and felsic volcaniclastic rocks.

The deposit is characterized by a tabular dipping massive sulphide body underlain by a stockwork vein system. The deposit has been folded and faulted, resulting in dislocated mineralised volumes. Hydrothermal alteration is typically chloritic/quartz sericitic and is restricted to the stockwork zones. CLC deposit shows zonation where copper rich volumes are flanked by zinc and lead mineralization. Lead, zinc and silver tend to concentrate in unevenly distributed polymetallic zones. Minor gold is noted in both polymetallic and copper rich areas. Pyrite is dispersed throughout.

Figure 8-1 A north-south isometric view of CLC mineralization



At CLC, post-deposition tectonism uplifted the deposit, exposing it to erosion and weathering with partial oxidation of some of the primary sulphides. An uppermost copper-leached gossan resulted with some immobile gold and silver and overlies the cementation zone of secondary sulphide mineralization (Figure 8-1).

The 3D modelling of the deposit and its different mineralization domains has taken into consideration the VMS nature of the exhalative and replacement processes for the mineralization for massive sulphides zones, together with hydrothermal alteration dominant in the stockwork areas. These have been overprinted and further divided into detailed mineralization style domains with the addition of the supergene alteration process, namely oxidation cap (gossan), supergene - cementation zone (secondary copper sulphides) and hypogene (primary copper sulphides, PMS, SMS and stockwork).

The interpretation of the deposit's geometry, geology, alteration, structure and mineralization domains honours the sample data included in the geological database.

ITEM 9 EXPLORATION

9.1 Introduction

Riomin Exploraciones, S.A. (RTZ) discovered the CLC deposit in 1994 and continued drilling for five years to define the deposit. Thereafter, exploration was discontinued until 2013, when CLC re-initiated exploration across their mining concession and surrounds.

CLC exploration maintains three main objectives:

- To find additional mill feed for CLC by identifying Mineral Resources in the current tenement package that will positively impact upon economics and extend the mine life.
- To identify an early-stage 3rd party exploration project with upside resource development potential to provide additional medium-term feed for CLC.
- To discover a standalone deposit within the IPB that is aligned with CLC mining and processing technology for creating longer term value.

These exploration activities are divided into three spatial areas:

1. The CLC mining concession, around 300 Ha and which covers the entire project area (plant, pit, deposit footprint)
2. The near mine exploration (NME) permits in areas close to the mining concession
3. The regional exploration (RE) permits are in more distant areas but within the Iberian Pyrite Belt.

9.2 Exploration in the CLC Mining Concession

Over the last 10 years several geophysical exploration works have been completed to improve deposit knowledge, increase known mineralization, and identify other massive sulphide targets. The equipment and techniques used for data acquisition and interpretation are considered of adequate quality for guidance of the exploration works, with no known bias or effect to the mineral resources estimate included in this report.

A 2D seismic survey was completed in 2017 for a north to south profile across the extents of the CLC deposit. The survey was done to identify reflectors related to massive sulphide bodies or dense stockwork. In addition, seismic data has been used to assist with detailed interpretation of local structural geology.

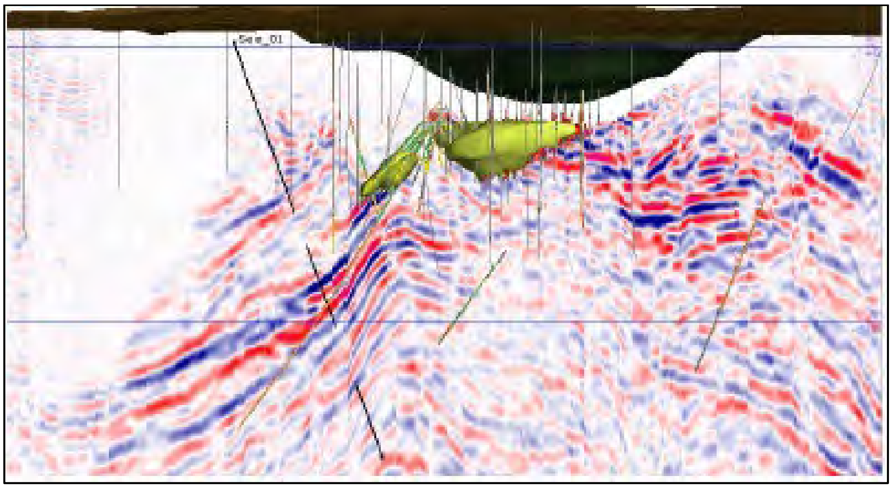
Other geophysical techniques have been used to study structural geology, geological model (petrophysics) and hydrogeological aspects. The following geophysical acquisitions are noted:

- An audiomagnetotelluric survey together with a 3D model was completed in 2015. Data from this survey was used in the interpretation and modeling of a structural geology model.
- Airborne electromagnetic survey conducted in 2017, covering the mining concession and NME area. The survey was carried out with an average penetration of up to 250 m below surface which provided data relevant to the deeper deposit.
- A gravimetric campaign was conducted to define continuity of massive sulphide mineralization dipping to the north.
- Down hole electromagnetic surveys were completed for more recent diamond drilled holes. Data was used to help refine the extents and orientation of the zones of intersected mineralization.
- A ground electromagnetic survey was completed in the time domain (Fixed Electromagnetic Loop) using 500 m by 500 m loops and a high current above 20A. The method allows for greater depth

penetration than an airborne survey and enables measurements from different frequencies, including very low frequencies as collected by SQUID antenna. The survey provided supplementary data to existing data sets.

Recent activities during 2023, include the drilling of SEIS_01 diamond hole (Figure 9-1) to a depth of 802.6 m to verify a set of seismic reflectors dipping similarly to and extending from the known geological continuity of the deposit.

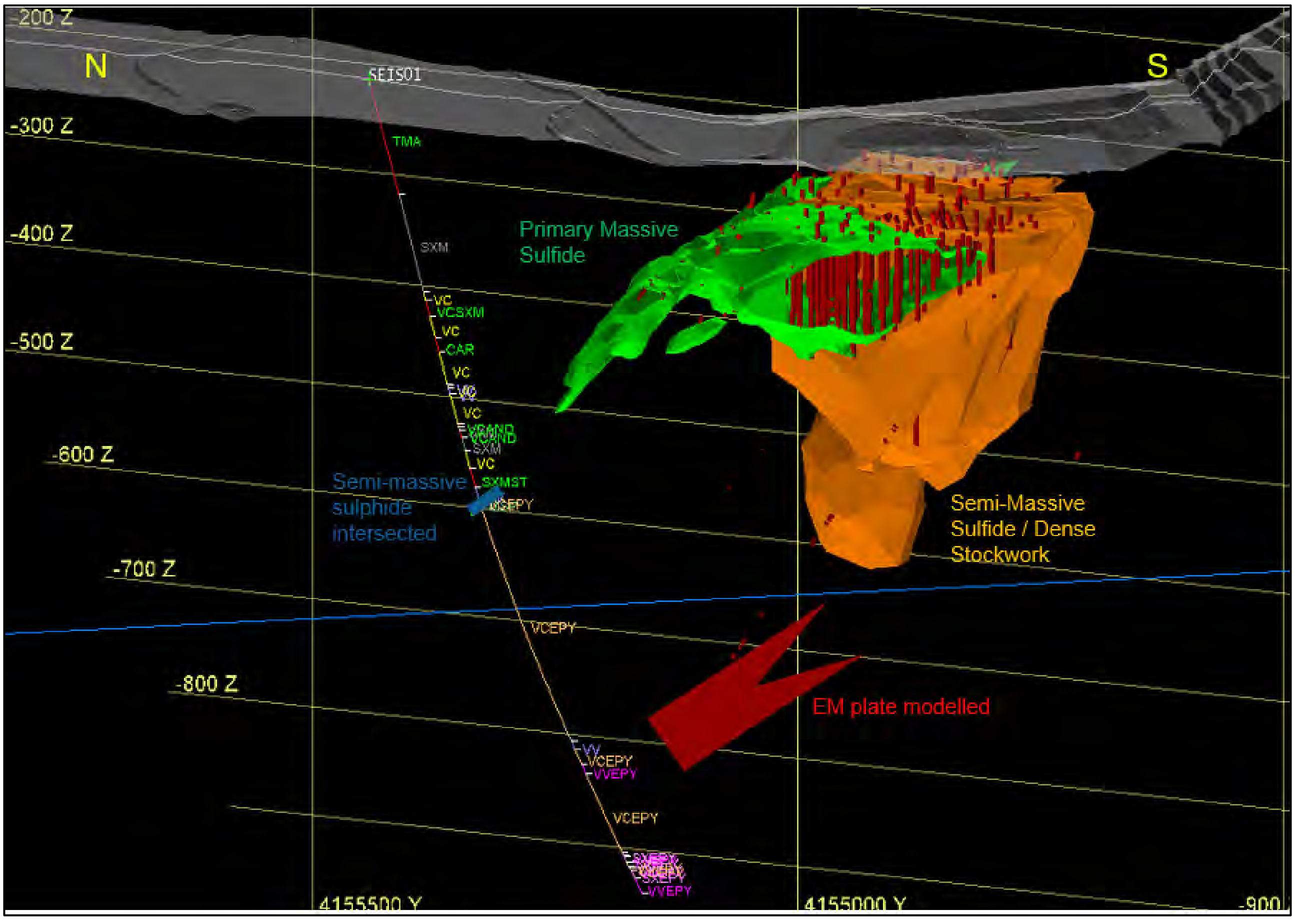
Figure 9-1 A north south cross section showing the diamond hole, SEIS_01, against seismic reflector amplitudes. The yellow ore shell represents the current PMS modelled mineralised volume.



At 408 m to 412 m SEIS_01 intersected semi-massive sulphide with an average copper grade of 2.45%. The mineralization occurs within a volcanic breccia typical of the footwall to the massive sulphide. While the intersected copper grades are good, the width of mineralization is narrower than expected from the strength of the seismic profile’s reflectors.

In addition, downhole geophysics, using a sonic probe and collection of electromagnetic data was completed to compare results to the 2D seismic profile (Figure 9-1 and Figure 9-2). Results suggested that the mineralised intercept was not related to a major conductive plate, however, data did suggest an off-hole anomaly at 690 m depths to the south-southeast. The anomaly will be investigated further as a potential exploration target.

Figure 9-2 **The location of the SEIS-01 diamond hole relative to the CLC PMS deposit. The intersection of sulphide is shown around 400 metres (blue) together with the EM plate anomaly modelled at 690m (red).**

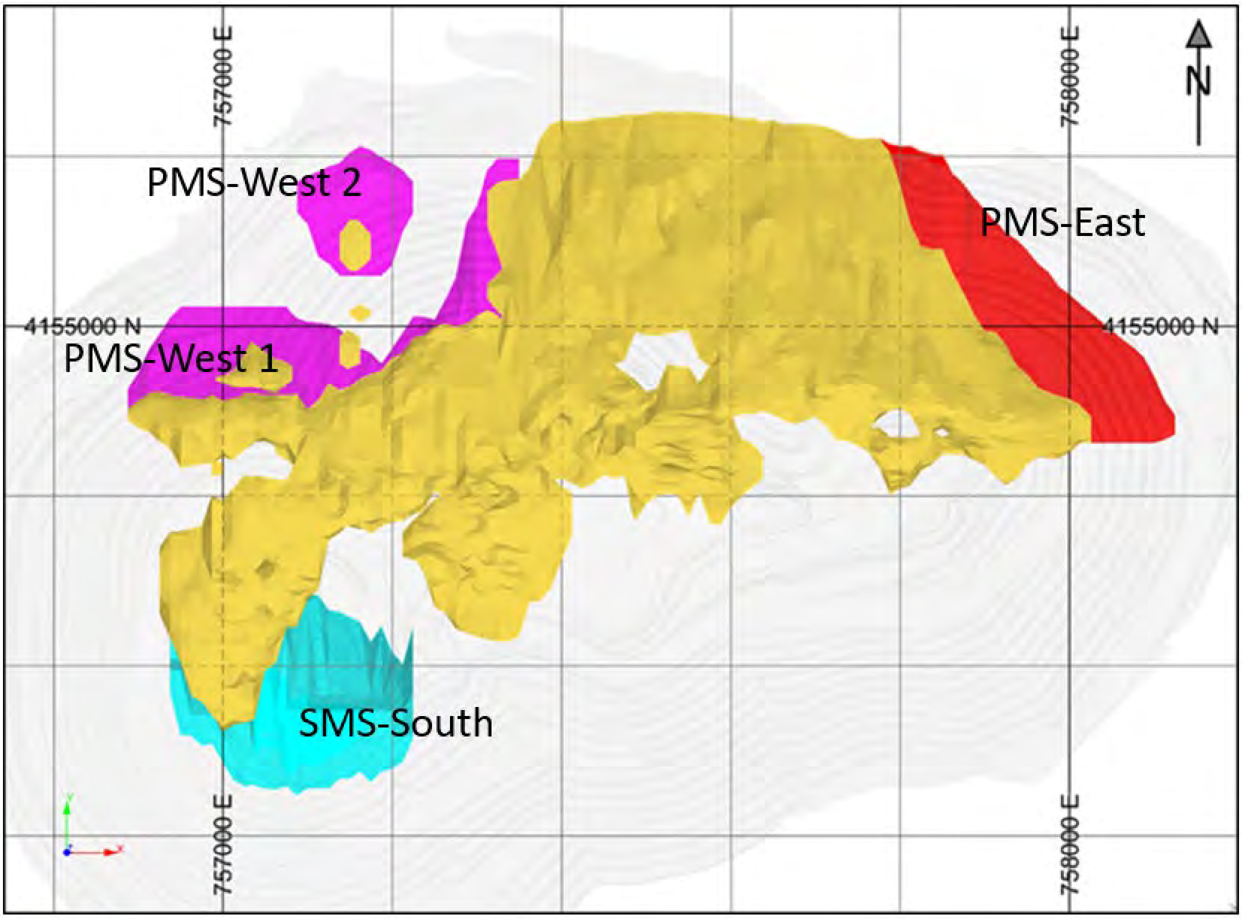


9.2.1 Exploration Target Mineralization Potential

The declared Inferred Mineral Resources are most likely to be converted to reserves with additional infill drilling and sampling. Inferred Mineral Resources comprise some 9.4 million tonnes with a Cu_{eq} grade of 1.66%. Of this, stockwork mineralization comprises 69% of the Inferred material, with 23% been massive sulphide and 8% been semi-massive sulphide.

Apart from the Inferred Mineral Resources an additional four target exploration mineralization areas were identified. These areas were coded into the block model and are the subject of further exploration work for increasing CLC Mineral Resources. The four areas have been named PMS West 1, PMS West 2, PMS East, and SMS South.

Figure 9-3 A 3D view of known CLC PMS mineralization together with the identified target exploration mineralization areas.

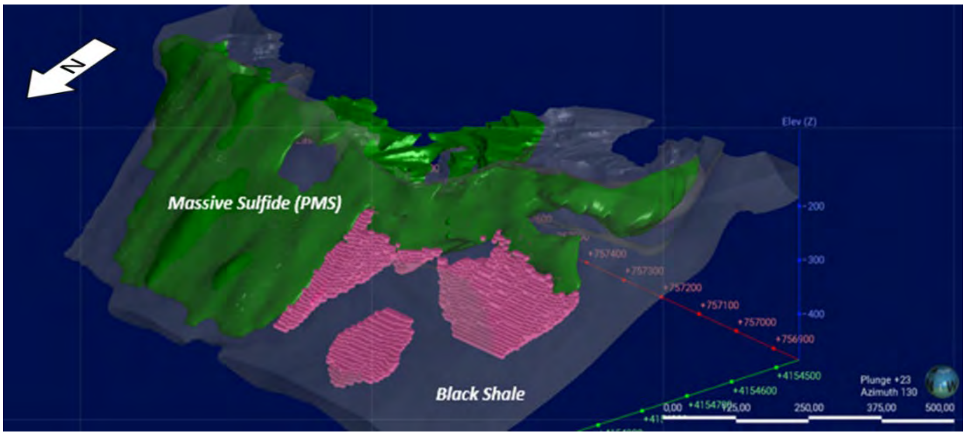


9.2.2 PMS West 1 and 2

PMS extension, in the west (Figure 9-3) of the deposit is supported by:

- The area been immediately adjacent to well-developed mineralization with limited geological evidence for closure
- Limited wide spaced drilling having indications for mineralization
- Boreholes CR803 and SEIS-01 have demonstrated some degree of mineralization continuity. Intercept widths are narrow but with reasonable grades and are therefore worth pursuing.
- Additionally, CR803 identified an EM anomaly and a conductive plate having an azimuth and dip consistent with local geology.

Figure 9-4 **Potential target mineralization extension of PMS West 1 and 2 areas (pink) relative to the current Mineral Resource PMS (green) and looking to the southwest.**

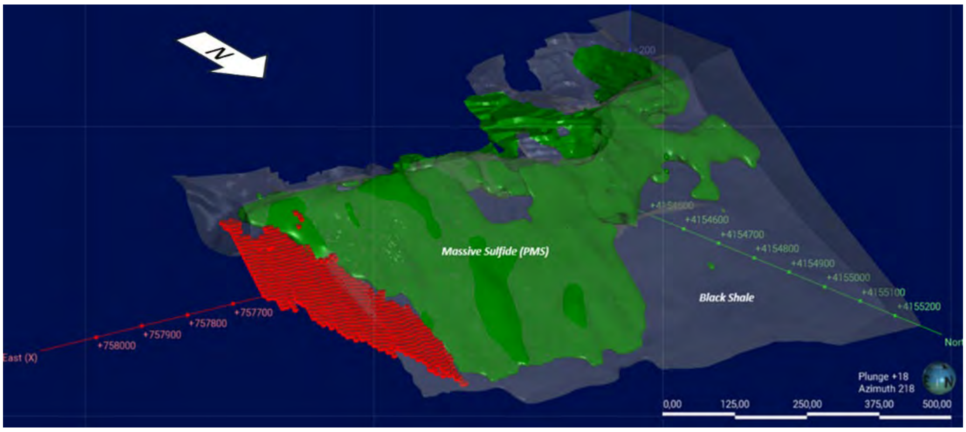


As per recent geological work, structural complexity is likely to be introducing variations to the sulphide thickness and position. A surface drilling campaign with a closer grid would be necessary to define this potential further. Alternatively, drilling from underground development may also be considered.

9.2.3 PMS NE

The primary massive sulphide to the northeast (Figure 9-5) of the deposit is characterized by thinning of the mineralised widths together with the appearance of a dense footwall stockwork dominated by pyrite mineralization. CR-133 intersects some copper mineralization in this area. Further drilling in this area is likely to intersect low grade mineralization with a risk of poorer continuity. The area is otherwise open with no drilling to close mineralization off.

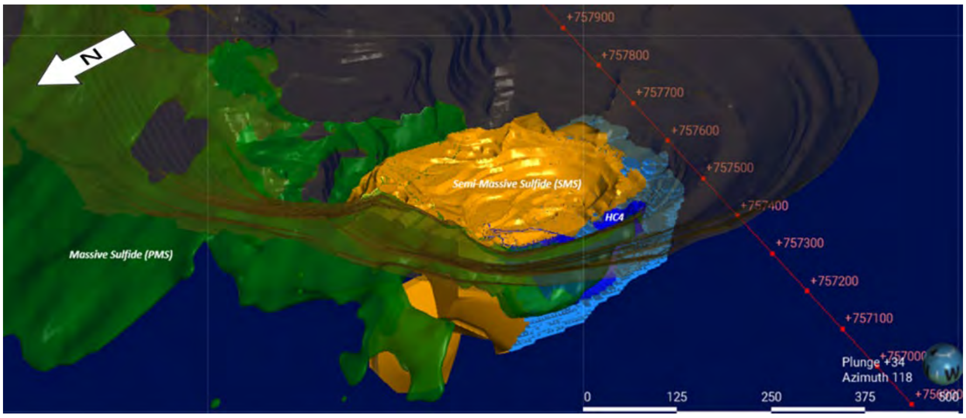
Figure 9-5 **Potential target mineralization extension of PMS NE (red) relative to known PMS (green) mineralization looking to the south southeast.**



9.2.4 SMS South

The semi-massive stockwork (SMS) is rich in copper mineralization and has been difficult to investigate with the previously active open pit mining. With part of the pit now filled, it is becoming feasible to access this area with infill drilling.

Figure 9-6 Potential target mineralization extension zone of SMS South (sky-blue) compared with Mineral Resource SMS (orange), PMS (green), HC4 (blue) and final pit.

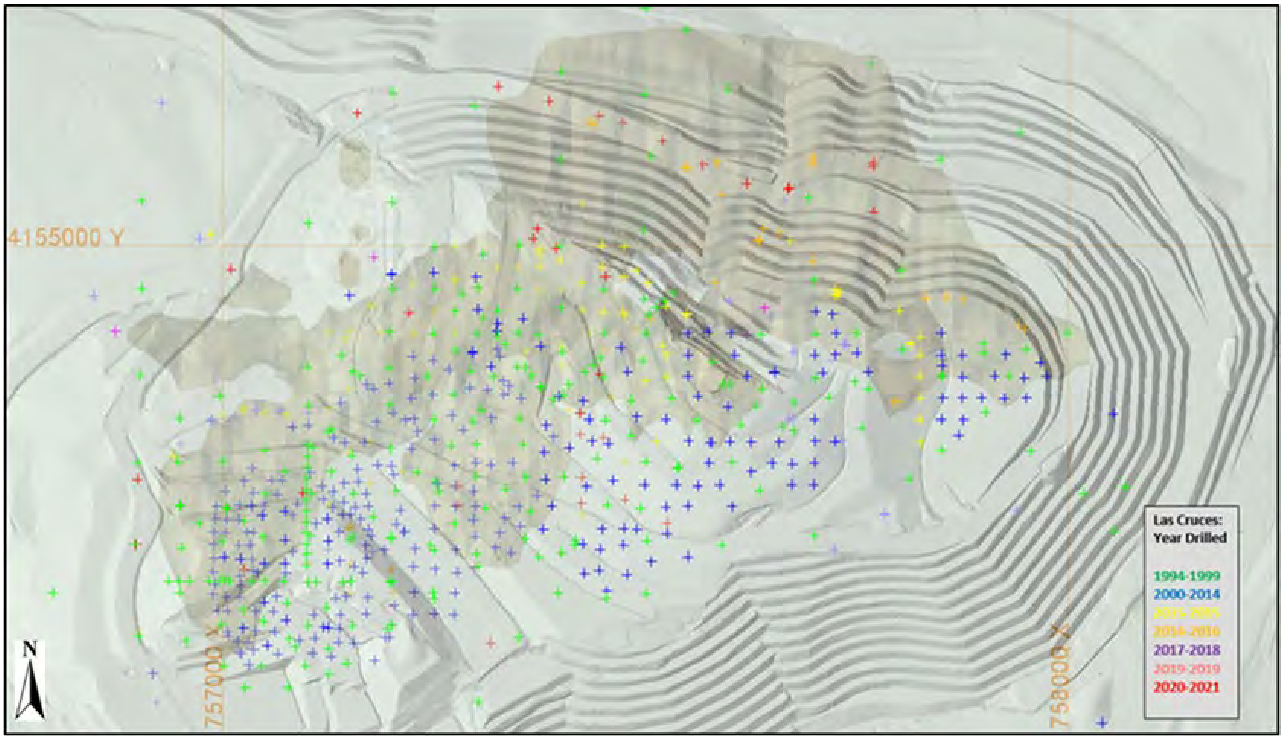


ITEM 10 DRILLING

Since the 2022 CLC Technical Report, a single exploration hole (SEIS-01) was drilled for targeting and guiding further drilling. The data derived from this hole does not warrant an update of the existing Mineral Resources estimates. The disclosed Mineral Resource estimates have used a total of 807 diamond drilled holes (Figure 10-1) for 159,100 m of core. These holes are comprised of:

- 277 diamond holes (82,352m) drilled between 1994 and 1999 by Riomin for defining mineralization extents.
- 34 diamond holes were drilled by CLC between 2000 and 2007 and were used for defining deposit detail and western extents. Six holes were included for condemnation.
- Between 2010 and 2014 an additional 261 holes were drilled as infill to establish the secondary sulphide limits.
- In 2015 a further 99 holes were drilled for added definition across the eastern portion of the secondary sulphide mineralization and to delineate the likely extents of the primary sulphide mineralization.
- An additional 130 holes were drilled between 2016 and 2021 as infill and to delimit the PPS mineralization extents. Four of these holes were for hydrogeological purposes.
- In 2018, an exploration adit was developed to the east from the northern slope of the open pit. The adit was used to provide additional drilling access to PPS mineralization. 12 diamond drilled holes for 1,280 m (391 samples) were drilled from this adit.

Figure 10-1 A plan view of the diamond drilled collars per year.



Drill grid spacing started at around 100 m and was progressively infilled to a spacing of 50 m and then 35 m for detail. From 2010 onwards, grid spacing was further reduced to 12 m across key zones of mineralization.

As far as practicable, holes were drilled to ensure highest angles of intersection with zones of mineralization. More than 50 percent of diamond holes were drilled vertically with the remainder drilled at an angle in multiple directions to test mineralization continuity and structural orientations.

Select information from blast hole sampling was used in the Mineral Resource estimates to guide delineation of mineralization. Blast hole data provided valuable short-range information for validating variogram models used during ordinary kriging.

Drillhole hole collar coordinates were surveyed by a professional surveyor using a theodolite and from 2011, a differential GPS. All diamond holes were surveyed down the hole using a multi-shot or single-shot instrument. Only minor downhole deviations were noted.

88% of diamond holes used a PQ core diameter (85 mm) and 12% used HQ core diameter (64 mm). Core recovery was good with 75% of drilled metres with recoveries greater than 90%. The relatively low number of poor recovery samples have an irregular spatial distribution and negligible risk to the estimate.

Drilled core was logged for lithology, structure and mineralization with core photographs available for all core from 2011.

Diamond core samples were taken at one metre intervals according to geology and mineralization breaks. Core samples submitted to the analytical laboratory were either full core or cut into halves or quarters using a diamond saw.

There are no known additional factors that could impact the accuracy and reliability of the drilling data and its results.

ITEM 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Overview

Diamond drilled core sampling, preparation and analysis was completed under supervision of the QP and lead geologists. Reputable laboratories were used for sample preparation and analysis. Transportation and sample handling was completed under proper supervision with sample storage in secure locked facilities. Quality assurance and Quality control (QAQC) has been practiced for the duration of the diamond drilling. All drill data is stored in a secure SQL database managed by Datashed front-end software. Checks between historical laboratory reports and stored data has verified that the original data is correctly replicated in the database.

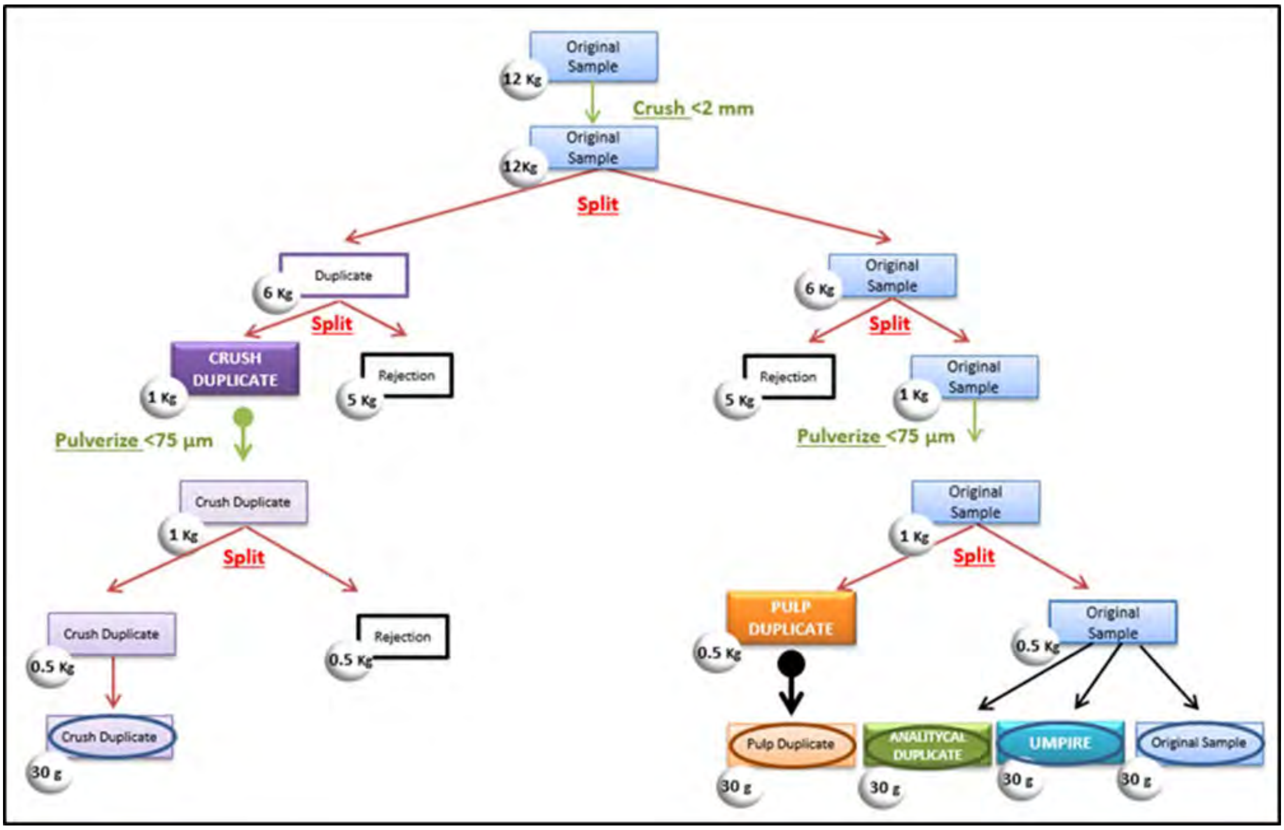
The QP, Mr Carmelo Gomez Dominguez, is not aware of any risks to the sample preparation, security, analytical procedures and results, and have verified that they are representative of the prevailing styles of mineralization. The sample data is believed suitable for use in this Mineral Resource estimate update.

11.2 Sample Preparation

Diamond drilled core of PPS mineralization is mostly HQ diameter with sample lengths between 1.5 to 2.0 m. Drilled core was transported from the drill site to the secure logging, sampling and storage facilities where resident geologists logged and marked core for sampling. Sample sheets were used to assign unique sample numbers. Bar coded sample numbering was introduced in recent years. The numbered and cut diamond core was bagged, sealed and then despatched to the preparation laboratory for crushing, splitting and pulverisation. Sample sheets were signed on delivery and receipt at each despatch and delivery point.

In summary, samples were prepared by crushing to 70% passing minus 2 mm size and then split twice into a 1 kg sample for pulverisation to 85% passing 75 microns. A 50 g pulp was split for digest and analysis. Sample preparation (Figure 11-1) included blind inserts of QAQC samples at regular intervals. CLC inserted a standard every 15 samples, a blank, a coarse crush and a pulp duplicate were inserted every 20 samples.

Figure 11-1 Workflow of sample preparation and QAQC insert process



11.3 Sample Analysis

Sample pulp aliquots were exposed to a four-acid digest followed by an Inductively Coupled Plasma (ICP-AES) analysis. A 30 g pulp sample was split and used to analyse for copper, lead, zinc and silver. Sulphur was analysed using the Leco furnace procedure. From diamond hole CR317 onwards, CLC introduced analysis for leachable copper which uses a single acid digest and an ICP-AES finish. The laboratories used that are listed below; are independent of FQM other than the AGQ-Mining laboratory, which whilst located on the CLC site, is managed independently of the issuer.

1. Anamet Laboratory in Avonmouth, England – holes CR001 to CR256, an accredited and independently managed (Anamet services) laboratory.
2. OMAC Laboratory based in Loughrea, Ireland - holes CR257 to CR277, an accredited, independently managed (OMAC Laboratories Ltd) laboratory.
3. ALS Geochemistry laboratory in Vancouver, Canada and Lima, Peru with sample preparation completed in Seville, Spain – holes CR278 to CR578, each accredited and independently managed (ALS) laboratories
4. AGQ-Mining. Labs and technological services AGQ – CLC on-mine laboratory, a non-accredited and independently managed by AGQ.
5. ALS Geochemistry laboratory in Vancouver, Canada and Lima, Peru with sample preparation completed in Seville, Spain – holes CR578 to CR766, each accredited and independently managed (ALS) laboratories.
6. SGS Geochemistry laboratory in Vancouver, Canada and ALS Geochemistry laboratory in Vancouver, Canada and Lima, Peru with sample preparation completed in Seville, Spain – holes CR766 to CR815, each accredited and independently managed (ALS) laboratories.

In addition, dry bulk density measurements were completed for half of the samples taken from mineralised zones and their host lithologies. Core was weighed in air and water and density calculated by dividing the weight in air by the difference between weight in air and weight in water. The impact of moisture and pore space on density was investigated and found to have negligible impact. The resulting density data has good spatial coverage and with quality control measurements of a standard sample. Dry bulk density data is believed representative of the prevailing lithologies and was used to estimate density for this Mineral Resource estimate.

11.4 Quality Assurance and Quality Control

Quality assurance and Quality control (QAQC) has been practiced for the duration of diamond drilling with several umpire laboratory checks completed. Established international (Anamet Inc.) and local laboratories (OMAC, AGQ and ALS laboratories) were employed for sample preparation and analysis, with resulting analytical data electronically uploaded into secure database storage systems. All laboratories are accredited in compliance with the International Standard ISO/IEC 17025:2005 2nd Edition. A detailed QAQC Study was completed by independent Golder Associates consultants in November 2013 for drillhole sampling between 2011 and 2012. The report details the sampling and QAQC practices as well as the QAQC results. Coarse crush and pulp duplicates were found to have acceptable to excellent precision. The submitted standards, or certified reference materials (CRM's), performed well suggesting good analytical accuracy. Umpire laboratory results also had very good precision.

Riomin (diamond drilling campaigns between 1994 to 1999), inserted two blind reference standards and field duplicates. The standards revealed limited deviation to the certified values and support accurate analysed values. The field duplicate values highlight an acceptable level of precision supporting precise analyses by minimising sampling error during homogenisation and splitting. No blank samples were submitted during the Riomin campaign for controlling contamination.

CLC diamond core sample QAQC included blind standards, duplicates, blank inserts and umpire checks. QAQC data was actively monitored for failure during the respective drilling campaigns. Sample preparation was completed by an independent off mine laboratory (ALS).

Regular laboratory audits were completed by the CLC geologists with improvements to sample preparation, digestion formula, instrument calibrations and QAQC limits and rules for rejection. Several laboratory audits were completed by the QP and site lead geologists during the different drilling campaigns. Crushing and pulverising regimes were controlled well above the minimum percent passing sizes.

The precision of sample analysis was assessed from the duplicate samples using scatter, HARD and QQ plots. Results from the most recent (2020 - 2021) infill drilling campaign's field duplicates of copper show that 80% of samples have copper and lead analysis that are within 10%. Similarly, the precision results from coarse crush and pulp duplicates highlights very good precision, with more than 90% of samples having copper and lead sample analysis within 10%.

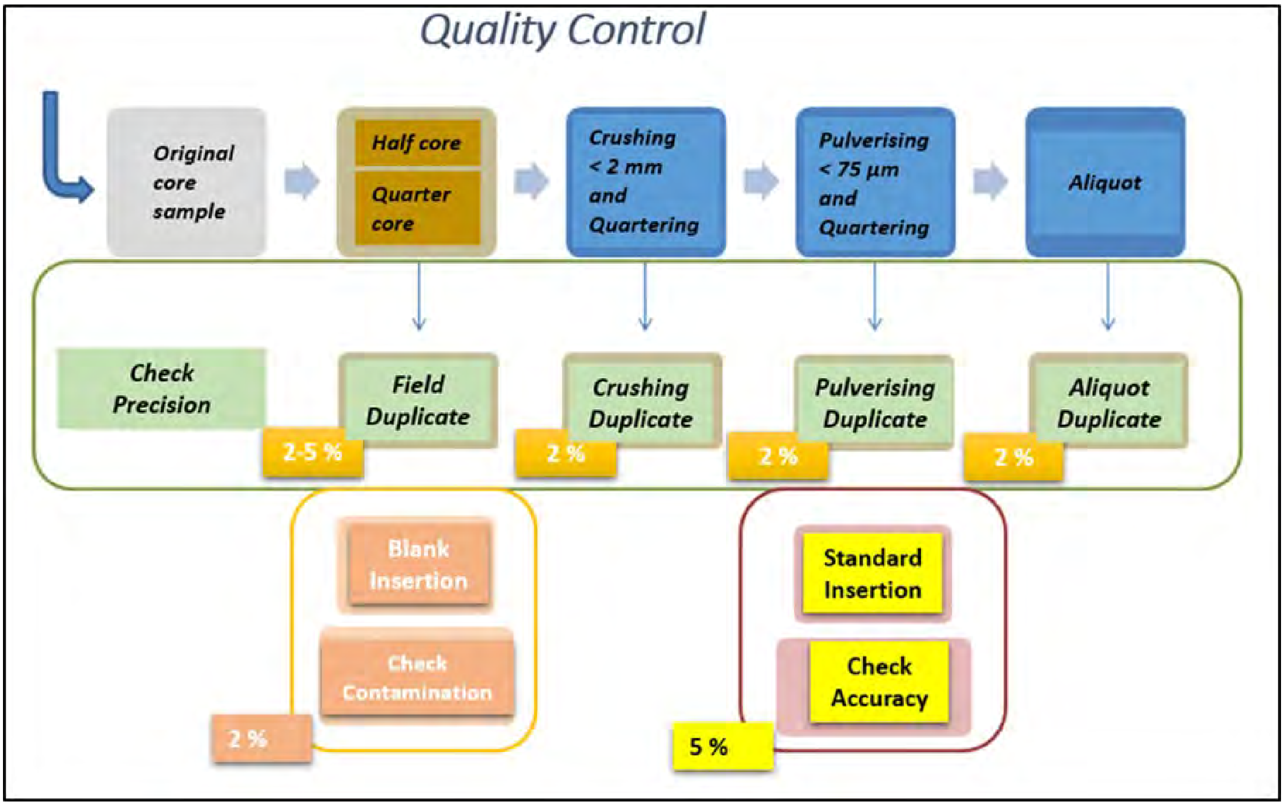
Standards, or Certified Reference Materials (CRM) were inserted every 25 samples to monitor the accuracy of analysis at the respective laboratories. Both CRM standards and matrix-matched standards (MMS) were submitted (Table 11-1).

Table 11-1 List of CRM's and MMS's used in CLC drilling campaigns QAQC.

Standard	Type	Mineralization
OREAS 904	CRM	Secondary copper sulphides
OREAS 95		
OREAS 96		
OREAS 901		
OREAS 620		Primary copper sulphides
OREAS 621		
OREAS 622		
OREAS 623		
OREAS 624		
GMB913-12		
STK-0	MMM	Primary copper sulphides
STK-1		
PZN		
PMS-5		

Standard results highlight limited evidence for sample number mixing and that returned values were within two standard deviations of the certified value. The QAQC process is summarised in Figure 11-2.

Figure 11-2 QAQC procedure for diamond core sampling



ITEM 12 DATA VERIFICATION

Mr Carmelo Gomez Dominguez (QP) and Mr Juan Manuel Escobar Torres have both worked as professional geologists at CLC for many years. During this time, both have gained familiarity and confidence in the available diamond drilled data, the geology models and the prevailing mineralization. They have been responsible for drilling, logging, sampling, database management, block model estimates, ore control, reconciliation, geological studies and this resource estimate. Mr Gomez believes that the geological understanding and data available for this CLC Mineral Resource estimate is of good quality and is representative of the prevailing mineralization relevant to the deposit.

Several verifications are hereby confirmed by Mr Gomez and Mr Escobar Torres:

1. Diamond drillhole collar coordinates were checked against the latest topographic survey data and were verified through visual observation and digital checks against database data. Spot checks were completed using a handheld GPS.
2. Sampling methods and data correspond to visual inspection of samples taken from stored core and samples are correctly represented against the original sample sheet records and the stored database data.
3. Laboratories used for sample analysis were audited during processing of the CLC samples to ensure adequate practices were followed.
4. Database validations were performed to:
    a) Ensure assay results reflect original assay certificates
    b) Investigate outlier values for assays data fields
    c) Checked for and addressed errors related to overlapping or duplicate logging and sampling records
    d) Check orientations and relative magnitudes of downhole survey data
    e) Confirmed that relevant metadata was recorded consistently and accurately.
5. QAQC data was investigated together with the process used for analysis. The methods were verified as robust for assuring assay accuracy, precision and that they limited contamination.
6. In-pit observations served to verify the prevailing geology and its association with the different styles of mineralization as per the logged data and 3D geology models.
7. Mining and run of mine stockpiling of mineralised material was verified through visual checks, grade control and reconciliation processes.

Review of drilling and logging procedures, sample preparation, analytical methods, database security and management, supports the assessment that data is good quality and is representative of in-situ PPS mineralization. The QP, Mr Gomez is confident that the information available is of a good standard for use in estimating the PPS mineralization at CLC.

ITEM 13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Core metallurgical test work began in 2014 and lasted five years. Testwork comprised two main stages: the first one aimed at defining the most suited technology to treat CLC polymetallic sulphides, with the second stage carried out at pilot plant scale to validate the selected technology. The resulting operating data from the pilot plant and other testing forms the basis of assumption for recovery estimates as summarised in Table 13-1.

13.2 Metallurgical Testing Programs

In 2014, differential flotation testing was undertaken by Wardell Armstrong International (WAI). Results showed that selectivity was too low to produce differential concentrates with acceptable recoveries. Specifically, copper and zinc separation was not efficient and zinc concentrates in particular could not reach acceptable quality. These results led to consideration of hydrometallurgical routes, including pressure and atmospheric ferric oxidation, for copper and zinc leaching as well as chloride leaching for silver and lead. The process scheme applied a previous bulk flotation to feed the leaching stage with a bulk rougher concentrate instead of direct ore. The ore characteristics did not allow for the concentrate to be upgraded via the secondary flotation cleaning step without significant value loss. Therefore only a rougher bulk concentrate was used going forward. This allows for reducing the hydrometallurgical plant capacity and consequently the CAPEX, with minor metal losses. Initial bulk flotation testing was carried out by WAI and AGQ Mining.

With regard to copper and zinc leaching, CLC performed bulk concentrates atmospheric leaching trials to evaluate the performance with a silver catalyst addition. Results proved to be positive and CLC utilised SGS-Lakefield for external testing of pressure and atmospheric leaching routes in 2015. Both yielded high copper and zinc recoveries. As demonstrated in CLC's initial tests, atmospheric leaching required a silver catalyst addition to enhance copper (chalcopyrite) dissolution, and zinc dissolution in some cases.

Considering the above results as well as noting that CLC already operated a copper leaching plant in atmospheric conditions, this was the route selected to minimize the CAPEX, including addition of recycled silver catalyst to the leaching reactors.

For lead and silver, chloride leaching was evaluated by CLC at its metallurgical laboratory in 2015, with very satisfactory results.

Laboratory test work carried out between 2014 and 2016 used different samples, each progressively more representative of the orebody as new blends were prepared. In 2016, specific laboratory testing was undertaken using a 3,077 kg massive sulphide sample composited out of 87 drill cores and representing around 67% of the total deposit. Results with this sample were used in the base case of the PMR and the defined processing was applied in the piloting.

The main objective of pilot plant testing was raising the Technology Readiness Level (TRL) from prior Technology Readiness Level TRL-5 to TRL-7 and 8. TRL represents a method for estimating the maturity of technologies during the developing phases.

In 2016, CLC constructed a 1,000 kg/h capacity pilot plant to run trials aiming at technology validation in steady state conditions. The piloting trials treated circa 5,000 tonnes of primary polymetallic ore. For this purpose, blending was prepared with material from its Primary Ore Stockpile and primary ore from open pit blasting (V1131). Around 4,200 tonnes of stockpiled ore and 800 tonnes of primary ore from the blasting were combined to compose the blend.

Pilot plant operation was 24 hours, seven days a week during the years 2016-2017. The pilot plant at CLC comprised the stages of grinding, flotation, copper and zinc sulphate leaching and silver and lead chloride leaching. The sulphate leaching liquor bearing copper and zinc produced at the CLC pilot plant was sent for external pilot testing on zinc purification (SX) and electrowinning, reaching positive results that confirmed the production of Special High Grade (SHG) quality Zn cathodes.

A second series of piloting trials was carried out in continuous running 24 hours, seven days a week, in the period from 2017 to 2018. The target was to develop and validate the novel silver and lead "dynamic cementation" technology in the pilot equipment dedicated to silver and lead recovery as cemented metals. The main advantage of this new cementation procedure is to produce cemented metals free of contamination from cementing metal agents. For this campaign, synthetic solutions prepared with litharge as well as a 500 tonnes sample of fresh massive sulphide ore from a blast were used.

A portion of produced silver cement was recycled and added as a catalyst to the atmospheric leaching reactors of the bulk concentrate, yielding very positive results; therefore, the use of recycled silver cement as a catalyst was validated in the pilot plant.

Lead metal cement recovered in the pilot plant was sent to Ruf (Germany) to produce metal briquettes of compacted lead. Briquettes of suitable quality for lead melting by offtaker were successfully produced..

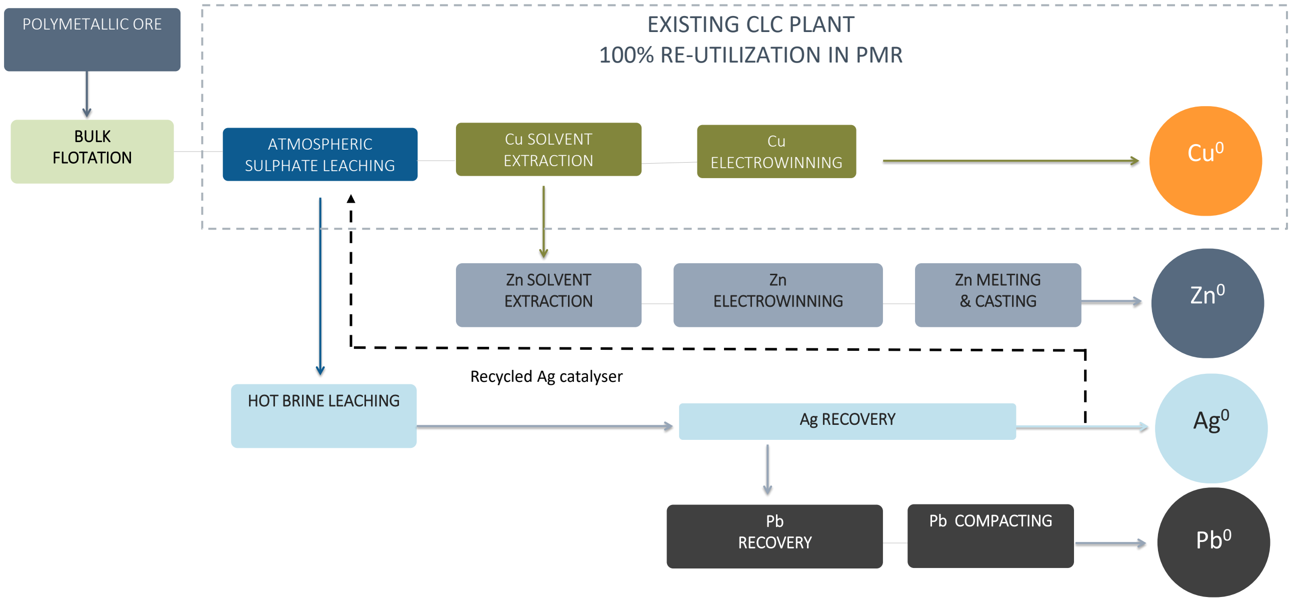
As a result of the trials described above, pilot plant results confirmed the PMR technology.

PMR flowsheet was defined with the following process stages:

- Crushing and grinding
- Bulk rougher flotation
- Silver catalyst atmospheric leaching (SICAL) for copper and zinc sulphides. To be noted is that the silver catalyst is recycled after recovery downstream in the silver plant. There is no fresh silver catalyst consumption.
- Copper purification by solvent extraction (SX) and recovery as LME grade "A" cathodes by electrowinning (EW).
- Zinc purification by SX after iron removal. Zinc is subsequently deposited as SHG-grade cathodes (EW) and produced as ingots in melting and casting section.
- Lead and silver extraction by leaching in hot brine solution.
- Silver recovery by "dynamic cementation" on lead plates to generate a silver cement product.
- Lead recovery by "dynamic cementation" on aluminium plates, with subsequent compacting of the lead cement to produce lead metal briquettes as final PMR product, suitable for external melting and casting to produce high purity lead ingots.

Summary flowsheet shown in Figure 13-1.

Figure 13-1 1 PMR Plant Process Flow



Following initial pilot plant operation, a blend was prepared to confirm the base case laboratory results with a sample of higher representativeness. The composed 1,550 kg sample was prepared with 114 drills representing approximately 84% of the total deposit, including massive as well as stockwork material (75:25%). The metallurgical results confirmed those adopted for the base case.

The table below summarizes the recovery scenarios for the PMR process.

Table 13-1 PMR Base Case Metals Recovery

Metals Recovery (%)				
	Cu	**Pb**	**Zn**	**Ag**
Flotation	90.5	81.0	94.0	74.1
Hydromet	94.1	98.0	94.1	70.0
Overall	85.1	79.4	88.4	51.9

Results from the pilot plant were used for the PMR plant conceptual engineering. Initial Engineering Cost Study (ECS) work was undertaken by Lycopodium Engineering in 2018. The ECS was fully updated by Lycopodium Engineering in 2023 with a +20 -10% accuracy estimate. The engineering work included the following deliverables:

- Plant Block Diagrams
- Plant Process Flowsheet Diagrams
- Mass and Energy Balances
- Equipment List
- Process Design Criteria
- Layouts
- Process Description
- Plant CAPEX Estimate

13.3 Metallurgical Samples

To supply representative samples for PMR metallurgical tests at the CLC Technology Department, samples of primary sulphide (massive and stockwork) were selected from existing boreholes, and from those of the different infill drilling campaigns, for the realization of blends. Representativeness of the total deposit increased as new blends were composed. Base case metallurgical results defined in 2016 utilized the PMS 4 composite sample. Samples PMS 5 and 6 include massive sulphides as well as stockwork.

Table 13-2 Blends used in PMR metallurgical testing

Sample (year)	% of deposit	Drills	Samples	% Cu	% Zn	% Pb	ppm Ag
PMS 1 (2014)	27	35	41	1.99	2.65	1.48	34
PMS 2 (2014)	27	35	1,552	0.62	3.70	1.53	30
PMS 3 (2015)	40	52	802	1.05	2.95	1.32	28
PMS 4 (2016)	67	87	2,395	1.07	2.98	1.36	28
PMS 5 (2017)	84	114	3,829	1.18	2.04	0.83	22
PMS 6 (2017)	84	114	3,829	0.87	2.30	1.04	23

13.4 Preliminary Metallurgical Testing

Preliminary testing was conducted to define the bases upon which to define the processing scheme and set the piloting operation.

Testwork at this stage established the ore characteristics for the comminution circuit design as well as the conditions and performance of the bulk flotation, maximizing metals recovery and optimizing the mass pull to suit the existing leaching plant capacity as much as possible.

Preliminary batch metallurgical testing on copper and zinc leaching in ferric sulphate media demonstrated that high extraction yields could be achieved. Laboratory trials performed by SGS in 2015 demonstrated that pressure leaching produced copper and zinc extractions of 97%. Trials undertaken by SGS and CLC on atmospheric leaching catalyzed by silver yields were equivalent or better, giving values of >97% for copper and 97% for zinc but with the much reduced circuit complexity. Copper extraction under atmospheric conditions requires the addition of silver catalyst, to offset the passivation effect caused by a deposited sulphur layer. Additionally, silver catalyst also improves zinc extraction in some tested materials. Leaching is undertaken at 90 degrees Celsius and in acidic conditions. Leaching time was determined at 17 hours.

Considering that CLC owns, and previously successfully operated, an atmospheric leaching reactor circuit, and that the whole industrial plant facilities will preferentially be utilised in the future PMR plant, the decision was to select catalysed atmospheric leaching of the bulk concentrate as preferred leaching technology for PMR process.

Preliminary lead and silver leaching batch tests in chloride media showed that lead and silver dissolution is efficient, with extraction yields above 98%. Conditions in the batch tests were set at 80 degrees Celsius and acidic media, with a leaching time of 2 hours.

Lead and silver recovery using cementation was successfully tested, offering cementation rates not below 98%. An important aspect of silver recovery is to provide the silver product required to be recycled as the effective catalyst to enhance copper and zinc leaching.

Following these preliminary tests, CLC completed further detailed laboratory experimental studies on representative samples to confirm and optimize the process conditions prior to proceeding to piloting. The produced lab results were very satisfactory, confirming the preliminary results.

13.4.1 Comminution Testwork

A programme of comminution testwork was undertaken by ALS Kamloops in 2017 on a series of primary sulphide and stockwork samples prepared and sent by CLC. The sample shipment had a total mass of 95 kg and comprised 11 samples of which six were ½ core samples and five had been pre-crushed to pass a six mesh (3.36 mm) screen aperture. Of the ½ core samples, three were primary sulphide (named as Masivo) and three were stockwork material. The Stockwork and Masivo 1/3, 2/3 and 3/3 samples of the table below were received as half core samples and were used to construct the Stockwork and Masivo composites. The samples for Bond Mill testing were received as material passing six mesh.

Table 13-3 Samples used for the comminution test work

Sample ID	Weight (kg)	Sample Form
Stockwork 1/3	9.4	½ Core
Stockwork 2/3	9.1	½ Core
Stockwork 3/3	7.9	½ Core
Masivo 1/3	8.8	½ Core
Masivo 2/3	8.8	½ Core
Masivo 3/3	10.5	½ Core
Primario Planta Piloto	8.0	<0.6 Mesh Bulk
Stock Primario	8.0	<0.6 Mesh Bulk
PMS-5	8.1	<0.6 mesh Bulk
PMS-6	8.1	<0.6 Mesh Bulk
STK-01	8.0	<0.6 Mesh Bulk

The three primary sulphide samples were combined to form a 'Masivo' composite and the three stockwork samples were combined to form a separate 'Stockwork' composite. Both composites were then subjected to a standard suite of tests to determine the JKSimMet and SMC test comminution parameters.

The remaining five samples were each subjected to a Bond Ball Work Index test using a closing screen size of 53 microns (µm). The two primary sulphide samples (PMS 5 and 6) returned BBWi values of 13.4 kWh/t and 12.5 kWh/t, while the stockwork sample returned a BBWi value of 14.3 kWh/t. The pilot plant primary feed sample and a single surface stockpile sample returned BBWi values of 11.2 kWh/t and 10.5 kWh/t respectively.

13.4.2 Flotation Testwork

A first milestone in flotation testwork was closed in 2016 after testing with a polymetallic sulphide ore sample representing around 67% of the deposit. The drill core used to develop the total blended sample mass of 3,077 kg originated from 87 drill holes and 2,395 separate samples, The testwork programme was aimed at optimising the flotation configuration and yielded optimised recoveries of 90.5% Cu, 94.0% Zn, 81.0% Pb, and 74.1% Ag to a concentrate mass pull of 35%. These values have been applied to the process design criteria.

13.4.3 Primary Leaching Testwork

A series of primary leaching tests were conducted at the CLC laboratory. Results for the sample representing around 67% of the total deposit yielded the expected results for copper and zinc dissolution assisted by silver catalyst. Copper and zinc dissolution achieved were of 98% and 95% respectively.

Further laboratory testing carried out after the pilot plant operation using blends covering around 84% of the deposit and including massive sulphides and stockwork confirmed the base case results, yielding 96% and 94%. Additionally, the stockwork sample was tested separately, giving leaching rates of 97% and 99% for copper and zinc respectively.

13.4.4 Secondary Leaching Testwork

Brine leaching laboratory initial testing for lead and silver carried out by CLC and the University of Seville in 2015 achieved lead and silver dissolution rates of 99% and 98% respectively. These results were consistently achieved in the subsequent testwork done as more representative samples were used.

13.4.5 Silver Cementation Testwork

Preliminary silver cementation testing was performed by CLC using a laboratory reactor containing a rotating lead disc in contact with a synthetic PLS solution. Achieved results for silver cementation efficiency with time at different disc rotation speed values reached total silver cementation, exceeding 99% value.

13.4.6 Lead Cementation Testwork

Cementation testing was performed by CLC using a laboratory reactor containing a rotating aluminium disk in contact with a synthetic lead PLS solution. Achieved results regarding lead cementation efficiency with the time at different rotation speed values proved full lead recovery, exceeding 98% value.

13.5 Pilot plant Testing

To perform the pilot testing, CLC designed and constructed a pilot plant in 2016 of a nominal capacity of 1,000 kg/h of ore crushed below 2 mm. The sample of c.5, 000 tonnes for the pilot plant comprised ore from the surface stockpile of primary massive sulphides and primary ore from an open pit blasting (V1131). Around 4,200 tons of stockpiled were blended with 800 tonnes of primary ore from the blasting. The pilot plant operational areas are as follows:

- Milling area, comprising a primary ball mill with closed circuit hydrocyclone, slurry pumps and a secondary ball mill. The milling product is the ground ore slurry feeding flotation.
- Bulk flotation area, with two conditioning tanks, eight 500-liter Outotec flotation cells distributed in four banks and a press filter for concentrate filtration. Flotation produces the slurry (or cake) of bulk concentrate that feeds the atmospheric leaching.
- Atmospheric copper and zinc sulphides leaching stage, including a vertical tower mill to grind the concentrate, a cascade of four fiberglass $4m^3$ agitated reactors with oxygen supply and electrical heaters. Solid-liquid separation equipment includes a thickener and a filtering and washing stage. The bulk concentrate slurry is reground and leached in the reactors. The pregnant liquid solution (PLS) bearing the leached copper and zinc is sent to the thickener and then to a filtration and washing stage where it is separated from the solid residue bearing the unleached lead and silver, now in sulphate form.

- Hot brine chloride leaching undertaken in a series of three 1m^3 fiberglass stirred reactors heated with steam. This stage generates a final tail as well as a PLS with the dissolved lead and silver. The tail is filtered and washed.
- Silver recovery stage by means of the "dynamic cementation" technique developed by CLC. This section included one fiberglass 700-liter agitated vessel where silver cement is produced.
- Lead cementation area, using the "dynamic cementation" technique by CLC. The produced lead cement is filtered and washed in a belt filter. Finally, lead cement is compacted to produce briquettes of high purity.

Figure 13-2 2 Pilot Plant Front View

Figure 13-3 3 Pilot Plant Aerial View

Figure 13-4 4 Pilot Plant Leaching Reactors

Pilot trials were run in continuous (24 hours, 7 days a week) in two phases:

1. July 2016 to April 2017. Testing and validation of the bulk flotation and the Cu, Zn, Pb and Ag leaching stages of the produced bulk concentrate.
2. July 2017 to May 2018. Testing and validation of the Pb and Ag dynamic cementation stages.

During the first phase, a total of c 5,000 ore tonnes was tested in the pilot plant. The relevant results for the pilot plant campaigns are as follows:

- Grinding product optimum size is around 35 microns P80.
- Bulk flotation recoveries of 79% Cu, 91% Zn, 73% Pb and 71% Ag were achieved.
- Sulphate leaching extractions are 93% Cu and 94% Zn. Residence time is of 17 hours, silver catalyst addition is set at 1.0 kg/t and temperature is of 90ºC. Concentrate to be reground to 10 microns before leaching.
- Chloride leaching extractions are of 96% Pb and 97% Ag. Residence time was of 2 hours and temperature was of 80ºC.

During the phase 2, several runs were undertaken with synthetic and real solutions aiming at developing and validating the novel “dynamic cementation” of silver and lead in the pilot plant. For that purpose, a 500 tonnes sample was prepared by CLC Geology department using fresh polymetallic sulphide ore from a blast. Synthetic solutions were prepared using litharge and doped with gypsum cake from the CLC existing plant to suit calcium levels. These materials were treated in the pilot plant through all the process stages, flotation, sulphate leaching and brine leaching, to generate a lead and silver liquor suited for the scoped metals recovery pilot testwork.

Produced results were very positive, recovering over 98% of both metals, lead and silver.

CLC pilot plant results led to validating the technology and defining the main process conditions to be applied in the conceptual engineering work and the PMR Project pre-feasibility study.

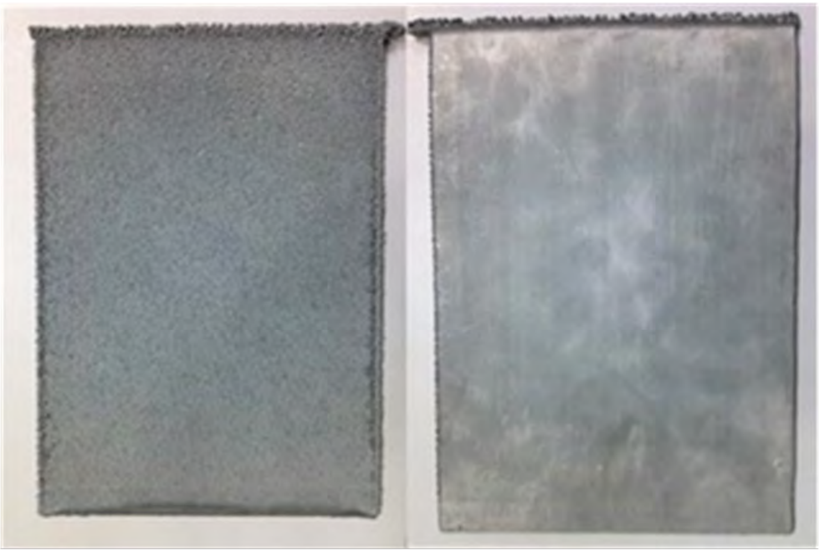
An important aspect of the piloting work aimed at the generation of commercial products during the trials. Achieving copper cathodes was not in the scope since the pilot plant PLS was suited to copper cathode production in the existing CLC plant.

Since CLC pilot plant did not include the equipment to generate the zinc and lead; the pilot tests to produce the final metals were undertaken by external companies.

Zn piloting was carried out at the Outotec facilities in Pori, Finland for two weeks. The feed sample was of two cubic metres of pregnant liquor solution (PLS) generated during the operation of the CLC pilot plant. Outotec pilot scheme comprised the stages of iron removal, Zn Solvent Extraction (SX) and electrowinning (EW) and was run in a continuous mini pilot plant. Iron removal from the PLS is needed before Zn SX since the extracting reagent (D2EHPA) is not selective for zinc versus iron. Iron removal is achieved by means of lime addition, which produces a cake of gypsum and iron hydroxides and an iron-free PLS suited to Zn SX.

Continuous piloting was done for two weeks at a rate of 1 kg/h of EW deposited Zn. The results confirmed that SHG-quality Zn cathodes could be obtained from the CLC pilot plant PLS, confirming the PMR technology validation for zinc.

Figure 13-5 5 Piloting: Zn SHG Grade Cathodes



The PMR lead commercial product is in form of lead briquettes, of a quality suitable for melting. In the CLC pilot trials conducted using fresh massive sulphide ore and synthetic solutions, 25 tonnes of metallic lead cement were produced by means of cementation. A 500 kg sample of cement was sent to a German company (RUF) that commercializes metal powder briquettes to test briquetting on the PMR lead cement. Trials were successfully done and good quality briquettes for melting were produced.

Figure 13-6 6 CLC Pilot Plant Lead Cement compacted as Briquettes

13.6 Effluent Treatment Definition

The effluent from the PMR plant is the raffinate bleed from the Zn SX stage. The specifications of this stream are similar to those of the existing CLC plant, with the difference that it does not contain iron.

Iron in the bleed feeding the effluent treatment plant is necessary to make sure of the efficient removal of impurities such as arsenic. Arsenic precipitation only occurs onto the surface of ferric ion precipitate.

To provide a robust effluent treatment process, the PMR circuit repulps a small proportion of the iron cake from the iron removal stage with the Zn SX raffinate bleed. This way, the effluent treatment plant feed bears the iron charge needed for efficient arsenic removal.

13.7 Ore Variability

Metallurgical laboratory testing was undertaken considering ore variability. This implies two different materials: primary sulphides and stockwork. PMR base case metallurgical testing utilised a composited sample approximately in line with ongoing mining expectations

Consequently, part of the laboratory testing for the PMR process was accomplished for individual samples. Specifically, PMS 5 and stockwork samples were tested for flotation and sulphate leaching. According to the results, both samples show similar yields, with the stockwork more amenable for flotation and leaching.

Table 13-4 Metallurgical performance of Base Case and confirmation testwork

		Flotation recovery %				Sulphate Leaching %	
	% deposit representativeness	Cu	Zn	Pb	Ag	Cu	Zn
PMS 5	84	93.1	94.9	79.8	72.3	95.6	94.1
Stockwork	17	95.0	94.9	85.3	86.9	97.1	99.7
Base case		90.5	94.0	81.0	74.1	94.1	94.1

The range of samples and composites evaluated through the various metallurgical programs have demonstrated the robustness of the PMR process. The ore fed into the circuit is limited only by the contained metal(s) and the inherent capacity of the final production unit, whether this be EW for Copper and Zinc or cementation for Lead and Silver. Equally, the nature of the process is unaffected by impurities such as Arsenic or Mercury.

13.8 PMR Batch Additional Testing

Following the pilot plant testwork, CLC undertook in 2018 further metallurgical testing at its laboratory to confirm the metallurgical performance data defined out of the previous laboratory and the pilot plant work. In this additional testing, a sample more representative of the deposit was used.

Following the pilot plant campaign(s), confirmatory flotation and sulphate leaching tests were conducted, in 2018, on a blended polymetallic supplied and stockwork (75:25%) ore sample labelled as PMS 5 of 1,150 kg and composed out of 114 drills, representing approximately 84% of the total deposit. Testing results confirmed those of the base case, yielding flotation recoveries of 93% Cu, 95% Zn, 79% Pb and 72% Ag. Sulphate leaching recoveries of 96% Cu and 94% Zn were achieved.

After these confirmation trials, the PMR metallurgical performances for the base case as follows:

Table 13-5 Base Case metals recovery

Metals Recovery (%)				
	Cu	**Pb**	**Zn**	**Ag**
Flotation	90.5	81.0	94.0	74.1
Hydromet	94.1	98.0	94.1	70.0
Overall	85.1	79.4	88.4	51.9

During the second half of 2022, metallurgical work was carried out to optimize plant OPEX. The most relevant outcome was the reduction in lime consumption by means of lowering the iron levels in leaching. This upside was confirmed through laboratory testing and metallurgical simulation tools (Metsim).

The base case recovery profiles utilized in the Engineering designs are considered conservative.

13.9 Summary of Key Process Design Parameters

The design basis for the concentrator plant has been developed on the basis of laboratory bench and pilot scale testwork carried out in 2016, 2017 and 2018 with appropriate scale-up factors applied to testwork results. For the hydromet plant, the early bench scale work has been used, supplemented by a number of results from the pilot testwork campaigns as well as the additional laboratory testwork conducted in 2018 and 2022. Standard chemical relationships have been used in the development of the Metsim mass balance and these have been ratified where applicable by the pilot testwork results.

In addition, independent testwork programs have been conducted by vendors, using representative samples prepared and supplied by CLC, to confirm the sizing and selection of various equipment (e.g. thickeners and filters). The results of these testwork programs have also been applied to the design criteria developed for the project.

On the basis of the testwork outcomes, a concentrate product P80 of 35 µm has been adopted as the target grind size from the milling circuit for subsequent feed to the bulk flotation circuit. Testwork has also demonstrated that a mass pull to concentrate of ~35%w/w may be anticipated and that further grinding of this concentrate to a target P80 of 10 µm will be required for optimum downstream leaching performance and metal recovery.

ITEM 14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The CLC PPS Mineral Resource estimate was prepared in September 2023 by Mr Carmelo Gomez Dominguez (QP) and Mr David Gray. Grades were estimated into a 3-dimensional (3D) geology block model using ordinary kriging. The Project limits and coordinates were based upon the UTM coordinate system known as ED50 projection Zone 29.

Deposit mineralization was delineated with vertically drilled holes and several fences of angled holes for improving structure and ore body orientation. Holes are spaced at 15 m intervals for most of the western portion and 25 to 50 m spacing in the central and eastern areas.

All available drill data was used for the geological model interpretation as well as the spatially related copper mineralization. Interpolation parameters were based upon the geology, styles of mineralization, drill hole spacing and geostatistical analysis of the data. Wireframe modelling and all aspects related to the block model estimates were completed using commercially available software packages (Datamine Studio RM version 1.10.69.01 and Snowden Supervisor version 8.14.3.0).

Mineral Resource estimates were classified according to geological continuity, QAQC, density data, drillhole grid spacing, grade continuity and confidence in the panel grade estimate and have been reported in accordance with the guidelines of the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014).

14.2 Geological and Mineralization model

3D wireframe models per geological domain were developed from the drill hole data by using string envelopes snapped and digitised along consecutive vertical sections. The string envelopes were linked with wireframes to form a 3D volume per domain. The mineralised domains, in stratigraphic order are:

1. Gossan Zone (Gossan)
2. Depleted Zone (DZ)
3. High Copper Main Zone (HC)
4. High Copper Small Zone (HC4)
5. Primary Massive Sulphides (PMS)
6. Semi Massive Sulphides (SMS)
7. Stockwork Zone (STWK)

The primary sulphide mineralisation was modelled as 3D wireframe volumes representing the lens shaped deposit, between 30 to 130 m thick with a strike length of 1,000 m and a down dip extent of 500 m. The PPS mineralization dips between 25° to 45° to the north.

Primary sulphide mineralization is overlain by a supergene profile with an upper gossan that overlies the cemented secondary sulphides. The respective zones have different concentrations of metals. Gold is concentrated in the gossan, but copper and zinc are depleted. In contrast, copper is enriched in the cemented secondary sulphide zones relative to the overlying gossan and underlying primary sulphide mineralization.

14.3 Available Data

A pre-mining topographic surface and survey of the final mined pit floor have been used as the upper limits of unmined Mineral Resources.

With drilling having started around 1994, 807 drill holes were available for informing this Mineral Resource estimate. 729 of these, contain sample intervals with assay results that are directly relevant to the deposit. The samples were routinely analysed for copper, lead and zinc. In addition, select samples were also analysed for gold, silver, iron and sulphur, plus sequential copper analysis. The additional elements have been estimated into the block model to support their economic contributions. This Mineral Resource estimate is focused on Polymetallic Primary Sulphides (PPS) and uses the supporting diamond drilling information. A general summary of the underlying drillhole database statistics is presented in Table 14-1.

Prior to de-surveying drillhole data into a 3D format, collar elevations were checked against the topographic surface with no corrections needed. Downhole survey data was investigated for excessive deviations with no risks identified. In addition, logging, sampling, and assay data was investigated for overlaps, gaps, and duplication. Sample values were checked to ensure no outlier samples. Good core recovery allowed all sample assay values to be included.

The validated and de-surveyed assay drillhole file was coded according to each of the mineralization domains. The coded drillhole data was used for compositing, statistical and geostatistical analysis, prior to grade estimation.

Table 14-1 Table of drillhole statistics per element of interest

Element	No. of samples	Total length (m)	Min.	Max.	Mean	St. Dev
Copper (%)	43,499	49,696	0.001	40.00	1.90	3.87
Lead (%)	46,972	55,665	0.005	44.46	0.70	1.83
Zinc (%)	44,351	52,814	0.005	36.91	0.96	2.18
Silver (ppm)	41,700	48,062	0.01	18,950	20.19	133.67
Gold (ppm)	12,123	13,356	0.005	353	0.80	5.09
Iron (%)	39,802	45,322	0.11	70.04	24.24	13.16
Sulphur (%)	39,534	45,065	0.01	64.19	25.48	16.92

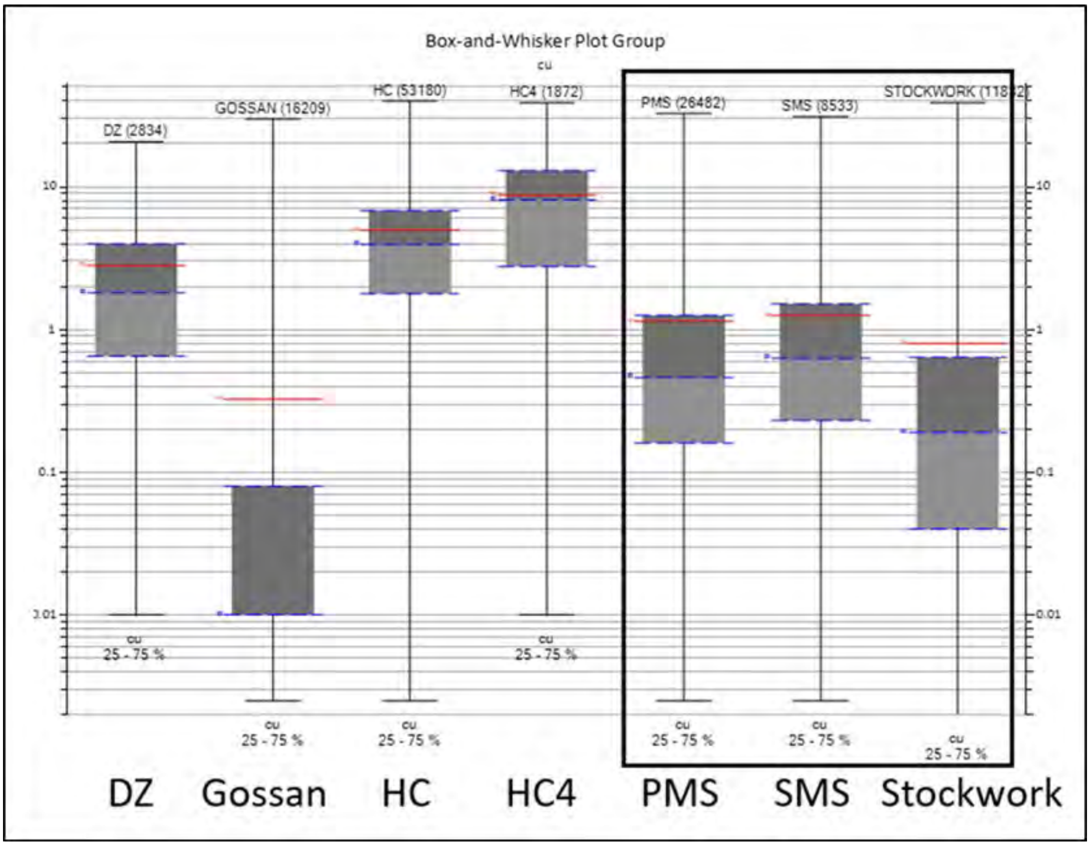
14.4 Sample Compositing

Downhole compositing of drillhole samples was completed in order to reduce the effect that varying sample length may have on grade values. A 5 m downhole composite length was chosen for stockwork and semi-massive sulphides domains, whereas a 2 m composite was chosen for PMS. Drillhole samples were composited so as not to cross geology boundaries.

14.5 Statistical Analysis

Exploratory data analysis has ensured minimal mixing of sample grade values per domain. If multiple populations were evident from data analysis, further geological constraints were added to limit unwanted mixing of data and to ensure accurate grade estimates per domain. The resulting sample mean grades and histogram distributions, per domain, are significantly different to each other. A box and whisker plot of summary copper statistics (Figure 14-1) presents all mineralised domains relative to the PPS domains (PMS, SMS and Stockwork). Higher grades are noted for the secondary sulphide DZ, HC and HC4 domains. Copper in the Gossan zone was depleted with values close to the analytical detection limit.

Figure 14-1 Box and whisker plot of copper assay values per geological domain with PPS domains highlighted.



14.6 Boundary Analysis

Boundary analysis was completed to identify the need for hard or soft boundary conditions during grade estimation of blocks. Significant grade changes (hard boundaries) were evident between Gossan and DZ, DZ and HC, HC and PMS and PMS and stockwork. Hard boundary conditions were therefore applied to all domain contacts for each element estimated into the block model. The respective domains estimated are summarized in Table 14-2 below.

Table 14-2 Summary of domain names used for block grade estimation

Domain	Comments
Gossan	Oxidized zone. Low in Copper, high in gold, silver, and lead.
DZ	Depleted Zone. Transitional zone between Gossan and HC. Thin layer of 1-6 metres. Copper partially depleted.
HC	High Copper. Zone of enrichment. Supergene mineralization. Rich in Copper secondary sulphides.
HC4	High Copper 4. Small, structurally controlled enrichment zone for secondary copper sulphides.
PMS	Primary massive sulphides. Hypogene mineralization. Copper, lead and zinc sulphides.
SMS	Semi-massive sulphides. Hypogene mineralization. Copper and zinc with lesser lead sulphides.
Stockwork	Irregular zone of stockwork vein type mineralization. Grades tend to be higher immediately below the HC and PMS domains.

14.7 Top Cutting

Histograms and log probability plots were used to identify outlier grades for all elements per domain to limit the influence of excessively high grades on block estimates. Top cuts impacted domain statistics with reduced variance and negligible impact on mean values and distributions. Top cuts were applied separately to each element and domain (Table 14-3).

Table 14-3 Summary table of top cut values applied to Polymetallic Primary Sulphides domains.

Domain	Element	Top Cut
PMS	Cu (%)	13.8
	Pb (%)	19.0
	Zn (%)	9.2
SMS	Cu (%)	13.8
	Pb (%)	12.5
	Zn (%)	20.5
Stockwork	Cu (%)	10.0
	Pb (%)	6.4
	Zn (%)	8.8

14.8 Variography

The following variogram methodology was employed using Snowden Supervisor v8.14.3.0:

- principal axes of anisotropy were determined using variogram fans from normal scores variograms
- directional normal scores variograms were calculated for each of the principal axes of anisotropy
- downhole normal scores variograms were modelled for each domain to determine the normal scores nugget effect
- variogram models were determined for each of the principal axes of anisotropy using the nugget effect from the downhole variogram
- the variogram parameters were standardised to a sill of one
- the variogram models were back transformed to the original distribution using a Gaussian anamorphosis and used to guide search parameters and complete ordinary kriging estimation
- the variogram parameters were standardised to the population variance for each domain to permit post-processing of the copper panel estimates to SMU estimates.

The multidirectional variogram model results are summarised in Table 14-4. Variogram models had low nugget values which were clearly defined by the close spaced data. Each of the domains variograms were modelled using two spherical structures. The ranges of influence were clearly visible from the variograms, providing confidence in domain data selections and grade continuity. Variogram models were completed for copper, leachable copper, iron, lead, sulphur, density, zinc and gold per domain.

Table 14-4 Variogram parameters for copper, lead and zinc per Polymetallic Primary Sulphides domain

Domain	Grade	Variogram rotation angles			Nugget	Spherical model 1 ranges				Spherical model 2 ranges			
		Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill
PMS	Cu	0	0	-20	0.067	30	6	15	0.434	161	26	141	0.499
	Pb	0	0	-20	0.203	51	14	81	0.364	190	204	246	0.433
	Zn	60	55	90	0.173	11	31	24	0.212	246	205	156	0.615
SMS	Cu	150	10	180	0.21	41	23	40	0.26	140	38	54	0.53
	Pb	-20	10	0	0.13	34	23	37	0.33	145	167	103	0.54
	Zn	150	145	180	0.1	27	70	21	0.29	140	80	118	0.61
Stockwork	Cu	0	40	0	0.1	15	13	17	0.45	27	24	30	0.45
	Pb	-60	10	0	0.24	9	6	22	0.18	21	22	47	0.58
	Zn	0	60	-90	0.16	22	41	23	0.57	83	42	66	0.27

14.9 Kriging Neighbourhood Analysis

Kriging neighbourhood analysis (KNA) optimised block size, ellipse dimensions, minimum and maximum numbers of samples to be used for grade estimation as well as the discretization parameters. Correctly defined sample selection parameters are essential to optimising the estimation process and help reduce the risk of conditional bias (overestimation of high grades and underestimation of low grades). An optimum parent block size of 20 m by 20 m by 20 m was selected together with a minimum of 8 samples and a maximum of 26 for the estimation routine.

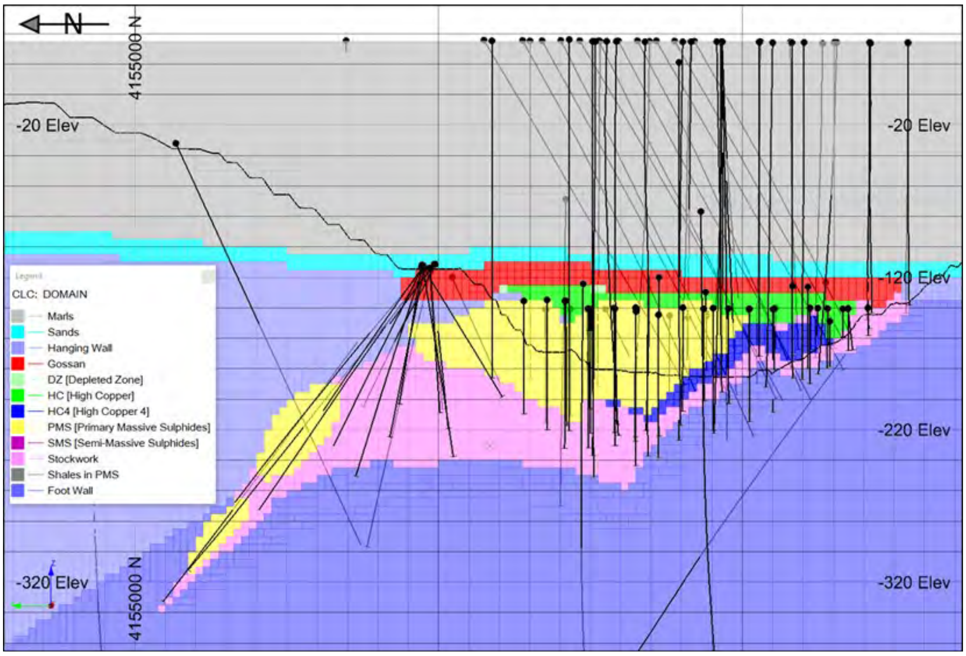
14.10 Block Model Setup and Limits

A 3D empty block model (Table 14-5) was compiled using a parent block size of 20 m x 20 m x 20 m and a sub-block size of 5 m x 5 m x 5 m for an accurate representation of mineralised volumes and to match the smallest mining unit (SMU). The block model was not rotated or unfolded. Local undulations in the mineralised horizons were considered by using dynamic anisotropy during estimation. Block model coordinates are expressed in ED50 coordinate system, Huso 29 (ED50-H29). Each block centroid was assigned domain code values as per the modelled wireframe volumes (Figure 14-2).

Table 14-5 Block model parameters – limits and dimensions

Block Model Settings		
X Origin		756,870
Y Origin		4,154,420
Z Origin		-540
Parent cell size	X	20 m
	Y	20 m
	Z	20 m
Minimum cell size	X	5 m
	Y	5 m
	Z	5 m
Number of parent blocks	X	59
	Y	45
	Z	32

Figure 14-2 North-South cross section showing the domains coded in the block model, diamond drillholes and the contour of the maximum excavation reached.



14.11 Grade estimate - Interpolation Parameters

Ordinary kriging (OK) was used to estimate copper, leachable copper, iron, lead, sulphur, density, zinc and gold into each parent block. OK was an appropriate estimation technique due to the near normal distributions and limited domain grade mixing. Interpolation parameters are summarized by domain in the Figure 14-2. Estimation into parent block used a discretisation of 4 (X points) by 4 (Y points) by 4 (Z points) to best represent the block volume shape.

Table 14-6 Summarised search ellipses for sample selection for main elements in PPS domains.

Domain	Grade	Search axis rotation			First pass search radius			# of composites		Second pass radius	Third pass radius
		Z	X	Z	X	Y	Z	Min.	Max.		
PMS	ag	Using dynamic anisotropy vectors	259	261	168	8	26	1.5	8	1.5	8
	au		60	74	68						
	cu		106	17	93						
	fe		91	83	138						
	pb		125	134	162						
	s		121	69	113						
	zn		162	135	102						
SMS	cu	150	10	180	92.4	25.1	35.6	8	26	1.5	8
	fe	10	25	-60	98.3	93.7	29.0				
	pb	-20	10	0	95.7	110.2	68.0				
	s	10	15	-50	199.3	79.9	56.8				
	zn	150	145	180	92.4	52.8	77.9				
Stockwork	cu	0	40	0	27.0	24.0	30.0	8	26	1.5	8
	fe	-20	10	-60	55.0	278.0	203.0				
	pb	-60	10	0	21.0	22.0	47.0				
	s	0	0	-40	67.0	132.0	109.0				
	zn	0	60	-90	83	42	66				

PMS grade estimation used dynamic anisotropy to modify search orientations to follow the meandering trend of mineralization. A mineralization trend surface was modelled from the vertical midpoint between the hangingwall and footwall surfaces of the PMS domain. Dynamic anisotropy optimises the sample selection routine and better honours grades continuity as the deposit changes strike and dip orientations.

14.12 Primary Massive Sulphides (PMS) Copper, Lead and Zinc estimate

Due to the variable spatial distribution of copper, lead and zinc mineralization, within the overall PMS domain, probabilistic methods (categorical indicators) were used to best define mineralised volumes. Categorical indicators were estimated into the 5 m x 5 m x 5 m sized block model (SMU).

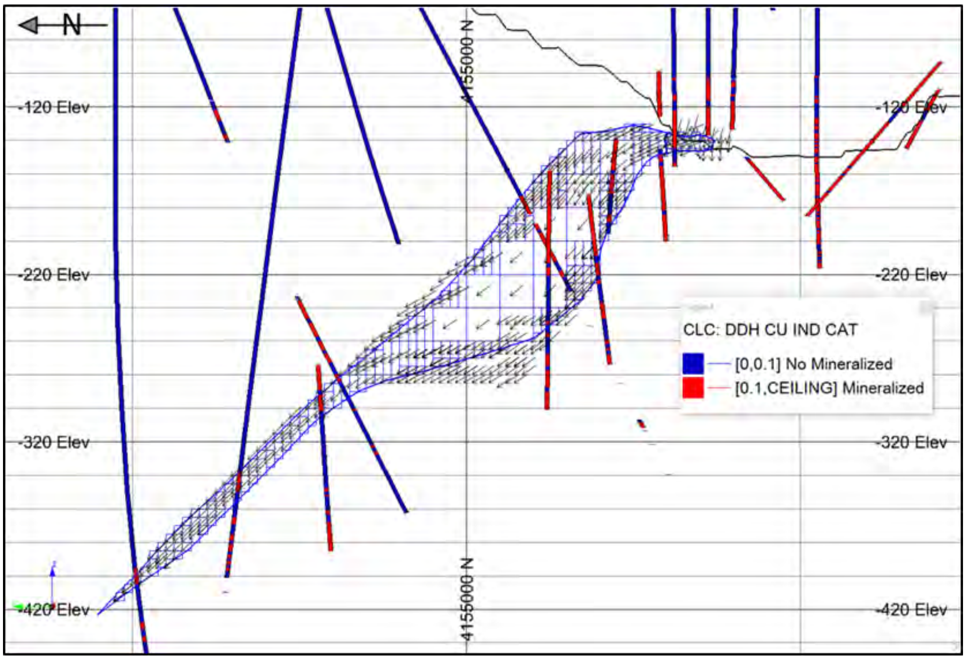
Each element had two indicator domains that were identified from analysis of each element's statistical distribution as well as a visual inspection of the spatial distribution of sample values. These indicator domains defined non-mineralised and mineralised volumes for each element.

Table 14-7 **PMS Copper, Lead and Zind Categorical indicator threshold values for mineralised and unmineralized domains.**

Domain	Element	Threshold	Definition	Indicator Value
PMS	Copper	< 0.1%	No Mineralised	0
		≥ 0.1%	Mineralised	1
	Lead	< 0.1%	No Mineralised	0
		≥ 0.1%	Mineralised	1
	Zinc	< 0.2%	No Mineralised	0
		≥ 0.2%	Mineralised	1

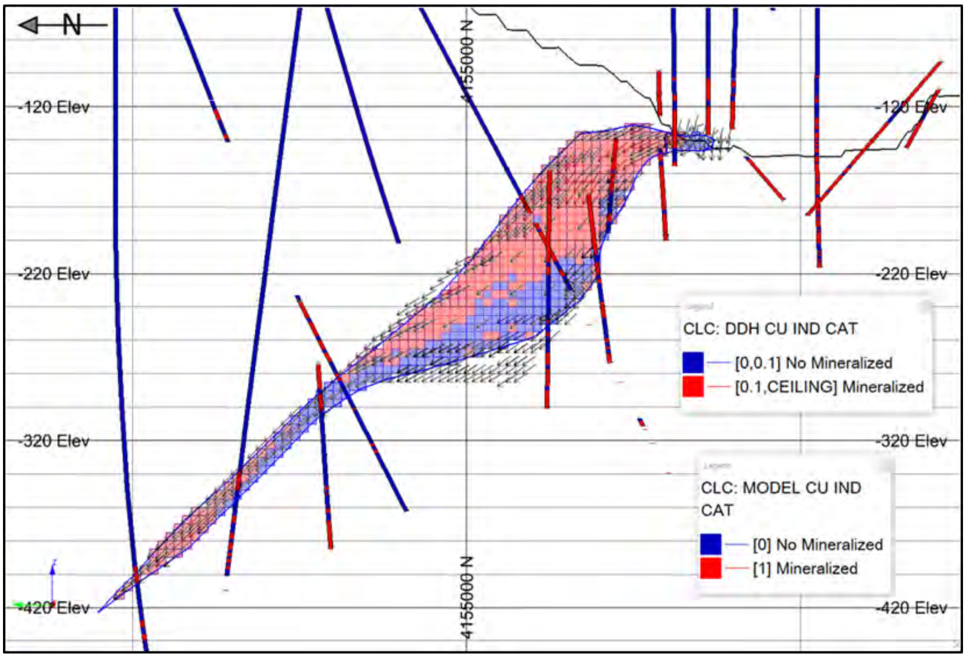
Variography was completed for each indicator with variogram models defined. Search orientations used for the categorical indicator estimates were guided by dynamic anisotropy (DA) vectors (Figure 14-3).

Figure 14-3 **Cross section showing copper indicator values in the drillholes and the dynamic anisotropy vectors in the block model for PMS**



Indicator values for copper, lead and zinc were estimated into the PMS domain of the block model. Mineralised volumes were identified from non-mineralised volumes where the probability value was greater than 50%. Blocks in the model were coded to mineralised and non-mineralised blocks.

Figure 14-4 **Cross section showing copper indicator values in the drillholes, dynamic anisotropy vectors and categorical indicator domains in the PMS domain block model.**



Copper, lead and zinc grades were estimated into the parent blocks as coded by the categorical indicator domain. Post processing of the parent block grade estimates into SMU sized grade estimates was completed using a change of support routine, localised uniform conditioning (LUC).

14.13 Semi-Massive Sulphides (SMS) Copper estimate

SMS grade estimates used a similar but separate probabilistic method for interpolating copper mineralised volumes within the SMS domain. Three different ranges were defined as indicators for representing the copper distributions, as detailed in Table 14-8. The Ultra Low-grade range was the less represented as most of the non-mineralised samples were already removed during the wireframing process.

Table 14-8 **Copper categorical thresholds for Copper mineralization within SMS domain**

Domain	Element	Threshold	Indicator	Definition
SMS	Copper	< 0.05%	ULGIND = 1	Ultra Low Grade (Not Mineralised)
		0.05% to 0.6%	LGIND = 1	Low Grade
		> 0.6%	HGIND =1	High Grade

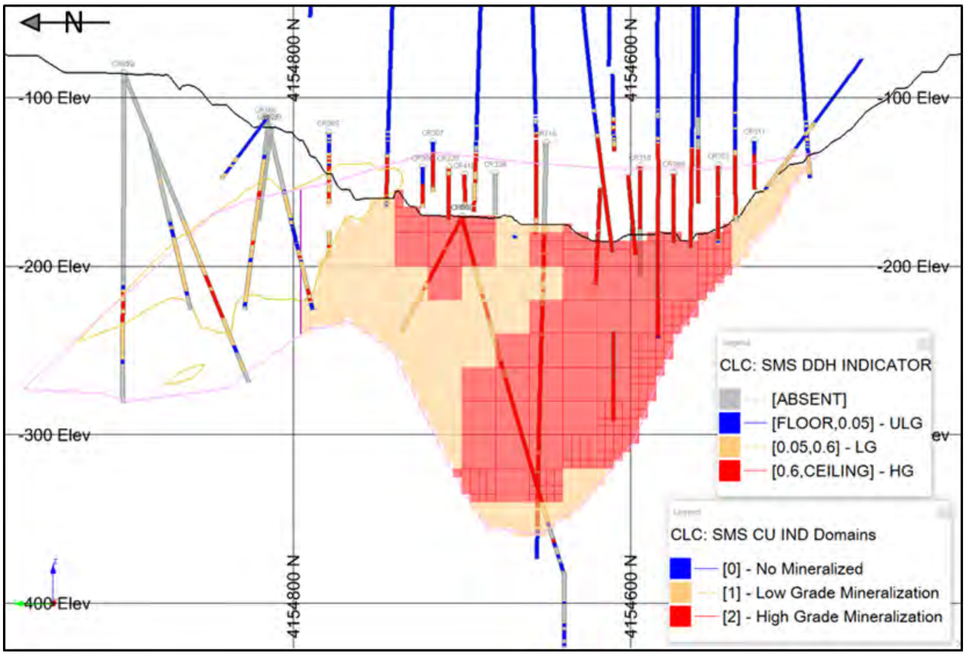
Indicator values were estimated into the block model and categorical domains were defined based on the combination of indicators probability values (Table 14-9 and Figure 14-5) as well as from visual and statistical analysis of the distributions.

Table 14-9 Categorical indicator threshold values for Copper within SMS domain for non-mineralised, low grade and high-grade domains

SMS Copper Domain	Value	Criteria	Definition
CUINDDOM	0	Default (ULG)	Ultra Low Grade (No Mineralised)
	1	LGIND ≥ 0.1 and HGIND < 0.6	Low Grade
	2	HGIND ≥ 0.6	High Grade

Copper grades were estimated within each categorical indicator domain using OK and then combined with the other element estimates.

Figure 14-5 Cross section showing copper indicator values in the drillholes and categorical indicator domains in the SMS domain block model.



14.14 Density Estimates

CLC drill data has a good number of drillhole samples with dry bulk density measurements which supports estimation of bulk density directly into the block model. The distribution of density data from the drillhole composites is listed in Table 14-10.

Table 14-10 Summary of density measurement statistics per domain

Domain	Dry bulk Density (t/m^3)		
	Minimum	Maximum	Mean
Gossan	1.51	4.99	2.49
DZ	1.96	4.8	3.09
HC	1.51	4.96	3.60
HC4	1.89	4.89	3.90
PMS	1.68	5.54	4.07
SMS	1.56	4.78	3.47
Stockwork	1.59	4.97	2.96

Density was estimated per domain using robust variograms and OK. Density ranges from 1.56 t/m^3 for areas in the SMS, 1.59 t/m^3 in the stockwork zones and up to 5.54 t/m^3 for parts of the PMS zone.

Table 14-11 Variogram parameters for density per domain

Domain	Variogram rotation angles			Nugget	Spherical model 1 ranges				Spherical model 2 ranges			
	Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill
Gossan	0	0	60	0.18	43	42	20	0.34	66	181	22	0.48
DZ	0	0	30	0.23	55	28	5	0.13	90	133	8	0.64
HC	60	30	150	0.12	13	16	8	0.31	77	103	72	0.57
HC4	-75	60	-20	0.11	13	10	17	0.25	76	31	18	0.64
PMS	-10	20	-5	0.18	85	54	13	0.45	293	179	25	0.37
SMS	-10	20	-5	0.18	85	54	13	0.45	293	179	25	0.37
Stockwork	-40	10	0	0.08	49	20	21	0.36	60	113	41	0.56

14.15 Post-processing by Localized Uniform Conditioning

All element grade estimates of PMS parent blocks were post processed using localised uniform conditioning (LUC). Uniform conditioning (UC) provides the proportions of parent block sub cells (SMU's) that are above a range of cut-off grades. LUC provides a spatial grade estimate per SMU block while maintaining the total metal estimated for the parent block. The LUC results provide an assessment of recoverable resources available per domain at the scale of mining and is particularly relevant in more widely drilled areas. The parent block size used was 20 m by 20 m by 20 m and the SMU block size was 5 m by 5 m by 5 m. Each parent block contains 64 SMU blocks. LUC models were validated by:

- Visual comparisons with drillhole sample grades and the OK sample grades.
- Checks that the SMU average grade is the same as the Parent grade.
- Checking that contained copper at a zero cut-off grade is the same for the OK estimates and the LUC estimates.

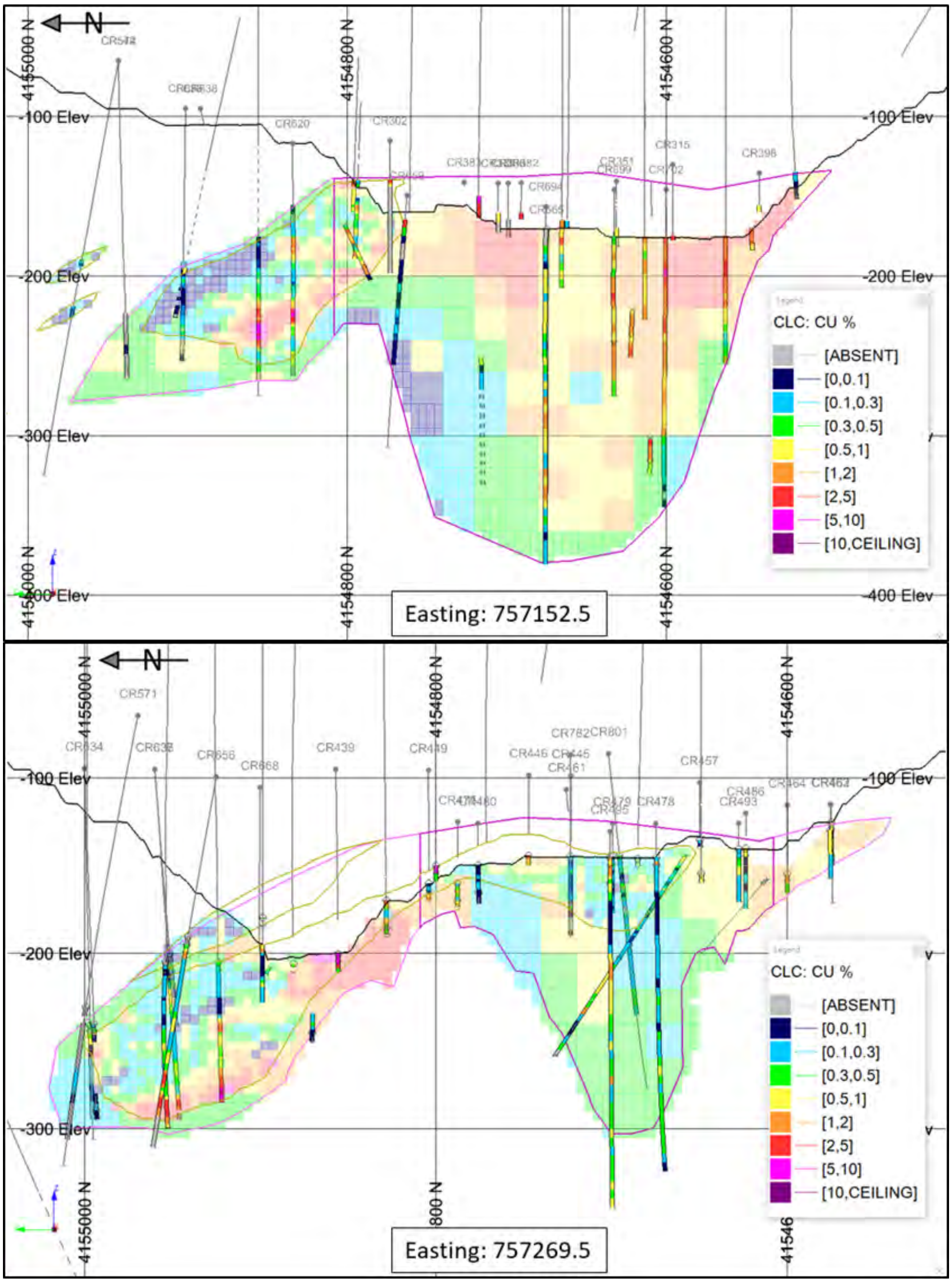
14.16 Validation of Block Model Estimates

The following validation steps were completed to ensure block grade estimates represent prevailing geology and input sample data:

- comparisons of wireframed volumes with block model volumes
- summary statistics comparing mean input composite grade and mean parent block estimate
- validation of sample grades and block grades in 2D cross sections
- comparisons of input sample data versus block model parent grade estimates and change of support (LUC) estimates using swath plots

The wireframes and block model volumes compared well and were within 1%. Visual validations suggest the grade tenor of the input data is accurately represented in the block model estimates. Figure 14-6 illustrates key vertical cross sections representing the block model SMU copper grades and the input drillhole composite copper grades for the PPS domains.

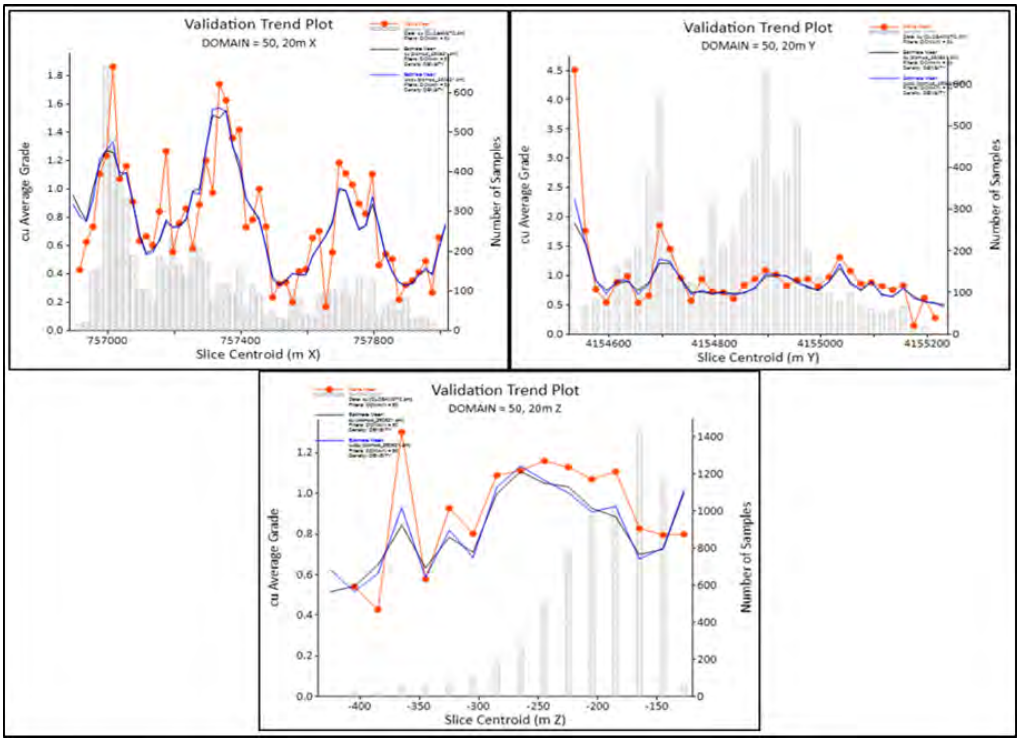
Figure 14-6 Vertical cross sections looking east and at an X easting of 757152.5 m and 757269.5 m for unmined Mineral Resources.



The parent (OK) and the LUC (SMU) estimates were compared at various cut-off grades using grade/tonnage curves with total copper metal preserved (the same) at a zero cut-off in both model estimates.

Swath plots (Figure 14-7) compare the mean grades of the input data with block estimates for consecutive widths in a particular direction (easting, northing or vertical). Grade variations from the OK parent block estimates and change of support (LUC) models are compared to those derived from the declustered input grade data. The swath plots (Figure 14-7) demonstrate good validation of parent and SMU estimates with the input sample data values.

Figure 14-7 Swath plots for copper in the PMS domain by easting, northing and elevation. Mean from drillhole composited samples are coloured red, OK parent block estimates are in black and LUC change of support estimates are in blue.



Summary statistics, visual validations and swath plots demonstrate the OK parent and LUC SMU estimates are consistent with the input drillhole composite data and are believed to constitute a good representation of the respective domains of mineralization.

14.17 Mineral Resource Classification

The Mineral Resource estimate was classified as Measured, Indicated and Inferred and reported using the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines, CIM November 2019).

Classification was guided by confidence in drillhole data, geological continuity, reliability of the geological model, quality of kriged estimates, positive metallurgical responses and an underground mine design with Mineral Reserves. Specifically, Mr Carmelo Gomez Dominguez (QP) considered the following criteria during classification:

- Verification of the tenement title
- Positive metallurgical processing results as tested and verified via pilot plant studies

An underground mine design of long hole and cut and fill stoping defining Mineral Reserves was used to guide the position of Mineral Resource classification

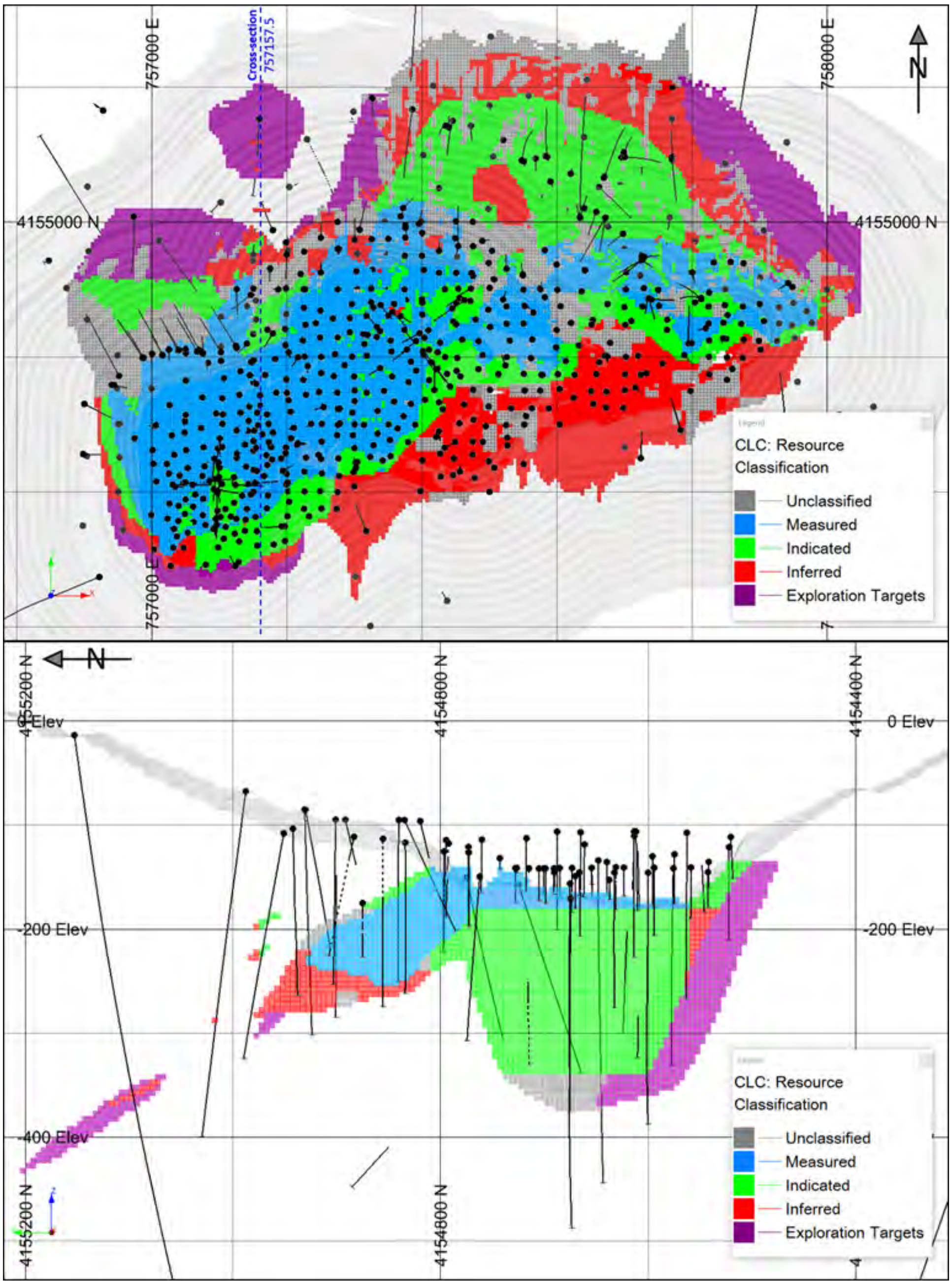
- Drilling, sampling, and geological process, standards and systems, as managed and implemented by Mr Escobar Torres and Mr Gomez Dominguez
- Good geological evidence for strong continuity of mineralization at the cut-off grade
- Drill core and some open pit exposure as evidence supporting the sample analytical values for copper, lead, zinc, silver and gold mineralization
- Good QAQC controls with results that verify robust sampling practices and analysis of the copper, lead, zinc, silver and gold grades used
- Good core sampling to determine dry in-situ bulk density for estimating density and supporting tonnages of mineralization
- Safe, secure databases providing validated diamond core logging, survey and sample analysis data
- Ordinary kriging slope of regression and kriging efficiency values that are indicative of sound confidence in the grade estimates.

Classification (Figure 14-8) was primarily based upon Kriging confidence, drillhole data and sample spacing, confidence in geological continuity and the potential for economic extraction:

- Measured Mineral Resources were deemed appropriate in areas where the drill grid spacing was less than 25 m. Kriging efficiency was greater than 80% and regression slope values were greater than 0.8.
- Indicated Mineral Resources were assigned to block estimates where the drill grid was between 25 m to 75 m, where the kriging efficiency was between 60 % to 80% and where regression slope values were greater than 0.6.
- Block estimates that did not meet the Measured or Indicated criteria and that were within 100 m of a single drillhole with geological continuity, were assigned to the inferred category. Typically, Inferred block estimates had a kriging efficiency greater than 40% and regression slope value greater than 0.4.
- In addition to the above criteria for Measured, Indicated and Inferred Mineral Resources, some blocks were removed from the Mineral Resources classification after further considering their potential for economic extraction:
  - Blocks within the Stockwork domain with copper grades below 0.30% TCu or located below the -350 m elevation.
  - Blocks in the SMS domain sitting below the -335 m elevation.
  - Blocks in the Gossan Domain located external to the optimized pit shell.

In summary, PPS domains are classified as Measured, Indicated and Inferred Mineral Resources. Secondary Sulphide domains are classified as Measured and Indicated Mineral Resources. Gossan is classified as Indicated Mineral Resources.

Figure 14-8 A Plan view (top) and a north-south cross-section (bottom) of Mineral Resource classification of PPS Mineralization domains with diamond drillholes location and exploration target areas.



14.18 Mineral Resource Reporting

The updated Mineral Resource estimate for Cobre Las Cruces, depleted as at 30th September 2023 is presented in Table 14-12 below. Polymetallic Primary Sulphides (PPS), including PMS, SMS and Stockwork, were reported at a copper equivalent grade cutoff of 1%. The copper equivalent formula is:

- $Cu_{eq}\% = [TCu\% + (Zn\% \times 0.333) + (Pb\% \times 0.221) + (Ag\ ppm \times 0.005)]$

The copper equivalent formula uses recent 2023 consensus metal prices, expected metallurgical recoveries and amounts payable by the smelter:

- Metal prices – Cu (3.77$/lb), Zn (1.21$/lb), Pb (0.94$/lb) and Ag (22.37/oz)
- Metal recoveries – Cu (85.1%), Zn (88.4%), Pb (79.4%) and Ag (51.9%)

- Smelter metal payable – Cu (100%), Zn (100%), Pb (95%) and Ag (95%)

The secondary sulphide domains, DZ, HC and HC4 have used a total copper cut-off of 1%. Gossan zone gold resources were reported using a 1 ppm gold cut-off grade.

There are no known factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which could materially affect this Mineral Resource statement.

Table 14-12 Detailed CLC Mineral Resource statement as at 30th September 2023. Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Mineral Resource statement reported at respective Material type cutoff grades as at 30th September 2023												
Classification	**Tonnes (Mt)**	**Grade**						**Metal**				
		Cu_{eq}(%)	**Cu (%)**	**Au (g/t)**	**Zn (%)**	**Pb (%)**	**Ag (g/t)**	**Cu (kt)**	**Au (koz)**	**Zn (kt)**	**Pb (kt)**	**Ag (Moz)**
Polymetallic Primary Sulphides (0.8% Cu_{eq} cutoff grade*)												
Measured	20.0	2.62	1.21	-	2.92	1.29	31.7	241		583	257	20.3
Indicated	21.4	1.97	1.13	-	1.65	0.79	23.4	242		353	169	16.1
Measured + Indicated	**41.4**	**2.29**	**1.17**	**-**	**2.26**	**1.03**	**27.4**	**484**		**936**	**427**	**36.5**
Inferred	9.4	1.66	1.08	-	0.94	0.61	26.1	1		1	1	7.9
Secondary Sulphides (1.0% Cu cutoff grade)												
Measured	0.9	-	6.23	-	-	-	-	0.54				
Indicated	0.1	-	2.51	-	-	-	-	0.02				
Measured + Indicated	**0.9**	**-**	**6.01**	**-**	**-**	**-**	**-**	**0.55**				
Inferred	-	-	-	-	-	-	-					
Gossan (1.0 g/t Au cutoff grade)**												
Measured	-	-	-	-	-	-	-					
Indicated	0.01	-	-	1.54	-	0.833	100.5		0.50		0.08	0.03
Measured + Indicated	**0.01**	**-**	**-**	**1.54**	**-**	**0.83**	**100.5**		**0.50**		**0.08**	**0.03**
Inferred	-	-	-	-	-	-	-					
*Resource estimates for the Polymetallic Primary Sulphides are reported on a cutoff grade of **0.8 % copper equivalent (Cu_{eq})**, based upon the following formula which accounts for metal price ($3.77/lb copper, $1.21/lb zinc, $0.94/lb lead and $22.37/lb silver), metallurgical recoveries and amounts payable by the smelter. Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability												
Cu_{eq} = [Cu% + (Zn% x 0.333) + (Pb% x 0.221) + (Ag g/t x 0.005)]												
**** Gossan is only reported within the optimized pitshell of the previous open pit project**												

Figure 14-9 **Grade tonnage curve for PPS Measured and Indicated Mineral Resources, depleted of mined material as at 30th September 2023.**

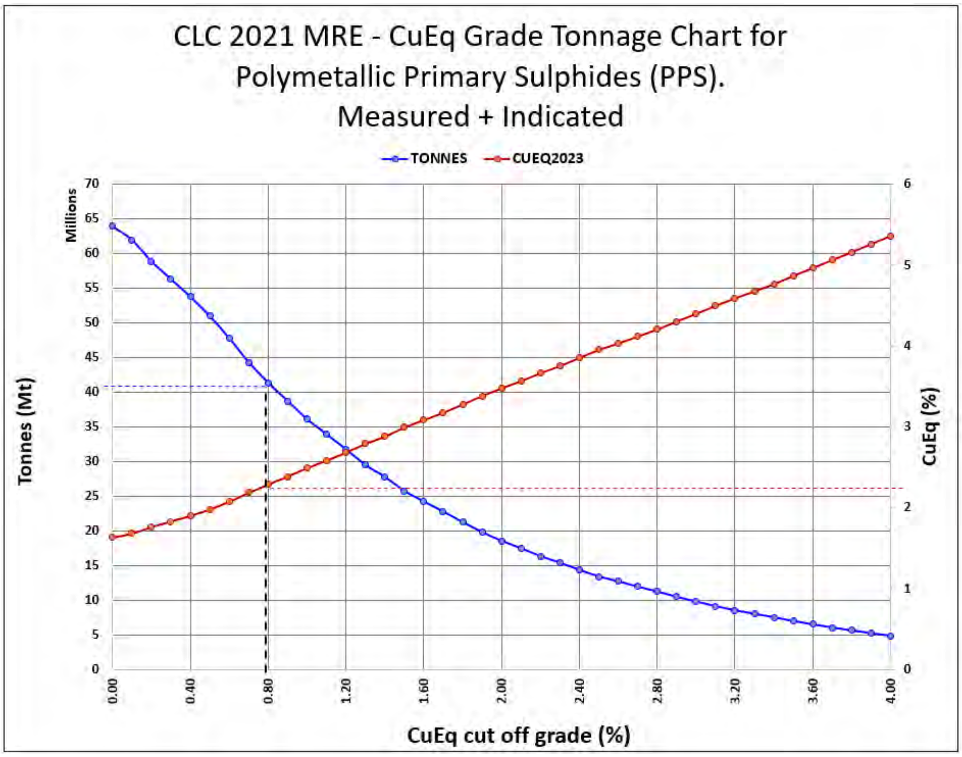


Table 14-13 **Grade tonnage curve data for PPS Measured and Indicated Mineral Resources, depleted of mined material as at 30th September 2023.**

Cutoff (Cu_{eq} %)	Cu_{eq} (%)	Volume (Mm^3)	Density (t/m^3)	Tonnes (Mt)	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)
0.50	1.98	13.46	3.79	50.97	1.02	1.92	0.88	24.06	0.41
0.60	2.07	12.55	3.81	47.78	1.07	2.03	0.92	25.09	0.42
0.70	2.18	11.58	3.83	44.33	1.12	2.15	0.98	26.31	0.43
0.80	2.29	10.74	3.85	41.38	1.17	2.26	1.03	27.37	0.44
0.90	2.39	9.99	3.88	38.73	1.22	2.37	1.08	28.27	0.45
1.00	2.49	9.26	3.90	36.11	1.26	2.50	1.14	29.47	0.45
1.10	2.58	8.67	3.92	33.94	1.30	2.61	1.19	30.34	0.46

In addition to the unmined Mineral Resources (Table 14-12), several stockpiles were generated during open pit mining of secondary sulphides. Current stockpiles include a stockpile for gossan and a stockpile for Polymetallic Primary Sulphides. Stockpiles are classified as Indicated Mineral Resources (Table 14-14).

Stockpile material was mined from areas with closely spaced grade control drilling. The PPS stockpile grades were derived from the open pit grade control model which used the same samples and grades as the Mineral Resource estimate. The marked-out polygons for mining included dilution and loss of metals with adjacent material types. The PPS stockpile therefore includes materials that are mostly from massive sulphides, but also from semi-massive (4.6%) and stockwork (14.2%) areas. The position of these materials on the PPS stockpile is not recorded. An ore control database details each PPS polygons data with a date, blast identity, material type, tonnes, moisture and grades in support of the assigned stockpile average grades.

The Gossan stockpiles were created during mining phases 1 to 4. The stockpile material used a 1 g/t gold cut-off grade with selective mining during phase 3 to limit dilution and loss. Blast hole sampling was used to guide mining of gossan material for stockpiles with an overall average grade assigned from mark-out polygons.

Table 14-14 CLC Mineral Resource statement of remaining stockpiles as at 30th September 2023.

Classification	Stockpile	Tonnes (Mt)	Grade					Metal				
			Cu (%)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)	Cu (kt)	Zn (kt)	Pb (kt)	Au (Moz)	Ag (Moz)
Indicated	Polymetallic Primary Sulphide (PPS)	5.0	1.19	2.21	1.63	-	-	60	111	82		
	Gossan	2.7	-	-	3.30	2.58	82.3			88	0.2	7.1
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability												

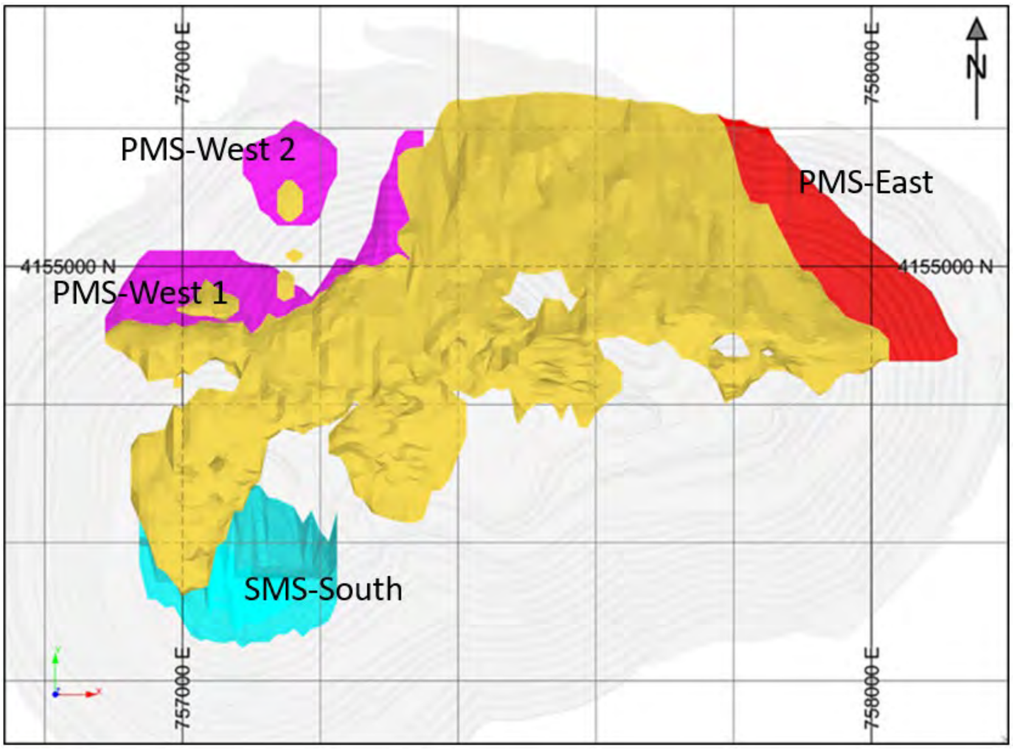
14.19 Exploration Target Mineralization Potential

Exploration target mineralization potential was identified in areas located adjacent to existing Mineral Resources. These areas were defined where immediately adjacent to areas having known geological and grade continuity as well as having geophysical data anomalies and diamond core sample assay data. The disclosed tonnes and grades of the exploration target mineralization potential is conceptual in nature. There is insufficient exploration information to define these as Mineral Resources. As such, it is uncertain if further exploration will result in the disclosed exploration target mineralization potential being defined as a Mineral Resource.

Four areas were identified and coded into the block model. Three of these areas (Figure 14-10) were located close to the existing PMS domain and one was close to the SMS domain. These areas were defined by:

- zones with gaps or wide drillhole spacing but that were near to mineralised intercepts,
- anomalies detected by geophysical data like down-hole EM and seismic profiles,
- proximity and potential for geological continuity/extension to known mineralised domains.

Figure 14-10 Areas of exploration target mineralization potential



The exploration target potential for each of these areas was evaluated from the nearest Mineral Resources estimates which were extrapolated into the exploration target volumes with a ±30% error range, as shown in Table 14-15.

Table 14-15 Summary of exploration target mineralization potential areas

Area	Volume (Mm^3)	Density (t/m^3)	Tonnes (Mt)	TCu (%)	Zn (%)	Pb (%)	Au (ppm)	Ag (ppm)
PMS-WEST_1	0.7 - 1.3	4 - 4.3	3.0 - 5.6	0.6 - 1.2	2.2 - 4.1	0.8 - 1.6	0.3 - 0.5	20 - 40
PMS-WEST_2	0.2 - 0.3	4 - 4.3	0.6 - 1.3	0.6 - 1.2	2.2 - 4.1	0.8 - 1.6	0.3 - 0.5	20 - 40
PMS-EAST	0.2 - 0.3	3.6 - 3.8	0.6 - 1.3	0.3 - 0.6	0.2 - 0.3	0.1 - 0.1	0.2 - 0.4	4 - 8
SMS-SOUTH	1.9 - 3.5	3.2 - 3.4	6.3 - 11.8	1.1 - 2.0	0.2 - 0.3	0.1 - 0.2	0.2 - 0.3	6 - 12
Total	**3.0 - 5.4**	**3.5 - 3.7**	**10.5 - 20**	**0.9 - 1.6**	**0.9 - 1.6**	**0.3 - 0.7**	**0.2 - 0.4**	**10 - 20**

14.20 Comparison with Previous Mineral Resource Estimate

The CLC 2021 Mineral Resource estimate reflects additional diamond drilling data and improved geological models from drill core logging and pit mapping. The 2023 reported figures included in this report, reflect an update on the copper equivalent formula as well as to the reporting copper equivalent cutoff grade used.

A comparison with the June 2015 Mineral Resource estimate has focused on the Polymetallic Primary Sulphides (PPS). For comparison purposes, all models were depleted as at 30th September 2023. Models from 2015 and 2022 are using the 2015 copper equivalent formula and are reported at the same copper equivalent cutoff grade of 1.0% Cu_{eq} (Table 14-16 and Table 14-17), whereas the 2023 figures are reported using the 2023 updated copper equivalent formula and copper equivalent cutoff grade of 0.8% Cu_{eq} (Table 14-18).

The stepwise changes are shown in the waterfall chart included below (Figure 14-11). In summary:

- The updated estimate has upgraded Inferred Mineral Resources to Indicated and Measured Mineral Resource categories. Only Inferred Mineral Resources within the Stockwork domain remain at the same classification. 41.38Mt of Measured and Indicated PPS Mineral Resource is available for conversion into Mineral Reserves.
- The increase in Measured and Indicated Mineral Resources is due to the addition of 229 diamond drillhole data located across the PMS and SMS domains.
- The added drilling has resulted in Inferred Mineral Resources located along the edges of the domains. 9.43Mtof Inferred Mineral Resource is tabled.

Overall, M+I+I tonnage has increased by 54%, total copper grade has increased by 1.1%, from 1.14% TCu to 1.15% TCu, and copper equivalent has decreased by 7.5%. Zinc and lead have decreased by 25.6% and 24.5% respectively. The changes are aligned with the previous estimates Inferred classification together with the additional data and improved geology.

Table 14-16 May 2015 Mineral Resource estimate depleted as of 2023, using 2015 copper equivalent formula and reported at 1.0% Cu_{eq} cutoff

Mineralization Style	Classification	Tonnes (Mt)	Cu (%)	Cu_{eq} (%)	Zn (%)	Pb (%)
Polymetallic Primary Sulphides (PPS)	Measured	-	-	-	-	-
	Indicated	-	-	-	-	-
	Meas + Ind.	-	-	-	-	-
	Inferred	33.0	1.14	2.35	2.71	1.26
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability						

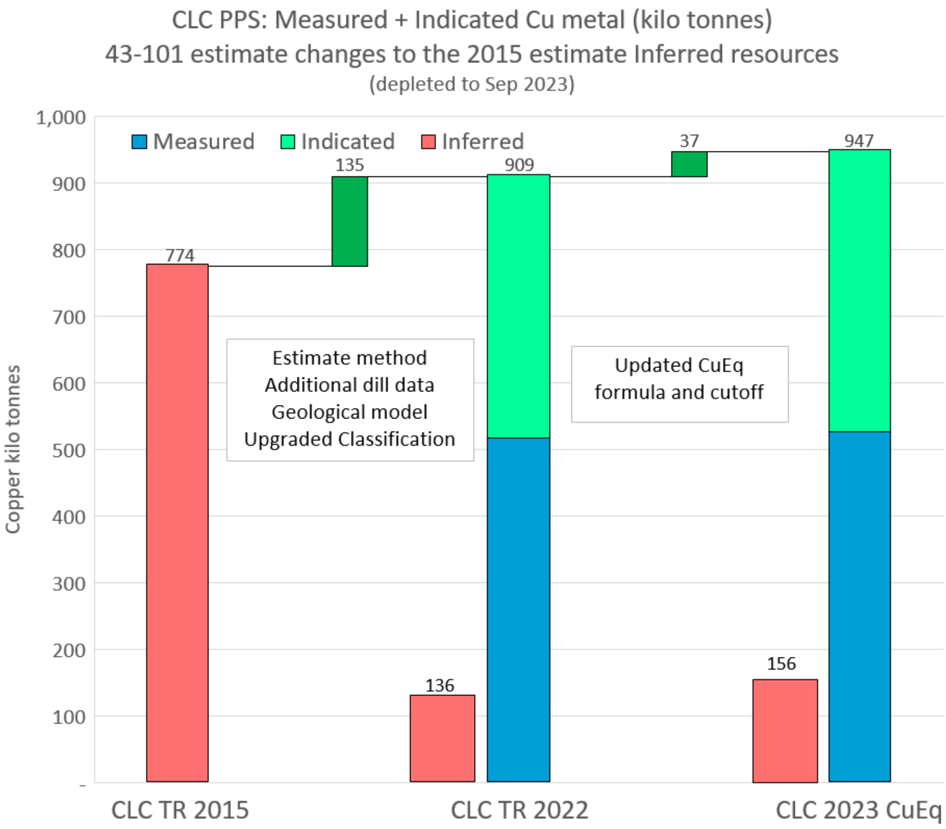
Table 14-17 January 2022 Mineral Resource estimate depleted as of 2023, using 2015 copper equivalent formula and reported at 1.0% Cu_{eq}

Mineralization Style	Classification	Tonnes (Mt)	Cu (%)	Cu_{eq} (%)	Zn (%)	Pb (%)
Polymetallic Primary Sulphides (PPS)	Measured	18.3	1.27	2.80	3.11	1.37
	Indicated	17.9	1.24	2.20	1.86	0.89
	Meas + Ind.	36.3	1.26	2.51	2.49	1.13
	Inferred	7.1	1.23	1.93	1.12	0.73
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability						

Table 14-18 2023 Mineral Resource estimate depleted as of 30th September 2023, using 2023 copper equivalent formula and reported at 0.8% Cu_{eq} cutoff

Mineralization Style	Classification	Tonnes (Mt)	Cu (%)	Cu_{eq} (%)	Zn (%)	Pb (%)
Polymetallic Primary Sulphides (PPS)	Measured	19.96	1.21	2.62	2.92	1.29
	Indicated	21.42	1.13	1.97	1.65	0.79
	Meas + Ind.	41.38	1.17	2.29	2.26	1.03
	Inferred	9.43	1.08	1.66	0.94	0.61
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability						

Figure 14-11 Waterfall chart showing changes to the 2015 estimate



ITEM 15 MINERAL RESERVE ESTIMATES

15.1 Methodology

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate has followed a conventional approach, commencing with stope optimisation techniques incorporating economic parameters and other Modifying Factors. The shapes developed during the stope optimisation process were then used to develop practical stope designs whilst a development design was then created to provide access to the stopes.

The development design took into account operational factors including;

- Initial decline development through the upper clay (marl) layer.
- Utilising a bottom-up approach to maximise mining recovery.
- Ore transport using a combination of Railveyor and underground haul trucks.
- Ventilation requirements.

The combined development and stope design then provided the level-by-level ore and waste mining inventories for the detailed production schedule that demonstrates viable underground mining. This schedule, which in turn provides the physical basis for cash flow modelling, is described in detail in Item 16.10.

15.2 Cut-off Value

A Net Value (NV) cut-off, derived from a break-even style calculation, was employed to define that part of the Mineral Resource eligible for conversion to a Mineral Reserve. Effectively, net value is the expected unit Net Smelter Return (NSR) less the mining, processing and general and administrative (G&A) costs.

The NSR for the PMR Project was calculated considering copper, lead, zinc and silver grades, and using metal prices, payable metal rates, metallurgical recoveries, and realisation costs. This calculation was applied to all blocks in the Mineral Resource block model to develop an NSR value model. An itemisation of the parameters used to develop the NSR value is presented in Table 15-1 and Table 15-2.

Table 15-1 NSR Model Parameter Description

ELEMENT	DESCRIPTION	UNIT
Cu, Zn, Pb, Ag	In-situ element grade	% (Cu, Zn, Pb) ; g/t (Ag)
RecElement	Element metallurgical recovery	%
PagElement	Fraction of metal payable	% of total metal
PrElement	Selling price of element	$/selling unit
Rec	Mining recovery of ore	%

Separate process recoveries were defined for each metal and are shown with selling prices and payabilities in Table 15-2.

Table 15-2 Element Selling Unit Prices, payable and recoveries

ELEMENT	SELLING PRICE	UNIT	PAYABLE (%)	PMS PROCESS RECOVERY (%)	STW PROCESS RECOVERY (%)
Cu	3.77	$/lb.	100	85.12	89
Zn	1.21	$/lb.	100	88.41	89
Pb	0.94	$/lb.	90	79.38	83
Ag	21.37	$/oz	90	51.88	63

The input costs for the NSR calculation and MSO are summarised in Table 15-3. Mining cost variables are the mining method, haulage distances, and mine development. DAF cost per tonne mined is approximately 20% higher than the LHOS mining cost.

An average (fixed and variable) process cost of $50.60/t was applied to all mineralisation apart from the high grade secondary ore (HC4), which had a processing cost of $40.44/t applied. For the purposes of the NSR calculation and MSO, sustaining capital was included in the processing costs.

Table 15-3 NSR Calculation – Cost Inputs

Cost Item	Mining Method	
Mining	**LHOS**	**DAF**
Mining Cost USD/t	21.96	28.16
Process USD/t	50.60	
Process (HC4) USD/t	40.44	
G & A USD/t	7.26	3.93

As a result of applying the above input parameters to the NSR calculation, the NSR cut-off values shown in Table 15-4 were calculated.

Table 15-4 NSR Cut-off (USD/t) by Mining Method and Ore Type

MINING METHOD	PRIMARY ORE	SECONDARY ORE
LHOS	78.42	-
DAF	81.24	71.25

15.3 Dilution and Recovery Estimates

In the evaluation of the Mineral Reserve, modifying factors were applied to both tonnages and grade of the designed mining shapes (LHOS, DAF production and ore development) to account for ore dilution and losses that are experienced at all mining operations.

Ore dilution includes over break into the hangingwall and footwall and into adjacent backfilled stopes. Potential diluting materials include, waste rock, Inferred Resource volume, and also backfill material. All dilutants have been assumed to carry no metal content in the estimation of the diluted Mineral Reserve grades.

Ore losses (i.e., mining recovery factor) are related to the practicalities of extracting ore under varying conditions, including difficult mining geometry, problematic rock conditions, losses in fill, and blasting issues.

The weight-basis factors used to generate the diluted and recovered ore tonnage are listed in Table 15-5.

Table 15-5 Mining Dilution and Recovery per Mining Method

MINING METHOD	DILUTION FACTOR (%)	RECOVERY FACTOR (%)
LHOS – T (primaries)	2	85
LHOS – T (secondaries)	14	85
LHOS – L	12	83
DAF	5	95
Ore development	0	100

The dilution factors were calculated from standard over break assumptions that were considered appropriate based on benchmarking of similar nearby underground mining operations utilising the LHOS and DAF methods.

15.4 Mining Shapes

15.4.1 Design Process

The Mineable Shape Optimizer (MSO) from the Deswik.SO mine planning software package has been used to produce sublevel LHOS mining shapes meeting both NSR cut-off and operational design criteria. The shapes obtained have been reviewed and manually re-designed to adjust them to ore boundaries.

For the design of DAF areas, iso-grade polygons have been first generated at the selected NSR cut-off value on a 5m vertical interval (matching the ore drive height). These boundary polygons were then manually adjusted to maintain an adequate standoff distance to both topographic surface and adjacent LHOS stopes. Once the mineable extents of the level were defined, design of both ore drives and access infrastructure was manually conducted.

The design criteria, as listed in Table 15-6, constrained the geometry of planned excavations to that which is achievable through the planned mining methods and according to geotechnical conditions. In both design cases and where required, the shapes were individually refined to minimize the amount of sub-economic material within the shape volume that is inseparable from profitable material due to the practical constraints of mining.

Table 15-6 Underground Design Parameters

Description	Width (m)	Length (m)	Height (m)	Diameter (m)
Stope				
LHOS - T	15	30	25	
LHOS - L	up to 20	30		
DAF	5	30	5	
Pillars in DAF	5			
Development				
Decline, ramp	6		6	-
Transport level	5.5		5.5	-
Workshop	6		6	-
Other ancillary infrastructure	5		5	-
Ore pass *	4	4	-	4
Vent raises	-		-	2.5 - 3.5
Paste fill line borehole	-		-	0.2

**Depending on raise method (long hole raise = 4m x4m; raise bore = 4m diameter)*

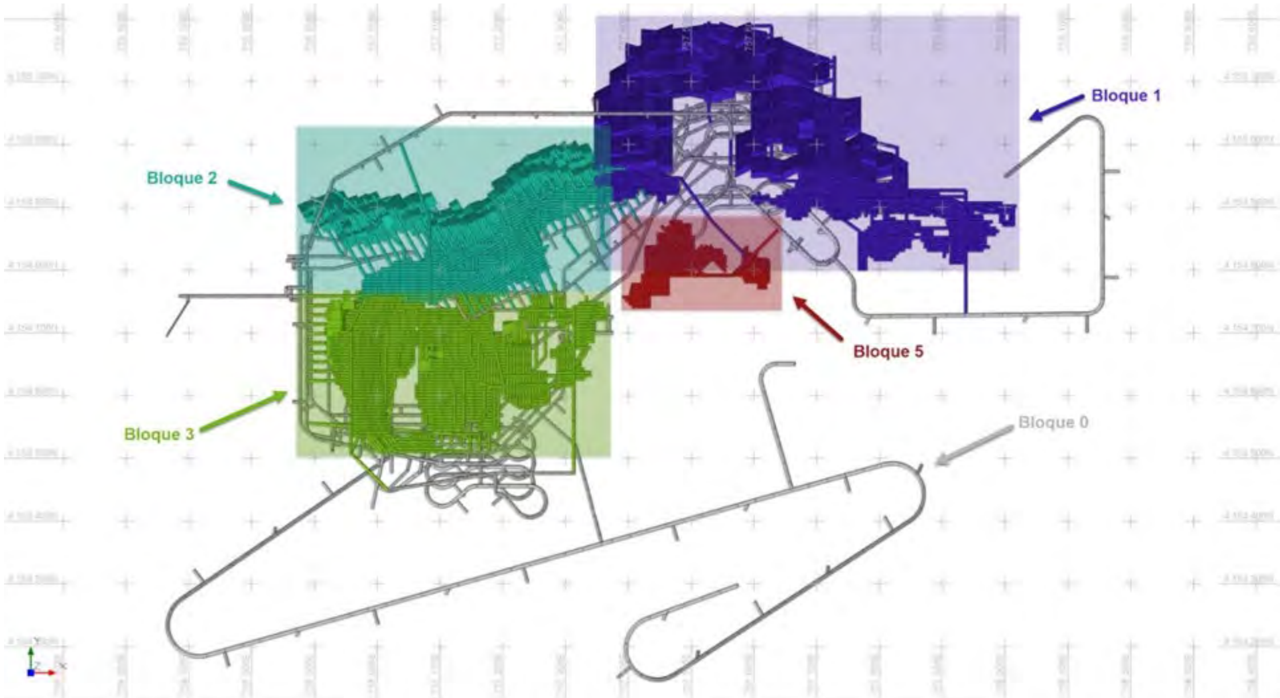
15.4.2 Stope Design Summary

The mining shapes resulting from the stope optimisation and design process were distributed across four discrete mine blocks (Figure 15-1), referred to as “Bloques” (i.e., Blocks). The four blocks are Block 1, Block 2, Block 3 and Block 5. Block 4 was originally used for the old “Satellite Mass” that has since been classified as part of the main primary massive sulphide domain (PMS) and has been included in Block 2[1].

The four zones extend over a 260-metre vertical distance, from - 410 mRL to -151 mRL and contain approximately 36.5 Mt of Mineral Resource at an average grade of 1.14% Cu, 1.00% Pb, 2.23% Zn and 26.00 g/t Ag.

[1] Block 0 is ore mined during pre-production underground development

Figure 15-1 CLC Underground Mining Zones - Top Plan View



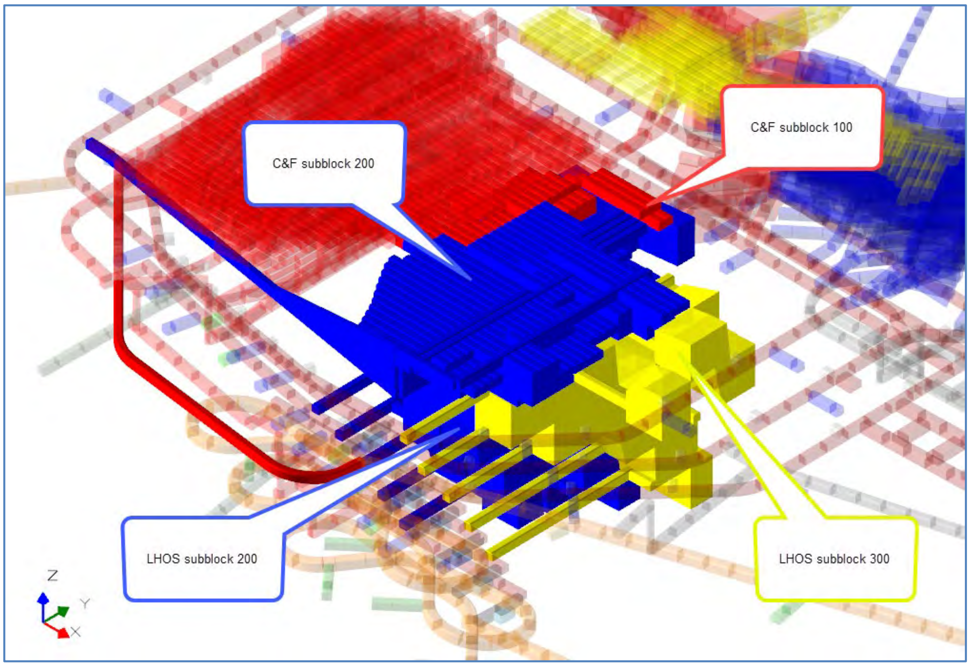
Block 1, or the East zone, is the easternmost part of the massive sulphides known as "La Manta." This block is located at the east end of the deposit and is the deepest delineated mining shape. Drilling results from 2021 indicate there is down dip and lateral extension potential for Block 1. Containing 10.1Mt of ore, Block 1 makes up 28% of the total underground mining inventory, with average grades of 0.91% Cu, 0.99% Pb, 1.98% Zn, and 29.24ppm Ag. The average NSR of Block 1 is $140/t.

Block 2 is the North-West area. Block 4, or the "Satellite Mass" discovered in 2018, has since been reclassified and included in Block 2, which is the highest value block (NSR $188/t) and has been prioritized in the mine schedule during the initial production years. Drill hole intersections in this area show a potential down dip extension of Block 2 ore which will be targeted in upcoming drilling campaigns.

Block 2 totals 11.7Mt, 32% of the total underground Reserve, with an average grade of 1.12% Cu, 1.28% Pb, 3.13% Zn and 30.41ppm Ag. It is noted that Inferred Resource located close to the designed stoping areas could add to the mining inventory, with minor additional development, should grade control drilling result in the Inferred Resource being upgraded to at least Indicated status.

Block 3 is located at the western end of the deposit and consists of the westernmost massive sulphides and a dense copper stockwork sub-vertical pipe known as the "Stockwork West" or "Semi-massive Sulphide". It is subdivided into two separate mining units by elevation. The bottom unit, located between -334 and -170 elevations, will be extracted with transverse LHOS mining. The upper unit, located between -210 and -180 elevations, is the 30m crown pillar required directly below the open pit mine to ensure the safe extraction of the underlying LHOS mining block. The upper unit will be partially recovered by DAF later in the LOM schedule. Block 3 totals 14.2Mt (39% of the mining inventory), with an average grade of 1.30% Cu, 0.71% Pb, 1.48% Zn, and 18.9ppm Ag with an average NSR of $152/t.

Figure 15-2 Isometric View of LHOS and DAF Mining Units within STW zone - Oriented North West



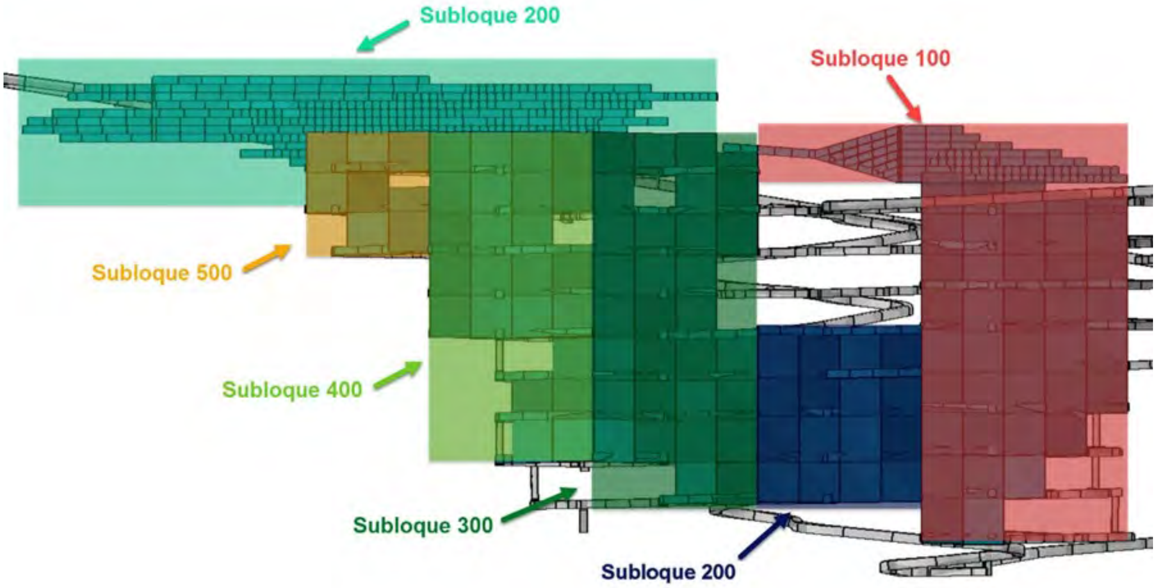
The Block 3 reported tonnage includes some high grade secondary ore from below the final pit floor, referred to as HC4, which contains 0.27Mt of ore (less than 1% of total), and has an average grade of 8.01% Cu, 0.78% Zn, 0.79% Pb and 24.6ppm Ag. HC4 is planned to be mined by DAF between -230 and -150 levels.

Block 5 is a small stockwork type ore zone located in the centre of the mineralised area and corresponds to the extension in depth of the former open pit Phase 5. This block contains 0.5Mt of ore (1% of the total inventory), with an average grade of 0.98% Cu, 0.70% Pb, 1.20% Zn, and 22.8ppm Ag, and with an average NSR of $125/t.

Finally, small quantities of ore will be mined during the development of the mine infrastructure. These have been grouped as Block 0 for scheduling purposes. Block 0 contains only 80kt of ore with an average grade of 0.79% Cu, 0.25% Pb, 0.54% Zn, and 16.5ppm Ag and an average NSR of $83/t.

Each of the four main blocks (1,2,3 &5) are further subdivided into sub-blocks, each with a maximum 100m span. Figure 15-3 shows an example of the sub-blocking in Block 1. This subdivision was introduced to provide a higher flexibility regarding mine sequencing as each sub-block is considered for scheduling purposes to be an independent mining unit.

Figure 15-3 Delineated LHOS Blocks and Sub-blocks within Block 1 Zone - plan view from North.



The mining inventory contained in Block 1 will be mined by Longitudinal LHOS. Blocks 2 and 3 will be mined by transverse LHOS.

Following the geotechnical stability recommendations, all sub-blocks have been vertically divided into sublevels on a 25m interval. Each sublevel in Block 1 was then split along its strike into individual longitudinal stopes with a maximum 30m length.

A crown pillar of 30m below the ultimate open pit surface will be maintained above the designed LHOS shapes. This is scheduled to be partially recovered with DAF mining sub-blocks, once stoping of the underlying LHOS sub-blocks has been completed. Final crown and rib pillars of 5m height will be maintained in these DAF zones to ensure stability of the area.

15.4.3 Reserve Estimation Process

Mineable shapes were designed using a combination of MSO and manual stope and development design. These mineable shapes were defined and reported using the Mineral Resource block models, based on the NSR break even cut-off values and Mineral Resource classifications.

Dilution and mining recovery factors were applied to the mineable shapes with a diluted stope grade calculated for each stope within the design. Stope shapes which had an average NSR value below the appropriate cut-off value were then excluded from the design and scheduling process.

Only Measured and Indicated Resources contained within the scheduled mining stopes were used for the estimation of the Mining Reserve.

15.5 Mineral Reserves Statement

15.5.1 Underground Reserves

The Mineral Reserve estimate for the CLC underground PMR Project has been estimated and classified in accordance with NI 43-101 requirements. The numbers in Table 15-7 are presented on a diluted basis for both mass and grades. The effective date of this Mineral Reserve estimate is 30th September 2023.

Table 15-7 CLC PMR Mining Inventory by Zone and Classification as at 30th September 2023 at \$3.77/lb Cu, \$1.21/lb Zn, \$0.94/lb Pb and \$21.37/oz Ag

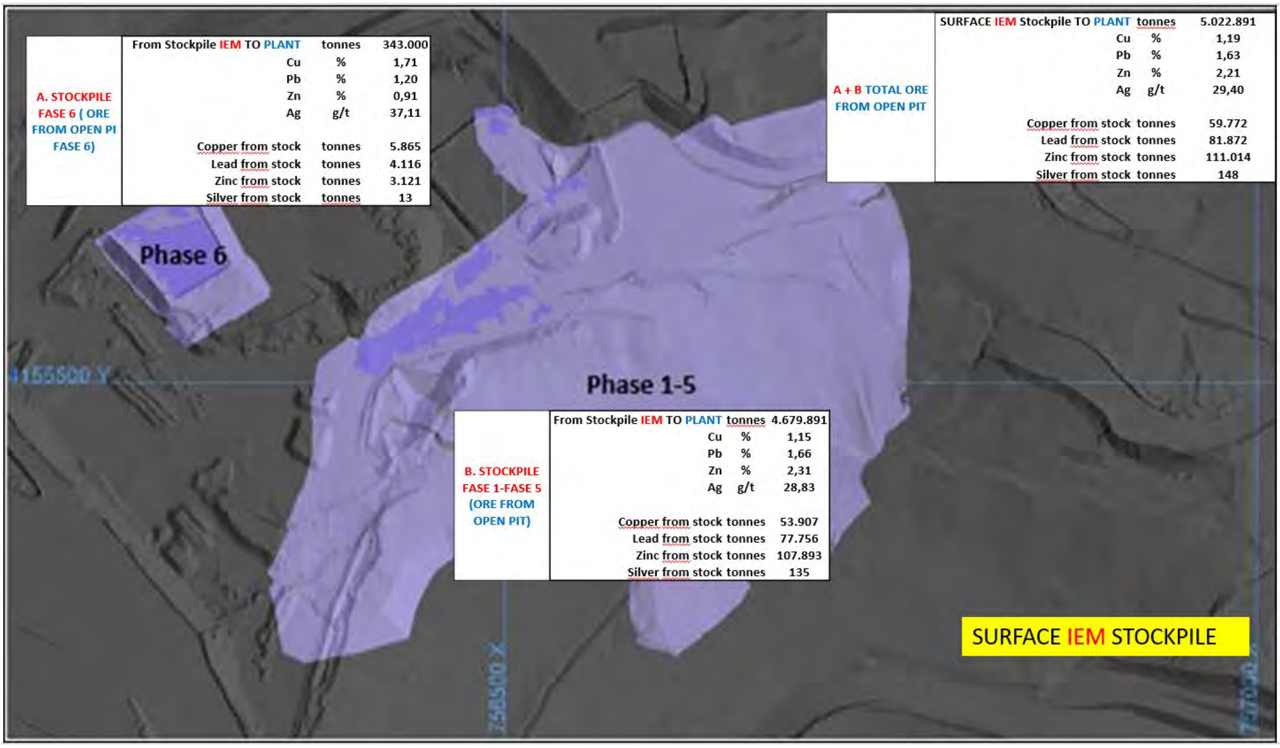
Zone	Class	Mass (kt)	Cu (%)	Pb (%)	Zn (%)	Ag (ppm)
Block 0	Proven	34.8	0.95	0.26	0.62	9.05
	Probable	44.8	0.67	0.24	0.48	22.30
	Total	**79.6**	**0.79**	**0.25**	**0.54**	**16.51**
Block 1	Proven	2,572.3	1.12	0.70	1.59	22.61
	Probable	7,551.6	0.84	1.09	2.11	31.50
	Total	**10,123.9**	**0.91**	**0.99**	**1.98**	**29.24**
Block 2	Proven	8,888.7	1.02	1.32	3.21	29.92
	Probable	2,772.3	1.44	1.15	2.87	31.98
	Total	**11,661.0**	**1.12**	**1.28**	**3.13**	**30.41**
Block 3	Proven	4,383.6	1.72	1.20	2.56	31.76
	Probable	9,806.5	1.11	0.49	1.00	13.09
	Total	**14,190.1**	**1.30**	**0.71**	**1.48**	**18.86**
Block 5	Proven	89.7	0.70	1.59	1.74	30.32
	Probable	401.7	1.04	0.50	1.08	21.12
	Total	**491.4**	**0.98**	**0.70**	**1.20**	**22.80**
Total	Proven	15,969.1	1.23	1.19	2.76	29,20
	Probable	20,577.0	1.05	0.80	1.66	22.57
	Total	**36,546.1**	**1.13**	**0.97**	**2.14**	**25.47**

Numbers may not add up due to rounding

15.5.2 Surface Primary Sulphide Stockpile

Between 2009 and 2021, approximately 5Mt of primary sulphide was mined from the open pit operations. This material has been managed and stockpiled and now constitutes another source of plant feed for the PMR Project. The Total Mineral Reserve table, Table 15-8, includes the tonnage and grade of the primary sulphide ore stockpiled at the CLC Mine Rock Storage Facility (MRSF).

Figure 15-4 Existing Primary Sulphide Stockpiles as at 30 2023



15.5.3 Mineral Reserves Statement

The Total Mineral Reserve estimate for the CLC PMR Project is listed in Table 15-8. The effective date of the Mineral Reserve estimate is September 30, 2023.

The stated Mineral Reserves are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues, to the best knowledge of the QP. There are no known mining, metallurgical, infrastructure, or other factors that materially affect this Mineral Reserve estimate, at this time.

Table 15-8 Mine and Stockpile Mineral Reserve Statement at $3.77/lb Cu, $1.21/lb Zn, $0.94/lb Pb and $21.37/oz Ag

Cobre las Cruces Mineral Reserve Estimate as at 30th September 2023									
		Grade				Contained Metal			
	Mt	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Cu (kt)	Pb (kt)	Zn (kt)	Ag (koz)
Underground (In-situ)									
Proven Reserve	16.0	1.23	1.19	2.76	29.20	196	190	441	150
Probable Reserve	20.6	1.05	0.80	1.66	22.57	216	165	342	149
Total	**36.6**	**1.13**	**0.97**	**2.14**	**25.47**	**413**	**355**	**782**	**299**
Stockpile									
Proven Reserve									
Probable Reserve	5.0	1.19	1.63	2.21	29.40	60	82	111	47
Total	**5.0**	**1.19**	**1.63**	**2.21**	**29.40**	**60**	**82**	**111**	**47**
Combined									
Proven Reserve	16.0	1.23	1.19	2.76	29.20	196	190	441	150
Probable Reserve	25.6	1.08	0.96	1.77	23.85	276	246	453	197
Total	**41.6**	**1.14**	**1.05**	**2.15**	**25.94**	**472**	**437**	**893**	**347**

Numbers may not add up due to rounding.

ITEM 16 MINING METHODS

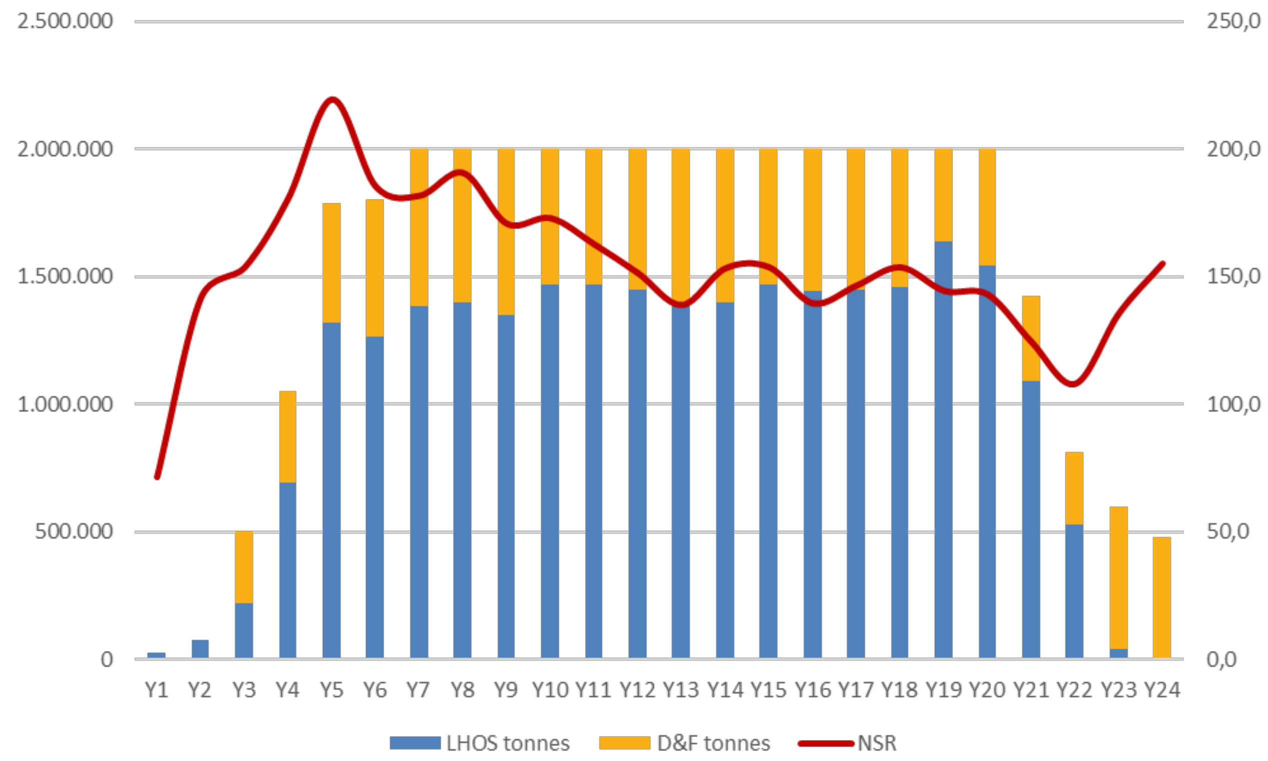
16.1 Introduction

16.1.1 Production Schedule

The Cobre Las Cruces (CLC) underground mine plan produces 2.0Mtpa of polymetallic ore through a combination of transverse and longitudinal Long Hole Open Stoping (LHOS) as well as Drift & Fill (DAF) extraction methods. The primary means of backfilling will be by cemented paste fill, delivered to the underground stope voids through a pipe network, to maximise both mining recovery and productivity.

The yearly production tonnage and grade is shown in Figure 16-1 . The development activity is inclusive of both stoping production and ore development in stoping zones.

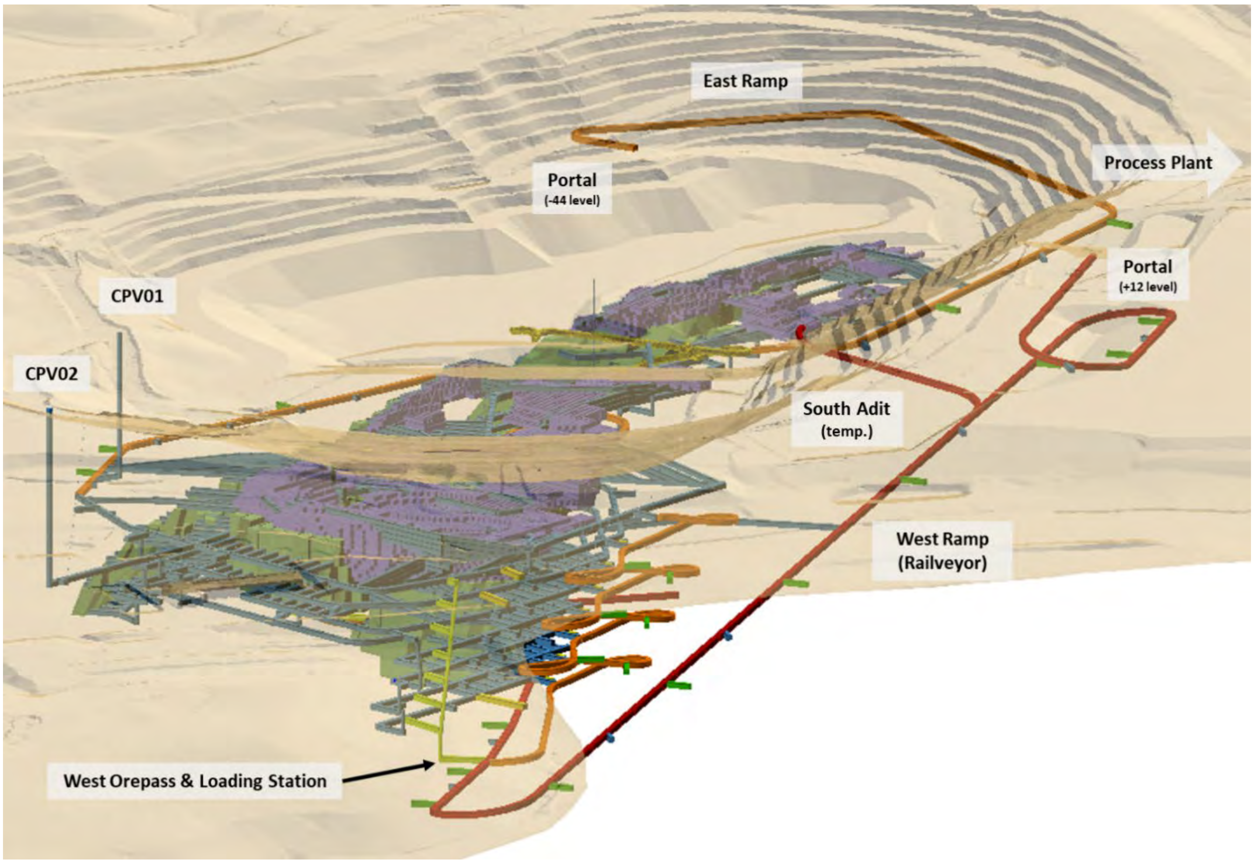
Figure 16-1 LOM Production Schedule by Mining Method (excl. Inferred)



Underground access will be via two declines.

- The East Ramp will descend from a portal located at -44 mRL in the existing open pit and will be used for personnel and equipment access.
- The West Ramp will descend from a portal located at the surface outside of the existing open pit and adjacent to the proposed new processing facilities. This ramp has been designed and will be constructed for the installation of an electric powered ore hauling system, known as a Railveyor, and will also act as an emergency egress route from the mine.

Figure 16-2 CLC Underground Project -View from South-West



A three-year pre-production development program, entailing up to 2,000m per quarter of advance, will be undertaken to both establish the mine infrastructure and provide access to the initial stoping levels. Ongoing development to reach and sustain 2.0Mtpa of ore production, will average approximately 7,500m per year during the first three years of production. The required development rate will decrease during years four and five to an average of 6,000 m per year. It will then ramp up again once mining of the DAF sub-blocks reaches planned capacity.

16.1.2 Mine Elevation Nomenclature

The surface topography around CLC is relatively flat. The original surface prior to open pit mining was characterised by gently rolling hills, with surface elevations ranging from 15 to 45 metres above sea level (ASL). The surface elevation of the proposed western portal is +12 ASL (12 mRL) and the surface elevation of the proposed ventilation fan next to the paste plant is 0 ASL (or 0 mRL). Mining level nomenclature in this Study is in terms of metres Reduced Level (mRL); for example, the 185 Level is -185m RL or 185m below sea level.

16.2 Mining Method and Sequence

16.2.1 Mining Method Selection

The selection of the applicable underground mining methods for the CLC PMR Project has utilised the modified Nicholas method (NM). The NM is a qualitative numerical method of assessing and ranking different mining methods. This method considers factors including:

- the deposit geometry (shape, thickness, and plunge),
- distribution of mineralization, and

- geotechnical attributes.

Each factor is scored following a standard criteria and then weighted to get a final overall score for each of the preferred mining methods. Figure 16-3 and Figure 16-4 summarise the two assessments done during the initial stage of the Project permitting process.

Figure 16-3 shows the NM scores for the massive sulphides (PMS) and stockwork (SMS) mineralization in good to average ground conditions (Geotechnical Domains DG3, DG4, & DG5; refer to Section 16.4).

Figure 16-3 PMR Project Nicholas Method Results – Good to Average Ground Conditions

Geometry	
General Shape	Tabular
Ore Thickness	Thick 30–100m
Ore Plunge	Intermediate 20°–55°
Grade Distribution	Uniform
Rock Substance Strenght	uniaxial strength/overburden Pressure
Hanging Wall	Strong >15
Ore Zone	Strong >15
Footwall	Strong >15
Fracture Frecuency	
Hanging Wall	Very Wide: RQD 70–100
Ore Zone	Wide: RQD 40–70
Footwall	Close: RQD 20–40
Fracture Strenght	
Hanging Wall	Moderate
Ore Zone	Weak
Footwall	Moderate

Scoring		
Method	Nicholas	Modif. Nicholas
Open-pit mining	43	40
Sublevel stoping	30	31
Sublevel caving	31	30
Cut-and-fill stoping	34	30
Square-set stoping	33	28
Block caving	30	27
Shrinkage stoping	28	27
Top slicing	28	27
Room-and-pillar mining	-17	-18

Figure 16-4 shows the NM scores for poor ground conditions (Geotechnical Domains DG1 and DG2; refer to Section 16.4). These ground conditions are expected near fault zones and the crown pillar.

Figure 16-4 PMR Project Nicholas Method Results – Poor Ground Conditions

Geometry	
General Shape	Tabular
Ore Thickness	Intermediate 10–30m
Ore Plunge	Intermediate 20°–55°
Grade Distribution	Uniform
Rock Substance Strenght	uniaxial strength/overburden Pressure
Hanging Wall	Strong >15
Ore Zone	Strong >15
Footwall	Strong >15
Fracture Frecuency	
Hanging Wall	Wide: RQD 40–70
Ore Zone	Close: RQD 20–40
Footwall	Close: RQD 20–40
Fracture Strenght	
Hanging Wall	Weak
Ore Zone	Weak
Footwall	Moderate

Scoring		
Method	Nicholas	Modif. Nicholas
Open-pit mining	40	38
Cut-and-fill stoping	40	35
Square-set stoping	39	34
Block caving	34	29
Room-and-pillar mining	29	30
Top slicing	29	26
Shrinkage stoping	29	27
Sublevel caving	29	28
Sublevel stoping	22	25

In both types of ground conditions considered, open pit mining was the method that received the highest score in the assessment primarily due to the deposit being relatively shallow. However, extension of the current open pit was not considered as an option due to environmental constraints and the need to acquire access to more land for waste rock and tailings storage facilities. As a result, the selected mining methods from the NM assessment were both underground mining methods. LHOS was selected for good to average ground conditions and will be the primary mining method. This will be supplemented by DAF which was selected for areas with poorer ground conditions.

Both methods have been used in the Iberian Pyrite Belt (IPB) for more than 50 years, in both massive sulphide and stockwork deposits. LHOS was first introduced in the 1980´s at the Alfredo Mine by Rio Tinto and has since been implemented in all of the modern underground operations in the region. Cemented paste backfill was introduced in the early 2000´s at the Neves Corvo mine (Portugal) and later adopted at the Aguas Teñidas and Magdalena mines in Spain.

16.2.2 Mining Block Definition

In order to facilitate the establishment of multiple production areas in the mine, the deposit was divided into four main mining zones, Blocks 1, 2, 3 & 5[2], each defined by geometrical and geological parameters. These blocks were then sub-divided into sub-blocks which were then vertically divided into sublevels. Each sublevel contains multiple LHOS and DAF mining shapes. Where the orebody thickness exceeds the maximum stope width, contiguous LHOS stopes have been designed parallel to each other across the thickness.

The use of paste fill means that, except for a crown pillar between the underground development and the existing open pit, no permanent pillars between stopes and/or sub-blocks will be required.

Several factors that impact value and cash flow were considered when defining the block elevations and development access, including;

- **Net Smelter Return (NSR) profile:** Block and sub-block layout impacts accessibility to higher NSR areas over time and hence the design layout assists with achieving a higher NSR earlier in the mine life. This is especially true for the higher-grade NW area of the Main Zone which has been targeted for mining in the early years of the mine-life.
- **Backfill type:** Cemented tailings in the form of paste is the backfill material of choice with curing time required by the cemented material to develop an adequate strength being the determinant for the start of other mining activities in the neighbouring areas. The sub-block layout allows greater sequencing and flexibility, thereby mitigating backfill-related sequencing constraints within smaller stope volumes.
- **Development equipment requirements**: Enabling and sustaining production in the different blocks and sub-blocks requires time and can be resource consuming. This is particularly true in the DAF areas, which rely entirely on the availability of development resources to open mining areas and to sustain production levels. The proposed sub-block division enables a better utilization of development resources over the LOM.

---

[2] A Block 0 is included in the production schedule to distinguish development ore from stoping ore

16.2.3 LHOS Cycle

A two-day delay between each of the activities in the stoping cycle has been considered to cater for common minor operational and equipment availability disruptions derived from normal operational conditions (e.g. shift changes and equipment repositioning).

A standard 35,000 tonne sized stope cycle is shown in Table 16-1. The average production rate of the stope is 495 tpd from the moment the development is completed. This includes drilling, cabling, mucking backfilling and other activities such as slotting and barricading.

Figure 16-5 Task Sequence in LHOS Mining

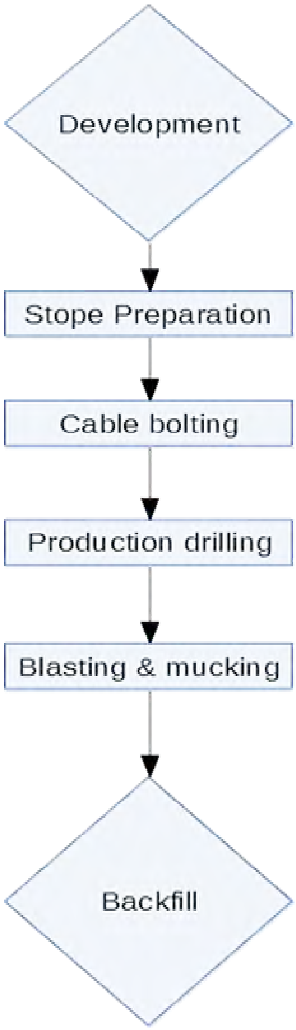


Table 16-1 Standard Stope Cycle

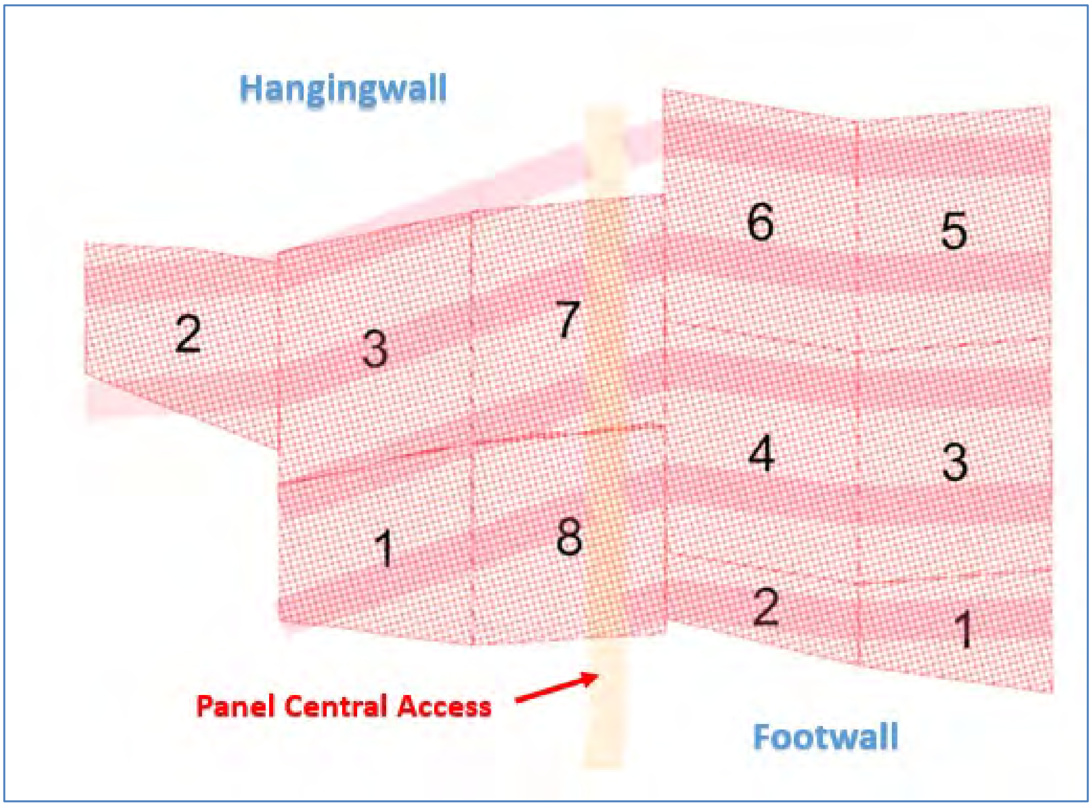
Standard Stope in the Main Zone (35kt)		
Cabling	725	m
	220	m per day
	3.3	days
Drilling	3,555	m
	220	m per day
	16.2	days
Mucking	35,660	t
	1,000	t per day
	35.7	days
Paste	10,550	m^3
	1,000	m^3 per day
	10.6	days
Other Activities & Delays	6	days
Total Stope Cycle	**72.0**	**days**
Average Production Rate	**495**	**t per day**

16.2.4 LHOS Sequence

The overall LHOS mining sequence is bottom-up. Sub-block sequences vary depending on the longitudinal or transverse variant used. The mining sequence in all Longitudinal LHOS sub-blocks will begin from the stopes located furthest from the access cross-cut and closest to the footwall on the first (lowest) level. Once the first stope is mined and the backfill cured, the extraction sequence within the same ore drive will continue with the adjacent stope, proceeding sequentially in retreat towards the cross-cut. A planned curing time of 28 days will be allowed between the end of the backfilling in a stope and the start of the stope preparation activities in the adjacent stope so adequate ground stability can be maintained.

The central stopes in the sub-block (7 and 8) will be maintained as a temporary central pillar to protect the cross-cut access. These stopes will be mined in retreat towards the footwall drive after all the other stopes in the same level and sub-block have been mined and backfilled. A sample stoping sequence is shown in Figure 16-6.

Figure 16-6 Sample Longitudinal LHOS Level Sequence, plan view



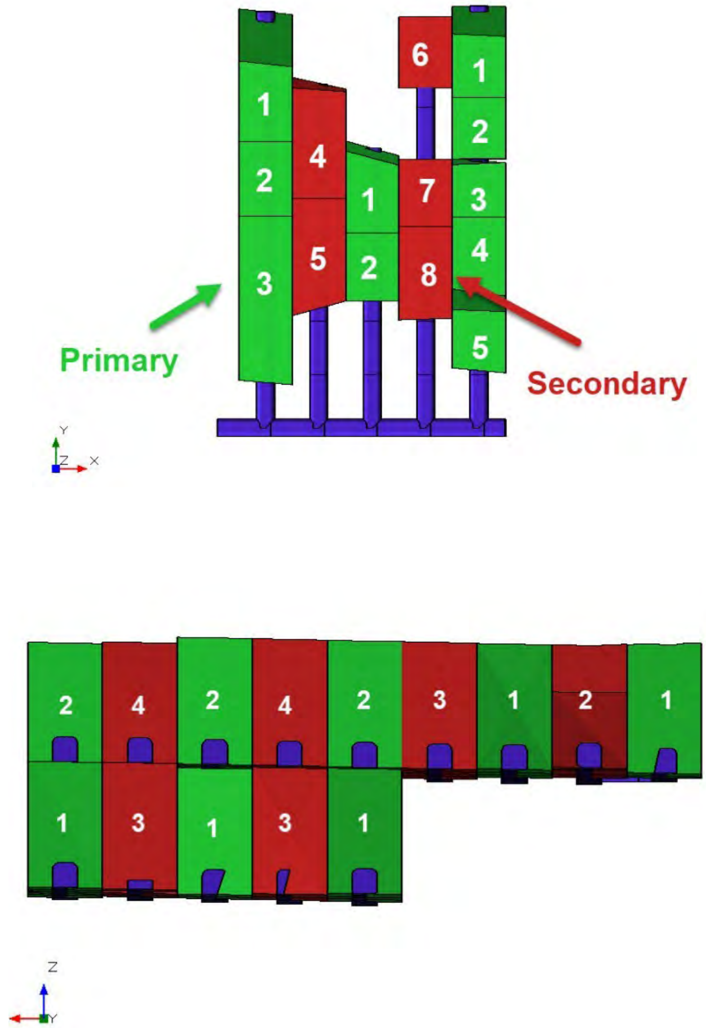
Where the thickness of the orebody allows the layout of contiguous stopes, those in the footwall side will be mined first. Once the stopes on the same drive have been mined and backfilled, the extraction on the next drive towards the hangingwall can begin, as shown Figure 16-6.

It is anticipated that the footwall to hangingwall sequence for Longitudinal LHOS will allow for a better ground support; generally, support cable bolts will be anchored into rock instead of in the cemented backfill, as would be the case with a hangingwall to footwall sequence. Dilution will also be reduced as only stopes at the far end of the orebody will have waste rock on the hangingwall.

Once an entire sub-block level is completely mined and backfilled, stoping can commence on the level above, with the described cycle repeated upwards within the sub-block until the topmost level of the sub-block is completed.

For transverse LHOS blocks, a retreat sequence from hangingwall to footwall will be employed within each sub-block for both safe and productive ore extraction (Figure 16-7). In a vertical profile, the sequence will be ascending pyramidal with primary and secondary stopes.

Figure 16-7 Sample Transverse LHOS Level Blocks sequence – plan & vertical views



A stope of an upper level can be mined after waiting the required 28 days curing time for the backfilled stope immediately below it. Secondary stopes can be mined after their neighbouring primary stopes are backfilled and the 28 days curing period has elapsed. It is also noted that the time allowed before secondary stope backfilling cannot exceed one year. This is to avoid potential issues caused by paste aging in the adjacent primary stopes.

16.2.5 DAF Cycle

The secondary mining method will be standard fully mechanized overhand DAF. This method will be employed to recover ore in the upper parts of the mine, either in crown pillars of LHOS sub-blocks, or in zones not amenable to be extracted by the main mining method due to unfavourable geotechnical conditions.

An access ramp will be constructed from the main infrastructure to the level of the base-lift in the DAF sub-block. A central access cross-cut will then be developed to the far end of the level footprint. Ore drives will then be perpendicularly developed from both sides of the cross-cut, extending to the mineable ore boundary limit.

Once completed, the drives will be backfilled with cemented paste fill. A seven-day delay between the completion of an ore drive backfill task and its successor start will be maintained to allow for adequate curing of the cemented fill material.

An average production rate for a typical 60m length drift is planned at 194tpd. Multiple production areas, each composed of up to eight contiguous headings, will be simultaneously mined in each lift, so that productivity in the lift can be maximised. Once a lift is completely mined and backfilled, the ramp backs will be stripped to take the next lift and the already described cycle commenced again.

16.2.6 Backfilling

The primary means of backfilling will be by cemented paste fill, delivered to the underground stope voids through a pipe network. This has several advantages over other backfill systems including:

- Better quality control of backfill placement: production of the paste material can be controlled and automated, so batches are always prepared to the required specifications.
- Reduction of underground traffic: since the material is delivered through a pipe system potential traffic congestion is avoided or reduced.
- Process tailings management: a portion of the processing tailings will be used to produce the paste fill material, thus reducing the size needed for a surface Tailings Storage Facility (TSF) and also minimizing an environmental impact.

Two cemented paste fill qualities will be utilized and will be produced from unclassified plant tailings mixed with 6% and 3% cementitious binder proportions to produce high and low strength quality backfill material, respectively. Overall, it is anticipated that 66% of the total paste production will be constituted by 6% binder content paste with an average of 5% cement content for the total backfill.

Laboratory testwork conducted by Golder Associates (2016 – 2018) determined that the optimal paste production recipe is flotation tailing fines mixed with a 5% proportion of cementitious binder. The Golder testwork indicates that the proposed recipe would meet the required seven-day, fourteen-day and twenty-eight -day strength criteria, as well as having the adequate rheology to flow through the paste distribution system.

16.2.7 Waste Management and Stope Filling

Waste rock will need to be disposed of throughout the mine life. Some of this waste, up to 300kt, may be used for stope backfilling in order to reduce haulage costs although no provision has been made in the production schedule to this effect. All the waste is scheduled to be transported to the in-pit facility. Unused headings in mined-out areas and / or nominated stockpile bays could be employed as temporary waste storages for this purpose. Nevertheless, most of the waste that will be produced in the early years of the LOM will need to be hauled to the surface and placed in the in-pit waste storage facility.

Table 16-2 lists the mass of waste rock to be mined each year from development headings over the LOM.

Table 16-2 Waste Rock Production over LOM

YEAR	WASTE ROCK MASS (Kt)
1	730
2	646
3	686
4	384
5	316
6	99
7	73
8	62
9	89
10	79
11	90
12	73
13	78
14	62
15	37
16	18
17	54
18	36
19	42
20	24
21	44
22	13
23	8
24	0.3
Total	**3,742**

16.3 Mining Design

16.3.1 Access Ramp and Infrastructure

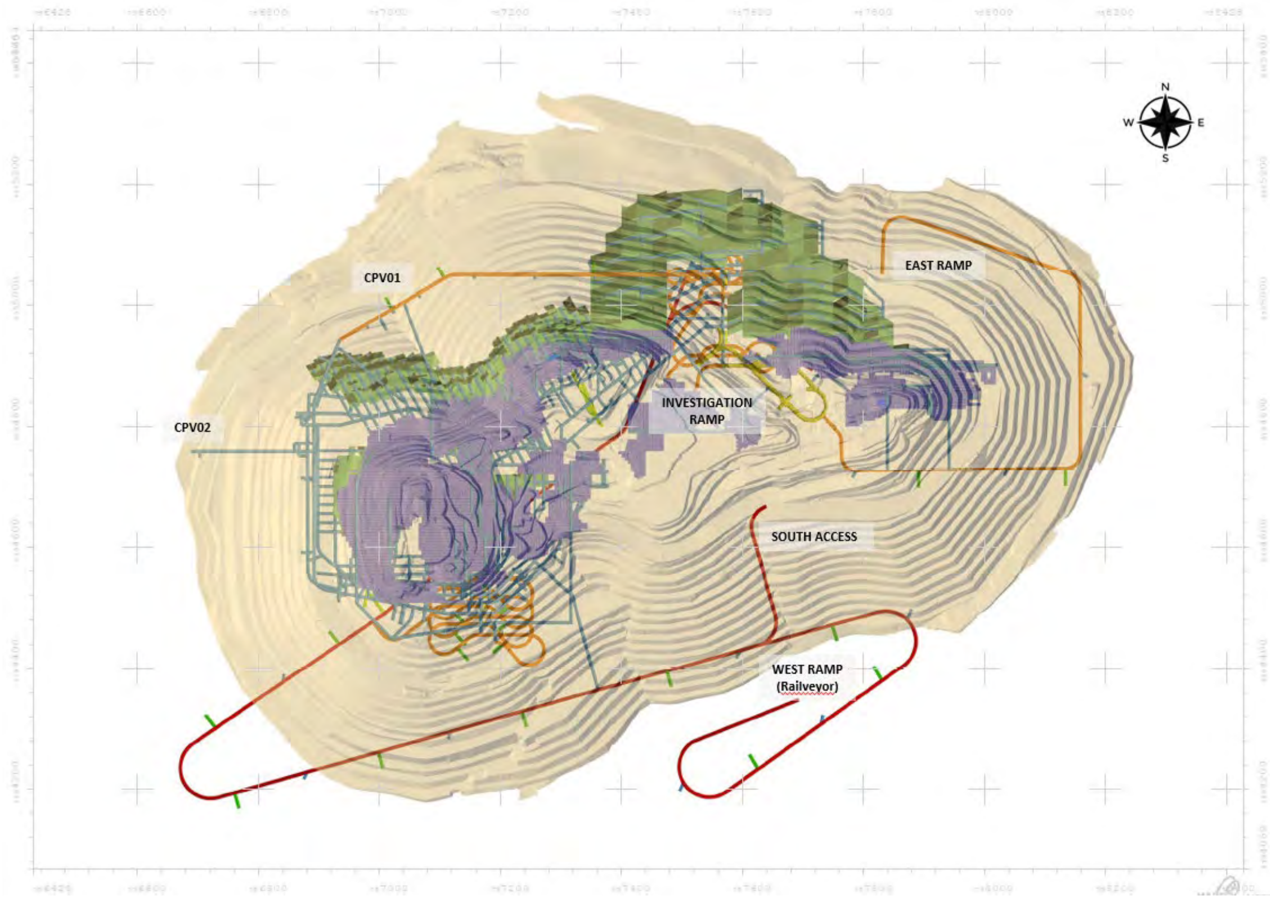
Two access declines will be constructed as follows (refer to Figure 16-8):

- The East Ramp, with a portal located in the easternmost part of the pit at –44 mRL, will provide access for mine equipment, personnel vehicles, mine services, and ventilation.
- The West ramp, with a portal located on the surface, at +12mRL, south of the pit and close to the new process plant. This ramp will provide ventilation and access for the ore transport system (the Railveyor) and will also become the second means of egress in case of emergency.

A third (internal) haulage ramp will connect the West zone deep levels and the Railveyor West loading station at the 335 level with the main 185 level.

The decline development plan enables the establishment of a primary ventilation circuit with no intermediate ventilation shafts required until the main ventilation infrastructure is developed.

Figure 16-8 Decline Location in Plan View



The upper section of both ramps will cross the soft rock (marl) covering the Paleozoic rock formation that hosts the CLC deposit. A double advance face approach will be used to accelerate the 1,500m of development in the marls including;

- **East Ramp**. A second face upwards will be developed from the existing Investigation Ramp to complete the East Ramp upper section. Once it is completed, the existing Investigation Ramp portal at -128 mRL will be sealed to close entry from the pit and complete the in-pit tailings facility construction.
- **West Ramp**. The 200m long South Access will be opened at -110 mRL to reach an intermediate point of the main decline. In this way mine deepening will be independent of the soft rock (Marls) section of development.

The main infrastructure, including declines, ventilation and other mine services, has been designed considering the following criteria:

- Maximum gradient of 12.5% for personnel and equipment ramps, with intersections at 7.5% maximum.
- Maximum gradient of 15% for the Railveyor ramp (West Ramp), with intersections at a 7.5% maximum.
- Minimum curve radius of 25m (East Ramp) and 50m (West Ramp).

- 20m long loading bays spaced at 500m apart along both declines.
- Dewatering sumps spaced at 500m apart along both declines.
- A central main pumping station located at pit bottom, with two temporary stations located in the east and west areas of the mine.
- Transport ramps designed with a figure eight layout, to reduce wear and tear on tyres and mechanical parts of vehicles
- Ventilation shafts accessible at all times from both top and bottom.
- Independent ventilation sub circuits for the east and west parts of the mine.

Details on the layout of the main infrastructure and ancillary excavations can be seen in Figure 16-8 and Figure 16-9.

Figure 16-9 Main and Ancillary Infrastructure Section View Oriented North

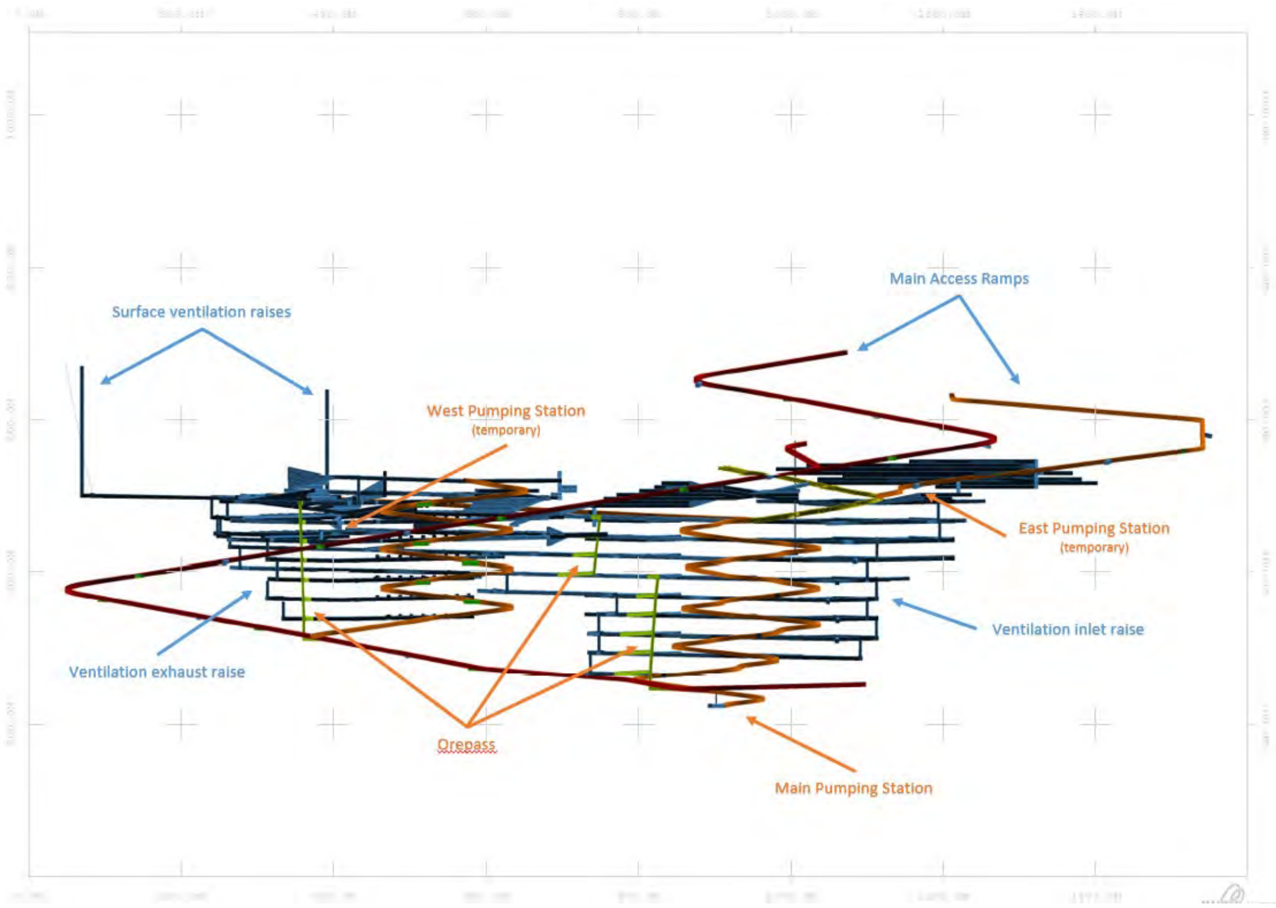


Table 16-3 Dimensions for Lateral and Vertical Development Section Profiles Respectively

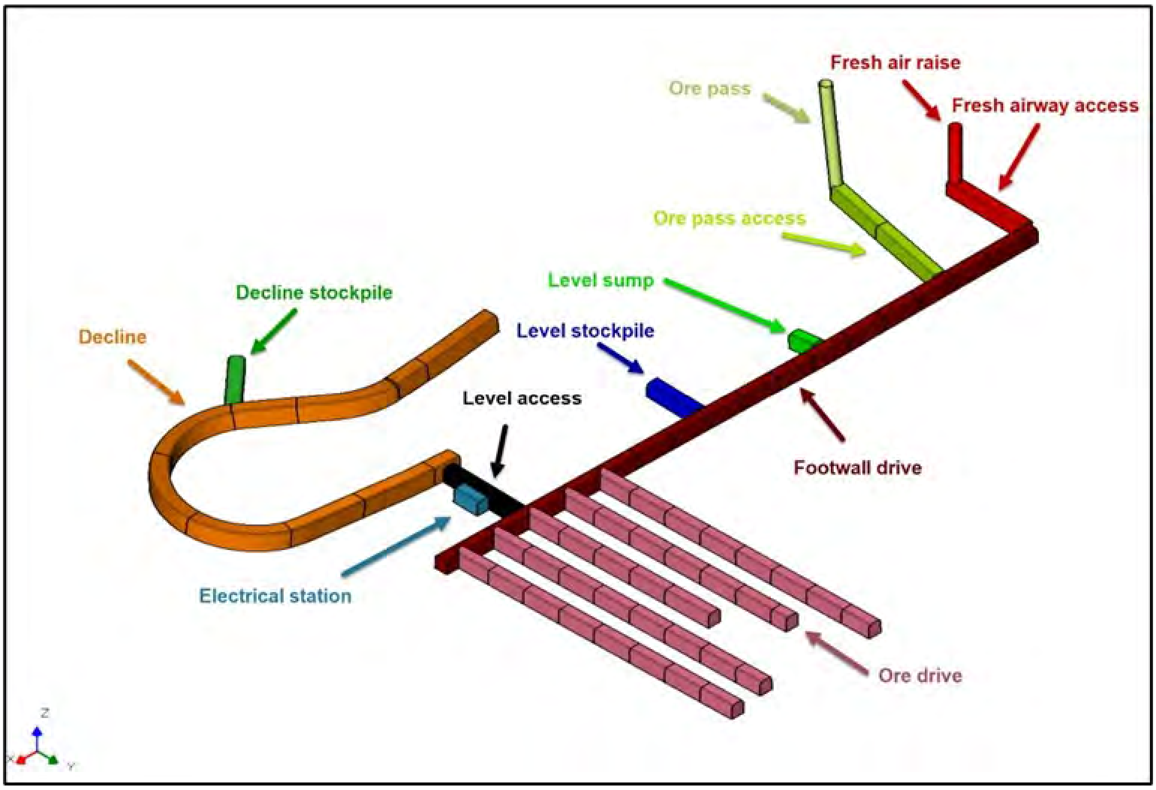
DEVELOPMENT TYPE	WIDTH (M)	HEIGHT (M)	DIAMETER (M)
Decline, ramp	6	6	-
Transport level	5.5	5.5	-
Workshop	6	6	-
Other ancillary infrastructure	5	5	-
Ore pass	-	-	4
Vent raises	-	-	3.5
Paste fill line borehole	-	-	0.2

16.3.2 Level Development – LHOS

Sublevels will be accessed from the haulage ramp system on a 25m vertical interval defined by the planned stoping heights. Footwall drives, following the general strike of the orebody zone, are designed with a minimum 2% gradient to ensure dewatering and are set back a minimum of 25m from the ore contact. These footwall drives will generally contain the excavations for exploration drilling, dewatering sumps, mucking bays, and ventilation raise accesses. Each level includes a 20m long mucking bay every 150m that can be used as re-mucks and an access to the ore pass. Ventilation raise accesses are located at both ends of a drive so as to be placed in the primary ventilation circuit, thus facilitating the exhaust of contaminated air.

Ramp development is set back a further 20m from the footwall drive, that is, 45m from the ore contact. Both are connected by the level access, which contains other infrastructure excavations such as electrical substations. Figure 16-10 below illustrates a typical transverse open stoping level layout. The layout ensures long-term geotechnical stability and provides adequate space for the placement of ancillary services between the ramp and level development.

Figure 16-10 Transverse LHOS - Typical Level Layout, South West Oriented Isometric View



The development section profile dimensions used in the level development designs are shown in Table 16-4.

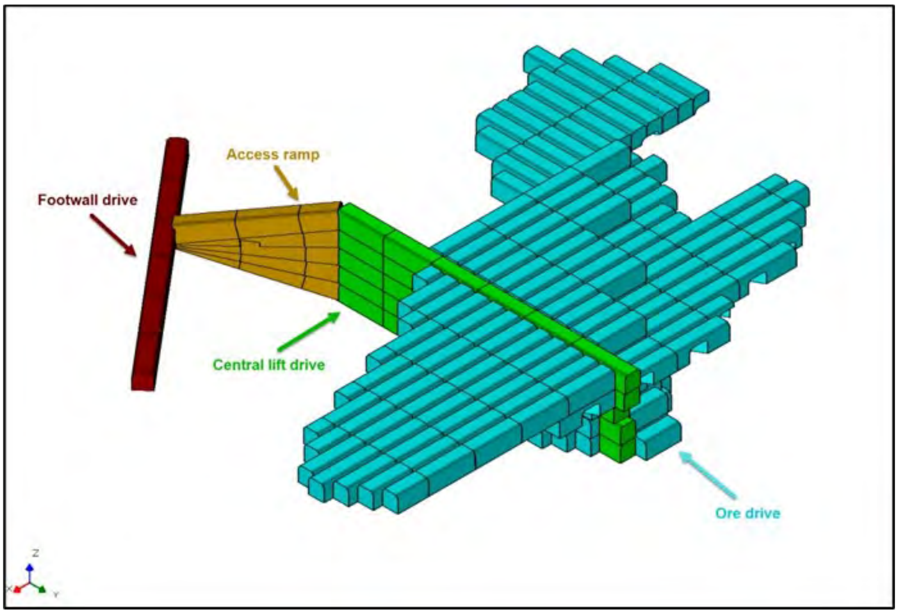
Table 16-4 LHOS Level Development Section Profile Dimensions

DEVELOPMENT TYPE	WIDTH (M)	HEIGHT (M)
Lateral (FW)	5.5	5.5
Ore access	5	5

16.3.3 Level Development - DAF

The DAF stopes are designed in arrays of level lifts accessible through a common cross-cut ramp. The number of lifts accessed through the same ramp is defined by the ramp layout and gradient, set to a maximum of 17%, to minimize infrastructure. The lifts in the same sub-block are mined following a standard DAF overhand sequence and are laid out from a central cross-cut with ore drives branching perpendicularly from it. The ore drives will be mined in retreat towards the access ramp and backfilled with cemented paste fill. Each lift in the sub-block will be accessed through an extension of the ramp, excavated once the cemented backfill has been cured. A sample sub-block layout is illustrated in Figure 16-11.

Figure 16-11 Drift & Fill Mining Block - Isometric View, Oriented South West



The development drive section dimensions employed in the DAF stoping areas are shown in Table 16-5.

Table 16-5 DAF Level Development Section Profile Dimensions

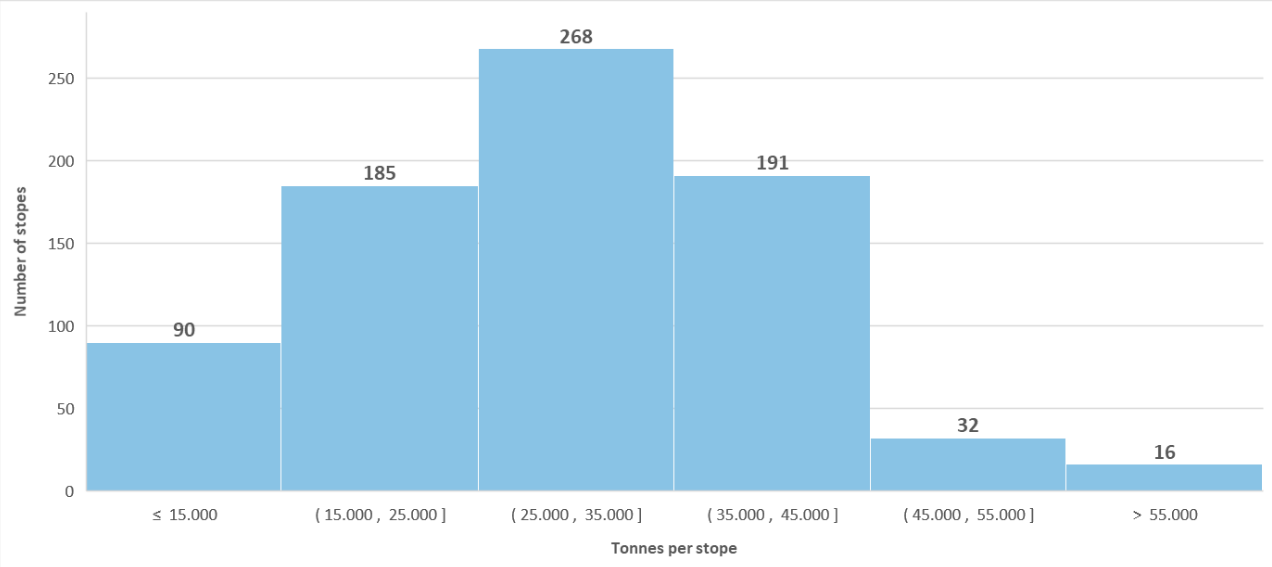
DEVELOPMENT TYPE	WIDTH (M)	HEIGHT (M)
All	5	5

16.3.4 Stope Design

A Mineable Shape Optimizer (MSO) from the Deswik.SO mine planning software package was used to devise conceptual mining shapes meeting both NSR cut-off and operational design criteria. Further information in relation to the mine planning and Mineral Reserve process is provided in Section 15.11.

The conceptual stope shapes obtained from the MSO were manually adjusted as necessary to minimize the amount of planned dilution and to meet practical mining constraints. A total of 782 LOM stopes have been designed ranging in size from 15kt to 75kt as shown in Figure 16-12 below. The average size of a stope is about 35,000 tonnes.

Figure 16-12 Stope Size Histogram



16.4 Geotechnical

16.4.1 Overview

Geotechnical information relative to the geometric layout and ground support requirements for portal locations, ore and waste development, stope design and geotechnical pillar design has been incorporated into the mine design process.

The geotechnical analysis and recommendations to date are based upon available laboratory testing and geotechnical drill core log information obtained from a series of drilling campaigns undertaken between 2012 and 2017. The tests included dry density, relative density, moisture content, uniaxial & triaxial compression tests, indirect tensile (Brazilian) strength tests, shear strength, slake durability tests, sonic speed tests and point load tests.

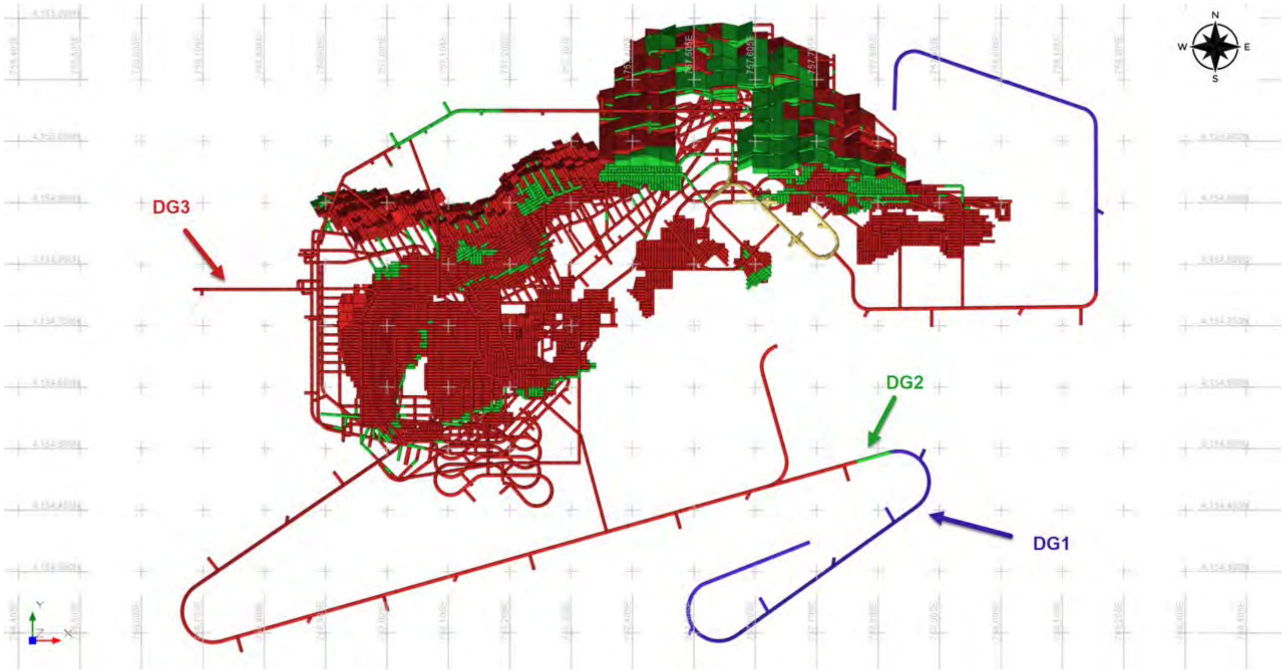
Geotechnical recommendations were provided to CLC in the 2018 and 2022 geotechnical reports prepared by Subterra Ingeniería (Madrid).

Based on the available geotechnical information, a systematic approach was used to define the distribution of domains:

- very poor ground (DG1) for soft rock or marl areas,
- poor ground (DG2) for fault areas, and
- fair to good ground (DG3) for the remainder.

Figure 16-13 shows the geotechnical domain distribution throughout the underground mine.

Figure 16-13 CLC Geotechnical Domain Distribution



This geotechnical recommendations will be updated on a regular basis incorporating new geotechnical information collected as the underground workings progress.

16.4.2 Rock Mass Properties

The underground geological domains of significant heterogeneous nature have been classified into five broad lithological groups, namely Gossan, Volcanics, Shales, Sulphides and Stockworks.

For each of these domains, a spatial Rock Mass Rating (RMR) distribution was interpreted from a series of plans and sections, using the available geotechnical drilling information. Since both the sulphide and stockwork materials constitute the bulk of the mining inventory the analysis has focused on these domains.

Table 16-6 below shows the distribution of RMR values across sulphide geotechnical samples. An RMR value of 55 was selected as representative for the sulphide domain and employed in the subsequent stope stability calculations.

Table 16-6 RMR Value Distribution in Sulphides

GEOTECHNICAL DOMAIN	RMR VALUES	FREQUENCY
DG5	81 – 100	0
DG4	61 – 80	33
DG3	41 – 60	57
DG2	21 – 40	8
DG1	< 20	1

Table 16-7 shows the distribution of RMR values across stockwork geotechnical samples. An RMR value of 60 was selected as representative for the stockwork domain and employed in the subsequent stope stability calculations.

Table 16-7 RMR Value Distribution in Stockwork

GEOTECHNICAL DOMAIN	RMR VALUES	FREQUENCY
DG5	81 – 100	0
DG4	61 – 80	56
DG3	41 – 60	37
DG2	21 – 40	8
DG1	< 20	0

Corresponding Q index values of 3.39 and 5.9 were calculated for the massive sulphide and stockwork domains respectively.

16.4.3 Discontinuities

Three discontinuity sets, listed in the next table, were identified and characterized.

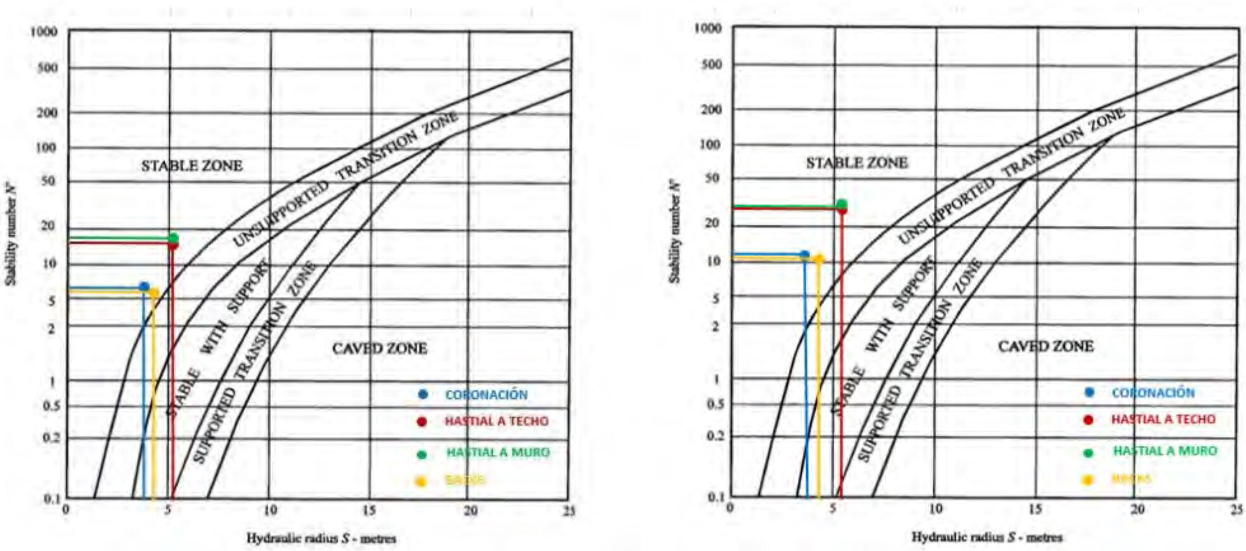
Table 16-8 Identified Joint Sets in Orebody

SET	TYPE	DIP	DIP DIRECTION
J1	Diaclase	55	75
J2	Diaclase	40	220
J3	Diaclase	62	259

16.4.4 Underground Rock Mechanics

The Mathew's Stope Stability Method and its revision by Villaescusa, extensively used in underground mining, were employed to assess the stope stability in the massive sulphide and stockwork domains for several stope heights and widths, namely 15, 25, 35 and 40m. Figure 16-14 shows the results of the Mathew's – Villaescusa stability graph for the main stope dimensions planned in massive sulphides and stockwork.

Figure 16-14 Mathew's – Villaescusa stability graph applied to stopes in massive sulphides (left) and stockwork (right). Subterra (2018)



An optimization of stope height and length dimensions, based on the hydraulic radius (HR) of the floor/roof, front/back and sidewall surfaces was also conducted. Whilst all the 3D combinations given by the Mathew's – Villaescusa method were found to be geotechnically sound, of the four stope heights assessed, the 25m stope height options gave a higher overall mining recovery. Based on this, the 25m (wide) x 25m (high) x 30m (long) case was selected as the base case for the stope design in both stockwork and sulphide domains.

16.4.5 Stand-off Distances

The observed stand-off distance for the hangingwall drives from the orebody contacts is 25m. Transport ramps are designed to be a minimum 20m from the hangingwall drives and 45m from the contacts.

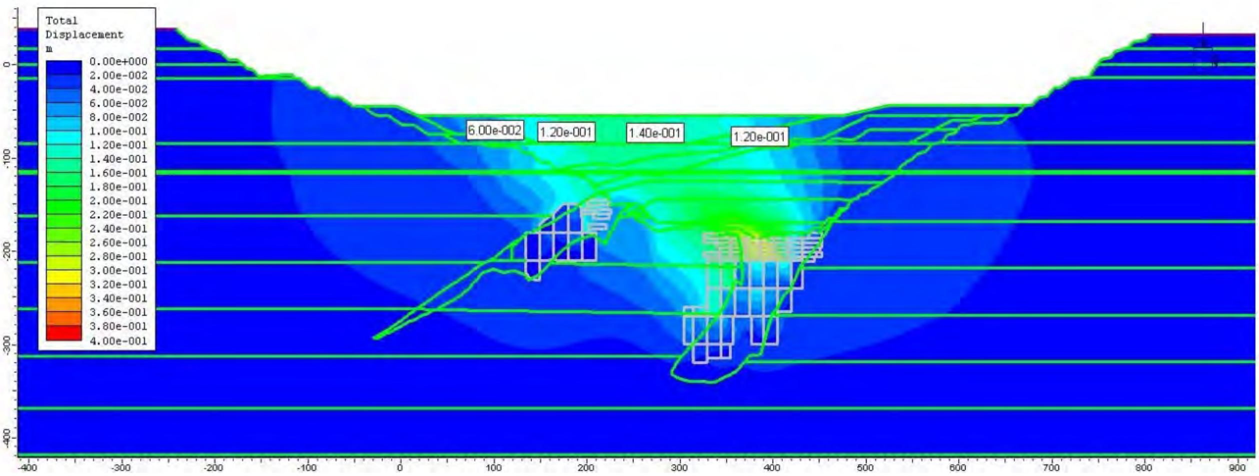
16.4.6 Geotechnical Pillars

Due to the selected mining methods and the use of cemented paste backfill, no temporary or permanent pillars are considered necessary inside stoping areas.

A 30m thick crown pillar between the underground mine and the backfilled open pit has been incorporated into the design. This pillar is larger than required for stability purposes and will be partially recovered by DAF stoping after the LHOS blocks immediately below are backfilled.

A finite element simulation was run to validate the crown pillar thickness under different mining recovery scenarios. The geotechnical review (Subterra, 2018), concluded that a cumulative displacement (subsidence) of 14cm induced in the pit backfill during the final stage of the recovery phase would not compromise the integrity of the in-pit tailings facility (Figure 16-15).

Figure 16-15 Cumulative displacements induced during the crown pillar recovery phase. Subterra (2018)



Optimization of the crown pillar size and recovery method remains as potential upside for the Project and will be reviewed during the production phase.

16.4.7 Ground Support Requirements

Ground support analyses for ore and waste development, including main decline ramps and ramp bays, were conducted resulting in the support recommendations listed in Table 16-9. Except for the portal area of the ramps, most of the support will be provided by a combination of shotcrete and 2.4m-long, 25mm-diameter grouted rebar rockbolts, installed following systematic transversal (T) and longitudinal (L) patterns. Ore and waste development in the DG5 geotechnical domain may not require systematic support.

Additional cable bolting support (6m-long twin cables with 2m spacing) will be installed in the backs of traffic passing bays and above intersections in the declines.

Table 16-9 Development Ground Support Recommendations (Subterra, 2022)

OPENING TYPE	CROSS SECTION	SIDE	GROUND SUPPORT	BOLTING PATTERN (M)	SHOTCRETE THICKNESS (CM)
Ramp - Unaltered marl	6x6	Walls & back	THN-29 steel arches & shotcrete	N/A	15
Ramp - Altered marl & aquifer	6x6	Walls & back	THN-29 steel arches & shotcrete	N/A	20
Ramp - DG1	6x6	Walls & back	THN-29 steel arches & shotcrete	N/A	12
Ramp - DG2	6x6	Walls & back	Grouted rockbolts, mesh & shotcrete	1.5 (T) x 1.5 (L)	7
Ramp & Ramp Bays - DG3	6x6	Walls & back	Grouted rockbolts & shotcrete	1.5 (T) x 1.5 (L)	5
Ramp & Ramp Bays - DG4	6x6	Walls & back	Grouted rockbolts & shotcrete	2 (T) x 1.5 (L)	5
Ramp & Ramp Bays - DG5	6x6	Back	Grouted rockbolts & shotcrete	2 (T) x 2 (L)	3
Ore & Waste Development - DG1	5x5	Walls & back	Shotcrete & mesh	N/A	7
Ore & Waste Development - DG2	5x5	Walls & back	Grouted rockbolts & shotcrete	1.5 (T) x 1.5 (L)	5
Ore & Waste Development - DG3	5x5	Walls & back	Grouted rockbolts & shotcrete	2 (T) x 1.5 (L)	5
Ore & Waste Development - DG4	5x5	Walls & back	Grouted rockbolts & shotcrete	1.5 (T) x 1.5 (L)	5
Ore & Waste Development - DG5	5x5	Walls & back	Grouted rockbolts	As required	N/A

16.5 Hydrogeology/Groundwater

16.5.1 Hydrogeological Model

The hydrogeological model indicates that dewatering from two aquifers will be required. The upper aquifer lies in the Miocene layers, which is a porous media. The lower aquifer is in Paleozoic horizons, a fractured media which is more heterogeneous and anisotropic.

To develop the hydrogeological model the following parameters were used:

- Historical information from an array of piezometers. These were located primarily in the Miocene layers.
- New information from the Paleozoic horizons, mainly sourced from new piezometers, oriented core drillholes (2019-2021) and a structural geological model supplying information on the fractured media.
- A hydrogeological review of selected drillholes from the 2020-2021 infill campaign, to identify and characterise structures.

The Hydrogeological Model was first developed in August 2018 (REDUX model) and subsequently updated in March 2020 and June 2022 (N REDUX model) as new data were logged and analysed. The model was reviewed and validated by KLM Consulting Services in March 2020 whilst the latest update was done in June 2022.

The model was calibrated with both regional groundwater information and piezometric information obtained from existing groundwater control boreholes. In addition, the model was also calibrated with the water balance in the open pit and the existing underground exploration ramp.

The N-REDUX confidence matrix shows that the level of knowledge of the modelled environment has been substantially improved since its first version, improving by 23 points compared to previous versions (Table 16-10).

Table 16-10 Confidence Matrix of N-REDUX 2020 and 2022 compared with original REDUX Model

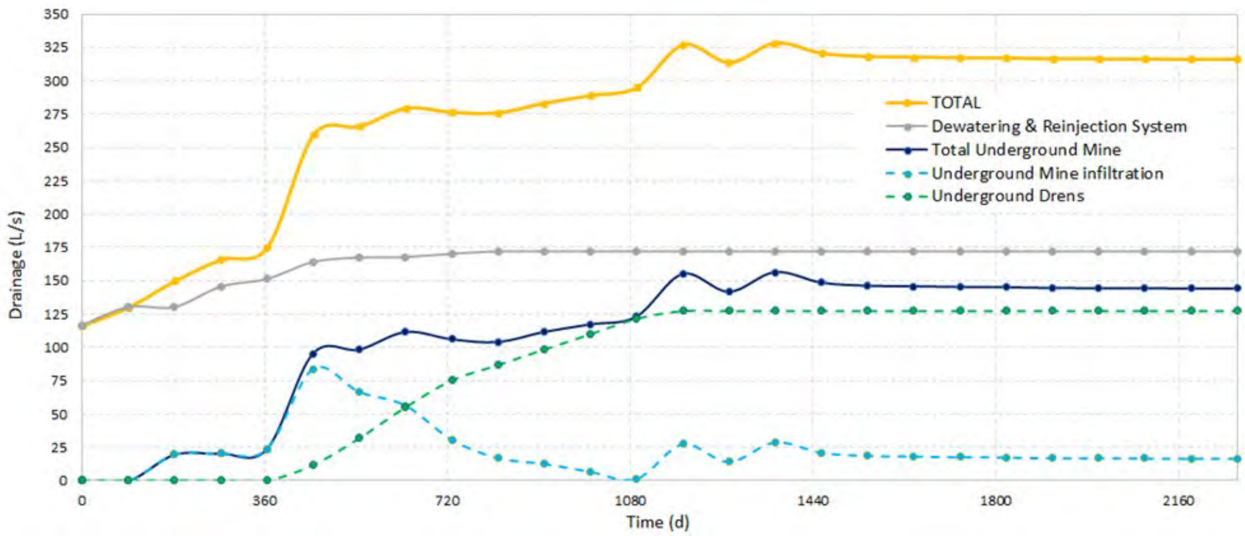
PMR UG MINE HYDROGEOLOGICAL MODEL CONFIDENCE MEASURE AGREEMENT	Nº Factors	REDUX (August 2018)	N-REDUX (March 2020)	N-REDUX (June 2022)
DATA	17	64%	78%	81%
INFORMATION	45	49%	74%	77%
KNOWLEDGE AND APPLICATION	49	63%	84%	86%
Overall Confidence Percentage	111	58%	78%	81%

Annual updates of the hydrogeological model are a compliance requirement of the local water authority.

16.5.2 Predictive Simulations and Inflow Estimates

Predictive water inflow simulations were based on the LOM plan mining sequence and extents. The development of the mine was modelled in successive steps with a Dewatering-Reinjection System (DRS) flow constant at 172 l/s (Figure 16-16 below).

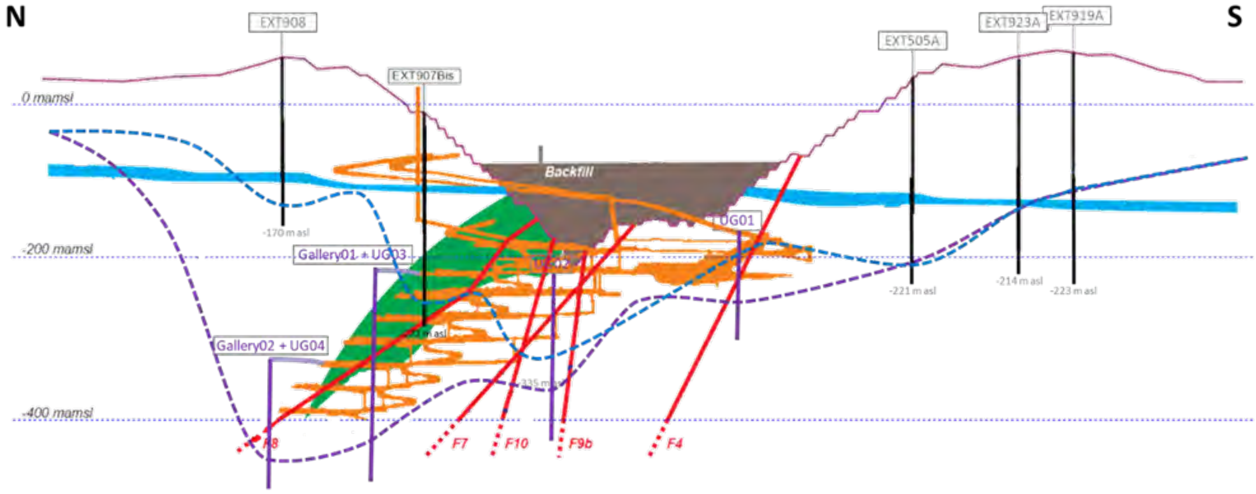
Figure 16-16 Predicted UG Mine Dewatering. CLC-Ayterra, 2022



A sensitivity analysis of the permeability of fractured Paleozoic rock was undertaken. The result of the interpreted underground annual average flow (underground drains plus underground inflow) is approximately 156 l/s during Year 3 and is expected to decrease progressively to 144 l/s in Year 5.

The proposed mine drainage concept is based on a ring of underground vertical drains complementing a ring of surface DRS wells. This drainage reduces hydraulic pressure, the risk of water inrushes and facilitates mining work in safe conditions. Figure 16-17 shows the conceptual approach to this strategy. Under this assumption, annual and peak mine water inflows have been estimated and are the basis for the underground pumping system capacity calculations.

Figure 16-17 Mine Drainage Strategy



The mine has been divided in two sectors, East and West, based on orebody geometry and the mine design. Figure 16-18 shows a plan view of the existing and planned surface and underground drainage wells.

Figure 16-18 Existing and planned drainage wells

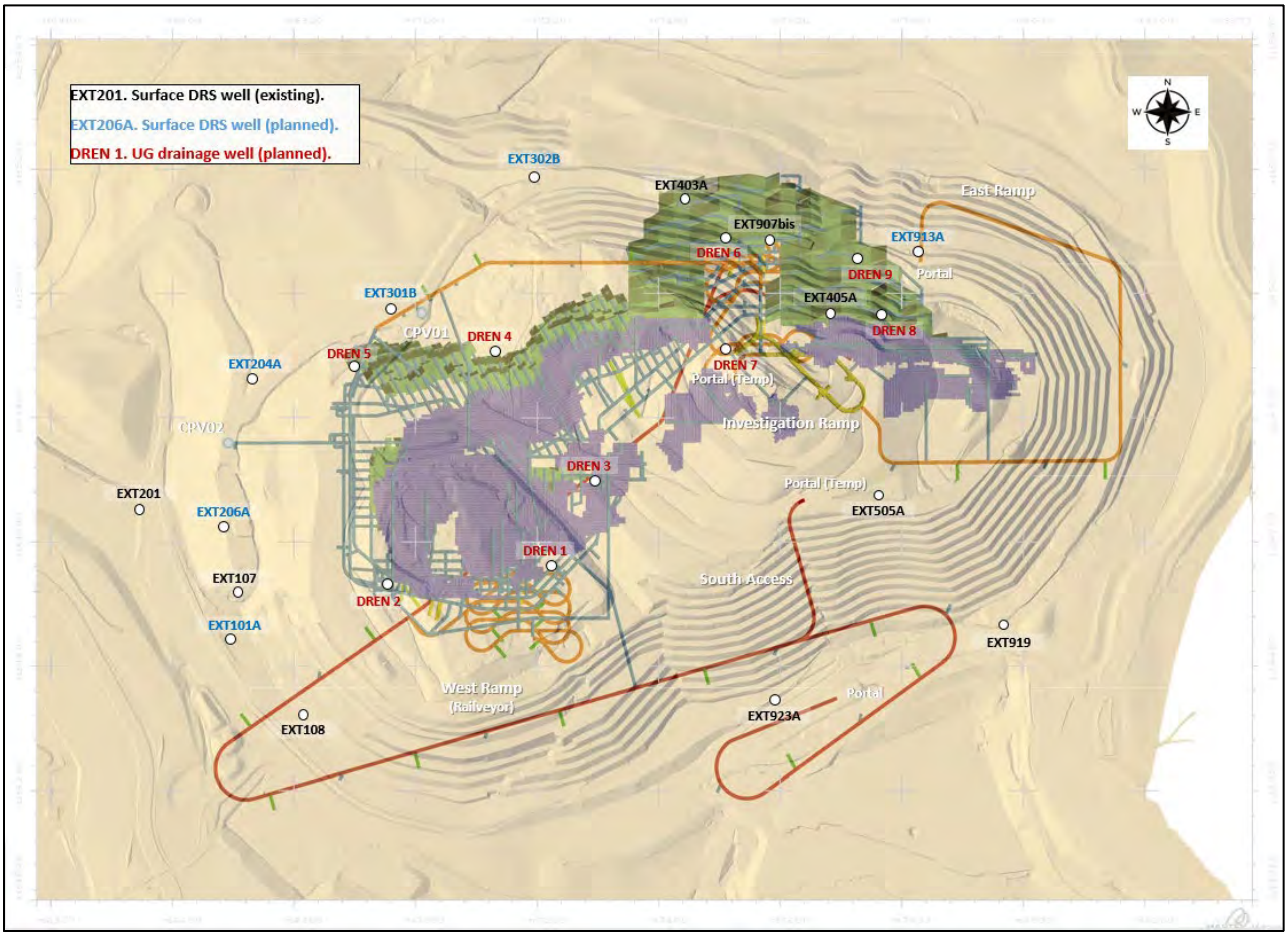


Table 16-11 below shows the LOM average and peak water in-flows for each sector. The maximum case peak mine water inflow is anticipated to be reached during short intervals only.

Table 16-11 Maximum UG Mine Water Inflow and Peak Predicted (CLC, 2022)

SECTOR	MAXIMUM ANNUAL AVERAGE UG MINE INFLOW (L/S)	MAXIMUM PEAK INFLOW (L/S)
EAST	88 (Year three)	58 (Year 1) *
WEST	69 (Year three)	34 (Year 3) *

* *Additional inflow over the average observed that time.*

16.6 Ventilation

The ventilation system was designed in accordance with the Spanish Mining Safety Regulations. In addition, a minimum fresh airflow of 0.067 $m^3/s/kw$ per diesel-powered kilowatt has been considered. The design is based on a negative pressure (pull) configuration, with two exhausting main fans installed on surface over two 3.5m-diameter independent shafts, working at the same time during the mine life (Figure 16-19). The two ramps serve as fresh air intakes.

Mine ventilation is managed in two proposed independent circuits. The PMS East and NW Zone will be supplied with fresh air from the East Ramp; each level will have two air entries, a fresh raise (FAR) placed at the east end of the level and the level access itself. The air outlet will be located at the western end of the level, through a return air way connected with the North Ventilation Shaft (CPV01).

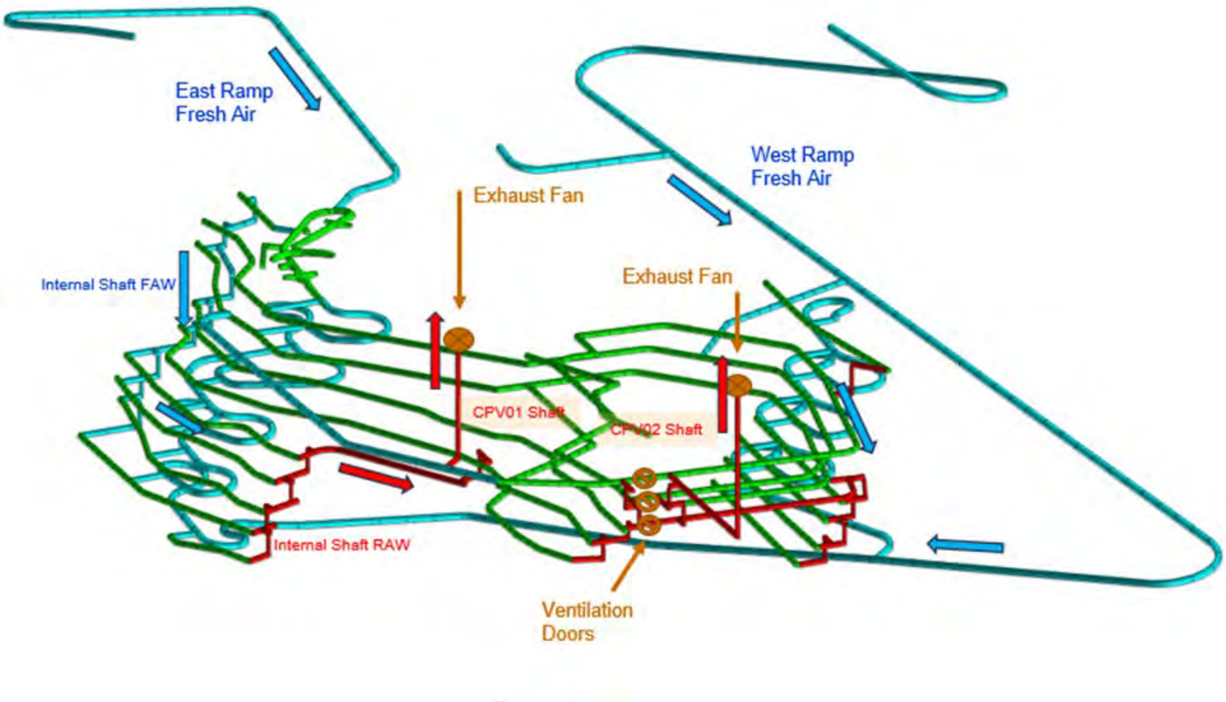
The SMS and PMS West Zones will be supplied with fresh air from the West Ramp. Airflow will be introduced to each level from the access and the exhaust will be made through a return air winze (RAW) that connects with the West Ventilation Shaft (CPV02). The contaminated airflow will be exhausted through the two ventilation shafts.

The East and West ramps will provide fresh air to the Railveyor loading level through a connection at the -435mRL level.

The amount of air flowing through each level in both the Main and SMS zones will be controlled by a combination of fans and regulators, installed at the end of the -210mRL level between CPV01 and CPV02. This will give flexibility to operate the system as two independent circuits providing redundancy in case of emergency.

Figure 16-19 shows an isometric view of the CLC ventilation system.

Figure 16-19 Isometric view of the Main Ventilation Network from the North West



16.6.1 Total Airflow Requirements

Airflow requirements were determined considering the point-of-use Diesel Engine Exhaust (DEE) dilution for the number of operating machines. An airflow allowance was also determined for underground infrastructure, leakage, and balancing inefficiencies.

Total airflow requirements were determined based on the anticipated concurrent activities during steady state production and development. A total of six scenarios were simulated, each representing the mine layout at different points along the LOM. The following requirements were considered:

- Trucking transport level, to receive a minimum airflow of 35m³/s.
- Levels with LHD hauling, to receive a minimum airflow of 20m³/s.
- Working levels without production mucking to have a minimum 20m³/s airflow.
- One stope per level and area.
- A maximum of two levels can be developed below main ventilation.
- Developed levels without activity to have a minimum 5m³/s airflow.

Considering working equipment, active production levels, leakages and network balancing, the maximum airflow requirement throughout the LOM will be 290m³/s during Year 4.

16.6.2 Auxiliary Ventilation

All mining areas not supplied by the primary ventilation circuit must be ventilated using auxiliary systems. The most effective method of providing the required airflow to these areas is typically with small-diameter, up to 1,400 mm, axial fans combined with limited-leakage, flexible ducting.

During access and level development work, distances up to 1,000m will be ventilated using auxiliary systems. The peak auxiliary airflow for development activity will consider the dilution and exhaust of the emissions from a 45t haulage truck and a loader; this will amount to 30m³/s of auxiliary airflow.

The -210mRL crosscut connects the East and West areas and closes the first ventilation circuit at the end of Year 1. In combination with the completion of the CPV01 shaft, it will provide fresh airflow to progress the mine deepening to the East and the development of the Railveyor ramp until the completion of the CPV02 about one year later.

16.6.3 Production Ventilation

An allowance of 15m^3/s airflow was made per each active stope to account for dust, blasting and diesel engine fumes exhaust. Modelling indicates that a single-stage 75kW, 900mm diameter fan, coupled to 1,000mm diameter, low-resistance, low-leakage ducting will supply the required airflow to a distance of at least 250m. A larger ducting diameter of 1,100mm can be employed for stopes requiring a longer forcing distance, up to 700m maximum.

A Ventsim ventilation model was developed for the Project for three primary purposes:

- Validating the proper operation of the ventilation circuit, so adequate airflow can be provided to all required areas.
- Ensuring compliance with design and legal criteria.
- Selecting fan units and calculating operating points and energy requirements.

Peak primary fan duties will occur at full production level in conjunction with maximum development activities in the lowest levels of each ventilation district.

16.6.4 Permanent Primary Fans

Throughout the LOM, there will be a multitude of settings for the ventilation circuit, depending on the type of activities and their location throughout the mine. The circuit has been modelled to reflect the peak primary fan operating conditions that can be reasonably expected. Primary fan requirements are summarized in the next table.

Table 16-12 Primary Fan Requirements

CPV01 North Fan	
Description	Specification
Duty	@ 200m^3/s @ 1,600 Pa
Type	Horizontal mount axial mine fan
Configuration	Exhausting fan
Fan Motor	400 kW

CPV02 West Fan	
Description	Specification
Duty	@ 150m^3/s @ 1,550 Pa
Type	Horizontal mount axial mine fan
Configuration	Exhausting fan
Fan Motor	300 kW

16.6.5 Emergency Preparedness

As part of the ventilation strategy, consideration has been given to the potential for mine emergencies. As such, the following criteria have been established:

- Both declines will be located in fresh airflow. The Railveyor decline acts as an egress route for emergencies.
- An emergency man-escape way is to be installed in the internal Fresh Air Raises (FAR).
- Return airflow from each level is to be directed towards the return circuit and not to be circulated through a second level.
- Fire ventilation doors will be placed in accordance with legal mining requirements and to provide means to isolate the ventilation circuits between areas.
- The maintenance workshop is to be directly connected to the return air circuit. This will avoid gases or potential fire fumes being leaked to the fresh air circuit.

16.7 Mobile Equipment Requirements

16.7.1 Pre-production Phase

During the pre-production phase approximately 31,000m of lateral development is required to complete the ramp-up production schedule. Development during this period will peak at approximately 2,000m per quarter and averaging approximately 1,900m per quarter.

Table 16-13 shows the supplied equipment for the pre-production development phase.

Table 16-13 UG Equipment Count During the Pre-production Phase

DESCRIPTION	TOTAL
Two boom jumbos	5
LHD 12.5 t	7
Bolter	5
Cable Bolter	1
Shotcrete sprayer	5
Transmixer	6
Production driller 100 mm	1
Mobile raiseborer (Easer L)	1
LHD 17 t	5
Haulage Truck, 45t	4
Haulage Truck, 60t	3

16.7.2 Production Phase

Table 16-14 summarizes the total peak equipment requirements during the production phase of the UG operation.

Table 16-14 UG Equipment Count during the Production Phase

DESCRIPTION	NUMBER
Two boom jumbos	3
LHD 12.5 t	4
Bolter	3
Cable Bolter	1
Shotcrete sprayer	3
Transmixer	4
Production driller 100 mm	3
Mobile raiseborer	1
LHD 17 t	9
Haulage Truck, 65t	4

16.7.3 Support Equipment

The ancillary equipment listed in Table 16-15 will be required to support all activities in the mine.

Table 16-15 Ancillary Equipment

DESCRIPTION	TOTAL
Emulsion Loader	5
Scissor Lift	5
Mobile breaker	2
Boom Truck	3
Personnel Carrier	6
UG Forklift	2
Grader	1
Service Truck	5
Pickup Truck	13
Fuel/Lubricant Truck	3

16.8 Underground Infrastructure

16.8.1 Mine Dewatering and Solids Handlings

Mine development has been designed to avoid unnecessary intersections with water-bearing structures. Pumping will manage water from the underground drains as well as the inflow.

The pumping capacity has been designed to cover both the maximum groundwater inflow expected through the LOM (average 156 l/s in Year 3 and a maximum peak predicted of +58 l/s in Year 1) plus the maximum equipment usage inflow (12 l/s). The maximum average flow from the mine and the maximum peak do not coincide in time. The total water to manage will be 176 l/s in Year 3. For this reason, pump capacity was designed to be larger than the total water inflow, such that the maximum required operation, relative to time, is limited to 80% or less. The design capacity for the pumping system is based on maximum inflows of 124 l/s for each of the mining areas (east and west).

Mine dewatering will be handled by a combination of submersible centrifugal pumps located throughout the working faces. The pumps will handle mine water to the nearest level sump throughout the mine to minimize pump size and power. These level sumps will feed via gravity through boreholes to the main pump station.

The main pump stations will all be equipped with the same pumps. This reduces the amount of different pump sizes required, and will simplify maintenance. Each pump station has 100% redundancy such that the loss of any single pump will not disable the system. Intermediary levels will drain via boreholes to the main pumping levels, before leaving the mine and reporting at the surface in a single lift.

Robust, dirty water, positive displacement diaphragm pumps with full redundancy in both main pump stations have been considered to ensure full availability of the pumping system. In order to get quotes, the pump suppliers were provided with specifications and instructions to install the same equipment in both pump stations. A total of two pumps will be needed in each pump station.

Figure 16-20 Positive Displacement Diaphragm Type Pump. Source : GEHO

Mine dewatering will commence using two separate circuits. Once the mine reaches the lowest level in Year 5, a new central pumping station will be placed at -450 mRL. This pumping station will be equipped with an isolation door that will prevent the pumps from flooding and give some buffer capacity in the event of issues affecting the dewatering system. The pumps will be accessible at all times through a raise equipped with a ladder way. Figure 16-21 shows the situation of the dewatering system and Figure 16-22 shows the concept design for the main pumping station.

Figure 16-21 Permanent dewatering circuit (Year 5): View to the North

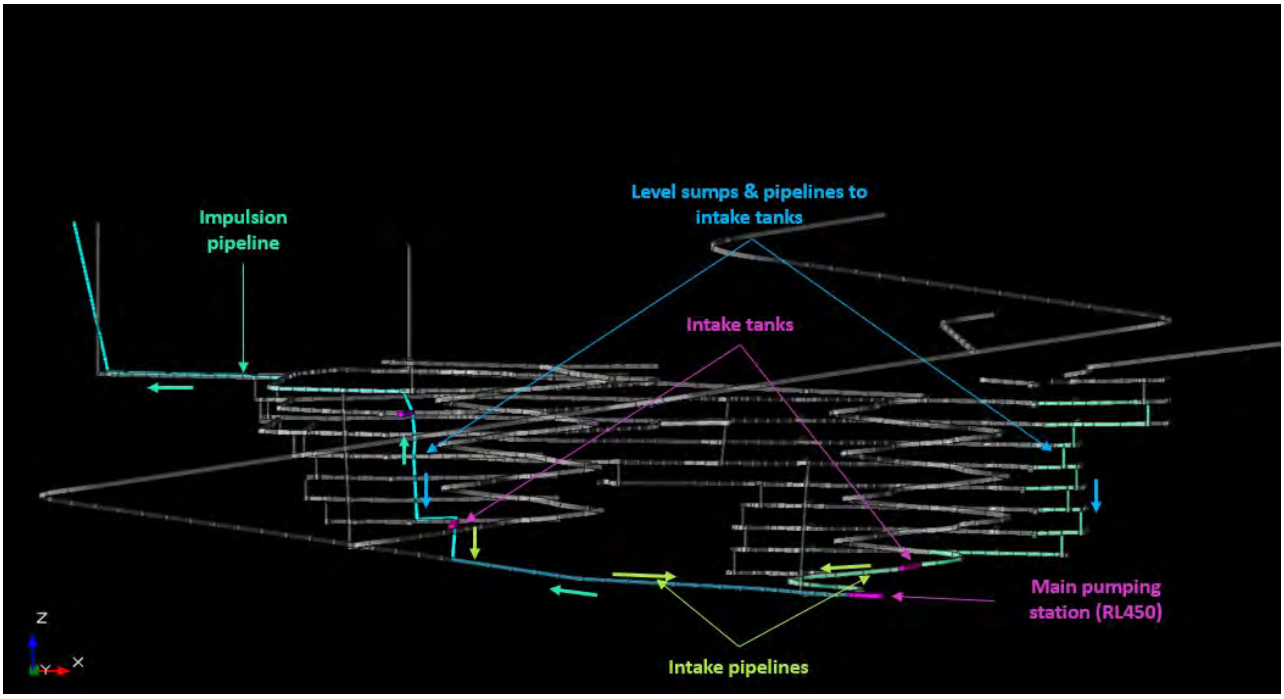
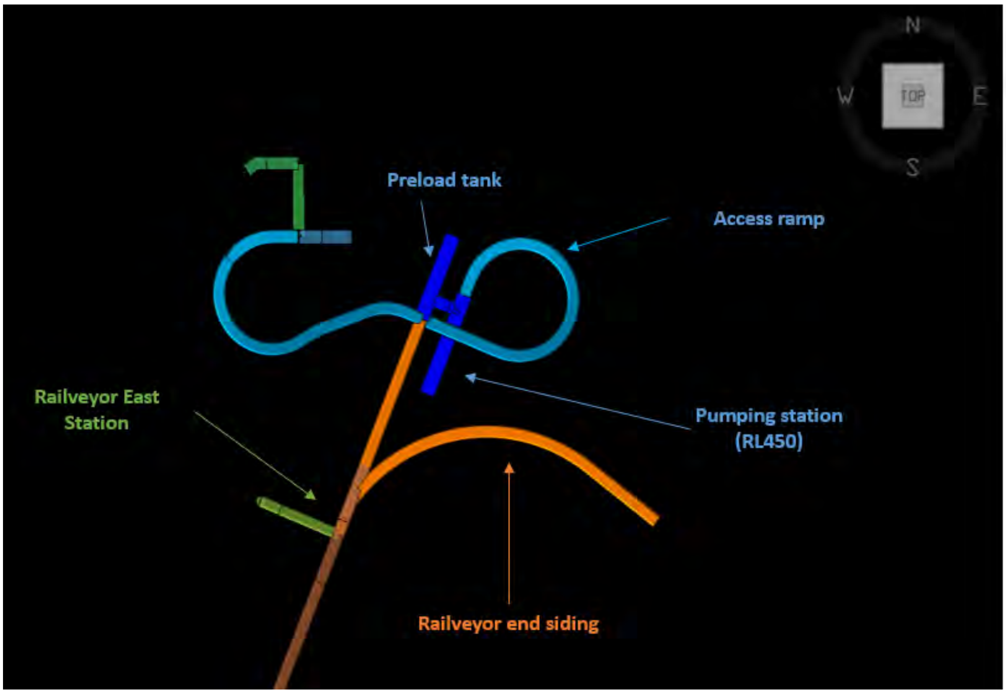


Figure 16-22 -450 mRL pumping station concept design (plan view)

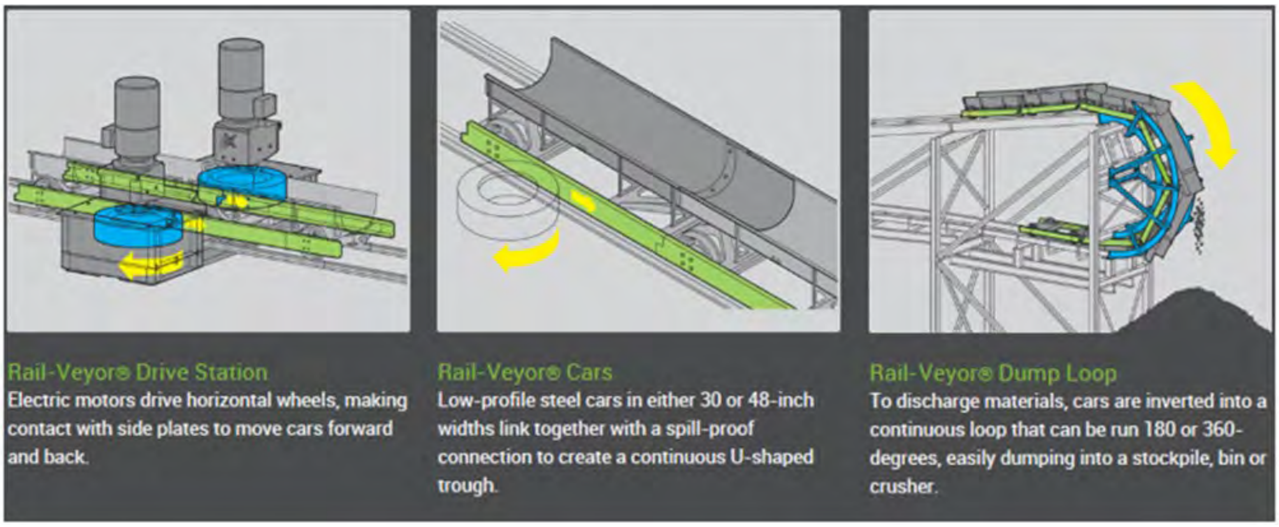


16.8.2 Material Handling: The Railveyor

The mine has been designed with a double ramp system. Mine equipment and light vehicles will enter via the fresh air ramp (East Ramp). Strategically located passing bays will allow for smooth traffic flow. The second ramp (West Ramp) is dedicated for the Railveyor ore handling system transport circuit and it has been designed to meet system specifications, i.e. long straight sections, few curves with a wide turning radius (50m) amenable to future system expansions, and 15% gradient.

The Railveyor is an innovative material handling system that combines the flexibility of truck hauling with the energy efficiency of conveyors and rail lines. The principle of the Railveyor system is to fill the cars from one of two loading stations and to move them forward using drive stations, discharging the material with a 180 degrees continuous loop. Figure 16-23 presents an overview of the proposed Railveyor system.

Figure 16-23 Railveyor Equipment Overview (WSP, 2019)



During the pre-production phase, development waste will be mucked out from the face with 12.5t payload load/haul/dump (LHD) machines to the nearest stockpile where it will be loaded into 45t trucks by the contractor. A new South Access and portal at -110 mRL in the existing pit will provide a shorter way to dispose of the mucked material at the in-pit waste storage facility.

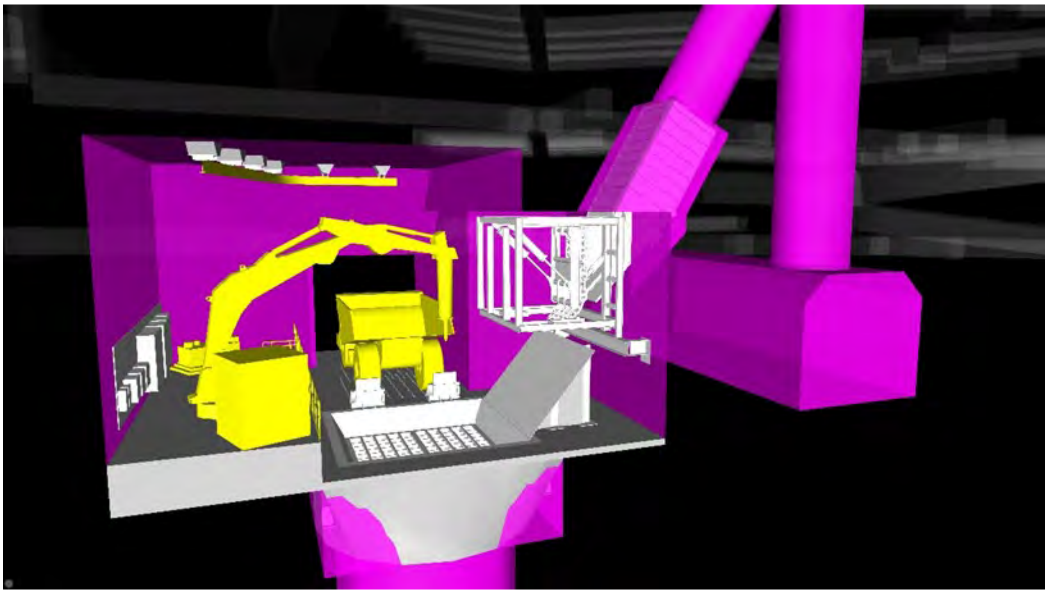
The design standard size remuck bay is 25m in length with 5.5m x 5.5m cross-section. It will have a higher section at the crossing point with the main drive to allow sufficient height to load the trucks. Considering that out of the 25 m, 10 m will be used for storage and the rest for the LHD manoeuvre, the approximate capacity of one stockpile is 500t (using 4.1 specific gravity and 30% swell factor). This is enough capacity for one full shift of LHD production and will provide sufficient flexibility.

LHOS ore will be mucked directly from the stopes to the nearest ore pass with 17t payload LHDs. Where the hauling distance exceeds 300m, the ore will be loaded into 65t trucks and hauled to the allocated ore pass. Three ore passes cover all the planned production areas with ore distribution approximately even at one third of the LOM production each. The East and West ore passes are directly connected with the two Railveyor loading stations at levels -360mRL (West) and -435mRL (East). The Centre ore pass is equipped with a truck chute station at level 335RL. This truck chute station is designed to reduce the required infrastructure while increasing the capacity. One truck will be needed to haul the ore between the chute station and the East ore pass transfer station at 150m. Connection with the West ore pass is under evaluation to give more flexibility and redundancy to the ore handling system.

The ore will be dumped through the transfer stations at different levels. Each transfer station will be equipped with a single cone plug. The cone plug can be put in place or removed by a pneumatic lifting device equipped with rope and sheaves operated by a pull cord.

The ore drops to one of two rock breaker stations (Figure 16-24). That is, one at level 410 (East) and one at level 335 (West). Each rock breaker station will be equipped with one 6 kJ hydraulic hammer to reduce oversize material and a 360 x 360mm grizzly. The hammer is designed to reach the entire grizzly surface. A metal removal device will be used to remove steel bars directly on the grizzly to prevent damaging the conveyor belt at the loading station.

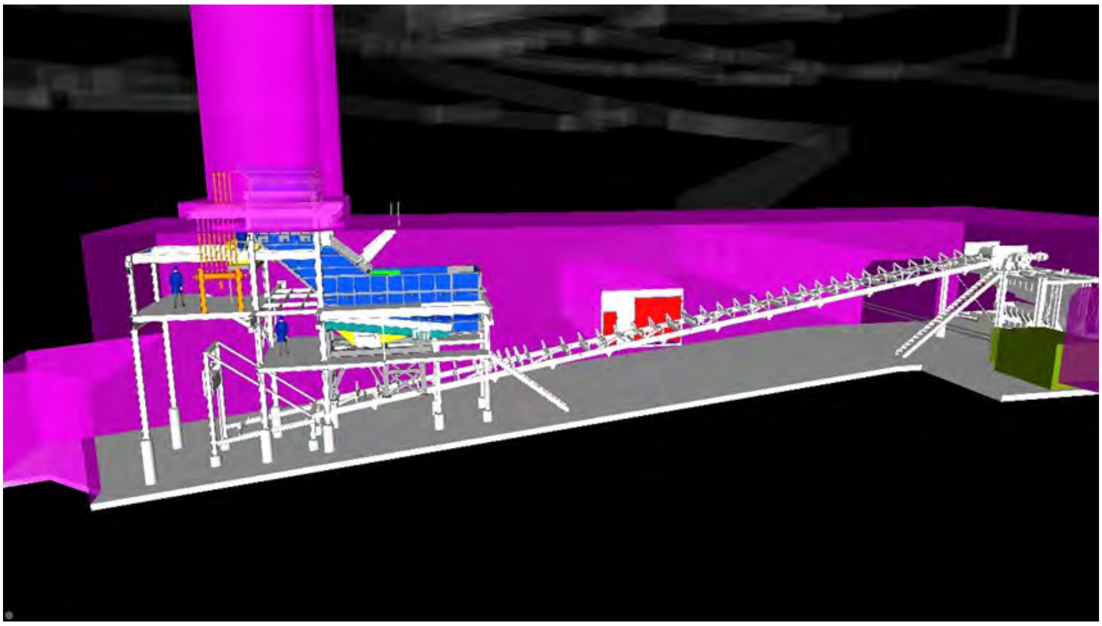
Figure 16-24 Rock breaker Station (WSP, 2019)



The grizzly can be fed by a truck access or by a press-frame control chute.

The two loading stations at levels 435RL and 360RL are composed of two structures designed to facilitate material handling (Figure 16-25). A 6mdiameter ore bin will be used to reduce the risk of downtime when material feeding is lower than the hauling capacity of the Railveyor. A vibrating feeder, supported on a steel structure, will be used to empty the ore bin.

Figure 16-25 Loading Station (WSP, 2019)



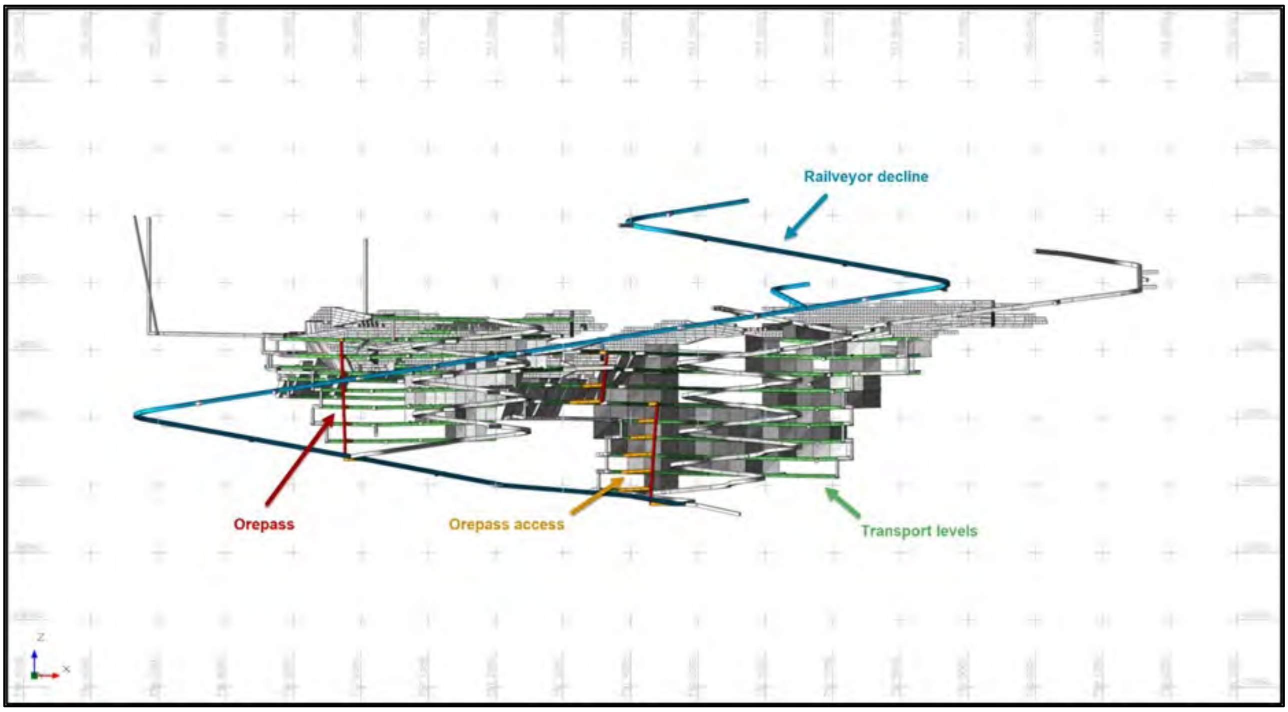
The Railveyor cars will be loaded using a grizzly feeder to separate the fines from the ore and preload the cars to reduce the impact of boulders, thus minimising damage. The feeder is supported by a two-floor high steel structure which also supports the conveyor head and the operator cab, and allows access for maintenance. The chute system is designed to reduce the risk of spillage when loading the cars and is also supported by the steel structure.

The material will be carried between the two structures by a conveyor to spread out the material on the grizzly feeder and facilitate the loading of the cars. The conveyor also allows the operator to regulate the loading speed when required to align the material in the cars and reduce the risk of spillage. The conveyor is equipped with a belt scale to regulate the loading of the cars, reducing the risk of overloading or under loading. A self-cleaning magnet is installed between the two main infrastructures to remove metal bars and to reduce the risk of a bar damaging a drive station. The feeder and conveyor combined system design capacity is 1,100 t/h.

The cars move on a single 962mm gauge track through the West ramp with a point-to-point configuration between the loading stations level and the primary crusher stockpile. Five by-passes allow crossing of the passing cars. Six units have been considered appropriate to achieve the mine production, each unit capable of hauling 74t of ore.

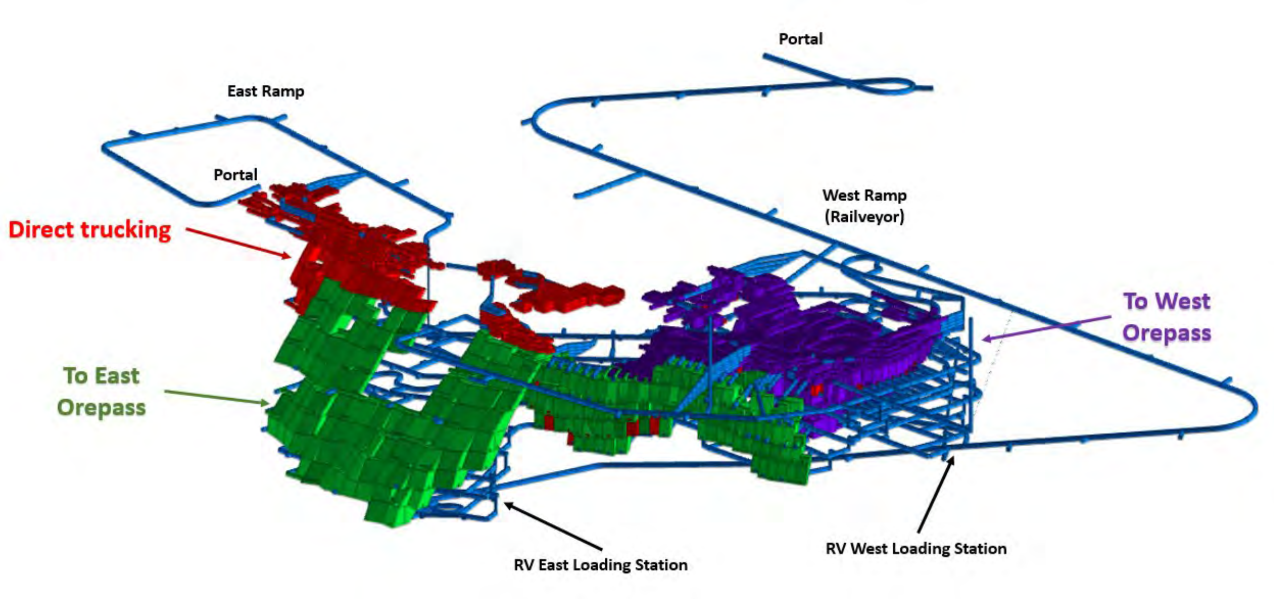
When the cars reach the discharge station at surface, they go through a 180° dump loop to unload the ore into the crusher hopper. The complete hauling system is shown in Figure 16-26. The system has an overall design capacity of 400 t/h.

Figure 16-26 Hauling System View from the South



Ore produced before the hauling system is fully commissioned and ore coming from levels 210RL and 185RL at the eastern edge of the orebody during the production period will be trucked directly to the plant. This totals 5% of the Reserve. Figure 16-27 shows the distribution of ore by destination.

Figure 16-27 Ore Distribution by Destination - Isometric View Oriented South East



16.8.3 Power Requirements and Electrical Distribution

The underground mining electrical supply consists of two main distribution systems, i.e. the Railveyor and the mining supply.

The Railveyor has its own distribution system. The surface facility for the Railveyor is composed of a 20kV overhead line that will follow the train rail to feed three surface substations. The third substation is close to the underground portal and voltage is planned to be lowered to 6.6kV for the underground substations.

The existing 20kV overhead line for the mine ring will be used to feed the paste plant substation. The paste plant substation will house 3MW emergency backup power for the underground supply. The underground mining supply to the paste plant will be via a borehole. Electrical power will be supplied to the UG mine through a 20kV/6.6kV substation located in the paste plant area (currently used for the open pit contractor facilities).

Mobile stepdown transformers will be installed in different level accesses to provide 690V supply to the equipment working in the area while development of the mine is underway. Lower voltage will be also available in permanent working areas (e.g. UG workshop) for small tools, computers and other ancillary uses.

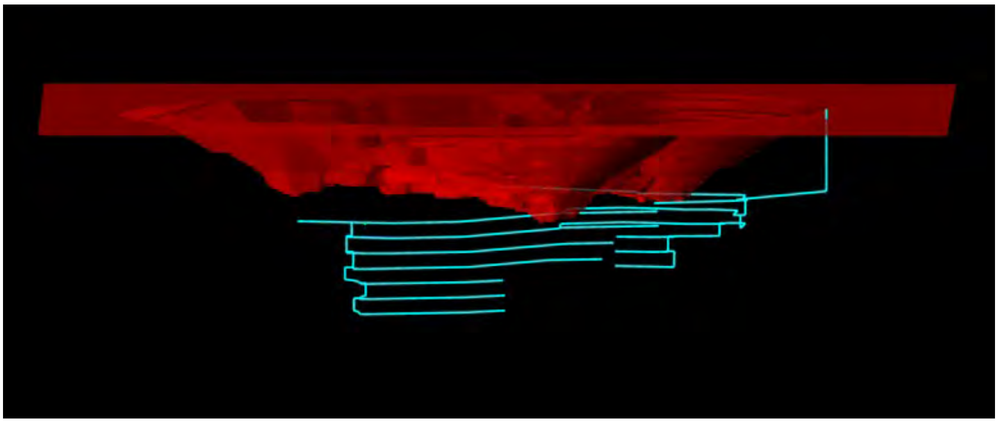
In the production phase it is planned to have one transformer per two (or maximum three) levels to allow maximum flexibility for the movement of production machines and auxiliary equipment.

16.8.4 Service Water Supply

Service water for drilling and dust control will be supplied via a 150mm PN 12.5 HDPE pipe installed within a 750mm service hole near the West shaft. The line will continue through the -210mRL level to the east and west areas of the mine and down to every level through the fresh air raises. The demand requirement has been estimated at 31m^3/h.

A 150m^3 tank positioned near the service hole collar will gravity feed the system. Pressure reducing valves (PRV) will be installed on every level to ensure correct pressure at the destination point.

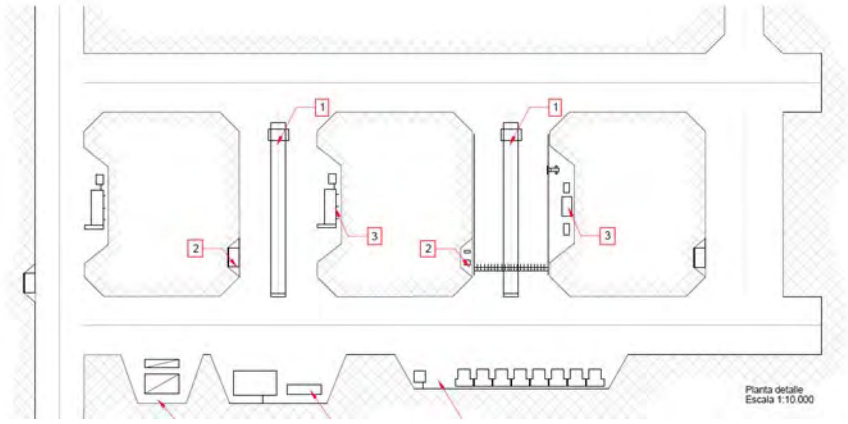
Figure 16-28 Fresh Water Network 3D view



16.8.5 Workshop and Stores

Due to the size of the mine an underground workshop will be necessary to avoid all the machinery having to travel to the surface for maintenance. The schematic design of the workshop is shown in Figure 16-29.

Figure 16-29 Underground Workshop

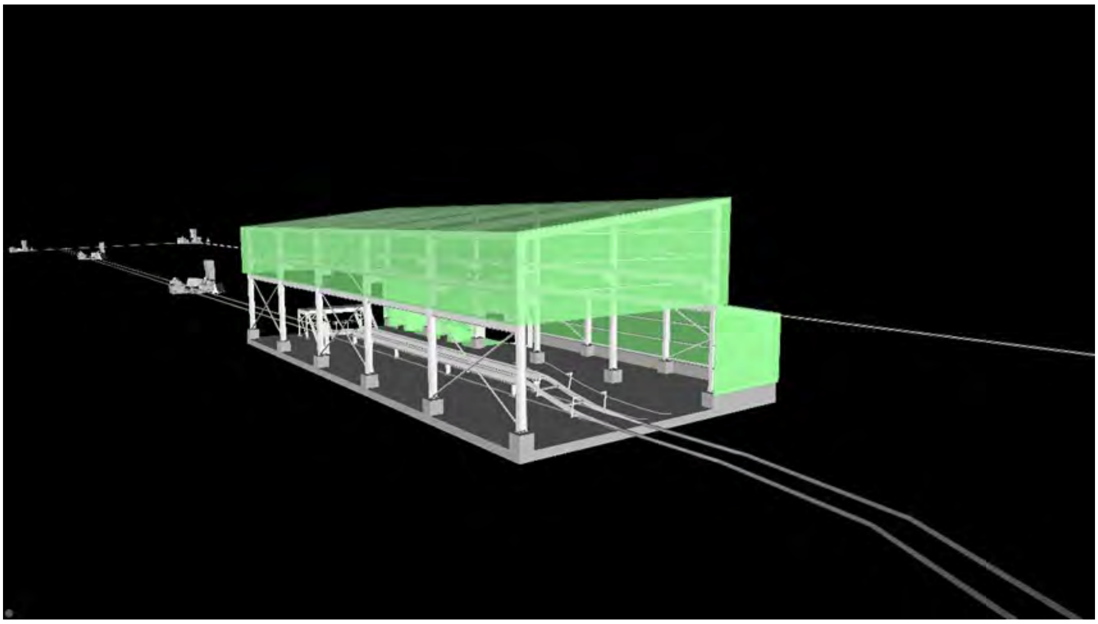


This design has two parallel accesses that connect with the main transport drive and are connected between with several maintenance bay openings. In this case, three bays are provided. This type of design has the advantage of being easily upgraded by adding more parallel bays. Each of these bays will be stripped laterally to accommodate tool storage or fuel and oil storage.

The workshop design has one entrance and one exit so the direction for the equipment movement is always the same. The shop has its own connection to the exhaust air system.

The Railveyor has its own maintenance bay located on surface. The maintenance siding is designed to access a full train length. The maintenance bay is equipped with an overhead crane and services to facilitate maintenance of the train cars. Offices, a bathroom, and a storage area are also included. A 3D view of the Railveyor maintenance bay is shown in Figure 16-30.

Figure 16-30 Railveyor Maintenance Station (WSP, 2019)



16.8.6 Explosives Magazine

A 10tonne explosive magazine will be installed on surface. Underground storage of gassed bulk emulsion will be assessed in future specific studies.

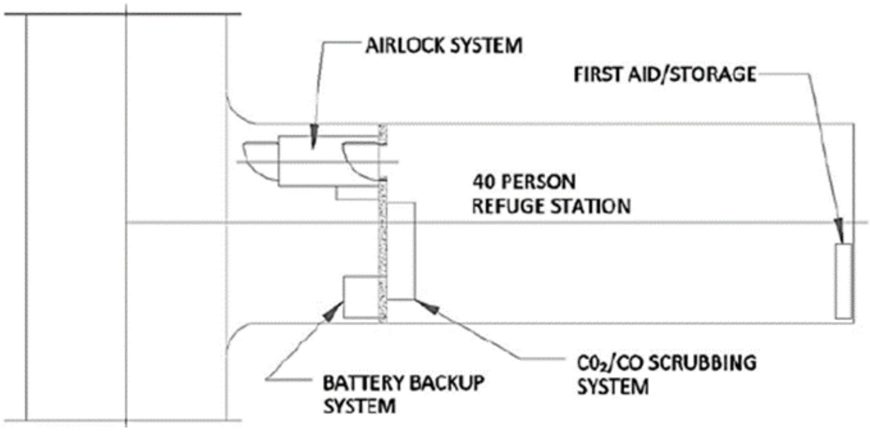
16.8.7 Refuge Stations

Portable stations are to be used for the development and operation of the mine. The proposed stations have capacity for 15 people and full autonomy for 24 hours. Figure 16-31 shows an example of a commercial unit of this type; three units will be necessary.

Figure 16-31 MineArc Refuge Chamber

Four permanent stations (Figure 16-32) will be built throughout the mine to serve areas not covered by the portable stations once they are moved away. Each refuge station will be located in a bay off a drift and will be separated from that drift by a concrete wall. Access to the station will be through an airlock system. The permanent stations will each accommodate 40 people and will be fully equipped for emergency preparedness.

Figure 16-32 Permanent Refuge Station



16.8.8 Communications

The intended communication system is via an LTE installed in all the main infrastructure of the mine. This system has enough bandwidth to support the data load needed for data, image and voice transmission. As well, it has the potential for remote controlling of equipment and automation.

The main communication equipment for the Railveyor includes two fibre optic cables between each substation. One of the fibre optic cables is for redundancy of the Railveyor and loading station communication. At each substation, a communication panel will feed the local equipment. In addition, the communication for the drive stations will use this communication network.

16.8.9 Emergency Generators

Emergency generators are considered for the emergency operation of the paste plant, the main fans, the pumping stations and the communications system. Small gensets will be also needed for the operation of the ventilation doors.

16.8.10 Paste Fill Distribution

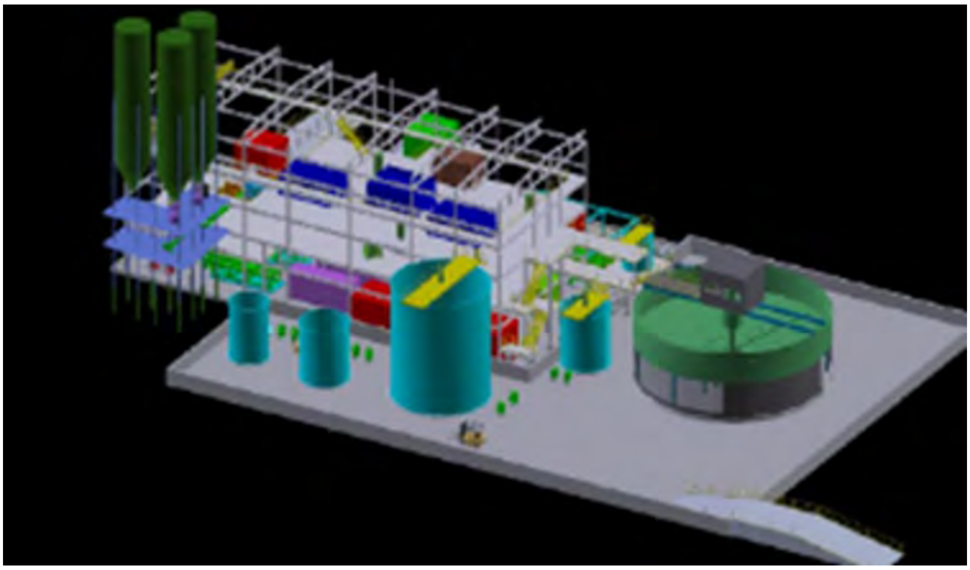
The proposed paste fill distribution system transports the paste from the surface plant to the underground stope voids through a pipeline system.

Initial studies assumed the use of two separate paste fill qualities, both produced from flotation tailings and mixed with 6% (high quality) and 3% (low quality) proportions of cementitious binder contents, respectively. Work conducted by Golder suggests the use of a single quality of paste fill using flotation tailings with 5% binder content. A high-level summary of the paste distribution system current assumptions is presented as follows:

- The paste fill is prepared at a plant located at the surface, which is anticipated to be placed SW from the ultimate pit.
- The paste plant pump feeds a paste line borehole close to the CPV02 shaft connecting the plant to the -185mRL transport level.
- Paste line boreholes connect vertically to the different levels. These would be drilled from accessible locations, such as dedicated bays or cuddies so as to facilitate maintenance tasks such as periodic pipe rotations.

- Pipe flow loops, aimed at lowering the paste flow speed, may be installed at periodic vertical intervals, should the dynamic pressure of the paste become too high and accelerate the tear and wear of the pipes.

Figure 16-33 CLC Paste Plant Model (Golder, 2018)



The pipe routing for the underground distribution system will be developed with consideration given to site conditions, previous engineering work and CLC preferences.

16.9 Manpower Requirements

A 365-day working year with three daily shifts, each totalling 7 working hours, has been adopted. This allows one hour for evacuation of blast fumes after end-of-shift blasting. However, the effective working time per shift per day will be less than 7 hours, once travel time and daily pre-shift briefings are accounted for. The effective working time per shift during production operations will be 6.25 hours.

The underground mining team will be organized into Mining and Maintenance operational groups, respectively. The Mining group is broken down into mining supervision, production and technical services, whilst the Maintenance group is broken down into supervision and maintenance teams.

The average hired personnel head count, by operational group in the full steady state production period (Y7 to Y15, will be approximately 212 people. A breakdown by group and subgroup is listed in Table 16-16.

Table 16-16 PMR Personnel

Group	Subgroup	Head Count
Mining	Supervision	11
	Production	116
	Technical Services	20
Maintenance	Supervision	14
	Maintenance	51

16.10 Development and Production Schedule

16.10.1 Production Rate

The production rate is constrained by the total mineable tonnage available, and also by mining practicalities and geotechnical sequencing rules. A single stope per level and block is to be mined at any time so that traffic congestion can be minimized and main ventilation needs per level are not excessive. Likewise, development under secondary ventilation is limited to a maximum of two levels from a level already in the primary ventilation circuit.

The mine production rates tested during scenario analysis were scaled up in annual tonnage steps to assess the maximum mine production potential while maintaining a stable production rate and an optimized grade profile over the LOM. Finally, a value of 1.8Mtpa or 90% of annual ore production has been set as the first target to be increased in the progression to 2.0Mtpa from Year 7.

16.10.2 Pre-production Development

The first stopes, located between the 210RL and 185RL of Block 3, will be mined at the end of Year 3. The development strategy targets initially the higher-grade ore in Blocks 3 and 2. It is followed by both Block 3 within the PMS zone and the deeper SMS zone. Sub-blocks once mined, will enable access to the overlaying DAF zones. Development and construction of significant mine infrastructure including the declines, material handling system and the Railveyor, primary ventilation circuit, surface ventilation shafts, paste delivery system, main pumping system and others will be accomplished during the pre-production period, both prior to and in parallel with the development of these zones.

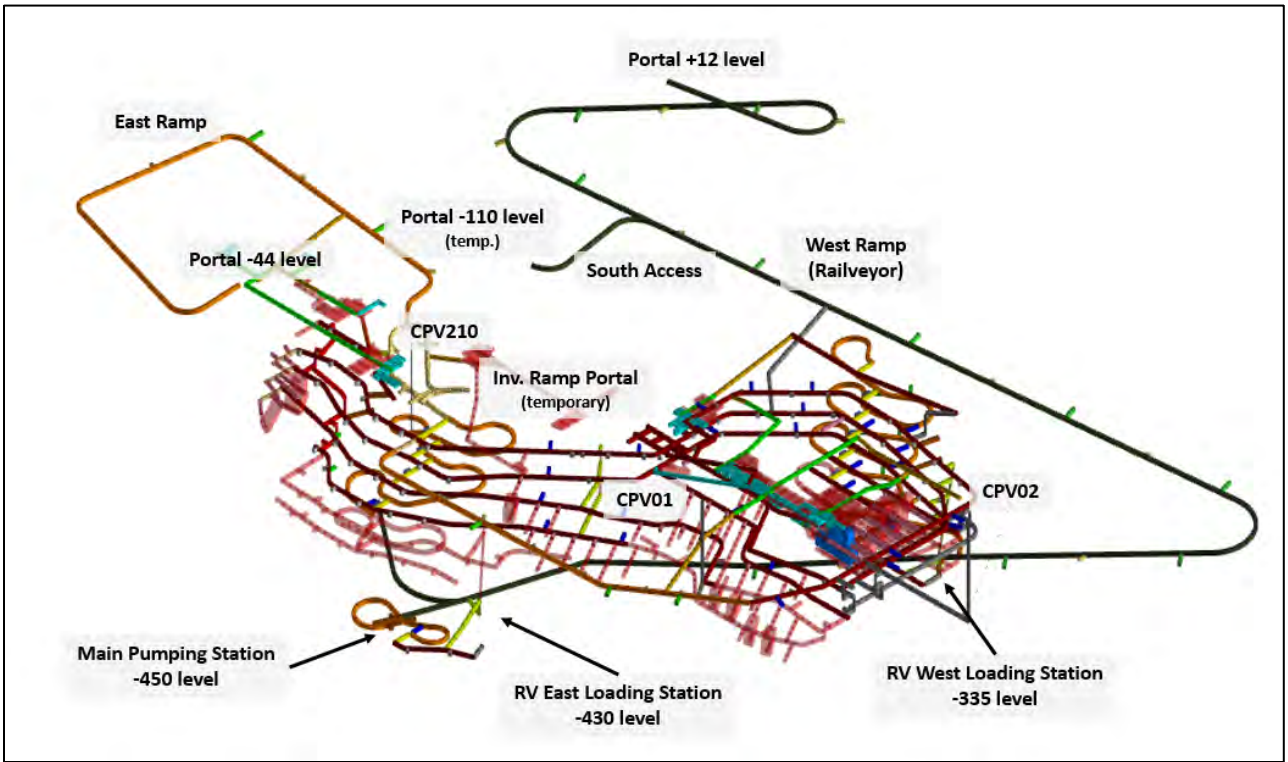
Utilisation of the existing 500m of underground investigation ramp has been adopted as a means of early access for development of the West Ramp. The underground development ramp is currently being rehabilitated and will be kept on care and maintenance until the underground development starts.

Figure 16-34 Rehabilitation Works at Existing Investigation Ramp

Development of the DAF areas will commence as early as possible into the LOM so that the lower productivity from these horizons can be offset by the more productive and less development resource intensive stoping areas. Critical path pre-production activities include:

- Main decline development from the surface portals to establish the main underground access and egress routes as well as the main ventilation intake and exhaust airways.
- Development of the material handling system, including the ore passes, rock breaker and loading stations, in addition to the installation of the Railveyor.
- Construction of the two return air raises required to start stoping production.
- Development of the 185RL, 210RL transport levels and paste drive access required to enable the supply of backfill material to all underground production areas.

Figure 16-35 Extent of the Mine at end of Y4 - Isometric View-Oriented South East



A development summary schedule up to the end of Year 5 is shown in Table 16-17.

Table 16-17 CAPEX Development Schedule

PERIOD	YEAR	DEVELOPMENT (METRES)
Pre-production	Y1 Q1	1,560
	Y1 Q2	2,243
	Y1 Q3	1,770
	Y1 Q4	2,096
	Y2 Q1	1,926
	Y2 Q2	1,910
	Y2 Q3	1,820
	Y2 Q4	1,457
	Y3 Q1	1,903
	Y3 Q2	1,758
	Y3 Q3	1,519
	Y3 Q4	1,182
	Y4 Q1	959
	Y4 Q2	998
	Y4 Q3	656
	Y4 Q4	670
Production	Y5 Q1	443
	Y5 Q2	646
	Y5 Q3	811
	Y5 Q4	398
	Y6	1,878
	Y7	533
	Y8	0
	Y9	0
	Y10	0
	Y11	0
	Y12	0
	Y13	0
	Y14	0
	Y15	0
	Y16	0
	Y17	0
	Y18	0
	Y19	0
	Y20	0
Total		**26,687**

16.10.3 Sustaining Development

Sustaining development, completed throughout the mine in parallel with ongoing CAPEX activities, is required to both supplement the stoping ore production as well as to provide timely access to new production areas so as to reach a 1.8Mtpa production level at a first stage and then increasing to 2.0Mtpa and sustaining this during the LOM.

Table 16-18 includes slot drives and excludes DAF production drives. OPEX development will be capitalized during the pre-production period.

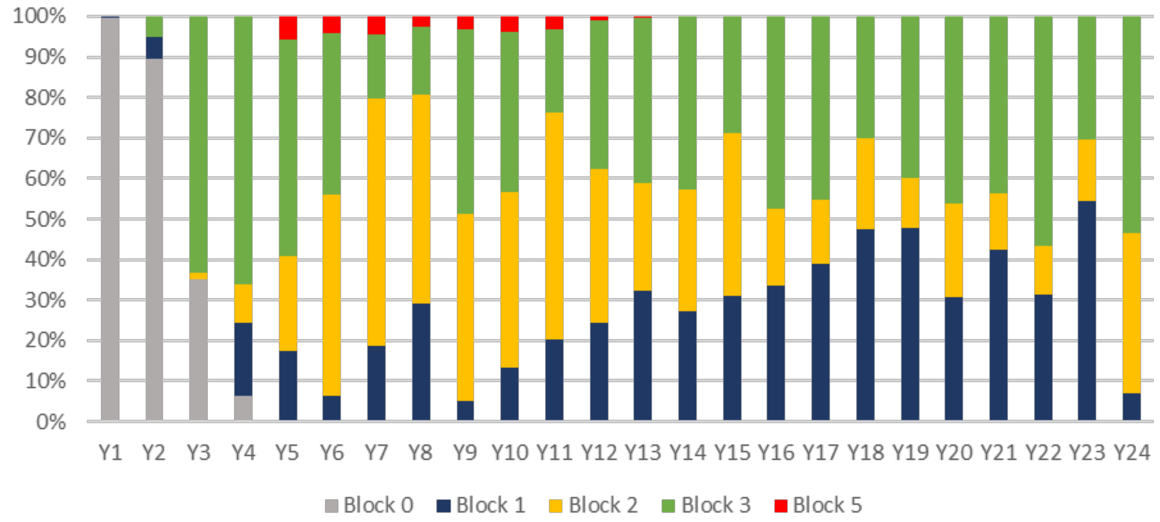
Table 16-18 Sustaining Development Schedule

PERIOD	YEAR	DEVELOPMENT (METRES)
Pre-production	Y1 Q1	40
	Y1 Q2	0
	Y1 Q3	25
	Y1 Q4	15
	Y2 Q1	72
	Y2 Q2	88
	Y2 Q3	178
	Y2 Q4	541
	Y3 Q1	943
	Y3 Q2	240
	Y3 Q3	479
	Y3 Q4	741
	Y4 Q1	1,032
	Y4 Q2	500
	Y4 Q3	801
	Y4 Q4	845
Production	Y5 Q1	1,113
	Y5 Q2	1,184
	Y5 Q3	1,081
	Y5 Q4	868
	Y6	3,584
	Y7	2,559
	Y8	1,801
	Y9	2,650
	Y10	1,306
	Y11	2,251
	Y12	3,751
	Y13	2,177
	Y14	2,124
	Y15	1,799
	Y16	743
	Y17	2,338
	Y18	1,435
	Y19	857
	Y20	1,364
	Y21	1,026
	Y22	565
	Y23	174
	Y24	4
Total		**43,292**

16.10.4 LOM Production Schedule

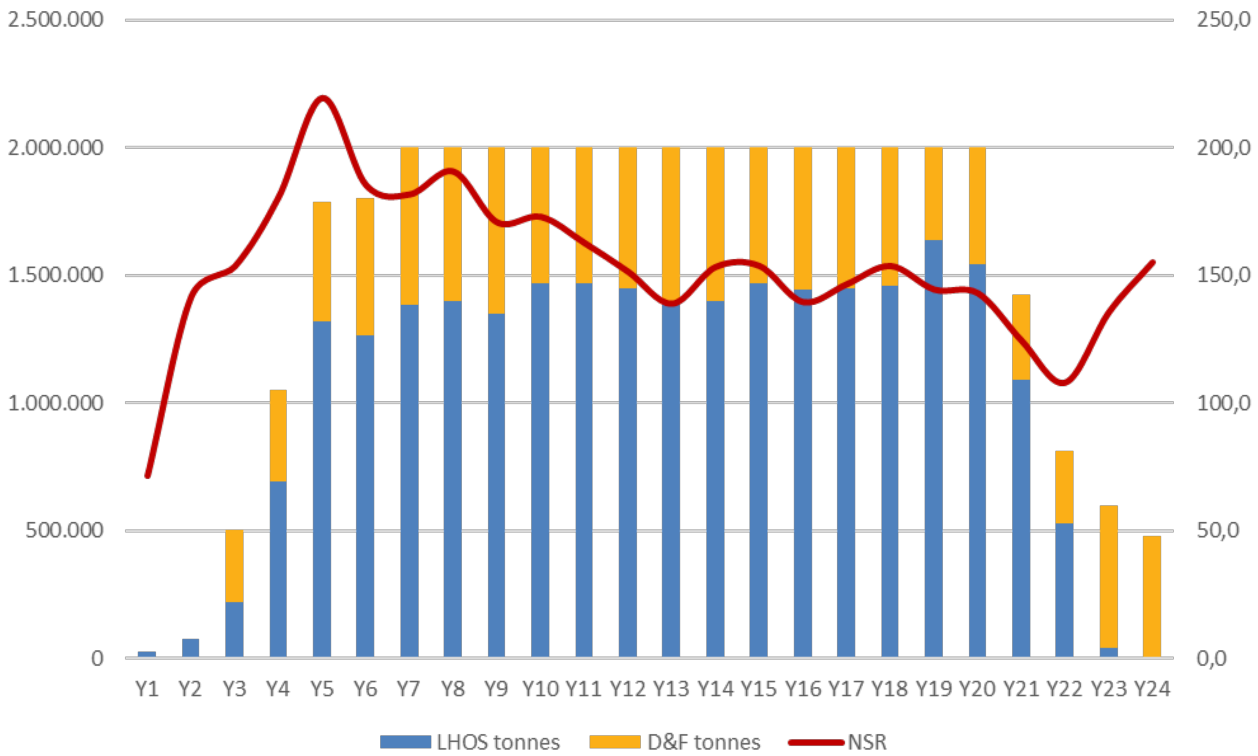
The nominal full 2.0Mtpa production rate is achieved in Year 7 with the bulk of the production sustained by both the PMS and the SMS zones. Figure 16-36 below shows the ore block contribution to LOM production by year. This excludes the scheduling of any Inferred Resources.

Figure 16-36 Contribution of Each Ore Block in the Life of Mine production (excl. Inferred)



The yearly production tonnage and NSR is shown in Figure 16-37. The development activity is inclusive of both DAF production and ore development in stoping zones.

Figure 16-37 LOM Production Schedule by Mining Method (excl. Inferred)



The average element grades and ore mass tonnage throughout the LOM are shown in Table 16-19.

Table 16-19 Mine Production Tonnage and Grades over LOM (excl. Inferred)

YEAR	ORE (KT)	NSR $/t	% CU	% ZN	% PB	PPM AG
1	24	71	0.60	0.70	0.25	10.42
2	75	142	1.01	1.83	0.76	31.45
3	503	154	1.11	2.03	0.92	23.46
4	1,050	181	1.28	2.47	1.10	28.93
5	1,785	220	1.57	2.93	1.37	36.39
6	1,800	185	1.32	2.43	1.20	31.64
7	2,000	182	1.27	2.47	1.13	32.79
8	2,000	191	1.35	2.65	1.17	26.45
9	2,000	171	1.10	2.57	1.11	28.21
10	2,000	173	1.02	2.77	1.32	31.42
11	2,000	163	1.03	2.41	1.12	29.55
12	2,000	151	1.08	2.07	0.88	21.47
13	2,000	139	1.01	1.89	0.74	19.07
14	2,000	153	1.06	2.23	0.88	21.64
15	2,000	154	1.17	1.85	0.89	23.17
16	2,000	140	1.07	1.66	0.75	21.35
17	2,000	147	1.04	1.98	0.86	22.34
18	2,000	154	1.09	2.05	0.95	24.77
19	2,000	144	1.04	1.86	0.82	24.98
20	2,000	143	1.01	1.87	0.87	26.71
21	1,425	124	1.05	1.23	0.59	18.02
22	814	108	0.92	1.03	0.58	14.47
23	597	136	1.03	1.58	0.75	16.55
24	477	155	1.35	1,40	0.61	18.58
Total	**36,546**	**160**	**1.13**	**2.14**	**0.97**	**25.47**

The main milestones to achieve during the development phase are described below.

YEAR 1

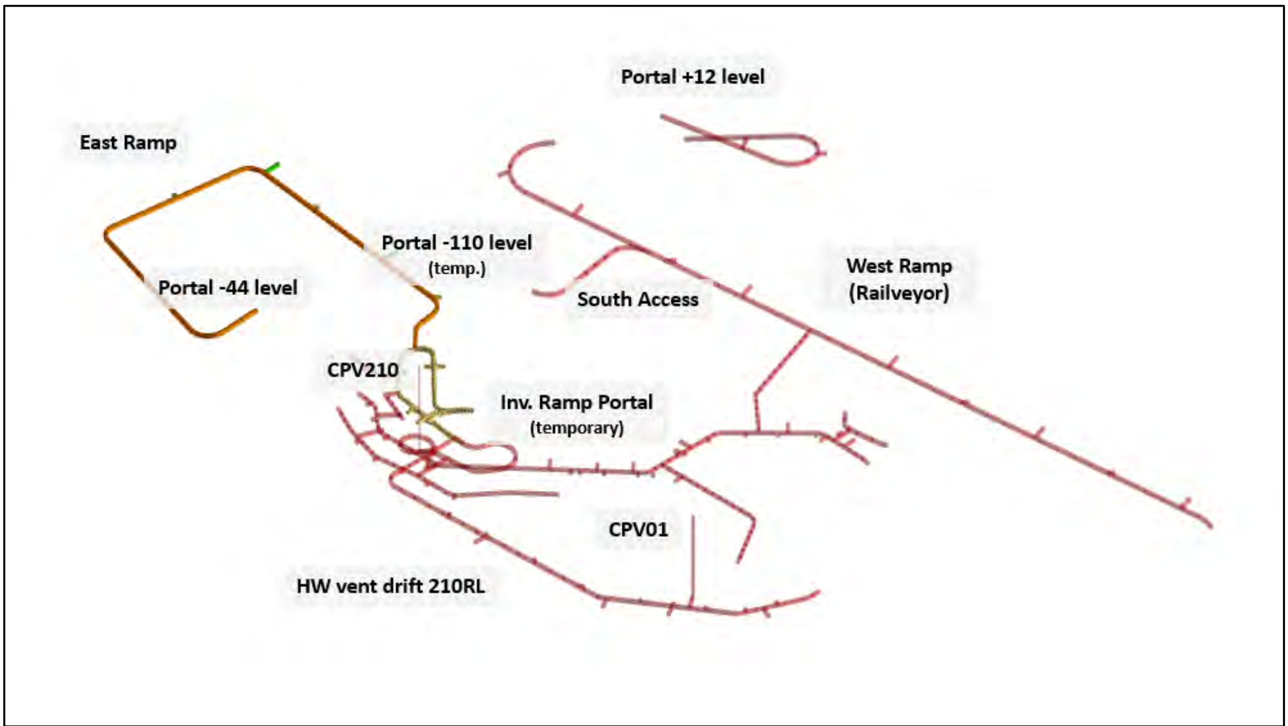
Underground development starts in two independent areas. The East Ramp progresses simultaneously down from the portal in marl at -44mRL and up from the existing investigation ramp with a connection achieved by end of the year.

The investigation ramp will be deepened to open the -210mRL and -235mRL levels. To improve the ventilation in that area a temporary raise will be deepened (CPV210) and a fan will be installed and commissioned on surface. At 210RL, the hanging wall ventilation drift will progress to reach the bottom of the CPV01 raise. This raise will be developed by the end of the year.

A new portal will be opened at the -110-pit level and declined to intersect the West Ramp at an intermediate point. From that point, development of the hauling ramp for the Railveyor will then progress upwards and downwards. A third advance face will progress from surface, crossing the soft rock formation.

Both areas will be connected by the end of the year through the main hauling drift at -210mRL. This will enable the first main ventilation circuit to be completed. The first main pumping station will be developed at -235mRL East.

Figure 16-38 Mine Development at End of Year 1 - Isometric View Oriented South East



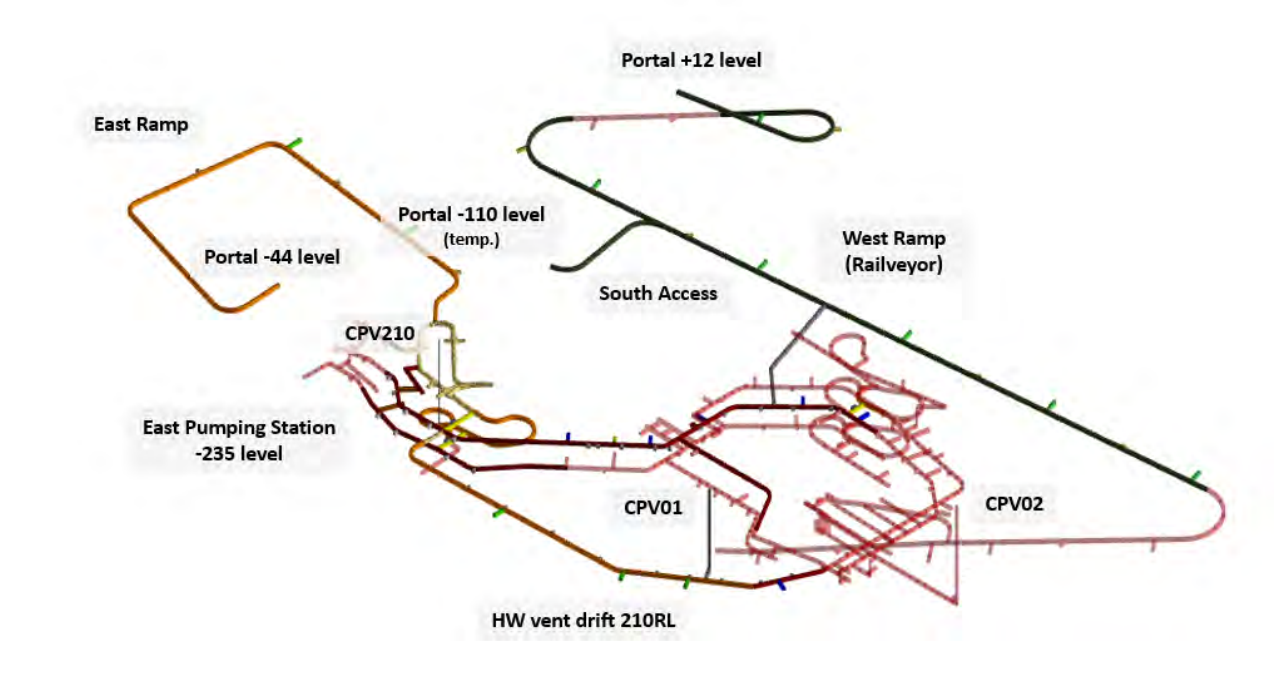
YEAR 2

Pre-production development works in Year 2 will be focused on three main areas:

- The upper section of the main hauling ramp for the Railveyor will be completed and connected with the surface. The ramp deepening will continue to reach the West Loading Station location at 335RL.
- Development at the east will be slowed to prioritise the preparation of the first areas to mine at the West (Block 3).
- A ventilation fan at the CPV01 will be commissioned. The development of the hanging wall ventilation drift will continue towards the CPV02 base and the raise will be completed by the end of the year.

The ramp portal at -128 mRL will be closed and the pit backfilled with Marls to prepare the in-pit tailings facility in advance of the processing plant start-up.

Figure 16-39 Mine Development at End of Year 2 - Isometric View Oriented South East



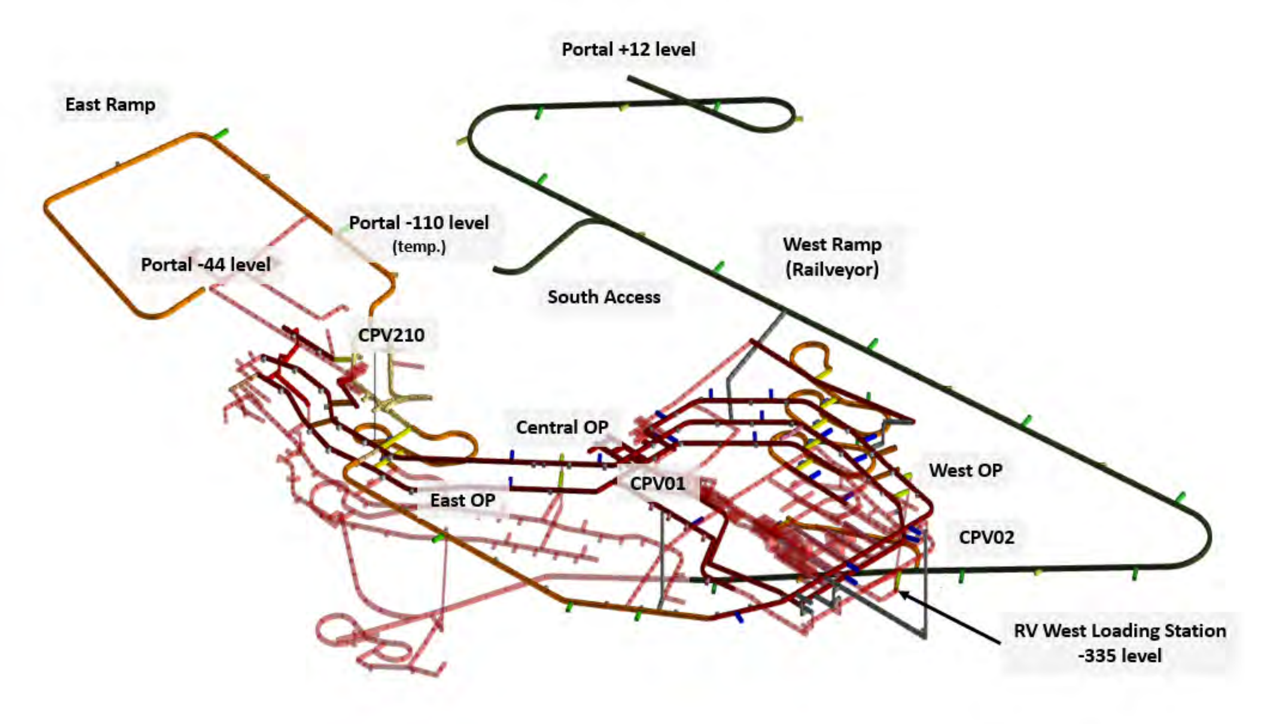
YEAR 3

During Year 3, pre-production mine development and deepening will advance in both the east and west sections. The East Ramp will reach -285mRL and the West Ramp will reach the East Loading Station location. The access ramp to the central (main) pumping station at -450mRL will be completed.

Other Milestones achieved in Year 3 include;

- CPV02 fan commissioned and the main ventilation circuit becomes fully operative.
- For the ore handling system, three ore passes will be raise bored and the West loading station will be under construction.
- Preparatory works will start at Block 2 (Northwest).
- Preparation of Block 3 will be completed and the service paste hole will be commissioned.
- Production stoping will commence and first ore will be hauled to the surface.

Figure 16-40 Mine Development at End of Year 3 - Isometric View Oriented South East

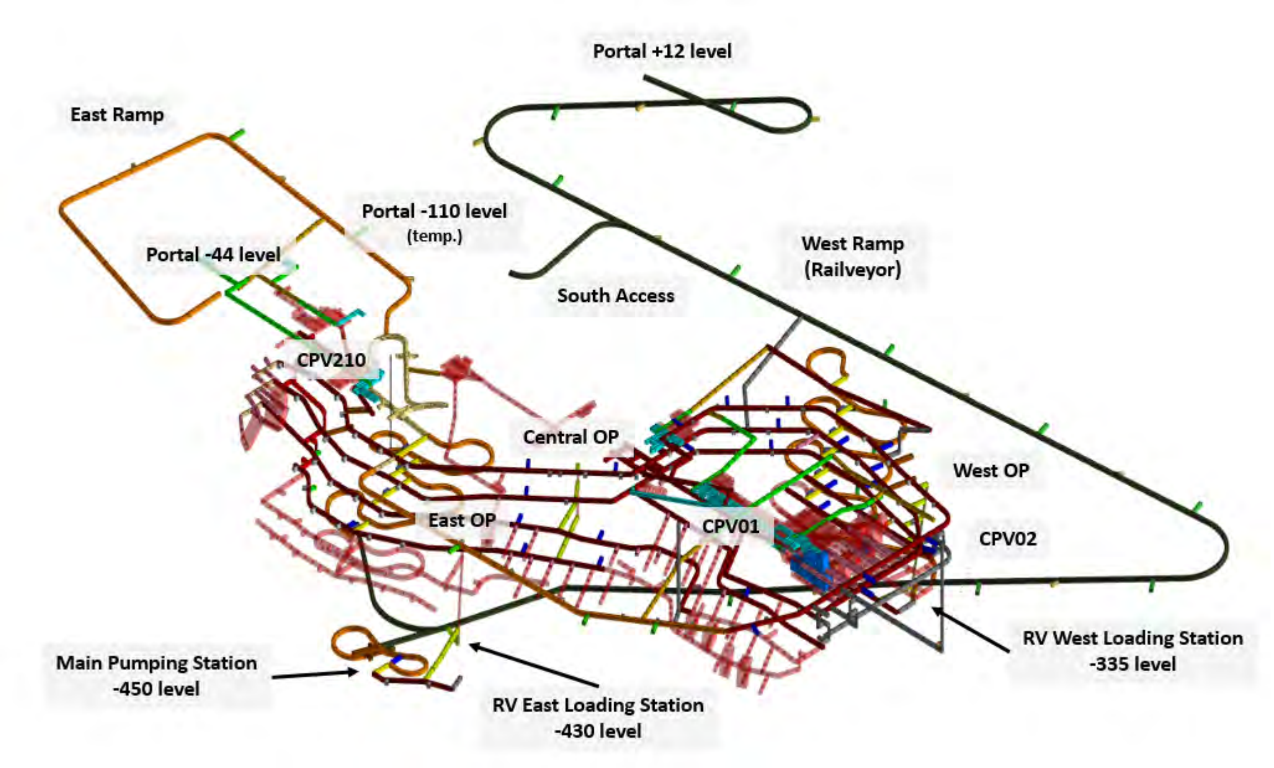


YEAR 4

During Year 4, production ramp up will continue primarily with ore from Block 3 plus some production from the top DAF levels at Blocks 1, 3 and 5. By the end of Year 4 the first stopes will be mined at Block 2 in addition to the eastern most section of Block 1.

At the beginning of the year, the Railveyor system will be commissioned down to the West Loading Station The final section to mine bottom and the east ore handling and loading facilities will be under construction ready to be commissioned early in Year 5. The dewatering system will be centralised in the main pumping station at -450mRL.

Figure 16-41 Mine Development at End of Year 4 - Isometric View Oriented South East



ITEM 17 RECOVERY METHODS

This section provides a brief description of the ore processing techniques, process plant equipment operation, and process plant support facilities. Process plant capital and operating costs are discussed in Item 22.

17.1 Introduction

The Cobre las Cruces mining and processing facility at Seville, southern Spain, has historically mined high grade secondary sulphide copper minerals (chalcocite) from an open pit operation which have then been treated in the process plant to yield ~72,000 tpa of LME Grade A copper cathode. Due to depletion of the secondary sulphides, it is intended to exploit the primary polymetallic sulphide (PMS) Mineral Resources from beneath the open pit via an underground mining operation.

These polymetallic sulphides contain large quantities of zinc, copper, lead, and silver. As the existing processing plant is not able to effectively treat these polymetallic sulphides to recover the pay metals, it is intended to expand and modify the existing plant and to install an additional processing facility, approximately 1 km from the existing plant site, to incorporate the following new and modified major unit processing operations:

- **Crushing and Milling** – installation of a new primary crushing and SABC milling circuit to process 2.2 Mtpa of run-of-mine ore and deliver a feed slurry with a P_{80} of 35 µm to the downstream flotation circuit.
- **Flotation** – installation of a new flotation circuit to recover a bulk flotation concentrate with a throw-away tail. The tailings will be filtered to yield a residue suitable for use as paste backfill and /or for disposal on a tailings storage facility.
- **Primary (ferric) Leaching** – installation of an additional pre-leach reactor and conversion of the existing leaching circuit to the silver catalysed atmospheric leaching (SICAL) process.
- **Copper pH Adjust** – installation of new copper pH adjust reactors and re-use of the existing gypsum thickener and clarifier for this circuit.
- **Copper Solvent Extraction** – implement some minor conversions to the existing copper solvent extraction circuit.
- **Copper Electrowinning** – implement some minor modifications to the existing copper electrowinning circuit.
- **Iron Removal** – installation of a new iron removal circuit to ensure removal of iron prior to zinc solvent extraction.
- **Zinc Solvent Extraction** – installation of a new zinc solvent extraction circuit, comprising separate primary and secondary zinc extraction circuits.
- **Zinc pH Adjust** – re-purpose and utilise equipment from the existing pre-neutralisation circuit to neutralise acid from a bleed of the primary zinc solvent extraction raffinate.
- **Zinc Electrowinning** – installation of a new zinc electrowinning circuit to produce up to 50,000 tpa super high grade (SHG) zinc cathodes.
- **Zinc Melting and Casting** – installation of new melting and casting facilities to process the cathodes from zinc electrowinning and produce up to 50,000 tpa of zinc ingots.
- **Secondary (Brine) Leaching** – installation of a new leaching circuit using an acidic brine lixiviant to extract lead and silver from the primary leach residue.

- **Silver Recovery** – installation of a new silver cementation and filtration circuit to recover silver from the secondary leach circuit pregnant leach solution (PLS), and a separate silver purification circuit to generate a saleable silver cement product.
- **Lead Recovery** – installation of a new lead cementation circuit to recover lead from the silver-depleted secondary leach PLS and produce lead metal briquettes.
- **Aluminium and Sulphate Removal** – installation of a new aluminium and sulphate removal circuit to process a bleed stream of recycle secondary leach lixiviant to remove excess aluminium and sodium sulphate, and produce a solid residue for disposal.
- **Effluent Treatment** – modify the existing effluent treatment plant to process the final plant effluent stream, produce a solid residue for disposal, and generate a purified effluent suitable for discharge.
- **Paste Backfill Plant** – installation of a new paste backfill plant, located midway between the existing and proposed new process plants, to provide cemented backfill for underground support.

In addition to these major processing steps, modifications to existing services and utilities will be required as well as the installation of new services and utilities at the proposed new plant site. The key design criteria for the proposed PMR Project are summarised in Table 17-1.

Table 17-1 Key Process Design Criteria

Description	Unit	Design Value
Life of Project	y	20
Ore Milled, dry	t/y	2,200,000
Design Ore Blend, Primary Sulphide: Stockwork (*)	ratio	60 : 40
Plant Operation	days/y	365
Plant Operating Hours	h/y	7,884
Milling Circuit Configuration	-	SABC
Milling Circuit Specific Energy	kWh/t	30.3
Milled Product Particle Size, P80	µm	35
Flotation Concentrate Mass Pull	%w/w	35
Primary Leach Feed Particle Size, P80	µm	10
Copper Recovery (overall)	%	85
Zinc Recovery (overall)	%	88
Silver Recovery (overall)	%	52
Lead Recovery (overall)	%	79
Peak / Average Copper Production over 20 Years	tpa	28,044 / 20,147
Peak / Average Zinc Production over 20 Years	tpa	54,229 / 39,477
Peak / Average Silver Production over 20 Years	tpa	42 / 29
Peak / Average Lead Production over 20 Years	tpa	24,737 / 17,265

(*) Design blend has a higher percentage of "stockwork" than indicated in the resource model. This ore type is harder and thus this blend imparts a degree of conservatism into the milling and crushing circuits to accommodate feed variability.

17.2 Plant Layout and Design Considerations

The proposed new process plant site will be located to the south of the existing open pit and adjacent to the existing mine haul road and has been located to suit the available usable area within a restrictive topography, while providing appropriate access for all supporting infrastructure. This proposed new plant location was once a waste stockpile which has subsequently been cleared and levelled to present a suitable flat site for construction.

The plant has been oriented to enable ore from the new underground workings to be delivered to surface, via a portal, by a Railveyor™ and discharged either into the ROM bin or to the emergency stockpile. The design and location of the ROM pad will enable ore from the emergency stockpile to be reclaimed by FEL, loaded into mine trucks and subsequently hauled and tipped into the ROM bin.

The existing mine haul road will enable ore from existing surface stockpiles to be delivered to the ROM pad access ramp and discharged into the ROM bin. The proposed new plant layout also enables the main process plant residues (secondary leach residue, iron removal residue, aluminium removal residue) to be back-loaded into trucks using a FEL and disposed into impoundment facilities via the existing mine haul road.

It is proposed that the new paste plant will be located adjacent to the existing mine vehicle workshop, approximately midway between the existing and new process plant sites. This location is deemed the most appropriate to facilitate pumping of cemented tailings to the underground workings and for pumping thickened flotation tailings to the existing TSF.

A layout of the existing process plant with a schematic of the new process plant superimposed on the site is presented in the figure below.

Figure 17-1 Existing and Proposed New Process Plant Layout

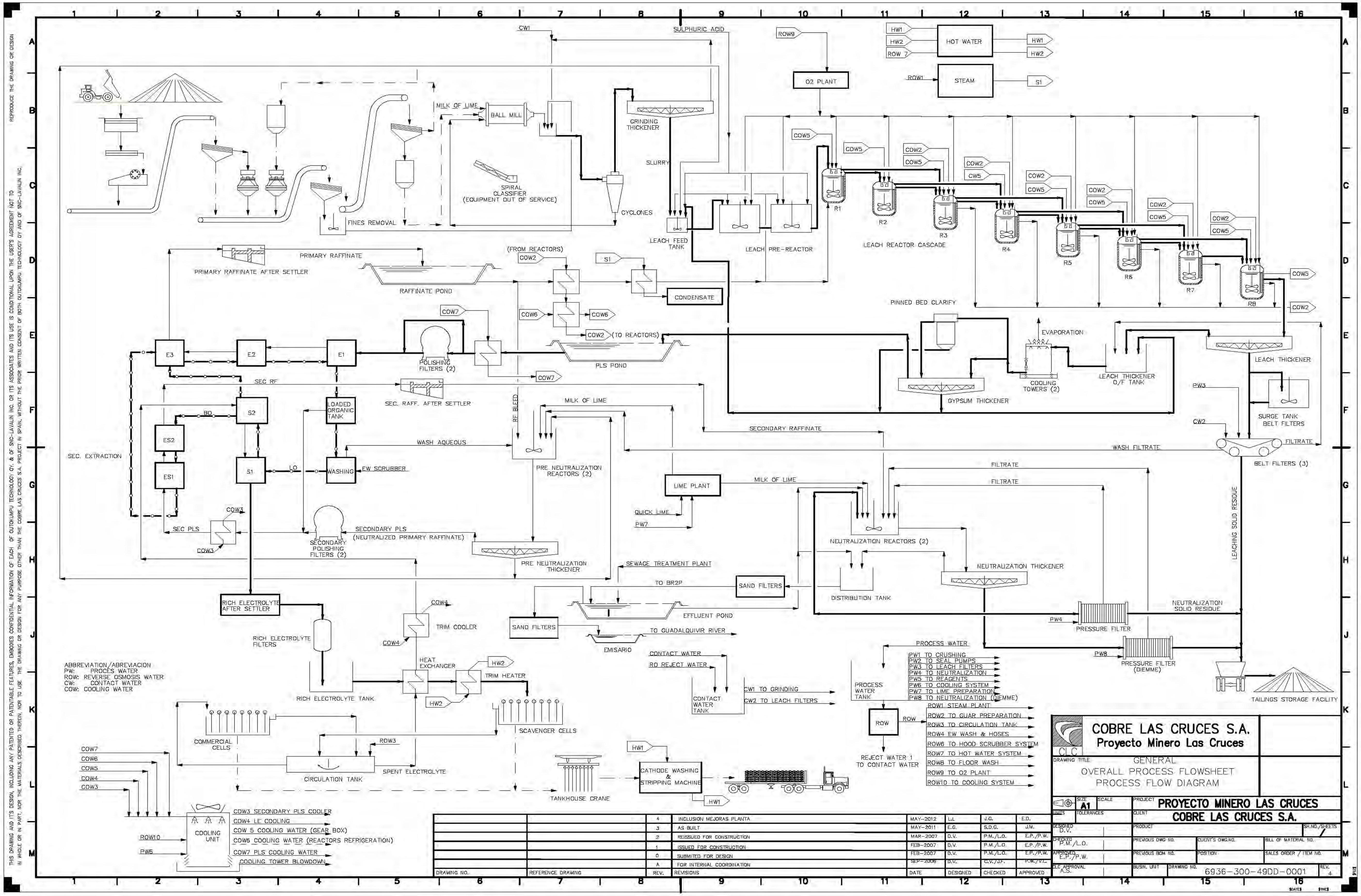
17.3 Current Processing Facility

The current processing facility is designed to operate 365 days per year, 24 hours per day and process 1.5 Mtpa of secondary ore to produce up to 72,000 tpa of cathode copper. The ore from the historical open cast mine ranged from 5% to 10% Cu with a nominal design criteria of 6.02% Cu. The copper in the secondary ore is mostly found in chalcocite with some minor amounts found in chalcopyrite, tennantite-tetrahedrite complex, and enargite.

The plant relies on an atmospheric leaching process to recover copper from the rich secondary sulphide chalcocite mineralization. A unique feature of the plant is the use of agitated reactor tanks to dissolve the copper under conditions of high temperature and high acidity. Oxygen is also added into the reactors to complete the reaction. The feed to the leaching reactor tanks is mined ore that has passed through three stages of crushing, a single stage of grinding and has then been thickened to eliminate as much process water as possible.

Once leached, the liquid is separated from the ground solids to become PLS, the feed for the solvent extraction (SX) area. In the SX area the copper is passed to an organic solution and then to the electrolyte that feeds the electrowinning cells. The electrowinning cells produce LME grade copper cathodes weighing approximately 50 kilograms each. An automated crane and stripping machine then harvests and packages the cathodes for shipment. A simplified process flow diagram for the plant is presented in the figure below.

Figure 17-2 Current Las Cruces Process Flowsheet



17.4 PMR Plant Process Description

The current processing plant described above requires expansion, modification and enhancement, including installation of a new processing facility approximately 1 km from the existing plant site, to incorporate the following new and modified major unit processing operations associated with the inclusion of PMR. A simplified block flow diagram of the proposed PMR Project flowsheet is presented in Figure 17-3.

17.4.1 Crushing and Stockpiling

Run-of-Mine (ROM) ore from underground will be direct tipped from a Railveyor™ into the rear section of a ROM bin. A facility will also be provided to enable the Railveyor™ to discharge onto an emergency stockpile, thereby allowing mining activities to temporarily continue if the process plant primary crushing circuit is off-line.

The ore will then be fed from the ROM bin, using an inclined apron feeder, to a vibrating grizzly. Grizzly oversize material will discharge to a single toggle primary jaw crusher which will reduce the ore product P_{80} to approximately 160 mm.

The crushed ore product will be conveyed to a covered conical stockpile. The stockpile will have a live capacity of ~6,700 tonnes, roughly equivalent to 24 hours of operation at the nominal milling throughput rate. The stockpile will have a dome cover to minimise dust generation around the site and will have front-end-loader (FEL) access from both sides.

Dust collection will be provided at the ore transfer points via a bag-house installation.

17.4.2 Ore Reclaim

Primary crushed mill feed will be reclaimed from the crushed ore stockpile at a controlled rate from dual variable speed reclaim apron feeders, discharging onto a mill feed conveyor.

A SAG mill ball receival bunker will be provided for receipt and storage of SAG mill grinding media. A ball charging hopper will be provided to facilitate ball loading onto the SAG mill feed conveyor using a FEL.

17.4.3 Milling, Classification, and Recycle Crushing

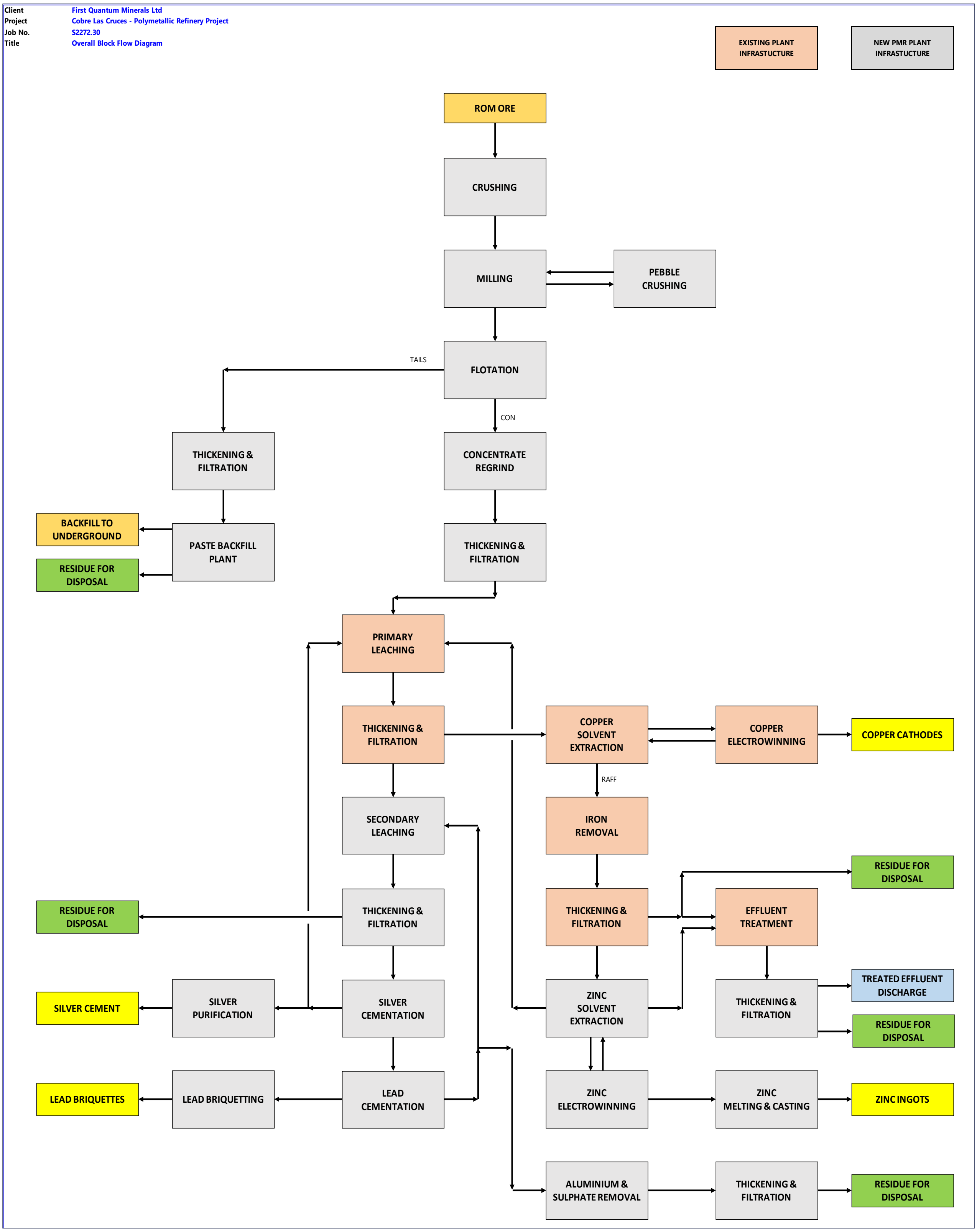
The purpose of the milling circuit is to produce a slurry with a target particle size P_{80} of 35 µm suitable for presentation to the downstream flotation circuit. The milling circuit will comprise separate SAG and Ball mills with intermediate pebble crushing (SABC).

SAG Milling

The grinding circuit will consist of a single SAG mill operating in closed circuit with a horizontal vibrating classification screen, followed by a Ball mill operating in closed circuit with a cyclone cluster. The requirement to operate the SAG mill in closed circuit is due to the fine product particle size (P_{80} of 35 µm) – operating the SAG mill in open circuit with the closed circuit Ball mill would be unlikely to consistently achieve the product target particle size.

The SAG mill will be a 6.71 m (22 ft) diameter x 5.50 m (18 ft) EGL grate discharge mill incorporating pebble ports.

Figure 17-3 Overall Block Flow Diagram



Recycle Crushing

Oversize material (washed pebbles and worn grinding media) from the SAG mill trommel will be directed to the pebble transfer conveyor. The tramp metal will be removed from the trommel oversize using a cross belt electromagnet mounted above the head pulley of the transfer conveyor. The pebbles will then discharge onto the pebble crusher feed conveyor before reporting to the pebble crusher feed bin.

The pebble crusher will comprise a re-purposed (from the existing process plant operation) crusher with a nominal 12 mm closed side setting, yielding a crushed product P_{80} of 15 mm.

Ball Milling

The Ball mill will be a pinion drive 6.10 m (20 ft) diameter x 9.20 m (30 ft) EGL overflow mill with a 6.2 MW motor and will produce a product P_{80} of 35 µm at the nominated 2.2 Mtpa (279 t/h) throughput and design ore blend (60%w/w polymetallic ore, 40%w/w stockwork ore).

17.4.4 Flotation

The purpose of the flotation circuit is to generate a flotation concentrate so that it may be leached in the downstream primary leaching circuit to solubilise copper and zinc values. The expected mass pull to flotation concentrate will be 35%w/w of new flotation feed, or equivalent to 770,000 tpa based on a new feed throughput of 2.2 Mtpa. The flotation circuit will comprise separate roughing and scavenging cells and a regrind facility to generate a bulk flotation concentrate with a target particle size P_{80} of 10 µm.

Rougher Flotation

Slurry will flow to the first in a bank of four rougher flotation tank cells. Each cell will have a nominal volume of 70 m^3, providing an approximate 25 minutes total residence time in the roughing circuit at the nominal 30%w/w solids pulp density. Concentrate from the cells will be directed to concentrate regrind.

Scavenger Flotation

Rougher tails slurry will discharge to the first in a bank of four scavenger flotation tank cells. Each cell will have a volume of 70 m^3, providing an approximate 25 minutes residence time in the scavenging circuit at the nominal 29%w/w solids pulp density. Concentrate from the cells will be directed to concentrate regrind.

Scavenger tails slurry will be transferred to the flotation tailings thickener located at the Paste Plant approximately 500 m away.

Concentrate Regrind

Combined rougher and scavenger concentrate slurry will be transferred to the concentrate cyclone cluster. Cyclone underflow will report to the concentrate regrind mill while cyclone overflow will report to the concentrate surge tanks.

The concentrate regrind mill, equipped with a 3.5 MW drive, will operate in open circuit using 2 mm diameter ceramic grinding media to yield a product particle size P_{80} of 10 µm. The regrind mill discharge will combine with the concentrate cyclone cluster overflow on a horizontal vibrating trash screen, equipped with 0.63 mm x 8.8 mm polyurethane panels, to remove oversize tramp material that could potentially contaminate the final concentrate and the process water system.

Concentrate Storage and Pumping

Trash screen undersize will gravitate to two mechanically agitated concentrate surge tanks. These tanks will provide ~4 hours surge capacity at nominal production rates between the flotation circuit and the concentrate thickener located at the existing plant site.

Concentrate Thickening

An existing 26 m diameter thickener will be used to de-water the flotation concentrate prior to filtration.

Concentrate Filtration

Thickened flotation concentrate will be filtered on three existing horizontal vacuum belt filters. The combined filtrate from the filters will report to an existing filtrate tank which will then be recycled. Filtered solids will discharge from each filter onto the concentrate filter discharge conveyor which in turn will discharge the filter cake onto the concentrate storage conveyor for subsequent transfer and discharge within the concentrate storage shed. The concentrate storage shed will provide storage capacity equivalent to two days of nominal throughput. Filtered concentrate will be reclaimed from the storage shed to the repulping facility using a FEL.

Flotation Tailings Thickening and Filtration

Scavenger flotation tails slurry will be directed to a 30 m diameter high rate tailings thickener located at the Paste Plant site.

Thickened slurry, at a target 70%w/w solids, will be delivered to the mechanically agitated paste filter feed tank for subsequent filtration. The filtrate will be returned to the tailings thickener while the filter cake will either be fed to the paste mixer or diverted onto the final tailings conveyor.

Tailings thickener overflow solution will flow to the adjacent process water tank where it will combine with the overflow solution returned from the concentrate thickener in the existing plant. The process water tank will be equipped with a rake mechanism and underflow pump to prevent accumulation of settled solids within the tank. The dilute underflow slurry from the process water tank will be returned to the tailings thickener. Process water pumps will return process water from the process water tank to the milling and flotation circuits. Make-up water to the process water tank will be pumped from the existing water treatment plant. Grey water from the new sewage treatment plant, contact water, and cooling tower blowdown will also be diverted to the process water tank.

17.4.5 Primary Leaching

The primary leaching circuit will be configured with three pre-leach reactors (two existing) and eight primary leach reactors (existing). The purpose of the primary leaching circuit will be to solubilise copper and zinc from their respective chalcopyrite and sphalerite minerals to yield a copper and zinc-rich pregnant leach solution (PLS). Lead and silver will also be liberated from their respective minerals during the leaching process and will precipitate as sulphates in the acidic sulphate lixiviant, reporting to the leach discharge solids.

The leaching process will be carried out at around 20%w/w solids pulp density using an acidic ferric sulphate lixiviant catalysed with silvers. The silver catalyst will be generated from the silver cementation circuit following the secondary leaching process further downstream. The principal reactions in the primary leaching circuit are as follows:

- $CuFeS_2 + 2Fe_2(SO_4)_3 \rightarrow CuSO_4 + 5FeSO_4 + 2S°$
- $ZnS + Fe_2(SO_4)_3 \rightarrow ZnSO_4 + 2FeSO_4 + S°$
- $PbS + Fe_2(SO_4)_3 \rightarrow PbSO_4 + 2FeSO_4 + S°$
- $FeS_2 + 7Fe_2(SO_4)_3 + 8H_2O \rightarrow 15FeSO_4 + 8H_2SO_4$
- $FeSO_4 + H_2SO_4 + ½O_2 \rightarrow Fe_2(SO_4)_3 + S°$

A minor fraction of pyrite will also be leached within the tank reactors according to the following chemical reaction:

- $FeS_2 + 7Fe_2(SO_4)_3 + 8H_2O \rightarrow 15FeSO_4 + 8H_2SO_4$

Pre-Leach Reactors

Flotation concentrate from the concentrate filtration area will be repulped with acidic primary raffinate solution from the downstream zinc solvent extraction circuit in a mechanically agitated repulp tank and pumped to the existing pre-leach feed tank.

Silver cement catalyst, zinc raffinate, and sulphuric acid, will also be added to the pre-leach feed tank. The slurry will then be pumped to the first in a series of three mechanically agitated atmospheric pre-leach reactors (two existing, one new) each with a nominal 1,000 m^3 capacity to provide approximately ten hours of leaching residence time.

Oxygen will be sparged into each of the reactors through a sparge ring located beneath the reactor agitator. Oxidation of the sulphide minerals will result in an exothermic reaction and cooling water will be applied to a submerged cooling coil in each reactor to control the slurry temperature to a target 80°C.

Primary Leach Reactors

Slurry discharge from the final pre-leach reactor will be pumped to the first of eight existing atmospheric primary leach reactors, arranged in a series cascade overflow configuration, to provide a nominal seven hours leaching residence time.

Submerged cooling coils in each of the reactors will enable the slurry temperature to be controlled to ~90°C. Acidic vapour from the primary leach reactors will be ducted to the primary leach cooling towers.

Slurry Cooling and Thickening

Discharge slurry from the final primary leach reactor will be partially cooled in a pair of existing slurry cooling towers which will also scrub the off-gas vapour from the pre-leach and primary leach reactors. The partially cooled slurry will then be pumped to an existing 20 m diameter leach discharge thickener.

The thickener underflow slurry will be pumped to the primary leach filtration circuit to separate the primary leach residue from the copper and zinc-rich PLS. The partially cooled overflow will be pumped to a pair of existing solution cooling towers to further cool the PLS and effect precipitation of gypsum. The cooling tower discharge will gravitate to the copper pH adjust circuit for partial neutralisation of the PLS prior to copper solvent extraction.

Primary Leach Filtration

Underflow slurry from the leach discharge thickener will be pumped to the existing primary leach filtration circuit to effect separation of the solids from the copper and zinc-rich PLS. The filtration circuit comprises a mechanically agitated filter feed tank delivering thickened slurry to three existing Larox tower filters. The filtrate will report to the filtrate tank and will be returned to the primary leach thickener.

The filter cake will be washed with treated water to maximise recovery of PLS. The filtrate from the filter cake wash cycle will be collected in a dedicated washate tank and pumped to a water recovery package to maximise recovery of clean water for recycle to the filter wash duty, thereby reducing the impact on the overall site water balance.

The washed filter cake from the primary leach filters will discharge into an existing storage bunker. A FEL will be used to load the primary leach residue into a hopper which will discharge onto conveyor and then into a repulp tank where it will be repulped with brine from the secondary leaching circuit. Alternatively, a FEL may be used to remove filter cake from the bunker and relocate to an adjacent storage area which will be extended to accommodate ~6,000 tonnes of primary leach residue to provide buffer capacity between the primary and secondary leaching circuits.

The repulped slurry will be pumped to the secondary leaching circuit located approximately 1.1 km from the repulp facility at the proposed new plant site.

Copper pH Adjust

PLS from the primary leach solution cooling towers will flow to the copper pH adjust reactors. Hydrated lime will be added to the reactors to achieve the target pH of typically ~1.5.

The discharge slurry will be pumped to an existing 26 m diameter thickener. The thickener underflow slurry will be pumped to the primary leach filter feed tank while a separate seed recycle pump will return a portion of the thickener underflow stream to the pH adjust reactors to improve particle growth and reduce the formation of scale.

The thickener overflow solution will report to a copper PLS pinned bed clarifier for removal of particulates from the thickener overflow stream prior to copper SX.

17.4.6 Copper Solvent Extraction (SX)

The primary leach solution will be sent to the existing plant Copper SX circuit for the recovery of copper from solution. The existing copper SX plant has operated at up to 72000tpa copper and has an inherent capacity well in excess of PMR requirements. The plant will remain fundamentally unchanged.

Details of the existing Copper SX can be found in previous versions of the 2015 CLC Technical.

The two products from the Copper SX are a rich electrolyte directed to Copper Electrowinning, and a zinc rich raffinate directed to Zinc Solvent Extraction.

17.4.7 Copper Electrowinning (EW)

The existing Copper EW circuit will be utilised as is without modification. This circuit has operated at up to 72000tpa finished product, well in excess of PMR requirements.

The EW circuit produces two primary products, copper cathode and a weak, or spent, electrolyte which is returned to the Copper SX circuit.

Details of the existing Copper EW can be found in previous versions of the 2015 CLC Technical Report.

17.4.8 Iron Removal

Prior to Zinc solvent extraction it is necessary to remove iron from the copper raffinate stream.

Air (oxygen) will be used to oxidise ferrous to ferric species and lime to neutralise both the acid in the incoming feed and the acid generated during the ferrous oxidation according to the following chemical equations:

- $4FeSO_4 + O_2 + 10H_2O \rightarrow 4Fe(OH)_3 + 4H_2SO_4$
- $H_2SO_4 + Ca(OH)_2 \rightarrow CaSO_4.2H_2O$

The precipitated ferric hydroxide and gypsum will be removed by thickening and filtration while the now iron-free zinc PLS will advance to the new zinc solvent extraction circuit.

Iron Removal Reactors

Raffinate from the copper solvent extraction circuit will be pumped to a series of four mechanically agitated atmospheric iron removal reactors arranged in a cascade overflow configuration. Each reactor will have an active volume of ~400 m^3 to provide a total six hour residence time. Plant air will be sparged into the reactors through a sparge ring located beneath the reactor agitator impeller. Hydrated lime will be added to the reactors to achieve a terminal target pH of 4.0.

Thickening and Clarification

Discharge slurry from the final iron removal reactor will be pumped to a new 40 m diameter high rate iron removal thickener.

Thickener overflow solution will report to a mechanically agitated overflow tank which will then be fed to the iron removal pinned bed clarifier for removal of particulates from the thickener overflow stream. The settled solids will be returned as a dilute slurry to the upstream thickener feed tank. The clarified solution will be pumped to the zinc solvent extraction circuit.

Filtration

Underflow slurry from the iron removal thickener will be pumped to a new filtration circuit to effect separation of the solids from the zinc-rich PLS. The filtration circuit will comprise three 120 m^2 horizontal vacuum belt filters. The filter cake will be washed with treated water to maximise recovery of PLS. Both the filtrate and washate streams will be returned to the thickener feed tank.

Washed filter cake from the filters will be conveyed using the iron removal residue conveyor to the common iron removal residue bunker. Approximately one third of the iron residue will be loaded by FEL into a hopper and repulped with zinc secondary raffinate (<0.1 g/L Zn). The resulting slurry will then be pumped overland to the effluent treatment plant, located at the existing plant site, where the residual iron will complex with soluble arsenic to form a stable ferric arsenate precipitate.

The remaining two thirds of the iron residue will be loaded by FEL into mine trucks and hauled to the TSF for subsequent disposal.

17.4.9 Zinc Solvent Extraction

Clarified iron-free zinc PLS will be pumped from the Iron Removal circuit to a new zinc solvent extraction facility comprising separate primary and secondary extraction circuits with intermediate acid neutralisation (zinc pH adjust).

The function of the zinc solvent extraction circuit is very similar to that of the upstream copper solvent extraction process. The zinc will be selectively extracted into an organic phase and then extracted back into a more concentrated aqueous phase so that it is suitable for electrowinning. An organic extractant will be mixed with a high flash-point diluent and contacted counter-currently with the incoming zinc PLS feed in the extraction mixer-settlers. At low acidity zinc transfers from the aqueous PLS phase onto the organic phase, generating an equivalent amount of sulphuric acid, according to the following reaction:

- $2RH_{(org)} + Zn^{2+}_{(aq)} + SO_4{}^{2-}_{(aq)} \rightarrow R_2Zn_{(org)} + H_2SO_{4(aq)}$

where RH represents the organic extractant.

After primary extraction the zinc depleted and acid rich aqueous phase (raffinate) will be returned to the primary leaching circuit while a raffinate bleed stream will be diverted to an acid neutralisation circuit before recovering the residual zinc through a secondary solvent extraction circuit. The high concentration of zinc in the PLS feed stream results in the generation of a high acid concentration which reduces the organic extraction efficiency. This results in a typical raffinate composition from the primary extraction circuit of ~16 g/L zinc and 45 g/L sulphuric acid. The raffinate bleed stream to the acid neutralisation circuit is required, therefore, to not only address the overall circuit water balance but also to neutralise the acid to allow the remaining zinc to be recovered.

The zinc loaded organic will then be stripped in the stripping section of the primary solvent extraction circuit at high acid concentration. Stripping is thus a reverse reaction to extraction, where the zinc transfers from the organic phase onto the aqueous phase consuming an equivalent amount of sulphuric acid, according to the following reaction:

- $R_2Zn_{(org)} + H_2SO_{4(aq)} \rightarrow 2RH_{(org)} + Zn^{2+}_{(aq)} + SO_4^{2-}{}_{(aq)}$

An additional feature of the zinc solvent extraction circuit is the requirement to remove ferric from the circulating organic which is achieved by contacting an organic bleed stream with a solution of hydrochloric acid.

The primary extraction, washing, stripping, secondary extraction, and iron stripping operations will each be performed in a series of mixer-settlers – the mixers provide the contact between the organic and aqueous phases while the settlers enable the two phases to separate based on their different physical and chemical properties. The organic is recovered and continuously circulates throughout the zinc solvent extraction circuit counter-currently to the direction of the aqueous flow.

Primary Extraction

Iron-free zinc PLS will be pumped from two 800 m^3 capacity surge tanks to the first of three zinc primary extraction mixer-settlers where it will be contacted counter-currently with a recirculating organic extractant. The raffinate from the final extraction mixer-settler will flow to an after-settler to remove entrained organic before reporting to two 800 m^3 capacity raffinate surge tanks.

Washing

Loaded organic from the zinc primary extraction stages will flow to a loaded organic tank from where it will be pumped to the first of three wash mixer-settlers. A small volume of RO water will then be used to wash the loaded organic and remove entrained contaminants prior to stripping. The wash raffinate will be pumped to the zinc pH adjust circuit while the washed organic will flow to the adjacent stripping mixer-settlers.

Stripping

The washed loaded organic will flow to the first of two zinc stripping mixer-settlers where it will be contacted counter-currently with spent electrolyte from the zinc electrowinning circuit. The stripped organic will overflow from the second stage strip settler to the third stage of the three secondary zinc extraction stages. The loaded strip liquor will flow to the zinc-rich electrolyte after-settler for separation and subsequent removal of entrained organic ahead of zinc electrolyte filtration.

Zinc pH Adjust

The Zinc pH Adjust circuit is located at the existing plant site and will utilise mostly existing equipment. A bleed stream of zinc primary raffinate will be pumped overland from the Zn solvent extraction circuit at the new plant site to the first of three mechanically agitated atmospheric reactors arranged in a series cascade configuration at the existing plant site. Powdered lime will be added to the reactors to achieve a target pH in the final reactor discharge solution of ~4.0.

The discharge slurry will be pumped to an existing 10 m diameter thickener. The thickener underflow slurry will be pumped to the primary leach filter feed tank while a separate seed recycle pump will return a portion of the thickener underflow stream to the pH adjust reactors to improve particle growth and reduce the formation of scale.

The thickener overflow solution will report to an overflow tank which will then be fed to a pair of secondary zinc PLS polishing filters for removal of particulates from the thickener overflow stream. Accumulated solids will be discharged from the polishing filters as required into a repulp tank and returned to the thickener feed tank. The clarified solution will flow to a surge tank before being pumped overland to the secondary zinc solvent extraction circuit at the new plant site.

Secondary Extraction

Clarified secondary zinc PLS from the zinc pH adjust circuit will be pumped to the first of three zinc secondary extraction mixer-settlers where it will be contacted counter-currently with a bleed stream of stripped organic from the zinc strip mixer-settlers. Raffinate from the final secondary extraction stage will flow to an after-settler to remove entrained organic before being pumped to the iron residue repulp tank in the iron removal circuit. Partially loaded organic will flow to the third extract stage of the zinc primary extraction circuit.

Iron Stripping and Organic Washing

A bleed stream of stripped organic from the second stripping stage will be contacted with a 6M solution of hydrochloric acid in a separate mixer-settler which results in transfer of most of the iron into the aqueous phase. The organic will then be washed with water to remove trace hydrochloric acid before returning the washed organic to the recirculating organic stream. The ferric loaded aqueous solution will be pumped to the hydrochloric acid recovery circuit.

Hydrochloric Acid Recovery

In the hydrochloric acid recovery circuit the aqueous solution from the iron stripping circuit will be contacted with concentrated sulphuric acid. The resulting off-gas will be scrubbed to regenerate the 6M hydrochloric acid solution which will be recycled to the iron stripping mixer-settler. The strip liquor, containing the stripped iron and other extracted metals, will be pumped to the secondary leaching circuit.

Crud Removal and Clay Treatment

Crud handling facilities similar to those for the copper solvent extraction circuit will be used to manage and treat the crud formed in the zinc solvent extraction circuit. Each mixer-settler in the primary and secondary zinc solvent extraction circuits will be provided with a dedicated air-operated crud extraction pump attached to a suction wand which will be manipulated by the field operator.

Acid activated bentonite clay will be mixed with the organic to adsorb degradation products and the crud slurry will be fed to the tricanter centrifuge. The solids will be discharged via a chute into a container, removed with a forklift, and discarded onto the tailings storage facility, while the organic will be returned to the zinc solvent extraction circuit.

17.4.10 Zinc Electrowinning (EW)

The function of the electrowinning circuit is to effect zinc metal deposition from a zinc-bearing solution onto permanent aluminium cathodes by the passage of direct electric current. The overall cell reaction may be represented as follows:

- $ZnSO_4 + H_2O \rightarrow Zn \downarrow + H_2SO_4 + ½O_2 \uparrow$

Where zinc is deposited at the cathode and oxygen is liberated at the anode.

All of the equipment associated with the zinc electrowinning circuit as part of the PMR Project will be new.

The high purity of loaded electrolyte will result in the production of super high grade (SHG) zinc cathodes.

Electrolyte Filtration

Strong electrolyte solution will be pumped through a bank of three dual media filters to remove entrained organic.

The filters will be individually back-washed with electrolyte. Filter backwash liquor will report to a backwash filtrate tank from where the aqueous will be returned to the strip section of the zinc solvent extraction circuit. The organic will be recovered and returned separately to the zinc solvent extraction crud treatment circuit.

Electrowinning

The organic-free rich electrolyte (90 g/L zinc) will report to the rich electrolyte tank from where it will be delivered to a bank of “scavenger” electrowinning cells to remove any residual organic. Electrolyte overflow from the cells will pass to the electrolyte circulation tank.

The electrolyte circulation tank will be equipped with electrolyte feed pumps which will deliver electrolyte to the banks of “commercial” electrowinning cells. Overflow from the commercial cells will return to the circulation tank. The circulation tank will be divided into separate ‘high’ and ‘low’ concentration sides by an internal baffle. The electrolyte from the electrowinning cells will flow back to the ‘high’ concentration side of the circulation tank while the ‘low’ concentration side will receive overflow solution from the commercial electrowinning cells. The spent electrolyte concentration will be 45 g/L zinc resulting in an overall zinc ‘bite’ across the tank house of 45 g/L.

A small flow of sulphuric acid will be added to the electrolyte circulation tank to maintain the required acidity and to replace the acid lost to the electrolyte bleed stream. The electrowinning circuit will deploy permanent cathode technology, using aluminium cathode blanks and silver alloyed lead anodes. Electrolyte will be introduced to each of the cells through an internal sparge pipe beneath the electrodes and will overflow a weir into the return electrolyte header.

Electric current will be delivered to each of the electrowinning cells from rectiformers via a busbar system. A trickle current (emergency) rectiformer will be provided to maintain delivery of sufficient electrical potential to ensure that zinc does not become delaminated from the cathode blanks if the main rectiformers are out of service.

A solution of strontium carbonate will be made-up in a mixing facility and dosed into the recirculating electrolyte to maintain a low lead content in the zinc cathode. Manganese sulphate solution will also be made-up in a mixing facility and dosed into the recirculating electrolyte to help passivate and protect the anode surface, while gelatine and liquorice will be added to respectively enhance zinc cathode appearance and improve the tank house atmosphere.

Acid Mist Removal

Evolution of oxygen at the cathode during zinc electrowinning results in the formation of small oxygen bubbles which, when they burst, leads to the generation of an acidic aerosol. Thus, the electrowinning cells will each be provided with a hood which will be ducted to the zinc electrolyte cooling towers. Here the acidic vapour will undergo scrubbing before venting the cleaned off-gas to atmosphere.

Cathode Harvesting and Stripping

The zinc electrowinning circuit will be designed in two rows of electrowinning cells, each with its own dedicated overhead crane. Loaded cathodes will be removed from the cells using the overhead crane and bale configuration and placed onto the inbound conveyor of a cathode stripping machine. The cathodes will then be advanced to the washing chamber where surface contaminants and electrolyte will be washed off the surface of the cathode and hanger bars using recirculating hot water. An exhaust system will draw vapour from the washing chamber using a centrifugal fan which will be ducted to the acid mist scrubbers.

The clean cathodes with then be passed to a stripping station via a pivot arm where the zinc metal is removed from the cathode blanks and stacked into a bundle of pre-determined weight

A zinc sheet will be removed from the top of the cathode bundle and transferred to the sampling conveyor of the sampling device. Samples will then be punched from several locations in the zinc sheet and discharged into a container, before returning the sheet back to the bundle on the stack discharge conveyor. The cathode bundle will then be weighed, labelled, strapped using high tensile steel straps, and removed using a forklift. The entire washing, stripping, sampling, and weighing process will be automated while bundle strapping and removal will be a manual operation.

The stripped cathode blanks will be picked up by the pivot arm and placed on the indexing outbound conveyor which will continuously index cathode blanks until it is full. The batch of stripped cathode blanks will then be transferred back to the zinc electrowinning cells by the overhead crane. An additional manually operated pivot arm will be provided which will be used for shifting rejected cathode plates to a reject rack. These will then be transported to dissolving cells where zinc stuck onto the surface of the cathode will be dissolved using spent electrolyte.

Anode Cleaning and Straightening

During the zinc electrowinning process, it will be required to periodically remove manganese dioxide from the surface of the anodes to prevent short-circuits with the adjacent cathodes. The anodes will be removed from the cells approximately every 40 days, cleaned with high pressure water jets, and straightened in a pneumatically operated washing and flattening machine before being returned to the cells.

Cell Cleaning

It will be required to clean the electrowinning cells several times each year to avoid short circuits caused by excessive accumulation of solids (the cleaning interval will be determined by the electrowinning current density and the $CaSO_4$ content in the electrolyte). The solids will be removed using a vacuum cleaning system which will discharge slurry into an agitated tank. The collected slurry will be returned to the primary leaching circuit.

17.4.11 Zinc Melting and Casting

The zinc melting and casting operation will receive bundles of zinc cathodes from the electrowinning circuits which will then be melted in an induction furnace and cast into ingots before despatch to market.

Zinc Melting

Zinc cathodes from the electrowinning circuit will be stacked in bundles after stripping and then transported by forklift to the zinc cathode bundle conveyor. The conveyor will then deliver the cathode bundles to the zinc cathode bundle elevator which will lift the bundles and then convey them to a tilting table. The tilting table will align the cathode sheets vertically before being gradually fed, using a hydraulic grab mechanism, to the zinc melting induction furnace via an opening at the top of the furnace. This entire feeding operation will be fully automated.

The zinc melting furnace will be designed for a throughput rate of 16 t/h of zinc cathodes. The cathodes will be melted under a layer of ammonium chloride to prevent the formation of an oxide film that may be formed during the melting operation. Ammonium chloride will be manually added from bags to the ammonium chloride hopper mounted above the furnace.

The furnace inspection and dross extraction doors, as well as the cathode feeding chutes, will be ducted to a bag house filter to ensure that the furnace operates under slight negative pressure and thus minimises dust emissions.

Zinc Casting

The zinc casting system will comprise two identical lines, each with 108 zinc ingot moulds. A pair of graphite zinc melting pumps will withdraw the molten zinc from the furnace and will pump to separate steel casting launder lined with heat resistant material. The molten zinc will flow from each casting launder to a zinc casting tundish which will then pour the molten zinc into the moulds on each line. The zinc slab casting machines will be provided with automatic zinc pouring and skimming devices to ensure that each mould is filled with the required amount of molten zinc (~25 kg).

When the moulds are full, they are conveyed to the slab bundle stacker located at the end of each casting line, while being cooled from the bottom to the top. As the moulds reach the end of the casting line, the zinc ingots are ejected from the moulds and automatically arranged to form a bundle. Each zinc ingot bundle is then conveyed to a common zinc ingot cooling unit where they are further cooled before being sampled, weighed, strapped, and labelled. The zinc ingot bundles are then loaded by forklift into shipping containers in preparation for despatch off-site.

Dross Treatment

Due to oxidation and other chemical reactions taking place during the melting process, approximately 1% of the cathode metal will be converted to a dross product composed mainly of zinc oxide (ZnO). This dross product is a dense powdery material that accumulates on the surface of the furnace bath during melting. Once removed from the bath surface the dross begins to cool which causes any molten zinc carryover to freeze into small beads. Separation of these metallic zinc particles from the dross oxide will be carried out in a dross treatment plant.

The furnace dross will be cooled and treated in an air-swept rotary grinding mill. The milling circuit will produce a metallic zinc stream for re-melting while the dross oxide will be removed as a dust, captured in containers and returned to the iron removal circuit.

17.4.12 Secondary Leaching

Unlike the primary leach, which is a silver catalysed sulphate acid leach, the secondary leaching process will use an acidic chloride lixiviant to solubilise the lead and silver species precipitated during primary leaching. The leaching process will be performed at atmospheric pressure and at ~80°C. The secondary leach discharge will be subjected to thickening and filtration, with the clarified liquor reporting to the silver and lead recovery circuits and the washed barren solids filter cake (secondary leach residue) reporting to tailings disposal. The principal reactions in the secondary leaching circuit are:

- $PbSO_4 + 4NaCl \rightarrow Na_2PbCl_4 + Na_2SO_4$
- $Ag_2SO_4 + 8NaCl \rightarrow 2Na_3AgCl_4 + Na_2SO_4$

Secondary Leach Reactors

Filter cake from the primary leach filters will be repulped with brine solution and pumped to the mechanically agitated secondary leach feed tank. This tank will also receive conditioned brine, sulphuric acid, and strip liquor from the zinc solvent extraction iron stripping circuit and will provide surge capacity ahead of the secondary leach reactors.

The slurry will then be delivered to the first of four mechanically agitated secondary leach reactors, arranged in a series cascade overflow configuration, providing a total two hour residence time. Supplemental sulphuric acid addition facilities to each of the reactors will be provided. Submerged heating coils in the feed tank and in each of the reactors will enable the slurry temperature to be maintained at ~80°C. Vapour from the reactors will be ducted to the brine cooling tower.

Secondary Leach Thickening and Polishing Filtration

Discharge slurry from the final secondary leach reactor will be pumped to a 20 m diameter secondary leach thickener. The thickener will be provided with a heating jacket to ensure that the temperature is maintained above 80°C, thereby preventing lead precipitation. The thickener underflow slurry will be pumped to the secondary leach filtration circuit while the thickener overflow will be pumped to a pair of polishing filters. Recovered fine solids will be returned to the secondary leach thickener, while the polishing filter filtrate will report to the silver recovery circuit.

Secondary Leach Filtration

Underflow slurry from the secondary leach thickener will be pumped to the secondary leach filtration circuit to separate the secondary leach residue from the silver- and lead-rich PLS. The filtration circuit will comprise three 100 m^2 horizontal vacuum belt filters. The filter cake will be washed with hot water to maximise recovery of PLS while preventing lead precipitation. Both the filtrate and washate streams will report to a common filtrate tank for subsequent return to the secondary leach thickener.

The washed filter cake (secondary leach residue, SLR) from each of the secondary leach filters will be conveyed to a common secondary leach residue bunker. A FEL will then load the SLR into trucks for subsequent disposal

Brine Preparation and Conditioning

The brine preparation and conditioning circuit will receive barren liquor from the downstream lead recovery process and combine this with clarified overflow liquor from the aluminium removal circuit in a mechanically agitated brine conditioning tank. The barren liquor will first be cooled in two cooling towers operating in series to evaporate excess water from the recycle brine liquor. A bleed stream from the brine conditioning tank will be delivered to a brine saturator to which salt (sodium chloride) crystals will be added from bulk bags. The resulting concentrated brine solution (~265 g/L NaCl) will overflow back to the brine conditioning tank.

A portion of the brine solution will be diverted to the mechanically agitated brine transfer tank for subsequent overland delivery to the primary leach residue repulp tank located at the existing plant site. The remainder of the brine solution will flow through heat exchangers to pre-heat the brine prior to secondary leaching.

17.4.13 Silver Recovery

The silver recovery circuit will be designed to recover silver from the pregnant secondary leach liquor by cementation. The silver will be cemented onto the surface of lead plates according to the following general reaction:

- $2Ag^{+}_{(aq)} + Pb^{\circ} \rightarrow 2Ag^{\circ} + Pb^{2+}_{(aq)}$

The resulting silver cement will be continuously removed from the lead surface by agitation and will fall to the bottom of the reactor for subsequent filtration. The majority of the filtered silver solids will be washed and packaged into bulk bags for subsequent return to the primary leaching circuit to catalyse the leaching process. The remainder of the silver cement will be diverted to a purification circuit for impurity removal and packaging as a filter cake into drums for despatch.

Silver Cementation

Pregnant leach solution from the secondary leach circuit polishing filters will report to the silver cementation feed tank. From here the solution will be pumped through silver cementation feed heat exchangers, to a series of 8 mechanically agitated conical bottomed silver cementation reactors operating in parallel. Each reactor will have a nominal 45 minutes residence time. The reactors will operate at atmospheric pressure and ~80°C and will each contain a series of lead plates. The depleted lead plates will be removed and replaced with new plates on a demand basis using a dedicated overhead crane. Off-gas will be ducted to the brine cooling tower.

Cementation reactor overflow liquor will be fed to a silver pinned bed clarifier for removal of particulates from the thickener overflow stream. The settled solids will be continuously removed from the bottom outlet cone of the clarifier and pumped as a dilute slurry to the downstream filter. The clarified solution will be adjusted to pH 2.2-2.4 using hydrated lime before being pumped to the lead cementation circuit. Both the clarifier feed and overflow tanks will be equipped with steam coils to maintain an elevated liquor temperature and prevent lead precipitation.

Filtration

Settled solids from the cone of each silver cementation reactor will be discharged continuously using peristaltic pumps to the mechanically agitated silver cement reactor underflow tank. Silver cement filter feed pumps will deliver a slurry of silver cement to a pair of vertical plate filter presses for subsequent solid-liquid separation and washing. The washed silver cement will discharge from each filter press into dedicated screw feeders, in turn discharging into the bi-directional silver cement transfer feeder. When operating in forward mode, the feeder will discharge silver cement into bulk bags at a bulk bag filling station. The silver cement bags will then be transferred by vehicle to the existing plant for subsequent addition to the primary leaching circuit.

Silver Purification

When the silver cement transfer feeder is selected to operate in a reverse direction, the feeder will discharge the silver cement into a series of trays which will then be manually loaded into the mercury retort oven in the adjacent silver purification circuit. The trays of silver cement will then be heated under vacuum in the retort oven for ~6 hours at 550°C to vaporise the mercury. During retorting, the mercury vapour will be condensed and the resulting liquid collected under water in a collection tank. Periodically the mercury will be tapped from the collection tank into small mercury flasks, sealed, and stored in a dedicated storage facility offsite.

After retorting, the trays of silver cement will be cooled, removed from the oven, and then the silver cement manually loaded into a pair of mechanically agitated and jacketed silver purification reactors. The crude silver cement will be leached in a 10%w/w solution of sulphuric acid at 50°C by applying saturated steam to the reactor jacket.

Following leaching the reactor contents will be filtered on a mobile vacuum filter, washed with RO water, and packaged into 10 litre drums within a secure area. Each drum will be sampled, weighed, and labelled before being despatched off site. The purified silver product will have an expected silver purity of 99.9%w/w (dry weight basis) and a moisture content of ~10%w/w.

17.4.14 Lead Recovery

The lead recovery circuit will be designed to recover lead from the silver-depleted secondary leach liquor by cementation. The lead will be cemented onto the surface of rotating aluminium discs according to the following general reaction:

- $3Pb^{2+}_{(aq)} + 2Al^{\circ} \rightarrow 3Pb^{\circ} + 2Al^{3+}_{(aq)}$

The resulting lead cement will be continuously removed from the surface of the aluminium discs and will fall to the bottom of each reactor. A submerged conveyor will transfer the recovered lead cement onto a surface conveyor for subsequent filtration and washing. The filtered and washed lead solids will be transferred to the to the compacting unit to produce pure lead briquettes.

Lead Cementation

The pH adjusted slurry from the silver cementation circuit will be pumped to a 20 m diameter high rate lead PLS thickener where it will be thickened to remove gypsum and iron hydroxides. The thickened underflow slurry will be pumped to the secondary leach thickener underflow tank where it will combine with the underflow slurry from the secondary leach thickener for subsequent filtration on the secondary leach filters.

The lead PLS thickener overflow will report to the mechanically agitated lead cementation feed tank equipped with a steam heating coil to maintain the solution temperature at ~80°C. From here the solution will be pumped to a bank of twelve lead cementators, each having a nominal one hour residence time and each will be equipped with a series of rotating aluminium discs and a submerged conveyor. The lead will cement onto the surface of the rotating discs, causing the discs to gradually dissolve, thereby increasing the aluminium tenor of the lead barren liquor.

The lead cement will fall to the bottom of the reactors which will then be recovered by the individual conveyors and discharged onto a common horizontal lead cement filter feed conveyor. The depleted aluminium discs will be removed and replaced with new discs on a demand basis using a dedicated overhead crane.

Off-gas (predominantly water vapour and hydrogen) will be captured and ducted to the brine cooling tower. Overflow liquor from the cementators will report to the lead barren liquor tank which will then be pumped, to the brine conditioning circuit. A barren liquor bleed will also be diverted to the aluminium (and sulphate) removal circuit to prevent accumulation of aluminium and other base metal impurities within the recirculating brine lixiviant.

Filtration and Storage

The lead cement filter feed conveyor will deliver the lead cement to a horizontal vacuum belt filter for subsequent washing and filtration. The filtered solids will be washed counter-currently with hot RO water to remove residual chlorides and the resulting filtrate and washate solutions will be pumped to the lead barren liquor tank.

17.4.15 Lead Briquetting

The washed lead cement will discharge from the filter into a lead briquetting machine where the cement lead will be compressed into lead briquettes. These will then be drummed for dispatch.

Aluminium and Sulphate Removal

The purpose of the aluminium and sulphate removal circuit will be to remove aluminium and other trace base metals as well as excess sodium sulphate from the recirculating brine lixiviant. This will be achieved in a series of reactors using hydrated lime addition according to the following main reactions:

- $2AlCl_3 + Ca(OH)_2 \rightarrow 2Al(OH)_3 + 3CaCl_2$
- $Na_2SO_4 + CaCl_2 \rightarrow CaSO_4 + 2NaCl$

The reactor discharge slurry will be thickened and filtered. The solids will be sent for disposal as dry stacked tailings, while the filtrate will be recycled to the brine conditioning circuit.

Aluminium Removal Reactors

Barren liquor bleed from the lead removal circuit will be delivered to the first of three mechanically agitated aluminium removal reactors, arranged in a series cascade overflow configuration, each with a nominal 90 minute residence time.

Hydrated lime will be added to the reactors to achieve a terminal target pH of 4.5. Submerged heating coils in each of the reactors will enable the slurry temperature to be maintained at ~70°C. Vapour from the reactors will be vented to atmosphere.

Aluminium Removal Thickening

Discharge slurry from the final aluminium removal reactor will be pumped to the feed tank of a 10 m diameter aluminium removal thickener. The thickener underflow slurry will be pumped to the aluminium removal filtration circuit to separate the solids residue from the brine lixiviant. A seed recycle pump will return a portion of the thickener underflow stream to the aluminium removal reactors to improve particle growth and reduce the formation of scale. The thickener overflow will be pumped via an overflow tank to the brine conditioning circuit.

Aluminium Removal Filtration

Aluminium removal thickener underflow slurry will be pumped to a mechanically agitated underflow tank and then delivered to a pair of vertical plate pressure filters. The filtered solids will be washed, squeezed and air-dried to maximise chloride removal, and then discharged into a residue bunker for subsequent back-loading by FEL into mine trucks for disposal on the tailings storage facility. The filtrate will gravitate back to the aluminium removal thickener feed tank.

17.4.16 Effluent Treatment

The purpose of the effluent treatment circuit will be to treat process liquors and generate a clean effluent stream suitable for discharge to the Guadalquivir River. The effluent treatment process will use lime to elevate the pH of the effluent and precipitate base metal impurities as insoluble metal salts, followed by thickening and filtration to yield a filter cake for dry stacking. Polishing filtration of the final effluent will remove trace solids prior to sampling and release of the treated effluent stream. The majority of equipment within the effluent treatment circuit is existing although there will be a requirement for two new effluent treatment reactors to be installed as part of the PMR Project.

Effluent Treatment Reactors

Iron residue from the iron removal circuit will be repulped with secondary zinc raffinate and pumped overland from the new plant site to the first of three existing mechanically agitated effluent treatment reactors located in the existing plant. An additional two reactors will be installed as part of the PMR Project to fulfil the residence time requirements for the increased flowrates. All five reactors will be arranged in a series cascade overflow configuration providing a combined three hour residence time. Milk of lime from the existing lime slaking and distribution circuit will be added to the tanks to achieve a terminal target pH of 9.0. The addition of sparge air to each of the reactors will oxidise the iron and assist with the precipitation of arsenic as a benign ferric arsenate complex.

Effluent Treatment Thickening

Discharge slurry from the final effluent treatment reactor will be pumped to the feed tank of the existing 20 m diameter effluent treatment thickener. The thickener underflow slurry will be pumped to the existing effluent treatment thickener underflow tank for subsequent filtration. A seed recycle pump will return a portion of the thickener underflow stream to the effluent treatment reactors to improve particle growth and reduce the formation of scale. The thickener overflow will gravitate to the existing effluent treatment thickener overflow tank ahead of polishing (sand) filtration.

Effluent Treatment Filtration

Thickener underflow slurry from the mechanically agitated effluent treatment thickener underflow tank will be pumped to a pair of effluent treatment pressure filters, one new and one existing. The filtered solids will be discharged into an effluent treatment residue bunker for subsequent back-loading by FEL into mine trucks for disposal on the tailings storage facility. The filtrate will gravitate back to the effluent treatment thickener.

Solution from the thickener overflow tank will be pumped through a series of existing sand filters to recover any fugitive fine solids from the thickener overflow stream. The filtrate from the sand filters will flow to the effluent treatment filtrate tank.

Existing filter backwash pumps will deliver filtered treated effluent to the sand filters on an intermittent basis and the backwash slurry will return recovered solids back to the effluent treatment thickener. The final treated effluent filtrate will be pumped to the existing effluent pond for sampling prior to release

17.4.17 Paste Backfill Plant

The purpose of the paste backfill plant will be to produce a paste product suitable for use as backfill in the underground mine workings for improved ground stability. The paste plant will be sized to generate ~1 Mtpa of backfill, from flotation tailings. Cement will be added to the solids to provide strength to the backfill product. The paste plant will be located approximately midway between the existing process plant site and the proposed new plant site. The new flotation tailings thickener and process water collection and distribution facilities will be located in the paste plant area to minimise the pumping distance of the thickened tailings to the paste plant filtration circuit.

Flotation tailings thickener underflow slurry will be pumped to a mechanically agitated paste filter feed tank. The slurry will be fed to a bank of three vacuum disc filters. The filtrate from the filters will gravitate to the adjacent paste filter filtrate tank for subsequent return to the process water tank.

The filter cake from each filter will discharge onto a single horizontal conveyor which will deliver the solids either to a twin shaft paste mixer, where they will be mixed with process water and cement, or to the final tailings conveyor. Cement will be delivered to site in road tankers and pneumatically off-loaded into a pair of storage silos adjacent to the paste mixing plant. The paste mixer will overflow to paste hoppers, each equipped with a high pressure positive displacement paste backfill pump to deliver the paste underground. Each pump will have a dedicated delivery line connected to air and water flushing facilities.

During normal operation, a significant portion of the filtered flotation tailings (~0.6 Mtpa) will not be required for backfill and this will be diverted to the final tailings conveyor. The conveyor will deliver the filter cake to a mechanically agitated tailings repulp tank where it will be repulped with process water and then pumped to the tailings storage facility.

Although the normal operation requires that ~60% of the filter cake be diverted to paste backfill, there will likely be frequent occasions when backfill cannot be utilised and thus the tailings repulp and pumping facilities will be designed to accommodate the entire flotation tailings filter cake tonnage.

17.5 Water Services

The following new and modified water services will be required to support the new PMR Processing facility;

- Process Water
- RO Water
- Potable Water
- Fire Water
- Cooling Water
- Gland Seal Water
- Contact Water
- Sewage Water

17.5.1 Reagents Services

The following new and modified reagent services will be required to support the new PMR Processing facility;

- Hydrated Lime
- Activator
- Depressant
- Frother
- Collector
- Flocculant
- Defoamer
- Sulphuric Acid
- Hydrochloric Acid
- Diluent

- Extractant
- Coagulant
- Anthracite
- Garnet
- Cobalt Sulphate
- Dextrin
- Bentonite
- Manganese Sulphate
- Strontium Carbonate
- Liquorice
- Gelatine
- Grinding Media
- Cement

17.5.2 Utilities

The following new and modified utilities will be required to support the new PMR Processing facility;

- Electrical Power
- Steam
- Plant and Instrument Air
- Low Pressure Air
- Oxygen Plant
- Emergency Power
- Spill Management
- Spill Management – Process Slurries
- Spill Management – Reagent Mixing and Storage

17.6 Mechanical Equipment

A summary mechanical equipment list has been developed for the proposed new and modified existing process plants which has formed the basis for compilation of the capital cost estimate. All equipment has been sized based on preliminary mass balance and design criteria data. A summary of the principal mechanical equipment items is presented in Table 17-2.

Table 17-2 Principal Mechanical Equipment

Description	Quantity	Size / Capacity
ROM Bin	1	250 m^3 capacity
Primary Crusher	1	Single toggle jaw crusher, 160 kW
Coarse Ore Stockpile	1	6,700 t live capacity, 24,000 t total capacity
SAG Mill	1	6.71 m dia x 5.50 m EGL, 4.7 MW
Ball Mill	1	6.10 m dia x 9.20 m EGL, 6.2 MW
Pebble Crusher	1	Hydrocone H4800, 250 kW (existing, repurposed)
Rougher Flotation Cells	4	TC-70 Tank Cell, 70 m^3
Scavenger Flotation Cells	4	TC-70 Tank Cell, 70 m^3
Flotation Tailings Thickener	1	30 m diameter
Concentrate Regrind Mill	1	3,500 kW
Concentrate Thickener	1	26 m diameter (existing, repurposed)
Concentrate Filters	3	Vacuum belt filters (existing, repurposed)
Concentrate Storage Shed	1	6,000 t total capacity
Primary Leach Reactors	8	350 m^3 live capacity, 6.9 m dia x 10.4 m S/W
Primary Leach Thickener	1	20 m diameter (existing, repurposed)
Primary Leach Filters	3	156 m^2, Larox Tower Filter (existing, repurposed)
Primary Leach Residue Storage	1	6,000 t total capacity
Copper pH Adjust Reactors	3	98 m^3 live capacity, 5 m dia x 6 m S/W
Copper pH Adjust Thickener	1	26 m diameter (existing, repurposed)
Copper PLS Clarifier	1	6 m diameter (existing, repurposed)
Copper SX Settlers	6	18.7 m W x 20.4 m L x 2.2 m H (existing)
Copper Electrolyte Filters	3	4.9 m dia x 7.4 m H (existing)
Copper Electrowinning Cells	144	1.33 m W x 9.18 m L x 1.77 m H (existing)
Iron Removal Reactors	4	373 m^3 live capacity, 7.8 m dia x 8.8 m H
Iron Removal Thickener	1	40 m diameter
Iron Removal Filters	3	120 m^2, 4.0 m W x 30 m L
Iron Removal Clarifier	1	6 m dia, 28.3 m^2
Zinc pH Adjust Reactors	3	2 x 28 m^3, 1 x 45 m^3 (existing, repurposed)
Zinc pH Adjust Thickener	1	10 m diameter (existing, repurposed)
Zinc SX Settlers	11	18.7 m W x 20.4 m L x 2.2 m H
Zinc Electrolyte Filters	3	4.9 m dia x 7.4 m H
Zinc Electrowinning Cells	34	1.3 m W x 12.0 m L x 1.8 m H
Zinc Melting Furnace	1	16 t/h, natural gas fired
Effluent Treatment Reactors	5	180 m^3, 5.8 m dia x 7.5 m H (3 existing, 2 new)
Effluent Treatment Thickener	1	20 m diameter (existing)
Effluent Treatment Filters	5	80 m^3/h (existing)
Secondary Leach Reactors	4	110 m^3 live capacity, 5.2 m dia x 6.2 m S/W
Secondary Leach Thickener	1	20 m diameter
Secondary Leach Filters	3	100 m^2, 4.0 m W x 25 m L
Silver Cementation Reactors	8	106 m^3 live capacity, 5.0 dia x 4.4 m S/W, 60° cone
Silver Cementation Clarifier	1	6 m dia, 28.3 m^2
Silver Cement Filters	2	19.6 m^2, vertical plate filter press
Lead PLS Thickener	1	20 m diameter
Lead Cementators	12	7.5 m L x 2.0 m W x 2.4 m H
Aluminium Removal Reactors	3	76 m^3 live capacity, 4.6 m dia x 5.2 m H
Aluminium Removal Thickener	1	10 m diameter
Aluminium Removal Filters	2	30 m^2, vertical plate filter press
Paste Plant Filters	3	156 m^2, rotary vacuum disc filters
Steam Boiler	1	20 MW water tube boiler, natural gas fired
Oxygen Plant	1	24.9 t/h, 350 kPag, (expanded by Others)

17.7 Electrical Design

The Project, including all mining and processing facilities, accommodation and infrastructure is estimated to have a maximum demand of 51.9 MW, with an average load of 47.5 MW. The largest loads will be the Zinc electrowinning with two rectifiers rated at 10.8 MW each. The largest motors will be the SAG mill and Ball mill with an installed motor power of 4.7 MW and 6.2 MW respectively. The mills will be provided with a VVVF drive which will minimise the impact on the power system during starting. A breakdown of power demand by load centre is provided in Table 17-3. Electrical power to meet the new demand below can be fully supplied from the local distribution network and may be supplemented by construction of an onsite solar plant.

Table 17-3 Power Demand by Load Centre

Substation No.	MCC No.	Description	Connected Load kW	Average Load kW	Average Load kW	Average Power MWh/year
311SUB9001	311MCC9001	Primary Crushing	833	692	575	5,042
320SUB9002	320MCC9001	Milling and Pebble Crushing	2,538	1242	1,096	9,520
320SUB9002	-	SAG Mill	4,700	3,690	3,509	30,708
320SUB9002	-	Ball Mill	6,200	5,122	4,871	42,625
320SUB9002	-	Regrind Mill	2,750	2,090	1,987	17,393
330SUB9001	325MCC9001	Flotation	3,141	1,818	1,708	14,886
330SUB9001	330MCC9001	Secondary Leaching	2,969	2,105	2,013	17,511
360SUB9001	360MCC9001	Silver Recovery	4,044	2,060	1,450	11,961
390SUB9001	390MCC9001	Paste Backfill Plant	2,819	1,647	1,381	12,071
350SUB9001	351MCC9001	Zinc Tank Farm	2,179	1,361	1,137	9,883
350SUB9001	-	Zinc Electrowinning	22,400	17,920	17,022	149,130
350SUB9001	361MCC9001	Zinc Melting and Casting	2,000	1,600	1,520	13,315
340SUB9001	340MCC9001	Zinc Solvent Extraction	1,244	944	884	7,738
355SUB9001	355MCC9001	Iron Removal	2,909	1,901	1,801	15,699
		Railveyor™	6,150	2,256	699	6,127
		Surface	1,036	1,014	1,083	8,878
		Underground	4,693	4,477	4,768	39,219
		New Plant Load	72,605	51,937	47,503	411,707,292
		Existing Load	37,645	27,640	10,813	90,699,571
		Total Load	**110,250**	**79,578**	**58,316**	**502,406,863**

ITEM 18 PROJECT INFRASTRUCTURE

CLC is an existing operation that has been open pit mining and processing of copper ore for more than fifteen years. The associated infrastructure as required by these operations is still in place and includes sealed roads, power lines and substations, a process plant, site offices, workshops, tailings dams, and waste storage facilities.

18.1 Roads, Rail, Airports and Ports

Road access to the operation is via the N-630 National Road, which passes 2km east of the mine and enables access to the site via existing SE-3410 sealed public road.

By rail, Seville is connected with the rest of the country by the Spanish national railways network. High speed trains have a regularly scheduled service from Seville to Madrid.

Port facilities are available at the Seville port on the Guadalquivir River, located 23 km from CLC and at Huelva Port, at a distance of 85km to the southwest of CLC.

Air transport is available at the Seville international airport, with connections to Madrid, Barcelona, Malaga and other European airports.

18.2 Dumps and Stockpiles

Waste rock storage dumps from the open pit operation include the South and West Dumps which were used for storing inert waste (marls & sandstone) and the North dump, which includes both plant tailings and rock waste. The North dump is lined to prevent and control acid drainage.

Marls are currently being returned to the mined-out pit from the South and West dumps to prepare the in-pit tailings cells. Stockpiles were maintained on the property to provide storage capacity between the mine and plant and for blending to a more consistent feed grade to the plant. A similar strategy will be adopted for the PMR underground operation.

18.3 Tailings Facility

The PMR process plant will generate two tailings streams:

- 1.4Mtpa flotation tailings, with 63% used as UG mine paste backfill and 37% disposed of into the in-pit tailings facility (IPTF).
- 0.7Mtpa of leaching tails disposed of into the IPTF.

The current IPTF strategy will be utilized to dispose, store and encapsulate plant tailings. The IPTF has key advantages over the previous surface dry stacking facility, including;

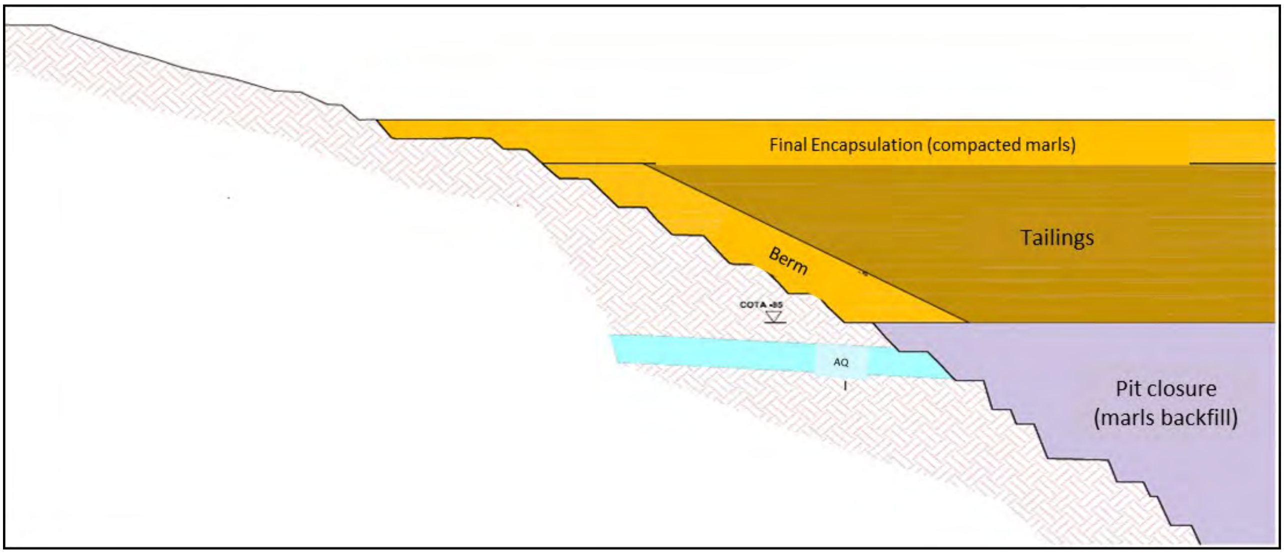
- It is compatible with the pit closure plan and the new facility will be developed over the open pit backfill at the -85 level.
- Reduces the disturbed surface footprint and issues associated with access to land, environmental and permitting versus the surface option.
- IPTF is the lowest CAPEX tailings disposal alternative for this Project.

The IPTF conceptual design is summarized in Table 18-1 and is shown in Figure 18-1 as a schematic view of the proposed facility.

Table 18-1 IPTF Design

Layer	Description	Thickness
Pit backfill (closure)	Compacted Marls (75% & 90% PM)	>75 m
Liner	HDPE	1.5 mm
Drainage layer	Crushed rock	0.5 m
Perimeter Berm	Compacted Marls (90% PM)	30m (top width)

Figure 18-1 IPTF Conceptual Design (Subterra, 2018)



Prior to the commencement of the operations at the IPTF, the pit closure process needs to be completed. A total of 3.4 Mm^3 of compacted marl will be disposed of once the upper section of the new East ramp intersects the Investigation Ramp and the existing in-pit portal can be closed in Year 2 of the LOM (refer to Chapter 4). The liner will be placed then over the base of the facility at -85 mRL in the pit. Once the drainage layer is in place, the facility will be ready for tailings placement. A compacted marl perimeter berm will be developed in 10m lifts around the pit wall as the tailings disposal progresses upwards.

The IPTF will be ready in advance of the plant start-up in Year 3. Different disposal methods have been assessed, concluding that dewatered tailings options are preferable as a means to increase the storage capacity and to simplify water management (Golder, 2017; Subterra, 2018). The solids content of the tailings to be placed in the pit will be within the range of 65% to 85%. Tailing disposal will be done from alternative discharge points to allow a uniform distribution. Supernatant water will be collected and incorporated into the water management circuit.

The total quantity of tailings for in-pit placement in the LOM plan is 33.3Mt dry, which is 94% of the initially approved capacity. The IPTF capacity can be expanded in the future by following a formal permitting procedure with the regional administration. The maximum theoretical capacity of the in-pit storage facility is more than 100Mt (dry), which is approximately 300% of the total tailings generated over the planned LOM.

A first cell has been prepared and operated from 2016 to 2023 to encapsulate waste rock and the tailings from the reprocessing project. This cell acts as a real scale trial of the future facility. Continuous geotechnical and hydrogeological monitoring during these years proves that the disposed material is properly encapsulated and validates the in-pit disposal plan selected for the PMR Project.

Figure 18-2 Inpit Tailings Disposal - August 2023

18.4 Existing Site Infrastructure

18.4.1 Existing Process Plants

The existing processing plant, designed to recover copper from secondary mineralization mined from the open pit will be fully re-used (with modifications) as part of the new PMR plant to exploit the primary mineralising extracted via underground means. This includes the copper and zinc leaching, copper recovery and effluent treatment inclusive of iron removal circuits.

An existing pilot plant covering a total area of 2000m^2 and encompassing a 450m^2 roofed structure that includes equipment and processes that reproduce, on a smaller scale, the PMR Process plant. This experimental plant can carry out tests at an industrial level; it includes primary and secondary grinding processes, flotation, primary and chloride leaching, filtration, cementation and auxiliary steam and oxygen supply systems.

Item 17 – (Recovery Methods) has further details of the existing processing plant and Item 13 (Mineral Processing and Metallurgical Testing) on the pilot plant.

18.4.2 Auxiliary Infrastructure

Significant auxiliary infrastructure has been developed on and around site to support the open pit operations between 2009 and 2021. This infrastructure remains in good working condition and is available to support the new PMR plant and underground operations with minimal alterations and additions. This existing infrastructure includes, but is not limited to:

- electrical sub-station
- emergency generators
- maintenance workshops
- spare parts warehouse
- truck weigh scale
- experimental pilot plant
- hazardous waste warehouse
- oxygen plant
- LNG plants
- mining workshop
- access control
- visitors' centre and CLC Mining Museum
- cafeteria
- laboratory
- medical centre
- changing rooms
- administrative buildings
- control and communication systems
- Internal roads & parking
- Water boilers
- Fire protection systems

Further details on mining related infrastructure are in Item 16 (mining Methods).

Figure 18-3 Existing Plant & Administration Infrastructure

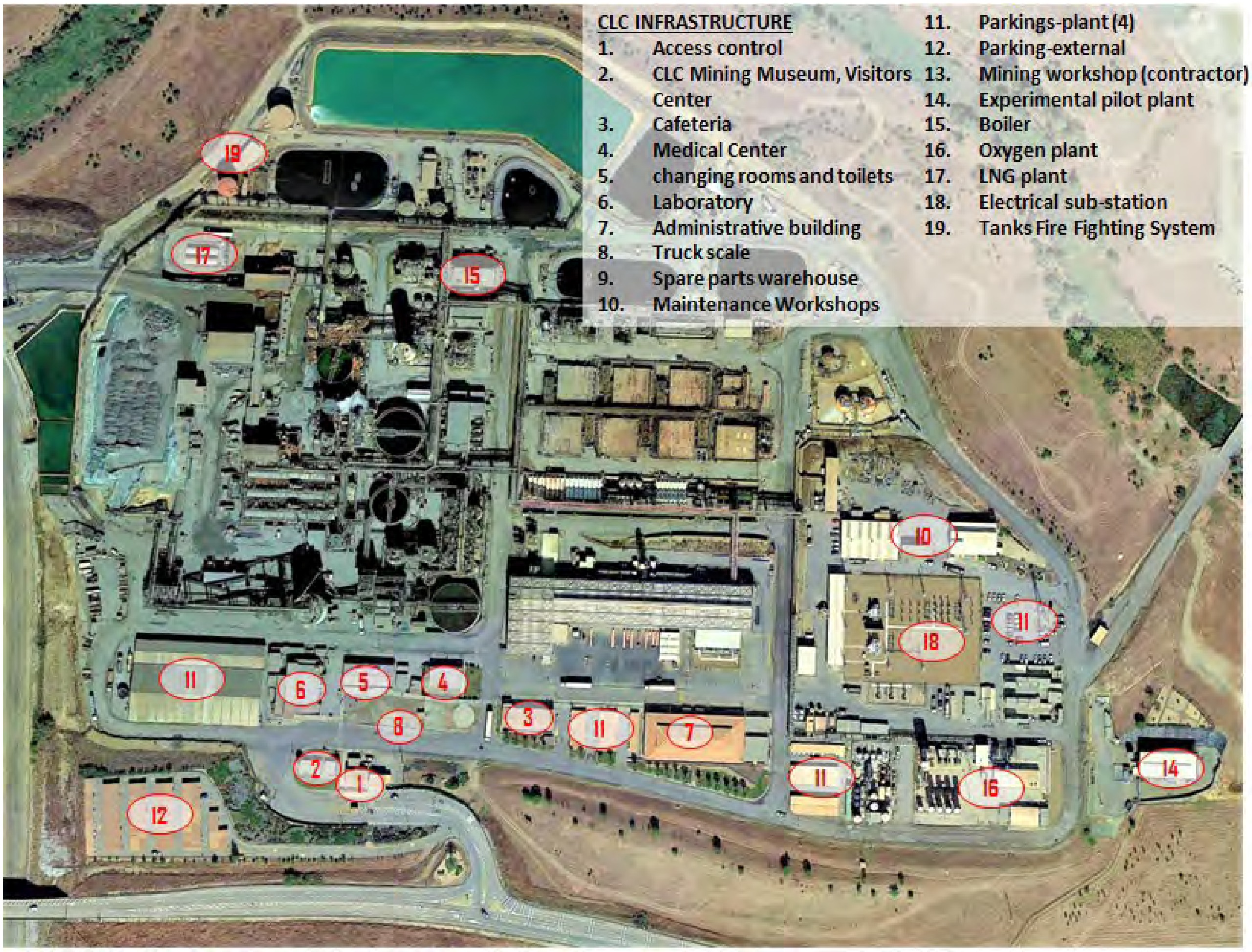


Figure 18-4 Existing Infrastructure - Other Areas

18.5 Electrical Supply Distribution

18.5.1 Existing 220 kV Substation and Electrical Rooms

Electrical power is supplied via a 4.2 km long dedicated 220 kV branch line that feeds a 220 kV outdoor substation with two transformers 220/20kV-50 MVA and a 20 kV main substation, adjacent and located to the east of the hydrometallurgical facility. Power is distributed internally through 20 kV power lines to the electrical rooms in plant, mine and water areas.

The distribution of energy to the various plant areas is carried out through six electrical rooms and transformation centres located in the Hydrometallurgy Plant, Mine and Water Plant as shown in Figure 18-6.

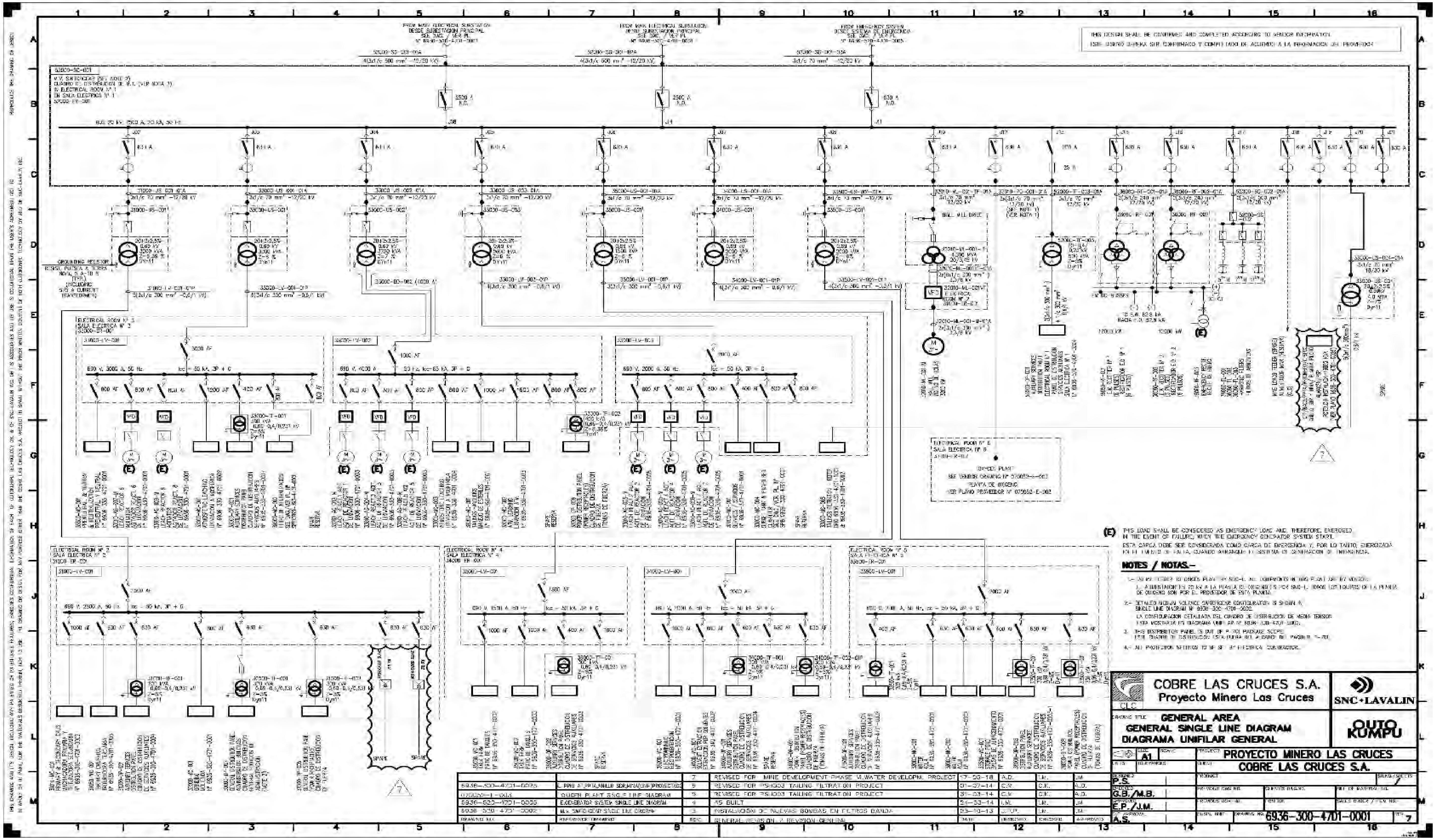
In addition, there are four generator sets for backup in case of a failure in the installation (for critical loads). A $10 m^3$ capacity fuel tank located nearby feeds these generators. All the low-tension motors are 690 V high efficiency units.

Figure 18-5 Electrical Substation Exterior View

Figure 18-6 Electrical Substation Interior Room

.

Figure 18-7 Existing Electrical Distribution Infrastructure in the Plant Area



New Electrical Infrastructure

18.5.2 Surface and Processing Plant

Power for the treatment plant will be sourced from twin circuit 20kV overhead power lines originating from the existing Electrical Room No.1. Each circuit will feed a primary substation within the treatment plant. From these two substations, power will be distributed at 20kV to substations throughout the plant.

Transportable packaged substations will be used to distribute loads within the treatment plant. The substations shall include:

- Transformers up to 3MVA
- Emergency Generators
- LV Motor Control Centres (MCCs):
- Variable Speed Drives & Starters:
- Emerson Controllers
- Uninterruptable Power Supply
- Fire Detection + Fire Suppression
- Power factor Correction
- New transformers in dedicated transformer compounds with neutral earthing resistors.

18.5.3 Underground Mining

The underground mining electrical supply consists of two distribution systems, i.e. for the Railveyor and the mining supply.

The Railveyor has its own distribution system. The surface facility for the Railveyor is composed of a 20kV overhead line that will follow the train rail to feed three surface substations. The third substation is close to the underground portal and voltage is planned to be lowered to 6.6KV for the underground substations.

The existing 20kV overhead line for the mine ring will be used to feed the paste plant substation. The paste plant substation will house 3MW emergency backup power for the underground supply. The underground mining supply to the paste plant will be via a borehole. Electrical power will be supplied to the mine through a 20kV/6.6kV substation located in the paste plant area (currently used for the open pit contractor facilities).

18.5.4 Load Flow Model

The Project, including all mining and processing facilities is estimated to have a maximum demand of 51.9MW with an average load of 47.5 MW (Table 18-2).

Table 18-2 Load Flow Summary

Substation No.	MCC No.	Description	Connected Load kW	Maximum Demand kW	Average Load kW	Av Power MWh/year
311SUB9001	311MC9001	PRIMARY CRUSHING	833	692	575	5042
320SUB9001	320MCC9001	MILLING+ CLASSIFICATION	2538	1242	1096	9520
320SUB9001	-	SAG MILL	4700	3690	3509	30708
320SUB9001	-	BALL MILL	6200	5122	4871	42625
320SUB9001	-	REGRIND MILL	2750	2090	1987	17393
330SUB9001	325MCC9001	FLOTATION	3141	1818	1708	14886
330SUB9001	330MCC9001	PRIMARY + SECONDARY LEACHING	2969	2105	2013	17511
360SUB9001	360MCC9001	SILVER RECOVERY	4044	2060	1450	11961
390SUB9001	390MCC9001	PASTE BACKFILL PLANT	2819	1647	1381	12071
350SUB9001	351MCC9001	ZINC ELECTROWINNING + TANK FARMS	2179	1361	1137	9883
350SUB9001	-	ZINC ELECTROWINNING	22400	17920	17022	149130
350SUB9001	361MCC9001	FURNANCE	2000	1600	1520	13315
340SUB9001	340MCC9001	COPPER SOLVENT EXTRACTION	1244	944	884	7738
340SUB9001	355MCC9001	IRON REMOVAL	2909	1901	1801	15699
		RAILVEYOR	6150	2256	699	6127
		SURFACE	1036	1014	1083	8878
		UNDERGROUND	4693	4477	4768	39219
		New Plant Load	**72,605**	**51,937**	**47,503**	**411,707**
		Existing Load	**37,645**	**27,640**	**10,813**	**90,699**
		Total Load	**110,250**	**79,578**	**58,316**	**502,406**

18.5.5 Solar Power Supply

Tenders have been sought for a photovoltaic plant that will be built specifically for the PMR plant by a third party. Power generated by the solar plant will be supplied directly to the CLC 20kV internal network with the remaining power demand supplied by the existing, dedicated 220kV line from the national grid (Figure 18-8).

The CLC site can provide 120ha of area for solar cell installation, sufficient to generate up to 166GWh per year or up to 30% of total site demand (Figure 18-9). As of 2023, 51% of the Spanish national power supply comes from green and renewable energy sources which means that approximately 64% of CLC’s power supply will become renewable.

Electrical distribution infrastructure will be designed to permit wide flexibility, allowing to consume the whole photovoltaic generation for the PMR or to get 100% of the CLC power demand from the national grid in the event of lack of solar power (Figure 18-8).

Figure 18-8 Electrical scheme with Solar Power supply

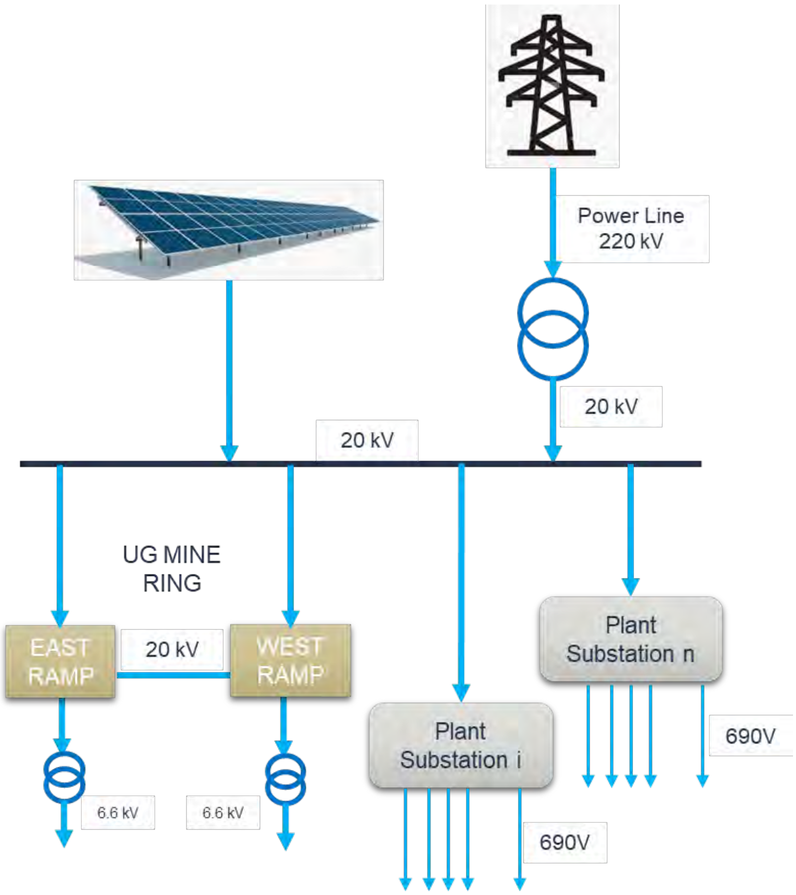


Figure 18-9 Surface available for the Solar Power plant (green)

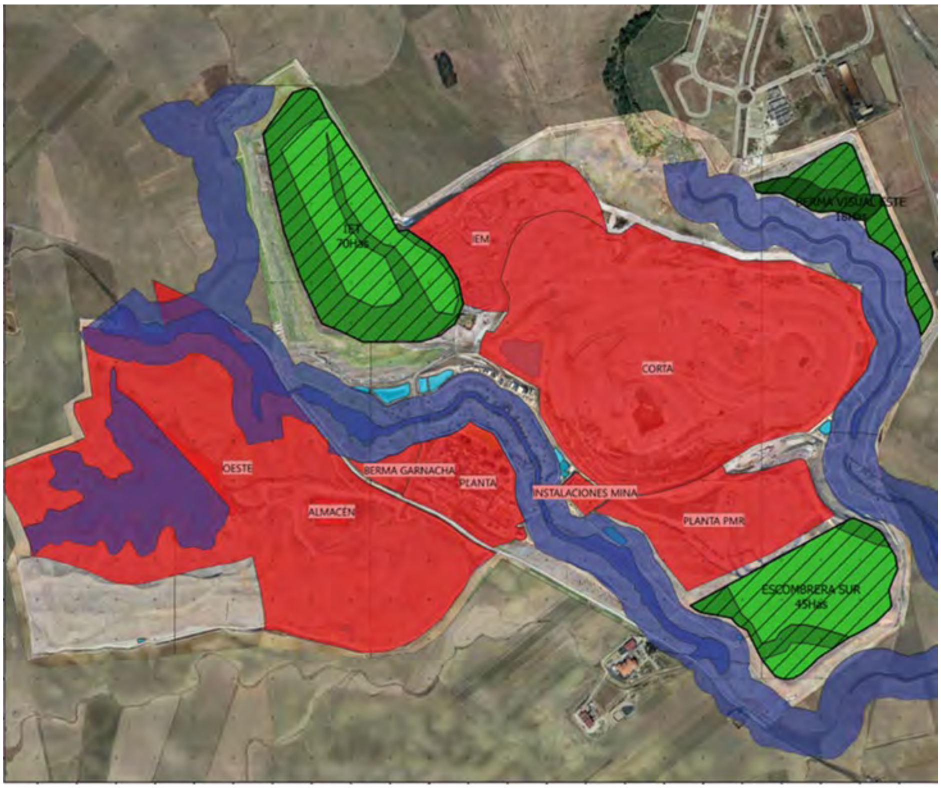
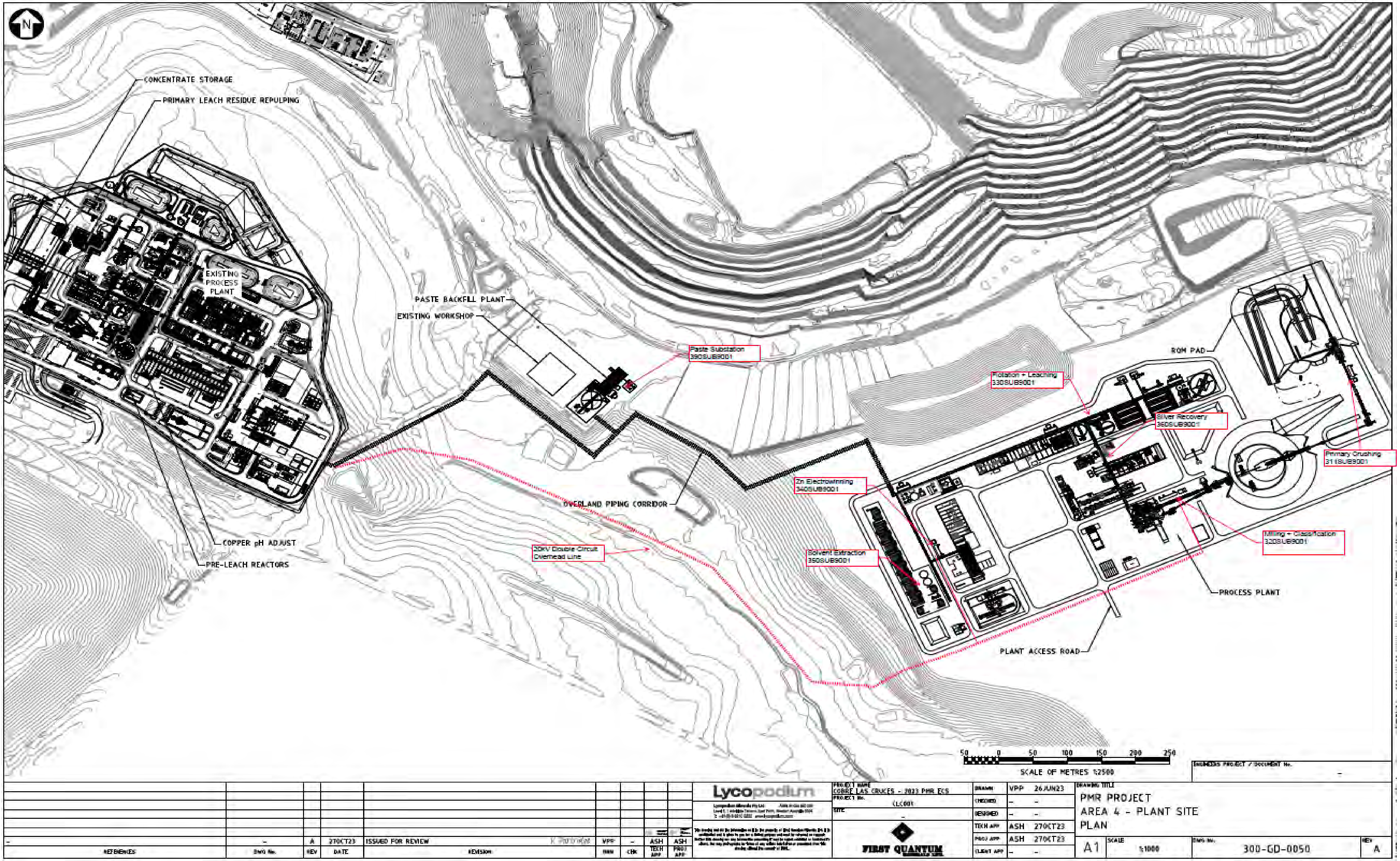


Figure 18-10 Existing and New Plant Sites



18.6 Water Management

18.6.1 Introduction

The scope of the CLC water management system includes underground and surface sources as well as external sources of water. Underground water is comprised of underground mine drainage and the Drainage and Reinjection System (DRS). Surface water takes into account the management of the rainwater and the streams around the Project whilst external sources of water include the San Jerónimo sewage plant in Seville.

The hydrogeological model indicates that dewatering from two aquifers will be required. The upper aquifer lies in the Miocene age strata which is a porous media. The lower aquifer is in Palaeozoic strata, a fractured media which is more heterogeneous and anisotropic.

The hydrogeological model was first developed in August 2018 (REDUX model) and subsequently updated in March 2020 and June 2022 (N REDUX model) as new data was logged and added. The model was reviewed and validated by KLM Consulting Services in March 2020 with the latest update completed in June 2022.

The model was calibrated with both regional groundwater information and piezometric information obtained from existing groundwater control boreholes. In addition, the model was calibrated with the water balance in the open pit and the existing underground exploration ramp.

18.6.2 Global Water Balance

The global water balance estimate is based on the following:

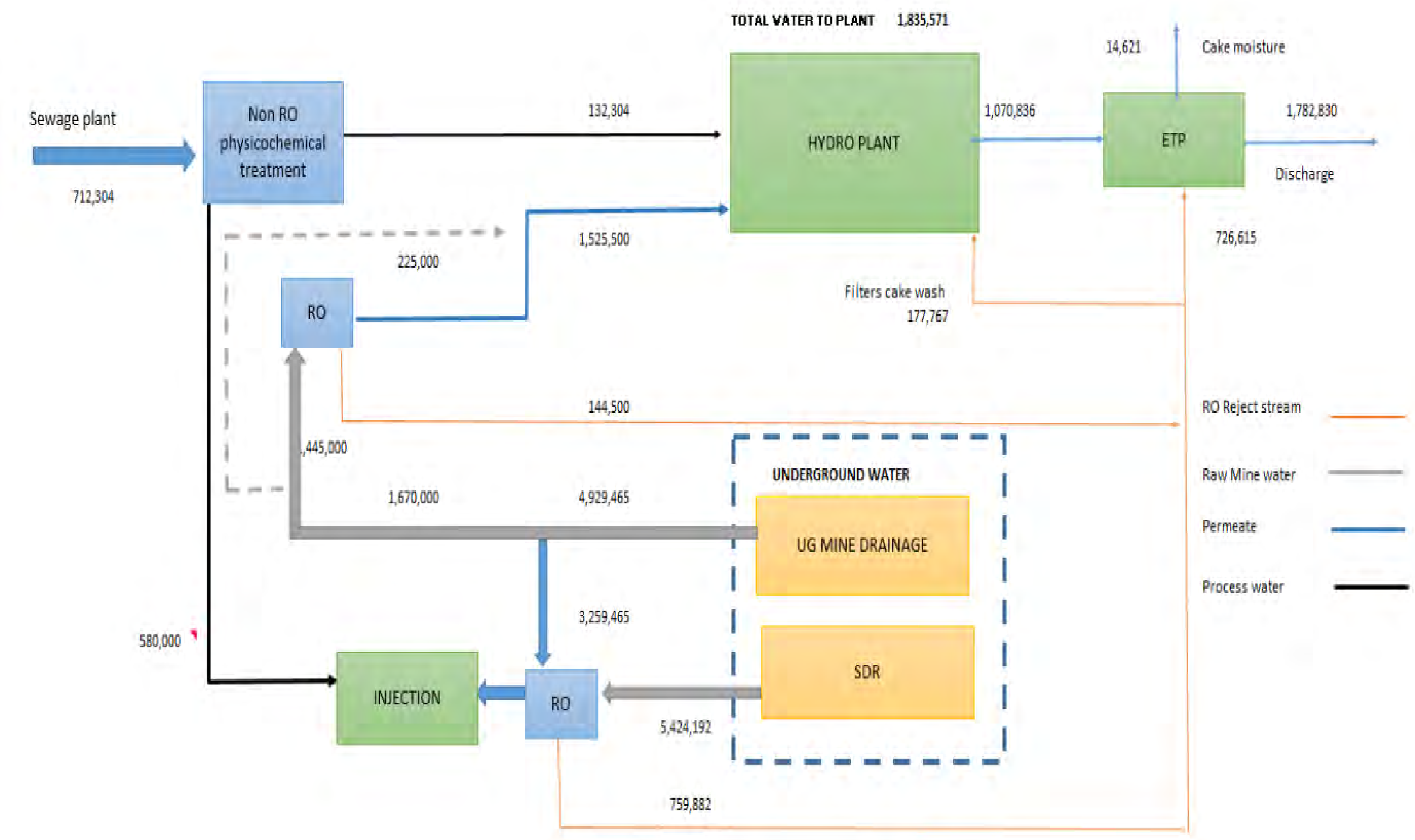
- Underground drainage and DRS volumes as per the hydrogeological model.
- Hydrometallurgical plant water feed as per the metallurgical mass balance, including water quality standards.
- Reinjection volumes calculated by the hydrogeological model to maintain the settled aquifer piezometer levels.
- Supplementary water from the San Jerónimo sewage treatment plant.
- Underground water RO treatment waste discarded as effluent and also partially used for tailings filters cake wash water.

Table 18-3 Main PMR Water Balance Volumes

Water Volumes		ML per annum
Underground Water	UG Mine Drainage	4,929,465
	DRS Extraction	5,424,192
Total reinjection	From UG	7,923,775
	External	580,000
Hydrometallurgical Plant Feed		1,835,775
Discharge		1,782,830

Water concession intakes are limited to 4.36M m^3/y with modelled requirements sitting comfortably within this limit at 3.35M m^3/y. Water concession discharges to the river are limited to 2.34 Mm^3/y with the hydrological model requirements sitting comfortably within this limit at 1.797 M m^3/y.

Figure 18-11 Global Water Balance



18.6.3 Water Treatment Plants and Ponds

A schematic chart of the water treatment plants and ponds is shown in Figure 18-11.

Treatment Plants

The treatment process to be used is determined by the final use of the stream:

- Supply to the hydrometallurgical plant from the mine or the San Jerónimo sewage treatment plant
- Environmental discharge to the river or aquifer recharge via the DRS.

Underground water for DRS or for process water makeup is treated by RO and the existing RO plants will require expansion to handle increased underground water volumes. The RO treatment plant process is as follows:

- Physical and chemical pre-treatment of solids, turbidity, and trace elements including arsenic and iron that may form precipitates and organics
- Ultrafiltration, consisting of filtration with membranes retaining particle sizes below 0.1 microns, to separate finer particles and bacteria.
- RO to remove dissolved solids.

External water from the San Jerónimo sewage plant is to be pumped to the CLC PSP pond. From this pond, water will be supplied to the following treatment ponds and treatment plants:

1. Pond C13: to supply the hydrometallurgical plant with process water. Water treatment deals with suspended solids and organic hydrocarbons and includes the following stages:
   - Disinfection with hypochlorite.
   - pH fitting with sodium hydroxide for metals precipitation
   - Precipitated solids removal in a decanter
   - Sand filtration

2. Pond C13 bis; to supply the reinjection volume to match unbalanced piezometric levels due to RO losses. Basic treatment stages are as follows:
   - Disinfection with hypochlorite
   - Sand filtration
   - Ultrafiltration
   - Ionic exchange for denitrification (??)

As a summary, all the PMR capacities for the treatment plants and facilities are listed inTable 18-4.

Table 18-4 PMR and Existing Water Treatment Plant Capacities

PMR Water Facilities and Plant Capacities							
	Balance	Existing Design Capacity		Extended capacity for PMR		PMR Capacity	
	m^3/y	m^3/y	l/s	m^3/y	l/s	m^3/y	l/s
UG water total flow	10,353,657						
UG water bypass to Hydro plant (no RO Feed)	225,000						
UG water: Physicochemical prior to RO	10,128,657						
UG water: RO plants		9,005,718	286	1,122,939	35	10,128,657	321
Reinjection systems (wells, pumps, pipes)	8,503,775	5,455,728	173	3,048,047	97	8,503,775	270
C-13 bis outlet (sewage for reinjection)	580,000	1.095.000	35			1.095.000	35
C-13 outlet (sewage for Hydro Plant)	132,304	1.204.000	38			1.204.000	38
ETP plant for discharge	1,797,451	1.208.000	38	574,830	18	1.786.160	57

Ponds

The main pond is a 250,000 m^3 new emergency pond, to be constructed in two separated bodies (DEAM 1 &2) to collect mine drainage peak flows. The others are all existing ponds at CLC.

In addition there are non-contact water ponds to collect and manage the rainfall in the wider CLC area prior to discharge to the natural streams. Leachates from the IPTF will be collected in ponds and sent to the water treatment plants.

ITEM 19 MARKET STUDIES AND CONTRACTS

19.1 Markets and Contracts

19.1.1 Overview

The annual revenues in the cash flow model were calculated using September 2023 Broker Consensus long term metal pricing information for copper, lead, zinc, and silver from a number of banks and financial service institutions. (Table 19-1).

Table 19-1 Element Selling Unit Prices and Payabilities

ELEMENT	SELLING PRICE	UNIT	PAYABLE (%)
Cu	3.80	$/lb	100
Zn	1.20	$/lb	100
Pb	0.95	$/lb	95
Ag	22.75	$/t oz	95

All metals produced have established markets, both domestically within Spain and internationally. Metal payabilities are based on historical actuals realised by FQM in global markets and have been provided by FQMs internal marketing team. The following metals premiums are assumed to be received based on final product qualities;

- Copper = $135.30/t Cu cathode
- Zinc = $198.00/t Zn ingot

As is the case with products from the Company's other operations, all products will be sold through the Company's internal marketing division who have an established customer base for copper and zinc products which constitute 84% of total estimated total revenue from the Project.

There are as yet, no contracts in place for the sale of products from the Project.

19.1.2 Copper

Cobre Las Cruces (CLC) has produced copper cathodes since 2009, by means of extracting and processing secondary ore from its adjacent open pit. Production of copper during the open cast exploitation of the secondary mineralization and retreatment of its high-grade tailings has always provided copper cathodes as final product which has been branded with London Metal Exchange certificate since 2016. The achievement of this certificate has allowed CLC to demonstrate a track record of outstanding high quality in its production as well as to facilitate sales agreement based on recognized quality standards. As a result, the Company has been able to apply higher premiums to its sales compared with market benchmarks and does not suffer penalties imposed by impurities or undesirable elements.

Based on testwork completed to date, the Company reasonably expects to maintain its LME certification in the exploitation of Primary ore and will continue to realize above market premiums.

The QP has reviewed available test work and is satisfied that the copper product will meet a saleable quality and that the corresponding payabilities are reasonable.

19.1.3 Zinc

The domestic Spanish metal market has the capacity to absorb all zinc production from the PMR Project, reducing the freight costs and making the operation of the new mine more profitable. Nevertheless, the final product is expected to achieve LME certification for Special High Grade (SHG) ingots of 99.995% Zn, allowing for ready delivery, if needed, to LME warehouses.

The QP has reviewed available test work and is satisfied that the zinc product will meet a saleable quality and that the corresponding payabilities are reasonable.

19.1.4 Lead

Final product as lead metal briquettes is expected to have a composition of at least 99.5% Pb with less than 0.5% moisture. Copper and bismuth content is expected to be lower than 68ppm and 7ppm respectively.

Its main use is in the manufacture of lead-acid batteries (2/3) and it is a material that is being replaced by new ones. It is also used in printing presses and in the production of alloys. Approximately 50% of total production is from scrap recovery. World production has doubled in 10 years, mainly due to world growth specially of emerging countries.

The QP has reviewed available test work and is satisfied that the lead product will meet a saleable quality and that the corresponding payabilities are reasonable.

19.1.5 Silver

Silver cement product specification is expected to be higher than 99% Ag content and a size between 20 and 25 microns based on the production of 3.5t during the pilot plant operation.

Silver has many industrial and technological uses, for example, electrical contacts, conductors, brazing alloys and solders. This accounts for more than half of the annual demand worldwide over the last five years. Additionally, silver is used in jewellery and coins. In this project, part of the silver produced will be reintroduced into the process itself as a catalyst for the chemical sector. It should be noted that this is a use with great potential and a long way to go in the market, promoting totally clean green industries. Sterling silver, which is 92.5% silver, is used for jewellery and silver tableware. Thanks to its reflectivity, silver is also used to make mirrors. The metal also has uses in photography and x-ray films, as silver bromide and iodide have a high sensitivity to light. Initial discussion have been held with three potential customers confirming the suitability of the product specification.

The QP is of the opinion that the silver cement product will provide a saleable product of the desired quality once the contained mercury in the test product is removed via retorting, a known and proven technology included in the PMR design. Following retorting the product payabilities are reasonable and it is further noted that silver is estimated to provide only 6% of total revenue for the project.

19.2 Contracts

19.2.1 Operating Supplies & Consumables

Long-term contracts for utilities and consumables were previously in place for electricity, oxygen, sulphuric acid, rehabilitation works and laboratory operation and are expected to be readily reentered as required to meet the PMR Project. Based off recent market enquiries, all consumables, reagents and other supplies are deemed to be readily available for delivery to the CLC operation.

19.2.2 Contract Mining Operations

A mining contractor will be engaged to complete the initial underground development including portals, access declines and initial stoping operations. This will include the provision of mobile mining equipment and operators, surveyors, maintenance services, consumables and field supervision. The intent is for CLC to provide engineering and technical services, including mine planning, grade control, geotechnical monitoring and design and construction supervision as per the previous open pit mining contract during the exploitation of secondary ore. Based on recent market enquiries, multiple experienced underground mining contractors have the capacity to meet this demand in the region.

19.2.3 Infrastructure Engineering and Construction Contracts

Engineering and construction contracts for the new PMR Plant, associated infrastructure as well as required modifications to the existing plant will be entered into after the final investment decision. The Company, through its numerous previous and current projects, maintains strong relationships with multiple globally recognised engineering services firms specialising in mineral processing and infrastructure.

ITEM 20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Setting

CLC's environmental management practices align with Spanish mining industry best practice and are strictly regulated and supervised by various government departments and agencies. The area surrounding the CLC operation is characterized by agricultural usage, mainly dryland cropping (wheat and sunflower), and with some plots of olive groves. Almost all of the area is delimited for hunting (hare, rabbit, partridge, thrush and turtledove) which is traditional in the surrounding municipalities. In general terms, the area is significantly anthropic as a result of intense human activity over centuries. The landform is predominantly characterised by very gentle slopes and a few ephemeral streams.

All of the small streams in the Project vicinity are tributaries of the Guadalquivir River, which is the major regional river of significance in terms of agricultural activities, social awareness and cultural heritage. The deposit is also geologically located below a confined aquifer and the protection of the aquifer along with the surface water around the tenure are the most significant environmental aspects to be managed for the Project.

CLC has implemented an integrated system to manage its activities in a systematic manner, providing a clear overview of all operational aspects of the Company, their interrelationship and related risks, in addition to optimizing compliance related work and achieving operational and environmental excellence. This includes being accredited under the international ISO 9000 and ISO 14000 standards which require a well-documented operational structure that is integrated with technical and administrative procedures to guide monitoring work and information reporting in an effective and coordinated manner.

20.2 Status of Environmental Approvals

Mining activities in Spain are subject to Spanish national, regional and local environmental laws. There are also regulations which regulate, among other things, air emissions, water discharges, soil contamination, waste management, management of hazardous substances, natural protected sites, protection of natural resources, heritage areas, endangered species and reclamation activities. Spain has adopted European Union directives pertaining to environmental matters into its domestic legislation and these rules impose environmental conditions on the management of, amongst other things, water, and waste and air emissions.

The valuable experience gained by CLC in environmental management during the open pit operations is considered a major strength for the PMR Project.

Under CLC's various licenses, it is required to comply with several environmental commitments including commitments to receiving and observing the terms of several permits. The key permits that CLC have obtained for the PMR Project are:

- **The Unified Environmental Authorization (AAU)**

  The AAU regulates mining activities and auxiliary facilities associated with mining at an environmental level. This authorization includes the original Declaration of Environmental Impact (DEI) of the Las Cruces Secondary Project, which was the formal statement from the regional authority that determined the environmental suitability of Las Cruces. The AAU and the DEI also establish the environmental conditions for the development and operation of Las Cruces regarding protective measures, mitigation and monitoring.

- **The Integrated Pollution Prevention and Control Authorization (IPPC- AAI)**

The IPPC-AAI regulates the polymetallic refinery in addition to discharge into the Guadalquivir River.

- **The PMR Water Concession**

  The PMR Water concession provides the Project with the necessary water intakes for the development of underground mining and polymetallic refinery operation. In addition, this permit contains the necessary water intake to continue with the artificial recharge of the aquifer using treated and regenerated wastewater. This injection provides an additional guarantee for the preservation of the aquifer located in the area affected by the mine, by maintaining the piezometric levels at the points and values set by the Administration. In order to process this Water Concession application, the Project had to be declared by the Mining Authority as a Project of "Higher Public Interest". Once this Declaration was obtained, the Guadalquivir Hydrographic Confederation (CHG, the water administration) favourably reported the exceptional authorization regime provided under article 4.7 of the Water Framework Directive for the mine and Polymetallic Refinery Project. This regime is essential to lower the aquifer level and to be able to exploit the mine in optimal safety and operating conditions. This new authorization was granted in March 2023.

- **The DRS Authorization**

  The DRS authorization issued by the CHG, regulates the extraction and re-injection of groundwater surrounding the Las Cruces open pit. The previous DRS Authorization for the Secondary Project has been modified and extended. This new authorization was granted in March 2023 and covers, among other things, the construction of new infrastructure associated with the PMR DRS.

The experience and information of environmental vectors, permits, and community aspects reflect CLC's previous operation and should be considered as a strength and a basis for developing the PMR Project.

20.3 Environmental Monitoring

The mining industry in Spain is highly regulated with a focus on constant monitoring and compliance combined with periodic checks and audits. As well as the CLC Environmental Department, responsible for ensuring the Company complies with all external regulations as well as internal rules, there are two government employees permanently on site monitoring environmental aspects of the operation. These are:

- Water Authority – a person employed by the CHG and stationed on site to monitor water quality and compliance with the conditions of water management and operations.
- Environmental Authority – a person employed by the Regional Environmental Authority and stationed on site to monitor environmental compliance.

In addition to that, remote sensors send real-time data directly to the Regional Environmental Authority offices concerning the characteristics of water discharge into the Guadalquivir River, specifically pH, temperature and water flow. These environmental control actions will continue to be maintained for the PMR Project.

20.4 Environmental Issues

After several upgrade projects, the reliability of the neutralization plant was increased markedly and currently the plant produces a very stable high-quality effluent with very few scattered and short periods of non-compliance. Therefore, limits for the discharge to the Guadalquivir River are being consistently met for the current Project.

CLC has received three Notices of Violation (NOV), served in 2014, 2017 and 2019, regarding loss of groundwater and insufficient compensation by the Dewatering-Reinjection System.

With regard to the 2014 NOV, so far CLC has achieved a reduction in the fine of up to €326,700 from the initial amount of €595,727. The Company has already paid the sum of €237,600 in advance without admitting any liability for the NOV issues and continues to challenge the penalty.

The 2017 case, in which a penalty of €1,496,882 was proposed, has been shelved provisionally and is pending possible reopening.

On the 2019 legal procedure, the penalty initially proposed (€1,535,078) has been reduced by €200,000 (CLC has made a part-payment of €800,000 without admitting any liability). Recently CLC has received a favourable response from the Supreme Court, and accordingly the penalty reduced.

For the PMR Project, the new Water Concession and DRS permit have addressed all the former issues so that under the new permit conditions, it is believed to be unlikely that similar breaches will reoccur.

20.5 Social and Community Related Requirements

CLC maintains good relationships with all four local municipalities (Gerena, Guillena, Salteras and La Algaba) directly affected by the Project. The Company has developed relationships with non-government associations in the area and has meetings with a neighbourhood group of representatives from each of the four municipalities as well as people from nearby properties. These meetings discuss developments, plans and other issues that may be of interest to the group.

In 2010, CLC created the Cobre Las Cruces Foundation (the “Foundation”) intended to encourage, promote and develop Environment and Sustainable Development, thereby enabling the Company to meet its objectives regarding social corporate responsibility. Since its start-up, the Foundation has become one of the most active entities of its kind in Andalusia, with more than 500 activities and/or projects performed to date.

The Foundation focuses on two spheres of activity: collaborations with other entities; and its own projects. In the first of these, the Foundation has provided support to other institutions at the regional, provincial and local level to boost education and training, social, charitable work, cultural, sporting and environmental initiatives. Since inception, the foundation has donated more than €10 million to the local economy.

At the time of writing this report there were no material social risks at CLC.

20.6 Mine Closure Provisions

A new Restoration Plan was developed and submitted in July 2019. This plan was approved in September 2022. It is based on the PMR Project scope recently submitted to the authorities and includes a basic Closure Plan for the underground mine project.

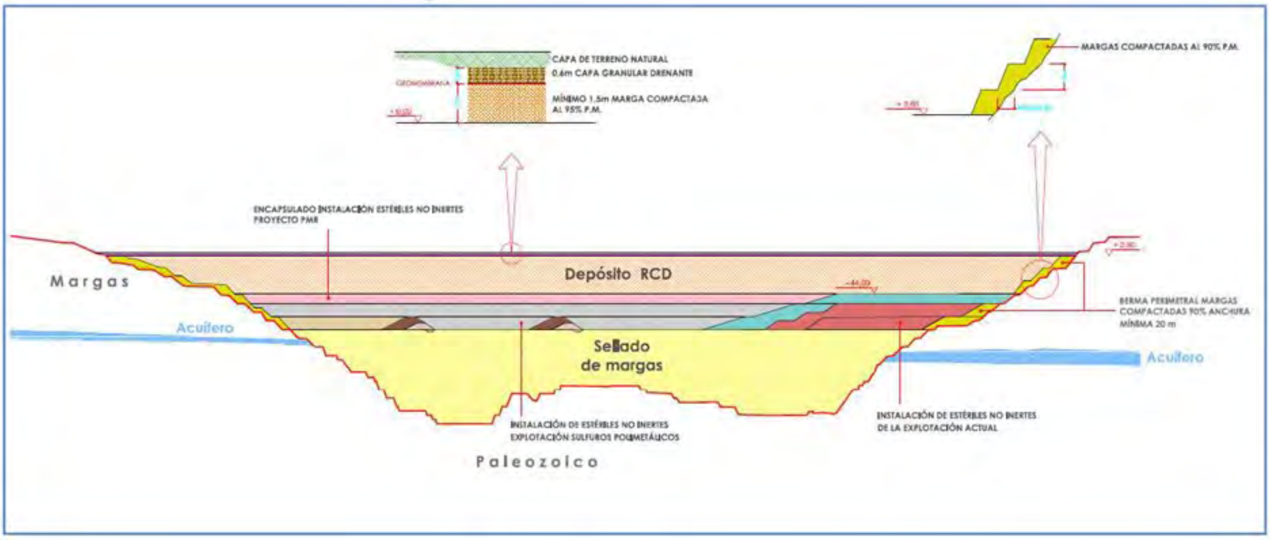
The PMR Environmental Restoration Plan for CLC is a continuation of the previous Open Pit Mine Restoration Plan. The current plan incorporates the results of that Restoration Plan plus modifications for Project optimizations, updates for the new underground mine project, and incorporation of all permit conditions specified in the DEI, AAU and AAI. The main modifications to the original restoration plan are:

1. A new Tailings Storage Facility (TSF) in the pit. This facility is placed over a partial backfill with marl, which creates a residual void completely lined with marl as a dry seal. The new facility, which includes both plant tailings and waste rocks, is lined to prevent and control the acid drainage potential. Currently, this facility is already in use. The restoration of the mining pit will be completed with a landfill for inert waste material, mainly from construction and demolition works. Tailings management during the operations phase is discussed in Item 18 Project Infrastructure.

2. The surface areas to be restored have been modified in line with the current underground mining plan.

3. The water supply pond (PSP) will be restored and will remain as a wetland since it has now become a lake ecosystem that is home to a multitude of aquatic species of birds and other fauna such as otters. Tailings management during the operations phase is discussed in Item 18 Project Infrastructure.

Figure 20-1 Construction Project to Encapsulate the Tailings Storage Facility (IPTF) in the Las Cruces Pit



CLC is not required by law to submit a Construction Project concerning the back-fill of the pit; however, a Performance Project must be registered with the authorities detailing the pit's use as a depository for discarded construction material. This plan was filed in 2012 and has been updated for the PMR Project.

The Las Cruces Mine has been and will continue undertaking on-going environmental restoration comprised of:

- Temporary planting for erosion protection;
- Final re-vegetation of areas completed during various project phases; and
- Removal of infrastructure at Project closure.

Temporary planting has been undertaken for erosion prevention and landscaping primarily to dump surfaces and stockpiled materials that are being covered subsequently with other materials or moved elsewhere. Permanent restoration is applied to the final reclaimed surfaces in dumps, mine waste storage, or areas that were used for temporary stockpiles.

Visual berms and waste piles are graded at slopes of approximately 4 horizontal to 1 vertical in all areas visible to the public. These slopes approximately match the rolling terrain in the Project area. Slopes are graded and seeded. The tailings disposal facility is capped and seeded as placement of the tailings progresses across the facility. Similarly, the pit is being partially backfilled behind the open mine operation as it proceeds across the proposed footprint of the pit. Backfilling of the pit consumes a portion of the marl produced over the life of the mine, limiting the size of surface waste dumps that require reclamation.

Typically, the Spanish government requires posting of a bond based on the total disturbed area of the mine. Since CLC is reclaiming the site throughout the life of the mine, the largest bonding requirements occur early in the Project, and gradually decrease as mining and reclamation progresses. The current environmental bond is placed at €28.3 million with an additional €5 million labour bond in place at start-up of the Secondary Project.

The environmental restoration will be carried out progressively during the Project life. The first two years being construction years where the primary activities will be pit backfill and the construction and restoration of anew visual berm that will hide the new treatment plant. The remaining years will include ongoing restoration where final proposed grades are reached. The post-mining activities include final capping, grading, re-vegetation, and other activities associated with reclamation of the waste dumps, livestock trails, power lines, tailings facility, mill site, water diversion and storage ponds, roads, and other facilities.

The overall cost of restoration and dismantling of current mine and plant facilities as at end 2022 totalled €75.2 million and comprises mine backfill, aquifer water control, geotechnical surveillance, environmental restoration, plant dismantling and employee severance cost.

Underground mining is planned for the PMR Project and is therefore expected to incur lower mine closure costs when compared with the recently completed secondary sulphide open pit mining. Asset retirement obligations include sealing the tailings facility, plant dismantling and aquifer water control as well as restoration of temporary visual berms. Total costs are to be assessed on finalization of PMR Mineral Reserve estimate.

Figure 20-2 Pit Situation before Final Encapsulation

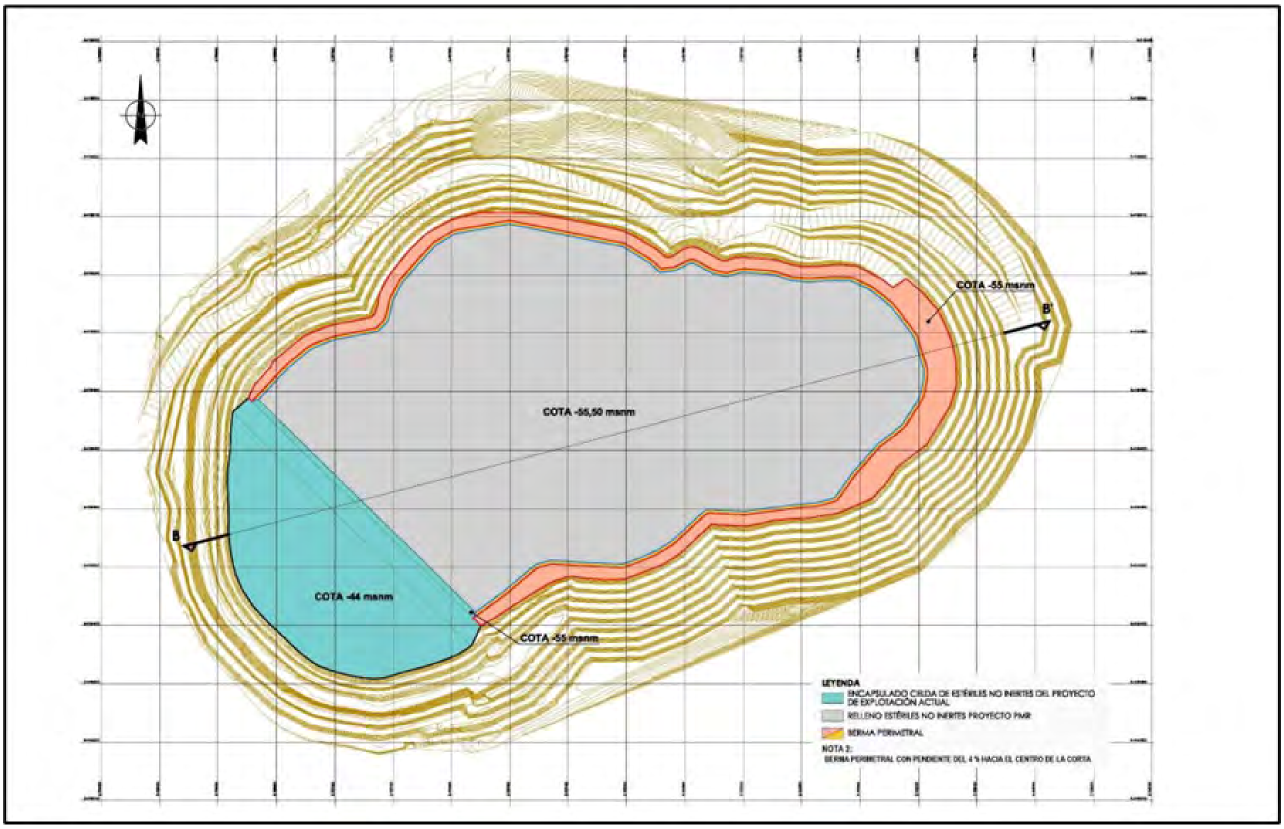


Figure 20-3 Pit Situation after IEC Final Encapsulation

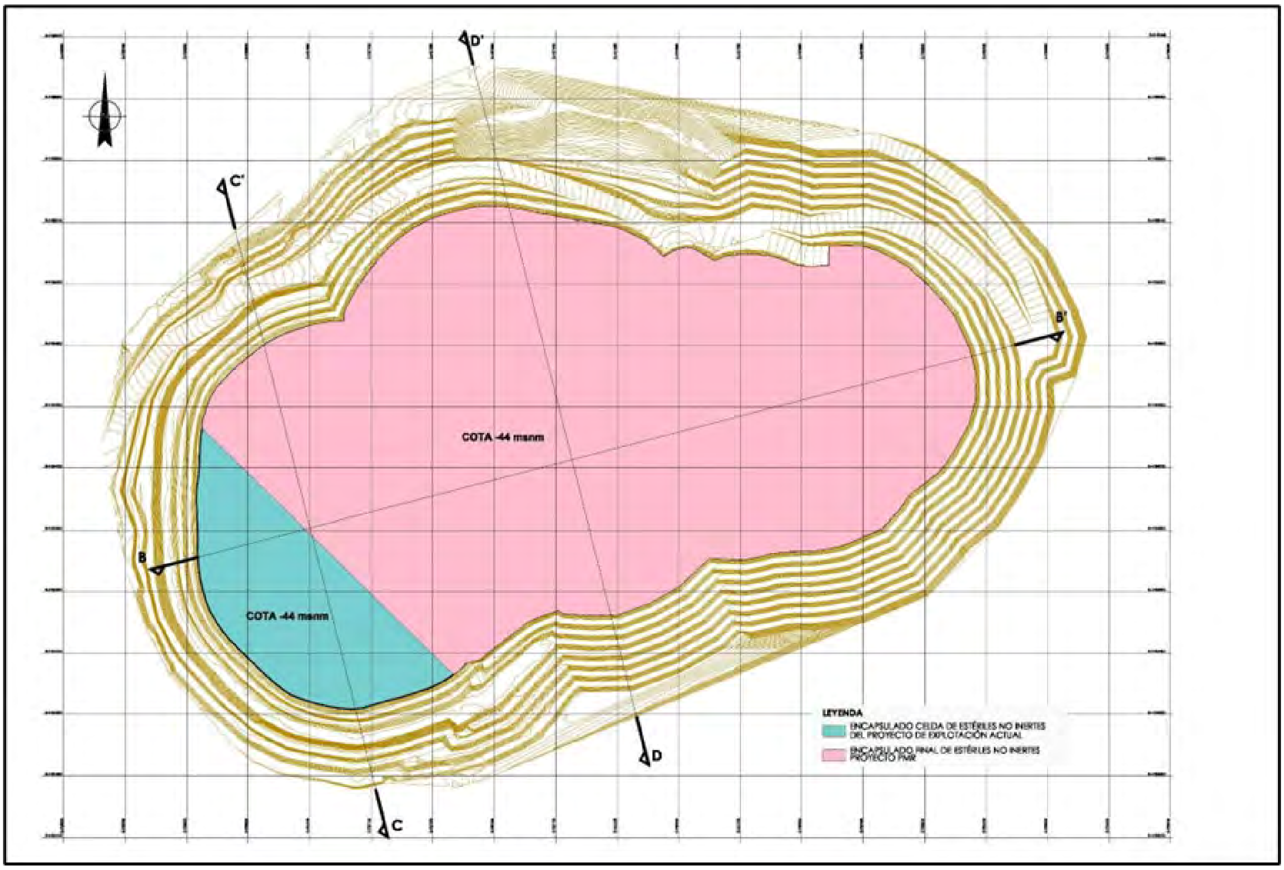


Figure 20-4 CLC Closure Plan (year +18)

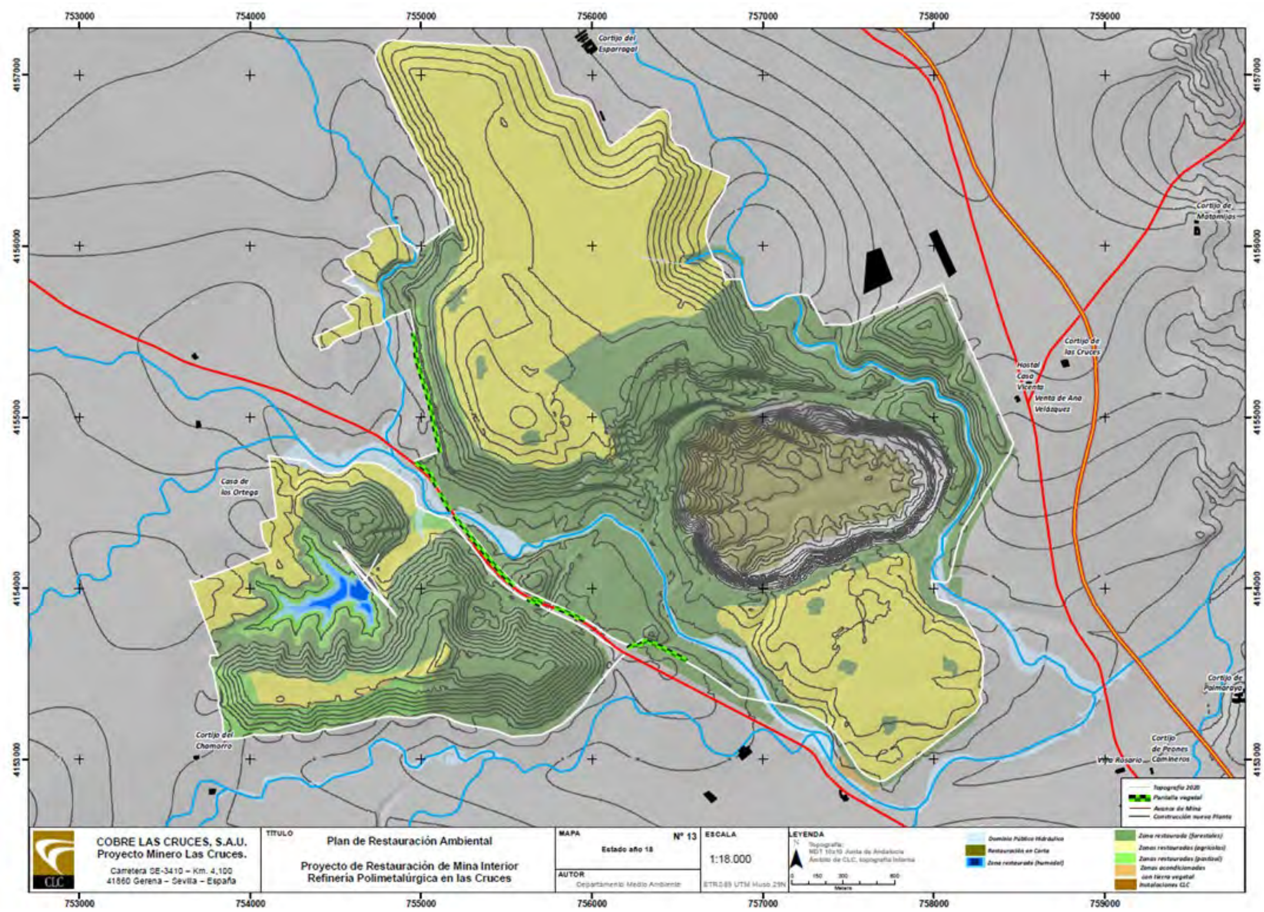
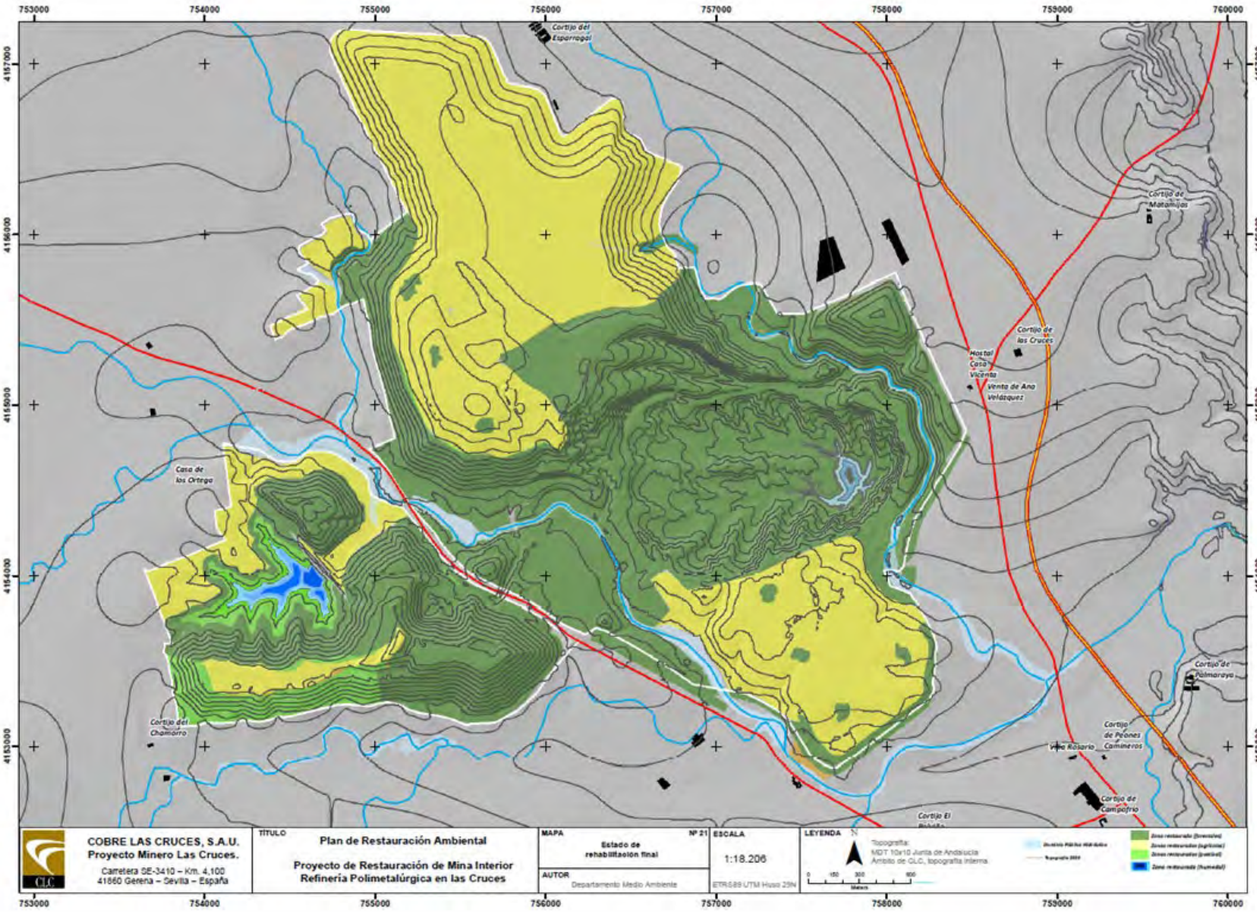


Figure 20-5 CLC Closure Plan (FINAL STATE)



ITEM 21 CAPITAL AND OPERATING COSTS

In the following commentary all converted costs referenced are in United States Dollars (USD) unless specified otherwise.

21.1 Capital Cost Estimates

21.1.1 Overview

The PMR Project will make use of the existing facilities at CLC but will need additional investment to develop and operate an underground mine and polymetallic processing plant. The total capital cost estimate for the Project is estimated at $846M, consisting of two major components, firstly the construction of the processing plant and secondly the development of the underground mine. This capital estimate includes contingency of 14% or $104M.

Table 21-1 PMR Total Capital (including contingency)

COST ITEM	$ Million
Mine	271
Plant	499
Owners and Indirect costs	28
Existing Cu SX/EW plant refurbishment	23
Water facility expansion	21
Other	4
Initial Project Capital	**846**
Sustaining Capital	6
Mine Closure Provision	98
Total CAPITAL	**950**

21.1.2 Basis of Capital Costs

The basis of the itemised costs in Table 21-1 are as follows;

- Capital estimate base date is third quarter 2023 US dollar terms.
- Escalation of 5% pa compound was applied to any quotes with a validity date prior to the base data.
- Estimates exclude:
  - foreign currency exchange fluctuations after the estimate base date
  - escalation after the estimate base date
  - transaction costs and taxes
  - financing costs
- Initial (capitalised) mining costs are derived from supplier consumable quotations, i.e. '*2022 Cost Mine Intelligence'* database rates and have been applied to a mining development profile by a reputable underground mine planning consultant (IGAN Consultores, Spain).
- Processing plant material take-off (MTO) quantities were developed by a reputable mineral processing consultant (Lycopodium) with supplier quotations (59% of direct plant capital by value) and recent (<6 months) Project database rates (26% of direct plant capital by value), and with the remainder from projects >6 months and escalated to the estimate base date.

- Capital estimates for the following four processing plant major vendor packages were provided by vendors on a lump sum basis. Each of these packages has been broken down in the capital cost estimate into the various engineering disciplines to facilitate the determination of an overall Project cost contribution by discipline;
  - Zinc Solvent Extraction vendor package.
  - Zinc Electrowinning vendor package.
  - Zinc Melting and Casting vendor package.
  - Lead Melting and Casting vendor package.
- Labour rates, for construction and capitalised mining costs, were based on supplier quotes and actual CLC rates and contemporary Spanish projects, and publicly available European Commission labour data.
- Capital estimates for the 'Railveyor' materials handling system were provided by the equipment supplier (Rail-Veyor Technologies) and applied to a MTO developed for the 2017 Railveyor concept study by WSP Canada. Adjustments to suit local conditions were made by CLC.
- Mining equipment capital estimates were based on supplier costs and applied to equipment profiles developed by CLC and IGAN.
- Electrical material take-off (MTO) quantities were developed by a reputable electrical engineering consultant (Process E&I) and supplier quotations applied to the MTOs.

21.1.3 Mine

Mining costs (Table 1-14) are related to the construction of a new underground mine located underneath the current open pit mine. With a capacity of 2.0Mtpa, total mine capital costs are $271M primarily for the development of underground access to the ore body and construction of mining infrastructure and facilities.

Table 21-2 Mine Capital Cost

COST ITEM	$ Million
Mine Development	99
Railveyor and Ore Handling System	43
Mobile Equipment Fleet	42
Capitalised operating costs	31
UG Main Dewatering Sumps	11
UG Electrical Distribution	8
UG Ventilation	4
Other	8
Contingency	25
Total Mine CAPITAL	**271**

21.1.4 Plant

Capital costs cover the construction of a new process plant with an annual processing capacity of up to 2.2Mtpa of ore and producing a steady state LOM average of ~41,000 tonnes of copper equivalent per annum over the life of mine in the form of four final products; copper, zinc, lead and silver (Table 1-15).

Table 21-3 Plant Capital Cost

COST ITEM	$ Million
Zinc Plants	127
Indirect costs	67
Electrical	48
Lead and Silver Plants	34
Crushing & Grinding	40
Flotation	27
Leaching	24
Iron Removal	21
Paste Plant	15
Water & Reagents	11
General	6
Aluminium Removal	4
Effluent Treatment	2
Contingency	73
Total Plant CAPITAL	**499**

21.1.5 Current plant refurbishment

Upon the completion of the tailings processing phase, an engineering survey was conducted which identified areas for refurbishment, obsolescence and modification of the existing copper cathode solvent extraction and electro winning process plant. An allowance of $22.8M including 10% contingency has been included.

21.1.6 Water

The existing DRS water facilities will be expanded to handle 8.4Mm^3 of water per annum with initial capital requirements of $20.6M including 10% contingency.

21.1.7 Mine Closure Provision

Two main costs are included for mine closure: sealing the tailings facility and dismantling the process plant. The cost of one year of operation of the water treatment facilities to allow operations in the tailing facilities has also been included (Table 21-4).

Table 21-4 Closure Provision

COST ITEM	$ Million
Previous open pit operation	67
Tailings facility sealing	14
DRS water treatment	9
Process Plant dismantling	7
Dump restoration	1
Total	**98**

21.1.8 Contingency

The capital cost estimate for the processing plant and related site infrastructure has been comprehensively developed by Lycopodium including benchmarking to their internal database and a contingency allowance assessed accordingly. Contingency factors of between 15% and 20% have been applied deriving an overall contingency of 17.0%.

The capital cost estimate for the underground mining and other surface infrastructure has been completed by the CLC team utilising budget quotes from reputable suppliers. A contingency factor of 10% has been applied after benchmarking overall Project contingency against similar recent projects at the PFS and FS stage.

21.2 Potential Capital reductions

The process plant capital estimate is $510.3M. Three potential capital reduction opportunities that were subsequently adopted to reduce the estimate by $11.65M to $498.65M;

1. Lead Smelting & Casting Removal ($10.2M)
2. Reorientation of crushing materials handling layout ($0.45M)
3. Reduction of lead cementers by 2 units to 10 units initially. ($1.0M)

Consequently Items 1 and 2 above do not form part of the PMR Project scope and are not planned to be constructed. Payabilities received for lead production has been adjusted down to reflect this change.

21.3 Operating Cost Estimates

21.3.1 Overview

The LOM of the Project is 24 years including initial development. The process plant at 46% is the largest contributor to operating costs followed by the mining operations at 24%, and then maintenance activities at 18% (Table 1-16).

Table 21-5 Steady State Operational Costs (excludes 1 years production ramp up and ramp down)

COST ITEM	LOM Average Annual Cost $ Million	LOM Average Annual Unit Cost $/ore t processed
Plant	90	41.61
Mine	45	21.27
Maintenance	31	14.43
G&A	15	6.93
Water	10	4.62
Royalty	3	1.39
Total	**194**	**90.25**

21.3.2 Basis of Operating Costs

Operating cost unit rates have been calculated utilising a range of sources including existing plant, infrastructure, current CLC utilities & administration costs, in addition to current market rates for consumables. Mining unit rates were benchmarked using the 2022 Costmine database, whilst consumable consumption rates in the plant were derived from existing data, pilot plant testing and other metallurgical testwork results. For the mine operations, estimates were benchmarked by mining consultants with knowledge of similar operations.

21.3.3 Process Plant Costs

Variable costs amount to over 80% of total plant costs. The main variable cost drivers for the process plant are utilities and reagents, specifically lime and electricity being the largest costs for the ore processing. Labour represents half of the fixed costs for the plant and 8% of total plant costs.

Table 21-6 lists the average annual total costs and unit cost per ore tonne processed.

Table 21-6 Average Annual and Unit Costs

COST ITEM	LOM Average Annual Cost $ Million	LOM Average Annual Unit Cost $/ore t processed
Other consumables	32	14.66
Utilities	27	12.26
Lime	17	8.08
Labour	7	3.43
External Services	3	1.47
Materials and supplies	2	0.94
Other	2	0.77
Total	**90**	**41.61**

21.3.4 Mining Costs

Total mine operating costs are $21.27t/t ore with the largest components being materials, supplies and consumables contributing over 65%. Labour costs are approximately 18% of mine operating costs. Table 21-7 lists average annual total costs and unit cost per ore tonne processed:

Table 21-7 Annual and Unit Costs per ore tonne processed

COST ITEM	LOM Average Annual Cost $ Million	LOM Average Annual Unit Cost $ /ore t processed
Consumables	15	7.11
Materials and supplies	15	7.05
Labour	8	3.88
Utilities	3	1.40
Other	4	1.83
Total	**45**	**21.27**

21.3.5 Water Costs

Water costs are linked to water activities that consist of draining, pumping, treating and reinjecting water flows in the mine site. Such operations are already performed by CLC and cost accounting is already in place for these activities. The water volumes for the underground mine are higher than those in the existing open pit mine and operating costs have increased accordingly.

Table 21-8 lists average annual total costs and unit cost per tonne treated.

Table 21-8 Annual and Unit Costs per ore tonne processed

COST ITEM	LOM Average Annual Cost $ Million	LOM Average Annual Unit Cost $/ore t processed
Contractors	4	1.72
Consumables	3	1.25
Utilities	2	0.68
External Services	1	0.39
Materials and supplies	0	0.25
Other	0	0.33
Total	**10**	**4.62**

21.3.6 Maintenance Costs

Maintenance costs have been calculated for process plant and water facilities. Table 21-9 lists the average annual total costs and unit cost per ore tonne processed.

Table 21-9 Maintenance Annual and Unit Costs per ore of tonne processed

COST ITEM	LOM Average Annual Cost $Million	LOM Average Annual Unit Cost $/ore t processed
Contractors	14	6.62
Materials and supplies	11	5.46
External Services	3	1.18
Labour	3	1.17
Total	**31**	**14.43**

21.3.7 G&A Costs

Average annual costs for G&A are $15M and include the activities of the following departments:

- General and Administration
- Human Resources
- Safety.
- Technology
- Environment

The main costs relate to external services like insurance, security, laboratory and consultancy, in addition to internal labour, and overall, equate to $6.93/t ore processed.

21.3.8 Distribution and Selling Costs

Royalties have been calculated per contractual terms and conditions with third parties and are attributable to copper production only when exceeding a certain price.

Metal end-product transportation costs and customer premiums by metal are also included, pertaining to potential customers that have been identified in the country. Customer premiums have been allocated to copper and zinc, $135.30/t copper cathode and $198/t zinc ingot.

Payables for copper and zinc are 100% while for lead and silver have been considered at 95% of metal production.

ITEM 22 ECONOMIC ANALYSIS

In accordance with 43-101, the economic analysis set out below does not include Inferred Mineral Resources.

The economic analysis in the form of a basic cashflow model is intended to support the Mineral Reserve estimate, and in order to demonstrate a positive cashflow for mining and processing. The development and expansion capital costs are included in the analysis for completeness. The model is provided pre-tax.

22.1 Methodology and key assumptions

22.1.1 Overview

The basic methodology adopted for the economic analysis was to initially tabulate the detailed production schedule physicals (ore and waste mined, ore processed, stockpiles reclaimed and the recovered metal profile for the production schedule). To reflect the gradual attainment of the designed plant recovery during plant ramp-up in the third year of the Project, downgrade factors were subsequently adopted and applied to the first year of the ramp-up period. Linked to the recovered metal profile then, were the payability assumptions and the metal prices, to arrive at annual gross revenues.

Initial and expansion capital costs, and then sustaining capital costs were deducted from the gross revenue, followed by operating costs. Treatment charges and refining charges (metal costs for silver and lead realisation) were next calculated, inclusive of royalties, to arrive at annual undiscounted net revenues.

Key assumptions for the base case economic analysis are as follows:

- The plant feed tonnes and grade account for mining dilution and mining recovery factors.
- The initial development capital is expended in Years 1 to 6.
- The assumed infrastructure capital spending intensity is 16% in Year 1, 60% in Year 2 and 12% in Year 3.
- Costs associated with underground decline (i.e., ramp) development, pit bottom development and pre-commercial ore extraction in Years 1 to 2 were assumed to be capitalised.
- Initial mine ore production starts in Year 1 at 0.02Mt, ramping up to 1.8 Mtpa for Years 5 to 6, then to 2.0 Mt from Year 7 onwards.
- Initial plant production starts in Year 3 at 1.5Mtpa ramping up to 2.2Mtpa in Year 4 onwards.
- The first year of processing features down-rated processing recoveries.
- Existing stockpiles of 5,023Mt are available to feed the plant and make up any shortfall between planned direct mine and plant feed.
- The annual operating costs (i.e., processing and G&A unit costs) were not profiled for each year.
- The metal costs were not profiled against varying concentrate grade, payability or other factors.
- Cashflows exclude an export levy, income tax, carbon tax, interest and depreciation.

By FQM definition, Project cash costs include all mining, processing, G&A and stockpile reclaim operating costs, excluding sustaining capital allowances, and are in terms of $/payable pound of copper. C0 cash costs exclude profits from the sale of by-products and take no account of metal costs (i.e., TCRC charges and royalties), whereas C1 cash costs include profits from the sale of by-products, and also take account of metal costs but not royalties.

22.1.2 Metal Pricing

The annual revenues in the cash flow model were calculated using July 2023 Broker Consensus long term metal pricing information for copper, lead, zinc, and silver from a number of banks and financial service institutions. (Table 22-1). Metal payabilities are based on historical actuals realised by FQM in global markets and have been provided by FQM's internal marketing team.

Table 22-1 Element Selling Unit Prices and Payabilities

ELEMENT	SELLING PRICE	UNIT	PAYABLE (%)
Cu	3.80	$/lb	100
Zn	1.20	$/lb	100
Pb	0.95	$/lb	95
Ag	22.75	$/t oz	95

22.1.3 Taxes

The model is provided pre-tax. Royalties, discussed below, are included.

22.1.4 Royalties

The applicable CLC concessions are subject to a contractual royalty of 1.5% of NSR. In modelling the royalty charge, revenue was assumed to be calculated on payable metal. The current owner of the royalty is Royal Gold. An export levy, as a net revenue deduction, was not included in the cashflow model.

22.2 Cashflow Model Inputs

22.2.1 Production schedule

The plant feed production schedule forming the basis of the Project cashflow model is shown below in Table 22-2.

Table 22-2 Production Schedule

YEAR	ORE (Ktpa)	Feed Grades				Production			
		CU GRADE (%)	ZN GRADE (%)	PB GRADE (%)	AG GRADE (PPM)	CU (Ktpa)	ZN (Ktpa)	PB (Ktpa)	AG (Ktpa)
1	-	-	-	-	-	-	-	-	-
2	-	-	-	-	-	-	-	-	-
3	1,485	1.14%	2.21%	1.41%	27.02	14	28	13	618
4	2,172	1.21%	2.39%	1.39%	28.88	22	46	24	1,046
5	2,200	1.56%	2.57%	1.31%	36.04	29	50	23	1,322
6	2,200	1.29%	2.41%	1.28%	31.13	24	47	22	1,142
7	2,200	1.26%	2.45%	1.18%	32.43	24	48	21	1,190
8	2,200	1.31%	2.58%	1.26%	26.88	25	50	22	986
9	2,200	1.11%	2.55%	1.16%	28.26	21	50	20	1,037
10	2,200	1.08%	2.59%	1.44%	30.58	20	50	25	1,122
11	2,200	1.03%	2.44%	1.13%	29.72	19	48	20	1,090
12	2,200	1.11%	2.12%	0.91%	21.92	21	41	16	804
13	2,200	1.01%	1.97%	0.79%	20.20	19	38	14	741
14	2,200	1.06%	2.28%	0.92%	22.53	20	44	16	826
15	2,200	1.17%	1.89%	0.96%	23.69	22	37	17	869
16	2,200	1.08%	1.72%	0.83%	22.03	20	33	15	808
17	2,200	1.05%	2.01%	0.93%	22.93	20	39	16	841
18	2,002	1.09%	2.05%	0.95%	24.78	19	36	15	827
19	1,999	1.04%	1.86%	0.82%	24.98	18	33	13	833
20	2,000	1.01%	1.87%	0.87%	26.71	17	33	14	891
21	1,424	1.05%	1.23%	0.59%	18.02	13	15	7	428
22	814	1.06%	1.30%	0.64%	16.17	7	9	4	220
23	597	1.03%	1.58%	0.75%	16.55	5	8	4	165
24	477	1.35%	1.40%	0.61%	18.58	5	6	2	148
Total	**41,569**	**1.14%**	**2.15%**	**1.05%**	**25.97**	**403**	**790**	**342**	**17,957**

22.2.2 Processing recoveries

Based on the metallurgical testwork, pilot plant results and the mine production schedules, the following average life of mine recoveries are incorporated in the cashflow modelling:

- Copper recovery of 85.12%, producing LME grade "A" copper cathodes
- Zinc recovery of 88.41%, producing Special High Grade (SHG) quality zinc cathodes
- Lead recovery of 79.38%, producing Lead Cement compacted as briquettes
- Silver recovery of 51.87%, with a grade of approximately 26 g/t Ag, producing silver cement

To reflect expected ramp up of initial metal recovery to the factors above, the following ramp-up factors were also applied to the CLC PMR model during Year 1 as follows:

- Copper recovery of 80.90%
- Zinc recovery of 84.03%
- Lead recovery of 63.53%
- Silver recovery to the copper concentrate of 47.92%,

The impact of this factoring is a reduction in the overall average life of mine copper recovery from 85.12% to 84.97%.

22.2.3 Capital and closure costs

Table 22-3 lists the initial Project development capital costs, inclusive of a contingency allowance. In the cashflow model the total amount is expended over the first three years. Additional capital provisions, thereafter are also listed in Table 22-3.

Sustaining capital costs for on-going mining equipment replacement over the life of the Project are included in the cashflow model and amount to $6M, or an average of $0.6Mpa over eleven years. A maintenance charge is provided to cover processing plant and infrastructure replacement within the operating expenditure, on the basis of 3.0% of the installed plant capital expenditure.

Table 22-3 PMR Total Capital

COST ITEM	$ Million
Plant	499
Mine	271
Owners and Indirect costs	28
Existing Cu SX/EW plant refurbishment	23
Water facility expansion	21
Other	4
Initial Project CAPITAL	**846**
Sustaining Capital	6
Mine Closure Provision	98
Total CAPITAL	**950**

22.2.4 Operating and metal costs

The overall average unit operating costs in the cashflow model are:

- mining ore and waste = $22.79/t mined
- ore mining, excluding the mine development period = $20.07/t ore tonne mined
- waste mining, excluding the development period = $124.65/t waste tonne mined
- stockpile reclaim = $1.70/t reclaimed
- tailings = $1.16/t tailings
- processing = $41.61/t processed
- plant maintenance = $14.43/t processed
- water treatment = $4.62/t processed
- G&A = $6.93/t processed

The overall average metal costs, including freight and royalties are:

- freight = $ 11/t Cu cathode sold, $92/t Cu eq
- royalties = $0.06/lb Cu cathode sold, $ 0.03/lb Cu eq

The following metals premiums assumed to received are;

- Copper = $135.30/t Cu cathode (based off previous CLC sales from open cut operations)
- Zinc = $198.00/t Zn ingot (provided by FQM internal marketing team)

22.3 Cashflow model outcomes

The summary cashflow model for the economic analysis is listed in Table 22-4.

The undiscounted pre-tax cashflow is $1,801M, with a pre-tax NPV reflecting an 8% discount rate equal to $519M. The pre-tax internal rate of return is 17.4%. A pre-tax NPV reflecting a 10% discount rate equal to $356M demonstrates the economic robustness of the Project.

The valuation excludes various subsidy and financing applications in progress, some of which are in the final stages of approval.

The Project is cashflow positive from Year 4 and payback on the initial development capital is in Year 8.

Table 22-4 CLC LOM Cashflow Summary

PHYSICALS	UNITS	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17	Yr18	Yr19	Yr20	Yr21	Yr22	Yr23	Yr24	Yr25
MINING (AFTER MINING DILUTION & RECOVERY)																											
Ore mined direct to plant	Mt	36,447.0	-	-	503.2	1,050.0	1,785.3	1,799.9	1,999.8	1,999.9	1,999.8	2,000.0	2,000.0	1,999.9	1,999.7	1,999.9	1,999.9	1,999.9	2,000.0	1,999.9	1,999.3	2,000.0	1,423.6	813.6	596.6	476.9	-
Ore mined to stockpile	Mt	99.0	23.8	75.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore reclaimed from stockpile	Mt	5,121.9	-	-	981.8	1,122.5	414.7	400.1	200.2	200.0	200.2	200.0	200.0	200.1	200.3	200.1	200.1	200.1	200.0	1.6	-	-	-	-	-	-	-
Waste mined	Mt	3,741.7	729.8	646.0	686.3	383.2	315.7	98.7	72.9	61.9	89.5	78.5	90.4	73.0	77.7	61.9	37.2	18.2	54.0	36.0	42.1	23.9	44.0	12.9	7.8	0.3	-
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)																											
Total direct feed	Mt	41,569.0	-	-	1,485.0	2,172.5	2,200.0	2,200.0	2,200.0	2,199.9	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,200.0	2,001.6	1,999.3	2,000.0	1,423.6	813.6	596.6	476.9	-
	% Cu	1.14%	-	-	1.14%	1.21%	1.56%	1.29%	1.26%	1.31%	1.11%	1.08%	1.03%	1.11%	1.01%	1.06%	1.17%	1.08%	1.05%	1.09%	1.04%	1.01%	1.05%	1.06%	1.03%	1.35%	0.00%
	% Zn	2.15%	-	-	2.21%	2.39%	2.57%	2.41%	2.45%	2.58%	2.55%	2.59%	2.44%	2.12%	1.97%	2.28%	1.89%	1.72%	2.01%	2.05%	1.86%	1.87%	1.23%	1.30%	1.58%	1.40%	0.00%
	% Pb	1.05%	-	-	1.41%	1.39%	1.31%	1.28%	1.18%	1.26%	1.16%	1.44%	1.13%	0.91%	0.79%	0.92%	0.96%	0.83%	0.93%	0.95%	0.82%	0.87%	0.59%	0.64%	0.75%	0.61%	0.00%
	g/t Ag	0.8	-	-	27.0	28.9	36.0	31.1	32.4	26.9	28.3	30.6	29.7	21.9	20.2	22.5	23.7	22.0	22.9	24.8	25.0	26.7	18.0	16.2	16.5	18.6	-
Cu insitu	kt	474.7	-	-	16.91	26.34	34.34	28.42	27.72	28.87	24.36	23.67	22.70	24.34	22.21	23.27	25.76	23.77	23.15	21.74	20.75	20.13	14.98	8.66	6.17	6.42	-
Zn insitu	kt	895.4	-	-	32.83	51.85	56.65	53.03	53.98	56.85	56.09	56.89	53.75	46.66	43.34	50.15	41.58	37.74	44.28	40.99	37.20	37.36	17.52	10.55	9.40	6.70	-
Pb insitu	kt	435.4	-	-	20.93	30.20	28.90	28.18	25.92	27.69	25.47	31.60	24.97	20.01	17.41	20.29	21.17	18.34	20.54	18.96	16.46	17.33	8.41	5.22	4.48	2.92	-
Ag insitu	koz	34,714	-	-	1,290	2,017	2,549	2,202	2,294	1,901	1,999	2,163	2,102	1,550	1,429	1,593	1,676	1,558	1,622	1,594	1,606	1,717	825	423	317	285	-
AVERAGE RECOVERIES																											
Cu	%	85%	-	-	81%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	-
Zn	%	88%	-	-	84%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	-
Pb	%	79%	-	-	64%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	79%	-
Ag	%	52%	-	-	48%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	-
METAL RECOVERED																											
Cu	kt	403.3	-	-	13.7	22.4	29.2	24.2	23.6	24.6	20.7	20.1	19.3	20.7	18.9	19.8	21.9	20.2	19.7	18.5	17.7	17.1	12.8	7.4	5.3	5.5	-
Zn	kt	790.1	-	-	27.6	45.8	50.1	46.9	47.7	50.3	49.6	50.3	47.5	41.2	38.3	44.3	36.8	33.4	39.1	36.2	32.9	33.0	15.5	9.3	8.3	5.9	-
Pb	kt	342.3	-	-	13.3	24.0	22.9	22.4	20.6	22.0	20.2	25.1	19.8	15.9	13.8	16.1	16.8	14.6	16.3	15.0	13.1	13.8	6.7	4.1	3.6	2.3	-
Ag	koz	17,955	-	-	618	1,046	1,322	1,142	1,190	986	1,037	1,122	1,090	804	741	826	869	808	841	827	833	891	428	219	165	148	-
CASHFLOWS	**UNITS**	**Total**	**Yr1**	**Yr2**	**Yr3**	**Yr4**	**Yr5**	**Yr6**	**Yr7**	**Yr8**	**Yr9**	**Yr10**	**Yr11**	**Yr12**	**Yr13**	**Yr14**	**Yr15**	**Yr16**	**Yr17**	**Yr18**	**Yr19**	**Yr20**	**Yr21**	**Yr22**	**Yr23**	**Yr24**	**Yr25**
PAYABILITY																											
Cu	%	100%	0%	0%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	0%
Zn	%	100%	0%	0%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	0%
Pb	%	95%	0%	0%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	0%
Ag	%	95%	0%	0%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	0%
Payable Metal Recovered																											
Cu	kt	403.3	-	-	13.7	22.4	29.2	24.2	23.6	24.6	20.7	20.1	19.3	20.7	18.9	19.8	21.9	20.2	19.7	18.5	17.7	17.1	12.8	7.4	5.3	5.5	-
Zn	kt	790.1	-	-	27.6	45.8	50.1	46.9	47.7	50.3	49.6	50.3	47.5	41.2	38.3	44.3	36.8	33.4	39.1	36.2	32.9	33.0	15.5	9.3	8.3	5.9	-
Pb	kt	325.2	-	-	12.6	22.8	21.8	21.3	19.5	20.9	19.2	23.8	18.8	15.1	13.1	15.3	16.0	13.8	15.5	14.3	12.4	13.1	6.3	3.9	3.4	2.2	-
Ag	koz	17,057.3	-	-	587.2	994.1	1,256.2	1,085.0	1,130.2	936.9	985.1	1,065.9	1,035.9	764.0	704.0	785.1	825.7	767.9	799.3	785.6	791.3	846.3	406.4	208.4	156.4	140.4	-
GROSS REVENUE																											
Cu	$/lb	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80	$ 3.80
Zn	$/lb	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20	$ 1.20
Pb	$/lb	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95
Ag	$/oz	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75	$ 22.75
Revenue after payability																											
Copper revenue	$M	$ 3,378.6	$ -	$ -	$ 114.6	$ 187.8	$ 244.9	$ 202.6	$ 197.6	$ 205.9	$ 173.7	$ 168.8	$ 161.9	$ 173.5	$ 158.4	$ 165.9	$ 183.7	$ 169.5	$ 165.1	$ 155.0	$ 148.0	$ 143.5	$ 106.8	$ 61.8	$ 44.0	$ 45.8	$ -
Zinc revenue	$M	$ 2,090.3	$ -	$ -	$ 73.0	$ 121.3	$ 132.5	$ 124.0	$ 126.3	$ 133.0	$ 131.2	$ 133.1	$ 125.7	$ 109.1	$ 101.4	$ 117.3	$ 97.2	$ 88.3	$ 103.6	$ 95.9	$ 87.0	$ 87.4	$ 41.0	$ 24.7	$ 22.0	$ 15.7	$ -
Lead revenue	$M	$ 681.1	$ -	$ -	$ 26.5	$ 47.7	$ 45.6	$ 44.5	$ 40.9	$ 43.7	$ 40.2	$ 49.9	$ 39.4	$ 31.6	$ 27.5	$ 32.1	$ 33.4	$ 29.0	$ 32.4	$ 29.9	$ 26.0	$ 27.4	$ 13.3	$ 8.3	$ 7.1	$ 4.6	$ -
Silver revenue	$M	$ 388.1	$ -	$ -	$ 13.4	$ 22.6	$ 28.6	$ 24.7	$ 25.7	$ 21.3	$ 22.4	$ 24.3	$ 23.6	$ 17.4	$ 16.0	$ 17.9	$ 18.8	$ 17.5	$ 18.2	$ 17.9	$ 18.0	$ 19.3	$ 9.2	$ 4.7	$ 3.6	$ 3.2	$ -
Total revenue	**$M**	**$ 6,538.1**	**$ -**	**$ -**	**$ 227.4**	**$ 379.4**	**$ 451.6**	**$ 395.8**	**$ 390.6**	**$ 403.9**	**$ 367.5**	**$ 376.0**	**$ 350.6**	**$ 331.6**	**$ 303.3**	**$ 333.1**	**$ 333.1**	**$ 304.2**	**$ 319.3**	**$ 298.7**	**$ 279.0**	**$ 277.5**	**$ 170.3**	**$ 99.4**	**$ 76.6**	**$ 69.2**	**$ -**
CAPITAL COSTS																											
Plant	$M	$ 499.4	$ 92.0	$ 379.0	$ 28.4	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Mine	$M	$ 270.9	$ 52.2	$ 70.0	$ 56.5	$ 61.7	$ 26.2	$ 3.7	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owners and Indirect costs	$M	$ 27.9	$ 7.9	$ 19.6	$ 0.4	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Existing plant refurbishment	$M	$ 22.8	$ -	$ 22.8	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Water facility expansion	$M	$ 20.6	$ 3.8	$ 16.5	$ 0.4	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Other	$M	$ 4.4	$ 0.3	$ 3.9	$ 0.1	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Initial Project CAPEX	**$M**	**$ 846.0**	**$ 156.2**	**$ 511.9**	**$ 85.8**	**$ 61.7**	**$ 26.2**	**$ 3.7**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**
Sustaining capex (fleet replacement)	$M	$ 5.9	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1.8	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ 0.5	$ -	$ -	$ -	$ -	$ -	$ -	$ -
ARO Closure costs	$M	$ 98.1	$ 8.9	$ 10.5	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 32.3	$ 46.3
Total Project CAPEX	**$M**	**$ 950.0**	**$ 165.1**	**$ 522.4**	**$ 85.8**	**$ 61.7**	**$ 26.2**	**$ 3.7**	**$ -**	**$ 1.8**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ 0.5**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ 32.3**	**$ 46.3**
OPERATING COSTS																											
Ore and waste mining	$M	$ 831.1	$ 0.0	$ 0.0	$ 25.3	$ 33.3	$ 42.7	$ 42.8	$ 43.7	$ 45.2	$ 46.5	$ 42.9	$ 44.5	$ 45.4	$ 44.2	$ 43.2	$ 43.0	$ 40.8	$ 44.6	$ 41.6	$ 40.9	$ 41.0	$ 31.3	$ 20.8	$ 15.2	$ 12.2	$ -
Stockpile reclaim and tailings	$M	$ 57.2	$ -	$ -	$ 3.3	$ 4.3	$ 3.2	$ 3.1	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.8	$ 2.2	$ 2.2	$ 2.2	$ 1.6	$ 1.7	$ 1.2	$ 1.0	$ -
Processing (inc. maintenance)	$M	$ 2,391.1	$ 8.4	$ 8.4	$ 97.7	$ 124.4	$ 125.6	$ 124.7	$ 124.0	$ 124.9	$ 123.9	$ 126.0	$ 123.5	$ 121.2	$ 119.9	$ 121.5	$ 121.2	$ 119.8	$ 121.1	$ 114.5	$ 113.2	$ 113.5	$ 91.8	$ 54.1	$ 37.0	$ 31.1	$ -
Water management & treatment	$M	$ 243.3	$ 12.5	$ 12.0	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 10.1	$ 6.6	$ -
General and adminstration	$M	$ 321.6	$ 6.6	$ 6.6	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 14.5	$ 10.9	$ 7.3	$ -
Total operating costs	**$M**	**$ 3,844.3**	**$ 27.4**	**$ 27.0**	**$ 150.9**	**$ 186.7**	**$ 196.1**	**$ 195.3**	**$ 195.1**	**$ 197.6**	**$ 197.8**	**$ 196.4**	**$ 195.4**	**$ 194.0**	**$ 191.5**	**$ 192.2**	**$ 191.6**	**$ 188.0**	**$ 193.1**	**$ 182.9**	**$ 180.9**	**$ 181.4**	**$ 149.3**	**$ 101.2**	**$ 74.4**	**$ 58.1**	**$ -**
METAL COSTS	$M	$ -																									
Metal Premiums	$M	$ -189.9	$ -	$ -	$ -6.6	$ -10.9	$ -12.5	$ -11.3	$ -11.4	$ -11.9	$ -11.4	$ -11.4	$ -10.8	$ -9.9	$ -9.1	$ -10.3	$ -9.2	$ -8.4	$ -9.4	$ -8.7	$ -8.0	$ -8.0	$ -4.3	$ -2.6	$ -1.9	$ -1.9	$ -
TCRCs and Freight	$M	$ 71.4	$ -	$ -	$ 2.6	$ 4.5	$ 4.7	$ 4.4	$ 4.3	$ 4.5	$ 4.3	$ 4.8	$ 4.2	$ 3.6	$ 3.2	$ 3.7	$ 3.4	$ 3.1	$ 3.5	$ 3.2	$ 2.9	$ 2.9	$ 1.4	$ 0.9	$ 0.6	$ 0.7	$ -
Royalites	$M	$ 51.4	$ -	$ -	$ 1.7	$ 2.9	$ 3.7	$ 3.1	$ 3.0	$ 3.1	$ 2.6	$ 2.6	$ 2.5	$ 2.6	$ 2.4	$ 2.5	$ 2.8	$ 2.6	$ 2.5	$ 2.4	$ 2.2	$ 2.2	$ 1.6	$ 0.9	$ 0.7	$ 0.7	$ -
Total Metal Costs	**$M**	**$ -67.2**	**$ -**	**$ -**	**$ -2.2**	**$ -3.6**	**$ -4.1**	**$ -3.8**	**$ -4.1**	**$ -4.3**	**$ -4.4**	**$ -4.1**	**$ -4.2**	**$ -3.7**	**$ -3.5**	**$ -4.1**	**$ -3.0**	**$ -2.8**	**$ -3.4**	**$ -3.1**	**$ -2.9**	**$ -2.8**	**$ -1.2**	**$ -0.8**	**$ -0.6**	**$ -0.6**	**$ -**
MINERAL RESERVE & RESOURCES CASHFLOW (PRE-TAX)																											
Undiscounted	$M	$ 1,800.7	$ -175.5	$ -546.7	$ -39.8	$ 123.3	$ 227.7	$ 205.3	$ 199.9	$ 207.8	$ 176.7	$ 182.5	$ 160.9	$ 142.4	$ 117.1	$ 142.1	$ 144.1	$ 120.8	$ 128.0	$ 119.6	$ 102.5	$ 99.1	$ 29.9	$ -1.0	$ 1.7	$ -20.5	$ -47.2
NPV 8%	$M	$ 518.5																									
NPV 10%	$M	$ 356.1																									
IRR	%	17.4%																									
MINERAL RESERVE & RESOURCES CASHFLOW (POST-TAX)																											
Tax paid	$M	$ 415.1	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 8.4	$ 32.8	$ 41.6	$ 37.9	$ 33.4	$ 27.9	$ 31.5	$ 33.2	$ 28.7	$ 29.1	$ 27.8	$ 24.1	$ 22.7	$ 15.1	$ 9.0	$ 6.3	$ 4.3	$ 1.2
Undiscounted	$M	$ 1,385.7	$ -175.5	$ -546.7	$ -39.8	$ 123.3	$ 227.7	$ 205.3	$ 199.9	$ 199.4	$ 143.9	$ 140.9	$ 123.0	$ 108.9	$ 89.2	$ 110.5	$ 110.9	$ 92.0	$ 98.9	$ 91.7	$ 78.4	$ 76.5	$ 14.8	$ -10.0	$ -4.6	$ -24.9	$ -48.4
NPV 8%	$M	$ 392.0																									
NPV 10%	$M	$ 257.3																									
IRR	%	15.7%																									

22.4 Sensitivity analysis

Economic model sensitivity analysis was completed on the metal prices, recoveries, as well as capital and operating cost estimates. The results indicate that the Project is robust whilst profitability is most sensitive to variations in metal price, grades, recoveries, operating costs, and capital costs in that order.

Figure 22-1 NPV_8 Sensitivity

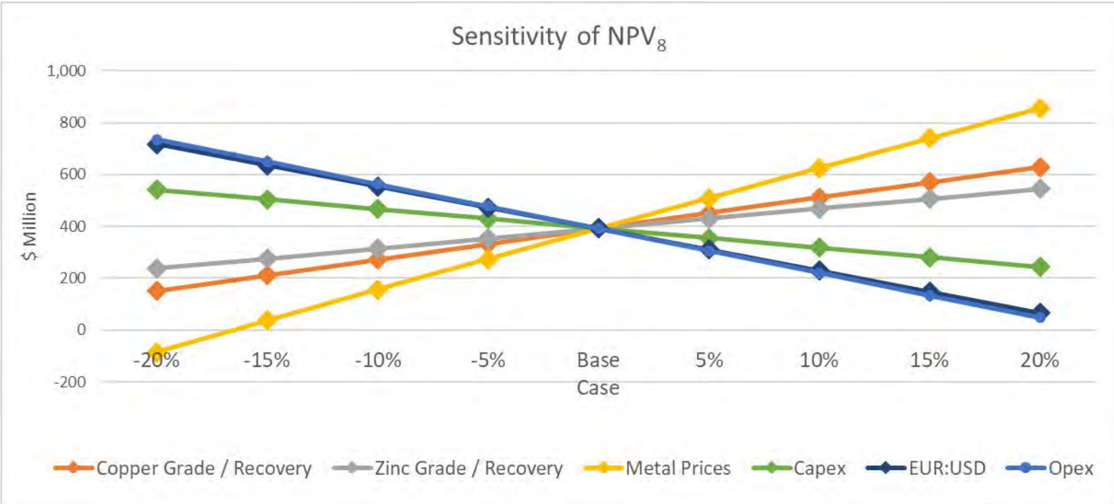


Figure 22-2 NPV_{10} Sensitivity

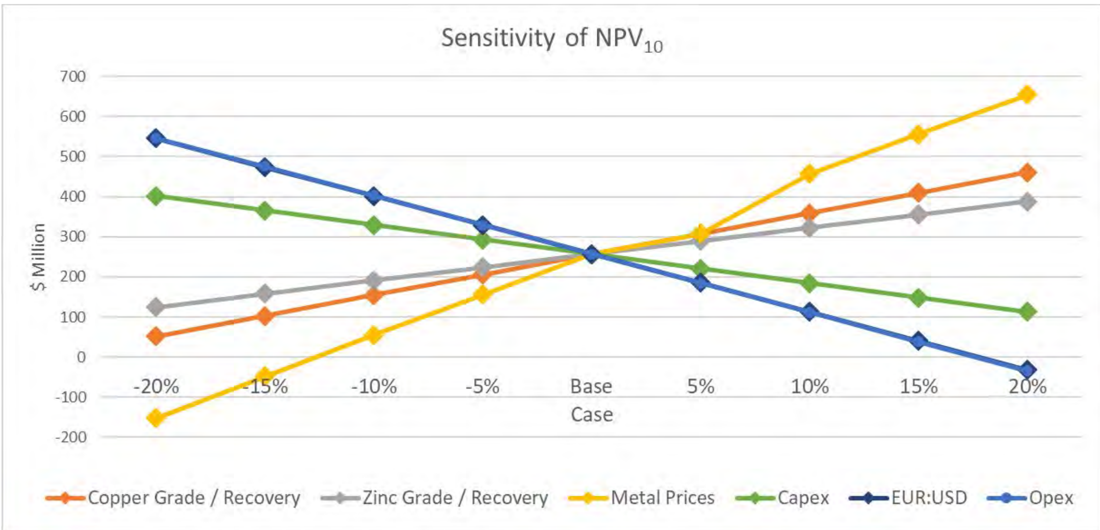


Figure 22-3 Pre-Tax IRR Sensitivity

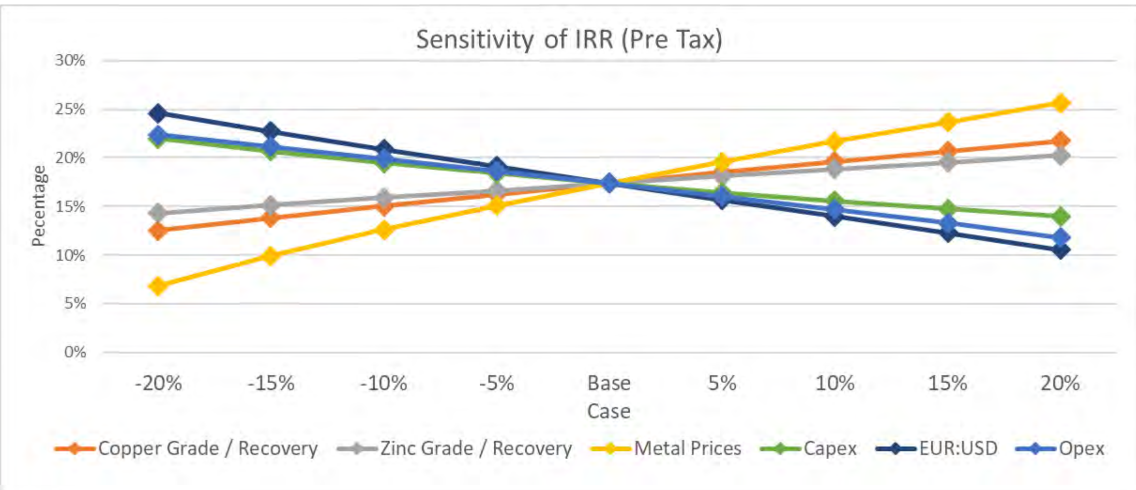
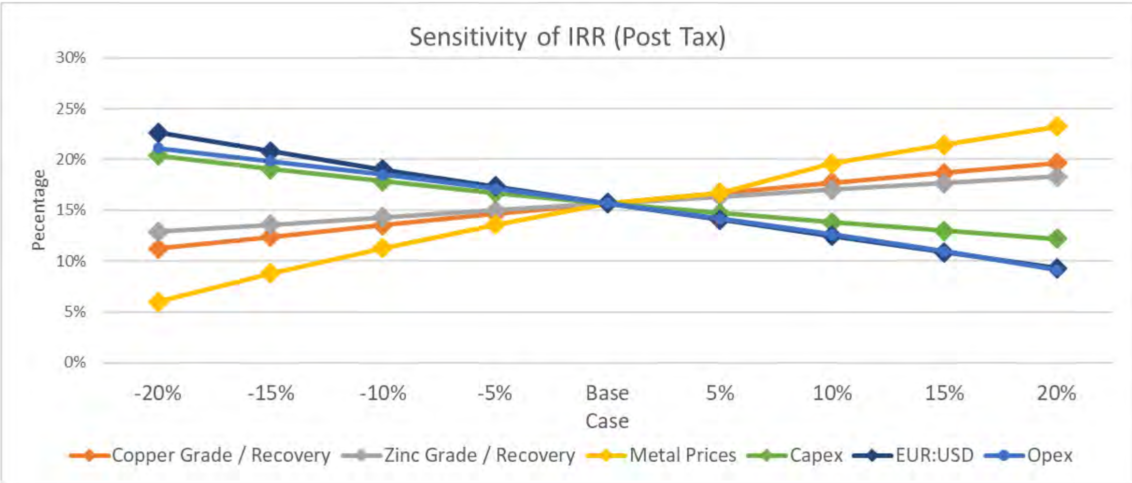


Figure 22-4 Post Tax IRR Sensitivity



ITEM 23 ADJACENT PROPERTIES

Apart from the properties already disclosed in Item 9 of this TR, the QPs have not identified any other adjacent properties that are material to the PPS Mineral Resource estimate.

The Cobre Las Cruces deposit is a blind deposit that does not outcrop, due to the 100 to 150 metres of sandstone and calcareous mudstone (marl) that was deposited on top of the deposit. To date, no other deposits have been found in the immediate area. As at the end of September 2023, there are no mining concessions granted or Resources identified or reported adjacent to the property.

ITEM 24 OTHER RELEVANT DATA AND INFORMATION

The QPs are not aware of any other relevant data or information not already presented in this TR.

ITEM 25 INTERPRETATION AND CONCLUSIONS

25.1 Risk Analysis and Management

A comprehensive identification of Project risks and the workshopping of management strategies will be completed as the Project engineering phase nears completion. The analysis information below is preliminary and relates to observations that are relevant at this current stage of Project engineering. As conventionally, the identified and grouped Project risks would ultimately be rated in terms of likelihood (or probability) of occurrence, and the consequence to the Project.

25.1.1 Mineral Resource Modelling and Estimation

Mineralization volumes

The current extents of mineralised volumes are supported by a reasonable grid of drill data in combination with geophysical information. Previous production experience, during which 5Mt of PPS were mined and stockpiled, is also in support of current 3D modelling of the mineralization.

The main risk identified in relation to mineralization volume is the presence of small lenses of barren-silicified shales within the PMS unit. These lenses are sporadic and intersected only by a few drill holes, suggesting that their size is smaller than the drilling grid. Its impact is of small significance to the global resources and the risk has been assessed as low. To mitigate this, close spaced grid drilling is recommended to be conducted ahead of mining.

PMS modelled volume gets thinner towards the North, in depth and towards its eastern limit, where it pinches out. Mineralization in these areas of Inferred resources could potentially be less continuous than interpreted due to its small vertical thickness and overall geological and structural setting. The potential risk to the Mineral Resources Estimate is low, with no impact to Mineral Reserves. Infill drilling in these areas will improve the 3D modelling of the limits of the mineralization.

Structural model

The Mineral Resource estimate will benefit from continued development of the 3D structural and geological model, improving predicted location and orientation of faults, shear zones, joints, pervasive ductile fabrics and key lithologies. Faults and key structures may have an impact on geology and mineralization continuity, mine stope designs and mining dilution. The impact of this risk to the Mineral Resource Estimates is considered low to moderate and within the context of the Mineral Resources Estimate classification.

Sulphide and gangue mineralogy

Mineralogical and petrographic knowledge acquired to date is reasonable, however, it is dependent on drill core samples, their location and drilling grid. Although overall mineralogical assemblages are typical of the ones found on most deposits along the Iberian Pyrite Belt, there is a potential risk associated with local variations of these assemblages which could impact the metallurgical performance of the materials. Additional drilling targeting specific areas and in alignment with metallurgical requirements, is recommended as the Project advances.

Dry bulk density

Dry bulk density shows a significant range of values from non-mineralised to semi-massive sulphides and to massive sulphides, as the presence of sulphides increases. Dry bulk density estimate is included in the Mineral Resources within each mineralization domain and is capturing local changes in a reasonable way.

The most typical bias within the industry on dry bulk density values is related to sample selection towards the more competent parts of the drill core. This type of bias is considered to be at a low level at CLC, as most of the core within the PPS domains is competent, with high core recovery values and excellent RQD index. Nonetheless, any local variance of dry bulk density values not captured by the drilling grid and sample selection could have a potential risk on showing lower or higher density values, which could translate into variances on tonnes and contained metal. The potential risk to the Mineral Resources is assessed as low and is within the context of the Mineral Resource's classification.

Additional dry bulk density samples and measures are recommended to be taken from future infill drilling programs and from existing core from areas showing significant variability.

Stockwork mineralization

The mineralization within the Stockwork domain is present as veins and disseminated forms. The frequency, size and orientation of the ore bearing veins varies, as does grade and contained metal.

Drill spacing has the risk of not providing enough support to the estimates in some areas and mineralization could have higher variances than predicted. This risk is captured in the Mineral Resource classification and is considered to be low to moderate within Measured and Indicated, and moderate to high in Inferred materials.

The prediction of size, orientation and grades of these swarms of veins is a challenge and it is recommended to conduct Infill diamond drilling in Indicated and Inferred areas in addition to close spaced Ore Control drilling in advance of mining.

25.1.2 Mine planning and Mineral Reserve estimate

Hydrogeology and mine dewatering

The hydrological model has been developed using data obtained prior to and during the 12 years of open pit mining. The model was calibrated with both regional groundwater information and piezometric information obtained from existing groundwater control boreholes. In addition the model was also calibrated with the water balance in the open pit and the existing underground exploration ramp. However, there remains risk that water inflows encountered during development of the mine exceed modelled outputs and may incur delays and extra costs. The underground dewatering system is designed with 20% spare capacity and 100% pump redundancy at each pump station. The likelihood is possible whilst the risk has been assessed as low to medium.

Mine geotechnical engineering

Geotechnical inputs are based on data obtained during open pit mining as well as extra data from the exploration decline and geotechnical drilling done for the underground project. The main risk identified geotechnically is how the crown pillar behaves during the partial recovery phase. This is a key part of the Project as it contains almost 30% of the metal contained in the Reserves. Further detailed geotechnical studies are required during the detailed engineering phase in order to mitigate this risk. However, given that the proposed drift and fill mining method includes paste back fill, the likelihood of this risk is considered low.

Faults and dilution

Faults have been mapped and incorporated into the stope design process however it is considered that undetected faults may impact stope hangingwall stability. Longitudinal stopes may have a greater risk of abandonment if the dilution becomes excessive. A change to transverse stope design will give some control in relation to managing dilution under these circumstances.

Ventilation

External review confirmed that there is a suitable distance between the proposed exhaust and intake sites (>1km separation). A Ventsim model was developed showing airflow quantities (total 270m^3/s flow rate) and velocity rates are in alignment with the peak equipment build up. At this point in time, the only risk related to ventilation that remains unaddressed is in relation to whether the system can be upgraded for future mine extensions.

Development schedule

The focus of the development schedule is to access the stoping areas and install the Railveyor. Up to 2,000 metres of development per quarter is scheduled over a three-year period leading up to plant commissioning. Delays in the development schedule could cause reduced production rates and increased ore haulage costs in the early years of the schedule. Detailed engineering designs and construction scheduling will be carried out to mitigate a risk which has been assessed as low impact.

Production schedule

Conventional underground mining methods have been selected, making use of equipment suited to the scale of production. As noted above, a considerable pre-production mine development is required to access the initial plant feed. To offset this initial hurdle, a sequence of stope development and production schedule have been devised which seek to mine higher grade ores from Block 2 first. In the initial years, delays, or lower than planned production rates, from the underground mining can be offset by reclaim from the surface stockpile to maintain feed to the process plant. Hence, the potential risk to the Project is considered to be low.

Mine operating costs

The estimated unit mining cost estimates have been built-up from first principles and have been checked against contractor quotes for stoping and development. This work has enabled the calculation of primary mining equipment requirements over the life of the mine. Whilst there is a risk of operating costs being underestimated, the risk and consequences have been assessed to be low.

25.1.3 Metallurgy

Metallurgical Testwork

The testwork performed to date has been undertaken in three phases commencing with conceptual, moving through laboratory and finally into a bespoke pilot plant. The aim of these phases was to confirm and verify the technology and process route. This work was undertaken by a combination of internal and external laboratories.

In the project phases I and II, laboratory and small-scale pilot plant tests were run to confirm whether the PMS was amenable to the proposed technologies. Tests were successful and resulted in the development of a provisional flowsheet. This led into Phase III of the Project to include in the scope a 1,000kg/hr pilot plant with the target to validate the confirmed technologies under continuous pilot scale trials.

The pilot plant campaign successfully demonstrated both the primary (copper-zinc SICAL) and subsequent secondary (lead-silver brine) leaching technologies as viable processing routes for the treatment of primary CLC sulphide ores. The pilot work also concluded that silver could be successfully recovered from secondary leach solution by cementation. Finally, the use of cementation for lead recovery was subsequently piloted and proved successful. These enhancements were subsequently included in the flowsheet for the PMR.

It is considered that sufficient test work has been completed to justify proof of process design concept, capital & operating cost estimates with stated contingency levels and to advance into detailed design.

Sampling and testing representivity

The early phase testwork was undertaken with a range of samples, mostly composed of drill core used for PMR Mine modelling, with a number of the approximately 114 drill cores selected. These cover 84% of the primary deposit defined. Additionally two composite samples (laboratory and Pilot Plant) were prepared out of the primary ore stockpiled on surface during the secondary copper ore mining.

To run the pilot plant a 5,000 tonnes sample was prepared by the geological team. Sample was taken from the PMS stockpile, homogenized and characterized. Lead and zinc were upgraded by blending with primary sulphides from the open pit. Assays are not exactly the same as those for the PMS samples prepared from drill cores (PMS 4,5 and 6 represent samples used for phases 1 and 2).

Table 25-1 Sampling and testing representivity

		Head grades				Bulk concentrate grades				Sulphate leaching%		chloride leaching%	
	% representativity of deposit	Cu%	Pb%	Zn%	Agppm	Cu%	Pb%	Zn%	Agppm	Cu	Zn	Pb	Ag
PMS4	67	1.05	1.32	2.95	30	3.49	3.45	9.83	86	97.5	94.5	97.60	98.00
PMS5	84	1.16	0.75	1.95	22	4.31	2.58	7.45	67	97.5	95.9	96.46	98.53
PMS6	84	0.82	1.00	2.16	23	2.52	2.41	6.33	53	95.9	97.7	97.56	98.02
Pilot plant	84% from stockpile	1.26	2.7	3.44	68	2.14	4.59	6.26	107	93	94	96.00	97.00
Base case		**1.14**	**1.1**	**2.25**	**27.8**	**2.92**	**2.52**	**6.00**	**58.3**	**94.0**	**94.1**	**98.0**	**98.0**

Whilst the samples prepared for the testwork programs were not entirely representative of the underground orebody, they do demonstrate the robustness of the PMR technology in responding to variations in feed mineralization and grades.

25.1.4 Processing

The PMR feed is provided from a single underground orebody, with minor blending from existing surface stockpiles. Whilst the stockpiled material may exhibit minor oxidation, the technology selected is unlikely to be affected.

Variability of plant feed

The PMR process plant is designed to process a feed "blend" within an envelope of assays which ensures the capacities of the key areas are not exceeded. While significant "unused" copper capacity exists due to only partial use of the existing facility, zinc, lead and/or silver and limited by design to mine plan grades. Within these grade constraints the process has been proven to possess considerable flexibility.

Equipment sizing

The PMR circuit is designed based on in house and external engineering understanding of the technology. The Company has significant expertise in crushing, grinding, flotation, filtration and copper leach/SX/EW to have confidence in the design. Zinc design is provided by well-known package suppliers with considerable experience. Ag and Pb cementation circuits are bespoke with both having been proven at pilot plant scale.

Design uncertainty

The silver and lead cementation circuits are bespoke and as such may present commissioning challenges. To mitigate this challenge, full-scale equipment is proposed to be piloted prior to the main plant commissioning.

25.1.5 Tailings Storage

The continued use of In-pit tailings disposal to dispose, store and encapsulate plant tailings in the In-pit tailings facility (IPTF) is considered a significant advantage of the Project when compared with previous surface dry stacking or conventional tailings disposal methods.

The method has been successfully operated and proven from 2016 to 2023 to encapsulate waste rock and the tailings derived from the reprocessing project. Continuous geotechnical and hydrogeological monitoring during these years confirmed that the disposed material is properly encapsulated and validates the in-pit disposal selected for the PMR Project.

25.1.6 Water Management

The expansion and continued successful operation of the site water management system to include new underground water sources is critical to the continued mining operations at CLC. The Drainage and Reinjection System (DRS) to maintain aquifer levels and water quality such that they are unaffected as a result of mining operations, is a strict legal permitting requirement and a community priority. This is evidenced by CLC receiving three Notices of Violation (NOV), served in 2014, 2017 and 2019, regarding loss of groundwater and insufficient compensation by the DRS.

After several upgrade projects, the reliability of the neutralization plant increased markedly and currently the plant produces a very stable high-quality effluent with very few scattered and short periods of non-compliance. The limits for the discharge to the aquifers and Guadalquivir River are being consistently met for the current open pit operations. This expertise and experience gained by CLC in operating the water management system during the open it operations can be considered as a strength and a basis for developing the PMR Project.

25.1.7 Infrastructure

CLC is an existing open pit mining operation and processing operation which has been established for more than fifteen years. The associated infrastructure as required by these operations is still in place and includes sealed roads, power lines and substations, process plant, site offices, workshops, tailings dams, and waste storage facilities.

In general, this infrastructure remains in good working condition and is suitable for reuse to support the PMR Project. Significant additional processing facilities and modifications to existing processing infrastructure is required and is discussed in more detail above.

Relatively modest additional surface infrastructure is required for the mining operations including ventilation fans, site offices, change rooms, light workshops, consumable storages, electrical power extensions and UG utilities distribution systems. This infrastructure is standard to the local mining industry and be reliably designed, manufactured and constructed by local contractors with some specialist equipment sourced from within the wider EU.

The two most significant infrastructure additions relate to power supply and underground materials handling via the Railveyor.

25.1.8 Power supply costs

Electrical power costs contribute ~30% of plant operating costs with large fluctuations in local power unit costs in recent years attributable to European gas prices. The most critical new infrastructure required is the construction of new photovoltaic solar farm to provide up to 30% of electrical demand. To minimise execution risk the plant will be constructed and operated by an external third party similar to other recently completed project in the area.

25.1.9 Environmental studies and permitting

All key permits for the PMR Project are in place including the Unified Environmental Authorisation, Integrated Pollution Prevention, and Control Authorization, PMR Water Concession, DRS Authorization, and the modified mining concession. The experience and information of environmental vectors, permits, and community aspects reflect CLC's previous operation and should be considered as a strength and a basis for developing the PMR Project within the requirements of these permits.

A remaining permit required to commence construction is the Construction Permit. A procedural process submitted to the local authorities is expected to be received with six months of submission. Whilst procedural in nature, the process and site layout design are required to be developed to an advanced stage and 'locked'. Design changes to these aspects can result in additional engineering costs and time delays in the granting of the Project construction permit.

25.1.10 Cost estimation

The capital cost estimate for the processing plant and related site infrastructure has been comprehensively assessed by Lycopodium including benchmarking to their internal database and with an appropriate contingency assessed accordingly. Contingency factors of between 15% and 20% have been applied deriving an overall contingency of 16.9%.

The capital cost estimate for the underground mining and other surface infrastructure has been compiled by the CLC team utilising budget quotes from reputable suppliers. A contingency factor of 10% has been applied after benchmarking overall Project contingency against similar recent projects at the PFS and FS stage.

These preliminary capital cost estimates are considered to be suitable for adoption at this stage of Project engineering and evaluation, and will be improved upon as the detailed engineering phase proceeds.

Operating cost unit rates have been calculated utilising a range of sources including existing plant, infrastructure, utilities and administration costs at CLC in addition to current market rates for consumables. Mining unit rates drew on a 2022 CostMine database, whilst consumable consumption rates in the plant were derived from existing data, pilot plant testing and other metallurgical testwork results. For the mine operations, estimates were completed by mining consultants with extensive knowledge of similar operations. It is generally considered that the North American basis of the CostMine database provides a conservative estimate for the IPB at this stage of project development.

25.2 Conclusions

25.2.1 Mineral Resource Estimate

The CLC PPS Mineral Resource estimate is detailed in the previous 2022 CLC Technical. The Mineral Resource statement has been updated using a lower Cu_{eq} cutoff grade of 0.8% as supported by this Technical Report's Mineral Reserve studies. Consequently, Measured and Indicated Mineral Resources have increased from 36.24 million tonnes to 41.38 million tonnes with the Cu_{eq} grade decreasing from 2.51% to 2.29%. The Mineral Resource classification has considered the relevant factors for reasonable prospects of eventual economic extraction.

The geologically modelled PPS mineralization and spatially related copper, zinc, lead, and silver grade estimates were supported by high-quality diamond-drilled core samples and detailed 3D geological modelling of the respective domains of mineralization. Estimates were completed using ordinary kriging with post processing of parent cells using localised uniform conditioning to ensure that block estimates were relevant to the scale of planned underground mining.

25.2.2 Mineral Reserve Estimate

A conventional approach has been adopted in the process of optimising a mine planning model derived from the Mineral Resource model, followed by detailed stope and development design, production scheduling, and Mineral Reserve estimation.

The Mineral Reserve estimate has been developed using a Net Value calculation including expected metallurgical recoveries obtained from the pilot process testing and modifying factors and reflects an achievable mining plan and production schedule for the Project, at this stage of evaluation.

Groundwater inflow has been identified as a mining risk and has been taken into account during the mine design process. The new dewatering strategy based on a combined approach from surface and underground drains is focused on depressing the water level in advance of mine development. A predictive tool has been developed to estimate the type and quantity of water inflow.

A Railveyor hauling system was considered appropriate for the Las Cruces underground project in late 2019. The system has developed since then and is now a proven technology with production rates achieved well over CLC targets.

Taking into account all the comments above, the technical risk related to mining is considered to be low to medium.

25.2.3 Metallurgical & Processing

The Polymetallurgical Refinery Project was initiated by CLC in order to process its primary sulphide mineralization and so extend the life of mine following depletion of the secondary copper ore in 2021. The Project commenced in 2014 by defining the actual technology and Mineral Resources available. The optimum technology route was established after a research period carried out by CLC in conjunction with external organisations, universities and institutions. The proposed process technology was validated in a pilot plant constructed and operated by CLC, treating 5,000 tonnes of primary ore during 2016 to 2017. The “dynamic cementation” process for silver and lead was piloted and validated during 2017-2018. Pilot plant results were used as the basis for the plant conceptual engineering.

Preferred locations have been identified and conceptual designs produced for the Project infrastructure including new hydrometallurgical plant, paste plant, overland piping, expanded water management and treatment equipment and new electrical equipment and power lines. The concepts for these are considered to be suitable for the Project at this stage of the engineering phase.

25.2.4 Cost Estimation and Economic Outcomes

Mine and process operating costs, plus general and administration costs (G&A) have been estimated from first principles utilising industry databases, supplier quotes and historical operating costs. Metal costs (i.e. including transport and refining charges (TCRCs)) have been advised by the Company’s own metals marketing group. The process plant and related infrastructure capital costs have been estimated by a globally experience metallurgical processing engineering company primarily by the means of consultant and / or vendor estimates. The order of accuracy of the capital cost estimates reflects the adoption of contingency factors of up to 13%.

ITEM 26 RECOMMENDATIONS

26.1 Geology and Mineral Resource Estimation

The QP, Mr. Carmelo Gomez Dominguez recommends the following for future Mineral Resource estimates:

- Continue to review and update the 3D geology and structural models.
- Complete an infill diamond drill program with two objectives:
  - upgrade Inferred Mineral Resources inside the current underground mining design to potentially increase the Mineral Reserve inventory, and
  - increase Mineral Resources in exploration/extension areas.
- Improve on the host rock model of the PPS (massive) to minimize future dilution and losses.
- Integrate hydrogeological and geotechnical data into the geological model.
- Work on improving the geometallurgical model for improved mine planning definition.
- PPS stockpile volumes and position should be confirmed with updated surveys.
- PPS stockpiles were built from 2011 and may be subject to a degree of oxidation. Stockpile grades and tonnes may benefit from review and an estimate update.

Some of this work is being done inhouse with the remainder outsourced to contractors and consultants. The total cost of this work is estimated to be $2.1M .

26.2 Mining and Mineral Reserves

Mineral Reserve studies have been finalized up to a feasibility stage. Studies include consideration of technical evaluations associated with mine designs, production schedules and the initial process plant designs and throughput profiles. The QP recommends that: -

- Drilling of additional holes to collect Hydrogeological and Geotechnical data should be undertaken prior to commencement of mining. The data collected should be used to confirm and if necessary, modify design parameters in preparation for engineering design. The drilling would be conducted by drilling contractors with data analysis by hydrological and geotechnical consultants at an estimated cost of $200,000.
- Once mining commences, hydrological and geotechnical reviews should be completed annually to confirm the mine design parameters, update 3D hydrological and geotechnical models, audit the installation of ground support in the active working areas of the mines, and assess the dewatering performance. This work would be conducted by hydrological and geotechnical consultants at an estimated cost of $120,000 per year.
- Develop specific drilling programs to precisely locate ore boundaries and assess ground conditions, for example at the Block 1 (La Manta) hangingwall. The cost of this work is estimated at $360,000 per year.
- Re-estimate Mineral Reserves following updates to Mineral Resource estimates and/or material changes to cost and financial inputs. This work would be completed by CLC personnel at no additional cost.

Stope shapes and production rates should be reviewed and additional life of mine schedule optimization be carried out. This work would be completed by CLC personnel at no additional cost

26.3 Mineral Processing Capital Cost

While budget pricing has been obtained for the majority of the large equipment items, the capital cost estimate for the ECS has also been compiled using pricing information for a number of large vendor packages. In general, there needs to be a greater understanding of the defined scope, battery limits, and service / utility requirements for each of these packages to ensure that all capital costs have been captured. These vendor packages include;

- Zinc Solvent Extraction
- Zinc Electrowinning
- Zinc Melting and Casting
- Paste Plant

An allowance for tie-ins to plant and equipment within the existing plant has been allocated in the capital cost estimate based on a preliminary assessment by CLC. It is recommended that a comprehensive evaluation be performed during the next project phase to ensure that all costs have been captured and to confirm that the proposed location and installation of new plant and equipment within the existing plant site is indeed practical.

The cost of this work is estimated at $1M.

ITEM 27 REFERENCES

1. Pincock, Allen & Holt; May 2005: *Independent Technical Audit of the Las Cruces Copper Project*, Southern Spain; internal consultant's report.
2. Ore Reserves Engineering; June 2012; *Technical Report for the Las Cruces Copper Mine. Andalusia, Spain*.
3. Inmet Mining; 2005 to 2009 *Annual Reports and AIFs*.
4. FQM; 2014, 2015 and 2021 *Annual Reports and AIFs*
5. *Geology and lithogeochemistry of the unique Las Cruces VMS deposit, Iberian Pyrite Belt* (2007). Carmen Conde et al.
6. *Ore Geology Reviews* (2015), The Las Cruces deposit, Iberian Pyrite Belt, Sapin. Lola Yesares et al
7. *Mineralium Deposita* (2017), *The high-grade Las Cruces copper deposit, Spain: a product of secondary enrichment in an evolving basin*. Fernando Tornos et al.
8. *The massive sulphide event in the Iberian Pyrite Belt: confirmatory evidence from the Sotiel-Coronada Mine (2006)*. F. Gozález et al. Geological Magazine, vol. 143. DOI 10.1017/S0016756806002573
9. *The Iberian Pyrite Belt.* In *Database on Global VMS Districts* (2000). Tornos F et al
10. *Estudio Geotécnico para el Proyecto de Explotación Subterráneo del Mineral Primario Polimetálico en la Mina de Las Cruces* (Gerena, Sevilla). Report No. 1623. Subterra Ingenieria, January 2018.
11. *Estudio de Alternativas de Accesos a la Mina Subterránea Las Cruces. Análisis de la Factibilidad de los túneles en margas*. Report No. 2145_01. Subterra Ingenieria, March 2022.
12. *Hydrogeological Model Estimation of PMR Underground Mine Drainage in Las Cruces. N-REDUX Model*. Prepared by Ayterra, reviewed by KLM Ground Water Consultants, March 2020.
13. *Modelo Hidrogeologico N-REDUX 2022. Bases de diseño y resultados obtenidos de drenaje a través del SDR y UG Mine*. CLC – Ayterra, September 2022.
14. *Estudio de Sensibilidad de Reservas*. Memorandum. IGAN, July 2023.
15. *Actualización de Reservas y Vida de Mina*. IGAN, September 2023.
16. *Actualización del LOM considerando Inferidos*. IGAN, November 2023.
17. *Las Cruces Underground Mine Feasibility Study*. CLC, March 2020.
18. *Actualización del Proyecto Minero Subterráneo (Nivel viabilidad)*. IGAN, 2018.
19. *Cobre Las Cruces. Preliminary Economic Assessment Underground Ore Handling System*. Report No. 191-11236-00. WSP, December 2019.
20. *Railveyor Engineering and Design Services Report*. First Quantum Minerals – Cobre Las Cruces. RVTG, April 2023.
21. *Bombeo Desagüe CLC*. IGAN, November 2022.
22. *Proyecto PMR. Estudio de Alternativas de Ventilación Principal*. IGAN, January 2022.
23. *Proyecto Basico Planta de Pasta*. Golder Associates, Sept. 2017. Report No. 1772564/1152.
24. *Laboratory Report for Dewatering Test and Long Term Strength Test on Las Cruces Mine Tailings*. Golder Associates, February, 2019.
25. *Proyecto Constructivo de la Instalación de Encapsulado de Estériles no Inertes en Corta*. Subterra, May 2018.
26. C*obre Las Cruces. Evaluación de Alternativas de Deposición de Estériles de Flotación*. Report No. 1779609/1138. Golder Associates, September 2017.
27. *FQM Las Cruces Underground Mining High Level Review* (May 2023), Snowden Optiro
28. *Cobre Las Cruces. Preliminary Economic Assessment Underground Ore Handling System. Report No. 191-11236-00. WSP, December 2019.*

29. *Railveyor Engineering and Design Services Report. First Quantum Minerals – Cobre Las Cruces. RVTG, April 2023.*
30. *Cobre Las Cruces Polymetallic Refinery (PMR) Project 2023 Engineering Study*, First Quantum Minerals, November 2023
31. *Cobre Las Cruces: Polymetallic Primary Sulphide Mineral Resources, Andalusia, Spain, NI 43-101 Technical Report, January 2022*, First Quantum Minerals, January 2022.
32. *Cobre Las Cruces Operation Andalucía, Spain*, *NI 43-101 Technical Report, June 2015,* First Quantum Minerals, June 2015.

ITEM 28 Certificates

Anthony Robert Cameron
Cameron Mining Consulting Ltd
20/F Central Tower, 28 Queen's Road, Central, Hong Kong
Tel +86 13521513186;
tony@cameronmining.com

I, Anthony Robert Cameron, do hereby certify that:

1. I am a Consultant Mining Engineer employed by Cameron Mining Consulting Ltd.
2. This certificate applies to the technical report entitled *"Cobre Las Cruces: Polymetallic Primary Sulphide Project"* dated effective 30th September 2023 (the "Technical Report").
3. I am a professional mining engineer having graduated with an undergraduate degree of Bachelor of Engineering (Mining) from the University of Queensland in 1988. In addition, I have obtained a First Class Mine Manager's Certificate (No. 509) in Western Australia, a Graduate Diploma in Business from Curtin University (Western Australia) in 2000, and a Masters of Commercial Law from Melbourne University in 2004.
4. I am a Fellow of the Australasian Institute of Mining and Metallurgy.
5. I have worked as a mining engineer for a period in excess of twenty-five years since my graduation from university. Over the last fifteen years I have worked as a consulting mining engineer on mine planning and evaluations for base metals operations and development projects worldwide.
6. I have read the definition of "*qualified person*" as set out in *National Instrument 43-101 – Standards of Disclosure for Mineral Projects* ("*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 am a "qualified person" for the purposes of *NI 43-101*.
7. I most recently personally inspected the Cobre Las Cruces property in June 2023 for 4 days.
8. I am responsible for the preparation of those portions of the *Technical Report* relating to Mineral Reserve estimation and Mining, namely Items 15 and 16, respectively. I am also responsible for the preparation of those items of the Technical Report not covered specifically by the other qualified persons.
9. I am independent (as defined by Section 1.5 of *NI 43-101*) of First Quantum Minerals Ltd.
10. I have not had prior involvement with the property that is the subject of the *Technical Report*.
11. I have read *NI 43-101* and *Form 43-101F1* and the *Technical Report* has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the *Technical Report* contains all scientific and technical information that is required to be disclosed and to make the *Technical Report* not misleading.

Signed and dated this 20th day of February 2024 at West Perth, Western Australia, Australia.

"Anthony Robert Cameron"

____

Anthony Robert Cameron

Robert Stone
First Quantum Minerals Ltd
18-32 Parliament Place, West Perth WA, 6005, Australia
Tel +61 8 9346 0100; rob.stone@fqml.com

I, Robert Stone, do hereby certify that:

1. I am Group Consulting Metallurgist employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled *"Cobre Las Cruces: Polymetallic Primary Sulphide Project"* dated effective 30th September 2023 (the "*Technical Report*").
3. I am a professional process engineer having graduated with an undergraduate degree of Bachelor of Science (Honours) from the Camborne School of Mines in 1984.
4. I am a Member of the Institute of Materials, Minerals and Mining (UK). I have been a Chartered Engineer through the Institute of Materials, Minerals and Mining since 1991.
5. I have worked as process engineer and metallurgist for a period in excess of thirty years since my graduation from university. For the last fifteen years I have been in the employ of First Quantum Minerals Ltd in both technical and managerial roles. Of these, seven years were as a manager of process plants producing copper in concentrate, copper as electrowon cathode, gold concentrate and cobalt metal by RLE. The remaining eight years were in a technical role as Consulting Process Metallurgist responsible for development of First Quantum Minerals Ltd projects worldwide including copper/cobalt in Central Africa, nickel in Australia and copper/molybdenum in Panama.
6. I have read the definition of "*qualified person*" as set out in *National Instrument 43-101 – Standards of Disclosure for Mineral Projects* ("*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 am a "*qualified person*" for the purposes of *NI 43-101*.
7. I most recently personally inspected the Cobre Las Cruces property described in the *Technical Report* in November 2022 for 3days.
8. I am responsible for the preparation of those portions of the *Technical Report* relating to mineral processing/metallurgical testing and recovery methods, namely Items 13 and 17.
9. I am not independent (as defined by Section 1.5 of *NI 43-101*) of First Quantum Minerals Ltd.
10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been in project planning and the preparation of engineering studies, commencing in 2013.
11. I have read *NI 43-101* and *Form 43-101F1* and the *Technical Report* has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the *Technical Report* contains all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Signed and dated this 20th day of February 2024 at West Perth, Western Australia, Australia.

"Robert Stone"

____

Robert Stone

Carmelo Gomez Dominguez
Group Principal Geologist, Mine and Resources
FQM (Australia) Pty Ltd.
18-32 Parliament Place, West Perth WA, 6005, Australia
Tel +61 893460117 ; Carmelo.Gomez@fqml.com

I, Carmelo Gomez Dominguez, do hereby certify that:

1. I am a Principal Geologist, Group Mine and Resources employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled *"Cobre Las Cruces: Polymetallic Primary Sulphide Project"* dated effective 30th September 2023 (the "*Technical Report*").
3. I am a professional geologist having graduated with a Bachelor of Science degree with Honours (2003) in Geology from Huelva University, Spain.
4. I am a Fellow of the European Federation of Geologists.
5. I have worked as a geologist for a total of 20 years since my graduation from university. I have over 14 years' experience in production geology. During the last ten years I have consulted to and held senior technical mineral resource positions in copper mining companies operating in Europe, Central Africa and worldwide.
6. I have read the definition of "*qualified person*" as set out in *National Instrument 43-101 - Standards of Disclosure for Mineral Projects* ("*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 am a "*qualified person*" for the purposes of *NI 43-101*.
7. I have worked at Cobre Las Cruces for 9 years, and most recently personally inspected the Cobre Las Cruces property described in the Technical Report in September and October 2023 for 4 days.
8. I am responsible for the preparation of those portions of the *Technical Report* relating to geology of the deposit and mineral resource estimation, namely Items 7-12 and 14.
9. I am not independent (as defined by Section 1.5 of *NI 43-101*) of First Quantum Minerals Ltd.
10. I have had prior involvement with the property that is the subject of the *Technical Report*. The nature of my prior involvement has been in the assurance of sampling QAQC, definition and optimization of estimation methods and the development of geology and mineralization models.
11. I have read NI 43-101 and Form 43-101F1 and the *Technical Report* has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Signed and dated this 20th day of February 2024 at West Perth, Western Australia, Australia.

"Carmelo Gomez Dominguez"

Carmelo Gomez Dominguez