Çayeli Operations

Çayeli Bakir, Rize Province, Türkiye

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Item 1 SUMMARY

This Technical Report on the Çayeli Operations (the Property or Operations) has been prepared by Qualified Persons (QPs) Richard Sulway, Michael Lawlor and Andy Briggs of First Quantum Minerals Ltd (FQM, the issuer or the Company). This report focusses on Mineral Resource and Mineral Reserve updates, specifically in relation to a newly defined copper and zinc deposit adjacent to the current mining operations.

It supersedes a Technical Report prepared for the Operation's former owners, issued in March 2006 and titled Technical Report on Mineral Resource and Mineral Reserve Estimates, Çayeli Mine, Republic of Türkiye (Roscoe Postle and Associates Inc (RPA), March 2006).

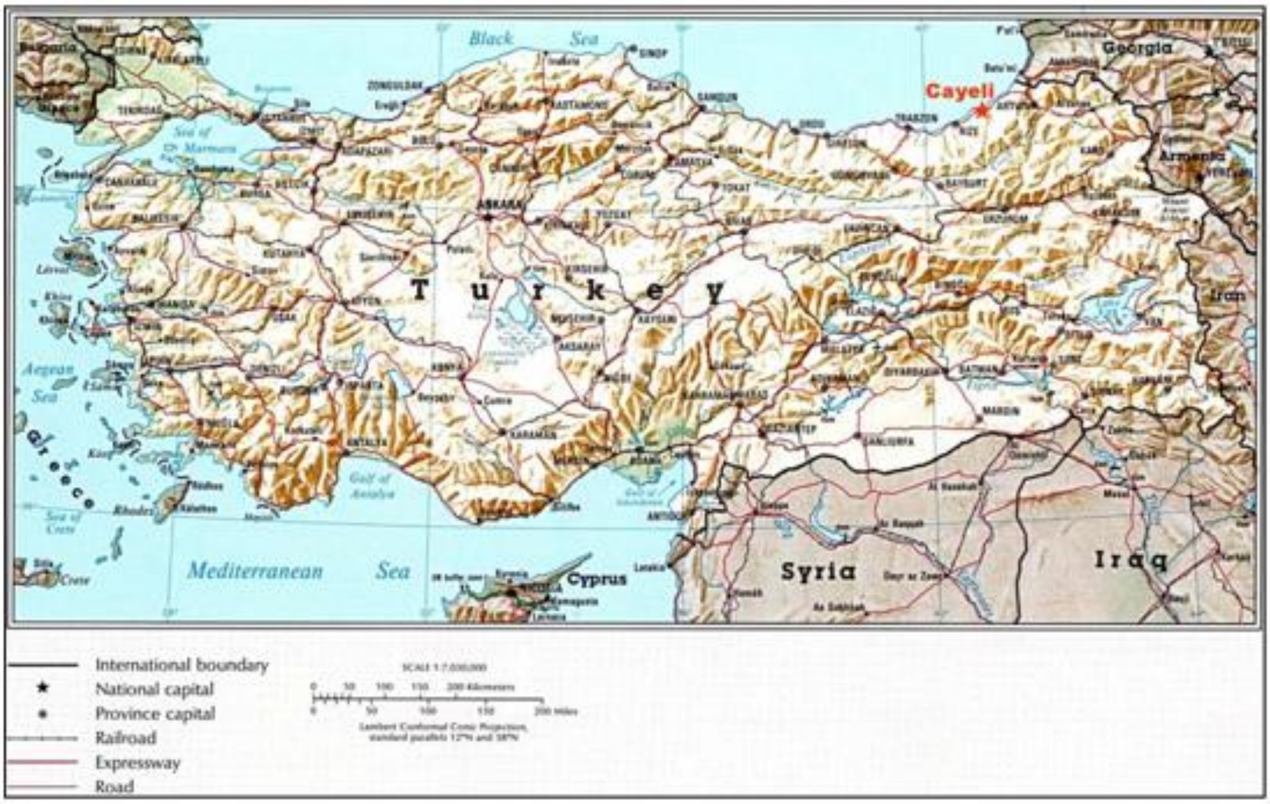
The effective date for the updated Çayeli Mineral Resource and Mineral Reserve estimates is 30th April 2025.

1.1 Property ownership, location and overview

Çayeli Bakir İşletmeleri A.Ş. (ÇBI) is a wholly owned subsidiary company of FQM. *Eti Maden İşletmeleri Genel Müdürlüğü* (*Eti*), a company wholly owned by the Government of Türkiye, holds the operating licence for the mine and has leased it to ÇBI.

The ÇBI Operations, comprising an underground copper and zinc mine, together with associated processing facilities, is located 8 km south of the town of Çayeli and approximately 18 km from the coastal city of Rize, which is on the Black Sea coast of north-eastern Türkiye.

Figure 1.1 Location of the Çayeli Operations, Rize Province, Türkiye



The current Operations commenced in late 1994, producing copper and zinc concentrates from ore mined by conventional underground methods in what is known as the Main Orebody. In 2021, with the Main Orebody in its final stages of production and nearing closure, three diamond holes were drilled targeting an anomaly detected by a land-based gravity survey (FQM, 2020), located approximately 300 m south of the

Main Orebody. The discovery hole (D0202167) intersected 33 m at an average grade of 2.4% Cu in what is now referred to as the South Orebody.

Exploration drilling proceeded from 2021, from surface positions and from underground cuddies. An underground exploration heading was also driven towards the South Orebody on the 1040 mRL sublevel, intersecting mineralisation in December 2023. Underground drilling from this and other sublevels continues to this day.

A new life of mine (LOM) plan has been produced focussing on production from the newly defined South Orebody and continuing through to 2036. This plan is supplemented with production from remnant areas within the Main Orebody, over an intermittent period between 2025 and 2036.

The existing ore processing plant consists of conventional crushing, grinding, selective flotation and pressure filtration facilities. The copper and zinc concentrates, containing silver and gold as by-products, are transported by road to the nearby port of Rize. Plant tailings are partly used to fill the underground voids after mixing with cement at a paste fill facility. The balance is discharged at depth into the Black Sea. These existing facilities will continue to be used for processing of the South Orebody feed.

1.2 Operations background

ÇBI is an existing operation that has been mining and processing copper and zinc ores for more than thirty years. The associated infrastructure as required by these operations remains in place and includes sealed roads, power lines and substations, a process plant, site offices, workshops, tailings/paste fill disposal and waste storage facilities.

The mined ore is truck hauled to surface and dumped into one of several ore blending bins. The ore is then selectively reclaimed into a bunker at the process plant, from where it is fed to a two-stage crushing facility. There are several ore types that are mined and then blended into either of a “Spec” Main Orebody, “Non-spec” Main Orebody, “Spec” South Orebody or “Non-spec” South Orebody feed[1].

The blended ore feed is crushed and then conveyed to a fine ore silo. From the silo, the crushed ore is conveyed further to the grinding unit, which includes two closed-circuit ball mills. The ground ore is then fed to the flotation system.

The copper and zinc flotation concentrates are subsequently fed to a thickener tank, followed by cleaning, filtration and final dewatering. Copper and zinc concentrates are loaded into road trucks and transported to the Rize port, approximately 26 km away from the Operations site.

That portion of the process tailings that is not used for underground paste filling is transferred via pipeline to a mixing tank on the Çayeli coast. The tailing is discharged from that tank into another pipeline extending out into the Black Sea.

Power to the Operations site is provided by a 31.5 kV line connected to the Turkish national grid system. Power is then distributed into the underground mine, to the milling circuit and into the plant and other site facilities, from two transformers.

1.3 Project approvals

Projects that are documented to have started production and/or commenced operations before the publication in February 1993 of the *Environmental Impact Assessment Regulation of Türkiye* (number 21489)

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[1] Spec ore types are typically copper rich and zinc poor. Conversely, Non-spec ore types are typically zinc rich, and contain bornite and lead mineralisation. Comprehensive definitions are provided in Sections 13.1 and 17.2.

are deemed to be exempt from submitting an Environmental Impact Assessment (EIA). ÇBI's exemption from the EIA process is applicable due to the Company being issued with an operating (business) licence in July 1987, which licence is a legal requirement for the commencement of operations. The Directorate of Mining Affairs and Natural Resources of Türkiye confirmed the EIA exemption on the 25th of May 2010.

The *Environmental Licence* conditions stipulate that in accordance with the *Mining Waste Regulations of Türkiye, a Waste Management Plan* must be updated and submitted every five years. Information to enable the Project Environmental Licence to be renewed was submitted to the *Ministry of Environment and Urbanisation* (the E&U Ministry) in November 2019.

In March 2021, ÇBI renewed its five year *Integrated Environmental Permit* that governs the environmental compliance requirements for integrated plants, which will be valid until March 2026. In October 2017, ÇBI also renewed the permit for operations at the Rize port concentrate storage and handling facility, which is valid until 2027.

An integral part of the waste management plan relates to the discharge of tailings into the Black Sea, and to that end, the E&U Ministry are to appoint a "special expertise commission" to review the extensive discharge monitoring records and the plan for continued discharge (Item 1.10). In the absence of that commission being appointed, the Ministry has formally confirmed in writing that the Company's operations "……. can continue within the scope of the current environmental permit process". This confirmation was given in March 2021 during the lead up to permit renewal.

During the 2025 environmental permit renewal process, the Ministry has reiterated its earlier position in writing, stating: *"…from the perspective of waste management legislation, there is no objection to submitting an environmental permit application*."

1.4 Production history

From the commencement of operations in 1994, up until the end of April 2025, total ÇBI production was 28.5 Mt of plant feed yielding 3.5 Mt of copper concentrate at an average grade of 21.7% Cu, and 1.7 Mt of zinc concentrate at an average grade of 49.2% Zn.

1.5 Geology and Mineral Resource

The main Çayeli mineralisation is located along the contact between a hanging wall of pyroclastics (tuffs) and basalts, and a footwall of rhyolite and felsic pyroclastic rocks. Hydrothermal alteration related to the formation of the deposit is restricted to the footwall stratigraphy and is in the form of clay (argillite) and chlorite. The mineralisation is a typical example of a volcanogenic massive sulphide (VMS) style of deposit, characterised by two main styles, i.e., massive sulphides and associated stockwork mineralisation. The key economic elements are Cu and Zn and to a lesser degree Ag as a by-product. Au is present but is very patchy and limited in its distribution.

A Mineral Resource estimate was compiled in May 2025 to incorporate all new South Orebody drilling assay data up to the 30th of April 2025. As part of this work, remnant areas of the Main Orebody were also modelled for inclusion in the Mineral Resource estimate. All remnant areas of the Main Orebody are above 800 mRL, below this level the Main Orebody is mined out.

The estimates were completed for copper (Cu), zinc (Zn), lead (Pb), Silver (Ag), gold (Au) and density. Drillhole samples were composited to 2 m intervals and coded according to their respective rock and mineralisation domains. As per previous site practices, oxidation was not modelled as all mineralisation is sited at depth and largely unaffected by surface weathering. All elements were estimated into a block model using optimised parameters for ordinary kriging while density was estimated into blocks where there was sufficient sample support.

Block estimates were validated per element and per domain using visual inspection, comparisons of mean values against input sample data, as well as multiple oriented grade trend validation slices. Estimate results per element and domain compare well with their input data. Grade trend plots of key elements show good reproduction of input sample grades.

Mineral Resource classification has been guided by:

- A Cu metal equivalent (Cu_{eq}) grade of 0.90%, determined using long term consensus metal prices, costs and metal recoveries.
- A floating stope style optimisation run on the South Orebody mineralisation for a minimum stope size of 6 m by 14 m by 25 m in the X, Y and Z directions respectively and a cut-off grade of 0.90% Cu_{eq}.
- Geological and grade continuity, drill grid spacing, drilling quality control results and mining constraints where relevant.

Mineral Resources have been classified as Inferred or Indicated; no Measured Mineral Resources have been reported. The Mineral Resource statement is provided in Table 1-1. While Au is payable in Non-spec Cu concentrate, it was excluded from the statement for two reasons:

1. The Au grade estimates are considered indicative, being reflective of 6 m composite samples.
2. The contribution to final concentrate return is relatively low due to a low plant recovery.

Table 1-1 Çayeli April 30th 2025 Mineral Resource statement using a 0.90% Cu_{eq} cut-off grade

Orebody	Classification	Tonnes (Mt)	Cu (%)	Zn (%)	Ag (ppm)
Main	Indicated	0.54	3.55	3.59	31.50
South	Indicated	8.79	1.33	2.48	9.09
	Inferred	0.46	0.64	2.31	6.39
	Total Indicated	**9.33**	**1.46**	**2.54**	**10.37**
	Total Inferred	**0.46**	**0.64**	**2.31**	**6.39**

Mineral Resources are reported inclusive of Mineral Reserves.

1.6 Metallurgy and mineral processing

The ÇBI processing facility has been in continuous operation for over thirty years, treating a range of ore types from the Main Orebody. Ore throughputs reached a peak of 1.34 Mtpa in 2014 and then steadily decreased to 691,000 tpa in 2024 due to ore depletion and a reduced number of operating stopes.

The main ore types treated have comprised 'yellow' ores containing high copper grades, and low (<4%) zinc grades, with and without bornite mineralisation, in addition to 'black' ores with high zinc content, and 'clastic' ores (with and without bornite mineralisation) which exhibited intergrowths of chalcopyrite within grains of sphalerite. Yellow ores have produced Spec copper and zinc concentrates, and clastic and black ores have produced Non-spec concentrates[2].

Significant historical data exists on the blending and treatment of these ores and historic recoveries and concentrate grades are expected to be achieved in years 2025 to 2027 and 2035, 2036 when treating the remnant material from the Main Orebody. The recently discovered South Orebody comprises two main ore

[2] Comprehensive definitions of ore types and processing specifications are provided in Sections 13.1.

types – 'yellow' ore and 'black' ores. The black ores could be further subdivided into high and low grade zinc ores. These ores will provide most of the plant feed for the remainder of the mine life.

Three composite samples of material from the South Orebody were tested by MRD, Hacettepe University, Ankara in 2024. Additionally, just over 5,000 t of mineralised mine development material from the South Orebody was successfully treated in the existing milling and flotation circuit, using existing operating parameters, and achieving good recoveries and concentrate grades.

Testwork indicated that the South Orebody ores, and particularly the yellow ores, had higher Bond ball mill and abrasion indices, suggesting higher energy requirements in the milling circuit, in addition to a higher consumption of grinding media.

In the treatment of both ore types, the operating parameters were like those already being employed in the treatment of Main Orebody plant feed, with the same grind size requirements and the same flotation reagent additions.

Flotation testwork highlighted that the yellow and black ores should not be blended during treatment. Yellow ores produced Spec copper concentrates, but the zinc feed grades (0.1% to 0.2% Zn) were too low to warrant treating this material through a zinc flotation circuit. Black ores produced a Non-spec copper concentrate and a Spec zinc concentrates. Recoveries and concentrate grades achieved in testwork are shown in Table 1-2.

Table 1-2 Recoveries and concentrate grades for the South Orebody plant feed

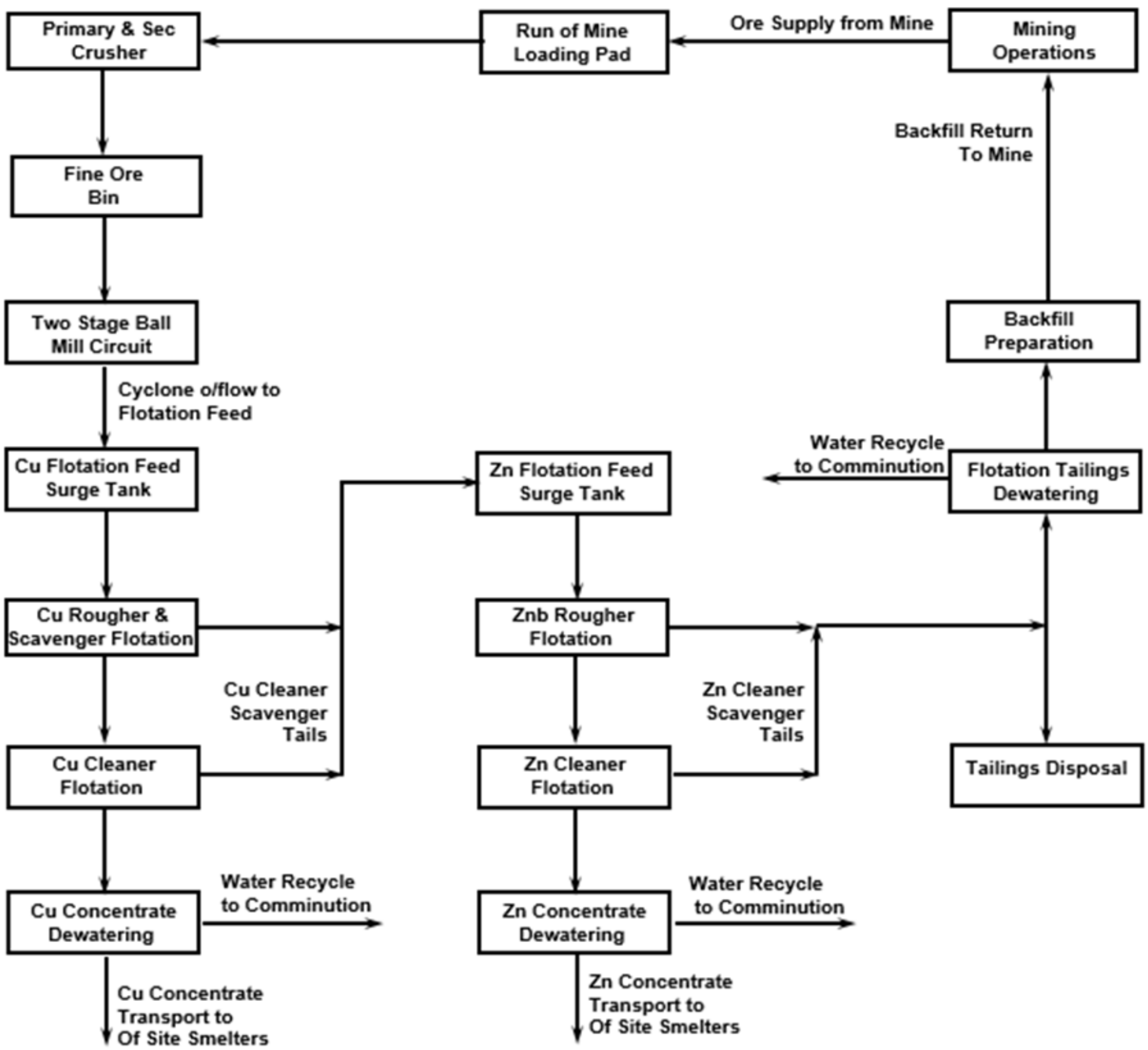
	Recoveries, %		Concentrate grades				
	Cu	Zn		% Cu	% Zn	Ag ppm	Au ppm
Footwall (Spec con.)	92.0		**Cu**	23.0	2.5	20.0	1.3
Zinc Ores (Blend 2)	60.0	75.0	**Cu**	19.0	10.0	40.0	5.0
(Non-spec Cu con.)			**Zn**	5.0	50.0	65.0	3.0

Note that a zero recovery is assigned for zinc for the footwall material despite the concentrate grade showing 2.5% Zn. This is because no payment is received for zinc in the copper concentrate.

The ÇBI ore processing facility consists of conventional crushing, grinding, selective flotation, and pressure filtration circuits. The facility is equipped with an online Yokogawa process control system and an SGS Expert System.

A simplified block flowsheet of the plant is presented in Figure 1.2.

Figure 1.2 Simplified block flowsheet for the ÇBI processing facilities



Since the ore types generated from the South Orebody are expected to be similar to the current ore types from the Main Orebody, but without the complications of bornite or clastic ores, the existing circuit at Çayeli does not require any modifications to the flowsheet, nor the introduction of new reagents to treat South Orebody ores.

No additional equipment will be required to handle the anticipated ore throughput or feed grades for both the remnant ores from the Main Orebody, and the longer-term feed from the South Orebody.

Due to the significant age of the equipment, a programme has been in place for some years to replace worn and corroded equipment, affecting equipment reliability, and posing a significant challenge for the processing plant. A capital allowance of $6.6 M has been provided for equipment replacement over the remaining life of mine and this includes provision for a new primary jaw crusher.

1.7 Mining and Mineral Reserve

An underground bulk mining method is in use at Çayeli, with the practice of backfilling to maximise the stoping of ore in a sequential extraction manner. Figure 1.3 shows a 3D view through the Main and South Orebodies. Ore production in the Deep Orebody below the Main 800 mRL sublevel was essentially completed by 2020.

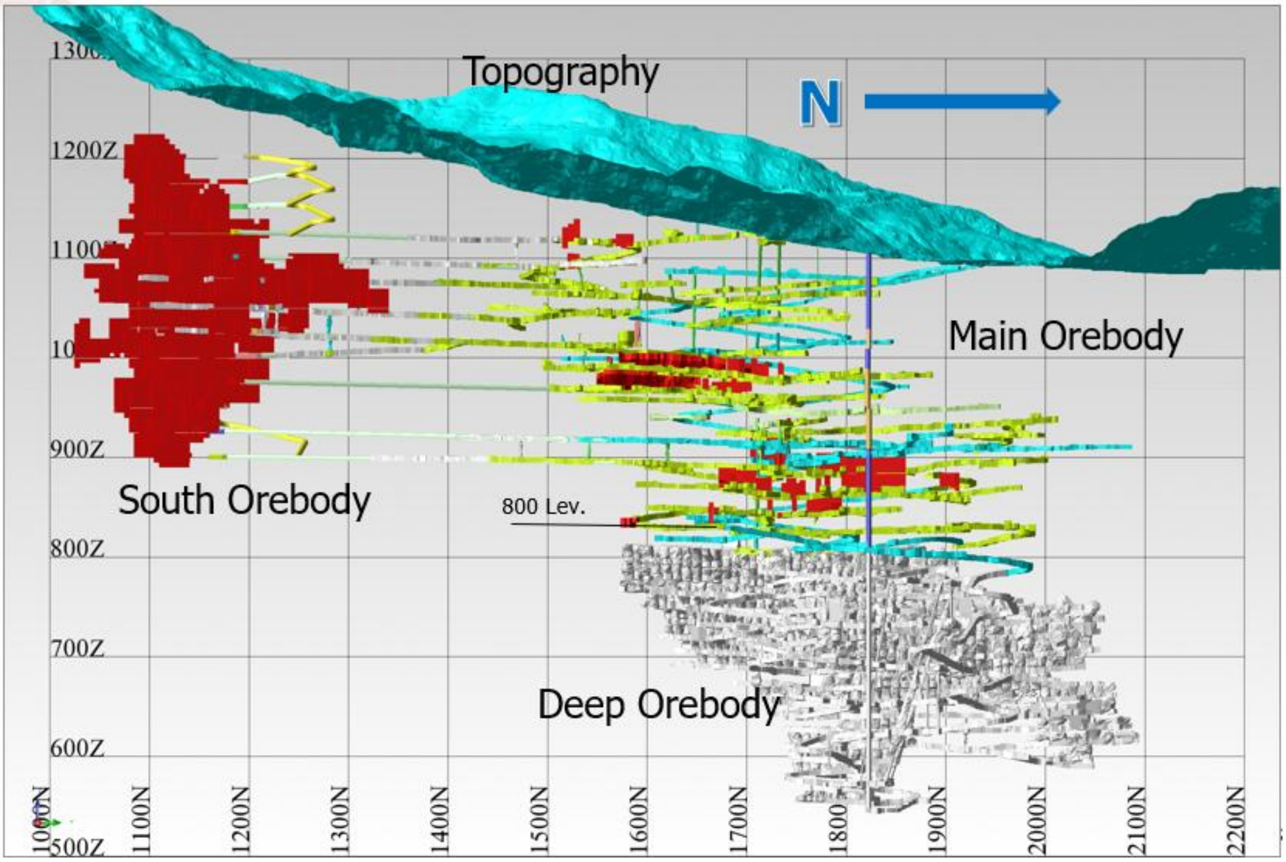
The existing underground mine, to the base of the abandoned Deep Orebody sublevels, reaches a depth of about 550 m below surface. The hoisting shaft has been decommissioned and backfilled, whilst leaving the ladderway compartments open for secondary egress and downcast ventilation. Access into the existing mine is now via a decline, which above the 800 mRL sublevel is positioned in the hangingwall. A new ramp from surface is being developed to access the upper South Orebody, and its completion is imminent.

Ore and waste development is carried out conventionally, using jumbo drills, front end loaders and articulated dump trucks. In zones where weak rock mass conditions are experienced, rock breakers are used in place of jumbo drilling and blasting.

The primary production method for the Main Orebody is conventional long hole stoping with extraction maximised by means of paste filling or with unconsolidated waste rock fill. The method is overhand (i.e., progressing from bottom-up in each stoping block), retreating from a mined slot rise, and featuring primary, secondary and tertiary stope sequencing.

The prevailing rock mass and host rocks can be classified as poor to fair quality, characterised by intense schistosity and foliation. These conditions require special attention and intensive ground support and reinforcement measures.

Figure 1.3 Longitudinal schematic (3D) view through the Main, Deep and South Orebodies



An NSR approach was used to define that part of the Mineral Resource eligible for conversion to a Mineral Reserve. Modelled blocks were eligible if the NSR value was equal to or in excess of the estimated overall life of mine (LOM) average operating cost.

The calculated NSR value accounted for metal prices, process recovery projections, TCRCs (i.e., treatment and refining charges), concentrate freight charges, and recovered metal payability. Unit operating costs were estimated in detail from mid-year 2024 budget/forecast costs accounting for fixed and variable components, specific production activities, consumables usage and unit charges, power and fuel consumption, operating and maintenance labour, general and administrative charges, and concentrate handling up to the Rize port.

The April 30th 2025 Çayeli Mineral Reserve estimate and statement is presented in Table 1-3.

Table 1-3 Çayeli April 30th 2025 Mineral Reserve statement, at $4.10/lb Cu, $1.20/lb Zn, $22.50/oz Ag

Orebody	Classification	Tonnes (Mt)	NSR ($/t)	Cu (%)	Zn (%)	Ag (ppm)
Main	Probable	0.58	244	3.12	3.08	26.57
South	Probable	6.74	131	1.37	2.28	8.92
	Total Probable	**7.31**	**139**	**1.51**	**2.34**	**10.31**

The inventory in Table 1-3 reflects individually designed ore development and stope openings, which are fully diluted (both "planned" and "unplanned") and adjusted for mining recovery.

Additional notes:

- Mineral Resources are reported inclusive of Mineral Reserves.
- For reasons associated with the compositing of gold samples, an indicative gold (Au) grade is not included in the Mineral Reserve statement.
- Whilst small discrepancies may occur in the figures due to rounding, the impact is not material.

1.8 Production schedule

Table 1-4 combines and summarises the annual production schedule information for ore development and stoping in both orebodies. This table reports the schedule figures after the application of both planned and unplanned dilution (and mining recovery losses).

Table 1-4 Annual schedule of combined mine development and stope tonnes and grades

Orebody	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Main Orebody ore development														
Dil'd and Rec'd tonnes	t	**65,968**	16,821	20,826	11,137								10,271	6,913
Av. NSR	$/t ore	**195**	216	216	80								226	217
Av. Cu	%	**2.48**	2.61	2.41	1.48								2.96	3.24
Av. Zn	%	**1.80**	3.04	0.95	2.69								0.29	2.13
Main Orebody stopes														
Dil'd and Rec'd tonnes	t	**510,219**	191,121	78,946	74,244								110,198	55,710
Av. NSR	$/t ore	**250**	317	208	144								247	230
Av. Cu	%	**3.20**	4.05	2.81	2.02								3.06	2.72
Av. Zn	%	**3.25**	5.02	0.61	0.50								3.20	4.66
Main Orebody total														
Dil'd and Rec'd tonnes	**t**	**576,187**	**207,942**	**99,772**	**85,381**								**120,469**	**62,623**
Av. NSR	**$/t ore**	**244**	**309**	**210**	**136**								**245**	**229**
Av. Cu	**%**	**3.12**	**3.93**	**2.73**	**1.95**								**3.05**	**2.78**
Av. Zn	**%**	**3.08**	**4.86**	**0.68**	**0.79**								**2.95**	**4.38**
South Orebody ore development														
Dil'd and Rec'd tonnes	t	**1,321,055**	56,117	166,598	126,730	164,148	153,044	114,843	178,359	130,639	59,358	84,394	50,218	36,607
Av. NSR	$/t ore	**135**	126	128	117	152	146	149	144	146	141	115	109	83
Av. Cu	%	**1.42**	1.37	1.39	1.11	1.51	1.53	1.52	1.61	1.74	1.47	1.00	0.87	1.02
Av. Zn	%	**2.33**	1.57	1.62	2.79	3.10	2.29	3.14	1.68	1.50	3.02	3.33	3.88	0.01
South Orebody stopes														
Dil'd and Rec'd tonnes	t	**5,415,010**	183,408	525,531	629,284	677,246	688,350	627,563	465,060	463,286	435,580	361,049	225,263	133,390
Av. NSR	$/t ore	**129**	123	114	121	129	131	147	135	149	138	120	110	92
Av. Cu	%	**1.36**	1.63	1.38	1.20	1.30	1.21	1.49	1.48	1.60	1.58	1.22	0.96	1.13
Av. Zn	%	**2.27**	0.02	0.95	2.49	2.58	3.29	2.81	2.00	2.16	2.22	2.25	3.14	0.09
South Orebody total														
Dil'd and Rec'd tonnes	**t**	**6,736,066**	**239,525**	**692,129**	**756,013**	**841,394**	**841,394**	**742,406**	**643,419**	**593,926**	**494,937**	**445,444**	**275,482**	**169,997**
Av. NSR	**$/t ore**	**131**	**123**	**117**	**120**	**133**	**134**	**147**	**138**	**148**	**138**	**119**	**109**	**90**
Av. Cu	**%**	**1.37**	**1.57**	**1.38**	**1.18**	**1.34**	**1.27**	**1.50**	**1.52**	**1.63**	**1.57**	**1.18**	**0.94**	**1.11**
Av. Zn	**%**	**2.28**	**0.38**	**1.11**	**2.54**	**2.68**	**3.11**	**2.86**	**1.91**	**2.01**	**2.32**	**2.45**	**3.28**	**0.08**
Combined ore development														
Dil'd and Rec'd tonnes	t	**1,387,024**	72,939	187,423	137,867	164,148	153,044	114,843	178,359	130,639	59,358	84,394	60,490	43,520
Av. NSR	$/t ore	**138**	146	138	114	152	146	149	144	146	141	115	129	104
Av. Cu	%	**1.47**	1.66	1.51	1.14	1.51	1.53	1.52	1.61	1.74	1.47	1.00	1.22	1.38
Av. Zn	%	**2.31**	1.91	1.54	2.78	3.10	2.29	3.14	1.68	1.50	3.02	3.33	3.27	0.35
Combined stopes														
Dil'd and Rec'd tonnes	t	**5,925,229**	374,528	604,477	703,527	677,246	688,350	627,563	465,060	463,286	435,580	361,049	335,461	189,100
Av. NSR	$/t ore	**140**	222	126	123	129	131	147	135	149	138	120	155	133
Av. Cu	%	**1.52**	2.86	1.57	1.29	1.30	1.21	1.49	1.48	1.60	1.58	1.22	1.65	1.60
Av. Zn	%	**2.35**	2.57	0.91	2.28	2.58	3.29	2.81	2.00	2.16	2.22	2.25	3.16	1.44
Combined total														
Dil'd and Rec'd tonnes	**t**	**7,312,253**	**447,467**	**791,901**	**841,394**	**841,394**	**841,394**	**742,406**	**643,419**	**593,926**	**494,937**	**445,444**	**395,950**	**232,621**
Av. NSR	**$/t ore**	**139**	**209**	**129**	**122**	**133**	**134**	**147**	**138**	**148**	**138**	**119**	**151**	**127**
Av. Cu	**%**	**1.51**	**2.67**	**1.55**	**1.26**	**1.34**	**1.27**	**1.50**	**1.52**	**1.63**	**1.57**	**1.18**	**1.59**	**1.56**
Av. Zn	**%**	**2.34**	**2.46**	**1.06**	**2.36**	**2.68**	**3.11**	**2.86**	**1.91**	**2.01**	**2.32**	**2.45**	**3.18**	**1.23**

A blended plant feed schedule accompanies the mining production schedule and this is summarised in Table 1-5. In this instance, there is a no cross-blending from the Main Orebody mined ore types to the corresponding plant feed. Similarly, there is no cross-blending from the South Orebody mined ore to the corresponding plant feed.

Table 1-5 Annual schedule of plant feed tonnes and grades

Oretype	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Main Orebody														
Spec														
Dil'd and Rec'd tonnes	t	**386,616**	**106,760**	**96,026**	**71,554**								**73,893**	**38,383**
Av. Cu	%	2.60	3.27	2.69	1.79								2.40	2.43
Av. Zn	%	0.62	0.95	0.62	0.09								0.51	0.93
Non-spec														
Dil'd and Rec'd tonnes	t	**189,571**	**101,181**	**3,746**	**13,827**								**46,576**	**24,241**
Av. Cu	%	4.17	4.63	3.74	2.77								4.07	3.33
Av. Zn	%	8.09	8.98	2.27	4.40								6.81	9.85
South Orebody														
Spec														
Dil'd and Rec'd tonnes	t	**4,800,095**	**227,147**	**589,881**	**540,915**	**546,554**	**512,599**	**486,215**	**473,362**	**452,850**	**353,489**	**294,308**	**152,778**	**169,997**
Av. Cu	%	1.67	1.63	1.57	1.51	1.77	1.75	1.82	1.69	1.86	1.78	1.60	1.38	1.11
Av. Zn	%	0.19	0.02	0.11	0.04	0.08	0.05	0.10	0.06	0.47	1.01	0.26	0.05	0.08
Non-spec														
Dil'd and Rec'd tonnes	t	**1,935,970**	**12,378**	**102,248**	**215,098**	**294,839**	**328,795**	**256,192**	**170,057**	**141,075**	**141,448**	**151,136**	**122,703**	
Av. Cu	%	0.64	0.42	0.34	0.36	0.56	0.53	0.88	1.04	0.89	1.05	0.36	0.41	
Av. Zn	%	7.46	6.96	6.87	8.83	7.50	7.87	8.12	7.06	6.98	5.57	6.72	7.30	
Combined orebodies														
Spec														
Dil'd and Rec'd tonnes	t	**5,186,711**	**333,907**	**685,906**	**612,469**	**546,554**	**512,599**	**486,215**	**473,362**	**452,850**	**353,489**	**294,308**	**226,671**	**208,380**
Av. Cu	%	**1.74**	**2.16**	**1.72**	**1.54**	**1.77**	**1.75**	**1.82**	**1.69**	**1.86**	**1.78**	**1.60**	**1.71**	**1.35**
Av. Zn	%	**0.22**	**0.32**	**0.18**	**0.04**	**0.08**	**0.05**	**0.10**	**0.06**	**0.47**	**1.01**	**0.26**	**0.20**	**0.23**
Non-spec														
Dil'd and Rec'd tonnes	t	**2,125,542**	**113,559**	**105,994**	**228,925**	**294,839**	**328,795**	**256,192**	**170,057**	**141,075**	**141,448**	**151,136**	**169,280**	**24,241**
Av. Cu	%	**0.95**	**4.17**	**0.46**	**0.51**	**0.56**	**0.53**	**0.88**	**1.04**	**0.89**	**1.05**	**0.36**	**1.41**	**3.33**
Av. Zn	%	**7.52**	**8.76**	**6.70**	**8.57**	**7.50**	**7.87**	**8.12**	**7.06**	**6.98**	**5.57**	**6.72**	**7.16**	**9.85**
TOTAL														
Dil'd and Rec'd tonnes	t	**7,312,253**	**447,467**	**791,901**	**841,394**	**841,394**	**841,394**	**742,406**	**643,419**	**593,926**	**494,937**	**445,444**	**395,950**	**232,621**
Av. Cu	%	**1.51**	**2.67**	**1.55**	**1.26**	**1.34**	**1.27**	**1.50**	**1.52**	**1.63**	**1.57**	**1.18**	**1.59**	**1.56**
Av. Zn	%	**2.34**	**2.46**	**1.06**	**2.36**	**2.68**	**3.11**	**2.86**	**1.91**	**2.01**	**2.32**	**2.45**	**3.18**	**1.23**

1.9 Infrastructure

In addition to the overview of existing Operations infrastructure in Item 1.2, further facilities will be required to sustain the extended Operations to 2036, and specifically to cater for new production from the South Orebody, e.g.:

- a new primary underground ventilation fan, plus auxiliary fans
- a new paste fill thickener
- a new primary mine dewatering pump
- a new primary jaw crusher
- replacement flotation cells and reagent tanks

These facilities and equipment are included in the capital cost provisions summarised in Item 1.11.

1.10 Tailings disposal

Owing to topographical constraints, there is no surface tailings disposal and storage facility within the Project licence area. Excess tailings, not required for paste fill in the mine, are piped to a mix tank on the Black Sea coast via a 7.5 km long overland pipeline. In what is referred to as Deep Sea Tailings (DST), the tailings are then piped out to sea through a 3 km long pipeline and discharged at a depth of 275 m. The Black Sea aquatic environment is anoxic below a depth of 150 m and does not support any form of marine life. The seawater is naturally rich in hydrogen sulphide and deficient in dissolved oxygen.

Since the commencement of operations, a total of 15.4 million tonnes of tailings has been discharged via DST placement.

For over twenty years a discharge monitoring programme has been carried out by the *Central Fisheries Research Institute* (SUMAE) affiliated with the *Ministry of Agriculture and Forestry.* There are ten offshore discharge sampling locations and seven sampling depth intervals down to 650 m. Twenty one different parameters have been analysed from each sample location and thousands of samples have been analysed since the commencement of monitoring. A report on 2023 quarterly sampling and analysis was prepared by MCG Engineering Consultancy from Istanbul (MCG, April 2024), in which it was concluded that:

"*it was observed that all metal matrix values were well below the values specified in the continental water general quality indicators ……., and no pollutant contribution related to mining terrestrial activities was detected*".

1.11 Capital costs estimate

The estimated capital costs over the remaining life of operations are summarised in Table 1-6.

Table 1-6 Capital costs, as at the end of February 2025

Department	Description	2025 ($,000)	2026 ($,000)	2027 ($,000)	2028 ($,000)	2029 ($,000)	2030 ($,000)	2031 ($,000)	2032 ($,000)	2033 ($,000)	2034 ($,000)	2035 ($,000)	TOTAL ($,000)
Mine	Initial Capex	$5,908											**$5,908**
	Sustaining		$5,940	$1,830	$1,675	$695	$3,540	$625	$535	$495	$400	$260	**$15,995**
	Subtotal	**$5,908**	**$5,940**	**$1,830**	**$1,675**	**$695**	**$3,540**	**$625**	**$535**	**$495**	**$400**	**$260**	**$21,903**
Mill	Initial Capex	$1,501											**$1,501**
	Sustaining		$1,546	$1,626	$796	$216	$195	$300	$200	$150	$150	$0	**$5,179**
	Subtotal	**$1,501**	**$1,546**	**$1,626**	**$796**	**$216**	**$195**	**$300**	**$200**	**$150**	**$150**	**$0**	**$6,680**
Plant	Initial Capex	$6,706											**$6,706**
	Sustaining		$6,640	$5,540	$530	$520	$510	$610	$510	$510	$510	$510	**$16,390**
	Subtotal	**$6,706**	**$6,640**	**$5,540**	**$530**	**$520**	**$510**	**$610**	**$510**	**$510**	**$510**	**$510**	**$23,096**
Administration	Initial Capex	$586											**$586**
	Sustaining		$1,023	$194	$151	$127	$125	$425	$461	$125	$100	$100	**$2,828**
	Subtotal	**$586**	**$1,023**	**$194**	**$151**	**$127**	**$125**	**$425**	**$461**	**$125**	**$100**	**$100**	**$3,414**
All	**Initial Capex**	**$14,701**											**$14,701**
	Sustaining		**$15,149**	**$9,190**	**$3,152**	**$1,558**	**$4,370**	**$1,960**	**$1,706**	**$1,280**	**$1,160**	**$870**	**$40,392**
	Subtotal	**$14,701**	**$15,149**	**$9,190**	**$3,152**	**$1,558**	**$4,370**	**$1,960**	**$1,706**	**$1,280**	**$1,160**	**$870**	**$55,093**

1.12 Operating cost estimate

Table 1-7 shows the overall LOM operating costs in total dollar terms, based on 2024 budgeted costs adjusted to reflect the February 2025 forecast physicals. These cost projections were adopted for mine planning purposes and for determining the overall annual and LOM costs for comparison against model block NSR calculations.

Table 1-7 Total operating costs estimate for mine planning, based on historical operating data

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
MINING COSTS															
Development in waste	$k	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	**$6,641.0**
Development in ore	$k	$5,537.6	$4,014.9	$5,499.2	$4,003.9	$3,263.3	$3,243.7	$1,615.2	$2,077.9	$1,670.2	$1,781.2	$1,510.3	$938.6	$849.2	**$36,005.2**
Stoping	$k	$2,531.3	$3,182.6	$3,374.2	$3,740.9	$3,922.6	$3,093.9	$2,937.7	$2,268.6	$2,090.7	$1,507.9	$1,574.3	$1,436.7	$1,458.6	**$33,120.0**
Services (ore + waste)	$k	$8,888.5	$9,507.5	$10,745.4	$10,745.4	$10,745.4	$8,888.5	$7,650.5	$6,412.6	$5,793.6	$4,555.7	$4,555.7	$3,936.7	$3,936.7	**$96,362.0**
Maintenance	$k	$3,748.1	$3,577.8	$3,833.3	$3,833.3	$3,833.3	$3,207.8	$2,627.0	$2,046.2	$2,001.5	$1,420.7	$1,599.4	$1,331.4	$1,331.4	**$34,391.4**
Additional mining labour	$k	$3,285.0	$3,197.8	$3,183.3	$3,161.5	$3,139.7	$2,627.4	$2,151.7	$1,676.0	$1,639.4	$1,163.7	$1,310.0	$1,090.5	$1,090.5	**$28,716.4**
subtotal	**$k**	**$24,501.4**	**$23,991.4**	**$27,146.2**	**$25,995.9**	**$25,415.1**	**$21,572.1**	**$17,493.0**	**$14,992.1**	**$13,706.3**	**$10,939.9**	**$11,060.6**	**$9,244.7**	**$9,177.2**	**$235,236.1**
	$/t	***$35.00***	***$31.99***	***$31.94***	***$30.58***	***$29.90***	***$30.82***	***$29.15***	***$29.98***	***$30.46***	***$31.26***	***$31.60***	***$30.82***	***$30.59***	***$31.16***
MILLING COSTS															
subtotal	**$k**	**$11,813.8**	**$12,491.1**	**$14,035.1**	**$14,228.2**	**$14,285.2**	**$12,317.6**	**$10,929.1**	**$9,540.6**	**$8,961.5**	**$7,529.2**	**$7,656.7**	**$6,916.4**	**$6,685.3**	**$137,389.7**
	$/t	***$16.88***	***$16.65***	***$16.51***	***$16.74***	***$16.81***	***$17.60***	***$18.22***	***$19.08***	***$19.91***	***$21.51***	***$21.88***	***$23.05***	***$22.28***	***$18.20***
PLANT COSTS															
subtotal	**$k**	**$7,281.5**	**$7,122.3**	**$7,187.6**	**$7,147.8**	**$7,108.0**	**$5,948.2**	**$4,871.2**	**$3,794.2**	**$3,711.4**	**$2,634.4**	**$2,965.8**	**$2,468.7**	**$2,468.7**	**$64,710.0**
	$/t	***$10.40***	***$9.50***	***$8.46***	***$8.41***	***$8.36***	***$8.50***	***$8.12***	***$7.59***	***$8.25***	***$7.53***	***$8.47***	***$8.51***	***$8.51***	***$8.66***
ADMINISTRATION COSTS															
subtotal	**$k**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$136,252.1**
	$/t	***$14.97***	***$13.97***	***$12.33***	***$12.33***	***$12.33***	***$14.97***	***$17.47***	***$20.96***	***$23.29***	***$29.95***	***$29.95***	***$34.94***	***$34.94***	***$18.05***
CONCENTRATE HANDLING COSTS															
subtotal	**$k**	**$921.4**	**$987.2**	**$1,118.9**	**$1,118.9**	**$1,118.9**	**$921.4**	**$789.8**	**$658.2**	**$592.3**	**$460.7**	**$460.7**	**$394.9**	**$394.9**	**$9,938.1**
	$/t	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***
TOTAL OPERATING COSTS															
Total	**$k**	**$54,999.1**	**$55,073.0**	**$59,968.7**	**$58,971.7**	**$58,408.1**	**$51,240.2**	**$44,564.0**	**$39,466.0**	**$37,452.4**	**$32,045.2**	**$32,624.8**	**$29,505.7**	**$29,207.1**	**$583,525.9**
	$/t	***$78.57***	***$73.43***	***$70.55***	***$69.38***	***$68.72***	***$73.20***	***$74.27***	***$78.93***	***$83.23***	***$91.56***	***$93.21***	***$98.63***	***$97.63***	***$77.37***
averages	***$/t***	***$72***						***$86***							***$77***

Table 1-8 lists the updated operating costs for the cashflow model. The total operating costs shown in this update reflect the unit costs from the 2024 budget estimate review multiplied by the new mine and plant feed schedule physicals as set out in Table 1-4 and Table 1-5, respectively.

Table 1-8 Total operating costs estimate for cashflow modelling

	UNITS	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
PRODUCTION PHYSICALS															
Development in waste	kt	16.7	92.9	103.9	86.7	55.7	46.7	67.5	59.2	20.0	41.0	37.2	15.9		**643.4**
Development in ore	kt	72.9	187.4	137.9	164.1	153.0	114.8	178.4	130.6	59.4	84.4	60.5	43.5		**1,387.0**
Stoping	kt	374.5	604.5	703.5	677.2	688.3	627.6	465.1	463.3	435.6	361.0	335.5	189.1		**5,925.2**
Total ore tonnes	**kt**	**447.5**	**791.9**	**841.4**	**841.4**	**841.4**	**742.4**	**643.4**	**593.9**	**494.9**	**445.4**	**396.0**	**232.6**		**7,312.3**
Total tonnes	**kt**	**464.2**	**884.8**	**945.3**	**928.1**	**897.1**	**789.1**	**710.9**	**653.1**	**514.9**	**486.4**	**433.2**	**248.5**		**7,955.7**
MINING COSTS															
Development in waste	$k	$474.0	$2,636.6	$2,948.7	$2,460.6	$1,580.8	$1,325.4	$1,915.7	$1,680.1	$567.6	$1,163.6	$1,055.8	$451.3		**$18,260.0**
Development in ore	$k	$1,652.4	$4,246.0	$3,123.3	$3,718.7	$3,467.1	$2,601.7	$4,040.6	$2,959.6	$1,344.7	$1,911.9	$1,370.4	$985.9		**$31,422.2**
Stoping	$k	$2,081.0	$3,358.7	$3,909.1	$3,763.1	$3,824.8	$3,487.0	$2,584.1	$2,574.2	$2,420.3	$2,006.1	$1,864.0	$1,050.7		**$32,923.1**
Services (ore + waste)	$k	$5,746.2	$10,953.4	$11,702.3	$11,489.3	$11,105.6	$9,768.7	$8,800.8	$8,085.4	$6,374.7	$6,021.9	$5,362.2	$3,076.6		**$98,487.0**
Maintenance	$k	$2,498.8	$3,577.8	$3,833.3	$3,833.3	$3,833.3	$3,207.8	$2,627.0	$2,046.2	$2,001.5	$1,420.7	$1,599.4	$887.6		**$31,366.8**
Additional mining labour	$k	$2,190.0	$3,197.8	$3,183.3	$3,161.5	$3,139.7	$2,627.4	$2,151.7	$1,676.0	$1,639.4	$1,163.7	$1,310.0	$727.0		**$26,167.4**
subtotal	**$k**	**$14,642.3**	**$27,970.2**	**$28,700.0**	**$28,426.5**	**$26,951.2**	**$23,018.0**	**$22,119.9**	**$19,021.5**	**$14,348.2**	**$13,688.0**	**$12,561.7**	**$7,179.0**		**$238,626.5**
	$/t	***$32.72***	***$35.32***	***$34.11***	***$33.78***	***$32.03***	***$31.00***	***$34.38***	***$32.03***	***$28.99***	***$30.73***	***$31.73***	***$30.86***		***$32.63***
MILLING COSTS															
subtotal	**$k**	**$8,372.8**	**$13,290.3**	**$13,994.5**	**$14,189.1**	**$14,189.1**	**$12,791.0**	**$11,413.8**	**$10,589.2**	**$9,463.1**	**$8,638.5**	**$7,893.8**	**$5,077.3**		**$129,902.5**
	$/t	***$18.71***	***$16.78***	***$16.63***	***$16.86***	***$16.86***	***$17.23***	***$17.74***	***$17.83***	***$19.12***	***$19.39***	***$19.94***	***$21.83***		***$17.77***
PLANT COSTS															
subtotal	**$k**	**$4,854.4**	**$7,122.3**	**$7,187.6**	**$7,147.8**	**$7,108.0**	**$5,948.2**	**$4,871.2**	**$3,794.2**	**$3,711.4**	**$2,634.4**	**$2,965.8**	**$1,645.8**		**$58,991.1**
	$/t	***$10.85***	***$8.99***	***$8.54***	***$8.50***	***$8.45***	***$8.01***	***$7.57***	***$6.39***	***$7.50***	***$5.91***	***$7.49***	***$7.08***		***$8.07***
ADMINISTRATION COSTS															
subtotal	**$k**	**$6,987.3**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$6,987.3**		**$118,783.9**
	$/t	***$15.62***	***$13.24***	***$12.46***	***$12.46***	***$12.46***	***$14.12***	***$16.29***	***$17.65***	***$21.18***	***$23.53***	***$26.47***	***$30.04***		***$16.24***
CONCENTRATE HANDLING COSTS															
subtotal	**$k**	**$589.0**	**$1,042.4**	**$1,107.5**	**$1,107.5**	**$1,107.5**	**$977.2**	**$846.9**	**$781.8**	**$651.5**	**$586.3**	**$521.2**	**$306.2**		**$9,625.1**
	$/t	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***		***$1.32***
TOTAL OPERATING COSTS															
Total	**$k**	**$35,445.7**	**$59,906.1**	**$61,470.6**	**$61,351.8**	**$59,836.8**	**$53,215.4**	**$49,732.8**	**$44,667.6**	**$38,655.1**	**$36,028.2**	**$34,423.4**	**$21,195.6**		**$555,929.1**
	$/t	***$79.21***	***$75.65***	***$73.06***	***$72.92***	***$71.12***	***$71.68***	***$77.29***	***$75.21***	***$78.10***	***$80.88***	***$86.94***	***$91.12***		***$76.03***
averages	***$/t***	***$74***						***$80***							***$76***

1.13 Economic analysis

In accordance with the Rules and Policies of the NI 43-101, the economic analysis does not include Inferred Mineral Resource as a source of revenue. Furthermore, and for reasons associated with the absence of a specific grade from the Mineral Reserve statement, there is no revenue assigned to the gold mineralisation.

The economic analysis in the form of a basic cashflow model is intended to support the Mineral Reserve estimate, and to demonstrate a positive cashflow for mining and processing. The development and expansion capital costs are included in the analysis for completeness. The model accounts for:

- long term metal pricing projections
- metal payabilities determined from historical actuals
- royalties payable to the Government of Türkiye, the local municipality and to *Eti*
- processing recoveries linked to the Reserves production plan physicals
- operating costs linked to the Reserves production plan physicals

The model is provided both pre-tax and post-tax and is summarised in Table 1-9 and Table 1-10, respectively. Specific comments regarding this summary are listed below:

- the pre-tax undiscounted cashflow is $307.2M, and post-tax is $251.1M
- pre-tax (net present value) NPV_{10} is $193.1M, and $155.7M post-tax
- pre-tax NPV_8 is $210.1M, and $169.8M post-tax
- other than the years post-closure, there are no negative annual cashflows, hence an (internal rate of return) IRR cannot be calculated
- similarly, there is no payback period relevant to the projected capital expenditure profile

Table 1-9 LOM cashflow summary, pre-tax

CASHFLOW SUMMARY PRE-TAX	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
MINERAL RESERVES CASHFLOW			8 months											8 months		
Gross revenue	**$'000**	**$1,135,773.8**	$110,898.3	$112,605.8	$117,742.3	$125,384.1	$126,509.5	$119,930.0	$95,485.3	$92,671.1	$72,995.1	$58,782.9	$69,301.5	$33,468.0		
Treatment, refining and freight charges (metal costs)	**$'000**	**$110,665.2**	$6,380.8	$4,956.7	$7,847.4	$10,304.7	$13,073.1	$13,778.3	$11,964.5	$10,933.7	$8,693.6	$8,316.9	$10,360.6	$4,054.9		
Net revenue	**$'000**	**$1,025,108.6**	**$104,517.5**	**$107,649.1**	**$109,894.9**	**$115,079.3**	**$113,436.5**	**$106,151.7**	**$83,520.7**	**$81,737.4**	**$64,301.5**	**$50,466.0**	**$58,940.9**	**$29,413.1**	**$0.0**	**$0.0**
Total operating costs																
Mining	**$'000**	**$238,626.5**	$14,642.3	$27,970.2	$28,700.0	$28,426.5	$26,951.2	$23,018.0	$22,119.9	$19,021.5	$14,348.2	$13,688.0	$12,561.7	$7,179.0		
Processing	**$'000**	**$129,902.5**	$8,372.8	$13,290.3	$13,994.5	$14,189.1	$14,189.1	$12,791.0	$11,413.8	$10,589.2	$9,463.1	$8,638.5	$7,893.8	$5,077.3		
Plant	**$'000**	**$58,991.1**	$4,854.4	$7,122.3	$7,187.6	$7,147.8	$7,108.0	$5,948.2	$4,871.2	$3,794.2	$3,711.4	$2,634.4	$2,965.8	$1,645.8		
Site administration	**$'000**	**$118,783.9**	$6,987.3	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$6,987.3		
Other direct costs	**$'000**	**$9,625.1**	$589.0	$1,042.4	$1,107.5	$1,107.5	$1,107.5	$977.2	$846.9	$781.8	$651.5	$586.3	$521.2	$306.2		
Other costs																
Royalty & mine taxes	**$'000**	**$91,262.1**	$11,633.2	$10,188.0	$9,514.7	$10,097.5	$9,766.4	$9,512.8	$7,092.5	$7,321.9	$5,379.6	$3,546.4	$4,836.2	$2,372.9		
Corporate costs	**$'000**	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Cash EBITDA	**$'000**	**$377,917.3**	**$57,438.5**	**$37,554.9**	**$38,909.6**	**$43,630.0**	**$43,833.2**	**$43,423.5**	**$26,695.4**	**$29,747.9**	**$20,266.8**	**$10,891.4**	**$19,681.3**	**$5,844.6**	**$0.0**	**$0.0**
Total capital costs (expansion)	**$'000**	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Total capital costs (mine development)	**$'000**	**$6,853.9**	$2,596.8	$2,809.7	$1,447.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Total direct capital costs	**$'000**	**$55,093.3**	$14,701.3	$15,149.0	$9,190.0	$3,152.0	$1,558.0	$4,369.5	$1,959.5	$1,705.5	$1,279.5	$1,159.5	$869.5	$0.0		
ARO costs	**$'000**	**$8,756.7**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$1,741.3	$5,223.9	$1,095.4	$696.2
Undiscounted cashflow pre-tax	**$'000**	**$307,213.4**	**$40,140.4**	**$19,596.3**	**$28,272.2**	**$40,478.0**	**$42,275.2**	**$39,054.0**	**$24,735.9**	**$28,042.4**	**$18,987.3**	**$9,731.9**	**$17,070.5**	**$620.7**	**-$1,095.4**	**-$696.2**

Table 1-10 LOM cashflow summary, post-tax

CASHFLOW SUMMARY POST-TAX	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
TECHNICAL REPORT CASHFLOW			8 months											8 months		
Gross revenue	**$'000**	**$1,135,773.8**	$110,898.3	$112,605.8	$117,742.3	$125,384.1	$126,509.5	$119,930.0	$95,485.3	$92,671.1	$72,995.1	$58,782.9	$69,301.5	$33,468.0		
Net revenue	**$'000**	**$1,025,108.6**	**$104,517.5**	**$107,649.1**	**$109,894.9**	**$115,079.3**	**$113,436.5**	**$106,151.7**	**$83,520.7**	**$81,737.4**	**$64,301.5**	**$50,466.0**	**$58,940.9**	**$29,413.1**	**$0.0**	**$0.0**
Cost of sales																
Mining	**$'000**	**$238,626.5**	$14,642.3	$27,970.2	$28,700.0	$28,426.5	$26,951.2	$23,018.0	$22,119.9	$19,021.5	$14,348.2	$13,688.0	$12,561.7	$7,179.0		
Processing	**$'000**	**$129,902.5**	$8,372.8	$13,290.3	$13,994.5	$14,189.1	$14,189.1	$12,791.0	$11,413.8	$10,589.2	$9,463.1	$8,638.5	$7,893.8	$5,077.3		
Other Direct	**$'000**	**$68,616.3**	$5,443.4	$8,164.7	$8,295.1	$8,255.3	$8,215.5	$6,925.4	$5,718.1	$4,576.0	$4,362.9	$3,220.8	$3,487.0	$1,952.0		
Site administration	**$'000**	**$118,783.9**	$6,987.3	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$6,987.3		
Depreciation	**$'000**	**$74,360.9**	$4,425.2	$5,804.3	$6,085.2	$6,788.6	$6,652.0	$7,692.3	$7,255.5	$7,842.2	$6,577.0	$5,005.5	$6,423.5	$3,809.5		
Amortisation	**$'000**	**$20,109.2**	$1,792.6	$2,021.2	$2,050.9	$2,142.0	$2,017.0	$2,061.6	$1,825.0	$1,845.0	$1,447.3	$1,012.6	$1,189.0	$705.1		
Total costs	**$'000**	**$650,399.2**	**$41,663.6**	**$67,731.6**	**$69,606.7**	**$70,282.4**	**$68,505.8**	**$62,969.3**	**$58,813.3**	**$54,354.8**	**$46,679.4**	**$42,046.2**	**$42,035.9**	**$25,710.2**	**$0.0**	**$0.0**
Gross profit	**$'000**	**$374,709.4**	**$62,853.9**	**$39,917.5**	**$40,288.2**	**$44,796.9**	**$44,930.7**	**$43,182.4**	**$24,707.5**	**$27,382.6**	**$17,622.2**	**$8,419.7**	**$16,905.0**	**$3,702.9**		
Expenses																
Corporate costs	**$'000**	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Royalty & mine taxes	**$'000**	**$91,262.1**	$11,633.2	$10,188.0	$9,514.7	$10,097.5	$9,766.4	$9,512.8	$7,092.5	$7,321.9	$5,379.6	$3,546.4	$4,836.2	$2,372.9		
ARO costs	**$'000**	**$8,756.7**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$1,741.3	$5,223.9	$1,095.4	$696.2
Earnings before income tax	**$'000**	**$274,690.5**	**$51,220.7**	**$29,729.5**	**$30,773.5**	**$34,699.4**	**$35,164.3**	**$33,669.6**	**$17,614.9**	**$20,060.8**	**$12,242.5**	**$4,873.3**	**$10,327.5**	**-$3,893.9**	**-$1,095.4**	**-$696.2**
Tax payable																
Book Depreciation	**$'000**	**$100,160.3**	$4,662.1	$8,770.7	$8,936.4	$9,759.7	$9,449.7	$10,551.1	$9,786.9	$10,403.0	$8,584.6	$6,412.0	$8,058.8	$4,785.2		
Tax Depreciation	**$'000**	**-$100,086.6**	-$6,951.4	-$8,463.7	-$8,676.1	-$9,489.7	-$9,195.4	-$10,291.2	-$9,556.9	-$10,170.3	-$8,402.2	-$6,284.3	-$7,909.2	-$4,696.4		
Taxable income	**$'000**	**$276,555.8**	**$48,931.5**	**$30,036.5**	**$31,033.8**	**$34,969.4**	**$35,418.6**	**$33,929.5**	**$17,845.0**	**$20,293.4**	**$12,425.0**	**$5,001.0**	**$10,477.1**	**-$3,805.1**	**$0.0**	**$0.0**
Regular corporate tax rate (less incentive)	**$'000**	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%		
Tax payable	**$'000**	**$56,072.2**	**$9,786.3**	**$6,007.3**	**$6,206.8**	**$6,993.9**	**$7,083.7**	**$6,785.9**	**$3,569.0**	**$4,058.7**	**$2,485.0**	**$1,000.2**	**$2,095.4**	**$0.0**	**$0.0**	**$0.0**
Undiscounted cashflow pre-tax	**$'000**	**$307,213.4**	$40,140.4	$19,596.3	$28,272.2	$40,478.0	$42,275.2	$39,054.0	$24,735.9	$28,042.4	$18,987.3	$9,731.9	$17,070.5	$620.7	-$1,095.4	-$696.2
Undiscounted cashflow post-tax	**$'000**	**$251,141.2**	**$30,354.1**	**$13,589.0**	**$22,065.5**	**$33,484.1**	**$35,191.5**	**$32,268.1**	**$21,166.9**	**$23,983.7**	**$16,502.3**	**$8,731.7**	**$14,975.1**	**$620.7**	**-$1,095.4**	**-$696.2**

Item 2 INTRODUCTION

2.1 Purpose of this Technical Report

This Technical Report on the Çayeli Operations (the Property or Operations) has been prepared by Qualified Persons (QPs) Richard Sulway, Michael Lawlor and Andy Briggs of First Quantum Minerals Ltd (FQM, the issuer or the Company).

This purpose of this Technical Report is to document Mineral Resource and Mineral Reserve updates, specifically in relation to a newly defined copper and zinc deposit adjacent to the current mining operations at Çayeli.

2.2 Terms of reference

This Technical Report has been written to comply with the reporting requirements of the Canadian Securities Administrators' National Instrument 43-101 *'Standards of Disclosure for Mineral Projects' and 'Form 43-101F1 Technical Report'* (NI 43-101 or the Instrument, 2011).

The effective date for the Mineral Resource and Mineral Reserve estimates is the 30th of April 2025.

2.3 Qualified Persons and authors

The Mineral Resource estimate was prepared by Richard Sulway of FQM who meets the requirements of a QP according to his Certificate of a Qualified Person attached in Item 28.

The Mineral Reserve estimate and mining aspects of this report were prepared and documented under the direction of Michael Lawlor of FQM who also meets the requirements of a QP according to his Certificate of a Qualified Person attached in Item 28.

Metallurgical testing, mineral processing and recovery aspects of this report were addressed by Andy Briggs of FQM, who additionally meets the requirements of a QP according to his Certificate of a Qualified Person attached in Item 28.

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

Table 2-1 QP details

Name	Position	NI 43-101 Contribution
Richard Sulway	**Group Principal Geologist, Mine and Resources**	**Author and Qualified Person**
BAppSc Hons (Geology), MSc, MAusIMM(CP)	FQM (Australia) Pty Ltd	Items 1,7 to 12, 14, 25 and 26
Michael Lawlor	**Mine Technical Advisor**	**Author and Qualified Person**
BEng Hons (Mining), MEngSc, FAusIMM	FQM (Australia) Pty Ltd	Items 1 to 6, 15, 16 and 18 to 26
Andrew Briggs	**Group Consulting Metallurgist**	**Author and Qualified Person**
BSc (Eng), ARSM, FSAIMM	FQM (Australia) Pty Ltd	Items 13, 17 and 21 (in respect of processing and G&A costs only), 25 and 26

2.4 Sources of information

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

Mining, metallurgy, processing and economic sources of information were gathered preparatory to, during and following the QP site visits.

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

2.5 Personal inspections

The QPs have each visited the site and carried out personal inspections at the times and for the durations as follows:

- Mr Richard Sulway visited the Çayeli Operations between the 26th of June and the 13th of July 2024. The key purpose of his site visit was to become familiar with, and verify the current practices and procedures used by the ÇBI geology department and associated entities including the drilling contractors and mine laboratory. An added purpose to this visit was to start collating the relevant documentation and data for the planned Mineral Resource estimate. Mr Sulway also inspected the underground workings, site laboratory, diamond core processing facilities, and core storage warehouses.
- Michael Lawlor visited the Çayeli Operations between the 18th of August 2024 and the 2nd of September 2024, and again between the 30th of April 2025 and the 15th of May 2025. At these times, Mr Lawlor inspected the underground mine workings and held numerous discussions with the mine technical and Operations staff. Whilst on site, Mr Lawlor also reviewed mine plans and schedules related to the proposed production from Main Orebody remnant areas and the newly defined South Orebody.
- Andy Briggs first visited the Çayeli Operations between the 6th and 9th of October 2013 for familiarisation of the metallurgical operations following FQM purchase of the Operations. He visited again between the 12th and 14th of May 2014, and between 23rd and 25th June 2023, to review operating results. Mr Briggs' most recent visit was between the 18th and 25th of August 2024, to discuss plans for the processing of the South Orebody plant feed. These discussions covered metallurgical testwork, anticipated copper and zinc recoveries, concentrate grades, in addition to operating and capital costs pertaining to the treatment of South Orebody plant feed. A review of the conditions of equipment in the existing plant was also carried out, in addition to the plans for equipment replacement to support the life of mine capital cost projections.

2.6 Conventions and definitions

Reference in this Technical Report to dollars or $, relates to United States dollars. Copper 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 and gold production is reported in (troy) ounces.

The conventional chemical abbreviation for copper of Cu is used throughout this report, whilst the abbreviation for zinc is Zn, silver is Ag and gold is Au. ASCu is used to denote Acid Soluble Copper and TCu is used to denote Total Copper.

Dollar references followed by "M" mean millions of dollars and followed by "B" mean billions of dollars.

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

Table 2-2 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, except for certain information included in Item 20 and in Item 22.

The permitting and environmental approvals information in Item 20 has been provided by the ÇBI environmental team and reviewed by the Company's environmental experts. The authors of this Technical Report have relied on this information for the purposes of providing an opinion on factors and risks which may affect access or title to the continuing operations.

The information in Item 22, provided by the Company's internal taxation advisors, relates to the applicable corporate tax rate in Türkiye, the estimated taxable income and the tax to be paid. The modelled taxes, and royalty payments, are net of advised VAT (value added tax) refunds. The authors of this Technical Report have relied on this information for the purposes of the economic analysis in Item 22.

Item 4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property description

The Çayeli Operations commenced producing copper and zinc concentrates in late 1994. Feed to the conventional flotation processing facility has been sourced from an underground mining operation featuring a primary mining method of sequenced transverse and longitudinal long hole stoping, with plant tailings paste and waste rock backfill.

From 1994, ore processing had risen to a peak of about 1.3 Mtpa in 2016 but declined thereafter as the original Main Orebody approached depletion. The remaining plant feed from this original source, mostly in remnant areas of the mine, is sufficient for a further five years of intermittent underground production. A newly defined resource in the adjacent footwall of the Main Orebody, and referred to as the South Orebody, is now planned as a replacement plant feed source continuing through to 2036.

The ore processing plant consists of conventional crushing, grinding, selective flotation and pressure filtration facilities. The copper and zinc concentrates, containing silver and gold as by-products, are transported by road to the port of Rize, which is 26 km away. Plant tailings are partly used to fill the underground voids after mixing with cement at a paste fill facility. The balance is discharged at depth into the Black Sea.

Figure 4.1 is a view looking towards the southwest over the Çayeli Operations.

Figure 4.1 View over the Çayeli Operations, Rize Province, Türkiye

The Çayeli Operations draw electrical power from the national grid and extract processing water from the adjacent Büyükdere River, supplemented with abstraction from several ground water wells.

4.2 Location of the Operations

The Operations are located 8 km south of the town of Çayeli and approximately 18 km from the coastal city of Rize, which is on the Black Sea coast of north-eastern Türkiye (Figure 4.2 and Figure 4.3). The mine and processing facilities are situated on the banks of the Büyükdere River, directly across from the small town of Madenli (Figure 4.4).

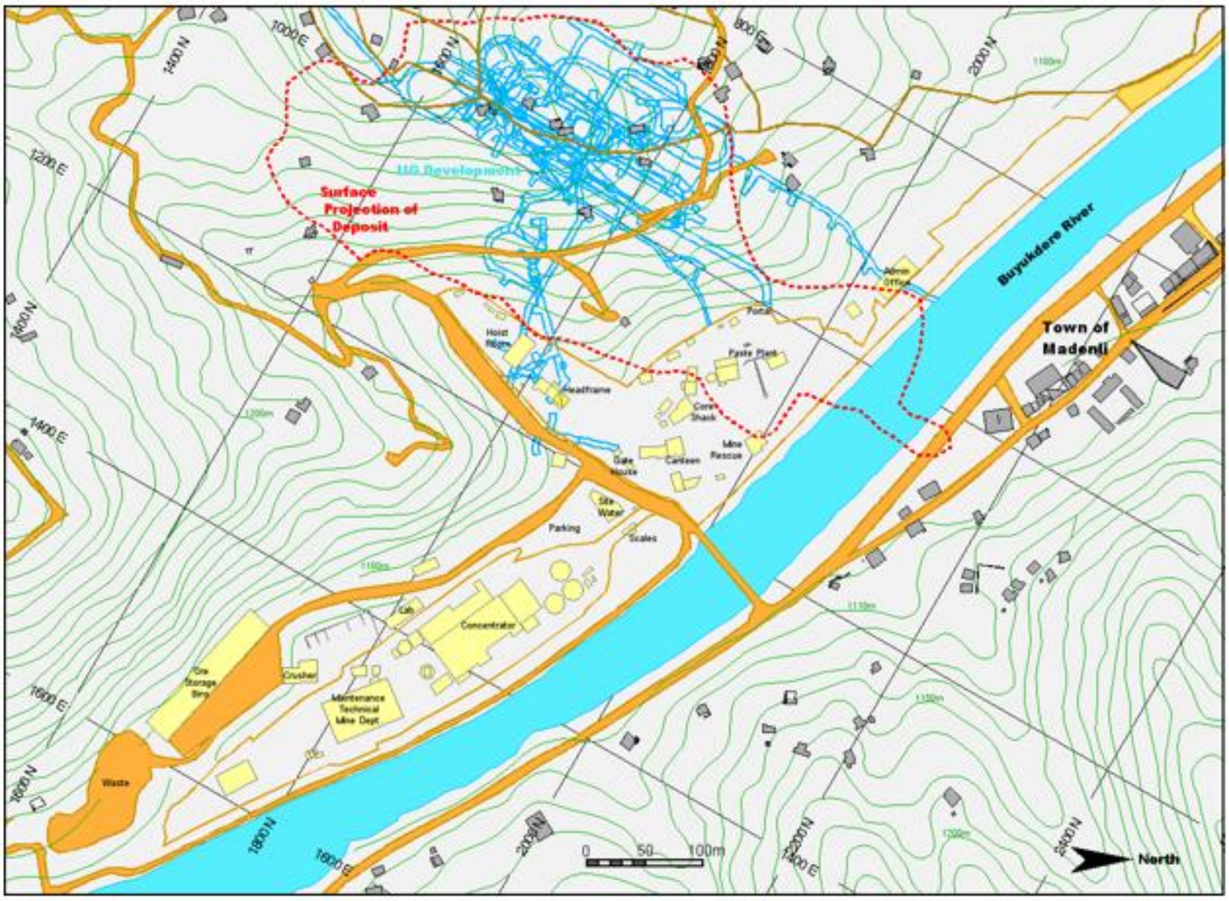
The plant site is positioned at about 100 m above sea level. The geographical coordinates of the location are 41° 2'26.06" North, 40°45'59.17" East.

Access to the Operations site is via the Karadeniz Highway which extends along the Black Sea coast from Samsun to the Georgian border. A sealed road connection exists between the highway and Madenli.

Figure 4.2 Location of the Çayeli Operations, Türkiye

Figure 4.3 Location of the Çayeli Operations, Rize Province, Türkiye

Figure 4.4 Site plan, Çayeli Operations, Rize Province, Türkiye

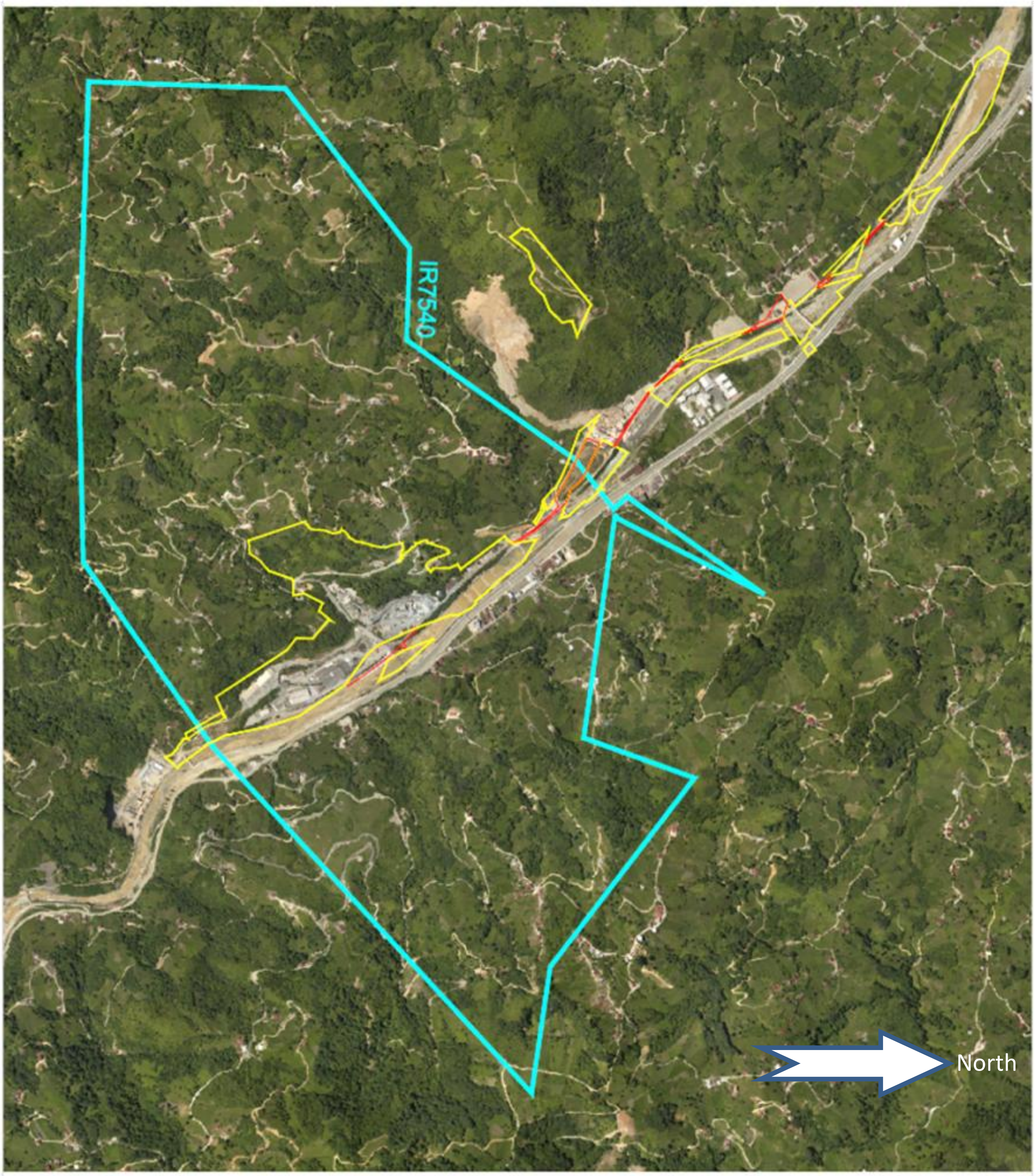


4.3 Mineral tenure and Property area

The mine and surrounding facilities are situated within a single operating licence, IR 7540, the area of which is 334 ha. The licence area is shown in Figure 4.5 and the expiration date is 29th July 2044.

Figure 4.5 shows several red and yellow polygons outside of the licence area, enclosing waste dumping and ore stockpiling sites. The yellow polygons are lands owned by ÇBI, whilst the red polygons are leased from the government. An approximate 200 ktonne mineralised waste dump emanating from South Orebody access development is located within one of these external areas.

Figure 4.5 IR 7540 licence area (source: ÇBI)



The Operations site is located in the middle of three administrative districts which are known as the Madenli Central District, the Çamlica District, and the Maden Districts.

4.4 Surface rights

ÇBI is a wholly owned subsidiary company of FQM. *Eti Maden İşletmeleri Genel Müdürlüğü* (*Eti),* a company wholly owned by the Government of Türkiye, holds the operating license for the mine and has leased it to ÇBI.

4.5 Royalties, payments and agreements

ÇBI pays a mine tax to the Government of Türkiye calculated as a percentage of the sales value of copper and zinc concentrate production, on a sliding scale royalty rate depending on the copper and zinc prices. A 25% mining tax escalation also applies, and the tax payable is net of operating costs and 38% of the calculated annual depreciation value.

In addition, *Eti* is entitled to a royalty predicated on 7% of ÇBI's net income (excluding depreciation), whilst a municipal tax is also payable on the same basis, at a rate of 0.2%.

4.6 Permitting

In March 2021, ÇBI renewed its five year *Integrated Environmental Permit* that governs the environmental requirements for integrated plants, which will be valid until March 2026. ÇBI's permit renewal application will need to be submitted before the 13th of September 2025. In October 2017, ÇBI renewed the permit for operations at the Rize Port concentrate storage and handling facility, which is valid until 2027.

For that proportion not used as paste fill in the underground mine, process plant tailings are discharged into the Black Sea in a process referred to as Deep Sea Tailings placement (DST). The Turkish government published a Mine Waste Regulation in June 2015, and this was subsequently enacted in June 2017, thereby permitting ÇBI to discharge tailings into the sea. The DST activities will need to be addressed in the Environmental Permit renewal application before which a "special expertise commission" must be appointed to review the extensive discharge monitoring records and the plan for continued discharge.

The Ministry of Environment and Urbanisation (the E&U Ministry) has advised that "*the Company's operations can continue within the scope of the current environmental permit process*". During the 2025 environmental permit renewal process, the Ministry has reiterated its earlier position in writing, stating: *"…from the perspective of waste management legislation, there is no objection to submitting an environmental permit application."*

Further information on the environmental licencing and permitting status is provided in Item 20.

4.7 Environmental liabilities

There are no known pre-existing environmental liabilities associated with the Property.

The primary future environmental liabilities at the Çayeli Operations will arise at closure and are related to the decommissioning and dismantling of the process plant and ancillary infrastructure, and the rehabilitation of the mine site and related facilities.

4.8 Factors and risks which may affect access or title

To the extent known, there are no significant factors or risks that may affect access, title to, the rights to or ability of ÇBI to continue operations at Çayeli.

The Company does not anticipate any challenge to the renewal of the *Environmental Permit* considering the monitoring programme that is in place, and there being no evidence of an adverse change in water quality to date. Nevertheless, the QP's are of the opinion that there remains some uncertainty as to when and if the formal permit renewal will eventually be forthcoming.

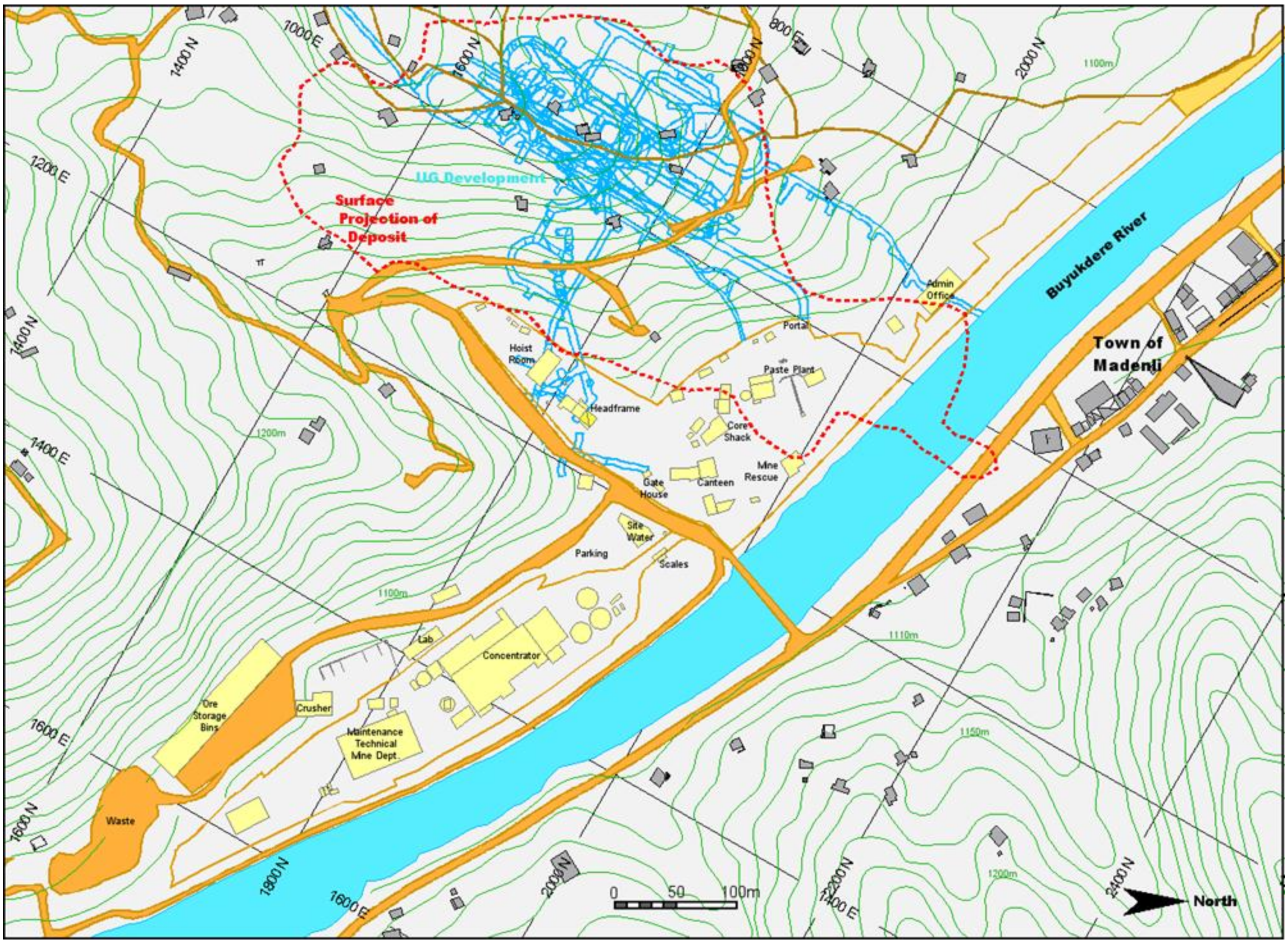
Item 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Accessibility

The Çayeli Operations are approximately 8 km inland via sealed road from the Black Sea coastal town of Çayeli. The Operations site is located adjacent to the town of Madenli (see Figure 5.1). The Karadeniz Highway provides an east-west transportation route along the Black Sea coast, from Samsun to the Georgian border. The port city of Rize is located 20 km west of the town of Çayeli.

An international airport is located approximately 100 km west of the town of Çayeli in the city of Trabzon, whilst a domestic airport is located nearby at Rize.

Figure 5.1 Location map of the Çayeli Operations and the nearby town of Madenli



5.2 Climate

The prevailing climate is moderated by the nearby Black Sea. Summers are typically hot and humid, whilst winters are generally wet. Rainfall is abundant in this area and averages over 2.5 m per year.

5.3 Physiography

The Operations site is located in the foothills of the Kaçkar Mountains which extend along the eastern portion of the southern Black Sea coast. This is a rugged mountain range with peaks reaching over 4,000 m in elevation.

There have been landslides in and around the Operations site, attributed primarily to the prevailing high rainfall, steep topography and clearing for tea plantations.

An appreciation for the physiography of the site can be gained from Figure 5.2. This photograph also shows the surface projections of the Main and South Orebodies.

Figure 5.2 Physiography of the Operations site

5.4 Vegetation

As a result of the relatively high rainfall, the landscape around Çayeli is typically lush with vegetation, including a wide variety of flowering plants such as rhododendrons and azaleas. Tea plantations are numerous throughout the region.

5.5 Population centres

The town of Madenli is located directly across the Büyükdere River from the Çayeli Operations. As at 2021, the population of Madenli was recorded as 2,700 people and that of Çayeli was 24,600 people (Wikipedia, 2021). A larger population centre at Rize, which is 28 km from Madenli, has a population of 120,000 people (Wikipedia, 2021).

5.6 Infrastructure

All the following surface infrastructure and facilities are located within the IR 7540 licence area:

- the original mine shaft collar, (backfilled) hoisting shaft and egress ladderway
- the existing mine decline portals
- mine ventilation shaft collars and exhaust fans
- the processing plant and associated primary crushers and ore storage
- the process water pond
- an assay laboratory
- an administration office, canteen and infirmary (with mine rescue)

- mine and plant workshop facilities
- a warehouse
- the paste fill plant
- a water pumping station

Electrical power is provided by a 31.5 kV line connected to the national grid. Process water is supplied from the Büyükdere River and from several ground water wells.

There is a mixing tank facility located on the coast, near the mouth of the Büyükdere River, from where tailings pumped from the processing plant is discharged via pipeline extending for 3 km out into the Black Sea.

5.7 Sufficiency of surface rights and concession extents

Considering the extents of the IR 7540 licence and the lands that are either leased or owned by ÇBI (Figure 4.5), there is sufficient area to accommodate the current and proposed expanded operations.

Item 6 HISTORY

6.1 Prior ownership

In 1981, ÇBI was created as a joint venture between *Eti,* Phelps Dodge Corporation (Phelps Dodge), and *Gama Endüstri A.Ş*. (Gama).

In 1988, Phelps Dodge sold its 49% share to Metall Mining (which later became Inmet Mining Corporation (Inmet)). In 2002, Inmet acquired Gama's 6% interest and, in 2004, Inmet purchased *Eti'*s 45% interest.

The Company acquired Inmet in 2013 and now owns 100% of the Operations. ÇBI is a wholly owned subsidiary of the Company.

6.2 Exploration and development history

The Eastern Pontides orogenic belt has a long mining history (thousands of years) as its proximity to the Black Sea coast gave early prospectors and miners good access to the coastal volcanogenic massive sulphide deposits. As an example, mining of the Damar deposit in the vicinity of the town of Murgul, about 30 km north-east of Çayeli, dates to 2000 BC (Akçay and Moon, 2004).

The main body of work which led to the development of the Çayeli mine started in 1967 when the Turkish Mineral Research and Exploration Institute (MTA) started exploration work in the area. Mining and exploration activities at Çayeli took place in two distinct periods pre-1967, and 1967 to 2025, leading to the development of the mine and later discovery of the South Orebody.

6.2.1 Pre 1967

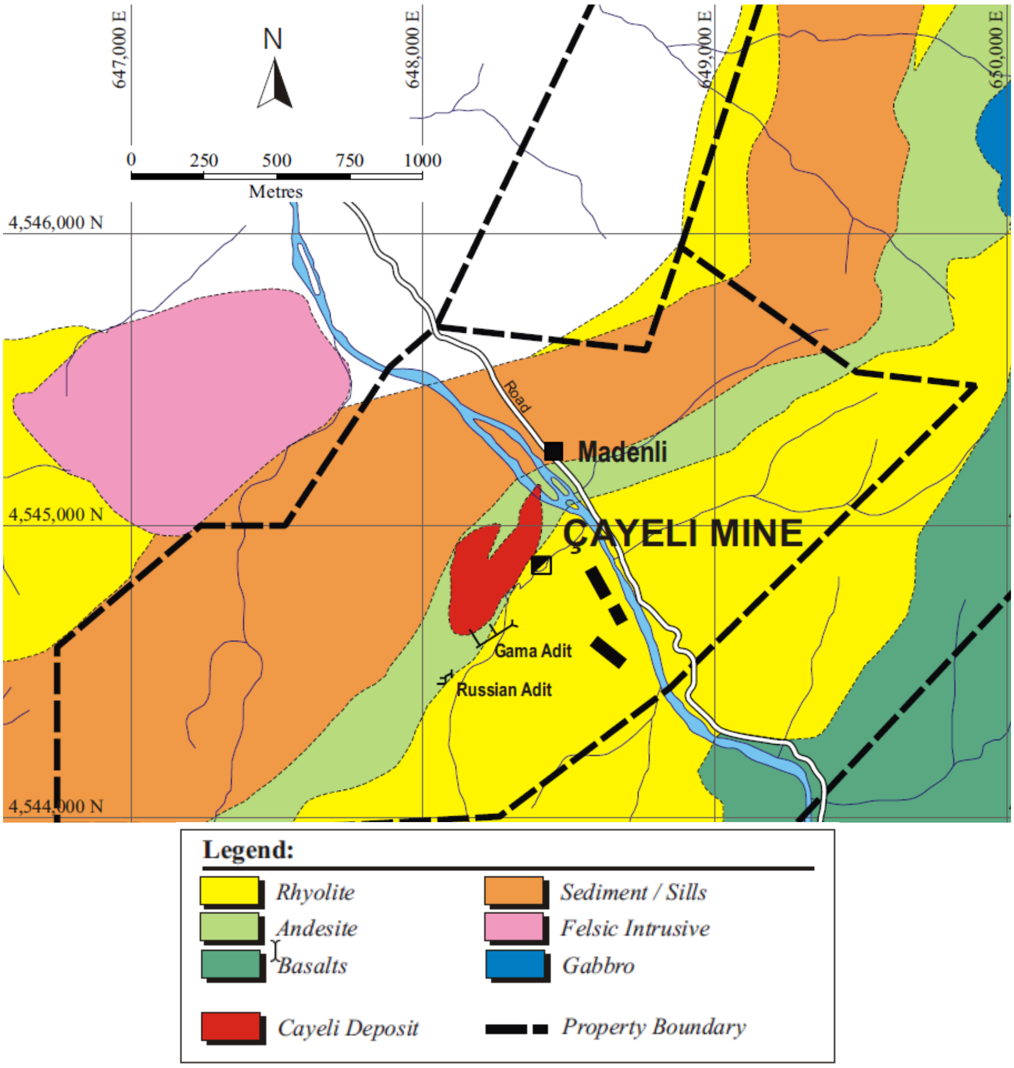
There is very little documentation describing mining activities in the region around the Çayeli mine prior to the involvement of the MTA. Various small-scale shafts and adits were completed between 1900 and 1955 and there are indications (crude smelting) of some minor production taking place (Yumlu, 2001, Karakuş 2008). It is likely that small scale mining/exploration activity in the area dates back hundreds if not thousands of years.

6.2.2 1967 to 2025

In 1967, the MTA undertook geophysical surveys, drilling (22 holes) around an old adit dating from the early 1900s and known as the "Russian Adit". The adit is located about 400 m south of the Çayeli hoisting shaft (now backfilled). The MTA also excavated a new adit into the sulphide mineralisation about 200 m to the north-east of the earlier drive, known as the "Gama adit". A map showing the location of these adits relative to the mine infrastructure is shown in Figure 6.1.

In the 1970s the area was geologically mapped at 1:1000 and 1:10000 scale (Altun, 1976).

Figure 6.1 2006 Location map showing the positions of the two old adits relative to the mine (source: Inmet)



In 1982, Phelps Dodge started work on the Çayeli deposit and initially, five diamond holes were drilled to check the validity of data from the MTA drilling which had intersected mineralisation. Arising from the drilling and subsequent assaying results, a new adit (the Gama Adit) and crosscut totalling 643 m of development was completed in order to collect a bulk sample for metallurgical testwork (Inmet, 2008).

Additional drilling was then completed both from surface and underground on twelve section lines at 40 m spacing. From the results of this additional drilling and sample assaying, it became apparent that the initial bulk sample was not truly representative, and another crosscut was developed to produce a more representative bulk sample for metallurgical testing (Inmet, 2008).

Further underground work and metallurgical testing were done between 1988 and 1991, and in 1990 a feasibility study was completed by Bechtel, Canada. Following on from this study, it was decided to develop the mine, with construction work on the concentrator starting in 1992. In March 1993 a portal was established and the driving of the hangingwall decline was commenced. Mill commissioning started in August 1994 and the first concentrate production occurred in November 1994.

The bulk of the drilling completed after 1994 was for underground grade control purposes. Detailed drilling of the South Orebody from surface and from underground started in 2021 following its discovery.

6.3 Previous Mineral Resource and Mineral Reserve estimates

The most recent formal Mineral Resource and Mineral Reserve estimates were prepared by Inmet in 2006, in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition and Standards on Mineral Resources and Mineral Reserves (CIM, 2019). The Mineral Resource and Mineral Reserve estimates and statements were published in the Inmet NI 43-101 Technical Report of the 30th of March 2006 (RPA, 2006) and are reproduced in Table 6-1 and Table 6-2, respectively.

Table 6-1 Mineral Resource at 31st December 2005, reported using a $35/t (ore) NSR[3] value cut-off

Category	Tonnes Mt	Cu %	Zn %	Au g/t	Ag g/t	Pb %	NSR $/t
Measured	1.91	3.08	3.09	0.50	22	0.18	57
Indicated	2.65	3.05	2.55	0.44	20	0.15	54
Meas.+Ind.	**4.56**	**3.06**	**2.78**	**0.46**	**21**	**0.16**	**55**
Inferred	1.07	3.34	6.29	N/A	N/A	N/A	N/A

Table 6-2 Mineral Reserve at 31st December 2005, reported using a $35/t (ore) NSR value cut-off

Category	Tonnes Mt	Cu %	Zn %	Au g/t	Ag g/t	Pb %	NSR $/t
Proven	4.70	3.77	5.85	0.59	44	0.30	73
Probable	6.90	3.57	5.88	0.53	52	0.36	69
Prov.+Prob.	**11.60**	**3.65**	**5.87**	**0.56**	**49**	**0.34**	**70**

Notes to the Mineral Resource statement were as follows:

- The Mineral Resource estimate was based on metal prices of $1.10/lb Cu, $0.55/lb Zn, $5.60/oz Ag and $450/oz Au
- Measured and Indicated Mineral Resources excluded Proven and Probable Mineral Reserves
- Mineralisation in the Russian adit (Inferred Resource) was not assayed for Au and Ag

Notes to the Mineral Reserve statement were as follows:

- The Mineral Reserve estimate was based on metal prices of $1.10/lb Cu, $0.55/lb Zn, $5.60/oz Ag and $450/oz Au

The 2006 Mineral Resource and Mineral Reserve estimates are now superseded (refer to Item 14 and Item 15, respectively).

6.4 Production history

The early production history for the period up to December 2005, is listed in Table 6-3.

[3] NSR is an acronym for Net Smelter Return. The meaning of this term is explained in Item 15.

Table 6-3 ÇBI production history up to 2005

Year	Feed Tonnes	Feed Grade		Recovery		Recovered		Concentrate		Concentrate Grade		NSR C/off
	(t)	Cu (%)	Zn (%)	Cu (%)	Zn (%)	Cu (t)	Zn (t)	Cu (t)	Zn (t)	Cu (%)	Zn (%)	$/t
1993												
1994	*140,000*	*4.40*	*8.10*									
1995	485,815	3.66	7.55	78.4	73.3	13,946	26,893	61,548	54,575	22.6	49.3	
1996	654,448	3.53	8.54	74.8	71.0	17,273	39,687	80,678	80,295	21.4	49.4	
1997	761,608	5.00	6.91	81.5	66.3	31,051	34,865	128,876	71,915	22.7	49.4	
1998	707,992	4.56	6.63	84.3	68.8	27,216	32,295	113,393	65,777	24.0	49.1	
1999	896,749	5.11	5.34	87.5	68.2	40,091	32,644	158,001	64,759	25.4	50.4	
2000	860,763	4.87	4.45	89.2	67.8	37,371	25,974	148,366	51,370	25.2	50.6	
2001	816,480	4.57	4.49	85.0	65.9	31,712	24,152	130,767	47,849	24.2	50.5	
2002	895,423	4.20	5.13	84.3	68.9	31,692	31,626	128,570	62,002	24.6	51.0	
2003	927,892	4.17	5.05	84.6	67.9	32,738	31,817	132,815	62,321	24.7	51.1	
2004	765,329	3.91	5.75	82.7	70.7	24,775	31,113	104,479	61,294	23.7	50.8	
2005	833,638	3.84	6.74	81.0	74.4	25,957	41,816	113,761	82,975	22.8	50.4	
TOTAL	**8,606,138**	**4.35**	**5.90**	**83.8**	**69.5**	**313,823**	**352,882**	**1,301,253**	**705,131**	**24.0**	**50.1**	

The production history from January 2006 (i.e., following the December 2005 Mineral Reserve statement) is listed in Table 6-4 and amounts to 19.6 Mt of plant feed yielding a total of 2.2 Mt of copper concentrate at an average grade of 20.3% Cu, and 0.9 Mt of zinc concentrate at an average grade of 48.5% Zn. The 2025 year to date production history is listed in Table 6-5.

Table 6-4 ÇBI production history 2006 to 2024

Year	Feed Tonnes	Feed Grade		Recovery		Recovered		Concentrate		Concentrate Grade		NSR C/off
	(t)	Cu (%)	Zn (%)	Cu (%)	Zn (%)	Cu (t)	Zn (t)	Cu (t)	Zn (t)	Cu (%)	Zn (%)	$/t
2006	932,873	3.87	5.68	81.6	71.7	29,438	38,025	126,855	75,407	23.2	50.4	46.0
2007	1,046,621	3.83	6.19	79.5	70.5	31,815	45,664	142,331	91,175	22.4	50.1	46.0
2008	1,108,590	3.68	6.10	78.7	70.4	32,151	47,634	149,045	96,249	21.6	49.5	56.0
2009	1,151,018	3.28	6.24	77.1	70.5	29,093	50,646	142,844	101,885	20.4	49.7	56.0
2010	1,147,083	3.21	6.29	76.4	70.6	28,119	50,925	138,417	104,261	20.3	48.8	56.0
2011	1,195,472	3.23	5.97	75.9	67.5	29,334	48,156	146,635	98,834	20.0	48.7	65.0
2012	1,218,490	3.35	5.21	78.6	65.1	32,060	41,290	160,135	84,815	20.0	48.7	65.0
2013	1,332,810	3.10	4.92	77.0	66.6	31,786	43,678	160,868	89,529	19.8	48.8	65.0
2014	1,341,067	2.79	4.35	80.8	63.6	30,253	37,092	155,248	77,173	19.5	48.1	65.0
2015	1,228,958	2.46	2.98	80.7	53.4	24,400	19,563	128,176	42,302	19.0	46.2	65.0
2016	1,285,271	2.33	1.63	88.0	38.9	26,383	8,130	131,339	17,179	20.1	47.3	65.0
2017	943,308	2.01	1.06	89.6	33.1	17,013	3,307	83,780	6,974	20.3	47.4	55.0
2018	1,006,737	2.28	1.46	87.5	31.9	20,077	4,708	105,538	10,239	19.0	46.0	55.0
2019	915,885	2.09	1.55	88.1	38.8	16,886	5,498	88,827	12,135	19.0	45.3	55.0
2020	776,650	2.04	1.62	86.0	36.6	13,646	4,599	72,195	10,433	18.9	44.1	55.0
2021	815,026	1.96	1.81	86.0	47.8	13,722	7,037	70,809	17,074	19.4	41.2	55.0
2022	720,208	1.71	1.14	87.8	35.8	10,832	2,948	55,197	7,488	19.6	39.4	55.0
2023	746,802	1.62	1.09	90.4	43.4	10,949	3,530	54,184	8,151	20.2	43.3	55.0
2024	691,328	1.77	1.11	90.1	37.7	10,992	2,896	53,330	7,209	20.6	40.2	55.0
TOTAL	**19,604,198**	**2.75**	**3.75**	**81.4**	**63.2**	**438,950**	**465,328**	**2,165,753**	**958,513**	**20.3**	**48.5**	**58.1**

Table 6-5 ÇBI production history for 2025

Month	Feed Tonnes	Feed Grade		Recovery		Recovered		Concentrate		Concentrate Grade		NSR C/off
	(t)	Cu (%)	Zn (%)	Cu (%)	Zn (%)	Cu (t)	Zn (t)	Cu (t)	Zn (t)	Cu (%)	Zn (%)	$/t
Jan-25	50,222	1.34	0.45	89.9	3.8	605	9	2,928	26	20.7	32.8	55.0
Feb-25	67,542	1.14	0.38	91.4	27.4	704	70	3,156	151	22.3	46.5	55.0
Mar-25	65,917	1.67	0.66	92.1	34.3	1,014	149	4,687	394	21.6	37.9	55.0
Apr-25	64,931	1.82	0.17	92.1	9.4	1,088	10	4,717	24	23.1	45.0	55.0
TOTAL	**248,612**	**1.50**	**0.41**	**91.5**	**23.2**	**3,410**	**238**	**15,488**	**595**	**22.0**	**3.5**	**55.0**

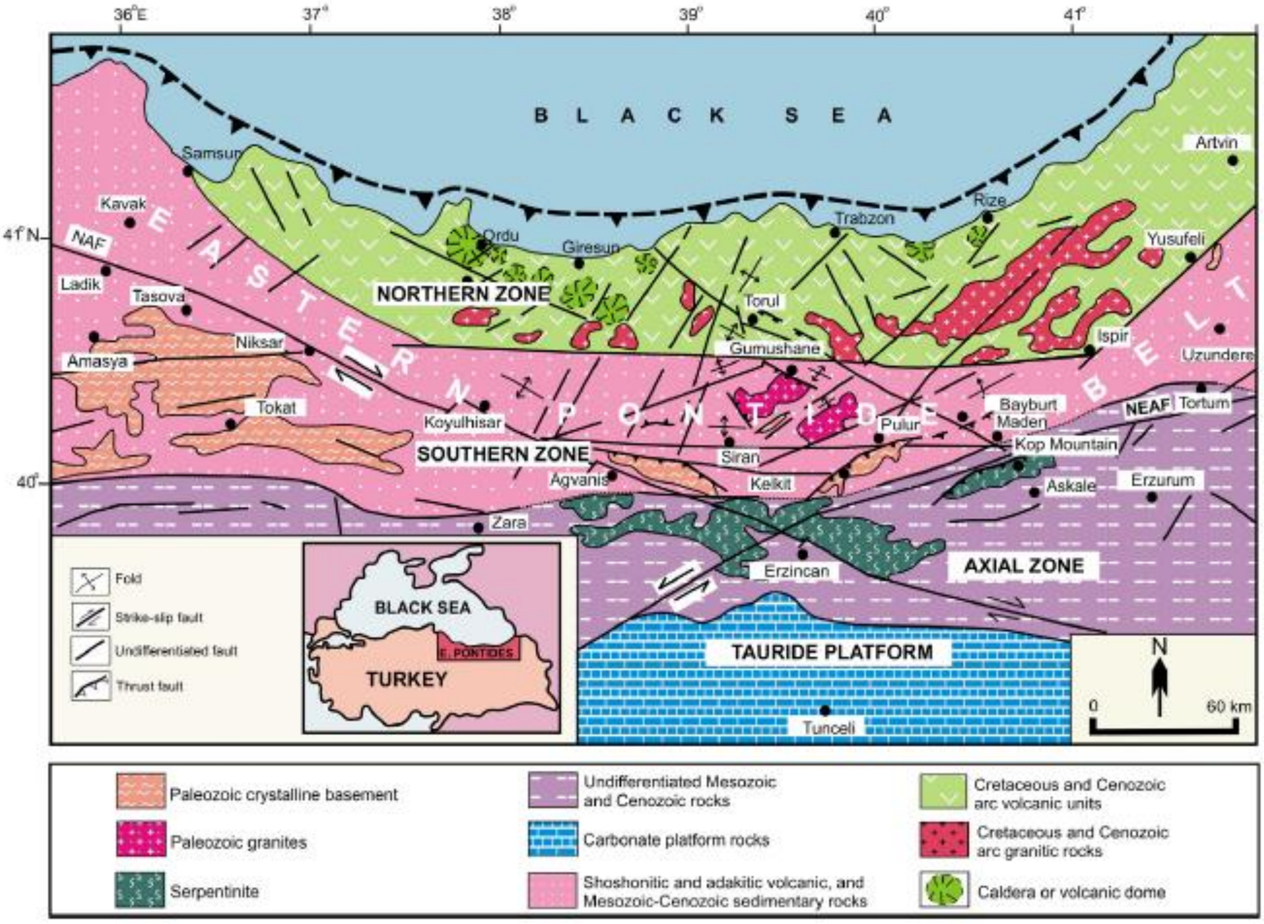
Item 7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional geology

The Çayeli mine is located in the Eastern Pontides Orogenic Belt ("EPOB"). The Pontıdes orogenic belt extends along the northern coast of Türkiye from just east of Istanbul to the border with Georgia in the east, a distance of over 1,100 km. The EPOB which hosts the mine strata runs parallel to the Black Sea coast for about 500 km, is about 200 km wide and forms part of the Alpine-Himalayan Belt. Many of the EPOB volcanic rocks are related to the convergence of the Eurasia and Gondwanaland plates (Kaygusuz *et al* 2015).

The geology is dominated by Late Cretaceous and to a lesser extent Tertiary (Eocene), well preserved fossil continental margin arc system, composed of calc-alkaline and tholeitic volcanic rocks, and flysch-type sediments. The Cretaceous magmatism that built much of the EPOB was generated in a subduction (continental margin arc) related setting. A geology map of the EPOB is shown in Figure 7.1.

Figure 7.1 Geological and tectonic map of the Eastern Pontides Orogenic Belt (Eyuboglu et al 2006)



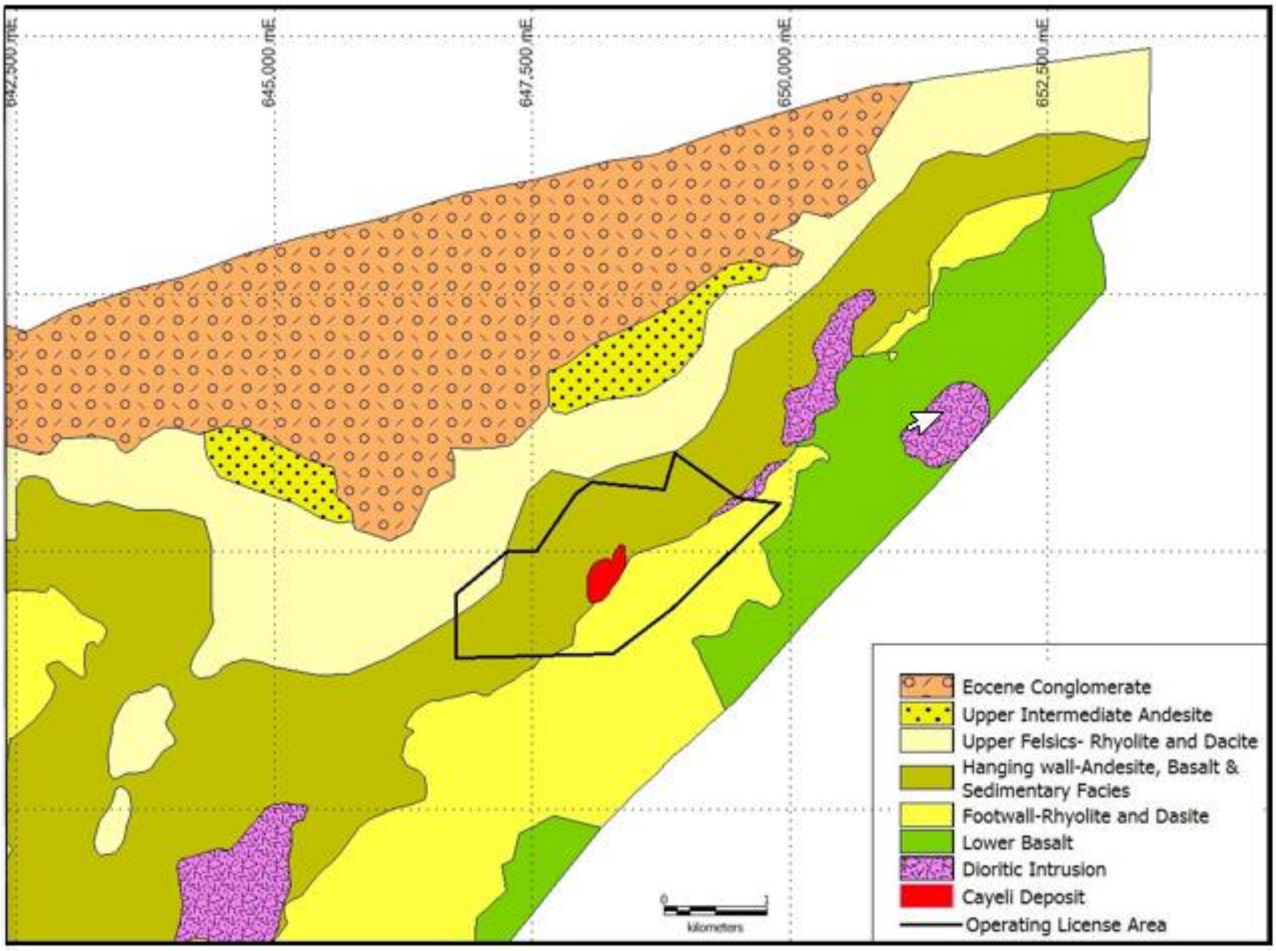
The EPOB can be divided into three zones (Northern, Southern and Axial zones) in recognition of the underlying geology and tectonic characteristics (Figure 7.1).

7.2 Local and Property geology

The main Çayeli mineralisation is located along the contact between a hanging wall of pyroclastics (tuffs) and basalts, and a footwall of rhyolite and felsic pyroclastic rocks. Hydrothermal alteration related to the formation of the deposit is restricted to the footwall stratigraphy and is in the form of clay (argillite) and chlorite.

A simplified geology map of the mineralisation (as defined in 1994) and surrounding area is shown in Figure 7.2. The South Orebody which is not shown in Figure 7.2, is located about 300 m from the current Main Orebody workings.

Figure 7.2 Çayeli, geology map (source: RPA, 2006)



7.3 Mineralisation

The Çayeli mineralisation is characterised by two main styles, i.e., massive sulphides and associated stockwork mineralisation. The key economic elements are Cu and Zn and to a lesser degree Ag as a by-product. Au is present but is very patchy, is limited in its distribution and is mostly low grade (<0.3 ppm). Au is not considered material to the economics of the Operation; similarly, Pb when present, has a very low tenor, typically <0.1%.

The main sulphides in the massive zones are pyrite, chalcopyrite, sphalerite and relatively minor amounts of bornite, galena and tetrahedrite. The massive sulphide zones are mostly concentrated along the contact zone between the footwall rhyolite and hangingwall mafics. Aside from a few mineralised small pods (probably due to faulting), the hangingwall basalts and associated intercalated mafic tuffs are essentially barren. The disseminated zones host mainly pyrite and chalcopyrite. Significantly lesser amounts of sphalerite and galena are located within the highly altered footwall rhyolite and lesser pyroclastics. Gangue minerals include barite, dolomite, quartz, sericite and kaolinite.

As is typical with volcanogenic massive sulphide (VMS) deposits, the Çayeli mineralisation is zoned in terms of Cu and Zn tenor. The lower grade Cu stockwork volumes are located adjacent to massive sulphide zones with relatively minimal overlap. A similar pattern is evident in the Zn mineralisation, however, the proportion of Zn rich stockwork in either orebody is relatively small.

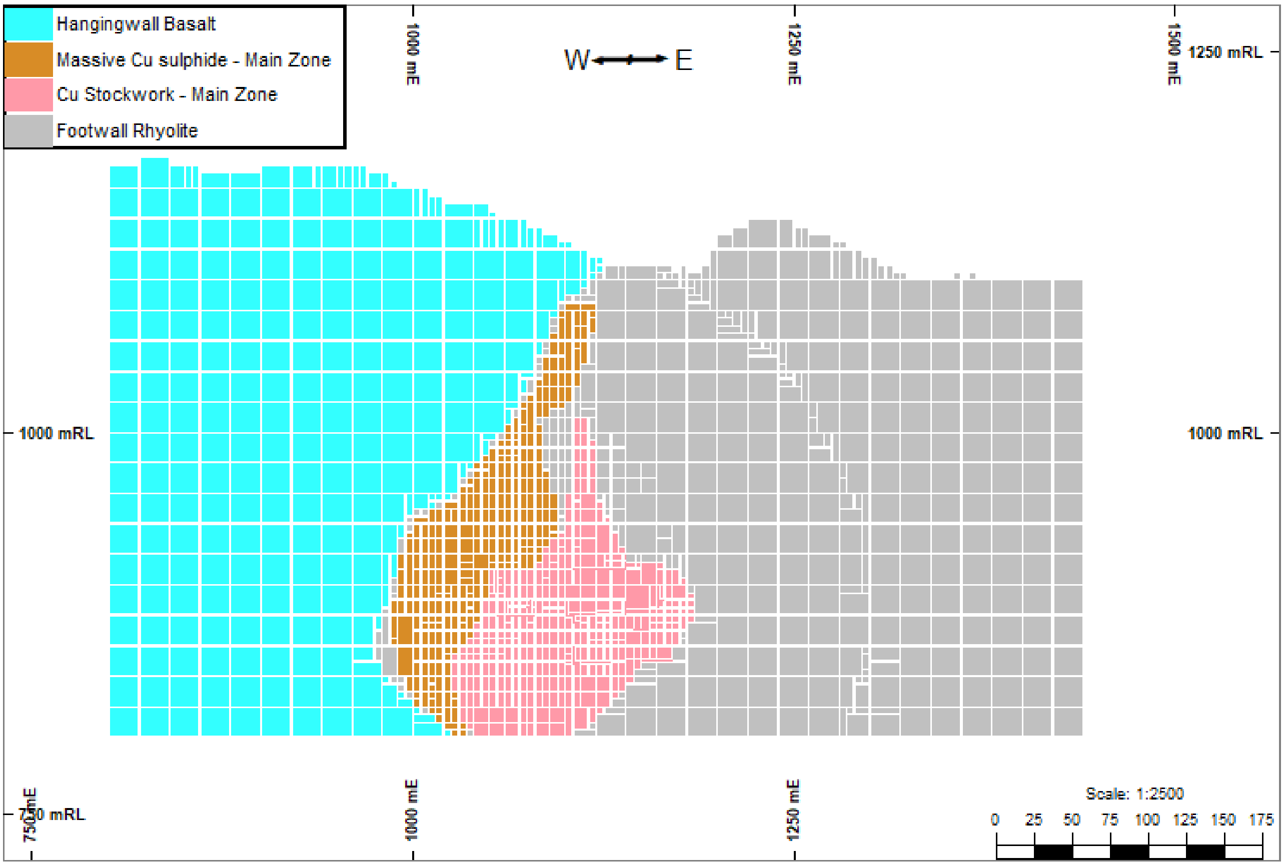
The general characteristics of the Main and South Orebody mineralisation are as follows:

1. The historic Main Orebody area consists of north-south striking, sub vertical lenses of massive sulphides. There is also a sub parallel adjacent zone of footwall stockwork to the east, with a strike length of about 900 m and dipping at about 60° steeply to the west. The mineralisation plunges at about 20° to the north and extends down dip for about 600 m before pinching out. The massive sulphide varies in thickness from a few m to 80 m, with an average thickness of about 30 m to 50 m. The strike and dip vary along the length of the ore body. The Cu/Zn massive mineralisation is largely coincident and varies in thickness (east-west) from about 1 m to 100 m, with the average thickness of about 30 m. The Cu stockwork mineralisation varies in thickness (east-west) from 1 m to 170 m with an average thickness of about 50 m. Zn rich stockwork is on average about 8 m thick. The ratio of massive sulphide to stockwork mineralisation (Cu and Zn combined) is about 50:50.
2. The South Orebody, unlike the Main Orebody, is dominated by stockwork mineralisation (75%) with the massive sulphide component making up the remainder. Similarly, yet unlike the Main Orebody, the Cu and Zn massive sulphide zones largely do not overlap and the general trend from massive sulphides to stockwork is from top to bottom (rather than east to west). The massive Cu sulphide mineralisation is located at the top, grading to Zn massive sulphides and disseminated Cu sulphides with depth. The mineralisation strikes north to south for about 280 m with a vertical extent of approximately 330 m. The Cu/Zn massive mineralisation varies in thickness (east-west) from about 2 m to 80 m with the average thickness of about 40 m. The Cu stockwork mineralisation varies in thickness (east-west) from 1 m to 130 m, with an average thickness of about 30 m, Zn rich stockwork is on average about 14 m thick.

The difference in morphology of the two deposits is not due to significant faulting as observations of the contact between the footwall and hangingwall strata show minimal disruption. There is evidence of faulting disrupting the contact; however, offsets are relatively small, being mostly less than 100 m.

Typical east-west sections through the block model for the Main and South Orebodies are shown in Figure 7.3 and Figure 7.4, respectively.

Figure 7.3 East-west Section (1800 mN) through the Main Orebody illustrating the Cu and Zn zones



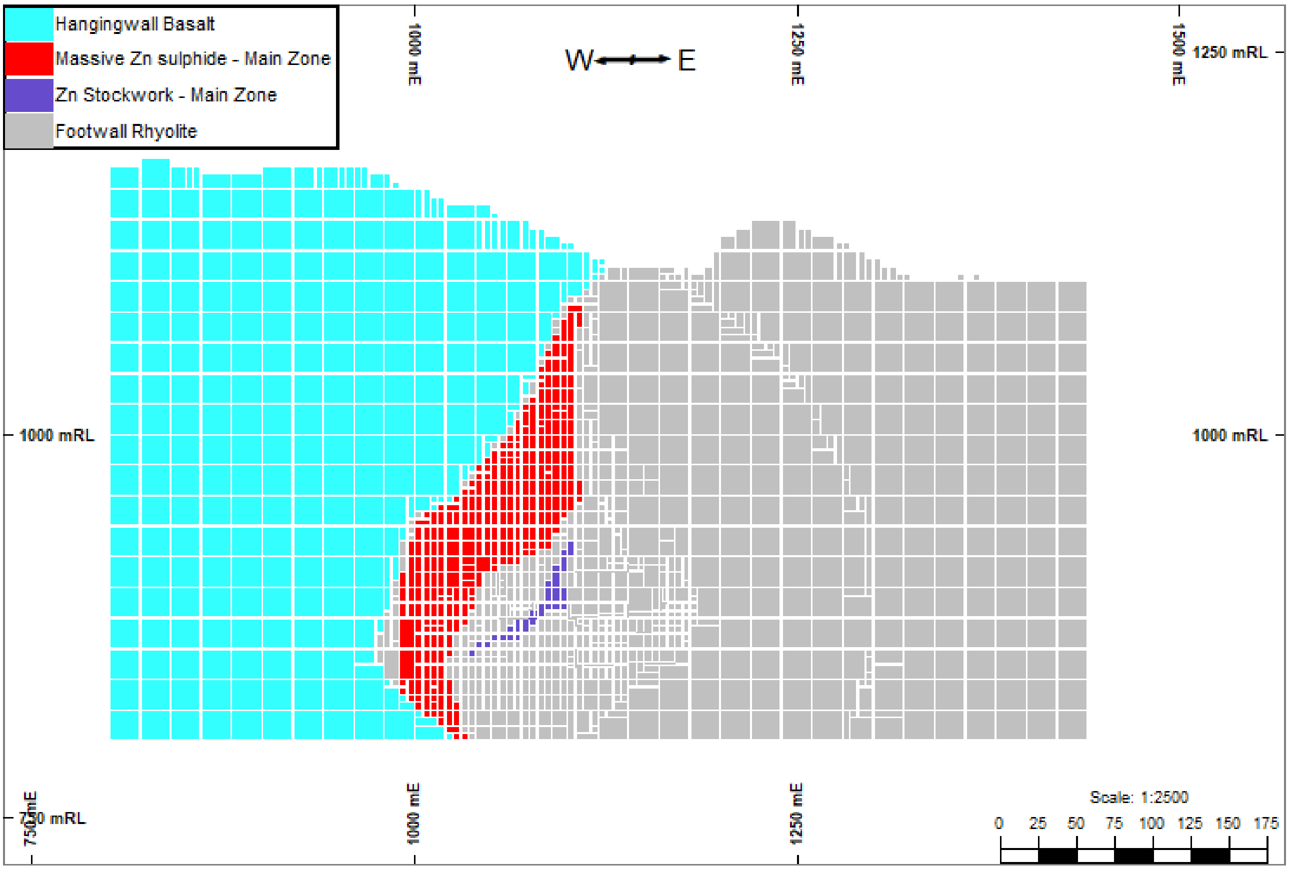
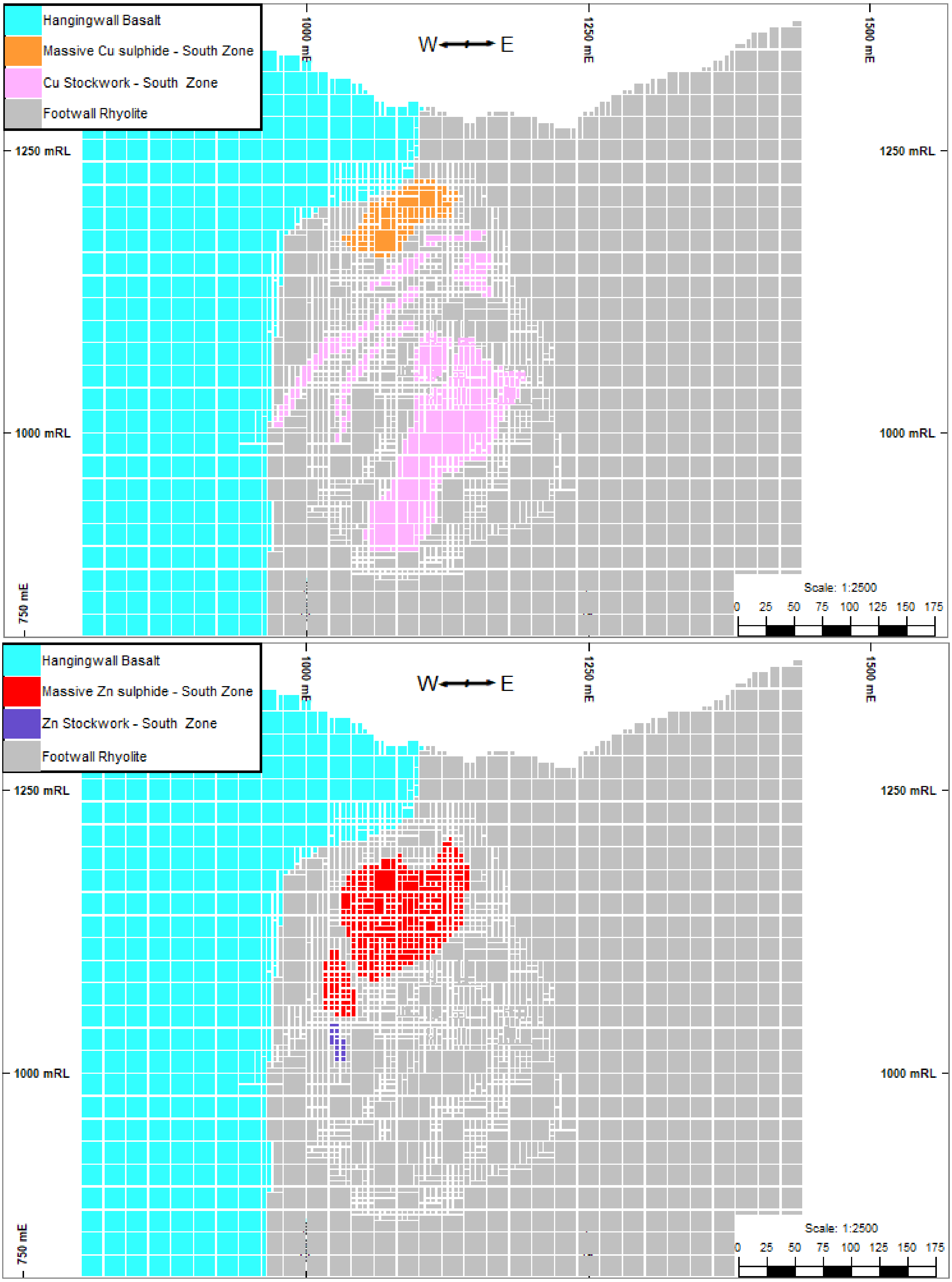


Figure 7.4 East-west Section (1100 mN) through the South Orebody illustrating the Cu and Zn zones



Item 8 DEPOSIT TYPE

The Çayeli mineralisation is a typical example of a volcanogenic massive sulphide (VMS) style of deposit. VMS deposits form at or near the sea floor, where circulating hydrothermal fluids in association with submarine volcanism are quenched through mixing with sea water and or pore waters in near seafloor lithologies.

The precipitated sulphides are typically rich in Cu, Zn, and Pb and lesser amounts of Au and Ag. VMS deposits form in extensional settings on the sea floor, and are associated with mid-ocean ridges, island arcs, back-arc spreading basins, and rifted continental margins. In other words, there is a significant spatial and temporal relationship in VMS deposits between magmatism, seismicity, and high-temperature hydrothermal venting. VMS deposits tend to occur as clusters of mineralised zones within regional districts.

The sulphide mineralisation occurs in the form of varying proportions of massive sulphides or stockwork style mineralisation. Massive sulphide zones in VMS deposits are typically stratiform and consist of greater than about 40% sulphides, namely pyrite, pyrrhotite, chalcopyrite, sphalerite, and galena. Other sulphides sometimes present include bornite and tetrahedrite.

The non-sulphide gangue mineralogy typically consists of quartz, barite, anhydrite, iron (Fe) oxides, chlorite, sericite, talc, and their metamorphosed equivalents (USGS, 2012). Stockwork mineralisation consists of disseminated sulphides within crosscutting veins hosted in pervasively altered host rock. Alteration types include argillic (kaolinite, alunite), argillic (illite, sericite), sericitic (sericite, quartz), chloritic (chlorite, quartz), and propylitic (carbonate, epidote, chlorite) types (Bonnet and Corriveau, 2007). Stockwork mineralisation typically forms adjacent to (below) the massive sulphide zone and is typically believed to represent fluid flow conduits below the sea floor.

VMS deposits exhibit a broad range of geological and geochemical characteristics. Cox and Singer (1986) used these geological characteristics to classify them into the following three types:

1. Cyprus – associated with marine mafic volcanics.
2. Besshi – associated with clastic and marine mafic volcanic rocks.
3. Kuroko – associated with marine felsic to intermediate volcanic rocks. This style of VMS deposit is named after the Kuroko deposit in Japan.

The Çayeli mineralisation is an example of a Kuroko style VMS deposit. Kuroko VMS deposits are characterised by the presence of massive stratiform sulphide mineralisation associated with volcanic rocks deposited in island arc settings.

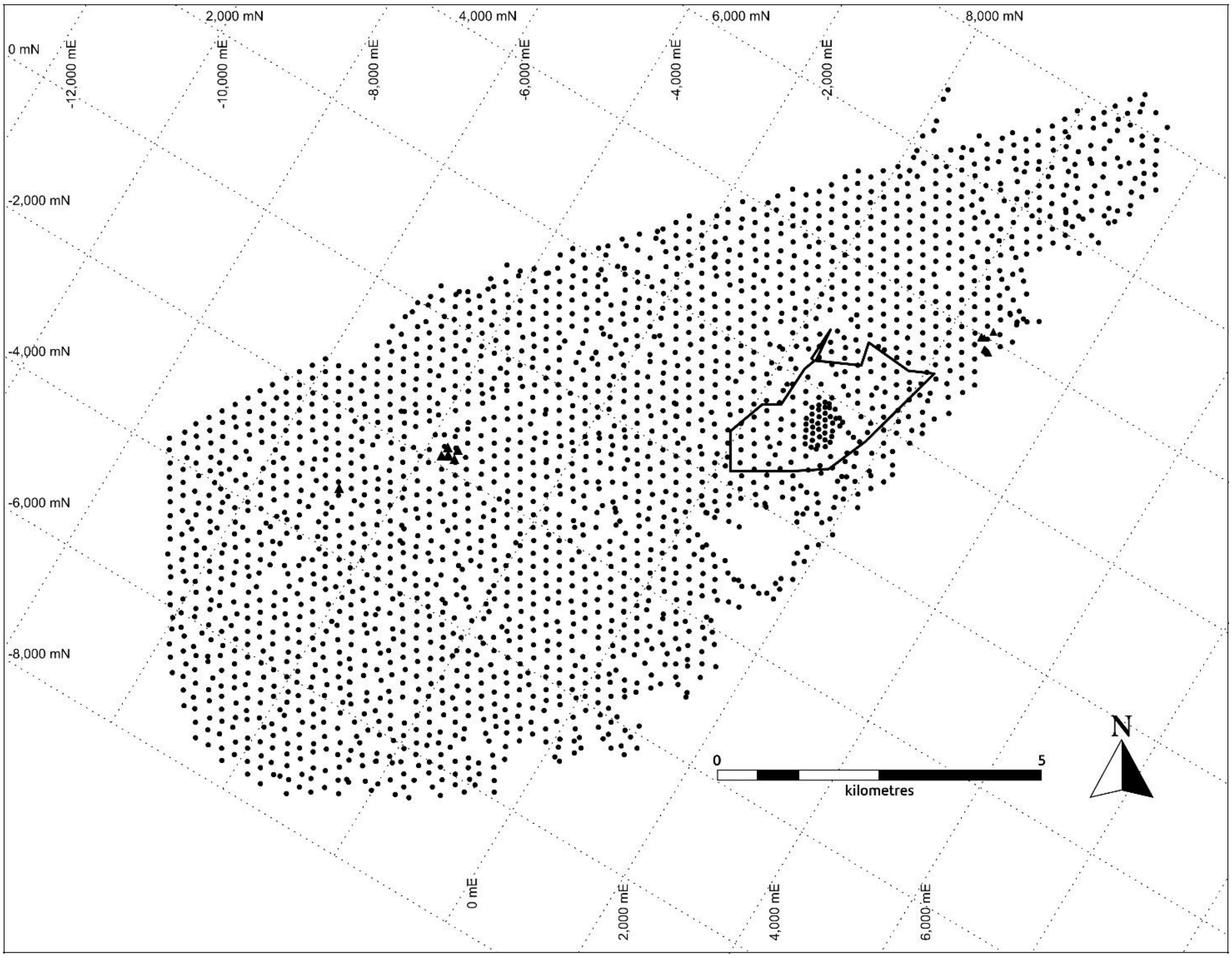
Item 9 EXPLORATION

9.1 Historical exploration

Other than drilling (discussed in Item 10), FQM exploration in the Çayeli district has consisted mainly of soil sampling and geological mapping, in addition to ground and geophysical surveys. The main exploration activities initiated by FQM since 2013 are summarised below.

The area was subject to extensive soil sampling in the period between August 2013 and March 2016, highlighting prospective areas which were already known. The sampling was done on a staggered 200 mE by 200 mN grid which was closed to 100 mE by 100 mN over the immediate Çayeli deposit area. An image of the sampling locations including the current lease boundary is shown in Figure 9.1.

Figure 9.1 Plan of soil sampling locations and the current lease boundary



The three major prospect areas (Armutlu, Gürgenli and Sarisu) were mapped in detail in 2013. In 2014, 19,000 m of drill core was subject to spectral analysis, the main finding of which was that the deposit does not have a broad alteration halo. Several surface and down hole Electromagnetic (EM) surveys were also completed with no success in identifying new massive sulphide targets.

In 2018, Complete MT (MT) solutions was retained by FQM to undertake acquisition, analysis and 3D inversion of data acquired in 2018 by Moombarriga Geoscience Pty Ltd, plus the 2009 Quantec data. The objective was to follow up on deep anomalies identified in the 2009 2D analysis, and to better locate their actual 3D locations for targeting purposes. The MT data analysis failed to identify potential targets worthy of further investigation; the mine anomaly was by far the most pronounced.

In late 2019, a land-based gravity survey was completed for the area covered in the 2009 and 2018 MT surveys. Gravity surveys are not impacted by electrical interference, a significant feature at site due to the surrounding infrastructure, and should therefore identify strong density contrasts associated with VMS deposits. Final processing of the 2019 gravity survey was delayed until June 2020 when an airborne LiDAR (Light Detection and Ranging) survey was flown to acquire reliable topography for the gravity terrain corrections (i.e., correcting for distortions due to topography changes).

Analysis of the gravity data in 2020 identified two anomalies, one north of the mine area and one to the southeast of the mine area. The northern target was dismissed due to a lack of supporting geochemical and MT signatures. The southeast target located some 300 m away from the existing mine workings remained the last potential target in the immediate mine area, although the corresponding MT anomaly was very subtle.

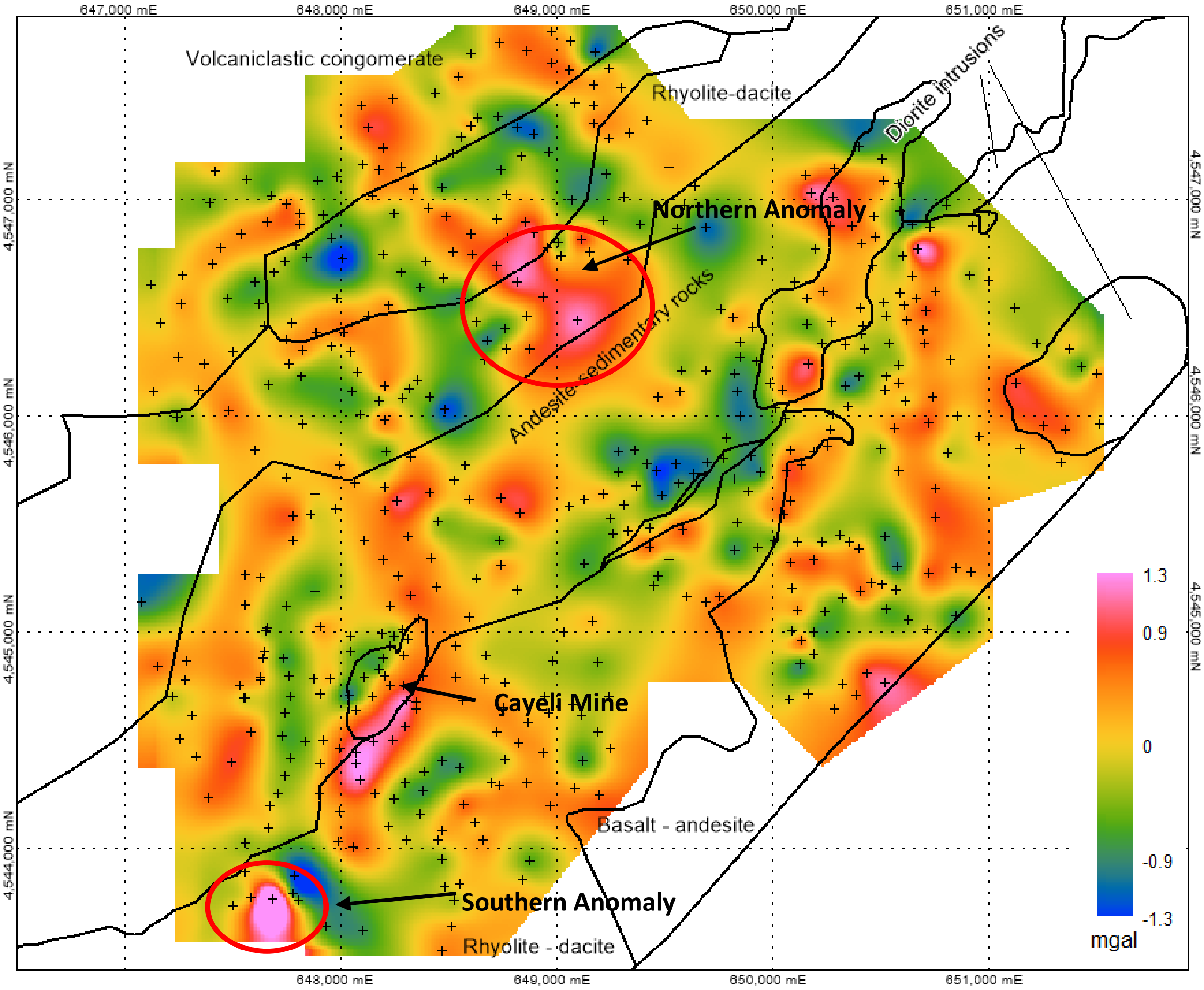
An image showing the gravity survey is shown in Figure 9.2. The crosses represent gravity survey points whereas the black lines delimit the key lithological boundaries. The map has been filtered to remove wavelengths greater than 1 km in length, in order to remove deep geological effects and highlight anomalies that might correspond to bodies at least one quarter the volume of the Main Orebody. The dark line corresponds to the approximate limit of the surface projection of the Çayeli mineralisation.

Around 2020, all but one of the tenements held by ÇBI were dropped as a cost saving measure. The remaining tenement (Lease IR740) covers the historic mine area and the immediate surrounds. It includes the southern anomaly identified in 2020.

9.2 Significant results

As part of a final round of exploration drilling in 2021, two diamond core holes were drilled from the underground workings to test the southern gravity anomaly. The discovery hole (D0202167) intersected 33 m at 2.4% Cu. This discovery hole led to the mobilisation of a surface rig, allowing continued drilling from both surface and underground sites from 2021 onwards. An underground development opening from the 1040 mRL sublevel intercepted the mineralisation in December 2023.

Figure 9.2 Gravity anomalies in the vicinity of the Çayeli mine (FQM, 2020)



9.3 Current exploration planning

Resumed near-mine exploration planning was addressed during 2024, focussing on the drill testing of multiple mineralisation targets over the next five years. These near-mine and regional VMS style targets are within 15 km of Çayeli and appear to be of potentially similar scale to the South Orebody.

VMS systems like Çayeli are known for having multiple disconnected lenses along a single geological contact horizon, with systematic drilling over time required to test for targets without surface expression or below the depth of useful geophysical penetration. FQM has planned a multiphase drill programme to test the near-mine mineralisation horizon over approximately 2 km in strike and up to 1.5 km in depth. This will be accompanied by downhole electromagnetic geophysics to enlarge the detection area and provide follow-up targets.

On the regional scale, the Company has agreed to an option over the Kaparyon deposit and intends to complete a programme of geophysical surveys and reconnaissance drilling.

Item 10 DRILLING

All drilling undertaken at Çayeli since the earliest exploration by the MTA has been core based drilling. There has been no percussion style drilling on the Property. Drilling completed to date in the Main and South Orebodies is summarised in Table 10-1. The Main and South Orebodies are delimited by the 1330 mN grid line.

Table 10-1 Çayeli core full (not coordinate clipped) drilling statistics - 1967 to April 2025

Deposit	Number of Holes Drilled	Total Metres Drilled
Main	2,673	124,800
South	107	21,800

The last core drilling in the Main Orebody area was completed in 2024. Except for some potential future regional exploration drilling, no further drilling in the vicinity of the old mine area workings is planned. All mine drilling completed and planned since that time is in the South Orebody area.

10.1 South Orebody drilling

Core drilling is conducted using combinations of HQ and NQ diameter drilling bits (63.5 mm and 47.6 mm core diameter respectively). HQ diameter core drilling is used from the outset and then if the drillhole length exceeds about 200 m in length, the core diameter is reduced to NQ. All drilling is currently double tube.

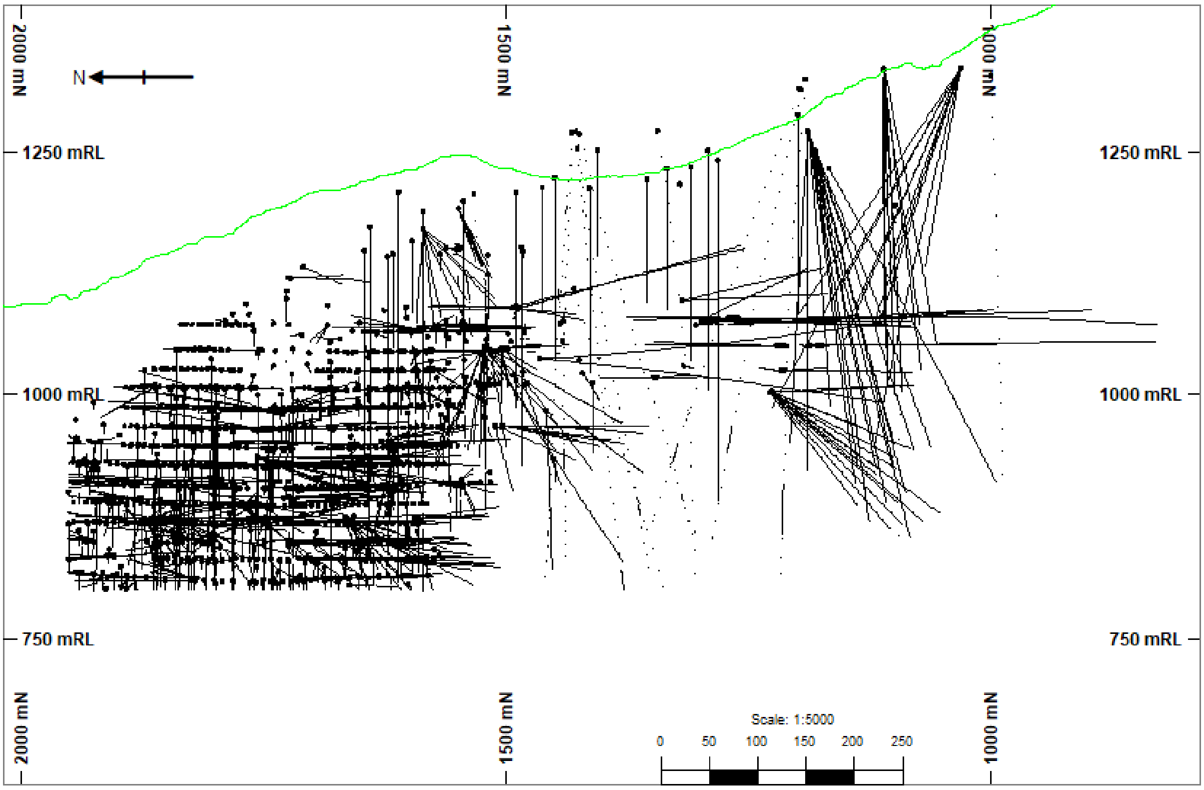
All planned surface drilling was completed in the period from 30th July 2022 to the 11th of May 2024, with all future mine drilling to be undertaken from within the underground workings. An image of the underground drilling equipment is shown in Figure 10.1; at the time the image was taken the rig was not operating.

The average core recovery for all drilling (South Orebody) completed up to May 2024 was 90%, which the Company considers to be a good outcome. In mineralised zones in the South Orebody (Cu>0.5%) the core recovery was 89%. Low core recovery is typically associated with sheared rocks and zones of “Black Ore” – massive sulphides which are typically Zn rich (>10% Sphalerite). There are no known relationships between core recovery and grade (Cu and Zn).

A north-south long section of the Çayeli drilling is shown in Figure 10.2. The drilling is clipped to south of 1950 mN and above 800 mRL to exclude drilling from mined out areas of the Main Orebody.

Figure 10.1 Underground diamond drilling (July 2024)

Figure 10.2 North-South section (1000 mN) showing the clipped Çayeli drilling and topography



10.1.1 Underground drilling survey

The mine surveyors paint fore and back sight lines in the drill cuddies and surrounds for the drillers to align each drillhole collar. The drillers set the desired dip of the drilling rods using an inclinometer. By surveying the top, end, and middle of the drilling boom, the dip and alignment (azimuth) are also checked by the mine surveyors once the rig is set up. These X, Y and Z coordinates are then provided to the geology department. All underground surveying is done using Leica total station equipment.

Downhole surveys are collected every 25 m down to the 50 m depth mark, and then every 50 m after that, using a single shot camera. The presence of magnetic minerals in the mine strata would distort the compass readings in this type of instrument. However, to date, magnetic interference on downhole survey measurements has not been identified as a material issue.

10.1.2 Diamond core archive

Currently, ÇBI rents two storage sheds on the outskirts of Madenli for the storage of processed core (Figure 10.3). The QP visited both sheds with ÇBI staff and found them to be secure (locked) and with sufficient space for future core storage. All core from drilling in the South Orebody is housed in these sheds, along with core from some older holes from the historic mine area. This means that the core is secure and readily available for review and further sampling, if and as required.

Figure 10.3 One of two storage sheds used for long term core storage

10.2 Other sampling

In addition to the sampling of diamond drilled core, “sludge sampling” is used to guide the mining and Resource development. This method involves collecting grab samples (3 kg to 4 kg) from the base of a jumbo heading, before the face is charged up and fired. Sometimes, the sample is collected while the heading is still being bored out, in which case the jumbo is shut down for safety reasons to enable the sampling process to be carried out. This is not the classic type of “sludge sampling” practiced at other mines where cuttings are collected directly below a boom in a bucket, as each hole is drilled.

The grab samples cannot be collected in a probabilistic manner (i.e. it is not possible) and as such the assayed grade results will be biased to some extent.

Item 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Since the start of mining at Çayeli, ÇBI has owned and operated a site laboratory which is responsible for the following services:

1. Grade analysis of diamond core and mine face sludge samples for Cu, Zn, Pb, Ag, Au and Fe.
2. Mineralogical studies of both diamond core and jumbo sludge drilling samples using analyses of polished mounts.
3. Grind size analysis of run of mine mill samples, analysis of feed and tails samples.
4. Analysis of external third-party samples. These samples consist of Cu and Zn concentrate grades. This aspect of work is minor in terms of the annual laboratory sample throughput, with approximately only twenty external samples being analysed each year.

The laboratory is accredited with the ISO/IEC 17025:2017 standard. The accreditation was last reviewed by TÜRKAK (Turkish Accreditation Agency) in 2023 and the certification renewed. TÜRKAK is affiliated with the Ministry of Foreign Affairs of Türkiye and is internationally recognised as the sole national accreditation body for the Turkish government.

11.1 Diamond core sampling

The core sampling method used is as follows:

1. Core is transported by the drilling contractor to the core storage shed on the surface, where it is then unpacked and arranged onto logging tables (Figure 11.1).
2. The core is usually clean but is washed if required. The core is then photographed, logged for lithology, mineralisation, core recovery and rock quality designation (RQD), using a nominal 2 m sample interval yielding approximately 4 kg to 6 kg per sample.
3. Prior to core cutting for sampling purposes, selected 10 cm to 20 cm lengths of core are chosen for dry bulk density determination. The method used is the conventional water displacement method.
4. The core is cut using a diamond saw into two halves. One half is returned to the core tray while the second half is ultimately submitted to the site laboratory for analysis.
5. Trays of logged and sampled half cut core are later transported to the core storage area where they are placed on wooden pallets (see Figure 10.3 in Item 10).
6. Core samples, which are delivered in pre-numbered plastic bags to the site laboratory, are marked with the drillhole and sample number as well as four duplicate paper sample tickets with the same number inside (Figure 11.2). A detailed printed sample submission sheet is also included and lists, amongst other items, the drillhole number, from and to intervals and logging codes etc.

Figure 11.1 Mine core sample processing facility

Figure 11.2 Core samples stacked at the laboratory sample preparation area

11.1.1 Sample security

The core processing facility is an area staffed by six geology technicians, with two technicians being available per shift. Technicians typically spend two to three hours a day doing underground work tasks while the rest of their shift is spent in the core shed area completing other tasks (e.g., logging, sampling etc). The geology technical team is supervised by the mine geologist.

11.2 Diamond core analysis

The core analysis method used is as follows:

1. The bagged core sample is crushed using a jaw crusher to 90% passing 1 cm. The crushed sample is typically split twice using a single tier 50:50 Jones splitter. Depending on the initial sample mass, the sample may be split a third or fourth time. The coarse split is placed in a steel tray with the ticketed sample number and then into one of two drying ovens at 105°C for two days (Figure 11.3).
2. Once dry, the approximately 300 g dry coarse crush samples are pulverised for 1 minute using a ring and puck mill to produce a pulp sample ground to 85% passing 75 µm.
3. Bagged pulp samples are initially analysed by X-Ray Fluorescence (XRF) for Cu, Zn, Ag, Pb and Fe. The detection limits for Cu and Zn are 0.01%. The goal is to quickly assess the key element grades from which those above a 0.25% detection limit are sent for further analysis using a two-acid digest (aqua regia) process with an Atomic Absorption Spectroscopy (Flame AAS – Au and Ag) or Inductively Coupled Plasma-Optical Omission Spectrometry (ICPOES – Cu, Zn, Pb and Fe) finish.

Figure 11.3 Drillhole sample drying oven

Results from XRF analysis and AAS/ICPOES analysis are typically available within two and five hours, respectively. Data is supplied to the mining and geology departments in the form of MS Excel files. Residual pulps from the core and sludge samples are kept for a period of six months in dedicated shelving in the laboratory oven drying room. After six months, the pulps are returned to the geology department for long term storage. All returned sample pulps from the South Orebody analyses are available in storage.

11.3 Sludge sample analysis

Sludge samples are bagged and delivered to the laboratory using a similar approach to core samples (i.e., labelled plastic bags). However, instead of a drillhole and sample number, the sludge bags are labelled with a location such as “1050S14” which translates to 1050 mRL level, South Orebody, number 14 drive, followed by a sample number. The geology technicians record the distance from the face to a known survey point of the drive wall (marked with paint) in separate documentation so that the sample can be later allocated mine grid coordinates for its approximate location.

With the need for a quick sample turn around (sludge samples are used to monitor underground development headings), a different approach is used to prepare a sample pulp for analysis. The steps are as follows:

1. The wet sample is placed in a large steel tray and then mixed and flattened using a trowel. The mixed flattened sample is divided into approximately ten to twelve similar areas and a scoop is taken from each area to produce approximately a 400 g split.
2. The sample is placed in a steel tray and placed on a hotplate for approximately 45 minutes to dry.
3. After drying, the samples are pulverised (no jaw crushing is required) using the same procedure as used to pulverise crushed and dried core samples.
4. The pulp is analysed using the same approach that is used with core samples.

11.4 Mineralogical assessment

Critical to managing the classification of mine production tonnages into plant feed categories is the determination of the sulphide composition, and where relevant, the texture of mineralised samples. The selection of samples for mineralogical assessment is based on the Cu and Zn grades. Currently (i.e., at the time of the site visit in July 2024 – Mr Richard Sulway), samples are selected for assessment if the Cu grade exceeds 0.6% and or when the Zn grade exceeds 1%. These threshold values are reviewed periodically, focussing on the prevailing metal prices at the time so that the analysis is focused on likely, imminent plant feed types. This work is done in the same building as the chemical analysis work.

The samples are set in resin and then prepared as polished mounts which are then analysed using a microscope under reflected light (Figure 11.4). The samples are processed at a rate of approximately ten sludge samples and fifty core samples each day.

Figure 11.4 Mineralogical assessment microscope and imaging software

11.5 Quality Assurance and Quality Control

11.5.1 Laboratory

The site laboratory has standard operating procedure (SOP) documents for the main laboratory functions. There is no documentation in English. Quality Control (QC) steps used by the laboratory consist of:

1. Participating in surveys (round robin analysis) of international laboratories from around the world, at a duration of every few years.
2. Routine submission of concentrate assays once a year to independent laboratories for analysis and comparison with site derived results.
3. Use of commercially certified reference materials (CRMs) purchased from Ore Research and Exploration Pty Ltd (OREAS) to check the calibration of the AAS and ICPOES instruments on a daily basis.
4. Approximately 10% of AAS and ICPOES analyses are repeated and any discrepancies are investigated.

11.5.2 Geology department

At the time of the July 2024 site visit there was:

- No SOP documentation for the main geology functions.

- No submission by the geology technicians of independent QC samples, namely standards, blanks duplicates etc.

Training of site staff is done largely by word of mouth, which while not ideal, is helped by a relatively low staff turnover on site. As an example, in the geology technician department, only one person had resigned (due to retirement) in the last ten years.

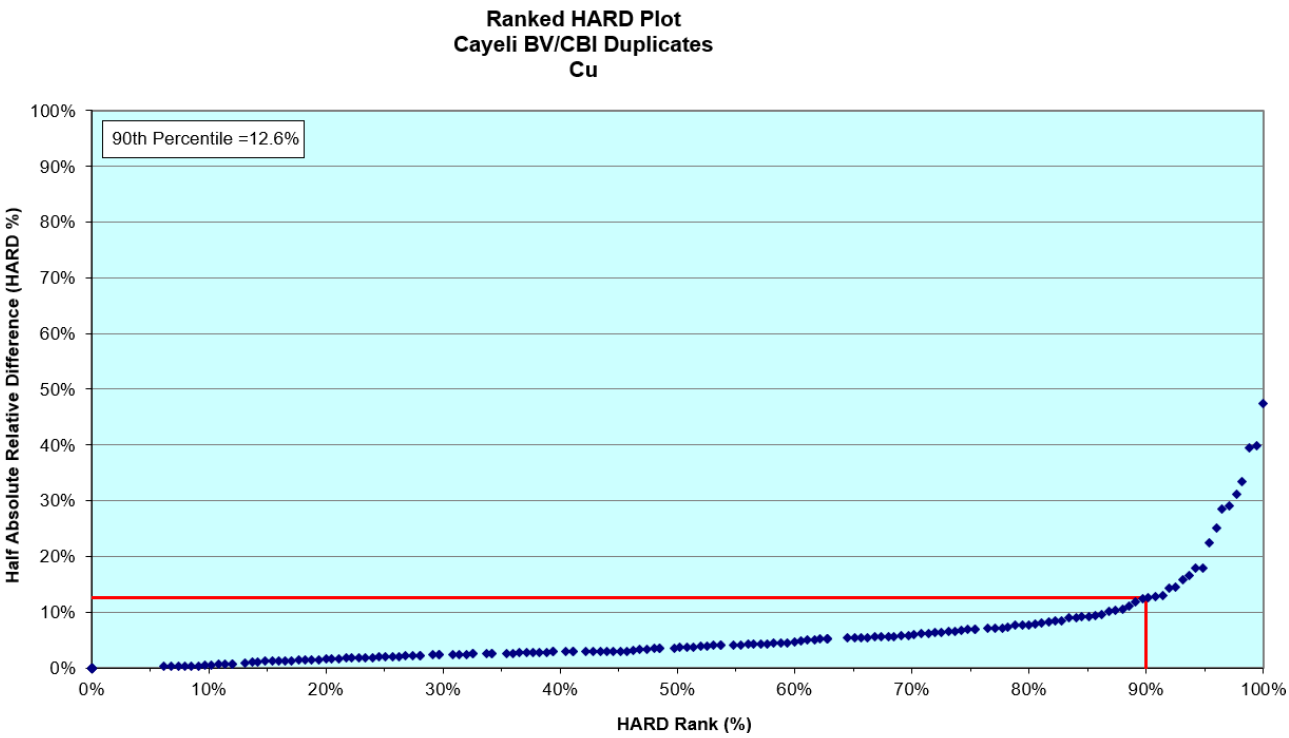
Among the recommendations from the site visit was to compile SOP documentation, to start submitting certified CRMs into the sample stream, and to start a pulp resampling programme.

Pulps were selected from storage (South Orebody samples) and sent with CRMS to the Bureau Veritas (BV) laboratory in Ankara for Cu, Zn, Pb and Ag analysis. At the time of writing, approximately 200 pulp results from BV and 35 CRM results (site laboratory) from five OREAS sourced standards were available for analysis; both programmes are ongoing. SOP documentation had yet to be compiled.

The QC data was analysed in May 2025 by the QP and the findings documented in a standalone report (FQM, 2025). The key findings summarised in this report were:

1. The results from the CMS submissions indicated good levels of accuracy were being achieved by the site laboratory.
2. The precision of the pulp results was mixed and ranged from good (Cu and Pb - Figure 11.5) to moderate (Zn) to poor (Ag). The low precision in the Zn and Ag data appears largely driven by grade biases between the ÇBI and BV results. These differences are still being investigated as some of the CRM values returned by BV are anomalous.
3. Start submitting blanks and duplicate samples into the mine sample stream.
4. Continue the pulp resampling programme until approximately 400 samples have been submitted (representing about 5% of the South Orebody drilling data in mid-2024).

Figure 11.5 Cu Hard Chart (BV and ÇBI)



11.6 Comments on sample preparation, security and analytical procedures

It is the opinion of the QP that:

1. The geological logging, sample security, collection, preparation and multi-element analysis undertaken at Çayeli is appropriate to the style of mineralisation.
2. While several minor issues were identified in the QC data, no material flaws were identified and the mine has operated successfully for over thirty years.
3. The collected dry density data is determined using standard industry methods and is appropriate for the deposit type.
4. The storage facilities used to house the residual samples and diamond core are both well organised and secure.

Item 12 DATA VERIFICATION

Over a period of about 2.5 weeks (26th June to the 13th of July 2024), Mr Richard Sulway (QP) visited the Çayeli Operations as part of a planned release of an updated Mineral Resource estimate and this accompanying NI 43-101 Technical Report. The key purpose of the site visit was to become familiar with, and verify the current practices and procedures used by the ÇBI geology department and associated entities, including the drilling contractors and mine laboratory. A secondary objective was to start collating the relevant documentation. The visit was coordinated and conducted with Mr Kadir Tolga Güngör (Senior Geologist) who has worked at the mine for eleven years.

During the visit two key issues were identified:

1. Most of the relevant geology site practices were not formally documented as SOP style documents.
2. Several areas were identified during the visit where there was scope for improving current site practices.

In response, the QP has documented the site geology and assay procedures as discussed during his visit, together with his recommended changes in a dedicated site visit report (FQM, 2025). This report was subsequently referenced as a basis for Item 11 in this Report.

The following verifications were undertaken as part of the site visit:

1. Collar coordinates from the Çayeli drilling data were validated against the current topographic surface, and no material discrepancies were noted.
2. A review of the site grade control modelling practices and associated data inputs (i.e., modelling parameters and wireframes) was undertaken.
3. All key geology and laboratory staff were interviewed as part of analysing the process route from core collection to the determination of the final analytical results.
4. The site laboratory, surface core shed facilities and underground mine were visited.
5. A half day tour of the Çayeli processing facility including the ore stockpiles was completed.

It is the opinion of the QP that the drilling data used to compile the Mineral Resource estimates described in this Report is of sufficient quality to adequately represent the in-situ mineralisation and provide the basis for the conclusions and recommendations reached in this Report.

Item 13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Çayeli ore types

Historically, the Çayeli mine has produced three main ore types:

- zinc-rich Clastic ore
- zinc-rich Black ore
- copper-rich Yellow ore

The Net Smelter Return (NSR) calculations, which form the basis for tabulation of the Mineral Resources and Reserves, continue to be calculated based on the characteristics of these ore types.

Drawing upon historic operating experience and mineralogical observation, each of the mined ore types are hauled to surface and dumped into separate storage compartments for blending or batch treatment through the processing facilities.

13.1.1 Main Orebody

Whilst originally, Yellow ore from the Main Orebody produced "Spec" copper and zinc concentrates, black ores produced 'non-spec' copper concentrates, and 'spec' zinc concentrates, the Clastic ore was a metallurgical challenge and produced "Non-spec" concentrate that was high in both copper and zinc.

As mining progressed deeper into the Main Orebody, several factors began to affect processing operations. Overall, the proportion of true Clastic ore, i.e. ore that exhibits a clastic texture and contains intergrowths of chalcopyrite within grains of sphalerite, increased. The presence of bornite in zinc-rich rocks (referred to as Black ore) made it difficult to separate copper and zinc in the plant, resulting in the production of Non-spec rather than Spec concentrates. The presence or absence of bornite is defined as material with greater than or less than a 40% probability of containing bornite.

The presence of lead grades greater than 0.8% also resulted in the production of a Non-spec concentrate. Finally, the increase in copper prices over past years has allowed for the economic extraction of significant volumes of footwall stockwork zone ore. Although typically lower in copper content than massive sulphides, these footwall zones contain little to no zinc, thereby yielding very favourable metallurgical properties.

The ore types for the Main Orebody are currently as follows:

- **Clastic ore**, comprising and characterised by:
  - exhibiting a clastic texture and containing intergrowths of chalcopyrite within grains of sphalerite, i.e.,
  - "CO – "; as above,
    - producing Non-spec concentrate
    - with bornite probability of <0.4
    - mineralisation with >0.8% Pb, >5.0% Cu and >4% Zn
    - mineralisation with >3% Cu and >5% Zn
  - "CO + (or BCO, Bornite Clastic Ore)"; as above,
    - producing Non-spec concentrate
    - with bornite probability of >0.4
    - mineralisation with >0.8% Pb, >5.0% Cu and >4% Zn
    - mineralisation with >3% Cu and >5% Zn

- **Black ore**, comprising and characterised by:
  - The presence of bornite in zinc-rich rocks, i.e.,
  - "BO",
    - producing Non-spec concentrate
    - with bornite probability of <0.4
    - mineralisation with <0.8%Pb, <5.0% Cu and >3.2% Zn
- **Yellow ore**, comprising and characterised by:
  - "YO – ",
    - producing Spec concentrate
    - with bornite probability of <0.4
    - mineralisation with >1.0% Cu and <4% Zn
  - "YO + (or BYO, Bornite Yellow Ore)",
    - producing Spec concentrate
    - with bornite probability of >0.4
    - mineralisation with >1.0% Cu and <4% Zn
  - "LYO",
    - producing Spec concentrate
    - mineralisation with <1.0% Cu and <4% Zn

13.1.2 South Orebody

The ore types in the South Orebody are similar to those in the Main Orebody. Two main ore types have been defined, a Black ore and a footwall type which is similar to Yellow ore:

- **Black ore**: comprising and characterised by:
  - "SBO",
    - producing Non-spec copper concentrate, and spec zinc concentrates
    - typically located above the 1050 mRL sublevel
    - mineralisation with >1.5% Zn
- **Yellow ore**, comprising and characterised by:
  - "SYO",
    - producing Spec copper concentrate, and no zinc concentrates
    - located typically on the footwall between the 900 mRL and 1200 mRL sublevels
    - mineralisation with >0.94%Cu and <1.5% Zn

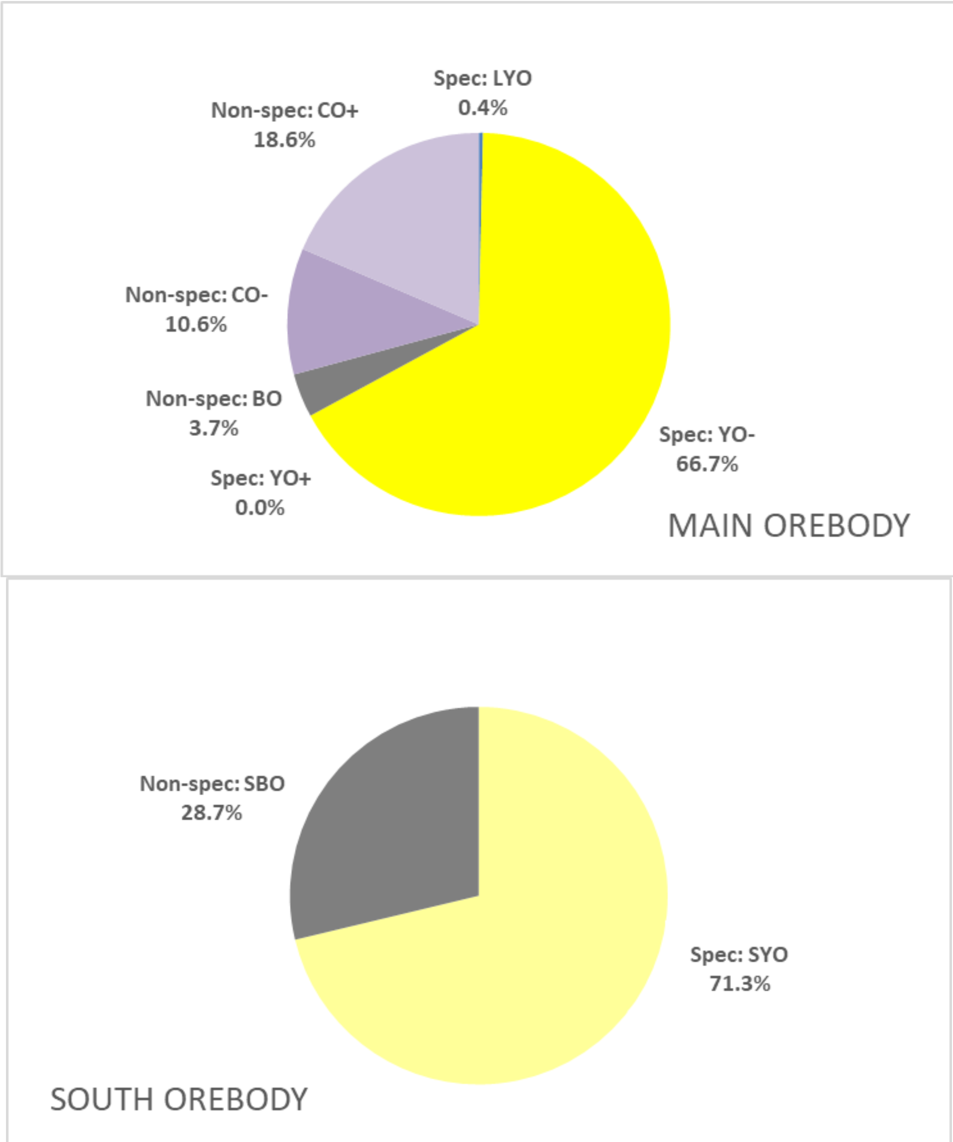
Below the 1000 mRL sublevel there is currently only limited drill samples and metallurgical information available. Further sampling and analysis are required for the deeper sublevels of the South Orebody.

13.1.3 Mined and blended ore types

Figure 13.1 is a pie-chart showing the relative proportions of each mined ore type as defined within the life of mine (LOM) mine production plan outlined in Item 16. In the Main Orebody, which is mined between 2025 and 2027, and in 2035 and 2036, Spec ore (YO-, LYO) is 67% of the mined total and Non-spec ore (BO, CO-, CO+) is 33%. Of note is that there is no Main Orebody Spec YO+ ore in this latest mine production plan.

In the South Orebody, which is mined from 2025 to 2036, Yellow Spec ore (SYO) is 71% of the mined total, and Black Non-spec ore (SBO) is 29% of the total mined.

Figure 13.1 Proportions of each mined ore type in the LOM production plan



In the past, plant feed from the Main Orebody was prepared by blending certain mined ore types according to their metallurgical composition and copper and zinc grades, producing Spec and Non-spec concentrates. Specifically, some of the Yellow (YO+ and LYO) ore, which is ordinarily Spec ore, was reclassified and cross-blended into the Non-spec feed.

In this latest production plan (Item 16.7), and in the absence of mined Spec YO+ ore, there is effectively no cross-blending of Main Orebody feed into the process plant (Item 16.8). The two distinct mined ore specification types from the South Orebody do not require blending.

13.2 Plant trial on ore from the South Orebody

Material from South Orebody mine development openings was found to be mineralised, and hence some of this material was processed as a plant trial.

The development material processed in the trial was not fully representative of the new orebody but could be used as an indication of the likely performance of this material in the existing processing plant.

The development material was processed over approximately 4.5 shifts of production at the existing processing plant, using the current operating conditions, and the results were evaluated both operationally and technically. Because of the low zinc grade, only the copper flotation circuit was operated. A summary of the data collected is presented in Table 13-1.

Table 13-1 Recovery data determined from the plant trial

	Tonnes Processed	Grades		Recovery	
		%Cu	%Zn	%Cu	%Zn
Feed	5,117	1.17	0.22	100.00	100.00
Cu concentrate	239	23.24	2.34	92.94	49.04
Final tail	4,878	0.09	0.12	7.06	50.96

The mill throughput rate was an average of 154 tph (the equivalent of 1.2 Mtpa), with a mill energy consumption of 12.8 kWh/t. The primary ball mill drew an average of 413 kW, and the secondary mill 1,555 kW. The grind was reported as 62.6% passing 36 µm.

Reagent consumptions were:

- Frother A-208 11.2 g/t
- Collector SIPX 38.7 g/t
- Lime 510.7 g/t
- Flocculant 0.78 g/t

The material was treated successfully through the existing circuit. Throughput was limited to 154 tph through the milling circuit, due to limitations in water addition and high densities. There were no issues with flotation, and no adverse conditions were encountered in the concentrate dewatering and drying areas.

A full plant survey was not undertaken during this trial.

13.2.1 Metallurgical sampling and testwork

Metallurgical testwork samples considered to be representative of the South Orebody were selected by the ÇBI geology team. The metallurgical testwork programme was then conducted by Mineral Research and Development (MRD, a part of the Hacettepe University in Ankara) under the direction of Professor Zafir Ekmekçi.

Coarse rejects of the selected drill cores were provided to MRD in July 2024. Halved samples from drill cores were selected for flotation testwork composites. Three composite ore samples were selected for testwork:

- Footwall (FW - copper rich, with low zinc content)
- Black ore zone, low zinc sample (Zn < 8%)
- Black ore zone, high zinc sample (Zn > 8%)

Head grades of the composite samples are listed in Table 13-2.

Table 13-2 Testwork sample head grade

Sample	Cu (%)	Zn (%)	Au (g/t)	Fe (%)	Pb (%)	Co (ppm)	Cr (ppm)	Mn (%)	Ni (ppm)	Ca (%)
Footwall ore, Cu rich	4.21	0.09	0.43	15.57	0.01	51	58.5	0.09	5.7	0.17
Black ore < 8% Zn	1.23	4.43	0.86	27.39	0.13	9.8	63.3	0.09	15.2	0.78
Black ore > 8% Zn	0.83	15.54	0.99	24.38	0.28	9.25	72.8	0.07	12.9	0.65

Two blended ore samples were also prepared to investigate the effects of blending different ore types:

- Blend 1; 75% Footwall (FW): 13% Low Zn:12% High Zn
- Blend 2: 50% Low Zn: 50% High Zn

Specific gravities of the Footwall, high Zn and low Zn blended samples were measured by a pycnometer and found to be 3.12, 3.90, and 4.09, respectively.

The following basic testwork was undertaken:

Comminution

- bond crushability
- bond abrasion test
- Bond Work Index determination
- kinetic grinding test
- specific gravity

Beneficiation

- mineral liberation analysis (MLA)
- flotation tests
- open cleaner flotation tests
- simulation study (to determine the plant recovery)

Two reports on the testwork were issued by MRD:

- Comminution Testwork Report for South Orebody, issued in July 2024 (Hacettepe Teknokent, 2024a)
- South Orebody Characterisation Testwork Flotation Studies, issued in December 2024 (Hacettepe Teknokent, 2024b)

13.2.2 Comminution testwork

Head grade analyses of the comminution composite samples, by x–ray fluorescence (XRF), gave the data presented in Table 13-3.

Table 13-3 Head grades for comminution composite testwork samples

Sample	Cu (%)	Fe (%)	Zn (%)	Pb (%)
Footwall ore, Cu rich	3.93	14.12	0.13	0.01
Black ore < 8% Zn	1.91	24.18	6.30	0.16
Black ore > 8% Zn	0.91	19.52	19.08	0.25

Table 13-4 presents the comminution testwork results for the three composite samples.

Table 13-4 Comminution testwork results

Test	Black Ore > 8% Zn	Black Ore < 8% Zn	Footwall (Cu rich)
Bond Grindability Work Index, kWh/t	8.81	10.14	15.42
Bond Crushability Work Index, kWh/t	8.09	7.98	NA
Los Angeles Test			11.7
Bond Abrasion Index (Ai), g	0.011	0.014	0.209
Levin Test, kWh/t	16.51	20.63	23.07
Specific Gravity	4.1	3.9	3.1
Bulk Density (from BWi), g/cm³	2.79	2.72	2.23

The data indicates that the footwall (Yellow ore) composite had the highest Bond Work Index (BWi), at 15.42 kWh/t, and the highest abrasion index.

When adopting these results, and assuming the following, the ball mill power requirement would be 2,455 kW:

- a maximum throughput of 850,000 tpa in the years 2027 to 2029[4]
- a feed to the ball mills of 80% passing 10 mm
- a grind size of 80% passing 38 µm (as per the current plant)

This power requirement compares with the current installed power in the two mills of 2,660 kW, indicating that the existing comminution circuit is adequate for the treatment of South Orebody feed at the maximum design throughput.

It is estimated that a maximum annual mill throughput would be 1.0 Mtpa, based on 145 tph and 7,000 hours per year (factoring in increased downtime and reliability issues due to the age of the facilities).

An abrasion index of 0.209 indicates a steel consumption in these mills of 2.5 kg/t for the footwall (Yellow) ores. By comparison, steel consumption for the treatment of the Main Orebody is currently being measured at 1.7 to 1.8 kg/t.

13.2.3 Mineralogy and flotation studies

Mineralogical and flotation characterisation of the South Orebody composite samples was undertaken by MRD.

Mineralogy

Chalcopyrite and sphalerite were the dominant copper and zinc minerals in all the samples, whilst the presence of secondary copper minerals was negligible. The footwall sample contained 25% pyrite, with other major non-sulphide gangue minerals being quartz, chlorite and siderite, plus a very small amount of Si-Al clay minerals. The cobalt content of this samples appeared to be unusually high.

Mineral liberation analyses indicated that the degree of liberation of pyrite and chalcopyrite were 89% and 85%, respectively in the footwall sample.

[4] 850,000 tpa is the maximum mine production, before the application of unplanned mining dilution and mining recovery (loss) factors.

The Black ores are massive sulphides containing about 56% to 66% pyrite. The degree of liberation of pyrite was 82% and thus suitable for selective separation. However, the degree of liberation of chalcopyrite and sphalerite, was only about 65% and 70%, respectively.

The locked particles were mostly in the form of binary particles associated with pyrite, and to some extent with each other (Cp/Sph). There was also about 9% to 10% complex (ternary) particles in these samples. The Cu/Zn selectivity and Cu recovery could therefore be low for the Black ore samples at a P_{80} of 38 µm. Finer grinding may be required to improve the flotation performance.

Flotation testwork

Flotation conditions and reagent additions at the ÇBİ flotation plant were taken as the starter point for the flotation testwork. The Spec ore conditions were applied on the Footwall ore sample, and Non-spec conditions on the Black ore samples. The flotation parameters were adjusted during the tests to optimise flotation performance.

Several rougher kinetic tests were undertaken on each ore type. Optimum flotation conditions (residence time, reagent dosages, pH, etc) were determined and one open cleaner flotation test was performed at this condition. Simulation studies were performed using the results of the open cleaner flotation tests to determine closed-circuit flotation performance of each sample. The simulation results for all three samples and the two blends are provided in Table 13-5 for copper flotation, and in Table 13-6 for zinc flotation.

Table 13-5 Copper flotation tests - concentrate grades and recoveries

Sample	Copper Concentrate								
	Grade				Recovery (%)				
	Cu, %	Zn, %	Au, g/t	Ag, g/t	Mass	Cu	Zn	Au	Ag
Footwall ore, Cu rich	28.83	0.55	1.1	20.45	14.59	93.68	73.55	33.04	74.13
Black ore < 8% Zn	29.02	8.6	2.39	24.22	3.09	69.27	6.1	14.44	3.04
Black ore > 8% Zn	15.75	18.06	4.25	92.23	1.63	29.64	1.86	8.68	3.69
Blend 1	30.3	7.16	1.13	19.66	6.86	60.23	19.17	9.92	13.26
Blend 2	25.13	10.04	5.17	68.5	2.89	69.05	2.88	19.67	6.45

Table 13-6 Zinc flotation tests - concentrate grades and recoveries

Sample	Zinc Concentrate								
	Grade				Recovery (%)				
	Cu %	Zn %	Au, g/t	Ag, g/t	Mass	Cu	Zn	Au	Ag
Footwall ore, Cu rich	-	-	-	-	-	-	-	-	-
Black ore < 8% Zn	3.67	55.79	0.39	90.56	6.38	18.05	81.54	4.91	23.45
Black ore > 8% Zn	1.65	54.88	2.3	75.16	25.81	49.05	89.32	74.23	47.52
Blend 1	12.92	28.38	0.89	33.62	5.35	20.02	59.22	6.12	17.67
Blend 2	1.29	58.43	1.91	67.89	14.93	18.27	86.6	37.58	33.04

It is noted that although the above testwork results show feed grade, recovery and the concentrate grade for gold, gold grades are not reported in the Mineral Resource and Reserve statements. This is due to the fundamental sampling and assaying issues mentioned in Item 14.

Flotation tests for the footwall (FW) sample

Since the FW sample contains chalcopyrite and only a very small amount of sphalerite, the recovery circuit comprised copper flotation only; no zinc flotation was required. A mixture of A208 (a dithiophosphate) at an addition rate of 15 g/t, with SIPX (sodium isopropyl xanthate) at 60 g/t, was the preferred collector and a pH 11.5 to 12 was required for pyrite depression.

A copper concentrate was produced after three stages of cleaner flotation assaying 33%, but at very low recovery. The calculated copper grade of the Cu cleaner 1 concentrate was also very high (27.4% Cu). This suggested that a good quality concentrate could be produced with a single stage of cleaner flotation.

The simulation of closed-circuit flotation performance with one stage of cleaner flotation indicated that a copper concentrate could be produced assaying 28.8% Cu, 0.5% Zn, 20 g/t Ag and 1.1 g/t Au at a copper recovery of 93.7%.

Flotation tests for the low zinc sample

For the low zinc ore sample, a mixture of A208:Aero3418A (a dithiophospine type of collector) was used as the collector in copper flotation, as per the existing plant conditions for treating the Black ore type. The pulp pH was adjusted to a pH of 11.5 by using lime. However, this selection of collector gave a very low copper recovery of 19% when applied to the South Orebody ores. The addition of sodium metabisulphite (MBS) was not effective, although a mixture 10 g/t A208 plus 40 g/t SIPX gave higher copper recoveries at high Cu/Zn selectivity.

For zinc flotation, the effects of $CuSO_4$ dosage for sphalerite activation were investigated. The Zn stage recoveries were similar at about 86% for both 600 g/t and 800 g/t $CuSO_4$ dosages. However, the highest Zn recovery and concentrate grades were achieved in the initial stages of rougher flotation with the higher copper sulphate dosage. Thus, 800 g/t $CuSO_4$ and 100 g/t SIPX were used in the open cleaner flotation test.

In the open cleaner flotation tests the copper concentrate grade achieved could be increased to 27% Cu, but the zinc grade was also very high (9.3%). In zinc flotation, a concentrate was produced at a grade of 58.86% Zn.

Simulation of the closed-circuit flotation performance suggested that a copper concentrate could be produced assaying 29% Cu, 8.6% Zn, 24 g/t Ag and 2.4 g/t Au at a copper recovery of 69%. A copper concentrate with high zinc contamination could be produced at an acceptable recovery from this complex ore sample. In the zinc flotation section, a concentrate was produced assaying 55.8% Zn at 81.5% recovery.

Flotation tests for the high zinc sample

The high Zn ore sample contained 0.83% Cu and 15.54% Zn. The Cu/Zn ratio was very low, which indicated that high zinc levels would be produced in the copper concentrate.

The copper flotation tests were performed at pH 11.5 in the presence and absence of MBS using a mixture of A208:3418A (as used in the existing circuit for treating Black ores from the Main Orebody) and A208:SIPX. As with the low zinc ores, copper recovery benefitted from the use of SIPX in the collector mixture. Cu/Zn selectivity was better in the presence of MBS.

High dosages of $CuSO_4$ and SIPX were tested in the zinc flotation section due to the high Zn feed grade.

The open cleaner flotation test was conducted at a pH of 11.8 using 1 kg/t MBS and 10 g/t A208 plus 40 g/t SIPX in the copper flotation test, and 1,300 g/t $CuSO_4$ plus 200 g/t SIPX in zinc flotation. Only one copper cleaner flotation stage could be tested due to insufficient sample mass. Two stages of zinc cleaner flotation were enough to produce a high quality zinc concentrate due to the very high head assay of the high Zn ore sample.

Simulation of closed-circuit flotation performance indicated that a copper concentrate could be produced assaying 15.7% Cu, 18% Zn, 92 g/t Ag and 4.25 g/t Au at a copper recovery of only 29%. This is a result of the low Cu to Zn ratio in the high zinc ores. In the zinc flotation section, a concentrate was produced assaying 54.9 % Zn, 75 g/t Ag and 2.3 g/t Au at a zinc recovery of 89%.

Flotation tests for the blended samples

Two samples of blended ores were prepared to investigate the effects on flotation performance when mixing different ore types, with the blends as follows:

- Blend 1: 75% Footwall:13% Low Zn:12% High Zn
- Blend 2: 50% Low Zn: 50% High Zn

Blend 1 represents the approximate proportions of the ore types in the South Orebody.

The results show that blending the FW ore sample with the Black ore samples reduced the quality of the copper concentrate and the copper recovery. The copper grade and recovery of the FW ore sample when treated alone were 28.83% Cu and 93.68%, respectively, with a low zinc content of 0.55%. In Blend 1, however, the copper recovery decreased to 60% and the zinc grade in the copper concentrate increased to 7% Zn.

When treating FW alone, zinc flotation was not performed due to a very low head assay (0.09%). The zinc assay increased to 2.6% Zn for Blend 1. However, the performance of the zinc flotation section was not satisfactory. A zinc concentrate could be produced assaying 28% Zn at 59% recovery, compared to concentrate grades of approximately 55% Zn, and recoveries of over 80% when the Black ores were treated alone. A significant amount (19%) of zinc was lost to the copper concentrate.

The results clearly showed that the FW ore sample should not be blended with Black ore samples. Footwall material should be treated separately in a campaign mode of operation. This has the added advantage of not having to operate the zinc flotation section of the plant for 75% of the material treated over the life of mine.

The Blend 2 ore sample comprised 50% low Zn and 50% high Zn ore samples, which is a similar ratio to the expected overall content of the South Orebody. Both ores have similar mineralogical characteristics and required similar flotation conditions and reagent additions when tested separately, except for higher reagent dosages applied to the high Zn sample because of the much higher zinc grade.

Reagent addition rates for the high Zn ore sample were thus applied to Blend 2. The copper and zinc assays of the Blend 2 were 1.05% Cu and 10.07% Zn, respectively.

The simulation results showed that a copper concentrate could be produced assaying 25% Cu and 10% Zn at a copper recovery of 69%. This was a slightly lower quality than that produced from the low Zn ore sample, but much better than the high Zn ore sample. These results showed that blending the high Zn ore sample, having very low Cu/Zn ratio with a higher Cu/Zn ration ore sample (low Zn) from the same ore type, improved the copper flotation performance.

In zinc flotation, a concentrate was produced assaying 58% Zn at a recovery of 86.6%. A high quality zinc concentrate could be produced from both the individual ore samples and from their blend.

Considering the above, it is recommended that there be only two feed types for processing from the South Orebody FW material and Blend 2 Black ore (50% low zinc and 50% high zinc).

The optimum regent additions for these feeds were found to be:

- FW, Yellow ore: for copper flotation, a grind P_{80} of 38 µm, pH 12.5 (with lime), 25 g/t MIBC as frother, 15 g/t A 208, and 60 g/t SIPX as collector

No zinc flotation:

- Blend 2, Black ore: for copper flotation, a of grind P_{80} of 38 µm, pH 12 (with lime), 25 g/t MIBC as frother, 1 kg/t sodium metabisulphite for Cu/Zn selectivity, 10 g/t A 208, and 40 g/t SIPX as collector

For zinc flotation, pH 11.8, 1.3 kg/t $CuSO_4$ for sphalerite activation, and 200 g/t SIPX collector.

13.2.4 Ore variability

No ore variability testwork has been performed on the individual ore types from the South Orebody.

13.2.5 Process recovery projections

The Table 13-7 listed recoveries and concentrate grades are projected for the treatment of South Orebody plant feed.

Table 13-7 Projected recoveries and concentrate grades

	Recoveries, %		Concentrate grades				
	Cu	Zn		% Cu	% Zn	Ag ppm	Au ppm
Footwall (Spec con.)	92.0		**Cu**	23.0	2.5	20.0	1.3
Zinc Ores (Blend 2) (Non-spec Cu con.)	60.0	75.0	**Cu**	19.0	10.0	40.0	5.0
			Zn	5.0	50.0	65.0	3.0

Note that the recoveries and concentrate grades listed above, are lower than those achieved in testwork because the testwork samples had much higher head grades than those scheduled in the LOM mine plan, e.g., for the footwall, the testwork sample was 4.2% Cu, vs the mine plan of 1.74% Cu.

Zinc recovery from footwall ore is not listed in the above table because no payment will be obtained for the low zinc grade in the copper concentrates. As noted in table 13.5, the actual recovery from testwork was 73.55% from a head grade of 0.09% Zn.

13.3 Treatment of South Orebody ores through the existing plant

As noted above, it is recommended that there be only two feed types for processing the feed from the South Orebody, i.e., FW Spec feed and Blend 2 Non-spec feed (50% low zinc and 50% high zinc). These would be treated on a campaign basis.

The geological description of the footwall material, the plant scale test on development material from the South Orebody footwall, and the recent testwork, all indicate that the South Orebody footwall material can be successfully treated through the existing circuit at Çayeli, to produce a Spec copper concentrate.

Past production data indicates that Çayeli previously treated over 1.3 Mtpa of ore grading over 3% Cu. The current LOM mine plan suggests a maximum throughput rate of approximately 850,000 tpa of ore[5], with the highest feed grade being 1.90% Cu, when treating footwall ore alone. The existing concentrate dewatering and handling circuits at Çayeli are thus expected to handle the concentrate quantities generated from this footwall material.

The blending of high and low zinc ores produces zinc feed grades that are typically higher than those seen in the past, but at a lower throughput rate.

[5] This maximum tonnage is before the application of unplanned mining dilution and mining recovery (loss) factors.

Historic production of zinc appears to have occurred in 2010, with a throughput of 1.147 Mtpa and a zinc grade of 6.29% (see Table 13-8). This produced 50,925 tonnes of zinc in concentrates for that year.

Table 13-8 Historic zinc production in peak years

Year	Ore	Head Grades		Zinc Tonnes	
	Tonnage	% Cu	% Zn	Feed	Concentrate
2007	1,046,621	3.83	6.19	64,786	45,664
2008	1,108,590	3.68	6.10	67,624	47,634
2009	1,151,018	3.28	6.24	71,824	50,646
2010	1,147,083	3.21	6.29	72,152	50,925
2011	1,195,472	3.23	5.97	71,370	48,156
2012	1,218,490	3.35	5.21	63,483	41,290
2013	1,332,810	3.10	4.92	65,574	43,678
2014	1,341,067	2.79	4.35	58,336	37,092

Future ore treatment as indicated in the latest production plan lists a maximum production of 20,590 tonnes of zinc in concentrates in 2029; i.e. treating 332,158 tonnes of zinc ore at a feed grade of 8.27% Zn and after 75% recovery. The year 2029, with feed from the South Orebody only, represents the highest tonnage of zinc ores to be treated, and the near-highest grade of zinc ores to be treated in the production plan.

These comparisons suggest that the existing circuit at Çayeli can successfully treat the zinc (Black) ores at the maximum anticipated throughput.

However, it should be noted that treatment of high zinc grade ores on their own may cause some difficulties in concentrate dewatering and handling due to the quantity of material to be handled. The high grade zinc ores must be blended with low zinc grade ores for successful treatment, or they must otherwise be treated at a reduced throughput rate to maintain the zinc production rate (in tonnes per hour) within plant design limits.

13.4 Conclusions

Item 15.3.2 includes a table of annual and life of mine average processing recoveries that were used for mine planning and as modifying factors for Mineral Reserve estimation. This information was derived from a preliminary production schedule and metallurgical information available at that time.

After the event, and now with the benefit of new metallurgical information, the following commentary provides updated projections that can be used for cashflow modelling.

13.4.1 Main Orebody plant feed

Plant feed from the Main Orebody remnant mining will be processed in Years 2025 to 2027, and in 2035 and 2036. Projected average recoveries and concentrate grades for this feed are based on historical data. These projections are listed in Table 13-9.

Table 13-9 Projected average recoveries and concentrate grades for Main Orebody plant feed

	Recoveries, %		Concentrate grades				
	Cu	Zn		% Cu	% Zn	Ag ppm	Au ppm
Yellow Ore (Spec con.)	92.0		**Cu**	22.0	2.4	45.0	1.5
		30.0	**Zn**	5.0	40.0	0.0	0.0
Black & Clastic Ores	84.0		**Cu**	17.0	12.0	94.0	1.7
(Non-spec Cu con.)		67.0	**Zn**	5.0	40.0	94.0	0.0

13.4.2 South Orebody plant feed

Ore mined from the South Orebody will be metallurgically very similar to ores mined from the Main Orebody, with footwall (FW) ores being similar to Yellow ores currently being treated, and the zinc ores being similar to the Black ores.

Three ore types from the South Orebody were tested:

- Footwall (FW), Yellow ore (SYO), copper rich, with low zinc content
- Black ore zone (SBO), low grade zinc (Zn < 8%)
- Black ore zone (SBO), high grade zinc (Zn > 8%)

Strictly speaking, the low and high grade zinc ores are not separate and must be blended as SBO only, separate from SYO plant feed.

Mineralogical investigations, a plant trial on South Orebody development material, and comminution and flotation testwork at Hacettepe University (2024a and 2024b) have confirmed that these ores can be successfully treated through the existing processing facilities at Çayeli, with no modifications required to the plant.

However, the following recommendations apply:

- The different ores must be campaigned through the plant, with FW (SYO) material treated separately from the zinc (SBO) ores.
- High and low grade SBO zinc ores should be blended in a 1:1 ratio (similar to their occurrence in the Main Orebody) to control zinc grades and also enable a Non-spec copper concentrate to be produced.
- To avoid overwhelming the zinc concentrate treatment circuit, the overall zinc feed grade should be controlled.

Table 13-10 lists the recommended overall average recoveries and concentrate grades for the treatment of South Orebody plant feed.

Table 13-10 Projected average recoveries and concentrate grades for the South Orebody plant feed

	Recoveries, %		Concentrate grades				
	Cu	Zn		% Cu	% Zn	Ag ppm	Au ppm
Footwall (Spec con.)	92.0		**Cu**	23.0	2.5	20.0	1.3
Zinc Ores (Blend 2)	60.0	75.0	**Cu**	19.0	10.0	40.0	5.0
(Non-spec Cu con.)			**Zn**	5.0	50.0	65.0	3.0

Item 14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The Çayeli Mineral Resource estimate was prepared in May 2025 by Mr Richard Sulway (QP), with Mr David Gray of FQM as a supporting author.

All available drill data was used for the geological model interpretation, as well as the spatially related copper mineralisation interpretations. Interpolation parameters were based upon the geology, styles of mineralisation, 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 – "Datamine" and Snowden Supervisor – "Supervisor").

Mineral Resource estimates were classified according to geological continuity, QAQC, density data, drillhole grid spacing, grade continuity and confidence in the panel grade estimate. The Mineral Resource estimates 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, 2019).

14.2 Data

The estimates cover the remnant mining areas in the Main Orebody and the newly defined South Orebody mineralisation.

14.2.1 Diamond drilling data

All drilling data (including logging, collar surveys, downhole surveys, assays and density results) is stored in a centralised MS Excel file with separate work sheets used to store each data group. Effectively, it is a relational style database. The ability to access and modify this file is restricted to the site Senior Geologist.

The data is validated both visually and again when it is desurveyed, using the site mining software package (MineSight3D). The software reports common table issues, such as overlapping from and to sample intervals and duplicate records.

The source drilling data used in the estimate is listed in Table 14-1.

Table 14-1 Supplied drilling data

Supplied Drilling Data	Description
Database-2025-03-20-Drillhole.xlsm	Collars, surveys, assays and logging data based on a nominal 2 m sample interval
Database-2024-02-05-Composite.xls	Assays table of gold assays prepared from 6 m composite samples.

Owing to its relatively low grade, patchy distribution, and poor processing recovery, gold is not considered economically significant in terms of the concentrates produced by the mine.

Composite samples are prepared for analysis from the pulverised 2 m samples. Typically, three sequential samples are composited to produce a single sample (nominal 6 m interval) for analysis.

The csv tables exported from the MS Excel file were imported into Datamine software and desurveyed (i.e., the sample tables were merged and the local grid coordinates were added to each sample interval). The desurveyed drilling database was clipped to the limits listed in Table 14-2 to exclude drilling from outside of the immediate Operations area and or parts of the Main Orebody which have been mined out. All drilling statistics and grade estimates in this Report are derived from the clipped drilling data.

Table 14-2 Drilling collar limits used for the 2025 Mineral Resource estimate

	Minimum	Maximum
Easting	250	1500
Northing	750	1950
RL	800	9999999

A north-south section through the Main and South Orebodies, and associated clipped drilling, is shown in Figure 14.1. The desurveyed drillhole files initially created for validation purposes and as inputs to the grade estimation process are listed in Table 14-3.

Figure 14.1 North-South Section (1000 mN) showing the clipped Çayeli drilling and topography

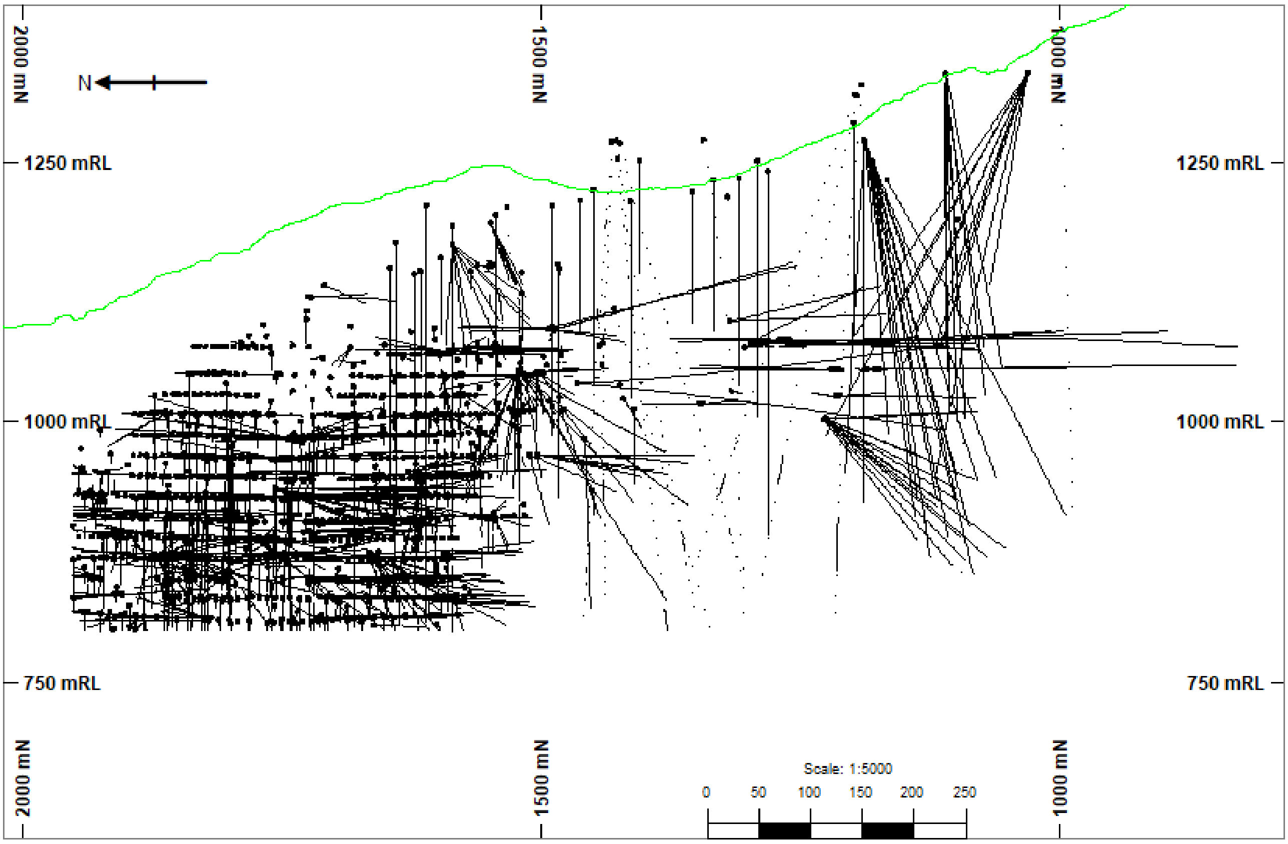


Table 14-3 Raw desurveyed Datamine drillhole files

File Name	Description
cay_hole	Includes grade data for Cu, Zn, Pb and Ag (2 m composites).
cay_hole_au	Includes grade data for Au only (6 m composites)
cay_hole_geol	Includes both the base metal grades and the geological logging. This drillhole file was used as a guide when compiling the geological interpretations (wireframe), particularly the hangingwall/footwall contact and the massive sulphide zones. This file was not used for grade estimation purposes.

14.2.2 Grade field conventions

All domain and grade fields were stored in an uppercase format for consistency. The five estimated grade fields are listed in Table 14-4.

Table 14-4 Çayeli grade attribute field names

Drilling/model field name	Database name
CUPCT	CU
ZNPCT	ZN
PBPCT	PB
AGPPM	AG
AUPPM	AU

14.2.1 Density data

Bulk density data has been collected from core samples which are representative over the life of the mine. The bulk density measurements were collected using variations of the water displacement method. This involves taking selected lengths of core (approximately 10 cm to 20 cm length), weighing them in air and then again while submerged in water. The basic formula used to calculate bulk density is shown in Figure 14.2.

Figure 14.2 Bulk density formula

Bulk Density=A/(A-B)

Note: A = weight of sample in air, B = weight of the sample in water.

As observed during the QP's 2024 site visit, the general approach being used to measure density and manage the resulting data is considered to be reasonable. The density results are stored in the assays table using the "from" and "to" interval which includes the downhole "from" and "to" depth of the actual piece of core used to measure each density value. As such, there is no need to compile a separate desurveyed density table for estimation purposes.

14.2.2 Sludge drilling data

As with the drilling data, the sludge data is stored in a dedicated MS Excel file managed by Mr Kadir Tolga Güngör. The data is stored in a pseudo drillhole format (assays, collars and surveys tables); the supplied file name was Database-2025-03-04-Sludge.xls. The grab samples cannot be collected in a probabilistic manner (i.e., it is not possible). The data was imported into Datamine and desurveyed and was used with the drilling data to compile the geological interpretation. However, due to the grade bias risk, the data was not used in the grade estimation process.

14.2.3 Local grid

Çayeli uses a local grid conforming to the UTM-ITRF coordinate system. The local grid is used to reduce the number of digits in the easting and northings, increase the RL values by 1,000 to avoid negative mine level numbers and to finally rotate the grid so that the eastings are roughly parallel to the strike of the orebody.

14.2.4 Surface topography

The surface topography was compiled from a LiDAR survey which was flown in 2020. This survey captured reliable topography data to enable gravity terrain corrections to be applied to the geophysical surveys that were completed in 2019. The Datamine files compiled from the LiDAR data are listed in Table 14-5.

Table 14-5 Datamine format topography surface (DTM) file

File name	Description
topo290724tr/topo290724pt	Single DTM surface covering the entire Çayeli Project area that is being modelled.

The surface file was used to exclude any blocks created by the volume modelling process lying above the current topographic surface. In other words, there are no “air” blocks incorporated into the model.

14.2.5 Drill hole database validation

The drilling database was checked whilst importing the files into Datamine during the desurveying process. Specifically, the checks that were completed during this process included identifying:

- overlapping sample intervals in the sample tables
- duplicate sample intervals in the sample tables
- duplicate records in the collars and surveys tables
- inconsistent from and to values in the sample tables
- absent and or missing collar or survey data

Several small discrepancies were identified and corrected prior to proceeding with the Mineral Resource estimate. No material flaws were identified.

14.2.6 Treatment of absent data

Absent field values were denoted as blank values in the database tables. In the case of numeric fields, these were treated as nulls when loading the data and storing it in Datamine tables.

14.2.7 Other data fields

An indicator field was estimated: BO_IND, containing indicator values of between 0 and 1. These values were determined by a visual estimate of the existence of bornite, and expressed as a percent. This field was estimated only for metallurgical purposes and is not relevant to the Mineral Resource estimate. The raw drilling data is stored in the assays table.

14.3 Modelling domains

14.3.1 Domaining criteria

The Çayeli orebody domains were interpreted by means of the following approach and criteria:

1. The block model was clipped to the topography surface (refer Item 14.2.4).
2. All domain wireframes were compiled on an east-west sectional basis using the explicit approach. Site compiled geology domain wireframes do not exist, as all site-based estimation is largely unconstrained.
3. The strings/wireframes compiled for the South Orebody were snapped to the drilling intercepts according to the sample and logging interval endpoints. In the case of the Main Orebody, the significant density of drilling in some areas meant at times that this was not possible, and in which case point locations were approximated.
4. All grade and density estimations were undertaken using wireframe-based domains with hard boundaries. Where values could not be estimated due to insufficient drilling information, mean values were assigned.
5. To date, during the life of the mine, oxidation has not been considered a material issue either to estimating Mineral Resource tonnages or to processing recovery. Hence, it is not modelled as part of the current site grade control models. Relevant to this, is that the South Orebody is about 100 m below the surface and as such is unlikely to be impacted by significant levels of oxidation. Oxidation was not interpreted or modelled as part of this Mineral Resource estimate.
6. The hangingwall basalts and tuffs are essentially un-mineralised. There are however two small pods of mineralisation known to exist in this rock unit. They are probably the result of slices of mineralisation being offset by faulting. The low tonnage is considered to be not economically significant. In the final model these pods were allocated mean grades and flagged as not classified.
7. A hanging wall surface was interpreted on east-west sections along the full strike length of the volume being modelled. This surface was used to:
   - delimit the hangingwall basalts/tuffs from the footwall rhyolite in the Mineral Resource model
   - provide a guide to the western limit of the mineralisation (excluding the two pods described previously)
8. The surface was interpreted by referencing the geological logging records (*cay_hole_geol*) drillhole file. This surface also provides an important guide for targeting future exploration drilling.
9. While there is a semi-natural break in the mineralised volumes between the Main and South Orebodies, at around 1350 mN, it is more a case of the mineralisation being patchy in this area rather than the area being barren. Further drilling in this area will very likely identify more mineralisation, however the zones will be relatively small. The "official" break between the Main and South Orebodies as defined by ÇBI is 1330 mN. This break northing is mainly used for reporting purposes. While a small fraction of the South Orebody crosses this line, the impact is negligible. A plan view of the Cu mineralisation wireframes from both the Main and South Orebodies, along with the 1330 mN break line, is shown in Figure 14.3.
10. There are distinct higher and lower grade Cu and Zn zones which correspond to massive sulphide and stockwork styles of mineralisation, respectively. As described in Item 7.3, this pattern is evident in both orebodies. The four zones in each of the orebodies were modelled using a combination of grade thresholds and geological logging. The grade thresholds where derived from histograms and probability plots of the un-domained drilling data. The domaining criteria are listed in Table 14-6.
11. The Pb and Ag grades were estimated based on the Zn domains. This decision was made on the basis of observed grade trends and linear correlations (i.e., Pearson's correlation matrix) between the various metals.
12. The Au mineralisation is both low grade and very patchy. Where small contiguous zones could be identified they were invariably associated with the Zn massive sulphide zones. No other material correlations could be identified. Au grade was estimated based on the massive Zn domains.

13. Analysis of the density data identified three distinct statistical domains namely:
    - Massive Cu and Zn sulphide in both deposits
    - Footwall rhyolite, both mineralised and unmineralised
    - Hangingwall mafic unit
14. Density was estimated for the Massive and Footwall zones. A mean density value was assigned to the Hangingwall unit as it is barren, and the data is very patchy in this area.

Figure 14.3 Plan view of the Main and South Orebodies showing the Cu massive and stockwork mineralisation wireframes

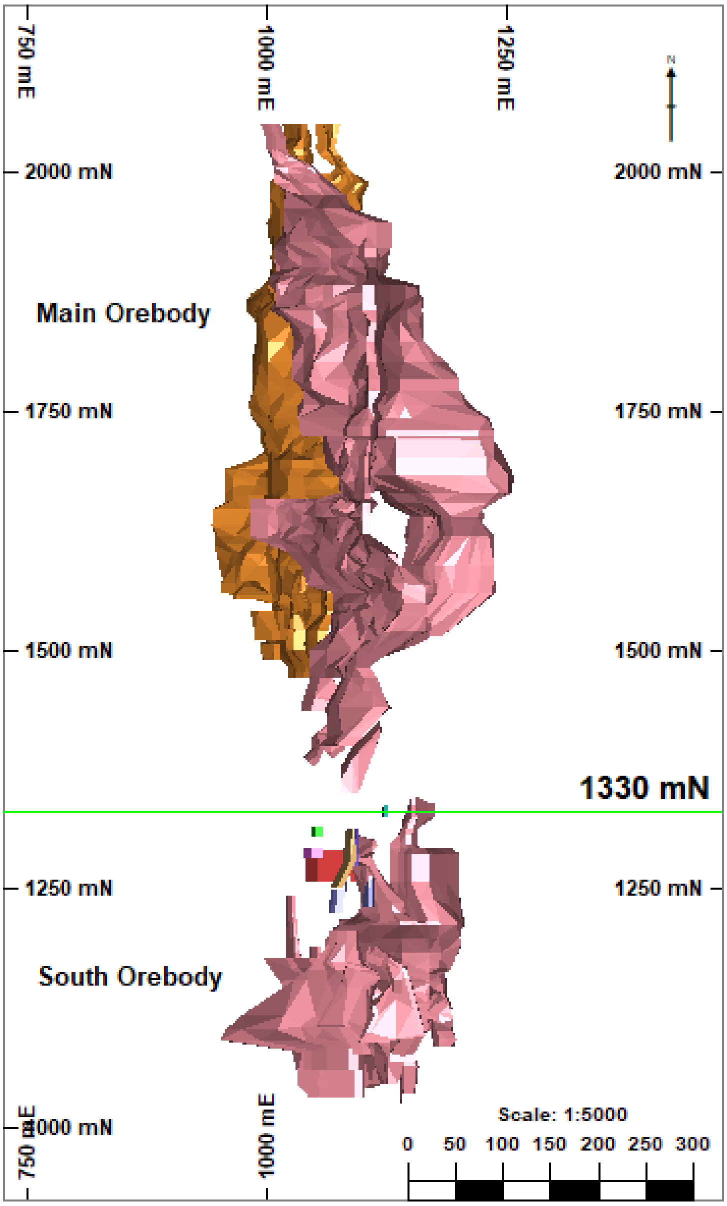


Table 14-6 Cu and Zn domaining criteria

Orebody	Mineralisation	Domaining criteria
Main Orebody	Massive Cu Mineralisation	1.5% Cu cut-off and geological logging
	Stockwork Cu Mineralisation	0.25% Cu cut-off
	Massive Zn Mineralisation	0.5% Zn cut-off and geological logging. Most values are >3 and relatively contiguous. The contact is less sharp than the South Orebody.
	Stockwork Zn Mineralisation	0.5% Zn cut-off. Grades are notably lower than the massive zone and much more patchy and irregular. The Zn stockwork tonnage in the Main Orebody is relatively minor compared to the Massive Zn mineralisation.
South Orebody	Massive Cu Mineralisation	1.5% Cu Cut-off and geological logging
	Stockwork Cu Mineralisation	0.25% Cu cut-off
	Massive Zn Mineralisation	0.5% Zn cut-off and geological logging. Most values are >3 and relatively contiguous.
	Stockwork Zn Mineralisation	0.5% Zn cut-off. Grades are notably lower than the massive zone and much more patchy and irregular.

14.3.2 Mineralisation, rock type wireframes and associated domains

The wireframe file names used to code the block model and the corresponding drilling with the mineralisation and rock type domains, are listed in Table 14-7. The corresponding CUMINDOM, ZNMINDOM and ROCK domain values with descriptions are listed in Table 14-8 and Table 14-9.

Table 14-7 Mineralisation, rock type wireframes and domain field names

File Names	Orebody	Domain Field Name	Description
mn_cumassive_mintr/ mn_cumassive_minpt	Main	CUMINDOM	Massive Cu Sulphide wireframe consisting of one main solid and seven minor solids
mn_custockworktr/ mn_custockworkpt	Main	CUMINDOM	Cu Stockwork mineralisation consisting of one main solid and two minor solids
mn_znmassive_mintr/ mn_znmassive_minpt	Main	ZNMINDOM	Massive Zn sulphide wireframe consisting of one main solid and nine minor solids
mn_znstockworktr/ mn_znstockworkpt	Main	ZNMINDOM	Zn Stockwork mineralisation consisting of two minor solids
su_cumassive_mintr/ su_cumassive_minpt	South	CUMINDOM	Massive Cu Sulphide wireframe consisting of one main solid
su_custockworktr/ su_custockworkpt	South	CUMINDOM	Cu Stockwork mineralisation consisting of one main solid and 17 minor solids
su_znmassive_mintr/ su_znmassive_minpt	South	ZNMINDOM	Massive Zn sulphide wireframe consisting of one main solid and one minor solid
su_znstockworktr/ su_znstockworkpt	South	ZNMINDOM	Zn Stockwork mineralisation consisting of nine minor solids
mafic_hw_contacttr/ mafic_hw_contactpt	Main and South	ROCK	Sub vertical open surface used to delimit the mafic hanging wall (basalts, tuffs) and the felsic foot wall (rhyolite)

Table 14-8 CUMINDOM and ZNMINDOM domain values

Orebody	CUMINDOM	ZNMINDOM	Description
Main	201	201	Massive sulphide, Cu and Zn, main zone
	601	601	Stockwork sulphide, Cu and Zn, main zone
	500	500	Host rhyolite, with weak or no mineralisation
South	101	101	Massive sulphide, Cu and Zn
	501	501	Stockwork sulphide, Cu and Zn, main zone
	502	502	Minor Cu and Zn Stockwork sulphide
	503	N/A	Minor Cu Stockwork sulphide
	504	N/A	Minor Cu Stockwork sulphide
	505	N/A	Minor Cu Stockwork sulphide
	506	N/A	Minor Cu Stockwork sulphide
	507	N/A	Minor Cu Stockwork sulphide
	508	N/A	Minor Cu Stockwork sulphide
	500	500	Host rhyolite, with weak or no mineralisation
Hangingwall Lens	801	802	Scatted small pod of mineralisation in the basalt/tuff zone
	802	802	Scatted small pod of mineralisation in the basalt/tuff zone

Table 14-9 ROCK Domain Values

Orebody	CUMINDOM	Description
Main and South	1000	Hangingwall basalts and tuff
	2000	Footwall rhyolite

14.3.3 Calculated mineralisation domains

Two additional domain names, AUMINDOM and DENSITYDOM, were defined using existing fields in order to estimate Au and DENSITY, respectively. The domain field names and values, and the formulae used to set them, are listed in Table 14-10.

Table 14-10 AUMINDOM and DENSITYDOM field definition

Domain field name	Domain field value	Description	Field definition formula
AUMINDOM	0	Hangingwall mafics	ROCK=1000
	101	South Orebody, massive Zn sulphide	ZNMINDOM=101
	201	Main Orebody, massive Zn sulphide	ZNMINDOM=201
	500	Footwall rhyolites, mineralised and not mineralised including Cu massive sulphides	AUMINDOM!=0 and AUMINDOM!=101 and AUMINDOM!=201
DENSITYDOM	0	Hangingwall mafics	ROCK=1000
	1	South Cu and Zn massive sulphide zones	CUMINDOM=101 or ZNMINDOM=101
	2	Main Cu and Zn massive sulphide zones	CUMINDOM=101 or ZNMINDOM=101
	9	Footwall rhyolites, mineralised and not mineralised	DENSITYDOM!=1 AND DENSITYDOM!=2 and DENSITYDOM!=0

14.4 Drill hole flagging and compositing

14.4.1 Drillhole flagging

The domain fields CUMINDOM, ZNMINDOM, AUMINDOM and DENSITYDOM were flagged into the desurveyed drillhole files as listed in Table 14-11.

Table 14-11 Drillhole flagging

Input drillhole filename	Output drillhole filename	Domain fields added
cay_hole	cay_hole_f	CUMINDOM, ZNMINDOM, ROCKDOM, DENSITYDOM
cay_hole_au	cay_hole_au_f	AUMINDOM

14.4.2 Drillhole compositing

There is little merit in considering compositing of the *cay_hole_au_f* file given the samples represent large composite values aggregated by combining smaller samples for analysis. Hence, compositing was not applied to this file.

In terms of the main assay drillhole file (*cay_hole_f),* and while the nominal sample interval is 2 m, the actual sample intervals used are quite variable. When excluding samples with absent Cu grade (i.e., removing long lengths of unsampled core), the percentage of core samples using the most common sample intervals, 1 m and 2 m, is 30% and 38% respectively. The *cay_hole_f* desurveyed drillhole file was composited to 2 m in line with the larger of the two dominant intervals used with the final file called *cay_hole_fc*.

Sample compositing occurs within the individual domains to ensure that no composite interval crosses the domain boundaries. To allow for uneven sample lengths within each of the domains, the Datamine composite process (COMPDH) was run using the variable sample length method (@MODE=1). This adjusts the sample intervals where required to ensure that all samples are included in the composite file (i.e. no residuals) while keeping the composite interval as close to the desired sample interval as possible.

14.5 Volume modelling

The model prototype was compiled as per the settings detailed in Table 14-12. The adopted maximum block sizes were considered appropriate for the general drillhole spacing (40 mE by 40 mN) in the South Orebody. The minimum block sizes were used to model the volumes of the underground drives and ventilation rises in the South Orebody area and the remnant stopes in the Main Orebody area. In areas not located near any remaining stopes (Main Orebody) or underground development (South Orebody) the minimum block size is typically 5m by 5m by 5m (X, Y and Z directions respectively).

Table 14-12 Çayeli volume model prototype

Model setting	Value
X Origin	800 mE
Y Origin	900 mN
Z Origin	800 mRL
Maximum Easting	1440 mE
Maximum Northing	1950 mN
Maximum Elevation (RL)	1500 mRL
Parent cell size – X	20 m
Parent cell size – Y	20 m
Parent cell size – Z	20 m
Minimum cell size – X	1 m
Minimum cell size – Y	1 m
Minimum cell size – Z	1 m

14.5.1 Depletion due to mining

Over thirty years of mining in the Main Orebody and three years of development with minimal stoping in the South Orebody means that no single approach to incorporating mining depletion into the volume model will be effective. As such a different approach for each of the two deposits was used as described in the following paragraphs.

Main Orebody

The Main Orebody solids (wireframes) reflecting thousands of mined development openings and stope wireframes, presented an impractical basis for modelling mining depletion from over thirty years of operations. The solution used was to effectively reverse the problem by instead coding the model with the remaining stopes (wireframe solids of unmined material which were scheduled to be mined). Everything outside these planned mining shapes was flagged as unclassified, as though effectively having been mined.

The site mine planning engineers supplied the remaining stope design solids in ten dxf files, one per sublevel on the 30th of May 2025. The dxf files were converted to a single Datamine format wireframe file (MOB_stopes_300525tr/MOB_stopes_300525pt)[6] and a block model of the planned stopes was built to a 1m by 1 m by 1 m resolution (X, Y and Z directions respectively). The stope models were built one at a time and combined due to overlapping wireframes.

[6] MOB refers to the Main Orebody

South Orebody

The depletion of the South Orebody, which has only been mined since December 2023, was a much simpler task when compared with the Main Orebody. Most of the South Orebody mining to date has consisted of level development with a negligible amount of stoping. The site engineers supplied mine surveys (wireframe solids) of all underground development as at the 25th of March 2025. The stope wireframes were combined with the surveyed development wireframes and modelled in the volume model to a 1 m by 1 m by 1 m resolution. Blocks inside the final model were depleted for mining by setting the DENSITY field to 0.001 and the Resource classification to unclassified.

14.5.2 Volume resolution as a function of drillhole spacing (SMUDRSCL field)

The drillhole spacing in the Main and South Orebodies is quite variable. As an example, in the mineralised volumes in the Main Orebody, the deposit is densely drilled often to an average drillhole spacing of about 10 m by 10 m by 20 m. Conversely in the South Orebody, the drillhole spacing is mostly much larger, being approximately 40 m by 40 m by 40 m, or greater. With ongoing development and drilling in the South Orebody, the orebody will be infilled drilled to a similar density as seen in the Main Orebody.

To align the volume resolution of the model based on the drilling density to the estimation process, a field called SMUDRSCL was coded into the model to delimit the different drilling densities as listed in Table 14-13.

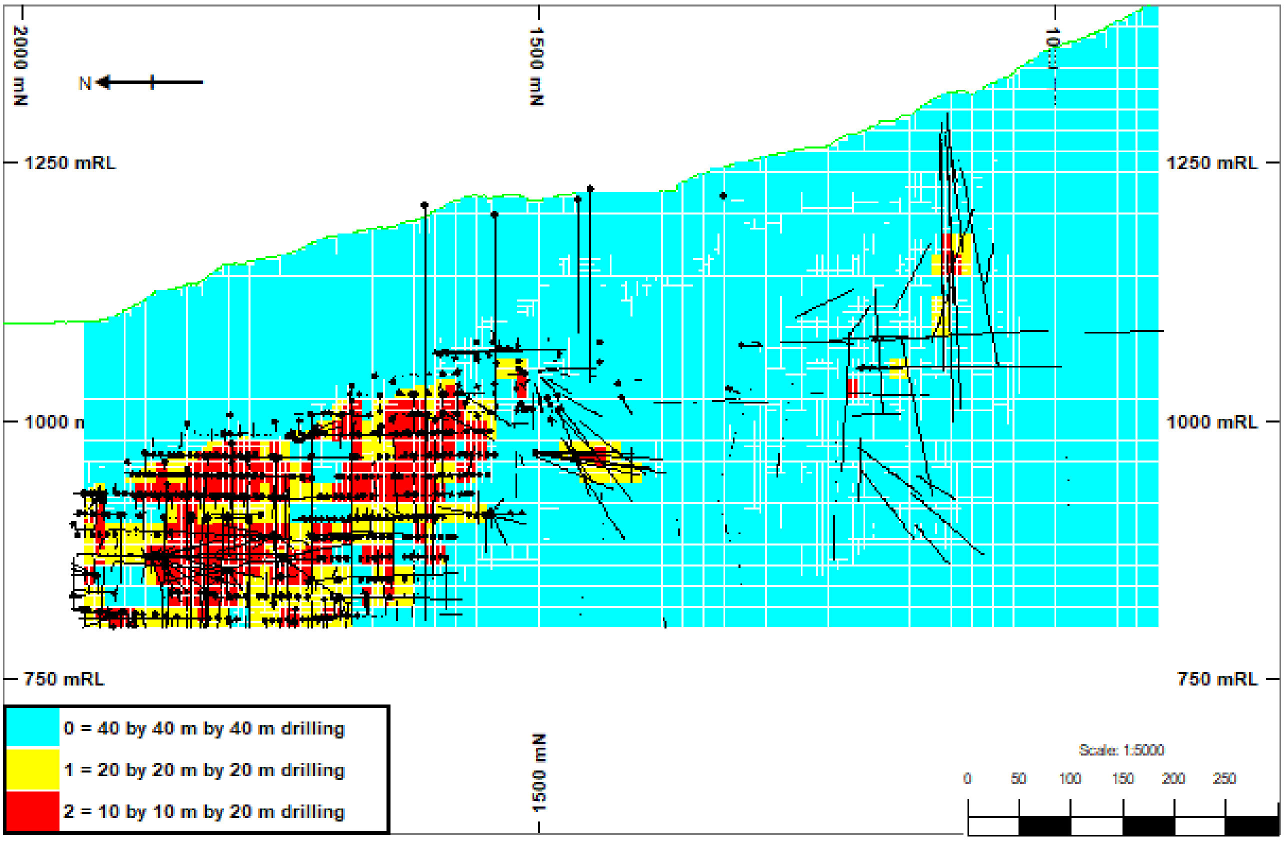
Table 14-13 SMUDRSCL field values and descriptions

SMUDRSCL field value	Parent block size for estimation (X, Y and Z)	Description
0	20 m by 20 m by 20 m	Typically, 40 m by 40 m by 40 m drillhole spacing, or larger away from the mineralised zones
1	10 m by 10 m by 10 m	Typically, 20 m by 20 m by 20 m drillhole spacing
2	5 m by 5 m by 10 m	Typically, 10 m by 10 m by 20 m drillhole spacing

In other words, volumes with closed spaced drilling are estimated using a smaller parent block size compared to volumes which are more sparsely drilled. This means that the grade resolution of the final model reflects the drillhole spacing at the time that the model was compiled. The alternative of using a fixed block size would result in over smoothing of the grade estimates in some parts of the model, depending on the block size that was chosen.

Figure 14.4 illustrates a long section through both orebodies showing blocks colour coded on the SMUDRSCL field value.

Figure 14.4 North-South section (10400 mN) showing the clipped (±25 m) Çayeli drilling, topography and a slice through the model coloured on SMUDRSCL



The SMUDRSCL field was set using a series of nearest neighbour estimates (Cu) with the decision criteria honouring the number of samples and drillholes identified for each block estimate. The actual estimated grade is irrelevant. The estimation settings were optimised through a process of trial and error and visual checking to produce a final set of parameters. The approach, by its nature, is an approximation but it does yield reasonable results. The parameters used are listed in Table 14-14.

All blocks were initially coded as SMUDRSCL=0 and then reassigned a value of 1 or 2 in instances where there was sufficient sampling data in the vicinity of each block to warrant the change.

Table 14-14 SMUDRSCL nearest neighbour parameter settings

SMUDRSCL value	Parent block size (X, Y and Z)	Search ellipse size (X, Y and Z)	Minimum number of samples	Minimum number of drillholes
0	N/A	N/A	N/A	N/A
1	20 m by 20m by 20 m	20 m by 20m by 20 m	30	14
2	10 m by 10m by 20 m	10 m by 10m by 20 m	17	3

14.6 Grade estimation

14.6.1 Estimation methodology

For grade estimation purposes, an assessment of grade data per domain identified the following estimation criteria:

- The Cu and Zn massive and stockwork sulphide domains were all estimated separately.
- All grades and density data are suitable for estimation using ordinary block kriging ("OK"). Some top cutting to stop the excessive smearing of relatively high grades into the surrounding relatively lower grade areas was required.
- All domain boundaries were set as "hard". From visual observations, most of the boundaries between the various mineralised domains are fairly sharp.
- Mineralisation domains (CUMINDOM, ZNMINDOM, AUMINDOM AND DENSITYDOM) were flagged in the drilling and model files used as inputs to the estimation process. This step was done to control the estimation process by using conventional zone control.
- Dedicated estimation fields were compiled by using the existing fields to govern each estimate run. This allowed the various estimation domains (grade and density) to be coded with rock type information.
- Search ellipse ranges were set in recognition of the 80% variance range in the various Cu variogram models.
- Mineralisation domains with less than fifty samples were allocated domain means.
- The density data in the hang wall domain (basalts) is problematic. Many historic holes have been allocated fixed density values more in line with mineralised rhyolite, rather than barren basalt. The reason for this is not known. The hangingwall strata was allocated a mean density value after much of the data from that domain was excluded to remove anomalous values. This is not a significant issue as most development and all stoping occurs in the footwall strata.
- For each domain estimate, three iterations were run to support estimation based on the SMUDRSCL values. In practice this means resetting the model prototype for each category, estimating the grades and then resetting the model prototype back to the default setting.
- Any blocks not estimated or assigned negative grades were allocated domain referenced mean grades. There is no absent grade or density data in the final model.
- The hangingwall domain (ROCKDOM=1000) was not estimated for grades due to limited sampling information and the fact that it is not host to any economic mineralisation.

14.6.2 Estimation methods

All density and grade estimates were compiled using Datamine software, specifically the COKRIG process, which is part of the advanced estimation model. All estimates were compiled using ordinary block kriging (OK) i.e. not co-kriging. This process was used as it is considerably faster than the older commonly used Datamine ESTIMA process as it is multithreaded. The names of the search, estimation and variogram parameter files used to estimate the grade and density data are listed in Table 14-15.

Table 14-15 Datamine estimation parameter file names

File name	Description
cay_spar	Search parameters for ordinary kriging
cay_epar_ok	Estimation parameters for ordinary kriging
cay_vpar_ok	Variogram parameters for ordinary kriging

14.6.3 Grade estimation domain fields

Four estimation domain fields (ESTFLAGCU, ESTFLAGZN, ESTFLAGAU and ESTFLAGASDN) were set in the model and drillhole files to control (i.e. apply domains) the estimation process. The field values in the model and drillhole files are summarised in Table 14-16 and Table 14-17. The criteria were set using the Datamine EXTRA process.

Table 14-16 Estimation domain field value criteria (ESTFLAGCU)

Field name	Field values	EXTRA formula	Comment
ESTFLAGCU	5101 5201 5500 5501 5601 5502 5503 5504 5505 5506 5507 5508 1801 1802	ESTFLAGCU = ROCKDOM + CUMINDOM	Used to estimate both mineralised and background grades. (5500) Domains 1000 and 5501 to 5508 were all allocated mean grades Domain was only used to estimate CU Domains 1801 and 1802 were allocated mean grades (two pods of hanging wall mineralisation)

Table 14-17 Estimation domain field value criteria (ESTFLAGZN, ESTFLAGAU and ESTFLAGDN

Field name	Field values	EXTRA formula	Comment
ESTFLAGZN	1000 5101 5500 5501 5502 5201 5601 1801 1802	ESTFLAGZN = ROCKDOM + ZNMINDOM	Used to estimate both mineralised and background grades. (5500) Domain was used to estimate Zn, Pb and Ag Domains 1000 and 5502 were allocated mean grades. Domains 1801 and 1802 were allocated mean grades (two pods of hanging wall mineralisation)
ESTFLAGAU	1000 5101 5201 5500	ESTFLAGAU = ROCKDOM + AUMINDOM	Used to estimate both mineralised and background grade (5500) Domain was only used to estimate Au Domains 1000 was allocated mean grades
ESTFLAGDN	1000 5001 5002 5009	ESTFLAGAU = ROCKDOM + DENSITYDOM	Domain was only used to estimate Density Domains 1000 was allocated mean grades

14.6.4 Drillhole files

Two separate drillhole files were used to compile the estimate; the drillhole file names are listed in Table 14-18. A separate file was used for the Au estimates due to the different composite interval used (6 m), compared to the *Cay_hole_fc_est* file (2 m). The larger composite interval and the fact the samples are composite results, meant that estimating grades using the SMUDRSCL field was considered to be of little value.

Table 14-18 Drill holes files used for the Mineral Resource estimate

Drillhole file	Description
cay_hole_fc_est	Used to estimate density and all grades excluding Au
cay_hole_auf_est	Au grade estimates

14.6.5 Search parameters

Defined search parameters for all grades adhered to the Cu variogram modelling for the stockwork domains in the Main and South Orebodies and the surrounding rhyolite host rocks. The basis for this decision was the variogram ranges and directions of continuity in these domains being more clearly defined than for the massive sulphide zones or the Zn stockwork zones. The key search parameter file settings are listed in Table 14-19.

Table 14-19 Search ellipse parameters

Estimation domain	Datamine rotation (Z, X, Z)	Axis lengths (m)			First pass			Second pass		
		X	Y	Z	Expansion Factor	Minimum Samples	Maximum Samples	Expansion Factor	Minimum Samples	Maximum Samples
Main Orebody Mineralisation Domains	0, 90, -90	40	40	100	N/A	6	18	2	6	18
South Orebody Mineralisation Domains	160, 90, 90	40	40	40	N/A	6	18	2	6	18
Rhyolite Host Rocks	0, 0, -90	100	50	100	N/A	6	18	0	N/A	N/A

Search criteria were optimised as follows:

- The selection of samples was clipped (using the Datamine MAXKEY parameter) so that a maximum of six composites per drillhole was used when estimating the mineralised zones, and three samples when estimating the host rocks. This constraint was applied to stop individual block estimates being informed from only one drillhole composite.
- Dynamic search volumes were applied to the minimum sample number limits to help with the grade interpolation process. These methods work by checking for each block estimate, that the minimum sample count criteria are met (i.e. a grade was estimated). Where this was not the case, the search ellipse was expanded by a factor and a second attempt was made to estimate the block grade. If this second attempt failed, then the affected blocks remained un-estimated.

14.6.6 Variogram modelling

All variograms (grades and density) were modelled using Datamine Supervisor software, adopting the approach described below. The model parameters are all recorded in the variogram parameter files (see Table 14-15).

Due to the positively skewed nature of some of the grade domains, normal scores variograms were modelled for all the domains. This method produces a clearer image of the ranges of continuity in skewed data sets. Downhole variograms were modelled to determine the nugget, followed by directional variography.

Variograms were modelled using the following general approach:

- variograms were standard Cartesian coordinates (Mine Grid)
- all variogram variances were standardised to a sill of one
- all grades were modelled using two or three structure spherical variograms
- the nugget and sill values were then back transformed to traditional variograms using the discrete gaussian polynomials technique (Guibal et al, 1987), prior to estimation

An example is shown in Figure 14.5.

14.6.7 Top-cuts

The presence of relatively high (>1.5) coefficient of variation ("CV") values for some of the grades estimated using ordinary kriging required top-cuts to be applied to prevent overestimation and smearing of the relatively high values (when compared with most of the results) into the surrounding blocks. Top-cutting of grades involved resetting the grades that exceed a top-cut value to the top-cut value, on a domain-by-domain basis.

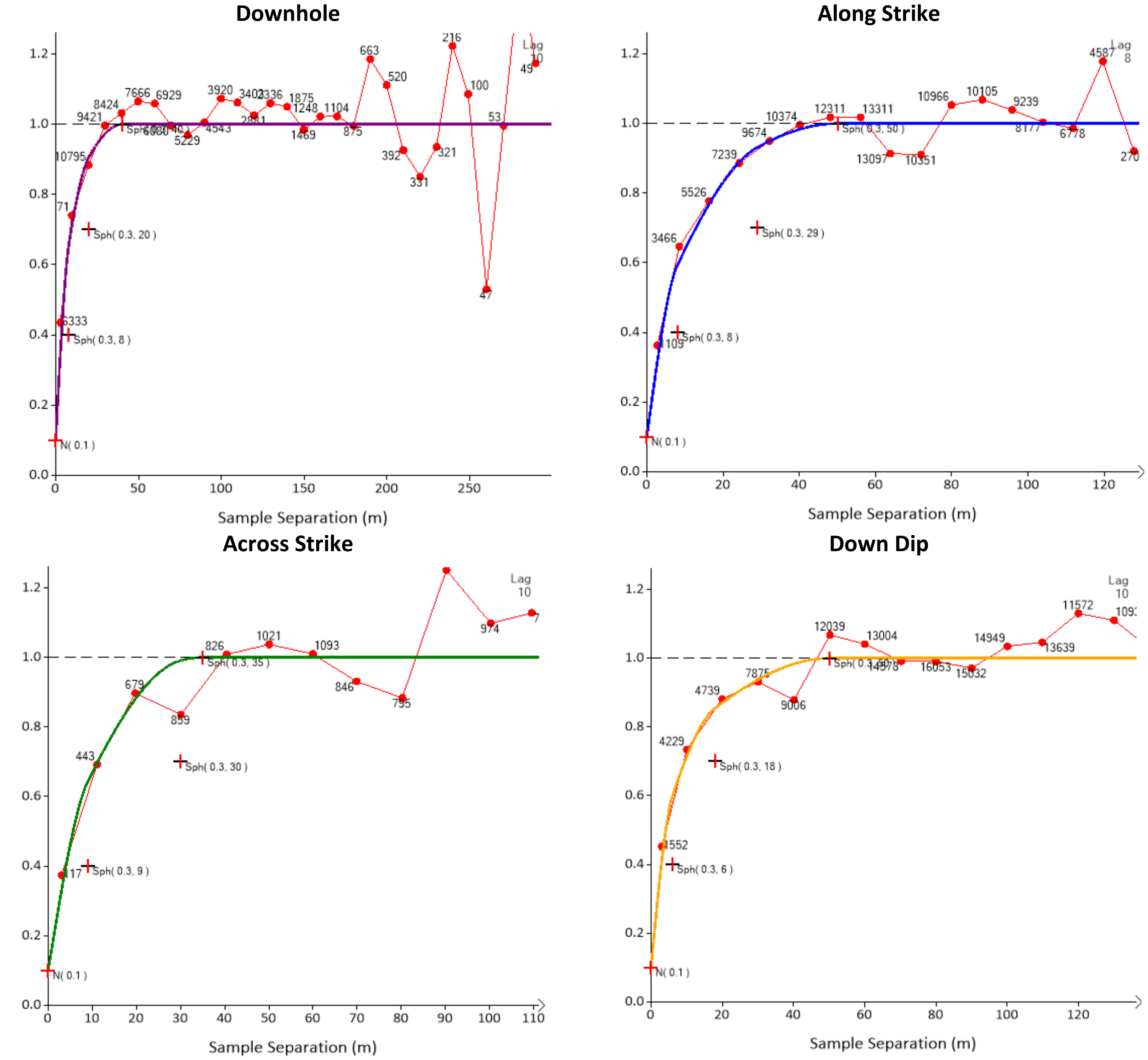
The density domains were not strongly skewed and as is typical with this data type, the distributions were approximately normally distributed. However, there were anomalous values (unusually high or low) identified in each of the domains.

Top-cut values for each domain were defined by an analysis of log-probability plots and histograms. The impact of the selected top-cut threshold was assessed from the co-efficient of variation (CV) and the number of samples that were cut. Top-cuts were applied as follows:

- Grade data was top-cut on a domain-by-domain basis by resetting values above a chosen threshold to the threshold value. The threshold was set to lower the CV to about 1.5 to 1.7 while minimising the percentage of data that is changed to ideally less than 5%.
- Density values that were deemed anomalous (high and or low) were reset as absent data (nulls) – not reset to different values. Unlike the grade data, unusually high or low density data will mostly represent errors rather than parts of a different statistical population (usual case with grades). The number of anomalous values identified and adjusted was relatively small and as such the overall risk is considered low.

Some domains which were not estimated (not materially economic or based on limited sample data) were still top-cut to stop assigned means being materially biased high. In such cases the top-cuts were sometimes quite "harsh" as a result of dealing with mixed populations, most notably the hangingwall mafics domain.

Figure 14.5 Example of normal scores variograms for Cu (ESTFLAGCU=5501)

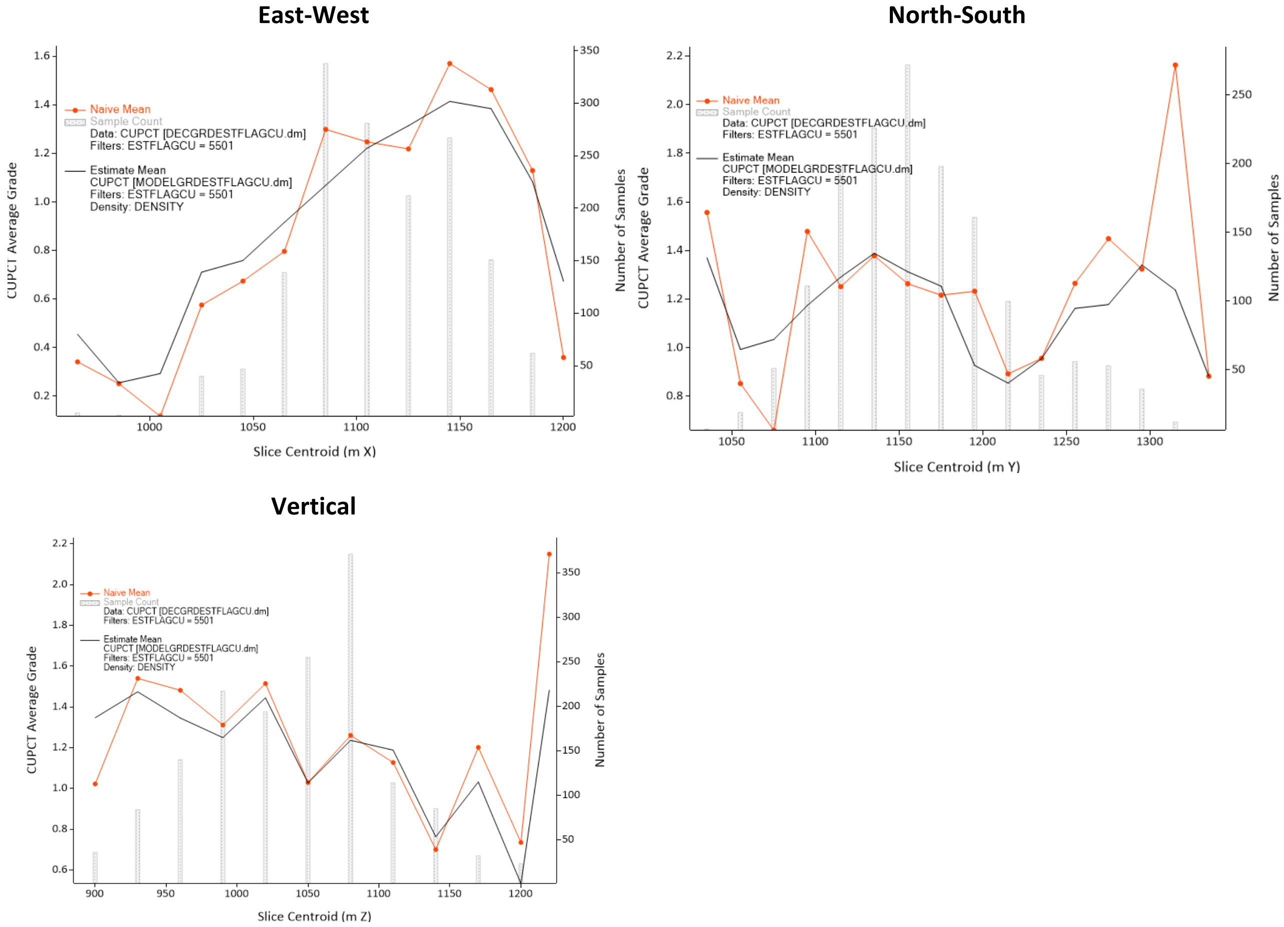


14.7 Block model validation

The grade and density estimates were validated using the following steps:

- Visual comparison of the drilling and the model data was undertaken on a section-by-section basis.
- By generation of grade trend plots for the model and grade fields for the grade estimation domain fields (ESTFLAGCU, ESTFLAGZN, ESTFLAGAU and ESTFLAGDN). An example is shown in Figure 14.6.
- By generation of univariate statistics (naive) from the drilling, on a domain-by-domain basis, and comparing this data with the corresponding model grades. Statistics were compiled from both mean and de-clustered (5 m cell based) drilling data.

Figure 14.6 Disseminated stockwork estimate, South Orebody (ESTFLAGCU=5501)



Assessment of the results yielded the following observations:

1. Visually, the drilling and model grades compare well.
2. Statistical comparison of the modelled key grade estimates with input drilling data, yielded acceptable to good differences, which for the mineralised domains were mostly less than 10%.
3. Grade trend plots of key elements show good reproduction of block grades when compared with sample input grades.

14.8 Mineral Resource classification

The Mineral Resource estimates were classified as Indicated and Inferred Mineral Resources, 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 and the CIM Definition Standards).

The classification was guided by confidences in the geology, estimation methods and results, geological continuity, the drillhole grid spacing and the quality of sample analysis.

14.8.1 Classification criteria

The 2025 Çayeli Mineral Resource estimate was classified according to the following criteria:

- The Mineral Resources were reported using a Cu metal equivalent (Cu_{eq}) grade of 0.90% based on the estimated Cu, Zn and Ag grades. While Au is also payable in concentrate form it was excluded from the equation for two reasons:
  - The Au estimates are considered indicative, being reflective of 6 m composite samples.
  - The contribution to final concentrate return is relatively low due to the low plant recovery (typically <20% in a 2024/2025 budget version of the life of mine plan).
- The adopted metal prices and recoveries are listed in Table 14-20 and are the values from a 2024/2025 budget version of the life of mine plan.

Table 14-20 Element prices and recoveries

Element	Price	Unit (USD)	Recovery (%)
Cu	$4.00	$/lb Cu	87.1
Zn	$1.20	$/lb Zn	68.5
Ag	$25.0	$/troy oz Au	55.2

  - The corresponding Cu equivalent equation is given in Table 14-21.

Table 14-21 Cu metal equivalent equation

$$Cu_{eq} = Cu + Zn * 0.236 + Ag * 0.006$$

- The mine plans are compiled on the basis of net smelter return (NSR) cut-off values, which are prone to change depending on the scheduling period. Reporting the Resource using a metal equivalent grade based on life of mine recoveries and prices, provides a more holistic view.
- Mining in the historic Main Orebody area is essentially a remnant mining operation extracting residual resources in the old mining sublevels. Risks associated with mining in and around old workings means that there will invariably be losses and potentially some gains when reconciled against the plan. It is for this reason that all remaining Resources in the Main Orebody were classified as Indicated Mineral Resources.
- To gauge the economic potential of the South Orebody, a floating stope style optimisation was run by adopting a minimum stope size of 6 m by 14 m by 25 m in the X, Y and Z directions respectively, and a cut-off grade of 0.90% Cu_{eq}. Any amount of internal waste was allowed as long as the cut-off grade criteria was met. A dedicated wireframe solid *(su_MDV_based_classtr/ su_MDV_based_classpt*) was manually compiled to delimit the potential stopes.
- Much of the South Orebody mineralisation is defined from relatively broad spaced drilling intersecting the mineralisation on several sublevels. The South Orebody was classified as Indicated and Inferred Mineral Resources. The Indicated portion is that mineralisation which fell inside the digitised stope optimisation wireframe. Several isolated pods of mineralisation were defined based on a few samples existing around the edges of the South Orebody mineralisation. These were all classified as Inferred Mineral Resources.
- During the 2024 July site visit (FQM,2024), a key concern identified with the geological data on which this estimate is derived, is that there is limited to no quality control (QC) data. All sample analysis is undertaken by a ÇBI on-site laboratory. A review of results from CRMs and pulp resampling work started after the July 2024 site visit has not identified any major flaws.

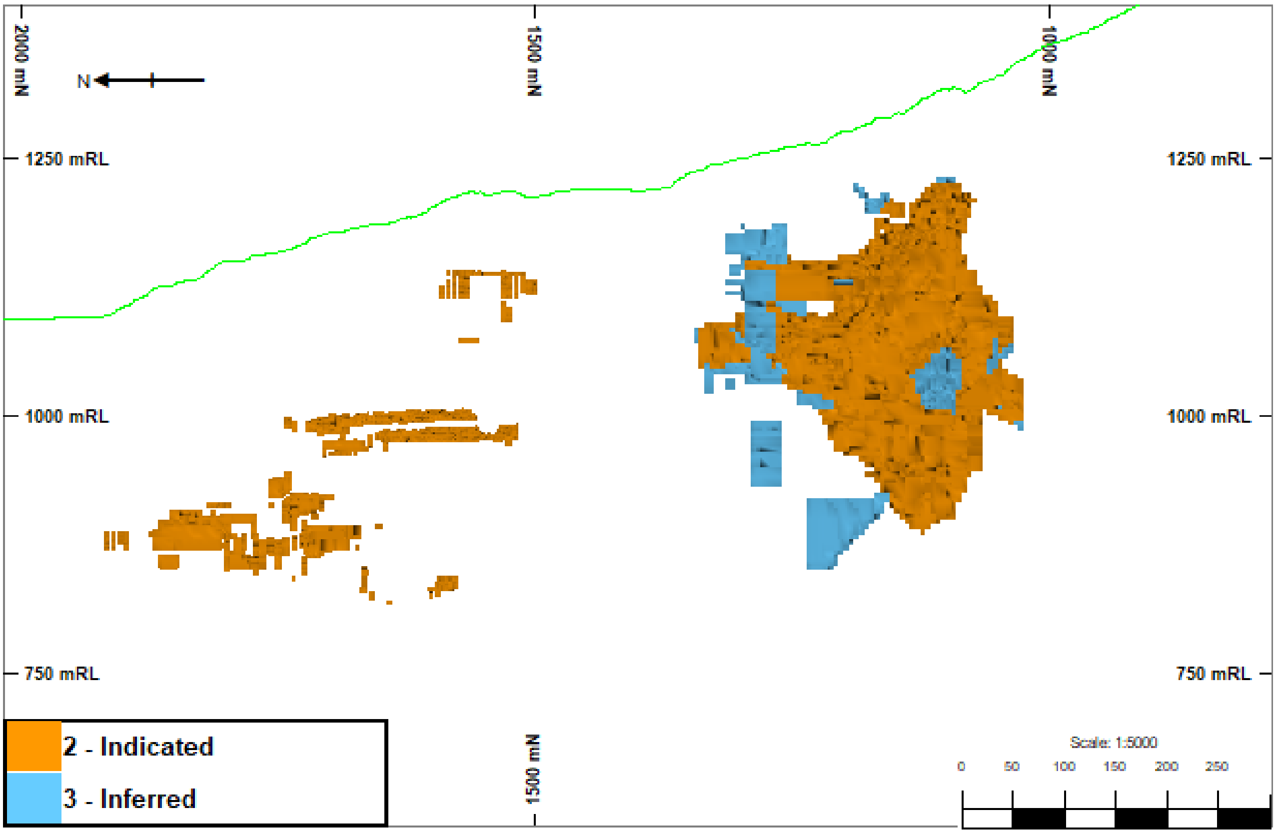
The classification was recorded in the Mineral Resource model using a field called RESCAT, which is described in Table 14-22. The final classified model is called cayresmod040625.dm.

Table 14-22 Resource classification model field (RESCAT) values

RESCAT	Description
2	Indicated
3	Inferred
4	Not classified

A long section of the classified Mineral Resource (Main and South Orebodies) is shown in Figure 14.7.

Figure 14.7 Çayeli Mineral Resource classification



14.9 Mineral Resource reporting

The April 30th, 2025, Çayeli Mineral Resource estimate statement (cayresmod040625.dm) is presented in Table 14-23 using a 0.90% Cu_{eq} cut-off.

Table 14-23 Çayeli April 30th, 2025, Mineral Resource statement using a 0.90% Cu_{eq} cut-off grade

Orebody	Classification	Tonnes (Mt)	Cu (%)	Zn (%)	Ag (ppm)
Main	Indicated	0.54	3.55	3.59	31.50
South	Indicated	8.79	1.33	2.48	9.09
	Inferred	0.46	0.64	2.31	6.39
	Total Indicated	**9.33**	**1.46**	**2.54**	**10.37**
	Total Inferred	**0.46**	**0.64**	**2.31**	**6.39**

Notes:

- Mineral Resources are reported inclusive of Mineral Reserves
- Mineral Resources that are not Mineral Reserves do not have a demonstrated economic viability as per the current Mineral Reserve conversion criteria
- Whilst small discrepancies may occur in the figures due to rounding, the impact is not material

14.9.1 Comparison with previous estimates

There was no comparison with any previous estimates undertaken due to:

1. Mining in the Main Orebody being essentially a remnant mining operation. Any comparison is essentially meaningless as the Main Orebody deposit is almost mined out.
2. The South Orebody Mineral Resource has not been previously disclosed in the public domain.

14.9.2 Potential factors which could impact Mineral Resource reporting

To the extent known, there are no significant risks relating to the Mineral Resource estimates, from legal, title, taxation, socio-economic, marketing and/or political factors. ÇBI's DST activities will need to be addressed as part of the Environmental Permit renewal. In the QP's opinion, there remains some uncertainty as to when and if the formal permit renewal will eventually be forthcoming.

Item 15 MINERAL RESERVE ESTIMATE

15.1 Methodology

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate has followed a conventional approach, commencing with the collation of economic parameters used to constrain the design of practical mine development and stope solids (i.e., practical mining shapes).

Bounded by economic considerations, these design solids were then produced in detail by honouring various planning rules and geotechnical parameters, whilst seeking to minimise the incorporation of internal waste dilution.

The completed development and stope design then provided the sublevel 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 Item 16.

Throughout the following commentary, and in the various tables and figures, gold (Au) grades are typically reported. Whilst this grade information is carried through the mine planning model and processes, it cannot be reported in the Mineral Reserve statement, for the reasons explained in Item 14.8.

The mine planning and design work required for this Mineral Reserve estimate was completed by the ÇBI mine planning team, with oversight by the mining QP.

15.2 Mine planning model

A block model suitable for mine planning purposes was produced from the Mineral Resource model described in Item 14. The sub-celled Mineral Resource model produced using Datamine software (reference: CAYRESMOD040625_CRB.dm) was reblocked for input to MineSight mine planning software. Reblocking was to a consistent 5 m x 5 m x 5 m block dimensions. Table 15-1 and Table 15-2 indicate that there is minimal loss of definition arising from this reblocking process.

Table 15-1 Mineral Resource model report

DEPOSIT	RESCAT	Mm^3	Mtonnes	DENSITY	CUPCT	ZNPCT	PBPCT	AGPPM	AUPPM	CUEQ
MAIN	2	0.2	0.5	3.44	3.55	3.59	0.13	31.48	0.37	4.59
MAIN	3	0.0	0.0	2.99	0.99	0.01	0.02	1.98	0.24	1.00
		0.2	**0.5**	**3.44**	**3.54**	**3.56**	**0.13**	**31.28**	**0.37**	**4.56**
SOUTH	2	2.7	9.0	3.36	1.32	2.45	0.04	9.03	0.71	1.95
SOUTH	3	0.1	0.5	3.12	0.64	2.31	0.04	6.37	0.31	1.22
		2.8	**9.4**	**3.35**	**1.29**	**2.44**	**0.04**	**8.90**	**0.69**	**1.91**
TOTAL	**2**	**2.8**	**9.5**	**3.37**	**1.49**	**2.54**	**0.05**	**10.79**	**0.68**	**2.16**
	3	**0.1**	**0.5**	**3.12**	**0.64**	**2.29**	**0.04**	**6.34**	**0.31**	**1.22**
		3.0	**10.0**	**3.36**	**1.38**	**2.53**	**0.05**	**10.59**	**0.67**	**2.11**

Table 15-2 Mine planning model report

DEPOSIT	RESCAT	Mm^3	Mtonnes	DENSITY	CUPCT	ZNPCT	PBPCT	AGPPM	AUPPM	CUEQ
MAIN	2	0.1	0.5	3.43	3.54	3.56	0.13	31.73	0.36	4.57
MAIN	3	0.0	0.0	3.00	0.99	0.01	0.02	1.99	0.24	1.00
		0.2	**0.5**	**3.43**	**3.53**	**3.55**	**0.13**	**31.60**	**0.36**	**4.56**
SOUTH	2	2.7	9.1	3.36	1.32	2.45	0.04	9.03	0.71	1.95
SOUTH	3	0.1	0.5	3.12	0.64	2.30	0.04	6.36	0.31	1.22
		2.9	**9.6**	**3.35**	**1.29**	**2.44**	**0.04**	**8.90**	**0.69**	**1.92**
TOTAL	**2**	**2.9**	**9.6**	**3.36**	**1.48**	**2.53**	**0.05**	**10.68**	**0.68**	**2.14**
	3	**0.1**	**0.5**	**3.12**	**0.64**	**2.29**	**0.04**	**6.34**	**0.31**	**1.22**
		3.0	**10.1**	**3.35**	**1.37**	**2.52**	**0.05**	**10.48**	**0.67**	**2.10**

15.2.1 Model reporting cut-off grade

Preparatory to mine planning, the Mineral Resource reporting cut-off grade was determined according to preliminary inputs as listed in Table 15-3. The listed metal prices and average recoveries were subsequently modified when carried through into the net smelter return (NSR) calculations described in the following items. Similarly, the listed unit operating costs were produced from a preliminary estimate and were also increased following a subsequent review. That being the case, the equivalent copper cut-off grade listed in Table 15-3 would provide a lower bound for defining the limits within which the Mineral Reserve was subsequently defined.

Table 15-3 Datamine mine planning model cut-off grade

	UNITS	TOTAL
Metal Prices		
Copper	\$/lb Cu	\$4.00
Zinc	\$/lb Zn	\$1.20
Silver	\$/oz Ag	\$25.00
Gold	\$/oz Au	
Process Recoveries		
Copper	%	87.1%
Zinc	%	68.5%
Silver	%	55.2%
Gold	%	
Operating Costs		
Incremental Ore Mining Cost	\$/t	\$21.49
Processing + Admin + Con Cost	\$/t	\$33.16
subtotal	\$/t	\$54.65
Mining Dilution (unplanned)		1.05
Net Return		
Copper net return	\$/lb Cu	\$2.83
Total net return	\$/lb Cueq	\$3.26
	\$/10kg	**\$71.90**
Cut-off Grade		
	% Cu	**1.04**
	% Cu_{eq}	**0.90**

15.2.2 Net Smelter Return equations

For the purposes of mine planning and Mineral Reserve definition, a set of equations were coded into the Mineral Resource model, with specific constants determined from an NSR calculation. The derivation of the following NSR equations is described in Item 15.3.4:

Main Orebody equations:

- Non-spec feed: NSR ($/t of ore) = Cu grade (%) * 7,589 + Zn grade (%) * 931
- Spec feed: NSR ($/t of ore) = Cu grade (%) * 7,631

South Orebody equations:

- Non-spec feed: NSR ($/t of ore) = Cu grade (%) * 6,105 + Zn grade (%) * 1,161
- Spec feed: NSR ($/t of ore) = Cu grade (%) * 8,064

15.3 Mineral Resource conversion

An NSR approach was used to define that part of the Mineral Resource eligible for conversion to a Mineral Reserve. Modelled blocks were eligible if the NSR value was equal to or in excess of the estimated overall life of mine (LOM) average operating cost.

The following items record the various inputs to the NSR calculations and summarise the operating costs against which the block NSR values were evaluated.

15.3.1 Metal prices

The adopted metal prices for Mineral Reserve estimation are as follows:

- Copper = $4.10/lb ($9,039/t)
- Zinc = $1.20/lb ($2,646/t)
- Silver = $22.50/oz
- Gold = $2,750/oz[7]

15.3.2 Processing recovery

Life of mine processing recovery projections were adopted as listed in Table 15-4. These recovery figures account for Spec and Non-spec plant feed from the Main Orebody and from the South Orebody, and are based on a preliminary production schedule completed in mid-2024 (refer also to Item 13).

Table 15-4 Preliminary processing recovery projections

METAL	UNITS	Average	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Copper	%	**87.1%**	89.5%	89.9%	89.6%	86.2%	87.4%	87.8%	85.9%	85.4%	84.0%	83.3%	82.6%	86.5%	86.5%
Zinc	%	**68.5%**	33.9%	59.6%	70.0%	69.7%	70.7%	73.5%	69.7%	68.3%	67.9%	69.7%	67.8%	65.5%	65.5%
Silver	%	**55.2%**	86.9%	64.4%	50.6%	37.5%	42.6%	44.6%	39.6%	46.6%	38.3%	36.5%	54.4%	49.3%	49.3%
Gold	%	**34.8%**	14.7%	31.1%	38.3%	32.1%	33.8%	36.2%	33.9%	42.5%	39.5%	33.3%	30.0%	33.2%	33.2%

[7] Gold price, processing recovery of gold, and gold TCRCs are carried only in the mine planning and cut-off grade estimation process.

15.3.3 Treatment, refining and freight charges

The adopted treatment, refining and freight charges (also referred to as "metal costs") were those as listed in Table 15-5.

Table 15-5 Treatment, refining and freight charges

Metal	Units	Value
Treatment		
Copper	$/t con.	21.25
Zinc	$/t con.	230.00
Refining		
Copper	$/lb	0.021
Zinc	$/lb	n/a
Silver	$/oz	0.40
Gold	$/oz	4.00
Freight		
Copper	$/t con.	52.56
Zinc	$/t con.	85.61

15.3.4 Net Smelter Return

Incorporating the various inputs described above, Table 15-6 provides NSR calculation examples for a notional model block with assumed 1.0% Cu and Zn grades. The examples show the separately calculated NSR ($/t ore) values for the Main Orebody and South Orebody Spec and Non-spec plant feed.

The NSR calculator was used to determine the following constants adopted for the NSR equations coded into the mine planning model (refer to Item 15.2.2):

Main Orebody equations:

- Non-spec feed constants:
  - copper = 7,589
  - zinc = 931
  - combined = 8,520 (i.e., NSR of $85.20/t at 1% Cu and 1% Zn)
- Spec feed constant:
  - copper = 7,631 (i.e., NSR of $76.31/t at 1% Cu)

South Orebody equations:

- Non-spec feed constants:
  - copper = 6,105
  - zinc = 1,161
  - combined = 7,266 (i.e., NSR of $72.66/t at 1% Cu and 1% Zn)
- Spec feed constants:
  - copper = 8,064 (i.e., NSR of $80.64/t at 1% Cu)

Table 15-6 Example NSR calculations

	MAIN OREBODY		SOUTH OREBODY	
	NON-SPEC ORE	SPEC ORE	NON-SPEC ORE	SPEC ORE
ASSUMPTIONS				
Feed Grade Cu (%)	**1.00%**	**1.00%**	**1.00%**	**1.00%**
Feed Grade Zn (%)	**1.00%**	**1.00%**	**1.00%**	**1.00%**
Copper in Copper Concentrate (%)	17.0%	22.2%	19.0%	23.0%
Zinc in Copper Concentrate (%)	*10.2%*	*2.4%*	*10.2%*	*2.4%*
Gold in Copper Concentrate (g/t)	1.33	0.36	5.00	1.55
Silver in Copper Concentrate (g/t)	124.50	36.14	40.00	20.00
Lead in Copper Concentrate (%)	*0.4%*	*0.2%*	*0.4%*	*0.2%*
Zinc in Zinc Concentrate (%)	40.0%		50.0%	
Silver in Zinc Concentrate (g/t)	226.60		65.00	
Copper Recovey (%)	84.8%	92.4%	60.0%	92.0%
Zinc Recovery (%)	63.0%	0.0%	75.0%	0.0%
Copper Price (c/lb)	**410.00**	**410.00**	**410.00**	**410.00**
Copper Price ($/mt)	**9,039**	**9,039**	**9,039**	**9,039**
Zinc Price (c/lb)	**120.00**	**120.00**	**120.00**	**120.00**
Gold Price ($/oz)	**2,750**	**2,750**	**2,750**	**2,750**
Silver Price ($/oz)	**22.50**	**22.50**	**22.50**	**22.50**
Copper Treatment Charge ($/t conc)	21.25	21.25	21.25	21.25
Copper Refining Charge(c/lb pay Cu)	0.021	0.021	0.021	0.021
Copper Freight ($/t conc)	52.56	52.56	52.56	52.56
Zinc Treatment Charge ($/t conc)	230.00	230.00	230.00	230.00
Zinc Freight ($/t conc)	85.61	85.61	85.61	85.61
Gold Refining Charge($/oz)	4.00	4.00	4.00	4.00
Silver Refining Charge ($/oz)	0.40	0.40	0.40	0.40
PAYABLE METALS				
Copper in Copper Concentrate ($/dmt)	72.10	79.80	51.38	79.54
Silver in Copper Concentrate Credit ($/dmt)	3.42	0.18	0.23	-0.29
Gold in Copper Concentrate Credit ($/dmt)	5.29	0.00	12.56	4.93
Zinc in Zinc Concentrate ($/dmt)	13.34		16.67	
Silver in Zinc Concentrate Credit ($/dmt)	1.06		-0.22	
Subtotal	**95.21**	**79.98**	**80.62**	**84.18**
PRICE PARTICIPATION				
Copper ($/dmt)	0.00	0.00	0.00	0.00
Zinc ($/dmt)	0.00	0.00	0.00	0.00
DEDUCTIONS				
Copper Treatment Charge ($/dmt)	-1.06	-0.88	-0.67	-0.85
Copper Refining Charges ($/dmt):				
Copper	-0.37	-0.41	-0.27	-0.41
Silver	-0.06	0.00	0.00	0.01
Gold	0.00	0.00	-0.02	0.00
Zinc Treatment Charge ($/dmt)	-3.62	0.00	-3.45	0.00
PENALTIES				
Zn+Pb in Cu Concentrate ($/dmt)	-0.57	0.00	-0.36	0.00
CIF VALUE ($/dmt)	**89.52**	**78.69**	**75.85**	**82.92**
FREIGHT				
Copper ($/dmt)	-2.86	-2.37	-1.80	-2.29
Zinc ($/dmt)	-1.47	0.00	-1.40	0.00
NSR (FOB) VALUE $/dmt)	**85.20**	**76.31**	**72.65**	**80.63**

15.3.5 Operating costs

Item 21.3 provides a detailed account of the estimation of operating costs for Mineral Reserve estimation, from which the respective departmental costs can be summarised as listed in Table 15-7. The physicals basis of this estimate is a preliminary production schedule completed in mid-2024 and updated in early 2025. This initial schedule shows a distinct peak ore production of 850,000 tpa in the period to 2030. Thereafter, the scale of ore production reduces to an average of about 400,000 tpa.

The annual operating costs vary each year reflecting this production profile, from an average of $72/t ore in the period to 2030, to an average of $84/t ore for the period from 2031. Without pre-empting a production schedule update, it is not possible to associate the annual operating costs with block NSR values when creating the development and stope solids for the mine design (refer to Item 15.4). Hence, the overall LOM average operating cost of approximately $77/t ore was adopted for mine design purposes.

By way of a sensitivity analysis, several charts have been prepared to test the impact of adopting an overall average operating cost for LOM planning and Mineral Reserve estimation.

Figure 15.1 shows the impact on annual ore production tonnes at different operating cost averages. In this figure the x-axis represents the ore production inventory at an average cost of $72/t ore to 2030 and then $84/t ore. The effect of adopting an overall $77/t average results in an approximate loss of 3% of the ore inventory up to 2030, and a gain of about 5% from 2031. The corresponding Figure 15.2 shows the impact on in situ copper metal at the same cost averages. In this instance the effect of adopting an overall $77/t average results in an approximate loss of 1% of the metal inventory up to 2030, and a gain of about 3% from 2031.

Table 15-7 Summary of estimated operating costs

	UNITS	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
PRODUCTION PHYSICALS															
Development in waste	kt	18.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0	234.0
Development in ore	kt	244.4	177.2	242.7	176.7	144.0	143.2	71.3	91.7	73.7	78.6	66.7	41.4	37.5	1,589.3
Stoping	kt	455.6	572.8	607.3	673.3	706.0	556.8	528.7	408.3	376.3	271.4	283.3	258.6	262.5	5,960.7
Total ore tonnes	**kt**	**700.0**	**750.0**	**850.0**	**850.0**	**850.0**	**700.0**	**600.0**	**500.0**	**450.0**	**350.0**	**350.0**	**300.0**	**300.0**	**7,550.0**
Total tonnes	kt	718.0	768.0	868.0	868.0	868.0	718.0	618.0	518.0	468.0	368.0	368.0	318.0	318.0	7,784.0
MINING COSTS															
Development in waste	$k	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	**$6,641.0**
Development in ore	$k	$5,537.6	$4,014.9	$5,499.2	$4,003.9	$3,263.3	$3,243.7	$1,615.2	$2,077.9	$1,670.2	$1,781.2	$1,510.3	$938.6	$849.2	**$36,005.2**
Stoping	$k	$2,531.3	$3,182.6	$3,374.2	$3,740.9	$3,922.6	$3,093.9	$2,937.7	$2,268.6	$2,090.7	$1,507.9	$1,574.3	$1,436.7	$1,458.6	**$33,120.0**
Services (ore + waste)	$k	$8,888.5	$9,507.5	$10,745.4	$10,745.4	$10,745.4	$8,888.5	$7,650.5	$6,412.6	$5,793.6	$4,555.7	$4,555.7	$3,936.7	$3,936.7	**$96,362.0**
Maintenance	$k	$3,748.1	$3,577.8	$3,833.3	$3,833.3	$3,833.3	$3,207.8	$2,627.0	$2,046.2	$2,001.5	$1,420.7	$1,599.4	$1,331.4	$1,331.4	**$34,391.4**
Additional mining labour	$k	$3,285.0	$3,197.8	$3,183.3	$3,161.5	$3,139.7	$2,627.4	$2,151.7	$1,676.0	$1,639.4	$1,163.7	$1,310.0	$1,090.5	$1,090.5	**$28,716.4**
subtotal	**$k**	**$24,501.4**	**$23,991.4**	**$27,146.2**	**$25,995.9**	**$25,415.1**	**$21,572.1**	**$17,493.0**	**$14,992.1**	**$13,706.3**	**$10,939.9**	**$11,060.6**	**$9,244.7**	**$9,177.2**	**$235,236.1**
	$/t	***$35.00***	***$31.99***	***$31.94***	***$30.58***	***$29.90***	***$30.82***	***$29.15***	***$29.98***	***$30.46***	***$31.26***	***$31.60***	***$30.82***	***$30.59***	***$31.16***
MILLING COSTS															
subtotal	**$k**	**$11,813.8**	**$12,491.1**	**$14,035.1**	**$14,228.2**	**$14,285.2**	**$12,317.6**	**$10,929.1**	**$9,540.6**	**$8,961.5**	**$7,529.2**	**$7,656.7**	**$6,916.4**	**$6,685.3**	**$137,389.7**
	$/t	***$16.88***	***$16.65***	***$16.51***	***$16.74***	***$16.81***	***$17.60***	***$18.22***	***$19.08***	***$19.91***	***$21.51***	***$21.88***	***$23.05***	***$22.28***	***$18.20***
PLANT COSTS															
subtotal	**$k**	**$7,281.5**	**$7,122.3**	**$7,187.6**	**$7,147.8**	**$7,108.0**	**$5,948.2**	**$4,871.2**	**$3,794.2**	**$3,711.4**	**$2,634.4**	**$2,965.8**	**$2,468.7**	**$2,468.7**	**$64,710.0**
	$/t	***$10.40***	***$9.50***	***$8.46***	***$8.41***	***$8.36***	***$8.50***	***$8.12***	***$7.59***	***$8.25***	***$7.53***	***$8.47***	***$8.51***	***$8.51***	***$8.66***
ADMINISTRATION COSTS															
subtotal	**$k**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$10,480.9**	**$136,252.1**
	$/t	***$14.97***	***$13.97***	***$12.33***	***$12.33***	***$12.33***	***$14.97***	***$17.47***	***$20.96***	***$23.29***	***$29.95***	***$29.95***	***$34.94***	***$34.94***	***$18.05***
CONCENTRATE HANDLING COSTS															
subtotal	**$k**	**$921.4**	**$987.2**	**$1,118.9**	**$1,118.9**	**$1,118.9**	**$921.4**	**$789.8**	**$658.2**	**$592.3**	**$460.7**	**$460.7**	**$394.9**	**$394.9**	**$9,938.1**
	$/t	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***	***$1.32***
TOTAL OPERATING COSTS															
Total	**$k**	**$54,999.1**	**$55,073.0**	**$59,968.7**	**$58,971.7**	**$58,408.1**	**$51,240.2**	**$44,564.0**	**$39,466.0**	**$37,452.4**	**$32,045.2**	**$32,624.8**	**$29,505.7**	**$29,207.1**	**$583,525.9**
	$/t	***$78.57***	***$73.43***	***$70.55***	***$69.38***	***$68.72***	***$73.20***	***$74.27***	***$78.93***	***$83.23***	***$91.56***	***$93.21***	***$98.63***	***$97.63***	***$77.37***
averages	***$/t***	***$72***						***$84***							***$77***

(Tabled ore tonnes are after the application of unplanned mining dilution and mining recovery (loss) adjustment).

Figure 15.1 Production ore tonnes sensitivity to operating cost averaging

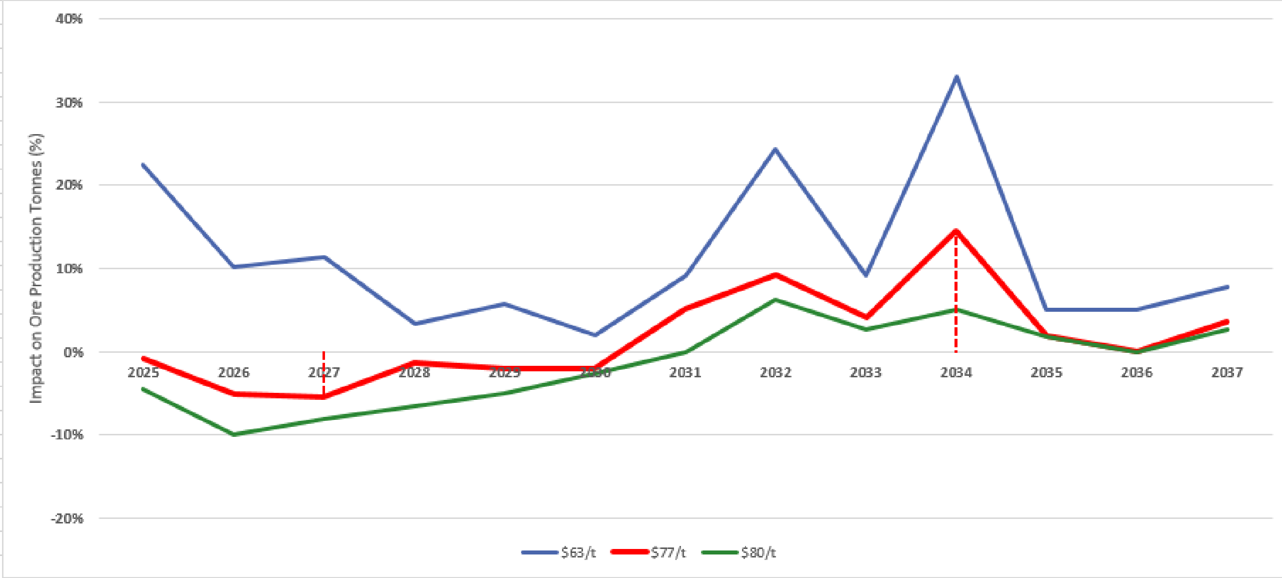
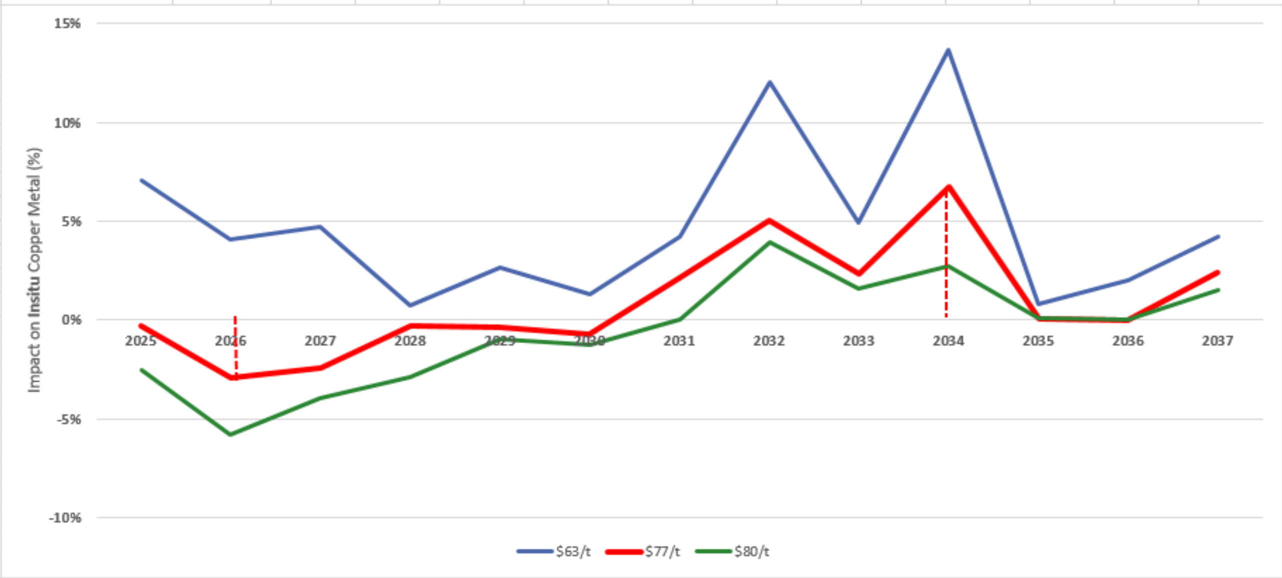


Figure 15.2 In situ copper metal sensitivity to operating cost averaging



15.3.6 Marginal cut-off grade

Relative to the adopted production schedule, LOM average metal grades, process recoveries and incremental ore mining and processing costs, an indicative overall marginal cut-off grade of 1.09% Cu (or 0.93% Cu_{eq}) can be calculated. The basis of this calculation is summarised in Table 15-8.

Table 15-8 Estimated marginal cut-off grades for mine planning

	UNITS	TOTAL
TOTAL PLANT FEED		
Total Ore	kt	**7,550**
Copper	%	**1.63**
Zinc	%	**1.80**
Silver	g/t	**7.98**
Gold	g/t	**0.54**
RECOVERIES		
Copper	%	**87.7%**
Zinc	%	**69.5%**
Silver	%	**52.4%**
Gold	%	**35.2%**
METAL PRICES		
Copper	\$/lb Cu	**\$4.10**
Zinc	\$/lb Zn	**\$1.20**
Silver	\$/oz Ag	**\$22.50**
Gold	\$/oz Au	**\$2,750**
PAYABILITY		
Copper	%	**95.0%**
Zinc	%	**80.5%**
Silver	%	**92.4%**
Gold	%	**61.9%**
OPERATING COSTS		
Incremental Ore Mining Cost	**\$/t**	**\$21.54**
Processing + Admin + Con Cost	**\$/t**	**\$46.13**
subtotal	**\$/t**	**\$67.67**
Total Operating Costs (NSR Costs)		
Total	**\$/t**	**\$77.29**
TCRCs		
Copper	\$/lb Cu	**\$0.18**
Zinc	\$/lb Zn	**\$0.29**
Silver	\$/oz Ag	**\$0.40**
Gold	\$/oz Au	**\$4.00**
ROYALTIES		
Copper	\$/lb Cu	**\$0.38**
Zinc	\$/lb Zn	**\$0.11**
Silver	\$/oz Ag	**\$2.09**
Gold	\$/oz Au	**\$255.47**
MARGINAL CUT-OFF GRADE		
Mining dilution factor (unplanned)		**1.05**
Total net return (recovered)	**\$/10kg**	**\$76.81**
Marginal Cut-off Grade	**% Cu**	**1.09**
Marginal Cut-off Grade	**% Cueq**	**0.93**

15.4 Mining dilution and recovery losses

Mining dilution and ore recovery losses are accounted for in the mine design and inventory reporting process. Further information on this topic is provided in Item 16.5.5.

15.4.1 Planned dilution

Planned dilution is incorporated when designing stope solids, in situations where internal waste must be included to produce practical mining shapes and geometries. In some circumstances, Inferred Resource model blocks and unclassified Resource model blocks can also be included, with both assigned as waste dilution. As far as possible, however, the stope solids are designed to minimise these inclusions.

15.4.2 Unplanned dilution

Unplanned dilution arises during the mining process when waste or other diluents are inadvertently extracted with the ore. This unplanned dilution can include material such as paste or waste backfill sloughing (over-breaking) from the side walls of stopes. Drawing upon historical records, the overall unplanned dilution factor is 105.1%. The dilution is assumed to carry a nil diluent grade.

15.4.3 Mining recovery losses

Inadvertent losses to the design production inventory can arise due to several circumstances, such as adverse ground conditions and inaccurate production drilling and blasting practices. Again, drawing upon historical trends, the overall mining recovery loss factor is 5.8% (i.e., 94.2% mining recovery).

15.4.4 Mine design

Item 16.4 provides an overview of the mining method, whilst Item 16.5 provides a summary of the mine planning design considerations, including geotechnical parameters and specific planning rules.

15.4.5 Design process

The mine design process, for the new South Orebody production sublevels, essentially involves a number of steps as follows:

- A template layout of crosscuts into the orebody is designed for every sublevel, commencing from the main access entry point for that sublevel. An example for the 1075 mRL sublevel is shown in Figure 15.3. The template layout is shown as light blue lines on this figure.
- MineSight software is then used to:
  - define the termination extents for each crosscut and the internal waste segments, and
  - define practical stope shapes (solids) for the ore between each sublevel.
- This definition involves delimiting the ore production solids by referencing the overall NSR value of the solid against the overall \$77/t operating cost. The NSR value of the individual 5 m x 5 m x 5 m blocks within each designed solid is predefined by the NSR equations listed in Item 15.2.2. The green coloured shapes in Figure 15.3 are an example of stope solids so defined.
- Care is taken to minimise the inclusion of Inferred and unclassified resource blocks, and waste dilution, when designing practical stope solids.
- In situations where isolated stope blocks can be identified, an assessment is made as to whether the cost of crosscut development to access these solids is warranted.

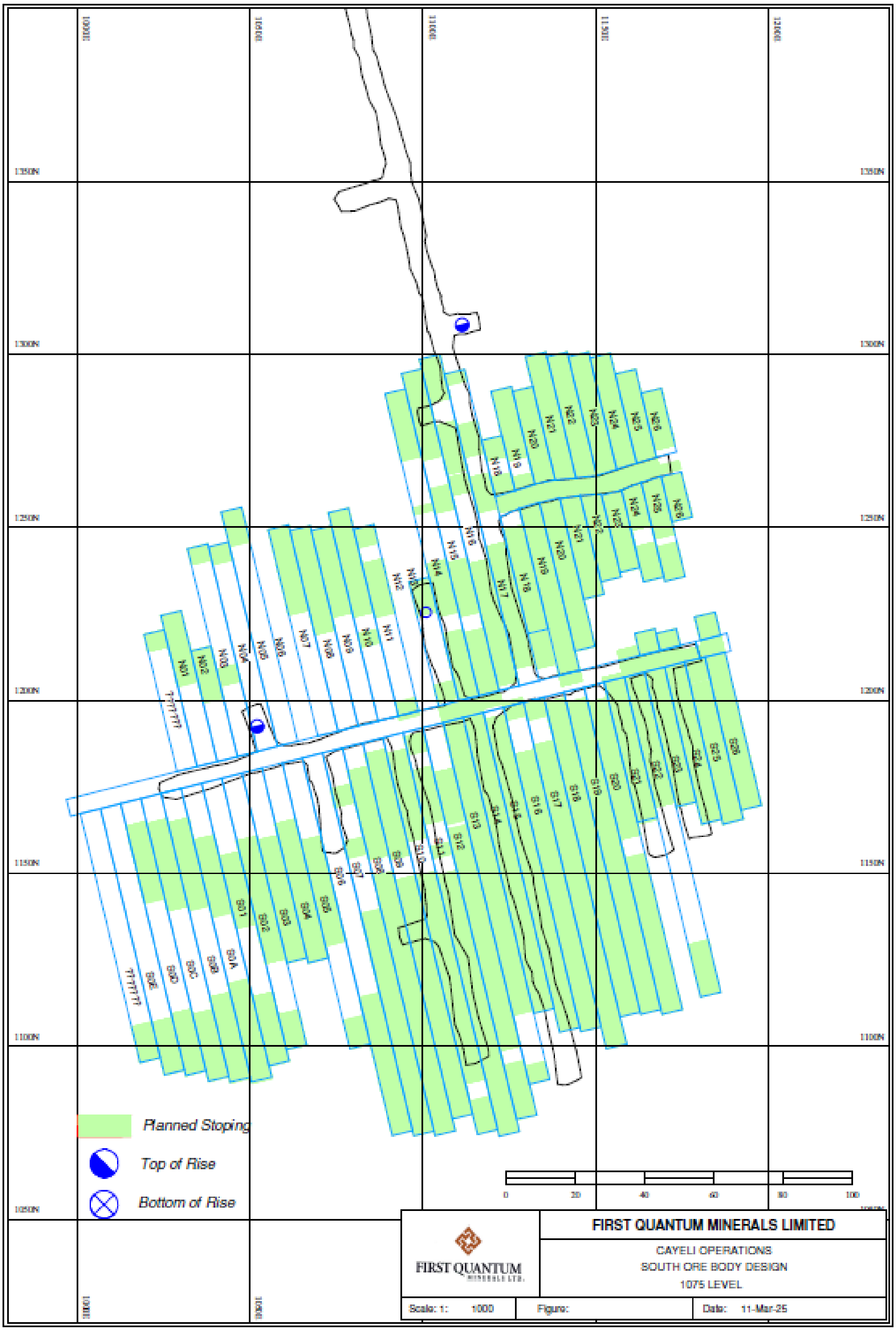
Several years of remnant mining is proposed for the Main Orebody, accessing favourably mineralised zones in several of the existing sublevels. In these instances, sublevel development is already in place, allowing for example:

- completion of primary and secondary development and stoping adjacent to areas mined in recent years
- tertiary (longitudinal) stoping of remnants above the sublevel, against previously completed and backfilled primary and secondary stopes
- crosscut development and stoping at extremities of the sublevel

Under these circumstances, the mine design process has had to consider the following:

- ground conditions in the vicinity of the remnants, the extent of any deterioration and the requirement for rehabilitation
- the accessibility of the remnant mining areas, i.e., sometimes necessitating excavation through backfilled openings
- the ability to ventilate the remnant areas

Figure 15.3 South Orebody, sublevel design example



15.4.6 Development and stope design layouts

Main Orebody

Figure 15.4 to Figure 15.11 show the existing sublevel development (blue solids) and the planned remnant mining stopes (red solids) between the 1100 mRL and 820 mRL sublevels.

Figure 15.4 Main Orebody, 1100 mRL and 1080 mRL sublevels

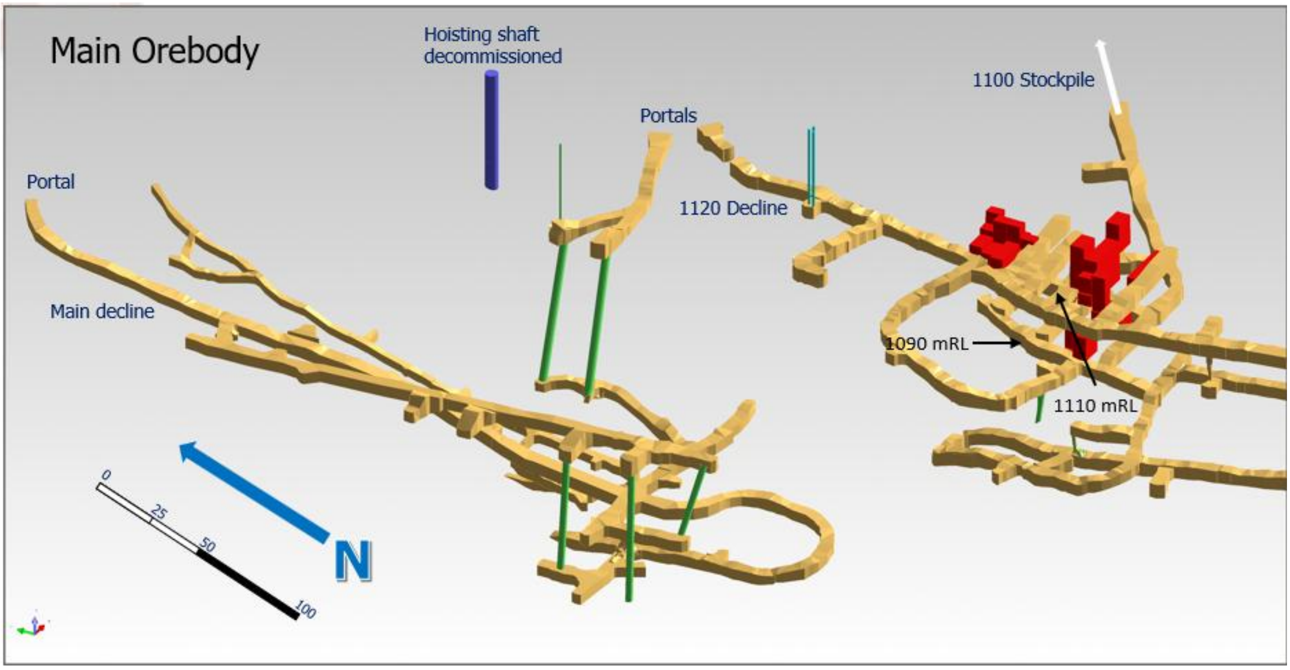


Figure 15.5 Main Orebody, 980 mRL and 960 mRL sublevels

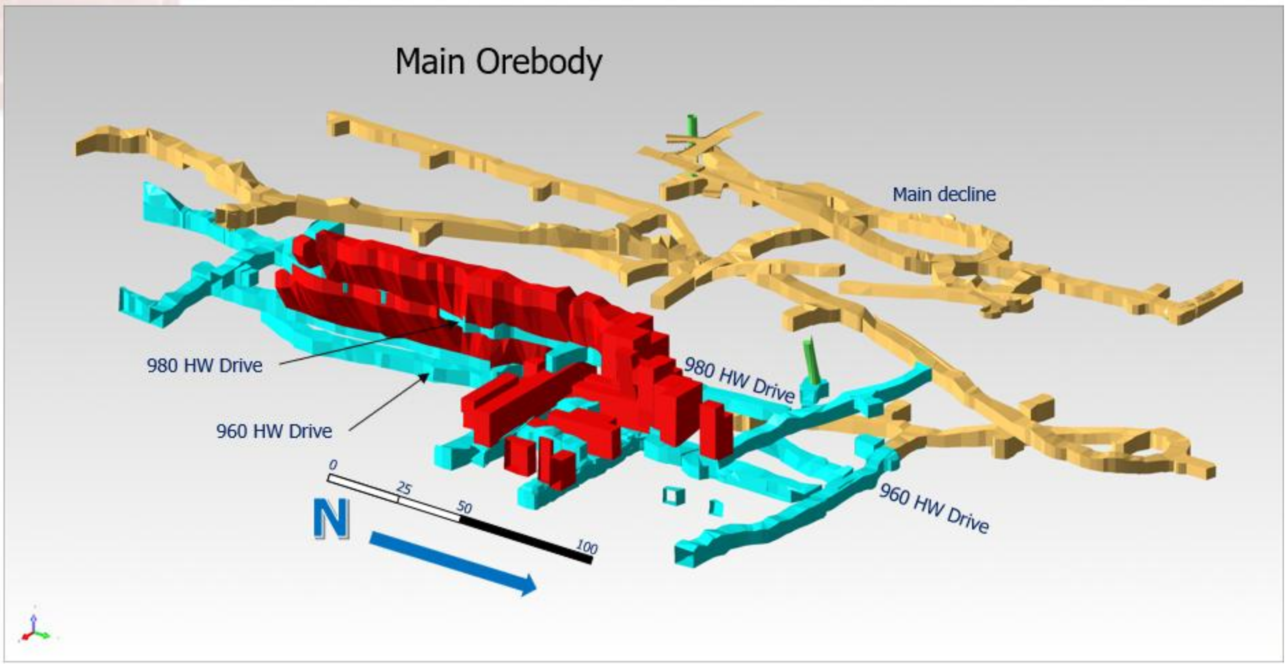


Figure 15.6 Main Orebody, 920 mRL sublevel

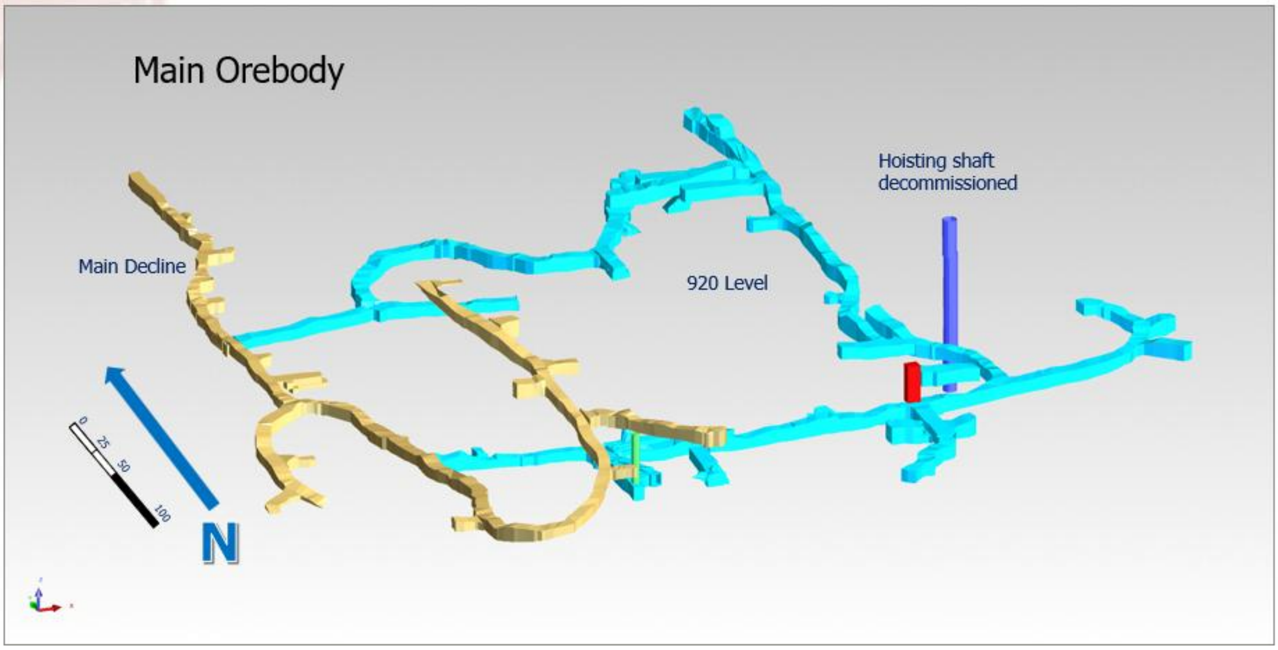


Figure 15.7 Main Orebody, 900 mRL sublevel

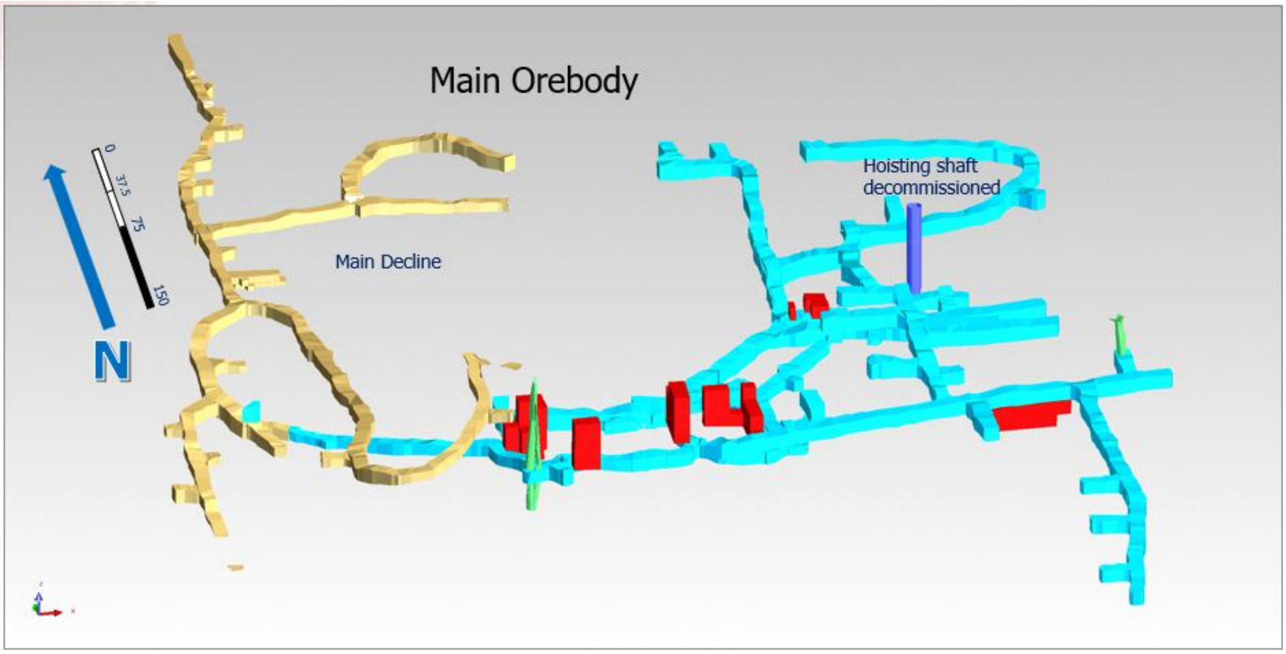


Figure 15.8 Main Orebody, 880 mRL sublevel

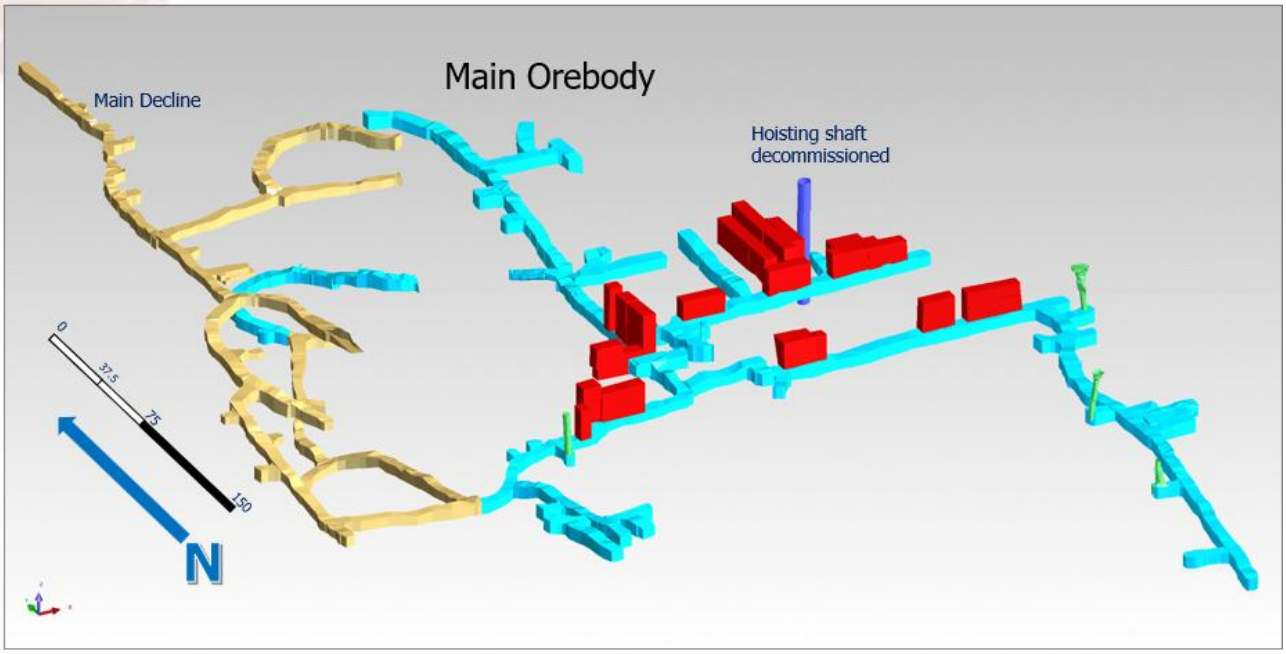


Figure 15.9 Main Orebody, 860 mRL sublevel

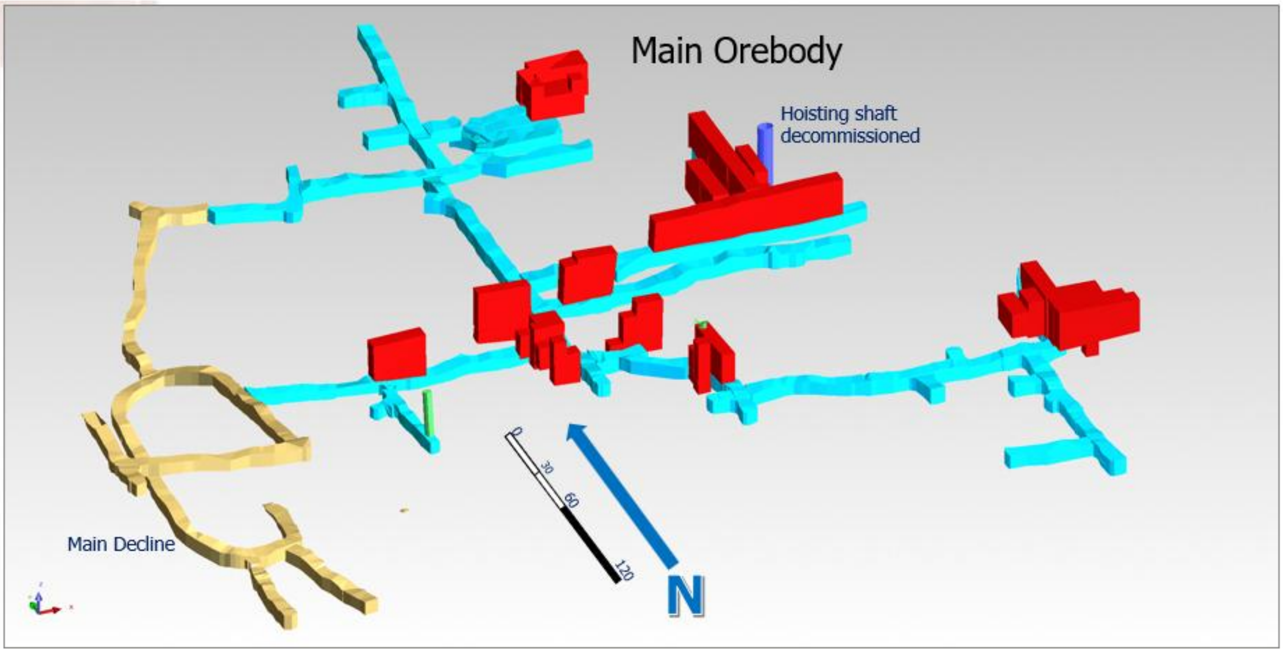


Figure 15.10 Main Orebody, 840 mRL sublevels

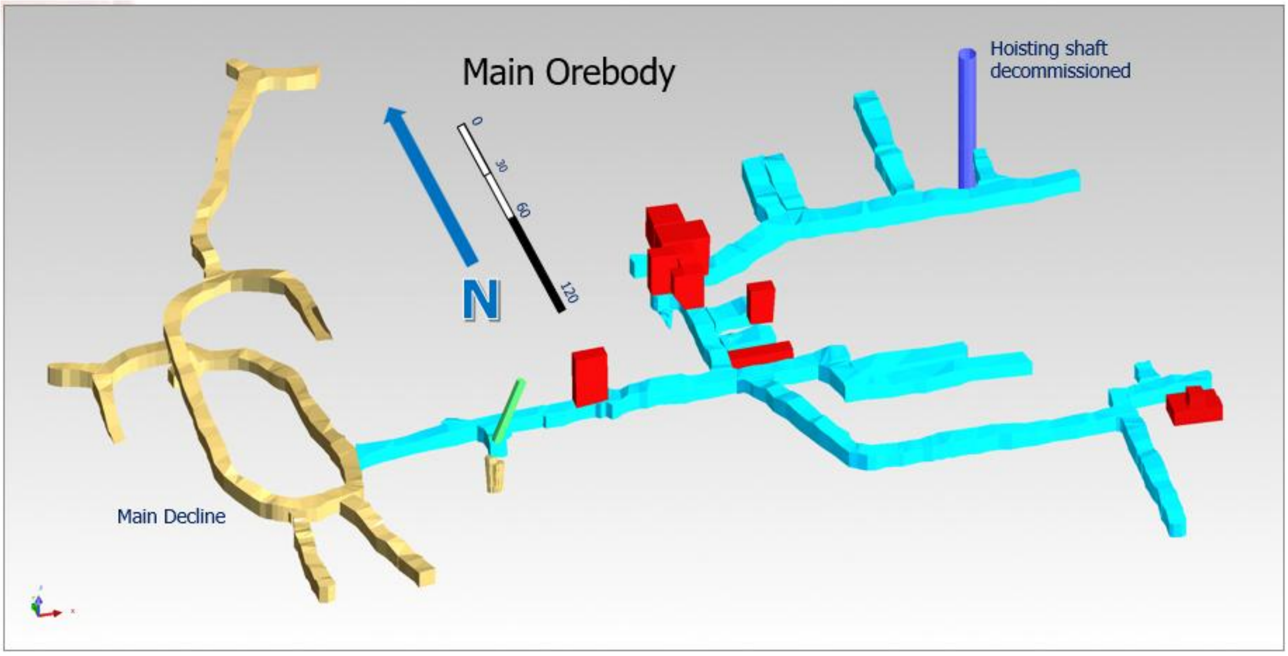
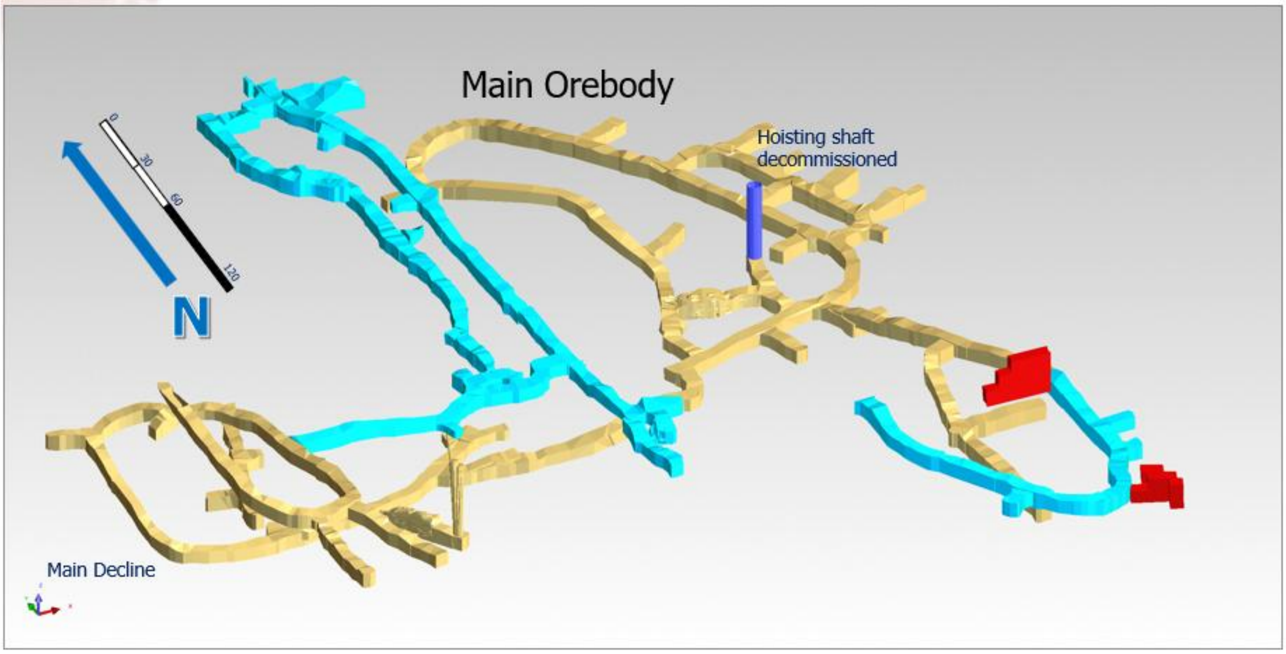


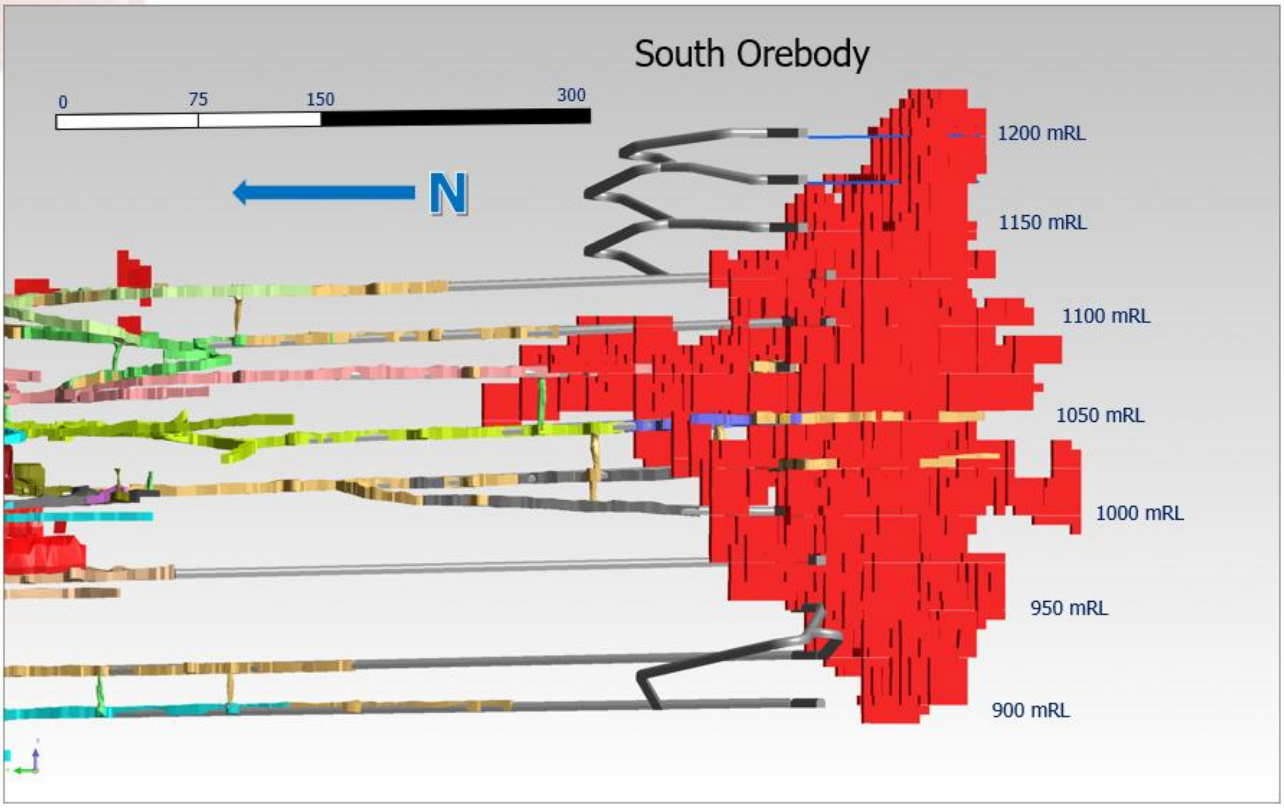
Figure 15.11 Main Orebody, 820 mRL sublevels



South Orebody

Figure 15.12 shows the access development, sublevels and the entire volume of planned stoping solids for the South Orebody, between 1200 mRL and 900 mRL. The completed development openings as at the beginning of May 2025, are shown coloured.

Figure 15.12 South Orebody, 1200 mRL to 900 mRL



15.5 Mining inventory

15.5.1 Main Orebody

Table 15-9 summarises the mining inventory for each of the Main Orebody sublevels. In total there are 36 design solids for ore development and 130 design solids for remnant stopes in the Main Orebody. This table lists the design inventory after accounting for planned dilution, unplanned dilution and mining recovery losses.

The planned dilution figure is approximately 24% of the total design solids inventory. This is not an unreasonable figure when considering remnant design stopes in the midst of old, back filled workings, the extents of which are likely to be uncertain. It is for this reason, and despite the fact that each of the sublevels has been extensively developed in ore, the Mineral Resource classification for the Main Orebody is Indicated rather than Measured. The planned diluent grades (and NSR value) for the Main Orebody reflect that unclassified Mineral Resource blocks rather than waste blocks are being incorporated into the design of regular and practical mining shapes.

The subsequently applied unplanned dilution and mining recovery factors, to arrive at the final inventory listed in Table 15-9, are as mentioned in Item 15.4.

Table 15-9 Main Orebody mining inventory, in ore development headings and stopes

MOB	Development headings					Stopes					TOTAL				
mRL	#	Tonnes	Av. NSR	Av. Cu%	Av. Zn%	#	Tonnes	Av. NSR	Av. Cu%	Av. Zn%	Tonnes	Av. NSR	Av. Cu%	Av. Zn%	Av. Ag%
1110	6	15,871	131	1.98	4.27	5	20,829	233	2.66	8.52	36,700	189	2.37	6.68	78.11
1090	2	2,416	267	2.91	5.18	3	6,854	426	4.73	6.04	9,270	384	4.26	5.82	73.53
980	1	2,523	379	4.97	1.07	26	107,479	403	5.30	6.10	110,002	402	5.29	5.98	57.73
960	5	15,235	333	3.67	2.36	21	101,952	370	4.53	5.46	117,187	365	4.42	5.06	36.70
920	0					1	1,062	344	4.50	0.05	1,062	344	4.50	0.05	5.47
900	5	6,324	212	2.95	0.63	8	30,500	282	3.49	3.39	36,823	270	3.40	2.92	25.93
880	5	11,642	177	2.32	0.17	21	88,158	162	2.02	1.93	99,800	164	2.05	1.73	7.43
860	7	7,090	101	1.56	0.09	34	130,378	149	2.04	0.33	137,468	147	2.02	0.32	1.28
840	3	3,908	153	2.00	0.04	7	13,631	152	2.50	0.73	17,539	152	2.39	0.58	9.16
820	2	1,634	101	1.33	0.01	4	14,596	128	1.67	0.01	16,230	125	1.64	0.01	0.60
TOTAL	**36**	**66,643**	**205**	**2.60**	**1.89**	**130**	**515,437**	**263**	**3.36**	**3.41**	582,081	256	3.28	3.24	27.91
Reserve											441,080	271	3.43	3.38	
Plan dilution											140,930	206	2.83	2.75	
Unp. dilution	**36**	**3,372**	**0**	**0.00**	**0.00**	**130**	**26,081**	**0**	**0.00**	**0.00**	29,453	0	0.00	0.00	0.00
Diluted	**36**	**70,015**	**195**	**2.48**	**1.80**	**130**	**541,519**	**250**	**3.20**	**3.25**	611,534	244	3.12	3.08	26.57
Recovered	**36**	**65,968**	**195**	**2.48**	**1.80**	**130**	**510,219**	**250**	**3.20**	**3.25**	576,187	244	3.12	3.08	26.57

15.5.2 South Orebody

Table 15-10 summarises the mining inventory for each of the South Orebody sublevels. In total there are 593 design solids for ore development and 1,002 design solids for stopes in the South Orebody. The number of development design solids in many cases, includes successive mining faces within the same development heading.

This table lists the design inventory after accounting for planned dilution, unplanned dilution and mining recovery losses.

The planned dilution figure is approximately 5% of the total design solids inventory. The planned diluent grades (and NSR value) for the Main Orebody reflect that waste blocks are being incorporated into the design of regular and practical mining shapes.

The subsequently applied unplanned dilution and mining recovery factors, to arrive at the final inventory listed in Table 15-10, are as mentioned in Item 15.4.

Table 15-10 South Orebody mining inventory, in ore development headings and stopes

SOB	Development headings					Stopes					TOTAL				
mRL	#	Tonnes	Av. NSR	Av. Cu%	Av. Zn%	#	Tonnes	Av. NSR	Av. Cu%	Av. Zn%	Tonnes	Av. NSR	Av. Cu%	Av. Zn%	Av. Ag%
1200	20	54,536	329	4.21	0.72	25	125,363	270	3.18	1.25	179,899	288	3.49	1.09	7.70
1175	48	93,348	193	2.06	5.25	60	289,324	250	2.81	4.24	382,673	236	2.63	4.49	16.07
1150	50	116,947	130	0.42	8.97	69	380,976	165	1.15	8.15	497,923	157	0.98	8.34	30.79
1125	58	144,233	124	0.47	8.05	100	548,801	124	0.41	8.39	693,034	124	0.42	8.32	26.93
1100	54	111,258	124	1.37	1.72	73	389,843	119	1.04	3.75	501,101	121	1.12	3.30	11.30
1075	80	182,955	122	1.32	1.62	140	728,963	122	1.29	2.03	911,918	122	1.30	1.95	5.54
1050	50	110,809	119	1.48	0.17	133	773,715	108	1.26	1.16	884,524	109	1.29	1.04	5.46
1025	60	138,544	170	2.10	0.06	115	614,937	144	1.81	0.05	753,481	149	1.87	0.05	2.32
1000	53	120,333	136	1.70	0.05	88	511,299	146	1.83	0.07	631,632	144	1.80	0.06	2.31
975	45	99,008	111	1.40	0.03	66	415,657	105	1.32	0.02	514,665	106	1.34	0.02	1.82
950	35	69,180	131	1.68	0.03	67	323,665	115	1.45	0.04	392,845	118	1.49	0.04	2.89
925	27	65,185	126	1.57	0.03	38	212,405	119	1.51	0.03	277,590	121	1.52	0.03	3.52
900	13	28,232	114	1.43	0.04	28	155,447	115	1.44	0.05	183,679	115	1.44	0.04	2.59
TOTAL	**593**	**1,334,568**	**142**	**1.49**	**2.45**	**1,002**	**5,470,396**	**136**	**1.43**	**2.38**	6,804,964	137	1.44	2.39	9.38
Reserve											6,453,019	144	1.51	2.52	
Plan dilution											351,945	0	0.18	0.11	
Unp. dilution	**593**	**67,529**	**0**	**0.00**	**0.00**	**1,002**	**276,802**	**0**	**0.00**	**0.00**	344,331	0	0.00	0.00	0.00
Diluted	**593**	**1,402,097**	**135**	**1.42**	**2.33**	**1,002**	**5,747,198**	**129**	**1.36**	**2.27**	7,149,295	131	1.37	2.28	8.92
Recovered	**593**	**1,321,055**	**135**	**1.42**	**2.33**	**1,002**	**5,415,010**	**129**	**1.36**	**2.27**	6,736,066	131	1.37	2.28	8.92

15.5.3 Combined orebody inventories

Table 15-11 summarises the combined mining inventory for all the mine sublevels. As a grand total there are 629 design solids for ore development and 1,132 design solids for stopes.

Table 15-11 Main and South Orebodies mining inventory, in ore development headings and stopes

	Development headings					Stopes					TOTAL				
	#	Tonnes	Av. NSR	Av. Cu%	Av. Zn%	#	Tonnes	Av. NSR	Av. Cu%	Av. Zn%	Tonnes	Av. NSR	Av. Cu%	Av. Zn%	Av. Ag%
After planned dilution															
MOB	36	66,643	205	2.60	1.89	130	515,437	263	3.36	3.41	**582,081**	**256**	**3.28**	**3.24**	**27.91**
SOB	593	1,334,568	142	1.49	2.45	1,002	5,470,396	136	1.43	2.38	**6,804,964**	**137**	**1.44**	**2.39**	**9.38**
Subtotal	**629**	**1,401,211**	**145**	**1.54**	**2.42**	**1,132**	**5,985,834**	**147**	**1.59**	**2.47**	**7,387,044**	**146**	**1.58**	**2.46**	**10.84**
After unplanned dilution															
MOB	36	70,015	195	2.48	1.80	130	541,519	250	3.20	3.25	**611,534**	**244**	**3.12**	**3.08**	**26.57**
SOB	593	1,402,097	135	1.42	2.33	1,002	5,747,198	129	1.36	2.27	**7,149,295**	**131**	**1.37**	**2.28**	**8.92**
Subtotal	**629**	**1,472,112**	**138**	**1.47**	**2.31**	**1,132**	**6,288,717**	**140**	**1.52**	**2.35**	**7,760,829**	**139**	**1.51**	**2.34**	**10.31**
After mining recovery adjustment															
MOB	36	65,968	195	2.48	1.80	130	510,219	250	3.20	3.25	**576,187**	**244**	**3.12**	**3.08**	**26.57**
SOB	593	1,321,055	135	1.42	2.33	1,002	5,415,010	129	1.36	2.27	**6,736,066**	**131**	**1.37**	**2.28**	**8.92**
TOTAL	**629**	**1,387,024**	**138**	**1.47**	**2.31**	**1,132**	**5,925,229**	**140**	**1.52**	**2.35**	**7,312,253**	**139**	**1.51**	**2.34**	**10.31**

15.6 Blended plant feed inventory

Table 15-12 lists the plant feed inventory. This table indicates that there is no cross-blending required for the mined ore from the Main Orebody. As expected, with only two ore types defined, there is no cross-blending for the South Orebody mined ore.

Table 15-12 Main and South Orebodies blended plant feed inventory

	Main Orebody									South Orebody			TOTAL
	Spec				Non-spec				MOB Total	Spec	Non-spec	SOB Total	
	Yellow Ore			Subtotal	Black Ore	Clastic Ore		Subtotal		Yellow Ore	Black Ore		
	LYO	YO-	YO+		BO	CO-	CO+			SYO	SBO		
After planned dilution													
Tonnes	2,077	388,493	0	**390,570**	21,510	61,987	108,014	**191,510**	**582,081**	4,849,192	1,955,772	**6,804,964**	**7,387,044**
Av. NSR	90	207	0	**206**	99	405	383	**358**	**256**	139	132	**137**	**146**
Av. Cu%	1.18	2.74	0.00	**2.73**	1.37	4.43	4.96	**4.39**	**3.28**	1.75	0.67	**1.44**	**1.58**
Av. Zn%	1.33	0.65	0.00	**0.65**	4.57	10.05	8.40	**8.50**	**3.24**	0.20	7.84	**2.39**	**2.46**
Av. Ag g/t	35.56	8.15	0.00	**8.29**	31.94	87.53	63.84	**67.92**	**27.91**	2.78	25.72	**9.38**	**10.84**
After unplanned dilution													
Tonnes	2,182	408,151	0	**410,333**	22,598	65,124	113,479	**201,201**	**611,534**	5,094,561	2,054,734	**7,149,295**	**7,760,829**
Av. NSR	86	197	0	**196**	95	385	365	**341**	**244**	132	126	**131**	**139**
Av. Cu%	1.12	2.61	0.00	**2.60**	1.30	4.22	4.72	**4.17**	**3.12**	1.67	0.64	**1.37**	**1.51**
Av. Zn%	1.27	0.62	0.00	**0.62**	4.35	9.57	7.99	**8.09**	**3.08**	0.19	7.46	**2.28**	**2.34**
Av. Ag g/t	33.84	7.75	0.00	**7.89**	30.40	83.32	60.76	**64.65**	**26.57**	2.65	24.48	**8.92**	**10.31**
After mining recovery adjustment													
Tonnes	2,056	384,560	0	**386,616**	21,292	61,359	106,920	**189,571**	**576,187**	4,800,095	1,935,970	**6,736,066**	**7,312,253**
Av. NSR	86	197	0	**196**	95	385	365	**341**	**244**	132	126	**131**	**139**
Av. Cu%	1.12	2.61	0.00	**2.60**	1.30	4.22	4.72	**4.17**	**3.12**	1.67	0.64	**1.37**	**1.51**
Av. Zn%	1.27	0.62	0.00	**0.62**	4.35	9.57	7.99	**8.09**	**3.08**	0.19	7.46	**2.28**	**2.34**
Av. Ag g/t	33.84	7.75	1.00	**7.89**	30.40	83.32	60.76	**64.65**	**26.57**	2.65	24.48	**8.92**	**10.31**

15.7 Mineral Reserve estimation and statement

The April 30th, 2025, Çayeli Mineral Reserve estimate and statement is presented in Table 15-13.

Table 15-13 Çayeli Mineral Reserve statement at April 30th 2025, \$4.10/lb Cu, \$1.20/lb Zn, \$22.50/oz Ag

Orebody	Classification	Tonnes (Mt)	NSR ($/t)	Cu (%)	Zn (%)	Ag (ppm)
Main	Probable	0.58	244	3.12	3.08	26.57
South	Probable	6.74	131	1.37	2.28	8.92
	Total Probable	**7.31**	**139**	**1.51**	**2.34**	**10.31**

Notes:

- Mineral Resources are reported inclusive of Mineral Reserves
- for reasons associated with the compositing of gold samples (Item 14.8), an indicative gold (Au) grade is not included in the Mineral Reserve statement
- Whilst small discrepancies may occur in the figures due to rounding, the impact is not material

Figure 15.13 to Figure 15.15 show a series of pie-charts for the Main Orebody Mineral Reserve, which indicate:

- the bulk of the Reserve tonnage (~80%) is located on the 980 mRL, 960 mRL, 880 mRL and 860 mRL sublevels
- this is also the case for the bulk of in situ copper metal (~83% on these four sublevels)
- the highest in situ zinc metal (~89%) is spread over the 1110 mRL, 980 mRL, 960 mRL and 880 mRL sublevels

Figure 15.13 Main Orebody, Mineral Reserve tonnage (%) by sublevel

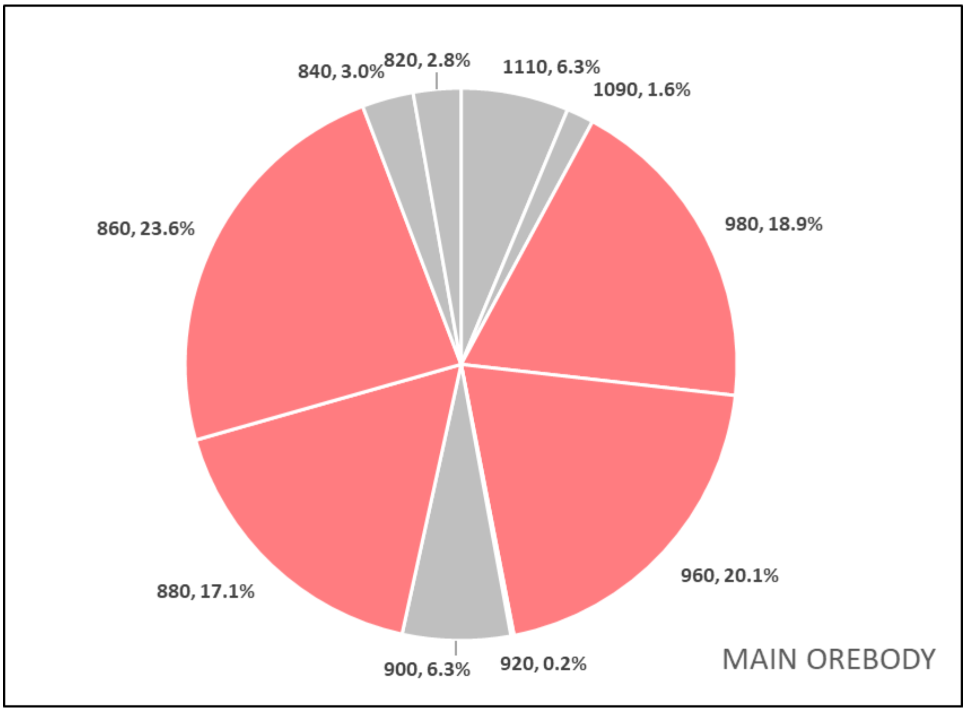


Figure 15.14 Main Orebody, Mineral Reserve in situ copper (%) by sublevel

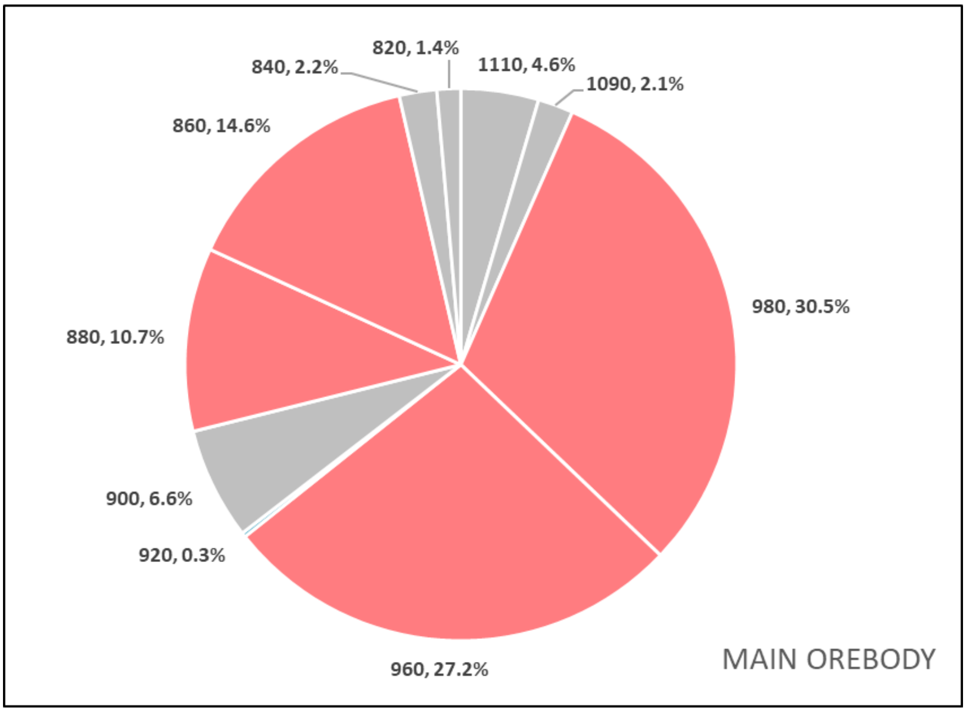


Figure 15.15 Main Orebody, Mineral Reserve in situ zinc (%) by sublevel

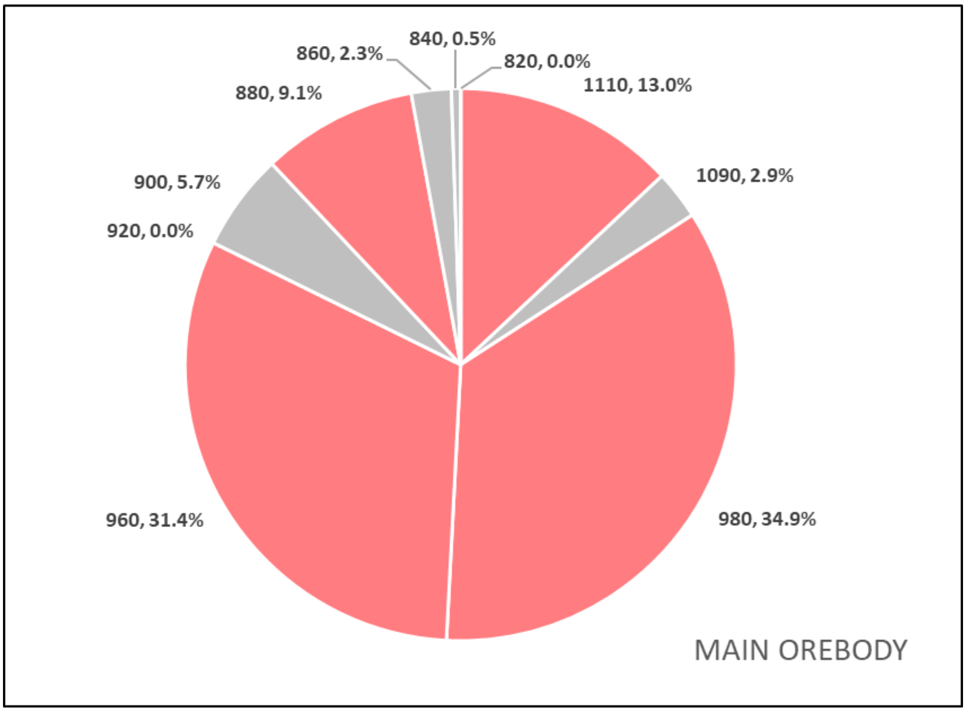


Figure 15.16 to Figure 15.18 show a series of pie-charts for the South Orebody Mineral Reserve, which indicate:

- the bulk of the Reserve tonnage (~47%) is located on four of the thirteen sublevels, i.e., the 1075 mRL, 1050 mRL, 1025 mRL and 1000 mRL sublevels
- this is also the case for the bulk of in situ copper metal (~50% on these same four sublevels)
- the highest in situ zinc metal (~92%) is spread over the upper five sublevels, i.e., the 1175 mRL, 1150 mRL, 1125 mRL, 1100 mRL and 1075 mRL sublevels

Figure 15.16 South Orebody, Mineral Reserve tonnage (%) by sublevel

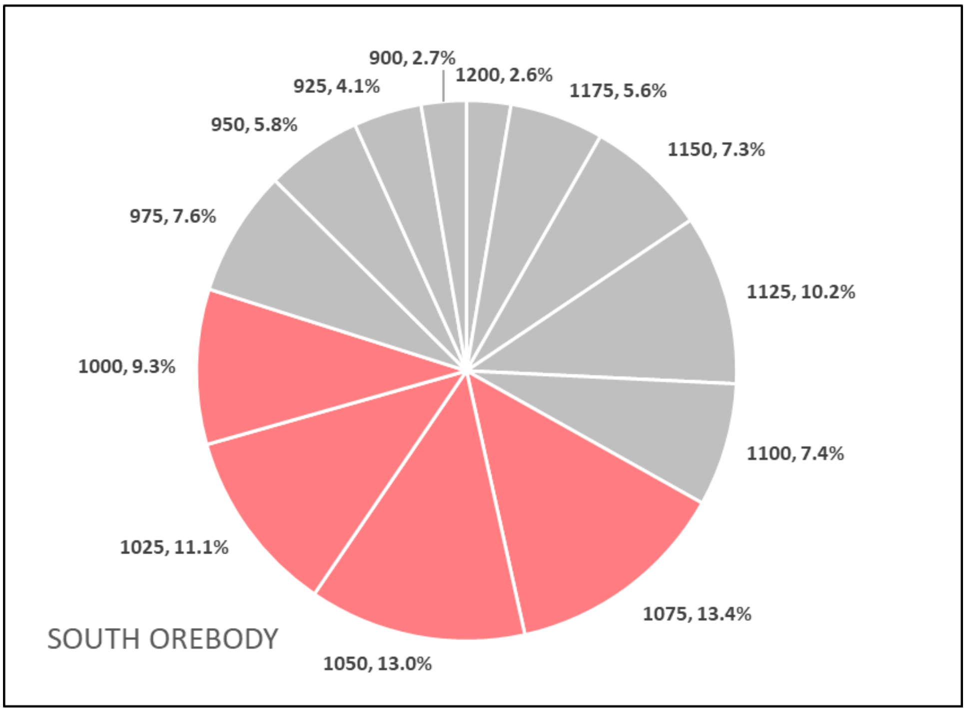


Figure 15.17 South Orebody, Mineral Reserve in situ copper (%) by sublevel

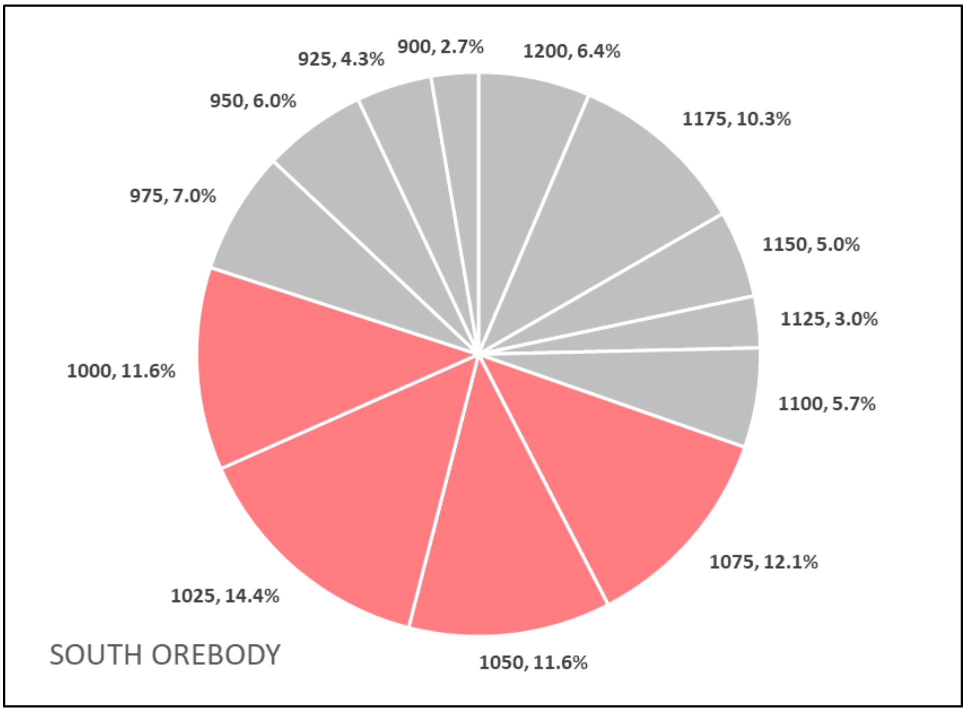
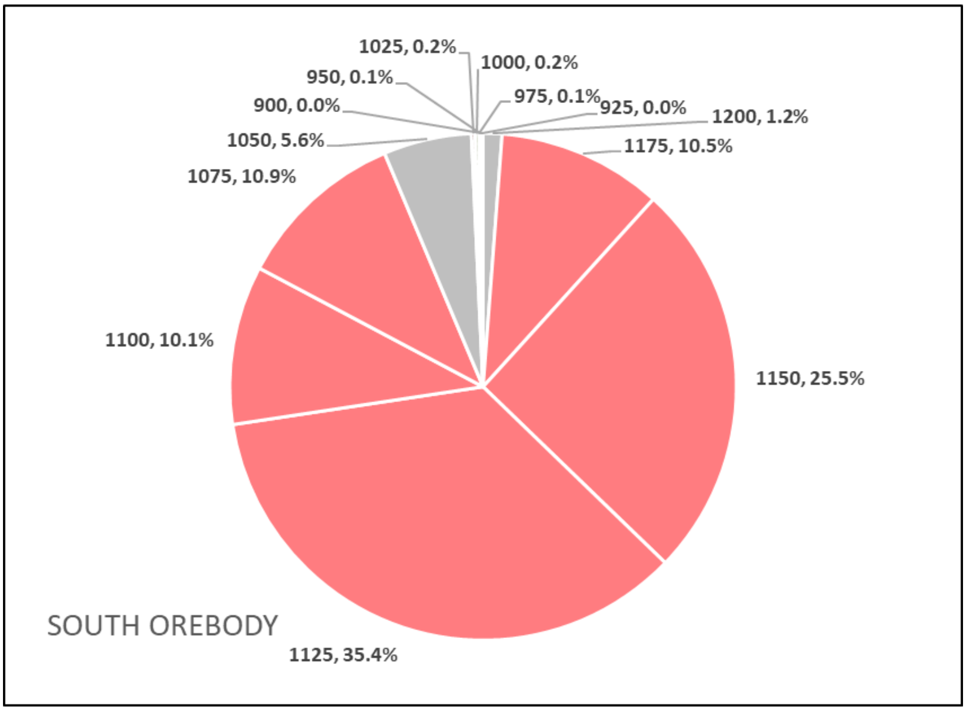


Figure 15.18 South Orebody, Mineral Reserve in situ zinc (%) by sublevel



15.7.1 Comparison with previous estimates

There is no comparison to be made with any previous estimates due to planned production from the Main Orebody being essentially a remnant mining operation. Any comparison is essentially meaningless as the Main Orebody deposit is almost mined out. This is the first South Orebody Mineral Reserve estimate that has been declared.

15.7.2 Potential factors which could impact Mineral Reserve reporting

To the extent known, there are no significant risks relating to the Mineral Reserve estimates, from mining, metallurgical and infrastructure factors. ÇBI's DST activities will need to be addressed as part of the Environmental Permit renewal. In the QP's opinion, there remains some uncertainty as to when and if the formal permit renewal will eventually be forthcoming.

Item 16 MINING METHODS

16.1 Introduction

This item provides information on the mining methods and operations as they currently exist, supplemented with additional information on the proposed mining and operational aspects for the newly defined South Orebody.

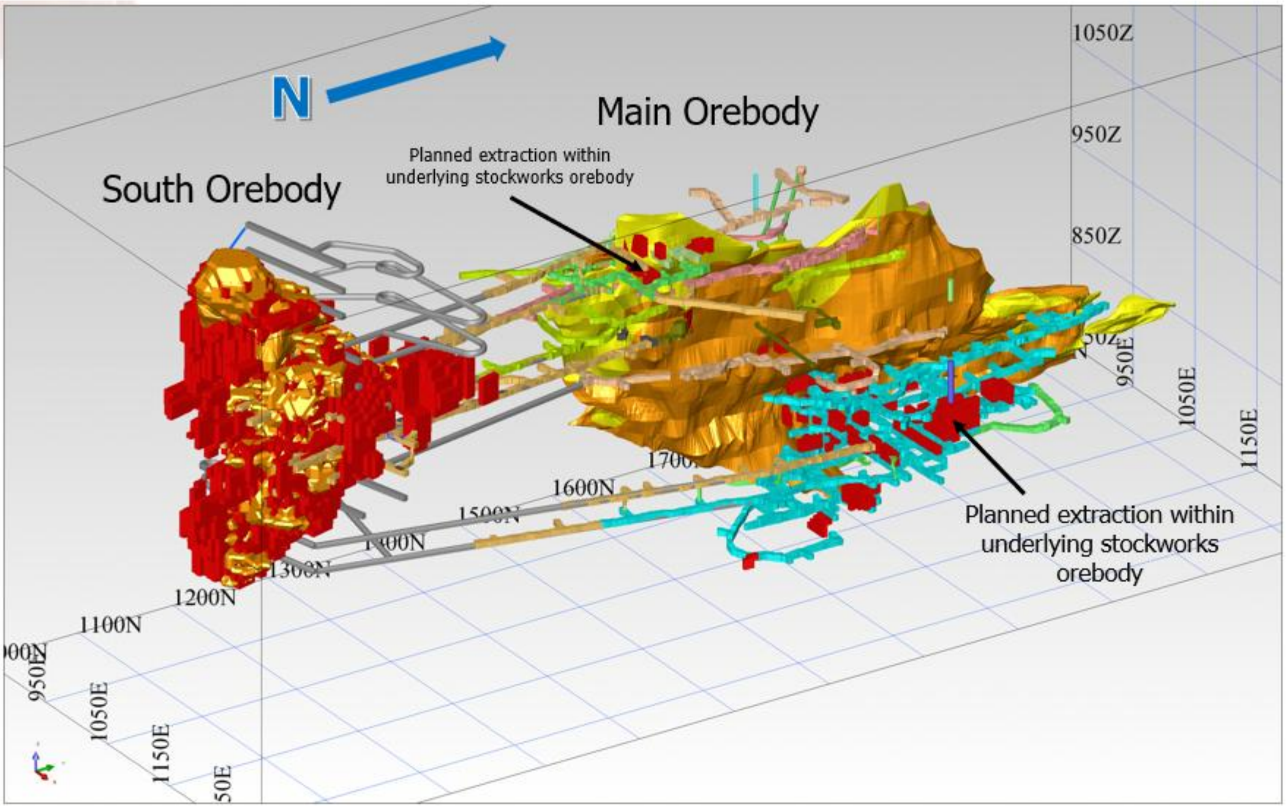
Included herein is a summary of the LOM production schedule as produced by the ÇBI mining team, with oversight by the mining QP. The supporting LOM blended feed schedule was produced by the ÇBI processing team, again with oversight by the mining QP.

16.2 Mining overview

An underground bulk mining method is in use at Çayeli, with the practice of backfilling to maximise the stoping of ore in a sequential extraction manner. Figure 16.1 shows an oblique view through the Main and South Orebodies. Ore production in the Deep Orebody below the Main 800 mRL sublevel was essentially completed by 2020.

Clastic and massive sulphide mineralisation is predominant on the western, hangingwall side of the Main and now depleted Deep Orebodies, transitioning to stockwork mineralisation towards the footwall. The position of the orebodies can be seen in Figure 16.1, specifically the proposed remnant mining areas in the Main Orebody and the newly defined mineralisation in the South Orebody.

Figure 16.1 Schematic view of the Çayeli Main and South Orebodies



The upper Main Orebody is separated from the lower, depleted Deep Orebody by a structure referred to as the Scissor Fault. The newly defined South Orebody is located approximately 300 m from the footwall of the Main Orebody, where shown in Figure 16.1.

The production sublevels which are the subject of the Mineral Reserve estimate are listed in Table 16-1.

Table 16-1 Mineral Reserve production sublevels

Orebody	Sublevel (mRL)	Ore dev't headings (#)	Stopes (#)
Main (remnants)	1110	6	5
	1090	2	3
	980	1	26
	960	5	21
	920	0	1
	900	5	8
	880	5	21
	860	7	34
	840	3	7
	820	2	4
	Subtotal	**36**	**130**
South	1200	20	25
	1175	48	60
	1150	50	69
	1125	58	100
	1100	54	73
	1075	80	140
	1050	50	133
	1025	60	115
	1000	53	88
	975	45	66
	950	35	67
	925	27	38
	900	13	28
	Subtotal	**593**	**1,002**

The existing underground mine, to the base of the abandoned Deep Orebody sublevels, reaches a depth of about 550 m below surface. The hoisting shaft has been decommissioned and backfilled, whilst leaving the ladderway compartments open for secondary egress and downcast ventilation. Access into the existing mine is now via a decline, which above the 800 mRL sublevel is positioned in the hangingwall. A new ramp from surface is being developed to access the upper South Orebody, and its completion is imminent.

The Main Orebody sublevels have been developed off the decline at varying vertical intervals. The sublevel vertical separation is notionally 20 m, allowing the layout of generic 15 m high by 6 m wide stope blocks. The continued stoping (and ore development) for the Main Orebody is now essentially a remnant mining activity.

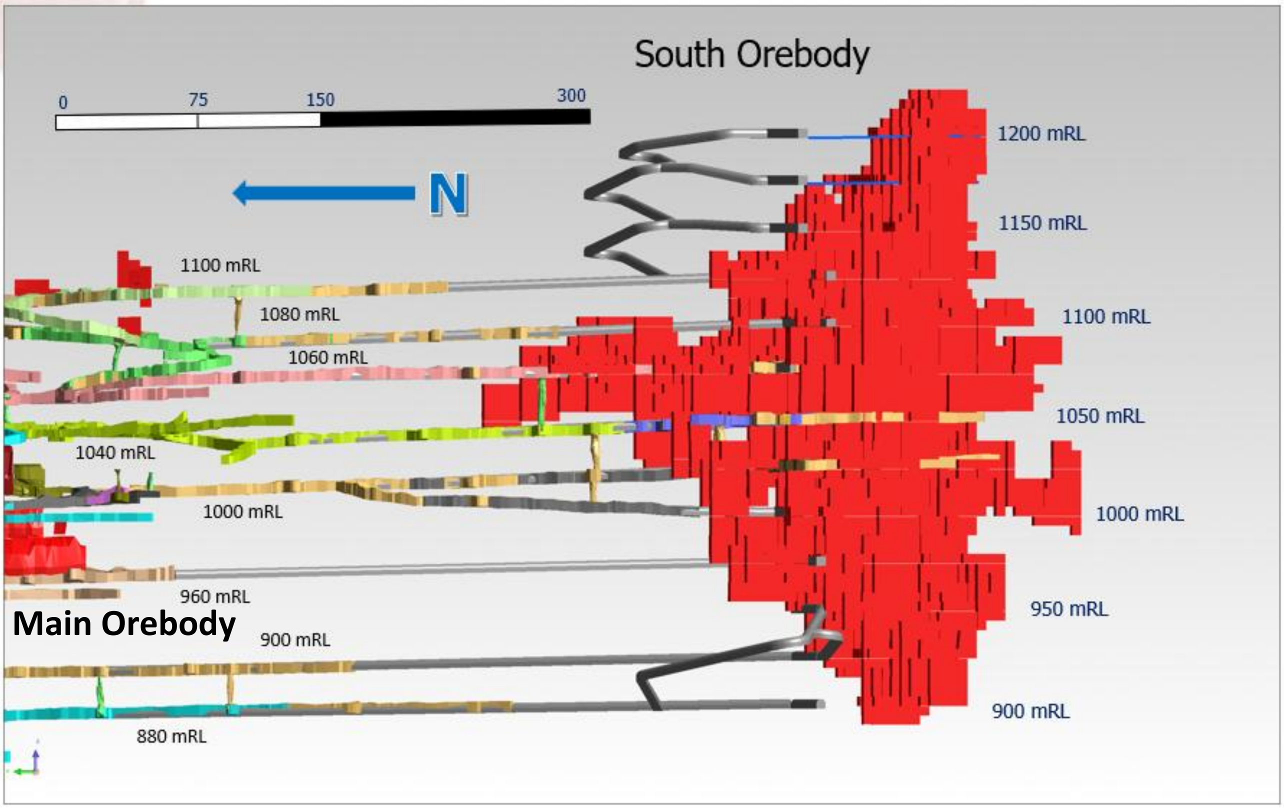
A similar bulk mining method is proposed for the South Orebody, into which some initial ore development has already been completed. The planned sublevel vertical separation is 25 m.

New access development has been mined from the Main Orebody across to the South Orebody (Figure 16.2). The Main to South connections are as follows:

- 1100 mRL sublevel to the 1125 mRL sublevel (in progress)
- 1080/1090 mRL sublevel to the 1100 mRL sublevel (in progress)
- 1060 mRL sublevel to the 1075 mRL sublevel (completed)
- 1040 mRL sublevel to the 1050 mRL sublevel (completed)
- 1000 mRL sublevel to the 1025 mRL sublevel (completed), with a ramp leading down to the 1000 mRL sublevel (in progress)
- 960 mRL sublevel to the 975 mRL sublevel (in progress)

- 900 mRL sublevel to the 925 mRL sublevel (in progress), with a ramp leading to the 950 mRL sublevel
- 880 mRL sublevel to the 900 mRL sublevel (in progress)

Figure 16.2 Oblique view of Main and South Orebody access development

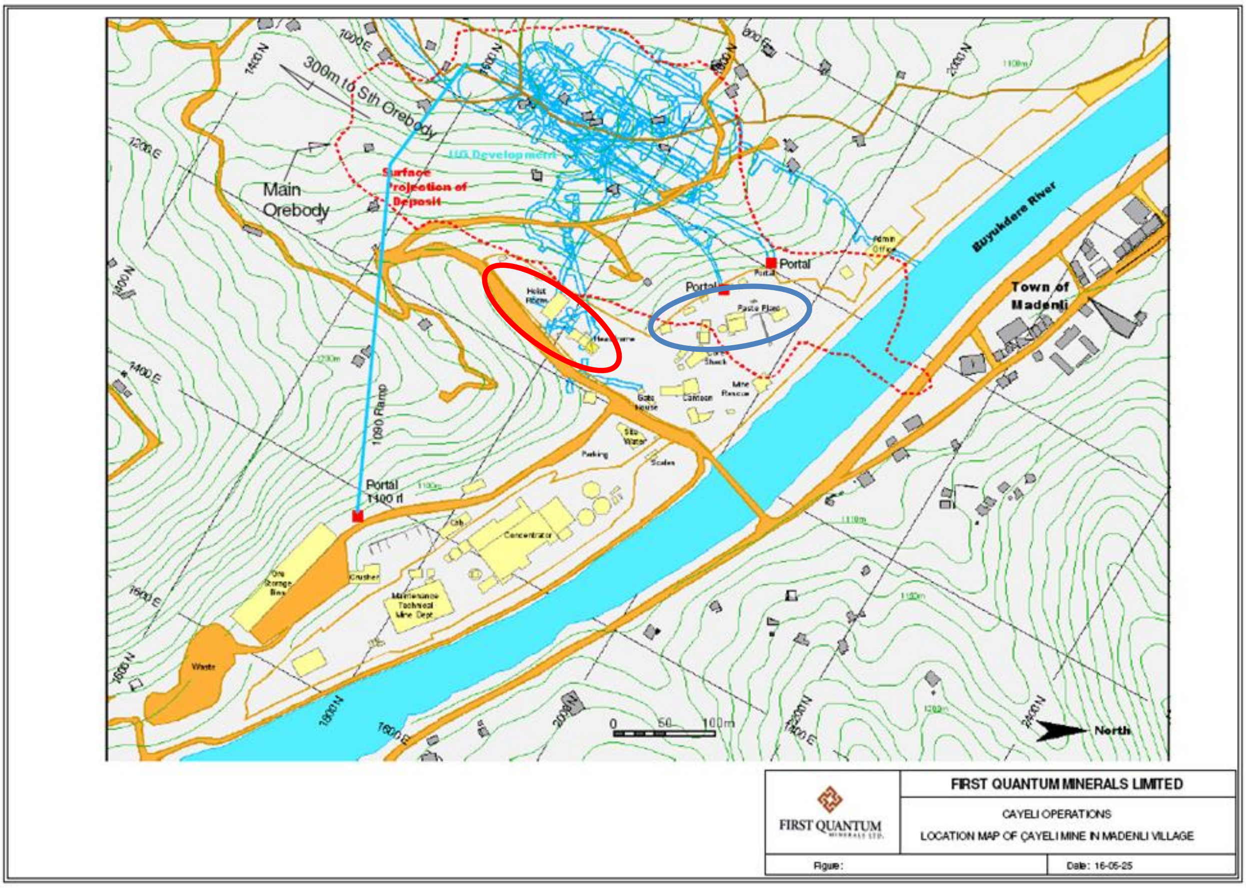


16.3 Mine site surface layout

In Figure 16.3, the Çayeli concentrator and mine administration facilities are apparent along the bank of the Büyükdere River. The surface projection of the existing Main Orebody extents is indicted by the dashed red line, whilst the location of the mine headframe and winder house (now decommissioned) are shown circled in red, and the paste fill plant location is circled in blue.

Although not shown in Figure 16.3, the projected surface location of the South Orebody is to the south-west of the old hoisting shaft.

Figure 16.3 Site layout plan



16.4 Mining method and operations

Ore and waste development is carried out conventionally, using jumbo drills, front end loaders and articulated dump trucks. In zones where weak rock mass conditions are experienced, rock breakers are used in place of jumbo drilling and blasting.

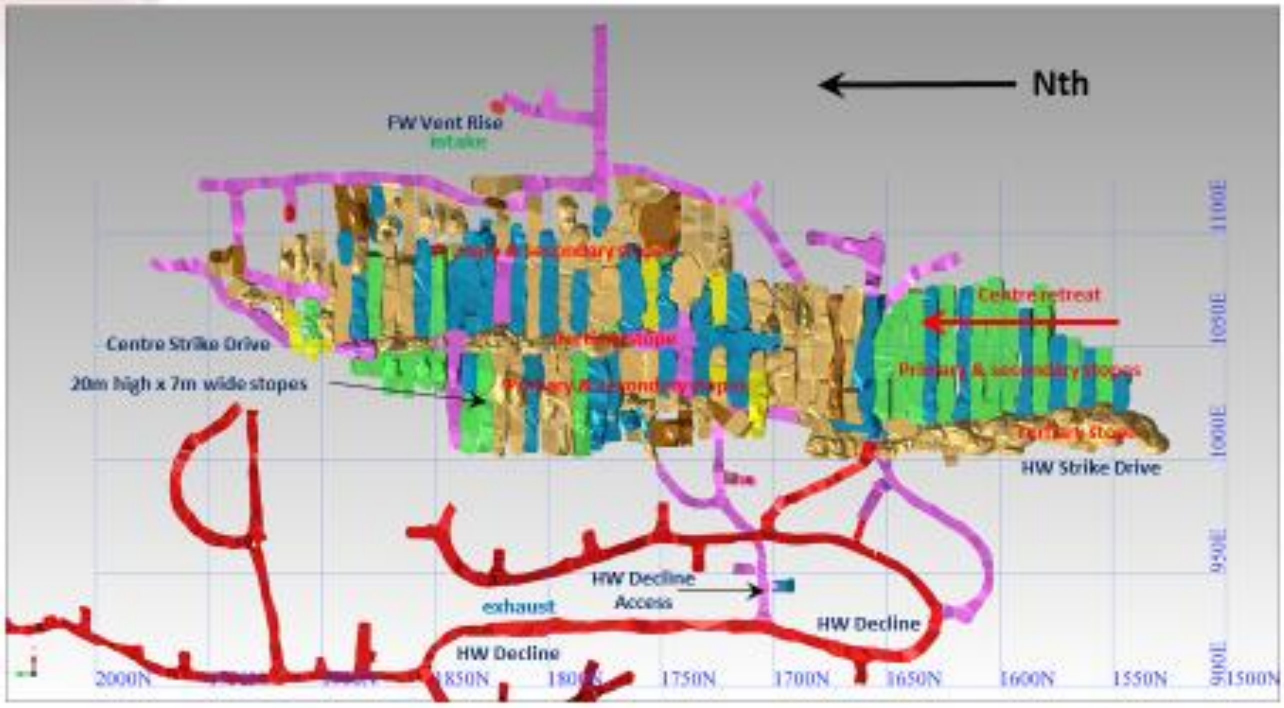
The primary production method for the Main Orebody is conventional long hole stoping with extraction maximised by means of paste filling or with unconsolidated waste rock fill. The method is overhand (i.e., progressing from bottom-up in each stoping block), retreating from a mined slot rise, and featuring primary, secondary and tertiary stope sequencing.

Depending on the sublevel elevation relative to the decline location, and the width of the economic mineralisation, historical extraction in the Main Orebody has proceeded either transversely from a hangingwall or footwall slot rise in the primary and secondary stopes, and then subsequently in a tertiary stope along the strike of the orebody. The primary and secondary stopes have been typically paste filled, whilst the tertiary stopes have been unconsolidated waste rock filled.

Figure 16.4 shows a typical layout of completed sublevel development and stoping, in this instance on the 940 mRL sublevel of the Main Orebody.

The stopes have been designed and sequenced for extraction without the need of sill pillars. In a general sense, lower primary stopes are mined underneath upper un-mined secondary stopes. In this way, stoping can proceed from bottom-up, over multiple operating sublevels of the mine.

Figure 16.4 Example (completed) development and stoping layout, 940 mRL sublevel, Main Orebody



A cut and fill mining method was applied successfully in 2019 in the upper mine sublevels of the Main Orebody to eliminate the risk of subsidence to various sublevels.

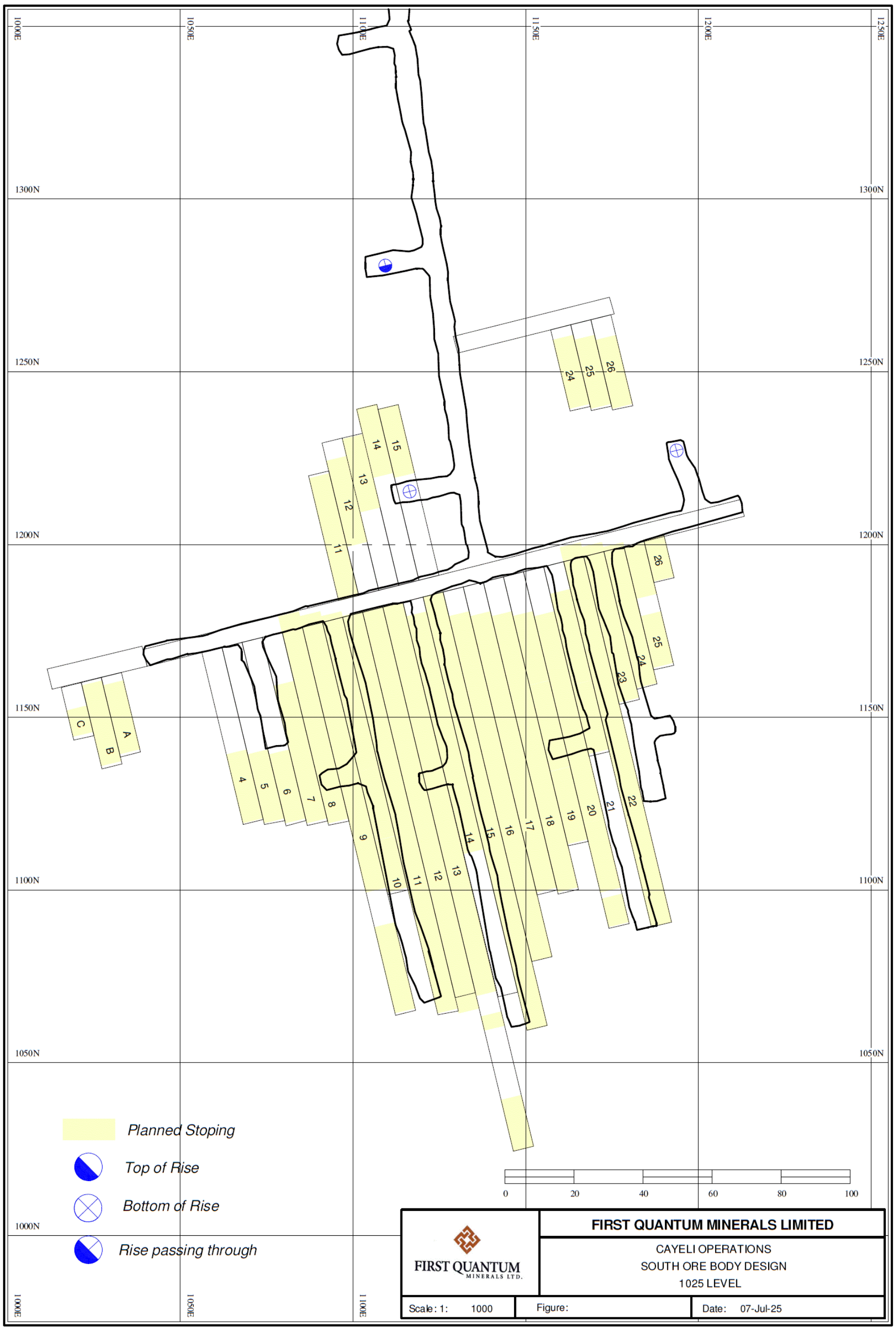
The proposed South Orebody mining method is similar to that for the Main Orebody, although the sequencing will differ due to a more systematic stope extraction plan arising from the new design layout (Figure 16.5).

16.4.1 Stope production sequence

Figure 16.6 shows the production sequence in a typical operating long hole stope. The sequence is as follows:

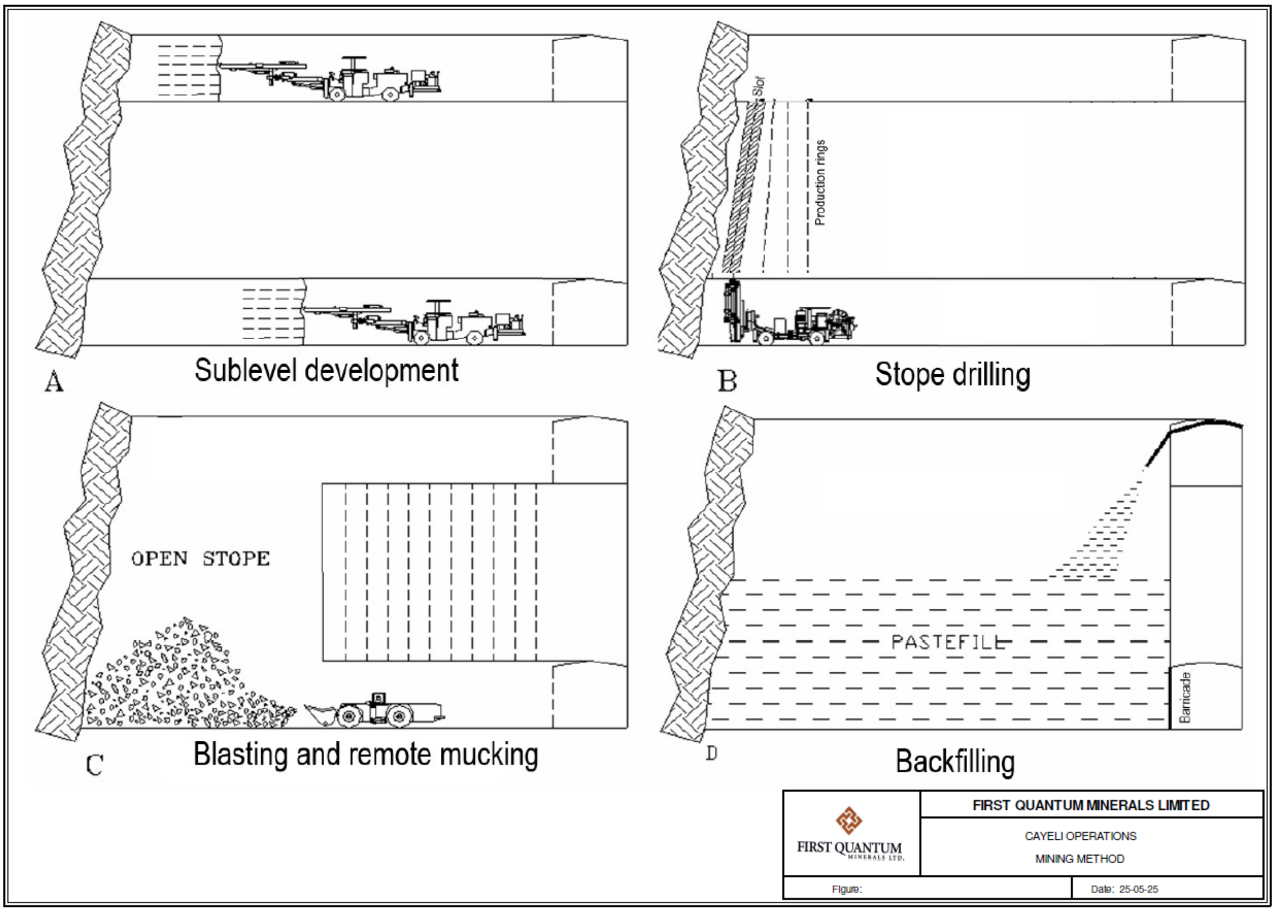
1. Sublevel openings in ore are developed with a jumbo, leading off from access crosscuts emanating from the decline.
    a) In the wider Main Orebody horizons, these sublevels typically had strike drives positioned in the centre of the orebody or along the hangingwall contact.
    b) In the narrower and now depleted Deep Orebody horizons, these sublevels were typically positioned along the footwall contact.
    c) The vertical spacing between sublevels is notionally 20 m in the Main Orebody and was 15 m in the Deep Orebody. In the South Orebody, the spacing is planned at 25 m.
2. From the sublevel ore drives, crosscuts are then developed with a jumbo, out to the footwall or hangingwall extremity of the orebody (View A).
    a) Depending on the orebody width, the length of the ore crosscuts varies from 10 m to 50 m.

Figure 16.5 Example (planned) sublevel development and stoping layout, 1025 mRL sublevel, South Orebody



3. Production drilling with a long hole drilling rig is then completed in a two-step process.
   a) A slot rise is long hole drilled and excavated between two sublevels (View B).
   b) Whilst during or after developing the slot rise, a series of overhead long hole rings are drilled between the slot position and the ore drive (View B).
4. The production blast holes are then pneumatically loaded and fired in sequential rings, to be followed by remote-controlled loading (mucking) of the blasted ore.
   a) Stope blasting is carried out sequentially with blasting of one to two rings at a time and mucking until the stope excavation is complete (View C).
   b) The remote-controlled loaders tram the ore to a truck loading bay at the decline entry to the sublevel.

Figure 16.6 Sequence of individual stope production activities



5. At the completion of mucking ore from a blasted stope, barricaded backfilling of the void follows as soon as possible (View D).
   a) The primary stopes are paste filled, with a cement content of up to 4.5% for the Main Orebody and 8.5% for the South Orebody (planned).
   b) The secondary stopes, with the exception of the brow to be exposed during tertiary extraction, are filled with 3.5% to 4.0% cement content paste fill.
   c) The secondary stope brows are filled with 4.0% cement content paste.
   d) The tertiary stopes are typically filled with unconsolidated waste from development openings, thereby reducing the tonnage of waste rock hauled to surface.
6. For geotechnical stability reasons, the maximum stope lengths along any sublevel (i.e., between initial and successively installed barricades) are:
   a) 25 m for primary transverse stopes
   b) 20 m for secondary transverse stopes
   c) 15 m for tertiary/longitudinal stopes

Item 16.5.3 describes multiple ore types that are defined within the underground mine. These ore types can be differentiated during mining and then truck hauled to surface, where there is a compartmented stockpile allowing preferential reclaim and blending of feed to the processing plant.

In susceptible areas of the mine, the jumbo development, followed by the production drilling, blasting and mucking activities are each completed on a sequential "just-in-time" basis without hiatus periods between each activity. This is in response to prevailing ground conditions that could lead to dynamic deformation of the openings and the need for remedial ground support installations.

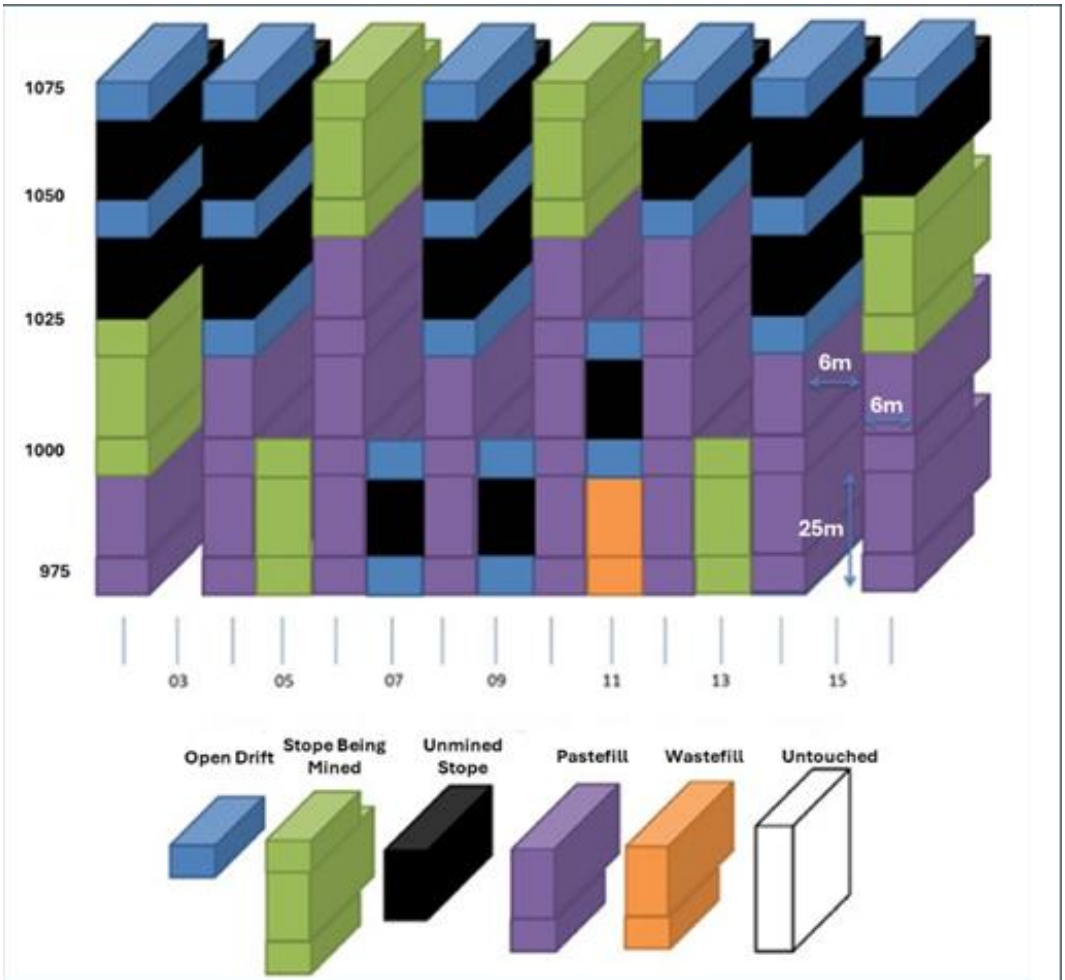
Further information on ore development, stope opening dimensions, and related mine design details are provided in Item 16.5. Further information on stope backfilling is provided in Item 16.4.3.

16.4.2 Sequencing across multiple sublevels

Figure 16.7 is a diagrammatic representation of the mining sequence as intended for the South Orebody ore development and stopes. Aspects of this sequence are:

- Primary stopes (e.g., SO2, SO4, SO8, S12, S14, S16):
  - lower stopes either backfilled or being mined above backfill
  - upper stopes being developed and not yet stoped
- Secondary stopes (e.g., S05, S07, S09, S11, S13):
  - lower stopes being developed or mined between backfilled primary stopes

Figure 16.7 Mining sequence across multiple sublevels (source: ÇBI, January 2025)



16.4.3 Stope backfilling

The primary means of stope backfilling is with the use of 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 that 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 is used to produce the paste fill, thus reducing the quantity to be disposed of external to the mine, and thereby minimising environmental impact

Several cemented paste fill recipes are utilised and are produced from unclassified plant tailings mixed with up to 4.5% cement for the Main Orebody and up to 8.5% cement for the South Orebody.

Drawing on extensive operating experience, the following best practices for paste backfilling are adopted:

- Stage 1, 8 m plug fill:
  - cure time of 2 days
  - use higher cement content for the plug regardless of stope type
- Stage 2, continuous filling:
  - maintain maximum fill rise rate of 50 m^3/hour

A number of other specifications apply to the cemented paste fill, as follows:

- target slump: 8.5 inches (21.5 cm); the operational range is within 7 inches to 10 inches (17.5 cm to 27 cm)
  - slump tests are conducted once per shift
  - fill samples are tested for compressive strength at day 2 and again at day 28
  - day 2 expected strength is 75 kPa to 125 kPa (self-support criteria)
  - 28 day strength targets vary by cement content and backfill zone type
- placement rate limits:
  - maximum height rise: 0.35 m/hour (about 8 m/day) (first stage)
  - minimum fill rate: 30 m^3/hour to avoid pipe blockage
  - stage 1 starts 2 m to 3 m above barricade level
- backfill stability and blast wait times:
  - below cemented paste fill: wait 28 days
  - adjacent to cemented paste fill (side walls): wait 21 days
  - in front of cemented paste fill (slot raise): wait 14 days
  - face advance beside cemented paste fill: wait 7 days
  - these times are valid under normal UCS strength development
  - if strength falls below expectations, these curing times are increased

Information on the surface paste fill plant and delivery of fill to the mine is provided in Item 17.5.8 and Item 16.6.8, respectively.

Waste rock will need to be disposed of throughout the mine life. Some of this waste 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 (Item 16.7).

16.4.4 Stope barricading

The stope paste fill barricades are of timber and steel rod construction; an example is shown in the Figure 16.8 photograph. Each barricade design is bespoke, taking into account the actual stope geometry and the required fill sequencing.

Some generic construction specifications, per barricade, are as follows:

- 5 x 1 m length ribbed iron bolts
- 34 x 2 m length ribbed iron bolts
- 50 x 4 m length ribbed iron bolts
- 27 lengths of 5 cm x 10 cm x 50 cm timber planks
- 14 sheets of 0.8 cm thick plywood

From field tests and operating practice, paste filling and barricade “rules” are adopted and include:

- no paste backfill is allowed without a completed barricade control form
- no filling is allowed without outlet pipes being in place
- for fill reticulation, one fill and two outlet holes/pipes are required for blind stopes
- improperly positioned boreholes must be re-drilled
- filling must always begin from the borehole closest to the barricade
- barricade shotcreting must be completed in the same shift
- surface water inflow must be blocked; necessary drainage must be piped through the barricade
- very small voids should be filled with unpressurised paste
- unknown voids must first be sealed with stage 2 backfill used as a plug

Continuous barricade pressure monitoring is critical for safe paste backfill operations and the optimisation of barricade design and paste fill curing specifications. Barricade pressure development is highly sensitive to fill sequencing, geometry, and material properties. An instrumented survey of a typical barricaded and filled stope showed that even with a consistent paste recipe and placement rate, barricade loading can vary significantly (Yumlu, 2008). Hence, routine monitoring of barricade installations includes:

- camera monitoring of each barricade
- backfill operations must be halted immediately if any of the following are observed:
  - barricade damage, visible cracking, excessive wetting, or failure to receive real-time camera feed.
  - blind stopes: reticulation pipe pressure exceeds 2 MPa, sudden pressure change, inability to verify open air holes after 90% filling, or backflow through air holes
  - open sublevel stopes: no visible flow from the pipe outlet, camera malfunction without a designated observer, or when paste is within 1.5 m of the floor

Figure 16.8 Photograph showing a typical fill barricade construction in the Main Orebody

16.5 Mine planning considerations

16.5.1 Planning “rules”

Table 16-2 lists the basic mining dimension details adopted for mine planning. In the case of the Main Orebody, the spacing between sublevels does vary, although for consistency, the planned remnant mining sublevels are referred to as though on a 20 m vertical spacing.

Table 16-3 lists the basic stope design considerations. For ease of design, an overall $77/t ore is the LOM estimated average operating cost which is referenced against the combined block NSR values that form a stope solid. In addition to geometrical constraints, other design considerations relate to ground conditions and, in the case of the Main Orebody remnants, the proximity to backfilled stopes.

The design maximum stope lengths (i.e., between barricades) are:

- primary transverse stopes = 25 m
- secondary transverse stopes = 20 m
- tertiary/longitudinal stopes = 15 m

Table 16-4 lists the stope drilling considerations, specifically for slot and production drilling rings.

Table 16-2 Mining dimensions

Sublevel intervals		
Main Orebody	(m)	20
South Orebody	(m)	25
Development		
Capital development	(m x m)	5 x 5
Waste development	(m x m)	5 x 5
decline gradient	(%)	10% to 14%
Ore development	(tonnes/m)	100
Main Orebody	(m width)	6
South Orebody	(m width)	6
	(m height)	5
Ore development spacing		
Main Orebody	(m horiz.)	6
South Orebody	(m horiz.)	6

Table 16-3 Stope design considerations

Design to operating cost NSR wireframe:			**incl. Zn, Ag**	
2025	($/t ore)	$79	(% Cu_{eq})	1.02
2026	($/t ore)	$73	(% Cu_{eq})	0.86
2027	($/t ore)	$71	(% Cu_{eq})	0.84
2028	($/t ore)	$69	(% Cu_{eq})	0.88
2029	($/t ore)	$69	(% Cu_{eq})	0.80
2030	($/t ore)	$73	(% Cu_{eq})	0.74
2031	($/t ore)	$74	(% Cu_{eq})	0.93
2032	($/t ore)	$79	(% Cu_{eq})	1.00
2033	($/t ore)	$83	(% Cu_{eq})	1.10
2034	($/t ore)	$92	(% Cu_{eq})	1.20
2035	($/t ore)	$93	(% Cu_{eq})	1.27
2036	($/t ore)	$98	(% Cu_{eq})	1.30
2037	($/t ore)	$97	(% Cu_{eq})	1.30
LOM average	**($/t ore)**	**$77**	**(% Cu_{eq})**	**0.95**
Design to geometrical constraints:				
minimum orebody dip (long hole design)	(degrees)	50		
minimum orebody width for tranverse stopes	(m)	15		
[revert to longitudinal at <15 m width]				

Table 16-4 Stope drilling considerations

Stoping physicals		
Slots		
Slot rising holes:		
hole length	(m)	12
no. of holes	(#)	14
Slot raise hole diameter	(mm)	76 to 150
Production Rings		
Burden		
slots	m	0.5
production rings	m	2 to 2.5
Spacing (collars)	cm	50 to 60
Production holes:		
MOB hole length	(m)	15
SOB hole length	(m)	20
No. of holes per row (ring)	(#)	5
Drilling diameter	(mm)	76
Production tonnes/drill metre	(t/ drill m)	6.7

16.5.2 Development and stope identifier

For each stoping block, and on every sublevel, there is a numbering sequence to identify development and stope openings. An example of this in the South Orebody, would be:

- Ore crosscut – 1025S07, where 1025 refers to the sublevel RL, and the next three figures are a sequential identifier
- Stope – S1025S07, which identifies the stope associated with the underlying crosscut

In the current LOM design, in the Main and South Orebody areas of interest, there are approximately 630 ore development openings and 1,130 individual stope openings (primary, secondary and tertiary).

16.5.3 Mined ore types

Referring to the mineralogical criteria listed in Item 13.1, there are eight ore types defined for the Çayeli orebodies, as follows:

Main Orebody:

- Spec Yellow Ore (YO -)
- Spec Yellow Ore (YO +)
- Spec Yellow Ore (LYO)
- Non-spec Clastic Ore (CO -)
- Non-spec Clastic Ore (CO +)
- Non-spec Black Ore (BO)

South Orebody:

- Spec Yellow Ore (SYO)
- Non-spec Black Ore (SBO)

The eight ore types are flagged in the Mineral Resource model (Item 14) according to the mineralogical criteria and can therefore be carried into the mine planning model and production plan.

The mined ore is selectively reclaimed from surface stockpiles to the processing plant to suit concentrate sale requirements and to optimise the NSR. There is currently a zinc concentrate, a Spec copper concentrate and a Non-spec copper concentrate.

16.5.4 Mine geotechnical engineering

Management of ground conditions

In relation to ground conditions, the orebody rock mass and host rocks can be classified as poor to fair quality, characterised by intense schistosity and foliation. These conditions require special attention and intensive ground support and reinforcement measures. Initially, the mine used conventional ground control measures such as rock bolting and meshing, with occasional shotcreting. These measures proved to be inadequate in dealing with the relatively dynamic deterioration of mine openings once exposed.

Subsequently, the ground support and reinforcement standards were upgraded, particularly on the hanging wall side of the Main Orebody. In response, the main decline and primary development at depth, below 800 mRL, was transferred from the hangingwall into the more competent footwall of the deposit.

Consultant geotechnical reviews and numerical modelling of mining-induced stresses have been carried out in the past. In combination with the findings from this work, operational experience has resulted in changes to the direction and sequencing of Main Orebody stopes to avoid high induced stress concentrations, as well as identifying maximum allowable stope spans.

Furthermore, operational experience over time has led to the adoption of the “just-in-time” sequencing of production activities in the Main Orebody, as described in Item 16.4. In these ground conditions, it was considered appropriate that broken ore loaded from the stopes was done so by remote control.

Main Orebody ground conditions

In common with other VMS style deposits, many of the Main Orebody rock types are pervasively altered and contain substantial amounts of clay. In some cases, the rock has been completely altered to clay. Table 16-5 summarises the major rock types and alteration progressing from the footwall to the hangingwall in the Main Orebody.

Table 16-5 Main Orebody rock and alteration types (source: AMC, 2014)

Rocktype	Description	Alteration Type
Rhyolite	Fine grained granular or spherultic texture	• Patchy clay • Chlorite • Clay
CVO	Coarse grained veins of pyrite and chalcopyrite in clay altered rhyolite	• Clay
Yellow ore	Pyrite and chalcopyrite clasts up to 20 cm in size in a sulphide matrix containing <10% sphalerite	
Black ore	Pyrite and chalcopyrite clasts from 2 mm to >20 cm in a matrix containing <10% sphalerite	
Clastic ore	Pyrite, chalcopyrite and sphalerite clasts from 2mm to >20cm in a sulphide matrix	
Tuff	Composed of variably compacted, green and white clay altered pumice clasts, enclosing scattered lithic clasts of red-brown and green-white rhyolite	• Clay
Basalt	Porphyritic basalt	• Chlorite • Haematite
Dykes	Dolerite	

The large-scale, undulating Scissor Fault separates the Main Orebody from the now abandoned Deep Orebody between the 700 mRL and 800 mRL sublevels (see Figure 16.9 example cross section). The thickness of the Main Orebody above the 800 mRL sublevel is attributed by Allen (2006) to fault repetition. Below the 800 mRL sublevel, the ore body thins and the strike length increases.

Figure 16.9 Location of the Scissor Fault, separating the Main and Deep Orebodies on section 1120 mN

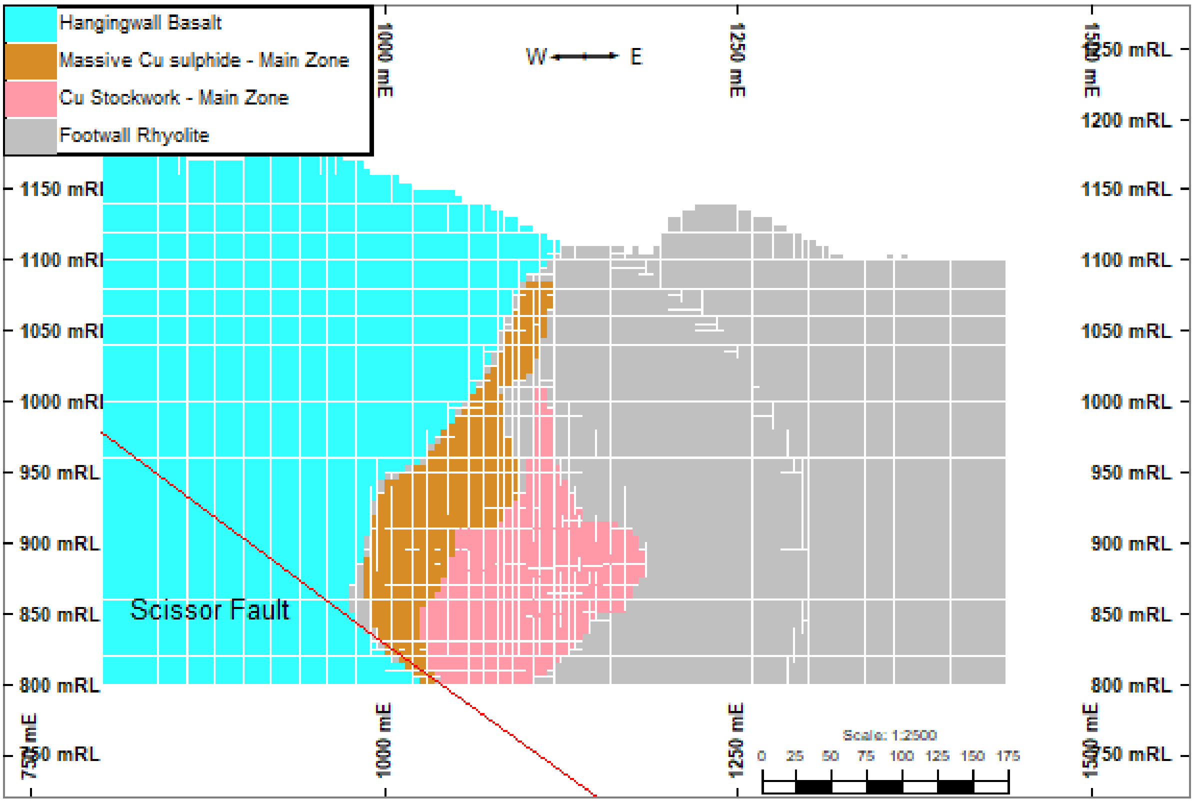


Table 16-6 lists the material properties used in the stability analysis models for the Main Orebody, referencing available laboratory test results, field estimates and geotechnical logging data. Properties for an interpreted Scissor Fault damage zone have been estimated, whilst the fault plane was added as a weakness zone in numerical models. Calibrated material properties are highlighted in orange based on existing stope performance considerations.

Table 16-6 Main Orebody geotechnical material properties (source: AMC, 2014)

Rock Unit	Intact Rock Properties				Rockmass Properties (Hoek-Brown)			
	Density (t/m^3)	E^1_{intact} (GPa)	UCS^2_{intact} (MPa)	mi[3]	GSI	m[3]	s[3]	E_{rm} (GPa)
Basalt (Host)	2.7	60.0	120.0	25	63	6.67	0.0164	35.3
Unaltered Rhyolite (FW)	2.7	60.0	120.0	25	63	6.67	0.0164	35.3
Patchy Clay Alt. Rhyolite (FW)	2.7	12.0	30.0	15	50	2.52	0.0039	3.7
Clay Altered Rhyolite	2.7	12.0	30.0	15	50	2.52	0.0039	3.7
CVO	2.6	1.5	15	5	90	7.34	0.1084	2.2
Ore (Yellow and Black)	2.6	15.0	35.0	25	40	2.93	0.0013	2.4
Ore (Clastic)	2.6	15.0	75.0	25	75	10.24	0.0622	12.2
Tuff	2.4	8.0	20.0	15	40	0.94	0.0013	2.4
Dyke	2.6	60.0	175.0	25	75	10.24	0.0622	48.9
Scissor Fault (Zone)	2.4	12.0	30.0	25	28	1.20	0.0001	0.4
Scissor Fault (Weak Plane)	2.4	12.0	8	25	28	1.20	0.0001	0.4

[1] Young's Modulus
[2] Unconfined Compressive Strength
[3] Hoek-Brown Constant

South Orebody ground conditions

The South Orebody mineralisation over several sublevels, varies in width between 50 m and 160 m in the North-South direction. The orebody has stockwork mineralisation in the footwall between the 1110 mRL and 1225 mRL sublevels, whilst there is only veined footwall mineralisation between the 900 mRL and 1100 mRL sublevels.

Clay, pyrite, chalcopyrite and calcite are evident throughout the South Orebody, and these have a significant effect on the intact rock strength. It has been determined in the drilling and on-site field studies that the strength decreases as the amount of clay and pyrite in the formation increases.

The following geotechnical characterisations are adopted for the South Orebody, according to the various lithologies expected (Ferid, April 2024):

- Unaltered rhyolite (UNRHY): High rock strength values and no clay alteration and pyrite mineralisation.
- Patchy clay altered rhyolite (PCLAY): Rhyolite with less than 15% clay alteration and containing pyrite and/or chalcopyrite, Pyrite and/or chalcopyrite are found in veins and scattered forms.
- Clay altered rhyolite (RCLAY): Rhyolite with more than 15% clay alteration and containing patchy pyrite grains. Due to its clay content, RCLAY strength values are expected to be low, especially when it interacts with groundwater.
- Chloritic rhyolite (RCHL): Rhyolite with chlorite alteration. It shows similar properties to patchy clayey or dense clayey rhyolite depending on the clay and pyrite mineralisation content.
- Pyrite ore (PO): Rhyolite with fine grained pyrite mineralisation. Rock strength decreases depending on clay alteration intensity.
- Chalcopyrite veined ore (CVO): High grade rhyolite with intense chalcopyrite mineralisation.
- Black ore (BO): Fine to coarse grained massive sulphide aggregates with >10% sphalerite.
- Clastic ore (CO): Ore zones generally consist of a dark grey sphalerite containing angular to sub-rounded clasts of pyrite, chalcopyrite, sphalerite and occasionally bornite and pyrrhotite.

From the work of Ferid (April, 2014):

1. The RCLAY unit is commonly encountered In the South Orebody. Uniaxial compressive strength (UCS) values for RCLAY samples range between 15 MPa and 30 MPa, whilst the corresponding UCS values derived from point load index tests fall within the 11 MPa to 40 MPa range[8]. The tensile strength of RCLAY is relatively high, typically ranging from 6 MPa to 7 MPa, resulting in a high tensile-to-compressive strength ratio for this unit.
2. For the PCLAY and RCHL units, UCS values range from 20 MPa to 80 MPa, whilst the corresponding point load-derived UCS values range from 80 MPa to 160 MPa. The Hoek-Brown constant (mi) has been measured as 43 for RCHL and 19 for PCLAY.
3. With data obtained from 17 development faces, adjusted rock mass rating (RMR) values for mine design were advised to be in the following ranges:
   a) PCLAY: RMR (adjusted)= 44 to 60
   b) RCHL: RMR (adjusted) = 45 to 50
4. These values can decrease by up to 10 points in the presence of chalcopyrite or pyrite veins. While locally massive and low-fractured zones within the RCLAY unit may yield higher RMR values, the high clay content negatively affects long-term behaviour, leading to adjusted RMR values in the range of 30 to 40.
5. Other lithologies, such as PO, CVO, brown tuff, flysch-like formations, and schistose, highly interbedded brown units, have adjusted RMR values typically in the range of 34 to 36. In locations where discontinuities are more prominent and rock mass quality decreases, RMR values can drop by an additional 10 points.
6. In parallel, the rock mass rating Q-System values for units such as RCHL and PCLAY are generally in the 1 to 10 range, but in weaker zones, they can decline to as low as 0.1 to 0.6. In highly disturbed zones of brown schistose, densely layered tuff units, and in the PO unit, Q-System values may drop to 0.1 to 0.08, and in severely fractured or disturbed sections, values as low as 0.028 have been observed.
7. The selection of parameters for design were primarily based on underground site visits and development face mapping. Additionally, existing operational data, including exploration drilling logs and imagery, were utilised. Due to the relatively higher degree of disturbance in exploration core samples, there may be misleading impressions of poorer rock mass conditions than what is present. Therefore, the assessments for mine design have been made with a conservative approach.

To date the Scissor Fault has not been identified in the South Orebody region.

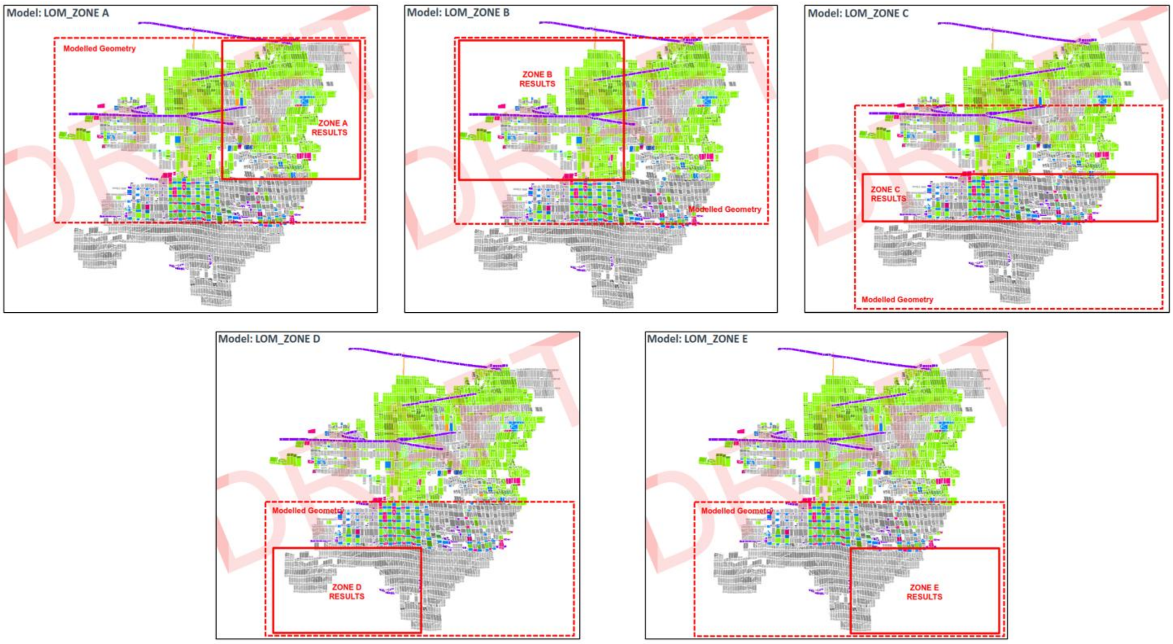
Mining induced stress conditions in the Main Orebody

In 2012, AMC Consultants (AMC, 2012) completed a 3D numerical analysis (linear-elastic boundary element method) of induced stress conditions in the Main Orebody. The modelling methodology incorporated a back analysis of several rock mass failures that had previously occurred.

[8] Internal microdefects can exist in core samples obtained using HQ-diameter drilling. The point load strength test, typically performed on smaller specimens, tends to yield higher strength values, suggesting a stronger material. This distinction is critical when selecting GSI (Geological Strength Index) values using the generalised Hoek-Brown failure criterion. If point load test results are used, lower GSI values should be considered; conversely, if UCS from uniaxial compressive tests is the basis, higher GSI values may be more representative, Ferid (April, 2014).

The numerical analysis was completed by subdividing the Main Orebody (as then developed) into five modelling volumes, to reduce model run times and hence the computation of induced stresses (Figure 16.10).

Figure 16.10 Main Orebody numerical modelling volumes (source: AMC, 2012)



The resulting modelled induced stresses were categorised by AMC (2012)) according to factors of safety, rock mass conditions and forecast behaviour (Table 16-7). Four critical areas were identified where stopes were forecast to be subject to a high stress, unconfined state and/or where critical transition areas were highlighted.

Table 16-7 Main Orebody categorisation of mining induced stresses (source: AMC, 2012)

Stress State	Factor of Safety		Rock Mass Conditions	Forecast Behaviour
Low Stress, Confined	FS > 1.4	Sigma 3 > 0	Favourable Conditions	No stress induced rock mass damage
Moderate Stress , Confined	1.0 > FS > 1.4	Sigma 3 > 0		Approaching rock mass failure – minor local rock mass damage
High Stress, Confined	FS < 1	Sigma 3 > 0	Rock Mass Failure	Fracturing of intact rock mass
High Stress, Unconfined	FS < 1	Sigma 3 < 0		Plastic deformation of fractured rock mass
Low Stress, Unconfined	FS < 1	Sigma 3 < 0		Unravelling of fractured rock mass

The identified critical areas of the mine were identified in 2012 as being in the now abandoned Deep Orebody below 800 mRL (AMC, 2012). By inference it is possible that in the current Main Orebody mine plan, there may be elevated stresses in the lower sublevels from 900 mRL to 840 mRL, within areas that have been extensively stoped.

Mining induced stress (deformation) conditions in the South Orebody

In 2024, an evaluation was conducted to determine the extent of potential induced stress conditions in the South Orebody (Ferid, August 2024). The author concluded:

- From depths of approximately 200 m, deformation issues could be anticipated in units such as CVO, PO, and in brown tuff with closely interbedded structures[9].

[9] Due to the undulating topography, the top of the South Orebody ranges in depth from 70 m to 140 m below surface.

- The RCLAY unit, due to its clay content, is expected to exhibit deformation especially in openings approaching a depth of 400 m, particularly in the presence of groundwater inflows.
- Similarly, faulted or sheared zones within other units are also considered susceptible to comparable deformation behaviour.
- The expected deformation, influenced by depth and rock mass strength, may result in deformations ranging from 20 cm to 30 cm, and to 60 cm or more, potentially requiring repeated maintenance and re-profiling of the excavation surface.
- In contrast, the RCHL and PCLAY units are generally expected to exhibit mild squeezing or non-deformation behaviour under similar conditions.

In the 2024 Study (Ferid, August 2024), the following observations were also provided:

- At elevations above the 1125 mRL sublevel, there is the potential for loosening in the rock mass and the potential for structurally controlled failures. Given the presence of the PO unit at these levels, the loosening behaviour may be most evident when excavating through this lithology.
- On the 1025 mRL and 1050 mRL sublevels, stress relaxation within the rock mass is expected. As production approaches these areas from lower sublevels, the risk of overstressing is projected to diminish. Similarly, early development of production on the 1025mRL and 1050 mRL sublevels could help mitigate overstress problems in the underlying levels.
- In scenarios where primary stopes are developed across three sublevels, there is the potential for reduced confining pressure in the intermediate secondary stopes, which could lead to loosening and structurally controlled movements, particularly in the roof areas.

Ground support and reinforcement

The standard ground support and reinforcement measures in every opening in the underground mine are shotcrete and cement grouted rebar. There are other support elements used for additional ground control such as steel fibred shotcrete, shotcrete arches, cable bolts, steel arches, split sets and wire mesh.

The following is a list of standard ground control measures installed in primary development:

- steel fibre reinforced shotcrete (SFRS) around the entire perimeter of mine openings
- standard bolting with fully cement grouted rebar bolts (2.4 m bar length)
- shotcrete arches with or without fully-grouted cable bolts (9.0 m length, “bulbed”, single stand cable design)
- steel sets (arches) in classified “poor’’ quality ground conditions

In development headings, a 7 cm to 10 cm thick SFRS layer is applied, from invert to invert, as soon as a blasted round is mucked out. Following shotcrete application, rebar rock bolts (2.4 m long) are installed on a regular 1 m x 1 m pattern, beginning at 1.5 m high from the floor.

At a practical distance behind advancing headings in poor rock conditions, shotcrete arches are installed at 1 m centres. In addition, cable bolts are also installed between the shotcrete arches. Each cable bolted row consists of bolts spaced at 2 m apart. Steel arches are used where shotcrete arches and cable bolts are insufficient.

Development in poor ground conditions, typically, cannot advance more than 50 m in the footwall and 30 m in the hanging wall from the last shotcrete arches, the last row of installed cable bolts, or from shotcrete arches. Occasionally, bolting and shotcreting is required to rehabilitate openings that need to remain accessible.

In the hangingwall, footwall and central ore drives, openings are supported with SFRS (with fibre addition at 15 kg/m^3) and reinforced with grouted rebar bolts. Where necessary, shotcrete arches at 1 m spacing are installed, as are six to seven cable bolts in rows which are 2 m apart. Crosscut openings in ore are typically supported by plain shotcrete and reinforced with up to 12 rebar bolts in rows spaced at 1 m apart.

Additional shotcrete arches or cable bolt reinforcements are installed at intersections and pillars in zones where sublevel extraction could impact on crosscuts stability.

Rock reinforcement patterns

Figure 16.11 is a diagram showing the typical, intensive rock reinforcement (bolting) patterns adopted at Çayeli.

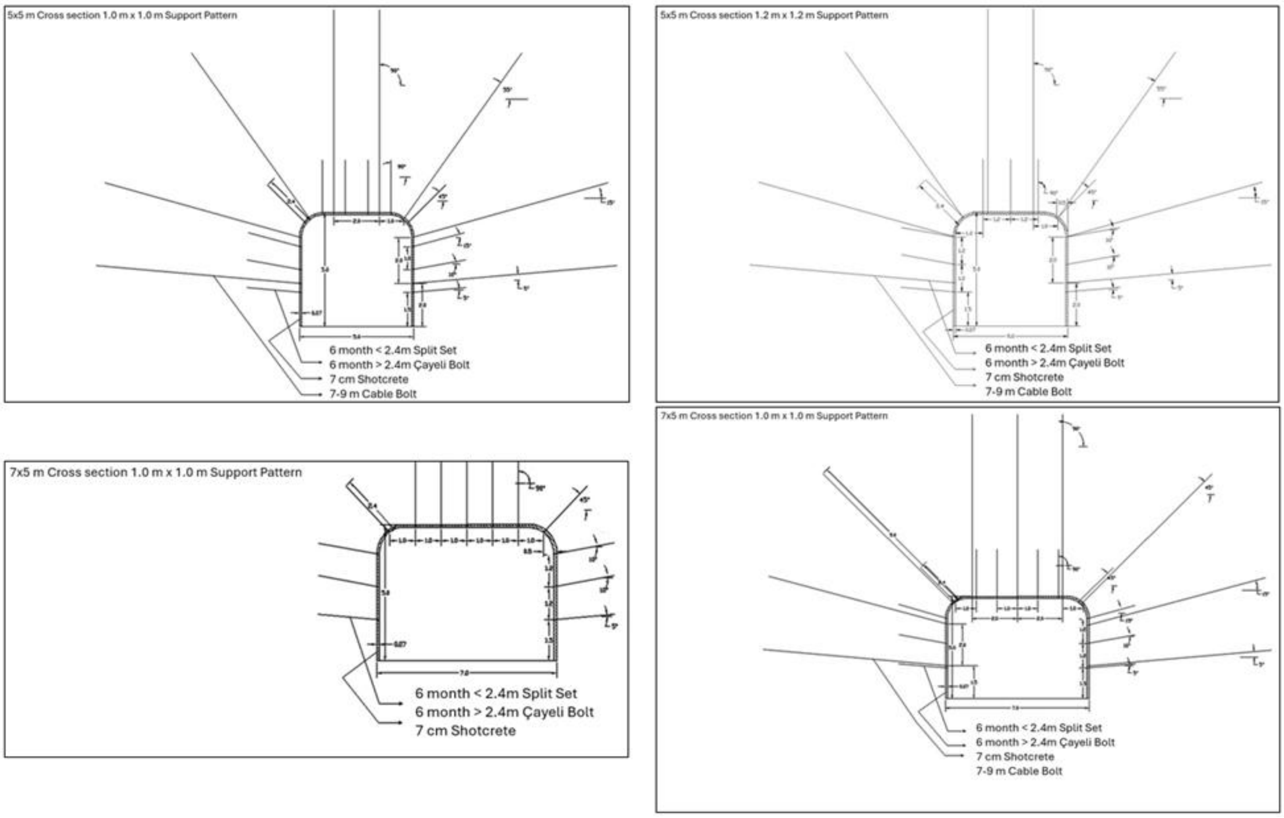
Shotcrete

The following specifications apply for each cubic metre of shotcrete:

- Cement : 420 kg (CEM II A-LL 42,5 R)
- W/C : 0.45 to 0.54 (moisture included)
- Aggregate : 1,570 kg (0-9 mm; %60 0-5mm - %40 5-9mm)
- Plasticiser : 5 kg (SIKA - Viscocrete 7177 HT - % 1.55 of cement weight)
- Hardening Acc, : 35 kg (MasterRoc -SA 160 - % 8.00 of cement weight)
- Silica Fume : 7 kg (MasterRoc -MS 685 - %1.59 of cement weight)
- Steel Fibre : 15 kg (if used 15kg sand weight will be reduced to accommodate steel fibre addition)
- Slump : 20 cm to 23 cm (at the plant exit)

The quality of the shotcrete is controlled and tested by taking cube samples from the ÇBI batch plant. The minimum required concrete strength is 35 MPa at the 28th day.

Figure 16.11 Typical rock reinforcement patterns (source: ÇBI 2025)



Ventilation raise support

Ventilation raises are being mined to service the South Orebody sublevels. These are typically 20 m long and mined to a 3.0 m diameter by long hole drilling and blasting. To maintain their integrity over the life of the mine, the raises are fully lined with steel segments (Figure 16.12).

16.5.5 Mining dilution and recovery losses

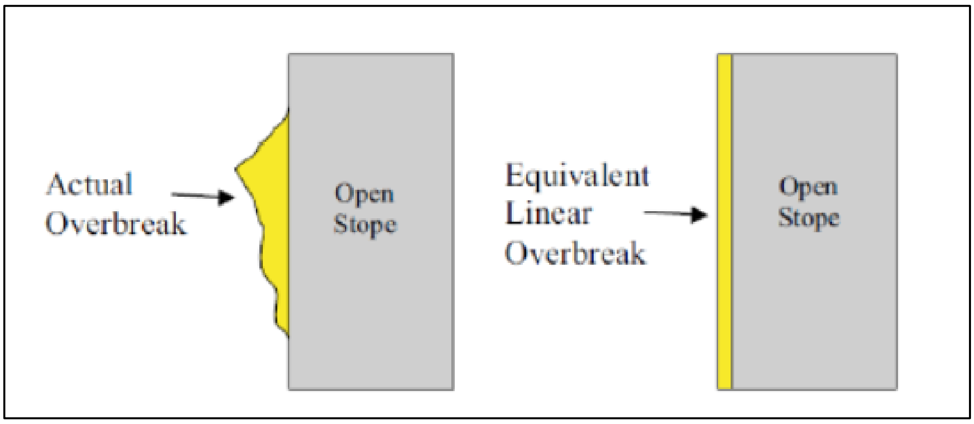
Drawing upon historical trends, a set of generic “unplanned” mining dilution and recovery loss factors are applied following the design of ore development and stope solids which take account of “planned” dilution. This sequence of design steps is described in the following commentary.

Cavity monitoring

A cavity monitoring system (CMS) has been used at ÇBI to assess the extent of unplanned dilution and ore loss, by comparing the as-mined voids against the design stope solids. The CMS records were reviewed by AMC Consultants in 2014 (AMC, 2014). The stope overbreak/equivalent linear overbreak slough (ELOS) was used to assess stope performance by comparing planned stope shapes to the CMS shapes as revealed by CMS. ELOS converts the true volumetric measurement into an average depth of slough over the entire stope surface. A simplified 2D example is shown in Figure 16.13 (AMC, 2014; after Oddie and Pascoe, 2005).

Figure 16.12 Mine ventilation raise, steel lining segments

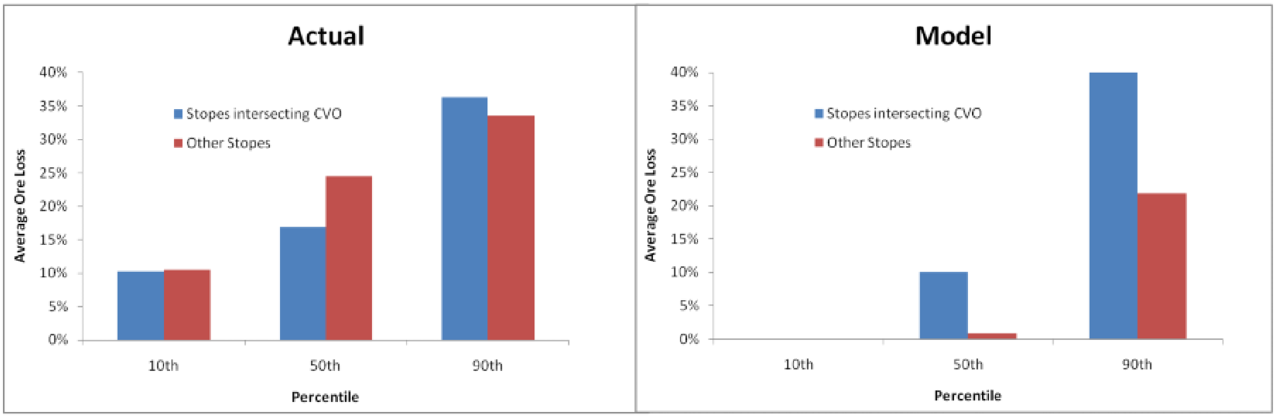
Figure 16.13 Unplanned mining dilution; equivalent linear overbreak (source: AMC, 2014)



Fifty one stopes were selected for ore loss assessment and comparison with the model; results can be summarised as follows:

- Actual data shows no correlation between the potential for ore loss and intersection with coarse grained veins of pyrite and chalcopyrite (chalcopyrite vein ore; CVO) in the clay altered rhyolite formation. This could be due to other contributing factors such as blasting and operational factors masking the geotechnical factors.
- Model result shows increased ore loss potential for stopes intersecting the CVO formation.

Figure 16.14 Actual overbreak vs modelled ore loss results comparison (source: AMC, 2014)



Planned and unplanned dilution

Planned dilution is incorporated through the deliberate inclusion of waste material in the design and layout of ore development and stoping. Such dilution also includes the incorporation of Inferred Mineral Resource blocks. The reason for these inclusions is to produce a practical mining geometry design, and in the case of a long hole stope design, to yield a reasonably contiguous series of production drill rings along the length of a stope crosscut or strike drive.

Depending on the extent of waste or assigned waste blocks, it may not be practical to separate or exclude them from the design. In which case, the NSR design approach dictates the viability of including all or some proportion of this planned dilution.

Unplanned dilution comes about through the unintended inclusion of waste during the process of ore development and stoping. Examples of this waste would be the inadvertent inclusion of backfill material when drawing ore from secondary and tertiary stopes, or otherwise from waste rock sloughing off the backs and walls of development and stope openings and then being mucked/transported with the ore.

Table 16-8 lists the overall unplanned dilution factors that are apparent after considering an historical design inventory of 670 stopes.

Table 16-8 Overall unplanned dilution factors

Stope configuration		Factor
Primary transverse stopes		
	incurring dilution from one end, footwall or hangingwall	1.79%
Secondary transverse stopes		
	incurring dilution from one end and sidewalls	7.34%
Tertiary/longitudinal stopes		
	incurring dilution from one end and sidewalls	7.92%
	Overall	**5.06%**

Unplanned ore losses

Deliberate and unintended ore losses are also factored into the design stoping inventory and these can be attributable to a number of factors relating to adverse ground conditions, sterilisation, under-breaks, etc. Table 16-9 lists the overall planned plus unplanned ore loss factors that are apparent after considering a design inventory of 670 stopes. The overall loss figure translates to 94.2% mining recovery.

Table 16-9 Overall unplanned ore loss factors

Stope configuration		Factor
Primary transverse stopes		
	incurring losses from one end, footwall or hangingwall	96.47%
Secondary transverse stopes		
	incurring losses from one end and sidewalls	92.35%
Tertiary/longitudinal stopes		
	incurring losses from one end and sidewalls	93.10%
	Overall	**94.22%**

16.5.6 Water inflow

In 2024 the total mine water pumped to surface was 68 kL/hour, on average. Process water from drilling and filled stope drainage was approximately 40% to 45%, with the balance therefore attributable to mine inflow.

16.5.7 Ventilation requirements

The ventilation requirements for the mine were calculated from the total utilised kW of underground diesel equipment. Table 16-10 lists the average ventilation requirements for equipment operating in the Main Orebody over the period 2022 to 2024. The utilisation factors were determined from ÇBI operational experience, whilst the ventilation airflow was calculated on the requirement of 1 kW/m^3/sec (which is equivalent to 3.6 m^3/min/kW).

The same utilisation factors and airflow requirement were adopted in estimating the ventilation requirements from 2025 (Table 16-11). The required LOM equipment numbers are as listed in Item 16.10.

There are currently 2 x 220 kW primary fans and an auxiliary 137 kW fan supplying air into the operating Main Orebody workings and the upper development headings of the South Orebody. Without a proposed new supplementary 400 kW fan, there would otherwise be a shortfall in supplying sufficient air for primary equipment operating in the South Orebody, when under full production.

An overview of the mine ventilation layout is provided in Item 16.6.4.

In those years when the Main Orebody and South Orebody are being mined at the same time, the ventilation network will likely become quite complicated, with airflow directions changing over time, along with the relative supply of fresh air as opposed to recirculated return air. The overall network will therefore feature a dynamic regulating arrangement of auxiliary ventilation fans and stopping/control devices.

There will be some future airflow supply flexibility (discussed in Item 16.6.4) if utilising the down-cast main shaft airway. Ultimately, however, the effective supply of air into the mine workings will require careful production plan sequencing to enable the effective use of trucks and LHDs operating on multiple sublevels.

Table 16-10 Ventilation requirements for the years 2022 to 2024

Equipment	Max No. Units #	Engine (kW)	Total (kW)	Utilisation (%)	Utilised (kW)	Vent. requirement (m^3/s)
Jumbos						
Atlas Copco 282 twin-boom	2	83	166	44.3	74	4.4
Sandvik DD420-40						
Production Drills						
Sandvik Solomatic DK430-7	1	110	110	28.2	31	1.9
Sandvik Solomatic DK431-7C	1	110	110	28.2	31	1.9
Cubex Megamatic ITH 5200	1	75	75	13.1	10	0.6
Bolters						
Robolter	2	113	226	55.6	126	7.5
Simba 127 (Cable Bolt Drilling)	1	56	56	1.7	1	0.1
LHDs (Kepçe)						
Toro 1400	3	252	756	56	423	25.4
Sandvik LH514	2	252	504	56	282	16.9
Sandvik LH517	1	252	252	56	141	8.5
Backhoe Loader (TakTak)						
Hidromek HMK 102S Alpha	3	75	225	61.9	139	8.4
Trucks (Kamyon)						
Wagner MT436B	10	207	2,070	48.6	1,006	60.4
Shotcrete Equipment						
Normet Spraymec 1050 WPC	3	145	435	62.2	271	16.2
Normet Transmixer (7.5 m3)	4	163	652	65.4	426	25.6
Utility Vehicles						
Paus Platforms	6	69	414	28.5	118	7.1
Paus ANFO Truck	2	69	138	28	39	2.3
Paus Crane	2	79	158	21.6	34	2.0
Paus Grouting Machine	1	69	69	0.6	0	0.0
Normet Veekmas Grader	1	90	90	11.3	10	0.6
Light Vehicles	21	106	2,226	47.2	1,051	63.0
TOTAL	**67**		**8,732**	**48.2**	**4,213**	**252.8**

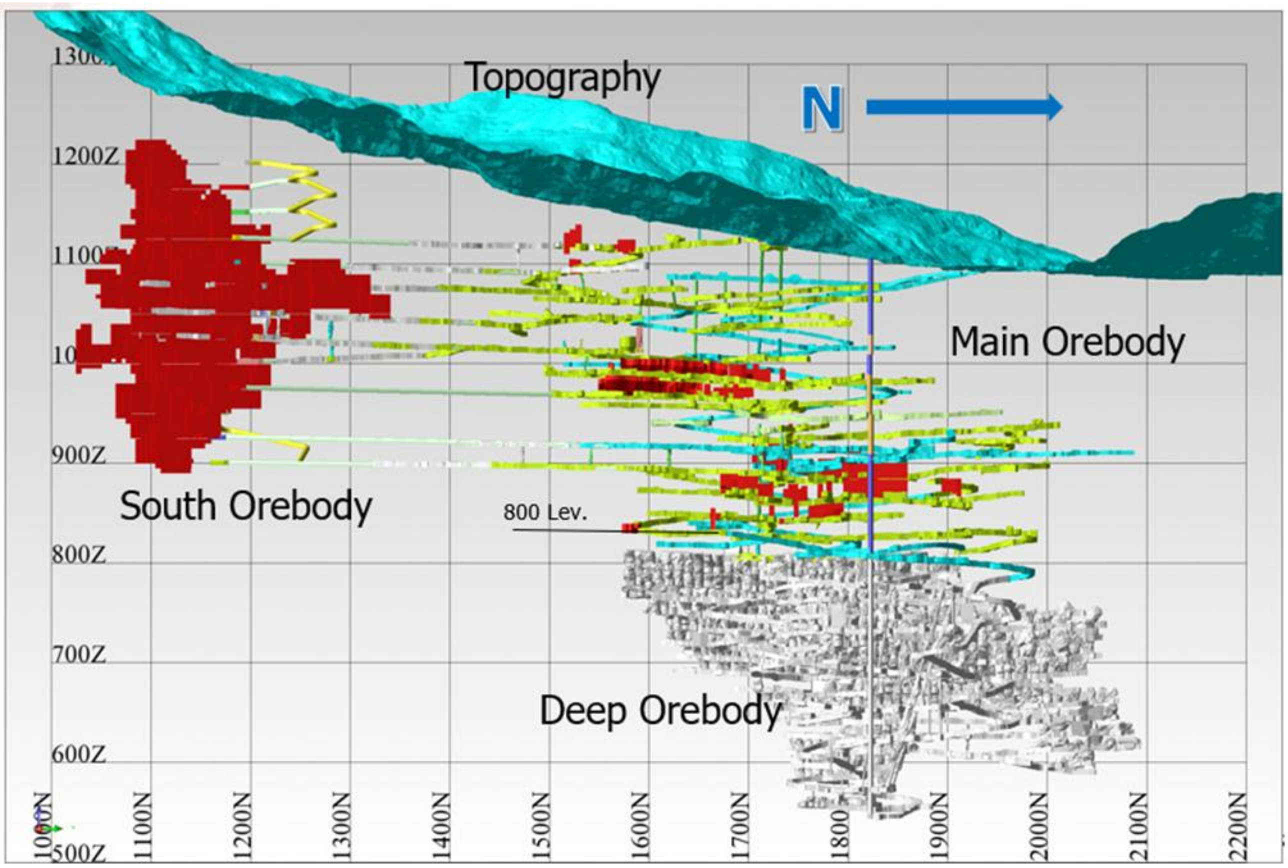
Table 16-11 Ventilation requirements for the years from 2025

Equipment	2025		2026		2027		2028		2029		2030		2031		2032		2033		2034		2035		2036		2037	
	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)	Units #	Vent. (m^3/s)
Jumbos																										
Atlas Copco 282 twin-boom	2	4.4	2	4.4	2	4.4	2	4.4	2	4.4	2	4.4	2	4.4	2	4.4	2	4.4	2	4.4	2	4.4	2	4.4	2	4.4
Sandvik DD420-40																										
Production Drills																										
Sandvik Solomatic DK430-7	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9
Sandvik Solomatic DK431-7C	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9	1	1.9
Cubex Megamatic ITH 5200	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6
Bolters																										
Robolter	3	11.3	3	11.3	3	11.3	2	7.5	2	7.5	2	7.5	2	7.5	2	7.5	2	7.5	2	7.5	2	7.5	2	7.5	2	7.5
Simba 127 (Cable Bolt Drilling)	1	0.1	1	0.1	1	0.1	1	0.1	1	0.1	1	0.1	1	0.1	1	0.1	1	0.1	1	0.1	1	0.1	1	0.1	1	0.1
LHDs (Kepçe)																										
Toro 1400	3	25.4	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9
Sandvik LH514	1	8.5	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9	2	16.9
Sandvik LH517	1	8.5	1	8.5	1	8.5	1	8.5	1	8.5	1	8.5	1	8.5	1	8.5	1	8.5	1	8.5	1	8.5	1	8.5	1	8.5
Backhoe Loader (TakTak)																										
Hidromek HMK 102S Alpha	5	13.9	4	11.1	4	11.1	4	11.1	4	11.1	4	11.1	4	11.1	4	11.1	4	11.1	4	11.1	4	11.1	4	11.1	4	11.1
Trucks (Kamyon)																										
Wagner MT436B	10	60.4	10	60.4	10	60.4	10	60.4	10	60.4	10	60.4	10	60.4	10	60.4	10	60.4	10	60.4	10	60.4	10	60.4	10	60.4
Shotcrete Equipment																										
Normet Spraymec 1050 WPC	3	16.2	3	16.2	3	16.2	3	16.2	3	16.2	3	16.2	3	16.2	3	16.2	3	16.2	3	16.2	3	16.2	3	16.2	3	16.2
Normet Transmixer (7.5 m3)	4	25.6	4	25.6	4	25.6	4	25.6	4	25.6	4	25.6	4	25.6	4	25.6	4	25.6	4	25.6	4	25.6	4	25.6	4	25.6
Utility Vehicles																										
Paus Platforms	6	7.1	6	7.1	6	7.1	6	7.1	6	7.1	6	7.1	6	7.1	6	7.1	6	7.1	6	7.1	6	7.1	6	7.1	6	7.1
Paus ANFO Truck	2	2.3	2	2.3	2	2.3	2	2.3	2	2.3	2	2.3	2	2.3	2	2.3	2	2.3	2	2.3	2	2.3	2	2.3	2	2.3
Paus Crane	2	2.0	2	2.0	2	2.0	2	2.0	2	2.0	2	2.0	2	2.0	2	2.0	2	2.0	2	2.0	2	2.0	2	2.0	2	2.0
Paus Grouting Machine	1	0.0	1	0.0	1	0.0	1	0.0	1	0.0	1	0.0	1	0.0	1	0.0	1	0.0	1	0.0	1	0.0	1	0.0	1	0.0
Normet Veekmas Grader	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6	1	0.6
Light Vehicles		0.0		0.0		0.0		0.0		0.0		0.0		0.0		0.0		0.0		0.0		0.0		0.0		0.0
TOTAL	**48**	**190.6**	**47**	**187.8**	**47**	**187.8**	**46**	**184.1**	**46**	**184.1**	**46**	**184.1**	**46**	**184.1**	**46**	**184.1**	**46**	**184.1**	**46**	**184.1**	**46**	**184.1**	**46**	**184.1**	**46**	**184.1**

16.6 Underground mine layout

Figure 16.15 is a schematic view showing the relationship between the Main and Deep orebodies, which are separated by the Scissor Fault. In the wider horizons, the Main Orebody has a steeper dip than the abandoned lower horizons of the Deep Orebody. The South Orebody, which is situated 300 m beyond the Main Orebody footwall, is not intersected by the Scissor Fault.

Figure 16.15 Longitudinal schematic (3D) view through the Main, Deep and South Orebodies



16.6.1 Mine access design

Underground mine layout

The original mine hoisting shaft was a 5.5 m diameter opening, fully concrete lined and operational to a depth of about 550 m. It was completed in 2006 and remained in use for hoisting until 2021, after which time it was abandoned and backfilled (i.e., with the ladderway remaining open).

Three decline access portals are shown on Figure 16.16:

- the northern-most portal is the entrance to the main decline
- the middle (south) portal is the entrance to the upper Main Orebody ramp, connecting to lateral access development across to the upper South Orebody
- the southern-most portal (the Stockpile portal) will be the primary access entrance, and exhaust airway, for the South Orebody[10]

[10] In progress, with surface breakthrough imminent

Main decline access

The main access to the Main Orebody for personnel, equipment, and materials delivery is now provided by a 12% to 15% gradient decline, of 25 m² cross sectional area. From a portal located at 1096 mRL (the northern most portal shown in Figure 16.16), the decline spirals across the Main Orebody hangingwall down to the 800 mRL sublevel, from where it crosses to the footwall side. Access below 800 mRL is no longer possible.

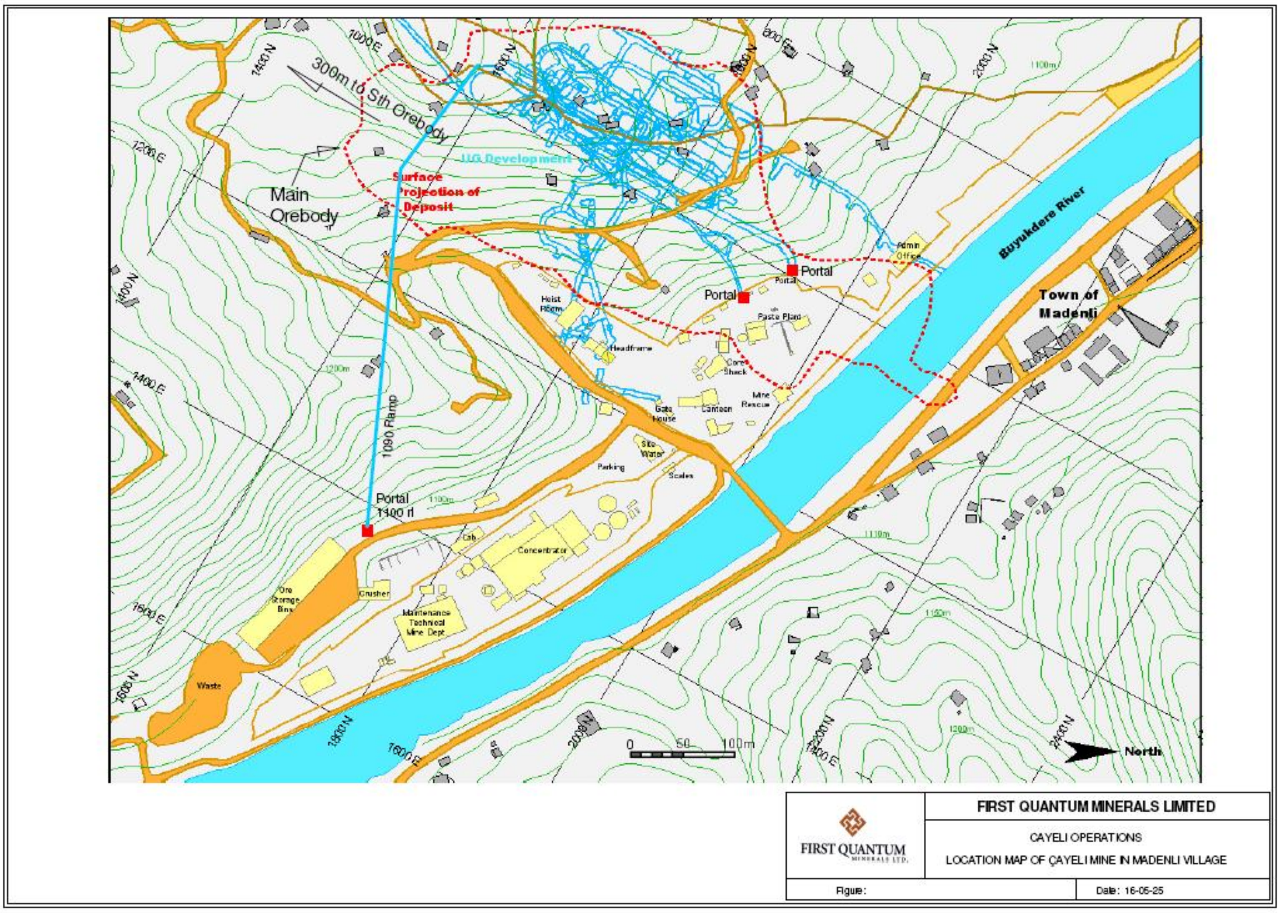
Access to the upper South Orebody

For the South Orebody layout, all access crosscuts and bypass ramps are located on the north side of the orebody. These development openings connect to a ramp servicing the upper South Orebody, from a portal referred to as the South portal (the centrally positioned portal shown in Figure 16.16).

Access to the lower South Orebody

Figure 16.16 also shows the location of the new access portal at 1100 mRL (*aka* 1120 mRL or the stockpile portal). When completed (imminent), the ramp from this portal will serve as the primary access to the lower South Orebody, and as the ventilation return airway to surface.

Figure 16.16 Location of mine access portals



16.6.2 Level development design

Commencing from sublevel development access leading away from the hangingwall decline, Main Orebody sublevels have been typically developed within the orebody, along strike. Existing openings will be used for mining remnant stopes on the 1110 mRL down to the 820 mRL sublevels.

In the wide Main Orebody accessed from the hangingwall decline, the sublevels are already developed.

The South Orebody sublevel development emanates from the access development driven across from the Main Orebody.

16.6.3 Stope design

Item 15.3 describes the NSR approach to the design of stoping blocks and the ore extraction limits.

In conjunction with this approach, there are several practical design guidelines adopted as follows:

- the minimum orebody dip for long hole stope design is 50°
- longitudinal stopes are applicable where the ore width is less than 15 m
- primary crosscut development in transverse stopes will be at 7 m width (6 m in the South Orebody) and 14 m centrelines
- these openings will be 5 m high and the height between sublevels will be 25 m (in the South Orebody)
- NSR cut-off grade criteria are applied to assess the economic viability of including “waste” production rings within any stoping inventory

16.6.4 Ventilation layout and airflows

Figure 16.17 shows a simplified ventilation diagram illustrating the current Main Orebody and South Orebody development, with airflow directions and quantities annotated.

The ventilation requirements of the mine are calculated based on the total utilised diesel engine power (in kW) of operating underground equipment. The requirements are outlined in Item 16.5.7.

Main Orebody

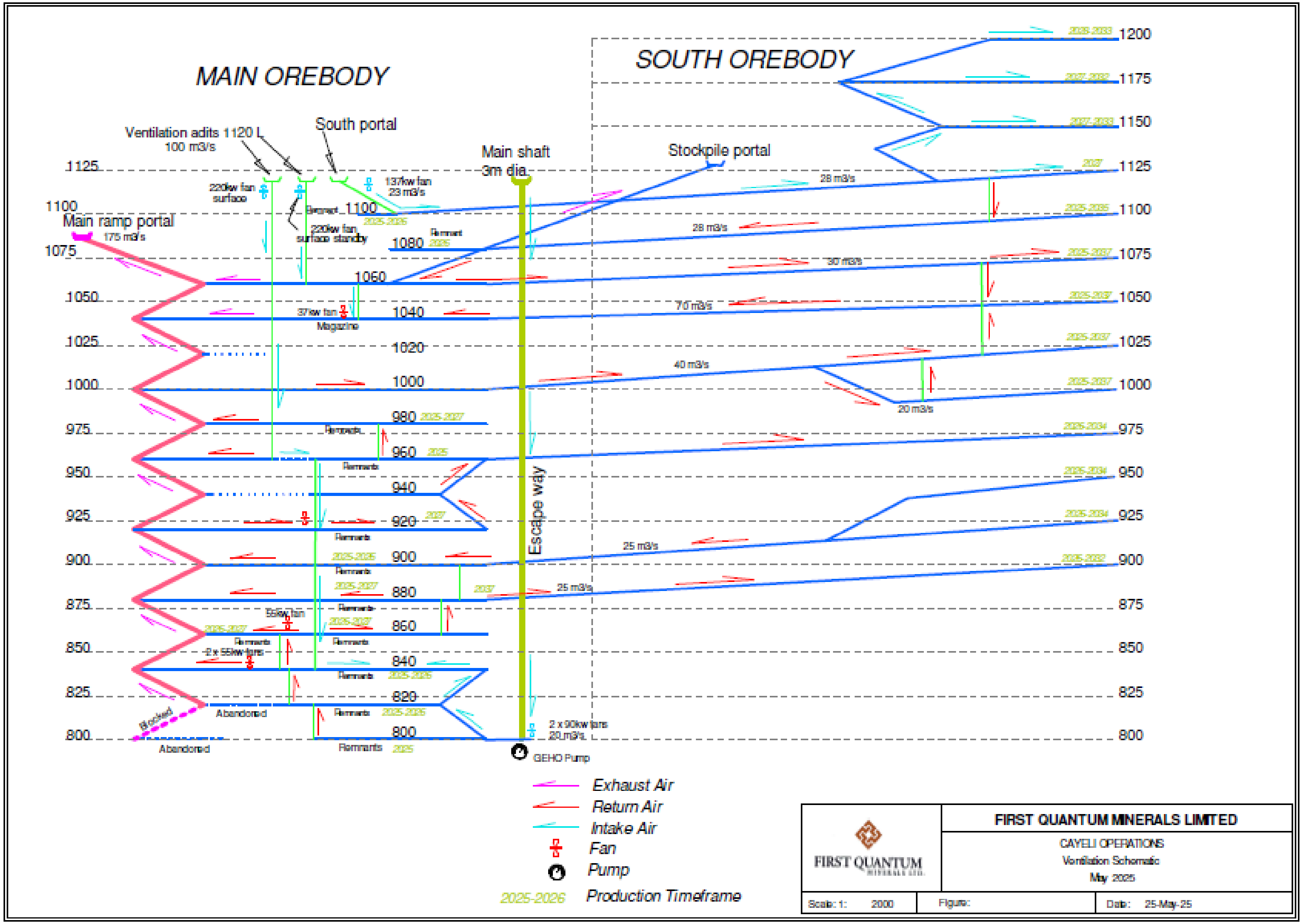
The Main Orebody workings are currently ventilated by means of two down-cast raises each of 3 m diameter. One of these raises (vent. raise #1) extends to the 840 mRL sublevel and the other (vent. raise #2) extends to the 1060 mRL sublevel. There are two 220 kW fans installed on the 1120 mRL sublevel, one at each raise collar. One of these fans operates at maximum capacity, whilst the other one is in reserve. The maximum capacity of these fans is 100 m^3/sec at 1,240 Pa.

From the base of ventilation raise #2 at 1060 mRL, air is supplied via a 37 kW auxiliary fan, connecting lateral and vertical development to the explosives magazine at the 1020 mRL sublevel. The airflow to the 1020 mRL sublevel is approximately 13 m^3/sec.

Apart from the down-cast ventilation supplied by the two 220 kW fans, there is an additional primary air intake of 20 m^3/s through the old main shaft ladderway. This 3.0 m diameter airway extends from surface to the 800 mRL sublevel, at which location there are two 90 kW fans installed.

The exhaust airway is the main ramp, where the airflow has been measured at 175 m^3/sec.

Figure 16.17 Ventilation network diagram; Main Orebody and South Orebody during development



Commencing from the top of the mine, the airflows into the Main Orebody sublevels are generally as follows:

1. Fresh air intake to the 1110 mRL and 1080/1090 mRL sublevel remnant stopes is directly via the South portal ramp.
2. Return air is recirculated through the 980 mRL and 960 mRL sublevels, emanating from an auxiliary fan on the 920 mRL sublevel, directing return air drawn from the decline and passing through a footwall ramp between the 920 mRL and 980 mRL sublevels.
3. Remnant stopes on the 900 mRL sublevel are ventilated by return air from the South Orebody 950 mRL and 925 mRL sublevels.
4. Remnant stopes on the 880 mRL sublevel are ventilated by return air from a footwall raise emanating on the 860 mRL sublevel.
5. Remnant stopes on the 800 mRL to 840 mRL sublevels are fresh air ventilated via a ramp ascending from the base of the old shaft on the 800 mRL sublevel.

South Orebody

Commencing from the top, the airflows into the South Orebody development sublevels are generally as follows:

1. Fresh air intake to the 1125 mRL and up to the 1200 mRL sublevel is directly from the South portal ramp, and via the 1100 mRL access development from the Main Orebody. A 137 kW fan is positioned underground, just beyond the South portal, and supplies 23 m^3/sec of airflow along the 1100 mRL access development.
2. All successively lower development sublevels are ventilated by return air either recirculating within the South Orebody, or via access connections from the Main Orebody sublevels.

The existing 220 kW fans, and the auxiliary 137 kW fan, cannot provide enough air to meet future requirements when the South Orebody comes into full production. Consequently, a new 400 kW primary fan is to be installed near-to and replacing the 137 kW fan. The new fan will force-ventilate the South Orebody workings, operating in parallel with the 220 kW fans.

As the South Orebody stoping proceeds below the 1125 mRL sublevel, fresh air from the 2 x 220 kW fan network will be drawn across from the Main Orebody 880 mRL sublevel and along the 900 mRL South Orebody sublevel. Fresh air into the future South Orebody workings could also be drawn from the down-cast ventilation shaft at the Main Orebody 960 mRL / South Orebody 975 mRL sublevels.

16.6.5 Underground power supply

The mine's main electrical substation is connected by a single 31.5 kV- 2 x 15 MVA rated underground power line from the *Çoruh Elektirik Dağıtım A.Ş.* substation north of the town of Madenli. This substation at Madenli is equipped with two 12 MVA transformers.

From one of these transformers, power is delivered to the underground workings by 6.3 kV lines and via:

- the shaft and the 940 mRL sublevel, and then through boreholes to 500 kVA substations located on the 880 mRL and 840 mRL sublevels of the Main Orebody
- the shaft and then to 500 kVA substations located on the 800 mRL sublevel of the Main Orebody
- the main decline, and connecting to 500 kVA substations located at:

- the 1060 mRL, 1040 mRL, 960 mRL and 940 mRL sublevels of the Main Orebody, and
- the 1075 mRL and 1060 mRL sublevels of the South Orebody

The feed from the substations to the electrical equipment is reduced to 400 V. A diagrammatic layout of the underground power distribution is shown in Figure 16.18.

16.6.6 Services

Four compressors located on surface provide compressed air for the mine. Each unit is powered by a 160 kW motor, delivering 450 L/sec of compressed air at 10 bar (1,000 kPa) pressure. Several pipelines, 100 to 200 mm in diameter, distribute compressed air to different parts of the mine. One of the lines is installed in the shaft, a second one in the main decline and a third one in a ventilation shaft.

Fresh water is supplied to the mine from a series of bores located on the bank of the Büyükdere River. The bore pumps have a supply capacity of 800 m^3/h, delivering water into the mine via pipelines installed in the shaft and in the main decline.

The mine uses wet shotcrete for ground support and this is prepared on surface at a batch plant located close to the decline portal. The batched shotcrete is transported in mixer trucks into the underground mine via the decline.

16.6.7 Mine dewatering

The current main dewatering pumps are installed on the 800 mRL sublevel. There are two main dewatering pumps. One of them is a 132 kW Geho model ZPM 700 pump which can handle 65 m^3/h of dirty water at an operating pressure of 42 bar (4,200 kPa). The other main pump is a Geho model ZMP 800 which is used as an emergency spare.

The total mine water pumped to surface is 68 m^3/h on average. Process water (i.e., from drilling and stope drainage) accounts for approximately 40% to 45% of the total mine water pumped to surface.

Once the lower Main Orebody sublevel remnants are mined, the dewatering pumps will be relocated to the 880 mRL sublevel.

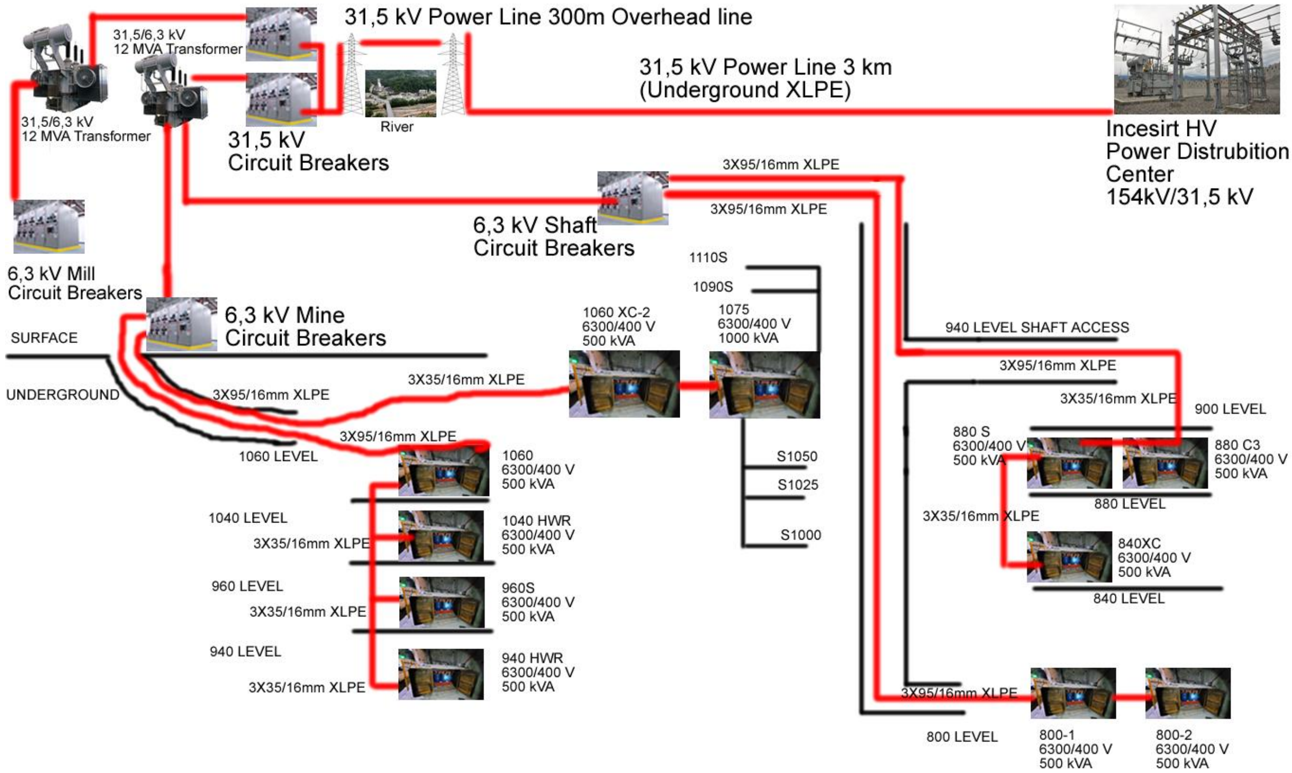
16.6.8 Paste fill reticulation

There are two paste fill distribution routes into the mine:

- via a steel cased borehole extending from surface to the 900 mRL sublevel of the Main Orebody
  - from there, another borehole extends to the 800 mRL sublevel
- via pipelines installed in the main decline and extending to the 1060 m sublevel of the Main Orebody
  - from there, several steel cased boreholes descend to various sublevels, down to the 820 mRL

Lateral pipelines extending along the sublevels are typically 4 inch (10.2 cm) diameter poly pipes.

Figure 16.18 Underground power distribution diagram



16.6.9 Secondary egress

Secondary egress from the mine is primarily via a retained ladderway in the backfilled hoisting shaft to 800 mRL (in fresh air drawn from surface by two primary fans). Internal ladderways connect various sublevels of both the Main and South Orebodies.

Ten refuge chambers are provided throughout the mine, of a design similar to that shown in Figure 16.19. One such chamber is used for training on surface. Several of these chambers are located along the access development leading across to the South Orebody. It is intended that as each of the Main Orebody remnant mining sublevels is completed, otherwise redundant refuge chambers will be relocated across to the South Orebody.

The communication system comprises telephone and radio installations within each chamber. In 2019, a camera system was installed in each chamber in order to monitor the welfare of refugees in any emergency situation.

16.6.10 Mining workshop

The original underground workshop, located on the 800 mRL sublevel of the Main Orebody, has been closed as it is no longer needed. Most of the major repairs to mining equipment are now carried out in a surface workshop. There is a smaller surface workshop used for light vehicle repairs and maintenance. Both workshops are located in the same building. These workshops are equipped to handle larger maintenance jobs and can accommodate several pieces of mobile equipment at the same time. The mine offices are located in the same building.

Figure 16.19 MineArc refuge chamber

16.6.11 Fuel storage

There is a diesel fuel station located near to the ramp portal on surface. An underground fuel station at the 800 mRL sublevel of the Main Orebody is no longer serviceable.

16.6.12 Explosives storage

An explosives magazine is located on the 1020 mRL sublevel of the Main Orebody. It comprises a 175 m long drift with 40 tonnes of licensed explosive storage capacity. It is intended that this magazine will be retained and used during mining of the South Orebody.

16.7 Mining production schedule

The following commentary summarises the annual mine development and stoping production schedule, and the LOM mining sequence.

16.7.1 Development metres

Table 16-12 lists the Main Orebody operating waste development metres, by sublevel. In this instance, there is no capitalised waste development required for access into the remnant stoping areas.

Table 16-12 Main Orebody; annual schedule of waste development metres

Sublevel	Category	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
1110 mRL	Capital	m	0												
1090 mRL		m	0												
980 mRL		m	0												
960 mRL		m	0												
920 mRL		m	0												
900 mRL		m	0												
880 mRL		m	0												
860 mRL		m	0												
840 mRL		m	0												
820 mRL		m	0												
	subtotal	m	0	0	0	0	0	0	0	0	0	0	0	0	0
1110 mRL	Operating	m	125	47		72									6
1090 mRL		m	18												
980 mRL		m	150										49	101	
960 mRL		m	55											55	
920 mRL		m	0												
900 mRL		m	28												28
880 mRL		m	61											11	50
860 mRL		m	234	74		160									
840 mRL		m	45		45										
820 mRL		m	28	28											
	subtotal	m	744	167	45	232	0	0	0	0	0	0	49	167	84
	Total	m	744	167	45	232	0	0	0	0	0	0	49	167	84

Table 16-13 lists the South Orebody capital and operating waste development metres, by sublevel.

Table 16-14 lists the Main Orebody ore development metres, whilst Table 16-15 lists the South Orebody ore development metres. The listed South Orebody capital development includes the access development headings leading across from the Main Orebody. The large number of headings listed in Table 16-15 partially reflects, that on some sublevels, there are several successively developed faces within individual headings.

Table 16-13 South Orebody; annual schedule of waste development metres

Sublevel	Category	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
1200 mRL	Capital	m	120			120									
1175 mRL		m	180			180									
1150 mRL		m	840	90	540	210									
1125 mRL		m	225	105	120										
1100 mRL		m	150	150											
1075 mRL		m	0												
1050 mRL		m	0												
1025 mRL		m	0												
1000 mRL		m	0												
975 mRL		m	375	180	195										
950 mRL		m	60		60										
925 mRL		m	360	285	75										
900 mRL		m	105	105											
	subtotal	m	2,415	915	990	510	0	0	0	0	0	0	0	0	0
1200 mRL	Operating	m	130				30	20	20	30	20	10			
1175 mRL		m	186			15	60	46		5	60				
1150 mRL		m	110			35				15	40	20			
1125 mRL		m	632		85	32	65	16	17	105	97	85	125	5	
1100 mRL		m	702		187	10	110	110	50	120	60	10	10	35	
1075 mRL		m	1,043		35	190	167	105	100	200	95	15	76	60	
1050 mRL		m	860		135	155	280	70	50	35	25	40	45		25
1025 mRL		m	332		52	85	40	30	30		30			15	50
1000 mRL		m	545		145	145	25	25	80	30	25			70	
975 mRL		m	540		95	15	90	80	100	65	95				
950 mRL		m	375		35	90		15		60	30	20	105	20	
925 mRL		m	90		25	25		20	10	10					
900 mRL		m	145		90	10		20	10		15				
	subtotal	m	5,690	0	884	807	867	557	467	675	592	200	361	205	75
	Total	m	8,105	915	1,874	1,317	867	557	467	675	592	200	361	205	75

Table 16-14 Main Orebody; annual schedule of ore development metres

Sublevel	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
1110 mRL														
Dev't metres	**m**	**159**	66		76									17
# headings	**#**	**6**	2		3									1
1090 mRL														
Dev't metres	**m**	**24**	24											
# headings	**#**	**2**	2											
980 mRL														
Dev't metres	**m**	**25**											25	
# headings	**#**	**1**											1	
960 mRL														
Dev't metres	**m**	**152**	53	99										
# headings	**#**	**5**	2	3										
920 mRL														
Dev't metres	**m**	**0**												
# headings	**#**	**0**												
900 mRL														
Dev't metres	**m**	**63**		48										15
# headings	**#**	**5**		4										1
880 mRL														
Dev't metres	**m**	**116**											79	38
# headings	**#**	**6**											3	3
860 mRL														
Dev't metres	**m**	**71**	11	24	36									
# headings	**#**	**7**	1	1	5									
840 mRL														
Dev't metres	**m**	**39**		39										
# headings	**#**	**3**		3										
820 mRL														
Dev't metres	**m**	**16**	16											
# headings	**#**	**2**	2											
Total														
Dev't metres	**m**	**666**	**170**	**210**	**113**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**104**	**70**
# headings	**#**	**37**	**9**	**11**	**8**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**4**	**5**

Table 16-15 South Orebody; annual schedule of ore development metres

Sublevel	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
1200 mRL														
Dev't metres	**m**	**545**				167	111	108	91	40	29			
# headings	**#**	**20**				5	4	4	3	3	1			
1175 mRL														
Dev't metres	**m**	**933**			5	194	148	208	161	129	61	28		
# headings	**#**	**48**			1	10	8	10	5	7	3	4		
1150 mRL														
Dev't metres	**m**	**1,169**			231	92	105	178	89	83	81	193	116	
# headings	**#**	**50**			9	4	4	7	3	4	4	11	4	
1125 mRL														
Dev't metres	**m**	**1,442**		279	113	199	143	157	146	81	42	131	150	
# headings	**#**	**58**		13	5	7	5	6	6	3	1	7	5	
1100 mRL														
Dev't metres	**m**	**1,113**		284	90	147	22	39	162	144	107	44	14	59
# headings	**#**	**54**		15	3	7	1	2	8	6	6	4	1	1
1075 mRL														
Dev't metres	**m**	**1,830**	256	109	252	233	75		250	192	119	82	73	189
# headings	**#**	**80**	11	5	9	11	4		12	8	7	3	5	5
1050 mRL														
Dev't metres	**m**	**1,108**	60	154	125	147	109		234	158		18	18	86
# headings	**#**	**50**	4	8	5	6	5		10	8		1	1	2
1025 mRL														
Dev't metres	**m**	**1,386**	109	254	188	209	121	20	248	139	40		23	36
# headings	**#**	**60**	5	11	9	8	5	1	9	7	2		1	2
1000 mRL														
Dev't metres	**m**	**1,203**	143	153	138	168	80	156	173	129	27		36	
# headings	**#**	**53**	7	7	7	7	3	7	6	4	2		3	
975 mRL														
Dev't metres	**m**	**990**		54	28	72	165	137	137	112	47	206	32	
# headings	**#**	**45**		3	1	3	7	7	7	5	3	8	1	
950 mRL														
Dev't metres	**m**	**692**		127	37	12	145	68	40	86	27	106	45	
# headings	**#**	**35**		6	2	1	6	3	2	5	1	7	2	
925 mRL														
Dev't metres	**m**	**652**		170	40	19	249	51	32	26	21	44		
# headings	**#**	**27**		7	2	1	8	2	2	2	1	2		
900 mRL														
Dev't metres	**m**	**282**		97	34		73	38	40					
# headings	**#**	**13**		5	1		3	2	2					
Total														
Dev't metres	**m**	**13,347**	**567**	**1,683**	**1,280**	**1,659**	**1,546**	**1,160**	**1,802**	**1,320**	**600**	**853**	**507**	**370**
# headings	**#**	**593**	**27**	**80**	**54**	**70**	**63**	**51**	**75**	**62**	**31**	**47**	**23**	**10**

16.7.2 Development in ore

The previous tables summarised the annual development in terms of metres advanced in waste and ore headings.

Table 16-16 and Table 16-17, summarise respectively, the annual ore development tonnage (in addition to NSR, Cu grade and Zn grade) for the Main Orebody and the South Orebody. The upper part of each table lists the designed tonnage and grades inclusive of planned dilution. The adjustments at the bottom of each table reflect the application of the generic unplanned dilution and mining recovery (loss) factors.

The respective information in these tables is also shown graphically in Figure 16.20 and Figure 16.21.

Table 16-16 Main Orebody; annual schedule of development ore tonnes and grades

Sublevel	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
1110 mRL														
Pl. diluted tonnes	t	**15,871**	6,590		7,618									1,664
Av. NSR	$/t ore	**131**	178		73									214
Av. Cu	%	**1.98**	1.99		1.41									4.52
Av. Zn	%	**4.27**	3.92		4.10									6.46
1090 mRL														
Pl. diluted tonnes	t	**2,416**	2,416											
Av. NSR	$/t ore	**267**	267											
Av. Cu	%	**2.91**	2.91											
Av. Zn	%	**5.18**	5.18											
980 mRL														
Pl. diluted tonnes	t	**2,523**											2,523	
Av. NSR	$/t ore	**379**											379	
Av. Cu	%	**4.97**											4.97	
Av. Zn	%	**1.07**											1.07	
960 mRL														
Pl. diluted tonnes	t	**15,235**	5,293	9,942										
Av. NSR	$/t ore	**333**	335	333										
Av. Cu	%	**3.67**	4.32	3.32										
Av. Zn	%	**2.36**	2.99	2.03										
920 mRL														
Pl. diluted tonnes	t	**0**												
Av. NSR	$/t ore	**0**												
Av. Cu	%	**0.00**												
Av. Zn	%	**0.00**												
900 mRL														
Pl. diluted tonnes	t	**6,324**		4,793										1,531
Av. NSR	$/t ore	**212**		137										447
Av. Cu	%	**2.95**		2.01										5.88
Av. Zn	%	**0.63**		0.13										2.19
880 mRL														
Pl. diluted tonnes	t	**11,642**											7,853	3,790
Av. NSR	$/t ore	**177**											192	145
Av. Cu	%	**2.32**											2.52	1.90
Av. Zn	%	**0.17**											0.06	0.40
860 mRL														
Pl. diluted tonnes	t	**7,090**	1,061	2,396	3,633									
Av. NSR	$/t ore	**101**	103	92	106									
Av. Cu	%	**1.56**	1.35	1.21	1.86									
Av. Zn	%	**0.09**	0.06	0.02	0.14									
840 mRL														
Pl. diluted tonnes	t	**3,908**		3,908										
Av. NSR	$/t ore	**153**		153										
Av. Cu	%	**2.00**		2.00										
Av. Zn	%	**0.04**		0.04										
820 mRL														
Pl. diluted tonnes	t	**1,634**	1,634											
Av. NSR	$/t ore	**101**	101											
Av. Cu	%	**1.33**	1.33											
Av. Zn	%	**0.01**	0.01											
Total														
Pl. diluted tonnes	**t**	**66,643**	**16,993**	**21,039**	**11,251**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**10,376**	**6,984**
Av. NSR	**$/t ore**	**205**	**227**	**227**	**84**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**238**	**228**
Av. Cu	**%**	**2.60**	**2.74**	**2.54**	**1.55**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**3.11**	**3.40**
Av. Zn	**%**	**1.89**	**3.19**	**1.00**	**2.82**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.31**	**2.23**
Unpl. diluted tonnes	**t**	**70,015**	**17,853**	**22,103**	**11,820**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**10,901**	**7,337**
Av. NSR	**$/t ore**	**195**	**216**	**216**	**80**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**226**	**217**
Av. Cu	**%**	**2.48**	**2.61**	**2.41**	**1.48**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**2.96**	**3.24**
Av. Zn	**%**	**1.80**	**3.04**	**0.95**	**2.69**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.29**	**2.13**
Recovered tonnes	**t**	**65,968**	**16,821**	**20,826**	**11,137**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**10,271**	**6,913**
Av. NSR	**$/t ore**	**195**	**216**	**216**	**80**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**226**	**217**
Av. Cu	**%**	**2.48**	**2.61**	**2.41**	**1.48**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**2.96**	**3.24**
Av. Zn	**%**	**1.80**	**3.04**	**0.95**	**2.69**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.29**	**2.13**

Table 16-17 South Orebody; annual schedule of development ore tonnes and grades

Sublevel	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
1200 mRL														
Pl. diluted tonnes	t	**54,536**				16,665	11,120	10,794	9,110	3,985	2,862			
Av. NSR	$/t ore	**329**				284	275	333	389	349	560			
Av. Cu	%	**4.21**				3.61	3.51	4.20	4.94	4.76	7.35			
Av. Zn	%	**0.72**				0.26	0.70	0.25	0.31	5.19	0.26			
1175 mRL														
Pl. diluted tonnes	t	**93,348**			490	19,441	14,809	20,811	16,102	12,863	6,078	2,755		
Av. NSR	$/t ore	**193**			312	190	207	203	163	186	253	106		
Av. Cu	%	**2.06**			4.22	1.64	2.09	2.32	1.65	2.31	3.20	1.34		
Av. Zn	%	**5.25**			5.02	7.35	5.75	4.50	5.04	4.78	3.81	0.12		
1150 mRL														
Pl. diluted tonnes	t	**116,947**			23,135	9,221	10,538	17,826	8,887	8,338	8,070	19,320	11,612	
Av. NSR	$/t ore	**130**			145	112	147	130	137	113	137	115	123	
Av. Cu	%	**0.42**			0.47	0.23	0.49	0.54	0.50	0.46	0.71	0.23	0.26	
Av. Zn	%	**8.97**			10.03	8.43	10.10	8.26	9.54	7.27	8.06	8.72	9.24	
1125 mRL														
Pl. diluted tonnes	t	**144,233**		27,874	11,297	19,940	14,323	15,732	14,581	8,140	4,155	13,145	15,047	
Av. NSR	$/t ore	**124**		141	117	122	141	133	130	119	112	97	103	
Av. Cu	%	**0.47**		0.64	0.47	0.30	0.38	0.46	0.50	0.69	0.18	0.36	0.46	
Av. Zn	%	**8.05**		8.35	7.19	8.77	10.30	8.75	8.30	5.88	8.65	6.50	6.46	
1100 mRL														
Pl. diluted tonnes	t	**111,258**		28,440	8,958	14,739	2,180	3,913	16,189	14,408	10,722	4,442	1,407	5,859
Av. NSR	$/t ore	**124**		149	143	134	83	169	107	128	89	91	92	77
Av. Cu	%	**1.37**		1.64	1.60	1.25	0.12	2.13	1.24	1.53	0.95	1.00	1.15	0.98
Av. Zn	%	**1.72**		1.63	2.43	3.22	6.49	0.01	1.06	0.95	2.42	1.06	0.01	0.01
1075 mRL														
Pl. diluted tonnes	t	**182,955**	25,568	10,927	25,152	23,328	7,506		25,009	19,223	11,895	8,177	7,267	18,903
Av. NSR	$/t ore	**122**	127	118	104	115	141		148	159	112	116	97	86
Av. Cu	%	**1.32**	1.18	1.50	1.14	0.98	1.61		1.81	1.99	0.99	0.84	1.25	1.06
Av. Zn	%	**1.62**	3.22	0.02	1.33	3.64	1.14		0.36	0.01	3.23	4.59	0.23	0.01
1050 mRL														
Pl. diluted tonnes	t	**110,809**	5,959	15,410	12,488	14,660	10,881		23,383	15,773		1,819	1,802	8,634
Av. NSR	$/t ore	**119**	81	121	100	133	135		126	127		77	104	100
Av. Cu	%	**1.48**	0.82	1.52	1.30	1.62	1.67		1.58	1.68		0.82	1.30	1.25
Av. Zn	%	**0.17**	1.73	0.05	0.04	0.41	0.05		0.02	0.02		0.01	0.01	0.01
1025 mRL														
Pl. diluted tonnes	t	**138,544**	10,868	25,369	18,786	20,801	12,073	2,005	24,805	13,912	4,026		2,315	3,585
Av. NSR	$/t ore	**170**	163	171	137	197	203	79	185	196	80		116	77
Av. Cu	%	**2.10**	2.02	2.13	1.70	2.44	2.33	1.00	2.31	2.45	1.11		1.45	0.92
Av. Zn	%	**0.06**	0.03	0.12	0.11	0.03	0.03	0.08	0.02	0.03	0.03		0.01	0.02
1000 mRL														
Pl. diluted tonnes	t	**120,333**	14,296	15,329	13,807	16,797	7,964	15,618	17,275	12,920	2,723		3,603	
Av. NSR	$/t ore	**136**	138	134	125	153	152	134	142	133	88		100	
Av. Cu	%	**1.70**	1.73	1.67	1.57	1.90	1.89	1.72	1.80	1.66	0.74		1.26	
Av. Zn	%	**0.05**	0.02	0.08	0.06	0.03	0.03	0.03	0.03	0.07	0.15		0.01	
975 mRL														
Pl. diluted tonnes	t	**99,008**		5,444	2,816	7,173	16,524	13,657	13,665	11,238	4,680	20,640	3,172	
Av. NSR	$/t ore	**111**		104	74	93	88	92	90	148	143	140	118	
Av. Cu	%	**1.40**		1.31	1.12	1.16	1.12	1.17	1.13	1.85	1.84	1.75	1.50	
Av. Zn	%	**0.03**		0.05	0.04	0.04	0.02	0.02	0.02	0.04	0.02	0.05	0.06	
950 mRL														
Pl. diluted tonnes	t	**69,180**		12,749	3,722	1,169	14,485	6,801	3,957	8,559	2,679	10,551	4,508	
Av. NSR	$/t ore	**131**		110	101	169	116	123	133	139	146	159	182	
Av. Cu	%	**1.68**		1.36	1.32	2.09	1.45	1.52	1.73	1.81	1.93	2.06	2.34	
Av. Zn	%	**0.03**		0.02	0.04	0.01	0.02	0.02	0.03	0.06	0.02	0.04	0.07	
925 mRL														
Pl. diluted tonnes	t	**65,185**		17,013	3,958	1,893	24,899	5,095	3,227	2,617	2,075	4,408		
Av. NSR	$/t ore	**126**		114	102	211	144	90	108	131	131	102		
Av. Cu	%	**1.57**		1.42	1.27	2.61	1.79	1.12	1.34	1.65	1.67	1.30		
Av. Zn	%	**0.03**		0.04	0.06	0.01	0.02	0.03	0.04	0.02	0.02	0.03		
900 mRL														
Pl. diluted tonnes	t	**28,232**		9,747	3,416		7,307	3,767	3,995					
Av. NSR	$/t ore	**114**		112	118		118	123	95					
Av. Cu	%	**1.43**		1.41	1.49		1.48	1.60	1.19					
Av. Zn	%	**0.04**		0.04	0.05		0.03	0.02	0.03					
Total														
Pl. diluted tonnes	**t**	**1,334,568**	**56,691**	**168,302**	**128,026**	**165,827**	**154,609**	**116,017**	**180,183**	**131,976**	**59,965**	**85,258**	**50,732**	**36,982**
Av. NSR	**$/t ore**	**142**	**132**	**135**	**122**	**159**	**153**	**156**	**151**	**153**	**148**	**121**	**115**	**87**
Av. Cu	**%**	**1.49**	**1.44**	**1.46**	**1.17**	**1.59**	**1.61**	**1.60**	**1.69**	**1.83**	**1.55**	**1.05**	**0.91**	**1.08**
Av. Zn	**%**	**2.45**	**1.65**	**1.70**	**2.93**	**3.25**	**2.41**	**3.30**	**1.77**	**1.57**	**3.17**	**3.49**	**4.08**	**0.01**
Unpl. diluted tonnes	**t**	**1,402,097**	**59,560**	**176,818**	**134,504**	**174,217**	**162,433**	**121,888**	**189,301**	**138,654**	**62,999**	**89,572**	**53,299**	**38,853**
Av. NSR	**$/t ore**	**135**	**126**	**128**	**117**	**152**	**146**	**149**	**144**	**146**	**141**	**115**	**109**	**83**
Av. Cu	**%**	**1.42**	**1.37**	**1.39**	**1.11**	**1.51**	**1.53**	**1.52**	**1.61**	**1.74**	**1.47**	**1.00**	**0.87**	**1.02**
Av. Zn	**%**	**2.33**	**1.57**	**1.62**	**2.79**	**3.10**	**2.29**	**3.14**	**1.68**	**1.50**	**3.02**	**3.33**	**3.88**	**0.01**
Recovered tonnes	**t**	**1,321,055**	**56,117**	**166,598**	**126,730**	**164,148**	**153,044**	**114,843**	**178,359**	**130,639**	**59,358**	**84,394**	**50,218**	**36,607**
Av. NSR	**$/t ore**	**135**	**126**	**128**	**117**	**152**	**146**	**149**	**144**	**146**	**141**	**115**	**109**	**83**
Av. Cu	**%**	**1.42**	**1.37**	**1.39**	**1.11**	**1.51**	**1.53**	**1.52**	**1.61**	**1.74**	**1.47**	**1.00**	**0.87**	**1.02**
Av. Zn	**%**	**2.33**	**1.57**	**1.62**	**2.79**	**3.10**	**2.29**	**3.14**	**1.68**	**1.50**	**3.02**	**3.33**	**3.88**	**0.01**

Figure 16.20 Main Orebody; annual schedule of development ore tonnes and grades

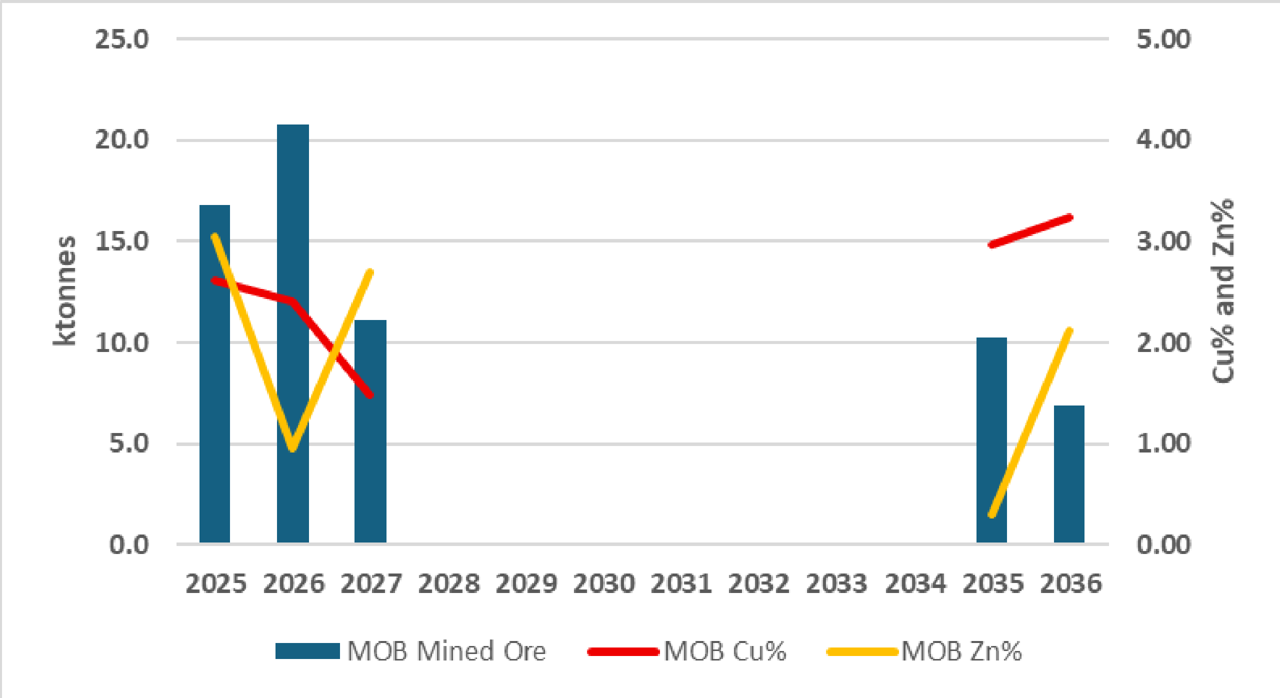
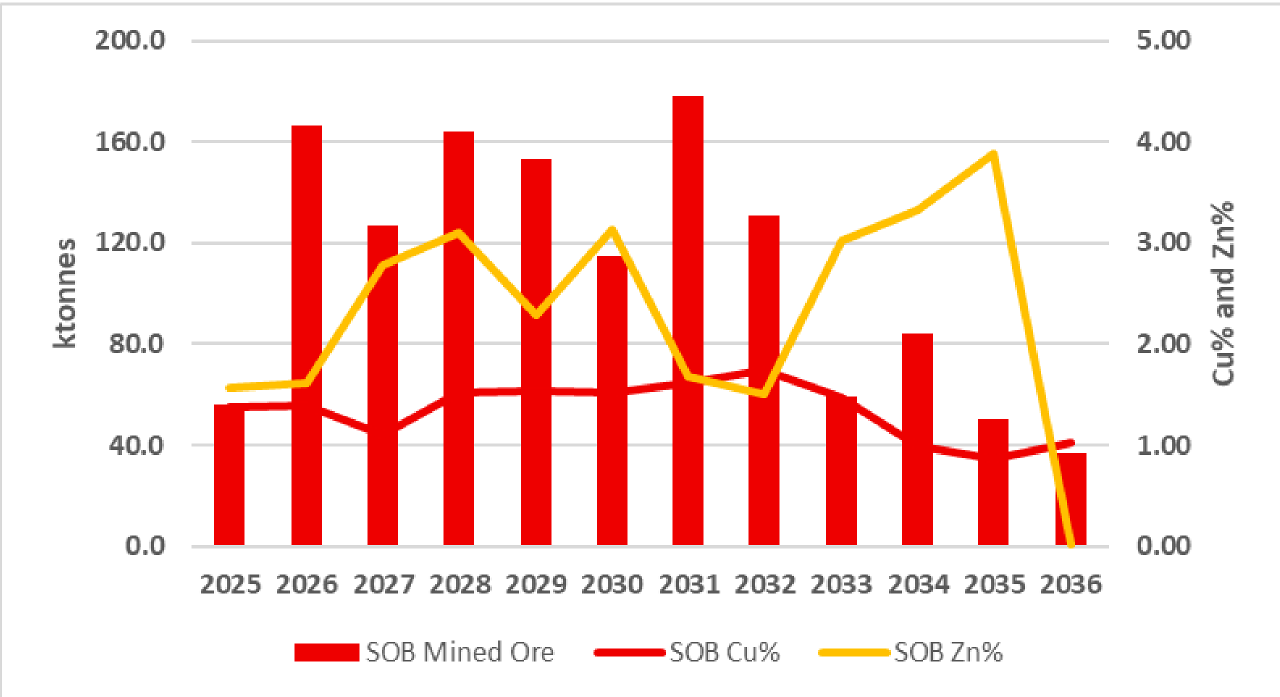


Figure 16.21 South Orebody; annual schedule of development ore tonnes and grades



16.7.3 Stoping ore

Table 16-18 and Table 16-19 summarise respectively, the annual stoping tonnage (in addition to NSR, Cu grade and Zn grade) for the Main Orebody and the South Orebody. The upper part of each table lists the designed tonnage and grades inclusive of planned dilution. The adjustments at the bottom of each table reflect the application of the generic unplanned dilution and mining recovery (loss) factors.

The respective information in these tables is also shown graphically in Figure 16.22 and Figure 16.23.

Table 16-18 Main Orebody; annual schedule of stope tonnes and grades

Sublevel	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
1110 mRL														
Pl. diluted tonnes	**t**	**20,829**	19,737											1,092
Av. NSR	**$/t ore**	**233**	240											117
Av. Cu	**%**	**2.66**	2.78											0.56
Av Zn	**%**	**8.52**	8.55											8.03
1090 mRL														
Pl. diluted tonnes	**t**	**6,854**	6,854											
Av. NSR	**$/t ore**	**426**	426											
Av. Cu	**%**	**4.73**	4.73											
Av Zn	**%**	**6.04**	6.04											
980 mRL														
Pl. diluted tonnes	**t**	**107,479**	58,492										48,986	
Av. NSR	**$/t ore**	**403**	389										420	
Av. Cu	**%**	**5.30**	5.42										5.15	
Av Zn	**%**	**6.10**	5.76										6.51	
960 mRL														
Pl. diluted tonnes	**t**	**101,952**	61,765	28,044									12,143	
Av. NSR	**$/t ore**	**370**	430	357									94	
Av. Cu	**%**	**4.53**	5.18	4.68									0.92	
Av Zn	**%**	**5.46**	7.57	1.39									4.12	
920 mRL														
Pl. diluted tonnes	**t**	**1,062**												1,062
Av. NSR	**$/t ore**	**0**												344
Av. Cu	**%**	**0.00**												5
Av Zn	**%**	**0.00**												0
900 mRL														
Pl. diluted tonnes	**t**	**30,500**		2,268	1,823								4,499	21,910
Av. NSR	**$/t ore**	**282**		232	194								97	333
Av. Cu	**%**	**3.49**		3.04	2.54								1.27	4.07
Av Zn	**%**	**3.39**		0.11	0.11								0.02	4.70
880 mRL														
Pl. diluted tonnes	**t**	**88,158**	10,245										45,696	32,217
Av. NSR	**$/t ore**	**162**	169										147	181
Av. Cu	**%**	**2.02**	2.29										1.93	2.06
Av Zn	**%**	**1.93**	0.15										0.11	5.08
860 mRL														
Pl. diluted tonnes	**t**	**130,378**	21,387	35,811	73,181									
Av. NSR	**$/t ore**	**149**	172	134	150									
Av. Cu	**%**	**2.04**	2.30	1.76	2.11									
Av Zn	**%**	**0.33**	0.12	0.05	0.54									
840 mRL														
Pl. diluted tonnes	**t**	**13,631**		13,631										
Av. NSR	**$/t ore**	**152**		152										
Av. Cu	**%**	**2.50**		2.50										
Av Zn	**%**	**0.73**		0.73										
820 mRL														
Pl. diluted tonnes	**t**	**14,596**	14,596											
Av. NSR	**$/t ore**	**128**	128											
Av. Cu	**%**	**1.67**	1.67											
Av Zn	**%**	**0.01**	0.01											
Total														
Pl. diluted tonnes	**t**	**515,437**	**193,075**	**79,754**	**75,003**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**111,325**	**56,280**
Av. NSR	**$/t ore**	**263**	**333**	**218**	**151**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**259**	**242**
Av. Cu	**%**	**3.36**	**4.25**	**2.95**	**2.12**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**3.21**	**2.86**
Av Zn	**%**	**3.41**	**5.27**	**0.64**	**0.53**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**3.36**	**4.90**
Unpl. diluted tonnes	**t**	**541,519**	**202,845**	**83,789**	**78,798**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**116,958**	**59,128**
Av. NSR	**$/t ore**	**250**	**317**	**208**	**144**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**247**	**230**
Av. Cu	**%**	**3.20**	**4.05**	**2.81**	**2.02**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**3.06**	**2.72**
Av. Zn	**%**	**3.25**	**5.02**	**0.61**	**0.50**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**3.20**	**4.66**
Recovered tonnes	**t**	**510,219**	**191,121**	**78,946**	**74,244**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**110,198**	**55,710**
Av. NSR	**$/t ore**	**250**	**317**	**208**	**144**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**247**	**230**
Av. Cu	**%**	**3.20**	**4.05**	**2.81**	**2.02**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**3.06**	**2.72**
Av. Zn	**%**	**3.25**	**5.02**	**0.61**	**0.50**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**3.20**	**4.66**

Table 16-19 South Orebody; annual schedule of stope tonnes and grades

Sublevel	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
1200 mRL														
Pl. diluted tonnes	**t**	**125,363**				17,537	37,324	27,744	17,836	4,280	14,610		6,031	
Av. NSR	**$/t ore**	**270**				242	256	277	282	594	235		236	
Av. Cu	**%**	**3.18**				3.00	3.22	3.33	3.46	2.39	2.93		3.04	
Av Zn	**%**	**1.25**				0.26	0.39	0.25	0.26	0.26	8.26		0.74	
1175 mRL														
Pl. diluted tonnes	**t**	**289,324**				71,774	31,953	48,781	42,071	28,380	50,611	4,651	11,103	
Av. NSR	**$/t ore**	**250**				211	206	344	203	322	282	118	122	
Av. Cu	**%**	**2.81**				2.44	2.01	4.07	2.36	3.34	3.31	1.15	0.88	
Av Zn	**%**	**4.24**				2.79	5.49	1.90	2.74	6.80	7.28	3.94	5.69	
1150 mRL														
Pl. diluted tonnes	**t**	**380,976**			5,148	72,717	44,695	76,198	42,762	37,872	31,352	39,143	31,089	
Av. NSR	**$/t ore**	**165**			222	157	150	174	224	180	164	118	142	
Av. Cu	**%**	**1.15**			2.16	0.75	0.94	1.32	2.53	1.59	1.32	0.31	0.32	
Av Zn	**%**	**8.15**			7.61	9.56	7.91	8.03	5.73	6.90	7.41	8.62	10.55	
1125 mRL														
Pl. diluted tonnes	**t**	**548,801**			102,166	35,127	96,131	103,827	58,347	48,600	25,436	29,206	49,962	
Av. NSR	**$/t ore**	**124**			131	130	130	135	123	122	107	100	96	
Av. Cu	**%**	**0.41**			0.45	0.60	0.30	0.34	0.46	0.28	0.87	0.45	0.31	
Av Zn	**%**	**8.39**			8.69	7.69	9.55	9.84	7.93	9.08	3.54	6.26	6.65	
1100 mRL														
Pl. diluted tonnes	**t**	**389,843**		54,147	55,409	47,472	92,429	9,269	29,042	23,328	26,922	29,067	14,312	8,445
Av. NSR	**$/t ore**	**119**		121	134	135	129	196	95	102	93	95	96	78
Av. Cu	**%**	**1.04**		0.88	1.30	1.18	1.09	2.17	0.74	0.79	1.07	0.63	1.06	0.96
Av Zn	**%**	**3.75**		5.36	3.06	4.18	4.00	1.81	3.82	5.14	1.29	4.59	1.21	0.01
1075 mRL														
Pl. diluted tonnes	**t**	**728,963**		129,144	98,031	88,719	91,870	41,751	29,250	59,254	47,646	40,084	40,799	62,416
Av. NSR	**$/t ore**	**122**		132	121	125	116	106	117	150	111	126	114	103
Av. Cu	**%**	**1.29**		1.87	0.74	1.40	0.71	1.04	1.12	1.83	1.34	1.35	1.40	1.23
Av Zn	**%**	**2.03**		0.08	5.62	1.21	5.61	2.49	1.38	0.65	0.50	1.81	0.08	0.19
1050 mRL														
Pl. diluted tonnes	**t**	**773,715**	92,447	136,620	61,867	116,369	26,154	56,819	26,162	61,210	90,399	56,427	16,119	33,122
Av. NSR	**$/t ore**	**108**	106	97	106	89	94	136	129	132	104	131	95	108
Av. Cu	**%**	**1.26**	1.51	1.05	1.39	0.72	0.89	1.71	1.67	1.76	1.12	1.44	1.22	1.37
Av Zn	**%**	**1.16**	0.02	1.57	0.04	3.13	1.79	0.22	0.01	0.02	1.56	1.98	0.01	0.01
1025 mRL														
Pl. diluted tonnes	**t**	**614,937**	92,837	85,902	66,996	96,735	44,237	40,229	8,025	70,410	28,481	39,823	10,491	30,771
Av. NSR	**$/t ore**	**144**	152	122	152	135	156	169	79	172	110	195	87	76
Av. Cu	**%**	**1.81**	1.92	1.55	1.96	1.69	1.98	2.10	1.02	2.16	1.38	2.42	1.09	0.97
Av Zn	**%**	**0.05**	0.03	0.08	0.03	0.10	0.06	0.02	0.06	0.02	0.06	0.02	0.02	0.02
1000 mRL														
Pl. diluted tonnes	**t**	**511,299**		89,206	85,920	34,837	71,758	54,276	65,048	33,097	62,488		14,669	
Av. NSR	**$/t ore**	**146**		139	142	150	156	139	152	177	141		85	
Av. Cu	**%**	**1.83**		1.75	1.80	1.92	1.94	1.73	1.91	2.20	1.76		1.16	
Av Zn	**%**	**0.07**		0.07	0.06	0.03	0.03	0.07	0.03	0.02	0.19		0.01	
975 mRL														
Pl. diluted tonnes	**t**	**415,657**			26,054	76,628	48,892	69,696	90,433	66,941	378	25,278	11,357	
Av. NSR	**$/t ore**	**105**			105	110	103	98	101	106	107	125	116	
Av. Cu	**%**	**1.32**			1.31	1.38	1.29	1.24	1.26	1.35	1.33	1.57	1.46	
Av Zn	**%**	**0.02**			0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.01	0.06	
950 mRL														
Pl. diluted tonnes	**t**	**323,665**		774	26,748	12,462	25,917	53,768	31,317	24,093	44,729	82,222	21,635	
Av. NSR	**$/t ore**	**115**		78	93	112	104	99	117	125	127	117	146	
Av. Cu	**%**	**1.45**		0.91	1.20	1.39	1.31	1.25	1.42	1.63	1.63	1.47	1.83	
Av Zn	**%**	**0.04**		0.13	0.03	0.04	0.03	0.02	0.02	0.05	0.04	0.05	0.07	
925 mRL														
Pl. diluted tonnes	**t**	**212,405**		3,160	57,634	13,796	48,710	31,845	12,831	8,606	16,983	18,841		
Av. NSR	**$/t ore**	**119**		78	121	82	108	104	137	135	193	115		
Av. Cu	**%**	**1.51**		0.93	1.52	1.03	1.36	1.40	1.74	1.71	2.41	1.48		
Av Zn	**%**	**0.03**		0.03	0.03	0.05	0.03	0.02	0.04	0.03	0.03	0.03		
900 mRL														
Pl. diluted tonnes	**t**	**155,447**		31,953	49,746		35,320	19,778	16,694	1,955				
Av. NSR	**$/t ore**	**115**		112	118		120	116	107	112				
Av. Cu	**%**	**1.44**		1.39	1.48		1.49	1.47	1.33	1.38				
Av Zn	**%**	**0.05**		0.05	0.05		0.05	0.03	0.03	0.02				
Total														
Pl. diluted tonnes	t	**5,470,396**	**185,284**	**530,906**	**635,720**	**684,173**	**695,390**	**633,982**	**469,817**	**468,025**	**440,035**	**364,742**	**227,567**	**134,754**
Av. NSR	$/t ore	**136**	**129**	**120**	**127**	**135**	**138**	**154**	**142**	**156**	**145**	**126**	**115**	**97**
Av. Cu	%	**1.43**	**1.71**	**1.45**	**1.26**	**1.37**	**1.28**	**1.57**	**1.55**	**1.68**	**1.66**	**1.29**	**1.01**	**1.19**
Av Zn	%	**2.38**	**0.02**	**1.00**	**2.61**	**2.71**	**3.46**	**2.96**	**2.10**	**2.27**	**2.33**	**2.36**	**3.30**	**0.10**
Unpl. diluted tonnes	t	**5,747,198**	**194,659**	**557,770**	**667,887**	**718,792**	**730,577**	**666,062**	**493,590**	**491,707**	**462,301**	**383,198**	**239,082**	**141,573**
Av. NSR	$/t ore	**129**	**123**	**114**	**121**	**129**	**131**	**147**	**135**	**149**	**138**	**120**	**110**	**92**
Av. Cu	%	**1.36**	**1.63**	**1.38**	**1.20**	**1.30**	**1.21**	**1.49**	**1.48**	**1.60**	**1.58**	**1.22**	**0.96**	**1.13**
Av. Zn	%	**2.27**	**0.02**	**0.95**	**2.49**	**2.58**	**3.29**	**2.81**	**2.00**	**2.16**	**2.22**	**2.25**	**3.14**	**0.09**
Recovered tonnes	t	**5,415,010**	**183,408**	**525,531**	**629,284**	**677,246**	**688,350**	**627,563**	**465,060**	**463,286**	**435,580**	**361,049**	**225,263**	**133,390**
Av. NSR	$/t ore	**129**	**123**	**114**	**121**	**129**	**131**	**147**	**135**	**149**	**138**	**120**	**110**	**92**
Av. Cu	%	**1.36**	**1.63**	**1.38**	**1.20**	**1.30**	**1.21**	**1.49**	**1.48**	**1.60**	**1.58**	**1.22**	**0.96**	**1.13**
Av. Zn	%	**2.27**	**0.02**	**0.95**	**2.49**	**2.58**	**3.29**	**2.81**	**2.00**	**2.16**	**2.22**	**2.25**	**3.14**	**0.09**

Figure 16.22 Main Orebody; annual schedule of stope tonnes and grades

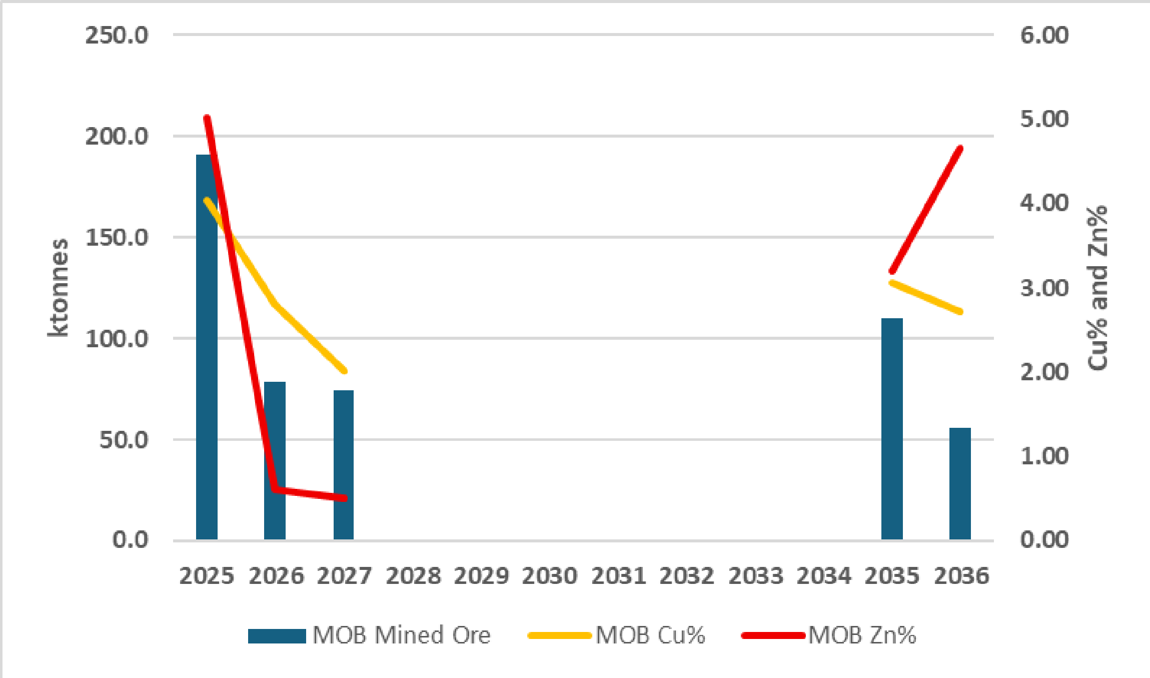
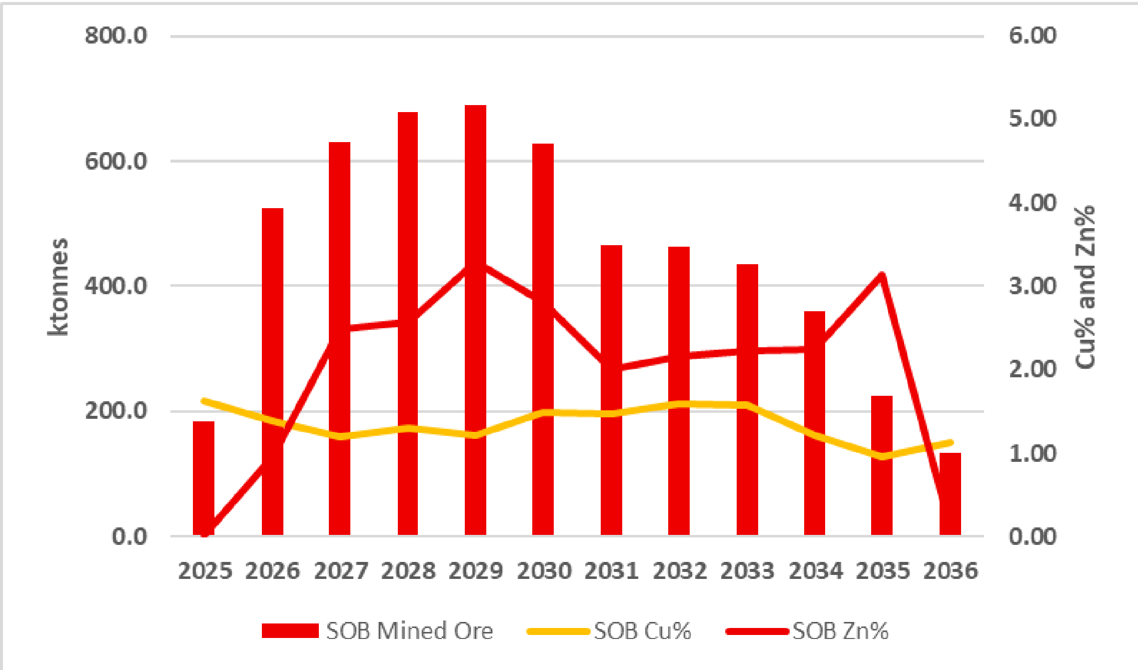


Figure 16.23 South Orebody; annual schedule of stope tonnes and grades



16.7.4 Combined production schedule

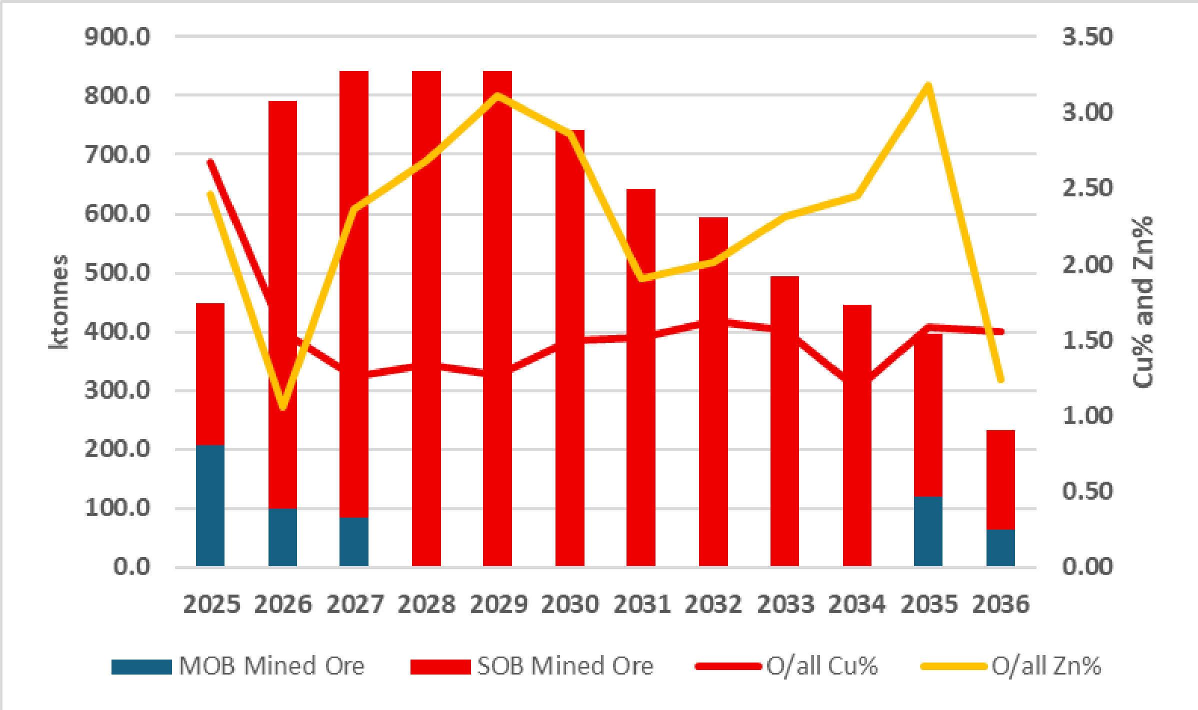
Table 16-20 combines and summarises the annual production schedule information for ore development and stoping in both orebodies. The information is also shown graphically in Figure 16.24.

As previously, this table reports the schedule figures after the application of both planned and unplanned dilution (and mining recovery losses).

Table 16-20 Annual schedule of combined development and stope tonnes and grades

Orebody	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Main Orebody ore development														
Dil'd and Rec'd tonnes	**t**	**65,968**	16,821	20,826	11,137								10,271	6,913
Av. NSR	**$/t ore**	**195**	216	216	80								226	217
Av. Cu	**%**	**2.48**	2.61	2.41	1.48								2.96	3.24
Av. Zn	**%**	**1.80**	3.04	0.95	2.69								0.29	2.13
Av. Ag	**g/t**	**20.74**	45.00	9.83	20.87								4.51	18.46
Main Orebody stopes														
Dil'd and Rec'd tonnes	**t**	**510,219**	191,121	78,946	74,244								110,198	55,710
Av. NSR	**$/t ore**	**250**	317	208	144								247	230
Av. Cu	**%**	**3.20**	4.05	2.81	2.02								3.06	2.72
Av. Zn	**%**	**3.25**	5.02	0.61	0.50								3.20	4.66
Av. Ag	**g/t**	**27.32**	40.56	8.18	1.66								34.08	29.85
Main Orebody total														
Dil'd and Rec'd tonnes	**t**	**576,187**	**207,942**	**99,772**	**85,381**								**120,469**	**62,623**
Av. NSR	**$/t ore**	**244**	**309**	**210**	**136**								**245**	**229**
Av. Cu	**%**	**3.12**	**3.93**	**2.73**	**1.95**								**3.05**	**2.78**
Av. Zn	**%**	**3.08**	**4.86**	**0.68**	**0.79**								**2.95**	**4.38**
Av. Ag	**g/t**	**26.57**	**40.92**	**8.52**	**4.16**								**31.56**	**28.59**
South Orebody ore development														
Dil'd and Rec'd tonnes	**t**	**1,321,055**	56,117	166,598	126,730	164,148	153,044	114,843	178,359	130,639	59,358	84,394	50,218	36,607
Av. NSR	**$/t ore**	**135**	126	128	117	152	146	149	144	146	141	115	109	83
Av. Cu	**%**	**1.42**	1.37	1.39	1.11	1.51	1.53	1.52	1.61	1.74	1.47	1.00	0.87	1.02
Av. Zn	**%**	**2.33**	1.57	1.62	2.79	3.10	2.29	3.14	1.68	1.50	3.02	3.33	3.88	0.01
Av. Ag	**g/t**	**9.13**	4.44	6.17	11.15	10.67	9.08	12.49	7.17	5.65	14.33	14.88	11.67	2.27
South Orebody stopes														
Dil'd and Rec'd tonnes	**t**	**5,415,010**	183,408	525,531	629,284	677,246	688,350	627,563	465,060	463,286	435,580	361,049	225,263	133,390
Av. NSR	**$/t ore**	**129**	123	114	121	129	131	147	135	149	138	120	110	92
Av. Cu	**%**	**1.36**	1.63	1.38	1.20	1.30	1.21	1.49	1.48	1.60	1.58	1.22	0.96	1.13
Av. Zn	**%**	**2.27**	0.02	0.95	2.49	2.58	3.29	2.81	2.00	2.16	2.22	2.25	3.14	0.09
Av. Ag	**g/t**	**8.87**	1.52	7.42	8.11	9.33	10.70	11.50	8.66	8.72	8.19	10.51	9.61	2.14
South Orebody total														
Dil'd and Rec'd tonnes	**t**	**6,736,066**	**239,525**	**692,129**	**756,013**	**841,394**	**841,394**	**742,406**	**643,419**	**593,926**	**494,937**	**445,444**	**275,482**	**169,997**
Av. NSR	**$/t ore**	**131**	**123**	**117**	**120**	**133**	**134**	**147**	**138**	**148**	**138**	**119**	**109**	**90**
Av. Cu	**%**	**1.37**	**1.57**	**1.38**	**1.18**	**1.34**	**1.27**	**1.50**	**1.52**	**1.63**	**1.57**	**1.18**	**0.94**	**1.11**
Av. Zn	**%**	**2.28**	**0.38**	**1.11**	**2.54**	**2.68**	**3.11**	**2.86**	**1.91**	**2.01**	**2.32**	**2.45**	**3.28**	**0.08**
Av. Ag	**g/t**	**8.92**	**2.21**	**7.12**	**8.62**	**9.59**	**10.40**	**11.66**	**8.24**	**8.04**	**8.92**	**11.34**	**9.98**	**2.17**
Combined ore development														
Dil'd and Rec'd tonnes	**t**	**1,387,024**	72,939	187,423	137,867	164,148	153,044	114,843	178,359	130,639	59,358	84,394	60,490	43,520
Av. NSR	**$/t ore**	**138**	146	138	114	152	146	149	144	146	141	115	129	104
Av. Cu	**%**	**1.47**	1.66	1.51	1.14	1.51	1.53	1.52	1.61	1.74	1.47	1.00	1.22	1.38
Av. Zn	**%**	**2.31**	1.91	1.54	2.78	3.10	2.29	3.14	1.68	1.50	3.02	3.33	3.27	0.35
Av. Ag	**g/t**	**9.68**	13.80	6.58	11.94	10.67	9.08	12.49	7.17	5.65	14.33	14.88	10.45	4.84
Combined stopes														
Dil'd and Rec'd tonnes	**t**	**5,925,229**	374,528	604,477	703,527	677,246	688,350	627,563	465,060	463,286	435,580	361,049	335,461	189,100
Av. NSR	**$/t ore**	**140**	222	126	123	129	131	147	135	149	138	120	155	133
Av. Cu	**%**	**1.52**	2.86	1.57	1.29	1.30	1.21	1.49	1.48	1.60	1.58	1.22	1.65	1.60
Av. Zn	**%**	**2.35**	2.57	0.91	2.28	2.58	3.29	2.81	2.00	2.16	2.22	2.25	3.16	1.44
Av. Ag	**g/t**	**10.46**	21.45	7.51	7.43	9.33	10.70	11.50	8.66	8.72	8.19	10.51	17.65	10.31
Combined total														
Dil'd and Rec'd tonnes	**t**	**7,312,253**	**447,467**	**791,901**	**841,394**	**841,394**	**841,394**	**742,406**	**643,419**	**593,926**	**494,937**	**445,444**	**395,950**	**232,621**
Av. NSR	**$/t ore**	**139**	**209**	**129**	**122**	**133**	**134**	**147**	**138**	**148**	**138**	**119**	**151**	**127**
Av. Cu	**%**	**1.51**	**2.67**	**1.55**	**1.26**	**1.34**	**1.27**	**1.50**	**1.52**	**1.63**	**1.57**	**1.18**	**1.59**	**1.56**
Av. Zn	**%**	**2.34**	**2.46**	**1.06**	**2.36**	**2.68**	**3.11**	**2.86**	**1.91**	**2.01**	**2.32**	**2.45**	**3.18**	**1.23**
Av. Ag	**g/t**	**10.31**	**20.20**	**7.29**	**8.17**	**9.59**	**10.40**	**11.66**	**8.24**	**8.04**	**8.92**	**11.34**	**16.55**	**9.28**

Figure 16.24 Annual schedule of combined development and stope tonnes and grades



16.7.5 Ore types and NSR

Figure 16.25 and Figure 16.26 show respectively, the annual ore production in terms of mined ore types (plus the corresponding overall annual NSR values), and the relative proportions of each ore type mined over the duration of the LOM. There is no Main Orebody YO+ ore type mined.

Figure 16.25 Annual production schedule in terms of mined ore type and NSR value

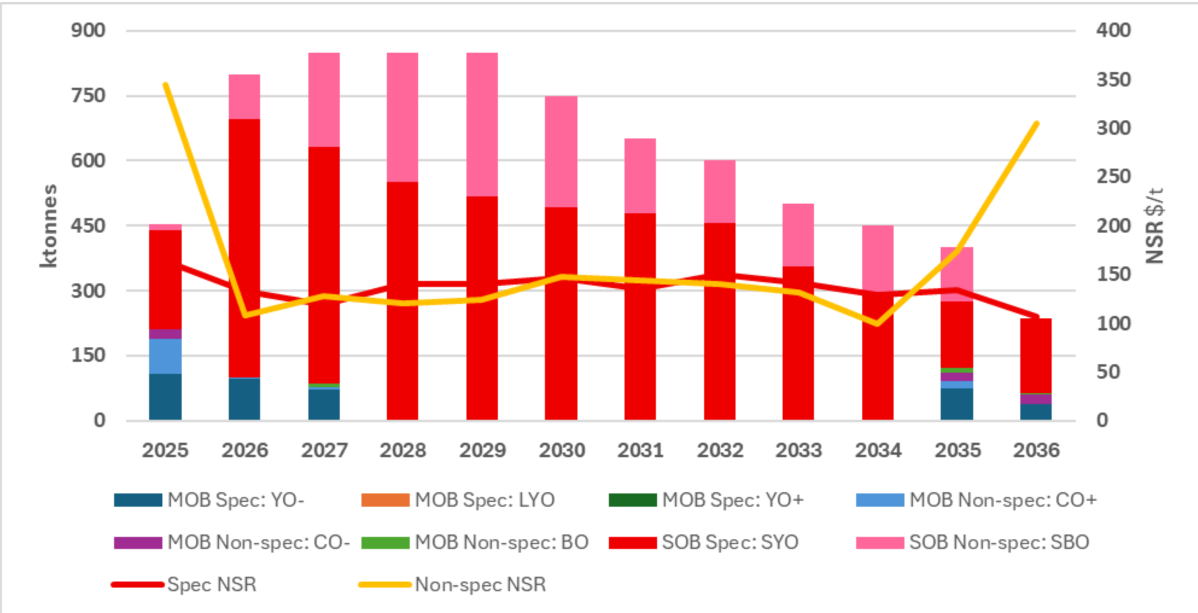
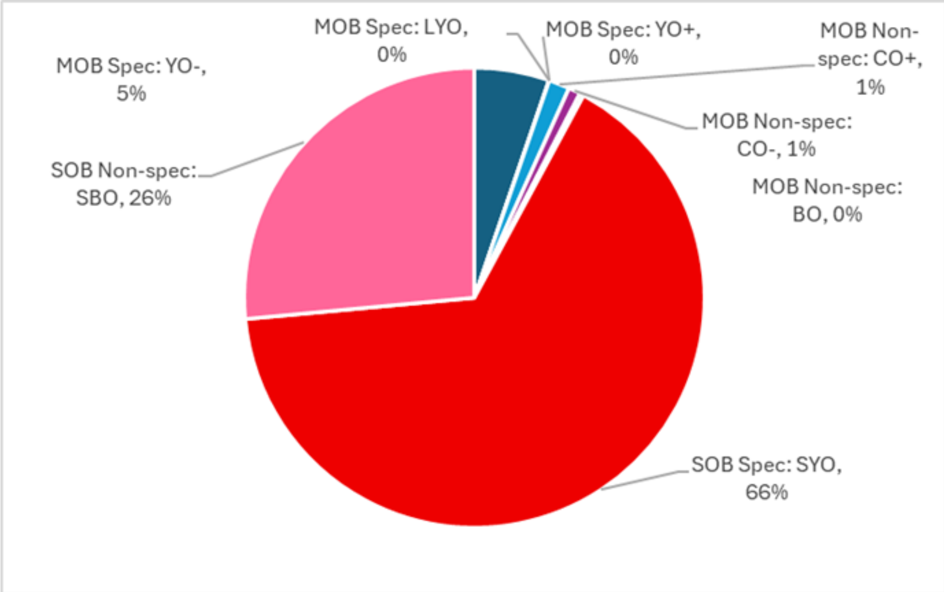


Figure 16.26 Proportion of ore types in LOM plan



16.8 Plant feed schedule

Table 16-21 lists the plant feed schedule. In this instance, there is a no cross-blending from the Main Orebody mined ore types to the corresponding plant feed. Similarly, there is no cross-blending from the South Orebody mined ore types to the corresponding plant feed.

Table 16-21 Annual schedule of plant feed tonnes and grades

Oretype	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Main Orebody														
Spec														
Dil'd and Rec'd tonnes	t	386,616	106,760	96,026	71,554								73,893	38,383
Av. Cu	%	2.60	3.27	2.69	1.79								2.40	2.43
Av. Zn	%	0.62	0.95	0.62	0.09								0.51	0.93
Non-spec														
Dil'd and Rec'd tonnes	t	189,571	101,181	3,746	13,827								46,576	24,241
Av. Cu	%	4.17	4.63	3.74	2.77								4.07	3.33
Av. Zn	%	8.09	8.98	2.27	4.40								6.81	9.85
South Orebody														
Spec														
Dil'd and Rec'd tonnes	t	4,800,095	227,147	589,881	540,915	546,554	512,599	486,215	473,362	452,850	353,489	294,308	152,778	169,997
Av. Cu	%	1.67	1.63	1.57	1.51	1.77	1.75	1.82	1.69	1.86	1.78	1.60	1.38	1.11
Av. Zn	%	0.19	0.02	0.11	0.04	0.08	0.05	0.10	0.06	0.47	1.01	0.26	0.05	0.08
Non-spec														
Dil'd and Rec'd tonnes	t	1,935,970	12,378	102,248	215,098	294,839	328,795	256,192	170,057	141,075	141,448	151,136	122,703	
Av. Cu	%	0.64	0.42	0.34	0.36	0.56	0.53	0.88	1.04	0.89	1.05	0.36	0.41	
Av. Zn	%	7.46	6.96	6.87	8.83	7.50	7.87	8.12	7.06	6.98	5.57	6.72	7.30	
Combined orebodies														
Spec														
Dil'd and Rec'd tonnes	t	5,186,711	333,907	685,906	612,469	546,554	512,599	486,215	473,362	452,850	353,489	294,308	226,671	208,380
Av. Cu	%	1.74	2.16	1.72	1.54	1.77	1.75	1.82	1.69	1.86	1.78	1.60	1.71	1.35
Av. Zn	%	0.22	0.32	0.18	0.04	0.08	0.05	0.10	0.06	0.47	1.01	0.26	0.20	0.23
Non-spec														
Dil'd and Rec'd tonnes	t	2,125,542	113,559	105,994	228,925	294,839	328,795	256,192	170,057	141,075	141,448	151,136	169,280	24,241
Av. Cu	%	0.95	4.17	0.46	0.51	0.56	0.53	0.88	1.04	0.89	1.05	0.36	1.41	3.33
Av. Zn	%	7.52	8.76	6.70	8.57	7.50	7.87	8.12	7.06	6.98	5.57	6.72	7.16	9.85
TOTAL														
Dil'd and Rec'd tonnes	t	7,312,253	447,467	791,901	841,394	841,394	841,394	742,406	643,419	593,926	494,937	445,444	395,950	232,621
Av. Cu	%	1.51	2.67	1.55	1.26	1.34	1.27	1.50	1.52	1.63	1.57	1.18	1.59	1.56
Av. Zn	%	2.34	2.46	1.06	2.36	2.68	3.11	2.86	1.91	2.01	2.32	2.45	3.18	1.23

16.9 Mining sequence

Following completion of the detailed development and stope designs, and the LOM mining production schedule, a comprehensive spreadsheet inventory was prepared for each of the two orebodies, showing the mining sequence for every design solid, on every sublevel and for each year of the LOM duration.

The complete output is too extensive to show either graphically or by means of a suitably comprehensive table. Figure 16.27, however, shows an example subset of the LOM mining plan, for a select set of sublevels, and showing the typical mining sequence for ore development headings and corresponding stopes. From this figure it can be appreciated that:

- stoping lags behind ore development, for varying durations
- stoping progresses upwards through the sublevels, without apparent under-cutting

Table 16-22 and Table 16-23, summarise respectively, the scheduled annual ore development and stoping tonnages for the Main and South Orebodies.

Figure 16.27 South Orebody, 1200 mRL to 1100 mRL, example subset from LOM mining sequence plan

Table 16-22 Mining sequence for the Main Orebody, listing annual ore tonnes mined from each sublevel, and according to development and stoping solids

Sublevel	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Main Orebody														
1110 mRL														
Ore development	t	**15,871**	6,590		7,618									1,664
Stoping	t	**20,829**	19,737											1,092
1090 mRL														
Ore development	t	**2,416**	2,416											
Stoping	t	**6,854**	6,854											
980 mRL														
Ore development	t	**2,523**											2,523	
Stoping	t	**107,479**	58,492										48,986	
960 mRL														
Ore development	t	**15,235**	5,293	9,942										
Stoping	t	**101,952**	61,765	28,044									12,143	
920 mRL														
Ore development	t													
Stoping	t	**1,062**												1,062
900 mRL														
Ore development	t	**6,324**		4,793										1,531
Stoping	t	**30,500**		2,268	1,823								4,499	21,910
880 mRL														
Ore development	t	**11,642**											7,853	3,790
Stoping	t	**88,158**	10,245										45,696	32,217
860 mRL														
Ore development	t	**7,090**	1,061	2,396	3,633									
Stoping	t	**130,378**	21,387	35,811	73,181									
840 mRL														
Ore development	t	**3,908**		3,908										
Stoping	t	**13,631**		13,631										
820 mRL														
Ore development	t	**1,634**	1,634											
Stoping	t	**14,596**	14,596											
Pl. diluted tonnes														
Ore development	t	**66,643**	**16,993**	**21,039**	**11,251**								**10,376**	**6,984**
Stoping	t	**515,437**	**193,075**	**79,754**	**75,003**								**111,325**	**56,280**
Total	t	**582,081**	**210,069**	**100,793**	**86,254**								**121,701**	**63,264**
Unpl. diluted tonnes														
Ore development	t	**70,015**	**17,853**	**22,103**	**11,820**								**10,901**	**7,337**
Stoping	t	**541,519**	**202,845**	**83,789**	**78,798**								**116,958**	**59,128**
Total	t	**611,534**	**220,698**	**105,893**	**90,619**								**127,859**	**66,465**
Recovered tonnes														
Ore development	t	**65,968**	**16,821**	**20,826**	**11,137**								**10,271**	**6,913**
Stoping	t	**510,219**	**191,121**	**78,946**	**74,244**								**110,198**	**55,710**
Total	t	**576,187**	**207,942**	**99,772**	**85,381**								**120,469**	**62,623**

Table 16-23 Mining sequence for the South Orebody, listing annual ore tonnes mined from each sublevel, and according to development and stoping solids

Sublevel	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
South Orebody														
1200 mRL														
Ore development	t	**54,536**				16,665	11,120	10,794	9,110	3,985	2,862			
Stoping	t	**125,363**				17,537	37,324	27,744	17,836	4,280	14,610		6,031	
1175 mRL														
Ore development	t	**93,348**			490	19,441	14,809	20,811	16,102	12,863	6,078	2,755		
Stoping	t	**289,324**				71,774	31,953	48,781	42,071	28,380	50,611	4,651	11,103	
1150 mRL														
Ore development	t	**116,947**			23,135	9,221	10,538	17,826	8,887	8,338	8,070	19,320	11,612	
Stoping	t	**380,976**			5,148	72,717	44,695	76,198	42,762	37,872	31,352	39,143	31,089	
1125 mRL														
Ore development	t	**144,233**		27,874	11,297	19,940	14,323	15,732	14,581	8,140	4,155	13,145	15,047	
Stoping	t	**548,801**			102,166	35,127	96,131	103,827	58,347	48,600	25,436	29,206	49,962	
1100 mRL														
Ore development	t	**111,258**		28,440	8,958	14,739	2,180	3,913	16,189	14,408	10,722	4,442	1,407	5,859
Stoping	t	**389,843**		54,147	55,409	47,472	92,429	9,269	29,042	23,328	26,922	29,067	14,312	8,445
1075 mRL														
Ore development	t	**182,955**	25,568	10,927	25,152	23,328	7,506		25,009	19,223	11,895	8,177	7,267	18,903
Stoping	t	**728,963**		129,144	98,031	88,719	91,870	41,751	29,250	59,254	47,646	40,084	40,799	62,416
1050 mRL														
Ore development	t	**110,809**	5,959	15,410	12,488	14,660	10,881		23,383	15,773		1,819	1,802	8,634
Stoping	t	**773,715**	92,447	136,620	61,867	116,369	26,154	56,819	26,162	61,210	90,399	56,427	16,119	33,122
1025 mRL														
Ore development	t	**138,544**	10,868	25,369	18,786	20,801	12,073	2,005	24,805	13,912	4,026		2,315	3,585
Stoping	t	**614,937**	92,837	85,902	66,996	96,735	44,237	40,229	8,025	70,410	28,481	39,823	10,491	30,771
1000 mRL														
Ore development	t	**120,333**	14,296	15,329	13,807	16,797	7,964	15,618	17,275	12,920	2,723		3,603	
Stoping	t	**511,299**		89,206	85,920	34,837	71,758	54,276	65,048	33,097	62,488		14,669	
975 mRL														
Ore development	t	**99,008**		5,444	2,816	7,173	16,524	13,657	13,665	11,238	4,680	20,640	3,172	
Stoping	t	**415,657**			26,054	76,628	48,892	69,696	90,433	66,941	378	25,278	11,357	
950 mRL														
Ore development	t	**69,180**		12,749	3,722	1,169	14,485	6,801	3,957	8,559	2,679	10,551	4,508	
Stoping	t	**323,665**		774	26,748	12,462	25,917	53,768	31,317	24,093	44,729	82,222	21,635	
925 mRL														
Ore development	t	**65,185**		17,013	3,958	1,893	24,899	5,095	3,227	2,617	2,075	4,408		
Stoping	t	**212,405**		3,160	57,634	13,796	48,710	31,845	12,831	8,606	16,983	18,841		
900 mRL														
Ore development	t	**28,232**		9,747	3,416		7,307	3,767	3,995					
Stoping	t	**155,447**		31,953	49,746		35,320	19,778	16,694	1,955				
Pl. diluted tonnes														
Ore development	t	**1,334,568**	**56,691**	**168,302**	**128,026**	**165,827**	**154,609**	**116,017**	**180,183**	**131,976**	**59,965**	**85,258**	**50,732**	**36,982**
Stoping	t	**5,470,396**	**185,284**	**530,906**	**635,720**	**684,173**	**695,390**	**633,982**	**469,817**	**468,025**	**440,035**	**364,742**	**227,567**	**134,754**
Total	t	**6,804,964**	**241,975**	**699,208**	**763,746**	**850,000**	**850,000**	**750,000**	**650,000**	**600,000**	**500,000**	**450,000**	**278,299**	**171,736**
Unpl. diluted tonnes														
Ore development	t	**1,402,097**	**59,560**	**176,818**	**134,504**	**174,217**	**162,433**	**121,888**	**189,301**	**138,654**	**62,999**	**89,572**	**53,299**	**38,853**
Stoping	t	**5,747,198**	**194,659**	**557,770**	**667,887**	**718,792**	**730,577**	**666,062**	**493,590**	**491,707**	**462,301**	**383,198**	**239,082**	**141,573**
Total	t	**7,149,295**	**254,219**	**734,588**	**802,391**	**893,010**	**893,010**	**787,950**	**682,890**	**630,360**	**525,300**	**472,770**	**292,381**	**180,426**
Recovered tonnes														
Ore development	t	**1,321,055**	**56,117**	**166,598**	**126,730**	**164,148**	**153,044**	**114,843**	**178,359**	**130,639**	**59,358**	**84,394**	**50,218**	**36,607**
Stoping	t	**5,415,010**	**183,408**	**525,531**	**629,284**	**677,246**	**688,350**	**627,563**	**465,060**	**463,286**	**435,580**	**361,049**	**225,263**	**133,390**
Total	t	**6,736,066**	**239,525**	**692,129**	**756,013**	**841,394**	**841,394**	**742,406**	**643,419**	**593,926**	**494,937**	**445,444**	**275,482**	**169,997**

16.10 Mining equipment

16.10.1 Equipment productivity

Table 16-24 lists the recorded 2023 and 2024 time usage figures for primary mining equipment items. In this table, effective use is defined as the product of availability and utilisation.

Table 16-24 Primary mining equipment usage, 2023 and 2024

Equipment	ID	2023 Availability (%)	2023 Utilisation (%)	2023 Effective use (%)	2024 Availability (%)	2024 Utilisation (%)	2024 Effective use (%)
Jumbos							
Sandvik DD420-40	415	73.9	67.9	**50.2**	77.2	49.9	**38.5**
Atlas Copco 282	417				67.5	50.5	**34.1**
Average		**73.9**	**67.9**	**50.2**	**72.4**	**50.2**	**36.3**
Production drills							
Cubex Megamatic	405	87.6	10.4	**9.1**	88.9	14.7	**13.1**
Simba 1257	406	90.8	1.9	**1.7**	93.9	0.9	**0.8**
Solomatic DK430-7	409	82.7	20.7	**17.1**	82.8	20.7	**17.1**
Solomatic DK430-7C	410	72.6	40.8	**29.6**	74.7	29.8	**22.3**
Average		**83.4**	**18.5**	**15.4**	**85.1**	**16.5**	**14.1**
Trucks							
Wagner MT436B	309	81.3	82.3	**66.9**	77.0	57.1	**44.0**
Wagner MT436B	312	89.5	83.0	**74.3**	83.4	25.3	**21.1**
Wagner MT436B	314	82.1	82.4	**67.7**	74.9	58.6	**43.9**
Wagner MT436B	316	80.7	83.3	**67.2**	84.8	41.7	**35.4**
Wagner MT436B	318	78.4	83.0	**65.1**	81.5	51.2	**41.7**
Wagner MT436B	319	77.3	82.5	**63.8**	78.0	55.9	**43.6**
Wagner MT436B	320	80.0	82.2	**65.8**	78.9	50.4	**39.8**
Wagner MT436B	321	77.9	83.0	**64.7**	83.5	61.8	**51.6**
Wagner MT436B	322	90.3	82.5	**74.5**	84.7	33.1	**28.0**
Average		**81.9**	**82.7**	**67.8**	**80.7**	**48.3**	**39.0**

The overall average effective usage values can be translated into a projection of the required development and stoping productivity going forward. Table 16-25 lists the current primary equipment numbers and the number of items consistent with the new capital purchases and replacements listed in Item 16.10.2. The information in this table for 2023 to 2025 comes from Pitram records; the year-to-date records for 2025 have been prorated to full year figures. The physicals for 2026 to 2035 are from an early 2025 production schedule forecast.

Table 16-25 Projected equipment productivity and equipment numbers

	Units	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Dev't metres drilled	m	*763*	*2,145*	*2,904*	2,741	2,596	2,155	1,664	1,567	1,227	854	1,150	563	735
Number of jumbos	#	*1*	*2*	*2*	2	2	2	2	2	2	2	2	2	2
Effective use of jumbos	%	***50.2***	***36.3***	***43.2***	**43.2**	**43.2**	**43.2**	**43.2**	**43.2**	**43.2**	**43.2**	**43.2**	**43.2**	**43.2**
Desired min. effective use	%	***60.0***	***60.0***	***60.0***	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**
Est. metres drilled	m	*912*	*3,543*	*4,029*	3,803	3,601	2,989	2,309	2,174	1,703	1,184	1,595	781	1,019
Production drill metres	m	*52,180*	*42,525*	*76,200*	83,850	88,917	82,913	89,790	68,846	58,994	42,184	37,768	46,520	36,496
Number of drills	#	*4*	*4*	*4*	4	4	4	4	4	4	4	4	4	4
Effective use of drills	%	***15.4***	***14.1***	***14.7***	**14.7**	**14.7**	**14.7**	**14.7**	**14.7**	**14.7**	**14.7**	**14.7**	**14.7**	**14.7**
Desired min. effective use	%	***60.0***	***60.0***	***60.0***	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**
Est. metres drilled	m	*203,406*	*181,490*	*310,486*	341,659	362,304	337,838	365,860	280,521	240,380	171,884	153,889	189,551	148,706
Tonnes o+w hauled	t	*680,550*	*683,480*	*946,419*	835,900	855,300	771,000	768,000	618,000	518,000	368,000	368,000	368,000	318,000
Number of trucks	#	*9*	*9*	*10*	10	10	10	10	10	10	10	10	10	10
Effective use of trucks	%	***67.8***	***39.0***	***53.4***	**53.4**	**53.4**	**53.4**	**53.4**	**53.4**	**53.4**	**53.4**	**53.4**	**53.4**	**53.4**
Desired min. effective use	%	***60.0***	***60.0***	***60.0***	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**	**60.0**
Est. tonnes hauled	m	*602,621*	*1,050,553*	*1,063,448*	939,263	961,061	866,338	862,966	694,419	582,053	413,505	413,505	413,505	357,322

In relation to the productivity of the two-jumbo development fleet, the current figures indicate that the effective use of these machines is impacted through relatively low utilisation. This reflects on the capacity of the jumbos however, as being sufficient to service the historic development metres. By extrapolation against future annual development metres, the same two-jumbo fleet ought to be adequate to maintain similar levels of productivity. If the utilisation was to be improved to the extent that a desired minimum effective use of 60% could be achieved, then the two-jumbo productivity could theoretically be increased by almost 40%.

In relation to the productivity of four production drills, the current figures again indicate that the effective use of these drills is impacted through relatively low utilisation. If the utilisation was to be improved to the extent that a desired minimum effective use of 60% could be achieved, then the production drilling productivity could be increased to more than adequately cater for the increased annual stoping activity.

In relation to the productivity of haul trucks, if a desired minimum effective use of 60% could be achieved to cater for longer ore hauls from the South Orebody, then the existing ten truck fleet ought to be adequate to service the haulage requirements for the longer term mine plan.

16.10.2 Equipment estimate

The required and replacement items listed in Table 16-26 are accounted for in the mining equipment capital cost provisions tabled in Item 21.2.1. The Table 16-26 list also shows the items to be replaced and/or scrapped according to the mine maintenance programme and considering such as:

- the current condition of the existing equipment, in terms of functionality, performance, failure frequency, availability of spare parts and the number of overhauls performed
- design life and expended operating hours
- chassis and electrical condition

The average age of trucks in the existing fleet is 21 years, whilst that of the loaders (LHDs) is 22 years. Operating hours have significantly exceeded design life and due to age, engine production has been discontinued and spare parts are now difficult to source.

Table 16-26 Mining equipment numbers

Type	Equipment numbers at 2024			Equipment numbers from 2025										
	Description	No. of Items	ID	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Jumbos	*Atlas Copco 282 twin-boom*	1	417	1	1	1	1	1	1	1	1	1	1	1
	Sandvik DD420-40	1	415	1	1	1	1	1	1	1	1	1	1	1
Production Drills	*Sandvik Solomatic DK430-7*	1	409	1	1	1	1	1	1	1	1	1	1	1
	Sandvik Solomatic DK431-7C	1	410	1	1	1	1	1	1	1	1	1	1	1
	Simba 1257	1	406	1	1	1	1	1	1	1	1	1	1	1
	Cubex Megamatic ITH 5200	1	405	1	1	1	1	1	1	1	1	1	1	1
Bolter	*Robolter*		new	1	1	1	1	1	1	1	1	1	1	1
		1	416	1	1	1	1	1	1	1	1	1	1	1
		1	414	1	1	1								
	subtotal	***2***		***3***	***3***	***3***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***
Shotcrete Equipment	*Normet Spraymec 1050 WPC*	1	110	1	1	1	1	1	1	1	1	1	1	1
		1	162	1	1	1	1	1	1	1	1	1	1	1
		1	164	1	1	1	1	1	1	1	1	1	1	1
	subtotal	***3***		***3***	***3***	***3***	***3***	***3***	***3***	***3***	***3***	***3***	***3***	***3***
	Normet Transmixer (7.5 m^3)	1	108	1	1	1	1	1	1	1	1	1	1	1
		1	109	1	1	1	1	1	1	1	1	1	1	1
		1	161	1	1	1	1	1	1	1	1	1	1	1
		1	163	1	1	1	1	1	1	1	1	1	1	1
	subtotal	***4***		***4***	***4***	***4***	***4***	***4***	***4***	***4***	***4***	***4***	***4***	***4***
LHDs	*Toro 1400*	1	210	1	1	1	1	1	1	1	1	1	1	1
		1	211	1	1	1	1	1	1	1	1	1	1	1
		1	212	1										
	subtotal	***3***		***3***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***
	Sandvik LH514	1	214	1	1	1	1	1	1	1	1	1	1	1
			216		1	1	1	1	1	1	1	1	1	1
	subtotal	***1***		***1***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***
	Sandvik LH517	1	217	1	1	1	1	1	1	1	1	1	1	1
Trucks	*Wagner MT436B*	1	309	1	1	1	1	1	1	1	1	1	1	1
		1	311	1	1	1	1	1	1	1	1	1	1	1
		1	312	1	1	1	1	1	1	1	1	1	1	1
			313											
		1	314	1	1	1	1	1	1	1	1	1	1	1
			315											
		1	316	1	1	1	1	1	1	1	1	1	1	1
			317											
		1	318	1	1	1	1	1	1	1	1	1	1	1
		1	319	1	1	1	1	1	1	1	1	1	1	1
		1	320	1	1	1	1	1	1	1	1	1	1	1
		1	321	1	1	1	1	1	1	1	1	1	1	1
		1	322	1	1	1	1	1	1	1	1	1	1	1
	subtotal	***10***		***10***	***10***	***10***	***10***	***10***	***10***	***10***	***10***	***10***	***10***	***10***
Utility Vehicles	*Paus Platforms*	1	118	1	1	1	1	1	1	1	1	1	1	1
		1	119	1	1	1	1	1	1	1	1	1	1	1
		1	120	1	1	1	1	1	1	1	1	1	1	1
		1	140	1	1	1	1	1	1	1	1	1	1	1
		1	141	1	1	1	1	1	1	1	1	1	1	1
		1	171	1	1	1	1	1	1	1	1	1	1	1
	subtotal	***6***		***6***	***6***	***6***	***6***	***6***	***6***	***6***	***6***	***6***	***6***	***6***
	Paus ANFO Truck	1	117	1	1	1	1	1	1	1	1	1	1	1
		1	145	1	1	1	1	1	1	1	1	1	1	1
	subtotal	***2***		***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***
	Paus Crane	1	115	1	1	1	1	1	1	1	1	1	1	1
		1	142	1	1	1	1	1	1	1	1	1	1	1
	subtotal	***2***		***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***	***2***
	Paus Grouting Machine	1	101	1	1	1	1	1	1	1	1	1	1	1
	Paus Oil Transfer	1	103	1	1	1	1	1	1	1	1	1	1	1
	McLean Sewage Truck	1	411	1	1	1	1	1	1	1	1	1	1	1
	Titan Platform	1	143	1	1	1	1	1	1	1	1	1	1	1
	Normet Veekmas Grader	1	221	1	1	1	1	1	1	1	1	1	1	1
Backhoe Loader	*Hidromek HMK 102S Alpha*	1	242	1										
		1	243	1	1	1	1	1	1	1	1	1	1	1
		1	244	1	1	1	1	1	1	1	1	1	1	1
		1	245	1	1	1	1	1	1	1	1	1	1	1
		1	246	1	1	1	1	1	1	1	1	1	1	1
	subtotal	***4***		***5***	***4***	***4***	***4***	***4***	***4***	***4***	***4***	***4***	***4***	***4***
		49		***51***	***50***	***50***	***49***	***49***	***49***	***49***	***49***	***49***	***49***	***49***

(grey)	means "scrapped"
(green)	means replaced

Item 17 RECOVERY METHODS

17.1 Introduction

The ÇBI ore processing facility was commissioned in August 1994, with the first concentrate production occurring in November 1994.

The facility has been in continuous operation for over thirty years, treating ore from the Main Orebody. Ore throughput rates have varied through the years, with a peak rate of 1.34 Mtpa reached in 2014. In recent years throughput has dropped as the Main Orebody has become depleted, with a throughput of 691,000 tonnes being achieved in 2024.

Discovery of the South Orebody will lead to a currently planned mine life extending to 2036, treating a mix of Main and South Orebody feed from 2025, and South Orebody ores exclusively from 2028. Plant throughput rates will peak at approximately 850,000 tpa in the years 2027 to 2029[11].

17.2 Plant feed types

The mineralogy of the ore types from the Main Orebody have been described in Item 13, as follows:

1. Yellow ore containing < 4% Zn. This ore is subdivided into three ore types: ore containing bornite, high grade ore at > 1% Cu, and low grade ore at <1% Cu
2. Black ore, < 5% Cu and > 3.2% Zn, and containing bornite
3. Clastic ore, which is metallurgically challenging and characterised by:
   - fine grained sphalerite interwoven with massive sulphides, with chalcopyrite, and containing high lead content
   - further subdivision into two types, i.e. with (CO +) and without (CO -) bornite.

Yellow ores can be processed to produce Spec copper and zinc concentrates, whilst clastic and black ores produces a Non-spec concentrate high in both copper and zinc, but which cannot be separated.

The South Orebody contains two ore types, which are similar to the black and yellow ores from the Main Orebody:

1. Footwall (yellow) ore, which is similar to Main Orebody yellow ore, but containing lower grades of copper
2. Black ores, which are differentiated as either low zinc (< 8% Zn) or high zinc (> 8% Zn)

With these South Orebody ores being similar in mineralogy to the yellow and black ores from the Main Orebody, their treatment in the existing processing facilities at Çayeli is not expected to cause any metallurgical concerns.

17.3 Processing and recovery operations

The ÇBI ore processing facility consists of conventional crushing, grinding, selective flotation, and pressure filtration circuits. The facility is equipped with an online Yokogawa process control system and also an SGS Expert System.

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[11] Stated peak rate is not adjusted for unplanned mining dilution and mining recovery losses

A simplified block flowsheet of the plant is presented in Figure 17.1, along with a more detailed pictorial flowsheet in Figure 17.2. A photograph of the Çayeli surface operations is provided in Figure 17.3.

The processing facilities can be itemised into the following sub-processes:

- Stockpiling of run-of-mine ore in one of eight covered sheds (one for each ore type, from the two ore bodies).
- Ore reclamation by front end loader.
- Primary crushing in open circuit using a jaw crusher.
- Screening of crushed ore on a double deck vibrating screen.
- Secondary crushing of screen oversize material in a cone crusher, with crusher discharge then combined with the jaw crusher discharge and returned to the double deck screen.
- Conveying of screen undersize to a fine ore bin.
- Milling of crushed ore in a primary ball mill in open circuit, followed by secondary ball milling in closed circuit with hydrocyclones:
  - the target grind size is a P_{80} of 50 µm (70% passing 36 µm)
  - scats generated from the primary ball mill are transported off-site for recrushing by a contractor, and returned to the ore sheds
- Rougher and scavenger flotation of cyclone overflow slurry for copper recovery.
- Cleaner flotation of the copper rougher concentrates to produce a final concentrate.
- Conditioning of the copper rougher scavenger and cleaner scavenger tailings prior to zinc flotation.
- Rougher and scavenger flotation of copper flotation tailings for zinc recovery.
- Cleaner flotation of the zinc rougher concentrates to produce a final concentrate.
- Thickening of zinc rougher scavenger and cleaner scavenger flotation tails (final tailings).
- Pumping of thickened tailings to the mine backfill preparation plant, with excess tailings being discharged into the Black Sea.
- Backfill preparation by thickening and filtering to 85% solids, with cement addition and pumping to the underground mine using positive displacement pumps.
- Dewatering of copper and zinc concentrates by thickening and pressure filtration, followed by bulk transportation to off-site smelters.

Ancillary facilities are as follows:

- Reagent make-up and dosing systems to support the milling and flotation operations.
- Process water reticulation systems.
- Compressed air systems to support instrumentation and for automatic valve activation.
- Low pressure air systems, provided by blowers, for the flotation cells.

Figure 17.1 Simplified block flowsheet for the ÇBI ore processing facilities

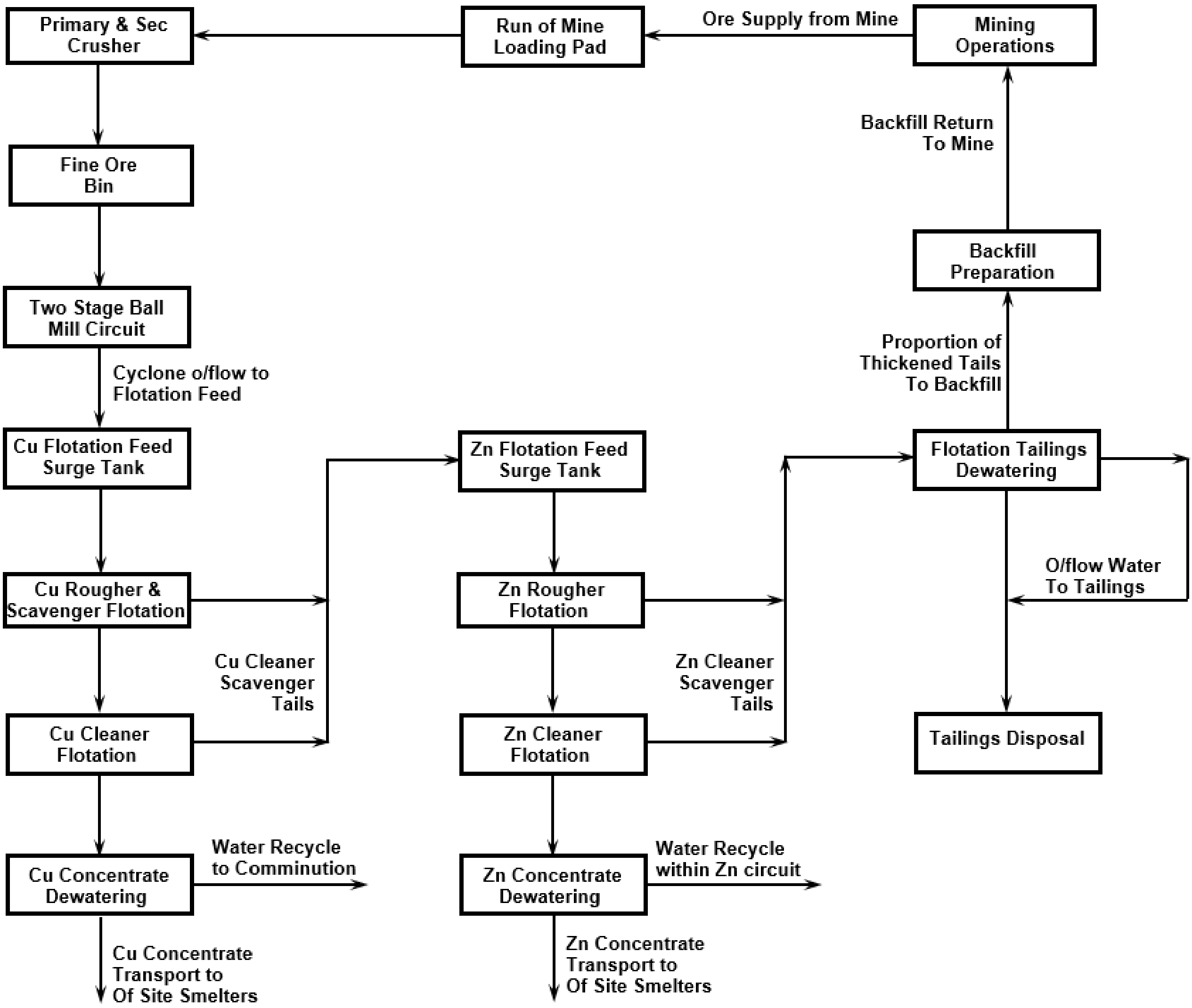


Figure 17.2 Pictorial flowsheet for the ÇBI ore processing facilities

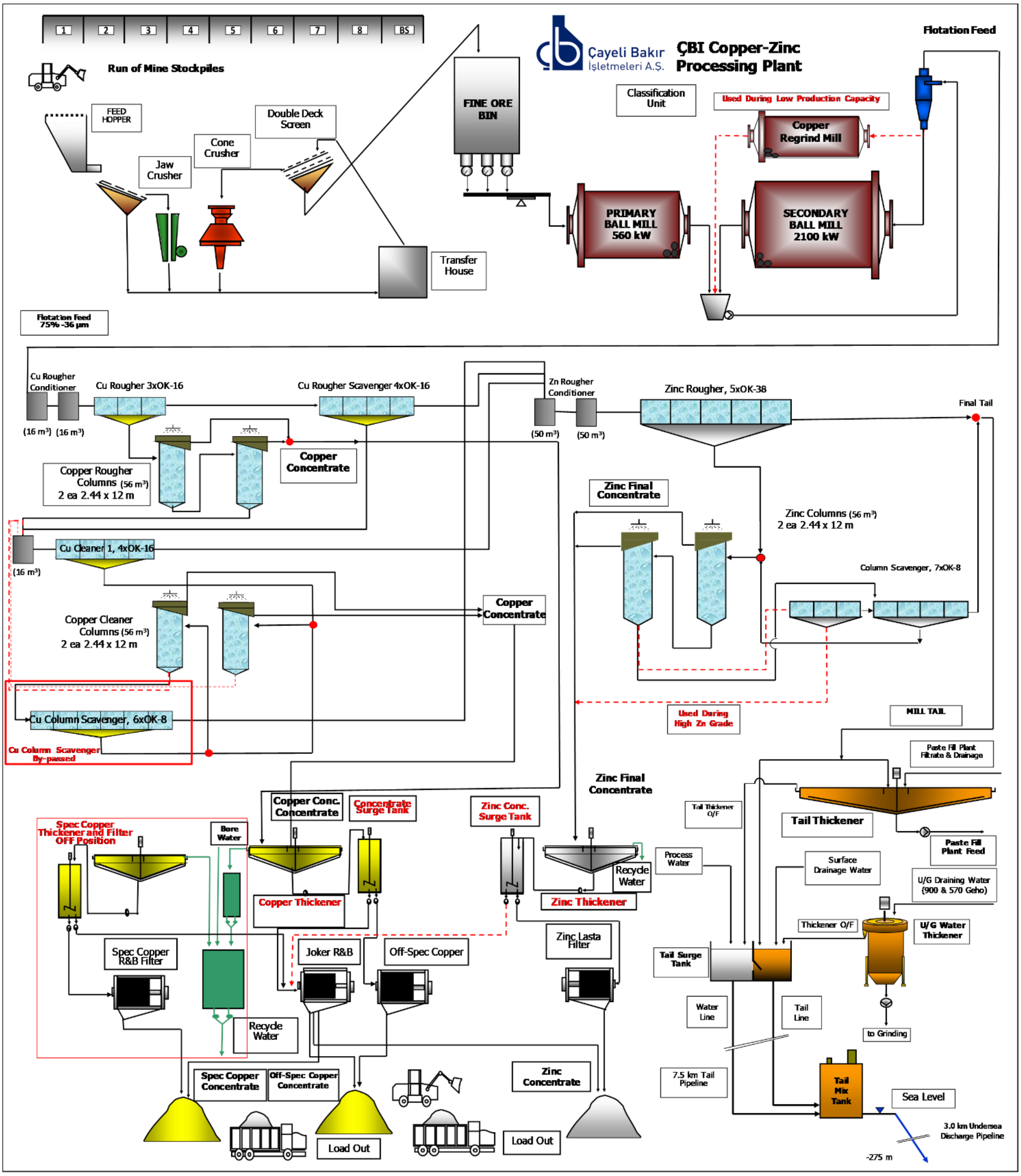
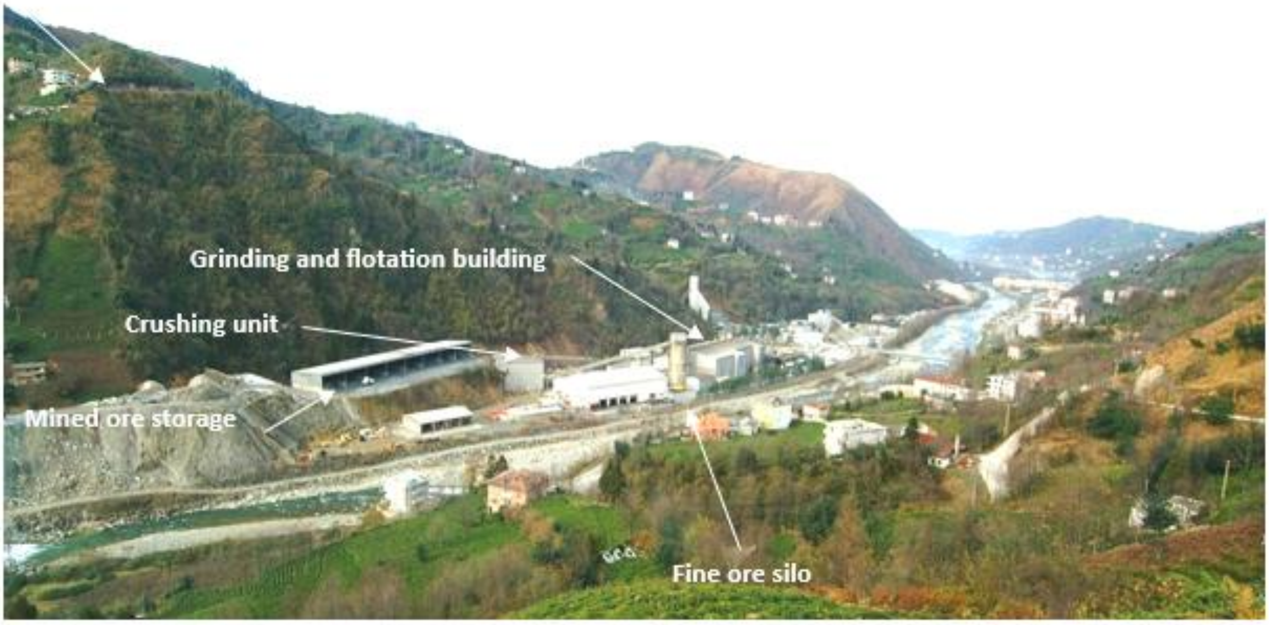


Figure 17.3 Çayeli Operations surface installations



17.4 Ore blending

17.4.1 Main Orebody

Ore selectively loaded and hauled from the Main Orebody is tipped into one of three Main Orebody surface storage compartments (i.e., of the eight compartments in total).

Plant feed is reclaimed by blending these ore types according to mineralogical composition, and by copper and zinc grades. Essentially, the mined ore types are blended into two plant feed types, depending on the secondary copper mineralisation and bornite minerals, the presence of which are determined by mineralogical studies on underground samples. If a mined ore type has either secondary copper minerals or bornite minerals, it is assigned the bornite attribute for processing.

Clastic ore is material that, from visual estimates, contains more than 15% sphalerite fragments in massive sulphides. These fragments are generally fine-grained and form intergrowths with chalcopyrite, which may affect the copper recoveries.

The Black and Yellow ore types are differentiated on the contained zinc grade and are mined separately to allow for optimal grade blending from the stockpile bins on the surface. These two ore types are referred to as Spec ore. Black ore is defined as material with more than 4.5% Zn and a Cu: Zn ratio of less than one. Yellow ore consists of copper-rich, zinc-poor sulphides. Yellow ore is comprised of approximately one-half massive sulphides and one-half stockwork material; stockwork mineralisation generally contains very little sphalerite.

Historically, two copper concentrates were produced from four different ore-type campaigns:

1. Spec campaign – yellow ore (YO +, YO -) and low grade yellow ore (LYO -)
2. Non-spec campaign – clastic ore (CO +, CO -) and black ore (BO)
3. Bornite yellow campaign (BYO)
4. Bornite clastic campaign (CO +, *aka* BCO)

In the future treatment of remnant ore from the Main Orebody between 2025 and 2027, and in 2035 and 2036 (Item 16; current life of mine production and plant feed schedule), there will be no separate BYO or BCO feed campaigns. Spec concentrate will be generated from campaign 1, whilst Non-spec concentrates will be generated from campaign 2.

Spec Cu concentrate will be concentrate produced from the combined treatment of yellow ores (i.e., YO - and LYO -; there is no YO+ in the current schedules) and will contain no zinc. Non-spec Cu concentrate will be concentrate produced from the combined treatment of clastic ore, bornite clastic ore and black ore (CO - , CO + (*aka* BCO) and BO). This concentrate will contain zinc and elevated levels of silver.

17.4.2 South Orebody

The South Orebody comprises two ore types only; footwall ore (SYO), which is very similar to yellow ore in the Main Orebody, and black ore (SBO). As concluded in Item 13.3, these two discrete ore types are not to be blended. Item 13.4, however, concludes that the SBO high and low zinc feed could be blended in a 1:1 ratio.

Hence, two different campaigns should be used for treating South Orebody ores, producing two copper concentrates, similar to the Main Orebody ore treatment:

1. Spec campaign – footwall yellow ore (SYO) treatment
2. Non-spec campaign – black ore (SBO) consisting of a blend of 50% high zinc and 50% low zinc feed

As is the case for Main Orebody feed, zinc concentrate will be generated from campaign 2 only.

17.5 Ore processing

17.5.1 Primary and secondary crushing

Crushing is carried out in two stages. Blended ore reclaimed from the surface ore bins is fed to the jaw crusher operating at a closed side setting (CSS) of 150 mm, and in open circuit, with crusher discharge being conveyed to a double deck screen.

The top screen oversize material (+ 30 mm) and the bottom screen oversized material (-30 + 18 mm) is fed to an HP 300 secondary cone crusher, operating with a CSS of 20 mm. The cone crusher operates in closed circuit with the double deck screen.

Discharge from the cone crusher and material from the jaw crusher are combined and returned to the double deck screen. Undersize material from the screen is conveyed to a 2,500 t capacity fine ore bin. A wet scrubber controls the dust levels in the crushing building.

17.5.2 Grinding

Crushed ore at nominally 80% passing 9 mm (100% passing 18 mm) is conveyed to a 560 kW, 3.2 m diameter by 4.3 m effective grinding length (EGL) ball mill operated in open circuit. Primary and secondary ball mill discharges are combined and pumped to a cyclone battery for classification. Cyclone underflow feeds a 2,100 kW secondary ball mill (4.4 m diameter by 7.2 m long) operating in closed circuit. Both mills are overflow mills, and are rubber lined. The primary mill ball charge comprises 90 mm and 80 mm balls, and the secondary mill 40 mm and 25 mm balls.

The milling facilities comprise four ball mills (see Figure 17.2). One of these mills is no longer operational, and the third ball mill is a secondary ball mill, smaller than the 2.1 MW mill described above, which can be brought into circuit in place of the larger mill when treating softer ores or for lower throughputs. This is achieved by swinging over the cyclone feed pipe from one set of cyclones to the other, to redirect cyclone underflow to the new operating mill.

Cyclone overflow, at 80% passing 50 µm (70% passing 36 µm), gravitates to the copper rougher circuit as flotation feed. Collector and lime are added to the primary ball mill if necessary; facilities exist to feed three different collectors to the mill feed hopper, to cater for the treatment of the different ore types. Depressant is also added to the primary ball mill.

Energy consumption for grinding Main Orebody ores averages 21 kWh/t; steel ball consumption averaged 1.82 kg/t of ore in 2023 and 1.67 kg/t in 2024. These numbers are projected to increase to 23.1 kWh/t and 2.5 kg/t respectively when treating South Orebody ores.

Scats comprising approximately 3% of the mill feed are generated by the first stage ball mill. These scats are transported off-site for crushing from -12 mm to between 1 and 2 mm by a contractor. Crushed scats are returned to the ore stockpiles.

17.5.3 Copper flotation

The copper flotation consists of several sets of conventional rougher and scavenger cells, in addition to rougher and cleaner column cells. Flotation feed slurry is 80% passing 50 µm, at a slurry density of about 30% solids.

The copper rougher circuit comprises three 16 m^3 Outokumpu mechanical cells followed by four 16 m^3 Outokumpu rougher scavenger cells. The pH in the copper circuit is adjusted to 11.8 to 12.0 with lime addition in the grinding mills. Reagents are added to the slurry feed in the mills or in the copper conditioning tank. Frother is added at various points throughout the flotation circuit.

Copper rougher concentrate is pumped to the copper rougher columns for cleaning. Two columns are installed, each being 12 m tall and 2.44 m diameter, and operating in parallel or in series depending on the concentrate grades achieved. Concentrate from the copper rougher columns is the final Spec copper concentrate.

Rougher scavenger concentrate and rougher column tails are pumped to a cleaner conditioning tank.

Overflow from the cleaner conditioner tank gravitates to the first of four 16 m^3 Outokumpu cells. Concentrate from these cells is pumped to the two copper cleaner columns, which are sized identically to the two rougher columns.

The cleaner column concentrate is combined with the rougher column concentrate as a final concentrate, and the tails returned to the cleaner conditioner. Previously, these tails were pumped to the first of six 8 m^3 Outokumpu cleaner scavenger cells; this circuit has now been permanently bypassed.

Copper cleaner tails are combined with rougher scavenger tails and comprise the feed to zinc flotation.

17.5.4 Zinc flotation

The copper rougher scavenger tail, and the copper cleaner tails are combined and pumped to two agitated zinc flotation conditioners. Copper sulphate is added to the first conditioner and collector to the second. Overflow from the second conditioner gravitates to five 38 m^3 Outokumpu flotation cells for zinc rougher flotation. The zinc rougher tail gravitates to the final tails pump box.

Zinc rougher concentrates are pumped to a cleaner conditioner from where they gravitate to the first of seven 8 m^3 Outokumpu cells for cleaning. Zinc cleaner concentrates are further cleaned in flotation columns, whilst the tails are transferred to the final tails pump box.

The zinc column circuit consists of two 12 m tall and 2.44 m diameter columns (identical in size to the copper columns), operating in series or in parallel, depending on the zinc feed grade. Columns tails are directed to the column scavenger circuit, consisting of seven 8 m^3 Outokumpu cells. Concentrate from these cells is returned to the column feed stream, whilst the tails are pumped to the final tails pump box.

Column concentrate is pumped to the zinc concentrate dewatering circuit as final zinc concentrate.

17.5.5 Concentrate production

When treating Main Orebody ores, two copper concentrates are produced from two different ore type campaigns:

- Spec Cu concentrates produced from treating yellow ores (YO - and LYO -)
  - yellow ore (YO +) is not present in the currently scheduled plant feed
- Non-spec Cu concentrates produced from:
  - Clastic ores (with and without bornite, i.e. CO + (*aka* BCO) and CO -)
  - Black ore (BO)

The Non-spec Cu concentrate contains Zn contamination, and hence the number of customers for this type of product is limited. Zinc concentrates are produced from clastic ores (CO + and CO -) and black ore (BO). All of these concentrates are considered to be within specification. No zinc concentrates are produced from yellow ores. Table 17-1 lists the average ore type recoveries and concentrate grades for Main Orebody plant feeds.

Table 17-1 Recoveries and concentrate production from Main Orebody ore feed

	Recoveries, %		Concentrate grades				
	Cu	Zn		% Cu	% Zn	Ag ppm	Au ppm
Yellow Ore (Spec con.)	92.0		**Cu**	22.0	2.4	45.0	1.5
		30.0	**Zn**	5.0	40.0	0.0	0.0
Black & Clastic Ores	84.0		**Cu**	17.0	12.0	94.0	1.7
(Non-spec Cu con.)		67.0	**Zn**	5.0	40.0	94.0	0.0

A similar range of concentrates is expected to be produced when treating South Orebody plant feed, as listed in Table 17-2.

Table 17-2 Recoveries and concentrate production from South Orebody ore feed

	Recoveries, %		Concentrate grades				
	Cu	Zn		% Cu	% Zn	Ag ppm	Au ppm
Footwall (Spec con.)	92.0		**Cu**	23.0	2.5	20.0	1.3
Zinc Ores (Blend 2)	60.0	75.0	**Cu**	19.0	10.0	40.0	5.0
(Non-spec Cu con.)			**Zn**	5.0	50.0	65.0	3.0

The BO ores occur as high and low zinc ores (high zinc being > 8%), and the zinc concentrates described in the table assume a 50:50 blend of high and low zinc ores, which was identified as being the optimum blend for treating black ores from the South Orebody. No zinc concentrates will be produced when treating footwall material.

Ore characteristics will have changed over time, up to the present day. However, these concentrate figures indicate that the plant is not constrained by capacity and will certainly be able to handle the anticipated production from the South Orebody.

Despite gold recoveries and concentrate grades being listed in Table 17-1 and Table 17-2, an overall average gold grade is not reported in the Mineral Resource and Reserve statements. The reason for this is explained in Item 14.

17.5.6 Copper and zinc concentrate dewatering and transport

The copper and zinc concentrate slurries are thickened in separate, but identical circuits. The Cu concentrate thickener is used for both the Spec concentrate and Non-spec concentrates that are recovered in separate campaigns.

The copper and zinc circuits both have conventional thickeners with a diameter of 16.0 m. Thickener underflow slurries are pumped to their respective filter feed surge tanks. Three tanks are installed, one per filter, each with a capacity of 100 m^3.

Pressure filters are used for filtration of the thickened copper and zinc concentrates. There are four plate and frame filters presses; one for Spec copper, one for Non-spec copper, one for zinc, and the fourth is a standby unit for either copper or zinc filtration. The final copper concentrates typically contain 11% moisture and final zinc concentrates contain 8% moisture.

Filtered copper and zinc concentrates drop directly down to a concrete reinforced 1,000 tonne capacity loadout area. Copper and zinc concentrates are loaded onto trucks for transport to the Rize port, approximately 28 km from the mine site. Each truck is weighed on both the site scale and the Rize port scale. The concentrate trucking is under a contract with a local contractor.

17.5.7 Port facilities

The Rize port facilities are located approximately 26 km from the Operations site. Views of the port are shown in Figure 17.4.

The facilities comprise three separate sheds for storage of the different concentrates, and a loading ramp for feeding concentrate to a series of conveyors leading to a ship loading conveyor. Transfer of concentrate from the sheds to the conveyors is by front end loader.

Washdown is collected and treated in a series of settling ponds with the addition of coagulant and flocculant. Water overflow from the ponds is analysed to ensure environmental compliance and discharged into the municipality wastewater system. The small amount of settled solids are added to the Non-spec copper concentrate.

Figure 17.4 Rize port facilities

17.5.8 Paste fill plant

The surface paste fill plant is designed to deliver paste to the underground mine at a maximum rate of 50 m^3/h. The plant building is positioned close to the mine decline portal and approximately 300 m away from the concentrator building.

Tailings from the zinc flotation circuit are pumped to the paste fill plant thickener at about 20% solids by weight, where they are thickened to 65% to 75% solids by weight. This thickener is a 16 m diameter conventional thickener and is located adjacent to the concentrate thickeners in the process plant. Underflow slurry from the thickener is pumped to an agitated surge tank ahead of two dewatering filters. Either one or both vacuum disc filters are operated, depending on the required paste fill rate.

The disc filters reduce the moisture content of the tailings to approximately 85% to 86% solids by weight. The filter cake drops onto a reversible belt conveyor, which delivers it to a surge hopper and a conditioner tank, where the cake is agitated and mixed with water or slurry until it forms a 7 inch to 8 inch (17.8 cm to 20.3 cm) slump, at about 80% to 84% solids. Cement is added to the paste mass; the cement content is determined by the specific requirement of each stope and ranges from 5% to 8.5%, with an average of 6.7%.

The final paste mass is pumped to the underground stope voids by Putzmeister positive displacement pumps, through one of four main pipelines.

A single operator controls the plant from a control room equipped with a computerised monitoring system. Every two hours, a sample is taken for the paste slump control.

TV cameras are used for monitoring paste placement in stopes.

17.5.9 Tailings disposal

Owing to topographical constraints there is no tailings disposal and storage facility within the Operations licence area.

Excess tailings not required for paste fill in the mine, are piped to a mix tank on the Black Sea coast via a 7.5 km long overland pipeline. In what is referred to as Deep Sea Tailings (DST), the tailings are then piped out to sea through a 3 km long pipeline and discharged at a depth of 275 m.

The Black Sea aquatic environment is anoxic below a depth of 150 m and does not support any form of marine life. The seawater is naturally rich in hydrogen sulphide and deficient in dissolved oxygen.

Tailings production in 2024 was 630,789 of which 45% was discharged into the Black Sea. Since the commencement of operations, a total of 15.4 million tonnes of tailings has been discharged via DST placement.

17.6 Consumables

A list of consumables and their consumption rates are presented in Table 17-3.

The consumption rates for treating Main Orebody ores have been derived from historic data, and for South Orebody ores from testwork. Consumption rates differ between the various ore types, and the numbers presented below represent a weighted average consumption, considering the proportion of each ore type in the process feed over the remaining life of mine.

Estimated annual consumption rates are based on a throughput rate of up to 850,000 tpa, and are indicative only, as the Main Orebody will not achieve this production rate in the remaining mine life, and the South Orebody will only produce at this level for several years.

Table 17-3 Reagent and steel ball consumption rates

Ore Type		MOB (2024 figures)		SOB	
Item		Consumption Rate, g/t ore	Annual Usage, tpa	Consumption Rate, g/t ore	Annual Usage, tpa
Grinding Media	**Total**	**1.673**		**2,499**	
Primary Mill Balls	80 mm	217	184	314	267
SBM Mill Balls	40 mm	727	618	1,100	935
SBM Mill Balls	25 mm	729	620	1,085	922
Reagents					
Collector - Cu and Zn	SIPX	41.6		92.4	79
Collector - Cu	3418A	8.0	7		
Collector - Cu	A-208	27.1	23	13.7	12
Frother	MIBC	26.6	23	31.6	27
Zn Activator	CuSO4	155.3	132	263.7	224
Deppressant	SMBS	131.0	111	121.1	103
Lump Lime Bulk		534.5	454	1000.0	850
Flocculant		2.6	2	2.5	2

The reagent requirements for treating the South Orebody ores differ in quantity, but are the same reagents as used currently for treating Main Orebody ores. No new reagent make-up facilities will be required for treating South Orebody ores.

17.7 Energy requirements

Power to the mine site is provided by a 31.5 kV line connected to the Turkish national grid system. This line comes into a substation located north of Madenli which is equipped with two 12 MVA transformers.

Power is then distributed to the milling circuit and into the plant and other site facilities from one of these transformers.

Energy consumption within the process plant for treating ores from the South Orebody will be similar to that in the past when treating ores from the Main Orebody, with an overall consumption rate of 45.2kWh/t of ore treated. However, the South Orebody ores are harder than those from the Main Orebody, and energy requirements for comminution will increase from an average of 21kWh/t to 23.1kWh/t.

No changes in the power reticulation infrastructure are required for the future treatment of ores from the south ore body.

17.8 Water usage

Process water for the plant is sourced from bores positioned alongside the Büyükdere River. The bores are monitored from the plant control room and the abstraction is recorded with ultrasonic flow metres.

Process water is also sourced from rainfall run-off and contact water emanating from the Operations (Including the underground mine), which is collected in several receiving ponds located around the Operations site. This reclaimed water is also used for dust suppression around the site.

Flotation tailings are thickened prior to being used for backfill with excess tailings being discharged to the Black Sea, as described in Section 17.5.9. Water from the tailings dewatering circuit is also discharged to final tailings and not recycled to the process plant, because residual flotation reagents from the zinc circuit would affect the operation of the copper flotation circuit.

Excess water from the process plant is discharged with tailings to the DST mix tank at the Çayeli coast (Figure 18.3) and then discharged to the sea.

Domestic wastewater is treated in a biological waste-water treatment plant with a capacity of 100 m^3/day. This water is added to the tailings stream and discharged to the Black Sea.

No increase in water consumption is expected with the treatment of ores from the South Ore Body, and no changes to the water reticulation circuits are envisaged.

17.9 Condition of the process plant equipment

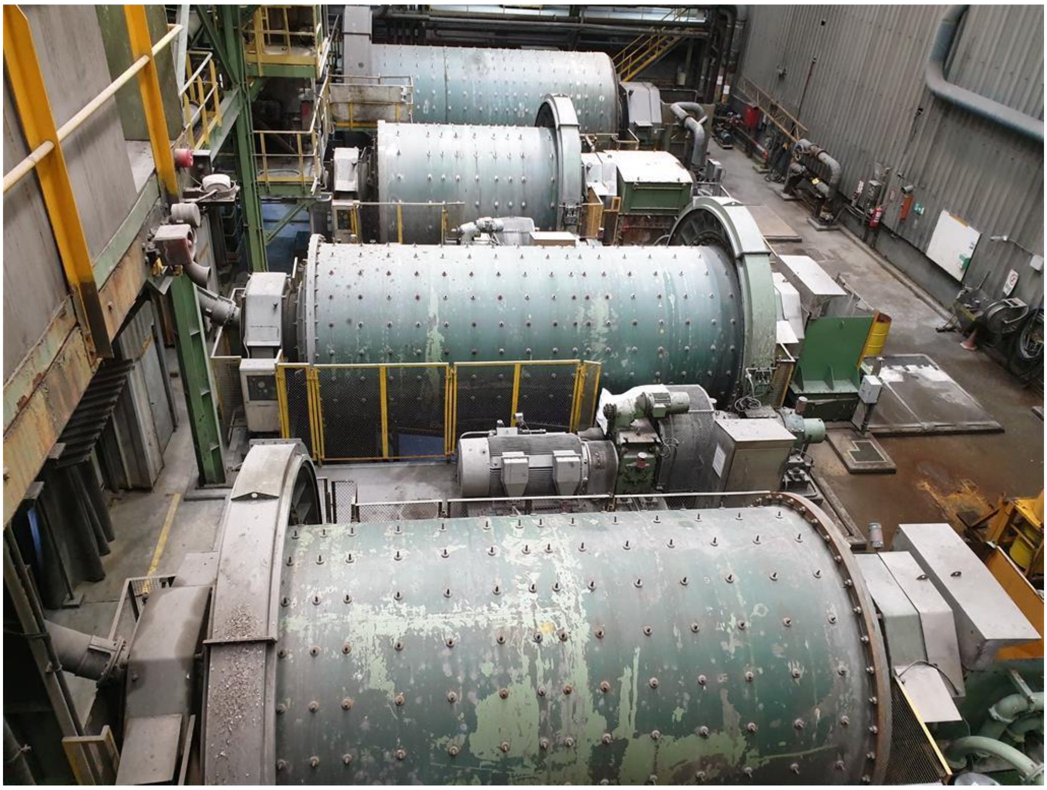
As noted previously, the processing facilities have been in operation since 1994. Whilst housekeeping and maintenance of the plant has been very good; it is inevitable that the condition of some equipment has deteriorated. There has been a programme in place for several years to refurbish or replace badly corroded equipment, and there is now a mix of almost new equipment within the original plant.

Figure 17.5 to Figure 17.9 are photographs of various areas of the circuit, taken in mid-2024, to illustrate the general condition of the facilities.

The primary jaw crusher installation appears to be in good condition, but the crusher itself is old, and spare parts are difficult to obtain. It is planned to replace this crusher in 2027; a capital provision is listed in Section 21.2.

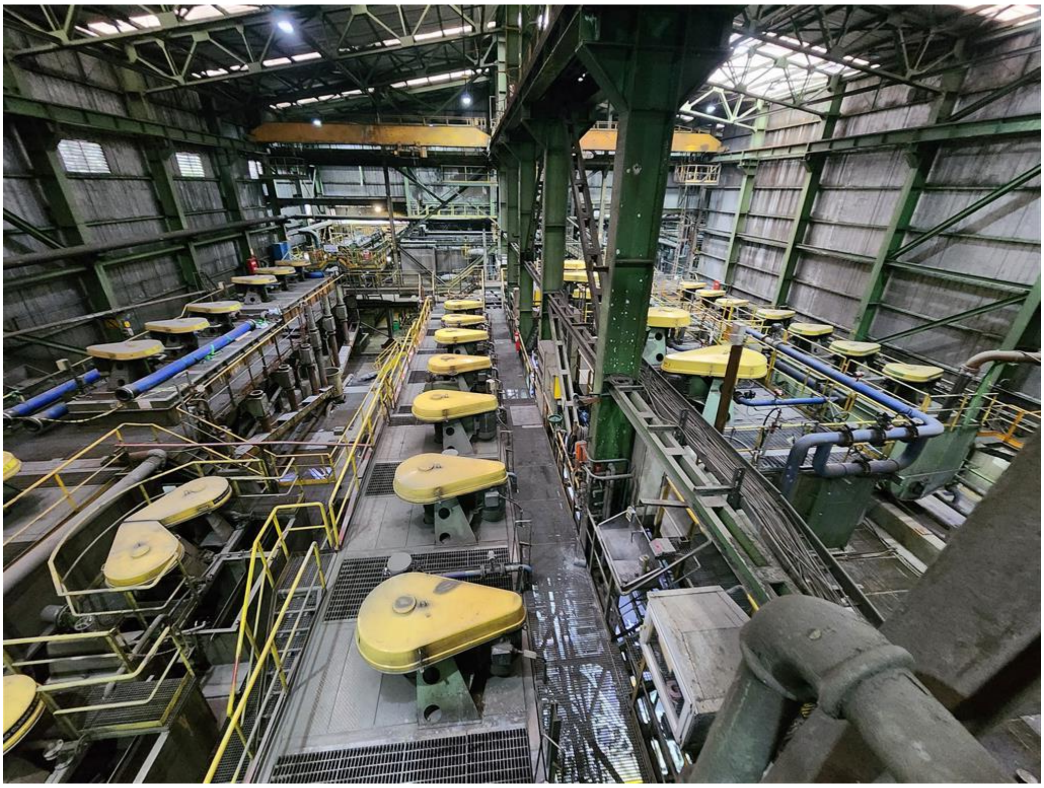
Figure 17.5 Primary crusher installation

Figure 17.6 Milling circuit



In the milling circuit, the smallest mill (second from the top in Figure 17.6) is the primary ball mill. The mill at the bottom of the picture, mill 4, is no longer being used.

Figure 17.7 Flotation circuit



The cells on the left of the picture in Figure 17.7, are for copper flotation, with zinc flotation on the right of the picture.

Figure 17.8 Cu Rougher scavenger cells in good condition

Figure 17.9 Cu rougher cleaner cells requiring replacement

Figure 17.10 Corrosion around the top of flotation columns; requiring replacement

17.10 Plant throughput and previous operating data

The current LOM production plan (see Item 16) describes the ore production profile from the Main and South Orebodies from 2025 through to 2036. This schedule is reproduced in Table 17-4. Before mining dilution and recovery adjustment, the peak throughput is 850 ktpa between 2027 and 2029.

Table 17-4 Annual production plan 2025 to 2036

Orebody	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Main Orebody total														
Dil'd and Rec'd tonnes	t	576,187	207,942	99,772	85,381								120,469	62,623
Av. Cu	%	3.12	3.93	2.73	1.95								3.05	2.78
Av. Zn	%	3.08	4.86	0.68	0.79								2.95	4.38
Av. Ag	g/t	26.57	40.92	8.52	4.16								31.56	28.59
Av. Au	g/t	0.33	0.44	0.25	0.15								0.37	0.27
South Orebody total														
Dil'd and Rec'd tonnes	t	6,736,066	239,525	692,129	756,013	841,394	841,394	742,406	643,419	593,926	494,937	445,444	275,482	169,997
Av. Cu	%	1.37	1.57	1.38	1.18	1.34	1.27	1.50	1.52	1.63	1.57	1.18	0.94	1.11
Av. Zn	%	2.28	0.38	1.11	2.54	2.68	3.11	2.86	1.91	2.01	2.32	2.45	3.28	0.08
Av. Ag	g/t	8.92	2.21	7.12	8.62	9.59	10.40	11.66	8.24	8.04	8.92	11.34	9.98	2.17
Av. Au	g/t	0.59	0.25	0.37	0.53	0.61	0.67	0.68	0.61	0.52	0.62	0.87	0.82	0.22
Combined total														
Dil'd and Rec'd tonnes	t	7,312,253	447,467	791,901	841,394	841,394	841,394	742,406	643,419	593,926	494,937	445,444	395,950	232,621
Av. Cu	%	1.51	2.67	1.55	1.26	1.34	1.27	1.50	1.52	1.63	1.57	1.18	1.59	1.56
Av. Zn	%	2.34	2.46	1.06	2.36	2.68	3.11	2.86	1.91	2.01	2.32	2.45	3.18	1.23
Av. Ag	g/t	10.31	20.20	7.29	8.17	9.59	10.40	11.66	8.24	8.04	8.92	11.34	16.55	9.28

The production profile shows a ramping down of plant feed from the Main Orebody from 2025 to 2027, and with some further remnant ore production in 2035 and 2036. Plant feed from the South Orebody increases from 2025 to 2026 by a factor of 290%, then by 109% and 111% respectively, to the first peak from that source in 2028.

The processing facilities will be required to handle the single source peak in 2028 and 2029. Up until that time, the different ore types will be by treated by campaigning ores that produce Spec and Non-spec concentrates. It is assumed that it will be possible to treat similar ores from the Main Orebody and the South Orebody together.

Other than the grouping of individually mined ore types into the respective Spec and Non-spec plant feed types, there is no “cross-blending” of Main Orebody mined ores between the Spec and Non-spec types, as has been evident in past life of mine production plans. Furthermore, there is no cross-blending required between the South Orebody plant feed types. As noted in Item 15.6, the mined and grouped ore types are not intermixed during surface reclaim operations and hence the as-mined groupings into Spec and Non-spec types are the same as the annual plant feed groupings as listed in in Table 17-5.

Table 17-5 Annual plant feed of Spec and Non-spec ores 2025 to 2036

	Units	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Combined orebodies Spec plant feed														
Dil'd and Rec'd tonnes	t	**5,186,711**	333,907	685,906	612,469	546,554	512,599	486,215	473,362	452,850	353,489	294,308	226,671	208,380
Av. Cu	%	**1.74**	2.16	1.72	1.54	1.77	1.75	1.82	1.69	1.86	1.78	1.60	1.71	1.35
Av. Zn	%	**0.22**	0.32	0.18	0.04	0.08	0.05	0.10	0.06	0.47	1.01	0.26	0.20	0.23
Av. Ag	g/t	**3.04**	5.55	3.91	1.84	2.63	2.60	3.01	2.25	2.21	3.12	3.91	3.86	3.24
Combined orebodies Non-spec plant feed														
Dil'd and Rec'd tonnes	t	**2,125,542**	113,559	105,994	228,925	294,839	328,795	256,192	170,057	141,075	141,448	151,136	169,280	24,241
Av. Cu	%	**0.95**	4.17	0.46	0.51	0.56	0.53	0.88	1.04	0.89	1.05	0.36	1.41	3.33
Av. Zn	%	**7.52**	8.76	6.70	8.57	7.50	7.87	8.12	7.06	6.98	5.57	6.72	7.16	9.85
Av. Ag	g/t	**28.07**	63.28	29.20	25.12	22.48	22.57	28.06	24.92	26.77	23.44	25.80	33.53	61.21
Combined orebodies plant feed														
Dil'd and Rec'd tonnes	t	**7,312,253**	**447,467**	**791,901**	**841,394**	**841,394**	**841,394**	**742,406**	**643,419**	**593,926**	**494,937**	**445,444**	**395,950**	**232,621**
Av. Cu	%	**1.51**	**2.67**	**1.55**	**1.26**	**1.34**	**1.27**	**1.50**	**1.52**	**1.63**	**1.57**	**1.18**	**1.59**	**1.56**
Av. Zn	%	**2.34**	**2.46**	**1.06**	**2.36**	**2.68**	**3.11**	**2.86**	**1.91**	**2.01**	**2.32**	**2.45**	**3.18**	**1.23**
Av. Ag	g/t	**10.31**	**20.20**	**7.29**	**8.17**	**9.59**	**10.40**	**11.66**	**8.24**	**8.04**	**8.92**	**11.34**	**16.55**	**9.28**

Annual average (diluted) copper feed grades reach a maximum of 1.63% Cu in 2032, but peak grades are 2.16% in Spec feed in 2025, and 4.17% Cu in Non-spec ores in the same year.

Similarly, average (diluted) zinc grades peak of 3.00% in 2029 and 2030, when treating a higher proportion of zinc ores from the South Orebody with elevated Non-spec zinc grades.

Operating data for the previous five years, when treating ores from the Main Orebody, is presented in Table 17-6.

Table 17-6 Plant operating data, 2019 to date

	2019	2020	2021	2022	2023	2024
Feed to Process						
Tonnes Milled	915,885	776,650	815,026	720,208	746,802	691,328
% Copper	2.09	2.04	1.96	1.71	1.62	1.79
% Zinc	1.55	1.62	1.81	1.14	1.09	1.07
Cu Concentrates - Total						
Tonnes Produced	88,827	72,195	70,809	55,197	54,184	54,257
Cu %	19.01	18.90	19.38	19.62	20.21	20.62
Zn %	6.96	8.27	8.18	6.70	5.91	6.56
Au, g/t	0.81	0.86	0.97	0.90	0.97	1.37
Ag, g/t	77	91	97	87	61	88
'Off-Spec' Cu Cons						
Tonnes Produced	58,373	53,854	58,639	32,243	24,041	24,194
Cu %	17.56	17.71	18.32	17.25	18.03	17.78
Zn %	8.72	9.66	9.75	9.51	9.6	11.53
Au, g/t	0.98	0.98	1.13	1.20	1.45	2.04
Ag, g/t	94	104.81	112	127	94	152
'Spec' Cu Cons						
Tonnes Produced	29,576	16,695	18,281	26,184	30,575	30,070
Cu %	21.83	22.75	22.59	22.51	21.92	22.9
Zn %	3.47	3.87	3.44	3.42	3.05	2.57
Au, g/t	0.49	0.47	0.44	0.52	0.56	0.84
Ag, g/t	44	45	46	47	35	36
Zn Concentrates						
Tonnes Produced	12,135	10,433	17,074	7,488	8,151	6,296
Cu %	3.70	4.47	4.51	5.69	3.20	4.79
Zn %	45.31	44.08	41.22	39.37	43.31	40.21
Ag, g/t	129	135	138	152	116	152
Tailings						
Tonnes Produced	814,924	694,022	727,143	657,524	684,467	630,775
Cu %	0.22	0.25	0.20	0.16	0.13	0.14
Zn %	0.31	0.29	0.26	0.24	0.22	0.21
Recoveries						
Cu %	88.1	86.0	86.0	87.8	90.4	87.9
Zn %	38.8	36.6	47.8	35.8	43.4	45

The maximum throughputs achieved historically by the Operations were:

1. Feed 1.3 Mtpa in 2013 and 2014
2. Cu concentrate production 159,500 tpa in 2013
3. Zn concentrate production 105,000 tpa in 2010

17.11 Conclusions

The ore types generated from the South Orebody are expected to be metallurgically similar to the current types from the Main Orebody, but without the complications of bornite or clastic ores.

Treatment of South Orebody ores through the existing circuit at Çayeli does not require any modifications to the flowsheet, nor the introduction of new reagents.

No additional equipment will be required to handle the anticipated ore throughput or feed grades. For both the Main and South Orebodies, the existing operating cells and idle cells will be sufficient.

However, the facilities are old, which affects equipment reliability, and this poses a significant challenge for the processing plant. Approximately half of all cells need to be changed due to corrosion or having reached their lifespan.

A programme has been in place for some years to replace worn and corroded equipment, and this programme extends into the current life of mine timeframe, with over $6.6M of expenditure planned for equipment replacement.

Item 18 PROJECT INFRASTRUCTURE

ÇBI is an existing operation that has been mining and processing copper and zinc ores for more than thirty years. The associated infrastructure as required by these operations remains in place and includes sealed roads, power lines and substations, a process plant, site offices, workshops, tailings/paste fill disposal and waste storage facilities.

18.1.1 Processing plant

The mined ore is truck hauled to surface and dumped into one of the several ore blending bins. The ore is then selectively reclaimed into a bunker at the process plant, from where it is fed to a two-stage crushing facility.

The crushed ore is conveyed to a fine ore silo. From the silo, the crushed ore is conveyed further to the grinding unit, which includes two closed-circuit ball mills. The ground ore is then fed to the flotation system.

The copper and zinc flotation concentrates are subsequently fed to a thickener tank, followed by cleaning, filtration and final dewatering. Copper and zinc concentrates are loaded into road trucks and transported to the Rize port, approximately 26 km away from the mine site.

18.1.2 Power supply

Power to the Operations site is provided by a 31.5 kV line connected to the Turkish national grid system. This line comes into a substation located north of Madenli which is equipped with two 12 MVA transformers.

Power is then distributed to the milling circuit and into the plant and other site facilities from one of these transformers.

Distribution within the mine is described in Item 16.6.5.

18.1.3 Paste backfill plant

Information on the existing surface backfill plant is provided in Item 17.5.8. The tailings preparation facilities for underground backfilling and deposition into the Black Sea are shown in Figure 18.1.

18.1.4 Tailings disposal

That portion of the process tailings that is not used for underground paste filling is transferred via pipeline to a mixing tank on the Çayeli coast. The tailing is discharged from that tank into another pipeline extending out 3 km to an oxygen-free and macro-life-free environment at a depth of 275 m in the Black Sea.

The Black Sea is the world's largest water body in which the bottom waters never mix with the shallower waters (a condition known as meromictic). As a result, the deeper waters are completely anoxic (devoid of oxygen). The deep water anoxic environment is saturated with hydrogen sulphur (H_2S) below a depth of about 175 m. Because of the overlying chemocline layers (halocline, thermocline) there is consequently no tailings release to the surface (Figure 18.2). The chemistry of the discharged tailings and that in the anoxic depths of the Black Sea are similar (ZnS, Fe_S2).

Figure 18.1 Çayeli tailings preparation system (source: FQM)

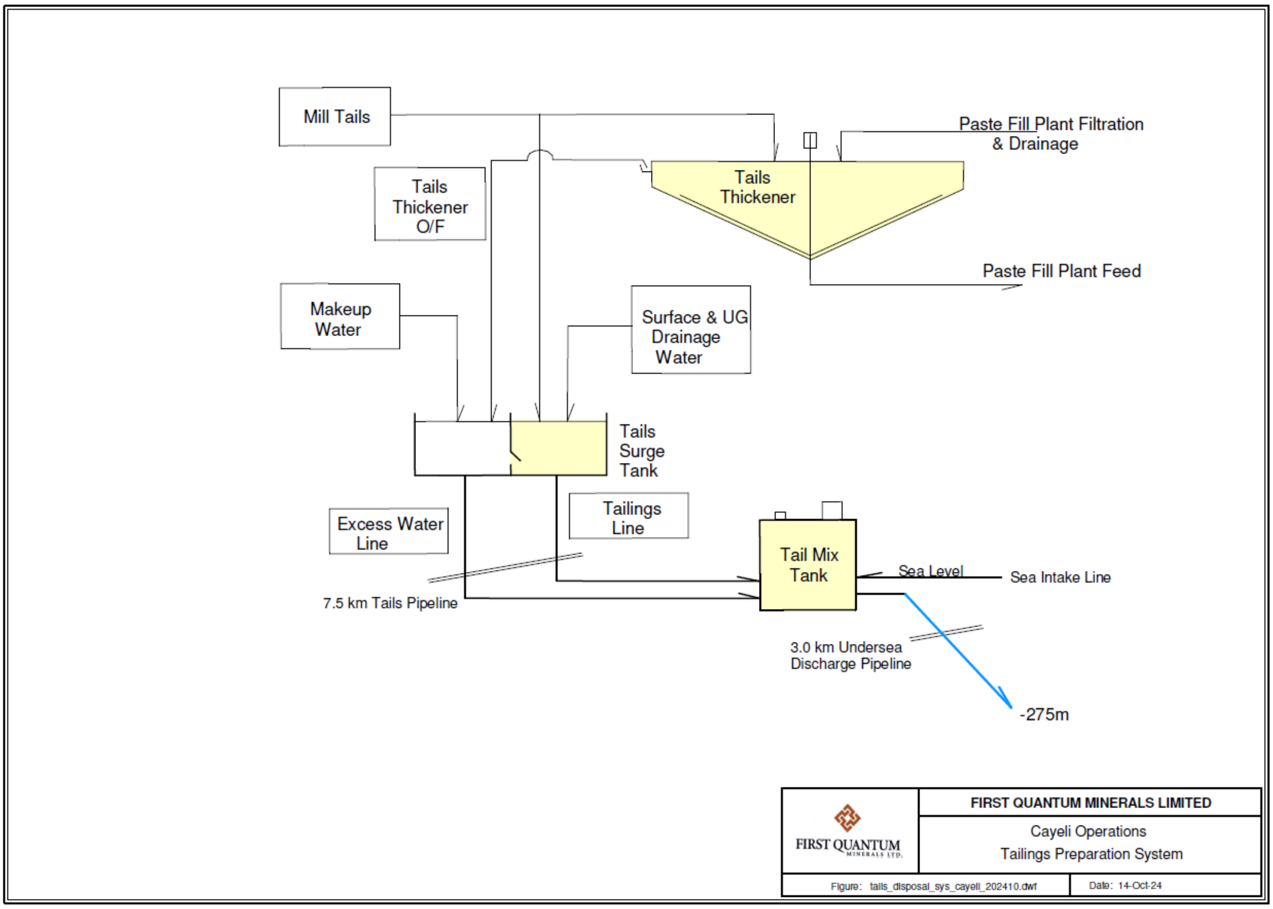
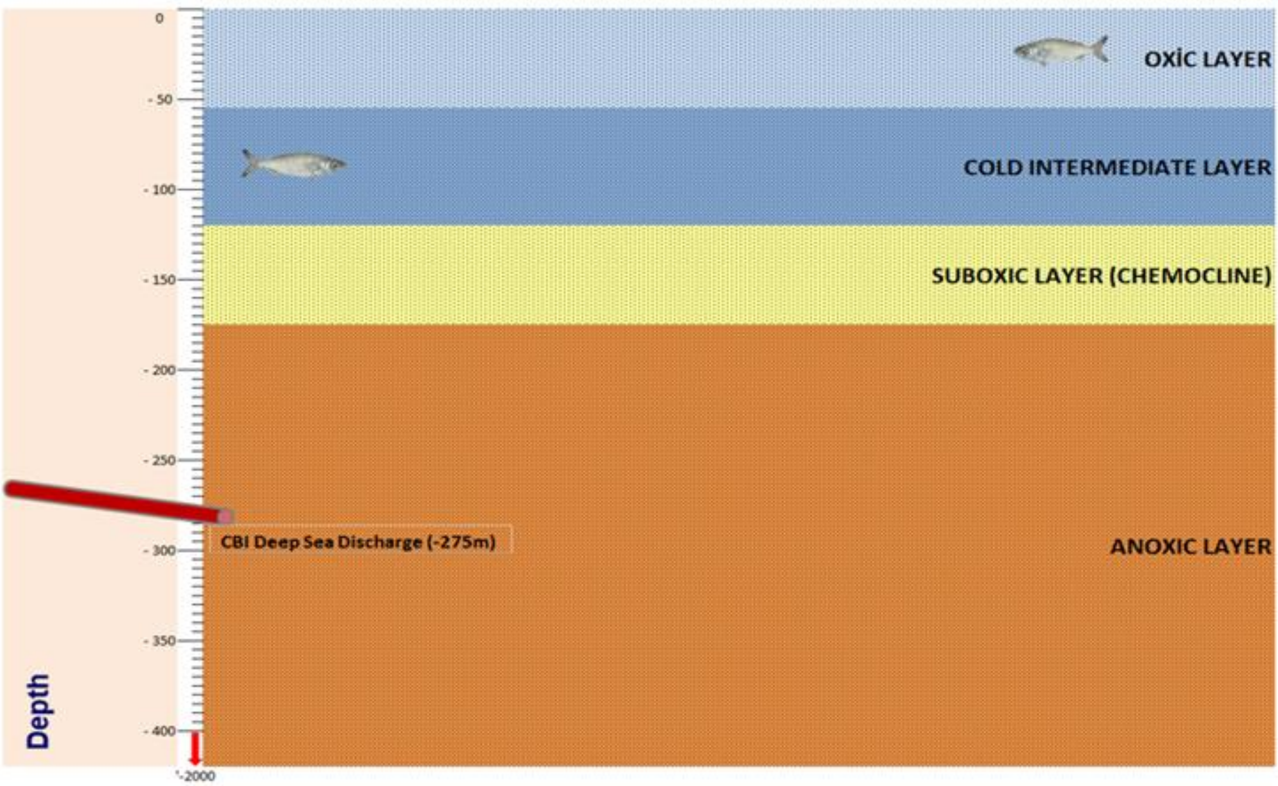
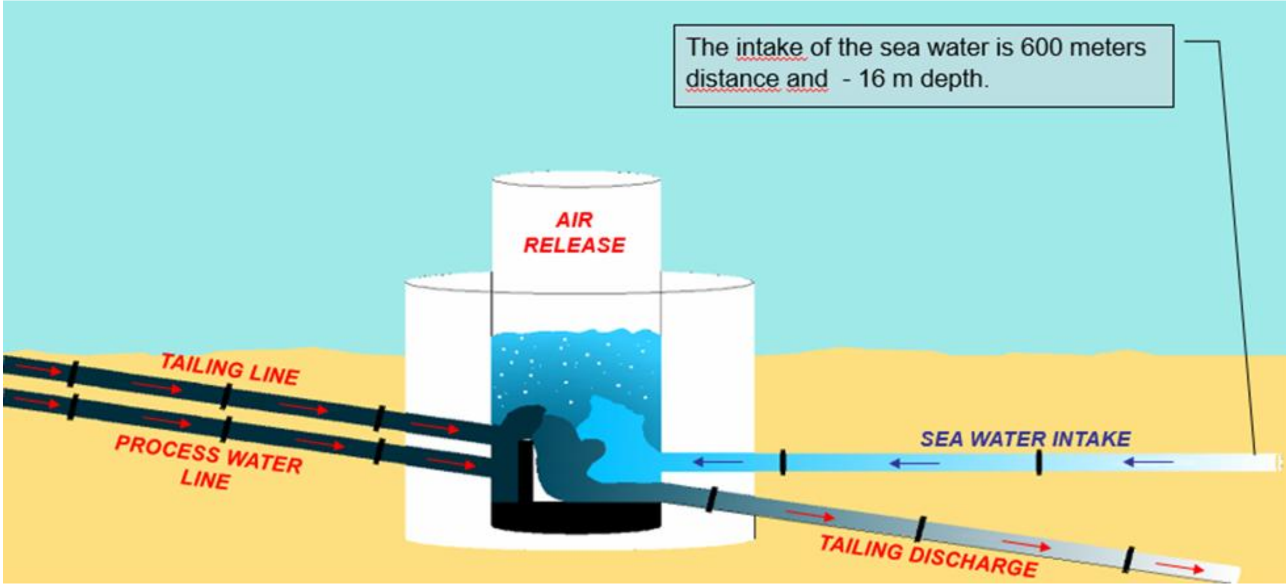


Figure 18.2 Black Sea water layers (source: ÇBI)



The primary purpose of the mix tank at the Çayeli coast is to release the air bubbles in the tailings piped from the processing plant. This prevents the tailing solids from rising to the surface after discharge at sea. Sea water is used to adjust the flow regime and protect the submerged pipeline from collapsing. A general arrangement of the mix tank is shown in Figure 18.3.

Figure 18.3 DST mix tank arrangement (source: ÇBI)



18.1.5 Port

Information on the existing port facilities is provided in Item 17.5.7.

18.1.6 Auxiliary infrastructure

Auxiliary infrastructure that already exists includes:

- an Operations administration building
- a mining technical department building, incorporating a changing and drying room
- an infirmary, incorporating mines rescue facilities
- a canteen building
- a surface maintenance workshop and warehouse
- a laboratory for sample preparation and analysis

Item 19 MARKET STUDIES AND CONTRACTS

Currently, all concentrate product is being sold through the Company's internal marketing division, FQM Trading.

The primary destination for Spec copper concentrate and for zinc concentrate is Europe, whilst Non-spec copper concentrate is sold to Chinese buyers.

Sales are made via long-term and spot contracts.

In view of the Property being an established operation, no further studies, analyses or QP review have been undertaken in respect of product marketing, commodity price projections, product valuations, market entry strategies or product specification requirements.

Item 20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental studies

An environmental management system is in place at ÇBI and includes a comprehensive sampling and monitoring programme to measure dust emissions, control noise levels, measure groundwater impacts, and monitor surface water discharge quality. This programme is in addition to the monitoring programme that is carried out for the DST operations, specific details of which are presented in Item 20.3.3.

The environmental permit conditions include a comprehensive list of required compliances. This list refers to numerous specific articles and annexes applicable to the permitting process. Essentially, however, the permit conditions set out the measurement frequency and methodology for managing air quality, noise control and waste water discharge.

20.2 Status of environmental approvals

20.2.1 Environmental Impact Assessment

Projects that are documented to have started production and/or commenced operations before the publication in February 1993 of the *Environmental Impact Assessment Regulation of Türkiye (*number 21489) are deemed to be exempt from submitting an Environmental Impact Assessment (EIA).

In 1992, Inmet had commenced construction work on a concentrator, with development of a new mining portal starting in March 1993 (Item 6.2.2). Mill commissioning started in August 1994 and the first concentrate production occurred in November 1994. Despite these dates, ÇBI's exemption from the EIA process is applicable due to the Company being issued with an operating (business) licence in July 1987, which licence is a legal requirement for the commencement of operations.

Accordingly, the Directorate of Mining Affairs and Natural Resources of Türkiye granted the EIA exemption on the 25th of May 2010.

20.2.2 Environmental Licence

The environmental licence conditions stipulate that in accordance with the *Mining Waste Regulations of Türkiye*, a *Waste Management Plan* must be updated and submitted every five years.

Information to enable the Operation's environmental licence to be renewed was submitted to the *Ministry of Environment and Urbanisation* (the E&U Ministry) in November 2019. This information included an SRK report (SRK, 2018) on the *Waste Management Plan* (which includes deep sea tailings discharge and mine paste filling) and another report by the *Karadeniz University Faculty of Marine Sciences* (*Karadeniz,* 2018).

In respect of DST, subsections of Article 22 of the *Mining Waste Regulations of Türkiye* state:

- (2) as a result of waste characterisation, mining waste defined as inert and non-hazardous can only be disposed of in the oxygen-free and non-living dead layer of the Black Sea
- (3) for the disposal of inert and non-hazardous mining waste in the Black Sea, there should not be a geographically, topographically and geologically suitable area in the terrestrial region within a radius of approximately 30 km from the centre of the mining activity

- (4) where the disposal of …. mining wastes in the …. Black Sea is planned, a detailed study and report by the *Karadeniz University Faculty of Marine Sciences* is to be submitted to the E&U Ministry, documenting:
  - the method of transport and discharge of the waste to be stored
  - discharge depth
  - density calculations of the waste and sea water
  - sedimentation and elevation range of the waste
  - assimilation capacity of the sea
  - hydrodynamic conditions

The *Karadeniz University* report (*Karadeniz,* 2018), observed that ÇBI's DST activities do not have a negative impact on the marine ecosystem of the region, and are similar in terms of biodiversity impacts to the findings of studies carried out in different regions of the Black Sea.

In respect of mine paste filling, a subsection of Article 15 of the *Mining Waste Regulations of Türkiye* states:

- (3) for the filling of underground galleries …. with mineral enrichment wastes (paste filling etc), in order to determine the acid mine drainage and long-term metal leaching potential of the wastes, and to demonstrate that all precautions have been taken….., a report should be prepared …. and submitted to the E&U Ministry, documenting:
  - all relevant information and documents
  - primarily static and kinetic tests
  - mineralogical, geochemical and hydrogeochemical examinations
  - hydrogeological, hydrological calculations and modelling
  - opinion letters from the institutions/organisations specified for the Deep Injection (D3) disposal method defined in Annex 2/A of the *Mining Waste Regulations of Türkiye*

The E&U Ministry response in August 2020 (E&U Ministry, 14th of August 2020) to the ÇBI 2019 submission acknowledges receipt of the SRK and university reports. In reply, they state that deep sea tailings deposition and mine paste filling "differ from other mine waste disposal methods" and additional studies need to be carried out. The E&U Ministry goes on to say that a "special expertise commission" must be appointed. ÇBI is waiting for this commission to be established.

When a favourable decision is forthcoming from this commission, ÇBI can then apply for renewal of the environmental licence. The E&U Ministry response letter (14th of August 2020) states that "since there may be a delay in starting the licencing process, the Company's operations can continue within the scope of the current environmental permit process".

20.2.3 Environmental Permit

The environmental permit conditions include a comprehensive list of required compliances, specifically for management of air emissions, noise control and waste water discharge.

ÇBI's *Environmental Permit Certificate for Wastewater Discharge, Noise Control and Air Emissions* was last renewed on the 12th of March 2021 and is valid for five years until March 2026. According to the *Environmental Permit and Licence Regulations of Türkiye,* a renewal application must be made at least 180 days before validity expires. This means that ÇBI's permit renewal application had to be submitted before the 13th of September 2025.

During the 2025 environmental permit renewal process, the Ministry has reiterated its earlier position in writing, stating: *"…from the perspective of waste management legislation, there is no objection to submitting an environmental permit application."*

20.3 Waste and tailings disposal, site monitoring and water management

20.3.1 Mine waste disposal

In general, very little mine waste is hauled to surface and dumped. Typically, mine waste from development openings is hauled directly to underground backfill locations.

Currently, and owing to a lack of suitably available backfilling locations, there is a surface stockpile of mineralised waste from the South Orebody access development. This stockpile is located to the north of the plant site and is referred to as the North Waste Area. The base of the stockpile is lined and all collected seepage is pumped back to the plant.

20.3.2 Process tailings used as mine backfill

Item 16.4 provides a description of the use of cemented tailings for mine backfill. Item 18.1.2 provides a description of the mine backfilling facilities.

Further to the monitoring information in Item 20.2.2, and regarding Article 15 of the *Mining Waste Regulations of Türkiye,* a new standard is being prepared by the *Ministry of Agriculture and Forestry* (A&F Ministry) in respect of the paste filling process and underground water monitoring.

ÇBI is currently monitoring water quality and water losses at monitoring points on six different sublevels of the mine.

20.3.3 Process tailings deposition into the Black Sea

Item 18.1.4 provides a description of the DST facility.

Before the discharge facility was commissioned, a comprehensive monitoring programme was established involving the routine analysis of water samples collected in the vicinity of the discharge point, and from locations where the Büyükdere and Beyazsu Rivers flow into the sea. For over twenty years this monitoring programme has been carried out by the *Central Fisheries Research Institute* (SUMAE) affiliated with the A&F Ministry. The ten sampling locations are shown in Figure 20.1, whilst the seven offshore sampling depth intervals are shown in Table 20-1.

Figure 20.1 Deep Sea Tailings water sampling locations (source: ÇBI)

Table 20-1 Offshore DST sampling depths

Measurement locations	1	2	3	4	5	6	7
Sampling depth (m)	Depth of the sea bottom (m)						
	200	275	225	372	385	361	650
2	X	X	X	X	X	X	X
25							
50							
75	X	X	X	X	X	X	X
100							
125							
150	X	X	X	X	X	X	X
175							
200	X	X	X	X	X	X	X
225			X				
250							
275		X					
300							
350				X	X	X	X

Twenty one different parameters are analysed from each of the DST water samples, including:

- temperature
- salinity
- light transmittance
- sigma-t
- electrical conductivity
- chlorophyll-a
- pH
- dissolved oxygen
- hydrogen sulphide
- alkalinity
- copper
- zinc
- mercury
- lead
- iron
- cadmium
- arsenic
- total dissolved solids
- total organic carbon
- petroleum derivatives

Some of the analyses are completed on board the sampling vessel, whilst the remaining ones are analysed at the SUMAE laboratory in Trabzon, and at the *University Faculty of Marine Sciences and the Scientific and Technological Research Council of Türkiye* (TÜBİTAK).

The SUMAE and TÜBİTAK analyses are routinely reported to the A&F Ministry. Thousands of samples have been analysed since the commencement of monitoring. A report on 2023 quarterly sampling and analysis was prepared by MCG Engineering Consultancy from Istanbul (MCG, April 2024), in which it was concluded that[12]:

"it was observed that all metal matrix values were well below the values specified in the continental water general quality indicators presented in (MCG) Table 11, and no pollutant contribution related to mining terrestrial activities was detected".

MCG Table 11 is reproduced in Table 20-2.

Table 20-2 MCG (2024) Table 11: General quality criteria of sea and continental water

	As (µg/L)	Cd (µg/L)	Cu (µg/L)	Fe (µg/L)	Hg (µg/L)	Mn (µg/L)	Pb (µg/L)	Zn (µg/L)
Sea Water	100	10	10	-	4	-	100	100
Continental Water	20	3	20	300	0.1	100	10	200

(*) General quality characteristics of sea water
(Official Gazette, regulation no. 25687 dated 31.12.2004; amendment dated 13.02.2008)

To safeguard against potential damage to the DST pipeline, there are warning signs along the surface route and nearby excavation work is prohibited. Fishing is prohibited in the vicinity of the pipeline.

20.3.4 Water management

Process water for the plant is sourced from bores positioned alongside the Büyükdere River. All bores are licenced and individually permitted for a particular rate of abstraction. The bores are monitored from the plant control room and the abstraction is recorded with ultrasonic flow metres. In accordance with the *Regulation on Water for Human Consumption,* routine monitoring samples are collected and analysed.

Domestic wastewater is treated in a biological waste water treatment plant with a capacity of 100 m^3/day. This water is pumped to the DST mix tank at the Çayeli coast (Figure 18.3) and then discharged to sea. The water quality is monitored over two-month periods to comply with the *Water Pollution Control Regulations of Türkiye,* with the results then entered into an on-line Integrated Environmental Information System (ECBS) database.

Ordinarily, operation of a wastewater treatment plant would be subject to environmental permitting requirements. The Operations are exempt however, since the permitting process applies only to new plants. On this basis, the ÇBI wastewater treatment plant operates under a *Treatment Plant Identity Certificate.*

In respect of rainfall run-off and contact water emanating from the Operations (including the underground mine), several receiving ponds are located around the Operations site. The reclaimed water is used in the process plant and for dust suppression. Water emanating from a batch concrete plant is also captured into a pond and reused in that plant.

Water monitoring samples are collected monthly from six underground mine locations, and tri-monthly from four surface monitoring bores. Approximately 100 water quality parameters are analysed using the services of an accredited environmental laboratory.

[12] translated from Turkish

20.3.5 Other environmental monitoring

Air quality

To comply with permit requirements, air quality measurements are carried out every two years using laboratories accredited by the E&U Ministry, and in accordance with the *Regulation on Control of Industrial Air Pollution.* Settled dust measurements are made at points determined by a selected accredited laboratory, recognising specific traffic routes and other locations where dust is typically created. Measurements are also recorded at the four site laboratory stacks, the primary crusher stack, and the limestone building stack.

On a more frequent monthly basis, certain designated control points around the site are also monitored for air quality.

Noise control

The proximity of the Operations site to residential areas requires an environmental monitoring programme aimed at limiting excessive industrial noise emissions. Emissions are monitored continuously for twenty fours a day by means of four permanently placed recorders around the site. Portable recorders are also used when required, e.g., in areas where construction activities are taking place. ÇBI's External Relations Department is responsible for liaising with the community when noise complaints are made.

To comply with permit requirements, an acoustic measurements report is periodically submitted, using a selected laboratory accredited by the E&U Ministry, and in accordance with the *Environmental Noise Control Regulation.* A report was last submitted in 2015 and relevant information subsequently included in the 2021 environmental permit renewal. There is no specific validity date for the acoustic report and the Provincial Directorate of Environment, Urbanisation and Climate Change may request that the report be updated with new environmental noise measurements depending on the increase in the number of settlements near to the Operations, changes in the activities of the Operation, environmental noise complaints, etc.

Waste management

The handling of waste materials, other than mined waste rock and process tailings, is managed according to the *Waste Management Regulations of Türkiye.* These regulations address the reduction of waste at the source, temporary storage and transport of the waste material to a final disposal or recycling facility.

The mandatory reporting requirements for waste management are addressed in an *Industrial Waste Management Plan*, a copy of which was submitted to and approved by the E&U Ministry in 2022. This approval remains valid for three years until late 2025.

There is also a *Zero Waste Regulation* to be complied with. ÇBI received a Basic Level Zero Waste Certificate in 2020 and this remains valid for five years until 2025.

Landslides

Landslide related ground movements are being monitored and an early warning system has been installed (Figure 20.2). A boundary defining the landslide prone area around the Operations site is shown in Figure 20.3.

Greenhouse gas (GHG) emissions

Mining and processing activities are not within the scope of the *Regulation on Monitoring Greenhouse Gas Emissions in Türkiye.* There are no relevant obligations in other national standards and directives.

Nevertheless, ÇBI has been calculating Scope 1 and 2 GHG emissions since 2007. In mid-2024, a consultancy agreement was entered into to establish an ISO 14064 GHG management system. ÇBI is intending to install three AC charging points in 2025 to support the growing use of electric vehicles. A project involving the installation of a 10 MW solar power plant is under consideration.

Figure 20.2 Landslide monitoring array (source: ÇBI)

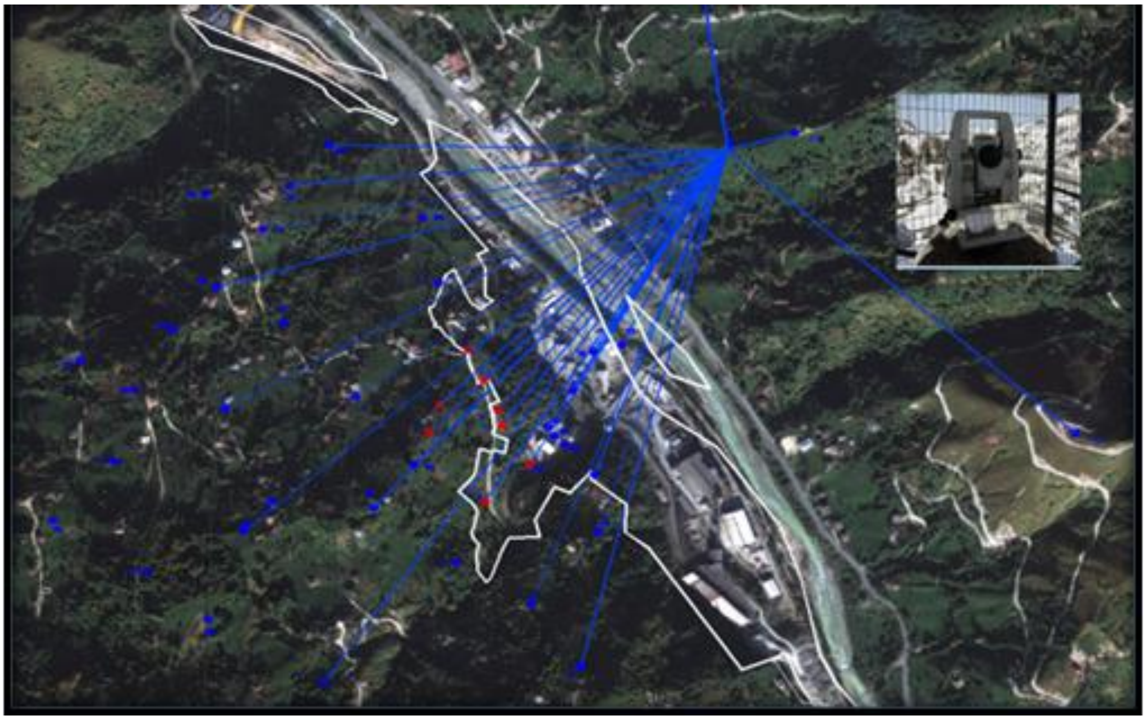
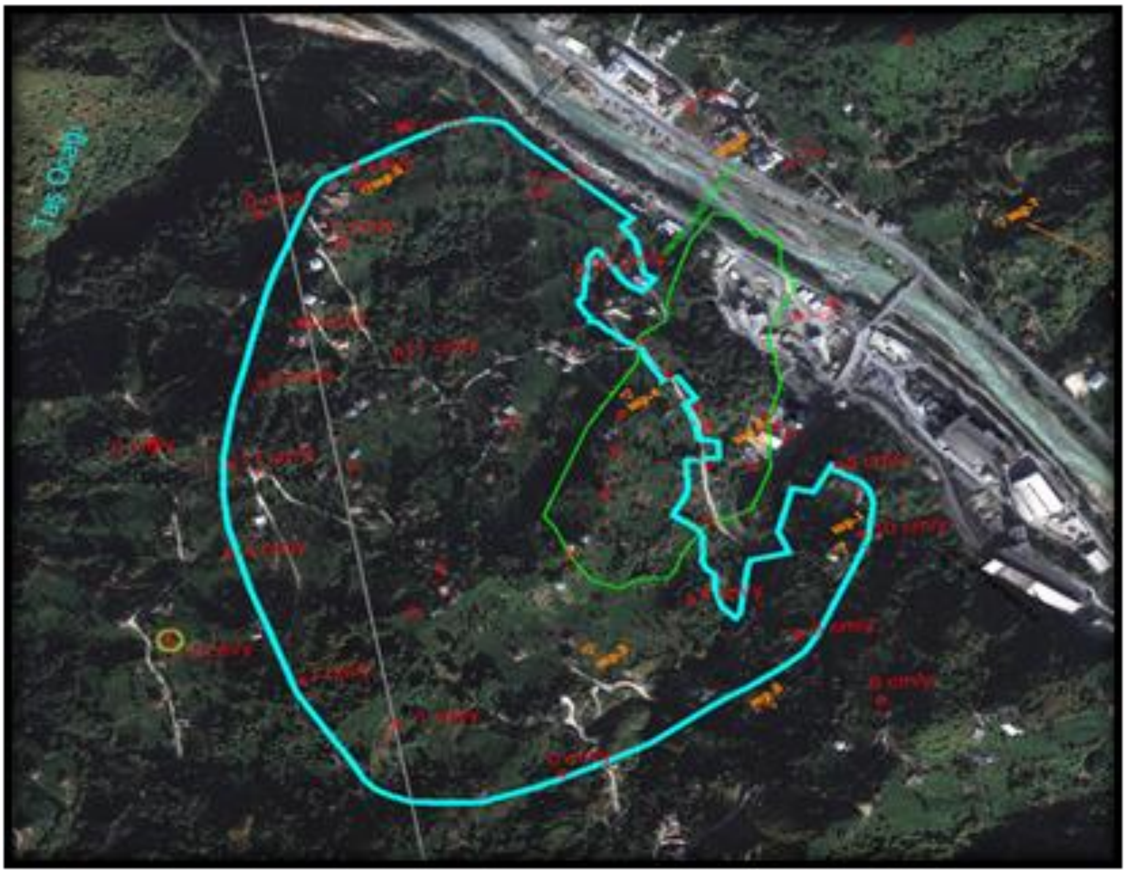


Figure 20.3 Defined landslide zone (source: ÇBI)



20.4 Social and community related requirements

In Item 5.3 it was mentioned that the steep slopes surrounding the Operations site have been subject to landslides. In particular, ground movements have occurred historically in the Maden and Çamlıca neighbourhoods located above the site.

A request for alternative housing sites has been the subject of meetings between ÇBI and the neighbourhood committees, the result of which has been the launch of a Near Mine Housing Project in 2010. This is a voluntary community resettlement project to which International Finance Corporation (IFC) Performance Standards have been applied. A construction fund of $21.9 M has been set up for the resettlement of affected families into 93 newly built homes. To date, resettlement into 85 completed homes has been achieved. In addition, two tea leaf collection stations and one mosque have also been constructed for community use.

Several livelihood structures have been donated to families whose resettlement has been completed so that they can continue their agricultural activities. To date, 81 livelihood structure donations have been made. Additionally, within the scope of the social support programme, support worth 300 thousand TL has been provided to approximately 50 families engaged in livestock farming.

20.5 Mine closure provisions

As of December 2007, government regulations require the submission of a formal Project Reclamation Plan (or a Mine Closure Plan, in other words). ÇBI, however, had produced a Mine Closure Plan routinely from 1995. This plan is updated periodically and an annual estimate of the asset retirement obligation (ARO) amount is recalculated.

The closure activities in this plan are broken down into five main components, and then further subdivided into miscellaneous components. The five main components are:

- the underground mine
- the process plant
- the general site facilities
- the DST facilities
- the Rize port concentrate storage and ship loading facility

For each component, the main decommissioning and closure items are highlighted along with their respective performance criteria, closure investigations and testwork, available closure options, closure actions, provisions for the demonstration of closure performance, and the cost of closure.

The plan anticipates a decommissioning and closure period of approximately 12 to 18 months for the underground mine and site infrastructure including the process plant, followed by three to four years for site rehabilitation and environmental compliance monitoring.

The base case closure scenario assumes that all infrastructure will be decommissioned, demolished and removed.

A recent comprehensive memorandum report was produced by SRK Consulting (SRK, December 2024) to document the asset retirement obligation estimate (or closure provision). The latest decommissioning and closure cost (ARO) estimate as itemised in Table 20-3 is $8.8 M.

Table 20-3 Estimated mine closure cost provision

DESCRIPTION	Labour US$M	Stores US$M	Equipment US$M	Other US$M	TOTAL US$M
Administration	$1.0	$0.0	$0.0		$1.0
Underground	$0.2	$0.2	$0.2		$0.6
Surface	$0.7	$0.2	$1.7		$2.6
Environmental	$0.1	$0.1	$0.2	$0.0	$0.3
Power				$1.2	$1.2
Hydrology				$1.4	$1.4
Total Direct Cost	**$1.9**	**$0.6**	**$2.0**	**$2.5**	**$7.0**
Contingency (25%)	**$0.5**	**$0.1**	**$0.5**	**$0.6**	**$1.8**
Grand Total Cost	**$2.3**	**$0.7**	**$2.5**	**$3.2**	**$8.8**

20.5.1 Water management following closure

Ground water management

SRK Consulting and Engineering (SRK, February 2024a) produced a numerical model and considered several possible closure scenarios in order to evaluate groundwater quality post-closure.

The modelling results showed that in all scenarios, the possible contaminant spread does not reach any water source downstream of the Operations site, even after 50 years.

Management of water in the underground mine

Following closure of the Operations and removal of surface infrastructure there will remain an abandoned underground mine which will gradually flood.

SRK (SRK, February 2024b) carried out an assessment of underground flooding scenarios and provided recommendations on post-closure water management and suitable monitoring procedures.

Basing their assessment on acid base accounting (ABA) data, and assuming a Main Orebody mined void, SRK considered several possible flooding scenarios and concluded that:

1. Flooding the mine by the natural rebound of groundwater inflows will take a long time, runs the risk of prolonged oxidation of exposed sulphide minerals, and will thereby lengthen the time-generation of elevated metal concentrations.
2. Accelerated flooding is the preferred scenario, minimising oxidation and the prolonged generation of elevated metal concentrations. This scenario would involve fast filling of the mine void using water drawn from a borefield.

To assist the closure efforts, SRK recommended the installation of bulkheads to close-off depleted areas of the mine and also the use of lime dosage. In terms of monitoring, SRK recommended the establishment of monitoring points at various locations throughout the mine. Depending on the rate of change of the water chemistry, it was advised that the frequency of monitoring could be reduced as the flooded water level trends towards a steady state.

According to the water quality values calculated as a result of SRK's geochemical modelling, the underground mine does not pose a concern in terms of pollutants other than SO_4 in the closure and post-closure periods.

The SRK report does not estimate the borefield requirements for the preferred accelerated flooding scenario water balance. Furthermore, the report ought to be updated to account for the future South Orebody mined void (including associated underground development).

20.6 Potential environmental issues

ÇBI's permit renewal application needed to be submitted before the 13th of September 2025, and could take some time to be approved thereafter. The DST activities will need to be addressed in this renewal application and the appointment of a "special expertise commission" could continue to linger. Although the E&U Ministry might honour their previous advice that the Operations can continue whilst waiting for the commission's deliberation, there is no certainty that the permit renewal will eventually be formally approved.

The ÇBI management team intends to address this risk through continued discussions with the Ministry during 2025. During the 2025 environmental permit renewal process, the Ministry has reiterated its earlier position in writing, stating: *"…from the perspective of waste management legislation, there is no objection to submitting an environmental permit application."*

Item 21 CAPITAL AND OPERATING COSTS

21.1 Capital expenditure requirements

The following commentary provides an overview of the major capital expenditure requirements. Item 21.2 provides an itemised breakdown.

21.1.1 Mining equipment

Mining capital expenditure is essential to enable the mine life to be extended according to the new plan. The justifications for this expenditure relate primarily to the age of the existing equipment and the supply difficulties related to maintenance parts that are either obsolete or difficult to procure, specifically:

- the average age of the existing haul truck fleet is 20 years
- truck maintenance costs are rising and availability is declining
- this is also the case with the LHDs, with working hours now reaching 50,000
- the backhoe loaders have high maintenance costs and require replacement after two years of service
- the concrete transmixers, shotcrete sprayers and platform lifts are of an age needing replacement due to increasing maintenance costs

21.1.2 Mine ventilation

The existing and proposed expansion to the mine ventilation network is described in Item16.6. The cost of developing the mine openings is covered elsewhere, whereas the capital expenditure for physical infrastructure is required for:

- a new 400 kW fan
- auxiliary fans for the South Orebody (required over the life of the mine)
- new ventilation bulk heads to regulate airflow
- ventilation raise steel supports

21.1.3 Paste backfill system

To cater for the increased production and mined void filling requirements, a new paste fill thickener and surface surge tank is required.

Various improvements and additions are also required to the existing infrastructure in the mine to enable paste backfill to be reticulated across to and throughout the South Orebody. The more significant capital requirement for the mine is for a new 110 bar (11 MPa) pump to augment the exiting 60 bar (6 MPa) pump.

21.1.4 Mine dewatering

Capital expenditure is required to replace the existing twenty year old primary mine dewatering pump and extend the main trunk line pipe over an estimated length to 1,300 m. Capital provisions are also made for:

- new auxiliary pumps for the South Orebody
- relocation of the Main Orebody Geho pump

21.1.5 Milling and processing equipment

There has been a continuous programme for the replacement of worn out and corroded equipment for several years, and this will be continued throughout the remainder of the mine life, with total expenditure of approximately $6.6M planned.

The major capital expenditure required for milling and processing equipment is as follows:

- a new primary jaw crusher
- flotation cell replacements, with auxiliary equipment such as flow meters, pneumatic valves and level controllers
- replacement reagent tanks, Cu column cells, Cu rougher/scavenger tails pumps and Cu lasta filter hydraulic cylinder
- a replacement particle size indicator
- a new atomic absorption spectrophotometer (AAS) and an upgrade to the online slurry analyser (Courier 6SL)

21.1.6 Surface slope stability project

The surface slope above the mine site needs to be stabilised due to ongoing land sliding, the causes of which are mentioned in Item 5.3. The regulatory requirement for continuous ground stability monitoring is described in Item 20.3.5.

Remediation measures are now required including earthworks and the installation of bored piles.

21.1.7 Plant capital

In ÇBI terminology, plant capital refers to provisions for general site infrastructure and equipment including major expenses for such as a sound deflection barrier adjacent to the new mine portal, and for light vehicle replacements.

21.1.8 Administration capital

ÇBI administration capital provisions cover environmental, security, health and safety and information technology requirements.

21.2 Capital cost estimates

Table 21-1 summarises the summary capital costs over the life of mine.

Table 21-1 Summary capital costs, as at the end of February 2025

Department	Description	2025 ($,000)	2026 ($,000)	2027 ($,000)	2028 ($,000)	2029 ($,000)	2030 ($,000)	2031 ($,000)	2032 ($,000)	2033 ($,000)	2034 ($,000)	2035 ($,000)	TOTAL ($,000)
Mine	Initial Capex	$5,908											**$5,908**
	Sustaining		$5,940	$1,830	$1,675	$695	$3,540	$625	$535	$495	$400	$260	**$15,995**
	Subtotal	**$5,908**	**$5,940**	**$1,830**	**$1,675**	**$695**	**$3,540**	**$625**	**$535**	**$495**	**$400**	**$260**	**$21,903**
Mill	Initial Capex	$1,501											**$1,501**
	Sustaining		$1,546	$1,626	$796	$216	$195	$300	$200	$150	$150	$0	**$5,179**
	Subtotal	**$1,501**	**$1,546**	**$1,626**	**$796**	**$216**	**$195**	**$300**	**$200**	**$150**	**$150**	**$0**	**$6,680**
Plant	Initial Capex	$6,706											**$6,706**
	Sustaining		$6,640	$5,540	$530	$520	$510	$610	$510	$510	$510	$510	**$16,390**
	Subtotal	**$6,706**	**$6,640**	**$5,540**	**$530**	**$520**	**$510**	**$610**	**$510**	**$510**	**$510**	**$510**	**$23,096**
Administration	Initial Capex	$586											**$586**
	Sustaining		$1,023	$194	$151	$127	$125	$425	$461	$125	$100	$100	**$2,828**
	Subtotal	**$586**	**$1,023**	**$194**	**$151**	**$127**	**$125**	**$425**	**$461**	**$125**	**$100**	**$100**	**$3,414**
All	**Initial Capex**	**$14,701**											**$14,701**
	Sustaining		**$15,149**	**$9,190**	**$3,152**	**$1,558**	**$4,370**	**$1,960**	**$1,706**	**$1,280**	**$1,160**	**$870**	**$40,392**
	Subtotal	**$14,701**	**$15,149**	**$9,190**	**$3,152**	**$1,558**	**$4,370**	**$1,960**	**$1,706**	**$1,280**	**$1,160**	**$870**	**$55,093**

21.2.1 Departmental cost itemisations

Mining department

Table 21-2 itemises the mining equipment overhaul and replacement requirements. Further information on mining equipment replacements is provided in Item 16.10.

Table 21-2 Mining department equipment capital

Description	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	TOTAL
Jumbos ($,000)												
Overhaul jumbos (27-415, 417)			$30				$50					**$80**
Jumbo drill drifter (27-415)			$60	$60				$60				**$180**
Subtotal	**$0**	**$0**	**$90**	**$60**	**$0**	**$0**	**$50**	**$60**	**$0**	**$0**	**$0**	**$260**
Production drills ($,000)												
Solomatic drill drifter				$80				$80				**$160**
Overhaul cubex (27-405)				$30								**$30**
Subtotal	**$0**	**$0**	**$0**	**$110**	**$0**	**$0**	**$0**	**$80**	**$0**	**$0**	**$0**	**$190**
Bolters ($,000)												
Bolter rock drill (HI 300)	$36											**$36**
Bolter enclosed cabin replacement (27-416)	$46											**$46**
Bolter new equipment	$808											**$808**
Bolter drifter						$80						**$80**
Overhaul bolter (27-414)		$50										**$50**
Subtotal	**$890**	**$50**	**$0**	**$0**	**$0**	**$80**	**$0**	**$0**	**$0**	**$0**	**$0**	**$1,020**
LHDs ($,000)												
Toro 1400 (27-210, 211, 212)	$998	$1,050				$1,000						**$3,048**
Toro 1400, 27-217, repower	$400	$400										**$800**
Engine replacements			$60	$60		$60		$60		$60		**$300**
Subtotal	**$1,398**	**$1,450**	**$60**	**$60**	**$0**	**$1,060**	**$0**	**$60**	**$0**	**$60**	**$0**	**$4,148**
Trucks ($,000)												
Wagner MT436B (27-314, 309, 316, 318, 319)	$750	$1,580	$790	$790								**$3,910**
Overhaul MT436B trucks (27-320, 321, 322)				$80	$80	$80						**$240**
Truck engine replacements		$60	$60	$60	$60	$60	$60	$60	$60	$60	$60	**$600**
Truck cabin replacements			$10	$10	$10		$10		$10	$10		**$60**
Truck rear chassis (27-320)				$35								**$35**
Subtotal	**$750**	**$1,640**	**$860**	**$975**	**$150**	**$140**	**$70**	**$60**	**$70**	**$70**	**$60**	**$4,845**
Shotcrete equipment ($,000)												
Normet Spraymec 1050 WPC (27-110, 162)	$760					$760						**$1,520**
Overhauls (27-162, 164)			$50		$50							**$100**
Replacement mixers (27-108, 109)		$650				$650						**$1,300**
Overhaul mixers (27-109,161,163)		$50	$50	$50								**$150**
Subtotal	**$760**	**$700**	**$100**	**$50**	**$50**	**$1,410**	**$0**	**$0**	**$0**	**$0**	**$0**	**$3,070**
Utility vehicles ($,000)												
Replacement ANFO charger (27-117)		$700										**$700**
Replacement platforms (27-118, 119)		$550				$550						**$1,100**
Overhaul platforms (27-120, 140, 141, 171)		$25	$25	$25	$25							**$100**
Paus platform cabin (120, 141, 142)		$10	$10	$10								**$30**
Subtotal	**$0**	**$1,285**	**$35**	**$35**	**$25**	**$550**	**$0**	**$0**	**$0**	**$0**	**$0**	**$1,930**
Backhoe loaders ($,000)												
Loader replacements	$150	$150	$150	$150	$300	$150	$300	$150	$300	$150	$150	**$2,100**
Subtotal	**$150**	**$150**	**$150**	**$150**	**$300**	**$150**	**$300**	**$150**	**$300**	**$150**	**$150**	**$2,100**
Light vehicles ($,000)												
Wet Brakes For LV's	$15	$15	$15	$15	$15	$15	$15	$15	$15	$15		**$150**
Subtotal	**$15**	**$15**	**$15**	**$15**	**$15**	**$15**	**$15**	**$15**	**$15**	**$15**	**$0**	**$150**
TOTAL	**$3,963**	**$5,290**	**$1,310**	**$1,455**	**$540**	**$3,405**	**$435**	**$425**	**$385**	**$295**	**$210**	**$17,713**

Correspondingly, Table 21-3 itemises the mining infrastructure provisions.

Table 21-3 Mining department infrastructure capital

Description	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	TOTAL
Infrastructure ($,000)												
Canopy for stockpile gallery entry	$165											**$165**
New South Orebody infrastructures	$80	$20	$20	$20	$20	$20	$20	$20	$20	$15	$10	**$265**
New drainage lines for the South Orebody	$250											**$250**
Ventilation shaft steel supports	$250	$350	$75	$75								**$750**
400 kW fan electrical system installation	$115											**$115**
Auxilary fan for South Orebody	$90	$40	$40	$40	$40	$30	$30	$30	$30	$25		**$395**
Jet fan	$2											**$2**
Ventilation bulkheads, doors and regulators	$30	$50	$50	$10	$10	$20	$20	$20	$20	$20	$15	**$265**
Relocation of mine compressor building	$20											**$20**
New compressor (Atlas Copco)		$65										**$65**
Lubrication station for mining equipment	$15											**$15**
Air pumps and submersible pumps	$30	$25	$25	$25	$25	$25	$20	$20	$15	$15	$15	**$240**
New Putzmeister pump and basic engineering	$820											**$820**
Relocation of Geho pump			$200									**$200**
Infrastructure supply for Minearcs	$10	$5	$5	$5	$5	$5	$5	$5	$5	$5	$5	**$60**
CAMs system for refuge stations		$45										**$45**
MPBX and smart cable	$15		$15		$15		$15		$15	$10		**$85**
Underground lifelines	$10	$5	$5	$10	$5	$10	$5	$10	$5	$10	$5	**$80**
Gas measurement device and sensors	$35	$5	$5	$5	$5	$5	$50	$5		$5		**$120**
Blast vibration monitoring device	$8						$25					**$33**
Various equipment		$10	$50			$20						**$80**
Replacement of change room lockers		$30	$30	$30	$30							**$120**
TOTAL	**$1,945**	**$650**	**$520**	**$220**	**$155**	**$135**	**$190**	**$110**	**$110**	**$105**	**$50**	**$4,190**

Milling and plant departments

The milling and plant department itemisations are listed in Table 21-4 and Table 21-5, respectively.

Administration department

The administration department itemisation is listed in Table 21-6.

Table 21-4 Milling department capital

Description	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	TOTAL
Crushing and Milling ($,000)												
Bin-whip overhaul		$75										**$75**
New Jaw Crusher			$400									**$400**
Crusher Plant Chute Modification	$60											**$60**
Mill Building Roof Revision	$100	$100										**$200**
New Compressor For Mill (Ingersol Rand)				$90								**$90**
Cyclone Replacement			$100									**$100**
Flotation ($,000)												
Cu Conditioner Tank Replacement	$45					$45						**$90**
Flotation Cell Replacements		$250	$250				$100					**$600**
Cu Columns Cell Replacement				$200								**$200**
Cu Rougher Scavenger Tail Pumps' Replacement				$75	$75							**$150**
Zn Conditioner Tank (2 Pcs) (60X2)		$120										**$120**
Fls Proflote Renting	$141	$141	$141	$141	$141	$150	$150	$150	$150	$150		**$1,455**
Concentrate Handling ($,000)												
Cu Final Concentrate Pump Replacement		$75										**$75**
Cu Lasta Filter Hydraulic Cylinder Replacement			$150									**$150**
Closing The Concentrate Field Filter Passage	$15											**$15**
Press Filter Plates		$50										**$50**
Port Transfer Chute Replacement	$20											**$20**
Final Tailings ($,000)												
Final Tail Pumps' Replacement	$100											**$100**
Tail Water Line Flowmeter	$15											**$15**
Spare Pipes For Tail Line		$50										**$50**
Reagents, Miscellaneous ($,000)												
Reagent Tank Revisions (Smbs, Floc, Lime)			$100	$100								**$200**
Sump Pump (Mud) (5 Ea. 37Kw, 3 Ea. 10Kw, 2 Ea. 8Kw)		$50	$50	$25			$50	$50				**$225**
27-452-101 Pumpline Upgrade With Flowmeter	$15											**$15**
27-452-003 And 27-452-004 Pumps Location Replacement	$15											**$15**
Analysers, Laboratory ($,000)												
Courier Upgrade		$550										**$550**
Particle Size Indicator (Psi) Replacement			$300									**$300**
New Aas		$0	$100									**$100**
Lab. Drying Oven		$50										**$50**
Lab. Scale Grinder		$35										**$35**
Lab. Fume Hood			$35									**$35**
Pastefill Circuit ($,000)												
Batch Plant And Pastefill Plant Operator Fitting Room	$15											**$15**
Pastefill Pump	$800											**$800**
PLC Upgrade For Pastefill Plant	$20											**$20**
Disc Filter Revision For Pastefill				$150								**$150**
Pastefill Conditioner Body Replacement				$15								**$15**
New Generator For Pastefill Plant	$90											**$90**
Miscellaneous ($,000)												
Processing Plant Offices & Bathrooms Upgrades	$50											**$50**
TOTAL	**$1,501**	**$1,546**	**$1,626**	**$796**	**$216**	**$195**	**$300**	**$200**	**$150**	**$150**	**$0**	**$6,680**

Table 21-5 Plant department capital

Description	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	TOTAL
Projects ($,000)												
A1-A1' Slope Stability Project	$5,000	$5,000	$5,000	$20	$20	$20	$20	$20	$20	$20	$20	**$15,160**
Alternative Village Road Project		$600										**$600**
Corrosion Management	$250	$100					$100					**$450**
Personal Tracking System Extension	$50	$20	$10	$10	$10							**$100**
Equipment ($,000)												
LX Replacements	$360	$60	$120	$120	$120	$120	$120	$120	$120	$120	$120	**$1,500**
Leasing Light Vehicle	$380	$350	$350	$350	$350	$350	$350	$350	$350	$350	$350	**$3,880**
Mini Loader Replacements (Bobcat S650 Model)For Mill And Workshop	$160											**$160**
PLC Replacement For Surface		$10	$10	$10								**$30**
Replacement Crane (27-115)		$440										**$440**
Telephone Central System			$15									**$15**
New Fire Diesel Pump		$40										**$40**
Transformator For Mill MCC			$15									**$15**
Leak Detection	$60											**$60**
Assorted Handtools	$24	$20	$20	$20	$20	$20	$20	$20	$20	$20	$20	**$224**
Infrastructure ($,000)												
Batch Plant Sand Bunker Replacement (X2)	$50											**$50**
Sound Barrier Construction In Front Of New Access Gallery	$100											**$100**
New Weight Scale For Security Office Area	$50											**$50**
Gabion Wall Construction And Relocation Of Service Pipes	$50											**$50**
Car Wash Station	$25											**$25**
Main Office Bathroom Restoration	$20											**$20**
Mill Plant Bathroom Restoration	$20											**$20**
Ac Charging Station For Electric Vehicles	$18											**$18**
Relocation Of The High Voltage Unit And The Low Voltage Unit	$70											**$70**
Relocation Of The Mine Generator And Fuel Tank	$10											**$10**
Light Vehicle Workshop Roof Replacement	$10											**$10**
TOTAL	**$6,706**	**$6,640**	**$5,540**	**$530**	**$520**	**$510**	**$610**	**$510**	**$510**	**$510**	**$510**	**$23,096**

Table 21-6 Administration department capital

Description	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	TOTAL
Environment ($,000)												
Hydrogeological Wells	$0	$636					$300	$336				**$1,272**
Current Monitoring Stations	$12											**$12**
Waste Storage Area	$9											**$9**
Camera Traps And Caution Signs	$3											**$3**
Compost Machine		$47				$5	$5	$5	$5	$5	$5	**$77**
Greenhouse (Nursery)		$21										**$21**
Others		$5	$5	$5	$5	$27	$27	$27	$27	$27	$27	**$181**
Security ($,000)												
North Area Wire Design	$7	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$7**
Health and Safety ($,000)												
Lifting Bags And Air Control Set.	$12											**$12**
Drager Pss 3000, Apparatus For Firefighters	$3											**$3**
Health Unit Furniture	$3											**$3**
Others		$5	$5	$5	$5	$27	$27	$27	$27	$27	$27	**$181**
IT ($,000)												
SAP Upgrade	$330	$130	$50	$40	$30	$30	$30	$30	$30	$30	$30	**$760**
Digital Transformation Projects Budget	$50	$50	$40	$30	$20							**$190**
Upgrade Computers, Laptops And Devices	$55	$60	$60	$50	$50	$25	$25	$25	$25			**$375**
Ehs Sofware Purchasing (INX etc)	$45	$45	$10	$5	$5	$5	$5	$5	$5	$5	$5	**$140**
Site Access Application Re-Modification	$15	$5	$5									**$25**
FQM Sharepoint Online Transition/Portal	$10	$10	$10	$8	$5	$5	$5	$5	$5	$5	$5	**$73**
Server And Server Os Upgrade In Data Center	$20	$1	$1	$1	$1	$1	$1	$1	$1	$1	$1	**$30**
Laptop For New Key Talent	$4	$5	$5	$5	$5							**$24**
Projectors for Meeting Rooms Voice Systems	$3	$3	$3	$2	$1							**$12**
Industrial Hygene System	$5	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$5**
TOTAL	**$586**	**$1,023**	**$194**	**$151**	**$127**	**$125**	**$425**	**$461**	**$125**	**$100**	**$100**	**$3,414**

21.2.2 Mine closure provision

A comprehensive memorandum report was produced by SRK Consulting (SRK, 2024) to document the asset retirement obligation estimate (ARO, or closure provision). The estimate was dated as at December 2024, assuming then current US dollar and Turkish lira exchange rates. Certain estimates made originally in 2018 were adjusted for inflation.

The 2024 estimate (Table 21-7) accounted for:

- administration costs, for persons involved in pre and post-closure monitoring
- underground mine costs, for closure of abandoned mine portals
- surface infrastructure costs, for dismantling of the process plant and other major infrastructure, and including dredging and clean-up at the Rize port
- environmental costs, for such as topsoil profiling and revegetation
- power costs to cover the pre and post-closure phases
- hydrological costs, for dewatering wells

Table 21-7 Closure provision

DESCRIPTION	Labour US$M	Stores US$M	Equipment US$M	Other US$M	TOTAL US$M
Administration	$1.0	$0.0	$0.0		$1.0
Underground	$0.2	$0.2	$0.2		$0.6
Surface	$0.7	$0.2	$1.7		$2.6
Environmental	$0.1	$0.1	$0.2	$0.0	$0.3
Power				$1.2	$1.2
Hydrology				$1.4	$1.4
Total Direct Cost	**$1.9**	**$0.6**	**$2.0**	**$2.5**	**$7.0**
Contingency (25%)	**$0.5**	**$0.1**	**$0.5**	**$0.6**	**$1.8**
Grand Total Cost	**$2.3**	**$0.7**	**$2.5**	**$3.2**	**$8.8**

21.2.3 Contingency

There are no contingency provisions in the various capital cost estimates, with the exception of 25% for the mine closure estimate.

21.3 Operating cost estimates

The following commentary relates to operating cost estimates that were produced preparatory to the update of the new life of mine production plan. Whilst adopting the originally estimated unit rates, the various total dollar cost estimates were subsequently revised and are now captured in the cashflow modelling described in Item 22.

21.3.1 Mining costs

Table 21-8 lists the physicals basis for the estimation of unit mine operating costs. This information relates to a preliminary LOM production schedule produced for budgeting in July 2024.

Table 21-8 Physicals basis for mine operating cost estimates, 2024 budget

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	LOM
Capital Development in Waste	m												**0**
Operating Development in Waste	m	0	180	180	180	180	180	180	180	180	180	180	**1,800**
Operating Development in Ore	m	1,857	1,382	1,543	1,945	1,484	1,387	1,047	674	970	383	555	**13,226**
Operating Development Total	m	1,857	1,562	1,723	2,125	1,664	1,567	1,227	854	1,150	563	735	**15,026**
TOTAL DEVELOPMENT	**m**	**1,857**	**1,562**	**1,723**	**2,125**	**1,664**	**1,567**	**1,227**	**854**	**1,150**	**563**	**735**	**15,026**
Development tonnes/m	t/m	149	100	100	100	100	100	100	100	100	100	100	**1,149**
Ore Development tonnes	**t**	**276,979**	**138,203**	**154,254**	**194,485**	**148,409**	**138,732**	**104,738**	**67,368**	**96,958**	**38,317**	**55,479**	**1,413,921**
Waste Development tonnes	**t**	**0**	**18,000**	**18,000**	**18,000**	**18,000**	**18,000**	**18,000**	**18,000**	**18,000**	**18,000**	**18,000**	**180,000**
TOTAL DEVELOPMENT	**t**	**276,979**	**156,203**	**172,254**	**212,485**	**166,409**	**156,732**	**122,738**	**85,368**	**114,958**	**56,317**	**73,479**	**1,593,921**
Production drill metres	drill m	67,146	83,850	88,917	82,913	89,790	68,846	58,994	42,184	37,768	46,520	36,496	**703,423**
Stope tonnes	**t**	**423,021**	**561,797**	**595,745**	**555,515**	**601,591**	**461,268**	**395,262**	**282,632**	**253,042**	**311,683**	**244,520**	**4,686,079**
Production tonnes/ drill m	t/dm	6.3	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	6.7	**6.7**
Ore tonnes	t	**700,000**	**700,000**	**750,000**	**750,000**	**750,000**	**600,000**	**500,000**	**350,000**	**350,000**	**350,000**	**300,000**	**6,100,000**
Ore + Waste tonnes (excl. capital dev't	t	**700,000**	**718,000**	**768,000**	**768,000**	**768,000**	**618,000**	**518,000**	**368,000**	**368,000**	**368,000**	**318,000**	**6,280,000**

All operating cost centres have fixed and variable cost components. The fixed cost items were estimated in total dollar terms for the period 2025 to 2029 and then prorated on mining tonnes thereafter. These costs account for mine operating labour, inclusive of overtime and certain worker benefits, and are approximately 46% of the total operating costs over the life of mine. The variable cost items were estimated for the period 2025 to 2035, with summary information provided as follows.

Mine development

Table 21-9 summarises the ore and waste development costs according to operational cost reporting centres relative to July 2024 budgeting and then updated to reflect February 2025 forecast physicals.

Table 21-9 Mine development cost estimate

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
DEVELOPMENT COSTS, 2024 BUDGET FIGURES															
Development in waste	m	0	180	180	180	180	180	180	180	180	180	180			**1,800**
Jumbo drilling	$/m	$34	$54	$59	$71	$92	$83	$86	$87	$72	$116	$86			**$81**
Blasting	$/m	$181	$193	$195	$202	$214	$214	$214	$214	$214	$214	$214			**$209**
LHD loading	$/m	$5	$9	$10	$12	$15	$13	$14	$14	$10	$21	$14			**$13**
Backhoe loading	$/m	$19	$30	$32	$39	$50	$48	$49	$49	$45	$56	$49			**$45**
General Haulage	$/m	$38	$10	$7	$1	$1	$2	$2	$4	$4	$1	$3			**$4**
Primary Haulage	$/m	$23	$6	$3	$0	$0	$0	$0	$0	$0	$1	$1			**$1**
Bolting	$/m	$174	$166	$172	$192	$225	$206	$214	$215	$186	$269	$215			**$206**
Cable bolting	$/m	$207	$289	$346	$362	$475	$464	$472	$476	$486	$530	$537			**$444**
Shotcreting	$/m	$614	$970	$1,199	$1,330	$1,614	$1,670	$1,757	$1,983	$2,251	$2,407	$3,181			**$1,836**
Subtotal	**$/m**	**$1,298**	**$1,725**	**$2,023**	**$2,208**	**$2,687**	**$2,701**	**$2,808**	**$3,042**	**$3,269**	**$3,616**	**$4,300**			**$2,838**
	$'000	**$0**	**$311**	**$364**	**$397**	**$484**	**$486**	**$506**	**$548**	**$588**	**$651**	**$774**			**$5,108**
Development in ore	m	1,857	1,382	1,543	1,945	1,484	1,387	1,047	674	970	383	555			**13,226**
Jumbo drilling	$/m	$34	$54	$59	$71	$92	$83	$86	$87	$72	$116	$86			**$70**
Blasting	$/m	$181	$193	$195	$202	$214	$214	$214	$214	$214	$214	$214			**$203**
LHD loading	$/m	$5	$9	$10	$12	$15	$13	$14	$14	$10	$21	$14			**$11**
Backhoe loading	$/m	$19	$30	$32	$39	$50	$48	$49	$49	$45	$56	$49			**$39**
General Haulage	$/m	$28	$10	$11	$13	$11	$14	$12	$13	$24	$3	$8			**$15**
Primary Haulage	$/m	$13	$7	$8	$12	$9	$11	$10	$9	$15	$4	$8			**$10**
Bolting	$/m	$174	$166	$172	$192	$225	$206	$214	$215	$186	$269	$215			**$195**
Cable bolting	$/m	$207	$289	$346	$362	$475	$464	$472	$476	$486	$530	$537			**$390**
Shotcreting	$/m	$614	$970	$1,199	$1,330	$1,614	$1,670	$1,757	$1,983	$2,251	$2,407	$3,181			**$1,488**
Subtotal	**$/m**	**$1,277**	**$1,727**	**$2,030**	**$2,232**	**$2,706**	**$2,725**	**$2,828**	**$3,060**	**$3,303**	**$3,621**	**$4,313**			**$2,422**
	$'000	**$2,371**	**$2,386**	**$3,132**	**$4,340**	**$4,016**	**$3,780**	**$2,962**	**$2,062**	**$3,203**	**$1,388**	**$2,393**			**$32,032**
DEVELOPMENT COSTS, 2025 UPDATE ESTIMATE															
Development in waste	m	180	180	180	180	180	180	180	180	180	180	180	180	180	**2,340**
Subtotal	**$/m**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**	**$2,838**
	$'000	**$511**	**$511**	**$511**	**$511**	**$511**	**$511**	**$511**	**$511**	**$511**	**$511**	**$511**	**$511**	**$511**	**$6,641**
Development in ore	m	2,444	1,772	2,427	1,767	1,440	1,432	713	917	737	786	667	414	375	**15,893**
Subtotal	**$/m**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**	**$2,265**
	$'000	**$5,538**	**$4,015**	**$5,499**	**$4,004**	**$3,263**	**$3,244**	**$1,615**	**$2,078**	**$1,670**	**$1,781**	**$1,510**	**$939**	**$849**	**$36,005**

In Table 21-9 jumbo drilling accounts for:

- drilling consumables such as bits, rods and shanks
- jumbo tyres, tubes and rims

Blasting accounts for:

- ANFO (in dry conditions; nominally 90% usage)
  - 200 kg per 3.75 m development round
- emulsion (in wet conditions; nominally 10% usage)
  - 60 kg per 3.75 m development round
- detonators and other consumables
  - 60 detonators per 3.75 m development round

LHD (load, haul dump machine) and backhoe loading accounts for:

- Equipment tyres, tubes and rims

General and primary haulage accounts for:

- ore haulage to the surface blending
- waste haulage to
- truck tyres, tubes and rims

Rock bolting accounts for:

- rock bolt installations
  - nominal 98% usage of rebar bolts; nominal 2% usage of split sets
  - 2.4 m bolt length

  - 1 m x 1 m pattern
- cement (for rebar bolts)
- consumables for the robolter machine such as bits, rods and shanks

Cable bolting accounts for:

- 9 m long, grouted single strand, bulbed cables
  - installed at 2 m x 2 m spacing in poor ground conditions
  - installed with shotcrete arches, also in poor ground conditions
- wire mesh installation
- miscellaneous consumables such as fasteners, hoses and fittings

Shotcreting accounts for:

- consumables such as:
  - cement at 0.425 t/m^3 of development opening; in dry conditions (90%)
  - sand at 0.795 t/m^3 of development opening; in dry conditions (90%)
  - plasticiser at 6.5 kg/m^3 of development opening; in dry conditions (90%)
  - accelerator at 35 kg/m^3 of development opening; in dry conditions (90%)
  - silica addition at 0.004 t/m^3 of development opening; in wet conditions (10%)
  - steel fibre addition at 15 kg/m^3 of development opening; in wet conditions (10%)
- application at 7 to 10 cm thickness in all development openings, plus shotcrete arches
- additional LOM average allowance of 62% for rehabilitation application

Fuel (and power) costs are accounted for under “Mine Services”.

Stoping

Table 21-10 summarises the stoping costs according to operational cost reporting centres, relative to July 2024 budgeting, and then updated to reflect 2025 forecast physicals.

In Table 21-10 solo drilling accounts for:

- drilling consumables such as bits, rods and shanks

Blasting accounts for:

- ANFO (in dry conditions; nominally 90% usage)
  - 500 kg in blind slots
  - 250 kg in open slots
  - 250 kg in production rings
- emulsion (in wet conditions; nominally 10% usage)
  - 750 kg in blind slots
  - 40 kg in open slots
  - 300 kg in blind production rings
  - 20 kg in open production rings

- P1000
  - 60 kg in open slots
  - 40 kg in open production rings
- detonators and other consumables
  - 25 detonators per blind slot
  - 12 detonators per open slot
  - 10 detonators in blind production rings
  - 5 detonators in open production rings

Table 21-10 Stoping cost estimate

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
STOPING COSTS, 2024 BUDGET FIGURES															
Stoping	t	423,021	561,797	595,745	555,515	601,591	461,268	395,262	282,632	253,042	311,683	244,520			**4,686,079**
Solo drilling	$/t	$0.49	$0.36	$0.36	$0.39	$0.36	$0.37	$0.36	$0.35	$0.38	$0.33	$0.35			**$0.37**
Blasting	$/t	$1.14	$1.08	$1.07	$1.08	$1.06	$1.06	$1.06	$1.06	$1.07	$1.06	$1.06			**$1.07**
LHD loading	$/t	$0.57	$0.42	$0.41	$0.44	$0.40	$0.41	$0.41	$0.40	$0.43	$0.38	$0.40			**$0.43**
General Haulage	$/t	$0.29	$0.41	$0.41	$0.37	$0.45	$0.45	$0.43	$0.54	$0.56	$0.30	$0.34			**$0.41**
Primary Haulage	$/t	$0.13	$0.28	$0.30	$0.34	$0.37	$0.34	$0.35	$0.35	$0.30	$0.41	$0.35			**$0.32**
Shotcreting	$/t	$0.37	$0.32	$0.15	$0.18	$0.15	$0.13	$0.16	$0.16	$0.15	$0.11	$0.18			**$0.19**
Barricading	$/t	$1.03	$0.73	$0.59	$0.64	$0.59	$0.57	$0.59	$0.60	$0.49	$0.49	$0.49			**$0.63**
Paste filling	$/t	$2.23	$2.04	$2.13	$2.13	$2.13	$2.13	$2.13	$2.13	$2.13	$2.13	$2.13			**$2.13**
Subtotal	**$/t**	**$6.25**	**$5.65**	**$5.41**	**$5.56**	**$5.51**	**$5.47**	**$5.50**	**$5.61**	**$5.50**	**$5.21**	**$5.31**			$5.56
	$'000	**$2,642.6**	**$3,172.9**	**$3,223.6**	**$3,089.6**	**$3,315.0**	**$2,523.2**	**$2,172.0**	**$1,584.4**	**$1,392.8**	**$1,623.0**	**$1,298.6**			$26,037.8
STOPING COSTS, 2025 UPDATE ESTIMATE															
Stoping	t	455,561	572,775	607,256	673,263	705,953	556,819	528,703	408,279	376,275	271,377	283,331	258,568	262,515	**5,960,675**
Subtotal	**$/t**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	**$5.56**	$5.56
	$'000	**$2,531.3**	**$3,182.6**	**$3,374.2**	**$3,740.9**	**$3,922.6**	**$3,093.9**	**$2,937.7**	**$2,268.6**	**$2,090.7**	**$1,507.9**	**$1,574.3**	**$1,436.7**	**$1,458.6**	**$33,120.0**

LHD loading accounts for:

- equipment tyres, tubes and rims

General and primary haulage accounts for:

- ore haulage to surface stockpile(s)
- waste haulage to minor surface storage sites (e.g., the North Waste Area, NWA)
- truck tyres, tubes and rims

Shotcreting accounts for:

- consumables as listed for development openings
- application to backfill barricades

Barricading accounts for:

- construction items such as timber planks, plywood and ribbed iron bolts

Paste filling accounts for:

- cement addition as follows:
  - Main Orebody (4.5% cement); 0.095 t/m^3
  - South Orebody (6.8% cement); 0.114 t/m^3

Fuel (and power) costs are accounted for under “Mine Services”.

Mine services

Table 21-11 summarises the mine services costs according to operational cost reporting centres, relative to July 2024 budgeting, and then updated to reflect 2025 forecast physicals.

Table 21-11 Mine services cost estimate

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
SERVICES COSTS, 2024 BUDGET FIGURES															
Common services	t o+w	700,000	718,000	768,000	768,000	768,000	618,000	518,000	368,000	368,000	368,000	318,000			**6,280,000**
Service hole drilling	$/t o+w	$0.51	$0.66	$0.69	$0.76	$0.76	$0.76	$0.75	$0.74	$0.74	$0.74	$0.74			**$0.71**
Cleaning	$/t o+w	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00			**$0.00**
Support Services (incl. vent ducting)	$/t o+w	$2.45	$2.96	$2.98	$3.26	$3.20	$3.35	$3.31	$3.24	$3.67	$3.23	$3.32			**$3.13**
Primary Vent.	$/t o+w	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00			**$0.00**
Roadworks	$/t o+w	$0.23	$0.30	$0.31	$0.34	$0.34	$0.34	$0.34	$0.34	$0.34	$0.34	$0.33			**$0.32**
Engineering	$/t o+w	$0.81	$1.00	$0.98	$1.08	$1.08	$1.12	$1.15	$1.15	$1.15	$1.11	$1.03			**$1.04**
Utilities (power)	$/t o+w	$1.53	$2.53	$2.92	$3.53	$3.84	$4.11	$4.38	$4.61	$4.89	$5.18	$5.42			**$3.62**
Facilities (fuel supply)	$/t o+w	$1.66	$2.12	$2.22	$2.46	$2.47	$2.45	$2.44	$2.40	$2.40	$2.40	$2.38			**$2.29**
Facilities (lubricants)	$/t o+w	$0.09	$0.12	$0.13	$0.14	$0.14	$0.14	$0.14	$0.14	$0.14	$0.14	$0.14			**$0.13**
Facilities (other)	$/t o+w	$0.46	$0.42	$0.42	$0.66	$0.49	$0.48	$0.59	$0.69	$0.52	$0.52	$0.54			**$0.52**
Geology	$/t o+w	$0.41	$0.50	$0.49	$0.54	$0.54	$0.54	$0.54	$0.54	$0.54	$0.54	$0.54			**$0.52**
Diamond drilling	$/t o+w	$0.79	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01			**$0.10**
Subtotal	**$/t o+w**	**$8.95**	**$10.62**	**$11.14**	**$12.80**	**$12.90**	**$13.32**	**$13.67**	**$13.87**	**$14.42**	**$14.23**	**$14.47**			**$12.38**
	$'000	**$6,264.5**	**$7,626.7**	**$8,556.6**	**$9,830.4**	**$9,904.7**	**$8,231.3**	**$7,081.6**	**$5,103.2**	**$5,305.6**	**$5,235.8**	**$4,602.7**			**$77,743.3**
SERVICES COSTS, 2025 UPDATE ESTIMATE															
Common services	t o+w	718,000	768,000	868,000	868,000	868,000	718,000	618,000	518,000	468,000	368,000	368,000	318,000	318,000	**7,784,000**
Subtotal	**$/t o+w**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**	**$12.38**
	$'000	**$8,888.5**	**$9,507.5**	**$10,745.4**	**$10,745.4**	**$10,745.4**	**$8,888.5**	**$7,650.5**	**$6,412.6**	**$5,793.6**	**$4,555.7**	**$4,555.7**	**$3,936.7**	**$3,936.7**	**$96,362.0**

In Table 21-11 service hole drilling accounts for:

- drilling consumables such as bits, rods and shanks

Cleaning refers to decline maintenance and this item accounts for:

- equipment tyres, tubes and rims

Support services account for:

- miscellaneous ground support and drilling services
- ventilation ducting and related fittings
- miscellaneous tools, fittings, hoses etc

Primary ventilation accounts for:

- minor allowances for equipment tyres, tubes and rims

Roadworks accounts for:

- an allowance for waste backfilling into abandoned development openings

Engineering accounts for:

- consulting services
- ground movement monitoring equipment
- various survey and monitoring services

Utilities (power) accounts for:

- power supply at a rate of $0.109/kW

Facilities (various) accounts for:

- diesel fuel supply at a rate of $1.03/L

- oils and lubricants
- miscellaneous contracted services
- supplies, food and catering

Geology accounts for:

- leases for buildings

Diamond drilling accounts for:

- underground diamond drilling in 2025, and with an allowance for minimal fixed costs thereafter

Mining maintenance

Table 21-12 summarises the mining maintenance costs according to operational cost reporting centres, relative to July 2024 budgeting, and then updated to reflect 2025 forecast physicals.

These costs are variable costs, whereas the corresponding fixed costs are itemised under maintenance charges in the plant costs (Item 21.3.3). The maintenance variable costs were estimated by ÇBI in total dollar terms for 2025 only, and then tonnage pro-rata estimated for the following years.

Table 21-12 Mining maintenance cost estimate

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
MAINTENANCE COSTS, 2024 BUDGET FIGURES															
Mined Tonnes (ore+waste)	t o+w	700,000	718,000	768,000	768,000	768,000	618,000	518,000	368,000	368,000	368,000	318,000			**6,280,000**
Consumables	$'000	$1,217.8	$1,249.1	$1,336.0	$1,336.0	$1,336.0	$1,075.1	$901.1	$640.2	$640.2	$640.2	$553.2			**$10,925.0**
Fuel and lubricants	$'000	$178.0	$182.6	$195.3	$195.3	$195.3	$157.1	$131.7	$93.6	$93.6	$93.6	$80.9			**$1,596.8**
Parts	$'000	$2,226.8	$2,284.1	$2,443.1	$2,443.1	$2,443.1	$1,966.0	$1,647.8	$1,170.7	$1,170.7	$1,170.7	$1,011.6			**$19,977.6**
Non-capital equipment	$'000	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.2	$0.2	$0.2	$0.2	$0.1			**$2.6**
Materials and inventory	$'000	$1.4	$1.4	$1.5	$1.5	$1.5	$1.2	$1.0	$0.7	$0.7	$0.7	$0.6			**$12.5**
Services	$'000	$123.7	$126.9	$135.7	$135.7	$135.7	$109.2	$91.5	$65.0	$65.0	$65.0	$56.2			**$1,109.6**
Other administrative costs	$'000	$0.2	$0.2	$0.3	$0.3	$0.3	$0.2	$0.2	$0.1	$0.1	$0.1	$0.1			**$2.1**
Total	**$'000**	**$3,748.1**	**$3,844.5**	**$4,112.3**	**$4,112.3**	**$4,112.3**	**$3,309.1**	**$2,773.6**	**$1,970.5**	**$1,970.5**	**$1,970.5**	**$1,702.7**			**$33,626.3**
$/t ore+waste	**$/t o+w**	**$5.35**	**$5.35**	**$5.35**	**$5.35**	**$5.35**	**$5.35**	**$5.35**	**$5.35**	**$5.35**	**$5.35**	**$5.35**			**$5.35**
MAINTENANCE COSTS, 2025 UPDATE ESTIMATE															
Mined Tonnes (ore+waste)	t o+w	718,000	768,000	868,000	868,000	868,000	718,000	618,000	518,000	468,000	368,000	368,000	318,000	318,000	**7,784,000**
Total	**$'000**	**$3,748.1**	**$3,577.8**	**$3,833.3**	**$3,833.3**	**$3,833.3**	**$3,207.8**	**$2,627.0**	**$2,046.2**	**$2,001.5**	**$1,420.7**	**$1,599.4**	**$1,331.4**	**$1,331.4**	**$34,391.4**
$/t ore+waste	**$/t o+w**	**$5.22**	**$4.66**	**$4.42**	**$4.42**	**$4.42**	**$4.47**	**$4.25**	**$3.95**	**$4.28**	**$3.86**	**$4.35**	**$4.19**	**$4.19**	**$4.42**

Mining maintenance labour

Hence Table 21-13 summarises the mining maintenance labour costs relative to the 2025 forecast physicals.

Table 21-13 Additional mining labour cost estimate

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
ADDITIONAL LABOUR COSTS, 2025 UPDATE ESTIMATE															
Mined Tonnes (ore+waste)	t o+w	718,000	768,000	868,000	868,000	868,000	718,000	618,000	518,000	468,000	368,000	368,000	318,000	318,000	**7,784,000**
Total	**$'000**	**$3,285.0**	**$3,197.8**	**$3,183.3**	**$3,161.5**	**$3,139.7**	**$2,627.4**	**$2,151.7**	**$1,676.0**	**$1,639.4**	**$1,163.7**	**$1,310.0**	**$1,090.5**	**$1,090.5**	**$28,716.4**
$/t ore+waste	**$/t o+w**	**$4.58**	**$4.16**	**$3.67**	**$3.64**	**$3.62**	**$3.66**	**$3.48**	**$3.24**	**$3.50**	**$3.16**	**$3.56**	**$3.43**	**$3.43**	**$3.69**

Mining cost summary

Table 21-14 summarises the LOM mine operating costs in total dollar and unit cost terms.

Table 21-14 Total mine operating cost estimate

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
2024 BUDGET FIGURES															
Development in waste	$'000	$0.0	$310.6	$364.1	$397.5	$483.7	$486.2	$505.5	$547.6	$588.5	$650.9	$774.0			**$5,108.5**
Development in ore	$'000	$2,370.8	$2,386.1	$3,132.0	$4,340.0	$4,016.3	$3,779.9	$2,961.8	$2,061.7	$3,202.6	$1,387.6	$2,392.8			**$32,031.5**
Stoping	$'000	$2,642.6	$3,172.9	$3,223.6	$3,089.6	$3,315.0	$2,523.2	$2,172.0	$1,584.4	$1,392.8	$1,623.0	$1,298.6			**$26,037.8**
Services (ore + waste)	$'000	$6,264.5	$7,626.7	$8,556.6	$9,830.4	$9,904.7	$8,231.3	$7,081.6	$5,103.2	$5,305.6	$5,235.8	$4,602.7			**$77,743.3**
Maintenance	$'000	$3,748.1	$3,844.5	$4,112.3	$4,112.3	$4,112.3	$3,309.1	$2,773.6	$1,970.5	$1,970.5	$1,970.5	$1,702.7			**$33,626.3**
Total	**$'000**	**$15,026.1**	**$17,340.8**	**$19,388.6**	**$21,769.7**	**$21,832.0**	**$18,329.7**	**$15,494.6**	**$11,267.4**	**$12,460.0**	**$10,867.8**	**$10,770.8**			**$174,547.4**
	$/t ore	***$21.47***	***$24.77***	***$25.85***	***$29.03***	***$29.11***	***$30.55***	***$30.99***	***$32.19***	***$35.60***	***$31.05***	***$35.90***			***$28.61***
2025 UPDATE ESTIMATE															
Development in waste	$'000	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	$510.8	**$6,641.0**
Development in ore	$'000	$5,537.6	$4,014.9	$5,499.2	$4,003.9	$3,263.3	$3,243.7	$1,615.2	$2,077.9	$1,670.2	$1,781.2	$1,510.3	$938.6	$849.2	**$36,005.2**
Stoping	$'000	$2,531.3	$3,182.6	$3,374.2	$3,740.9	$3,922.6	$3,093.9	$2,937.7	$2,268.6	$2,090.7	$1,507.9	$1,574.3	$1,436.7	$1,458.6	**$33,120.0**
Services (ore + waste)	$'000	$8,888.5	$9,507.5	$10,745.4	$10,745.4	$10,745.4	$8,888.5	$7,650.5	$6,412.6	$5,793.6	$4,555.7	$4,555.7	$3,936.7	$3,936.7	**$96,362.0**
Maintenance	$'000	$3,748.1	$3,577.8	$3,833.3	$3,833.3	$3,833.3	$3,207.8	$2,627.0	$2,046.2	$2,001.5	$1,420.7	$1,599.4	$1,331.4	$1,331.4	**$34,391.4**
Mining maintenance labour	$'000	$3,285.0	$3,197.8	$3,183.3	$3,161.5	$3,139.7	$2,627.4	$2,151.7	$1,676.0	$1,639.4	$1,163.7	$1,310.0	$1,090.5	$1,090.5	**$28,716.4**
Total	**$'000**	**$24,501.4**	**$23,991.4**	**$27,146.2**	**$25,995.9**	**$25,415.1**	**$21,572.1**	**$17,493.0**	**$14,992.1**	**$13,706.3**	**$10,939.9**	**$11,060.6**	**$9,244.7**	**$9,177.2**	**$235,236.1**
	$/t ore	***$35.00***	***$31.99***	***$31.94***	***$30.58***	***$29.90***	***$30.82***	***$29.15***	***$29.98***	***$30.46***	***$31.26***	***$31.60***	***$30.82***	***$30.59***	***$31.16***

The 2024 budgeted ÇBI haulage costs do not explicitly specify a variance relating to the different mine sublevels in operation throughout the LOM period. However, the overall annual costs reflect an observation that numerous sublevels and stopes are in operation in any one year, as indicated from budget LOM scheduling in 2024. For example, Table 21-15 lists a total of 69 sublevel headings being developed at different times during 2025, spread from top to bottom of the mine. Similarly, Table 21-16 lists a total of 71 stopes in operation during the same period. This suggests that the annual mine development and stoping cost estimates reflect overall average ore and waste haul profiles from multiple operating sublevels.

Table 21-15 2024 budget LOM scheduling; mined development headings between 2025 and 2028

	2025		2026		2027		2028	
	# headings	# months	# headings	# months	# headings	# months	# headings	# months
1200L	3	3	6	5	2	2	2	2
1175L	5	5	6	5	2	2	2	2
1150L	6	4	4	4	4	4	2	2
1125L	7	4	2	2				
1100L	6	4						
1075L	4	3	5	5	2	2	4	4
1050L	7	5	6	6	2	2	4	4
1025L	9	4	8	7	4	4	4	4
1000L	11	8	4	4	2	2		
975L	8	6	4	4	2	2		
950L								
925L	1	1						
900L	2	2						
	69		**45**		**20**		**18**	

Table 21-16 2024 budget LOM scheduling; operating stopes between 2025 and 2028

	2025		2026		2027		2028	
	# stopes	# months	# stopes	# months	# stopes	# months	# stopes	# months
1200L	3	3	4	4	4	4	2	2
1175L	3	2	5	5	6	6	5	5
1150L	9	6	8	8	2	2	5	5
1125L	6	6	6	5	1	1	2	2
1100L	6	6						
1075L	5	5	4	4	2	2		
1050L	9	5	6	5	6	6		
1025L	8	4	10	8	6	6		
1000L	10	8	5	5	4	4		
975L	8	7	7	7	3	3		
950L	1	1						
925L	1	1						
900L	2	2						
	71		**55**		**34**		**14**	

In the preceding commentary it was mentioned that the fuel costs (i.e., associated with ore and waste haulage) were included with the mine services costs, rather than being allocated to development and stoping costs. With the fuel (and lubricant) costs reallocated, Figure 21.1 shows the complete operating cost components for mine development. Figure 21.2 shows the complete operating cost components for stope production.

In relation to mine development, the combined diesel fuel and haulage costs are substantially outweighed by the intensive ground support costs. In the case of stope production, the combined diesel fuel and haulage costs are comparable with the combined costs for extensive barricade construction and paste filling.

Figure 21.1 Operating cost chart (LOM averages) for mine development

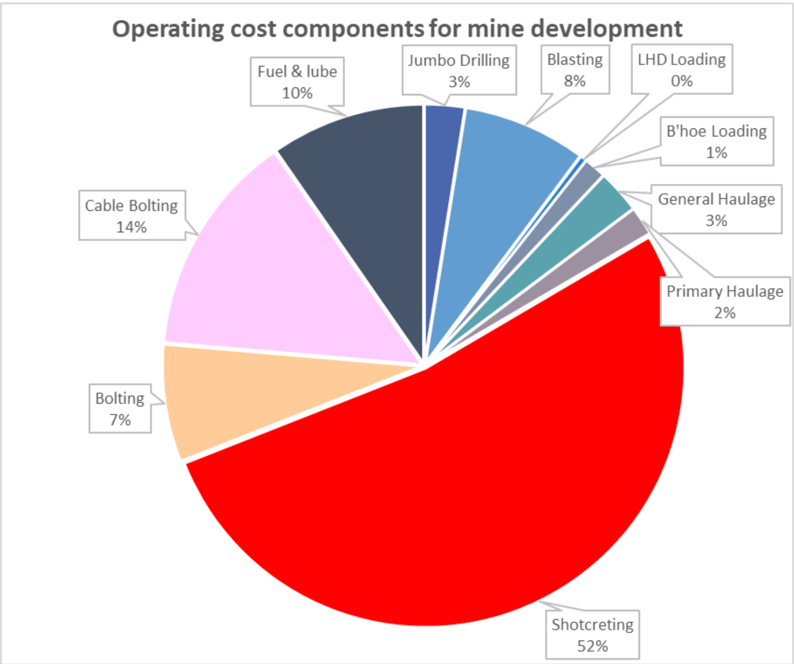
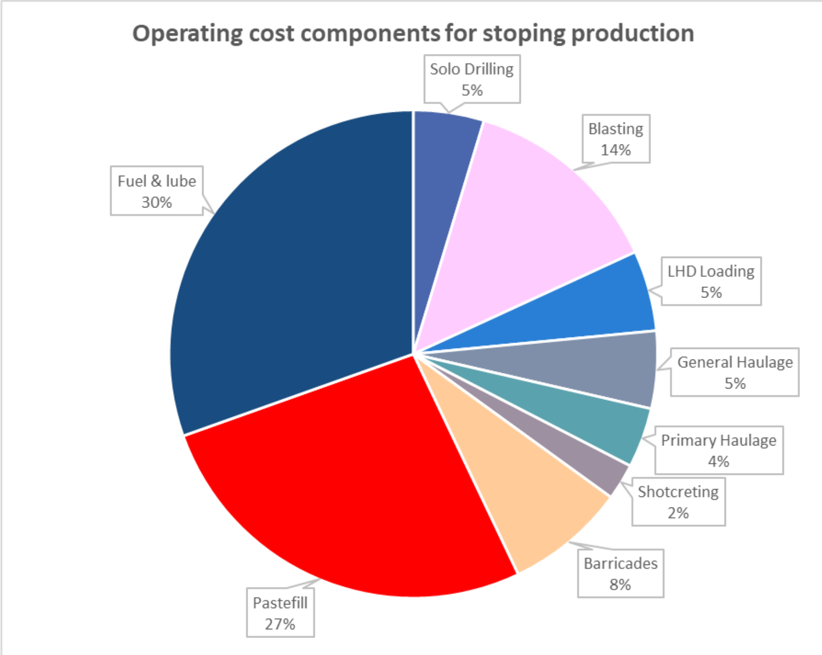


Figure 21.2 Operating cost chart (LOM averages) for stoping production



21.3.2 Milling (and processing) costs

Unit operating costs for Çayeli milling and processing have been estimated using actual costs for treatment of the Main Orebody, and from testwork data for mill and reagent consumptions for the South Orebody. The costs have been divided into the following discrete cost centres for the process plant:

- operating consumables
- energy
- operating labour
- maintenance
- concentrate handling costs

Details on the derivation of the operating costs are described below.

Table 21-17 below provides the yearly process operating costs. These are calculated using the actual cost in $/t calculated in Table 21-18 and Table 21-19 and the respective tonnages of ores from the Main Orebody and South Orebody treated per year.

Table 21-17 Milling (and processing) cost estimate, 2024 budget figures

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	LOM
MILLING COSTS													
Main Orebody feed	t	328,584	129,490									53,480	**511,554**
Crusher Plant Liners & Panels	$k	$75.4	$29.7									$12.3	**$117.4**
	$/t	*$0.23*	*$0.23*									*$0.23*	***$0.23***
Mill Liners	$k	$112.9	$44.5									$18.4	**$175.8**
	$/t	*$0.34*	*$0.34*									*$0.34*	***$0.34***
Grinding Media	$k	$781.6	$308.0									$127.2	**$1,216.8**
	$/t	*$2.38*	*$2.38*									*$2.38*	***$2.38***
Reagents	$k	$302.3	$119.1									$49.2	**$470.7**
	$/t	*$0.92*	*$0.92*									*$0.92*	***$0.92***
Power	$k	$1,643.5	$647.7									$267.5	**$2,558.7**
	$/t	*$5.00*	*$5.00*									*$5.00*	***$5.00***
South Orebody feed	t	371,416	570,510	750,000	750,000	750,000	600,000	500,000	350,000	350,000	350,000	246,520	**5,588,446**
Crusher Plant Liners & Panels	$k	$116.9	$179.5	$236.0	$236.0	$236.0	$188.8	$157.3	$110.1	$110.1	$110.1	$77.6	**$1,758.5**
	$/t	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	***$0.31***
Mill Liners	$k	$175.0	$268.8	$353.3	$353.3	$353.3	$282.7	$235.6	$164.9	$164.9	$164.9	$116.1	**$2,632.9**
	$/t	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	***$0.47***
Grinding Media	$k	$1,210.9	$1,860.1	$2,445.2	$2,445.3	$2,445.3	$1,956.2	$1,630.2	$1,141.1	$1,141.1	$1,141.1	$803.7	**$18,220.2**
	$/t	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	***$3.26***
Reagents	$k	$412.3	$633.3	$832.5	$832.5	$832.5	$666.0	$555.0	$388.5	$388.5	$388.5	$273.6	**$6,203.4**
	$/t	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	***$1.11***
Power	$k	$2,231.4	$3,427.5	$4,505.8	$4,505.8	$4,505.8	$3,604.7	$3,003.9	$2,102.7	$2,102.7	$2,102.7	$1,481.0	**$33,574.2**
	$/t	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	***$6.01***
Fixed costs													
Operating Labour	$k	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	**$33,000.0**
	$/t	*$4.29*	*$4.29*	*$4.00*	*$4.00*	*$4.00*	*$5.00*	*$6.00*	*$8.57*	*$8.57*	*$8.57*	*$10.00*	***$5.41***
Maintenance charges													
Covered under "Plant" costs	$k	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	**$0.0**
	$/t	*$0.00*	*$0.00*	*$0.00*	*$0.00*	*$0.00*	*$0.00*	*$0.00*	*$0.00*	*$0.00*	*$0.00*	*$0.00*	***$0.00***
subtotal	**$k**	**$10,062.2**	**$10,518.2**	**$11,373.0**	**$11,373.0**	**$11,373.0**	**$9,698.4**	**$8,582.0**	**$6,907.4**	**$6,907.4**	**$6,907.4**	**$6,226.7**	**$99,928.6**
subtotal	***$/t***	***$14.37***	***$15.03***	***$15.16***	***$15.16***	***$15.16***	***$16.16***	***$17.16***	***$19.74***	***$19.74***	***$19.74***	***$20.76***	***$16.38***

Details of the derivation of these costs are presented in Table 21-18 and Table 21-19, including consumption rates, costs and annual costs in terms of $/t treated.

Consumable costs

Table 21-18 below details the consumable costs for treatment of the Main Orebody feed. These costs are derived from the actual costs for the year 2023. Grinding media and reagent consumption rates have been back calculated using the actual costs for the year and reflect the average consumption rates from treating the range of different ore types.

Table 21-18 Consumable costs for the Main Orebody

Main Ore Body Figures							
Item		Cost to Site US$/Unit	Unit	Consumption Rate	Annual Consumption	TOTAL	
						US$/a	US$/t
Crusher Plant Liners & Panels		**$0.23**	**$/t**				
Primary Crusher Liner	Jaw Crusher	$0.04	$/t		750,000	$32,756	$0.04
Secondary Crusher Liner	Cone Crusher	$0.09	$/t		750,000	$67,770	$0.09
Screen Panels	Double Deck S.	$0.10	$/t		750,000	$71,650	$0.10
Mill Liners		**$0.34**	**$/t**				
PBM	Rubber	$0.25	$/t		750,000	$186,068	$0.25
SBM	Rubber	$0.10	$/t		750,000	$71,715	$0.10
Cu Regrind	Rubber	$0.00	$/t		750,000	$0	$0.00
Subtotal Liners						**$429,959**	**$0.57**
Grinding Media							
PBM Mill Balls	80 mm	$1.51	kg	229.29 g/t ore	172 t	$259,667	$0.35
SBM Mill Balls	40 mm	$1.26	kg	802.74 g/t ore	602 t	$758,589	$1.01
SBM Mill Balls	25 mm	$1.29	kg	791.40 g/t ore	594 t	$765,678	$1.02
Regrind Mill Balls	20 mm	$0.00	kg	0.00 g/t ore	0 t	$0	$0.00
Subtotal Grinding Media						**$1,783,934**	**$2.38**
Reagents							
Collector - Cu and Zn	SIPX	$2.28	kg	54.99 g/t ore	41 t	$94,033	$0.13
Collector - Cu	3418A	$16.21	kg	7.34 g/t ore	6 t	$89,230	$0.12
Collector - Cu	A-208	$7.18	kg	24.37 g/t ore	18 t	$131,252	$0.18
Frother	MIBC	$3.01	kg	29.17 g/t ore	22 t	$65,860	$0.09
Zn Activator	CuSO4	$2,219.80	t	140.52 g/t ore	105 t	$233,948	$0.31
Deppressant	SMBS	$396.73	t	118.14 g/t ore	89 t	$35,152	$0.05
Lump Lime Bulk		$66.85	t	730.44 g/t ore	548 t	$36,622	$0.05
Flocculant	NEWFLOC 6335A & Enfloc 5300ah	$2.14	kg	2.46 g/t ore	2 t	$3,953	$0.01
Subtotal Reagents						**$690,050**	**$0.92**
Electricity							
Electric Power		$5.00	$/t		750,000 t	$3,751,372	$5.00
Subtotal Electricity						**$3,751,372**	**$5.00**
TOTAL						**$6,655,315**	**$8.87**

Similar costs for the South Orebody are shown in Table 21-19. Consumption rates have been obtained from testwork data described in two reports from Hacettepe University (2024a, 2024b).

The South Orebody consists of three distinct ore types – Footwall, Low Zinc, and High Zinc ores. Reagent consumptions differ between the ore types; weighted average consumption rates have been estimated using a LOM mix of 75% footwall, 13% low zinc and 12% high zinc ores.

Energy costs

Energy costs for the Çayeli processing facilities averaged $5.00 per tonne in 2023, when treating Main Orebody feed.

These costs were adjusted for each of the South Orebody ore types, reflective of the ball mill work indices from Hacettepe testwork, and assuming that 60% of the energy consumption occurs in the milling circuit. The average energy cost was then calculated using the ratio of ore types in the LOM feed. A figure of $6.01/t for energy costs is included in Table 21-19.

Table 21-19 Consumable costs for the South Orebody

South Ore Body Figures							
Item		Cost to Site US$/Unit	Unit	Consumption Rate	Annual Consumption	TOTAL	
						US$/a	US$/t
Crusher Plant Liners & Panels		**$0.31**	**$/t**				
Primary Crusher Liner	Jaw Crusher	$0.06	$/t		750,000	$44,899	$0.06
Secondary Crusher Liner	Cone Crusher	$0.12	$/t		750,000	$92,893	$0.12
Screen Panels	Double Deck S.	$0.13	$/t		750,000	$98,211	$0.13
Mill Liners		**$0.47**	**$/t**				
PBM	Rubber	$0.34	$/t		750,000	$255,044	$0.34
SBM	Rubber	$0.13	$/t		750,000	$98,300	$0.13
Cu Regrind	Rubber	$0.00	$/t		750,000	$0	$0.00
Subtotal Liners						**$589,348**	**$0.79**
Grinding Media				**2.50 kg/t ore**			
PBM Mill Balls	80 mm	$1.51	kg	0.31 kg/t ore	235,714 kg	$355,928	$0.47
SBM Mill Balls	40 mm	$1.26	kg	1.10 kg/t ore	825,240 kg	$1,039,803	$1.39
SBM Mill Balls	25 mm	$1.29	kg	1.08 kg/t ore	813,582 kg	$1,049,520	$1.40
Regrind Mill Balls	20 mm	$0.00	kg	0.00 kg/t ore	0 kg	$0	$0.00
Subtotal Grinding Media						**$2,445,251**	**$3.26**
Reagents							
Collector - Cu and Zn	SIPX	$2.28	kg	92.42 g/t ore	69 t	$158,042	$0.21
Collector - Cu	3418A	$16.21	kg		0 t	$0	$0.00
Collector - Cu	A-208	$7.18	kg	13.73 g/t ore	10 t	$73,938	$0.10
Frother	MIBC	$3.01	kg	31.64 g/t ore	24 t	$71,435	$0.10
Zn Activator	CuSO4	$2,219.80	t	263.69 g/t ore	198 t	$439,003	$0.59
Deppressant	SMBS	$396.73	t	121.07 g/t ore	91 t	$36,024	$0.05
Lump Lime Bulk		$66.85	t	1000.00 g/t ore	750 t	$50,138	$0.07
Flocculant	NEWFLOC 6335A & Enfloc 5300ah	$2.14	kg	2.46 g/t ore	2 t	$3,953	$0.01
Subtotal Reagents						**$832,534**	**$1.11**
Electricity							
Electric Power		$6.01	$/t		750,000 t	$4,505,841	$6.01
Subtotal Electricity						**$4,505,841**	**$6.01**
TOTAL						**$8,372,974**	**$11.16**

Labour costs

Labour costs for processing were supplied by ÇBI in August 2024. Costs for 53 supervisors and operators were estimated to be $3.0M pa, as detailed in Table 21-20.

Table 21-20 Process Labour Costs

Mill	Day Shift	Per Shift	Total	$pa (each)	$pa total
Plant Manager	1		1	190,000	190,000
Process Superintendent	1		1	150,000	150,000
Sr. Process Engineer	1		1	130,000	130,000
Process Engineer	3		3	65,000	195,000
Jr. Process Engineer	1		1	40,000	40,000
Area Supervisor	3		3	75,000	225,000
Shift Supervisor		1	4	70,000	280,000
Operator	1	6	25	42,000	1,050,000
Pastefill Operator	5		5	42,000	210,000
Chief Chemist	1		1	80,000	80,000
Chemist	2		2	60,000	120,000
Laboratory Technicians	6		6	55,000	330,000
Totals	**25**	**28**	**53**		**3,000,000**

These costs are fixed at $3.0M pa but vary in terms of $/t as the treatment rate varies.

Milling (and processing) cost summary

Table 21-21 summarises the estimated costs for the milling (processing) department, originally based on July 2024 budgeting, and then updated to reflect February 2025 forecast physicals.

Table 21-21 Milling (and processing) cost estimate, updated to reflect February 2025 forecast physicals

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
MILLING COSTS															
Main Orebody feed	t	373,086	244,462	109,610	24,871						19,116		24,669	125,598	**921,412**
Crusher Plant Liners & Panels	$k	$85.6	$56.1	$25.2	$5.7						$4.4		$5.7	$28.8	**$211.5**
	$/t	*$0.23*	*$0.23*	*$0.23*	*$0.23*						*$0.23*		*$0.23*	*$0.23*	***$0.23***
Mill Liners	$k	$128.2	$84.0	$37.7	$8.5						$6.6		$8.5	$43.2	**$316.7**
	$/t	*$0.34*	*$0.34*	*$0.34*	*$0.34*						*$0.34*		*$0.34*	*$0.34*	***$0.34***
Grinding Media	$k	$887.4	$581.5	$260.7	$59.2						$45.5		$58.7	$298.7	**$2,191.7**
	$/t	*$2.38*	*$2.38*	*$2.38*	*$2.38*						*$2.38*		*$2.38*	*$2.38*	***$2.38***
Reagents	$k	$343.3	$224.9	$100.8	$22.9						$17.6		$22.7	$115.6	**$847.8**
	$/t	*$0.92*	*$0.92*	*$0.92*	*$0.92*						*$0.92*		*$0.92*	*$0.92*	***$0.92***
Power	$k	$1,866.1	$1,222.8	$548.3	$124.4						$95.6		$123.4	$628.2	**$4,608.7**
	$/t	*$5.00*	*$5.00*	*$5.00*	*$5.00*						*$5.00*		*$5.00*	*$5.00*	***$5.00***
South Orebody feed	t	326,914	505,538	740,389	825,129	850,000	700,000	600,000	500,000	450,000	330,884	350,000	275,331	174,402	**6,628,587**
Crusher Plant Liners & Panels	$k	$102.9	$159.1	$233.0	$259.6	$267.5	$220.3	$188.8	$157.3	$141.6	$104.1	$110.1	$86.6	$54.9	**$2,085.8**
	$/t	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	*$0.31*	***$0.31***
Mill Liners	$k	$154.0	$238.2	$348.8	$388.7	$400.5	$329.8	$282.7	$235.6	$212.0	$155.9	$164.9	$129.7	$82.2	**$3,122.9**
	$/t	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	*$0.47*	***$0.47***
Grinding Media	$k	$1,065.8	$1,648.2	$2,413.9	$2,690.2	$2,771.3	$2,282.2	$1,956.2	$1,630.2	$1,467.2	$1,078.8	$1,141.1	$897.7	$568.6	**$21,611.4**
	$/t	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	*$3.26*	***$3.26***
Reagents	$k	$362.9	$561.2	$821.9	$915.9	$943.5	$777.0	$666.0	$555.0	$499.5	$367.3	$388.5	$305.6	$193.6	**$7,358.0**
	$/t	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	*$1.11*	***$1.11***
Power	$k	$1,964.0	$3,037.2	$4,448.1	$4,957.2	$5,106.6	$4,205.5	$3,604.7	$3,003.9	$2,703.5	$1,987.9	$2,102.7	$1,654.1	$1,047.8	**$39,823.1**
	$/t	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	*$6.01*	***$6.01***
Fixed costs															
Operating Labour	$k	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	$3,000.0	**$39,000.0**
	$/t	*$4.29*	*$4.00*	*$3.53*	*$3.53*	*$3.53*	*$4.29*	*$5.00*	*$6.00*	*$6.67*	*$8.57*	*$8.57*	*$10.00*	*$10.00*	***$5.17***
Maintenance charges															
Additional labour	$k	$1,853.5	$1,678.0	$1,796.8	$1,795.8	$1,795.8	$1,502.8	$1,230.7	$958.6	$937.7	$665.6	$749.3	$623.7	$623.7	**$16,212.1**
	$/t	*$2.65*	*$2.24*	*$2.11*	*$2.11*	*$2.11*	*$2.15*	*$2.05*	*$1.92*	*$2.08*	*$1.90*	*$2.14*	*$2.08*	*$2.08*	***$2.15***
subtotal	**$k**	**$11,813.8**	**$12,491.1**	**$14,035.1**	**$14,228.2**	**$14,285.2**	**$12,317.6**	**$10,929.1**	**$9,540.6**	**$8,961.5**	**$7,529.2**	**$7,656.7**	**$6,916.4**	**$6,685.3**	**$137,389.7**
subtotal	***$/t***	***$16.88***	***$16.65***	***$16.51***	***$16.74***	***$16.81***	***$17.60***	***$18.22***	***$19.08***	***$19.91***	***$21.51***	***$21.88***	***$23.05***	***$22.28***	***$18.20***

To note with Table 21-21, and consistent with the absence of mining labour costs when tabulating the 2024 budget figures, additional labour costs relative to the February 2025 forecast physicals have now been included.

21.3.3 Plant (maintenance) costs

In ÇBI accounting parlance, "plant" costs refer to the plant and site maintenance costs. Table 21-22 lists the July 2024 budgeted bottom-line for plant costs in 2025, after deducting and reallocating the relevant maintenance costs to the mining cost centre. The projected costs for 2026 to 2035 were prorated on the feed tonnes for each year. This table also shows the updated estimate based on the February 2025 forecast physicals.

Table 21-22 Plant costs estimate

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
PLANT COSTS, 2024 BUDGET FIGURES															
	$k	$3,533.4	$3,437.0	$3,689.4	$3,689.4	$3,689.4	$2,932.3	$2,427.5	$1,670.3	$1,670.3	$1,670.3	$1,417.9			$29,827.3
	$/t	*$5.05*	*$4.91*	*$4.92*	*$4.92*	*$4.92*	*$4.89*	*$4.85*	*$4.77*	*$4.77*	*$4.77*	*$4.73*			*$4.89*
PLANT COSTS, 2025 UPDATE ESTIMATE															
	$k	$7,281.5	$7,122.3	$7,187.6	$7,147.8	$7,108.0	$5,948.2	$4,871.2	$3,794.2	$3,711.4	$2,634.4	$2,965.8	$2,468.7	$2,468.7	$64,710.0
	$/t	*$10.40*	*$9.50*	*$8.46*	*$8.41*	*$8.36*	*$8.50*	*$8.12*	*$7.59*	*$8.25*	*$7.53*	*$8.47*	*$8.51*	*$8.51*	*$8.66*

These maintenance costs account for labour, power, consumables, fuel, lubricants, spare parts, services and miscellaneous lesser charges. The costs for "services" specifically account for contracted maintenance and electrical services.

21.3.4 General and administration costs

Table 21-23 shows the July 2024 budgeted annual fixed cost for 2025 projected without change through to 2037. These are G&A charges (i.e., general and administration) for operations management personnel; the projection assumes that the management headcount remains constant for the LOM period.

Table 21-23 Administration costs estimate

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
ADMINISTRATION COSTS, 2024 BUDGET FIGURES															
	$k	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9			$115,290.2
	$/t	*$14.97*	*$14.97*	*$13.97*	*$13.97*	*$13.97*	*$17.47*	*$20.96*	*$29.95*	*$29.95*	*$29.95*	*$34.94*			*$18.90*
ADMINISTRATION COSTS, 2025 UPDATE ESTIMATE															
	$k	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$136,252.1
	$/t	*$14.97*	*$13.97*	*$12.33*	*$12.33*	*$12.33*	*$14.97*	*$17.47*	*$20.96*	*$23.29*	*$29.95*	*$29.95*	*$34.94*	*$34.94*	*$18.05*

These administration costs are those costs associated with the non-operating groups that support the Mining and Processing departments. They include accounting, human resources, logistics, information technology, legal, environmental, safety, security, community relations, training, and other administrative functions.

These administration costs account for:

- salaries and allowances (73% of the total)
- consulting and services contracts (14% of the total)

21.3.5 Concentrate handling costs

Table 21-24 shows the July 2024 budgeted costs associated with concentrate handling at the port of Rize in 2025. The projected costs for 2026 to 2037 were prorated on the feed tonnes for each year and updated to reflect the February 2025 forecast physicals.

Table 21-24 Concentrate handling costs estimate

		2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
CONCENTRATE HANDLING COSTS, 2024 BUDGET FIGURES															
	$k	$921.4	$921.4	$987.2	$987.2	$987.2	$789.8	$658.2	$460.7	$460.7	$460.7	$394.9			$8,029.4
	$/t	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*			*$1.32*
CONCENTRATE HANDLING COSTS, 2025 UPDATE ESTIMATE															
	$k	$921.4	$987.2	$1,118.9	$1,118.9	$1,118.9	$921.4	$789.8	$658.2	$592.3	$460.7	$460.7	$394.9	$394.9	$9,938.1
	$/t	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*

These costs include concentrate road transport from the mine, temporary concentrate storage, reclaim and ship loading.

21.3.6 Total operating costs

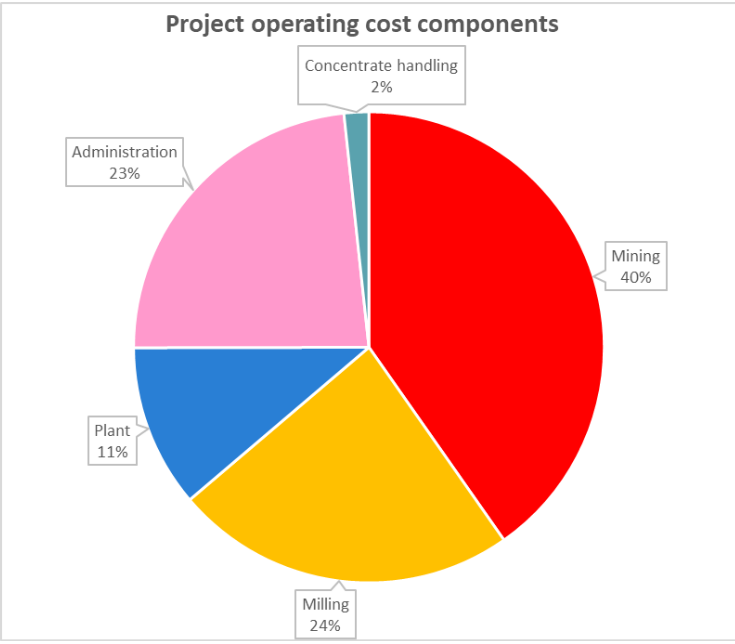
Table 21-25 shows the overall LOM operating costs in total dollar terms, based on 2024 budgeted costs adjusted to reflect the February 2025 forecast physicals. The denominator for the unit costs is the LOM production schedule tonnages.

Table 21-25 Total operating costs estimate

	UNITS	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	LOM
MINING COSTS															
subtotal	$k	$24,501.4	$23,991.4	$27,146.2	$25,995.9	$25,415.1	$21,572.1	$17,493.0	$14,992.1	$13,706.3	$10,939.9	$11,060.6	$9,244.7	$9,177.2	$235,236.1
	$/t	*$35.00*	*$31.99*	*$31.94*	*$30.58*	*$29.90*	*$30.82*	*$29.15*	*$29.98*	*$30.46*	*$31.26*	*$31.60*	*$30.82*	*$30.59*	*$31.16*
MILLING COSTS															
subtotal	$k	$11,813.8	$12,491.1	$14,035.1	$14,228.2	$14,285.2	$12,317.6	$10,929.1	$9,540.6	$8,961.5	$7,529.2	$7,656.7	$6,916.4	$6,685.3	$137,389.7
	$/t	*$16.88*	*$16.65*	*$16.51*	*$16.74*	*$16.81*	*$17.60*	*$18.22*	*$19.08*	*$19.91*	*$21.51*	*$21.88*	*$23.05*	*$22.28*	*$18.20*
PLANT COSTS															
subtotal	$k	$7,281.5	$7,122.3	$7,187.6	$7,147.8	$7,108.0	$5,948.2	$4,871.2	$3,794.2	$3,711.4	$2,634.4	$2,965.8	$2,468.7	$2,468.7	$64,710.0
	$/t	*$10.40*	*$9.50*	*$8.46*	*$8.41*	*$8.36*	*$8.50*	*$8.12*	*$7.59*	*$8.25*	*$7.53*	*$8.47*	*$8.51*	*$8.51*	*$8.66*
ADMINISTRATION COSTS															
subtotal	$k	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$136,252.1
	$/t	*$14.97*	*$13.97*	*$12.33*	*$12.33*	*$12.33*	*$14.97*	*$17.47*	*$20.96*	*$23.29*	*$29.95*	*$29.95*	*$34.94*	*$34.94*	*$18.05*
CONCENTRATE HANDLING COSTS															
subtotal	$k	$921.4	$987.2	$1,118.9	$1,118.9	$1,118.9	$921.4	$789.8	$658.2	$592.3	$460.7	$460.7	$394.9	$394.9	$9,938.1
	$/t	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*	*$1.32*
TOTAL OPERATING COSTS															
Total	$k	$54,999.1	$55,073.0	$59,968.7	$58,971.7	$58,408.1	$51,240.2	$44,564.0	$39,466.0	$37,452.4	$32,045.2	$32,624.8	$29,505.7	$29,207.1	$583,525.9
	$/t	*$78.57*	*$73.43*	*$70.55*	*$69.38*	*$68.72*	*$73.20*	*$74.27*	*$78.93*	*$83.23*	*$91.56*	*$93.21*	*$98.63*	*$97.63*	*$77.37*
averages	$/t	*$72*						*$84*							*$77*

Figure 21.3 shows the overall proportion of operating costs to the LOM average total.

Figure 21.3 LOM average operating cost components



21.4 Metal costs

The following metal costs (TCRCs) were confirmed preparatory to the update of the new life of mine production plan:

- copper treatment: $39/dmt of concentrate (dry metric tonne)
- zinc treatment: $230/dmt of concentrate
- copper refining: $0.039/lb of metal in concentrate
- gold refining: $1.87/oz (troy) of metal in concentrate
- silver refining: $0.19/oz (troy) of metal in concentrate

Concentrate freight charges are:

- copper: $61.30/dmt of concentrate
- zinc: $110.00/dmt of concentrate

Item 22 ECONOMIC ANALYSIS

In accordance with the Rules and Policies of the NI 43-101 (Canadian Securities Administrators, 2011), the economic analysis set out below does not include Inferred Mineral Resource as a source of revenue. Furthermore, and for reasons associated with the absence of a specific grade from the Mineral Reserve statement, there is no revenue assigned to gold mineralisation.

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 and post-tax.

Certain information in this item, provided by the Company's internal taxation advisors, relates to the applicable corporate tax rate in Türkiye, the estimated taxable income and the tax to be paid.

22.1 Methodology and key assumptions

22.1.1 Overview

The basic methodology adopted for the economic analysis described herein was to initially tabulate the detailed production schedule physicals (ore and waste mined, blended ore processed, and the recovered metal profile for the production schedule). Linked to the recovered metal profile then, were the various payability formulae and the metal prices, to arrive at annual gross revenues.

Treatment charges and refining charges (metal costs for copper, zinc and silver) were next calculated, in addition to concentrate freight charges, to arrive at annual undiscounted net revenues. Annual net return after royalties was subsequently calculated, accounting explicitly for the *Eti* royalty and for government and municipal mining taxes.

Total capital costs, including those that could be construed as sustaining costs were deducted from the annual net return, followed by operating costs.

Key assumptions for the economic analysis are as follows:

- There is no long-term stockpiling and reclaim of mined ore, hence the annual mined tonnes and grades are the same as the respective plant feed tonnes and grades.
- There is no cross-blending of Spec and Non-spec ore from the mine when delivered into the plant.
- The plant feed tonnes and grade account for mining dilution and mining recovery factors.
- Under the capital cost itemisation, there is no provision for deferred mine development expenses.
- There is no provision for corporate overheads.

22.1.2 Metal pricing and payability

The annual revenues in the cash flow model were calculated referencing consensus long term metal pricing information for copper, zinc, and silver from several banks and financial service institutions. The respective LOM constant, rather than varying annual prices, are as follows:

- Copper price = \$4.25/lb (\$9,370/t)
- Zinc price = \$1.20/lb (\$2,646/t)
- Silver price = \$30.00/oz

Metal payability rates have been provided by FQM's internal metals marketing team and are as follows:

- Copper LOM average payability = 95.4%, derived from a formula referencing copper concentrate grade, zinc concentrate grade deductions and copper metal price
- Zinc LOM average payability = 83.6%, derived from a formula referencing zinc concentrate grade
- LOM average payability of silver in Spec copper concentrate = 16.3%, derived from a formula referencing silver grade in Spec concentrate, a fixed grade reduction value of 15.98 g/t Ag, and a fixed reference payability of 97.0%
- LOM average payability of silver in Non-spec copper concentrate = 73.9%, derived from a formula referencing silver grade in Non-spec concentrate, a fixed grade reduction value of 15.98 g/t Ag, and a fixed reference payability of 97.0%
- LOM average payability of silver in zinc concentrate = 0.1%, derived from a formula referencing silver grade in zinc concentrate, a fixed grade reduction value of 93.00 g/t Ag, and a fixed reference payability of 77.5%

22.1.3 Taxes

The cashflow model includes an estimate of the taxable income, as determined by the Company's internal taxation advisors.

In this model, earnings before income tax (EBIT) have been calculated as:

- EBIT = gross profit less expenses, where
  - gross profit = net revenue (i.e., gross revenue less treatment, refining and concentrate freight charges) less cost of sales, and where
    - cost of sales = the sum of mining, processing administration and other direct operating costs, plus estimated book depreciation and amortisation charges, and where
    - book depreciation and amortisation have been calculated from an initial 2025 opening balance valuation of existing property, plant and equipment to which is incrementally added the annual capital costs for replacement items.
  - expenses = the sum of royalties, mining tax, asset retirement accruals and retirement/severance accruals

Taxable income has been calculated as:

- EBIT plus book depreciation, less tax depreciation, where:
  - tax depreciation has the initial 2025 opening balance valuation adjusted for the then prevailing Turkish lira to US dollar exchange rate.

Tax payable has been calculated on the taxable income according to:

- Turkish corporate tax rate of 25%, less a 5% export incentive rate.

22.1.4 Royalties

ÇBI pays a mine tax to the Government of Türkiye calculated as a percentage of the sales value of copper and zinc concentrate production, on a sliding scale royalty rate, depending on the copper and zinc prices. A 25% mining tax escalation also applies, and the tax payable is net of operating costs and 38% of the calculated annual depreciation value. On this basis, the adopted royalty rates are 7.5% multiplied by the proportion of copper concentrate production tonnes to the total, plus 4.0% multiplied by the proportion of zinc concentrate production tonnes to the total. The sliding scale of mining tax (royalty) rates is listed in Table 22-1; the applicable incentivised rates are shown in italics.

Table 22-1 Sliding scale of government mining tax (royalty) rates

State right (%)	Incentivised (%)	Copper price $/t	Zinc price $/t
1%	0.5%	< 5,000	< 1,000
2%	1.0%	5,001-5,300	1,001-1,250
3%	1.5%	5,301-5,600	1,251-1,500
4%	2.0%	5,601-5,900	1,501-1,750
5%	2.5%	5,901-6,200	1,751-2,000
6%	3.0%	6,201-6,500	2,001-2,250
7%	3.5%	6,501-6,800	2,251-2,500
8%	***4.0%***	6,801-7,100	***2,501-2,750***
9%	4.5%	7,101-7,400	2,751-3,000
10%	5.0%	7,401-7,700	3,001-3,250
11%	5.5%	7,701-8,000	3,251-3,500
12%	6.0%	8,001-8,300	3,501-3,750
13%	6.5%	8,301-8,600	3,751-4,000
14%	7.0%	8,601-8,900	4,001-4,250
15%	***7.5%***	***8,901-9,800***	4,251-5,000
16%	8.0%	9,801-10,700	5,001-5,750
17%	8.5%	10,701-11,600	5,751-6,500
18%	9.0%	11,601-12,500	6,501-7,250
19%	9.5%	12,501-13,400	7,251-8,000
20%	10.0%	13,401-14,300	8,001-8,750
21%	10.5%	14,301-15,200	8,751-9,500
22%	11.0%	15,201-16,100	9,501-10,250
23%	11.5%	16,101-17,000	10,251-11,000
24%	12.0%	17,001-17,900	11,001-11,750
25%	12.5%	≥ 17,901	≥ 11,751

In addition, Eti receives a royalty equivalent to 7% of ÇBI's net income (excluding depreciation), and a municipal tax is also payable on the same basis, at a rate of 0.2%.

22.2 Cashflow model inputs

22.2.1 Production schedule

The plant feed production schedule forming the basis of the LOM cashflow model is shown in Table 22-2. The corresponding annual plant feed schedule is shown in Table 22-3.

22.2.2 Processing recoveries

The modelled annual and overall processing recoveries are also listed in Table 22-3.

22.2.3 Total metal sold and gross revenue

The modelled annual schedule of metals sold is listed in Table 22-4. This table also records the respective, annually varying concentrate grades and tonnages, in addition to the revenue before the deduction of treatment, refining and freight charges.

22.2.4 Metal costs and royalty payments

Table 22-5 lists the modelled annual metal costs (treatment/refining (TCRCs) and freight charges) applicable to the respective concentrate and sold metal tonnages. These inputs have changed relative to those used in defining the Mineral Reserves cut off grade (referred to in Item 21.4). The LOM average unit costs are now:

- copper treatment charge = $67.81/t concentrate
- zinc treatment charge = $215.49/t concentrate
- copper refining charge = $0.075/lb
- silver refining charge = $0.17/oz
- copper concentrate freight = $42.01/t concentrate
- zinc concentrate freight = $46.12/t concentrate

Table 22-6 lists the royalty payments attributable to each of three parties mentioned in Item 22.1.4.

Table 22-2 Life of mine production schedule

P H Y S I C A L S	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
DEVELOPMENT IN ORE (Dil'd & Rec'd)			8 months											8 months		
TOTAL ORE DEVELOPMENT	*m*	**14,013**	**737**	**1,893**	**1,393**	**1,659**	**1,546**	**1,160**	**1,802**	**1,320**	**600**	**853**	**611**	**440**		
	t	**1,387,024**	**72,939**	**187,423**	**137,867**	**164,148**	**153,044**	**114,843**	**178,359**	**130,639**	**59,358**	**84,394**	**60,490**	**43,520**		
Copper	**%**	**1.47**	**1.66**	**1.51**	**1.14**	**1.51**	**1.53**	**1.52**	**1.61**	**1.74**	**1.47**	**1.00**	**1.22**	**1.38**		
Zinc	**%**	**2.31**	**1.91**	**1.54**	**2.78**	**3.10**	**2.29**	**3.14**	**1.68**	**1.50**	**3.02**	**3.33**	**3.27**	**0.35**		
Silver	**g/t**	**9.68**	**13.80**	**6.58**	**11.94**	**10.67**	**9.08**	**12.49**	**7.17**	**5.65**	**14.33**	**14.88**	**10.45**	**4.84**		
Gold	**g/t**															
P H Y S I C A L S	**UNITS**	**TOTAL**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**	**2037**	**2038**
STOPE PRODUCTION (Dil'd & Rec'd)			8 months											8 months		
TOTAL STOPE PRODUCTION	**t**	**5,925,229**	**374,528**	**604,477**	**703,527**	**677,246**	**688,350**	**627,563**	**465,060**	**463,286**	**435,580**	**361,049**	**335,461**	**189,100**		
Copper	**%**	**1.52**	**2.86**	**1.57**	**1.29**	**1.30**	**1.21**	**1.49**	**1.48**	**1.60**	**1.58**	**1.22**	**1.65**	**1.60**		
Zinc	**%**	**2.35**	**2.57**	**0.91**	**2.28**	**2.58**	**3.29**	**2.81**	**2.00**	**2.16**	**2.22**	**2.25**	**3.16**	**1.44**		
Silver	**g/t**	**10.46**	**21.45**	**7.51**	**7.43**	**9.33**	**10.70**	**11.50**	**8.66**	**8.72**	**8.19**	**10.51**	**17.65**	**10.31**		
Gold	**g/t**															
P H Y S I C A L S	**UNITS**	**TOTAL**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**	**2037**	**2038**
TOTAL ORE AND WASTE MINED			8 months											8 months		
Waste development mined	**m**	**8,849**	1,082	1,919	1,549	867	557	467	675	592	200	410	372	159		
	t	**884,900**	108,200	191,900	154,900	86,700	55,700	46,700	67,500	59,200	20,000	41,000	37,200	15,900		
Ore development mined	**m**	**14,013**	737	1,893	1,393	1,659	1,546	1,160	1,802	1,320	600	853	611	440		
	t	**1,387,024**	72,939	187,423	137,867	164,148	153,044	114,843	178,359	130,639	59,358	84,394	60,490	43,520		
Stope ore	**t**	**5,925,229**	374,528	604,477	703,527	677,246	688,350	627,563	465,060	463,286	435,580	361,049	335,461	189,100		
Development + stope ore	**t**	**7,312,253**	447,467	791,901	841,394	841,394	841,394	742,406	643,419	593,926	494,937	445,444	395,950	232,621		
All development + stope ore	**t**	**8,211,425**	**556,413**	**985,718**	**997,703**	**929,853**	**898,659**	**790,282**	**712,746**	**654,462**	**515,545**	**487,306**	**433,770**	**248,966**		

Table 22-3 Life of mine plant feed schedule

PHYSICALS	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
TOTAL BLENDED FEED TO PLANT			No cross-blending of Spec and Non-spec from the mine													
TOTAL BLENDED ORE	**t**	**7,312,253**	**447,467**	**791,901**	**841,394**	**841,394**	**841,394**	**742,406**	**643,419**	**593,926**	**494,937**	**445,444**	**395,950**	**232,621**		
Copper	**%**	**1.51**	**2.67**	**1.55**	**1.26**	**1.34**	**1.27**	**1.50**	**1.52**	**1.63**	**1.57**	**1.18**	**1.59**	**1.56**		
Zinc	**%**	**2.34**	**2.46**	**1.06**	**2.36**	**2.68**	**3.11**	**2.86**	**1.91**	**2.01**	**2.32**	**2.45**	**3.18**	**1.23**		
Silver	**g/t**	**10.31**	**20.20**	**7.29**	**8.17**	**9.59**	**10.40**	**11.66**	**8.24**	**8.04**	**8.92**	**11.34**	**16.55**	**9.28**		
Gold	**g/t**															
Contained metal																
Copper	t	**110,302**	11,941	12,300	10,618	11,291	10,705	11,109	9,752	9,668	7,764	5,257	6,276	3,620		
Zinc	t	**171,302**	11,023	8,368	19,866	22,549	26,147	21,268	12,286	11,952	11,464	10,930	12,577	2,872		
Silver	oz	**2,424,929**	290,587	185,684	221,023	259,388	281,449	278,223	170,525	153,583	141,986	162,395	210,657	69,429		
Gold	oz															
AVERAGE RECOVERIES																
Copper	%	**87.8%**	88.1%	91.1%	89.5%	87.4%	86.8%	85.5%	86.2%	87.9%	85.9%	88.6%	87.8%	90.0%		
Zinc	%	**69.8%**	67.4%	63.3%	73.6%	73.5%	74.2%	73.3%	73.3%	61.8%	51.6%	69.7%	69.7%	67.9%		
Silver	%	**38.2%**	40.5%	34.9%	41.3%	38.8%	39.5%	34.9%	38.3%	37.5%	33.0%	28.4%	44.7%	46.0%		
Gold	%															
Metal recovered																
Copper	**t**	**96,876**	**10,518**	**11,202**	**9,507**	**9,863**	**9,289**	**9,499**	**8,407**	**8,494**	**6,667**	**4,661**	**5,510**	**3,259**		
Zinc	**t**	**119,573**	**7,432**	**5,299**	**14,626**	**16,574**	**19,401**	**15,597**	**9,005**	**7,383**	**5,913**	**7,621**	**8,771**	**1,951**		
Silver	**oz**	**925,137**	**117,616**	**64,839**	**91,344**	**100,754**	**111,224**	**97,119**	**65,349**	**57,592**	**46,894**	**46,195**	**94,247**	**31,964**		
Gold	**oz**															
Metal grade in copper concentrate																
Copper	%	**21.99%**	19.88%	22.65%	22.37%	22.53%	22.47%	22.33%	22.41%	22.58%	22.37%	22.66%	20.35%	21.22%		
Zinc	%	**3.64%**	2.76%	2.83%	3.19%	3.30%	3.50%	3.76%	3.61%	3.29%	3.68%	3.13%	5.61%	5.01%		
Silver	g/t	**27.63**	41.23	26.74	20.87	22.37	22.67	23.35	22.97	22.10	23.15	21.69	55.86	34.33		
Gold	g/t															
Metal grade in spec copper concentrate																
Copper	%	**22.88%**	22.50%	22.77%	22.86%	23.00%	23.00%	23.00%	23.00%	23.00%	23.00%	23.00%	22.53%	22.66%		
Zinc	%	**2.38%**	1.05%	2.50%	2.50%	2.29%	2.50%	2.50%	2.50%	2.50%	2.50%	2.50%	2.50%	2.73%		
Silver	g/t	**19.21**	7.48	25.98	17.54	20.00	20.00	20.00	20.00	20.00	20.00	20.00	15.31	17.21		
Gold	g/t															
Metal grade in non spec copper concentrate																
Copper	%	**18.03%**	17.01%	19.19%	18.12%	19.00%	19.00%	19.00%	19.00%	19.00%	19.00%	19.00%	17.28%	17.00%		
Zinc	%	**10.36%**	10.50%	11.99%	9.26%	10.88%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	11.72%		
Silver	g/t	**67.02**	86.12	47.60	49.97	40.00	40.00	40.00	40.00	40.00	40.00	40.00	113.12	84.74		
Gold	g/t															
Metal grade in zinc concentrate																
Zinc	%	**48.86%**	40.71%	49.92%	49.68%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	47.23%	40.00%		
Copper	%	**6.38%**	3.83%	7.99%	5.82%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	11.23%	12.73%		
Silver	g/t	**67.85**	80.86	65.37	66.39	65.00	65.00	65.00	65.00	65.00	65.00	65.00	76.44	95.73		
Gold	g/t															

Table 22-4 Life of mine payable metals and gross revenue

REVENUE	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
PAYABLE METALS																
Copper in concentrate																
Concentrate grade	%	**22.0%**	19.9%	22.6%	22.4%	22.5%	22.5%	22.3%	22.4%	22.6%	22.4%	22.7%	20.4%	21.2%		
Concentrate produced	t	**440,479**	52,916	49,464	42,494	43,782	41,347	42,537	37,521	37,619	29,804	20,566	27,070	15,358		
Payability	%	**95.4%**	95.0%	95.6%	95.5%	95.5%	95.5%	95.5%	95.5%	95.6%	95.5%	95.6%	94.9%	95.2%		
Total copper sold	t	**92,445**	9,990	10,709	9,081	9,423	8,873	9,069	8,029	8,116	6,367	4,455	5,231	3,102		
Copper in concentrate (Spec-ore)																
Concentrate grade	%	**22.9%**	22.5%	22.8%	22.9%	23.0%	23.0%	23.0%	23.0%	23.0%	23.0%	23.0%	22.5%	22.7%		
Concentrate produced	t	**315,167**	39,487	42,843	30,932	28,440	25,190	27,859	27,604	28,683	21,286	13,588	15,497	13,758		
Payability	%	**95.4%**	95.0%	95.6%	95.5%	95.5%	95.5%	95.5%	95.5%	95.6%	95.5%	95.6%	94.9%	95.2%		
Total copper sold	t	**66,166**	7,455	9,275	6,610	6,121	5,406	5,940	5,907	6,189	4,547	2,943	2,994	2,779		
Copper in concentrate (Non Spec-ore)																
Concentrate grade	%	**18.0%**	17.0%	19.2%	18.1%	19.0%	19.0%	19.0%	19.0%	19.0%	19.0%	19.0%	17.3%	17.0%		
Concentrate produced	t	**125,312**	13,429	6,621	11,562	15,342	16,157	14,679	9,917	8,936	8,518	6,978	11,573	1,600		
Payability	%	**95.4%**	95.0%	95.6%	95.5%	95.5%	95.5%	95.5%	95.5%	95.6%	95.5%	95.6%	94.9%	95.2%		
Total copper sold	t	**26,279**	2,535	1,433	2,471	3,302	3,467	3,130	2,122	1,928	1,820	1,511	2,236	323		
Zinc in concentrate																
Concentrate grade	%	**48.9%**	40.7%	49.9%	49.7%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	47.2%	40.0%		
Concentrate produced	t	**244,746**	18,257	10,615	29,440	33,148	38,802	31,193	18,011	14,766	11,826	15,242	18,568	4,878		
Payability	%	**83.6%**	80.3%	84.0%	83.9%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	83.1%	80.0%		
Total zinc sold	t	**99,993**	5,972	4,450	12,271	13,922	16,297	13,101	7,565	6,202	4,967	6,402	7,285	1,561		
Silver in copper concentrate																
Silver grade in spec concentrate	g/t	**19.21**	7.48	25.98	17.54	20.00	20.00	20.00	20.00	20.00	20.00	20.00	15.31	17.21		
Silver grade in non-spec concentrate	g/t	**67.02**	86.12	47.60	49.97	40.00	40.00	40.00	40.00	40.00	40.00	40.00	113.12	84.74		
Silver in spec concentrate	oz	**223,679**	7,105	39,857	21,498	24,822	23,037	22,773	20,543	21,645	16,149	12,106	7,801	6,343		
Silver in non-spec concentrate	oz	**167,590**	63,048	2,671	7,009	6,660	7,099	9,158	7,167	5,088	6,030	2,237	40,814	10,608		
Subtotal silver produced	oz	**925,137**	**117,616**	**64,839**	**91,344**	**100,754**	**111,224**	**97,119**	**65,349**	**57,592**	**46,894**	**46,195**	**94,247**	**31,964**		
Payability on spec concentrate	%	**16.3%**	0.0%	37.3%	8.6%	19.5%	19.5%	19.5%	19.5%	19.5%	19.5%	19.5%	0.0%	6.9%		
Payability on non-spec concentrate	%	**73.9%**	79.0%	64.4%	66.0%	58.2%	58.2%	58.2%	58.2%	58.2%	58.2%	58.2%	83.3%	78.7%		
Average payability	%	**17.3%**	42.3%	**25.6%**	**7.1%**	**8.7%**	**7.8%**	**10.1%**	**12.5%**	**12.5%**	**14.2%**	**7.9%**	**36.1%**	**27.5%**		
Silver in spec concentrate	oz	**44,675**	0	14,881	1,850	4,840	4,492	4,440	4,005	4,220	3,149	2,360	0	439		
Silver in non-spec concentrate	oz	**123,803**	49,809	1,721	4,624	3,880	4,135	5,335	4,175	2,964	3,512	1,303	33,997	8,349		
Subtotal silver sold	oz	**168,478**	**49,809**	**16,602**	**6,474**	**8,719**	**8,627**	**9,775**	**8,180**	**7,184**	**6,661**	**3,663**	**33,997**	**8,788**		
Silver in zinc concentrate																
Silver grade in concentrate	g/t	**67.85**	80.86	65.37	66.39	65.00	65.00	65.00	65.00	65.00	65.00	65.00	76.44	95.73		
Silver in zinc concentrate	oz	**533,868**	47,463	22,310	62,837	69,272	81,088	65,188	37,639	30,858	24,714	31,852	45,632	15,014		
Payability on zinc concentrate	%	**0.1%**	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	2.2%		
Subtotal silver sold	oz	**332**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**332**		
Total Metal Sold																
Copper	t	**92,445**	**9,990**	**10,709**	**9,081**	**9,423**	**8,873**	**9,069**	**8,029**	**8,116**	**6,367**	**4,455**	**5,231**	**3,102**		
Zinc	t	**99,993**	**5,972**	**4,450**	**12,271**	**13,922**	**16,297**	**13,101**	**7,565**	**6,202**	**4,967**	**6,402**	**7,285**	**1,561**		
Silver	oz	**168,811**	**49,809**	**16,602**	**6,474**	**8,719**	**8,627**	**9,775**	**8,180**	**7,184**	**6,661**	**3,663**	**33,997**	**9,120**		
Gold	oz															
GROSS REVENUE																
Metal prices																
Copper	Cu $/lb	**$4.25**	$4.25	$4.25	$4.25	$4.25	$4.25	$4.25	$4.25	$4.25	$4.25	$4.25	$4.25	$4.25		
Zinc	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		
Silver	Ag $/oz	**$30.00**	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00		
Gold	Au $/oz															
Copper revenue	$'000	**$866,172.6**	**$93,605.7**	**$100,335.8**	**$85,084.0**	**$88,291.3**	**$83,136.9**	**$84,977.0**	**$75,227.5**	**$76,048.5**	**$59,654.9**	**$41,737.4**	**$49,008.6**	**$29,064.9**		
Zinc revenue	$'000	**$264,536.9**	**$15,798.3**	**$11,771.9**	**$32,464.0**	**$36,831.2**	**$43,113.9**	**$34,659.7**	**$20,012.3**	**$16,407.0**	**$13,140.4**	**$16,935.5**	**$19,273.0**	**$4,129.6**		
Silver revenue	$'000	**$5,064.3**	**$1,494.3**	**$498.1**	**$194.2**	**$261.6**	**$258.8**	**$293.2**	**$245.4**	**$215.5**	**$199.8**	**$109.9**	**$1,019.9**	**$273.6**		
Gold revenue	$'000															
Total Gross Revenue	$'000	**$1,135,773.8**	**$110,898.3**	**$112,605.8**	**$117,742.3**	**$125,384.1**	**$126,509.5**	**$119,930.0**	**$95,485.3**	**$92,671.1**	**$72,995.1**	**$58,782.9**	**$69,301.5**	**$33,468.0**		

Table 22-5 Life of mine metal costs (treatment, refining and freight charges)

METAL COSTS	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
METAL COSTS																
TCRCs																
Treatment Charges																
Copper	**$/t con**	**$67.81**	$25.77	$21.60	$30.00	$39.00	$50.70	$65.91	$85.00	$85.00	$85.00	$85.00	$85.00	$85.00		
	$/t Cu	**$308.34**	$129.65	$95.38	$134.10	$173.13	$225.67	$295.16	$379.36	$376.45	$379.96	$375.08	$417.60	$400.52		
	$/lb Cu	**$0.14**	$0.06	$0.04	$0.06	$0.08	$0.10	$0.13	$0.17	$0.17	$0.17	$0.17	$0.19	$0.18		
	$/t ore	**$3.36**	$3.05	$1.35	$1.52	$2.03	$2.49	$3.78	$4.96	$5.38	$5.12	$3.92	$5.81	$5.61		
	$'000	**$24,589.1**	$1,363.7	$1,068.4	$1,274.8	$1,707.5	$2,096.3	$2,803.6	$3,189.3	$3,197.6	$2,533.3	$1,748.1	$2,301.0	$1,305.5		
Zinc	**$/t con**	**$215.49**	$82.62	$70.00	$91.00	$118.30	$153.79	$199.93	$279.44	$279.44	$279.44	$279.44	$279.44	$279.44		
	$/t Zn	**$441.07**	$202.96	$140.23	$183.17	$236.60	$307.58	$399.85	$558.88	$558.88	$558.88	$558.88	$591.60	$698.60		
	$/lb Zn	**$0.20**	$0.09	$0.06	$0.08	$0.11	$0.14	$0.18	$0.25	$0.25	$0.25	$0.25	$0.27	$0.32		
	$/t ore	**$6.06**	$3.37	$0.94	$3.18	$4.66	$7.09	$8.40	$7.82	$6.95	$6.68	$9.56	$13.10	$5.86		
	$'000	**$44,330.4**	$1,508.4	$743.0	$2,679.1	$3,921.4	$5,967.3	$6,236.4	$5,032.9	$4,126.2	$3,304.7	$4,259.1	$5,188.7	$1,363.1		
Treatment charges	**$'000**	**$68,919.5**	**$2,872.1**	**$1,811.5**	**$3,953.9**	**$5,628.8**	**$8,063.6**	**$9,040.0**	**$8,222.2**	**$7,323.8**	**$5,838.0**	**$6,007.2**	**$7,489.7**	**$2,668.6**	**$0.0**	**$0.0**
Refining Charges																
Copper	**$/t Cu**	**$165.35**	$52.91	$53.35	$81.35	$137.35	$165.35	$165.35	$165.35	$165.35	$165.35	$165.35	$165.35	$165.35		
	$/lb Cu	**$0.075**	$0.024	$0.024	$0.037	$0.062	$0.075	$0.075	$0.075	$0.075	$0.075	$0.075	$0.075	$0.075		
	$/t ore	**$2.09**	$1.18	$0.72	$0.88	$1.54	$1.74	$2.02	$2.06	$2.26	$2.13	$1.65	$2.18	$2.20		
	$'000	**$15,285.4**	$528.6	$571.3	$738.7	$1,294.3	$1,467.1	$1,499.6	$1,327.5	$1,342.0	$1,052.7	$736.5	$864.9	$512.9		
Zinc	**$/t Zn**	**$0.00**	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00		
	$/lb Zn	**$0.00**	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00		
	$/t ore	**$0.00**	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00		
	$'000	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Silver	**$/oz Ag**	**$0.17**	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17		
	$/t ore	**$0.01**	$0.02	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.01	$0.01		
	$'000	**$28.3**	$8.3	$2.8	$1.1	$1.5	$1.4	$1.6	$1.4	$1.2	$1.1	$0.6	$5.7	$1.5		
Gold	**$/oz Au**	**$1.68**	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68		
	$/t ore	**$0.00**	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00		
	$'000	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Refining Charges	**$'000**	**$15,313.7**	**$536.9**	**$574.1**	**$739.8**	**$1,295.7**	**$1,468.6**	**$1,501.2**	**$1,328.9**	**$1,343.2**	**$1,053.9**	**$737.2**	**$870.6**	**$514.4**	**$0.0**	**$0.0**
Freight Charges																
Copper	**$/t con**	**$42.01**	$42.10	$42.00	$42.00	$42.00	$42.00	$42.00	$42.00	$42.00	$42.00	$42.00	$42.00	$42.00		
	$/t Cu	**$191.02**	$211.79	$185.45	$187.73	$186.44	$186.95	$188.08	$187.45	$186.01	$187.75	$185.33	$206.34	$197.90		
	$/lb Cu	**$0.09**	$0.10	$0.08	$0.09	$0.08	$0.08	$0.09	$0.09	$0.08	$0.09	$0.08	$0.09	$0.09		
	$/t ore	**$2.53**	$4.98	$2.62	$2.12	$2.19	$2.06	$2.41	$2.45	$2.66	$2.53	$1.94	$2.87	$2.77		
	$'000	**$18,505.3**	$2,227.6	$2,077.5	$1,784.8	$1,838.8	$1,736.6	$1,786.6	$1,575.9	$1,580.0	$1,251.8	$863.8	$1,137.0	$645.1		
Zinc	**$/t con**	**$46.12**	$40.76	$46.50	$46.50	$46.50	$46.50	$46.50	$46.50	$46.50	$46.50	$46.50	$46.50	$46.50		
	$/t Zn	**$94.40**	$100.13	$93.15	$93.60	$93.00	$93.00	$93.00	$93.00	$93.00	$93.00	$93.00	$98.45	$116.25		
	$/lb Zn	**$0.04**	$0.05	$0.04	$0.04	$0.04	$0.04	$0.04	$0.04	$0.04	$0.04	$0.04	$0.04	$0.05		
	$/t ore	**$1.54**	$1.66	$0.62	$1.63	$1.83	$2.14	$1.95	$1.30	$1.16	$1.11	$1.59	$2.18	$0.98		
	$'000	**$11,275.9**	$744.2	$493.6	$1,369.0	$1,541.4	$1,804.3	$1,450.5	$837.5	$686.6	$549.9	$708.7	$863.4	$226.8		
Freight Charges	**$'000**	**$29,781.1**	**$2,971.8**	**$2,571.1**	**$3,153.7**	**$3,380.2**	**$3,540.9**	**$3,237.1**	**$2,413.4**	**$2,266.6**	**$1,801.7**	**$1,572.5**	**$2,000.4**	**$871.9**	**$0.0**	**$0.0**
TOTAL TCRCs AND FREIGHT																
Copper	$/t Cu	**$664.71**	$394.36	$334.18	$403.18	$496.92	$577.97	$648.59	$732.16	$727.81	$733.05	$725.76	$789.29	$763.77		
	$/lb Cu	**$0.30**	$0.18	$0.15	$0.18	$0.23	$0.26	$0.29	$0.33	$0.33	$0.33	$0.33	$0.36	$0.35		
	$/t ore	**$7.98**	$9.21	$4.69	$4.51	$5.75	$6.30	$8.20	$9.47	$10.30	$9.77	$7.52	$10.87	$10.59		
Zinc	$/t Zn	**$535.48**	$303.08	$233.38	$276.76	$329.60	$400.58	$492.85	$651.88	$651.88	$651.88	$651.88	$690.05	$814.85		
	$/lb Zn	**$0.24**	$0.14	$0.11	$0.13	$0.15	$0.18	$0.22	$0.30	$0.30	$0.30	$0.30	$0.31	$0.37		
	$/t ore	**$7.60**	$5.03	$1.56	$4.81	$6.49	$9.24	$10.35	$9.12	$8.10	$7.79	$11.15	$15.29	$6.83		
Silver	$/oz Ag	**$0.17**	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17	$0.17		
	$/t ore	**$0.01**	$0.02	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.01	$0.01		
Gold	$/oz Au	**$1.68**	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68	$1.68		
	$/t ore	**$0.00**	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00		
Total TCRCs AND FREIGHT	**$'000**	**$114,014.4**	**$6,380.8**	**$4,956.7**	**$7,847.4**	**$10,304.7**	**$13,073.1**	**$13,778.3**	**$11,964.5**	**$10,933.7**	**$8,693.6**	**$8,316.9**	**$10,360.6**	**$4,054.9**	**$0.0**	**$0.0**

Table 22-6 Life of mine royalty payments

ROYALTIES	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
ETI ROYALTY																
Royalty rate	%	**7.0%**	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%		
Minimum	$	**$666.7**	$666.7	$666.7	$666.7	$666.7	$666.7	$666.7	$666.7	$666.7	$666.7	$666.7	$666.7	$666.7		
Regular tax rate	%	**20.0%**	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	120.0%		
Total plant feed tonnes	t	**7,312,253**	447,467	791,901	841,394	841,394	841,394	742,406	643,419	593,926	494,937	445,444	395,950	232,621		
Net revenue	**$'000**	**$1,025,108.6**	**$104,517.5**	**$107,649.1**	**$109,894.9**	**$115,079.3**	**$113,436.5**	**$106,151.7**	**$83,520.7**	**$81,737.4**	**$64,301.5**	**$50,466.0**	**$58,940.9**	**$29,413.1**		
Operating costs																
Mining	$'000	**$238,626.5**	$14,642.3	$27,970.2	$28,700.0	$28,426.5	$26,951.2	$23,018.0	$22,119.9	$19,021.5	$14,348.2	$13,688.0	$12,561.7	$7,179.0		
Milling	$'000	**$129,902.5**	$8,372.8	$13,290.3	$13,994.5	$14,189.1	$14,189.1	$12,791.0	$11,413.8	$10,589.2	$9,463.1	$8,638.5	$7,893.8	$5,077.3		
Plant	$'000	**$58,991.1**	$4,854.4	$7,122.3	$7,187.6	$7,147.8	$7,108.0	$5,948.2	$4,871.2	$3,794.2	$3,711.4	$2,634.4	$2,965.8	$1,645.8		
Site Administration	$'000	**$118,783.9**	$6,987.3	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$6,987.3		
Other Direct Costs (Con. Handling at Rize Port)	$'000	**$9,625.1**	$589.0	$1,042.4	$1,107.5	$1,107.5	$1,107.5	$977.2	$846.9	$781.8	$651.5	$586.3	$521.2	$306.2		
Total operating costs	**$'000**	**$555,929.1**	**$35,445.7**	**$59,906.1**	**$61,470.6**	**$61,351.8**	**$59,836.8**	**$53,215.4**	**$49,732.8**	**$44,667.6**	**$38,655.1**	**$36,028.2**	**$34,423.4**	**$21,195.6**		
Revenue less operating costs	**$'000**	**$469,179.5**	**$69,071.7**	**$47,743.0**	**$48,424.3**	**$53,727.5**	**$53,599.7**	**$52,936.3**	**$33,788.0**	**$37,069.8**	**$25,646.4**	**$14,437.8**	**$24,517.5**	**$8,217.5**		
Royalty to be paid	**$'000**	**$26,480.5**	**$3,868.0**	**$2,673.6**	**$2,711.8**	**$3,008.7**	**$3,001.6**	**$2,964.4**	**$1,892.1**	**$2,075.9**	**$1,436.2**	**$808.5**	**$1,373.0**	**$666.7**	**$0.0**	**$0.0**
GOVERNMENT MINING TAX																
Copper Production (payable)	t Cu	**92,445**	9,990	10,709	9,081	9,423	8,873	9,069	8,029	8,116	6,367	4,455	5,231	3,102		
Zinc Production (payable)	t Zn	**99,993**	5,972	4,450	12,271	13,922	16,297	13,101	7,565	6,202	4,967	6,402	7,285	1,561		
Copper concentrate production (payable)	t Cu	**420,289**	50,262	47,284	40,590	41,831	39,495	40,615	35,834	35,947	28,461	19,657	25,697	14,617		
Zinc concentrate production (payable)	t Zn	**204,518**	14,669	8,914	24,700	27,844	32,594	26,202	15,129	12,404	9,934	12,803	15,423	3,902		
Copper proportion	% prop.	**48%**	77%	84%	62%	60%	55%	61%	70%	74%	74%	61%	62%	79%		
Zinc proportion	% prop.	**52%**	23%	16%	38%	40%	45%	39%	30%	26%	26%	39%	38%	21%		
Metal prices																
Copper	$/t	**$9,370**	$9,370	$9,370	$9,370	$9,370	$9,370	$9,370	$9,370	$9,370	$9,370	$9,370	$9,370	$9,370		
Zinc	$/t	**$2,646**	$2,646	$2,646	$2,646	$2,646	$2,646	$2,646	$2,646	$2,646	$2,646	$2,646	$2,646	$2,646		
Copper Tax (post incentive)	%	**7.50%**	**7.50%**	**7.50%**	**7.50%**	**7.50%**	**7.50%**	**7.50%**	**7.50%**	**7.50%**	**7.50%**	**7.50%**	**7.50%**	**7.50%**		
Zinc Tax (post incentive)	%	**4.00%**	**4.0%**	**4.0%**	**4.0%**	**4.0%**	**4.0%**	**4.0%**	**4.0%**	**4.0%**	**4.0%**	**4.0%**	**4.0%**	**4.0%**		
Mining Tax Escalation	%	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**		
Calculated Mining Tax Rate	**%**	**7.10%**	**8.4%**	**8.7%**	**7.7%**	**7.6%**	**7.4%**	**7.7%**	**8.1%**	**8.3%**	**8.2%**	**7.6%**	**7.7%**	**8.5%**		
Operating costs																
Mining	$'000	**$238,626.5**	$14,642.3	$27,970.2	$28,700.0	$28,426.5	$26,951.2	$23,018.0	$22,119.9	$19,021.5	$14,348.2	$13,688.0	$12,561.7	$7,179.0		
Milling	$'000	**$129,902.5**	$8,372.8	$13,290.3	$13,994.5	$14,189.1	$14,189.1	$12,791.0	$11,413.8	$10,589.2	$9,463.1	$8,638.5	$7,893.8	$5,077.3		
Plant	$'000	**$58,991.1**	$4,854.4	$7,122.3	$7,187.6	$7,147.8	$7,108.0	$5,948.2	$4,871.2	$3,794.2	$3,711.4	$2,634.4	$2,965.8	$1,645.8		
Site Administration	$'000	**$118,783.9**	$6,987.3	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$6,987.3		
Other Direct Costs (Con. handling at Rize Port)	$'000	**$9,625.1**	$589.0	$1,042.4	$1,107.5	$1,107.5	$1,107.5	$977.2	$846.9	$781.8	$651.5	$586.3	$521.2	$306.2		
Total operating costs	**$'000**	**$555,929.1**	**$35,445.7**	**$59,906.1**	**$61,470.6**	**$61,351.8**	**$59,836.8**	**$53,215.4**	**$49,732.8**	**$44,667.6**	**$38,655.1**	**$36,028.2**	**$34,423.4**	**$21,195.6**	**$0.0**	**$0.0**
Book depreciation - per FQM taxation advisors	$'000	**$100,160.3**	$4,662.1	$8,770.7	$8,936.4	$9,759.7	$9,449.7	$10,551.1	$9,786.9	$10,403.0	$8,584.6	$6,412.0	$8,058.8	$4,785.2		
Tax deductions																
Trucking cost to Mine per tonne ore	$/t	**$3.20**	$3.37	$3.02	$2.99	$3.04	$3.04	$3.10	$3.19	$3.21	$3.44	$3.49	$3.59	$3.93		
Mill cost per tonne ore	$/t	**$17.77**	$18.71	$16.78	$16.63	$16.86	$16.86	$17.23	$17.74	$17.83	$19.12	$19.39	$19.94	$21.83		
Plant contribution per tonne ore (20 %)	$/t	**$1.61**	$2.17	$1.80	$1.71	$1.70	$1.69	$1.60	$1.51	$1.28	$1.50	$1.18	$1.50	$1.42		
Conc. handling per tonne ore	$/t	**$1.32**	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32		
Admin contribution per tonne ore (20 %)	$/t	**$3.25**	$3.12	$2.65	$2.49	$2.49	$2.49	$2.82	$3.26	$3.53	$4.24	$4.71	$5.29	$6.01		
Depreciation per tonne (38 %)	$/t	**$5.21**	$3.96	$4.21	$4.04	$4.41	$4.27	$5.40	$5.78	$6.66	$6.59	$5.47	$7.73	$7.82		
Total sales cost (net of operating costs)	**$/t**	**$108.55**	**$202.10**	**$106.85**	**$102.09**	**$107.65**	**$105.83**	**$112.23**	**$97.66**	**$104.50**	**$94.37**	**$78.30**	**$110.24**	**$84.77**		
Revenue less operating costs	**$'000**	**$793,733.9**	**$90,433.9**	**$84,611.6**	**$85,896.5**	**$90,572.5**	**$89,047.2**	**$83,319.3**	**$62,835.7**	**$62,062.9**	**$46,706.0**	**$34,880.2**	**$43,649.6**	**$19,718.6**		
Royalty to be paid	**$'000**	**$63,194.1**	**$7,584.3**	**$7,345.2**	**$6,631.1**	**$6,907.6**	**$6,586.8**	**$6,381.7**	**$5,074.7**	**$5,121.8**	**$3,850.0**	**$2,668.1**	**$3,375.9**	**$1,666.8**	**$0.0**	**$0.0**
MUNICIPAL MINING TAX																
Tax rate	%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%		
Revenue less operating costs	**$'000**	$793,733.9	$90,433.9	$84,611.6	$85,896.5	$90,572.5	$89,047.2	$83,319.3	$62,835.7	$62,062.9	$46,706.0	$34,880.2	$43,649.6	$19,718.6		
Royalty to be paid	**$'000**	**$1,587.5**	**$180.9**	**$169.2**	**$171.8**	**$181.1**	**$178.1**	**$166.6**	**$125.7**	**$124.1**	**$93.4**	**$69.8**	**$87.3**	**$39.4**	**$0.0**	**$0.0**
Total Royalties	**$'000**	**$91,262.1**	**$11,633.2**	**$10,188.0**	**$9,514.7**	**$10,097.5**	**$9,766.4**	**$9,512.8**	**$7,092.5**	**$7,321.9**	**$5,379.6**	**$3,546.4**	**$4,836.2**	**$2,372.9**	**$0.0**	**$0.0**

22.2.5 Capital and closure costs

Table 22-7 lists the modelled capital costs for the LOM cashflow model, summarised from the information presented in Item 21.1.

The scheduled (ARO) closure cost expenditures listed in the table amount to the $8.8M itemised in Table 21-7.

22.2.6 Operating costs

Table 22-8 lists the updated operating costs for the LOM cashflow model. The total $ shown in this update reflect the unit costs from the 2024 budget estimate review (Item 21.3) multiplied by the new schedule physicals as set out in Table 22-2. Site administration and processing fixed costs for 2025 and 2036 have been adjusted for production part-years.

22.3 Cashflow model summary

Table 22-9 provides a cashflow summary to support the Mineral Reserves production schedule. The undiscounted cashflow, pre-tax, for the expanded Operations is $307.2M, with an NPV reflecting a 10% discount rate equal to $193.1M. At a discount rate of 8% the NPV is $210.1M.

The undiscounted cashflow, post-tax for the expanded Operations is $251.1M, with an NPV reflecting a 10% discount rate equal to $155.7M. At a discount rate of 8% the NPV is $169.8M.

In each year of the respective cashflow models there is a positive cashflow (excluding years post-closure) and hence an internal rate of return is not relevant.

22.4 Cashflow model sensitivity analysis

Table 22-11 presents a cashflow model sensitivity analysis. Several model variants were analysed, namely:

- metal recovery on copper and zinc, varying separately and together
  - varying from 95% to 105% of the base case annual processing recovery rate
  - [would have a similar impact as varying the copper and zinc plant feed grades by the same factors]
- copper and zinc metal prices, varying separately and together
  - varying from 90% to 110% of the base case prices
- TCRCs, total operating costs and total capital costs
  - varying separately between 90% and 110% from the base case costs

Table 22-11 indicates that:

- the least sensitive variables are the capital costs and the TCRCs
- the more sensitive variables are metal price and processing recovery, since both are analogous to feed grade and revenue variability
- operating costs are a sensitive variable, likely attributable to the relatively high fixed cost components within the mining (services), processing labour and mine administration cost centres

Table 22-7 Life of mine capital expenditure

CAPITAL COSTS	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
DEFERRED EXPANSION CAPITAL COSTS																
Deferred development expenses capitalised	$'000	$0.0														
Subotal Expansion Capital	$'000	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
MINE DEVELOPMENT COSTS	$/t	$28.38														
Mine development	m	2,415	915	990	510	0	0	0	0	0	0	0	0	0		
	t	241,500	91,500	99,000	51,000	0	0	0	0	0	0	0	0	0		
Subtotal mine development	$'000	$6,853.9	$2,596.8	$2,809.7	$1,447.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
CAPITAL COSTS																
Mine	$'000	$21,903.0	$5,908.0	$5,940.0	$1,830.0	$1,675.0	$695.0	$3,540.0	$625.0	$535.0	$495.0	$400.0	$260.0	$0.0		
Mill	$'000	$6,680.0	$1,501.0	$1,546.0	$1,626.0	$796.0	$216.0	$195.0	$300.0	$200.0	$150.0	$150.0	$0.0	$0.0		
Plant	$'000	$23,096.0	$6,706.0	$6,640.0	$5,540.0	$530.0	$520.0	$510.0	$610.0	$510.0	$510.0	$510.0	$510.0	$0.0		
Administration	$'000	$3,414.3	$586.3	$1,023.0	$194.0	$151.0	$127.0	$124.5	$424.5	$460.5	$124.5	$99.5	$99.5	$0.0		
Subtotal capital	$'000	$55,093.3	$14,701.3	$15,149.0	$9,190.0	$3,152.0	$1,558.0	$4,369.5	$1,959.5	$1,705.5	$1,279.5	$1,159.5	$869.5	$0.0	$0.0	$0.0
OTHER CAPITALISED COSTS																
ARO costs	$'000	$8,756.7	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$1,741.3	$5,223.9	$1,095.4	$696.2
Subtotal other capital	$'000	$8,756.7	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$1,741.3	$5,223.9	$1,095.4	$696.2
Total capital costs	$'000	$70,703.9	$17,298.1	$17,958.7	$10,637.4	$3,152.0	$1,558.0	$4,369.5	$1,959.5	$1,705.5	$1,279.5	$1,159.5	$2,610.8	$5,223.9	$1,095.4	$696.2

Table 22-8 Life of mine operating cost summary

OPERATING COSTS	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Mining																
Development in waste	$/m	$2,838														
	$'000	$18,260.0	$474.0	$2,636.6	$2,948.7	$2,460.6	$1,580.8	$1,325.4	$1,915.7	$1,680.1	$567.6	$1,163.6	$1,055.8	$451.3		
Development in ore	$/m	$2,242														
	$'000	$31,422.2	$1,652.4	$4,246.0	$3,123.3	$3,718.7	$3,467.1	$2,601.7	$4,040.6	$2,959.6	$1,344.7	$1,911.9	$1,370.4	$985.9		
Stoping	$/t	$5.56														
	$'000	$32,923.1	$2,081.0	$3,358.7	$3,909.1	$3,763.1	$3,824.8	$3,487.0	$2,584.1	$2,574.2	$2,420.3	$2,006.1	$1,864.0	$1,050.7		
Common Services	$/t o+w	$11.99														
	$'000	$98,487.0	$5,746.2	$10,953.4	$11,702.3	$11,489.3	$11,105.6	$9,768.7	$8,800.8	$8,085.4	$6,374.7	$6,021.9	$5,362.2	$3,076.6		
Maintenance + add'n labour	$/t o+w	$7.01	8 months											8 months		
	$'000	$57,534.2	$4,688.8	$6,775.6	$7,016.6	$6,994.8	$6,973.0	$5,835.2	$4,778.7	$3,722.2	$3,640.9	$2,584.4	$2,909.5	$1,614.6		
Subtotal mine operating costs	$/t ore	$32.63	$32.72	$35.32	$34.11	$33.78	$32.03	$31.00	$34.38	$32.03	$28.99	$30.73	$31.73	$30.86	$0.00	$0.00
	$'000	$238,626.5	$14,642.3	$27,970.2	$28,700.0	$28,426.5	$26,951.2	$23,018.0	$22,119.9	$19,021.5	$14,348.2	$13,688.0	$12,561.7	$7,179.0	$0.0	$0.0
Processing			8 months											8 months		
	$/t	$17.77	$18.71	$16.78	$16.63	$16.86	$16.86	$17.23	$17.74	$17.83	$19.12	$19.39	$19.94	$21.83		
	$'000	$129,902.5	$8,372.8	$13,290.3	$13,994.5	$14,189.1	$14,189.1	$12,791.0	$11,413.8	$10,589.2	$9,463.1	$8,638.5	$7,893.8	$5,077.3		
Plant																
	$/t	$8.07	$10.85	$8.99	$8.54	$8.50	$8.45	$8.01	$7.57	$6.39	$7.50	$5.91	$7.49	$8.02		
	$'000	$58,991.1	$4,854.4	$7,122.3	$7,187.6	$7,147.8	$7,108.0	$5,948.2	$4,871.2	$3,794.2	$3,711.4	$2,634.4	$2,965.8	$1,645.8		
G&A																
	$/t	$16.24	$15.62	$13.24	$12.46	$12.46	$12.46	$14.12	$16.29	$17.65	$21.18	$23.53	$26.47	$30.04		
	$'000	$118,783.9	$6,987.3	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$6,987.3		
Conc. Handling																
	$/t ore	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32	$1.32		
	$'000	$9,625.1	$589.0	$1,042.4	$1,107.5	$1,107.5	$1,107.5	$977.2	$846.9	$781.8	$651.5	$586.3	$521.2	$306.2		
Total operating costs	$'000	$555,929.1	$35,445.7	$59,906.1	$61,470.6	$61,351.8	$59,836.8	$53,215.4	$49,732.8	$44,667.6	$38,655.1	$36,028.2	$34,423.4	$21,195.6	$0.0	$0.0

Table 22-9 Life of mine cashflow summary, pre-tax

CASHFLOW SUMMARY PRE-TAX	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
MINERAL RESERVES CASHFLOW			8 months											8 months		
Gross revenue	**$'000**	**$1,135,773.8**	$110,898.3	$112,605.8	$117,742.3	$125,384.1	$126,509.5	$119,930.0	$95,485.3	$92,671.1	$72,995.1	$58,782.9	$69,301.5	$33,468.0		
Treatment, refining and freight charges (metal costs)	**$'000**	**$110,665.2**	$6,380.8	$4,956.7	$7,847.4	$10,304.7	$13,073.1	$13,778.3	$11,964.5	$10,933.7	$8,693.6	$8,316.9	$10,360.6	$4,054.9		
Net revenue	**$'000**	**$1,025,108.6**	**$104,517.5**	**$107,649.1**	**$109,894.9**	**$115,079.3**	**$113,436.5**	**$106,151.7**	**$83,520.7**	**$81,737.4**	**$64,301.5**	**$50,466.0**	**$58,940.9**	**$29,413.1**	**$0.0**	**$0.0**
Total operating costs																
Mining	**$'000**	**$238,626.5**	$14,642.3	$27,970.2	$28,700.0	$28,426.5	$26,951.2	$23,018.0	$22,119.9	$19,021.5	$14,348.2	$13,688.0	$12,561.7	$7,179.0		
Processing	**$'000**	**$129,902.5**	$8,372.8	$13,290.3	$13,994.5	$14,189.1	$14,189.1	$12,791.0	$11,413.8	$10,589.2	$9,463.1	$8,638.5	$7,893.8	$5,077.3		
Plant	**$'000**	**$58,991.1**	$4,854.4	$7,122.3	$7,187.6	$7,147.8	$7,108.0	$5,948.2	$4,871.2	$3,794.2	$3,711.4	$2,634.4	$2,965.8	$1,645.8		
Site administration	**$'000**	**$118,783.9**	$6,987.3	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$6,987.3		
Other direct costs	**$'000**	**$9,625.1**	$589.0	$1,042.4	$1,107.5	$1,107.5	$1,107.5	$977.2	$846.9	$781.8	$651.5	$586.3	$521.2	$306.2		
Other costs																
Royalty & mine taxes	**$'000**	**$91,262.1**	$11,633.2	$10,188.0	$9,514.7	$10,097.5	$9,766.4	$9,512.8	$7,092.5	$7,321.9	$5,379.6	$3,546.4	$4,836.2	$2,372.9		
Corporate costs	**$'000**	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Cash EBITDA	**$'000**	**$377,917.3**	**$57,438.5**	**$37,554.9**	**$38,909.6**	**$43,630.0**	**$43,833.2**	**$43,423.5**	**$26,695.4**	**$29,747.9**	**$20,266.8**	**$10,891.4**	**$19,681.3**	**$5,844.6**	**$0.0**	**$0.0**
Total capital costs (expansion)	**$'000**	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Total capital costs (mine development)	**$'000**	**$6,853.9**	$2,596.8	$2,809.7	$1,447.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Total direct capital costs	**$'000**	**$55,093.3**	$14,701.3	$15,149.0	$9,190.0	$3,152.0	$1,558.0	$4,369.5	$1,959.5	$1,705.5	$1,279.5	$1,159.5	$869.5	$0.0		
ARO costs	**$'000**	**$8,756.7**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$1,741.3	$5,223.9	$1,095.4	$696.2
Undiscounted cashflow pre-tax	**$'000**	**$307,213.4**	**$40,140.4**	**$19,596.3**	**$28,272.2**	**$40,478.0**	**$42,275.2**	**$39,054.0**	**$24,735.9**	**$28,042.4**	**$18,987.3**	**$9,731.9**	**$17,070.5**	**$620.7**	**-$1,095.4**	**-$696.2**

Table 22-10 Life of mine cashflow summary, post-tax

CASHFLOW SUMMARY POST-TAX	UNITS	TOTAL	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
TECHNICAL REPORT CASHFLOW			8 months											8 months		
Gross revenue	**$'000**	**$1,135,773.8**	$110,898.3	$112,605.8	$117,742.3	$125,384.1	$126,509.5	$119,930.0	$95,485.3	$92,671.1	$72,995.1	$58,782.9	$69,301.5	$33,468.0		
Net revenue	**$'000**	**$1,025,108.6**	**$104,517.5**	**$107,649.1**	**$109,894.9**	**$115,079.3**	**$113,436.5**	**$106,151.7**	**$83,520.7**	**$81,737.4**	**$64,301.5**	**$50,466.0**	**$58,940.9**	**$29,413.1**	**$0.0**	**$0.0**
Cost of sales																
Mining	**$'000**	**$238,626.5**	$14,642.3	$27,970.2	$28,700.0	$28,426.5	$26,951.2	$23,018.0	$22,119.9	$19,021.5	$14,348.2	$13,688.0	$12,561.7	$7,179.0		
Processing	**$'000**	**$129,902.5**	$8,372.8	$13,290.3	$13,994.5	$14,189.1	$14,189.1	$12,791.0	$11,413.8	$10,589.2	$9,463.1	$8,638.5	$7,893.8	$5,077.3		
Other Direct	**$'000**	**$68,616.3**	$5,443.4	$8,164.7	$8,295.1	$8,255.3	$8,215.5	$6,925.4	$5,718.1	$4,576.0	$4,362.9	$3,220.8	$3,487.0	$1,952.0		
Site administration	**$'000**	**$118,783.9**	$6,987.3	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$10,480.9	$6,987.3		
Depreciation	**$'000**	**$74,360.9**	$4,425.2	$5,804.3	$6,085.2	$6,788.6	$6,652.0	$7,692.3	$7,255.5	$7,842.2	$6,577.0	$5,005.5	$6,423.5	$3,809.5		
Amortisation	**$'000**	**$20,109.2**	$1,792.6	$2,021.2	$2,050.9	$2,142.0	$2,017.0	$2,061.6	$1,825.0	$1,845.0	$1,447.3	$1,012.6	$1,189.0	$705.1		
Total costs	**$'000**	**$650,399.2**	**$41,663.6**	**$67,731.6**	**$69,606.7**	**$70,282.4**	**$68,505.8**	**$62,969.3**	**$58,813.3**	**$54,354.8**	**$46,679.4**	**$42,046.2**	**$42,035.9**	**$25,710.2**	**$0.0**	**$0.0**
Gross profit	**$'000**	**$374,709.4**	**$62,853.9**	**$39,917.5**	**$40,288.2**	**$44,796.9**	**$44,930.7**	**$43,182.4**	**$24,707.5**	**$27,382.6**	**$17,622.2**	**$8,419.7**	**$16,905.0**	**$3,702.9**		
Expenses																
Corporate costs	**$'000**	**$0.0**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0		
Royalty & mine taxes	**$'000**	**$91,262.1**	$11,633.2	$10,188.0	$9,514.7	$10,097.5	$9,766.4	$9,512.8	$7,092.5	$7,321.9	$5,379.6	$3,546.4	$4,836.2	$2,372.9		
ARO costs	**$'000**	**$8,756.7**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$1,741.3	$5,223.9	$1,095.4	$696.2
Earnings before income tax	**$'000**	**$274,690.5**	**$51,220.7**	**$29,729.5**	**$30,773.5**	**$34,699.4**	**$35,164.3**	**$33,669.6**	**$17,614.9**	**$20,060.8**	**$12,242.5**	**$4,873.3**	**$10,327.5**	**-$3,893.9**	**-$1,095.4**	**-$696.2**
Tax payable																
Book Depreciation	**$'000**	**$100,160.3**	$4,662.1	$8,770.7	$8,936.4	$9,759.7	$9,449.7	$10,551.1	$9,786.9	$10,403.0	$8,584.6	$6,412.0	$8,058.8	$4,785.2		
Tax Depreciation	**$'000**	**-$100,086.6**	-$6,951.4	-$8,463.7	-$8,676.1	-$9,489.7	-$9,195.4	-$10,291.2	-$9,556.9	-$10,170.3	-$8,402.2	-$6,284.3	-$7,909.2	-$4,696.4		
Taxable income	**$'000**	**$276,555.8**	**$48,931.5**	**$30,036.5**	**$31,033.8**	**$34,969.4**	**$35,418.6**	**$33,929.5**	**$17,845.0**	**$20,293.4**	**$12,425.0**	**$5,001.0**	**$10,477.1**	**-$3,805.1**	**$0.0**	**$0.0**
Regular corporate tax rate (less incentive)	**$'000**	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%		
Tax payable	**$'000**	**$56,072.2**	**$9,786.3**	**$6,007.3**	**$6,206.8**	**$6,993.9**	**$7,083.7**	**$6,785.9**	**$3,569.0**	**$4,058.7**	**$2,485.0**	**$1,000.2**	**$2,095.4**	**$0.0**	**$0.0**	**$0.0**
Undiscounted cashflow pre-tax	**$'000**	**$307,213.4**	$40,140.4	$19,596.3	$28,272.2	$40,478.0	$42,275.2	$39,054.0	$24,735.9	$28,042.4	$18,987.3	$9,731.9	$17,070.5	$620.7	-$1,095.4	-$696.2
Undiscounted cashflow post-tax	**$'000**	**$251,141.2**	**$30,354.1**	**$13,589.0**	**$22,065.5**	**$33,484.1**	**$35,191.5**	**$32,268.1**	**$21,166.9**	**$23,983.7**	**$16,502.3**	**$8,731.7**	**$14,975.1**	**$620.7**	**-$1,095.4**	**-$696.2**

Table 22-11 Pre-tax cashflow model; sensitivity analysis

SENSITIVITY ANALYSES	PRE-TAX CASHFLOW			DELTA
	UCF ($M)	NPV_{10} ($M)	NPV_8 ($M)	UCF (%)
Copper price * 0.90; zinc price * 0.90	$209.3	$133.4	$144.8	68%
Copper price * 0.90; zinc price * 1.00	$232.3	$147.1	$159.8	76%
Operating costs * 1.10	$251.6	$159.9	$173.6	82%
Copper recovery * 0.95; zinc recovery * 0.95	$263.1	$165.9	$180.4	86%
Copper recovery * 0.95; zinc recovery * 1.00	$272.6	$171.7	$186.7	89%
Operating costs * 1.05	$279.4	$176.5	$191.8	91%
Copper price * 1.00; zinc price * 0.90	$284.4	$179.5	$195.1	93%
TCRCs * 1.10	$297.6	$187.8	$204.1	97%
Copper recovery * 1.00; zinc recovery * 0.95	$297.8	$187.4	$203.8	97%
Capital costs * 1.10	$300.3	$188.2	$204.8	98%
TCRCs * 1.05	$302.4	$190.5	$207.1	98%
Capital costs * 1.05	$303.8	$190.7	$207.4	99%
Base Case	**$307.2**	**$193.1**	**$210.1**	**100%**
Capital costs * 0.95	$310.7	$195.6	$212.7	101%
TCRCs * 0.95	$312.0	$195.8	$213.0	102%
Capital costs * 0.90	$314.1	$198.1	$215.3	102%
Copper recovery * 1.00; zinc recovery * 1.05	$316.6	$198.9	$216.3	103%
TCRCs * 0.90	$316.8	$198.5	$216.0	103%
Copper price * 1.00; zinc price * 1.10	$330.1	$206.8	$225.1	107%
Operating costs * 0.95	$335.0	$209.8	$228.3	109%
Copper recovery * 1.05; zinc recovery * 1.00	$341.9	$214.6	$233.4	111%
Copper recovery * 1.05; zinc recovery * 1.05	$351.2	$220.3	$239.7	114%
Operating costs * 0.90	$362.8	$226.4	$246.5	118%
Copper price * 1.10; zinc price * 1.00	$382.0	$239.2	$260.3	124%
Copper price * 1.10; zinc price * 1.10	$404.9	$252.8	$275.3	132%

Item 23 ADJACENT PROPERTIES

There are no adjacent properties that are relevant to the Mineral Resource estimate described in this Technical Report.

Item 24 OTHER RELEVANT DATA AND INFORMATION

The QPs are not aware of any other relevant data or information not already presented in this Technical Report.

Item 25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource modelling and estimation

The discovery of the South Orebody is a significant material outcome enabling the life of the Çayeli Operations to be extended for in excess of ten years.

The South Orebody has a strike length of about 250 m and represents a relatively small exploration drilling target. The potential to identify additional mineralisation within the existing tenement boundary north and south of the existing workings remains.

In terms of the existing along-strike drilling sited away from the mine workings, the sampling and assaying were extremely selective. It is limited to the relatively better mineralised zones and as such it appears likely that historic workers were only targeting massive sulphide and not stockwork mineralisation. Interestingly, historic drillholes C1003 and C1506 intersected the lower parts of the South Orebody but C1003 was not sampled. In contrast, C1505 was only partially sampled with 21 m of recovered core assaying at 1.62% Cu (at 432 m to 454 m depth below collar).

The challenge remaining is in determining what is the most cost-effective way to drill-out these potential areas. The steep terrain and numerous houses, farms and tea plantations make surface drilling difficult. Underground drilling would probably be more effective but in many cases, requires costly underground development to provide suitable drilling sites. It is likely that a combination of both will be needed.

In the QP's opinion, outside of the Main Orebody remnants, deposit risk in terms of geological and grade continuity is relatively low given the bulk of the mineralisation in the South Orebody is dominated (75%) by broad scale stockwork mineralisation.

25.2 Mine planning and Mineral Reserve estimation

The detailed planning work for the Çayeli LOM production plan has been completed by the ÇBI mining team, and in particular by individuals with extensive site-specific planning and operational experience. The approach to determining the appropriate mining cut-off grade and NSR criteria has been based on the accumulated experience from many years of producing budget and forecast production plans for the mine. QP overview has provided the means by which offset mine operating costs have been thoroughly scrutinised, updated and projected to suit the new production plan.

This new production plan features over 1,100 individual long-hole stope designs, spread across both orebodies. These have all been designed to account for and minimise, the incorporation of planned mining dilution. The QP considers that the quantity and grade of the diluent material is reasonable for the style of deposit and mining method, particularly for the planned remnant stopes of the Main Orebody.

Considering the number of ore development and stope openings in the LOM production plan, the extraction sequencing work is time consuming and laborious. Nevertheless, the ÇBI team draws upon extensive experience in devising a sequence that has been cognisant of:

- geotechnical conditions
- established cemented backfilling and barricading practices
- adherence to mine planning "rules" based on site-specific operational experience

The above considerations and modifying factors have been taken into account when estimating and stating the Mineral Reserve.

The following information relates to risks and uncertainties around the Mineral Reserve estimate and related technical matters.

25.2.1 Mining and primary equipment

There is considered to be minimal risk attributable to the mining method and to the primary equipment in use at Çayeli and as proposed for future mining. The method and equipment are conventional and suitable for the scale of the extended mining operations. Furthermore, the long-hole stoping production method with requisite barricading and paste backfilling, is a well-established practice at Çayeli, managed by experienced personnel, and with the benefit of continuing technical review and improvement.

Approximately 46% of the scheduled mine production in 2025 will be sourced from remnant mining in the Main Orebody. In the following two years this proportion will reduce to 13% and 10%, respectively. In the final two years of LOM production, the average proportion from the Main Orebody will be 29.5%.

There is an elevated mining dilution and recovery risk associated with remnant mining in the Main Orebody, although in this instance, the design stopes have accounted for the inclusion of non-ore diluents. The planned dilution from the Main Orebody remnants, whilst elevated when compared with the South Orebody dilution, is considered to be reasonable when considering the location of these remnants.

Ordinarily, an inventory associated with already developed underground openings, could be classified as Proven ore where the presence of payable/economic mineralisation is readily evident. Although the Main Orebody remnants are located within already developed sublevels of the mine, the decision to classify them as Probable ore stems from the preceding Indicated Mineral Resource classification.

Whilst now approaching the end of their useful operating life, major items of primary equipment will be replaced over time. A review of the proposed new equipment purchases relative to operational performance to date, suggests that the replacement and additional fleet ought to be adequate for the extended mine life.

25.2.2 Mine ventilation

A review of the existing and proposed extension of the mine ventilation network was carried out to assess if the planned Main Orebody and South Orebody stoping areas could be adequately ventilated and therefore able to be mined and reported as Mineral Reserves. This review was completed in conjunction with an extrapolation and estimation of the annual airflow requirements over the life of mine.

In concluding that the proposed network should be sufficient, the following qualifying comments are made:

1. Whilst not a flawed system, the ventilation layout appears to be relatively unusual because of the force-ventilating near-surface underground fans, with additional drawn-down fresh air flow through the partially backfilled main shaft, and with exhaust through the main decline.
2. In considering the ventilation requirements to cater for expanded production, the unstable landform above the Main and South Orebodies does not avail itself for the installation of surface exhaust fans, and hence the proposal for a supplementary force-ventilating fan positioned underground beyond the new South portal.
3. Without the proposed new 400 kW fan, it appears there would otherwise be a shortfall in supplying sufficient air for primary equipment operating in the future.
4. The network, particularly within the remnant areas of the Main Orebody, is relatively complicated and airflow directions on the orebody sublevels will change over time.
5. The network will therefore require a dynamic regulating arrangement of auxiliary ventilation fans and stopping/control devices.

6. During development, the new South Orebody, below the 1125 mRL sublevel, will be ventilated by means of return air circulating from other sublevels and along access headings across from the Main Orebody.
7. Indicative airflow quantities have been shown for the Main Orebody and South Orebody sublevels. These quantities have been modelled using conventional ventilation simulation software.
8. Within the available detail of the new production schedule update, however, the projected number of diesel operated machines operating on these sublevels at any one time, cannot readily be ascertained.
9. It is assumed therefore, that these flow quantities as-modelled, could potentially be a limitation on the number of trucks and LHDs that can be in operation and hauling/loading on multiple sublevels at the same time.
10. Whilst careful production sequencing will be required, particularly when producing from both orebodies at the same time, there is some flexibility within the overall ventilation system through the possibility of drawing supplementary fresh air from the old main shaft airway.

25.3 Metallurgy and mineral processing

Plant feed from the Main Orebody remnant mining will be processed in Years 2025 to 2027, and in 2035 and 2036. Projected average recoveries and concentrate grades for this feed are based on historical data. These projections are listed in Table 25-1.

Table 25-1 Projected average recoveries and concentrate grades for Main Orebody plant feed

	Recoveries, %		Concentrate grades				
	Cu	Zn		% Cu	% Zn	Ag ppm	Au ppm
Yellow Ore (Spec con.)	92.0		**Cu**	22.0	2.4	45.0	1.5
		30.0	**Zn**	5.0	40.0	0.0	0.0
Black & Clastic Ores	84.0		**Cu**	17.0	12.0	94.0	1.7
(Non-spec Cu con.)		67.0	**Zn**	5.0	40.0	94.0	0.0

Ore mined from the South Orebody will be metallurgically very similar to ores mined from the Main Orebody, with footwall (FW) ores being similar to Yellow ores currently being treated, and the zinc ores being similar

Mineralogical investigations, a plant trial on South Orebody development material, and comminution and flotation testwork at Hacettepe University (2024a and 2024b) have confirmed that these ores can be successfully treated through the existing processing facilities at Çayeli, with no modifications required to the plant.

However, the following recommendations apply:

- The different ores must be campaigned through the plant, with FW (SYO) material treated separately from the zinc (SBO) ores.
- High and low grade SBO zinc ores should be blended to control zinc grades, enabling a Spec zinc concentrate and a Non-spec copper concentrate to be produced.
- To avoid overwhelming the zinc concentrate treatment circuit, the overall zinc feed grade should be controlled.

Table 25-2 lists the recommended overall average recoveries and concentrate grades for the treatment of South Orebody plant feed.

Table 25-2 Projected average recoveries and concentrate grades for the South Orebody plant feed

	Recoveries, %		Concentrate grades				
	Cu	Zn		% Cu	% Zn	Ag ppm	Au ppm
Footwall (Spec con.)	92.0		**Cu**	23.0	2.5	20.0	1.3
Zinc Ores (Blend 2)	60.0	75.0	**Cu**	19.0	10.0	40.0	5.0
(Non-spec Cu con.)			**Zn**	5.0	50.0	65.0	3.0

The existing facilities have been in operation for over thirty years, and a programme of replacement of old and corroded equipment has been in place for several years. A list of further equipment upgrades and replacements has been made for the remaining LOM, and the cost of this has been estimated at $6.6 M.

25.4 Tailings disposal

That portion of the process tailings that is not used for underground paste filling is transferred via pipeline to a mixing tank on the Çayeli coast. The tailing is discharged from that tank into another pipeline extending out into the Black Sea (DST). Tailings generated from the processing of South Orebody ore will be disposed of in the same way. The QP considers that there is minimal risk that the method of underground paste filling would be unsuitable for the extended life of mine.

Currently, there is no suitable surface tailings storage possibility to replace or augment DST. Comments on the DST risk are provided in Item 25.7.

25.5 Water management

Water inflows to the Operations are managed through adherence to the *Water Pollution Control Regulations of Türkiye.* Water flowing into the underground mine is pumped to surface into one of several receiving ponds. This also applies to rainfall run-off and contact water collected and contained across the site. Water is reclaimed from these ponds for use in the mine, the process plant, the concrete batch plant and for dust suppression. Domestic wastewater is treated and pumped out to sea via the DST mix tank.

In view of the regulated monitoring and water quality sampling that takes place, the QP is of the opinion that the Operations are not at risk from non-compliance to water management regulations.

25.6 Infrastructure

New underground infrastructure is required for development and production from the South Orebody. This includes ramps and access development, ventilation airways and associated equipment, new pumping stations, and additions to the paste fill distribution system.

Apart from a new primary crusher, the only other significant infrastructure required for the processing plant is in relation to replacement cells and tanks. No additional equipment will be required to handle the increased ore throughput and the feed grades indicated in the new life of mine production plan. The existing operating cells and idle cells will be sufficient to cater for the combined feed from the Main and South Orebodies.

25.7 Environmental studies and permitting

ÇBI's Environmental Permit renewal application needed to be submitted before the 13th of September 2025. The DST activities will need to be addressed in this renewal application and the appointment of a "special expertise commission" could continue to linger.

ÇBI management does not anticipate any challenge to this permit renewal considering the DST monitoring programme that has been in place for many years, and there being no evidence of an adverse change in water quality to date. Furthermore, during the current renewal process, the Ministry has reiterated its earlier position in writing, stating: *"…from the perspective of waste management legislation, there is no objection to submitting an environmental permit application."*

Although the E&U Ministry might honour their previous advice that the Operations can continue whilst waiting for the commission's deliberation, there is no certainty that the permit renewal will eventually be forthcoming. The ÇBI management team intends to address this risk through continued discussions with the Ministry during 2025.

25.8 Cost estimation

The cost estimation completed for the Mineral Reserves NSR valuation and cashflow modelling indicates that approximately 65% of all projected costs for the Çayeli life of mine to 2036 are associated with operating expenses, primarily mining (27%) and then processing (15%) costs, followed by combined plant and G&A costs.

Approximately 10% of the total projected costs relate to capital provisions, and the remaining 25% cover TCRCs and royalties.

The operating costs have a direct bearing on cut-off grades and the NSR estimates that are fundamental to defining the Mineral Reserves. Some retrospective comments on the estimation of the major operating costs are offered as follows:

- The mining costs have been estimated in detail by referencing 2024 budget provisions, as derived from a comprehensive "bottom-up" account of consumables unit costs and consumption rates. The estimate included power supply/usage, diesel fuel supply/usage, explosives, ground support, ground engaging tools, ventilation items, paste filling and barricading. All of which could be recast into development costs (ore and waste), stoping costs, services costs and maintenance costs. This itemisation was able to be split into fixed and variable components.
- Similarly, the processing costs were estimated by referencing the unit cost of consumables and average consumption rates for these consumables. In relation to the new South Orebody production, consumption rates were also derived from recent testwork by Hacettepe University (2024a, 2024b). The processing cost projections were also able to be split into fixed and variable components.
- In regard to the overall fixed costs, the mining cost estimate showed a value of 45% of the total costs, and 33% of the respective total for processing costs.
- An analysis was completed to assess the impact of haulage costs on the total mine operating costs, with findings as follows:
  - in relation to mine development, the combined diesel fuel and haulage costs are substantially outweighed by the intensive ground control costs incurred throughout the mine
  - in the case of stope production, the combined diesel fuel and haulage costs are comparable with the combined costs for extensive barricade construction and paste filling
  - these indications also support an observation that haulage profiles in any one year, can be over multiple sublevels, rather than being confined to specific upper or lower horizons
- Without pre-empting a production schedule with explicitly defined ore and waste mining sources and destinations, and notwithstanding the above findings, it was not feasible to assign operating costs which would vary annually and over the full range of operating sublevels[13]. Accounting for the current

[13] This may be possible with an alternative mining software approach (refer to recommendations in Item 26.2).

ÇBI circumstances, another analysis was carried out to assess the impact of using overall average rather than variable operating costs. Some specific outcomes were evident when assigning costs to a preliminary mining physicals schedule, e.g.:

- adopting overall average operating costs results in an approximate loss of 3% of the ore inventory up to 2030, and a gain of about 5% from 2031
- adopting overall average operating costs results in an approximate loss of 1% of the metal inventory up to 2030, and a gain of about 3% from 2031
- these figures indicate that there is possible conservatism in the early years of production when producing from the combined Main and South Orebodies, offset by possible optimism in the latter years when producing at higher throughputs from the South Orebody only

An overall conclusion drawn from the above commentary is that the mining and process operating costs, which are the larger of the Operations expenditure items for the purposes of this Technical Report, are relatively high cost items, with a significant fixed cost component. The adoption of a unit cost average, particularly for mining variable costs, does not appear to be untoward considering the analyses that have been undertaken.

Considering the relatively high actual NSR values for designed stopes (fully diluted and recovered), relative to the adopted overall average $77/t operating cost, there appears to be minimal risk from applying an average cost in these reporting circumstances, and which averages are largely derived from recent actual unit costs and consumption rates.

Item 26 RECOMMENDATIONS

26.1 Geology and Mineral Resource estimation

With the Main Orebody almost depleted, the remaining near-mine development and exploration (drilling) will be focused largely in and around the South Orebody area. The extents of mineralisation in the vicinity of the South Orebody are only partially defined by the existing drilling and underground development.

For the South Orebody, the recommended primary opportunities (i.e., drilling targets), in the QP's opinion are:

1. The area between the South and Main Orebody, which has only limited drilling coverage to date.
2. At depth (<1000mRL) below the currently defined South Orebody Cu and Zn mineralisation envelopes.

To minimise the operational risk of the geology function, the recommendations of the QP's July 2024 site visit report (FQM, 2024) should be implemented. The key recommendations not yet fully implemented are:

- Compile detailed standard operating procedures (SOP) documentation for all key practices.
- Implement a documented QC routine to submit samples (CRMS'S blanks, etc) to monitor the sample preparation and analysis at the site laboratory. The data needs to be routinely analysed and documented.
- Send 400 returned sample pulps to a certified third party laboratory as a check on the quality of the grade data collected in the South Orebody area.
- Implement an industry standard digital database for storing drilling data as opposed to the current practice of using MSExcel.
- Implement the use of computerised tablets for record items like logging and density data removing the need to used manual handwritten logs.

26.2 Mining and Mineral Reserve estimation

Item 15 and Item 16 described a conventional mine planning approach to the design of underground development and stoping limits, bounded by economic considerations and cognisant of various site-specific operational planning rules and geotechnical parameters. An impression could be gained that, with the number of individual design solids involved, there may be an opportunity to improve the efficiency and timeliness of the planning process. In the manner of continuous improvement, therefore, it is recommended that consideration be given to:

- Use of stope optimisation techniques to incorporate varying operating costs rather than life of mine average costs into the NSR value calculations, or otherwise optimising on varying copper equivalent block grades (i.e., where the equivalence takes account of varying process recovery, metal payability, operating costs, metal costs and royalties).
- Updating the cavity monitoring system records of stope overbreak and ore loss, especially for the newly commenced extraction in the South Orebody. This update would be useful for refining the generic unplanned mining dilution and recovery adjustments that would be applied for future Mineral Reserve estimates.
- Use of a suitable activity or resource based mine scheduling software package. Such software could possibly integrate the extensive operational rules and constraints that have been accumulated over many years at Çayeli. The objective would be to potentially improve upon the current manual process,

whilst effectively balancing multiple operational matters such as orderly sequencing, ventilation constraints, equipment utilisation etc.

26.3 Metallurgy and mineral processing

The treatment of the various ore types from the Main Orebody is well understood at Çayeli, and there are no additional recommendations for the treatment of the remnant Main Orebody ores.

The two main ore types from the South Orebody have been shown to be similar to the corresponding Main Orebody ore types, and can thus be treated successfully in the existing circuits. However, the following recommendations apply:

- The different ores must be campaigned through the plant, with footwall (SYO) material treated separately from the zinc (SBO) ores.
- High and low grade SBO zinc ores should be blended to control zinc grades, enabling a Spec zinc concentrate and a Non-spec copper concentrate to be produced.
- To avoid overwhelming the zinc concentrate treatment circuit, the overall zinc feed grade should be controlled.

No ore variability testing has been performed on the individual ore types from the South Orebody. As the orebody is developed, additional testing should be undertaken to ensure the homogeneity of the material and to prevent undue surprises during future processing.

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Item 28 Certificates

Richard Sulway
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I, Richard Sulway, do hereby certify that:

1. I have been employed by First Quantum Minerals Ltd since 2022 as a Group Principal Geologist, Mine and Resources. Prior to this time, in the period 2015 to 2021 I was retained as an independent contractor by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled "Çayeli Operations, Çayeli Bakir, Rize Province, Türkiye, NI 43-101 Technical Report", dated effective 30th April 2025 (the "Technical Report").
3. I am a professional geologist having graduated with a Bachelor of Applied Science degree with Honours (1989) in Applied Geology from the University of Technology Sydney. I have a Master's degree (Geological Data Processing) from the University of New South Wales, Sydney (1995) and I am a Member of the Australasian Institute of Mining and Metallurgy with Chartered Professional status, MAusIMM(CP).
4. I have worked as a geologist for a total of thirty four years since my graduation from university. During this period I have gained over 20 years of experience in Mineral Resource estimation both when working for First Quantum Minerals Ltd (10 years) and as an employee for a Perth based mining consulting firm (10 years).
5. 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.
6. I visited the Çayeli Operations between the 26th of June and the 13th of July 2024.
7. I am responsible for the preparation of those portions of the Technical Report relating to reliance on other experts, drilling, sample preparation, analyses and security, data verification and Mineral Resource estimates (namely Items 3, 10, 11, 12 and 14).
8. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd given my long association with The Company (10 years).
9. I have had no prior involvement with the Property that is the subject of the Technical Report.
10. I have read NI 43-101 disclosure document and Form 43-101F1. The Technical Report has been prepared in compliance with that instrument and form.
11. 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 for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this XXX day of XXXXX 2025 at West Perth, Western Australia, Australia.

Richard Sulway

Michael Lawlor
First Quantum Minerals Ltd
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Tel +61 8 9346 0100; mike.lawlor@fqml.com

I, Michael Lawlor, do hereby certify that:

1. I am a Mining Technical Advisor employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled "Çayeli Operations, Çayeli Bakır, Rize Province, Türkiye, NI 43-101 Technical Report", dated effective 30th April 2025 (the "Technical Report").
3. I am a professional mining engineer having graduated with an undergraduate degree of Bachelor of Engineering (Honours) from the Western Australian School of Mines in 1986. In addition, I have obtained a Master of Engineering Science degree from the James Cook University of North Queensland (1993), and subsequent Graduate Certificates in Mineral Economics and Project Management from Curtin University (Western Australia).
4. I am a Fellow of the Australasian Institute of Mining and Metallurgy.
5. I have worked as a mining and geotechnical engineer for a period in excess of thirty five years since my graduation from university. Within the last fifteen years I have held senior technical management positions in copper mining companies operating in Central Africa, and before that, as a consulting mining engineer working on mine planning and evaluations for metalliferous 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 visited the Çayeli Operations between the 18th of August and the 2nd of September 2024, and again between the 30th of April and the 15th of May 2025. During these visits, I inspected the underground mine workings and held numerous discussions with the mine technical and operations staff. Whilst on site, I also reviewed mine plans and schedules related to the proposed production from the Main Orebody remnant areas and from the newly defined South Orebody.
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, and for Items 1, 2, and 18 to 26.
9. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.
10. I have had no 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 for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this xxxxx day of xxxxxxxx 2025 at West Perth, Western Australia, Australia.

M. Lawlor

Michael Lawlor

Andrew Briggs
First Quantum Minerals Ltd
18-32 Parliament Place, West Perth, Western Australia, 6005
Tel +61 8 9346 0100; andy.briggs@fqml.com

I, Andrew Briggs, do hereby certify that:

1. I am a Group Consulting Project Metallurgist employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled "Çayeli Operations, Çayeli Bakır, Rize Province, Türkiye, NI 43-101 Technical Report", dated effective 30th April 2025 (the "Technical Report").
3. I am a professional metallurgist having graduated in 1974 from the Imperial College (Royal School of Mines), London, with a BSc (Eng) First Class in Metallurgy.
4. I am a Fellow of the Southern African Institute of Mining and Metallurgy.
5. I have worked as a process engineer and metallurgist since graduation in 1974 (51 years); the first 13 years of which were in operating positions up to Metallurgical Manager in the gold mining industry. This was followed by 19 years in engineering companies in Process Design for projects worldwide, and finally 18 years with First Quantum Minerals Ltd as a Process Consultant.
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 last visited the Çayeli Operations between the 18th and 25th of August 2024. Previous visits were in October 2013, May 2014 and June 2023.
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, respectively. I am also responsible for the estimates in Item 21 pertaining to processing, plus general and administration costs.
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 was in providing operational support and advice on technical matters to the ÇBI processing team.
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 for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this xxxxxx day of xxxxxxxx 2025 at West Perth, Western Australia, Australia.

Andrew Briggs