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Technical Report

Technical Report, Skouries Project, Greece

Eldorado Gold Corporation

Halkidiki Peninsula, Central Macedonian Province, Northern Greece

In accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators

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1 Summary

1.1 Introduction

This Technical Report on the Skouries Property (Property) located on the Halkidiki Peninsula in northern Greece has been prepared by AMC Mining Consultants (Canada) Ltd. (AMC) of Vancouver, Canada on behalf of Eldorado Gold Corporation (Eldorado) of Vancouver, Canada. AMC and its consultants are independent of Eldorado. It has been prepared to a standard that is in accordance with the requirements of National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects of the Canadian Securities Administrators (CSA). It incorporates the work of other consultants who are also independent qualified persons (QPs), and as such, is an independent NI 43-101 Technical Report.

The report is an update to the "Technical Report, Skouries Project, Greece" prepared by Eldorado with an effective date of 1 January 2018. The main purpose of this Technical Report is to report the results of the "Skouries Feasibility Study" (Feasibility Study) and to support the Eldorado annual statement of Mineral Resources and Mineral Reserves.

Eldorado is an international gold mining company based in Vancouver, British Columbia. It is listed on the Toronto Stock Exchange as "ELD" and the New York Stock Exchange as "EGO".

Currency used throughout this report is the lawful currency of the United States (US$) unless otherwise stated.

1.2 Location and ownership

Eldorado, through its 95% owned subsidiary Hellas Gold SA (Hellas Gold), owns the Property. Hellas Gold was acquired as part of the acquisition of European Goldfields Limited (EGL) completed in February 2012.

The Property is located within the Kassandra Mines complex, located on the Halkidiki Peninsula of northern Greece. The complex is located approximately 100 kilometres (km) east of Thessaloniki and comprises a group of mining and exploration concessions covering 317 squared kilometres (km^2), of which the Property is part. The Properties within the complex include the Olympias Mine currently in production, Stratoni Mine on care and maintenance, and the Skouries copper-gold porphyry deposit under development but currently on care and maintenance.

1.3 Property description

The Skouries project (Skouries Project) is a gold-copper porphyry deposit to be mined using a combination of conventional open pit and underground mining techniques. The mineral processing facilities will produce a gold-copper concentrate.

The Property is situated at an elevation range of 350 metres above sea level (masl) to 620 masl near the village of Megali Panagia in the prefecture of Halkidiki, northern Greece. It is approximately 7.2 km from the road connecting the villages of Megali Panagia and Palaiochori. The area is centred on co-ordinates 4745300 E and 4481400 N of the Greek Reference System EGSA '87, at approximately Latitude 40°29' and Longitude 23°42'. The location is classified according to Greek Seismic Code NEAK 2000 (modified in 2003) as Zone II.

The Property consists of concession numbers OP03, OP04, OP20, OP38, OP39, OP40, OP48, and OP57, which have a combined area of 55.1 km^2. Hellas Gold has been granted mining rights over these concessions until 6 April 2024. The concessions are conditionally renewable for a further two consecutive periods of 25 years each. Hellas Gold owns a small portion of private land within the concessions, is granted use of forestry land and is in negotiation for the remaining 0.3% of the total area required.

The Environmental Impact Study (EIS) for the Kassandra Mines Mineral Deposits Project (Kassandra Project) includes an area of 26,400 hectares (ha), in north-eastern Halkidiki (Macedonia Region). The Kassandra Project includes the Skouries, Olympias and Stratoni sites. The Skouries Project covers approximately 250 ha of the Kassandra Complex.

The EIS considers the potential impact on the local and regional environment as it relates to:

- Open pit and underground workings.
- Tailings impoundment.
- Process plant.
- Infrastructure necessary for the Project operation.

1.4 Permitting

The technical study submitted to the Ministry of Environment (MOE) for the Project was initially approved in February 2012. After numerous supplements relating to flotation plant, Tailings Management Facility (TMF) arrangements and "auxiliary temporary facilities", approval by the MOE was granted in 2013 - 14. An updated technical study covering amended aspects of the process plant and associated infrastructure was submitted to the MOE in December 2015, and this was approved in May 2016.

Subsequently, an updated specific technical study for the flotation plant was submitted to the MOE and approved on 11 November 2016. An update of the installation permit for the flotation plant was submitted by August 2016 and this was approved on 3 September 2019.

An Investment Agreement (IA) which amends the 2003 Transfer Agreement and provides a modernized legal and financial framework to allow for the advancement of Eldorado's investment in the Kassandra Mines was ratified in early 2021. After the 2019 Greek Parliamentary elections, when Eldorado initiated talks with the newly established government, outstanding routine permits were released.

Hellas Gold has provided a €50.0 million (M) Letter of Guarantee to the MOE as security for the due and proper performance of rehabilitation works in relation to the mining and metallurgical facilities of the Kassandra Mines project and the removal, cleaning, and rehabilitation of the old, disturbed areas from the historical mining activity in the wider area of the project. Additionally, a Letter of Guarantee to the MOE, in the amount of €7.5M, has been provided as security for the due and proper performance of the Kokkinolakkas TMF.

1.5 History

There is a long history of mining in the region dating back to 350 to 300 BC and continuing through the Roman, Byzantine, and Ottoman periods. There is limited historic development at the Skouries site.

In modern times, the Skouries deposit was initially drilled by Nippon Mining and Placer Development (Placer) during the 1960s. Placer also carried out limited underground development from an adit. The deposit was subsequently drilled in the 1970s by the Hellenic Fertiliser Company. TVX Gold Incorporated (TVX) began a drilling program in August 1996 to confirm the deposit and to explore it at depth. A subsequent infill drilling program was conducted in 1997 with the objective of improving the evaluation of Indicated Mineral Resources in the deeper high-grade zone. EGL acquired the Property in 2004, audited the TVX program and prepared a pre-feasibility study in

2006. The pre-feasibility study reflected an open pit operation followed by an underground mine using sub-level caving (SLC) underground mining methods at a production rate of 6.5 million tonnes per annum (Mtpa).

A later study prepared by EGL incorporated the use of sub-level open stoping (SLOS) with tailings backfill. This methodology formed the basis of the approved EIS. The Mineral Reserves estimated from this work formed the basis for those quoted in the 2011 EGL Technical Report.

1.6 Deposit geology

The Skouries deposit is centred on a small porphyry stock that has a surface expression of approximately 200 metres (m) in diameter. Skouries is typical of a gold-copper pencil porphyry. Mineralization occurs in stockwork veins, veinlets and disseminated styles typical of a porphyry, which has a sub-vertical, pipe-like shape.

Mineralization has been tested to a depth of 920 m from surface and the results show the orebody is open at depth. Potassic alteration and copper-gold mineralization also extend into the country rock; approximately two thirds of the Measured and Indicated Resources are hosted outside of the porphyry, with about a 50:50 split in gold-equivalent ounces.

1.7 Drilling

Diamond drillholes are the sole source of subsurface geologic and grade data for the Skouries Project. Resource delineation drilling was carried out in two major campaigns: in 1996 – 98 by then owner TVX and in 2012 to 2013 by Eldorado.

TVX drilled a total 72,232 m of core in 121 drillholes using NQ (47.6 millimetres (mm)) diameter core. Holes reached a maximum depth of 1,013 m.

Eldorado conducted two drill campaigns on the Skouries Project in 2012 and 2013: a 34-hole, infill drilling program comprising 6,922 m and a 10-hole, 6,617 m confirmation program. The confirmation program was completed to test the core of the main mineralized portion of the deposit to compensate for the lack of a drillcore record from the earlier TVX campaign. These confirmation drillholes confirmed the earlier results and are not included in the current Mineral Resource estimation.

1.8 Sample preparation and analysis

The majority of the core samples for the Skouries Project originated from the 1996 – 98 drill campaign by TVX. Eldorado has reviewed the TVX studies and quality control / quality assurance (QA/QC) procedures and agrees with the conclusions that the drill data are acceptable to be used for Mineral Resource estimation. The QP concurs with this conclusion on the pre-Eldorado data having reviewed the reports. The background and QA/QC results of the Eldorado work were reviewed in detail under the QP supervision, replotted and deemed suitable for estimation purposes.

Confidence in the data is also provided by the results of Eldorado’s confirmation drill program.

1.9 Metallurgical testwork

Metallurgical testwork and studies were performed by Lakefield Research, Canada on composites selected from core samples of the major rock types, covering mineralogy, grinding and flotation. This testing was carried out to support the original 2007 design completed by Aker Kvaerner. Based upon this information, the criteria for process plant and infrastructure design were established.

Additional testwork was completed by Outotec in 2007, mostly at its laboratory in Pori, Finland, to give additional design confidence. This included flash flotation, gravity gold recovery, concentrate settling and filtration.

Further supplementary testwork was undertaken by FLS Knelson in 2013 on gravity gold recovery and by Wardell Armstrong in 2015 on flotation concentrate. Solvay (formerly Cytec), in 2016, and Bureau Veritas Commodities Canada, in 2017, worked on selective flotation of copper from pyrite-rich ore. In 2014, Orway Mineral Consultants (OMC) reviewed the testwork conducted by Aker Kvaerner to design the Skouries grinding circuit and conducted comminution circuit modelling studies using circuit simulations.

1.10 Mineral Resources

The Mineral Resource estimate for the Skouries deposit was developed using assays and data from surface diamond drillholes. The estimate was made from a three-dimensional (3D) block model based on initial outlines derived by a method of probability assisted constrained kriging (PACK). The estimation, for both gold and copper, was within what is termed the 0.1% Cu PACK shell. The block size for the Skouries model was selected based on mining selectivity considerations and is 5 m x 5 m x 10 m.

Copper and gold grades are highest in the porphyry. The gold to copper ratios are also markedly different between the intrusive and non-intrusive units. Generally, the coefficient of variance (CV) values for copper in all units is relatively low reflecting the porphyry style mineralization of the deposit. Gold CV values are higher, especially in the schist unit, reflecting some influence by local extreme grades. These were mitigated by a gold grade cap equal to 20 grams per tonne (g/t), applied to the assay data prior to compositing.

The assays were composited into 4 m fixed-length downhole composites and were back-tagged by the mineralized shell and lithology units. The compositing process and subsequent back-tagging was reviewed and found to have performed as expected. Modelling consisted of grade interpolation by ordinary kriging (OK). A two-pass approach was instituted for interpolation. Nearest-neighbour (NN) grades were also interpolated for validation purposes.

As part of this reporting, the QP reviewed and validated the model by performing visual, statistical, and graphical checks in the form of a series of swath plots and checking reporting. On this basis, the QP is comfortable with the validity of the model.

The Mineral Resources of the Skouries deposit were classified using logic consistent with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards (2014). The mineralization of the Skouries deposit satisfies sufficient criteria to allow classification into Measured, Indicated, and Inferred Mineral Resource categories.

Reasonable grade and geologic continuity are demonstrated over most of the Skouries deposit, which is drilled generally on 40 m to 80 m, spaced sections. A two-hole rule was used where blocks containing an estimate resulting from two or more samples, all within 80 m and from different holes, were classified as Indicated Mineral Resources. For Measured Mineral Resource classification, a three-hole rule was applied where blocks contained an estimate resulting from three or more samples, all within 50 m and from different holes.

All remaining model blocks containing a gold grade estimate were classified as Inferred Mineral Resources.

The demonstration of Reasonable Prospects for Eventual Economic Extraction (RPEEE) was handled for both the open pit and underground portions of the deposit by creating potentially mineable shapes. In each case a long-term gold price of US$1,800/oz and copper price of US$3.50/lb were selected for the determination of Mineral Resource cut-off grades and pit shell. A gold equivalent (AuEq) calculation was used to combine the value of the two payable metals. The cut-offs used for defining the shapes were 0.3 g/t AuEq for open pit and 0.7 g/t AuEq for underground where AuEq is determined by AuEq = Au g/t + 1.25* Cu%. The parameters for cut-off grade calculations are listed in Table 1.1.

Table 1.1 Economic parameters for RPEEE evaluation

Description	Units	Open pit	Underground
Gold price	US$/oz	1,800	1,800
Copper price	US$/lb	3.50	3.50
Mining cost	US$/t processed	4.10	19.50
Process cost	US$/t processed	8.48	8.48
Filter plant cost	US$/t processed	2.13	2.13
IEWMF and water management	US$/t processed	0.13	0.13
G&A	US$/t processed	2.78	2.78
Overall costs	US$/t processed	17.62	33.02
Mill Au recovery	%	86.7	86.7
Mill Cu recovery	%	91.5	91.5
Cut-off used	AuEq g/t	0.3	0.7

The potentially mineable shapes representing volumes that have a reasonable expectation of being mined were determined as follows. Volumes that lie within both the 0.1% Cu PACK shell and the open pit shell and are predominantly above a cut-off grade of 0.3 g/t AuEq are assigned to the open pit reporting shape. Volumes that lie outside the open pit shell and lie within the 0.1% Cu PACK shell and are predominantly above a 0.7 g/t AuEq cut-off grade are assigned to the underground resource reporting shape. Volumes within both the open pit and underground resource reporting shapes are reported in their entirety; this includes some isolated blocks that are below the assigned cut-off, but that lie within the volumes deemed to be reasonably mineable. Similarly, isolated blocks that are above the cut-off grades, but that lie outside of the expected mineable volumes are omitted from the Mineral Resource estimate.

The Skouries Mineral Resources as of 30 September 2021 are shown in Table 1.2. The economic parameters and AuEq factors used are defined in the footnotes.

Table 1.2 Skouries Mineral Resources, as of 30 September 2021

Category	Tonnes (kt)	Au (g/t)	Cu (%)	Contained Au (koz)	Contained Cu (kt)
Open pit Mineral Resources					
Measured	50,641	0.62	0.42	1,013	214
Indicated	14,151	0.22	0.22	99	32
Measured & Indicated	64,791	0.53	0.38	1,112	246
Inferred	784	0.16	0.18	4	1
Underground Mineral Resources					
Measured	40,073	1.14	0.63	1,467	252
Indicated	135,109	0.56	0.46	2,452	620
Measured & Indicated	175,182	0.70	0.50	3,919	872
Inferred	66,873	0.38	0.40	811	265
Total Mineral Resources					
Measured	90,714	0.85	0.51	2,479	466
Indicated	149,260	0.53	0.44	2,551	652
Measured & Indicated	239,974	0.65	0.47	5,030	1,118
Inferred	67,657	0.37	0.40	814	267

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Open pit Mineral Resources are constrained by a semi-optimized pit that is strongly permit and crown pillar constrained and are reported at a 0.3 g/t AuEq cut-off.
- Underground Mineral Resources are those outside the pit shell and are reported at a 0.70 g/t AuEq cut-off.
- AuEq = Au g/t + 1.25 * Cu%, based on US$1,800/oz Au and US$3.50/lb Cu, and recoveries of 86.7% for gold and 91.5% for copper.
- Mineral Resources are stated inclusive of Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Numbers may not compute exactly due to rounding.

Source: Eldorado, re-reported by AMC and approved by the QP.

The QP has validated the Mineral Resources. The data, methodology and analysis described in this report are considered sufficient for reporting Mineral Resources.

There is no difference between the Mineral Resources reported in September 2020 and September 2021 and both statements are made on the same basis. There has been no production from the deposit, hence no depletion from the block model.

1.11 Mineral Reserves

The Mineral Reserves at Skouries comprise an open pit and an underground component.

Block model items transferred from the geology model for mine planning included estimated grades for copper and gold as well as Mineral Resource classification. Measured and Indicated Mineral Resources have been used to define the pit limits and for reporting of Mineral Reserves for scheduling. Inferred Mineral Resources were not used in the determination of Mineral Reserves.

The open pit optimization was carried out using MineSight® mine planning software. The Skouries open pit is constrained by the existing EIS boundary on surface and a potential underground mining crown pillar, which limits the pit depth to 420 masl. In addition to the physical boundary constraints, the open pit design and overall size is also affected by a requirement to provide construction materials for the integrated extractive waste management facility (IEWMF).

The Mineral Reserves for the deposit were estimated using a gold price of US$1,300/oz and copper price of US$2.75/lb. The open pit Mineral Reserves are reported using a US$10.60/t net smelter return (NSR) cut-off value. The open pit combined Proven and Probable Mineral Reserves are 59.6 million tonnes (Mt) with an average grade of 0.57 g/t Au and 0.40% Cu.

The underground contribution to Mineral Reserves has been evaluated at a diluted NSR cut-off of US$33.33/t, incorporating unplanned diluting material of 5% for porphyry stopes and 5.5% for schist stopes that is assumed to carry no metal value, and assuming an overall mining recovery of 95%.

The Mineral Reserves for the underground deposit have been estimated to be 87.5 Mt with an average grade of 0.90 g/t Au and 0.58% Cu.

The combined Mineral Reserves for the Skouries Project, as of 30 September 2021, are stated in Table 1.3. These represent the sum of the open pit and the underground Mineral Reserves. The cut-offs for the Mineral Reserves are NSR based with US$10.60/t used in the open pit portion and US$33.33/t for the underground estimate.

Table 1.3 Skouries Mineral Reserves, as of 30 September 2021

Category	Tonnes (kt)	Au (g/t)	Cu (%)	Contained Au (koz)	Contained Cu (kt)
Proven	73,101	0.87	0.52	2,053	381
Probable	74,014	0.66	0.48	1,576	359
Proven & Probable	**147,116**	**0.77**	**0.50**	**3,630**	**740**

Notes:
- Cut-off value applied, Open Pit: US$10.60/t ore; Underground: US$33.33/t ore.
- Gold Price: US$1,300/oz.
- Metallurgical Gold Recovery: 92.62 – 17.5 x oxide (%) – 22 x $e^{(-1.2 \times \text{Au Grade (g/t)})}$.
- Copper Price: US$2.75/lb.
- Metallurgical Copper Recovery: 99.41 – 56 x oxide (%) – 41 x $e^{(-338 \times \text{Cu Grade (\%)})}$.
- Mining Recovery, Open Pit: 100%, Underground: 95%.
- Mining Dilution, Open Pit: 0.0%; Underground - Ore Development: 5.0%, Porphyry Stopes: 5.0%, Schist Stopes: 5.5%.
- Numbers may not compute exactly due to rounding.

Source: Mining Plus (MP) and approved by the QPs.

1.12 Mine production schedule

The Project is designed as a two-phase mining operation. Phase 1 consists of a combined open pit and underground mine, operating over nine years. Phase 2 consists of mining from the underground mine for a further 11 years. The total life-of-mine (LOM) is 20 years.

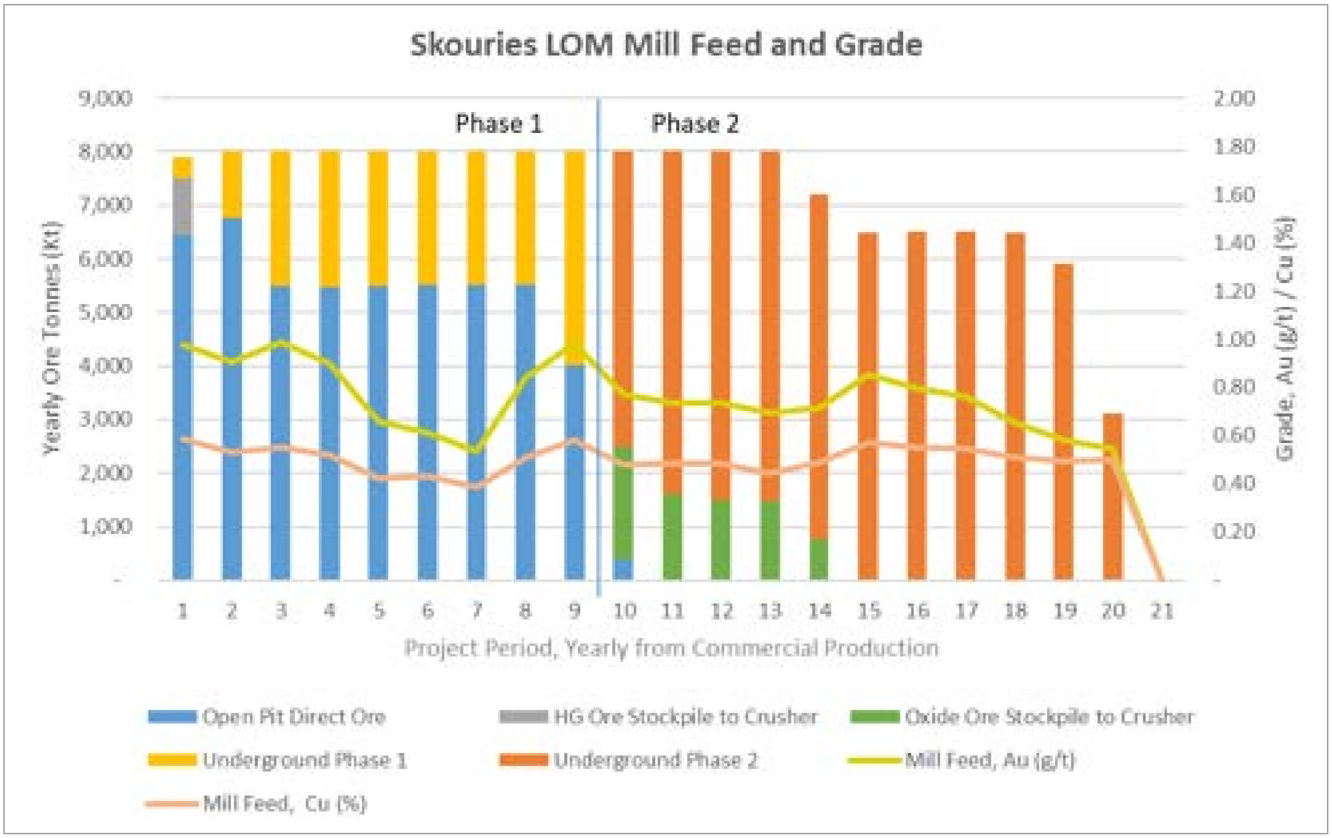
The production schedule has been developed to balance the materials volumes, metal production and capital expenditures over time, with consideration for the capacity of the surface tailings and waste management facilities

The LOM ore production schedule is shown in Figure 1.1.

Phase 1 mill feed is scheduled at 8.0 Mtpa, consisting of a nominal 5.5 Mtpa from the open pit mine combined with approximately 2.5 Mtpa from the underground mine. At the start of the mine life, during the initial two-year underground mine ramp-up period, the open pit feed rate is variable in order to maintain the 8.0 Mtpa mill feed. During Phase 1, 10.6 Mt of low-grade ore is stockpiled to be rehandled for mill feed during Phase 2. Phase 1 is completed at the end of the open pit mine life in Year 9.

Phase 2 mine production, from Year 10 to the end of the LOM, is provided from the underground mine. Phase 2 mine development begins in Year 4 to allow a seamless ramp up from the Phase 1 production of 2.5 Mtpa. During the first four years of Phase 2, the mill feed rate of 8.0 Mtpa is maintained by reclaiming low-grade ore stockpiled during Phase 1, at a rate that balances the mill feed to 8.0 Mtpa through Year 13. From Year 15 on, Phase 2 mill feed is maintained at a nominal feed rate of 6.5 Mtpa, solely from underground mine production, which tails off in Years 19 and 20.

Figure 1.1 Skouries LOM ore production schedule



Source: MP 2022.

1.13 Mining methods

1.13.1 Open pit

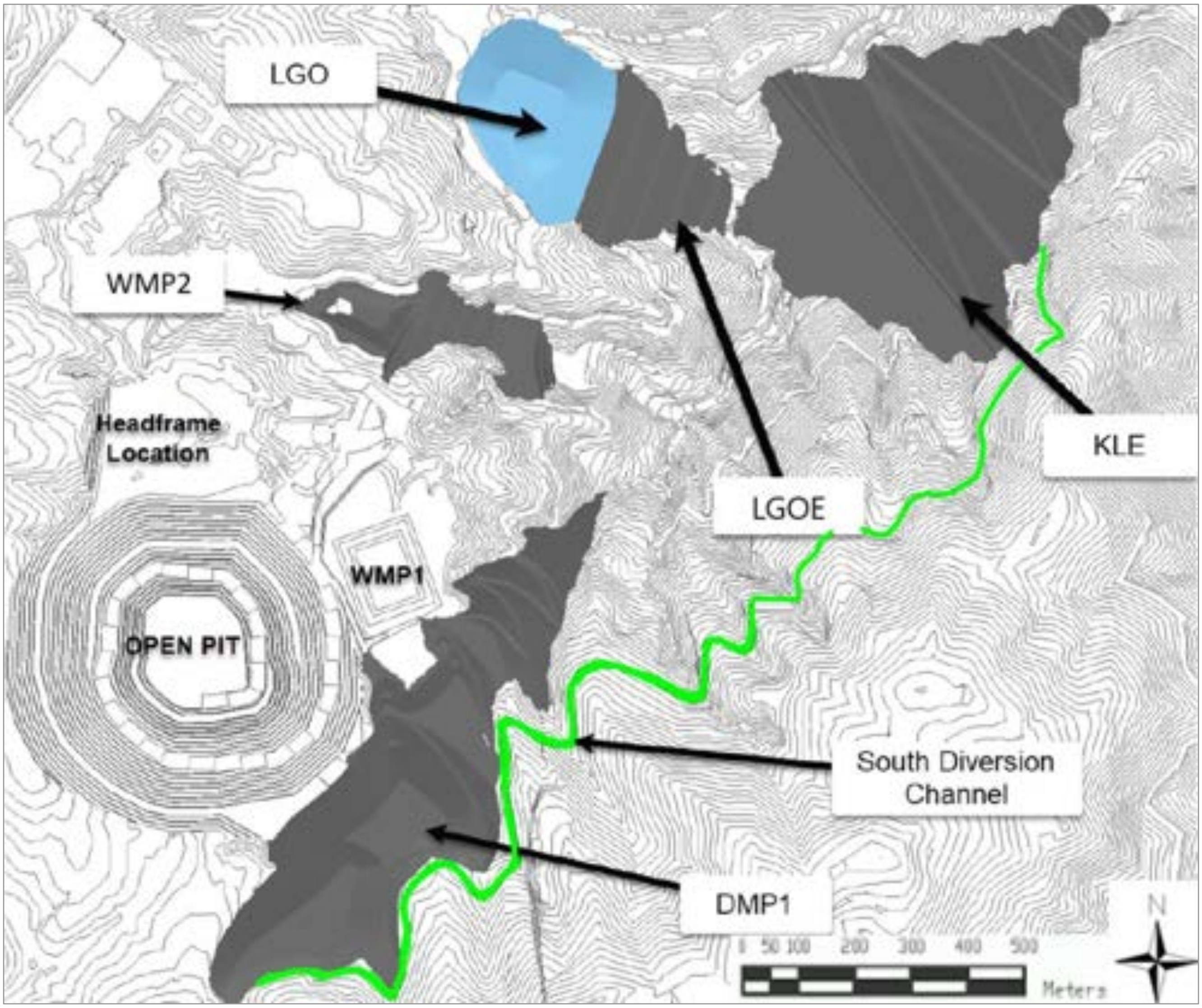
Open pit mining will be by conventional truck-shovel operation, with an ore production rate of approximately 5.5 Mtpa, at a waste to ore stripping ratio of approximately 0.90:1. The mining sequence will consist of drilling, blasting, loading, and hauling of ore and waste materials for processing and waste disposal. Based on the modelled rock types, approximately 17% of the mined material is amenable to free digging; this material will not be blasted.

Direct feed ore from the open pit will be hauled to the Skouries processing plant. A portion of low-grade ore (LGO) will be hauled directly to the plant, and an additional portion will be hauled to the low-grade ore stockpile (LGOS) where it will be re-handled during Phase 2 of the Project.

Waste material will be hauled directly to one of the material management structures within the IEWMF. The structures internal to the IEWMF are the LGO embankment, J5, Capping Rock Dump1, Cofferdam Karatza Lakkos (KL) Embankment, and South Diversion Channel (Figure 1.2).

Drilling operations will be carried out continuously as part of the normal mining operation. Once full mine production is reached, drilling and blasting of approximately 1 Mt (dry) per month will be required to maintain projected production levels.

Figure 1.2 Final pit design showing dumps and embankments



Notes:
- WMP1: Water Management Pond 1
- WMP2: Water Management Pond 2
- KLE: KL Embankment
- DMP1: Waste Rock Dump 1
- LGO: Low Grade Ore Stockpile
- LGOE: Low Grade Ore Stockpile Embankment

Source: MP 2022.

The primary haul roads are designed at 25 m width, based on a 90 tonne (t) haul truck. Other haul roads, to be used by contractor trucks, are designed for 55 t articulated haul trucks with an overall roadway width of 15 m.

The number of haulage units was determined by calculating cycle times in Haulage© from MinePlan© using annual haul cycle profiles from MinePlan©. Haulage calculations were carried out based on the designated 90 t and smaller 55 t trucks. A maximum truck speed limit of 50 km/h was set for flat or inclined roads, reducing to 15 km/h near shovel and dump points and 15 km/h around switchback corners. On the downhill segments, speeds were limited to a maximum of 25 km/h.

A tonnage factor for each material type was used to determine actual payload versus theoretical maximum payload for each truck class. These factors were based on experience from operations at other sites.

The open pit mine production schedule has been developed using a planned average annual ore production rate of 5.5 Mtpa. The actual yearly rate varies according to the ore production ramp-up schedule for the underground Phase 1, which will offset open pit ore. An open pit mining operation of 350 days per year consisting of three, eight-hour shifts operating 7 days a week is envisaged. The open pit production schedule is shown in Figure 1.3.

Figure 1.3 Open pit production schedule

	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Waste	6,590	8,809	5,942	7,733	7,075	7,456	6,903	4,950	2,952	1,347	303 K	66 K	0 K	0 K	0 K	0 K	0 K
Ore to Stock	0 K	44 K	227 K	1,967	1,853	1,373	1,998	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K
Stockpile (HGO) to Crusher	0 K	0 K	0 K	1,163	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K
Stockpile (LGO) to Crusher	0 K	0 K	100 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	0 K	2,105	1,624	1,500	1,469	664 K
Mine to Crusher	0 K	0 K	900 K	6,434	6,772	5,493	5,477	5,486	5,514	5,505	5,506	4,006	390 K	0 K	0 K	0 K	0 K
Au (g/t)	0.000	0.000	0.747	0.908	0.792	0.742	0.692	0.288	0.358	0.404	0.511	0.692	0.295	0.194	0.213		
Cu (%)	0.000	0.000	0.397	0.559	0.492	0.453	0.439	0.285	0.328	0.333	0.389	0.472	0.260	0.213	0.222		

Source: MP 2022.

1.13.2 Underground mining

The Skouries orebody that extends below the bottom of the open pit is amenable to a bulk underground mining method and has been evaluated under several different design approaches since the late 1990s, including block caving, SLC, and SLOS. SLOS has been confirmed as the most appropriate underground mining method for a number of reasons including:

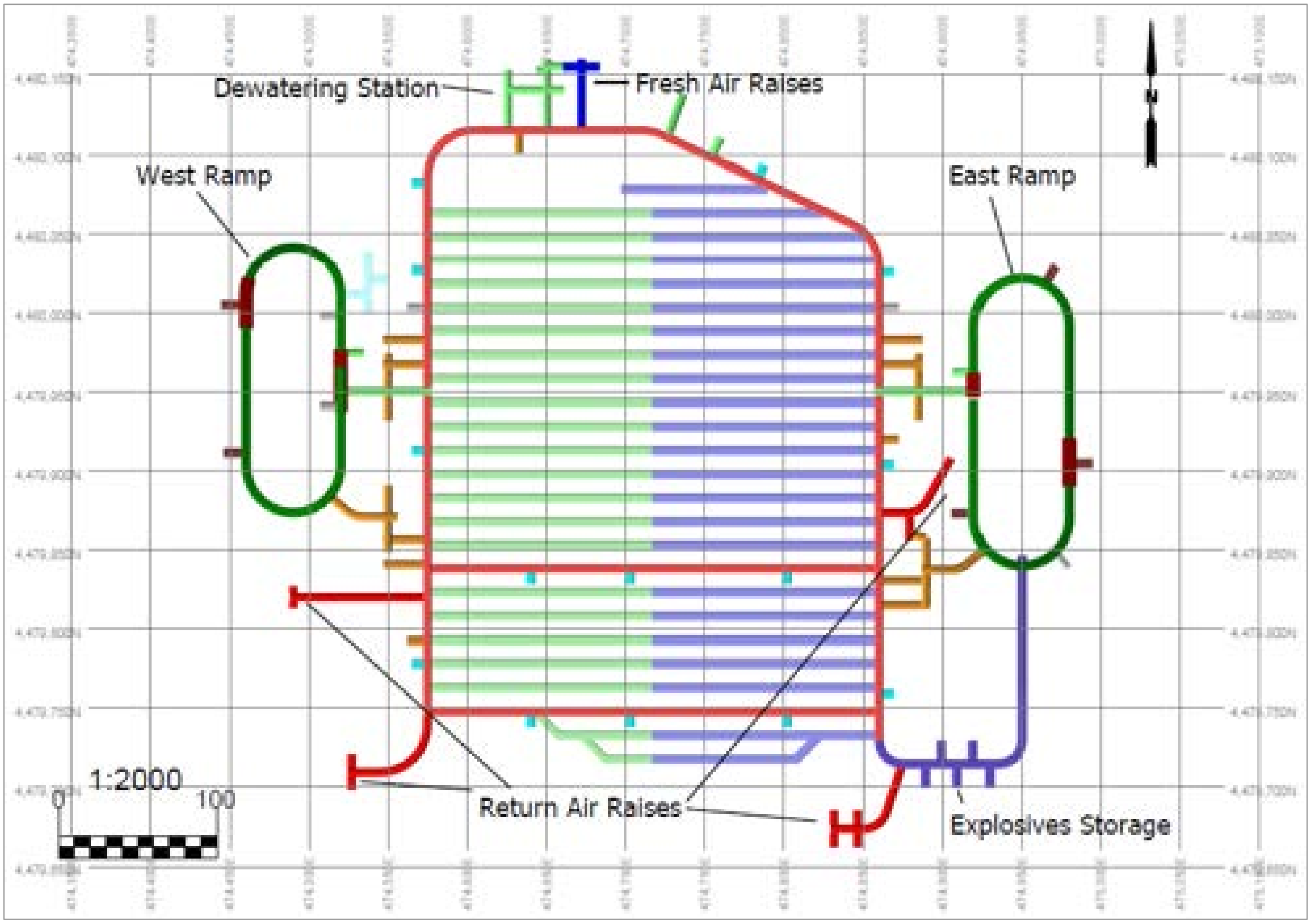
- The geo-technical stability of the final reclaimed land after closure of the Project.
- The minimization of land-take needed for the surface tailings.
- The ability to backfill the depleted open pit.

The majority of the stoping is considered to take place in reasonable quality rock mass. The stope stability assessment has indicated that, for stoping in the porphyry, a 60 m sub-level interval (60 m stope height plus 5 m top drive development) can largely be viable without significantly compromising stope wall stability if the length of the stope does not exceed 30 m. Of the stopes that will be extracted in the schist, only half of these excavations will expose schist in the stope sidewalls as secondary stopes will expose the paste backfill within the primaries.

Stope back stability assessments were conducted using the NGI-Q stability graph as well as the stability graph method to determine appropriate stoping spans. Stope span has been limited to 15 m. Thus, the standard stope dimensions were set to 65 m high x 30 m long x 15 m wide in porphyry stopes, 65 m high x 20 m long x 15 m wide for primary stope design in schist material, and 65 m high x 30 m long x 15 m wide for secondary stope design in schist material.

All levels in both phases have similar designs. Peripheral development (Ring-drives) will provide access to all sides of the orebody and terminate at return air raise (RAR) locations. Ore drives for stope extraction will traverse the orebody east to west on 15 m centres, developed incrementally to meet the production schedule and mining sequence. Both ramps are planned to be used to haul ore, with the orebody divided into East and West in order to maintain a stope extraction sequence from the centre out. The typical sub-level arrangement for the underground mine is shown on Figure 1.4.

Figure 1.4 Typical sub-level arrangement (230 Level)



Source: MP 2022.

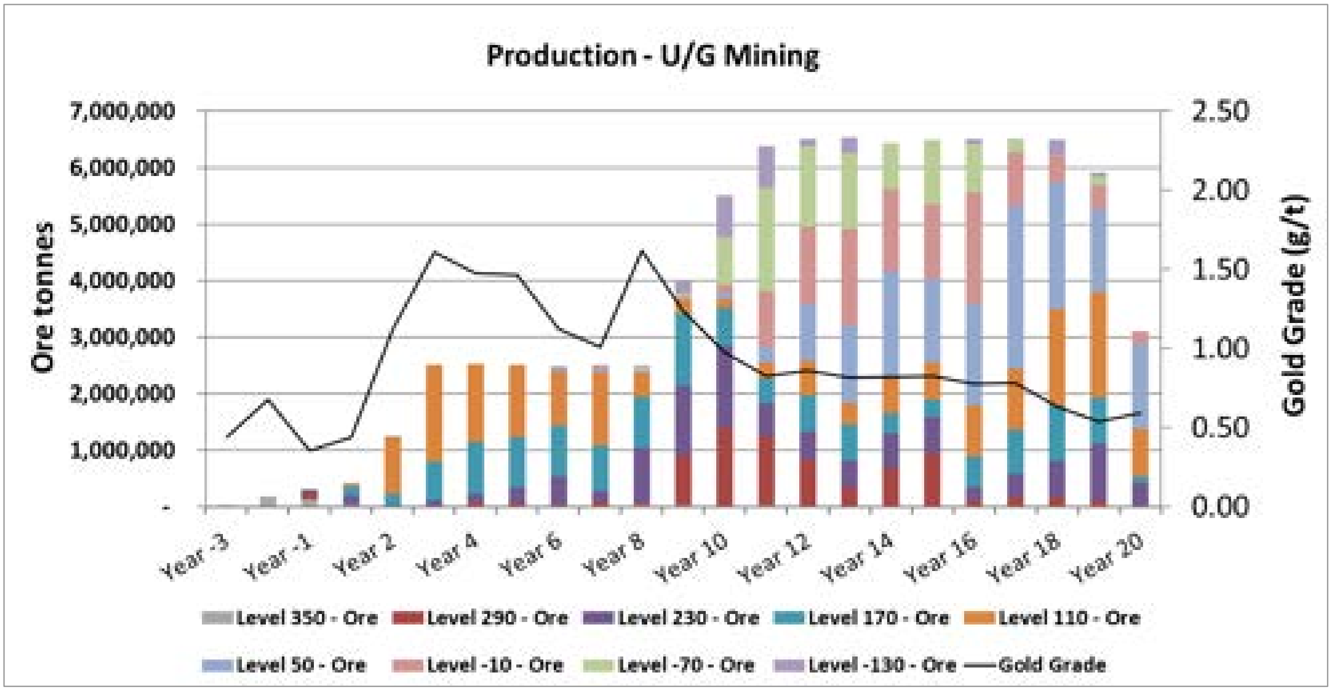
The underground portion of the Skouries Project will begin from the existing ramp from the surface to 385 masl. The ramp is currently developed to 35 m above the first production level, 350L. Mining will proceed to the 350L to establish major infrastructure and services. The 350L will serve as the mucking horizon for two test stopes, which are situated in the Crown Pillar and within the mining limits to enable a mineralized and accurate representation of the mining to be completed in Phase 1.

For Years -3 through to Year 2, underground mining efforts will focus on developing the access ramp and further establishing the levels and services for production, while also developing a second portal and ramp to the surface. In Year 4, the development will begin in preparation for Phase 2. This development will entail the dual ramp systems to -130L, the major underground workshop, fuel bay and excavations for the materials handling systems.

Underground mining will be by conventional underground mining techniques. The mining sequence will consist of drilling, blasting, loading, and hauling of ore and waste materials. During Phase 1, ore will be hauled to the surface crusher by truck. During Phase 2 ore will be hoisted to surface by a shaft. In Year 4, the shaft headframe construction will commence, and shaft excavation will begin in Year 6. Excavation of the shaft will continue through Year 8, with the entire materials handling system projected for completion six months prior to the beginning of Phase 2 in Year 10.

The underground production schedule is shown in Figure 1.5.

Figure 1.5 Underground production schedule



Source: MP 2022.

The design of the Skouries mine includes provision for remote mining technology (RMT), which has an impact on the cycle times of stopes and the productivity of equipment. This technology includes tele-remote operation of mechanized equipment by an operator located on surface or in a remote area of the underground mine. RMT is considered a best available technology, and Skouries mine is uniquely positioned to benefit from the improvements in mining process due to the simple repetitive nature of the design and the availability of highly skilled technical workers.

1.13.3 Underground materials handling

The materials handling strategy for Phase 1 is based on truck haulage of run-of-mine (ROM) ore directly to surface from the loading bays via the dual ramp system.

The Phase 2 materials handling will involve shaft hoisting of ore to surface. There are no vertical production nor development ore or waste passes included in the mine design; all broken rock will be loaded using load haul dumps (LHDs) and transported via the ramps in haul trucks. Shaft hoisting is critical to enable a ramp-up to full Phase 2 production of 6.5 Mtpa from the Phase 1 production of 2.5 Mtpa. In order to hoist the material by shaft, underground crushing will be implemented. During Phase 2, all stope ore and some late development ore will be hoisted to surface via the shaft. Development waste will continue to be hauled to surface via the dual ramp system, but these quantities are expected to be minimal.

1.14 Recovery methods

For the first nine years of operation, the ore will be extracted from the open pit mine as well as from the underground mine for a total mill feed rate of 8.0 Mtpa. From the tenth year of operation until the depletion of Mineral Reserves, the plant will process ore extracted from the underground mine at an average of around 6.5 Mtpa tailing off in Years 19 and 20. During years 10 to 14, previously stockpiled oxide ore will be re-handled to maintain mill feed at 8.0 Mtpa.

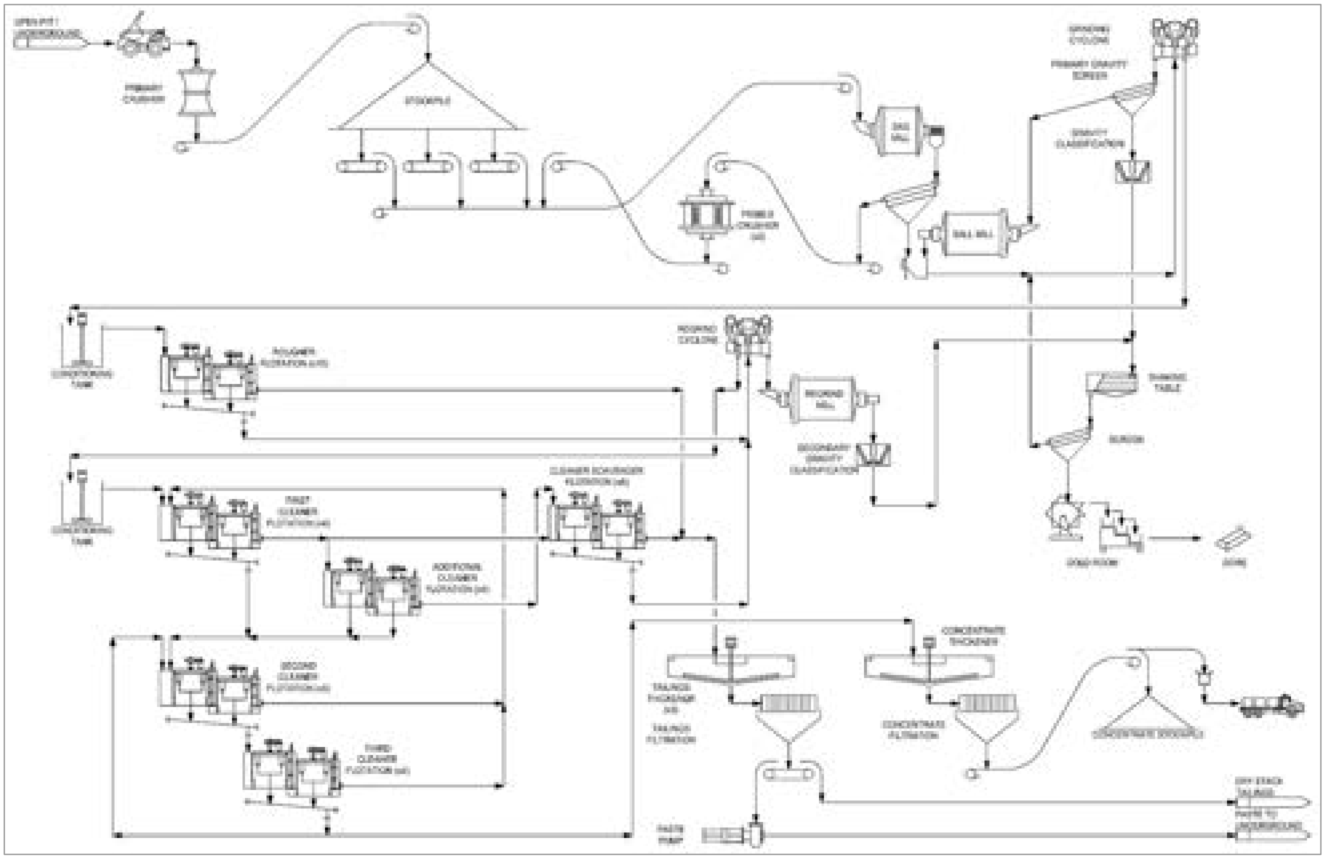
The plant will process the copper / gold ore at a projected LOM average head grade of 0.50% copper and 0.77 g/t gold. Anticipated LOM average payable recoveries are 87% for copper and 81% for gold. The mill will produce a flotation concentrate that contains an average of 26% copper and 27 g/t gold. Metallurgical tests have shown that the ore contains a small amount of palladium (Pd), which will be collected into the copper / gold concentrate during flotation.

The process plant design provides for a nominal 8.0 Mtpa of ore throughput. The Skouries simplified process flow diagram (PFD) is presented in Figure 1.6. While gravity classification, secondary gravity classification and gold room circuits have been designed, installation has been deferred pending confirmation of the need for gravity concentration to meet designed overall gold recoveries.

The unit operations comprise of:

- Primary crushing and crushed ore stockpile.
- SABC grinding and pebble crushing.
- Flotation and regrinding.
- Flotation concentrate and tailings thickening.
- Flotation concentrate filtering, storage and loadout.
- Tailings filtration, conveying and paste fill production.
- Reagent preparation and services.

Figure 1.6 Simplified process flow diagram



Source: Eldorado 2022.

1.15 Project infrastructure

1.15.1 Waste management

The principal waste streams generated from the Project are overburden and waste rock from open pit mining, waste rock from underground development, and tailings from mineral processing operations. Overburden and waste rock will be stored on surface; tailings will be used underground as paste backfill, with excess being stored on surface. The project mine plan and material balance have been developed such that the majority of overburden and waste rock are used for construction and the remaining material will be placed in a waste rock dump located upstream from the IEWMF. The waste management plan was developed to provide for surface storage capability of waste streams in the IEWMF within one watershed.

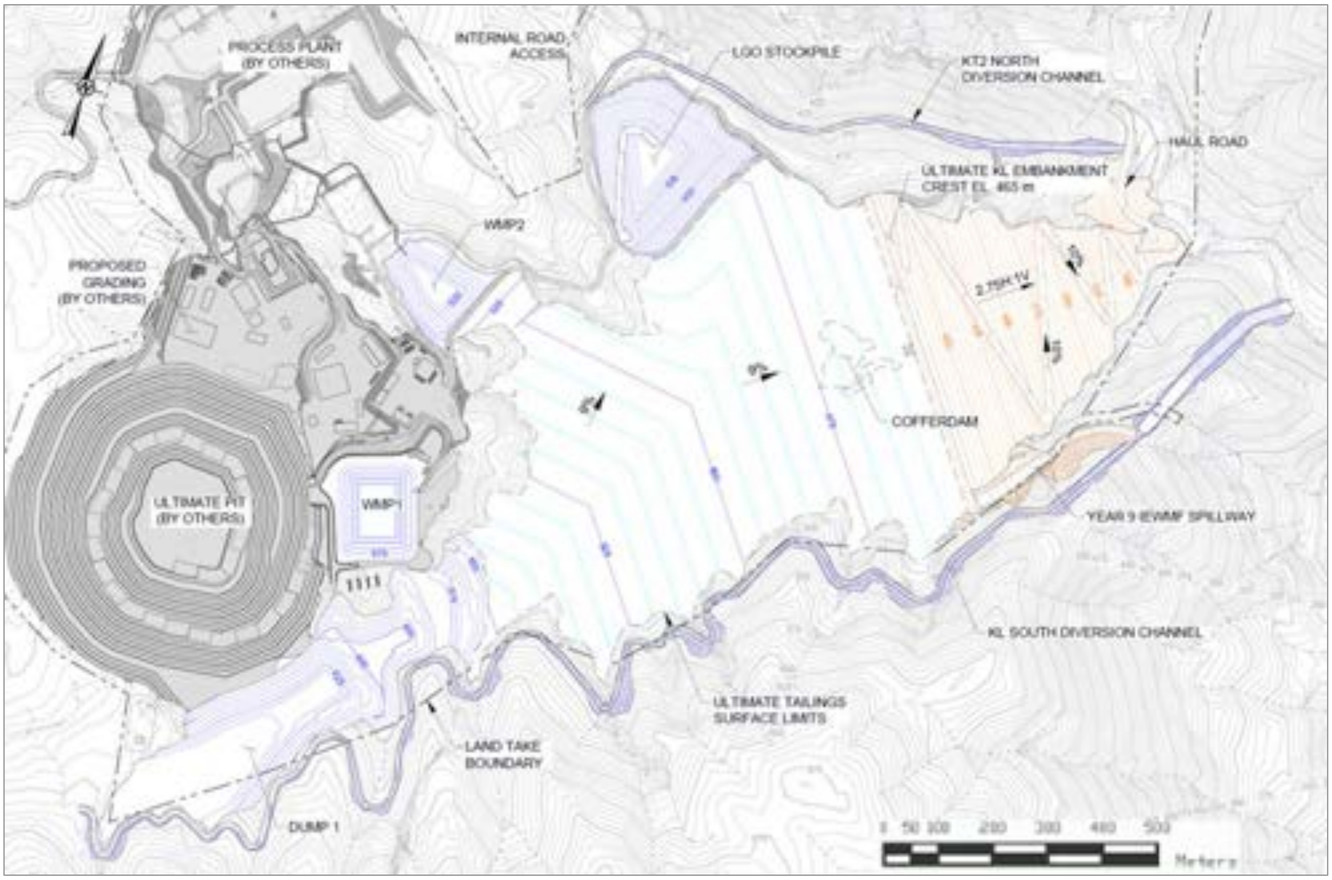
The compact footprint uses industry best practice to minimize disturbance to the natural environment, including surface water and groundwater impacts. The major waste management components are shown in Figure 1.7.

During Phase 1, most of the waste rock from mining activities will be used as a source of construction or borrow materials for the IEWMF embankment, cofferdam, contact water ponds (WMP1 and WMP2), LGO stockpile, process pads, and site infrastructure. Tailings will be deposited underground as paste backfill or above ground as filtered tailings in the IEWMF.

During Phase 2, tailings will be deposited underground as paste backfill and as filtered tailings in the open pit, allowing the IEWMF to be decommissioned and progressively reclaimed after Phase 1.

Surplus waste rock material will be stockpiled during Phase 1 in Dump No.1 and re-handled as closure cover materials for the IEWMF tailings surface and above the backfilled tailings surface in the open pit.

Figure 1.7 Phase 1 IEWMF site layout (Year 10)



Source: Golder 2022.

1.15.2 Water management

The water within the Project site can be classified as contact water and non-contact water. Non-contact water is surface water that is diverted around the mine facilities without being exposed to mine infrastructure using a series of diversion drainage ditches, and groundwater resulting from mine dewatering. Contact water includes groundwater and surface water that has been exposed to mine infrastructure and the process water.

A 3D groundwater flow model was developed for the Project utilizing site specific data from field investigations to estimate the rates of groundwater inflow to the open pit and underground mine development throughout the operations period; the extent of groundwater depressurization associated with the mine dewatering; and evaluate the potential reinjection rates for groundwater from non-contact and treated-water sources.

A site wide water balance (SWWB) model was developed for the Project to simulate water transfer throughout the entire mine operations.

The general results of the SWWB show that there will be enough water at site to supply mine water requirements. Excess contact water is expected to be generated especially in Phase 1, and will need to be managed with on-site storage capacity and water treatment prior to discharging to the groundwater aquifer through reinjection wells.

Water quality predictions were developed during the pre-feasibility design stage of the project. These predictions were not updated as part of the Feasibility Study. Recommendations for further geochemical characterization and refinement of source terms to be completed as part of the detailed design have been made.

Surplus contact water above the overall Project water demand will be transferred to a centralized water treatment plant (WTP) for treatment. The treatment process will consist of proven treatment techniques designed to treat water for discharge to the aquifer through reinjection wells.

Pumping systems will be used to transfer water through different water management facilities for re-use in the mine process and / or for treatment prior to discharge.

1.15.3 Transportation and logistics

The Project is well situated to take advantage of Greece's modern transportation network for shipment of construction and operations freight.

The main access road connects the process plant and mining area with the national road network. The major regional centre of Thessaloniki is approximately 80 km away and is accessed by highway EO 16. Thessaloniki has an international airport and one of Greece's largest seaports. It is linked to the rest of Greece by Greece's National Roadway, which has been extensively modernized in the last 20 years. Access to Europe and Turkey is provided by the highway and rail infrastructure.

1.15.4 Power supply

The Project site substation is fed from a new overhead, 6 km long 150 kilovolts (kV) transmission line connected to the national power grid. Hellas Gold signed an agreement with the Independent Electricity Transmission Operation for Greece (ADMIE) in 2015 that sets out the terms and conditions for connecting to the Greek power grid.

The high voltage substation constructed for the Project has a power capacity of 51 megawatts (MW).

1.16 Market studies and contracts

The Skouries processing plant will produce a gold-copper concentrate that is expected to be marketable to a large number of downstream smelters, refineries, traders, and sales agents.

In the future, the commissioning of the Olympias Phase III process facility and associated infrastructure, including a port, flotation plant, and waste management facilities, will allow for more cost-effective shipment of Skouries concentrate. As of the effective date of this report, the proposed Olympias Phase III developments have not been fully defined and a schedule for implementation has not been finalized. As such, this report considers that all concentrates over the life of the Skouries Project will be sold at competitive market rates to third parties.

No off-take agreements have been signed by Eldorado or Hellas Gold with potential concentrate off-takers at the time of preparation of this report; however, several indicative non-binding proposed term sheets have been received from European and global copper smelters.

1.16.1 Contracts

Construction of the Skouries Project has been ongoing since 2012. The Project is being executed using a standard engineering, procurement and construction management (EPCM) methodology. This Technical Report envisages several contracts relating to the initial development stages of the mining of the open pit and underground. In both cases, the contracts serve to implement the initial portions of the development, while allowing for a transition to owner operated mining as the Project matures.

1.17 Environmental

The EIS for the Kassandra Project includes the Skouries, Olympias, and Stratoni sites.

The EIS considers the potential impact on the local and regional environment as it relates to development, operation, and reclamation of the Project. The EIS was submitted in August 2010 and approved in July 2011. Hellas Gold plans to submit a revision to the EIS, incorporating changes to the tailings management plan.

1.17.1 Baseline studies

Three baseline studies relating to Skouries have been to support the EIS; the combination of these comprehensive studies has defined the ecological background conditions of the Project area and the wider study area.

The Project area is almost entirely forested, showing high density tree growth and flora and fauna diversity. In the wider area, there is small scale agricultural activity but no large cities or industrial infrastructure. There are some minor, naturally occurring elevated metals concentrations in the soils and pressure on water quality due to man-made activities in the way of unregulated landfills and wastewater discharge.

1.17.2 Impact assessment

The EIS concluded that, during construction and operations, there are site specific impacts; however, in general the impacts are considered reversible through the use of best practice during construction and operations, and proper decommissioning and reclamation at the end of the Project. In the wider study area, there are negligible impacts on the environment or surrounding villages.

Hellas Gold runs an extensive regional monitoring program covering air, water, noise, and vibration; this program will continue through the LOM and post closure.

There is currently high unemployment in the region, which is partly due to reduction in mining activities and lack of development. The Project will have a positive impact on employment in the region. Hellas Gold has committed to maximizing local employment.

The Ministry of Culture has performed archaeological investigations and identified two archaeological sites on the Skouries Project site; however, with design and relocation efforts the impacts are negligible.

Hellas Gold has an obligation to hire 90% of the workforce locally. Other than the commitment to maximize local employment there are no specific social obligations attached to the Project. However, Hellas Gold has a policy of assisting local communities that are stakeholders in its projects and will continue with this policy. In addition, a Stakeholder Engagement Plan (SEP) has been developed by Hellas Gold and the management of Eldorado Gold with the aim of providing a structure for communication and consultation with all identified stakeholders, taking into consideration Greek, European, and international law and best practice.

The closure and environmental rehabilitation program include comprehensive criteria for closure and reclamation of the Project. Plans have been developed, including for decommissioning, closure, and reclamation of the affected areas. At the end of reclamation, the site will be graded and returned to a morphology resembling the surrounding area, reclaimed with topsoil and reforested. Progressive reclamation will be undertaken.

Hellas Gold has provided the Mining & Industrial Minerals Directorate of the Ministry of the Environment (MDMOE) with a Letter of Guarantee for €57.5M in favour of the Greek State, as an assurance that the funds necessary for rehabilitation projects will be available. Hellas Gold has also provided insurance coverage in accordance with Presidential Decree 148/2009 (Government Gazette 190/A/29.9.2009) for environmental liability.

1.18 Project metrics

Key project metrics are summarized in Table 1.4.

Table 1.4 Project metrics

Item	Unit	Value
Total UG Ore	kt	87,519
Total OP Ore	kt	59,596
Total Ore Milled	kt	147,115
Gold Grade	g/t	0.77
Copper Grade	%	0.50
Gold recovered and payable	%	81
Copper recovered and payable	%	87
Revenue split by commodity	Gold	44.9
Revenue split by commodity	Copper	55.1

Source: AMC.

1.19 Capital and operating costs

All cost estimates are presented in US Dollars (US$).

The total Project capital cost includes the remaining cost to complete the Project construction until commercial production is achieved ('initial capital'), and subsequent sustaining capital costs spread out over the remaining 20 years of the mine life.

Capital costs are summarized in Table 1.5. Sunk costs to the end of 2021 are not included in the capital cost estimate. The accuracies of the cost estimates are consistent with the standards outlined by the Association for the Advancement of Cost Engineering (AACE). The cost estimate is a feasibility-level estimate categorized as AACE Class 3.

Direct costs were developed from a combination of budget quotes, material take-offs, existing contracts, Project-specific references, and historical benchmarks. Indirect and owners' costs were estimated using a combination of existing commitments, calculated project requirements, and historical benchmarks. Contingency was applied to each cost item in the estimate, based on the level of engineering definition and reliability of its unit rates.

Table 1.5 Capital cost summary

Area	Cost (US$ M)
Development capital (pre-production)	
Underground Phase 1 development	123
Open pit	99
Process and infrastructure	390
IEWMF and water management	158
Power Line	9
Owners Cost	66
Total pre-production development capital	**845**
Development capital (Phase 2 underground)	**172**
Sustaining capital	
Underground	569
Open pit	21
Process and infrastructure	190
IEWMF and water management	81
Sub-total sustaining capital	**861**
Ramp up period (costs net of production)	-19
Addback spares	5
Total sustaining capital	**847**
Total capital (development and sustaining capital)	**1,863**

Note: Numbers may not compute exactly due to rounding.

The operating cost estimate provides the LOM operating costs associated with mining, the process plant, tailings filtration plant, backfill plant, WTP, water systems, and general and administrative (G&A) facilities. The operating cost includes all on-site costs from mining through to the production of copper concentrate, including tailings filtration, tailings compaction, and paste production.

The operating cost estimate has been developed on a year-by-year basis in accordance with Eldorado's envisaged operations and mine plan. The estimated total costs by cost centre and cost category are presented in Table 1.6.

A €/US$ exchange rate of 1.2 was used for the preparation of the operating costs. The cost per tonne averages for the open pit and underground mining are calculated based on the tonnages mined for the production years of those phases. The non-mining cost centre expenditures are averaged based on the process plant ore throughput for the production years. The operating cost excludes cost associated with pre-production years.

Table 1.6 Operating costs

Cost centre	Production years total cost (US$)	Production years cost per tonne of production ore (US$/t)
Open pit mining	244,815,387	4.24*
Underground mining	1,681,025,005	19.32*
Process plant	1,247,247,282	8.54
Tailings filtration plant	314,300,479	2.15
Backfill plant	27,506,378	0.19
Water system	20,007,884	0.14
G&A	409,139,670	2.80
Subtotal mining	**1,925,840,391**	**13.18**
Subtotal non-mining	**2,018,201,653**	**13.81**
Total	**3,944,042,045**	**26.99**

1.20 Economic analysis

The economic analysis is based on the Mineral Reserves production schedule, mill feed, metal recoveries and the capital and operating costs. The Project case metal prices used in the economic model are US$1,500/oz Au and US$3.85/lb Cu. The economic model was also evaluated at the respective Mineral Reserve gold and copper prices of US$1,300/oz and US$2.75/lb. The model makes use of a first principles build-up in Euros, with values then converted to US$. All reporting is in US$.

The after-tax cash flow analysis shows that the Skouries Project provides a robust return on the remaining capital to complete the Project scope and bring the Project into commercial production. An internal rate of return (IRR) of 19% on an after-tax basis is achieved with the project case metal prices of US$1,500/oz Au and US$3.85/lb Cu. Using those metal prices, the net present value (NPV) of the Project is estimated to be US$1,273M using a discount rate of 5%, with a payback of the remaining capital expenditure achieved in 3.7 years from the start of commercial production.

A test of economic extraction for the Skouries Mineral Reserves is demonstrated by means of a sensitivity analysis (see below). At the Mineral Reserve metal prices of US$1,300/oz Au and US$2.75/lb Cu the Project shows positive economics. The after-tax IRR is 9.8% and the NPV is estimated to be US$354M using a 5% discount rate, with a calculated payback period of 8.1 years from start of Commercial Production.

Corporate income tax rates in Greece are 22% of net earnings. Income from operations can be offset by operating costs and by depreciation of capitalized items according to a schedule of depreciation based on the type of asset.

1.20.1 Sensitivity analysis

The economic model was subjected to a sensitivity analysis to determine the effects of changing metal prices, operating costs and capital costs on the projected financial returns, as shown in Table 1.7 and Table 1.8.

The sensitivity analysis shows that the Project is most sensitive to metal prices, followed by operating costs and then capital costs. The copper concentrate grade is the least sensitive. The sensitivity ranges show that the Project is also robust when evaluated using lower metal price assumptions, or higher operating and capital costs. Positive cash flows and positive NPV are maintained at metal prices of US$1,125/oz Au and US$2.89/lb Cu (except for when the NPV is discounted at 8%), operating and capital cost increased by 25% individually, or concentrate grade reduced by 25%.

Table 1.7 Metal price sensitivity analysis

		Sensitivity ranges					
Parameters	**Units**	**Reserve case**	**-25%**	**-12.5%**	**Project case**	**+12.5%**	**+25%**
Gold price	US$/oz	1,300.00	1,125.00	1,312.50	**1,500.00**	1,687.50	1,875.00
Copper price	US$/lb	2.75	2.89	3.37	**3.85**	4.33	4.81
Results (after tax)							
NPV 0%	US$M	1,104	834	1,818	**2,726**	3,596	4,451
NPV 5%	US$M	354	195	755	**1,273**	1,772	2,261
NPV 8%	US$M	105	-16	401	**788**	1,161	1,526
IRR%	%	9.8	7.7%	14.1%	**19.0%**	23.4%	27.3%
Payback period	years	8.1	8.8	5.3	**3.7**	3.1	2.7
Taxation	US$M	253	209	417	**667**	913	1,154
Royalties	US$M	87	79	120	**193**	308	444

Table 1.8 Capital and operating costs sensitivity analysis

		Sensitivity ranges				
Parameter	**Units**	**-25%**	**-12.5%**	**Project case**	**+12.5%**	**+25%**
LOM capital costs	US$M	1,397	1,630	**1,863**	2,096	2,329
Results (after tax)						
NPV 0%	US$M	3,100	2,913	**2,726**	2,538	2,349
NPV 5%	US$M	1,578	1,426	**1,273**	1,121	968
NPV 8%	US$M	1,064	926	**788**	651	512
IRR	%	26.4	22.3	**19.0%**	16.3	14.1
		Sensitivity ranges				
Parameter	**Units**	**-25%**	**-12.5%**	**Project case**	**+12.5%**	**+25%**
LOM operating costs	$/t ore	20.24	23.62	**26.99**	30.36	33.74
Results (after tax)						
NPV 0%	US$M	3,495	3,110	**2,726**	2,338	1,950
NPV 5%	US$M	1,696	1,484	**1,273**	1,061	849
NPV 8%	US$M	1,097	943	**788**	634	478
IRR	%	22.4	20.8	19.0	17.2	15.3

1.21 Other relevant data and information

The Skouries Project has been under construction since 2012 and the capital costs incurred to the end of 2021 are sunk costs and are not included in the capital cost estimate. The sunk costs are used in the economic evaluation as they form a portion of depreciable assets used to estimate net earnings and tax payable.

1.22 Conclusions

It is concluded that the Skouries work completed to date, including exploration, site development and study work leading to current Mineral Resource and Mineral Reserve estimates, has demonstrated the strong technical and economic viability of the Project.

1.22.1 Mineral Resources and Mineral Reserves

It is the opinion of the QPs that the information and analysis described in this report are sufficient for reporting Mineral Resources and Mineral Reserves.

The assessment shows that the orebody is open at depth. This is considered an opportunity for the Project that may ultimately result in an increase in Mineral Resources and, possibly, Mineral Reserves.

1.22.2 Mining

The Skouries Project is designed as a two-phase mining operation. Phase 1 consists of a combined open pit and underground mine, operating over nine years. Phase 2 consists of mining from the underground mine only, for an additional 11 years. The total ore producing LOM is 20 years.

Phase 1 mill feed is 8.0 Mtpa, consisting of a nominal 5.5 Mtpa from the open pit mine combined with approximately 2.5 Mtpa from the underground mine. Phase 2 mine production, from Year 10 to the end of the LOM, is provided from the underground mine at 6.5 Mtpa, with supplementary ore coming from the Phase 1 open pit low grade ore stockpile in Years 10 to 14, allowing mill feed to be maintained at or close to 8 Mtpa, for that period.

Open pit mining will be by conventional truck-shovel operation. SLOS has been confirmed as the most appropriate underground mining method. Production stopes will be backfilled with cemented paste fill. Shaft conveyance of ore will be utilized in Phase 2 to facilitate achievement of the projected production rate.

1.22.3 Metallurgical recovery

Significant metallurgical testwork and analysis has been completed to confirm the process designs and substantiate projected recoveries. The QP has reviewed and validated historical data obtained from Eldorado and has a high degree of confidence in the process designs and the stated recoveries.

1.22.4 Infrastructure

Tailings will be used underground as paste backfill, with excess being stored on surface. Surplus waste rock material will be stockpiled during Phase 1 in Dump No.1 and re-handled as closure cover materials for the IEWMF tailings surface and above the backfilled tailings surface in the open pit.

The principal waste streams generated from the Project are overburden and waste rock from open pit mining, waste rock from underground development, and tailings from mineral processing operations. Overburden and waste rock will be stored on surface; tailings will be used underground as paste backfill, with excess being stored on surface. The project mine plan and material balance have been developed such that the majority of overburden and waste rock are used for construction and the remaining material will be placed in a waste rock dump located upstream from the IEWMF.

Non-contact water is groundwater and surface water that has not been exposed to mine infrastructure. Contact water includes groundwater and surface water that has been exposed to mine infrastructure, as well as process water. Excess contact water is expected to be generated especially in Phase 1 and will need to be managed with on-site storage capacity and water treatment prior to discharge the groundwater aquifer through reinjection wells.

The main access road connects the process plant and mining area with the national road network. The major regional centre of Thessaloniki is approximately 80 km away and is accessed by highway EO 16. Thessaloniki has an international airport and one of Greece's largest seaports.

The Skouries Project site substation is fed from a new overhead 6 km long 150 kV transmission line connected to the national power grid. The high voltage substation constructed for the Skouries Project has a power capacity of 51 MW.

1.22.5 Metal sales

The Skouries processing plant will produce a gold-copper concentrate that is expected to be marketable to a large number of downstream smelters, refineries, traders, and sales agents.

1.22.6 Capital and operating costs and financial model

The accuracies of the cost estimates are consistent with the standards outlined by the AACE. The cost estimate is a feasibility-level estimate categorized as AACE Class 3.

Direct costs were developed from a combination of budget quotes, material take-offs, existing contracts, Project-specific references, and historical benchmarks. Indirect and owners' costs were estimated using a combination of existing commitments, calculated project requirements, and historical benchmarks. Contingency was applied to each cost item in the estimate, based on the level of engineering definition and reliability of its unit rates.

The capital cost estimate does not include sunk costs. Total capital cost is estimated to be US$1,863M and total operating cost over the LOM is estimated to be US$3,944M.

The after-tax cash flow analysis shows that the Skouries Project provides a robust return on the remaining capital to complete the Project scope and bring the Project into commercial production. An IRR of 19% on an after-tax basis is achieved with the project case metal prices of US$1,500/oz Au and US$3.85/lb Cu. Using those metal prices, the NPV of the Project is estimated to be US$1,273M using a discount rate of 5%, with a payback of the remaining capital expenditure achieved in 3.7 years from the start of commercial production.

A test of economic extraction for the Skouries Mineral Reserves is demonstrated by means at this sensitivity analysis. At the Mineral Reserve metal prices of US$1,300/oz Au and US$2.75/lb Cu the Project shows positive economics. The after-tax IRR is 9.8% and the NPV is estimated to be US$354M using a 5% discount rate, with a calculated payback period of 8.1 years from start of Commercial Production.

The sensitivity analysis shows that the Project is most sensitive to the metal prices, followed by operating costs and then capital costs.

1.22.7 Permitting

Hellas Gold has obtained the critical permits required to proceed with the Project. An IA which amends the 2003 Transfer Agreement and provides a modernized legal and financial framework to allow for the advancement of Eldorado's investment in the Kassandra Mines was ratified in early 2021. After the 2019 Greek Parliamentary elections, when Eldorado initiated talks with the newly established government, outstanding routine permits were released.

1.22.8 Environmental

The EIS for the Kassandra Project concluded that, during construction and operations, there are site specific impacts. In general, however, the impacts are considered reversible through the use of best practices during construction and operations, and appropriate decommissioning and reclamation at the end of the Project. In the wider area, there are negligible impacts on the environment or surrounding villages.

Hellas Gold has provided the MDMOE with a Letter of Guarantee for €57.5M in favour of the Greek State, as an assurance that the funds necessary for rehabilitation projects will be available. Hellas Gold has also provided insurance coverage in accordance with Presidential Decree 148/2009 (Government Gazette 190/A/29.9.2009) for environmental liability.

1.22.9 Site investigations

The report QPs recommend and support the work that Hellas Gold has indicated it will continue to complete on site during detailed design, development,t and operations, including:

- Geotechnical investigations of the shaft and ventilation raises.
- Completion of exploration and studies aimed at increasing available Mineral Resources through expansion of the underground mine at depth.
- Optimization of development to enable test stoping to be completed as soon as possible.
- Optimization of the mine plan with respect to owner supplied and contractor fleet for waste haulage.
- Filtered tailings and LGO fills, in-place densities.
- Supplemental geochemical testing and source terms refinement.
- Treatability studies to confirm chemical doses, residuals produced, and effluent quality.
- Tailings testing (filtration and geotechnical properties).
- Supplemental geotechnical investigation for the WMP1, WMP2, IEWMF, surface water diversions and LGO stockpile foundations.
- Assessment of existing surface water channels.

1.23 Recommendations

Study work and site development and construction completed to date have provided a strong technical and economic basis for proceeding with full development of the Skouries Project. It is recommended to continue with that development while undertaking the work described below. The activities involve optimization and confirmatory work that will not affect the development schedule as presented. The work is to be completed by Skouries operations personnel and is recommended to be undertaken as part of future design, construction, and operation activities. The majority of the recommended studies have been included into the capital cost estimates for the Feasibility Study, however an additional US$2.7M is estimated to complete additional studies including geotechnical, geochemical and tailings testwork.

1.23.1 Geology

When in favourable underground locations to do so, diamond drilling should be carried out to refine the knowledge of the deposit and assist in design and planning. In addition, results of drilling indicate that the orebody is open at depth, with potential for Inferred Resources to be converted to Indicated Resources through further exploration. This is considered an opportunity for the Project, and further exploration at depth should be completed during operations.

1.24 Mining

The QP makes the following recommendations for the open pit mine:

- Perform geotechnical assessments on the Open Pit and Capping Rock Dump No.1 areas up to a feasibility (FS) level to determine interaction risks.
- Rainfall impacts on pit and waste roads to be recorded during pre-production years to better inform maintenance costs and productivity differences.

The QP makes the following recommendations for the underground mine:

- In situ stress measurements to be taken once development has reached more competent rock in the general area in which test stoping has been proposed.
- Maintain dimensions of 15 m wide x 30 m long as the primary stope design basis until actual stope conditions are experienced and understood.
- Maintain recommended ground support design parameters, with ultimate proving to be achieved in the field.
- Perform confirmatory testwork on Skouries tailings for the backfill paste.
- Perform rheology testwork on tailings to verify the expected paste friction losses for the system.

1.24.1 Engineering

Engineering for a large portion of the Project has been done at a detailed level. Further detailed engineering and procurement for the underground mine, filtration plant, IEWMF and water management is required for completion of Project development. It is recommended that these activities be undertaken in parallel with development and construction work on site so that Project implementation is not delayed.

1.24.2 IEWMF and ancillary facilities

Engineering and design for the IEWMF and ancillary facilities (WMP1, WMP2, LGO stockpile, cofferdam, and waste Dump No. 1) in support of the Feasibility Study has been advanced to support this Technical Report; however, approximately 90% of the supporting documents have only been completed in draft format (with final completion pending Eldorado and third-party review), and the remainder are under development (with scheduled completion by February 2022).

Subsequent to completion of the Feasibility Study, more in-depth third-party reviews and risk assessments are recommended, including a FMEA risk assessment of the feasibility-level design, a formal constructability review, safety in design review, and a review by a geotechnical or Independent Technical Review Board (ITRB).

1.24.3 Water management design and water balance

Excess contact water is expected at the project site, especially during Phase 1 of the project. A management strategy including temporary on-site storage, and treatment and discharge is included in the design to mitigate risk associated with excess contact water. However, management of excess contact water remains a potential risk for the Project. The following should be evaluated during the next project stage to further reduce the risk associated with excess contact water:

- Opportunities to increase diversion of non-contact water groundwater as part of mine dewatering.
- Opportunities to increase on-site water storage capacity.
- Opportunities to optimize alignment of some water management structures, such as IEWMF spillways (e.g., move spillways to north side of the IEWMF).

1.24.4 WTP

There are other trade-off evaluations and optimizations that should be carried out to potentially reduce costs and improve operational efficiency.

1.24.5 Pipeline and pumping systems

After finalizing the water balance modelling and tailings management strategy, the following can be further reviewed to optimize the pipeline and pumping scope at the detailed design stage:

- Selection of materials of construction for buffer tanks (steel versus concrete).
- Selection of pump types for specific services (vertical turbine pump versus submersible pump).
- Optimization of pipeline sizing, routing, and alignment.
- Optimization of electrical room design (use common electrical room to the fullest extent and modularize / fabricate electrical rooms off site to achieve on-site “plug-and-play” strategy).

Contents

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Abbreviations and acronyms

Abbreviations & acronyms	Description
>	Greater than
<	Less than
$	United States dollar
$/lb	US dollar per pound
$/oz	US dollar per ounce
$/t	US dollar per tonne
€	Euro
%	Percentage
°	Degree
°C	Degrees Celsius
µm	Micrometre
3D	Three-dimensional
AACE	Association for the Advancement of Cost Engineering
AAS	Atomic absorption spectroscopy
Acme	Acme Labs
ADL	Analytical detection limit
ADMIE	Independent Electricity Transmission Operation for Greece
Aktor	Aktor Enterprises Limited
AMC	AMC Mining Consultants (Canada) Ltd.
ANFO	Ammonium Nitrate fuel oil
AONB	Area of Outstanding Natural Beauty
ARO	Asset retirement obligation
Au	Gold
AuEq	Gold equivalent
BH	Bench Height
BFA	Bench Face Angle
BV	Bureau Veritas
BW	Berm Width
Capex	Capital expenditure
Cementation	Cementation Canada Inc.
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	Centimetre
CP	Competent Person
CRM	Certified Reference Material
CSA	Canadian Securities Administrators
Cu	Copper
Cu_2O	Copper oxide
CV	Coefficient of variance
DCS	Distributed control system
DEE	Diesel engine exhaust
DMP1	Waste Rock Dump 1
dmt	Dry metric tonne
E	East
EGL	European Goldfields Limited

Abbreviations & acronyms	Description
EIA	Environmental Impact Assessment
EIS	Environmental Impact Study
Eldorado	Eldorado Gold Corporation
EMSG	Empirical modified stability graph
EPCM	engineering, procurement and construction management
FAR	Fresh air raise
Feasibility Study	Skouries Feasibility Study
FS	Feasibility Study
Fluor	Fluor Mining and Metals
G&A	General and Administration
g	Gram
g/t	Grams per tonne
GAL	Golder Associates Ltd.
GAUSA	Golder Associates USA Inc
h	Hour
ha	Hectare
Hellas Gold	Hellas Gold SA
HDPE	High density polyethylene
HGO	High grade ore
HO	High oxide
HS	High sulphides
IA	Investment Agreement
ICP-ES	Inductively Coupled Plasma Emission Spectroscopy
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
IDF	Inflow design flood
IEWMF	Integrated extractive waste management facility
IRA	Inter-Ramp Angle
IRAR	Internal return air raise
IRR	Internal rate of return
ISO	International Organization for Standardization
ITH	In-the-hole percussive hammer
ITRB	Independent Technical Review Board
JMD	Joint Management Decision
k	Kilo (thousand)
Kassandra Project	Kassandra Mines Mineral Deposits Project
kg	Kilogram
KL	Karatza Lakkos
KLE	Karatza Lakkos Embankment
km	Kilometre
km^2	Squared kilometre
km/h	Kilometres per hour
koz	Thousand ounces
kPa	Kilopascal
kt	Thousand tonnes
kV	Kilovolt
kVA	Kilovolt-amp

Abbreviations & acronyms	Description
kW	Kilowatt
kWh	Kilowatt hour
kWh/t	Kilowatt hour per tonne
L	Level
lb	Pound
LGO	Low-grade ore
LGOE	Low-grade ore stockpile embankment
LGOS	Low-grade ore stockpile
LHD	Load haul dump
LO	Low oxide
LOM	Life-of-mine
LS	Low sulphides
LV	Low-voltage
M	Million
m	Metre
m^2	Square metre
m^3	Cubic metre
m^3/h	Cubic metre per hour
m^3/s	Cubic metre per second
Ma	Million years / mega annum
masl	metres above sea level
MDMOE	Mining & Industrial Minerals Directorate of the Ministry of the Environment
MIBC	Methyl IsoButyl Carbinol
min	Minute (time)
MineFill	MineFill Services Inc
mm	Millimetre
MP	Mining Plus
MO	Medium oxide
MOE	Ministry of Environment
Moz	Million ounces
Mt	Million tonnes
Mtpa	Million tonnes per annum
MVA	Mega volt amp
MW	Megawatt
N	North
NaHS	Sodium hydrosulfide
NE	North-east
NI 43-101	National Instrument 43-101
NN	Nearest neighbour
Non-PAG	non-potentially acid generating
NPV	Net present value
NSR	Net smelter return
NW	North-west
OK	Ordinary kriging
OMC	Orway Mineral Consultants
OP	Open pit

Abbreviations & acronyms	Description
Opex	Operating expenditure
oz	Troy ounce
PACK	Probability assisted constrained kriging
PAG	Potentially acid generating
Pd	Palladium
PFD	Process flow diagram
PFS	Pre-Feasibility Study
Placer	Placer Development
PPC	Public Power Corporation
ppm	Parts per million
Property	Skouries Property
QA/QC	Quality assurance / quality control
QP	Qualified Person
RAR	Return air raise
RMR	Rock Mass Rating
RMT	Remote mining technology
ROM	Run-of-mine
RPD	Relative paired difference
RPEEE	Reasonable Prospects for Eventual Economic Extraction
S	South
SABC	SAG mill - ball mill - pebble crusher
SAG	Semi-autogenous grinding
SD	Standard deviation
SE	South-east
SEP	Stakeholder Engagement Plan
SGS	SGS S.A.
SIPX	Sodium IsoPropyl Xanthate
Skouries Project	Skouries project
SLC	Sub-level caving
SLOS	Sub-level open stoping
SMMP	Serbo-Macedonian Metallogenic Province
SMU	Selective mining unit
SPH	Spherical
SRCP	Steel wire reinforced pipe
SRK	SRK Consulting
SW	South-west
SWCC	Soil-water characteristics curve
SWWB	Site wide water balance
t	Metric tonne
t/h	Tonne per hour
t/m^2	Tonne per square metre
t/m^3	Tonne per cubic metre
TMF	Tailings Management Facility
tonne	Tonne = 1,000 kg
tpa	Tonnes per annum
tpd	Tonnes per day

Abbreviations & acronyms	Description
TVX	TVX Gold Incorporated
UCS	Unconfined compressive strength
UG	Underground
US$	United States dollar
V	Volt
VAT	Value added taxation
VFD	Variable frequency drive
W	West
w/w	Weight for weight
WBM	Water balance model
WBS	Work breakdown structure
WMP	Water management pond
wmt	Wet metric tonne
WTP	Water treatment plant

2 Introduction

2.1 General and terms of reference

This Technical Report on the Skouries Property (Property) located in Halkidiki Peninsula in northern Greece has been prepared by AMC Mining Consultants (Canada) Ltd. (AMC) of Vancouver, Canada on behalf of Eldorado Gold Corporation (Eldorado) of Vancouver, Canada. It has been prepared to a standard which is in accordance with the requirements of National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects, of the Canadian Securities Administrators (CSA).

The report is an update to the "Technical Report, Skouries Project, Greece prepared by Eldorado with an effective date of 1 January 2018". The main purpose of the Report is to report the results of the "Skouries Feasibility Study" (Feasibility Study) and to support the Eldorado annual statement of Mineral Resources and Mineral Reserves.

The Property is located within the Kassandra Mines complex, located on the Halkidiki Peninsula of northern Greece. The complex comprises a group of mining and exploration concessions, covering 317 squared kilometres (km^2), located approximately 100 kilometres (km) east of Thessaloniki of which the Property is part. The Properties within the complex include Olympias Mine currently in production, Stratoni Mine on care and maintenance, and the Skouries copper-gold porphyry deposit under development but currently on care and maintenance. This Report is specific to the Skouries Property.

2.2 The Issuer

Eldorado is an international mid-tier gold mining company based in Vancouver, British Columbia, with operations and projects in Greece, Turkey, and Canada. Eldorado, through its 95% owned subsidiary Hellas Gold SA (Hellas Gold), owns the Kassandra Mines Complex on the Halkidiki Peninsula, as well as the 100% owned Perama advanced property, also in Greece. Hellas Gold was acquired as part of the acquisition of European Goldfields Limited (EGL) completed in February 2012.

Eldorado is listed on the Toronto Stock Exchange as "ELD" and the New York Stock Exchange as "EGO".

2.3 Report authors

The names and details of persons who prepared this Report, or who have assisted the Qualified Persons (QPs) in its preparation, are listed in Table 2.1. The QPs meet the requirements of independence as defined in NI 43-101.

Table 2.1 Persons who prepared or assisted in preparation of this Technical Report

QPs responsible for the preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of Eldorado**	**Date of last site visit**	**Professional designation**	**Sections of Report**
Mr JM Shannon	General Manager / Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	28 May 2019	P.Geo.	2 - 12, 14, 23, and parts of 1, 25, 26, and 27
Mr G Methven	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng.	20, 22, 24 and parts of 1, 15 - 16, 25, 26, and 27
Mr J Battista	Principal Mining Engineer	Mining Plus	Yes	15-16 September 2021	MAusIMM (CP)	Parts of 1, 15, 16, 25 and 26
Mr M Molavi	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	7-9 September 2016	P.Eng.	Parts 1, 16, 18, 25, and 26
Mr D Maeda	Engineering Manager	Fluor Canada Ltd.	Yes	3-5 November 2021	P.Eng.	21 and parts of 1, 18, 25 and 26
Mr R Kiel	Director, Civil Engineer	Golder Associates USA Inc.	Yes	28 June to 1 July 2021	P.E.	Parts of 1, 18, 25 and 26
Mr P Chiaramello	Senior Water Resources Engineer	Golder Associates Ltd.	Yes	No visit	P.Eng.	Parts of 1, 18.4, 25 and 26
Mr R Chesher	General Manager / Principal Mining Consultant	AMC Consultants Pty Ltd	Yes	28 May 2019	FAusIMM	13, 17, 19 and parts of 1, 25 and 26
Other Experts who have assisted the QPs						
Expert	**Position**	**Employer**	**Independent of Eldorado**	**Visited site**	**Sections of Report**	
Ms D Nussipakynova	Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	No	14	
Mr HA Smith	Senior Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	28 May 2019	Peer reviewer	
Ms K Zunica	Senior Geologist	AMC Consultants (UK) Limited.	Yes	No	11	
Mr S Gregerson	Manager of Engineering Studies	Eldorado Gold Corporation	No	Last in September 2021	21 and 22	
Mr C Keogh	Director Underground Mining	Eldorado Gold Corporation	No	Yes	15 and 16	
Mr S McKinley	Manager, Mine Geology & Reconciliation	Eldorado Gold Corporation	No	Yes	3 - 12	
Mr P Zimmerman	Director of Engineering	Eldorado Gold Corporation	No	Yes	All	

Note shared responsibilities as follows:

- Section 1: Responsibility taken by QP responsible for relevant discipline as per body of report.
- Sections 15 and 16: G. Methven for underground, J. Battista for open pit, M. Molavi for underground infrastructure.
- Section 18: D. Maeda for site infrastructure, R. Keil and P. Chiaramello for tailings and water issues.
- Section 25 and 26: Responsibility taken by QP responsible for relevant discipline as per body of report.

2.4 Sources of information

A principal source of information for this Report is the Technical Report titled 'Technical Report, Skouries Project Greece, effective date: 1 January 2018' (2018 Eldorado Technical Report). A prior report by EGL filed in 2011 (2011 EGL Technical Report) was also referenced.

Parties additional to AMC who have supplied information that was used for the development of this Report include Fluor Mining and Metals (Fluor), Mining Plus (MP), MineFill Services Inc (MineFill), Cementation Canada Inc. (Cementation), and Golder Associates Ltd. (GAL).

2.5 Other

An inspection of the Property was carried out by J.M. Shannon, H.A. Smith, and R. Chesher, all of AMC, on 28 May 2019. This inspection included review of representative drill core, data collection facilities, mine site including open pit area, partially completed processing plant, and general plant site, including dry stack tailings area under construction and flood control system.

Currency used throughout this report is US$, unless stated otherwise. Where applicable, the conversion factor shown in Table 2.2 has been used, or as otherwise stated.

Table 2.2 Exchange rates

Currency code	Currency name	Exchange rate
US$	United States Dollar	US$1.00 = US$1.00
€	Euro	€1.00 = US$1.20

This Report has an effective date of 22 January 2022.

3 Reliance on other experts

The QPs have relied, in respect of legal aspects, upon the work of the Experts listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant sections of the Report.

The following disclosure is made in respect of this Expert:

- Ministry of Development and Natural Resources, Mining and Industrial Minerals Directorate, Departments A & D.

Report, opinion, or statement relied upon:

- Appendix III - Table 1, list of concessions, dated 4 December 2003, attached to deed No. 22138/2003, Athens, 12 December 2003.

Extent of reliance:

- Full reliance.

Portion of Report to which disclaimer applies:

- Section 4.2 Land Tenure.

The following disclosure is made in respect of this Expert:

- ENVECO S.A., Environmental Protection, Management and Economy Consultants.

Report, opinion, or statement relied upon:

- ENVECO S.A., 2010, Environmental Impact Assessment (EIA) of the Mining-Metallurgical Facilities of Company Hellas Gold in Halkidiki.

Extent of reliance:

- Full reliance.

Portion of Report to which disclaimer applies:

- Section 20, except for Section 20.3 which is supplied by Eldorado.

The following disclosure is made in respect of this Expert:

- Philip Yee, Chief Financial Officer, Eldorado.

Report, opinion, or statement relied upon:

- Calculation of Corporate taxes and depreciation provided in the Skouries Economic model (SKR-006-Skouries 43-101 Overall Financial Model).

Extent of reliance:

- Full reliance.

Portion of Report to which disclaimer applies:

- Section 22.7.

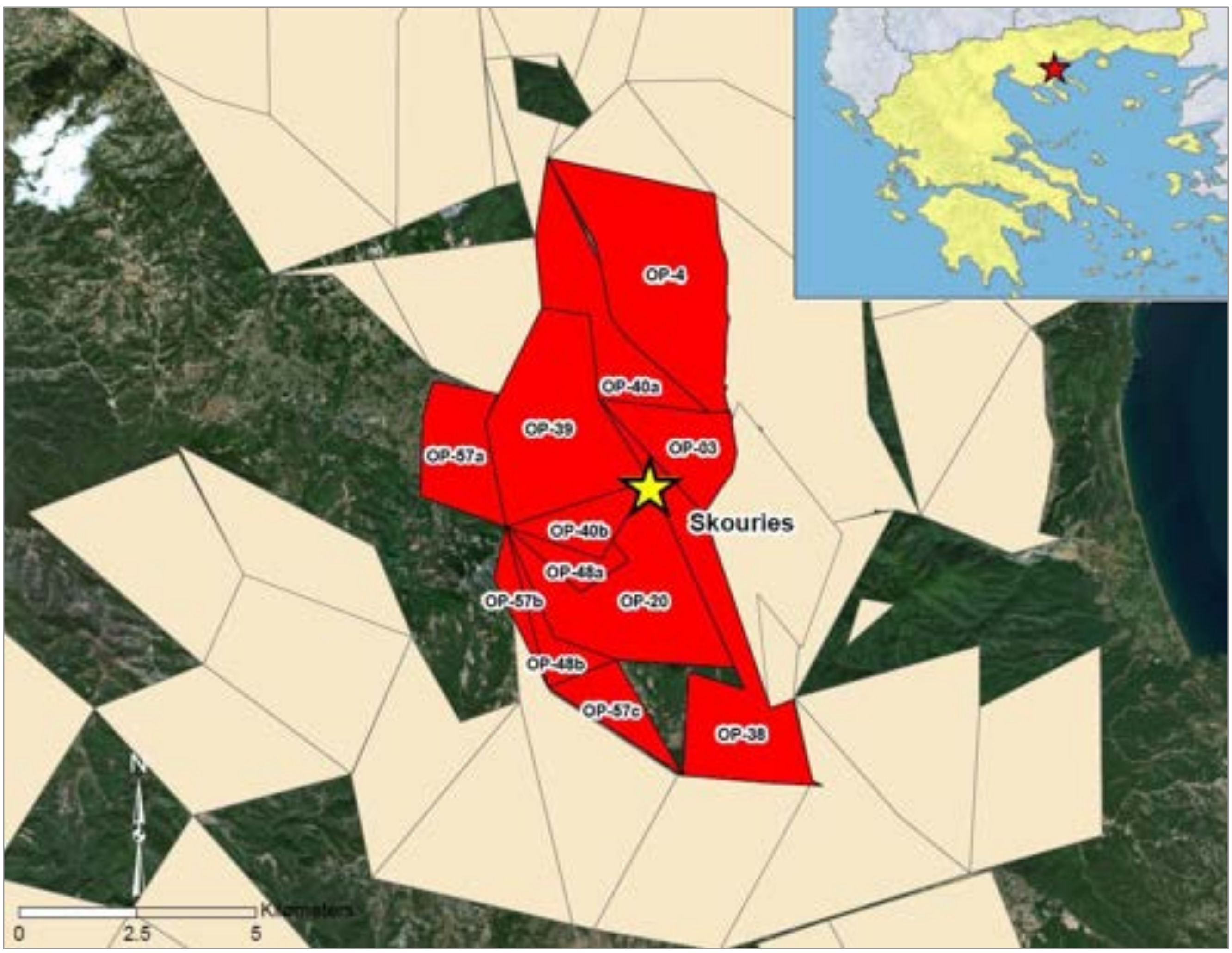
4 Property description and location

4.1 Property location

The Property is located within the Kassandra Mines complex found within the Halkidiki Peninsula of the Central Macedonia Province in northern Greece. The complex is comprised of a group of mining and exploration concessions, covering 317 km^2, located approximately 100 km east of Thessaloniki. The concessions include the Olympias Mine, and the Madem Lakkos and Mavres Petres Mines that are collectively known as Stratoni. The Olympias and Mavres Petres mines are currently in production and the Skouries copper-gold porphyry deposit is under development.

The Property is situated at an elevation range of 350 metres above sea level (masl) to 620 masl near the village of Megali Panagia in the prefecture of Halkidiki, northern Greece. It is approximately 7.2 km from the road connecting the villages of Megali Panagia and Palaiochori. The area is centred on co-ordinates 4,745,300 E and 4,481,400 N of the Greek Reference System EGSA '87, approximately at Latitude 40°29' and Longitude 23°42'. The location is classified according to Greek Seismic Code (NEAK 2000, modified in 2003) as Zone II. Figure 4.1 shows the concessions of the Kassandra Mines complex in beige, with the Skouries concessions shaded in red.

Figure 4.1 Kassandra Mines concessions and Skouries property



Source: Eldorado 2022.

The concessions forming the Property are listed in Table 4.1. With the exception of two concessions (OP03 and OP04), all concessions are held 100% by Hellas Gold or its wholly owned subsidiary Macedonia Copper Mines. Eldorado holds a 100% controlling interest in Hellas Gold.

4.2 Land tenure

The Property is located within concession numbers OP03, OP04, OP20, OP38, OP39, OP40, OP48, and OP57, which have a combined area of 55.1 km^2. Hellas Gold has been granted mining rights over these concessions until 7 April 2024 and can be extended twice for durations of 25 years each. A request for a 25-year renewal of their expiration has already been submitted in May 2020. With the IA, (see Section 4.3), there is a provision that the concessions will be renewed for 25 years after 2026. Therefore, the application will be resubmitted in 2026. Hellas Gold has ownership of a small portion of private land within the concessions.

Table 4.1 List of concessions

Concession #	Name	Tenement type	Ownership	Area (km^2)	Valid until
OP03	Hellas Gold	Mining Concession	75%	3.25	7 April 2024
OP04	Hellas Gold	Mining Concession	75%	9.13	7 April 2024
OP20	Macedonia Copper Mines	Mining Concession	100%	9.68	7 April 2024
OP38	Macedonia Copper Mines	Mining Concession	100%	7.04	7 April 2024
OP39	Macedonia Copper Mines	Mining Concession	100%	10.00	7 April 2024
OP40a	Macedonia Copper Mines	Mining Concession	100%	5.34	7 April 2024
OP40b	Macedonia Copper Mines	Mining Concession	100%	1.52	7 April 2024
OP48a	Macedonia Copper Mines	Mining Concession	100%	1.08	7 April 2024
OP48b	Macedonia Copper Mines	Mining Concession	100%	1.52	7 April 2024
OP57a	Macedonia Copper Mines	Mining Concession	100%	3.37	7 April 2024
OP57b	Macedonia Copper Mines	Mining Concession	100%	0.97	7 April 2024
OP57c	Macedonia Copper Mines	Mining Concession	100%	2.20	7 April 2024
Total				**55.10**	

Notes:
- The joint owners of concessions OP3 and OP4 are shown as "Heirs of Stefanos Dimopoulos".
- Macedonia Copper Mines is a 100% subsidiary of Hellas Gold.

4.3 Investment Agreement

Subsequent to the 2019 Greek Parliamentary elections, Eldorado initiated talks with the newly established government, following which outstanding routine permits were released. In February 2021, the Company announced its wholly-owned subsidiary, Hellas Gold S.A., entered into an Investment Agreement (IA) with the Hellenic Republic to govern the further development, construction and operation of the Kassandra Mines.

The IA amends the 2003 Transfer Agreement and provides a modernized legal and financial framework to allow for the advancement of Eldorado's investment in the Kassandra Mines. The amendments to the Transfer Agreement in the IA became legally effective on 23 March 2021 following ratification by the Hellenic Parliament and publication in the Greek Government Gazette. The IA is governed by Greek law. Its initial term continues to 2051 and may be extended by an additional 25 years subject to certain conditions.

Hellas Gold is required to use commercially reasonable endeavours to implement a revised investment plan that is annexed to the IA, subject to the timely issuance of all relevant required permits. Key terms of the revised investment plan include:

- Completion of construction at Skouries and transition of the project into production.

- Expansion of Olympias to 650,000 tonnes per annum (tpa).
- Upgrades to the port facilities at Stratoni to allow for bulk shipment of concentrates.
- Further investment in exploration at Mavres Petres-Stratoni.

There are numerous other clauses and provisions, some of which are summarized below.

Hellas Gold will undertake further studies of on-site gold processing methods.

The IA includes investor protection mechanisms, similar to other large-scale foreign investment agreements in Greece.

The IA establishes a contractual regime for Hellas Gold to apply for, and receive, permits and licences required for the implementation of the investment plan.

Over the term of the IA, Hellas Gold will establish a corporate social responsibility program to support certain community, cultural, social, environmental, and charitable purposes that benefit the communities in the regions near the Kassandra Mines.

4.4 Permitting

The technical study submitted to the Ministry of Environment (MOE) for the Project was initially approved in February 2012. After numerous supplements relating to flotation plant, Tailings Management Facility (TMF) arrangements and "auxiliary temporary facilities", approval by the MOE was granted in 2013 - 14. An updated technical study covering amended aspects of the process plant and associated infrastructure was submitted to the MOE in December 2015, and this was approved in May 2016.

Subsequently, an updated specific technical study for the flotation plant was submitted to the MOE and approved on 11 November 2016. An update of the installation permit for the flotation plant was submitted by August 2016 and this was approved on 3 September 2019.

Permitting is discussed in detail in Section 20.

4.5 Royalties

Based on current Greek legislation, royalties are applicable on active mining titles and payable to the Greek state. The royalty is calculated on a sliding scale tied to international gold and base metal prices and $/€ exchange rates (refer Section 22.4). At an exchange rate of €1.12:US$1 and price ranges of US$1,237 - 1,460/oz Au and US$5,617 - 6,516/tonne Cu, Hellas Gold would pay a royalty of approximately 2.0% on Au revenues and 0.5% on Cu revenues.

During the term of the IA, Hellas Gold will pay to the Greek State a 10% increase in royalty rates for all contained metals (for example, the 2% royalty would become a 2.2% royalty in aggregate). The increased royalty will cease to be payable if and when a metallurgy plant is constructed at the Kassandra Mines and is in commercial production.

The corporate income tax rate is set at 24%.

4.6 Environmental liabilities

The closure and environmental rehabilitation activities for the Kassandra Mines complex of which Skouries is a part, relate to the following facilities:

- Open pit and underground mine.

- Integrated extractive waste management facility (IEWMF).
- Process facilities and infrastructure.

To meet the requirements of the reclamation program, decommissioning, closure, and reclamation of the affected areas must be undertaken.

Hellas Gold has provided a €50.0 million (M) Letter of Guarantee to the MOE as security for the due and proper performance of rehabilitation works in relation to the mining and metallurgical facilities of the Kassandra Mines complex, and the removal, cleaning and rehabilitation of the previously disturbed areas from the historic mining activity in the wider area of the project. Additionally, a Letter of Guarantee to the MOE, in the amount of €7.5M, has been provided as security for the due and proper performance of the Kokkinolakkas TMF.

Hellas Gold has also provided insurance coverage in accordance with Presidential Decree 148/2009 (Government Gazette 190/A/29.9.2009) for environmental liability.

4.7 Other

To the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Property.

5 Accessibility, climate, local resources, infrastructure, and physiography

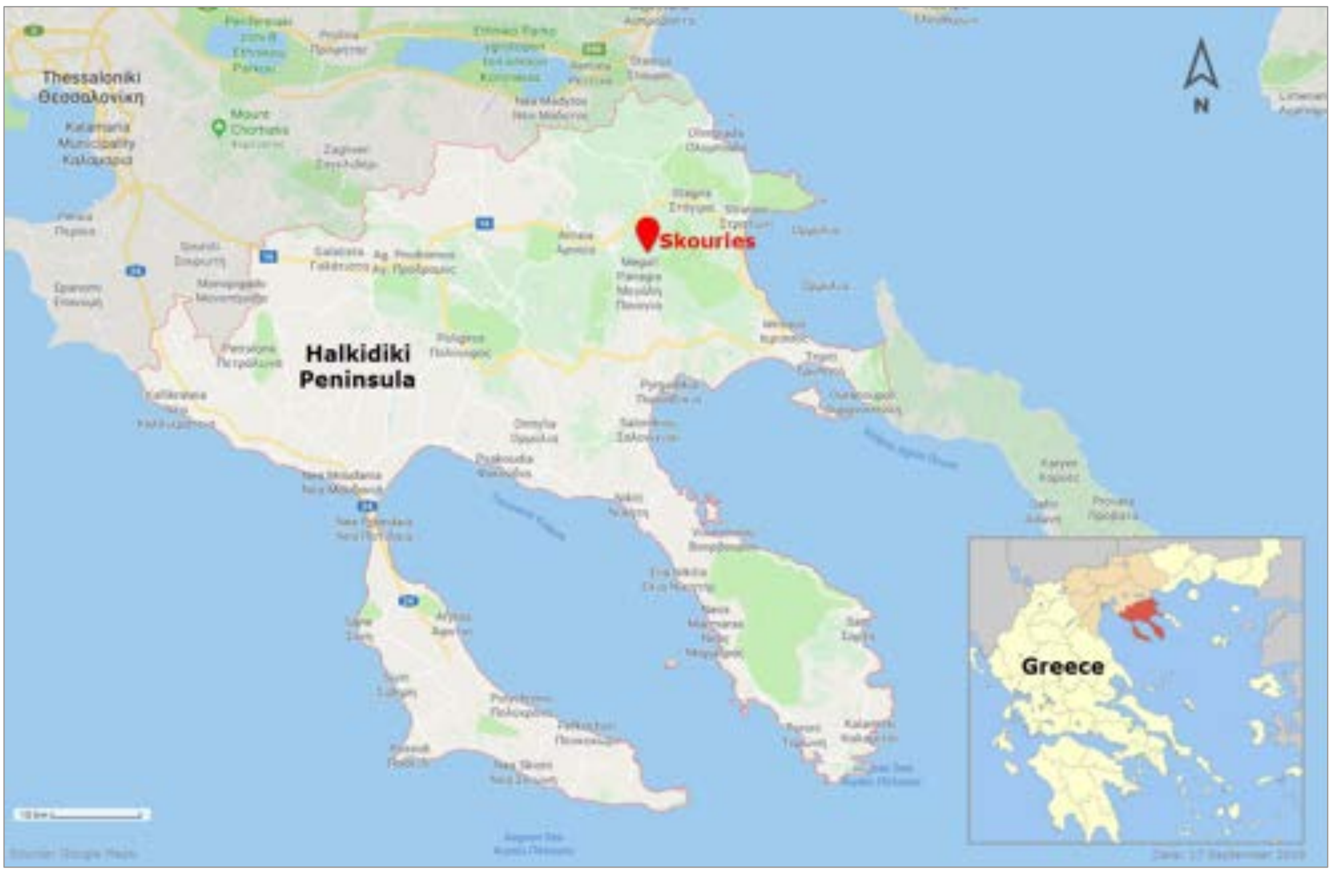
5.1 Location and accessibility

The Property is located within the Aristoteles municipality and Northern Macedonia region, about 100 km by road from Thessaloniki, the second largest city in Greece. Thessaloniki has one of the largest ports in Greece and an international airport.

The Property is readily accessible year-round by the national road network. The national road network in the area is among the best in northern Greece, with a major highway (E90) extending east from Thessaloniki to approximately 25 km north of the property. From the highway, the site is accessed by the regional road network to within 7 km of the site. A new access road has been constructed linking the Project site to the regional road network.

The Project is situated approximately 11 km south-west of Stratoni, 11 km south of the town of Palaeohori, and 3 km north-east of the village of Megali Panagia.

Figure 5.1 Location of Skouries property



5.2 Infrastructure and local resources

The area is well served by main power supplied via the Public Power Corporation (PPC). A high voltage 150 kilovolts (kV) overhead power line, connected to the main grid, is planned and will feed the main substation of the Skouries plant. Communications are good, broadband is available, and Hellas Gold also has a back-up microwave phone link located at Stratoni.

There is sufficient water available to support proposed operations from recirculated clean water from milling operations and boreholes. Groundwater levels are estimated to be some 50 metres (m) to 100 m below surface around the deposit.

The local area has a history of mining and there is a ready pool of skilled and unskilled labour.

5.3 Climate and physiography

The Project site is located in a sub-mountainous region characterized by hills dissected by steeply eroded valleys. The elevation ranges from 356 masl to 685 masl. The steep valleys drain towards the east and south.

The area is heavily wooded with oak, beech, and pine being the principal species. Regionally there is small-scale agriculture. The main agricultural products of the region are wines, honey, olives, and oil.

The Halkidiki Peninsula climate is generally mild with high rainfall. Typically, over 300 days or around 3,000 hours of sunshine are recorded annually. Average temperatures have limited fluctuations during the year. The lowest temperatures occur during December to February, ranging between 3.5 degrees Celsius (°C) to 19°C, while the highest temperatures occur during summer months, ranging between 23°C and 34°C. Temperatures below 0°C are limited to the mountainous areas. There are no seasonal restrictions on the operations.

5.4 Surface rights

The Project requires a total of 452.3 hectares (ha) of contiguous land. Hellas Gold owns 7% of this area. A further 92.7% of the required land (comprising private and public forestry) is granted in accordance to Greek law to Hellas Gold for its use. The remaining 0.3% is in the process of expropriation. This land requirement relates to the site layout diagrams given in this report for the mine, process plant, access and internal roads, tailings and waste rock storage facilities and other associated infrastructure. There are no properties of significance adjacent to the Skouries Project site and the surrounding area is mainly forested.

6 History

6.1 Introduction

There is a long history of mining in the area. Ancient mining reached a peak during the time of Philip II of Macedon and Alexander the Great, during the period 350 to 300 BC. The lead-rich ores from the Madem Lakkos mine at Stratoni were smelted for silver and the Olympias ores were processed for their high gold content. It has been estimated, from the volume of ancient slags, that about 1 million tonne (Mt) of ore were extracted from each locality during this period. It is believed that by 300 BC, the bulk of the ores above the water table at Olympias had been exploited, though the Stratoni mine continued in production through the Roman, Byzantine, and Ottoman periods. Ancient mining is less well documented at Skouries.

6.2 Ownership and work carried out

Milestones in the history of the Property are shown in Table 6.1, and elaborated on in the text in Sections 6.2.1, 6.2.2, and 6.2.3.

Table 6.1 Summary of the history of the Property

Year	Commentary
1960s	Initial drilling by Nippon Mining and Placer Development (Placer).
1970s	Drilling carried out by Hellenic Fertilizer Company.
1996 – 97	Ownership transferred to TVX, exploration drilling tested extensions at depth; in-fill drilling program carried out.
1999	TVX Gold Incorporated (TVX) issued Mineral Resource estimation; initial Feasibility Study completed.
2004	Aktor Enterprises Limited (Aktor) acquired mining concessions holding 317 km^2 including the Olympias and Skouries deposits together with the remaining Kassandra Mines assets through its subsidiary Hellas Gold.
	The Hellas Gold acquisition of the Kassandra Mines was ratified by parliament and passed into law in January 2004 (National Law no. 3220/2004).
	EGL acquired its initial ownership percentage interest in Hellas Gold from Aktor through its wholly owned subsidiary European Goldfields Mining (Netherlands) B.V.
2006	EGL prepared a bankable Feasibility Study based on an open pit operation to a depth of 240 m followed by underground mining.
2007	EGL increased share ownership of Hellas Gold to 95% (with 5% held by Aktor).
2011	Environmental Impact Study (EIS) approved by Greek government.
2012	Eldorado acquired the project through the acquisition of EGL.

6.2.1 1960 to 1990s

The Skouries deposit was initially drilled by Nippon Mining and Placer during the 1960s and subsequently in the 1970s by the then owners of the deposit, the Hellenic Fertiliser Company. Placer also carried out limited underground development from an adit. Details of this work are not available, and they have not been used in the Mineral Resource estimate.

6.2.2 TVX Gold (1996 to 2004)

TVX began a drilling program in August 1996 to confirm the deposit and to explore it at depth. A subsequent infill drilling program was conducted in 1997 with the objective of improving the evaluation of Indicated Mineral Resources in the deeper high-grade zone.

A Mineral Resource estimate was completed as part of a feasibility study initiated by TVX with SRK Consulting (SRK) and Kvaerner Metals in September 1998, with an updated EIS in February 1999. A summary of the 1998 Mineral Resource estimate is included in Table 6.2.

Table 6.2 Historical Mineral Resource estimate

Category	Tonnes (Mt)	Au (g/t)	Cu (%)
Measured	180.4	0.83	0.55
Indicated	10.8	0.61	0.47
Inferred	14.8	0.6	0.45
Total Measured and Indicated	**191.2**	**0.82**	**0.55**

Source: TVX / Kvaerner 1998.

6.2.3 European Goldfields (2004 to 2012)

EGL acquired the property in 2004 and audited and reviewed the 1998 Mineral Resource statement; it was concluded that the Mineral Resource was classified according to the definitions and guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) respecting Mineral Resources and Mineral Reserves. The historical Mineral Resources were reported at a nominal 0.4 grams per tonne (g/t) Au cut-off.

EGL prepared a feasibility study in 2006 based on an open pit operation followed by an underground mine accessed by a vertical shaft and surface access ramp. The selected underground mining method was sub-level caving (SLC) at a production rate of 7.0 million tonnes per annum (Mtpa).

In 2007, SRK was retained by EGL to undertake an engineering study, applying open pit and SLC underground mining methods based on the 1998 TVX/2004 EGL geological model, to update the 2006 Feasibility Study estimation of Mineral Reserves based on higher metal selling prices. The 2007 NI 43-101 Technical Report Mineral Reserves are summarized in Table 6.3.

Table 6.3 Historical Proven and Probable Mineral Reserve estimate

Category	Ore (Mt)	Grade Au (g/t)	Grade Cu (%)
Open Pit Mineral Reserves			
Proven	42.5	0.71	0.46
Probable	9.7	0.60	0.39
Subtotal	**52.2**	**0.69**	**0.45**
Underground Mineral Reserves			
Proven SLC	32.4	1.07	0.62
Proven development	2.6	1.16	0.66
Probable SLC	55.1	0.81	0.57
Probable development	3.9	0.90	0.62
Subtotal	**94.0**	**0.91**	**0.59**
All Sources			
Proven	77.5	0.87	0.54
Probable	68.7	0.78	0.55
Total	**146.2**	**0.83**	**0.54**

Source: EGL 2007.

A later study investigated the possible use of sub-level open stope (SLOS) with tailings backfill. The objective of the study was to review the possibility of mining the deposit by the SLOS method as an alternative method to SLC, using tailings as backfill to minimize the amount of surface tailings disposal and to reduce the potential subsidence area and so minimize the overall environmental impact of the Project. This methodology, along with limiting the open pit depth to 420 masl, was the basis of the approved EIS submitted in July 2011. The Mineral Reserves estimated from this work formed the basis for those reported in the 2011 EGL Technical Report. The 2011 Mineral Reserves are summarized on Table 6.4.

Table 6.4 Historical Proven and Probable Mineral Reserve estimate

Proven and Probable Mineral Reserves	Tonnes (Mt)	Au (g/t)	Cu (%)	Contained Au (Moz)	Contained Cu (kt)
Open pit	47.0	0.70	0.44	1.046	210
Underground	91.4	0.86	0.57	2.544	526
Total Mineral Reserves	**138.4**	**0.81**	**0.53**	**3.590**	**736**

Source: EGL 2011.

The QP for this Technical Report has not done sufficient work to classify the above-referenced historical estimates and they have been superseded by more recent estimates. They are, therefore, not current Mineral Resources or Mineral Reserves and the Issuer is not treating the historical estimates as current Mineral Resources or Mineral Reserves.

6.3 Production

There has been no documented production from the Property.

7 Geological setting and mineralization

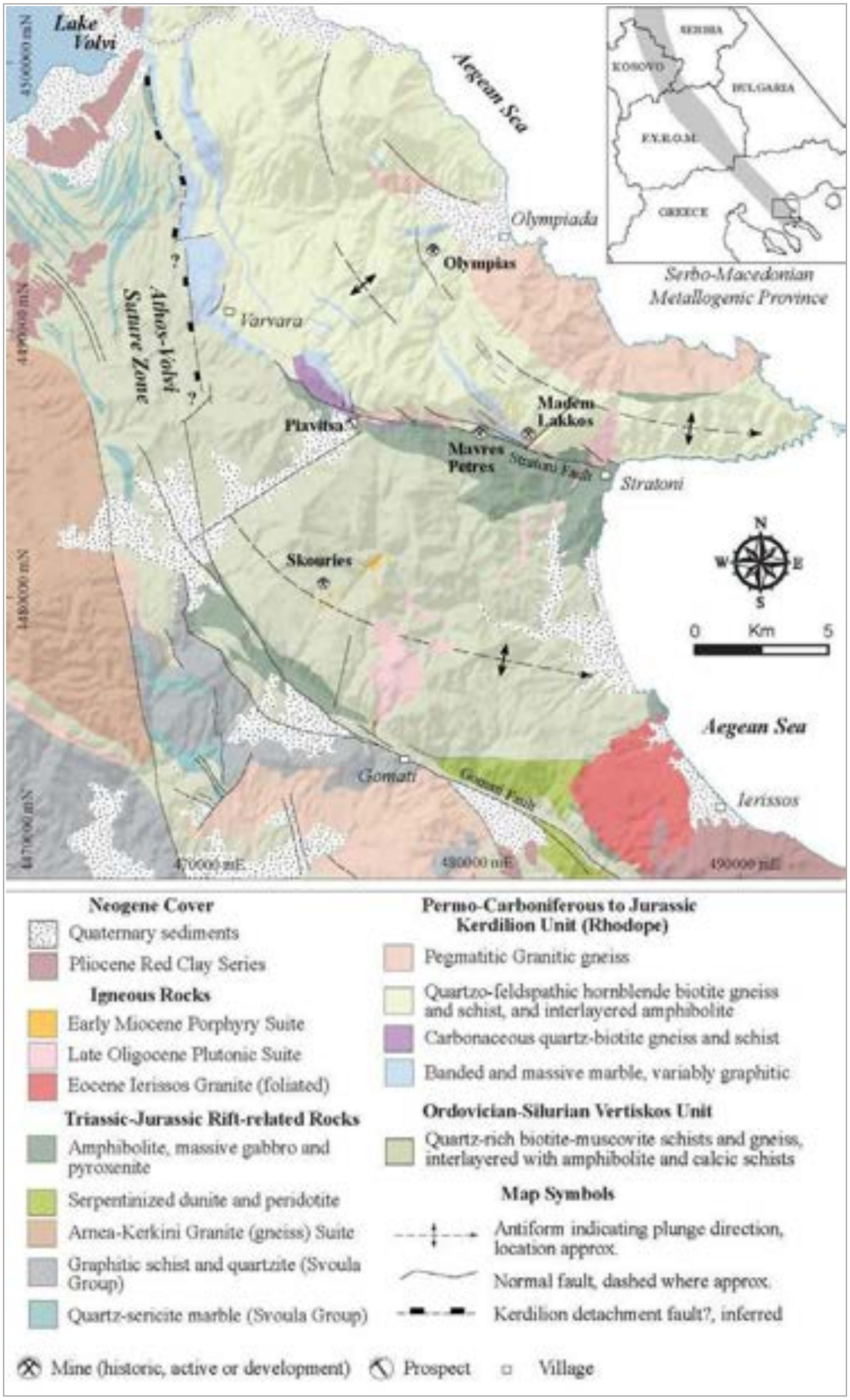
7.1 Regional geology

The tectonic structure of Greece consists of elongated tecto-magmatic belts of variable metamorphic grade which trend north-west (NW) to south-east (SE). These broadly coincide with the trend of the main mountain ranges of the country. These zones represent successive episodes of subduction, resulting from the north-east movement of the African plate during the Tertiary period. The rocks that comprise these orogenic zones consist of gneiss, schist and acid igneous intrusives. These rocks host the mineral deposits of the Kassandra Mining District.

The Western Tethyan orogenic belt in south-east Europe contains several major metallogenic provinces including the Serbo-Macedonian Metallogenic Province (SMMP) that hosts the Kassandra mining district and the Skouries deposit (Janković, 1997). The Western Tethyan orogen comprises a series of magmatic belts that broadly young to the south from Cretaceous to Paleogene subduction related arc magmatism through to post-collisional Neogene magmatism (Richards, 2015). In Northern Greece, the orogeny formed from the Late Cretaceous to early Eocene convergence of the Serbo-Macedonian Apulian and Pelagonian microcontinents to the previously accreted Rhodope continental fragments on the Eurasian margin (Pe-Piper and Piper, 2006). Crystalline basement within the Kassandra mining district includes the upper litho-tectonic SerboMacedonian Vertiskos unit and the lower litho-tectonic Kerdilion unit exposed within the southern Rhodope metamorphic core complex. Figure 7.1 shows the geological map of the Kassandra Mining District.

The SMMP forms a NW-trending zone of base and precious metal deposits including a large Au-endowment (~ 25 Moz) that is associated with Oligocene to Miocene magmatic complexes including porphyry (Skouries, Greece; Illovitza, Bucim, Republic of North Macedonia; and Tulare, Serbia) and carbonate replacement deposits (Olympias, Mavres Petres, Madem Lakkos, and Piavitsa, Greece), as well as the Plavica high sulphidation epithermal deposit, Republic of North Macedonia. The mineral deposits formed during postcollision- extension and emplacement of intermediate to felsic magmas with high K calcalkalic- to shoshonitic composition and localized ultra-potassic mafic magmas (Borojevic, Sostaric et al. 2012; Siron et al., 2016). The heterogeneity of the Cenozoic magmas likely resulted from crystal fractionation, assimilation and mixing of melted depleted mantle metasomatized by earlier subduction processes, and partial melting of lower crustal rocks.

Figure 7.1 Geological map of the Kassandra mining district



Source: Modified from Siron et al. 2016.

7.2 Local geology

The historically mined Madem Lakkos, currently mined Mavres Petres, and undeveloped Piavitsa deposits occur along the east-west oriented, moderate south-dipping Stratoni fault zone, a major structural feature and important mineralizing corridor in the centre of the region as shown on Figure 7.1. The mylonitic to brittle fault zone extends over 12 km from the coast at Stratoni to the village of Varvara in the west. Marble lenses entrained within the fault are separated from their likely footwall equivalents by a minimum of 250 m at the Mavres Petres deposit based on drill core data and cross section interpretation. The fault separates the Kerdilion unit to the north from the Vertiskos unit to the south with gneiss and marble in the footwall and amphibolite and schist in the hanging wall. The fault zone crosscuts the lower portion of the late Oligocene (25.4 ± 0.2 Ma) Stratoni granodiorite stock but is cut by a Miocene glomerophyric monzonite porphyry dike at Piavitsa (20.62 ± 0.13 Ma) constraining major fault movement and related hydrothermal mineralization to the late Oligocene to early Miocene (Siron et al., 2016).

Metamorphic rocks of the Kerdilion unit consist of quartzo-feldspathic hornblende-biotite gneiss, marble, amphibolite, localized bodies of megacrystic plagioclase-microcline orthogneiss, and fine-grained to aplitic granite gneiss (Kalogeropoulos et al. 1989; Nebel et al. 1991; Gilg and Frei 1994). The marble units host the carbonate-replacement deposits. The lithologies have an arcuate geometry, striking in a north-south direction in the north and becoming east-west near the Stratoni fault (Siron et al., 2016). Middle Jurassic to Early Cretaceous zircon U-Pb and Pb-Pb ages from granitic gneisses of the Kerdilion unit range from 164 Ma to 134 Ma and are interpreted as primary igneous ages (Himmerkus et al., 2011). The hosting lithologies, however, are likely Carboniferous to Permian based on inherited zircon cores derived from the orthogneisses (Himmerkus et al., 2011). Pegmatitic dikes and sills occur throughout the Kerdilion unit and represent anatectic partial melting of the metamorphic rock from about the middle Paleocene to middle Eocene (Wawrzenitz and Krohe 1998; Kalogeropoulos et al., 1989). The pegmatites are largely absent south of the Stratoni fault.

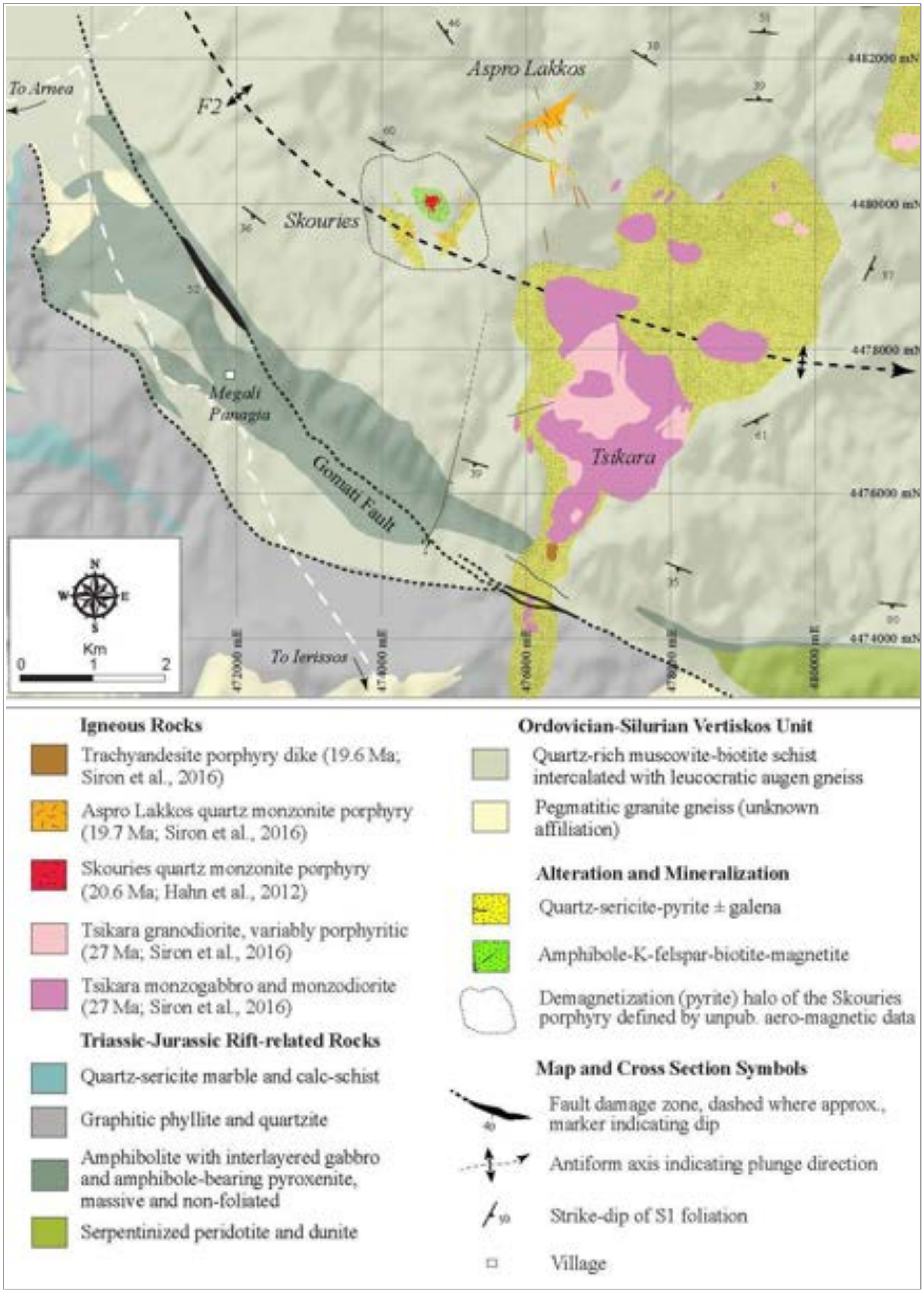
Graphitic garnet-bearing quartz-biotite gneiss and schist are spatially associated with the Stratoni fault zone and amphibolite with variably serpentinized pyroxenite occurs within the hanging wall. The Vertiskos unit occurs south of the Stratoni fault and hosts the Skouries porphyry deposit (Figure 7.2). The unit is a monotonous sequence of quartz-rich feldspathic to muscovite-biotite-bearing gneiss and schist. Minor calcareous schist, marble, and amphibolite are also thinly interlayered within the metamorphic sequence. Zircon U-Pb ages show that the micaceous schist ranges from Neoproterozoic (686 – 576 Ma) to Ordovician (464 – 450 Ma), which is consistent with the age of the Pan-African Pirgadikia and Vertiskos units of the Serbo-Macedonian terrane (Himmerkus et al., 2006, 2007).

Cretaceous to mid-Eocene ductile deformation accompanied by lower amphibolite-grade metamorphism and overprinting retrograde greenschist metamorphism affected the Kerdilion and Vertiskos units (Figure 7.2). A regionally prominent penetrative shallow dipping S1 foliation is defined by alignment of peak metamorphic minerals (e.g., biotite or amphibole). Subsequent high-strain transposition resulted in tight to isoclinal F2 folds locally accompanied by subparallel axial planar S2 cleavage. A later lower-strain deformation event superimposed a spaced and steeply dipping S3 foliation on the pre-existing fabrics. This event is associated with km-scale upright and open east-plunging F3 folds evident as district-scale antiforms in the footwall of the Stratoni fault (Siron et al., 2016).

A series of discrete magmatic events is recognized in the region including the Triassic Arnea granite suite (228 ± 5.6 Ma) within the Vertiskos unit, and Late Cretaceous to early Eocene (68 ± 1 Ma to 53 ± 4 Ma) subduction-related calc-alkaline granites at Ierissos, Ouranoupolis, and Grigoriou on the Athos Peninsula. These granitic intrusions exhibit a weak tectonic fabric suggesting emplacement during the waning stages of regional deformation. Post-collision Oligocene-Miocene magmatism coincides with the main mineralizing events in the Kassandra mining district. Late Oligocene magmatism ranges from early monzogabbro to monzodiorite to later-stage granodiorite (Siron et al., 2016). These intrusions typically display medium-grained equigranular textures through to porphyritic phases with crowded textures dominated by feldspar phenocrysts. Most unaltered late Oligocene intrusions are high-K calc alkaline, and the intrusions occur along an NNE-trending structural corridor defined by the alignment of igneous centres and orientation of dikes. A suite of early Miocene intrusions, including Skouries, have porphyritic textures and are quartz monzonite to syenite in composition. Phenocrysts are prismatic consisting of plagioclase and megacrystic K-feldspar, fine-grained euhedral biotite and relict amphibole. Rounded quartz phenocrysts occur in minor abundance and K-feldspar and quartz comprise the groundmass with accessory zircon, magnetite, and pyrite. The quartz monzonites belong to the high-K calc-alkaline to weakly shoshonitic magma series. Late Miocene stocks and dikes were controlled by pre-existing structures such as fold axes and faults.

Figure 7.2 is a geological map of the Skouries deposit and its local setting that has been modified from Siron et al., 2016.

Figure 7.2 Geological map of the Skouries deposit and surrounding area



Source: Modified from Siron et al. 2016.

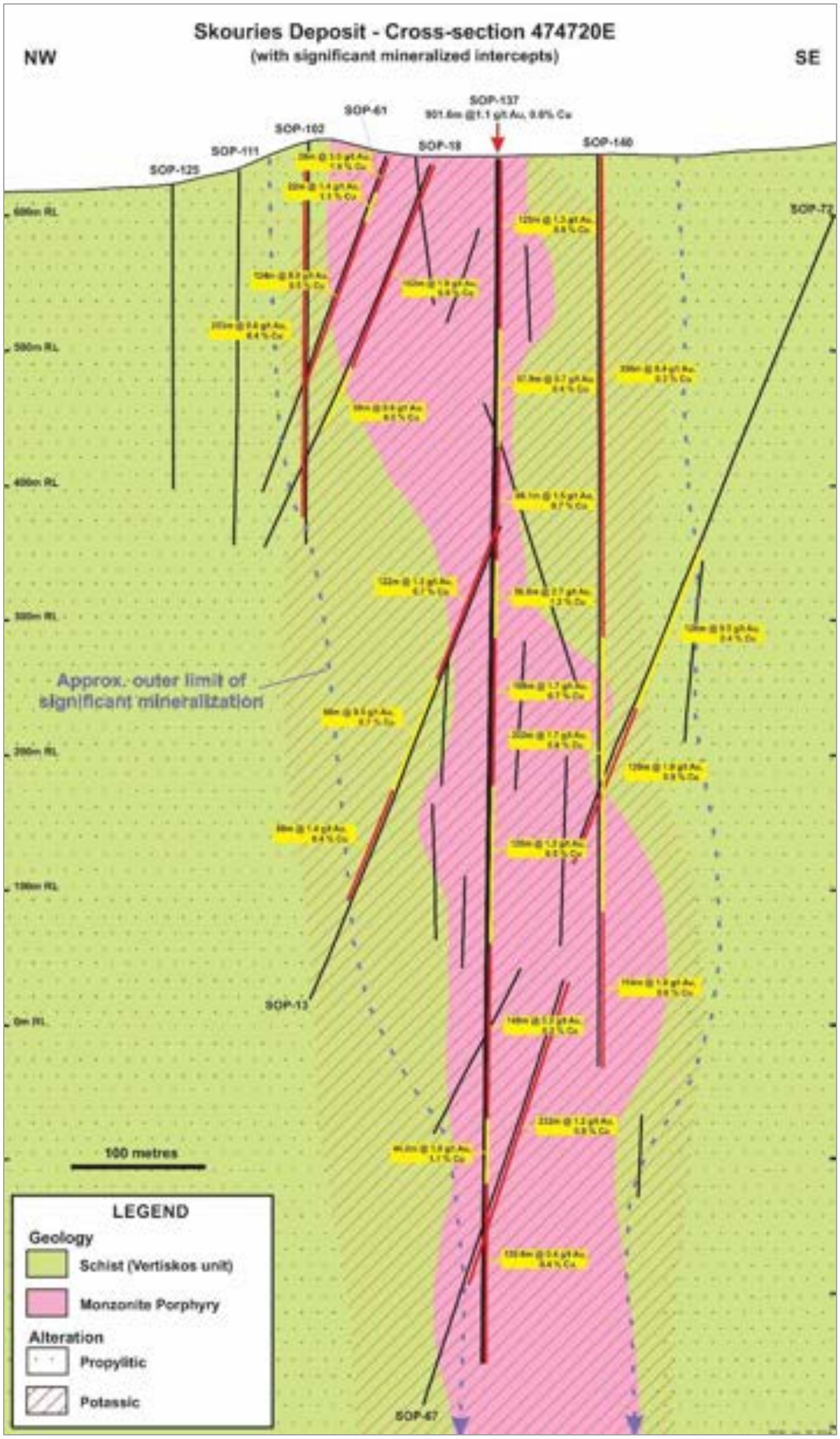
7.3 Deposit geology

The Skouries deposit is centred on a small (less than 200 m in diameter), pencil-porphyry stock that intruded schist and gneiss of the Vertiskos unit. The mineralized porphyry intrusion plunges steeply to the south-southwest and obliquely crosscuts the moderate to steeply north-east dipping limb of a district-scale F2 antiform. Mineralization has been tested to a depth of 920 m from surface as shown on Figure 7.3. Surface exposures and drill data indicate that the porphyry stock has a subtle north-east elongate geometry. The porphyry is characterized by at least four intrusive phases that are of probable quartz monzonite to syenite composition (Kroll et al. 2002; Frei, 1995) but contain an intense potassic alteration and related stockwork veining that overprints the original protolith. Potassic alteration and copper-gold mineralization also extend into the country rock; approximately two thirds of the Measured and Indicated Mineral Resources are hosted outside the porphyry with about a 50:50 split in gold-equivalent ounces. The potassic alteration is characterized by potassium feldspar overgrowths on plagioclase, secondary biotite replacement of igneous hornblende and biotite, and a fine-grained groundmass of K-feldspar-quartz with disseminated magnetite. Four main stages of veining are recognized:

- Early stage of intense quartz-magnetite stockwork (pre-ore stage).
- Quartz-magnetite veinlets with chalcopyrite ± bornite (initial ore stage).
- Quartz-biotite-chalcopyrite ± bornite-apatite-magnetite veinlets (main ore stage).
- Localized, late stage set of pyrite ± chalcopyrite-calcite-quartz veins (post-ore stage).

Dating by Hahn et al. (2012) confirms the coeval timing of the Skouries intrusion (20.56 ± 0.48 Ma; LA-ICP-MS single grain zircon U-Pb) and potassic alteration (19.9 ± 0.9 Ma; Ar-Ar biotite).

Figure 7.3 Geological cross section through the Skouries porphyry Au-Cu deposit



Source: Eldorado 2022.

8 Deposit types

8.1 Deposit model

Skouries is typical of a gold-copper porphyry deposit. Mineralization occurs in stockwork veins, veinlets and disseminated styles typical of a porphyry, and has a subvertical, pipe-like shape. The multi-phase monzonite to syenite porphyries intruded into metamorphic basement rocks. Both igneous and metamorphic rocks contain high temperature potassic alteration (K-feldspar-biotite) and stockwork quartz-magnetite-chalcopyrite-bornite veins. The potassic zone in the surrounding country rock is surrounded by a high temperature inner propylitic alteration characterized by amphibole. The deposit, however, lacks extensive phyllic or argillic-advanced argillic zones typical of many porphyry systems. This may, in part, reflect a deeper level of erosion and the focused nature of the magmatic-hydrothermal system.

9 Exploration

Exploration work at the Property completed by Eldorado has focused on evaluating potential for additional porphyry mineralization within the surrounding area. This has included geological mapping, geochemical sampling (soil and outcrop) and geophysical survey programs, as well as drill-testing of new targets generated from this work. Detailed geological mapping of fresh outcrop areas generated during early construction has been complete in several phases beginning in 2014. Historical soil sampling completed by previous owners has been infilled and extended, with the immediate deposit area now covered at a sample spacing of 50 m x 50 m and the surrounding property at 200 m x 200 m, with anomalous zones at a closer spacing. In November 2020, Eldorado in collaboration with the EU funded Smart Exploration program carried out a SkyTEM312HP survey over the majority of the Halkidiki license area. In total this comprised 79 N-S flight lines spaced at ~200 m with a transmitter height of 40 – 55 m for a total of 1,465 km. The survey recorded magnetic, electromagnetic, and digital elevation data. This was subsequently processed and delivered as sections and inversion models and used for further exploration targeting.

In 2019, reconnaissance drilling was conducted at the Rian Prospect (9 drillholes, 1,078 m), a base metal vein showing discovered during mapping of the tailings management facility area. A drilling program testing new targets at the Tsikara prospect, a granodiorite to monzodiorite complex with a large quartz-sericite alteration anomaly located two to four kilometres southeast of Skouries was conducted in 2017 with 10 drillholes (4,453 m) completed.

10 Drilling

10.1 Drilling progress

Diamond drillholes are the sole source of subsurface geologic and grade data for the Skouries Project. Delineation drilling of the deposit was carried out in two major campaigns: between 1996 and 1998 by then owner TVX, and in 2012 – 13 by Eldorado. This data is summarized in Table 10.1. The locations of these drillholes are shown on a collar plan map in Figure 10.1.

TVX drilled a total 72,232.5 m of core in 121 drillholes using NQ (47.6 millimetres (mm)) diameter core. Holes reached a maximum depth of 1,013 m. Hole deviation was measured by Sperry Sun nominally every 50 m depth. All of the drill core from this period was removed from site prior to Eldorado obtaining the Project through the acquisition of EGL.

Eldorado conducted two drill campaigns on the Skouries Project in 2012 and in 2013. This comprised: 1) a 34-hole, infill program comprising 6,922 m of drilling designed to upgrade all resources within the pit shell to Measured or Indicated categories; and 2) a 10-hole, 6,617 m confirmation program designed to test the core of the main mineralized portion of the deposit to compensate for the lack of a drillcore record from the earlier TVX campaign. These confirmation drillholes are not included in the current Mineral Resource estimation; they were only used to assess the resultant block grades in the resource model as a confirmation / verification activity (see also Section 12).

Table 10.1 Summary of diamond drilling programs

Campaign	Drillhole series	Purpose	Used in current Resource	No. of DHs	Total drilling (m)	Avg. depth (m)	Max. depth (m)
1996 - 98	SK-08 to -30	Infill	Y	23	15,501.00	674.0	1001.0
	SOP-01 to -98	Infill	Y	98	56,731.50	578.9	1013.0
	Total			**121**	**72,232.50**		
2012 - 13	SOP-99 to -132	Infill	Y	34	6,921.60	203.6	300.1
	SOP-134 to -143	Confirmation	N	10	6,617.00	661.7	901.6
	Total			**44**	**13,538.60**		
	Grand total			**165**	**85,771.10**		

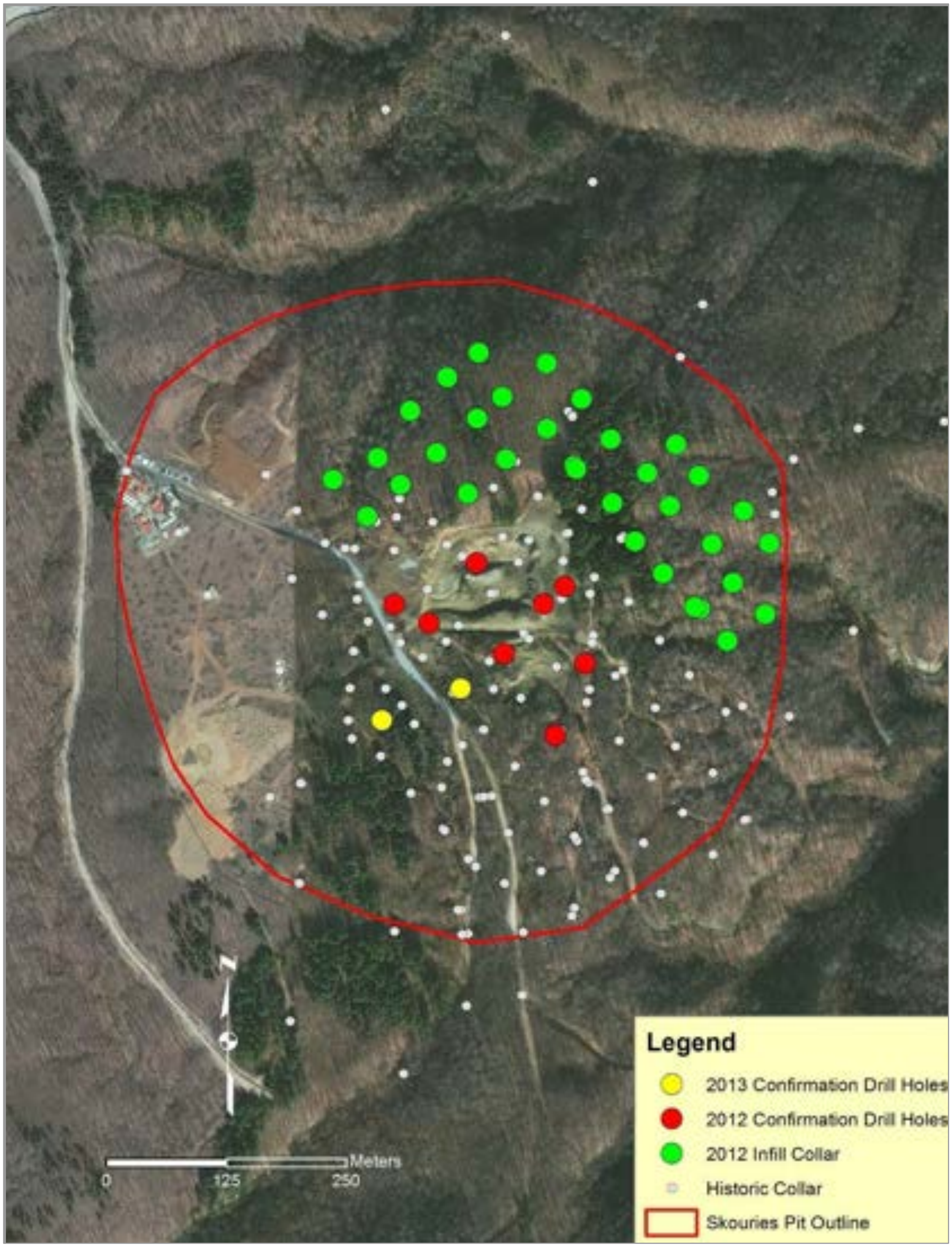
10.2 Sampling procedures

All diamond drilling carried out by Eldorado was done with wireline-equipped drill rigs with up to four rigs being employed. Core was generally HQ size, which is 63.5 mm nominal core diameter, or NQ, which is 47.6 mm nominal core diameter. Some deep drillholes required a reduction to NQ size to complete the drillhole. Drillers placed the core into sturdy, locally made, wooden core boxes with each box holding about 4 m of HQ core. The driller kept track of the drilling depth and placed wooden marker blocks at the end of each run to indicate the depth from the collar. These marker blocks were nailed into the boxes.

The core boxes were delivered to the logging site at the Stratoni mine area. The core was logged in detail on paper logging sheets, and the data were then entered into the database. Sample numbers were written on wooden core boxes allowing gaps in numbering sequence for control of sample insertion. Sample information (sample number, drillhole ID, sample depth, etc.) was recorded in durable sample tag books; copies of the sample tags were stapled in the core boxes at the beginning of each sample interval. The entire lengths of the drillholes were sampled with sample lengths being a nominal 2 m.

Geology and geotechnical data were collected from the core and core was photographed (wet) before sampling. Samples consisted of half-cores cut using a diamond-blade saw. The core cutting and sampling was done within the logging site. The cut samples were then sent to the Eldorado Canakkale preparation facility in Northwest Turkey.

Figure 10.1 Map of drillhole collars



Source: Eldorado 2022.

10.3 Surveys

TVX used a Sperry Sun multi-shot tool to measure drillhole deviation downhole. Station intervals were, generally, approximately 50 m and, occasionally, up to twice that distance.

The Eldorado drilling was surveyed downhole using a gyro system, with measurements being taken on a 5 m interval or, sometimes, 10 m.

Collar surveys were carried out by the project team both for set-up and for final collar pick-up.

10.4 Core recovery

Core recovery was very good to excellent. Holes drilled mostly in schist had slightly lower recovery than those drilled in the porphyry. The TVX historic recovery average was 91%. Eldorado's pit infill drilling, mainly in schist units, averaged 91%, whereas the deep confirmation drillholes that tested the bulk of the copper and gold mineralization of the deposit (in the porphyry) averaged 96% core recovery.

10.5 Bulk density

Samples taken for assay from drillholes were measured for specific gravity, with results tabulated by rock type. The specific gravity for non-porous samples (the most common type) was calculated using the weights of representative samples in water (W2) and in air (W1). As the samples are generally non-porous, specific gravity equates to bulk density, which is calculated by W1/(W1-W2).

From the historical work carried out by EGL, bulk density values of 2.64 t/m^3 for the porphyry and 2.73 t/m^3 for schist samples were calculated. A total of 483 samples was measured by EGL, of which 101 were porphyry and 382 were schist. Quality control measures included using outside laboratories and waxed and non-waxed samples. More recent measurements by Eldorado conducted as checks yielded very similar averages.

Overall, both the historic and recent Skouries drill programs and data capture were performed in a competent manner and there were no drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the drilling programs.

11 Sample preparation, analyses, and security

11.1 Introduction

The majority of the samples for the Skouries Project originated from the 1996 - 98 drill campaign by TVX. The procedures for sampling, analysis and security for this work were described in the 2011 EGL Technical Report and are summarized here. Eldorado reviewed the associated studies and data and agreed with the conclusions that the drill data are acceptable to be used for Mineral Resource estimation. The QP concurs with this conclusion, having reviewed the data and reports. The background and quality assurance / quality control (QA/QC) results of the Eldorado work are discussed in more detail in Section 11.3.

Eldorado also carried out a confirmation drill program in 2013, which is discussed in Section 12.3.

11.2 Commentary on TVX work

11.2.1 TVX sample preparation and assaying

The cut core samples were prepared in a several laboratories as follows:

- I.G.M.E. (the Greek Geological Survey) at Xanthi, I.G.M.E. at Athens (both International Organization for Standardization (ISO) accredited), and TVX at Stratoni (at the time ISO 9002 accredited), the latter by TVX personnel.
- Stratoni laboratory by TVX personnel (at the time ISO 9002 accredited).
- Skouries sample preparation laboratory located at Madem Lakkos by TVX personnel (at the time ISO 9002 accredited).

In all cases gold, total copper, soluble copper with citric and sulphuric acid, and silver assays were completed by the ALS-Geolab laboratory in Santiago Chile. This was chosen as the main laboratory. It should be noted that soluble copper assays were generally completed for samples within the first 100 m from the surface. Copper was determined by an aqua regia digest and atomic absorption spectroscopy (AAS). Gold was normally assayed on a 50 g sample utilizing fire assay with an AAS finish. However, as coarse gold is known to occur in the deposit, a study was conducted utilizing screen fire assay using a 170 mesh screen and assaying the −170 mesh fraction and combining with the results from the retained fraction.

11.2.2 TVX QA/QC program

The QA/QC programs carried out by TVX consisted of the use of specified duplicate assays by a different laboratory and “blind” coarse reject checks.

TVX’s umpire duplicate program entailed the submission of approximately 5% of mineralized pulps to an independent umpire laboratory. Initially, SGS S.A. (SGS) in France was used for this program and later, the Chemex Laboratory in Vancouver, Canada was used (certified under ISO 9001). The purpose of this analysis was to detect any biases between laboratories, as well as to calculate the assaying error and see if it was within industry standards.

The blind checks on the coarse rejects consisted of resubmitting 5% of the coarse rejects of samples in ore to the current sample preparation laboratory for splitting. These were then resubmitted, pulverized, and re-assayed by the main laboratory, ALS-Geolab in Chile. The aim of this work was to validate the complete sample preparation and assaying procedure, as well as to calculate the total error involved.

Due to a large scatter in gold assay results, a coarse gold study was carried out and made the following conclusions:

- No biases were detected for 50 g or 100 g assays relative to screen fire assays, which are considered the most reliable assay method.
- 50 g assays are reliable up to grades of approximately 2.8 g/t Au.
- 100 g assays are reliable up to grades of approximately 5.0 g/t Au.
- Coarse gold is associated with porphyry and not with schist.

Following a study on laboratory bias and sample preparation, Kvaerner Metals concluded that the TVX assay results for the Skouries deposit are within acceptable error limits.

11.3 Eldorado sample preparation and assaying

This section deals with Eldorado's sampling and QA/QC work in 2012 and 2013, after Eldorado acquired the Skouries Project. No additional data has been collected since 2013.

11.3.1 Sample preparation

The cut samples were sent to the Eldorado Canakkale preparation facility in north-west Turkey. There the samples were crushed to 90% minus 3 mm and prepared according to the following protocol:

- A 1 kilograms (kg) subsample was riffle split from the crushed minus 3 mm sample and pulverized to 90% passing 75 µm (200 mesh).
- A 200 g subsample was split off by taking multiple scoops from the pulverized 75 µm sample.
- The 200 g subsample was placed in a kraft envelope, sealed with a folded wire or glued top, and prepared for shipping. The rest of the pulverized sample was then stored in plastic bags.

11.3.2 Sample batches

- All equipment was flushed with barren material and blasted with compressed air between each sampling procedure. Regular screen tests were done on the crushed and pulverized material to ensure that sample preparation specifications were being met.
- The sample batches were arranged to contain regularly inserted control samples. A Certified Reference Material (CRM), a duplicate, or a blank sample were inserted into the sample stream at every 8th sample. The duplicates were used to monitor precision; the blank sample can indicate sample contamination or sample mix-ups and the CRM was used to monitor accuracy of the assay results.

11.3.3 Sample pulps

- The sample pulps were sent from the Çanakkale facility to Acme Labs (Acme), now Bureau Veritas (BV) analytical laboratory in Vancouver, Canada. The BV laboratory in Vancouver is an independent laboratory and is ISO 9001 certified and conforms with the requirements of ISO/IEC 17025:2017, RG-MINERAL (accredited laboratory No 720). All samples were assayed for gold by 30 g fire assay with an AAS finish, with Au values above 10 parts per million (ppm) determined by a gravimetric finish. Copper was determined by using an aqua regia digestion with an Inductively Coupled Plasma Mass Spectrometry (ICP-MS) analysis. For values over 10,000 ppm the analysis was by Inductively Coupled Plasma Emission Spectroscopy (ICP-ES).

11.4 Eldorado QA/QC program

Eldorado's QA/QC program comprised the use of CRMs, duplicates, and field blanks. The number of samples, duplicates, CRMs, and blanks are presented in Table 11.1. The QA/QC compliance varied between 6% and 8% for the different programs and sample types.

Table 11.1 Summary of QA/QC sampling for 2012 - 2013

Year	No. of holes	Metres	Samples	Duplicates	CRM	Blank
2012	34	6,921.6	3,306	197	208	241
2012	8	5,152.0	2,584	157	171	190
2013	2	1,465.0	733	54	55	58
Totals	**44**	**13,538.6**	**6,623**	**408**	**434**	**489**

11.4.1 Assay results for Certified Reference Material

Assay results were provided to Eldorado in electronic format and as paper certificates. Numerous CRM samples were used for both Au and Cu. The Au CRM grade range covered values between 0.26 g/t and 16.0 g/t, whereas the Cu CRM grade range covered values between 0.47% and 1.6% (Table 11.2).

The following comments are made regarding CRMs:

- CRMs are inserted to check the analytical accuracy of the laboratory.
- CRMs should be obtained for all economic minerals. For each economic mineral, there should be three corresponding standards:
  - At around the expected cut-off grade of the deposit.
  - At the expected average grade of the deposit.
  - At a higher grade.
- CRMs should represent approximately 5% of the total samples assayed. CRM results should be reviewed immediately upon receipt of assay.

It is noted that all criteria above have been adhered to for the Skouries Project.

Table 11.2 Summary of CRM expected values and standard deviation

CRM	Expected value and 2 standard deviations (SD)				No. assays
	Au (g/t)	2 x SD	Cu (g/t)	2 x SD	
CDN-CGS-26	1.64	0.11	1.58	0.07	67
CDN-CM-13	0.74	0.094	0.786	0.036	31
CDN-CM-17	1.37	0.13	0.791	0.04	58
CDN-CM-23	0.549	0.06	0.472	0.026	76
CDN-FCM-6	2.15	0.16	1.251	0.064	39
CDN-FCM-7	0.896	0.084	0.526	0.026	37
CDN-GS-12A	12.31	0.54			31
CDN-GS-15B	15.98	0.71			23
CDN-GS-2L	2.34	0.24			19
CDN-GS-P3C	0.263	0.02			17

The following pass / fail criteria were established by Eldorado:

- Automatic batch failure if the CRM result is greater than the round-robin limit of three standard deviations (SD).
- Automatic batch failure if two consecutive CRM results are greater than two SDs on the same side of the mean.

If the batch failed, it was re-assayed until the contained control samples passed. Override allowances were made for samples testing weakly- or non-mineralized material. Batch pass / failure data were tabulated on an ongoing basis, and charts of individual reference material values with respect to round-robin tolerance limits were maintained.

The following recommendations are made in regard to the current pass / fail criteria:

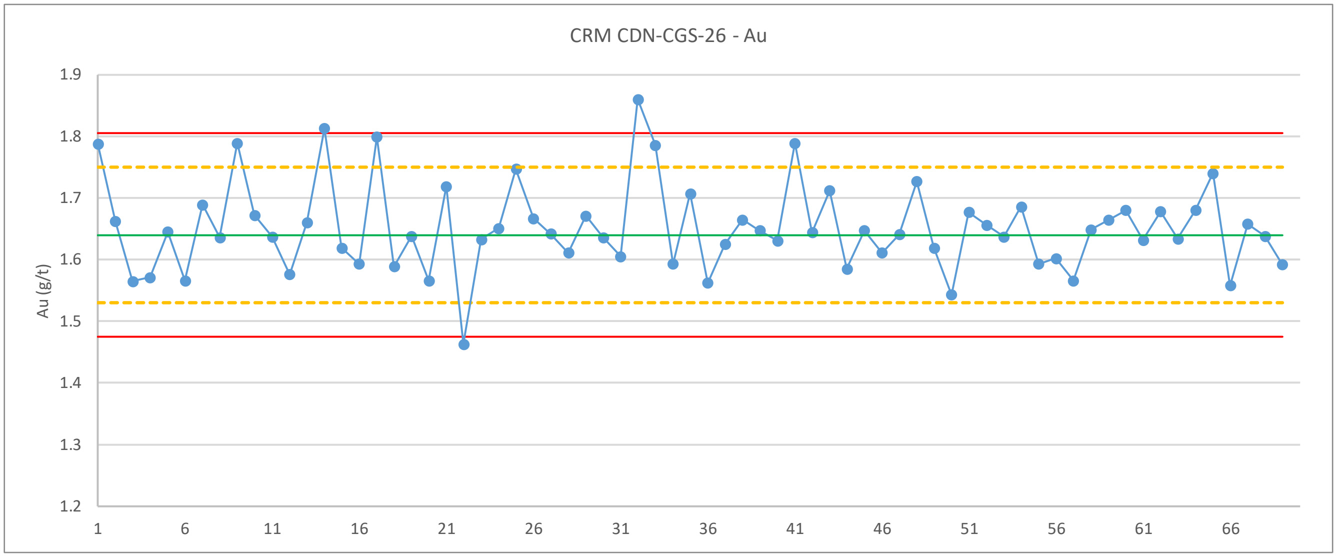
- Assay batches with two consecutive CRMs outside two SDs should be re-run, regardless of the side of the mean on which they fall.
- As CRM data accumulates over time, results should be reviewed for biases in the data.

Control charts are presented for selected CRMs (Figure 11.1 to Figure 11.9). These are for CRM CDN-CGS-26 Au and Cu, CRM CDN-CM-17 Au and Cu, CRM CDN-CM-23 Au and Cu, CRM CDN-FCM-7 Au and Cu, CRM CDN-GS-2L Au.

Note the legend for each CRM Figure is as follows:

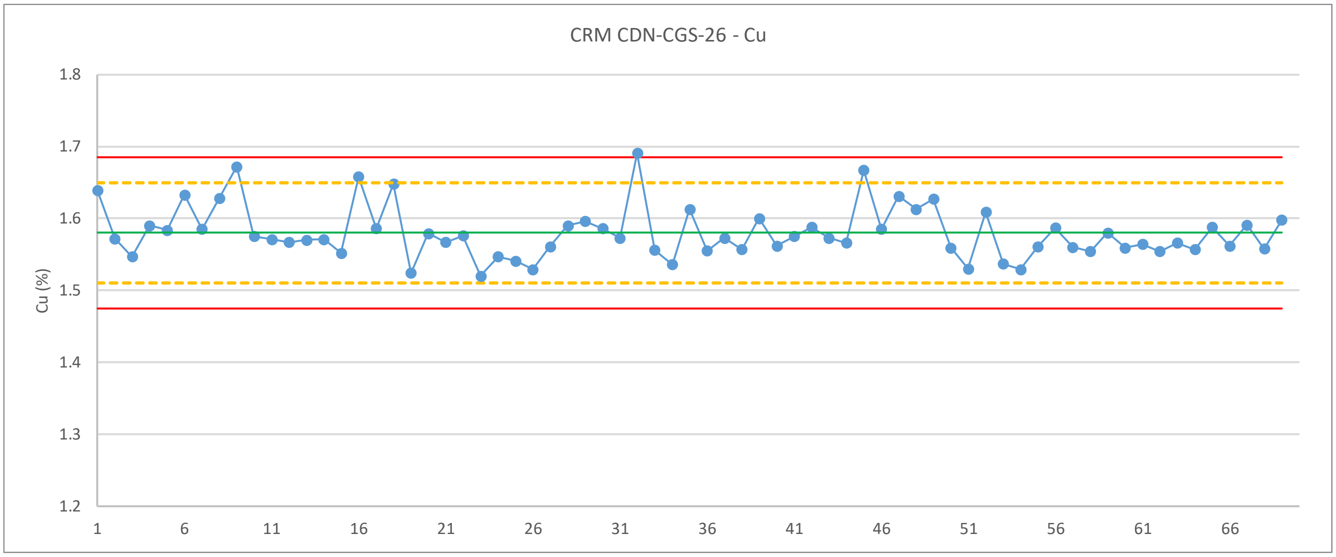
- The red line indicates three times the SD.
- The yellow dashed line indicates two times the SD.
- The green line represents the expected value.
- The blue dots connected by the solid blue line are the results of the assayed samples.

Figure 11.1 CRM CDN-CGS-26 Au



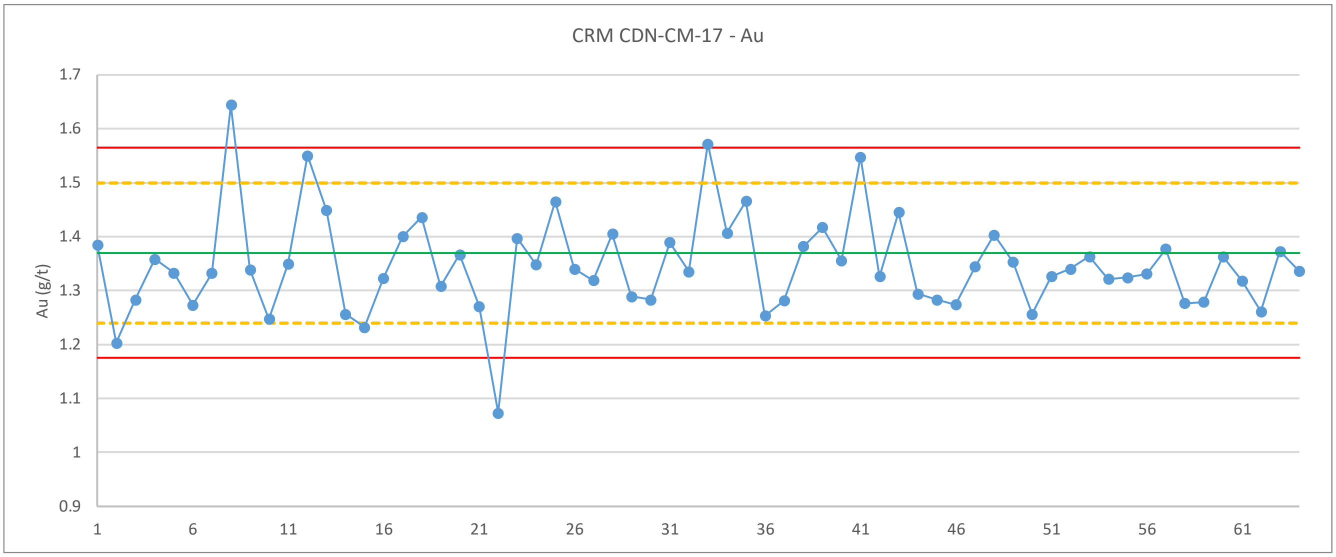
Source: AMC from Eldorado data.

Figure 11.2 CRM CDN-CGS-26 Cu



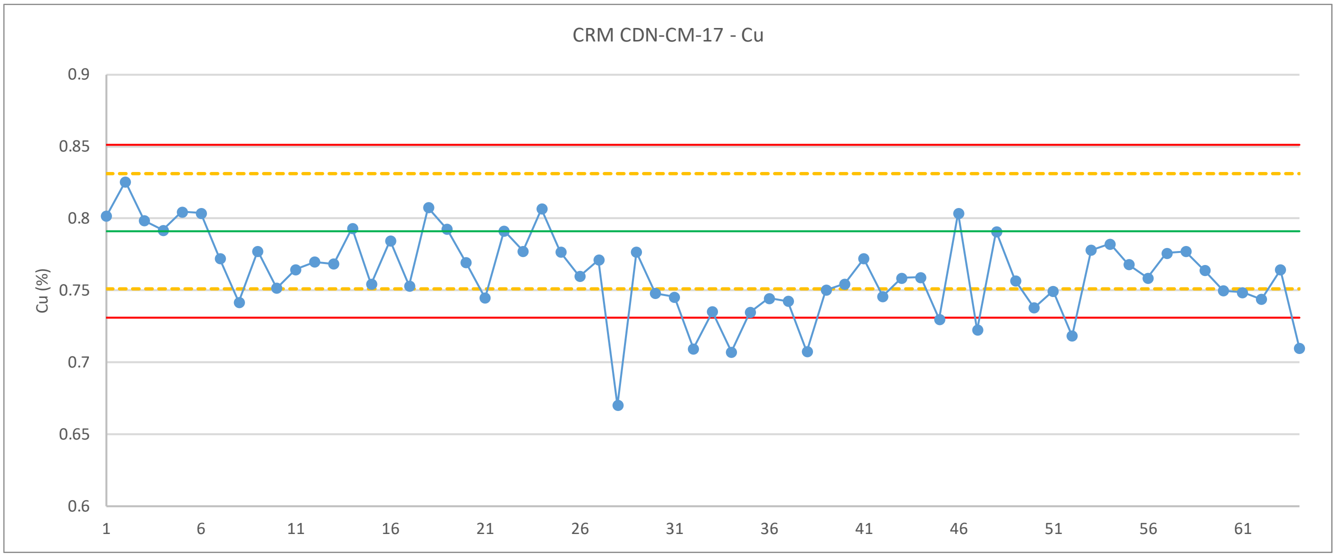
Source: AMC from Eldorado data.

Figure 11.3 CRM CDN-CM-17 Au



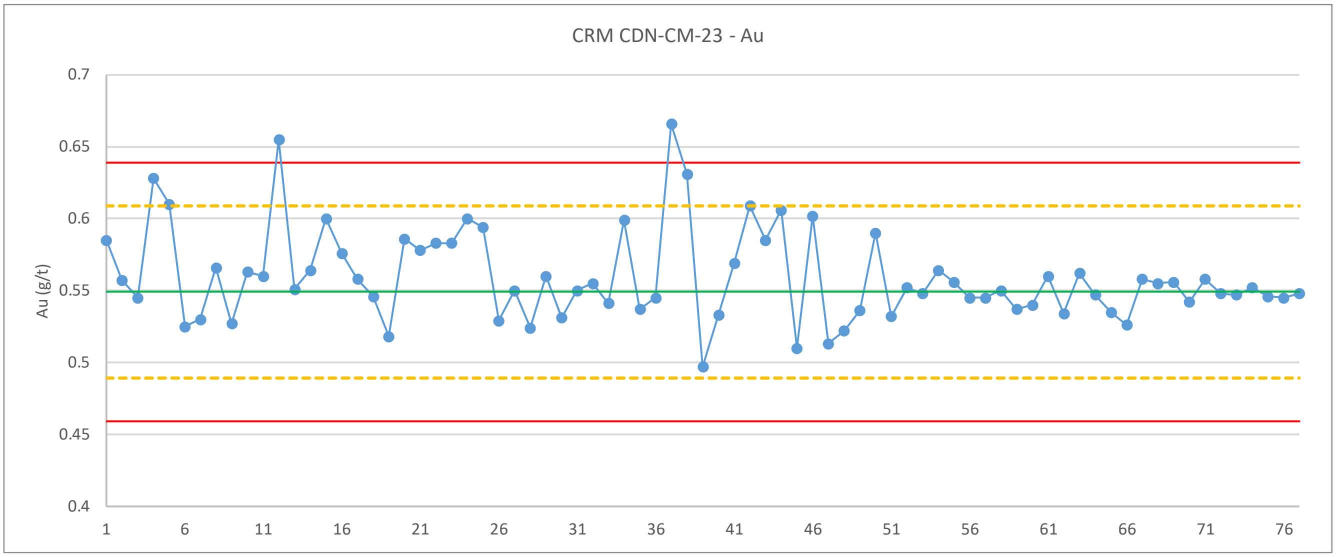
Source: AMC from Eldorado data.

Figure 11.4 CRM CDN-CM-17 Cu



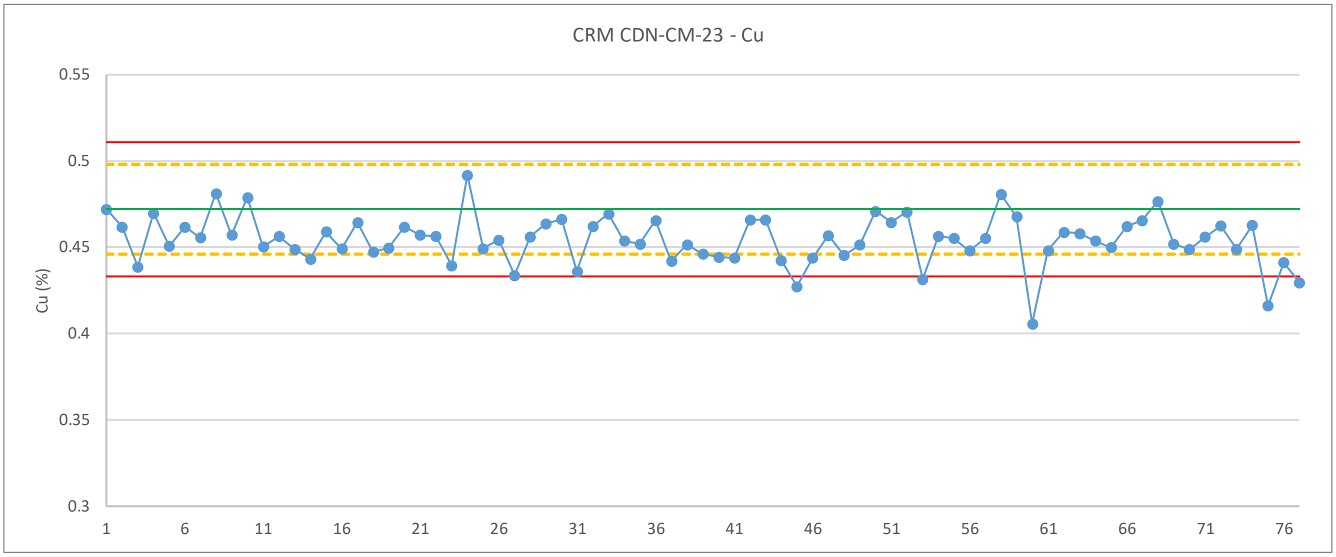
Source: AMC from Eldorado data.

Figure 11.5 CRM CDN-CM-23 Au



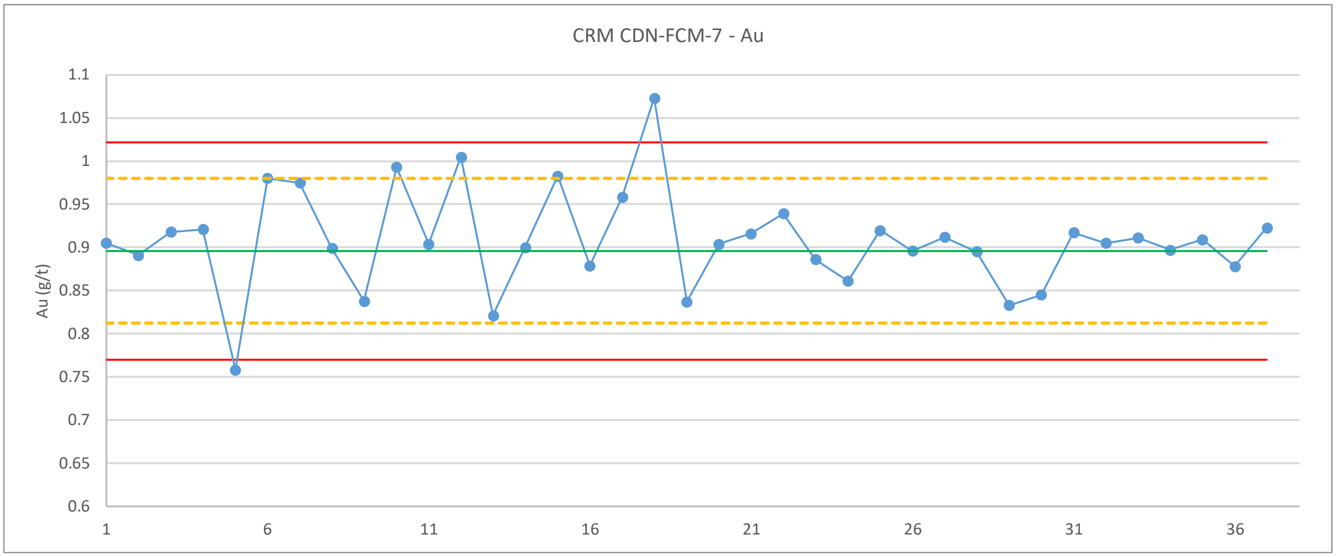
Source: AMC from Eldorado data.

Figure 11.6 CRM CDN-CM-23 Cu



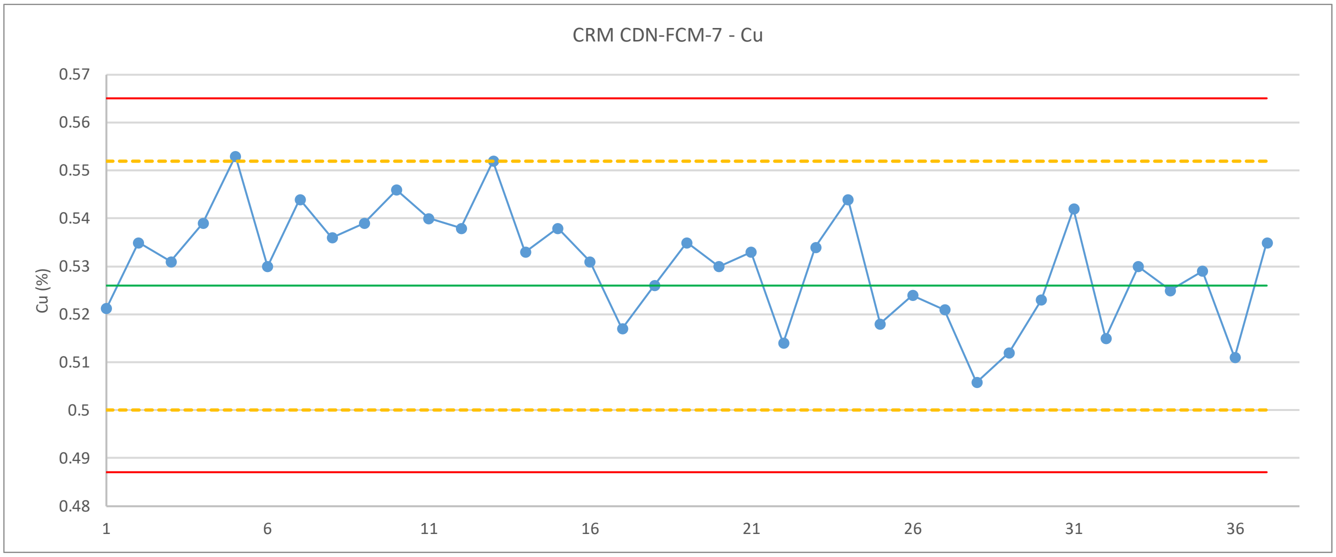
Source: AMC from Eldorado data.

Figure 11.7 CRM CDN-FCM-7 Au



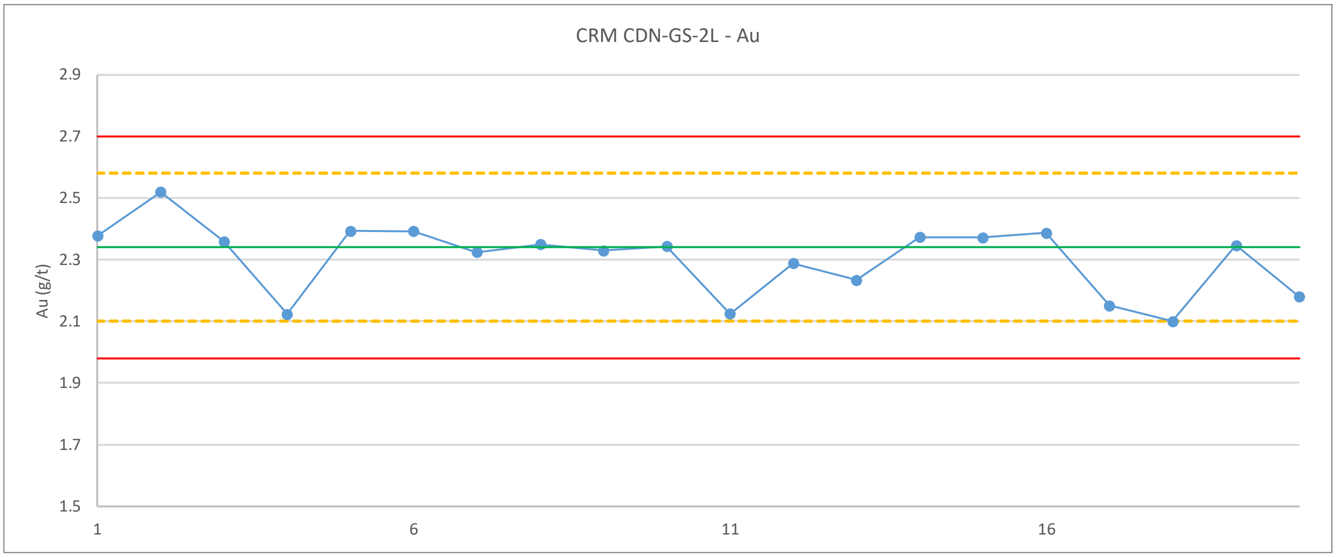
Source: AMC from Eldorado data.

Figure 11.8 CRM CDN-FCM-7 Cu



Source: AMC from Eldorado data.

Figure 11.9 CDN-GS-2L Au



Source: AMC from Eldorado data.

Table 11.3 summarizes the results of the CRM assays. It contains the number of samples for each CRM that exceeded either 2 or 3 SDs from the expected value. Based on the pass / fail criteria used by Eldorado and AMC, all CRMs except CDN-FCM-7 (Cu) and CDN-GS-2L (Au) had at least one fail.

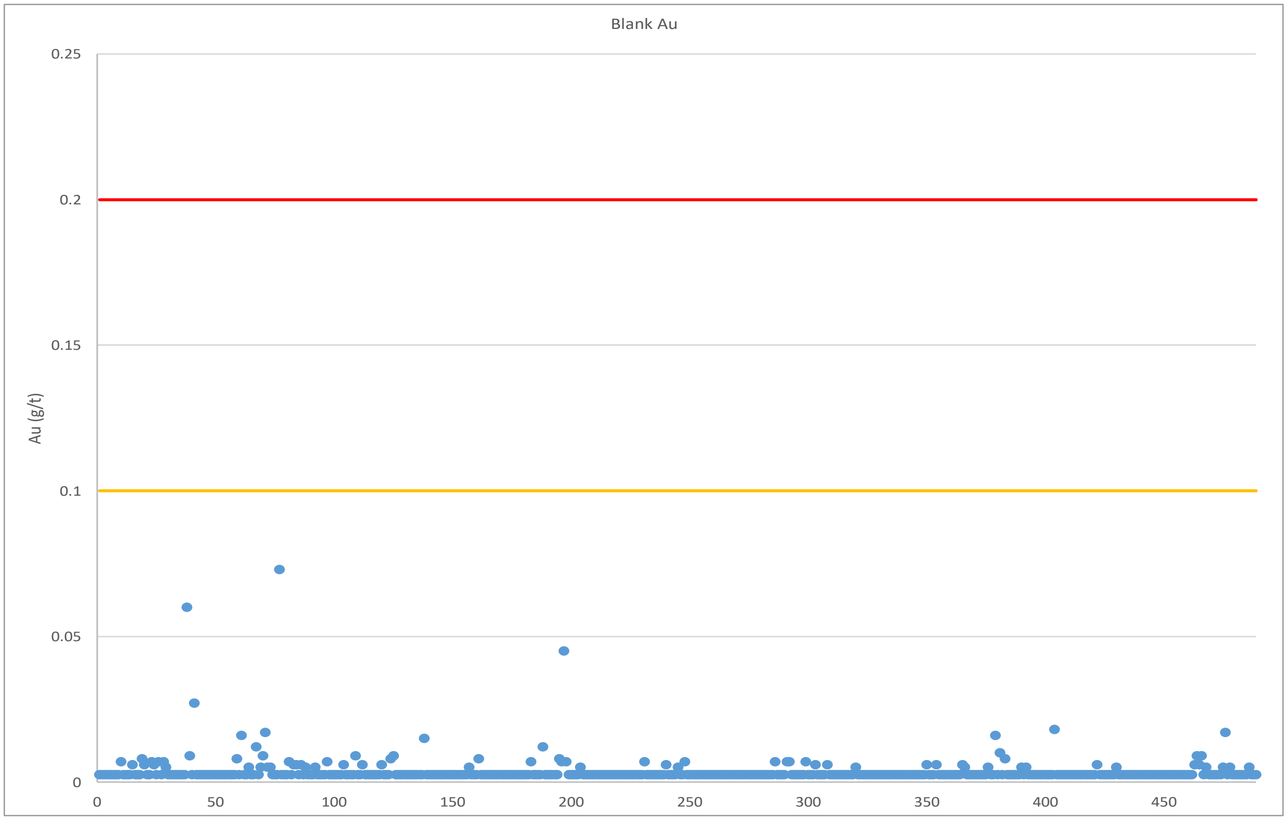
Table 11.3 Summary of CRM assays outside 2 or 3 standard deviations

CRM	No. assays	No. assays outside 2 or 3 standard deviations			
		Au (g/t)		Cu (%)	
		> 2 SD	> 3 SD	> 2 SD	> 3 SD
CDN-CGS-26	67	5	3	3	1
CDN-CM-13	31	3	-	3	3
CDN-CM-17	58	4	3	14	8
CDN-CM-23	76	2	2	11	5
CDN-FCM-6	39	2	3	4	4
CDN-FCM-7	37	2	2	-	-
CDN-GS-12A	31	7	1		
CDN-GS-15B	23	1	1		
CDN-GS-2L	19	-	-		
CDN-GS-P3C	17	2	2		

11.4.2 Assay results for blank samples

Blank samples monitor for sample contamination during sample preparation and the assay process. Eldorado samples were analyzed using a method with an analytical detection limit (ADL) for gold of 0.01 g/t. A threshold of 0.1 g/t (10 times the detection limit) was used by Eldorado as the failure limit for blank material. Based on these parameters, samples show no evidence of contamination (Figure 11.10).

Figure 11.10 Results of blank sample analysis



Note: Legend same as for CRM Figure 11.1 to Figure 11.9.
Source: AMC from Eldorado data.

11.4.3 Assay results for duplicate samples

Duplicate samples monitor sampling variance (including that arising from sample preparation), analytical variance and geological variance.

Eldorado regularly submitted coarse reject duplicates to monitor analytical precision. A total of 408 coarse reject duplicates was submitted between 2012 and 2013.

Duplicates should constitute around 5% of the samples submitted to the laboratory. Approximately 6% of samples submitted by Eldorado were coarse duplicates.

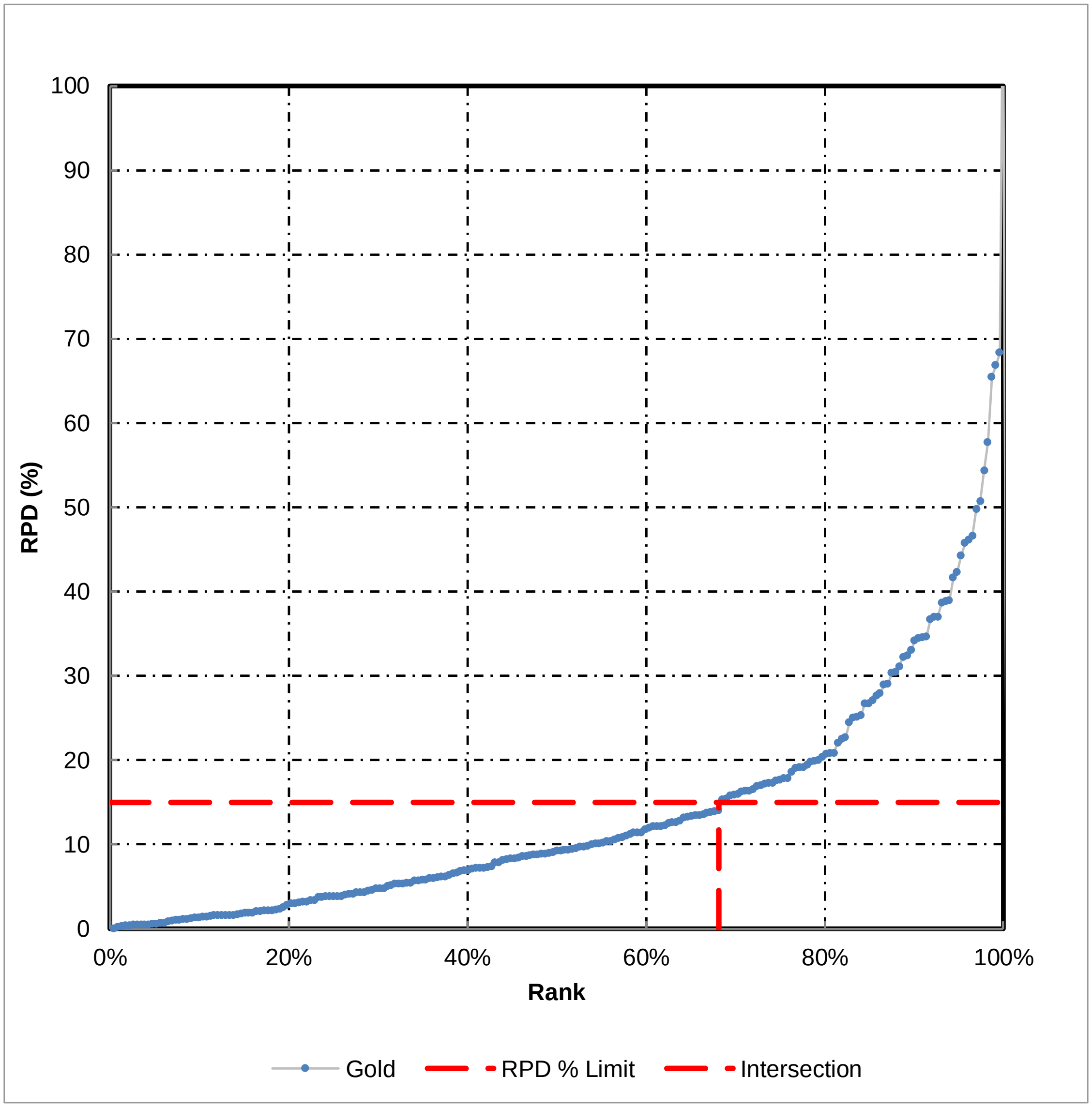
Unmineralized samples should not be sent as duplicates because assays near the detection limit are commonly inaccurate.

Duplicate data can be viewed on a scatterplot but should also be compared using the relative paired difference (RPD) plot. This method measures the absolute difference between a sample and its duplicate. It is desirable to achieve 80% to 85% of the pairs having less than 15% RPD between the original assay and check assay (Stoker, 2006). Sample pairs should be excluded from the analysis if the combined mean of the pair is less than 15 times the detection limit (Kaufman and Stoker 2009). Removing the low values ensures that there is no undue influence on the RPD plots due to the higher variance of grades likely near to the detection limit, where precision becomes poorer (Long et al., 1997).

To generate the RPD plots, AMC used a detection limit of 0.01 g/t Au and 0.001% Cu.

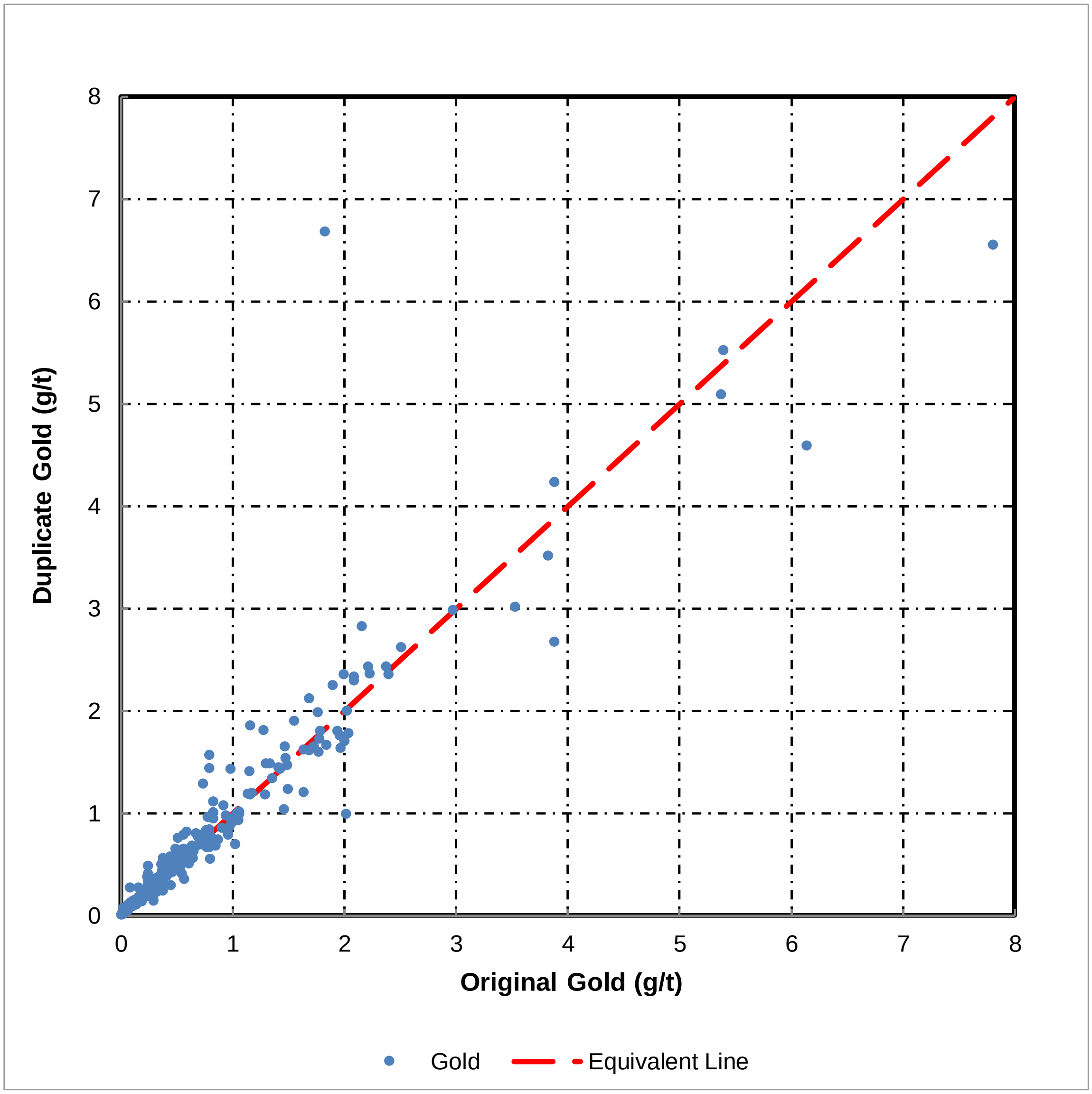
Figure 11.11 and Figure 11.12 show the RPD and scatterplots for the gold coarse reject duplicates. Table 11.4 presents the summary statistics for the gold duplicate data.

Figure 11.11 RPD plot for coarse reject duplicates Au



Source: AMC from Eldorado data.

Figure 11.12 Scatter plot for coarse reject duplicates Au



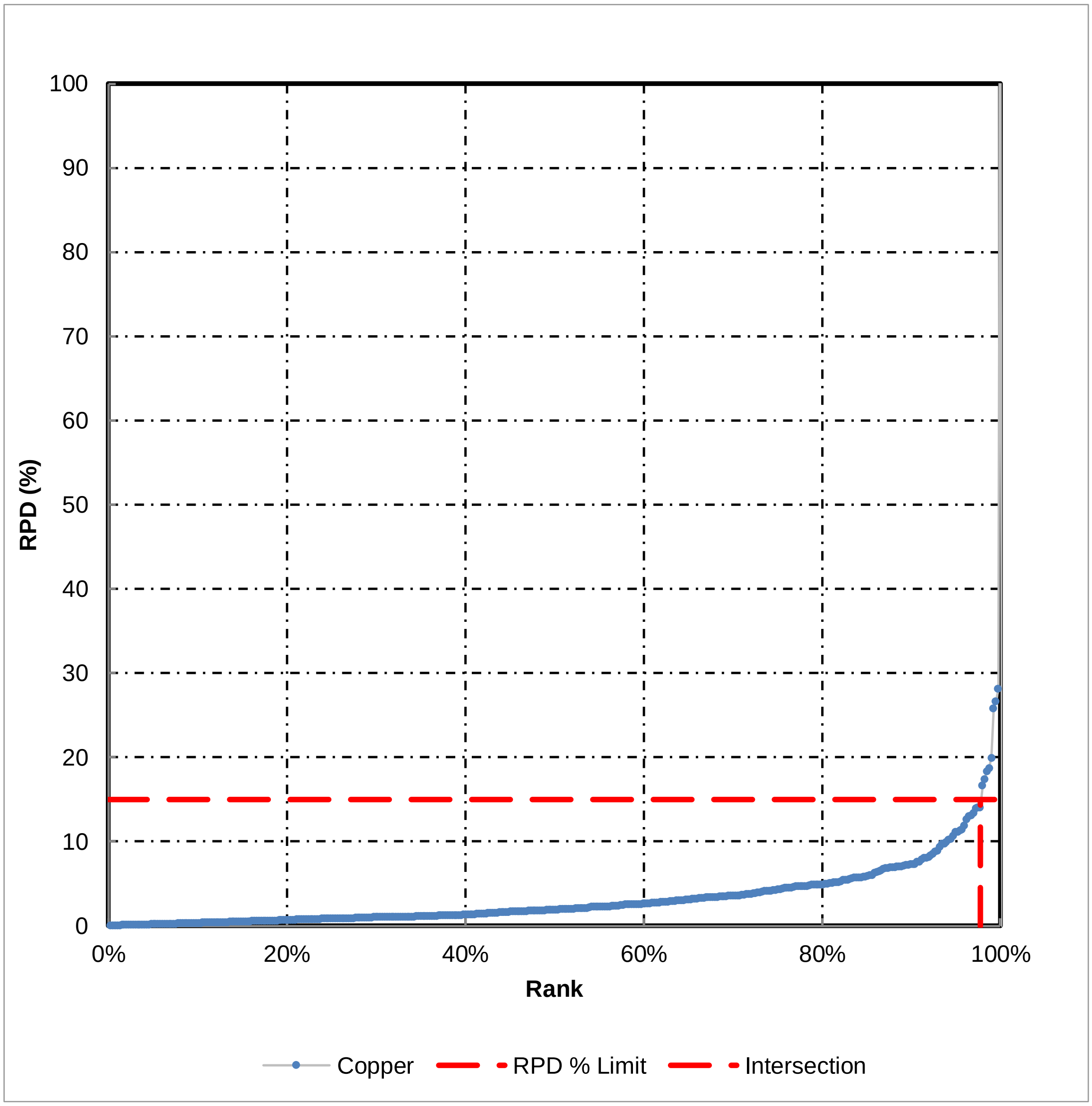
Source: AMC from Eldorado data.

Table 11.4 Summary statistics for coarse reject duplicates Au

Gold (g/t)	Primary	Duplicate
Number of samples	408	408
Number of samples > 15 times detection limit	232	232
Mean	0.54	0.55
Maximum	7.81	6.67
Minimum	0.01	0.01
Pop Std Dev.	0.86	0.86
CV	1.61	1.58
Cor. Coeff.	0.94	
Bias (all data)	-2.23%	
Percent samples >15% RPD	**68.10**	

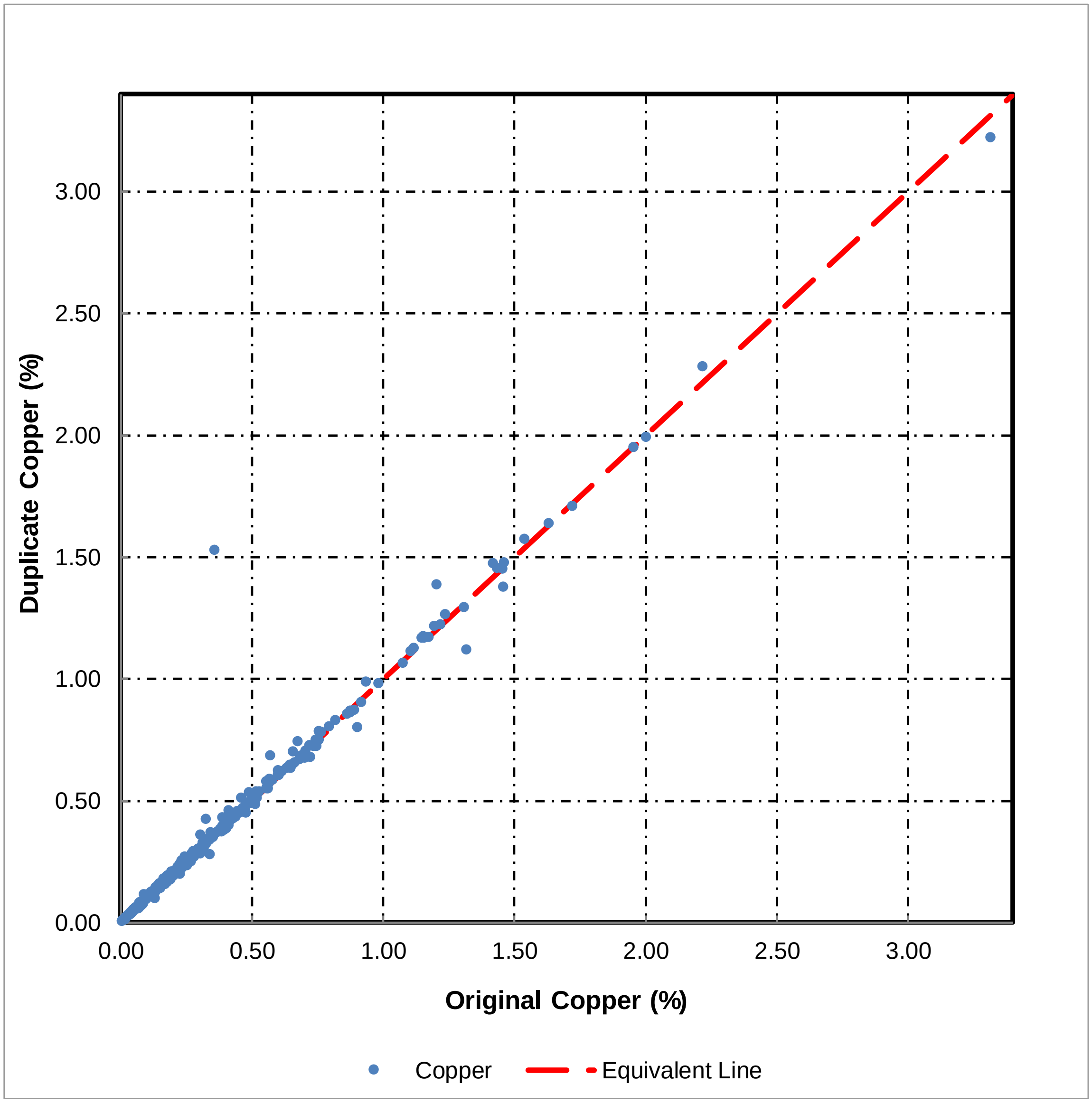
Figure 11.13 and Figure 11.14 show the RPD and scatterplots for the copper coarse reject duplicates. Table 11.5 presents the summary statistics for the copper duplicate data.

Figure 11.13 RPD plot for coarse reject duplicates Cu



Source: AMC from Eldorado data.

Figure 11.14 Scatter plot for coarse reject duplicates Cu



Source: AMC from Eldorado data.

Table 11.5 Summary statistics for coarse reject duplicates Cu

Copper (%)	Primary	Duplicate
Number of samples	408	408
Number of samples > 15 times detection limit	397	397
Mean	0.33	0.33
Maximum	3.32	3.22
Minimum	0.00	0.01
Pop Std Dev.	0.38	0.39
CV	1.16	1.16
Cor. Coeff	0.99	
Bias (all data)	-1.18%	
Percent samples >15% RPD	**97.73**	

The following is noted in regard to the coarse reject analysis:

- Only 68% of samples are above the 15% RPD line for gold.
- There is moderate deviation from the 1:1 line for gold assays above 1 g/t.
- No significant bias is apparent for the coarse reject duplicates for gold.
- 98% of samples are above the 15% RPD line for copper.
- There is little deviation from the 1:1 line for copper.
- No significant bias is apparent for the coarse reject duplicates for copper.

It is recommended to include pulp duplicates in addition to coarse duplicates in future QA/QC programs. Repeat pulp assays quantify the precision of the analytical procedure. It is considered by Eldorado that the pattern in the gold coarse reject duplicates represents readily liberated gold grains throughout the sample preparation process. After investigation, the QP recommends that Eldorado consider modifying the sample preparation protocol to try and minimize this effect.

11.4.4 Results for external check assays

No samples were sent to an external laboratory. The QP recommends that any future QA/QC program includes such samples.

11.5 Conclusions

In the QP's opinion, the sampling, sample preparation, security, and analytical procedures adopted by Eldorado for its drilling programs meet accepted industry standards, and the QA/QC results confirm that the assay results may be relied upon for Mineral Resource estimation purposes.

12 Data verification

12.1 Introduction

Data verification was carried out by the QP or under his supervision, and comprised the following:

- Review of available data.
- Review of QA/QC protocols, and QA/QC performance (documented in Section 11).
- Discussions and interviews with Eldorado personnel regarding data collection and database compilation.
- Discussions with Eldorado regarding database checks and validation.
- A site visit to the project on 28 May 2019.
- Cross checking of assays within the Eldorado sample database with original lab certificates, (done under the QP's supervision).

As drilling on the property has not been done since 2013, the QP was unable to observe Eldorado procedures first-hand. The QP considers the discussions carried out and the review of data an adequate assessment of the Skouries database.

12.2 Database checks

Checks to the entire drillhole database were undertaken by Eldorado, which consisted of checks of original assay certificates and survey data. Any discrepancies found were corrected and incorporated into the resource database. Eldorado therefore concluded that the data supporting the Skouries Project resource work is sufficiently free of error to be adequate for estimation. The QP made checks of the assay certificates against the database, which is discussed below.

Data verification carried out by the QP on the database consisted of a check of the assay data from the Eldorado drilling completed in 2012 and 2013. The data set provided consisted of the assay certificates from Acme, and these values were checked against what was recorded in the database as being used for the estimation. There was a total of 7,128 records in the database and 1,216 of these were verified from the certificates, which is 17.1% of the total. There were no errors found.

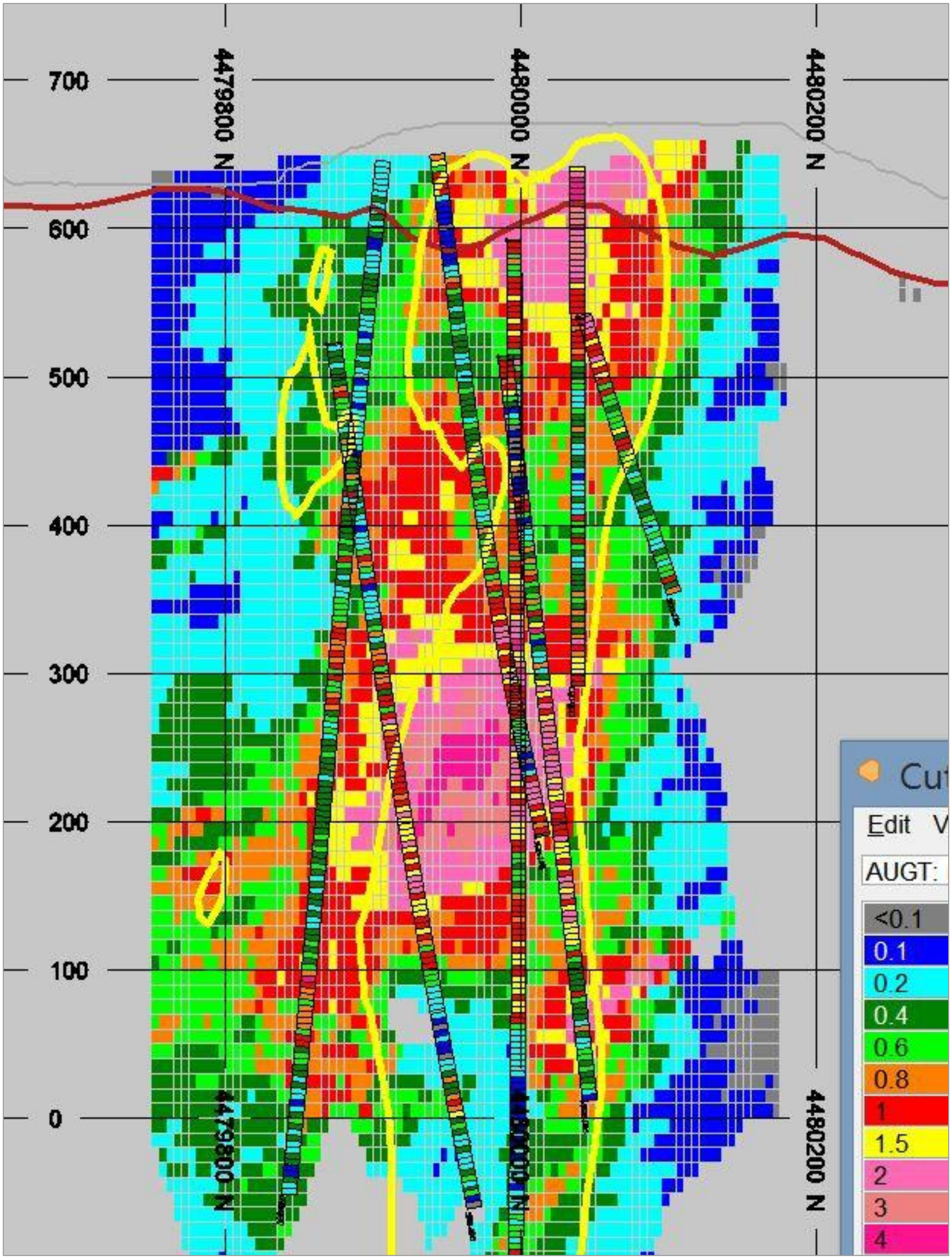
12.3 Analysis of confirmation drilling

The historical TVX data was available to Eldorado on purchase of the Property, however the core from the drilling was not available as it had been shipped overseas and ultimately destroyed. Because of this, it was decided by Eldorado to drill ten deep holes as a confirmatory measure.

This program of confirmation drilling was completed in 2013. These holes redrilled the mineralization previously tested by the 1990s work by TVX from which no core remained. Eldorado compared the two data sets by re-estimating the Mineral Resource using the 1990s drillholes and 2012 infill drilling and then visually comparing the generated block model to the confirmation drillhole assay results. These comparisons are shown in Figure 12.1 for the gold estimates and in Figure 12.2 for the copper estimates. Note the yellow lines in these figures are the outline of the porphyry body. The confirmation drillhole grades match the block model grades very well on a visual basis. Thus, Eldorado was able to verify the results obtained from the 1996 – 98 drill campaign, despite having none of that drillcore available. The QP concurs with the Eldorado verification, given the confirmation drilling backed up the earlier results.

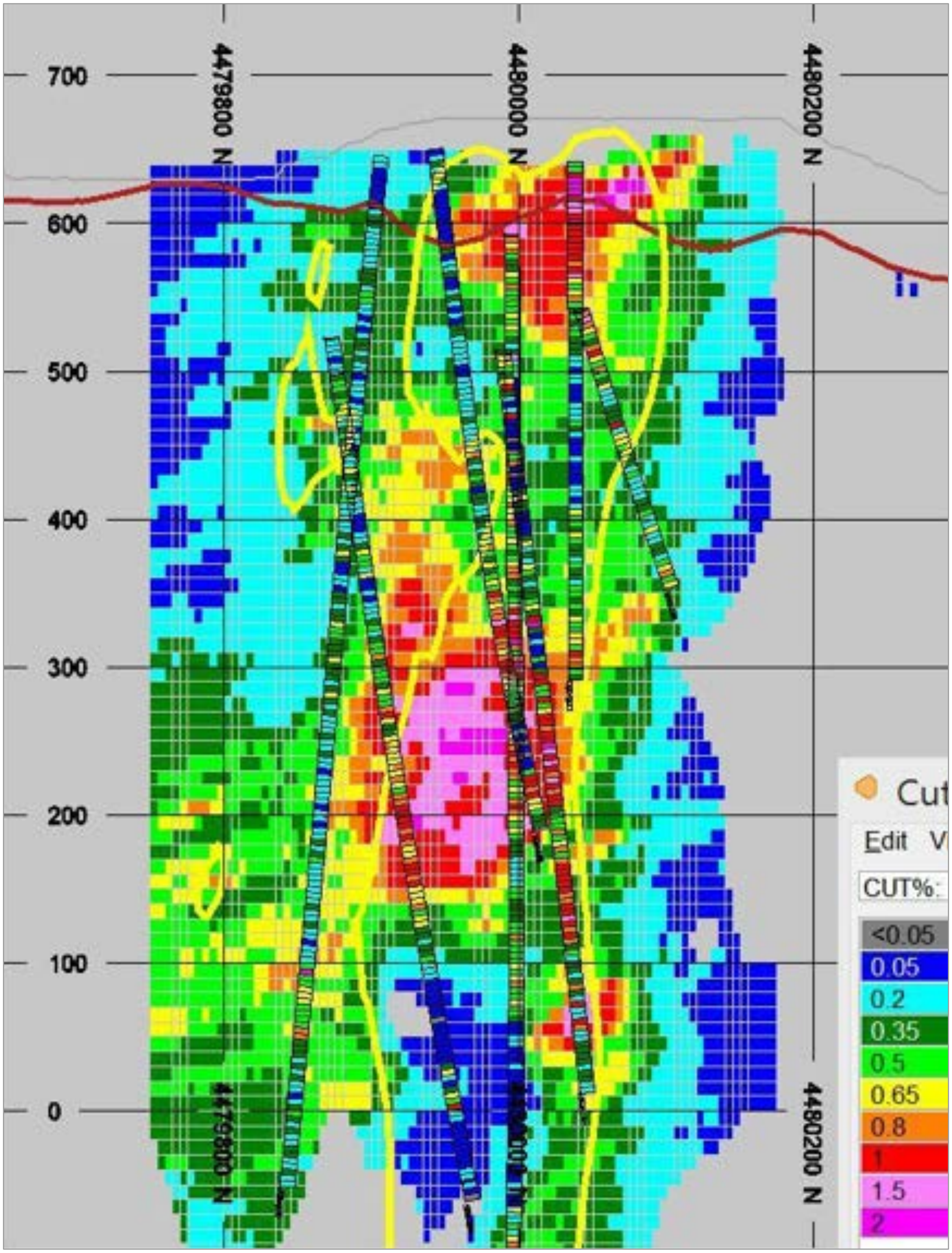
Note that the data from the ten deep holes were not included in any estimate, and only used to assess the resultant block grades in the model as a confirmation / verification activity.

Figure 12.1 Comparison of gold grades for confirmation drillholes and block model



Source: Eldorado 2022.

Figure 12.2 Comparison of copper grades for verification drillholes and block model



Source: Eldorado 2022.

12.4 Concluding statement

In the opinion of the QP the exploration data is acceptable for Mineral Resource estimation.

13 Mineral processing and metallurgical testing

13.1 Metallurgical testwork

Metallurgical testwork and studies were performed by Lakefield Research, Canada on composites selected from core samples of the major rock types covering mineralogy, grinding and flotation. This testing was carried out to support the original 2007 design completed by Aker Kvaerner. Based upon this information, the criteria for process plant and infrastructure design were established.

Additional testwork was completed by Outotec in 2007, mostly at its laboratory in Pori, Finland, to give additional design confidence. This included flash flotation, gravity gold recovery, concentrate settling and filtration.

Further supplementary testwork was undertaken by FLS Knelson in 2013 on gravity gold recovery and by Wardell Armstrong in 2015 on flotation concentrate. Solvay (formerly Cytec) in 2016 and Bureau Veritas Commodities Canada in 2017 worked on selective flotation of copper from pyrite-rich ore. In 2014, Orway Mineral Consultants (OMC) reviewed the testwork conducted by Aker Kvaerner to design the Skouries grinding circuit and conducted comminution circuit modelling studies using circuit simulations.

Mineralization of the sulphide ore primarily comprises chalcopyrite veinlets with subordinate bornite, disseminated chalcopyrite and bornite. Variable amounts of digenite, chalcocite, covellite, molybdenite and pyrite occur together with trace amounts of galena and sphalerite. Magnetite occurs both as disseminations and in quartz veinlets. Gold mineralization occurs as native gold associated with gangue minerals and ranges in size from a few microns (µm) to 160 µm. Gold also occurs as blebs within sulphides, particularly in bornite and chalcocite. It correlates strongly with copper. Palladium (Pd) was identified during metallurgical testing and could add by-product value to the ore. The oxide zone occurs from surface to 30 m to 70 m depth, and occasionally deeper, and consists mainly of malachite, cuprite, secondary chalcocite and minor azurite, covellite, digenite and native copper.

Five ore zone samples representing open-pit oxide, mixed, and underground fresh ore were tested by Aker Kvaerner. JKTech Drop Weight Tests and semi-autogenous grinding (SAG) Power Index tests were conducted to determine SAG mill design parameters. Bond comminution tests were conducted to determine standard ball mill parameters, such as Bond Ball Mill Work Index and Bond Abrasion Index. Using these test data, a SAG mill - ball mill - pebble crusher (SABC) circuit capable of 8 Mtpa throughput while treating softer ore, and 6.75 Mtpa while treating harder underground fresh ore, was designed and supplied by Outotec.

OMC confirmed the 8 Mtpa throughput capability of the grinding circuit in SABC configuration while treating oxide ore and mixed (oxide and fresh) ore.

Extensive flotation testwork was undertaken to enable metal recoveries to be correlated with the mine plan. This was based on systematic sampling to verify metallurgical response throughout the Mineral Resource and to understand variability. The final stage of the laboratory flotation testwork established the response of the open pit sulphide and oxide ores.

The oxide ore testwork was divided into three material types: high oxide (HO), medium oxide (MO), and low oxide (LO), depending on the acid soluble copper content. This was determined by citric acid leach / analysis (CuS) and sulphuric acid leach / analysis (CuL). The degree of oxidation was defined by the ratio of dissolved copper from these acid leaches / analyses to the total copper analysis (CuT). The selected ranges for the three material types are as follows:

- Low oxide
  - CuS / CuT ratio from 0.05 to 0.17
  - CuL / CuT ratio from 0.17 to 0.38
- Medium oxide
  - CuS / CuT ratio from 0.17 to 0.50
  - CuL / CuT ratio from 0.38 to 0.77
- High oxide
  - CuS / CuT ratio > 0.50
  - CuL / CuT ratio > 0.77

The open pit sulphide ores exhibited similar flotation characteristics to the underground sulphide ore. The open pit sulphide ore samples were divided into those associated with the oxide ore (low sulphides - LS), and those unassociated with the oxide ore (high sulphides - HS).

Following the establishment of best flotation conditions, one locked cycle test was performed on a representative ore sample from each of the three oxide ore types. In all tests, the flotation stages mimicked the same flowsheets as that used during the earlier testing of primary sulphide ores. Sodium hydrosulphide was used as a sulphidizer agent to float the oxide ores.

The oxide ores to be processed in the first year of operation and re-handled in Phase 2 have significantly lower copper and gold recoveries compared to the sulphide mineralization. These recoveries are estimated to be approximately 50% for copper and 70% for gold. These values can be compared to the life-of-mine (LOM) average recoveries of 87.2% for copper and 82.4% for gold when sulphide ore is floated. The testwork has shown that the oxide copper minerals and the associated gold can be recovered by conventional sulphidizer activated flotation. Mineralogical investigations indicated that the oxide ore copper losses were mainly due to very fine sulphides locked in gangue rather than non-floating oxide copper minerals. It has been deemed to not be cost effective to grind to the fineness required to liberate these locked copper sulphides, particularly as the timeframe involved will be limited.

The results of all the locked cycle flotation tests for both sulphide and oxide ore samples were evaluated to establish a relationship between recovery and head grade for both copper and gold. This led to the development by Aker Kvaerner of equations to predict the expected recoveries of copper and gold to flotation concentrate as a function of the ore head grades. Although largely based on flotation data produced from testwork by Lakefield Research, these recovery equations are accepted for the process plant as currently proposed, i.e., including any gold recovered by gravity concentration circuits. The equations developed are of the mathematical form:

$$y = a - ce^{(-bz)}$$

Where y represents copper or gold recovery; a, b, and c are constants, e is natural e, and z is the respective copper or gold head grade.

The equation passes through the origin at zero recovery and zero grade and places a limit to the maximum recovery attainable. The Aker Kvaerner-derived recovery equations were further developed by SRK in its mining pre-feasibility study of November 2005. Their current forms are given below:

- Copper Recovery [oxide] (%) = 43.4 – 41.0 x e $^{(-338 \text{ x Cu Head Grade \%})}$
- Gold Recovery [oxide] (%) = 75.1 – 22.0 x e $^{(-1.2 \text{ x Au Head Grade g/t})}$
- Copper Recovery [non-oxide] (%) = 99.4 – 41.0 x e $^{(-338 \text{ x Cu Head Grade \%})}$
- Gold Recovery [non-oxide] (%) = 92.6 – 22.0 x e $^{(-1.2 \text{ x Au Head Grade g/t})}$

The methodology used in deriving these equations is described in the Aker Kvaerner 2007 Cost and Definition study.

These equations have been applied to the Project mine planning and are accepted as projecting reasonable values for copper and gold recoveries.

Initial preliminary bench scale gravity gold concentration tests were carried out by South-West Metallurgical and demonstrated the viability of recovering free gold from the primary grinding circuit.

Further gold gravity concentration testwork was undertaken in 2013 by FLSmidth and Knelson and confirmed the applicability of centrifugal concentrators for gold recovery in both the primary grinding circuit and the regrind circuit. This testwork is the basis for the current gravity concentration circuit design; however, further testing in 2021 has indicated that the gravity concentration circuit will not be required to meet designed plant performance, and, at this time, the gravity concentration circuit will not be installed. Eldorado will conduct detailed testing in the plant after start-up to evaluate the need for the gravity concentration circuit to reach designed performance levels.

The Outotec testwork in 2007 was focused on evaluating the installation of a flash flotation unit cell to treat the primary grinding circuit cyclone underflow. The objective was to recover gold and copper in coarse mineral particles before over-grinding may occur. The testwork showed that flash flotation configured in this manner could recover mineralized values as predicted but would probably not significantly impact on overall gold / copper recoveries. Therefore, although space has been allocated in the grinding area for a unit cell for flash flotation, it is not included in the current design. Nevertheless, retrofit for a flash flotation cell can take place if this circuit is proved to be beneficial in later years of the Project.

The testwork also demonstrated that the number of flotation concentrate cleaning stages needs to be increased from two to three in order to achieve the targeted concentrate grade of 26% copper during periods when low grade ore is processed.

The 2015 testwork by Wardell Armstrong International investigated reduction of the fluoride content in the copper flotation concentrate. The testwork concluded that the use of guar gum as a slimes / clay dispersant / depressant in the copper cleaning circuit would keep the fluoride levels in the copper concentrate at or below the expected smelter penalty level.

14 Mineral Resource estimates

14.1 Introduction

The Mineral Resource estimates for the Skouries copper-gold deposit were originally prepared by Ms Susan Lomas of Lions Gate Geological Consulting Inc. in 2014. This work was reviewed and modified by Eldorado and its employee Mr Stephen Juras P.Geo. took responsibility for the model and reporting. Latterly Mr Sean McKinley P.Geo. also of Eldorado has acted as internal QP. For this report, Ms. Dinara Nussipakynova of AMC has reviewed and validated the model and Mr J.M. Shannon of AMC has taken responsibility for the Mineral Resource estimate.

The Skouries modelling was completed using GEMS software and the estimation was carried out by Eldorado in MineSight®. The validation and review of the modelling by AMC was carried out in Datamine™ software. The Mineral Resource estimate for the Skouries deposit used data from surface diamond drillholes. The block model cell size is 5 m east by 5 m north by 10 m high.

The Skouries Mineral Resource estimate, on 30 September 2021, is shown in Table 14.1. This is a combined summary for the open pit and underground. The open pit Mineral Resources are reported at a cut-off of 0.3 g/t gold equivalent (AuEq) and the underground portion at a cut-off of 0.7 g/t AuEq. The estimates for open pit and underground are reported separately in Section 14.13.

Table 14.1 Summary of Mineral Resources, on 30 September 2021

Category	Tonnes (kt)	Au (g/t)	Cu (%)	Contained Au (k oz)	Contained Cu (k tonnes)
Measured	90,714	0.85	0.51	2,479	466
Indicated	149,260	0.53	0.44	2,551	652
Measured & Indicated	239,974	0.65	0.47	5,030	1,118
Inferred	67,657	0.37	0.40	814	267

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Open pit Mineral Resources are constrained by a semi-optimized pit that is strongly permit and crown pillar constrained and are reported at a 0.3 g/t AuEq cut-off.
- Underground Mineral Resources are those outside the pit shell and are reported at a 0.70 g/t AuEq cut-off.
- AuEq = Au g/t + 1.25 * Cu%, based on US$1,800/oz Au and US$3.50/lb Cu, and recoveries of 86.7% for gold and 91.5% for copper.
- Mineral Resources are stated inclusive of Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Numbers may not compute exactly due to rounding.

Source: Eldorado, re-reported by AMC and approved by the QP.

The QP is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, mining, metallurgical, infrastructure, or other relevant factors other than those disclosed herein.

The main steps of the geological estimation process are discussed below.

14.2 Data used

Surface diamond drillholes alone were used to model the deposit. No drilling has been carried out since 2013 and a summary of the drilling is shown in Table 10.1. Confirmation drillholes completed by Eldorado to assess historical data were not included in the estimate. A total of 155 drillholes for 79,154 m were used for the Mineral Resource estimate.

14.3 Lithological domains

A three-dimensional (3D) geological model of the Skouries porphyry was constructed by Eldorado to aid subsequent grade modelling. This model was not used to constrain mineralization (discussed below). The porphyry model is shown as the yellow outline in Figure 14.1 to Figure 14.4.

14.4 Mineralization domains

As with many porphyry deposits, using only lithology or alteration-based domains to constrain grade interpolation is not appropriate due to the common overlapping nature of copper and gold mineralization. It has been Eldorado's experience in these deposit types to create 3D mineralized or grade domains based on initial outlines derived by a method of probability assisted constrained kriging (PACK). The threshold values of 0.10% Cu and 0.2 g/t Au were determined by inspection of histograms and probability curves as well as indicator variography. Shell outline selection was done by inspecting contoured probability values. Inspection in plan and section showed a high degree of similarity between the Cu and Au shells. Eldorado decided to interpolate both grades inside a single PACK shell to avoid the potential of some model blocks having only one grade (Cu or Au) interpolated. The Cu shell was chosen to be the interpolation domain for Cu and Au grades.

14.5 Data analysis

The lithologic and mineralized domains were reviewed to determine appropriate estimation or grade interpolation parameters. Several different procedures were applied to the data to determine whether statistically distinct domains needed to be defined. The lithology categories porphyry and schist (including all non-porphyry units) were investigated within the mineralized shell.

Descriptive statistics, histograms, and cumulative probability plots, box plots, and contact plots were completed for copper and gold. Results obtained were used to guide the construction of the block model and the development of estimation plans. The data analyses were conducted on assay data that were converted to 4 m downhole composites. The statistical properties from this analysis are summarized in Table 14.2.

Copper and gold grades are highest in the porphyry. The gold to copper ratios are also markedly different between the intrusive and non-intrusive units. Within the porphyry the Au:Cu ratio is close to 2:1, whereas in the schist (or all non-intrusive units) the ratio is virtually 1:1. Generally, the coefficient of variation (CV) values for copper in all units are relatively low, reflecting the porphyry style mineralization of the deposit. Gold CV values are higher, especially in the schist unit, reflecting some influence of local extreme grades.

Table 14.2 Skouries deposit statistics for 4 m composites – Cu and Au data

Lithology	Mean	CV	q25	q50	q75	Max	No. of comps
Within PACK Shell – Cu %							
Porphyry	0.65	0.81	0.05	0.51	2.77	6.27	3,356
Schist	0.33	0.80	0.04	0.26	1.86	13.22	13,378
All Units	0.39	0.91	0.04	0.30	2.84	13.22	16,734
Within PACK Shell – Au g/t							
Porphyry	1.21	1.19	0.06	0.87	7.02	28.28	3,356
Schist	0.38	1.35	0.02	0.23	3.94	20.05	13,378
All Units	0.55	1.57	0.02	0.29	7.87	28.28	16,734
Outside PACK Shell							
All Units – Cu %	0.06	0.65	0.02	0.06	0.14	0.71	1,186
All Units – Au g/t	0.05	2.65	0.01	0.04	0.17	4.49	1,186

Note: This table is based on an uncapped composite file provided by Eldorado.
Source: Eldorado, checked by AMC.

14.6 Evaluation of extreme grades

Extreme grades were examined for copper and gold, mainly by histograms and cumulative probability plots. Generally, the distributions do not indicate a problem with extreme grades for copper. For gold, local areas display extreme grades. These were mitigated by a gold grade cap equal to 20 g/t, applied to the assay data prior to compositing.

14.7 Variography

Variography, a continuation of data analysis, is the study of the spatial variability of an attribute. Eldorado prefers to use a correlogram, rather than the traditional variogram, because it is less sensitive to outliers and is normalized to the variance of data used for a given lag. Correlograms were calculated for copper and gold inside the copper PACK shell. Correlogram model parameters and orientation data of rotated axes are shown in Table 14.3 and Table 14.4.

Copper and gold display two structures: a long-ranged, SW-NE trending, near vertical to steeply E-dipping, steeply W-plunging structure, and a much shorter-ranged structure, especially for gold, that is nearly omni-directional. The nugget effects are low for both, reflective of the deposit type.

Table 14.3 Correlogram parameters for Skouries deposit

	Model	Nugget Co	Sills		Rotation angles						Ranges					
			C1	C2	Z1	Y1'	Z1''	Z2	Y2'	Z2''	Z1	Y1	X1	Z2	Y2	X2
PACK Shell - Cu	SPH	0.250	0.251	0.499	-41	24	3	-109	-12	66	18	33	36	289	170	124
PACK Shell – Au	SPH	0.250	0.279	0.471	-88	46	-24	-87	-15	118	15	17	27	261	121	163

Notes: Models are spherical (SPH). The first rotation is about Z, right hand rule; the second rotation is about Y', right hand rule; the third rotation is about the rotated Z", right hand rule.
Source: Eldorado.

Table 14.4 Azimuth and dip angles of rotated correlogram axes, Skouries deposit

	Axis azimuth						Axis dip					
	Z1	Y1	X1	Z2	Y2	X2	Z1	Y1	X1	Z2	Y2	X2
PACK Shell – Cu	131	39	129	19	44	133	66	1	-24	78	-11	5
PACK Shell - Au	178	105	210	357	328	60	44	-17	-41	75	-13	-7

Notes: Azimuths are in degrees. Dips are positive up and negative down.
Source: Eldorado.

14.8 Model set-up

The block size for the Skouries model was selected based on mining selectivity considerations for both open pit and underground mining. It was assumed that the smallest block size that could be selectively mined as ore or waste, referred to as the selective mining unit (SMU), was approximately 5 m x 5 m x 10 m. Block model parameters are outlined in Table 14.5.

Table 14.5 Block model parameters

	Minimum (m)	Maximum (m)	Block size (m)	Number of blocks
East	474,177	475,252	5	215
North	4,479,186	4,480,585	5	280
Elevation	-640	860	10	150

The model is not rotated.

The assays were composited into 4 m fixed-length downhole composites. The composite data were back-tagged by the mineralized shell and lithology units (on a majority code basis). The compositing process and subsequent back-tagging were reviewed and found to have performed as expected.

Bulk density data were assigned to the model by general rock type. The allotted values of 2.64 t/m^3 and 2.73 t/m^3 represented historical averages for the intrusive units (porphyry domain) and non-intrusive units (schist domain), respectively. More recent measurements, conducted as checks by Eldorado, yielded very similar averages.

A 30 m to 70 m thick, near surface oxidation zone of sulphide minerals has occurred at Skouries. Specific sub-units within this zone, namely overburden and red clay, contain no appreciable metal values. Model grades in these sub-units were reset to zero after grade interpolation.

14.9 Estimation

Modelling consisted of grade interpolation by ordinary kriging (OK). Nearest-neighbour (NN) grades were also interpolated for validation purposes. Blocks and composites were matched by estimation domain.

The search ellipsoid was oriented 150 m along the X axis, 150 m along the Y axis and 200 m along the Z axis. No rotation was applied.

A two-pass approach was instituted for interpolation. The first pass required a minimum of two holes from the same estimation domain, whereas the second pass allowed a single hole to place a grade estimate in any un-interpolated block from the first pass. This approach enabled most blocks to receive a grade estimate within the domains, including the background domains. Blocks received a minimum of two and maximum of three composites from a single drillhole (for the two-hole minimum pass). Maximum composite limit was 15, whereas the minimum in the two-hole case was set to four.

The interpolation domains comprised the Cu PACK shell and background (defined as any blocks outside of the PACK shell). The contact between the two was treated as a hard boundary, meaning that composite data must lie within the same domain as the model block to be interpolated.

These parameters were based on the geological interpretation, data analyses, and correlogram analyses. The number of composites used in estimating grade into a model block followed a strategy that matched composite values and model blocks sharing the same ore code or domain. The minimum and maximum numbers of composites were adjusted to incorporate an appropriate amount of grade smoothing.

14.10 Eldorado validation

14.10.1 Visual inspection

Eldorado completed a detailed visual validation of the Skouries Mineral Resource model. The model was checked for proper coding of drillhole intervals and block model cells, in both section and plan. Coding was found to be properly completed. Grade interpolation was examined relative to drillhole composite values by inspecting sections and plans. The checks showed good agreement between drillhole composite values and model cell values. Examples of representative sections and plans containing block model grades, drillhole composite values, and domain outlines are shown in Figure 14.1 to Figure 14.4. In these figures, the PACK shell is shown by the outer green outline, the Porphyry unit by the thick yellow line, the reddish-brown line demarcates the oxide – sulphide contact, and the bold white line represents the open pit design.

14.10.2 Model checks for bias

The block model estimates were checked for global bias by comparing the average metal grades (with no cut-off) from the model with means from NN estimates. The NN estimator declusters the data and produces a theoretically unbiased estimate of the average value when no cut-off grade is imposed and is a good basis for checking the performance of different estimation methods. Results, summarized in Table 14.6, show no problems with global bias in the estimates.

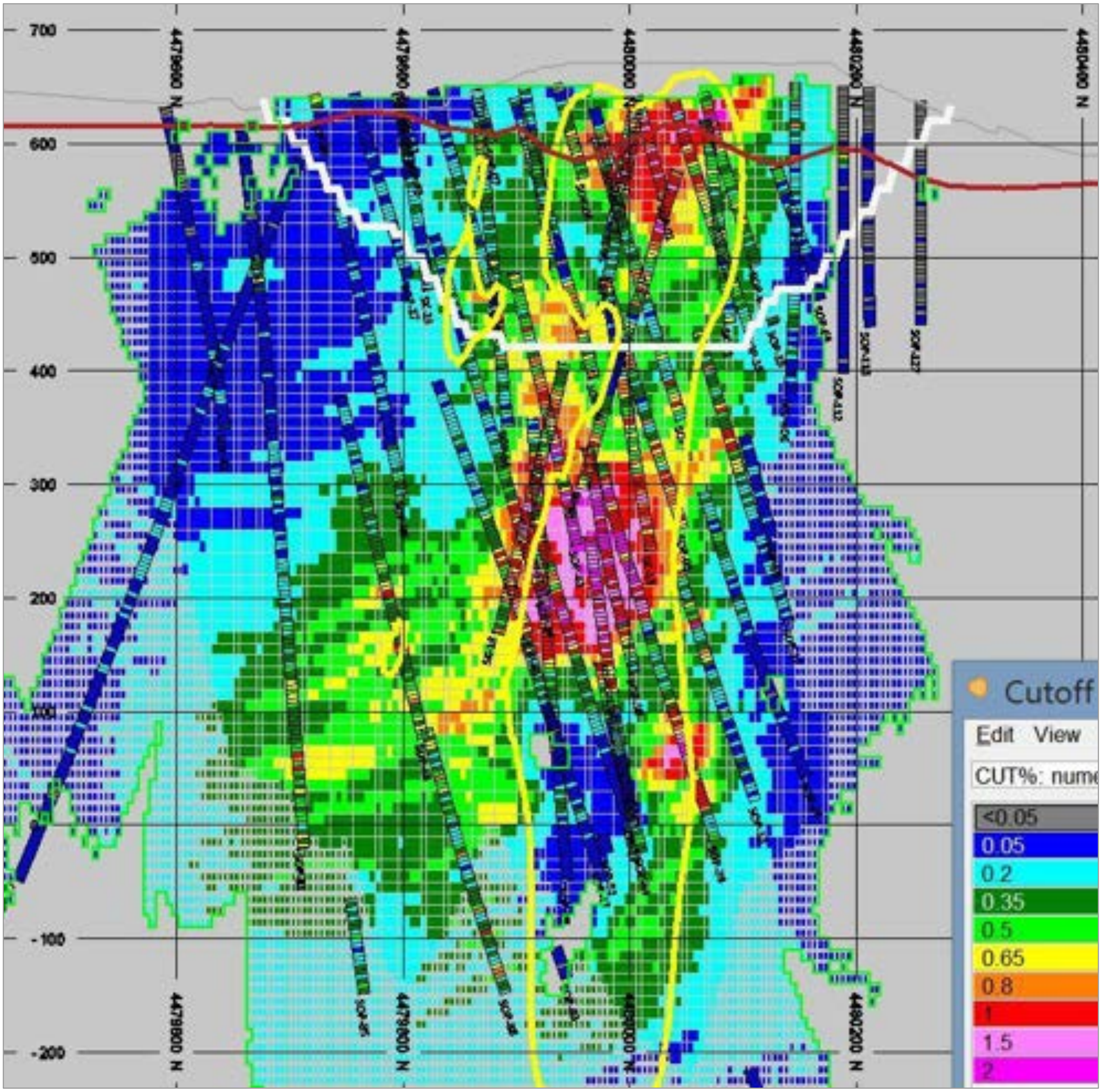
Table 14.6 Global model mean gold values

	NN estimate	Kriged estimate	% difference
PACK Shell - Cu	0.363	0.363	+0.0
PACK Shell - Au	0.455	0.461	+1.3

Source: Eldorado.

The model was also checked for local trends in the grade estimates by grade slice or swath checks. This was done by plotting the mean values from the NN estimate versus the kriged results for benches (in 5 m swaths) and for northings and eastings (both in 20 m swaths). The kriged estimate should be smoother than the NN estimate, thus the NN estimate should fluctuate around the kriged estimate on the plots. The observed trends displayed by the swath plots in Figure 14.5 behave as predicted and show no significant trends of gold or copper in the estimates in the Skouries model.

Figure 14.1 Skouries section showing copper block model values and drillhole composites grades

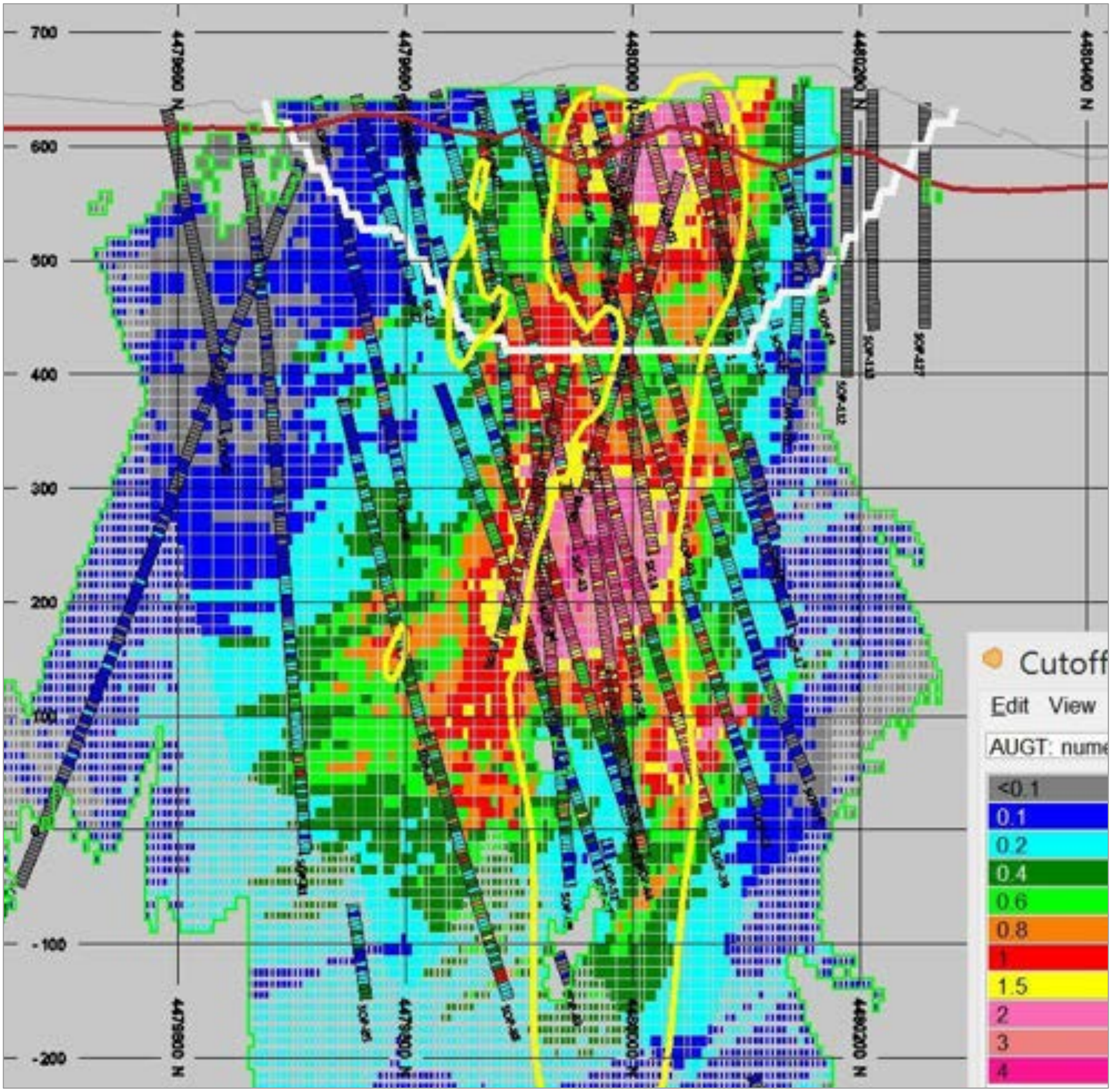


Notes:

- North-south section, Line 474700E.
- The PACK shell = outer green outline; the Porphyry unit = thick yellow line; the oxide – sulphide contact = reddish-brown line; the open pit design=bold white line. Small model blocks denote the Inferred Mineral Resource.

Source: Eldorado 2022.

Figure 14.2 Skouries section showing gold block model values and drillhole composites grades

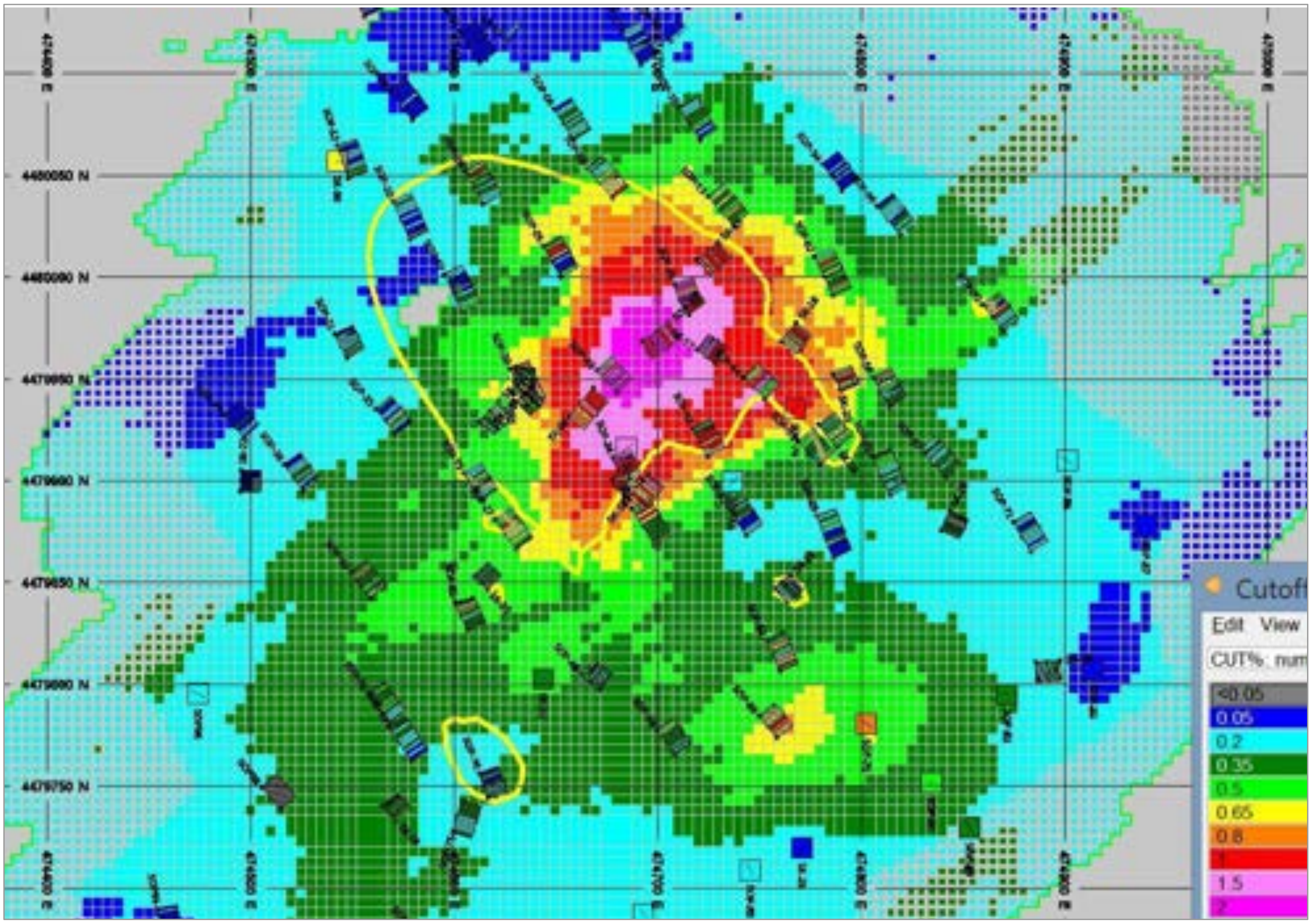


Notes:

- North-south section, Line 474700E.
- The PACK shell = outer green outline; the Porphyry unit = thick yellow line; the oxide – sulphide contact = reddish-brown line; the open pit design=bold white line. Small model blocks denote the Inferred Mineral Resource.

Source: Eldorado 2022.

Figure 14.3 Plan view showing copper block model values and drillhole composites grades

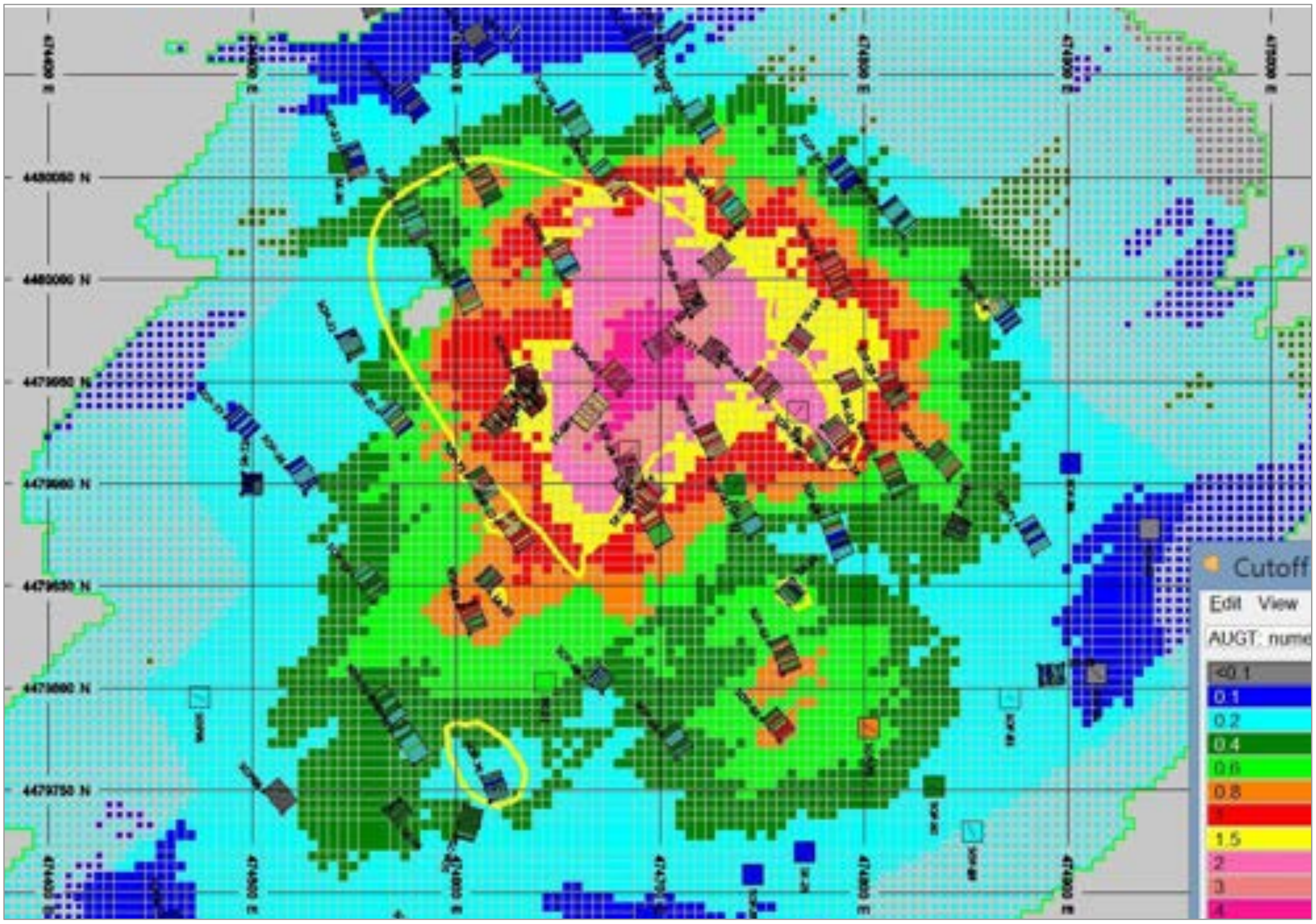


Notes:

- Plan view at 245 m elevation.
- Porphyry unit = thick yellow line, and small model blocks denote the Inferred Mineral Resource.

Source: Eldorado 2022.

Figure 14.4 Plan view showing gold block model values and drillhole composites grades

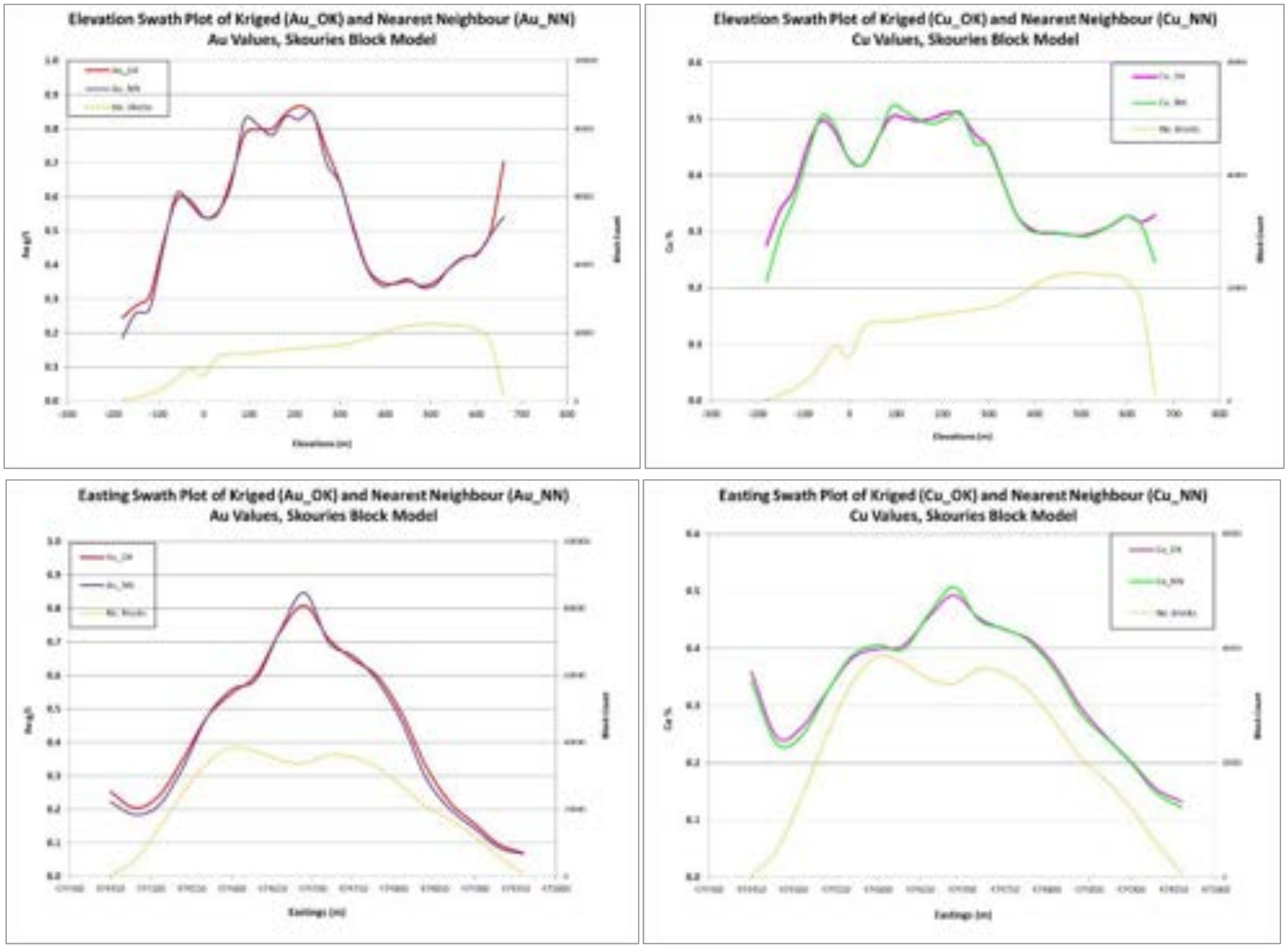


Notes:

- Plan view at 245 m elevation.
- Porphyry unit = thick yellow line, and small model blocks denote the Inferred Mineral Resource.

Source: Eldorado 2022.

Figure 14.5 NN swath plots of Skouries block model gold and copper values



Source: Eldorado 2022.

14.11 AMC validation

The following files were received from Eldorado for the validation exercise:

- Assays.csv, Collars.csv, Survey.csv, Alteration_Primary-ELD drilling ONLY.csv, Litho_Primary - ELD drilling ONLY.csv, Mineralization_Primary - ELD drilling ONLY.csv, Drillhole data
- Skouries_4m_comps
- SKU_Pit Resources_BM.zip, SKU_UG_Resource _BM.zip
- 2019 open pit.dxf
- OP_RO_REPORTING_SHAPE.dxf
- UG_RO_REPORTING_SHAPE.dxf
- UG_RO_REPORTING_SHAPE_INT_WASTE.dxf
- SKU _OP_Resource_2020.xls
- SKU _UG_Resource_2020.xls

AMC reproduced reported tonnes and grades to ensure the correct block model was received, and then validated the block model in three ways, as follows:

1 Conducted an additional visual check of raw drillhole versus block model grades.
2 Completed a statistical comparison of block model mean grades and composited grades.
3 Completed swath plots of drillholes versus the block model.

Visual checks between raw and estimated values on a series of cross sections showed good agreement.

Table 14.7 and Table 14.8 show the statistical comparison of block model and composites grades by classification.

Table 14.7 AMC comparison of block model and composite gold grades

Au (g/t)						
Class	**Measured**		**Indicated**		**Inferred**	
File	**Composites**	**Model**	**Composites**	**Model**	**Composites**	**Model**
No of records	7,121	170,962	8,472	428,507	1,141	637,561
Minimum	0.01	0.03	0.01	0.03	0.01	0.00
Maximum	20.00	7.94	20.00	3.85	1.83	2.16
Mean	**0.82**	**0.72**	**0.36**	**0.35**	**0.22**	**0.20**
Median	0.54	0.56	0.21	0.27	0.17	0.18
Variance	1.11	0.42	0.32	0.09	0.04	0.01
Coeff. variation	1.31	0.89	1.76	0.83	1.07	0.61

Source: AMC.

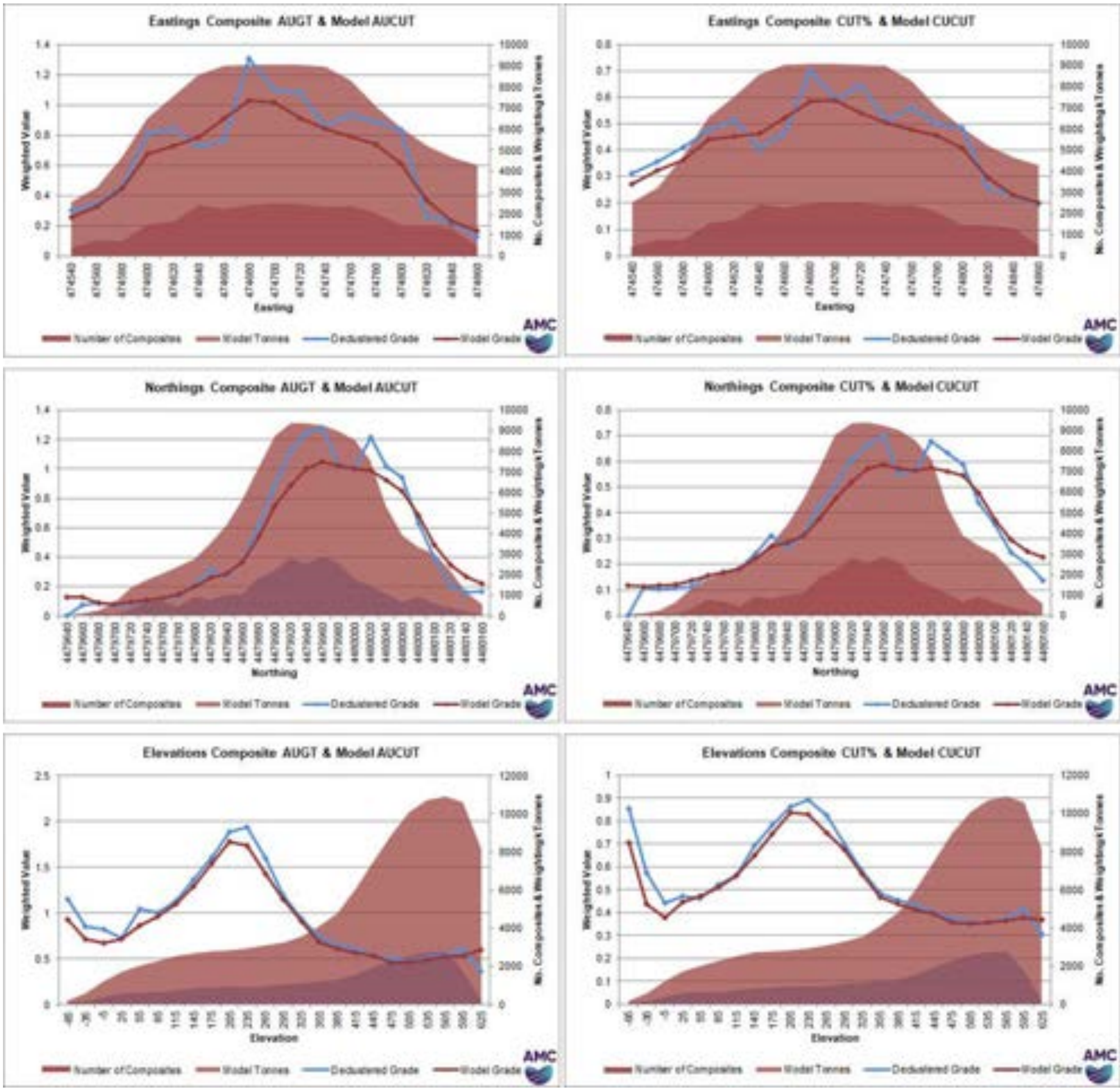
Table 14.8 AMC comparison of block model and composite copper grades

Cu (%)						
Class	**Measured**		**Indicated**		**Inferred**	
File	**Composites**	**Model**	**Composites**	**Model**	**Composites**	**Model**
No of records	7,121	170,962	8,472	428,507	1,141	637,561
Minimum	0.01	0.07	0.01	0.05	0.03	0.00
Maximum	6.27	3.14	13.22	2.38	1.25	1.03
Mean	**0.49**	**0.45**	**0.33**	**0.32**	**0.26**	**0.24**
Median	0.38	0.39	0.26	0.29	0.23	0.23
Variance	0.18	0.09	0.08	0.03	0.02	0.01
Coeff. variation	0.86	0.64	0.87	0.53	0.60	0.47

Source: AMC.

Figure 14.6, Figure 14.7, and Figure 14.8 show the AMC swath plots of gold and copper grades by classification.

Figure 14.6 AMC swath plots gold and copper grades in the Measured class

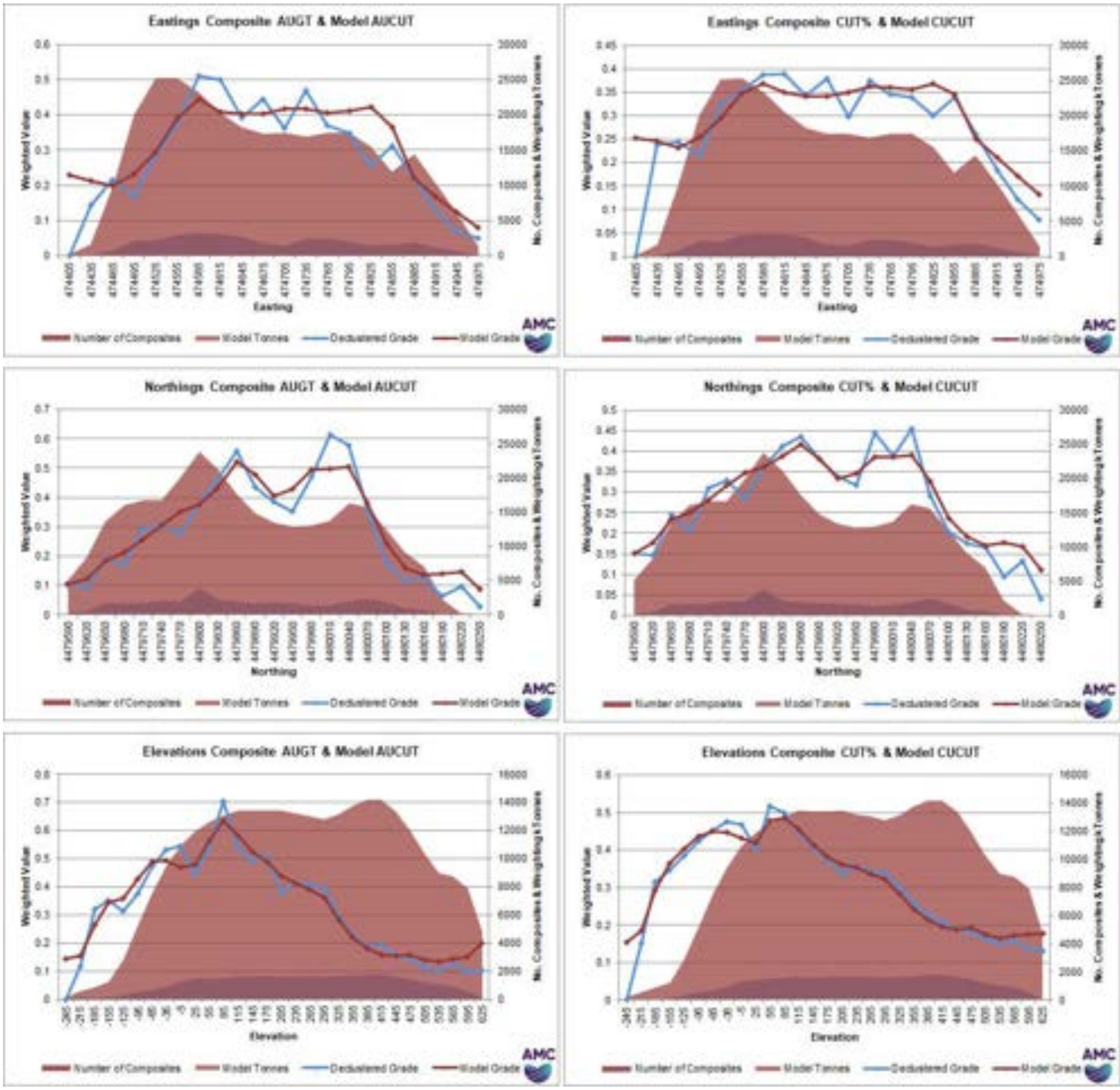


Notes:

- Heading on plot refer to file names, which are the composite files and the model files.
- "Composite AUGT" is gold in g/t cut to 20 g/t, "Model AUCUT" refers to the block grades.
- "Composite CUT%" is copper % uncut, "Model CUCUT" refers to the block grades.

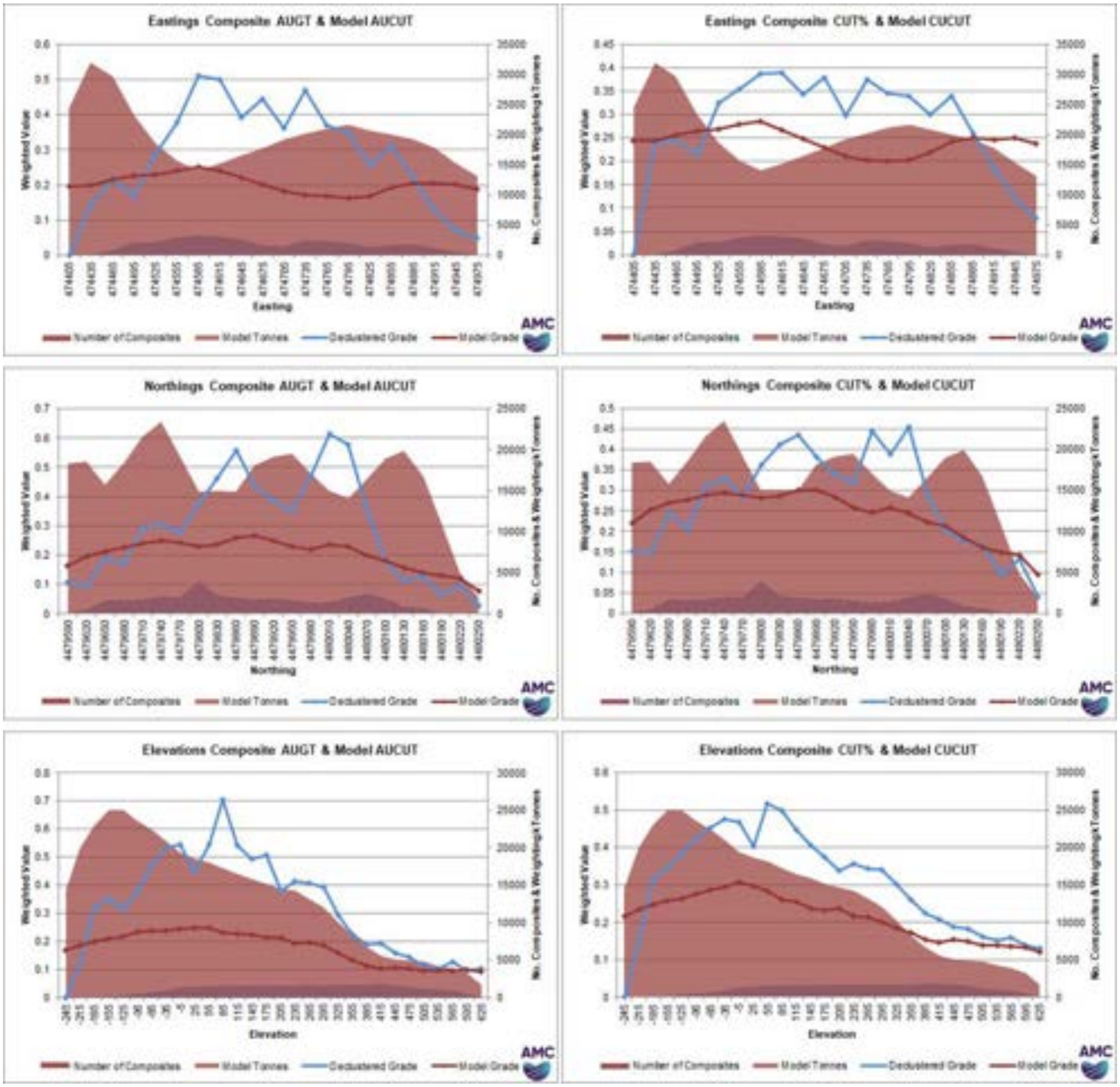
Source: AMC.

Figure 14.7 AMC swath plots gold and copper grades in the Indicated class



Note: Same notes as for Figure 14.6.

Figure 14.8 AMC swath plots gold and copper grades in the Inferred class



Note: Same notes as for Figure 14.6.

The swath plots for Measured and Indicated Mineral Resources generally show good agreement between the composites and the block grades. The plots for the Inferred Mineral Resources show less agreement but the estimate is based on considerably less drilling.

It is noted that the reporting in the 2021 statement is carried out using the process described in Section 14.13, which addresses the Reasonable Prospects for Eventual Economic Extraction (RPEEE). This is done by creating potential mining shapes using the chosen AuEq cut-off. After establishing these shapes, reporting out is at a zero cut-off, thus capturing all the material within the mining shapes. These are named RMAIN=1 for the open pit and RMAIN=3 for underground. The

Mineral Resources for the open pit were also reported at a 0.3 g/t cut-off out of the open pit shell, ignoring the mining shapes, and the difference was not material.

There is no allowance for a possible crown pillar below the pit in the Mineral Resources.

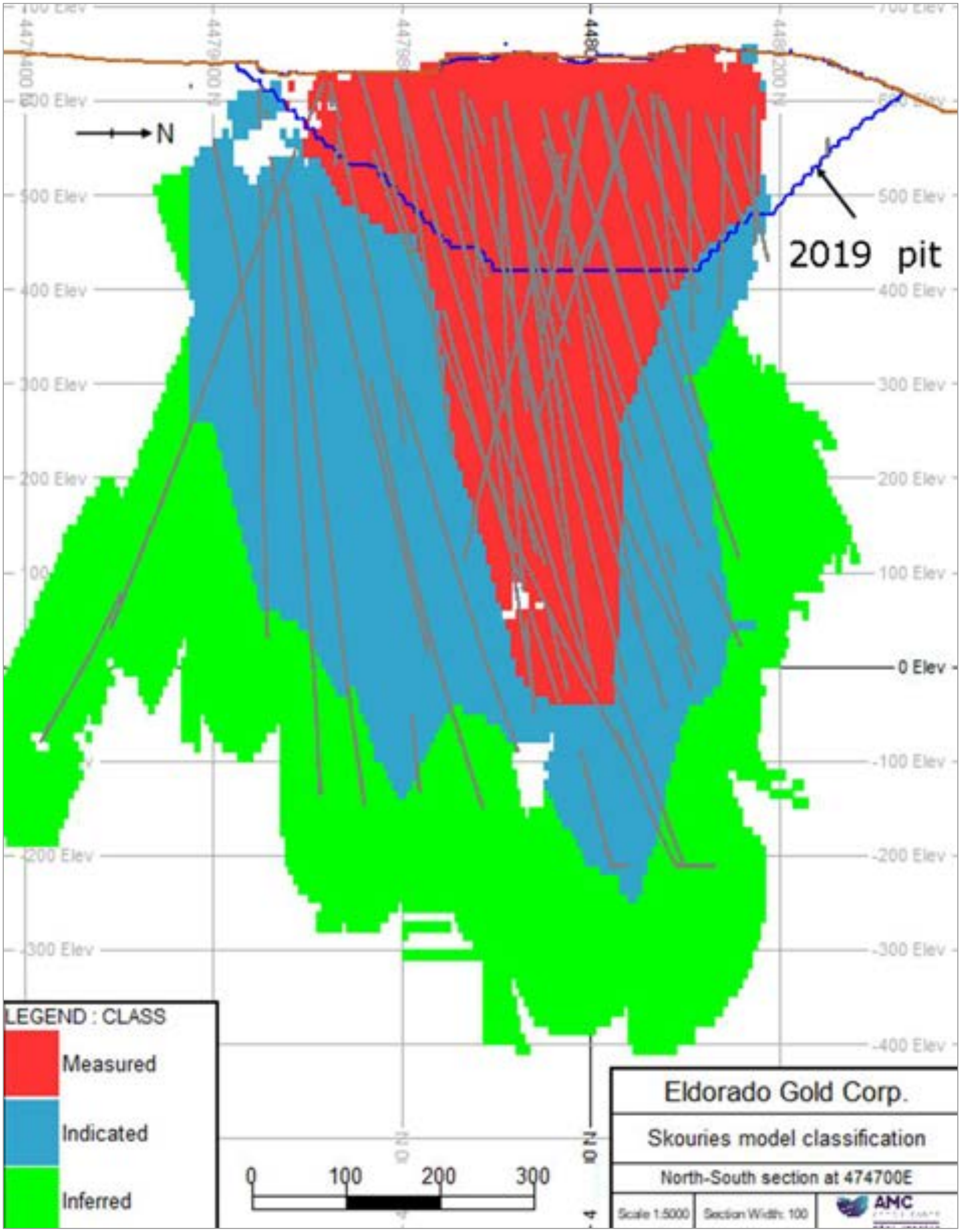
14.12 Mineral Resource classification

The Mineral Resources of the Skouries deposit were classified using the CIM Definition Standards (2014). The evaluation of the mineralization of the Skouries deposit satisfies sufficient criteria to allow classification into Measured, Indicated, and Inferred Mineral Resource categories.

Inspection of the Skouries model and drillhole data on plans and sections, combined with spatial statistical work and investigation of confidence limits in planned annual and quarterly production, contributed to the set-up of various distance to nearest composite protocols to help guide the assignment of blocks into Measured or Indicated Mineral Resource categories. Reasonable grade and geologic continuity are demonstrated over most of the Skouries deposit, which is drilled generally on 40 m- to 80 m-spaced sections. A two-hole rule was used where blocks containing an estimate resulting from two or more samples, all within 80 m and from different holes, were classified as Indicated Mineral Resources. For Measured Mineral Resource classification, a three-hole rule was applied where blocks contained an estimate resulting from three or more samples, all within 50 m and from different holes.

All remaining model blocks containing a gold grade estimate were classified as Inferred Mineral Resources. Figure 14.9 shows a vertical section of the block model classification and the composites used in the estimation.

Figure 14.9 North-south section showing classification



Source: AMC.

14.13 Mineral Resource

14.13.1 Economic basis

For Mineral Resource reporting the demonstration of RPEEE was handled independently for the open pit and underground portions of the deposit. In each case a long-term gold metal price of US$1,800/oz and long-term copper price of US$3.50/lb were selected for the determination of resource cut-off grades or values. This guided execution of the next step, where constraining surfaces or volumes were created to control Mineral Resource reporting.

Only material within a semi-optimized pit shell, the January 2019 pit supplied by Eldorado, is reported as open pit Mineral Resources; all other material was considered as reporting to underground. While the pit shell itself not fully optimized and is strongly permit constrained in depth and surface extent, prices of US$1,800/ounce for gold and US$3.50/lb for copper were used in the development of the shell.

The underground Mineral Resources were constrained by 3D volumes whose design was guided by the reporting cut-off grade or value, contiguous areas of mineralization and viability of mining. These shapes were also constrained within 100 m of any planned development. Only material internal to these volumes was eligible for reporting.

Note that the metal recovery values are significantly higher than projected in Section 17 (82.4% Au and 87.9% Cu). This is a timing issue and the cut-off grade and hence the resource estimates are not very sensitive to gut off grade.

Table 14.9 Economic parameters for RPEEE evaluation

Description	Units	Open pit	Underground
Gold price	US$/oz	1,800	1,800
Copper price	US$/lb	3.50	3.50
Mining cost	US$/t processed	4.10	19.50
Process cost	US$/t processed	8.48	8.48
Filter plant cost	US$/t processed	2.13	2.13
IEWMF and water management	US$/t processed	0.13	0.13
G&A	US$/t processed	2.78	2.78
Overall costs	US$/t processed	17.62	33.02
Mill Au recovery	%	86.7	86.7
Mill Cu recovery	%	91.5	91.5
Cut-off used	AuEq g/t	0.3	0.7

14.13.2 Reporting

The Skouries Mineral Resources as of 30 September 2021 are shown in Table 14.10. Mineral Resources are reported using resource reporting shapes representing volumes that have a reasonable expectation of being mined. Volumes that lie within both the 0.1% Cu PACK shell and the open pit shell and that are predominantly above a cut-off grade of 0.3 g/t AuEq are assigned to the underground resource reporting shape. Volumes that lie outside the open pit shell, but that lie within the 0.1% Cu PACK shell and that are predominantly above a 0.7 g/t AuEq cut-off grade are assigned to the underground resource reporting shape. Volumes within both the open pit and underground resource reporting shapes are reported in their entirety; this will include some isolated blocks that are below the assigned cut-off, but that lie within the volumes deemed to be reasonably viable to mine. Similarly, isolated blocks that are above the cut-off grades, but that lie outside of the expected mining viability volumes are omitted from the Mineral Resource estimate. The gold equivalent formula used is: AuEq = Au (g/t) + 1.25 * Cu (%), based on metal prices and recoveries stated in Table 14.9.

Table 14.10 Skouries Mineral Resources, at 30 September 2021

Category	Tonnes (kt)	Au (g/t)	Cu (%)	Contained Au (koz)	Contained Cu (kt)
Open pit Mineral Resources					
Measured	50,641	0.62	0.42	1,013	214
Indicated	14,151	0.22	0.22	99	32
Measured & Indicated	64,791	0.53	0.38	1,112	246
Inferred	784	0.16	0.18	4	1
Underground Mineral Resources					
Measured	40,073	1.14	0.63	1,467	252
Indicated	135,109	0.56	0.46	2,452	620
Measured & Indicated	175,182	0.70	0.50	3,919	872
Inferred	66,873	0.38	0.40	811	265
Total Mineral Resources					
Measured	90,714	0.85	0.51	2,479	466
Indicated	149,260	0.53	0.44	2,551	652
Measured & Indicated	239,974	0.65	0.47	5,030	1,118
Inferred	67,657	0.37	0.40	814	267

Notes:
- CIM Definition Standards (2014) were used for reporting the Mineral Resources.
- Open pit Mineral Resources are constrained by a semi-optimized pit that is strongly permit constrained and are reported at a 0.3 g/t AuEq cut-off.
- Underground Mineral Resources are those outside the pit shell and are reported at a 0.70 g/t AuEq cut-off.
- AuEq = Au g/t + 1.25 * Cu%, based on US$1,800/oz Au and US$3.50/lb Cu, and recoveries of 86.7% for gold and 91.5% for copper.
- Mineral Resources are stated inclusive of Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Numbers may not compute exactly due to rounding.

Source: Eldorado, re-reported by AMC and approved by the QP.

14.14 Comparison to the previously reported estimate

There is no difference between the Mineral Resources reported in September 2020 and September 2021 and both statements are made on the same basis. There has been no production from the deposit, hence no depletion from the model.

15 Mineral Reserve estimates

The Mineral Reserves at Skouries comprise an open pit and an underground component. These are described, respectively, in Sections 15.1 and 15.1.5. The QP for the open pit is John Battista, MAusIMM (CP) of MP and for the underground Gary Methven, P.Eng. of AMC.

15.1 Open pit Mineral Reserve estimate

The open pit Mineral Reserves include key assumptions and economic considerations leading to pit limit selection and the reporting of Mineral Reserves used for mine planning and scheduling as described in Section 16. The assumptions and economic considerations reported below were used only in the preliminary optimization of the open pit and have been refined in the economic model as outlined in Section 22, without any impact on the Mineral Reserves.

15.1.1 Open pit optimization

The open pit optimization was carried out using MineSight® mine planning software. A series of unsmoothed pit shells was created using a Lerchs-Grossman algorithm with revenue factors declining from unity. The unsmoothed pit shells were then used as a guide for developing detailed designs to be used in production scheduling and reserve reporting. The Skouries open pit is constrained by the existing EIS boundary on surface and the underground mining crown pillar, which limits the pit depth to 420 masl. In addition to the physical boundary constraints, the open pit design and overall size are also affected by a requirement to provide construction materials for the IEWMF and provide tailings disposal volume after the IEWMF closure.

15.1.1.1 Economic parameters applied to mine design

Metal prices

Base case pit optimization metal prices are as follows:

- Copper: US$2.75/lb
- Gold: US$1,300/oz

Smelter terms and offsite costs

Copper and gold will be recovered by flotation methods, and report to a single copper / gold concentrate. The basis for pit optimization was the net smelter return (NSR) revenue per tonne of ore calculated for each block in the resource model. Metal prices described above and offsite costs for concentrate transportation, treatment and refining are used in the analysis. The NSR calculation, smelter terms and offsite costs are summarized in Table 15.1. The table shows an estimate of NSR for copper and gold in a project-wide average sample within the sulphide zone of the resource block model. Table 15.1 is valid for both the Open Pit and Underground reserve NSR calculation.

Table 15.1 Example of NSR calculation

Test block NSR calculation	Units	Test block values
Copper head grade	%	0.503
Gold head grade	g/t	0.767
Oxide ore	%	-
Metallurgical recovery		
Copper recovery	%	91.91%*
Gold recovery	%	83.86%**
Metal pricing		
Copper price	US$/lb	$2.75
Gold price	US$/ounce	US$1,300.00
Copper concentrate		
Copper concentrate grade	%	26.0%
Moisture content	%	9.0%
Contained copper	lb/dmt	573.20
Contained gold	g/dmt	29.69
Payable copper	lb/dmt	551.15
Payable gold	g/dmt	28.72
Concentrate - recovery based	dmt/t ore	0.01777
Gross value concentrate	US$/dmt	2,715.96
Copper concentrate handling		
Mine to port concentrate freight cost	US$/wmt	50.00
Ocean concentrate freight cost	US$/wmt	25.00
Total concentrate handling	US$/wmt	75.00
Total	US$/wmt	**82.42**
	US$/t ore	**1.46**
Copper concentrate treatment and refining		
Deduction for copper	unit	1.0%
Copper payment	%	96.15%
Treatment charges	US$/dmt	$82.00
Gold to concentrate	%	82.0%
Gold deduction - adjustment	%	0.3%
Gold payment concentrate	%	97.0%
Copper refining cost	US$/payable lb	0.082
Gold refining cost	US$/payable oz	6.00
Total treatment and refining	US$/dmt	**132.73**
	US$/t ore	**2.36**
Copper NSR		
NSR	US$/dmt	**2,763.08**
	US$/payable lb Cu	5.01
NSR before royalty - excluding doré †	**NSR US$/t**	**49.10**
Royalty gold	%	1.65%
Royalty copper	%	0.55%
Gold value for royalty calculation	NSR US$/t	26.77
Copper value for royalty calculation	NSR US$/t	28.01
Total value for royalty	**NSR US$/t**	**54.78**
Royalty gold	US$/t	0.4417
Royalty copper	US$/t	0.154
NSR after royalty	NSR US$/t	48.50

Notes:
* Cu recovery (%) = 99.41 – 56 x oxide (%) – 41 x $e^{(-338 \times \text{Cu Grade } (\%))}$.
** Au recovery (%) = 92.62 – 17.5 x oxide (%) – 22 x $e^{(-1.2 \times \text{Au Grade } (g/t))}$.
† Doré included in NSR calculation, but gravity circuit removed during the course of the Feasibility Study. Reserve NSR calculation results in 0.3% higher NSR after royalty.

Concentrate transportation costs were estimated at US$75.00/wet metric tonne (wmt) preliminarily. Final concentrate transportation costs are lower than the preliminary estimate at US$19.55/wmt, which has built conservatism into the NSR calculation. The concentrate is assumed to have a moisture content of 9.0%.

Treatment charges for copper concentrate were estimated to be US$82.00/dry metric tonne (dmt) and refining charges were estimated to be US$0.0820/payable pound of copper. Copper concentrate grade will average 26.0%, and the typical concentrate terms call for a one (1) unit deduction, which results in a 96.15% payability for copper in concentrate.

The NSR calculations allow for the accounting of:

- Ore grades (Cu and Au), thus taking into account the variability in the metal content of the deposit.
- Ore mill recoveries, which vary according to grade and oxidation.
- Contained metal in concentrate.
- Deductions and payable metal value.
- Metal prices.
- Freight, smelting, and refining charges.

Royalty charges of 1.65% for gold and 0.55% for copper were applied to the reserve NSR calculation which correlate to US$1,300/oz gold and US$2.75/lb copper, respectively.

Changes in the NSR parameters from the pre-feasibility study (PFS) to the feasibility study (FS) are provided in Table 15.2.

Table 15.2 NSR variation

Metal pricing	**Unit**	**PFS**	**FS**	**Variation**
Copper price	US$/lb	2.50	2.75	+$0.25
Gold price	US$/ounce	1,200.00	1,300.00	+$100.00
Copper concentrate				
Copper concentrate grade		26.00%	26.00%	-
Moisture content	%	9.00%	9.00%	-
Copper concentrate handling				
Mine to port concentrate freight cost	US$/wmt	59.80	50.00	+$15.20
Ocean concentrate freight cost	US$/wmt		25.00	
Copper concentrate treatment and refining				
Deduction for copper	unit	1.00%	1.00%	-
Copper payment	%	96.20%	96.15%	-0.05%
Treatment charges	US$/dmt	97.35	82.00	-15.35
Gold to concentrate	%	82.10%	82.00%	-0.10%
Gold deduction - adjustment	%	100.00%	99.70%	-0.30%
Gold payment concentrate	%	96.60%	97.00%	+0.40%
Copper refining cost	US$/payable lb	0.097	0.082	-0.025
Gold refining cost	US$/payable oz	5.50	6.00	+0.50
Royalty				
Royalty gold	%	2.00%	1.65%	-0.35%
Royalty copper	%	0.50%	0.55%	+0.05%

Onsite operating costs and increments

The pit limit analysis included general and administration (G&A), processing, mining, and tailings handling costs. An incremental haulage cost of US$0.046/t ore / bench was added for each 10 m bench below the open pit entrance at 620 masl. No updates have been made by Mining Plus to the 2018 PFS pit limit analysis. The adjustments in metal pricing were to the positive, and the most economic pit was not selected due to permitting restrictions. The open pit is considered economically robust as the design limits it corresponds to are at a lower copper and gold price to current metal price assumptions.

15.1.1.2 Metallurgical parameters

Process selection

The processing method at Skouries is primary crushing followed by grinding and conventional flotation of copper concentrate to be smelted and refined off-site. Provision has been made for a gravity circuit to capture free gold as a concentrate to be refined to doré, but this is not part of the current plan.

Process recovery

The processing recovery used to develop the NSR model for mine planning was based upon lock cycle testwork for oxide and sulphide (Section 13). The results were used to develop metallurgical recovery equations relating to grade and oxidation level and are as follows:

- Recovery (Cu) = 99.41 – 56 x % oxide – 41 x $e^{(-338 \text{ x Cu Head Grade \%})}$
- Recovery (Au) = 92.62 – 17.5 x % oxide – 22 x $e^{(-1.2 \text{ x Au Head Grade g/t})}$

The copper recovery is capped at 95% and the gold recovery is capped at 90%.

Concentrate grade

The copper concentrate grade for copper was estimated to be 26.0% Cu with 9.0% moisture content. Gold concentrate grade was calculated based upon the gold head grades and the estimated recovery to concentrate.

15.1.1.3 Block model

General

The resource block model developed by Eldorado is described in Section 14. The block model and surfaces for topography, subsurface oxidation and the geology were imported to a 3D model. The MineSight® mine planning block model limits and block dimensions are shown in Table 15.3. The differences in Limits and number of blocks are due to design requirements within the MineSight® software. Additional waste material areas of the surface works were required in the design as a result the limits of the block model were extended to encompass them, resulting in additional blocks.

Table 15.3 Block model limits

Parameters	Units	Values		
Model limits 2015 - 2017		**Minimum**	**Maximum**	**Length**
Limits X	metres	473,877	475,552	1,675
Limits Y	metres	4,478,985	4,480,785	1,800
Limits Z	metres	640	860	1,500
Block size				
Size X	metres	5.0		
Size Y	metres	5.0		
Size Z	metres	10.0		
Number of blocks				
Number X	blocks	335.0		
Number Y	blocks	360.0		
Number Z	blocks	150.0		
Total blocks	**blocks**	**18,090,000**		

Block model items transferred from the geology model for mine planning included estimated grades for copper and gold as well as resource classification. Additional items were populated in the MineSight® model for rock codes, alteration, mining restrictions, slope codes for design purposes, recovery, ore %, net value and possible scheduling destinations.

Resource classification

The resource model includes Measured, Indicated, and Inferred resources. Measured and Indicated resources have been used to define the pit limits and for reporting of reserves for scheduling. Inferred resources were not used in the mine plan.

15.1.1.4 Mining recovery

Mining recovery is assumed to be 100%. No mining loss factors were applied to the Mineral Reserves for the following reasons:

- The deposit shows good lateral and vertical continuity at the cut-off values applied for scheduling.
- There is a broad width to the ore zones on individual benches.
- A detailed grade control program will be implemented.

15.1.1.5 Mining dilution

Internal dilution was incorporated in the resource model by virtue of the compositing and interpolation method used to obtain the block grades. No additional dilution factor was applied in pit optimization.

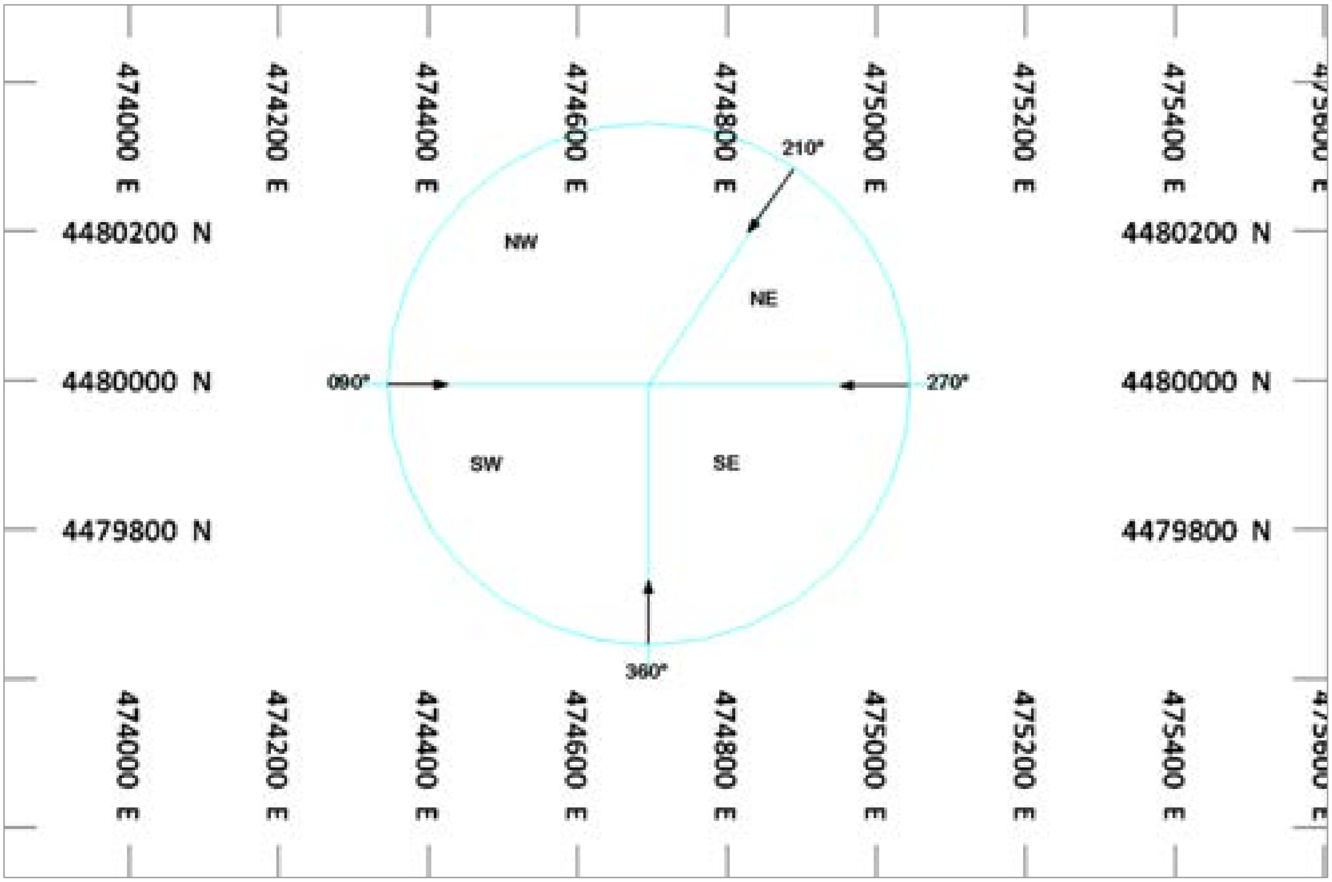
15.1.1.6 Reserves classification

Conversion of Mineral Resource categories from Measured to Proven and Indicated to Probable Mineral Reserves. Inferred Mineral Resources were considered as waste.

15.1.1.7 Wall slope design

The wall slope design was completed prior to the 2017 PFS report. No additional information has been acquired to alter the wall slope design at this time, and Mining Plus concurs with the previous design. Inter-ramp wall slopes angles were assigned by sector that were further subdivided into “Red Clay” zone, “Modified Overburden”, “Weak to Hard Rock Transition” and “Sulphide”. The slope sector orientation and design parameters applied for pit optimization and design are shown in Figure 15.1 and Table 15.4.

Figure 15.1 Slope sector orientation



Source: 2018 Eldorado Technical Report.

Table 15.4 Slope sector parameters

Sector	Units	SLOP2 Code	BFA	BH	BW	IRA
All	Red Clay	1	50	10	6	32
All	Mod. Ovb	2	55	10	6	35
All	WR-HR Trans.	3	65	10	6	42
NE (facing 210 - 270)	Sulphide	4	70	10	6	45
SE (facing 270 - 360)	Sulphide	5	70	10	6	45
SW (facing 360 - 90)	Sulphide	6	70	10	6	45
NW (facing 90 - 210)	Sulphide	7	70	10	6	45

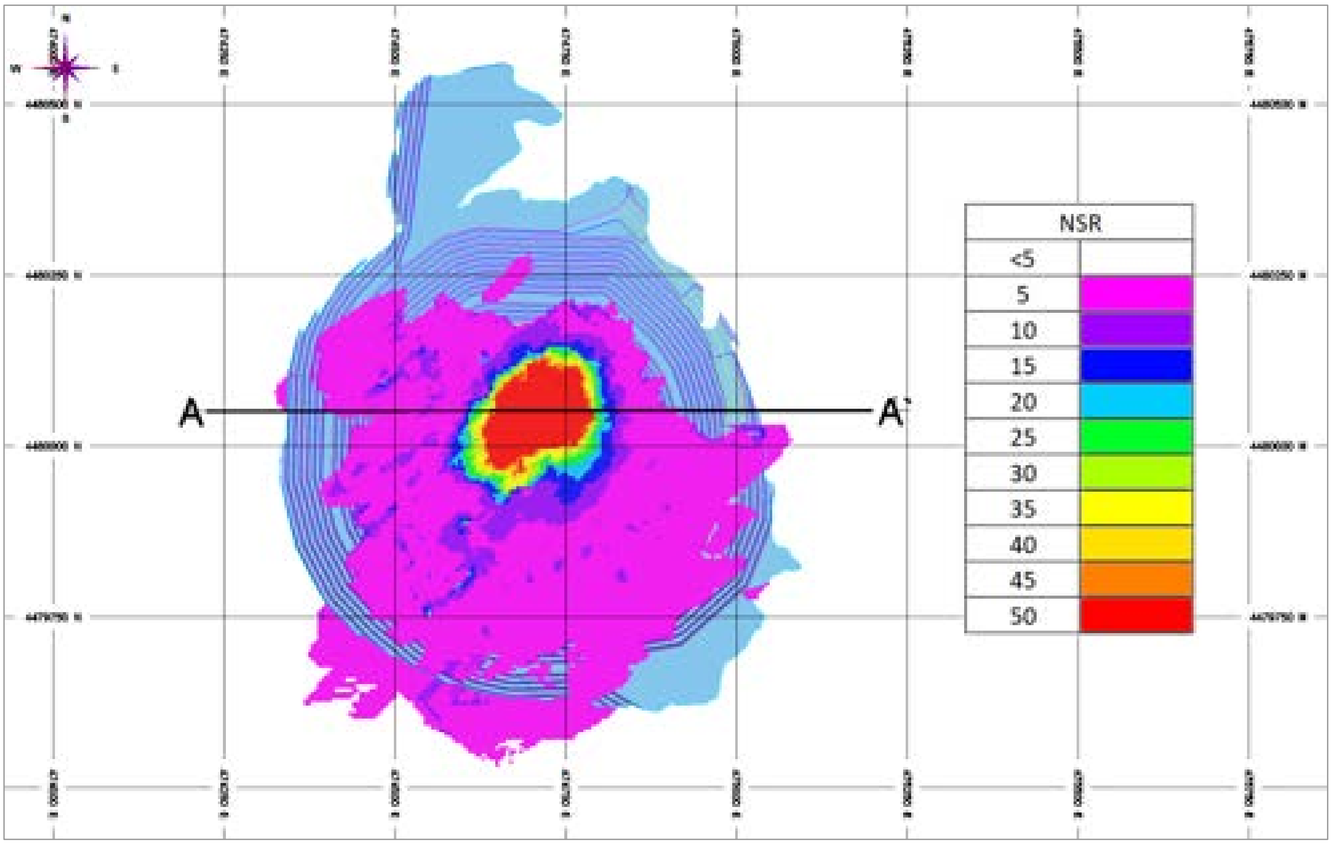
Notes:
- BFA-Bench Face Angle
- BH-Bench Height
- BW-Berm Width
- IRA-Inter-Ramp Angle

15.1.1.8 Pit limit analysis

The pit limit analysis was completed prior to the 2018 Eldorado Technical Report. The changes in metal pricing or royalties for this Technical Report do not alter the pit limit. The overall size of the pit is constrained by the EIS boundary and underground pillar location constraints, and MP concurs with the previous analysis.

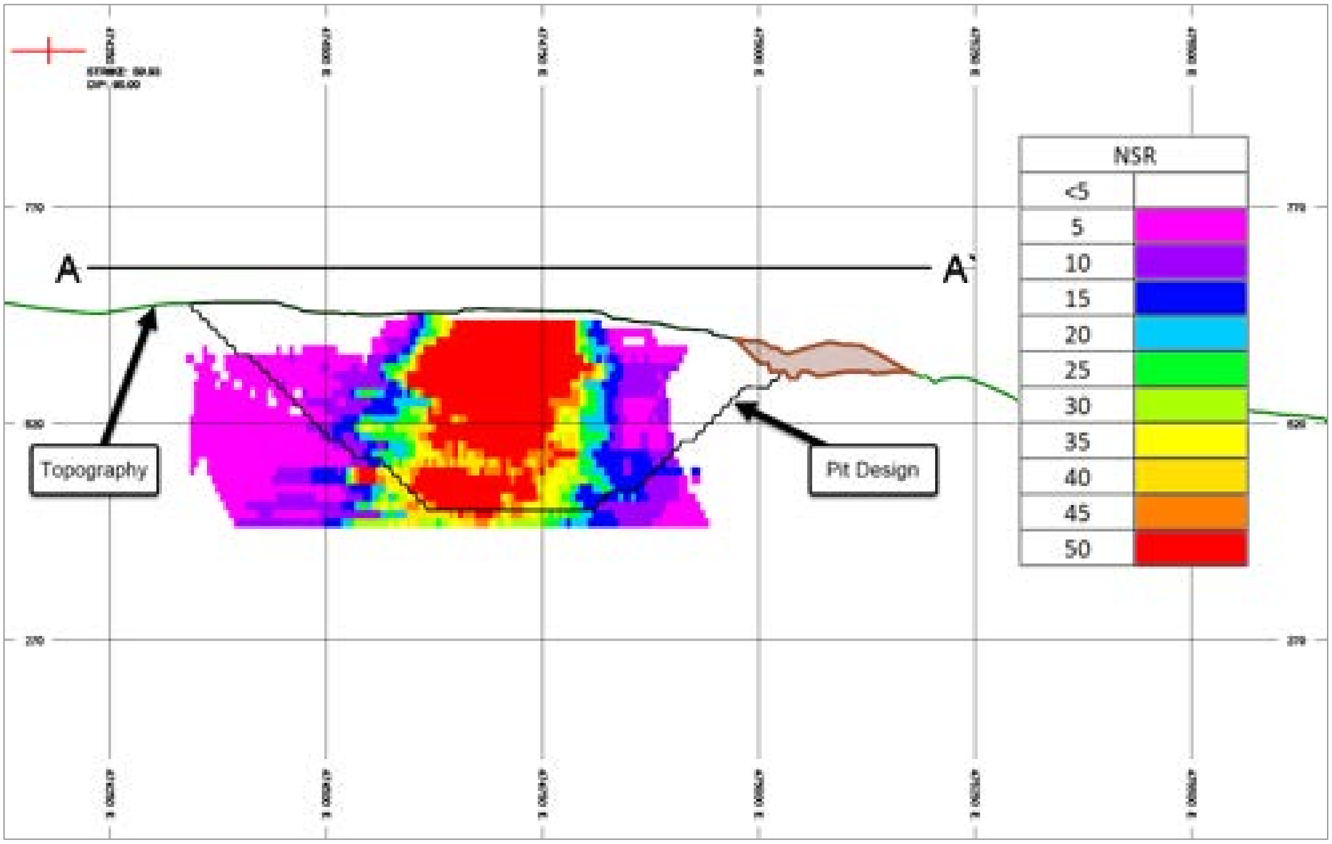
The pit limits are relatively insensitive to metal prices within the ultimate constrained pit limits. The nested pit limits used to develop the pit design are shown in Figure 15.2 and Figure 15.3. Using the NSR value heat maps in each figure, it can be seen that the orebody is centrally located in the pit and a clear delineation between ore and waste is present.

Figure 15.2 Bench plan NSR Lerchs-Grossman pit limits



Source: MP 2022.

Figure 15.3 Cross section NSR values

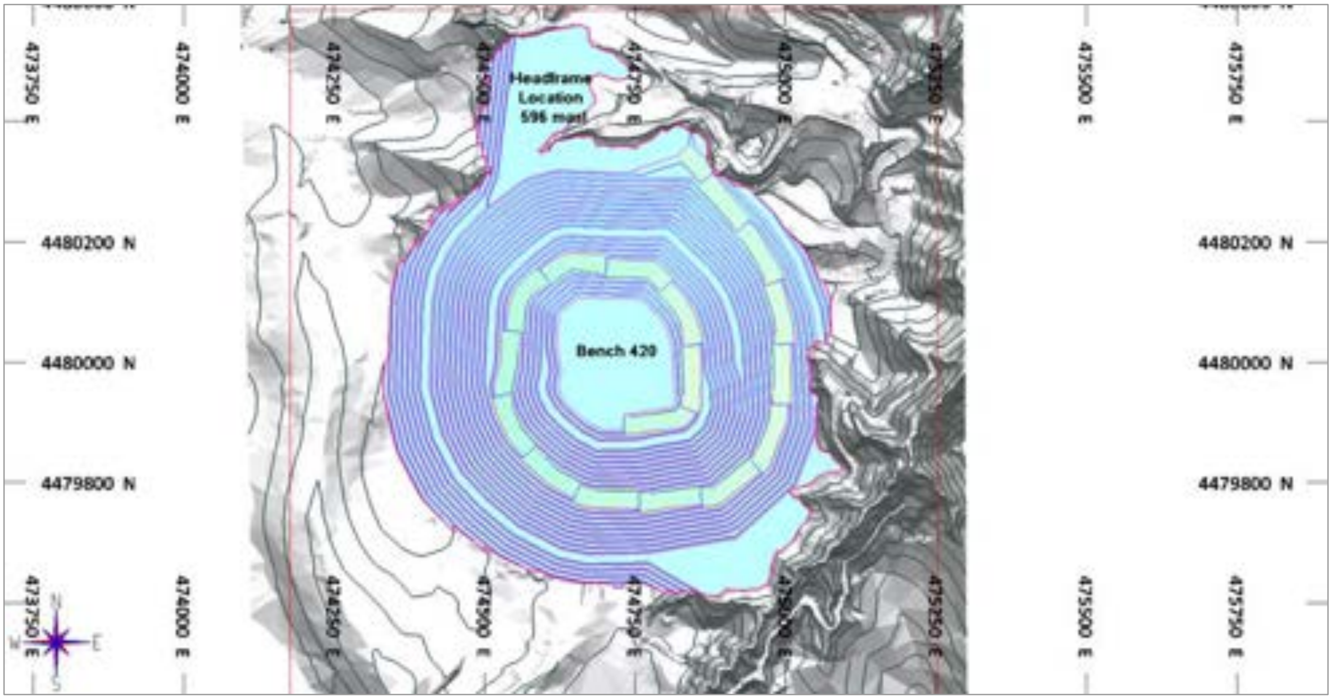


Source: MP 2022.

15.1.2 Pit design

The ultimate pit configuration is shown in Figure 15.4. This design was completed for the 2018 PFS. The design closely follows the Lerchs-Grossman pit limit on the west and southwest side where the EIS boundary restricts the pit. The pit design has been expanded to the east to provide additional construction materials needed for the IEWMF. The pit has also been expanded to the north at 596 masl to provide additional space surrounding the headframe location for Phase 2 of the underground mine.

Figure 15.4 Final pit design



Source: MP 2022.

Open pit optimization and final pit design were evaluated and completed during the previous 2018 pre-feasibility study. Although these were defined with price assumptions that are considered conservative today, the EIS and UG crown pillar location constraints to the final pit do not allow expansion of the final pit. As such, the current design is still considered valid.

15.1.3 Cut-off value

The open pit cut-off value has been split into two distinct timeframes, production, and low-grade ore reclamation.

The onsite operating costs used for cut-off value evaluation during the production years includes G&A, processing, paste backfill, water management and tailings handling costs. These direct feed costs were calculated to be US$13.44/t ore. Included is the cut-off value at Eldorado's direction is the sustaining capital to be spent throughout the LOM that can be attributed to open pit tonnes being milled, which adds an additional US$1.69/t ore. The direct feed open pit cut-off value is US$15.13/t ore.

The low-grade ore reclamation cut-off value will include the variable costs associated with processing the low-grade tonnes (US$9.14/ t ore), along with the operating costs of reclaiming those tonnes (US$1.22/t ore). A small sustaining capital cost is attributed to those tonnes (US$0.26/t ore). The low-grade tonne reclamation cut-off value is US$10.62/t ore.

15.1.4 Open pit Mineral Reserves

The Mineral Reserves for the deposit were estimated within the ultimate pit design using a gold price of US$1,300/oz and copper price of US$2.75/lb. The Mineral Reserves are reported using a US$10.60/t NSR cut-off. The Proven and Probable Mineral Reserves are 59.6 Mt with an average grade of 0.57 g/t Au and 0.40% Cu. The Mineral Reserves are summarized in Table 15.5.

The Mineral Reserve estimate was based upon economic parameters, geotechnical design criteria and metallurgical recovery assumptions. Changes in these assumptions (Table 15.2) have a very minor impact on the in-pit reserve estimate since the pit is largely constrained by permitting and the crown pillar location, not by economic parameters. The crown pillar lower elevation of 350 masl was chosen after consideration of pillar stability, underground ground support requirements, and geotechnical interaction with excavated and backfilled stopes.

Table 15.5 Skouries open pit Mineral Reserves, as of 30 September 2021

Phase 01					
Category	**Ore (kt)**	**Grade Au (g/t)**	**Grade Cu (%)**	**Contained Au (koz)**	**Contained Cu (kt)**
Proven	25,091	0.79	0.48	639	121
Probable	2,638	0.34	0.28	29	7
Proven & Probable	27,730	0.75	0.46	668	128
Phase 02					
Category	**Ore (kt)**	**Grade Au (g/t)**	**Grade Cu (%)**	**Contained Au (koz)**	**Contained Cu (kt)**
Proven	23,455	0.48	0.38	364	89
Probable	8,413	0.22	0.24	59	20
Proven & Probable	31,868	0.41	0.34	423	109
Total pit					
Category	**Ore (kt)**	**Grade Au (g/t)**	**Grade Cu (%)**	**Contained Au (koz)**	**Contained Cu (kt)**
Proven	48,546	0.64	0.43	1,002	210
Probable	11,051	0.25	0.25	88	27
Proven & Probable	59,597	0.57	0.40	1,091	238

Notes:
- Cut-off value applied: NSR of US$10.60/t ore
- Gold Price: US$1,300/oz
- Metallurgical Gold Recovery: 92.62 – 17.5 x oxide (%) – 22 x $e^{(-1.2 \times \text{Au Grade (g/t)})}$
- Copper Price: US$2.75/lb
- Metallurgical Copper Recovery: 99.41 – 56 x oxide (%) – 41 x $e^{(-338 \times \text{Cu Grade (\%)})}$
- Mining Recovery: 100.0%
- Mining Dilution: 0.0%
- Numbers may not compute exactly due to rounding.

15.1.5 Tonnes below cut-off in plan

The cut-off used in the plan is US$10.60/t ore, however the low-grade tonne cut-off value determined in Section 15.1.3 is US$10.62/t ore. This difference is negligible and does not affect the Mineral Reserve estimation.

15.2 Underground Mineral Reserve estimate

Work completed by MP in 2021/2022 provided the underground contribution for the combined Skouries Mineral Reserves. Updated economic input parameters for the determination of Mineral Reserves were provided by Eldorado during the course of the work and AMC accepted those economic parameters as reasonable.

15.2.1 Dilution and recovery factors

In the evaluation of underground Mineral Reserves, modifying factors were applied to the tonnages and grades of all in situ mining shapes to account for dilution and ore losses that are common to all mining operations. Due to stope blasting adjacent to fill surfaces, unplanned ore dilution will consist mainly of paste backfill. Paste backfill is assumed to carry no recoverable metal values.

Rock overbreak as a significant diluting material was not considered for the following reasons:

- The potential overbreak of primary stopes into secondary stopes is internal to the orebody and will not impact the forecasted averages for tonnes and grade over time.
- The tonnage that would be generated by overbreak of peripheral stopes into the country rock at the boundary of the orebody is negligible when compared to the tonnage of the orebody itself.

The major source of dilution is overbreak into backfilled stopes, which is inevitable and inherent in mines that employ SLOS. It has been estimated that 5.0% to 5.5% (by weight) will enter the mill. The sources of paste backfill dilution are a function of the various types of backfill exposure that will be created during the mining process:

- Primary stopes will expose the paste backfill of the preceding primary stopes in the line of retreat.
- Secondary stopes will expose the paste backfill of preceding secondary stopes in the line of retreat, and the paste backfill walls of adjacent primary stopes.
- Stopes on the 50 Level under the 110 Level sill will expose paste backfill in the roof of the excavation of a stope above, that has been previously extracted and backfilled.

A minor amount of dilution will also originate from the waste rock that is placed to provide a trafficable surface on the paste backfill of a filled stope and is included within the estimate.

The overall dilution by weight was calculated as 5.3%, as shown in Table 15.6.

Table 15.6 Dilution factor assumptions

Excavation area	Unit	Assumed dilution
Porphyry stope	%	5.0
Schist stope	%	5.5
Development	%	5.0
Average dilution by weight	%	5.3

Ore losses (mining recovery factors) are related to the practicalities of extracting ore under varying conditions, including difficult mining geometry, problematic rock conditions, losses of ore into backfill, and blasting issues. Mining recovery has been estimated to be 95% by weight in the compilation of the underground Mineral Reserves.

15.2.2 Cut-off value

The cut-off value supporting the estimation of underground Mineral Reserves is based on projected operating costs for Phases 1 and 2. The projected operating costs indicated that NSR cut-offs of US$37.49/t ore and US$34.42/t ore would cover all site costs and G&A costs on a breakeven basis for Phase 1 and 2, respectively. The following Table 15.7 includes the cut-off value buildups. The pit tailings placement cost has been incorporated into the Tailings Filtration Plant cost.

Table 15.7 Phase 1 and 2 cut-off value estimates

Function	Phase 1 * average (US$/t ore)	Phase 2 ** average (US$/t ore)
Mining supervision & management	0.74	0.43
Safety & training	1.05	0.27
Ore & waste development	3.16	2.06
SLOS	5.89	5.01
Fixed plant consumables	0.28	1.20
Backfilling	3.57	3.67
Mine rehabilitation	0.32	0.29
Mine general	4.77	3.47
Maintenance supervision & management	0.46	0.23
Maintenance labour	1.31	0.86
Technical services	0.57	0.39
Mine operating costs	22.12	17.87
Site operating costs	12.81	14.72
Site-wide Sustaining Capital	2.57	1.83
Total costs	**37.49**	**34.42**

Notes: Numbers may not add exactly due to rounding.
* Production period ore tonnes only (From 01 January 2027 through 31 December 2033).
** 1 January 2034 through LOM.

The Skouries underground Mineral Reserves also include weakly mineralized development material for which mining costs will have been sunk and only the unit cost of milling (originally estimated at US$10.60/t) remains to be expensed to recover saleable metal. This marginal material represents a small fraction (approximately 0.5%) of the overall underground Mineral Reserves.

The site operating costs have been calculated by prorating the overall site costs based on the percent of ore mined from the underground. An example is the unit process plant costs being higher in Phase 2, due to less tonnes on average are being processed annually. This is due to the majority of the tonnes coming from the Underground in Phase 2.

The NSR parameters on which the estimated Mineral Reserves are based are presented in Table 15.1.

15.2.3 Consideration of marginal ore

Marginal stope material is located within potential stopes within the extents of the sub-level development. This marginal material amounts to approximately 11 Mt that could be extracted at a cost of $31.1/t. This material was included in the 2018 Eldorado Technical Report estimate of Mineral Reserves but has been excluded in this Mineral Reserve estimate. The Marginal Stope material being removed from the underground Mineral Reserves is the main contributing reason for the reduction of ore tonnes from the 2018 PFS.

15.2.4 Underground Mineral Reserve

In the 2018 PFS, the underground contribution to Mineral Reserves was evaluated at an NSR cut-off of US$33.33/t, incorporating 5% external diluting material by weight that is assumed to carry no metal value and assuming an overall mining recovery of 95%.

In this FS study the same cut-off was carried through the project, which does not match the Phase cut-offs noted in Section 15.2.2. See Section 15.2.5 for a detailed explanation of the difference and the assessment of impact on the Mineral Reserve estimation. The underground Mineral Reserves estimate, as of 30 September 2021, is presented in Table 15.8. A reduction of 10.6 Mt is present in this Mineral Reserve estimation compared to the 2018 PFS.

Table 15.8 Skouries underground Mineral Reserves, as of 30 September 2021

Category	Ore (kt)	Grade Au (g/t)	Grade Cu (%)	Contained Au (koz)	Contained Cu (kt)
Proven	24,556	1.33	0.69	1,051	170
Probable	62,964	0.74	0.53	1,488	331
Proven & Probable	87,519	0.90	0.57	2,539	502

Notes:

- Cut-off value applied: NSR US$33.33/t ore
- Gold Price: US$1,300/oz
- Metallurgical Gold Recovery: 92.62 – 17.5 x oxide (%) – 22 x $e^{(-1.2 \times \text{Au Grade (g/t)})}$
- Copper Price: US$2.75/lb
- Metallurgical Copper Recovery: 99.41 – 56 x oxide (%) – 41 x $e^{(-338 \times \text{Cu Grade (\%)})}$
- Mining Recovery: 95%
- Mining Dilution, Ore Development: 5.0%, Porphyry Stopes: 5.0%, Schist Stopes: 5.5%
- Numbers may not compute exactly due to rounding.

15.2.5 Tonnes below cut-off in plan

The Skouries project has included some material that is below the indicated cut-off values of US$37.49 and US$34.42 in Section 15.2.2 for Phase 1 and Phase 2, respectively. A summary of the tonnes and metal content planned to be extracted by year is shown below in Table 15.9. In total, 4.1% of the tonnes in the mine plan are below the Phase 1 or 2 cut-off value discussed in Section 15.2.2, however only 2.2% of the contained gold and 2.9% of the contained copper are within those tonnes. These tonnes are multiple years away from being mined as they are on the periphery of the orebody, and the QP considers that they do not materially impact the estimation of Mineral Reserves or the overall economics of the project.

Table 15.9 Below cut-off value tonnes in mine plan

Phase	Year	Ore (t)	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz)	Contained Cu (t)
Phase 1	1	-	-	-	-	-
	2	-	-	-	-	-
	3	-	-	-	-	-
	4	-	-	-	-	-
	5	-	-	-	-	-
	6	47,633	0.62	0.38	944	181
	7	166,114	0.55	0.41	2,958	673
	8	69,712	0.62	0.37	1,399	255
	9	94,659	0.55	0.41	1,668	388
Phase 2	10	162,012	0.61	0.32	3,168	521
	11	140,506	0.57	0.34	2,587	481
	12	140,126	0.59	0.33	2,678	464
	13	166,377	0.46	0.42	2,471	699
	14	400,870	0.47	0.41	6,034	1,657
	15	283,511	0.50	0.40	4,522	1,128
	16	164,768	0.42	0.45	2,232	741
	17	-	-	-	-	-
	18	284,552	0.45	0.43	4,143	1,213
	19	590,712	0.45	0.43	8,618	2,516
	20	617,034	0.43	0.44	8,626	2,704
	Total	**3,328,587**	**0.49**	**0.41**	**52,047**	**13,621**

15.3 Mineral Reserve summary

The combined Mineral Reserves for the Skouries Project, as of 30 September 2021, are stated in Table 15.10. These represent the sum of the open pit (Table 15.5) and the underground Mineral Reserves (Table 15.8). As previously mentioned, the reporting cut-offs for the Mineral Reserves are NSR-based, with US$10.60/t used in the open pit estimate and US$33.33/t for the underground estimate.

Table 15.10 Skouries, underground and open pit Mineral Reserves, 30 September 2021

Category	Ore (kt)	Grade Au (g/t)	Grade Cu (%)	Contained Au (koz)	Contained Cu (kt)
Proven	73,101	0.87	0.52	2,053	381
Probable	74,014	0.66	0.48	1,576	359
Proven & Probable	**147,116**	**0.77**	**0.50**	**3,630**	**740**

Notes:

- Cut-off value applied, Open Pit: US$10.60/t ore; Underground: US$33.33/t ore
- Gold Price: US$1,300/oz
- Metallurgical Gold Recovery: 92.62 – 17.5 x oxide (%) – 22 x $e^{(-1.2 \times \text{Au Grade (g/t)})}$
- Copper Price: US$2.75/lb
- Metallurgical Copper Recovery: 99.41 – 56 x oxide (%) – 41 x $e^{(-338 \times \text{Cu Grade (\%)})}$
- Mining Recovery, Open Pit: 100%, Underground: 95%
- Mining Dilution, Open Pit: 0.0%; Underground - Ore Development: 5.0%, Porphyry Stopes: 5.0%, Schist Stopes: 5.5%
- Numbers may not compute exactly due to rounding.

Source: MP and approved by the QPs.

16 Mining methods

16.1 Introduction

The Skouries Project is designed as a two-phase mining operation. Phase 1 consists of a combined open pit and underground mine, operating over nine years. Phase 2 consists of mining from the underground mine only, for an additional 11 years. The total ore producing LOM is 20 years.

The QP for the open pit is John Battista, MAusIMM (CP) of MP and for the underground Gary Methven, P.Eng. of AMC.

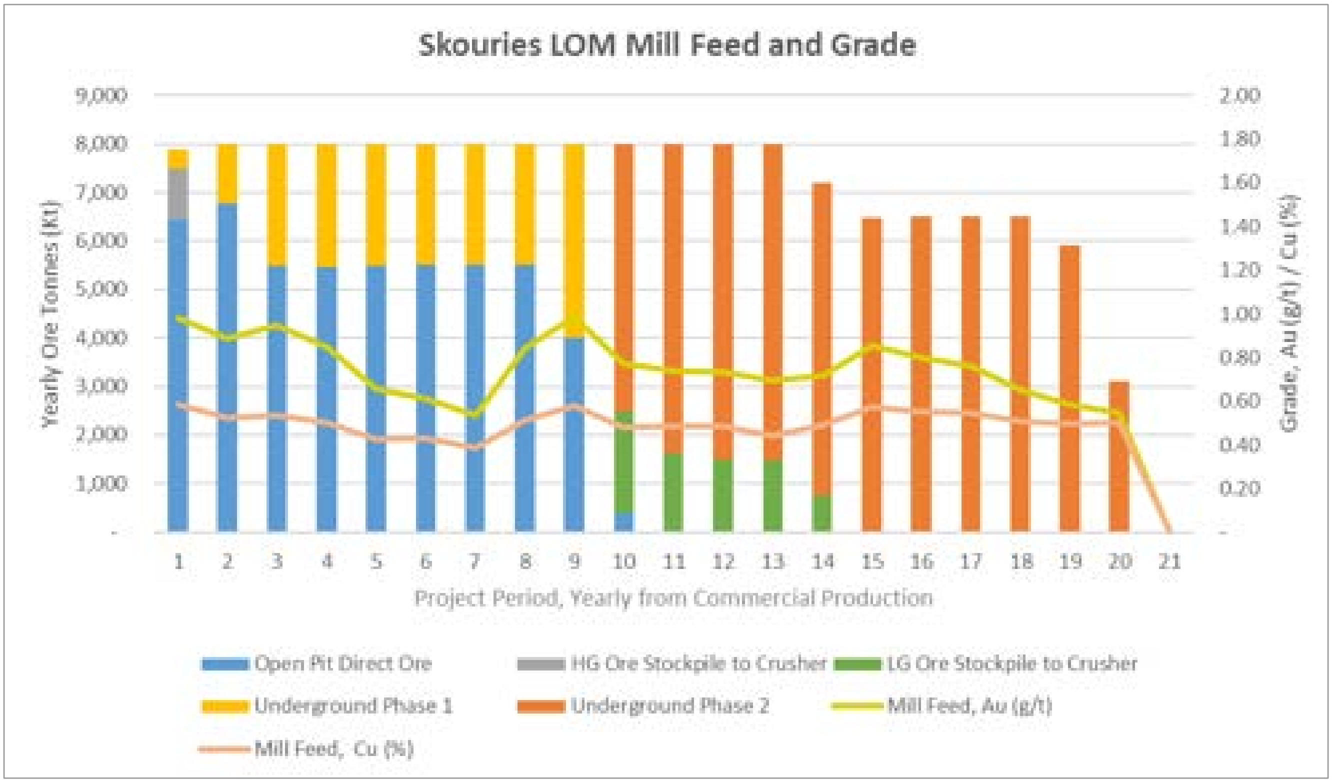
The LOM production schedule is found in Table 16.1. This is different from the mill feed schedule as the low-grade ore stockpile is deferred.

Table 16.1 LOM production schedule and grade

Period	Open Pit			Underground			Total		
	Tonnes (Kt)	Au (g/t)	Cu (%)	Tonnes (Kt)	Au (g/t)	Cu (%)	Tonnes (Kt)	Au (g/t)	Cu (%)
-3	-	-	-	27	0.46	0.34	27	0.46	0.34
-2	60	0.53	0.31	181	0.68	0.51	241	0.65	0.46
-1	1,763	1.00	0.49	302	0.36	0.30	2,066	0.90	0.46
1	8,400	0.73	0.49	404	0.45	0.36	8,804	0.72	0.48
2	8,625	0.67	0.43	1,228	1.13	0.59	9,853	0.72	0.45
3	6,865	0.63	0.40	2,507	1.61	0.78	9,373	0.89	0.50
4	7,475	0.56	0.38	2,523	1.47	0.72	9,998	0.79	0.46
5	5,486	0.29	0.28	2,514	1.47	0.74	8,000	0.66	0.43
6	5,514	0.36	0.33	2,486	1.12	0.62	8,000	0.60	0.42
7	5,505	0.40	0.33	2,495	1.01	0.59	8,000	0.59	0.41
8	5,506	0.51	0.39	2,494	1.62	0.82	8,000	0.86	0.52
9	4,006	0.69	0.47	3,994	1.24	0.68	8,000	0.96	0.57
10	390	0.89	0.52	5,505	0.98	0.57	5,895	0.97	0.57
11	-	-	-	6,375	0.82	0.54	6,375	0.82	0.54
12	-	-	-	6,500	0.86	0.54	6,500	0.86	0.54
13	-	-	-	6,531	0.81	0.54	6,531	0.81	0.54
14	-	-	-	6,443	0.82	0.54	6,443	0.82	0.54
15	-	-	-	6,491	0.83	0.56	6,491	0.83	0.56
16	-	-	-	6,503	0.78	0.55	6,503	0.78	0.55
17	-	-	-	6,496	0.78	0.56	6,496	0.78	0.56
18	-	-	-	6,496	0.63	0.51	6,496	0.63	0.51
19	-	-	-	5,909	0.54	0.47	5,909	0.54	0.47
20	-	-	-	3,115	0.59	0.51	3,115	0.59	0.51
Total	**59,596**	**0.57**	**0.40**	**87,519**	**0.90**	**0.57**	**147,115**	**0.77**	**0.50**

The LOM ore mill feed rate from the mining operation is shown in Figure 16.1.

Figure 16.1 Skouries LOM ore production schedule



Source: MP 2022.

Phase 1 total mill feed rate is 8.0 Mtpa, consisting of a nominal 5.5 Mtpa from the open pit mine combined with 2.5 Mtpa from the underground mine. At the start of the mine life, during the initial two-year underground mine ramp-up period, the open pit feed rate is variable in order to maintain the 8.0 Mtpa mill feed. During Phase 1, 8.0 Mt of low-grade oxide ore is stockpiled to be rehandled for mill feed during Phase 2. Phase 1 is completed at the end of the open pit mine life in Year 9.

Phase 2 mine production, from Year 10 to the end of the LOM, is provided from the underground mine. Phase 2 mine development begins in Year 4 in order to allow a seamless ramp-up from the nominal Phase 1 production rate of 2.5 Mtpa. During the first four years of Phase 2, the mill feed rate of 8.0 Mtpa is maintained by reclaiming oxide ore stockpiled during Phase 1, at a rate which balances the mill feed to 8.0 Mtpa through Year 13. From Year 15, Phase 2 mill feed is maintained at a nominal feed rate of 6.5 Mtpa, solely from underground mine production, which tails off in Years 19 and 20.

16.2 Open pit

16.2.1 Open pit operational and construction phases

The Skouries Open Pit Mine was designed with two operational phases, as summarized below:

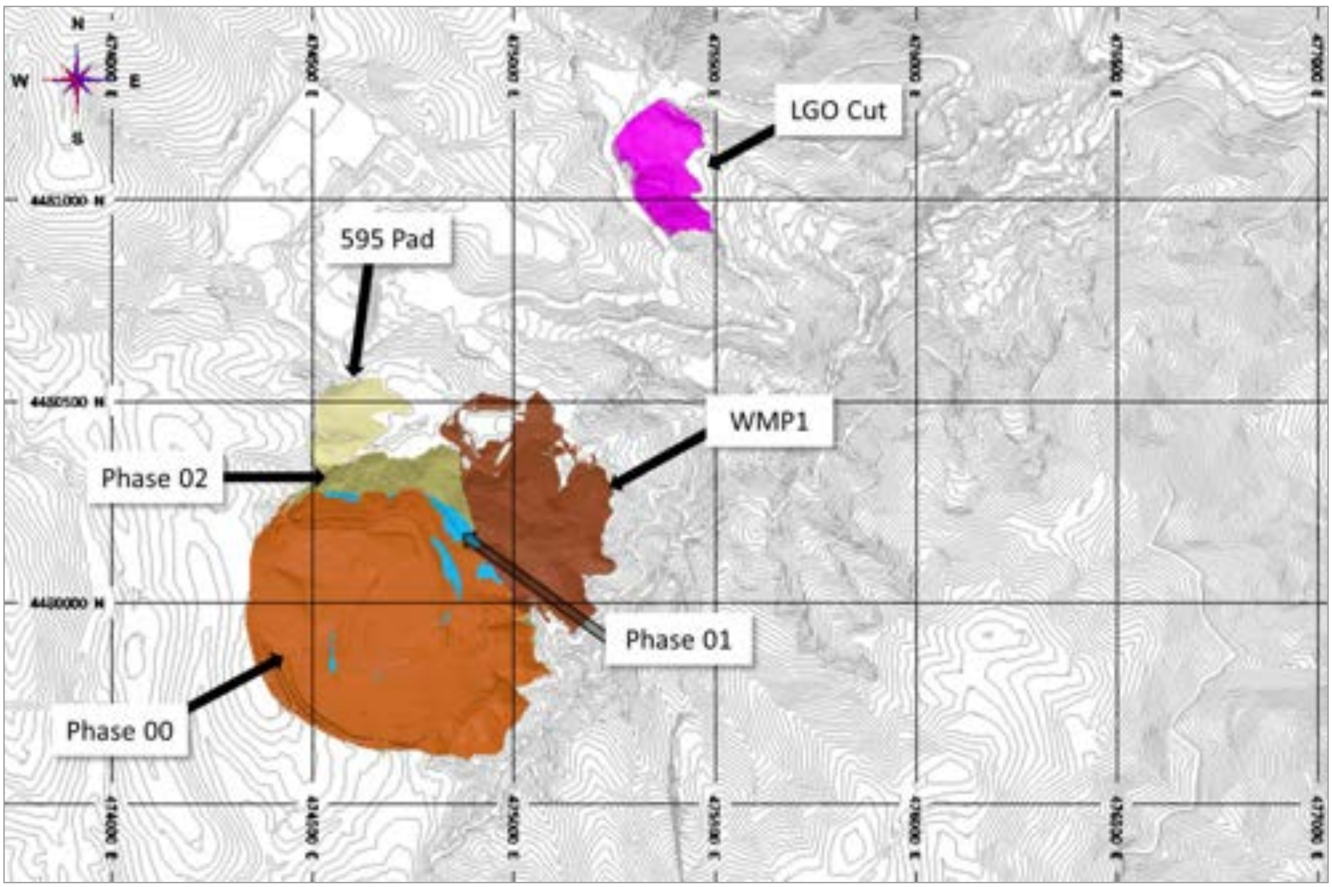
- PH 1: Pit Phase 1 mining
  - Includes free-digging fill material covering the pit
- PH 2: Pit Phase 2 mining
  - Results in Final Pit

Other interim phases were defined for construction purposes and used for open pit scheduling and waste material balance:

- WMP1: Water Management Pond 1 Cut
  - Includes 580 and 565 pads
- 595: Shaft Headframe Pad Cut at 595 masl
- LGO Cut: Low Grade Ore Stockpile Cut

A Phase 00 was considered during the completion of the mine schedule. This phase includes infill material allocated above the surface of Phases 1 and 2 from previous excavation in the surrounding areas, and includes an internal sub-phase dedicated mainly to waste required for the construction of embankments. Phase 00 allows for a reduction in waste stripping and mineralized tonnes mined during the pre-production years. Figure 16.2 shows the operational and construction phases of the open pit.

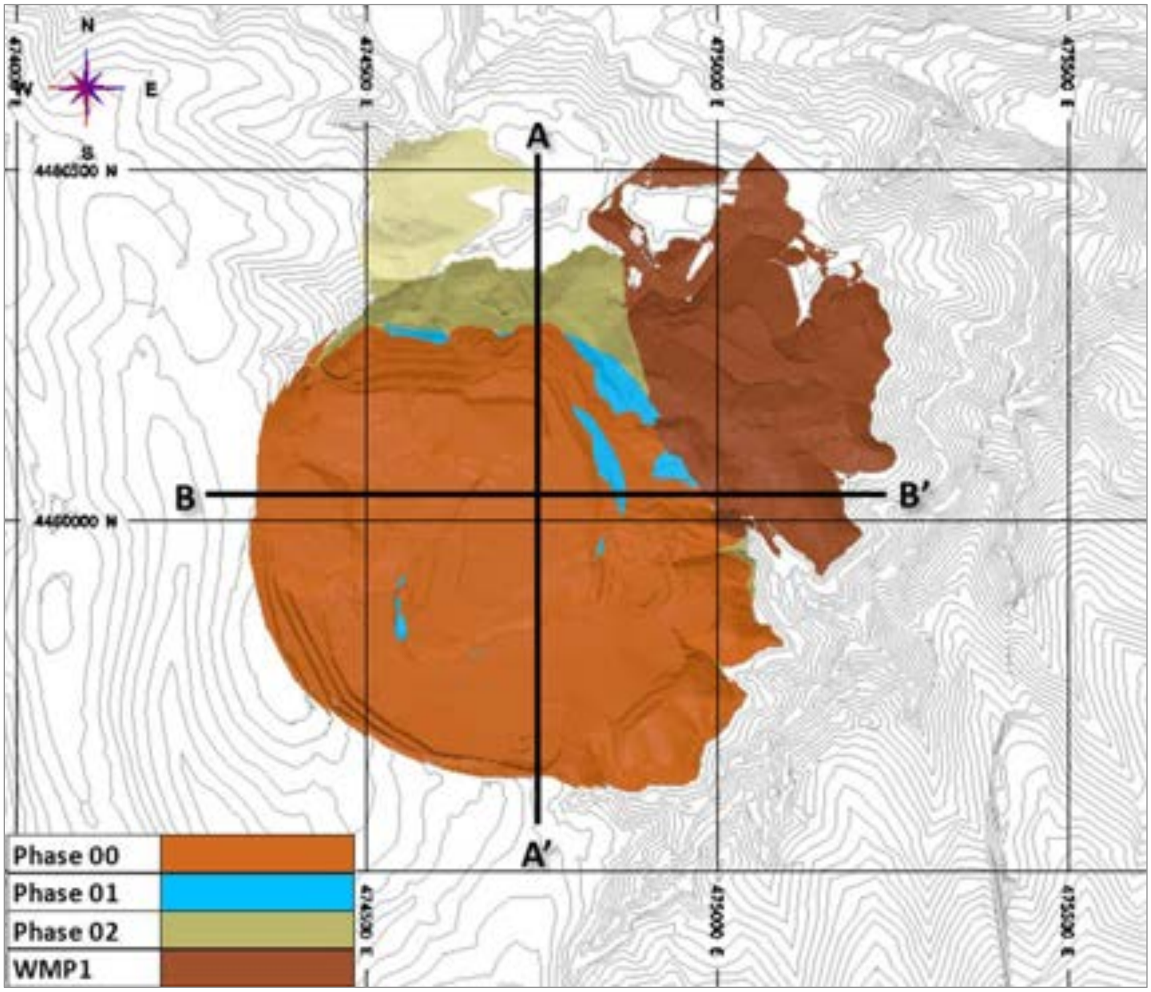
Figure 16.2 Operational and construction phases



Source: MP 2022.

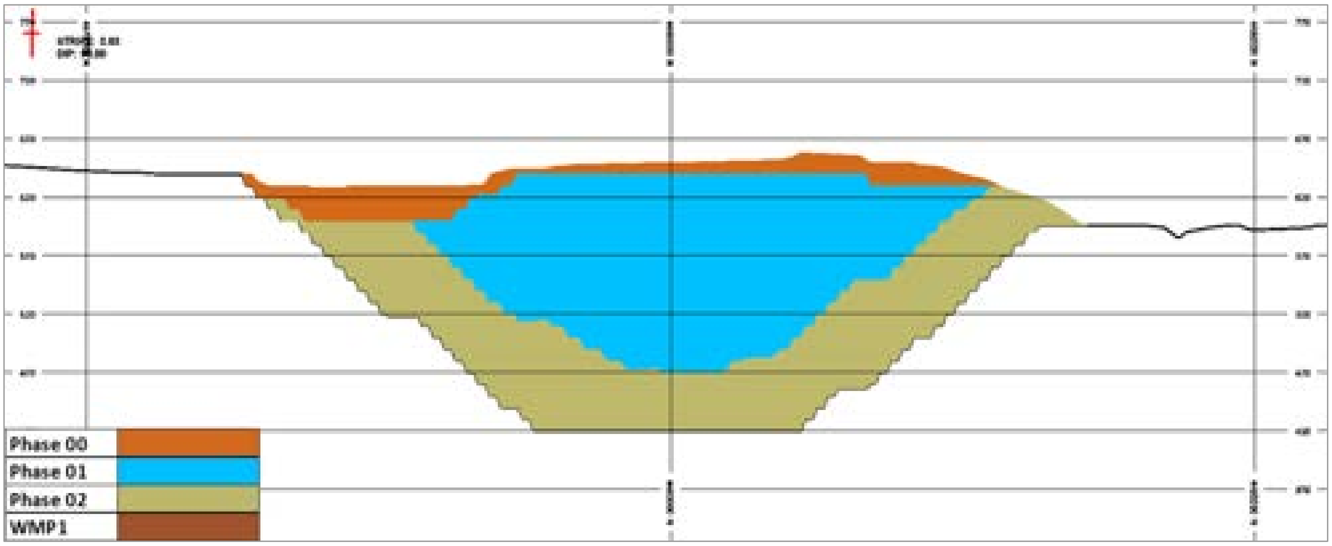
Figure 16.3 shows a plan view of the various phases and Figure 16.4 and Figure 16.5 show cross sections A-A and B-B, respectively.

Figure 16.3 Plan view of open pit and WMP1 area



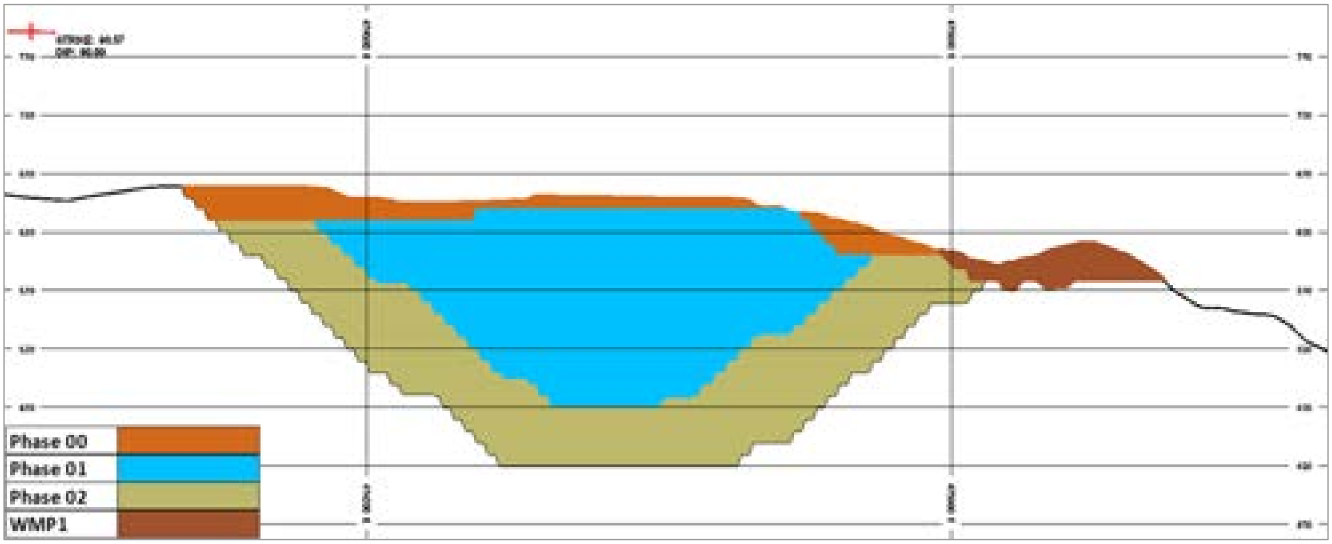
Source: MP 2022.

Figure 16.4 N-S Cross Section A-A′



Source: MP 2022.

Figure 16.5 W-E Cross Section B-B’



Source: MP 2022.

The material content by operational and construction phases is shown in Table 16.2 by ore and waste.

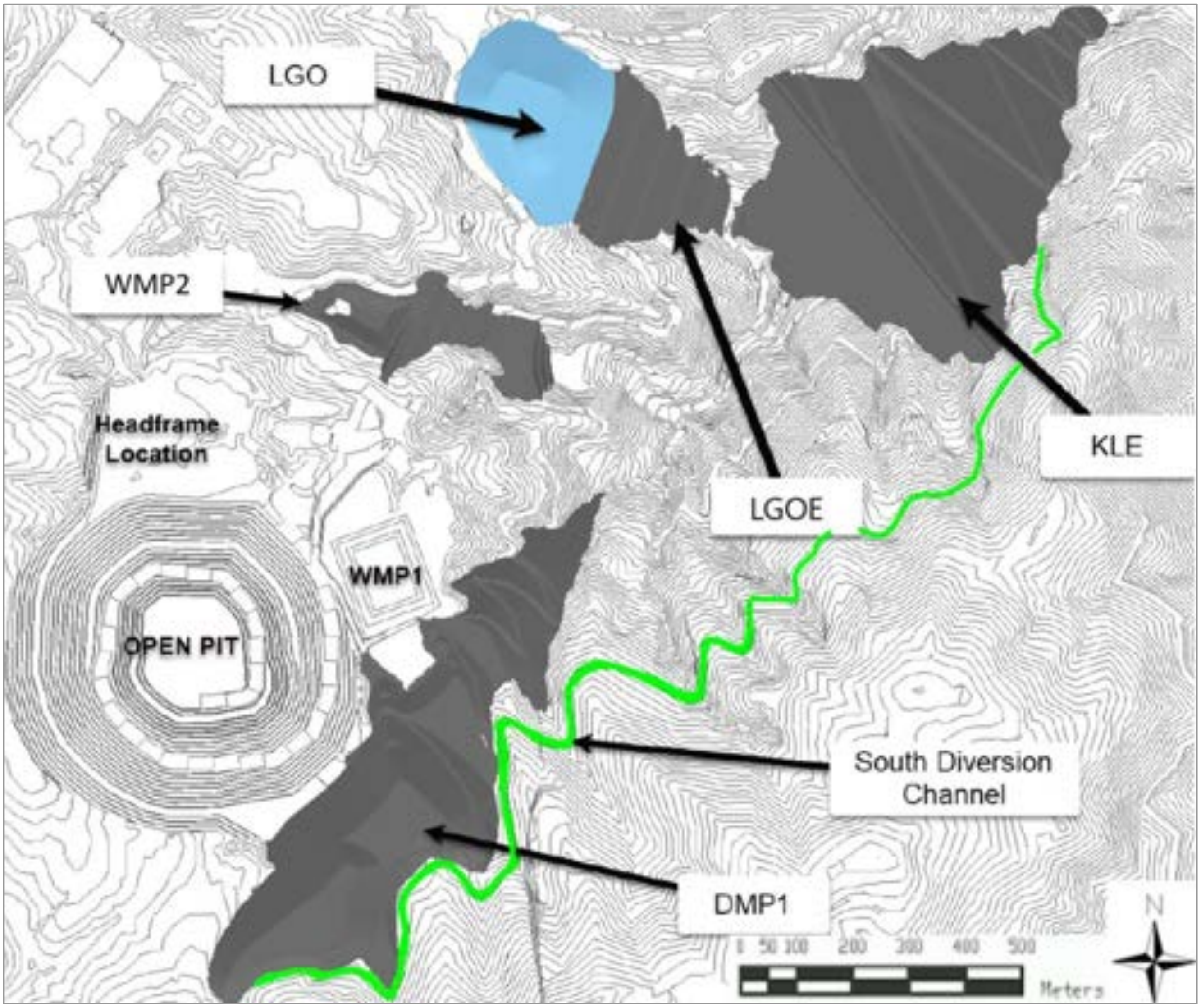
Table 16.2 Material content by operational and construction phases

PHASE (OP)	Ore tonnage (kt)	Gold (g/t)	Copper (%)	Oxide (%)	Waste tonnage (kt)	Total tonnage (kt)	Stripping ratio
WMP1					6,068	6,068	
LGO Cut					1,375	1,375	
595					688	688	
SDC					258	258	
PH 00	91	0.43	0.28	69.7	11,865	11,956	130.38
PH 1	27,649	0.75	0.46	17.5	8,169	35,818	0.30
PH 2	31,857	0.41	0.34	0.0	31,958	63,815	1.00
Total material	**59,597**	**0.57**	**0.40**	**8.2**	**61,407**	**121,004**	**1.03**

The Low-Grade Ore will be placed in a stockpile and the waste quantities shown in Table 16.2 will be placed in the locations listed below and shown in Figure 16.6.

- WMP1: Water Management Pond 1
- WMP2: Water Management Pond 2
- KLE: Karatza Lakkos Embankment
- DMP1: Capping Rock Dump 1
- LGO: Low Grade Ore Stockpile
- LGOE: Low Grade Ore Stockpile Embankment
- SDC: South Diversion Channel

Figure 16.6 Construction dumps and embankments



Source: MP 2022.

The WMP1, LGO Cut, 595 Pad and fill material in PH 1 are the first areas to be mined in the pre-production stage in Years -3 and -2, with the aim of enabling the platforms for mining components.

Phase 1 consists of a combined open pit and underground mine, operating for nine years. Phase 2 consists of mining the underground mine only, for a further 11 years. The total LOM is 20 years.

The production schedule has been developed to balance the materials volumes, metal production and capital expenditures over time with consideration for the capacity of the surface tailings and waste management facilities. See Table 16.3 for a full accounting of the waste material balance.

Table 16.3 Waste material balance

Period	Waste excavation (kt)								Waste deposition (kt)					
	WMP1	LGO Cut	595	PH 00	PH 1	PH 2	SDC	UG	WMP2	Coffer	KLE	DMP1	LGOE	SDC
-3	6,068	-	-	522	-	-	-	-	1,655	123	4,475	-	337	-
-2	-	1,375	688	6,745	-	-	129	211	-	-	5,368	117	3,651	12
-1	-	-	-	4,597	1,344	-	129	445	-	-	2,154	3,970	380	12
1	-	-	-	-	5,170	2,563	-	370	-	-	5,976	2,126	-	-
2	-	-	-	-	1,552	5,523	-		-	-	4,918	2,157	-	-
3	-	-	-	-	102	7,354	-		-	-	5,666	1,791	-	-
4	-	-	-	-	2	6,901	-	-	-	-	5,382	1,521	-	-
5	-	-	-	-	-	4,950	-	-	-	-	5,034	84	-	-
6	-	-	-	-	-	2,952	-	-	-	-	1,586	1,366	-	-
7	-	-	-	-	-	1,347	-	-	-	-	632	715	-	-
8	-	-	-	-	-	303	-	-	-	-	-	303	-	-
9	-	-	-	-	-	66	-	-	-	-	-	66	-	-
Subtotal	**6,068**	**1,375**	**688**	**11,865**	**8,169**	**31,958**	**258**	**1,026**	**1,655**	**123**	**41,191**	**14,047**	**4,369**	**24**
Total	**61,407**								**61,407**					

A low-grade ore stockpile (NSR values from US$10.62/t ore to US$18.00/t ore) will be built up during Phase 1, with ore above NSR value US$18.00/t being fed directly to the crusher. The stockpile will receive 7.2 Mt in Years 1 through 4; this material will be held until the end of the open pit mine life to enable 8 Mtpa to be milled until the underground mine ramps up to its full 6.5 Mtpa. US$18.00/t ore is selected as the cut-off between low grade and direct feed ore based on the available higher grade tonnes in-pit that are projected to be mined in Years 1 through 4.

16.2.2 Open pit mining method

Open pit mining will be by conventional truck-shovel operation, with an ore production rate of approximately 5.5 Mtpa, at a waste to ore stripping ratio of 1.03 The mining sequence will consist of drilling, blasting, loading, and hauling of ore and waste materials for processing and waste disposal. Based on the modelled rock types, approximately 17% of the mined material is amenable to free digging; this material will not be blasted.

Direct feed ore (greater than NSR US$18.00/t value) from the open pit will be hauled to the crusher. As per Figure 16.10, a portion of low-grade ore will be hauled to the LGOS, where it will be re-handled during Phase 2 of the Project; in later years the low-grade ore will be hauled directly to the crusher as it is mined.

Waste material will be hauled directly to one of the material management structures within the IEWMF. The structures internal to the IEWMF are the LGO embankment, WMP2, Capping Rock Dump1, Cofferdam KL Embankment, and South Diversion Channel.

16.2.3 Open pit drilling and blasting

Drilling operations will be carried out continuously as part of the normal mining operation. Once full mine production is reached, drilling and blasting of approximately 1 Mt (dry) per month will be required to maintain production levels. Drilling and blasting activities will be carried out by Hellas Gold, with bulk explosives and associated blasting accessories being delivered to site as needed by an explosives contractor.

Waste material classified as red clay and overburden will not be drilled or blasted as it is considered free dig material. All other waste and ore material types (weak rock and hard rock) will be drilled and blasted using the specifications outlined in Table 16.4. These parameters will be investigated and updated as the operation is brought into production and throughout the life of the open pit to optimize costs, fragmentation, overbreak and ore movement.

The drilling unit selected for the open pit will have the capability to drill with a rotary only arrangement for soft materials that may be encountered early in the mine life, and the ability to drill with an in-the-hole percussive hammer (ITH) for the hard rock materials that will make up the majority of the drilled rock later in the Project. A single pass drill string and mast of 12 m will be used. Each hole within the ore or on the boundary of the ore zone will be sampled for grade control and planning purposes. The drill cuttings can be sampled directly as the hole is being drilled and it is not necessary to split the sample into an upper or lower bound, as the orebody is deemed to be continuous over the 10 m sampled interval.

Table 16.4 Open pit production drilling and blasting specifications

Item	Specification
Bench height	10 m
Sub-drill	1.8 m
Burden	5.5 m
Spacing	6.1 m
Stemming	6.0 m
Borehole diameter	216 mm
Hard rock powder factor	0.67 kg/m^3
Weak rock powder factor	0.40 kg/m^3
Explosive type	70 / 30 Blended Emulsion / ANFO mix
Delay type	None

Wall control blasting will be used adjacent to final and intermediate pit walls to prevent over breakage of the wall and to maintain overall wall stability and safety. Wall control blasts will use pre-split holes along the row closest to the final wall and two adjacent buffer rows. The pre-split holes will be drilled at an approximately 70° angle and parallel to the pit wall using a 114 mm hole diameter. The two buffer rows will have a lower powder factor than a regular production hole and no subdrill. The buffer rows will be drilled using a 216 mm hole diameter.

16.2.4 Open pit hauling

The primary haul roads are designed at 25 m width, based on a 90 t haul truck. Other haul roads, to be used by contractor trucks, are designed for 55 t articulated haul truck with an overall roadway width of 15 m. Table 16.5 outlines the road widths for each truck class. Road grades are limited to 10% in-pit and ex-pit for the 90 t trucks and approximately 12.5% for the contractor trucks. Runaway lanes will be constructed as required for safe operations. The smaller 55 t haul truck has been selected for waste as the route which the waste needs to travel over to the various dump areas is winding along steep terrain. Figure 16.7 provides a picture of the type of trucks selected.

Figure 16.7 Graphic showing type of trucks selected (CAT 777 90 t and Volvo A60H 55 t trucks)

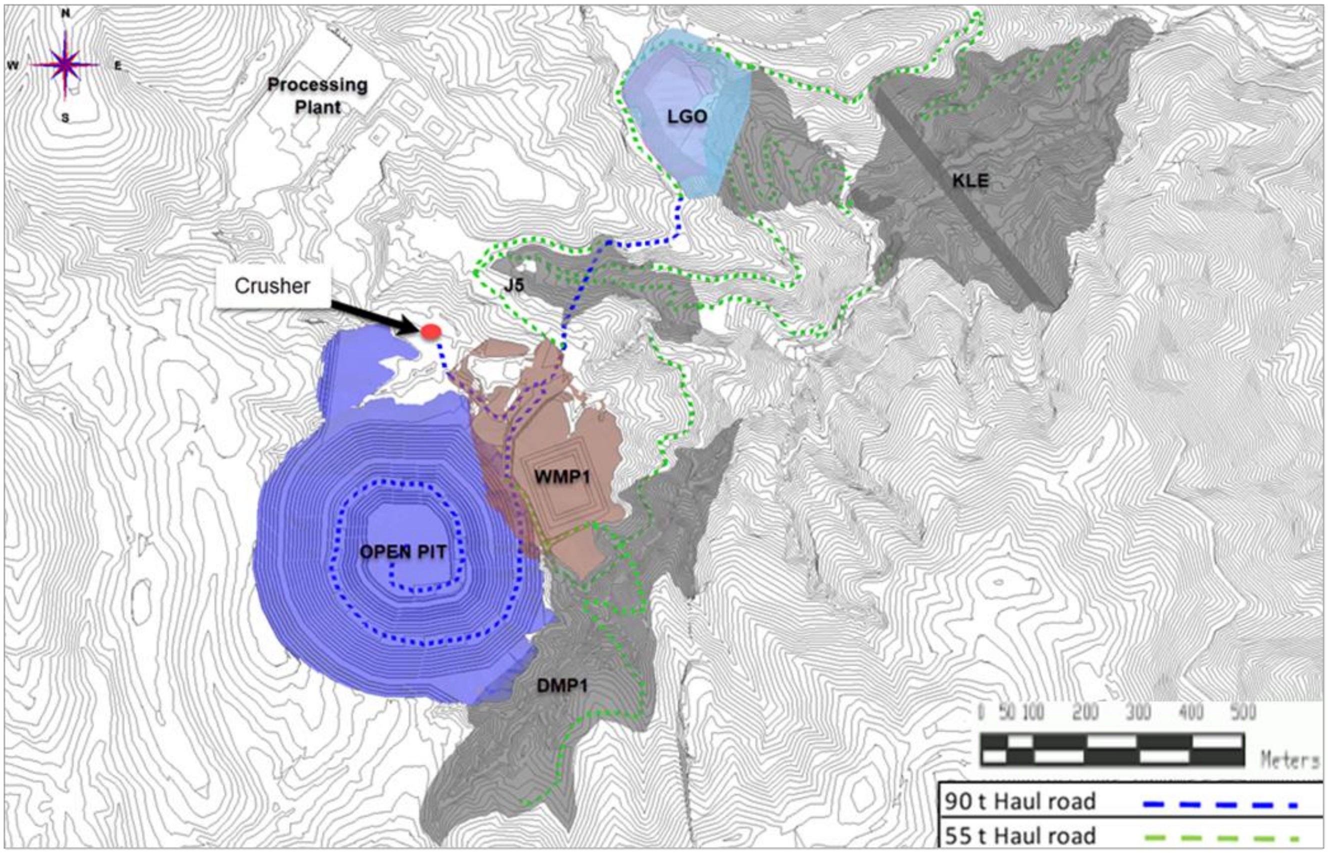
Source: MP 2022.

Table 16.5 Haul road design widths

Haul truck class / size	Vehicle width (m)	One-way traffic road width (m)	Two-way traffic road width (m)
55 t- Contractor	4.0	8.3	15
90 t-Owners	6.5	13.5	25

A material movement study was conducted to determine truck, loader, and support equipment requirements. The study concluded that the 90 t trucks would transport ore and the 55 t trucks would transport the waste. As can be seen in Figure 16.8, the routes shown as dashed lines in blue are to be constructed wide enough to allow access for the 90 t trucks that will transport ore from the open pit to the crusher, as well as ore from the open pit to the stockpile (LGO) and, ultimately, from the stockpile to the crusher. The dashed lines in green represents the roads where mainly 55 t trucks will transport waste from the open pit to the various dump sites.

Figure 16.8 Haulage routes of open pit material



Source: MP 2022.

The number of haulage units was determined by calculating cycle times in Haulage© using annual haul cycle profiles from MinePlan©. Haulage calculations were carried out based on the designated 90 t and smaller 55 t trucks. A maximum truck speed limit of 50 km/h was set for flat or inclined roads, reducing to 15 km/h near shovel and dump points and 15 km/h around switchback corners. On the downhill segments, speeds were limited to a maximum of 25 km/h.

A tonnage factor for each material type was used to determine actual payload vs theoretical maximum payload for each truck class. These factors were based on experience from operations at other sites and QP agrees with those values. Table 16.6 lists the factors used, and subsequent truck tonnages used for determining the number of units required.

Table 16.6 Material fill factors and adjusted haul truck capacities

	Hard rock (C)	Overburden (B1)	Weak rock (B2)	Red clay (A)
In situ density (t/m^3)	2.7	2.2	2.2	2.0
Swell	50%			
Broken density (t/m^3)	1.8	1.5	1.5	1.3
Material fill factor	1	0.7	0.85	0.65
55 t	55 t	38.5 t	46.8 t	35.8 t
90 t	90 t	63 t	76.5 t	58.5 t

16.2.5 Open pit loading

The primary mining loading fleet will consist of a conventional 12 m^3 diesel hydraulic excavator and two front-end loaders with 8.5 m^3 and 12 m^3 buckets. The strategy for the loading fleet is to place the most productive unit, the excavator, on an ore face to maximize its utilization. At the same time, the slightly lower productivity front-end loader will be placed on the waste faces where it will have the ability to move between material types in different parts of the pit to help manage the construction requirement of the embankment and other IEWMF structures. The specifications of the primary loading fleet were selected to match the 90 t haul trucks that are used for open pit hauling, and were determined based on the number of passes, the material loose density, and height clearance of the machines. Other than in the pre-production period, it is envisioned that 90 t trucks would transport ore and the 55 t trucks would transport waste; both will be loaded in the open pit.

16.2.6 Open pit mining fleet

The primary open pit fleet was sized to match the overall production schedule. The fleet was calculated yearly based on the number of hours of assumed mechanical availability and utilization according to the work schedule parameters shown in Table 16.7. The assumed maximum equipment utilization varies from 6,000 to 6,600 hrs/yr depending on the type of equipment. Due to the approximately nine-year mine life and annual operating hours, it is estimated that no major equipment will need to be replaced or purchased, however repairs and rebuilds have been budgeted as necessary. The maximum number of units required in any given year of the Project are shown in Table 16.8. The fleet size generally will decrease over time as the quantity of waste material moved in the later years of the Project reduces.

Table 16.7 Open pit work schedule parameters

Parameter	Value
Mechanical availability	85%
Utilization	88%
Shift length	8 h
Productive hours per shift	7 h
Shifts per day	3

Table 16.8 Open pit owner primary mining fleet

Equipment type	Class / size	Max number of units
Ore haul trucks	90 t	5
Ore haul cycle average	13.2 min	
Waste haul trucks	55 t	19
Waste haul cycle average	30.5 min	
Excavators	12 m^3	1
Front end loaders	12 m^3	1
Front end loaders	8.5 m^3	1
Blasthole drills	Rotary and ITH, 114 mm to 216 mm	2
Pre-split drill	Top Hammer 114 mm	1

Support equipment estimates depend on the numbers of larger equipment (excavator and trucks) and activities required, and similar assumptions to the previous estimates at PFS have been used. Table 16.9 summarizes total support equipment requirements, which are generally fixed throughout the life of the Project.

Table 16.9 Open pit owner support mining fleet

Equipment type	Class / size	Total quantity
Dozer	CAT D8	3
Wheel dozer	CAT 834K	1
Grader	CAT 12M & 16M	2
Small excavator	CAT 345	1
Water truck	30,000 L	2

16.2.7 Open pit personnel

The work schedule assumes 350 days of 24 h/day mining operation. 15 days are assumed to be lost due to inclement weather and holidays. Operations and mining personnel will work on three 8 h shifts per day as shown in Table 16.7. The mine will be split between Owner and Contractor labour, with a major shift in Year 1 to mostly Owner labour except for the waste haulage trucking.

Staff and labour requirements over the LOM for operations and maintenance departments are summarized in Table 16.10.

Table 16.10 Open pit annual personnel requirements

Description	Year														
	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Mine staff															
Mine Manager	0	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Production Supervisor	2	2	2	4	4	4	4	4	4	4	4	2	2	2	2
Drill-blast Supervisor	0	3	4	4	4	4	4	4	4	4	4	0	0	0	0
Chief Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Trainer	2	4	4	4	4	4	4	4	4	4	4	2	2	2	2
Mining Engineer	2	1	2	2	2	2	2	2	2	2	2	2	2	2	2
Senior Geologist	1	1	2	2	2	2	2	2	2	2	2	1	1	1	1
Pit Geologist	1	1	2	2	2	2	2	2	2	2	2	1	1	1	1
Grade Control Technician	2	2	4	4	4	4	4	4	4	4	4	2	2	2	2
Mining Technician	2	1	2	4	4	4	4	4	4	4	4	2	2	2	2
Chief Surveyor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Survey Technician	1	1	2	2	2	2	2	2	2	2	2	1	1	1	1
Surveyor Assistant	1	1	2	2	2	2	2	2	2	2	2	1	1	1	1
Mine Clerk	1	1	2	2	2	2	2	2	2	2	2	1	1	1	1
Subtotal	19	23	33	37	37	37	37	37	37	37	37	20	19	19	19
Mine - operations															
Drill Operator*	12	8	12	12	12	12	0	12	8	8	8	8	0	0	0
Drill Helper / Sampler	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Blasting Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Blaster Helper	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Explosives Truck Operator	4	4	4	4	4	4	4	4	4	4	4	4	4	0	0
Operator of Explosives Magazine	4	4	4	4	4	4	4	4	4	4	4	4	4	0	0
Shovel / Loader Operator*	8	4	4	8	8	8	8	8	8	8	4	4	4	4	4

Description	**Year**														
	-3	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**
777 Haul Truck Operator*	0	0	0	20	16	12	12	8	8	8	12	8	8	4	4
A60h Truck Operator**	76	76	56	64	52	52	48	36	20	8	4	4	*0*	*0*	*0*
Wheel Dozer Operator	4	4	4	4	4	4	4	4	4	4	4	0	0	0	0
Track Dozer Operator	12	12	12	12	12	12	12	12	12	12	8	4	4	4	4
Water Truck Operator	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Grader Operator	8	8	8	8	8	8	8	8	8	8	8	4	4	4	4
Subtotal	138	130	114	146	130	126	110	106	86	74	66	50	38	24	24
Mine - support															
Small Excavator Operator	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Dispatch	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Support Labour	2	6	6	6	6	6	6	6	6	6	6	6	6	2	2
Subtotal	10	14	14	14	14	14	14	14	14	14	14	14	14	10	10
Mine - maintenance															
Maintenance Manager	0	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Mine Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Maintenance Supervisor	2	2	2	4	4	4	4	4	4	4	4	2	2	2	2
Maintenance Clerk	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mechanic Heavy Duty	8	8	8	8	16	16	16	14	14	14	14	10	8	6	6
Mechanic Light Duty	4	4	4	4	8	8	8	8	8	8	8	6	4	4	4
Electrician	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Machinist	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Welder	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Tireman	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Lube / Fuel Serviceman	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Apprentice / Helper	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Washramp Attendant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Subtotal	32	33	33	35	47	47	47	45	45	45	45	37	32	30	30
Total	**199**	**180**	**182**	**232**	**236**	**236**	**236**	**190**	**166**	**170**	**162**	**125**	**103**	**83**	**83**

Notes:
* Labor provided by contractor through the end of Year 1.
** Labor provided by contractor for LOM.

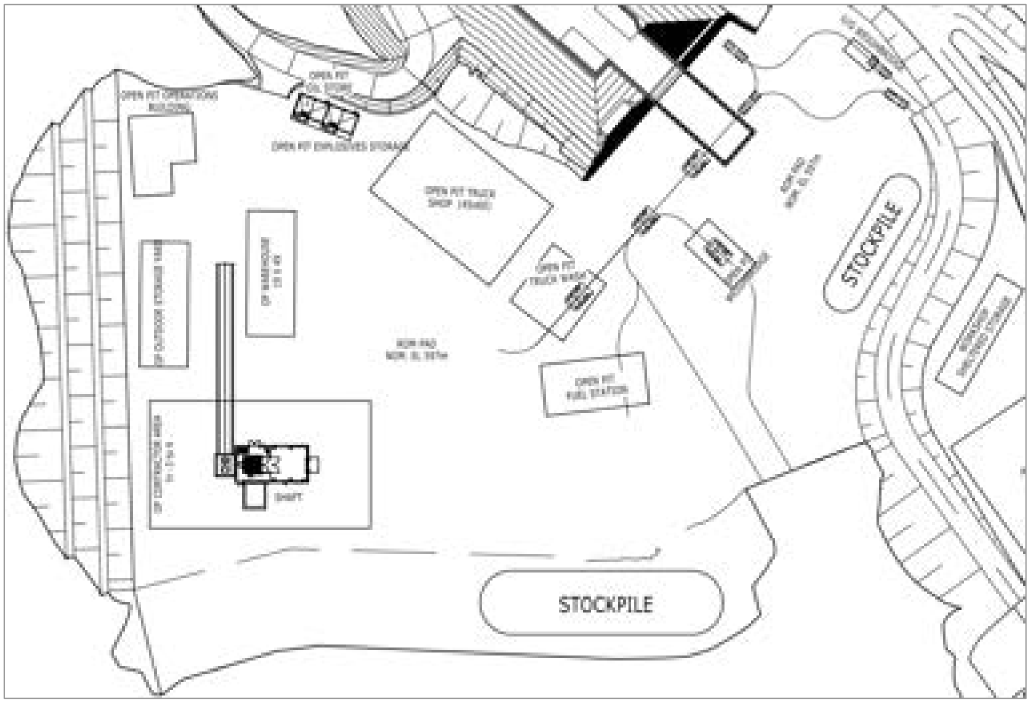
16.2.8 Open pit mine infrastructure

Mine infrastructure design, including ancillary facilities and services, has been fully developed to support the Phase 1 open pit mine production.

16.2.8.1 Ancillary facilities

The location of the facilities is shown on Figure 16.9. Surface ancillary facilities are situated to be close to the open pit access ramp and primary crusher dump pocket. The ancillary facilities include the production services building, surface workshop, warehouse, and surface fuel storage.

Figure 16.9 Open pit mine infrastructure



Note: Schematic not to scale.
Source: Fluor 2022.

16.3 Underground mining

The Skouries orebody that extends below the bottom of the open pit is amenable to bulk underground mining methods and has been evaluated under several different design approaches since the late 1990's, including block caving, sub-level caving and SLOS. SLOS has been confirmed as the most appropriate underground mining method for several reasons including:

- The geo-technical stability of the final reclaimed land after closure of the Project.
- The minimization of land-take needed for the surface tailings.
- The ability to backfill the depleted open pit.

16.3.1 Underground geotechnical

16.3.1.1 Selected rock mass design parameters

The rock mass classification was conducted based on geotechnical core logging and core photograph review using the Rock Mass Rating (RMR) system. The rock mass quality design parameter Q was determined from typical RMR data provided in the geotechnical database for the porphyry domains and information provided by the ramp development for the schists, as shown in Table 16.11.

Table 16.11 Q values for the analysis

Rock unit	Lower bound			Upper bound		
Porphyry	Poor	Intermediate	Good	Poor	Intermediate	Good
	0.1	1.1	5.9	1.1	5.9	31.3
Schist	Very Poor	Poor	Fair	Very Poor	Poor	Fair
	0.2	1.1	3.4	1.1	3.4	10.3

Note: Q determined using the relationship RMR = 9* Ln(Q) + 44.
Source: AMC 2016.

16.3.1.2 Geologic structure and argillic alteration

An assessment of geological structure and argillic alteration (which manifests itself in time-dependent deterioration in strength of core) was made from information provided in the geotechnical database and from information on alteration provided by HG (Hellas Gold, 2015). Table 16.12 is a summary of the main clay-filled discontinuities and faulting data for the porphyry and schist.

Table 16.12 Summary of the main discontinuities and faulting data for the porphyry and schist

Geology	Dip (°)	Dip direction (°)	Comments
Porphyry	87	253	Clay filled less than 5 mm
	88	309	
	89	274	Faults
	87	172	
Schist	71	33	Foliation
	80	24	Clay filled less than 5mm
	72	64	
	90	110	
	84	55	Clay filled greater than 5 mm
	84	55	Faults
	88	101	

The argillic alteration is postulated to be associated with faulting rather than being a pervasive alteration feature (Rhys, 2013). SRK (SRK, 2015) made the following points with regards to clay expansion:

- Dramatic reduction in rock mass quality often observed within days / weeks of drilling.
- Most prevalent in schist, proximal (but not always) to fault zones (and lesser to contact zones).
- Associated with late-stage hydrothermal activity (kaolinite / montmorillonite).
- No clear spatial zoning; patchy distribution in drill record.
- Structural model does not clearly assist with clay expansion zoning.

16.3.1.3 In situ stress

In situ stress measurements were undertaken in early 2017 but were not successful as difficulties with Poor ground and other issues resulted in limited data. It is recommended that planning proceed on the basis of the assumed stress field from the World Stress Map, but that further measurements be taken once development has reached more competent rock in the general area in which test stoping has been proposed. The orientations and magnitudes of the principal stresses derived from the World Stress Map are provided in Table 16.13; these are used as base case in situ stress for geotechnical design and analysis.

Table 16.13 Principal stress magnitudes and orientations: major principal stress vertical

Stress component	Stress orientation (azimuth / plunge)	Stress magnitude (MPa)	Equation
$\sigma_1 = \sigma_v$	275°/84°	0.0265 z	Stress is depth dependent
$\sigma_2 = \sigma_H$	095°/06°	0.0172 z + 0.065	Approx. average between σ_1 and σ_3
$\sigma_3 = \sigma_h$	185°/00°	0.0087 z + 0.033	Uniaxial strain model

Note: z is the depth in metres below surface level.

In contrast to findings from the World Stress Map, the kinematics of regional plate tectonics discussed by Okay (Okay et al, 1999) suggest that the regional maximum principal stress is likely to be horizontal. As such, another hypothetical case of in situ stress is considered and provided in Table 16.14.

Table 16.14 Principal stress magnitudes and orientations: major principal stress horizontal

Stress component	Stress orientation (azimuth / plunge)	Stress magnitude (MPa)	Equation
$\sigma_1 = \sigma_H$	090°/00°	0.053z	$\sigma_1=2\sigma_3$
$\sigma_2 = \sigma_h$	180°/00°	0.0265z	$\sigma_2=\sigma_3$
$\sigma_3 = \sigma_v$	270°/90°	0.0265z	$\sigma_3=\gamma z$

Note: z is the depth in metres below surface level.

16.3.1.4 Stope design

The empirical modified stability graph (EMSG) method developed by Mathews et al. (1981) and Potvin (1988) was used to estimate stable stope design parameters. The majority of the stoping is considered to take place in reasonable quality rock mass. Table 16.15 provides a summary of Q' and parameters A, B, and C to determine the modified stability number N' used for assessment of stope walls along strike. The maximum allowable hydraulic radii (unsupported) were then determined for stope walls based on N'.

Table 16.15 Parameters used for the stability graph method[1]

Stope walls	Q'[2]	A	B	C	N'	Hydraulic radius
AMC, 2016- excavation design - porphyry upper bound	31.3	1.0	0.4	8.0	100.3	10.3
AMC, 2016- excavation design - porphyry lower bound	5.9	1.0	0.4	8.0	18.9	10.3

Notes:
[1] AMC 2016 values were retained for this study.
[2] Q'=Q/ESR, where ESR = 1

The stope stability assessment has indicated that, for stoping in the porphyry, a 60 m sub-level interval (60 m stope height plus 5 m top drive development) can largely be viable without significantly compromising stope wall stability if the length of the stope does not exceed 30 m. Of the stopes that will be extracted in the schist, only half of these excavations will expose schist in the stope sidewalls, as secondary stopes will expose the paste backfill within the primaries. For primary stoping in schists, the maximum unsupported hydraulic radius of stope wall along strike is 7.6, with preliminary stope lengths being set at 20 m for 65 m high stope. The potential impact of exposing schist sidewalls is an increase in dilution.

Stope back stability assessments were conducted using the NGI-Q stability graph as well as the stability graph method to determine appropriate stoping spans. Stope span has been limited to 15 m. Thus, the standard stope dimensions were set to 65 m high x 30 m long x 15 m wide in porphyry stopes, 65 m high x 20 m long x 15 m wide for primary stope design in schist material, and 65 m high x 30 m long x 15 m wide for secondary stope design in schist material.

The assessment indicates that stope backs of 15 m span would be stable without cable bolt support in ground of Good and Intermediate rock mass quality. Within Poor rock mass ground conditions and without cable support, the stoping span would have to be reduced to the order of 7.5 m.

Additional work was conducted to assess both the stope spans and sub-level intervals using these same techniques and assessing the need for and viability of cable bolt support. The results of this work suggest that there may be areas at Skouries where increased stope spans (> 15 m) could be viable with significant cable bolt support. However, the current design of 15 m wide x 30 m long stopes allows for some factor of safety and it is recommended that these dimensions be maintained as the primary width / length stope design basis until actual stope conditions are experienced and understood. Provision for cable bolting has also been made in the mine plan for all stope brows and draw points, and in secondary stoping where blast damage and mining deformations are likely to impact ground conditions.

16.3.1.5 Vertical development design

Ground conditions for vertical development using raise boring were considered during the rock mass assessment. (Note – the use of Alimak Climbers for raise mining has been prohibited at Skouries due to the possible exposure of mine personnel to Poor ground in the raises.) The Poor near-surface weathered ground conditions dictate that partial removal of overburden / weathered rock and pre-support of remaining weathered rock must be done to facilitate stability of the raise walls immediately following boring.

The pre-support methodology involves the installation of steel-reinforced concrete piles around the perimeter of the planned raise location. The maximum length of piles was established to be 44 m and limited to 1.1 m in diameter. Planning and costing proceeded on the basis of pre-supporting to 44 m depth following partial excavation of the weathered material.

Once the raisebore reaming is complete, a remote spraying robot is lowered to apply the design thickness of 10 centimetres (cm) shotcrete over the entire length of the raise. This is the planned extent of ground support required for all raises to surface.

Permanent raise development is required for the primary ventilation circuits. It is planned to raise bore the ventilation raises from 350 Level to the surface. The inter-level raises will be excavated using a raise boring machine and shotcrete will be applied using a robotic arm. Vertical development sizes used in the mine design are summarized in Table 16.16.

Table 16.16 Vertical development dimensions

Description	Size (m)
RARs to surface	3.5 diameter
FARs to surface	3.5 diameter
IRARs	3.5 diameter
Internal FARs	3.5 diameter
Ore bins (coarse and fine)	6.0 diameter
Shaft	8.2 m (7.6 m finished) diameter

16.3.1.6 Test stoping

Two test stopes have been planned in low-mineralized waste rock between the 410 Level and the 350 Level to provide proof of concept for the 65 m high x 15 m wide x 30 m long basic stoping units in both porphyry and schist rock types. The location was selected on the basis that it is close to current development, thus providing the earliest opportunity to complete the testwork and to corroborate key stoping parameters. The development of the 290 Level and 170 Level will be delayed until the stope dimensions are accepted by the regulatory body.

16.3.1.7 Development ground support design

Indicative ground support requirements were estimated for typical drive dimensions and rock types. The recommended rockbolt spacings and lengths are presented in Table 16.17. Ground support requirement does not consider geological structure or possible kinematic failures which will need to be assessed once the ground has been exposed and specific data acquired.

Table 16.17 Recommended rock bolt design

Drive dimensions	Lithology	Rock mass quality	Depth of EDZ	Dead weight of EDZ	Support pressure at FOS=1	Support pressure at FOS=2	Rock bolt spacing	Rock bolt length	Cable-bolt length	Max. length of firing round
			(m)	(t)	(t/m²)	(t/m²)	(m)	(m)	(m)	(m)
5.5 m W × 5.5 m H	Porphyry	Good / Fair	0.7	35	2.0	4.0	2.0	2.4	-	5
		Poor	0.9	43	2.5	5.0	1.5 - 2.0	2.4	-	3
		Very Poor	1.1	52	3.0	6.0	1.5	2.4	-	1.5
6.0 m W × 5.5 m H	Porphyry	Good / Fair	0.8	42	2.2	4.4	2.0	2.4	-	5
		Poor	1.0	52	2.8	5.6	1.5 - 2.0	2.4	-	3
		Very Poor	1.3	63	3.3	6.6	1.5	2.4	-	1.5
5.5 m W × 5.0 m H (Ore X-cuts)	Porphyry	Good / Fair	0.7	35	2.0	4.0	2.0	2.4	-	5
		Poor	0.9	43	2.5	5.0	1.5 - 2.0	2.4	-	3
		Very Poor	1.1	52	3.0	6.0	1.5	2.4	-	1.5
5.5 m W × 5.5 m H	Schist	Fair	1.5	69	4.1	8.2	2.0	3.0	-	5
		Poor	1.9	87	5.1	10.2	1.5 - 2.0	3.0	-	3
		Very Poor	2.3	104	6.1	12.2	1.5	3.0	5	1.5
6.0 m W × 6.0 m H (decline)	Schist	Fair	1.6	84	4.4	8.8	2.0	3.0	-	5
		Poor	2.0	104	5.5	11.0	1.5 - 2.0	3.5	-	3
		Very Poor	2.4	125	6.6	13.2	1.5	3.5	5	1.5

Note: The above numbers are design parameters that will be proved in the field. The Skouries decline is currently successfully using bolt lengths of 3 m.

16.3.1.8 Development through paste fill

The proposed SLOS mining method involves development in paste fill to recover ore located in the 110 Level sill. This type of development has been practiced at many operations around the world and typically, ground support comprises mesh-reinforced shotcrete of minimum 100 mm thickness, sprayed floor-to-floor. A 10 m thickness of paste fill with increased cement content will be planned for the stopes that are located in areas where re-development in fill is required.

16.3.1.9 Crown pillar stability assessment

Based on the updated mine design of open pit and crown pillar mining option and the LOM schedule, 3D, elasto-plastic numerical modelling was undertaken (AMC, 2021) using FLAC3D to investigate underground and open pit interactions and the crown pillar stability through the undercutting process for two in situ stress scenarios described above, as presented in Table 16.13 (Case 1) and Table 16.14 (Case 2). The FLAC3D model consists of the pit and three sub-levels of LOM stopes (350, 290 and 230 Level) underneath the pit.

The modelling results indicate that:

- Both the pit slopes and underground excavation would be stable for both in situ stress cases:
  - Changes in displacement and shear strain on the final pit wall of Case 1 is not significant throughout pit undercutting.
  - Magnitudes and extents of displacement and shear strain on the final pit wall for Case 2 are greater than Case 1 and tend to increase as mining proceeds.
  - High displacement and shear strain are observed near the base of the final pit in Case 2. Localized bench scale failure near the base of the pit may be expected.
  - Rib pillars (secondary stopes) tend to be overstressed for Case 2 or at lower mining levels, which may induce rock mass damage in the ore drives. Rehabilitation would be expected during the mine life.
- Crown pillar stopes upon sill exposure should be stable throughout the undercutting sequence for both in situ stress cases.
  - No deep sliding shear or tensile plane is formed within the fill stopes upon sill exposure.
  - Localized instability in the sill is projected with estimated average depths of sill exposure failure (mining stress induced failure) being 0.4 m and 0.7 m for Case 1 and Case 2, respectively.
  - The average dilution from sill exposure (mining stress induced) is projected to be less than 2% by volume for both cases.

The realized dilution from cured backfill will be dependent on blasting practices and geomechanical performance of the cured backfill. The QP recommends that site operations focus on QA/QC to ensure that the strength of backfill placed in the stopes meets the design requirement and on drilling and blasting practices to minimize the blasting effects of dilution in the production stage. Drilling and blasting with stand-off of approximately 1.0 m from the CMS fill shape will reduce the blast damage dilution and increase the stability of the exposed fill.

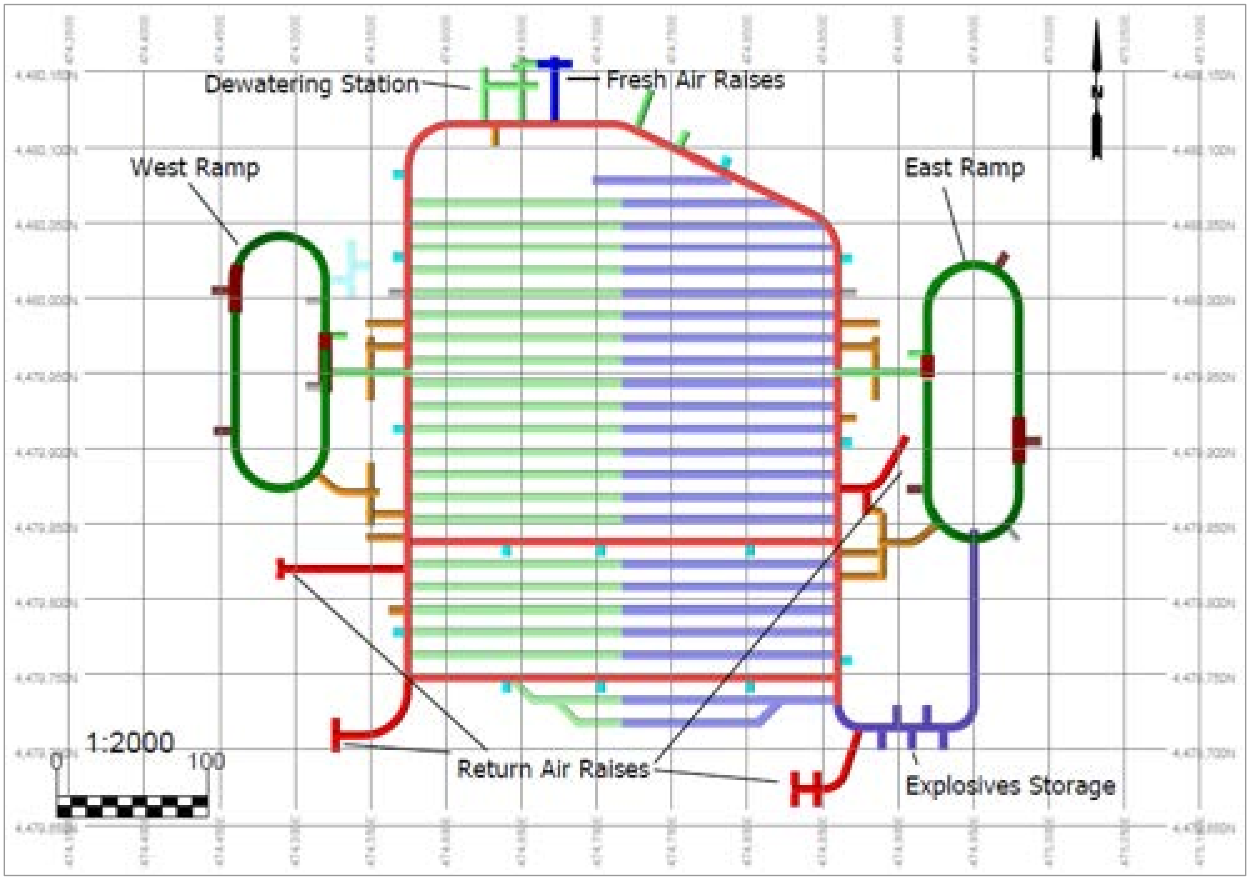
16.3.2 Sub-level design

All levels in both phases have similar designs (see Figure 16.10). Peripheral development (ring-drives) will provide access to all sides of the orebody and terminate at return air raise (RAR) locations. Ore drives for stope extraction will traverse the orebody east to west on 15 m centres, developed incrementally to meet the production schedule and mining sequence. Both ramps are planned to be used to haul ore, with the orebody divided into East and West areas to maintain a stope extraction sequence from the centre out.

Level development priority is to complete the inter-level fresh air raises (FAR), dewatering sumps, and electrical substation to support ongoing level and ramp development. A contact water sump and pumping facility will be located on the north side of each level, whereas non-contact water sumps will be located next to the contact water facilities on 350 Level and 230 Level only. Boreholes for passing electrical conduit and dewatering pipes will be drilled between levels.

The development on the ring-drives will include truck loading bays, compressor cut-outs, portable substation excavations, paste fill cut-outs, stope accesses, and appropriately spaced dewatering nests. The dual ramp system will provide secondary emergency egress for all levels. Definition drilling will be accomplished from the ring-drives or from the ramps.

Figure 16.10 Typical sub-level arrangement (230 Level)



Source: MP 2022.

16.3.3 Lateral development standards

The lateral development standards consider mobile equipment size and clearance requirements, provision of ventilation ducting and other services, ground support requirements, and the ultimate use of the excavation. All development has been sized to accommodate the largest anticipated equipment including an allowance of one-metre minimum clearance on each side. The ore drives (6 m wide) are designed to provide adequate room for efficient production drilling. Table 16.18 shows the standard lateral development dimensions.

Table 16.18 Standard lateral development dimensions

Name	Lateral development	
	Width (m)	Height (m)
Ramp (primary and secondary declines)	6.0	6.0
Ramp passing bays	9.0	7.5
Level access drifts	6.0	5.7
Level development (U-drives)	5.5	5.7
Stope ore drifts (ore drive), overcuts and undercuts	6.0	5.0
Haulage drifts	6.0	6.0
Raise and shaft access drifts	5.5	5.5
Large airway drifts (including slashed portion of TVX adit)	7.0	7.0
Large workshop development (Phase 2)	12.0	10.0

16.3.4 Underground production design

Anticipated ground conditions have played a major role in sizing the stopes. All stopes are designed at 60 m high by 15 m wide, in a primary-secondary transverse arrangement. Primary stopes will be developed and then backfilled in segments of 20 m or 30 m in length, depending on the rock type: 30 m long in porphyry and 20 m long in schist material due to expected less favourable ground conditions. Secondary stopes will lag the primary stopes by at least 60 m transversely and will be mined and then filled in 30 m long segments, regardless of the rock type. Unlike the test stopes, production stopes will be backfilled with paste fill. The stoping methodology is the same for both Phase 1 and Phase 2.

16.3.5 Explosive products

16.3.5.1 Bulk explosive products

Emulsion is a commonly utilized bulk explosive product that can be used for lateral development and can also be pumped both vertically upwards and downwards into stoping blastholes. Bulk emulsion is the primary explosive product that will be used for both the lateral development and stoping at Skouries underground mine.

Emulsion explosives are detonated through the use of detonators that are inserted into boosters (cartridged explosives) in both stoping and development.

16.3.5.2 Detonators

Non-electric detonators are the most basic detonation system and are proposed for development and stoping. They are widely available, relatively simple to use, and are less expensive than electronic detonators.

Electronic detonators are proposed for blast initiation (starter cap) of lateral development and stopes, with non-electric detonators utilized in all other applications of the development and stoping. This reduces cost of blasting and the complexity of the charge process while ensuring precision of blast initiation timing.

16.3.5.3 Stemming

Stemming is a gravel (10 mm to 20 mm) product poured on top of a column of charge in production stopes. It helps maintain the integrity of the remaining portion of a downhole that it not blasted, reduces the impact of the blast on the top sill drive, and assists with confining the explosive energy in the blasthole.

As a generally accepted empirical design rule, total stemming length is recommended to be 20 times the hole diameter to produce good confinement. Production blast design for Skouries envisions a 2.5 m stemming column per downhole, based on a 127-mm blasthole diameter.

16.3.6 Stope layout and design

The Skouries stopes are to be drilled from both the top sill and bottom sill using a standard slot and slash method to open the stope. The slot raises in the open stopes are developed by blasting towards a 1.2 m diameter raisebored void. The initial relief raise blast is designed at 2.6 m x 2.6 m and consists of the following:

- Eight downholes drilled from the top sill that are 127 mm in diameter.
- Eight upholes drilled from the bottom sill that are 89 mm in diameter.

The diameter of the upholes is smaller to facilitate the cohesiveness of the emulsion when the holes are charged.

Once the initial void has been created, adjacent slot blastholes are fired towards the relief raise to create further void. Once the slot is fully developed to the full width of the stope, the main production rings will have a complete free face to blast towards.

Production drill rings will consist of both uphole fans of 19 m (vertical distance) and downhole fans of 40 m (vertical distance) to cover the entire 60 m stope height. This layout has the following advantages:

- Maximizes drilling accuracy.
- Minimizes hole-loading and blasting issues.
- Minimizes dilution.
- Optimizes fragmentation.

Since upholes and downholes are fired together, a 1.0 m stand-off ensures that there is enough interaction between up and down holes to ensure that no unbroken ore bridge exists at the toes of the rings as well as mitigates the risk of up and down holes intersecting. All upholes are drilled from the bottom sill and are 89 mm in diameter. All downholes are drilled from the top sill and are 127 mm in diameter.

16.3.7 Underground mining schedule

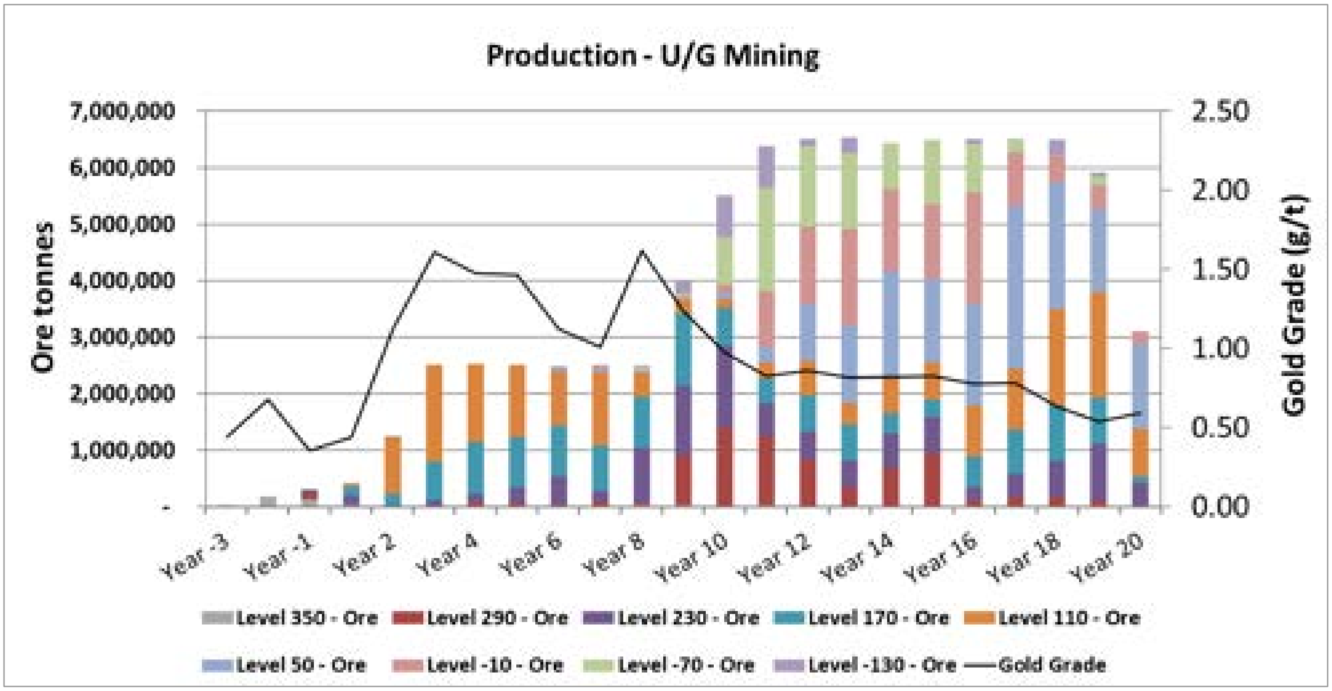
The underground portion of the Skouries Project will begin from the existing ramp from the surface to 385 masl. The ramp is currently developed to 35 m above the first production level, 350L. Mining will proceed down to the 350L to establish major infrastructure and services. The 350L will serve as the mucking horizon for the two test stopes, which are situated in the Crown Pillar and within the mining limits to enable a mineralized and accurate representation of the mining to be completed in Phase 1. The drill drive for these stopes will be established at 410L off a previously excavated remuck in the existing ramp. Excavation on 350L or below will be restricted to essential development only, which will result in a seven-month delay in all development while the test stopes are completed, as shown by the reduction in early waste development metres in Figure 16.11. Upon approval of the stope dimension sub-level design, development of the access ramp will restart.

Figure 16.11 Annual development meters

Source: MP 2022.

For Years -3 through to Year 2, underground mining efforts will focus on developing the access ramp and further establishing the levels and services for production, while also developing a second portal and ramp to the surface. This second ramp will allow for the anticipated 2.5 Mtpa to be produced in Phase 1 from the underground mine, as shown in Figure 16.12. This production rate will be sustained from Year 4 through Year 8.

Figure 16.12 Annual underground production plan



Source: MP 2022.

In Year 4, the development will begin in preparation for Phase 2. This development will entail the dual ramp systems to (-)130L, the major underground workshop, fuel bay and excavations for the materials handling systems that are described in Section 16.3.10. In Year 4, the shaft headframe construction will commence, and shaft excavation will begin in Year 6. Excavation of the shaft will continue through Year 8, with the entire materials handling system projected for completion six months prior to the beginning of Phase 2 in Year 10.

A production ramp-up will begin in Year 9 for Phase 2, as the Open Pit mine is nearing its end of life, culminating in a 6.5 Mtpa ore production rate starting in Year 11 and continuing through to Year 18. Stoping will continue on all levels throughout Phase 2 until they are each exhausted.

The end of the mine life will consist of four levels reducing down to one level producing ore, leading to the reduced annual tonnage in Year 20.

16.3.7.1 Stope cycle time and mine production rate

Overall stope cycle times, without and with remote mining technology (RMT), were estimated from an analysis of the various stoping activities. Cycle times do not include ore development or pre-support (cable bolting) as these activities are scheduled to be completed before the production area is required for stoping. The results of the analysis indicated that, without RMT, each 30 m long stope will produce, on average, 1,017 tonnes per day (tpd) during the mining cycle (drilling, blasting, mucking, but not backfilling) for Phase 1 and Phase 2. A separate cycle time was determined for the 20 m long stopes projected to be required in schist material, resulting in an estimate of 936 tpd for Phase 1 and Phase 2 as shown in Table 16.19.

The average number of active stopes required to achieve the 2.5 Mtpa in Phase 1 is between seven and nine at any given time, of which 30% will be in the backfill cycle. Phase 2 continues to ramp up production to 6.5 Mtpa and the total number of active stopes required to achieve this is 17 to 26 stopes in any part of the cycle. The range of stope numbers reflects the fact that drilling of the next stope in the cycle can start whilst the former stope is being backfilled and cured. The higher number reflects no overlap in those activities whilst the lower number reflects a full overlap.

Table 16.19 Stope cycle time without RMT

Parameter	**Unit**	**Primary / secondary stopes (30 m Length)**	**Primary stopes (20 m Length)**
Stope height	m	60	60
Stope width	m	15	15
Stope length	m	30	20
Recovery	%	95%	95%
Dilution	%	5.00%	5.50%
Undiluted tonnes	t	72,900	48,600
Ore development tonnes	t	2,430	1,620
In situ longhole tonnes	t	70,470	46,980
Stope tonnes recovered, including dilution	t	70,294	47,086
Slot raise drilling	m	1,030	1,030
Ring drilling	m	3,668	2,201
Number of blasts	each	6	6
Average load and haul rate	t/day	2,037	2,030
Volume to backfill	m^3	26,935	18,039
Fill factor	%	97%	97%
Total fill required	m^3	26,127	17,498
Plug volume - 10 m height	m^3	4,500	3,000
Pouring rate	m^3/day	4,800	4,800
Curing time	days	14	14
Operational contingency for lost time	%	25%	25%
Cycle times with contingency			
Longhole drilling	days	20.5	15.8
Stope blasting	days	5.4	5.4
Stope production mucking	days	43.2	29.0
Ancillary activities	days	3.8	3.8
Backfilling	days	29.4	27.5
Total cycle time	**days**	102.3	81.5
Total production rate - excluding fill	t/day	1,017	938
Total production rate - with fill	**t/day**	687	578

This study includes the implementation of RMT, which has an impact on the cycle times of stopes and the productivity of equipment. This technology includes tele-remote operation of mechanized equipment by an operator located on surface or in a remote area of the underground mine. The primary benefit is the ability to operate more hours in a day as the equipment does not need to be stopped during the shift change and travel time for employees to reach underground.

RMT implementation at the Skouries mine is anticipated in Year 1 for development drill jumbos, load haul dumps (LHDs) and production drills, and Year 9 for production trucking (subject to appropriate technology being available). To assess productivity improvements with RMT, a revised cycle time for stopes was prepared and is shown in Table 16.20. This cycle takes into consideration all stope activities, with longer durations of operation assumed on a daily basis for the LHDs and production drill rigs.

This study includes the implementation of RMT, which has an impact on the cycle times of stopes and the productivity of equipment. This technology includes tele-remote operation of mechanized equipment by an operator located on surface or in a remote area of the underground mine. The primary benefit is the ability to operate more hours in a day as the equipment does not need to be stopped during the shift change and travel time for employees to reach underground.

RMT implementation at the Skouries mine is anticipated in Year 1 for development drill jumbos, LHD and production drills, and Year 9 for production trucking (subject to appropriate technology being available). To assess productivity improvements with RMT, a revised cycle time for stopes was prepared and is shown in Table 16.20. This cycle takes into consideration all stope activities, with longer durations of operation assumed on a daily basis for the LHDs and production drill rigs. The projected productivity improvements are 20% for stope mucking, and 9% and 15% for production drilling in 20 m stopes and 30 m stopes, respectively.

RMT is considered a best available technology, and Skouries mine is uniquely positioned to benefit from the improvements in mining process due to the simple, repetitive nature of the mine design and the availability of highly skilled technical workers.

Table 16.20 Stope cycle time with RMT

Parameter	Unit	Primary / secondary stopes (30 m Length)	Primary stopes (20 m Length)
Stope height	m	60	60
Stope width	m	15	15
Stope length	m	30	20
Recovery	%	95%	95%
Dilution	%	5.00%	5.50%
Undiluted tonnes	t	72,900	48,600
Ore development tonnes	t	2,430	1,620
In situ longhole tonnes	t	70,470	46,980
Stope tonnes recovered, including dilution	t	70,294	47,086
Slot raise drilling	m	1,030	1,030
Ring drilling	m	3,668	2,201
Number of blasts	each	6	6
Average load and haul rate	t/day	2,543	2,030
Volume to backfill	m^3	26,935	18,039
Fill factor	%	97%	97%
Total fill required	m^3	26,127	17,498
Plug volume - 10 m height	m^3	4,500	3,000
Pouring rate	m^3/day	4,800	4,800
Curing time	days	14	14
Operational contingency for lost time	%	25%	25%

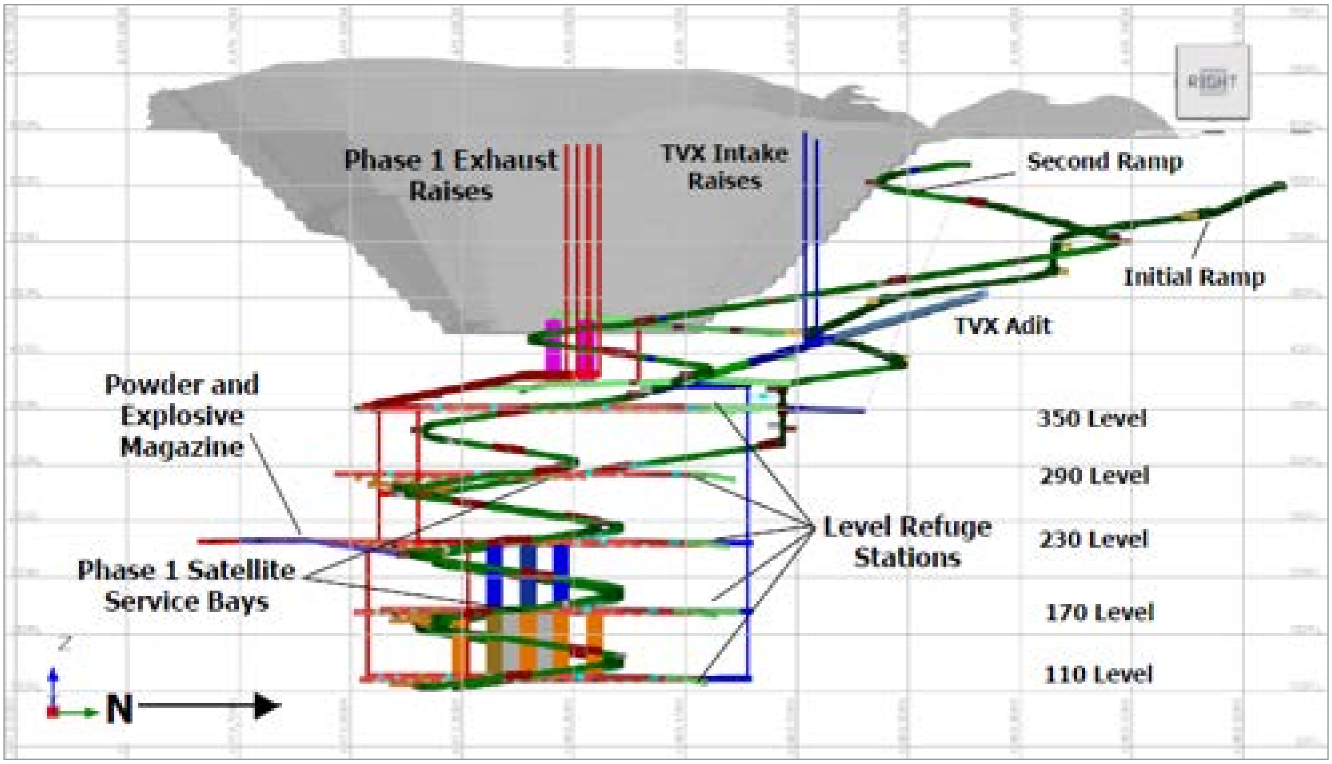
Parameter	Unit	Primary / secondary stopes (30 m Length)	Primary stopes (20 m Length)
Cycle times with contingency			
Longhole drilling	days	17.5	14.3
Stope blasting	days	5.4	5.4
Stope production mucking	days	34.6	23.2
Ancillary activities	days	3.8	3.8
Backfilling	days	29.4	27.5
Total cycle time	**days**	90.6	74.3
Total production rate - excluding fill	t/day	1,223	1,095
Total production rate - with fill	**t/day**	776	634

16.3.8 Underground materials handling

16.3.8.1 Phase 1 materials handling

The material handling strategy for Phase 1 is based on truck haulage of run-of-mine (ROM) ore directly to surface from the level loading bays via the dual ramp system as shown in Figure 16.13. The blasted production ore from the stopes will be loaded into haul trucks by LHDs at remuck / loading points off the production level U-drives. Ore is then hauled by trucks to the surface and will be crushed on surface by the same crusher that processes the open pit ore. The maximum rate at which material can be moved to surface is constrained by the number of haulage trucks that can use the ramps taking into consideration other ramp traffic. This volume of material must also account for development activities generated from the Phase 2 work, which must begin in Year 4.

Figure 16.13 View looking west of general access and development infrastructure for Phase 1



Source: MP 2022.

16.3.8.2 Phase 2 materials handling

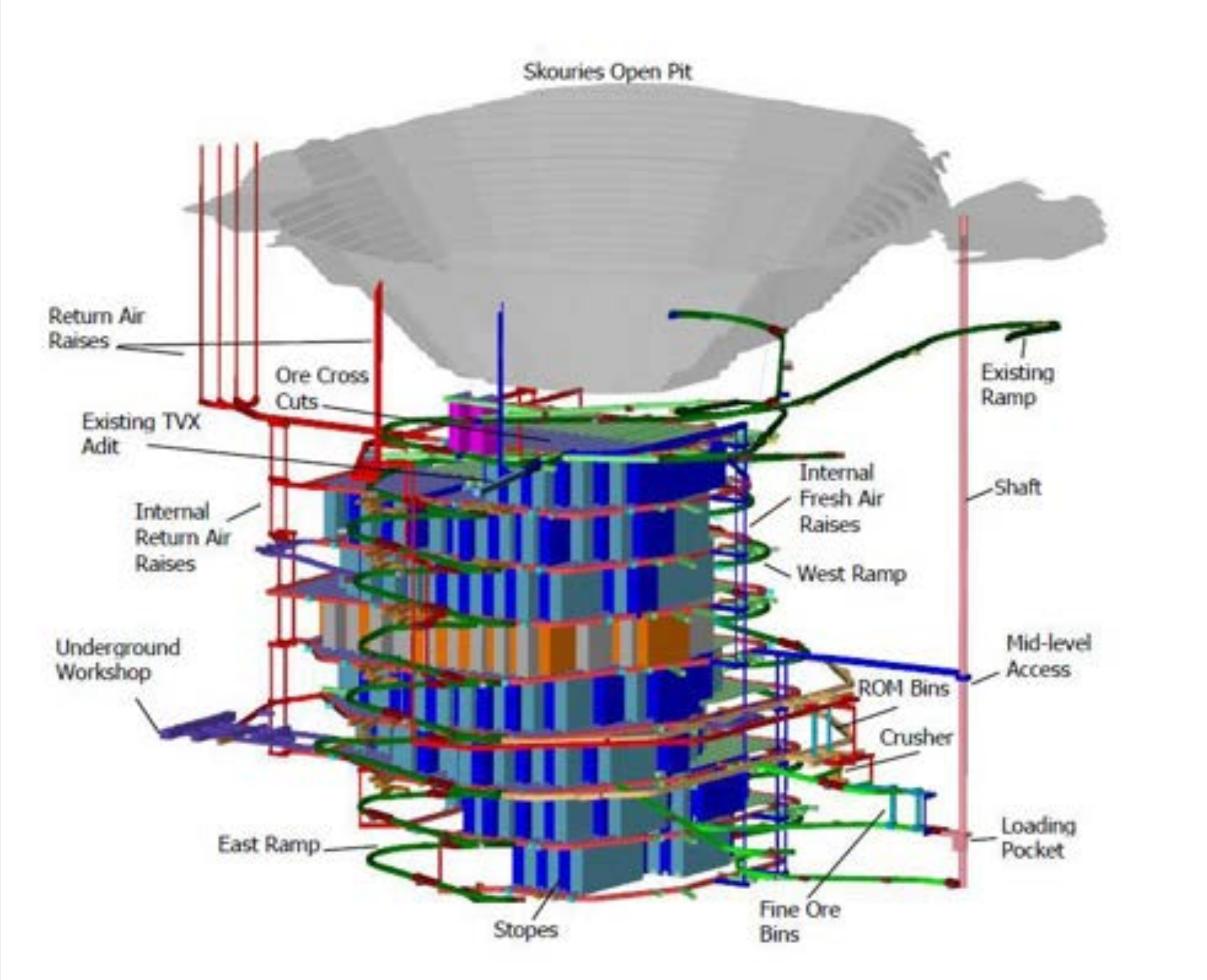
The Phase 2 materials handling will involve shaft hoisting of ore to surface. Shaft hoisting is critical to enable a ramp up to the maximum production of 6.5 Mtpa from the Phase 1 production of 2.5 Mtpa. During Phase 2, all stope ore and some late development ore will be hoisted to surface via the shaft. Development waste will continue to be hauled to surface via the dual ramp system, but these quantities are expected to be minimal. There are no vertical production nor development ore or waste passes included in the mine design; all broken rock will be loaded using LHDs and transported via the ramps in haul trucks. Vertical coarse and fine ore bins form part of the crushing arrangements and crushed ore is transferred horizontally using apron feeders and conveyors.

To allow hoisting of ore to surface, ROM ore will be crushed underground. The materials handling infrastructure will therefore include ROM and fine ore bins and underground crushing. The Phase 2 access and infrastructure arrangement of the materials handling system is shown on Figure 16.14.

The blasted production ore from the stopes will be loaded into haul trucks by LHDs at remuck / loading points off the production level U-drives. Ore is then hauled by trucks to one of two ROM bin dump pockets accessed from the 65 Level or directly to the crusher dump pocket on the 5 Level. The coarse ore bin dump pockets are each equipped with a grizzly and rock breaker, and the crusher tipping points are equipped with a stationary rock breaker. Generally, ore from 50 to 350 Level will report to the main dumps (for transfer to the crusher) and ore from -10 to -130 Level plus some late development ore will report directly to the crusher tipping points.

The gyratory crusher is fed from the central crusher apron feeder from the coarse ore bins or by direct tipping by haul trucks. Prior to the commencement of Phase 2 production, a shaft will be sunk from surface to -117 Level. The shaft will be excavated to 8.2 m diameter and will have a final finished diameter of 7.6 m. A loading pocket will be established at approximately -61 elevation and the shaft will be fitted with counterbalanced 37 t skips on cable rope guides and powered by a friction hoist.

Figure 16.14 View looking south of general access and development infrastructure for Phase 2



Source: MP 2022.

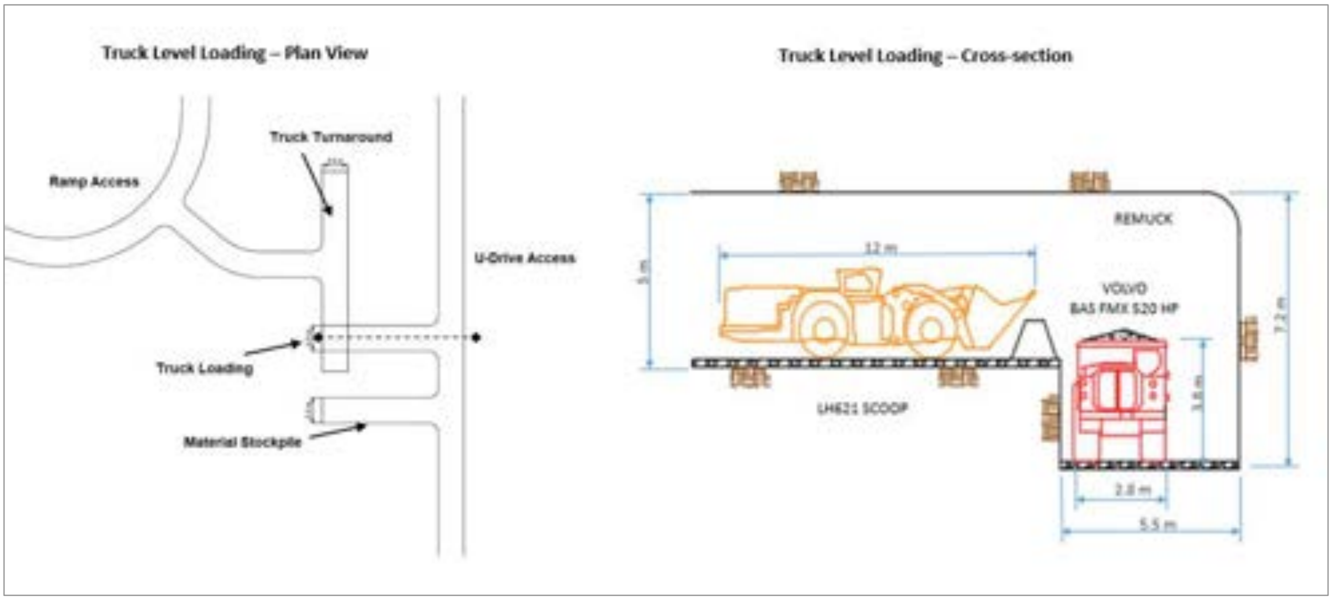
The production hoist will be a multi-rope friction hoist and rated for 6.5 Mtpa hoisting capacity and, under normal conditions 6.0 Mtpa is expected to be hoisted via the shaft. The hoist will be operated in fully automated mode apart from during maintenance or should emergencies occur. Secant piles will be used for the headframe foundations. A foundation pile cap is required to support the load from the headframe and bin house onto the secant piles.

The shaft headframe will be a concrete tower superstructure. The production skips will dump into the bin house. Ore will transfer from the bin house onto the surface stockpile feed conveyor, which feeds the existing fine ore stockpile.

16.3.8.3 Remuck and loading bays

Ore removed from the stopes will be hauled by 21 t loaders to either a truck loading bay or to a remuck bay with a storage capacity of 550 t. The average tramming distance to the loading point is 200 m. There will be one remuck and loading point on both the east and west sides of the production level in Phase 1 of the mine. The design of the loader tipping point is such that the bucket will only be raised high enough to clear the bumper block as shown in a representative view in Figure 16.15.

Figure 16.15 Representative plan and section of level re-muck and truck loading



Source: MP 2022.

16.3.9 Underground access

Primary access to the underground workings will be through a dual ramp system from the surface. The initial west ramp has been developed approximately 1,300 m at a -12.6% gradient to an elevation of 385 masl. The ramp will continue at a nominal -15% gradient and, at a point just above the 350 Level, a drift will be driven from the ramp to breakthrough into the existing TVX adit, which was developed in the late 1990s as an exploration drive. The primary ramp will continue in an elongated spiral configuration tracking the west side of the orebody until it reaches just below 110 masl.

The TVX adit will be rehabilitated from its portal and then slashed to a final dimension of 7 m by 7 m. The slashing will take place from a point where a pair of 3.5 m diameter FARs will eventually be located to the breakthrough of the drift from the primary ramp. From this point, a secondary ramp tracking the east side of the orebody will be driven at a nominal -15% gradient until it reaches just below 110 masl. Initially, the secondary egress route will be provided by this east ramp and the TVX adit to surface. Prior to the commencement of full production from the Phase 1 underground, this secondary east ramp will be extended to a permanent surface portal and will become the new secondary egress route. As each ramp is developed, regularly spaced passing bays, remuck bays, and dewatering sumps and electrical substations will be excavated.

Level access drifts will be established from both the ramps on 60 m vertical intervals as defined by the planned stope height. The final elevation of the ramps in Phase 1 is 110 masl; and in Phase 2, both ramps will be driven down to -130 masl.

16.3.10 Underground mine infrastructure

Mine infrastructure including ancillary facilities and services have been designed to support both the Phase 1 and Phase 2 underground mine production. During Phase 1, the ancillary facilities can be economically located on surface. During Phase 2, the ancillary facilities are developed underground to suit the mining and material handling methods. Underground services including egress, dewatering, ventilation, compressed air, power, communications and control are initially developed during Phase 1 and expanded on during Phase 2 as the mine level development continues.

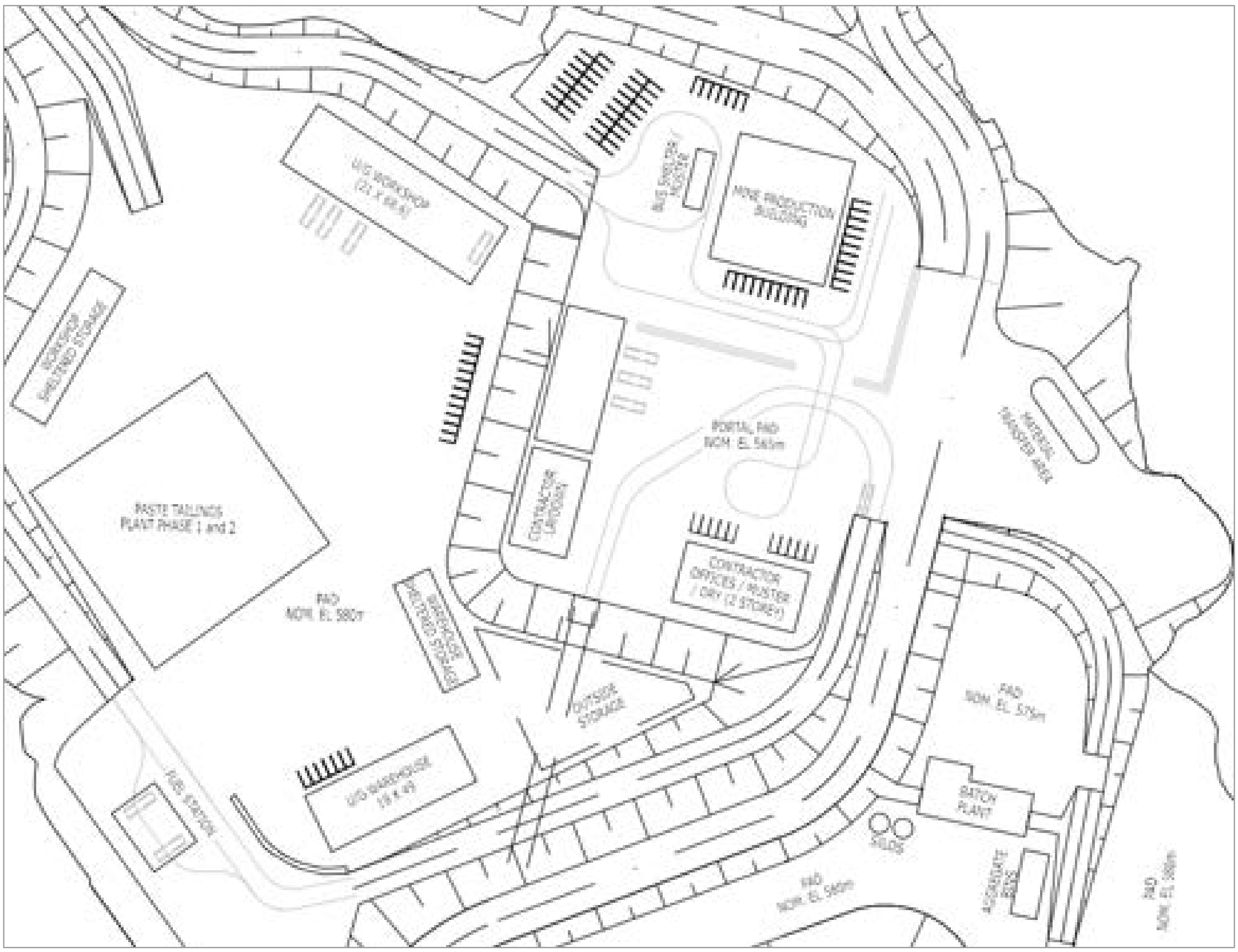
16.3.11 Phase 1 infrastructure

16.3.11.1 Ancillary surface facilities

The majority of the ancillary facilities for Phase 1 are located on surface. The location of the facilities is shown in Figure 16.16. Surface ancillary facilities include the production services building, the surface workshop and warehouse, the surface fuel and concrete / shotcrete batch plant systems, and power supply.

Underground ancillary facilities in Phase 1 are limited to satellite service bays, level portable refuge stations and explosives storage.

Figure 16.16 Mine surface facilities



Note: Schematic not to scale.
Source: Eldorado 2022.

16.3.11.2 Mine services

Mine services for Phase 1 include compressed air, ventilation controls, dewatering (contact and non-contact), process water and electrical power controls and communications.

Compressed air

In Phase 1, two permanent compressors will be located at the return end of each production level; one to supply production drills in the two west quadrants and the other to supply production drills in the two east quadrants of the U-Drive. There will be a small local shop compressor installed in each of the Phase 1 Satellite Work Bays on 290L and 170L. There is an onboard compressor for the Paralos Water Filtration System for clearing the filter cake from the filter drum cloth.

Ventilation controls

Phase 1 permanent ventilation controls consist of fire doors at flammable storage sites (230L Explosive Magazines and 290L and 170L Satellite Work Bays) and regulator doors to control ventilation flow from the northern intake side to the southern return side of the levels. There is a double conventional door airlock in the 370L West-East connector drive, roll up doors where vehicle access is required to the exhaust or fresh air drifts and louvre regulators to control return air flow at the southern ends of the U-Drives. The main drivers of the ventilation system are surface exhaust fans pulling air through the mine, whereas auxiliary and level intake fans are used to direct the fresh air to the working areas.

16.3.11.3 Dewatering (non-contact and contact) and process water

The mine dewatering system allows contact water and non-contact water to be pumped out of the mine via separate systems. The Skouries mine is in an area of relatively high rainfall, which leads to high inflows of ground water during the Phase 1 of mining. During Phase 2, inflow drops significantly due to open pit dewatering and low groundwater at depth. Estimated inflows were based on hydro-geological modelling completed in 2017 and updated in 2020.

Non-contact dewatering

All sources of non-contact water come from either advanced drilling boreholes or ramp and level perimeter boreholes. Non-contact water has not been exposed to any contaminants in the mine and is kept strictly separate from contact water. It is assumed that 70% of the ground water inflow will be captured by boreholes as non-contact water and the remaining 30% will report to the contact water sumps. Phase 1 non-contact water will be collected at advanced sumps 10 m above each level and from dewatering boreholes in the ramps and on the levels to be transferred via pipes and drainage boreholes to level non-contact sumps on 110L and 230L. These sumps pump up to the main non-contact water sump on 350L. Some non-contact water will be drawn from this reservoir for use in mine operations; the remaining non-contact water will be pumped to surface via twin 200 mm pipes in the West Decline to a surface receiving pond.

Process water

Initial process water will be piped into the mine from the West Portal. Once the non-contact dewatering main sump on 350L and the first advanced non-contact sump at 360L and initial dewatering boreholes have been connected to 350L, process water for mine operations will be drawn mostly from underground. The surface freshwater line will remain to supply make-up water as required.

Contact dewatering

Contact water comprises groundwater that has drained into the mine workings (not captured by boreholes) and process water that has been used by mine operations (such as for drilling or dust suppression) and may contain contaminants. This contact water is kept strictly separate from noncontact water. Phase 1 contact water is collected in open ditches and sumps in West and East declines, the TVX Adit and Levels between 350L and 110L. Levels above 170L drain to the 170L contact U-Drive sump, decline sumps pump to the nearest Level U-Drive sump and the 170L Satellite Work Bay sump transfers to the 170L U-Drive sump. The 110L U-Drive sump pumps up to the 170L U-Drive sump. There is a Paralos water Filtration Plant located on 170L that uses a drum filter and microfabric to remove particles down to 20 µm from the contact water. This filtered "clean" water is pumped via boreholes up to the main 350L contact water sump and from there is pumped via twin 200mm pipes to the surface contact water receiving pond. From here, the contact water is pumped to the mine water treatment pond. Sludge from the filters will be loaded into a mine truck box and disposed of on surface.

16.3.11.4 Power controls and communication

Electrical power distribution

For the permanent power supply, the entire underground mining electrical power will be fed from the 20 kV sub-station located at the West Portal. The power is then delivered to the 10 megawatts (MW) underground mine main sub-station also located near the West Portal. There is a second 10 MW sub-station as a Phase 1 back-up and to power up Phase 2. The underground medium voltage distribution will be 20 kV N.R.G. The underground new installation low voltage will be 690 N.R.G. (existing pumps will remain on the 400 V T.N.S. system during development, before eventually being decommissioned.) The West Portal 10 MW sub-station also feeds the East Portal sub-station.

There will be four (4) permanent mine electrical feeder lines:

- West
- Central
- East
- Shaft

The West Feeder supplies 20 kV power to the 370 W Electrical Sub-Station and the 370 W West-East Switch. This switch connects Phase 1 West and East Declines electrically. Normally, the switch remains in the open position, but if there is an upper mine failure the East may be energized from the West and vice versa. The West Feeder also feeds power to the West Side of each production level and the West Satellite Work Bays on 290L and 170L. The existing 400 V equipment in the West Decline is also supplied from the West Feeder.

The Central Feeder supplies 20 kV power to the main dewatering sumps and Level intake fans from 350L down to M130L. This line is independent of West and East Feeders during Phase 1 and is connected to the Phase 2 Shaft Feeder at 110L. If the Central Line fails during Phase 2, the dewatering pumps and level intake fans may be supplied from the shaft feeder.

The East Feeder supplies 20 kV power to the East Decline from the East Portal to 370L West-East Switch and down the East Decline to M130L. The East Feeder also supplies the East Side of each Production Level from 350L to M130L. Phase 2 services below 110L are supplied via the East Feeder.

Surface underground mine services also receive power from the East Feeder, including:

- Underground Mine Production Building
- Surface Fuel Storage Station
- Phase 1 Fresh Air Raise No1 Collar (Stench Injection, Raise Inspection Equipment & Lights)
- Phase 1 Fresh Air Raise No2 Collar (Raise Inspection Equipment & Lights)
- Phase 1 Return Air Raise Fans
- Phase 2 Return Air Raise Fans

The (Phase 2) Shaft Feeder is supplied independently from the Surface ROM Pad Sub-Station near the surface crusher. This line supplies 20 kV power to the production friction hoist, the auxiliary drum hoist and the headframe area on surface, and the shaft infrastructure underground to M117L Shaft Bottom. The Shaft Feeder has a connection to the Central Feeder at 110L Shaft Station, which under normal circumstances will remain open, but power may be supplied to the Central Line Dewatering Pumps and Intake Fans in an emergency from the shaft. Also, if the shaft power line fails, in an emergency the Central Line could supply limited power to the hoists via the shaft line.

Communications network

When the mine infrastructure on 350 Level has been commissioned, a fiber optic communications backbone will be established between surface and 350 Level. This will enable high volume data communication between surface and underground to allow fixed equipment control. The fixed plant automation design concept provides for a central control base on surface with modular expansion as the mine grows and develops, with a series of PLC control systems deployed based on process type and geographical zone for ventilation and dewatering infrastructure.

In parallel with the installation of the fiber optic hard wired network, a composite powered fiber optic wireless network will be installed throughout the mine to enable Wi-Fi wireless tracking and tagging of personnel and vehicles via wireless nodes; VoIP wireless voice and text communications with personnel and vehicles, and video cameras, gas and vibration monitoring will also be enabled via the fiber optic wireless network.

Mobile equipment automation during Phase 1 will be limited to tele remote control of production LHDs and production drills. The fiber optic hard wired and wireless backbone developed during Phase 1 will allow implementation of automation as the technology becomes available.

16.3.12 Phase 2 infrastructure

16.3.12.1 Ancillary facilities

In Phase 2, trucks will no longer be hauling ore to surface regularly; an underground workshop will be located on 50 Level south of the east ramp to service the mobile fleet underground. A borehole will be installed to deliver fuel directly to an underground fuel bay on 65L, near the main ROM dumps. This location for the fuel bay allows for convenient fuelling of mobile equipment adjacent to a main travelway. It also allows the fuel bay to be isolated by means of fire doors and for any smoke or fumes to be exhausted directly to the return airway in the event of a fire.

The majority of Phase 2 ancillary facilities are located underground; the location of the facilities is shown in Figure 16.14. Underground facilities include the underground workshop, a permanent refuge station / lunchroom, and materials handling. Services development will continue with egress, dewatering, ventilation compressed air, power, network communications, and control being developed by level.

Additional personnel will be required to achieve the increased production rate in Phase 2. Facilities for the additional manpower, including mine change house and administration, will be provided by utilizing the open pit production services buildings that will be available for underground mine use on completion of the open pit mining at the end of Phase 1. The open pit mine change house and administration facilities are sized to ensure there is capacity for the additional underground mine production personnel and support staff.

The existing surface primary crusher will be decommissioned, and Phase 2 crushing will take place underground. Surface facilities that are added include the head frame and new fine ore stockpile feed conveyor.

16.3.12.2 Emergency preparedness

Consideration has been given to the possibility of mine emergencies. As such, the following criteria have been established:

- In general, ramps will be in fresh air once developed.
- In each ramp, escape may be either up the ramp or down the ramp to a safe area.
- A permanent refuge station is planned for the 50 Level shop.
- Other portable refuge chambers are required for flexibility of location at the most appropriate points in the mine. Specifically, they are recommended where walking distance to a safe area exceeds 750 m.
- The shaft auxiliary hoist is fitted with a cage (capacity 14 persons) and an emergency back-up generator to ensure a means of egress via the fresh air production shaft during Phase 2.

A permanent refuge station, also serving as a lunchroom, will be established in the workshop area. This will provide refuge during an emergency for 40 persons. The remaining personnel working underground, namely the production, development, and service crews, are provided refuge by means of mobile self-sufficient rescue chambers on the levels. These refuge chambers will be independent of the external reticulated compressed air supply.

Prior to the development of the east ramp, the TVX adit will serve as the second egress from the mine. The route of travel for secondary egress is to access 350 Level via the west ramp and then follow the U-drive to the TVX adit. Personnel will drive or walk out through the adit to surface. When the East Decline from the East Portal breaks through at 370L the East Portal will provide the secondary means of egress via the East Decline.

A stench gas warning system will have release points at the portal and the FAR collar. When activated, this system will release stench gas into the main fresh-air system allowing the stench gas to permeate rapidly throughout the mine workings. Stench gas may be released in the event of mine emergencies, including fire, serious accident, or injury.

Wireless communications will be provided throughout all permanent declines and drives and to initial ore drive intersection development.

The primary purposes of fire doors is to prevent noxious gases from reaching workers should they be trapped underground and to prevent fire from spreading. Fire doors are required to isolate the 290L and 170L satellite work bays, 65L fuel bay, 50L lubrication bay and 50L workshop areas.

16.3.13 Ventilation

The function of the ventilation system is to dilute / remove airborne dust, diesel emissions, explosive gases, and to maintain oxygen and temperatures at levels necessary to ensure safe production throughout the life of the mine. The design is based on an exhausting “pull” configuration, with permanent exhaust surface fans located at the collar of the RARs. Fresh air is drawn into the mine through both the west and east ramp portals as well as the FARs, and the shaft in Phase 2. Distribution of air on the levels is accomplished by secondary distribution fans at the internal FARs and regulators on the internal return air raises (IRARs). Auxiliary fans with ducting deliver air to workplaces not ventilated with flow-through ventilation.

16.3.13.1 Design criteria and basis for design

The criteria and basis for design of the ventilation system for the Skouries Project have been defined based upon the following:

- "Mining and Quarrying Operations Regulations", 23 May 2011, Ministry of the Environment, Energy, and Climate Change, Hellenic Republic (the Regulations).
- Site criteria, such as projected personnel and equipment requirements.
- Global best practice.

16.3.13.2 Ventsim modelling criteria

The ventilation network has been modelled in industry standard "Ventsim™" software. Various factors are input into the model. The ventilation model serves three primary purposes:

- To validate the operability of the ventilation circuit ensuring airflow can be provided to all the required areas and during all phases of the mine.
- To ensure compliance with design criteria.
- To determine fan duties and energy requirements to ensure the deepest levels of the mine are adequately ventilated.

The Ventsim™ model was designed for the 2.5 Mtpa production rate in Phase 1 and the 6.5 Mtpa production rate in Phase 2. The key aspects considered in the ventilation modelling are as follows:

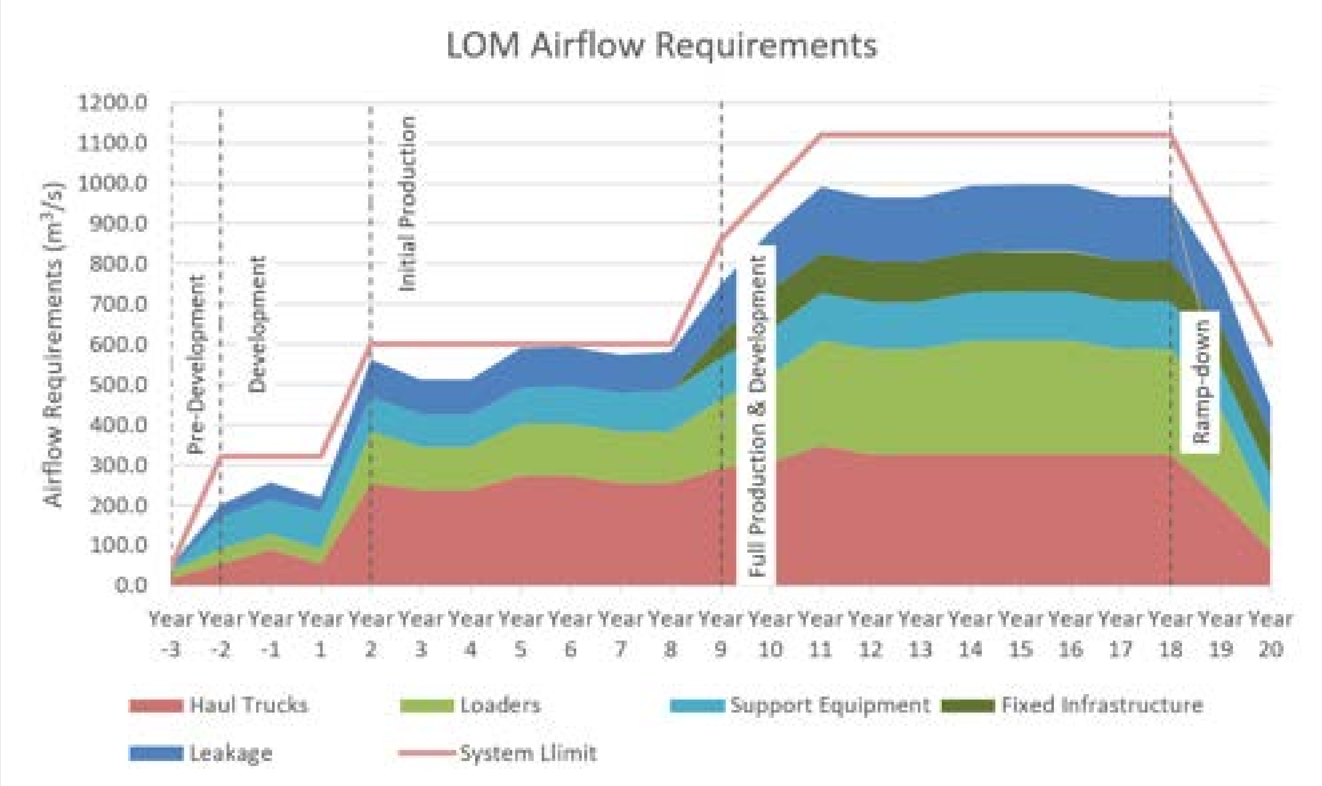
- As per geotechnical guidance, all raise-bored airways are restricted to 3.5 m diameter.
- The exhaust air from the workshop area travels via transfer drift to the 50 Level IRAR. The powder magazine storage area is ventilated on 230 Level via IRARs. The fuel bay on 65 Level is ventilated utilizing the dedicated return air transfer drift for crusher exhaust.
- Once the circuit was established in Ventsim™, balancing of the airflows was done using simulation regulators to achieve the estimated airflows in each working area of the mine.

16.3.13.3 Airflow requirements

Airflow requirements were determined based on the diesel engine exhaust (DEE) dilution provided at point of use for the number of required mining areas. An airflow allowance was also determined for underground infrastructure and balancing inefficiencies. Total airflow requirements were determined based on the anticipated concurrent activities and working places during steady-state production and development.

Peak airflow requirements reach 1,000 m^3/s in Year 16 of the project schedule. The modular nature of the RARs with individual fans means that the system can be readily adapted to the required flow rates by turning the fans on or off as required. Each fan is designed to provide approximately 130 m^3/s to 150 m^3/s depending on the system resistance at that stage. Figure 16.17 shows airflow demand over the life of project. Primary airflow is on the critical path for the underground mine as the ventilation system from a positive "push" system from the TVX cannot provide air to the deeper reaches of ramp development.

Figure 16.17 Project ventilation demand



Source: MP 2022.

16.3.13.4 Ventilation strategy

The proposed LOM layout for the Skouries underground project will support a negative pressure ventilation circuit. Primary fans located at the collars of the eight RARs will exhaust air from the mine. Intake air is supplied by the shaft, two declines, and two FARs. Airflow on the production levels is controlled primarily by regulators that control the quantity of air leaving to the RARs. Booster fans on the levels control airflow between the levels and the declines.

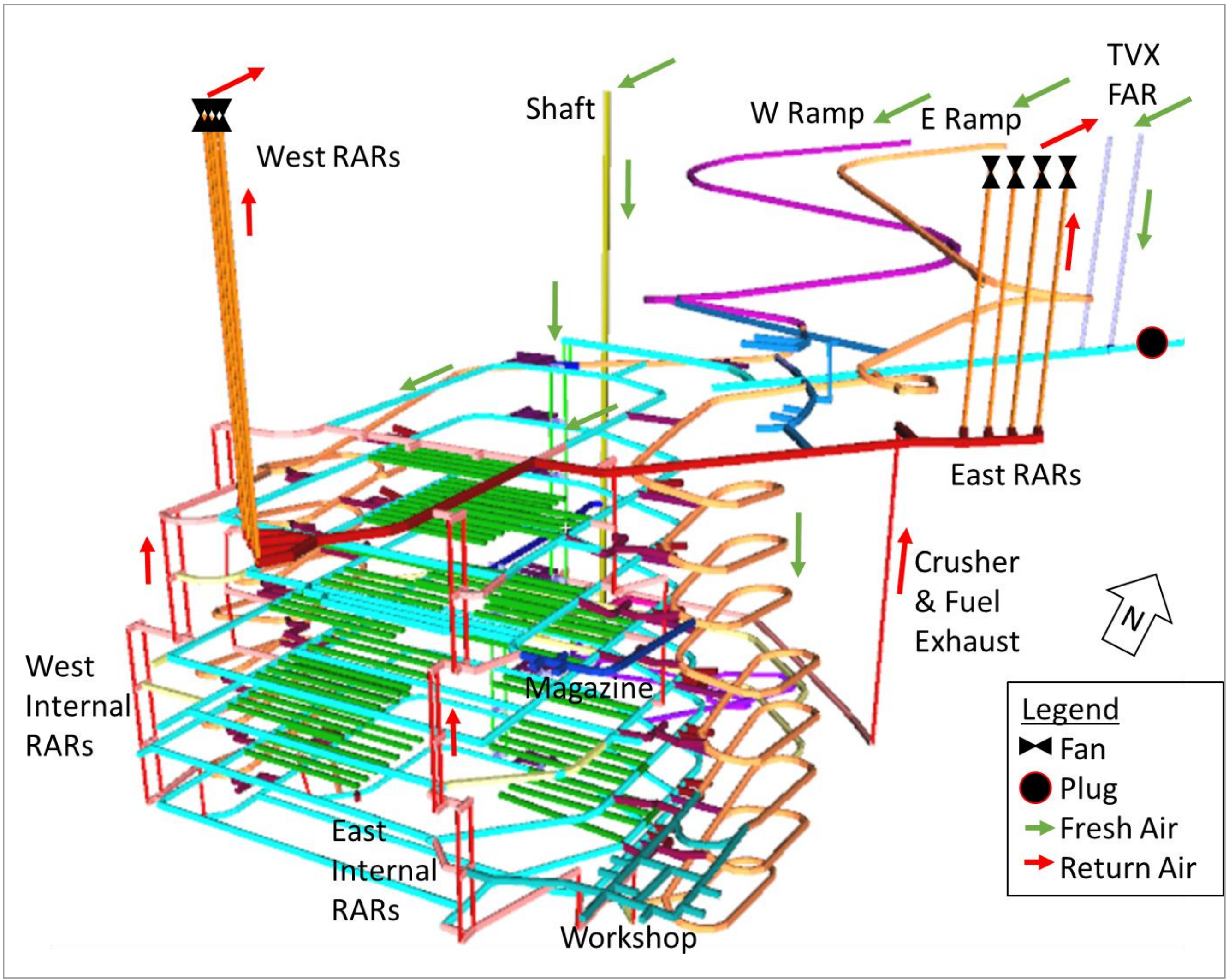
Figure 16.18 shows an isometric view of the LOM ventilation circuit, which functions as follows:

- The surface exhaust fans impart a negative pressure to the mine ventilation system.
- Air is drawn up both IRARs and into an air transfer drift on 350 Level.
- Regulators are located on each level in the accesses to the IRAR.
- Return air from crusher, workshop, and magazine storage area exhausts to 350 masl before discharging to surface via main RARs.

Table 16.21 includes the intake or exhaust airflow values that correspond to the LOM ventilation circuit at steady state.

Figure 16.19 shows a plan view of the typical level ventilation strategy. The main exhaust fans will force the air through the mine and regulators installed at the IRARs will determine single-pass ventilation quantities on each level. The regulators are located outside of the U-Drive and Automation Drift to allow for unimpeded travel along the level. Similarly, the booster fans located in the internal FARs replace the need for flaps on the level access. The booster fans control airflow on the levels but also allow for the declines to remain ventilated with fresh air. Auxiliary fans will ventilate the workface, using the U-Drive as a fresh air source.

Figure 16.18 Primary airflow schematic

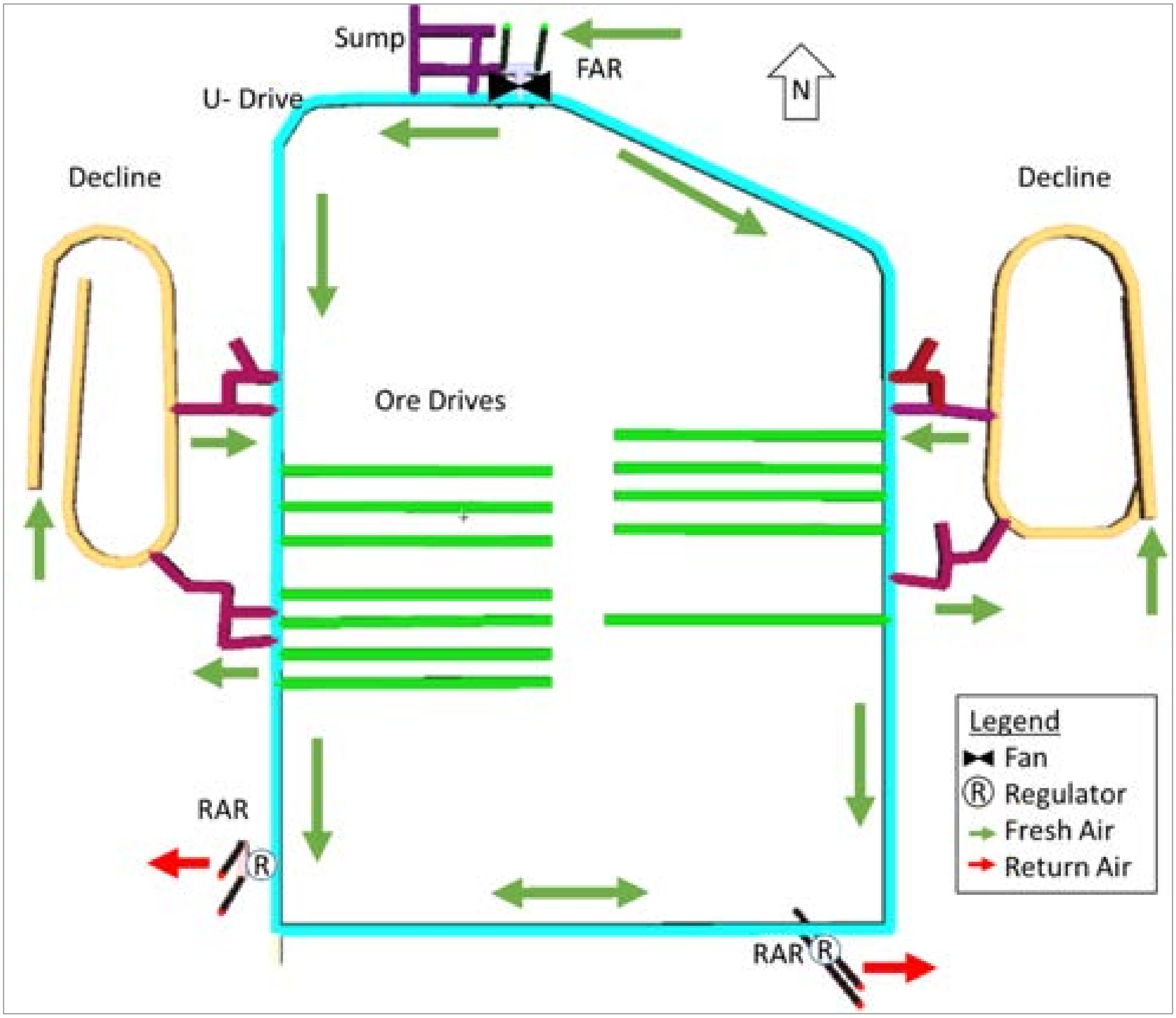


Source: MP 2022.

Table 16.21 LOM airflow values

Location	Air type	Air quantity (m^3/s)
West Ramp	Intake	265
East Ramp	Intake	188
TVX FAR	Intake	269
West RAR	Return	549
East RAR	Return	548
Shaft	Intake	338

Figure 16.19 Typical level ventilation system



Source: MP 2022.

During normal conditions in Phase 1, fresh air will be delivered down the west and east ramps, and through the TVX drive and internal FARs. This will be achieved by operation of the mine-air surface exhaust fans to draw a steady volume of air in through the portals in conjunction with secondary distribution fans located at each FAR access.

Once the shaft is established for Phase 2, it will form part of the intake airway system in addition to the fresh air supplied through the TVX raises and ramps from surface. An air transfer drift from the shaft at 110 Level will provide fresh air from surface to a second network of internal FAR connecting from 110 Level to the levels below.

An exhaust fan at the bottom of the crusher chamber will ensure that dust-infused air at the tipple will flow down the crusher and be exhausted through a return air drift at 65 masl via an internal exhaust raise. Regulators are in place on the crusher dump level on either side to balance the airflow on the haulage drives on that level. A RAR from 65 masl to 350 masl is in place to ensure that exhaust air from the crusher and fuel bay is exhausted to surface via a set of Phase 2 RARs.

The workshop on 50 Level and magazine storage area on 230 Level will be supplied with fresh air from the east ramp. The exhaust air from these locations will return to 350 Level via a set of IRARs on the east side of the deposit.

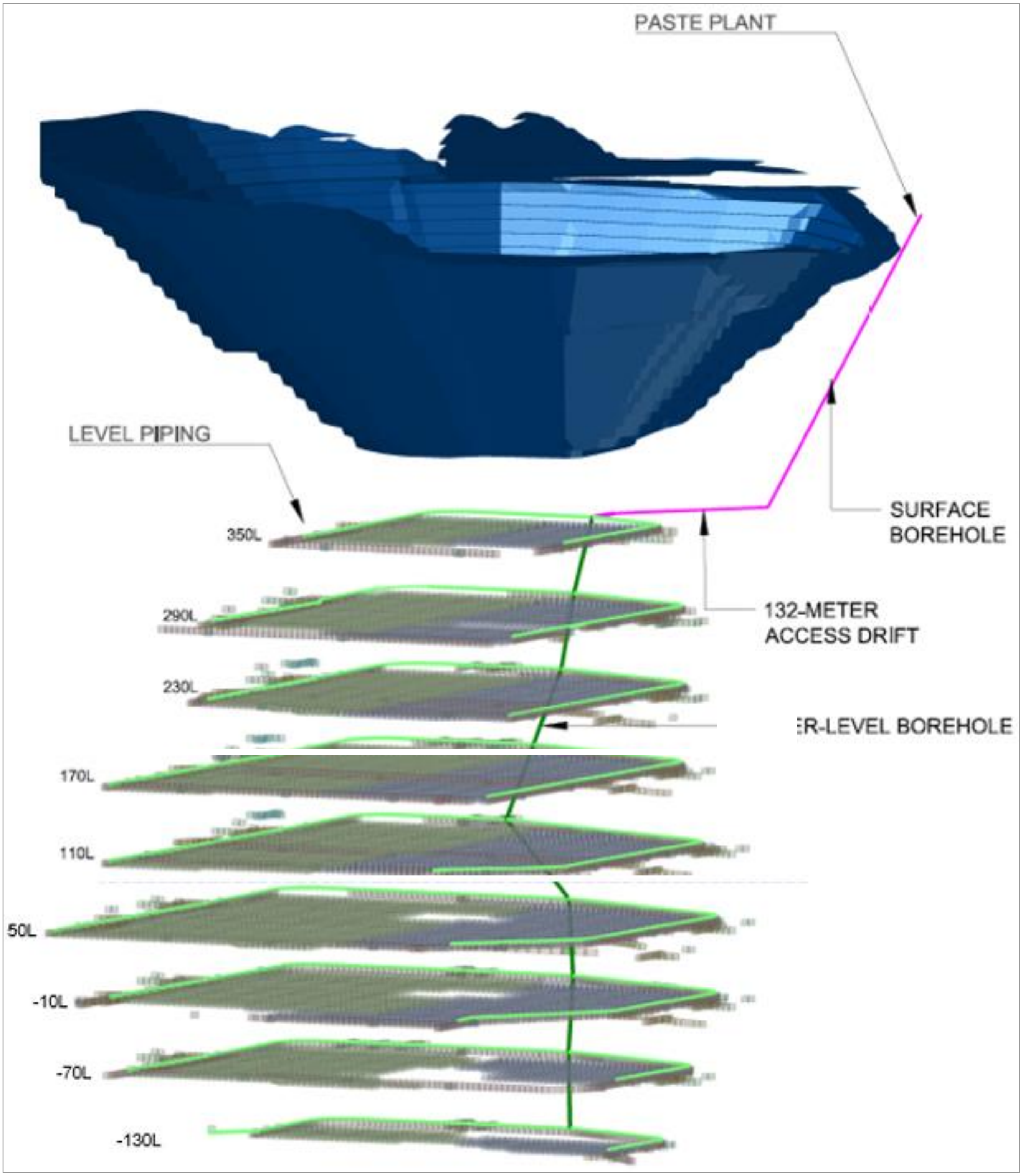
16.3.14 Backfill

A paste backfill plant and distribution system have been designed for Skouries underground Phase 1 and Phase 2. The Skouries paste fill system will combine moist de-slimed vacuum-filtered tailings cake and pre-wetted cement slurry to produce 200 m^3/h (Phase 1) and 400 m^3/h (Phase 2) of paste fill at an average 70.5% solids by weight in a plant located on the eastern rim of the open pit. The paste will be delivered by gravity and optional pump assistance via boreholes and pipelines to the stopes. Apart from the two trial stopes mined before the paste plant commissioning, all stopes will be filled with paste fill. One plant will service Phase 1 and a second, identical paste plant will be built for Phase 2, with each plant able to operate independently.

Typical stopes are 15 m wide and 30 m long and mined on a 60 m sub-level interval and will require 27,000 m^3 of paste fill. Arched shotcrete barricades will be built in the drawpoint to each stope and the paste fill will be placed in a near continuous single pour requiring 135 hours of active filling time to complete. Short interruptions to filling (up to one hour) can occur, but longer delays will require a flush out of the underground reticulation system. The paste fill system will have an overall utilization of 51% for Phase 1 and 67% for Phase 2. The paste plant will shut down for periods from one to ten days, depending on availability of the next stope to fill. Fill preparation activities and planned maintenance will take place during these down times.

Paste fill will be prepared and delivered at the maximum practical density to each stope. This will vary depending on the location and elevation of the stope being filled and will be monitored via pressure signals from the surface pumps and the underground reticulation system. The operators will control the density by adjusting trim water in the paste mixer to achieve stable delivery pressures and by monitoring for any risk of impending blockage. The paste fill reticulation system for Phase 1 is shown in Figure 16.20. In Phase 2, two additional surface boreholes will be drilled, and the internal boreholes extended down to -70 Level. Trunk delivery pipelines on each of the sub-levels will be extended to stoping crosscuts to enable filling of Phase 2 stopes.

Figure 16.20 Phase 1 and Phase 2 isometric schematic of paste reticulation



Source: MineFill 2022.

The stopes will be mined in primary and secondary sequence and most stopes will have more than one fill exposure (sequential, not simultaneous). The fill strengths have been designed to take into account the geometry of the cured fill exposures and vertical strength zoning has been applied in the fill to optimize binder consumption.

Most primary stopes will be exposed during mining on three faces (exposures occur in sequence, never simultaneously). Initially, at a minimum of 14 days after the last paste fill is placed in the stope, the cured fill will be exposed in the narrow, 15 m wide by 60 m high face. Later, as the secondary pillars are extracted on either side, the wider, cured fill faces 30 m long by 60 m high will be exposed. Secondary stopes will be extracted on retreat between the filled primary stopes. Only the narrow faces will be exposed, and lower fill strengths will be required.

The high-strength 10 m thick undercut sills apply to all stopes mined off 110 Level, which will have stopes extracted below in Phase 2.

All primary and secondary stopes, other than the final stope extracted in any given row, will have one narrow fill face exposed over 15 m wide and 60 m high at a minimum of 14 days curing. The binder requirements have been optimized for primary and secondary stopes to ensure that the cured strength achieved matches the larger of the 14-day narrow exposure or subsequent greater-than-28 day exposure for other stopes. The summary of unconfined compressive strength (UCS) and binder recipes is provided in Table 16.22.

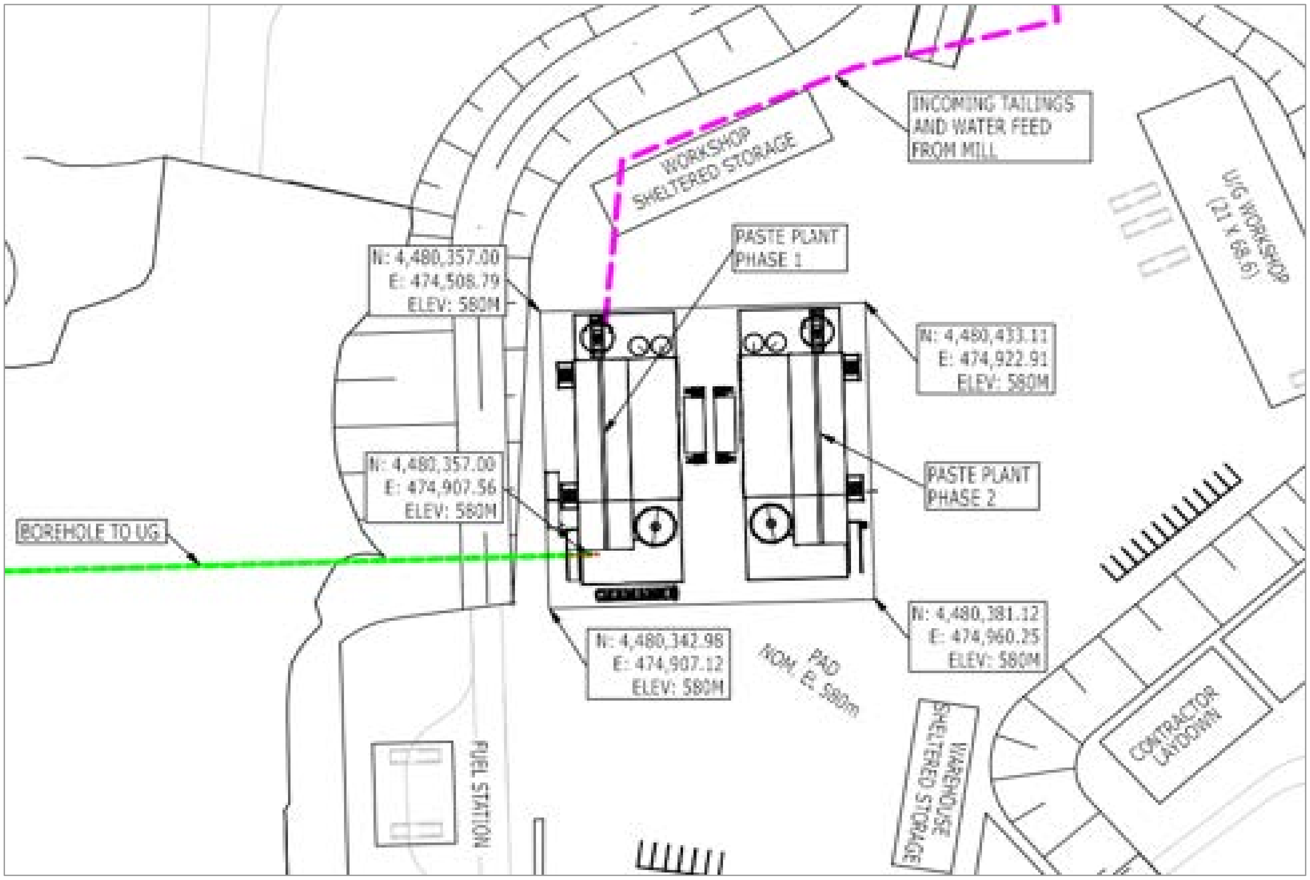
Table 16.22 Summary of average paste fill binder recipes

Stope type	Sill / no sill	Average 14D UCS (kPa)	Average 28D UCS (kPa)	Average binder (%)
Primary	No sill	275	525	5.6%
Secondary	No sill	275	275	5.3%
Primary	Sill	425	600	6.1%
Secondary	Sill	425	425	6.1%
Weighted LOM average				**5.6%**

16.3.14.1 Paste backfill plant

The paste system consists of two operating areas. The first is located at the Mill and is comprised of the desliming equipment for removal of a portion of the fine particles from the full-stream tailings. The second is the Paste Plant itself, located on the 580 masl pad as shown in Figure 16.21. A portion of the thickener feed from the Skouries mill will be deslimed using a hydrocyclone cluster to achieve an underflow at 55% solids content by weight. The cyclone underflow is then transported via pipeline to the paste plant where it is received into an agitated storage tank with two hours storage capacity. The overflow from the desliming cyclones reports to the tailings thickener, where it is thickened and dewatered with the remainder of the tailings for dry stacking. Due to the unavailability of tailings to complete testwork, it is noted that the deslime process design is based on previous experience with similar plants and additional physical testing should be completed during future engineering phases. The paste plant further dewaters the cyclone underflow using vacuum disc filters (two in operation and one on standby) after which the vacuum disc filter cake is conveyed by conveyor to the paste mixer. The paste mixer utilizes a shaft mixer to combine dewatered tailings from the filter cake conveyor and the pre-wetted cement from the binder system to bring the paste to the target rheology.

Figure 16.21 Site plan for Phase 1 and Phase 2 paste plants



Note: Schematic not to scale.
Source: MP 2022.

The Phase 1 paste plant is designed to mix deslimed vacuum filtered tailings cake with pre-wetted cement, to produce 200 m^3/h of paste. The plant is a continuous mixing operation with paste delivered underground via gravity with the option of diverting paste to utilize a positive displacement pump where required. In Phase 2, a second plant identical to Phase 1 is envisaged that will be capable of 200 m^3/h, resulting in a total paste plant throughput of 400 m^3/h.

Cement will be delivered to site in bulk and pneumatically unloaded into the 1,000 t binder silo, which is fitted with fan assisted dust collecting and continuous level monitoring. The silo will provide adequate storage to fill the largest high-strength sill stope without having to refill the silo.

The paste rheology (slump) is controlled by metering the tailings filter cake and the pre-wetted cement via a vortex mixer. The total solids content entering the paste mixer will be metered by each respective system’s instrumentation and a calculated solids content and throughput will be shown on the paste plant control panel. The target slump and solids content will be correlated to the required strength for the targeted underground stope and the transport rheology.

Regular samples will be taken and measured for rheology (slump) properties and placed into curing cylinders for strength testing. A fully equipped QC laboratory within the paste plant will include a temperature-controlled humidified curing chamber, compressive testing equipment, mixing equipment and slump measurement equipment. A database of QC results will be maintained relating paste quality to paste runs and stopes filled to enable performance assessment of the paste fill for planning purposes.

16.3.14.2 Paste borehole and pipe design

Phase 1 paste fill will be delivered underground via two boreholes (one operating and one standby) drilled from surface to a dedicated cross cut on 350 Level. The boreholes will be drilled 260 m long at 60° dip at nominal bore (NB) 300 mm and will be cased to NB 200 mm. The duty borehole will have a ceramic lined steel 200NB casing and will be utilized for gravity feed backfilling. The standby borehole will have unlined 200NB Schedule 80 casing and will be utilized for pumped feed backfilling. The holes will be drilled first with HQ diamond drill bits and then reamed out to final diameter of 311 mm (12.25”). In Phase 2, one more duty borehole plus a standby hole are planned to be drilled.

Two 60 m underground boreholes have been designed from a centrally-located stub drive off the perimeter access (north side) to each of the sub-levels; this will be repeated on each sub-level. The holes will be drilled at 200 mm NB. The ground is expected to be competent, thus unlined holes may be used. A short 200NB Schedule 80 steel toe and collar pipe will be grouted into the hole for adapting the unlined boreholes to the pipework. In Phase 2, this borehole system will be extended down to -70 Level.

On each of the paste delivery levels, a Schedule 80 black steel 200NB pipe will be installed at the paste access drift and along the level where the paste pressure exceeds the operating limit of the steel wire reinforced pipe (SRCP). Where the pressure is less than 4.35 MPa, SRCP pipe will be used to deliver paste along the level to the stope crosscuts. At each level, valves will be remotely controlled by the paste plant, which will allow the plant to direct the flow to the required stope and perform regular clean out functions as well as emergency dumps if required.

For each cross cut, DN225 SDR11 high density polyethylene (HDPE) pipes will be installed from the trunk line connector to the top of the stope to be filled. As stopes are filled and the production retreats down the cross cut, the HDPE lines can be removed and reused as required. It has been assumed that 200 m of HDPE pipe will be required per level.

16.3.14.3 Barricades and pouring regime

At the completion of production from a stope, a cavity monitoring survey will be conducted to provide the actual stope dimensions for ore reconciliation and backfill preparation purposes. A suitable location for the draw point fill barricade will be selected and prepared in accordance with the certified barricade design. At the completion of fill preparation, the stope will be available for paste filling to start.

Paste fill will be retained in each stope using a structural arched shotcrete barricade constructed in the drawpoint. The barricade will be located approximately 1.5 times the drift height back from the brow of the stope.

For typical stope sizes at Skouries with a plan area of 450 m^2, and a filling rate of 200 m^3/h, the fill rate of rise will be approximately 0.44 m/h. Depending on the selected mix recipe and bulkhead design, the filling strategy may be one continuous pour, or by way of a plug pour followed by a rest period then a continuous secondary pour to fill the stope.

A single continuous pour is preferred and has the benefit of reduced time per stope fill run, nominally taking place over five to seven days without the need to flush lines. Minor interruptions to the supply of paste may be tolerated without flushing but extended delays will require a flush cycle and restart of the paste fill line. Filling a 60 m high stope will require in the order of 135 hours. Initial barricade pressures will be monitored to verify design assumptions, and adjustments to design, if required, will be made.

16.3.15 Mobile equipment requirements

Table 16.23 lists the owner mobile mining equipment fleet required to develop and sustain 2.5 Mtpa of ore production over the 10 years of Phase 1 activity. The final equipment numbers required to ramp up production to 6.5 Mtpa in Phase 2 are also provided. The peak numbers refer to the maximum number of units in any year for the reported period. Specific model references by manufacturer are listed below by way of examples of suitable equipment to meet envisaged requirements. Other models from alternative manufacturers will also be considered at the time of purchase.

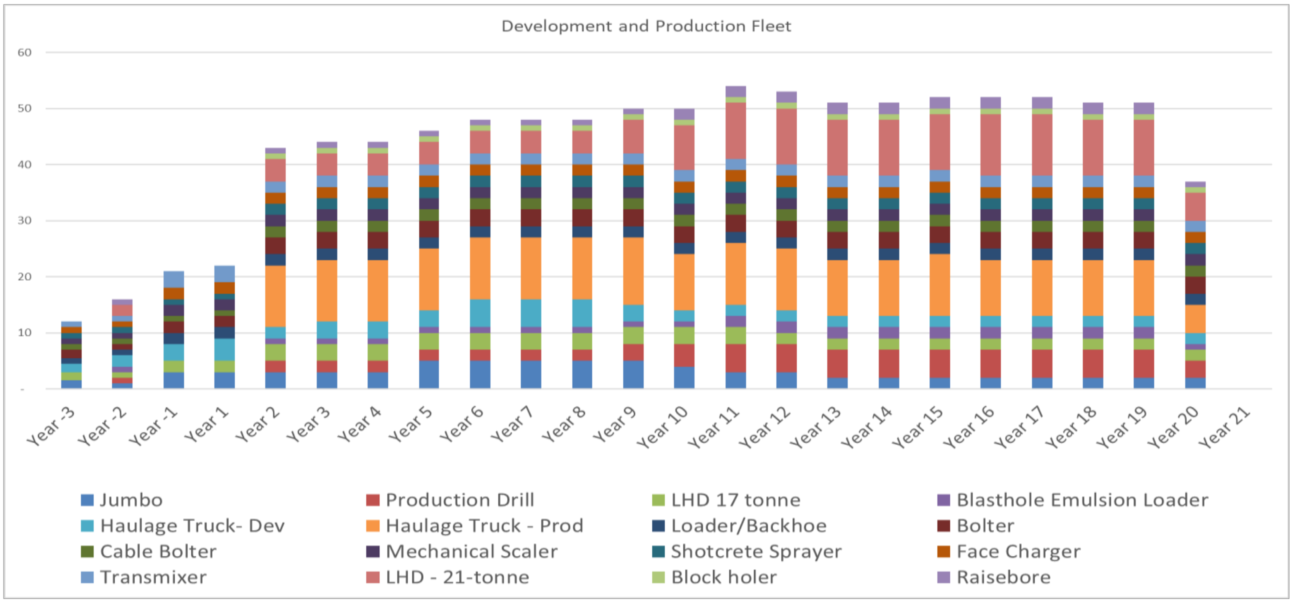
Given that Phase 1 material handling relies on a significant fleet of haulage trucks, particular attention was given to the selection of suitable vehicles for the haulage of development and production material to surface. The evaluation of rigid body dump trucks has shown that they carry a considerable economic advantage over traditional mine trucks when capital, operating, and replacement costs are considered. The BAS FMX 10 x 4 truck was selected for the study and is reflected in the capital and operating cost estimates.

The size of the haulage fleet was based on estimated cycle times from each level applied against the tonnage scheduled from each level. The aggregate engine hours dictate the instantaneous (active) fleet size and the total fleet size considering availability (estimated at 84%). The following parameters were used in the estimation of aggregate engine hours:

- Payload: 49.6 t (BAS FMX 10 x 4)
- Payload: 40.4 t (Volvo A45G)
- Average haul speed: 10 km/h loaded, 15 km/h empty
- Spot time: 0.33 minutes
- Loading time: 5 minutes (3 LHD passes per load)
- Dumping time: 2 minutes
- Lost time for traffic conflicts: 10%

Figure 16.22 provides a summary of the development and production equipment fleet by year.

Figure 16.22 Development and production equipment fleet (excluding service fleet)



Source: MP 2022.

Table 16.23 Mobile equipment requirements

Equipment type	Phase 1 peak	Phase 2 peak	Manufacturer	Model
Development				
2-Boom jumbo	5	5	Sandvik	DD422i
LHD 17-tonne	4	3	Sandvik	LH 517
Haulage truck - development	5	3	Volvo	A45G FS
Loader / backhoe	2	2	JCB	3X17
Bolter	4	3	Sandvik	DS411-C
Cable bolter	2	2	Sandvik	DS421
Mechanical scaler	2	2	Normet	Scamec 2000 M
Shotcrete sprayer	2	2	Normet	Spraymec LF050 VC
Face charger	2	2	Normet	Charmec LC605DEV
Transmixer	3	2	Normet	Utimec LF700
Production				
Production drill - Longhole	2	5	Sandvik	ITH DU 422i
Blasthole emulsion loader	1	2	Normet	Charmec LC605
Haulage truck - production	12	11	Volvo	BAS FMX 520HP 10x4
LHD 21-tonne		11	Sandvik	LH 621
Blockholer	1	1	Maclean	BH3
Raiseborer (Stopes)	1	2	Atlas Copco	34RH C QRS
Services				
Personnel carrier	4	3	Normet	LF100 &C122 Cassette
Scissor lift truck	2	2	Normet	UTILIFT MF 540
Service truck	2	1	Normet	MF 400 Lube
Boom truck	2	1	Normet	LF 130 Material
Emulsion transport	2	1	Normet	LF100 &C600 Emulsion Cassette
Explosives transport	1	1	Normet	LF100 &C600 Emulsion Cassette
Motor grader	2	2	Paus	PG10HA
Fuel truck	2	1	Normet	Utimec LF 1000
Pick-up truck	24	24	Nissan	Navara
Forklift	2	1	Komatsu	FD45T-10
Telehandler	2	1	Manitou	MT 1335 HA
Water truck	2	2	Normet	Utimec LF 1000
Total underground	**95**	**98**		

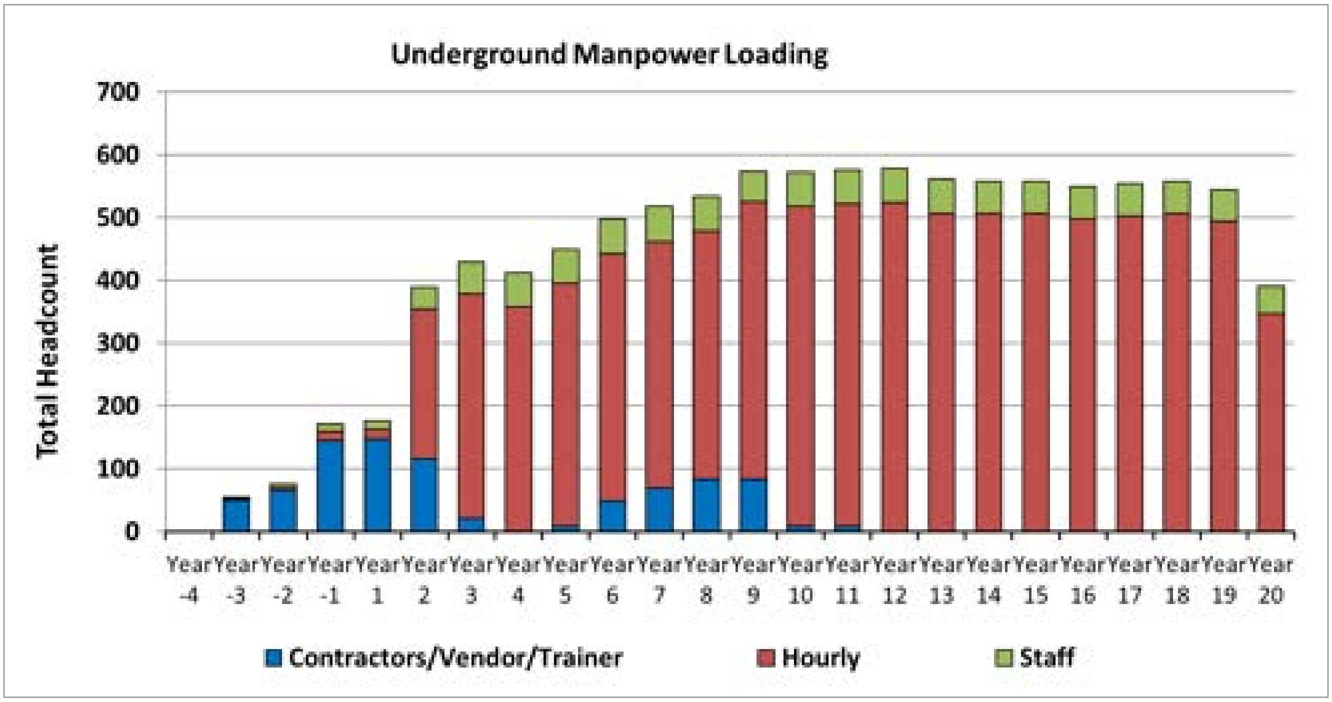
16.3.16 Underground personnel

The work schedule assumes 350 days of 24 h/day mining operation. Fifteen days are assumed to be lost to inclement weather and holidays. Operations and mining personnel will work on three 8 h shifts per day. The underground operations will be supported by contract labour through the beginning of production stoping at the end of Year 2, which will then switch to an owner-operated team. An additional group of contractors will be present during construction of the Phase 2 materials handling system. The contractor will be utilized to excavate, support, and construct the shaft headframe, shaft, loading pocket, ore bins, crusher, and conveyors.

A training program will be utilized with vendor and expatriates training up the Greek workforce at the outset of production mining. This will be done over the course of 12 months beginning in Year 2.

The staff, labour, and contractor requirements over the LOM for operations and maintenance departments are summarized in Figure 16.23.

Figure 16.23 Personnel requirements



Source: MP 2022.

17 Recovery methods

17.1 Process design basis

The original process plant and infrastructure design for the Project was prepared by Aker Kvaerner, (subsequently called Aker Solutions and Jacobs, and now part of the Worley Group) and presented in a 2007 Cost and Definition study (the Cost and Definition study). Subsequent to the Cost and Definition study, Outotec was retained on a design & supply contract to complete the basic engineering for the Project process facilities. Outotec's scope of work included the supply of process equipment within its manufacturing range, including the SAG mill, primary ball mill, regrinding ball mill, flotation cells, and plant control system. In parallel to this, ENOIA S.A. Athens (design and application engineers), Omikron Kappa (infrastructure engineering), GAL (environmental and tailings storage), and Paterson & Cooke (slurry pipeline engineering) completed the basic and detailed engineering for the Project, under the supervision of Hellas Gold and Eldorado.

The layout of the plant was developed from the design basis defined in the Cost and Definition study and has incorporated many design improvements. Eldorado has decided to defer installation of the gravity gold recovery system consisting of "gravity classification", "secondary gravity classification" and the "gold room" (refer to Figure 17.1). Eldorado has reconsidered the need for the secondary gravity circuit after regrind mill due to the reduced efficiency of gravity recovery methods at a P_{80} size of 34 µm and has conducted testwork to verify that acceptable gold recovery is achieved in rougher flotation without recovery of gravity-recoverable-gold prior to flotation. Eldorado intends to verify by plant testing that the gravity gold recovery system is not required once the plant is in operation. To facilitate installation at a later date if required, equipment already purchased will be retained on site and the plant layout will not be altered from the configuration that includes gravity gold recovery.

For the first ten years of operation, the ore will be extracted from the open pit mine as well as from the underground mine for a total mill feed tonnage of 8.0 Mtpa. From the eleventh year until the thirteenth year of the operation, the plant will process ore extracted from the underground mine at an average tonnage of 6.4 Mtpa, while also processing re-handled oxide ores stockpiled in Phase 1 for a total mill feed tonnage of 8.0 Mtpa. From the fourteenth year until the end of the mine life, the plant will process exclusively underground ore at an approximately 6.2 Mtpa rate.

The main design parameters of the process plant are indicated in Table 17.1. The plant will process the copper / gold ore at a projected head grade of 0.50% copper and 0.77 g/t gold (LOM averages). Anticipated LOM average payable recoveries for sulphide ore are 87% for copper and 81% for gold, respectively. The mill will produce a copper / gold concentrate that contains an average of 26% copper and 27 g/t gold. Metallurgical tests have shown that the ore contains a small amount of Pd, which will be collected into the copper / gold concentrate during flotation.

Table 17.1 Basic design parameters of Skouries process plant

	Parameter	Design basis
1	LOM	23 years
2	Maximum treated ore annually	8.0 Mtpa
3	Days of operation per year	350 d/y
4	Maximum daily ore production from mining (open pit plus underground)	24,000 tpd
5	Open pit primary crushing product particle size	80% passing 150 mm
6	Maximum daily ore treatment	24,000 tpd
7	Specific energy requirement of SAG mill grinding	7.1 kWh/t
8	Specific energy requirement of primary ball mill grinding	7.1 kWh/t
9	Particle size of flotation feed	80% passing 120 µm
10	LOM average gold grade in the mill feed	0.77 g/t
11	LOM average copper grade in the mill feed	0.50%
12	LOM overall payable gold recovery	81%
13	LOM payable copper recovery	87%
14	LOM average gold grade in flotation concentrate	27 g/t
15	Average copper grade in flotation concentrate	26%

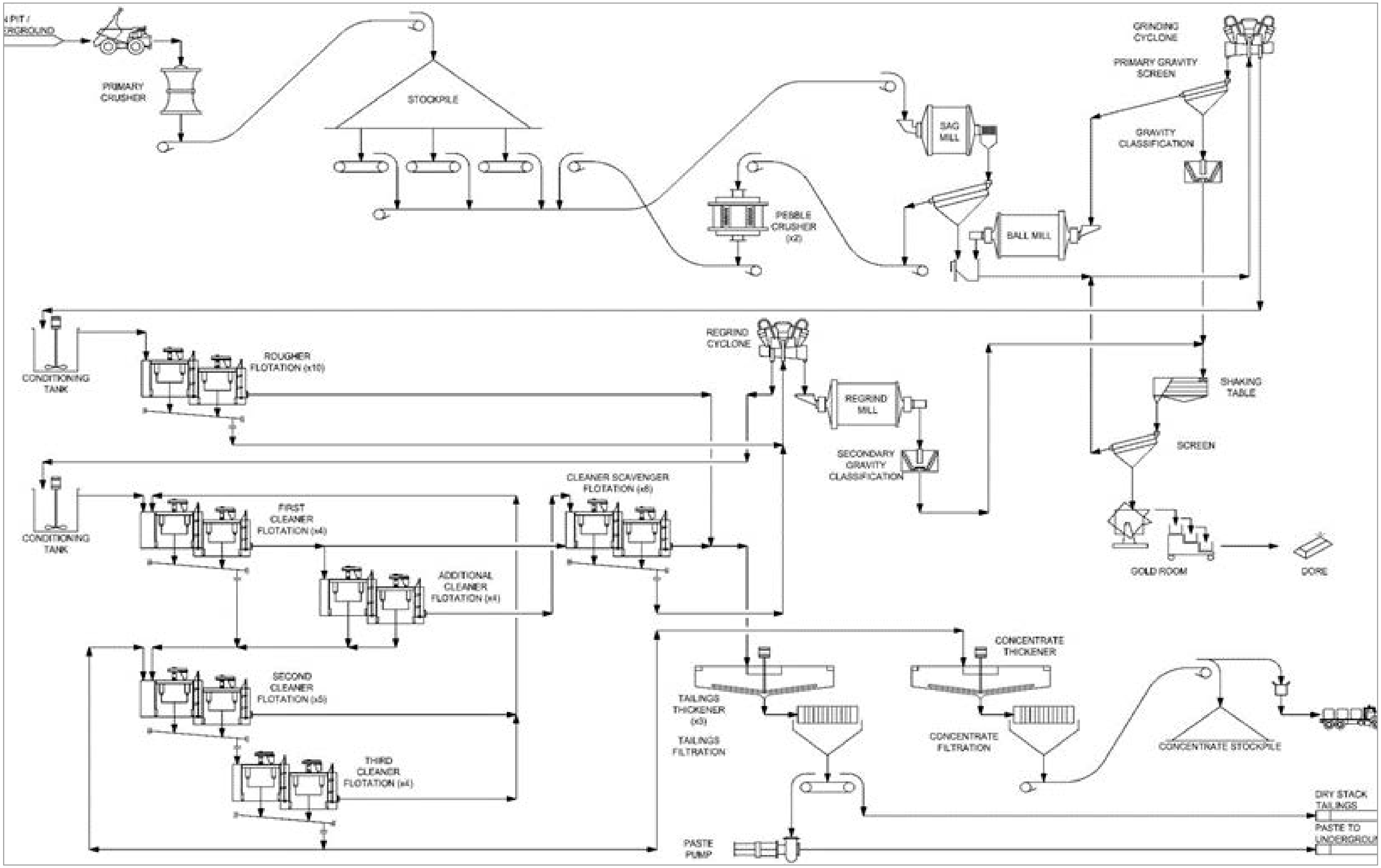
17.2 Process description

The process plant design has been developed using the results of extensive metallurgical testwork. It is based on a conventional flowsheet for the treatment of porphyry copper ores and, as such, offers a well proven design. The comminution circuit comprises primary crushing and an SABC grinding circuit, with gravity recoverable gold able to be extracted prior to ball milling (see above regarding deferment of gravity gold recovery). Following comminution, a flotation circuit consisting of rougher cells with three-stage cleaning of rougher concentrate will be used. This design, coupled with the straightforward ore metallurgy and acceptable design margins, offers a robust, low risk processing solution to the treatment of Skouries ore. The size and type of selected equipment are well proven in the industry and present minimal risk.

The process plant design provides for a nominal 8.0 Mtpa of ore throughput. The Skouries simplified process flow diagram (PFD) is presented in Figure 17.1. The unit operations comprise the following:

- Primary crushing and crushed ore stockpile.
- SABC grinding and pebble crushing.
- Flotation and regrinding.
- Flotation concentrate and tailings thickening.
- Flotation concentrate filtering, storage and loadout.
- Tailings filtration, conveying and paste fill production.
- Reagent preparation and plant services.

Figure 17.1 Simplified process flow diagram



Note: Gravity circuit and gold room shown but currently deferred.
Source: Eldorado 2022.

17.3 Primary crushing and stockpile

17.3.1 Primary crushing

The ore is delivered by 90-t haul trucks from the open pit and 50-t haul trucks from underground to the primary crushing station dump pocket. The ore is then crushed by the gyratory primary crusher and the crushed ore is discharged via the primary crusher discharge feeder.

The transportation of the crushed ore from the primary crusher discharge to the covered, conical, crushed-ore stockpile is implemented using a belt conveyor system that is comprised of the primary crusher sacrificial conveyor and the primary crusher discharge conveyor. An over-belt magnet at the discharge of the sacrificial conveyor removes scrap metal before ore enters the ore stockpile.

An automatic dust suppression system will be installed at the crusher feeding point and at the crusher discharge feeder to prevent dust emission from the ore unloading and crushing operations.

The belt conveyors are equipped with blockage detectors for discharge chutes (alarm and trip), zero speed switch in case of belt slippage (trip), and belt alignment switches for belt position (alarm and trip).

17.3.2 Ore stockpile and conveying

The ore stockpile is covered and has a live storage capacity of 24,000 t, equivalent to one day of production. The stockpile area total storage capacity will be approximately 80,000 t, equivalent to slightly over three days of production.

The ore is extracted from the ore stockpile using three variable speed apron feeders, which are of 1,215 t/h total combined capacity and each one is driven by a 75 kilowatt (kW) motor. The feeders are located beneath the stockpile in a tunnel.

The feeders discharge onto a belt conveyor system, which is comprised of the ore reclaim sacrificial conveyor and the SAG mill feed conveyor, which reports to the SAG mill feed chute.

The feeders have been designed and located to maximize live storage capacity of the stockpile. The feeding rate to the mill is controlled by the control system in the plant.

An automatic dust suppression system will be installed at the apron feeders to prevent dust emission. The spray water is collected in the inclined reclaim tunnel floor.

The belt conveyors are again equipped with blockage detectors for discharge chutes (alarm and trip), zero speed switch in case of belt slippage (trip), and four belt alignment switches for belt position (alarm and trip) and will be covered and guarded.

17.4 Grinding circuit and pebble crusher

The primary grinding circuit is designed to reduce the feed ore with a particle size of 80% passing 150 mm to a product with a particle size of 80% passing 120 µm. The size reduction is achieved by a two-stage wet grinding circuit comprising a SAG mill driven by variable speed motors, a ball mill (with a fixed speed motor) and a pebble crushing circuit (SABC circuit).

The transfer size of the material between the SAG mill and the ball mill is in the range of 80% passing between 2.0 mm and 3.6 mm. The SAG mill has a diameter of 9.75 m and an effective grinding length of 4.57 m and is driven by two 4.8 MW motors with variable frequency drives (VFD) and including auxiliary lubrication circuits. The grinding media consists of 125 mm diameter balls made from high quality forged steel. The SAG mill liners are Cr-Mo alloy cast steel.

Two shorthead type cone crushers will be installed for the crushing of the oversize pebbles (+12 mm) produced by the SAG mill. Total crushing capacity is 254 t/h and each cone crusher will be driven by a 450 kW motor.

The primary ball mill has a diameter of 7.01 m and an effective grinding length 9.75 m and is driven by two 4.8 MW motors. The grinding media consists of 60 mm diameter balls made from high quality forged steel. The ball mill will be lined with rubber liners and lifters.

Both mills will be equipped with trunnion bearings and drive gear lubrication, drive protection and cooling systems. The SAG mill product flows to a vibrating screen with 12 mm openings. The oversize particles (> 12 mm) are transferred via the pebble conveyor No. 1 to the pebble crushers. The crushed product will be transferred back onto the SAG mill feed conveyor via the pebble conveyor No. 2.

The undersize particles (< 12 mm) flow to the concrete SAG and ball mill sump, where they are mixed with the ball mill product. In this sump, process water of controlled quantity is also added to control cyclone feed density and ball mill circulating load.

The slurry product of both mills is pumped to a 660 mm diameter hydrocyclone cluster. The hydrocyclone overflow slurry with solids content of, typically, 35% w/w and particle size of 80% passing 120 µm comprises the feed of the flotation circuit. The cyclone underflow slurry is directed to the ball mill feed chute. The grinding circuit operation is controlled by an automated control system to ensure that the product size for all types of ore is 80% passing 120 µm at the maximum daily ore feed of 24,000 t.

Any spillages are collected in floor sumps and directed back via pumps to the appropriate points of process.

The main plant control room is located in an elevated area adjacent to the grinding and flotation sections to provide a panoramic view of operations and quick access to the plant.

17.5 Flotation and regrinding

17.5.1 Flotation circuit

Flotation is carried out in six stages, as shown in Figure 17.1:

- Rougher.
- 1st cleaner.
- Cleaner scavenger.
- 2nd cleaner.
- 3rd cleaner.
- Additional cleaning, in order to produce a clean copper / gold concentrate while achieving satisfactory recoveries.

The process equipment is of the latest proven technology and is completely automated. The flow of concentrate from one area to the other is implemented using gravity, where possible, in order to minimize the pumping and, consequently, energy consumption.

The flotation circuit is fed from the cyclone overflow, which is of particle size 80% passing 120 µm. The cyclone overflow slurry is directed into a 160 m^3 conditioning tank, where it is mixed with flotation reagents. From there it flows into the rougher flotation bank. The conditioning tank is also used to increase surge storage capacity.

The flotation cells are mechanically agitated and air-blown. The level of the slurry is controlled by dart valves located at the outlet of each pair of flotation cells. Flotation air is produced by two main multistage centrifugal air blowers, one duty and one standby, and a third centrifugal air blower of lower capacity. The airflow to each rougher flotation cell is individually controlled. Variable speed slurry pumps will be installed in the flotation circuits. Pump speeds and, hence, capacities are controlled according to level measurements in the respective pump sumps. For each duty, two pumps will be installed (one in operation and one on standby).

The flotation circuit operation is appropriately automated, including automatic sampling of the main streams and continuous on-stream analysis. Any spillages are collected in floor sumps and, via pumps, are recycled back to the production circuit.

The rougher flotation circuit is comprised of a bank of ten 160 m^3 cylindrical tank cells. The rougher flotation concentrate is directed to the regrinding circuit, to be reground to 80% passing 34 µm. The regrind cyclone overflow feeds the 1st cleaning flotation circuit. The rougher flotation tailings are directed to the tailings thickeners. Additional reagent dosing (SIPX or AP3418, MIBC or DF250 frother, promoter AP5540, sulphidizer if necessary and lime) is provided to different cells as required by the process measurement and control system.

The 1st cleaning flotation circuit is comprised of the first cleaner conditioning tank, of 50 m^3 net capacity, and four 50 m^3 flotation cells. The 1st cleaning flotation concentrate is directed to the flotation cells of the 2nd cleaning circuit, whereas the tailings of the 1st cleaning flotation stage are fed either to the cleaner scavenger flotation circuit or to the additional cleaning circuit. Additional reagent dosing (SIPX or AP3418, MIBC or DF250 frother, promoter AP5540, sulphidizer if necessary and lime) is provided to the first cleaner conditioning tank, as required, by the process measurement and control system.

The cleaner scavenger flotation circuit is comprised of six 50 m^3 cells. The scavenger flotation concentrate is directed to the regrinding circuit, where it is mixed with the rougher flotation concentrate. The tailings of the cleaner scavenger stage are mixed with the rougher flotation tailings before being directed to the tailings thickeners. Additional reagent dosing (SIPX or AP3418, MIBC or DF250 frother, promoter AP5540, sulphidizer if necessary, guar gum and lime) is provided to the cells in the cleaner scavenger flotation circuit, as required, by the process measurement and control system.

The 2nd cleaning flotation circuit is comprised of five 10 m^3 cells. The 2nd cleaning flotation concentrate is directed either to the cells of the 3rd cleaning circuit or to the concentrate thickener. The tailings of the 2nd cleaning flotation are recycled back to the 1st cleaning flotation circuit. Additional reagent dosing (MIBC or DF250 frother, sulphidizer if necessary and guar gum) is provided to the cells of the 2nd cleaning flotation circuit, as required, by the process measurement and control system. Lime solution is added to the second cleaner concentrate pump box.

The 3rd cleaning flotation circuit is comprised of four 10 m^3 cells. The 3rd cleaning flotation concentrate is directed to the concentrate thickener. Additional reagent dosing (sulphidizer and guar gum) is provided to the cells of the 3rd cleaning flotation circuit, as required, by the process measurement and control system.

The additional cleaning flotation circuit is comprised of four 10 m^3 cells. The additional cleaning flotation concentrate is directed to either the 2nd cleaning circuit or the regrinding circuit. The tailings of the additional cleaning circuit are directed to the cleaner scavenger flotation stage.

Centrifugal pumps located in the floor sumps collect any spillages and recycle them back to the appropriate points of the process.

17.5.2 Regrinding circuit

Concentrate regrinding is carried out by the regrind ball mill. The regrind ball mill has a diameter of 4.60 m and effective grinding length of 7.00 m. The mill is driven by a 2.25 MW motor and is equipped with lubrication, drive protection and cooling systems. The regrind mill has rubber lining and the regrinding media consists of forged steel balls of, typically, 25 mm diameter.

The regrind ball mill operates in closed circuit with a cluster of 14 hydrocyclones (250 mm diameter), 10 duty and four on standby. The operation of the regrind circuit will be controlled by the central distributed control system (DCS) of the plant.

The regrinding circuit is fed with the concentrates of rougher and cleaner scavenger flotation as well as the regrind gravity concentration tailings and additional cleaner concentrate. These are directed to the hydrocyclone cluster. The regrind cyclone cluster overflow will flow to the 1st cleaning circuit. The regrind cyclone underflow feeds the regrind ball mill.

17.6 Thickening

17.6.1 Concentrate thickening

The final concentrate of the 3rd cleaning flotation stage is directed to a concentrate thickener of 12 m diameter. Flocculant solution is added to increase the solids sedimentation rate. The solids concentration in the thickener underflow is typically 60% w/w. The underflow is pumped to filter presses for filtration.

17.6.2 Tailings thickening

The flotation tailings are equally distributed to three high-rate tailings thickeners of 26 m diameter. Flocculant solution is added to increase the solids sedimentation rate. The solids concentration in the thickener underflow is typically 55% to 65% w/w. During the underground mine development phase, the thickener underflow is filtered, with the filter cake transported by truck for dry-stack disposal. Once underground mining has commenced, a portion of the thickener underflow will report to the paste backfill plant and the remaining portion to the tailings disposal as dry-stack. The overflow from the three thickeners is recirculated as process water in the plant.

17.7 Concentrate filtering, storage, and loading

The concentrate thickener underflow is pumped to filter presses, for filtration to achieve the targeted moisture content. The filter cake is transferred with loaders to an adjacent covered concentrate storage space. There is also a second filter press in the building for spare capacity, as well as a concentrate bulk bag loading system.

The concentrate is transported offsite to smelter customers via closed cargo trucks.

17.8 Tailings filtration

Tailings from the thickening facility will be combined into a distribution box to split the slurry evenly into the three tailings stock tanks. Each stream will gravitate into one of the agitated tailings stock tanks, which will provide buffer storage ahead of the filtration stage. The slurry from the tailings stock tanks will be pumped to one of the tailings filter presses with dedicated centrifugal pumps during normal operation. An additional press will be available to provide standby capacity or offline for planned maintenance. The filters will produce a dry filter cake with a target moisture content of approximately 13% w/w and filtrate containing a small amount of fine solids.

During Phase 1, the dry tails material will be diverted to the IEWMF. During Phase 2, the filtered tailings will be conveyed from the filter plant to the open pit for permanent placement.

17.9 Flotation reagents

The reagents used in the flotation circuit are the following:

- Sodium IsoPropyl Xanthate (SIPX) or Aerophine 3418 as the primary collector.
- Aeropromoter MX-5010 (or an alternative thionocarbamate equivalent to the now discontinued Aeropromoter 5540 on which the original testwork for the Project was based) as the promoter.
- Methyl IsoButyl Carbinol (MIBC) as the main frother, and Dowfroth 250 as an auxiliary frother (polyalcohols).
- Guar Gum (polysaccharide) as a dispersant to reduce the amount of gangue minerals (fluoride) in the final copper concentrate.
- Hydrated lime as a pH modifier and pyrite depressant in the flotation circuit.
- Sulphidizer reagent (NaHS) will also be used only during the treatment of oxidized mineral from the open pit.

The process for each reagent is described in detail in the following paragraphs.

17.9.1 Chemical reagents system

17.9.1.1 Sodium IsoPropyl Xanthate (SIPX)

SIPX will be delivered in industrial bags of around 800 kg each, of which 90% are active pellets.

The bags are transported by an electric hoist in the plant building, where SIPX pellets will be dosed from a hopper to a mechanically agitated mixing tank. Fresh water will be added in the mixing tank for making up a solution of 10% w/w concentration.

The mixed SIPX solution will be transferred by pump from the mixing tank into a storage tank, which provides 48-hour live storage capacity for the requirements of the flotation plant.

SIPX solution is dosed continuously to the respective dosing points via metering pumps and a dosing system with magnetic flow meters and control valves. Reagent dosing rates are set and controlled by the main control system in the plant.

17.9.1.2 Aeropromoter MX-5010 (or equivalent) as promoter

Aeropromoter MX-5010 (or an alternative thionocarbamate equivalent to the now discontinued Aeropromoter 5540) is delivered in barrels of 200 L. On average, five barrels per day are pumped into a storage tank, which provides 54-hour live storage capacity for the requirements of the plant. The promoter is dosed as a 100% solution without dilution.

The promoter is dosed continuously to the respective dosing points of the flotation circuit via metering pumps. Reagent dosing rates are set and controlled by the main control system in the plant.

17.9.1.3 Frothers

Both MIBC and DF250 will be delivered in barrels of 200 L. On average, five barrels per day will be pumped into a storage tank, which provides 58-hour (for MIBC) and 71-hour (for DF250) live storage capacity for the requirements of the plant. The frothers are dosed as a 100% solution without dilution.

The frothers are dosed continuously to the respective dosing points of the flotation circuit via metering pumps. Reagent dosing rates are set and controlled by the main control system in the plant.

17.9.1.4 Guar gum

The guar gum is added to the flotation slurry and acts as a dispersant to reduce the amount of fine gangue mineral particles that are recovered via entrainment in the final copper concentrate. This improves quality and marketability of the copper concentrate. It is delivered in powder form and is mixed with water in an agitated mixing tank. The prepared solution is then transferred by pump into a storage tank, from which it is dosed to the respective dosing points of the flotation circuit via metering pumps. Guar gum solution preparation and distribution takes place in a dedicated building.

17.9.2 Sulphidizer system

The sulphidizer, i.e., sodium hydrosulphide NaHS, is used to regenerate the surface of the nonsulphur copper minerals such as cuprite (Cu_2O), azurite ($2CuCO_3.Cu(OH)_2$), chrysocolla ($CuSiO_3.2H_2O$), and malachite ($CuCO_3.Cu(OH)_2$) so that they can be floated with xanthate or aerophine collectors. It is also used in the flotation of the oxidized sulphur-containing copper minerals. In both cases, the sulphidizer (NaHS) acts as an activator; this process is known as "sulphidation".

Sodium hydrosulphide is delivered as flakes in bulk bags and is prepared in a separate area next to the reagent room. An adjacent temporary storage building will be available as required, given the fact that the sulphidizer will be used only during the treatment of oxidized minerals from the open pit.

The sulphidizer solution is pumped from the mixing tank to the storage tank. The solution is dosed continuously to the respective dosing points of the flotation circuit via metering pumps, flow meters and control valves. Reagent dosing rates are set and controlled by the main control system in the plant.

17.9.3 Lime system

Hydrated lime is delivered with tanker trucks (carrying 20 t to 30 t payload) into two 150 m^3 storage silos. Silos will be equipped with weight measurement and dust removal bag filters on the top, to prevent dust emission during unloading. Two silos cover the lime requirements for 18 days of operation for sulphide ore treatment and seven days for oxide ore treatment. The lime is transferred via screw feeders to mechanically agitated mixing tanks, each of 25 m^3 capacity. Water will be dosed to the mixing tank prior to addition of lime powder from the silos. Milk of lime concentration is 15% w/w.

Milk of lime is pumped from the mixing tank to the 94 m^3 storage tank and is dosed continuously to the respective dosing points of the flotation circuit via pumps and control valves. Reagent dosing rates are set and controlled by the main control system in the plant.

17.10 Flocculant system

Polyelectrolyte flocculant solution is added to concentrate and tailing slurries to increase the settling rate of solids. The flocculant solutions are prepared in three different systems, each located in a different section of the plant. One system serves the concentrate thickener, the second serves the tailings thickeners, and the third serves the mine water clarifier.

All flocculant solution concentrations will be 0.25%.

17.11 Process and fresh water management and storage systems

17.11.1 Process water system

A storage tank (6,100 m^3) is used for storage of process water. The water comes mainly from boreholes as well as from the overflows of the tailing thickeners and the mine water clarifier. The water is distributed to the relevant locations via a pump station, with three process water pumps (two running / one standby) and a 610 mm distribution pipeline.

17.11.2 Utility water system

A storage tank (1,000 m^3) is used for storage of utility / fresh water. Process water is recycled within the plant using a pond and thickeners, as shown in Figure 7.1. Make-up water is provided from boreholes and from the mine water clarifier. The water is then distributed via pumps to the various locations for system make-up, cooling systems, pumps gland seals, and dust suppression systems.

17.11.3 Firefighting water system

A storage tank (850 m^3) is used for storage of firefighting water. The water comes mainly from the boreholes and is distributed where required via a main 254 mm pipeline. The firefighting water distribution is effected by the firefighting pump station, which is comprised of an electrically driven fire water pump, a diesel driven fire water pump and a fire water jockey pump.

17.11.4 Potable water system

A storage tank (25 m^3) is used for storage of potable water. The water comes from the boreholes and is treated and distributed where required via the necessary pumps and a main 100 mm pipeline.

17.12 Plant infrastructure and utilities

The process plant will be supported by a comprehensive operational complex comprising maintenance facilities, stores, changehouses, administration offices, first aid centre, messing facilities, site security system, and process plant mobile equipment fleet.

Total power consumption of the infrastructure, open pit and processing plant is projected to be 43.5 MW, of which the process plant consumption is 36.8 MW. The 20 kV switchgear has power factor correction units installed, providing power factor control within 0.95 and 1. Therefore, the total power consumption at 0.95 would be 45.8 mega volt amps (MVA). The Greek power authority, ADMHE, will be required to supply 50 MVA of power to the site.

Multiple UPS systems providing 30-minute back-up will be installed in main process areas to control system supplies and essential instrumentation, and for monitoring purposes. These facilities will be supplied from essential services low-voltage (LV) switchboards, with diesel generator back-up.

An emergency diesel generator rated at 1,700 kilovolt-amps (kVA) will be connected to the essential services LV switchboard to supply critical drives and equipment in the event of power failure at the main process plant. Emergency lighting will also be fed from the emergency diesel generators; also 1.5-hour internal battery lighting for instantaneous emergency lighting upon a power failure.

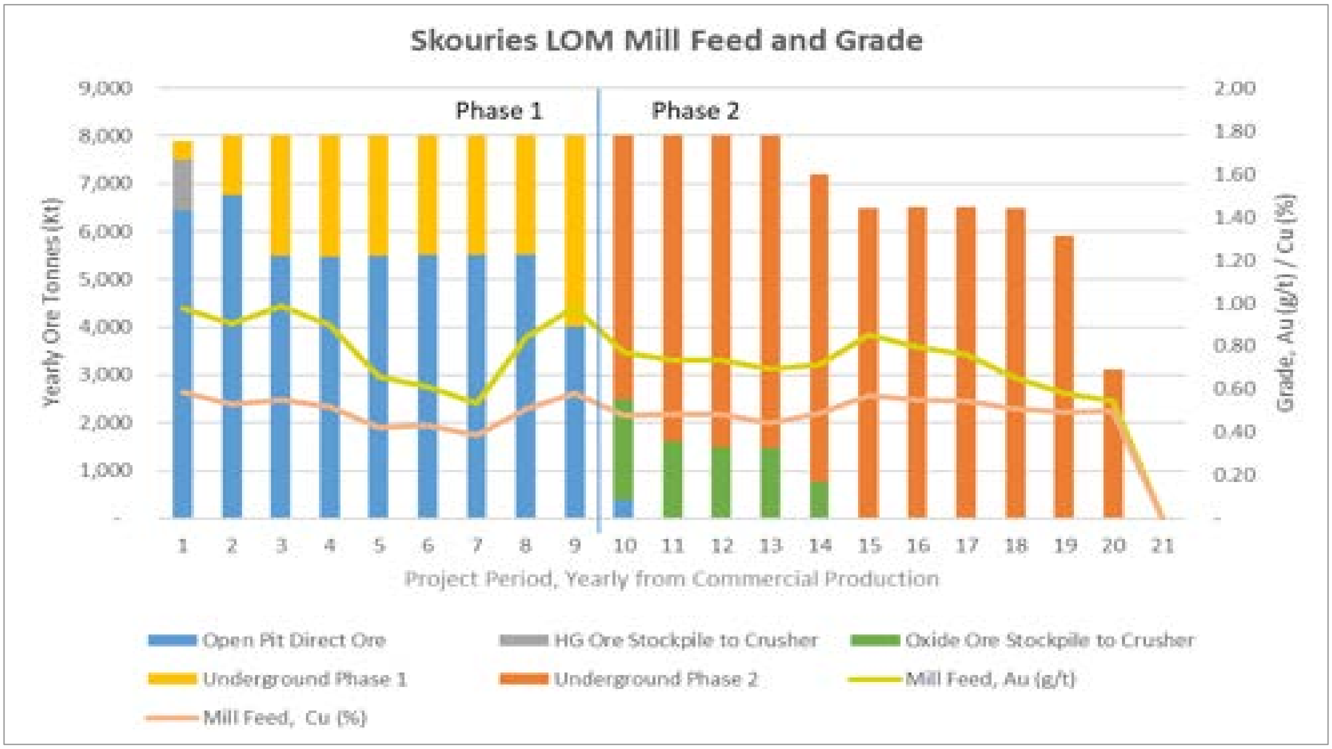
Additionally, a 1,700 kVA emergency diesel generator will be located at the thickener tanks, with a further 110 kVA unit at the primary crusher, a 1,700 kVA unit at the filter plant and a 450 kVA unit for the supply of the potable water package and for administration buildings.

The process plant and site infrastructure have been located on a site that provides the best balance between geotechnical constraints and location relative to the open pit, underground shaft, and ore transfer conveyor system. The plant is of compact design but provides sufficient room for maintenance access and for the installation of major equipment packages.

17.13 Mill feed schedule

Figure 17.2 shows the LOM mill feed schedule of tonnes and metal grades.

Figure 17.2 LOM mill feed and grade



Source: Mining Plus 2022.

As noted in Section 16, the Phase 1 mill feed consists of an average of approximately 5.5 Mtpa from the open pit together with about 2.5 Mtpa from underground, with a total average mill feed of 8.0 Mtpa. In the initial underground mine ramp-up period at the start of mine life, the open pit feed rate is around 6.5 Mtpa. In the first five years of Phase 2 (to Year 14), oxide ore that has been stockpiled in Phase 1 will supplement the ore from underground. From Year 15 to Year 19, Phase 2 average annual mill feed is maintained at or around 6.5 Mt, solely from underground mine production. The final year of production and mill feed is projected at 3.1 Mt. Both gold and copper head grades are seen to be higher in the early years of mine life.

18 Project infrastructure

18.1 Open pit mine infrastructure

Mine infrastructure, including ancillary facilities and services, has been fully designed to support the Phase 1 open pit mine production. Surface ancillary facilities are situated to be close to the open pit access ramp and primary crusher dump pocket. The ancillary facilities include the production services building, the surface workshop and warehouse, and surface fuel storage.

18.2 Underground mine infrastructure

Mine infrastructure, including ancillary facilities and services, has been fully developed to support both Phase 1 and Phase 2 underground mine production. During Phase 1, the ancillary facilities can be economically located on surface. During Phase 2, the ancillary facilities will be developed underground to suit the mining and material handling methods. Underground services, including egress, dewatering, ventilation, compressed air, power, and communications and control are initially developed during Phase 1 and expanded during Phase 2 as the mine level development continues. Further details of these facilities can be found in Section 16.

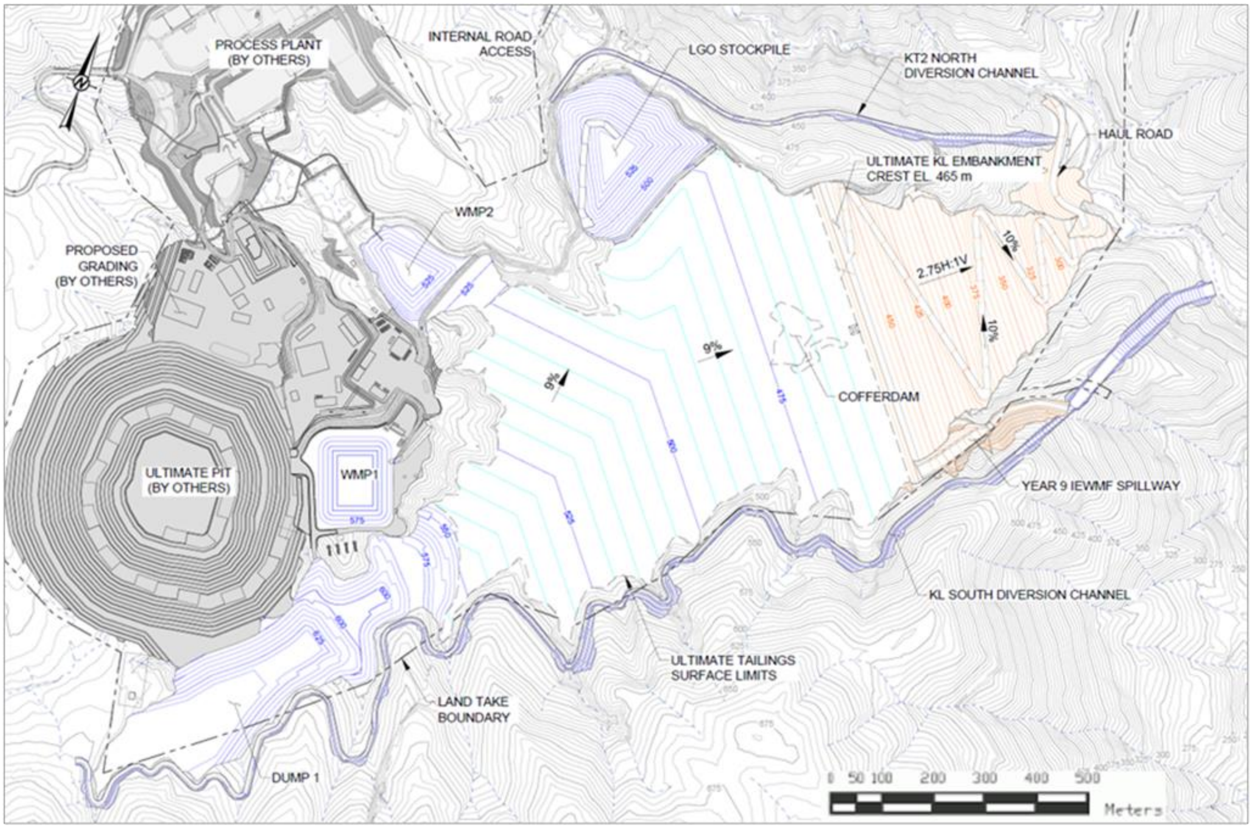
18.3 Waste management

The principal waste streams generated from the project are overburden and waste rock from the open pit mining and underground development, and tailings from the mineral processing operations. Overburden and waste rock will be stored on surface and used to construct project infrastructure. Based on the 28 September 2021 mine plan, ore processing will generate approximately 144.4 Mt of tailings, of which 38.1 Mt will be used underground as paste backfill. The remainder will be filtered and stacked for storage on surface, with 63.4 Mt stored in the IEWMF and 42.9 Mt placed as backfill into the open pit. The project will be mined in two operational phases over a 20-year LOM.

Phase 1 will be developed using both open pit and underground mining techniques over the first 12 years of operation (Year -3 through Year 9), with some ore and waste mined in Years -3 to -1 during the development phase. Overburden and waste rock from the Phase 1 mining activities will be used as the dominant source of construction material for the IEWMF Karatza Lakkos (KL) embankment and cofferdam, WMP1 (primarily in cut), WMP2, LGO stockpile embankment, process pads, and other site infrastructure. Some tailings produced during Phase 1 will be deposited underground as paste backfill; the balance will be stored above ground on surface as filtered and compacted tailings in the IEWMF. Surplus waste rock material will be stockpiled during Phase 1 in Dump No. 1 and re-handled as closure cover materials for the IEWMF tailings surface and above the backfilled tailings surface in the open pit.

During Phase 2, mining will consist entirely of underground mining for an additional 11 years. Tailings will be deposited underground as paste backfill and on surface as filtered and compacted tailings in the open pit. Underground mine development will be essentially completed during Phase 1, resulting in only a small amount of waste rock being generated during Phase 2. Any waste rock that is generated from underground mining activities during Phase 2 will be used underground as backfill (refer to Section 16 for details). Major waste management component areas are shown on Figure 18.1.

Figure 18.1 IEWMF site layout



Source: Golder 2022.

The majority of waste rock from the Phase 1 mining activities will be used as a source of construction or borrow materials for the IEWMF embankment, cofferdam, WMP1, WMP2, LGO stockpile, process pads, and site infrastructure. Borrow materials include red clay, general overburden waste, and waste rock with varying strength or "hardness" that will be designated for placement in different zones of the IEWMF KL embankment. The red clay will be used as the impermeable layer in the upstream portion of the KL dam section. Drain and filter materials in the dam section are required to be sourced from off-site borrow sources. Surplus waste rock material will be stockpiled during Phase 1 operation in Dump No. 1 and re-handled as closure cover materials for the IEWMF tailings surface. Dump No. 1 is located southeast of the open pit and WMP1, and upslope from the IEWMF tailings storage area, as shown on Figure 18.1. Based on the 28 September 2021 mine plan, the total waste rock volume produced during Phase 1 from the open pit is 60.1 Mt, with an additional 2.9 Mt produced from the underground operations in Phase 1 and a relatively small amount (135,350 tonnes) produced from the underground operations during Phase 2.

Details of the waste management strategy for the two phases of mining are described below.

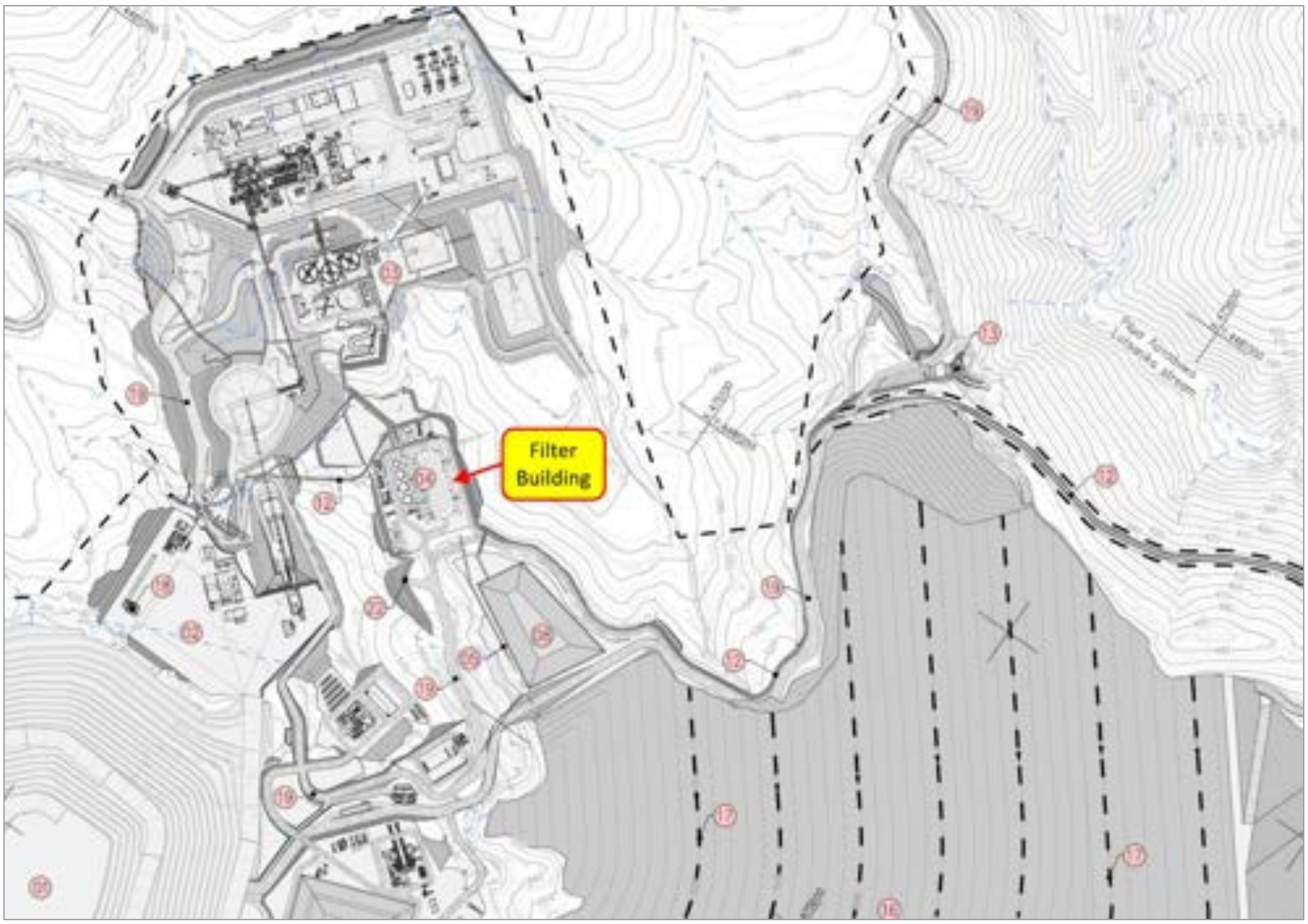
18.3.1 Phase 1 – IEWMF tailings disposal

During Phase 1, tailings will be dewatered to paste for underground backfill, with the remainder dewatered by filtering and stacked in placed in the IEWMF. Tailings will be thickened and filtered prior to placement within the IEWMF. Thickening and filtration processes are described in Section 17. The filtered tailings will be transported to the IEWMF via a conveyor system and then placed, spread, and compacted in lifts with stacking equipment, dozers, and compactors. The final tailings surfaces will have a maximum grade of 9% draining towards the dam and to the spillway that will be located at the south abutment of the KL embankment.

18.3.1.1 Phase 1 – Filtration facility design

The current filtration facility design is the completion of a previous conceptual design by Knight Piésold in 2020. This conceptual design called for waste generated at the beneficiation plant to be filtered in a filtration facility. The facility is located in the vicinity of the beneficiation plant thickeners and close to the flotation waste disposal site. The facility consists of a filter press system to filter the flotation waste generated and a portable conveyor belt array for carrying the filtered tailings to the basin of the IEWMF. The location of this filtration facility on the plant layout is indicated in Figure 18.2 and the layout of the filtration facility is illustrated in Figure 18.3.

Figure 18.2 Filtration facility location on plant layout



Source: Fluor 2022.

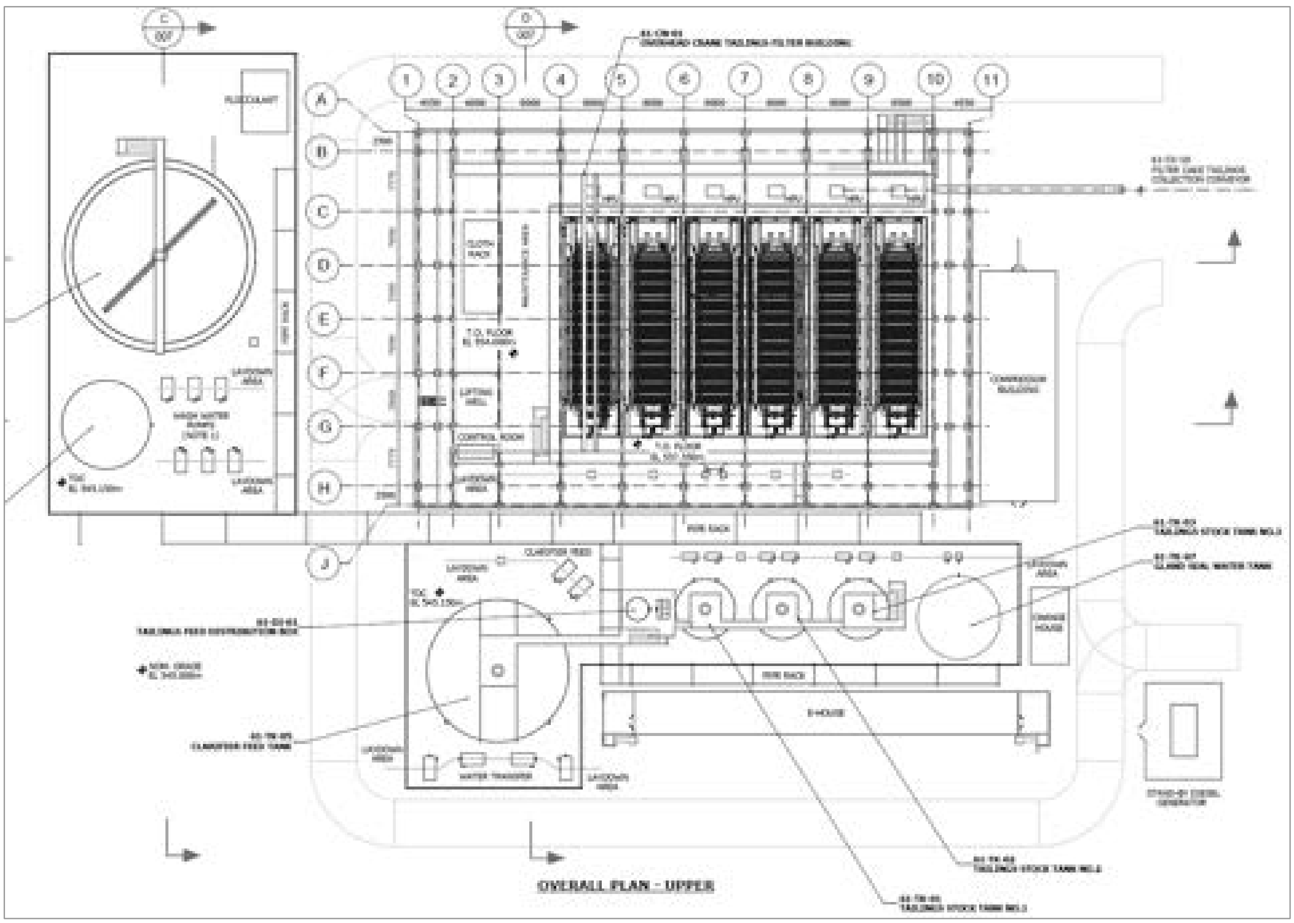
The tailings filtering equipment will be in an enclosed building, fitted with all required ancillaries for the operation and maintenance of the filter presses. The building will be equipped with an overhead crane and lifting wells and provide access on multiple levels to the filter presses and ancillary equipment for operation, inspection, and maintenance. A special laydown and maintenance area will be provided on the northern end of the building for filter cloth storage, repair, and replacement. The lower floor will be open on the sides and allow access for forklifts and maintenance vehicles.

The discharged filter cake moisture levels will be continuously monitored, and any deviations from required moisture set points will be relayed back to the control system to regulate the filter operation. Material with a moisture content outside these bounds will be labelled "off-specification".

When the filter material is labelled off-specification, it can fall into two categories. In the first category, the material is too wet to be placed in the IEWMF but is still conveyable. The material in this category will be conveyed to the bypass area in the IEWMF. Here it will be rehandled before being placed in the IEWMF.

In the second category, the material will be too wet to be conveyed on a belt conveyor and will be in the transformative stage between cake and slurry. The material in this category will be discharged into the trench behind the filter presses. This will be done by reversing the belt feeders after the material is discharged into the hoppers. The material in the trench will be recovered by a bobcat and fed back into the filtration system at the tailings stock tank for refiltration.

Figure 18.3 Filtration facility layout

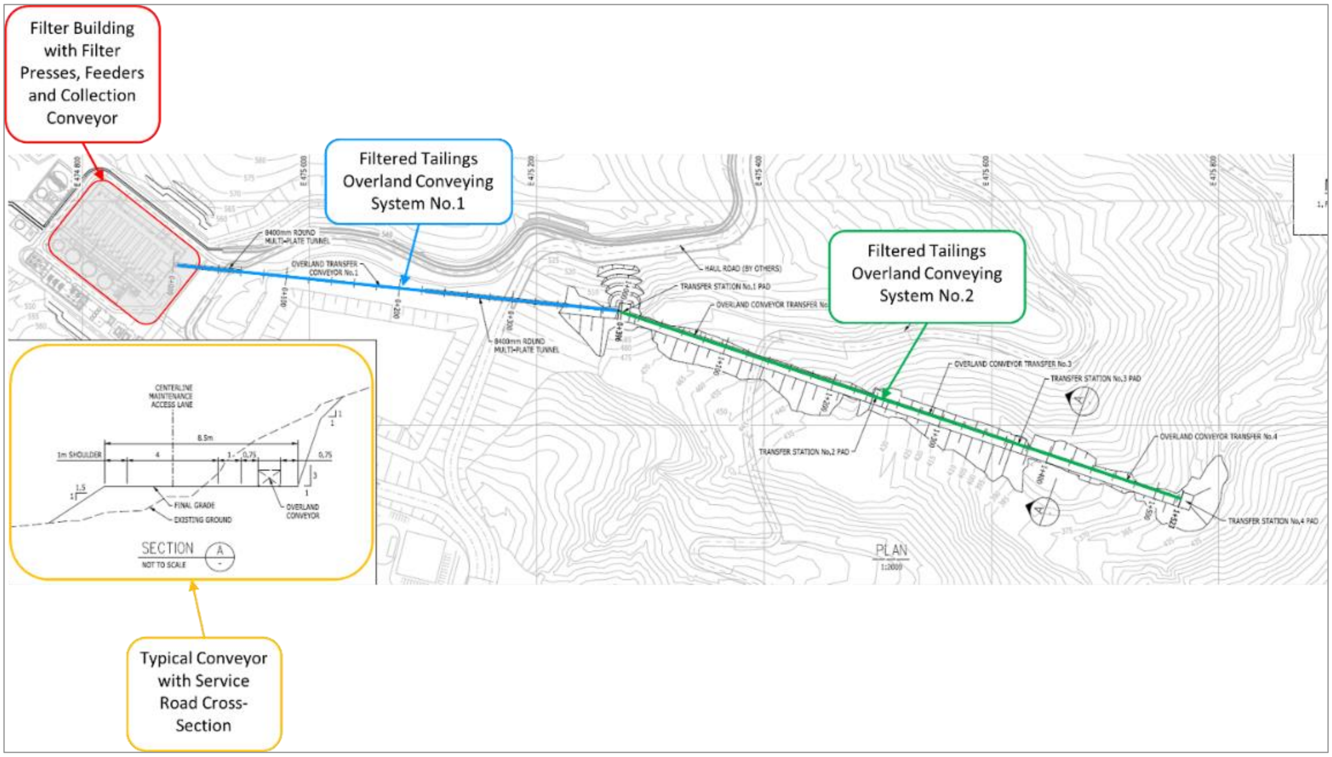


Source: Fluor 2022.

18.3.1.2 Phase 1 – Tailings handling

After filtration, the tailings will be conveyed to the IEWMF on the path as illustrated in Figure 18.4. At the IEWMF, the tailings will be spread and compacted as per the geotechnical requirements. The valley at the bottom of the IEWMF is narrow with steep sides, which, combined with the filtered tailings production rate and the maximum allowable lift height, indicates that the spreading equipment will have to be relocated to a new lift daily in the initial months. This will complicate the operation of both the spreading equipment and the overland conveying system and expose the mine to potential downtime. To mitigate these risks, the spreading operation is split into two stages. The first stage will consist of a dump truck operation, and the second stage will consist of a spreading system operation.

Figure 18.4 Conveying path to IEWMF



Source: Fluor 2022.

The spreading system operation will be done with portable conveyors and spreading equipment to accommodate the rapid rise of the tailings dump. The equipment will be reused during the second phase, when the tailings will be used to fill the open pit before rehabilitation of the area. The proposed portable conveying and spreading equipment will have sufficient redundancy to ensure the specified availability can be achieved.

18.3.1.3 Phase 1 – IEWMF tailings facility

The proposed IEWMF will cover approximately 852,000 m^2, of which the tailings impoundment area behind the KL embankment will cover 615,000 m^2, and will be located in the Karatza Lakkos Valley immediately northeast of the proposed open pit and the process facilities. The IEWMF will fill the Karatza Lakkos Valley and includes the KL embankment on the northeast side and tailings filter stack behind the dam. The KL embankment is a cross-valley embankment dam raised by the downstream method of construction.

The IEWMF will be subject to large earthquakes and storm events and is designed to withstand extreme events. The ultimate IEWMF configuration, with a crest at 465 m, will provide a total tailings storage capacity of 63.4 Mt. This capacity is based on an assume tailings density of 1.7 t/m^3, which must be confirmed by field testing.

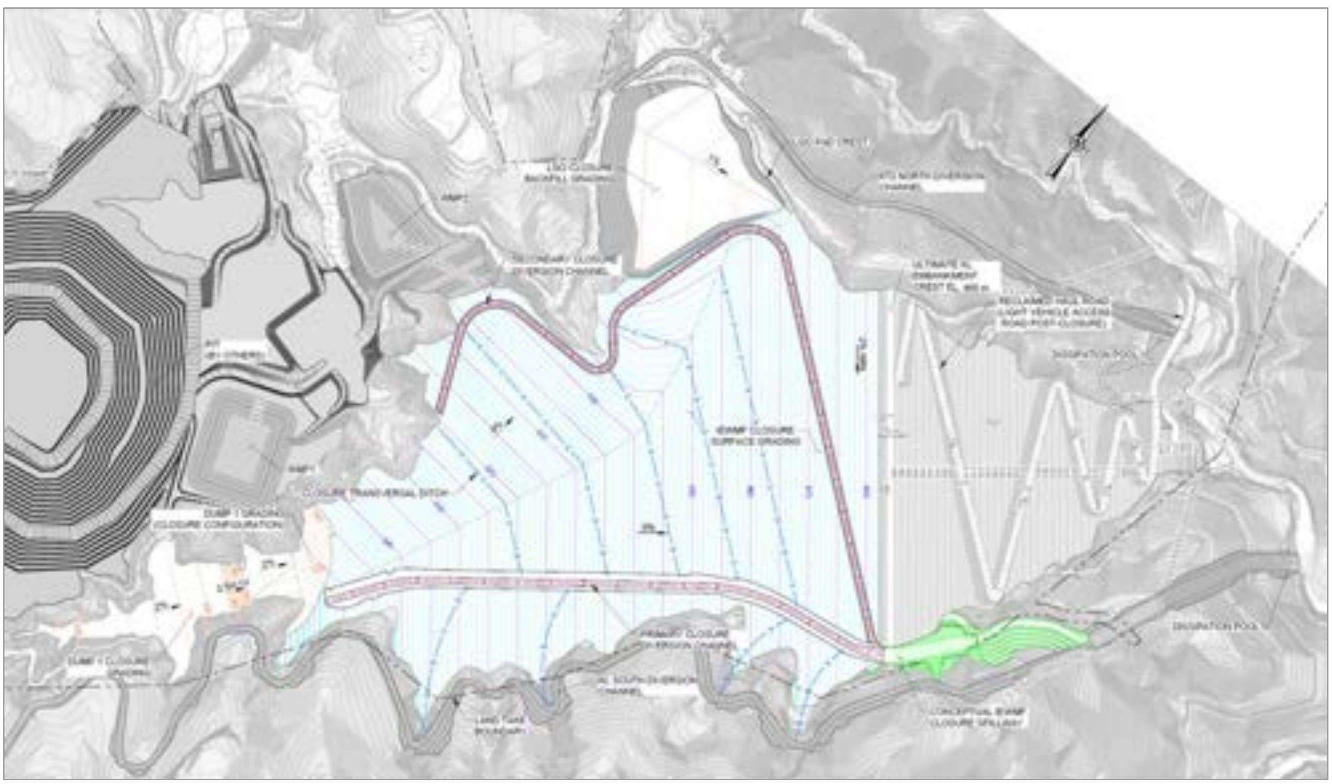
The initial KL embankment will be constructed as a cofferdam and starter dam, then raised in stages in advance of the tailings in Phase 1, with construction completed at the end of Year 8. The embankment will be constructed of primarily ROM waste rock and overburden materials as well as clay for the low permeable layers using the downstream construction method. Filters and drainage layers are sourced from off-site quarries. The KL embankment will have a maximum overall upstream slope of two horizontally to one vertically (2H:1V), with a downstream slope of 2.75H:1V selected for long-term geotechnical stability and closure. The KL embankment will have a maximum height of about 190 m. The ultimate crest width will be 15 m. The dam is protected by diversion channels and intermediate spillways during operation, and by a closure spillway.

The IEWMF is designed to store runoff from tributary areas and to safely store and pass direct precipitation on the facility resulting from the inflow design flood (IDF) storm event, with design allowances for wave run-up due to wind action. Spillways will be constructed on the south end of the KL embankment to protect the dam during an IDF. Spillways will protect each stage of the dam.

To support construction-level design and permitting, a detailed geotechnical monitoring plan will be prepared that defines the roles and responsibilities of key stakeholders (owner, operator, engineer) for safe and stable IEWMF construction and operation. Monitoring will be accomplished through both measurements of monitoring points (e.g., survey monuments, piezometers readings), and visual observations of surface conditions.

The IEWMF closure plan is shown in Figure 18.5. The IEWMF tailings will be covered with 2.5 m of waste rock and 0.5 m of topsoil. Capping of the IEWMF will be progressive starting in Year 9 of Phase 1 on areas where the tailings have reached final elevation.

Figure 18.5 IEWMF closure configuration (approximate end of Phase 1)



Source: Golder 2022.

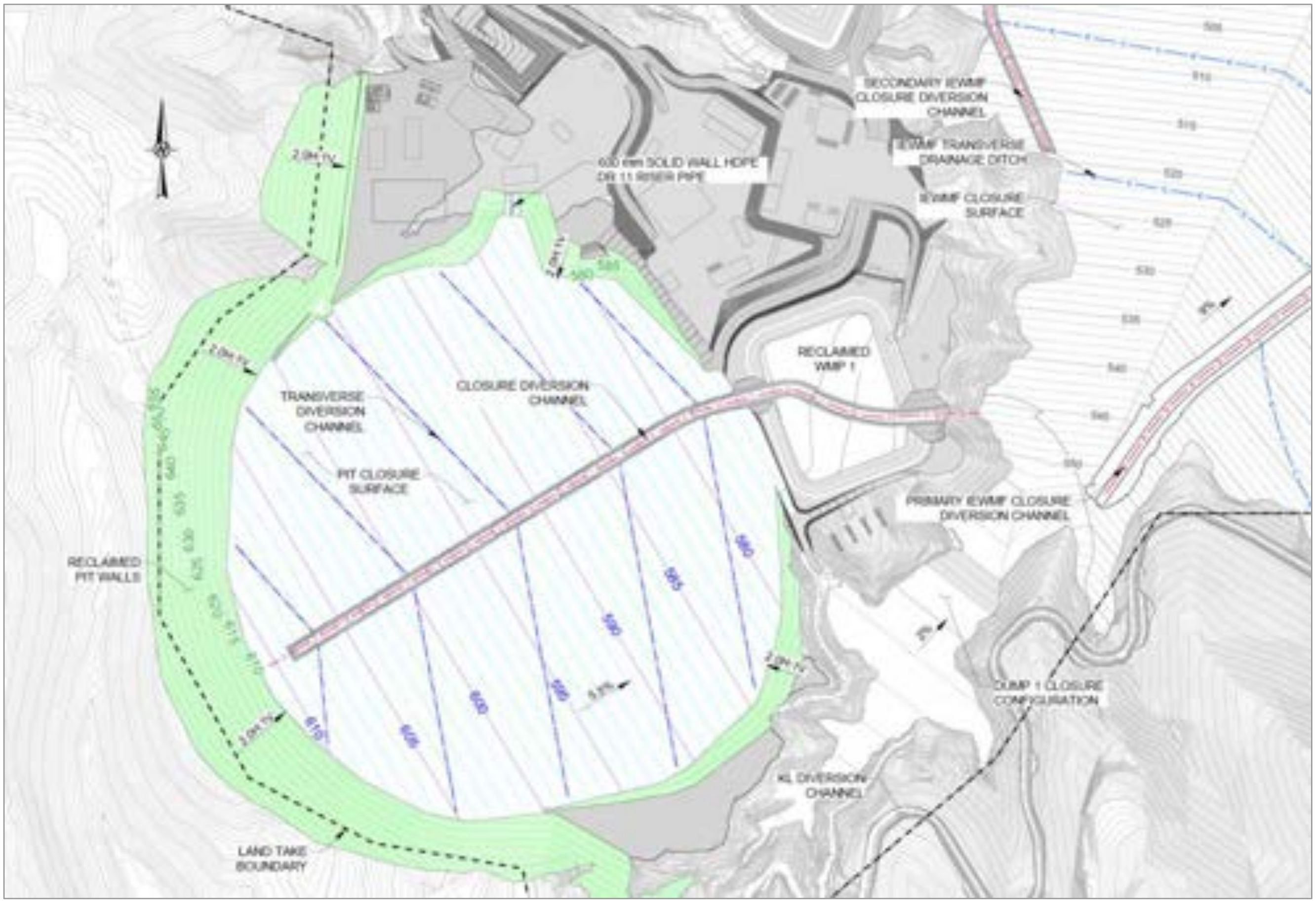
18.3.2 Phase 2 – In-pit tailings disposal

Ore processing from the underground mining operations during Phase 2 will generate approximately 72.5 Mt of tailings, of which 42.9 Mt will be backfilled into the mined-out open pit as a filter stack. The remaining 29.6 Mt will be used as paste backfill for underground mining. Filtered tailings will be transported via a series of overland conveyors along the haul road from the filter plant to the pit for disposal. The tailings will be deposited in the open pit and spread and compacted, progressing across the open pit from the haul ramp. Overland conveyors, grasshoppers, and support equipment used during Phase 1 in the IEWMF will be utilized during Phase 2 where practical.

Groundwater inflows to the open pit will be collected using vertical wet wells, a base underdrain and wet well sump, and an intermediate underdrain layer that will direct all inflows to the wet wells. Contact water will be managed in WMP1 and WMP2 during Phase 2.

The Phase 2 closure plan is shown on Figure 18.6. The open pit tailings surface will be closed and reclaimed to meet environmental commitments.

Figure 18.6 Phase 2 final closure configuration



Source: Golder 2022.

18.4 Water management

Surface water and groundwater within the project site can be classified into two categories, non-contact water and contact water.

Non-contact water is groundwater and surface water that is diverted around the mine facilities without being exposed to mine infrastructure. The intercepted non-contact water will be used to supplement water supply to processing operations and to provide dust suppression, with excess discharged to the environment.

Contact water includes groundwater and surface water that has been exposed to mine infrastructure, and the process water. Contact water is classified as either "potentially acidic contact water", which includes groundwater and surface water that has been exposed to materials characterized as "potentially acid generating" (PAG), or "non-acidic contact water", which includes groundwater and surface water that has been exposed to materials characterized as "non-potentially acid generating" (non-PAG).

The main objectives of the water management plan include the following:

- Implement key water management infrastructure to facilitate mining activities (dewatering for open pit and underground mining).
- Minimize non-contact water from entering mine facilities and construction areas, to a practical extent, by implementing diversion systems and installation of dewatering wells.
- Maximize reuse of contact and non-contact water and provide a continuous supply for mine processing operations.
- Manage excess contact water in an environmentally sound way and within the framework of the environmental permit for the Project.
- Provide a tool for effective site-wide, holistic, and proactive water management planning.

18.4.1 Non-contact water management systems

Non-contact water from undisturbed catchments around the mine facilities will be collected through a series of passive diversion systems. The diversion systems will include the KL south diversion channel and KT2 diversion system, that will include the existing channels around the processing facilities area that will be extended to discharge to the KT2 diversion system extension. The non-contact water collected by these systems will be conveyed downstream of the project site to Karolakkos Creek. The concrete lined diversion channels will have a base width ranging from 2 m to 5 m, and a depth ranging from 1 m to 1.5 m to safely convey the estimated peak flows during storm events with a return period of up to 50 years. The discharge to Karolakkos Creek will occur through engineered discharge structures for erosion control.

The mine dewatering and depressurization system will be comprised of a network of surface dewatering wells around the perimeter of the open pit and dewatering / depressurization wells drilled from the underground mine workings. The main objective of this system is to intercept groundwater in the proposed mine development area before it reaches the open pit and the underground workings. The non-contact water from the dewatering well network will be used for processing and dust suppression (mainly to supply mobile water trucks servicing the service / haul roads). The excess non-contact water will be reinjected to the groundwater system through wells located in the upper Lotsaniko Valley. Pumping and pipelines systems will be used to convey non-contact water from the source to the end users or to the reinjection field.

18.4.2 Contact water management system

The contact water will be collected and used for mine processing operations and dust suppression (on areas with downstream water collection). Excess contact water will be managed through evaporators, and through treatment and disposal. Treated contact water will be reinjected to the groundwater system through wells located along site access road to the south of the Project site. Disposal of treated contact water to the surface water system near the project site will occur in emergency conditions only to reduce the risk associated with water accumulation at the IEWMF.

Methods for disposal of treated contact water have been developed in accordance with the environmental permitting requirements. Additionally, Hellas Gold supports the development of community projects that may benefit from receiving non-contact and / or treated water from the project site. This would be possible only when and where these projects will be underway.

Contact water within the project site will be collected at source and centrally managed in WMP2. The main sources of contact water at the project site include runoff and seepage from the IEWMF, runoff and seepage to the open pit and underground development, and seepage from the LGO stockpile. Contact water in WMP2 will be mainly used for process water and dust suppression. Dust suppression will consist of two types: the pumped / piped sprinkler system for the IEWMF area and open pit, and mobile water trucks servicing the site roads.

A series of pumping and pipeline systems will be used to transfer contact water between various collection / storage facilities and contact water users through the project site. Most of the transfer pipelines will be HDPE pipes; the remainder will be made of carbon steel to accommodate the pumping pressure requirements.

The pumping and pipeline systems will be designed to support the overall water management strategy. The contact water will be collected, stored, and distributed to various end users to ensure there is no cross-contamination with any non-contact water systems.

18.4.3 Site wide water balance

A site wide water balance model (WBM) was developed for the project using Goldsim™ modelling software to simulate water transfer throughout the entire mine operations.

The primary objectives of the water balance are as follows:

- Estimate the availability of contact and non-contact water for process supply.
- Estimate the required on-site storage volumes for contact water.
- Assess water treatment requirements.
- Estimate the amount of excess non-contact and contact water requiring disposal.
- Optimize water management infrastructure of the mine site (including pumping rates and pond storage levels).

The general results of the WBM show that available site water will be sufficient to supply the mine. Excess non-contact and contact water is expected to be generated (especially during Phase 1) and will need to be managed (see Sections 18.4.1 and 18.4.2 for description of management of excess non-contact and contact water, respectively). During Phase 2, because of a reduction in underground mine dewatering rates and reduction in surface contact water runoff due to reclamation of the IEWMF, the amount of excess non-contact and contact water is expected to significantly reduce.

The water storage facilities at site (mainly WMP1 and WMP2) will be used to manage seasonal variability of contact water.

The results of the WBM show that disposal of treated contact water through injection wells would be require for approximately three months each year (especially during Phase 1). The discharge of treated contact water to the surface waterbodies (Karolakkos Creek, through KT2 diversion system extension) is expected to be required only in the occurrence of extreme rainfall events.

18.4.4 Water quality and treatment

Preliminary geochemical characterization of waste rock, ore, and tailings was completed in 2014 and 2015 using standard static and kinetic geochemical testing methods (GAUK, 2014 and 2015). The samples chosen for this study were considered representative of the waste rock and tailings materials associated with the project and mine plan, waste management strategy, and water management concept at the time. Subsequent water quality predictions were completed for the project to support a pre-feasibility study in 2017 (GAL, 2017). Conservative mass loading and drainage chemistry predictions were developed assuming aggressive leach test results were representative of drainage chemistry and that the source terms are constant (e.g., not depleting). These drainage predictions were then used to calculate conservative mass loading in the water management concept, including the inflow water quality to the water treatment plant (WTP) (GAL, 2017). A geochemical data gap analysis was conducted in 2021 that indicated specific geochemistry and water quality data gaps related to regulatory requirements and requirements to support the design of the IEWMF (GAL, 2021). Recommendations to address the geochemistry and water quality data gaps were provided and are subject to ongoing investigations.

The contaminants of concern in the contact water include total suspended solids, arsenic, chromium (VI), copper, lead, molybdenum, nickel, and selenium. A WTP design (GAUSA, 2021) was completed based on the outcome of the SWWB assessment on the current understanding for treatment requirements to include two parallel trains at 70 m^3/h and 180 m^3/h and to provide the flexibility to provide treatment for a range of expected flows. All contact water sources will be directed to the WTP through a series of collection ponds, including WMP1, WMP2, and two process water storage ponds. Treated water will normally be discharged to groundwater via reinjection wells; however, WTP capacity to treat to surface water quality standards, also allows for emergency discharge of treated water at flow rates up to 250 m^3/h to surface water. The treatment trains include membrane treatment (ultrafiltration and reverse osmosis), high-pH and low-pH chemical precipitation, clarification, ultrafiltration, anaerobic biological treatment (selenium removal) and aerobic biological treatment for effluent polishing. Treated brine is recombined with the permeate from the reverse osmosis system for discharge.

18.4.5 Reinjection wells

Dewatering will be required to support the development and operation of the open pit and the underground mine. A numerical groundwater flow model developed during earlier stages of design for the Project was used to provide estimates of groundwater inflows to the open pit and underground workings throughout the LOM in support of the WBM (see Section 18.4.3).

The excess non-contact water will be reinjected in the bedrock aquifer through injection wells located in the upper Lotsaniko Valley. Reinjection of excess non-contact water has been modelled based on the seven currently operating and eight proposed reinjection wells due to be commissioned prior to commencement of operations. Based on a preliminary evaluation of maximum allowable re-injection per well, the completed array will have a capacity of approximately 3,700 m^3/day, using a reinjection rate per well of 10 m^3/h.

During normal operations, when contact water volumes on site start to accumulate beyond the capacity of site demand and other water disposal methods, excess contact water is to be treated and reinjected at a new well array to be constructed along the site access road to the south of the site. Reinjection of treated contact water has been modelled based on a minimum of seven wells having the same yields as the operational reinjection wells at the site, resulting in a capacity matching the normal WTP operational rate (1,680 m^3/day). The model assumes that all seven wells are available at the beginning of mining operations. However, it is expected that the wells will be constructed sequentially during the first months / years of operations, which activities will also be used to confirm the actual number and location of the wells.

18.5 Transportation and logistics

The Project is well situated to take advantage of Greece’s modern transportation network for shipment of construction and operations freight.

The main access road connects the process plant and mining area with the national road network. The access road follows the alignment of an existing forestry road and will be upgraded to an asphalt-paved, 7 km long, 7.5 m wide, bidirectional road with adequate drainage to allow all season use.

The Project is 4 km from the village of Palaiochori, with access by a two-lane highway. From Palaiochori, the major regional centre of Thessaloniki is approximately 80 km away via highway EO 16. Thessaloniki has an international airport and one of Greece's largest seaports. Thessaloniki is linked to the rest of Greece by Greece's National Roadway, which has been extensively modernized in the last 20 years. Access to Europe and Turkey is provided by highway and rail infrastructure.

The port of Thessaloniki is one of the busiest Greek seaports and one of the largest ports in the Aegean Sea basin, with a total annual traffic capacity of 16 Mt of dry bulk and liquid bulk cargo. The cargo terminal has a total storage area of one million square metres and specializes in the handling of a wide range of bulk cargo. The recently expanded and modernized container port is the second largest container port in Greece and has a large oil and gas terminal and one of the largest passenger terminals in the Aegean Sea.

The port of Thessaloniki has served the Project during construction to date and will continue to do so for receipt of freight for remaining construction and during operation for shipment of operating consumables to the Project site. The port at Stratoni will be upgraded as part of the Olympias upgrade project and will serve the Project once commissioned, receiving concentrate from the Skouries flotation plant via the national road network.

18.6 Power supply

The high voltage substation is located in the NW corner of the Skouries plant site. The substation has a power capacity of 51 MW. The incoming overhead line voltage of 150 kV is stepped down via two power transformers rated at 50 MVA. The substation provides all power distribution to the site at 20 kV.

For the permanent power supply, the entire underground mining electrical power will be fed from the 20 kV substation located at the filter plant. The power is then delivered to the 10 MW underground mine main substation located near the mine portal. The underground medium voltage distribution voltage will be 20 kV.

Hellas Gold signed an agreement with the Independent Electricity Transmission Operation for Greece (ADMIE) in 2015 that sets out the terms and conditions for connecting to the Greek power grid.

Hellas Gold is obliged to construct all the system extension works required to connect the substation from the boundary of the Skouries facilities to the existing Greek power grid, including the 6 km long 150 kV overhead connecting transmission line that connects the site substation to the Stagira and Nikiti 150 kV transmission line. Greek legislation states that Hellas Gold is obliged to transfer the ownership and the possession of the system extension works (150 kV line) to ADMIE at the completion of commissioning.

An emergency diesel generator rated at 1,700 kVA will be connected to the essential services LV switchboard to supply critical drives and equipment in the event of power failure at the main process plant. Emergency lighting will also be fed from the emergency diesel generators, as well as 1.5 hours internal battery lighting for instantaneous emergency lighting upon a power failure.

Additionally, a 1,700 kVA emergency diesel generator will be located at the thickener tanks; other generators will be a 110 kVA at the primary crusher, a 1,700 kVA at the filter plant and a 450 kVA for the supply of the potable water package and administration buildings. There is also an emergency generator for the Hoist that will be operational in Phase 2.

18.7 Explosives magazines

For surface consumption, explosives are delivered to site and stored. For underground consumption, explosives storage facilities are located to the south-east of the 230 L mine infrastructure area in a dedicated access drift, leading directly to a RAR. The underground storage facilities comprise: two emulsion bays, a cartridge magazine, a detonator magazine, a truck turnaround, and a stub for the Phase 2 ventilation breakthrough.

18.8 Fuel and lube supply

The surface fuel storage arrangement includes three separate tanks, each with a capacity of 20,000 L. Each tank receives fuel from a small site transfer tanker of approximately 10,000 L capacity, which will deliver fuel daily as required. There are two tanker truck transfer locations on opposite sides of the fuel storage station. Tanker truck transfer locations will have a bunded concrete slab on grade to prevent spills from escaping into the environment. Delivery tankers will park there. Surface trucks will also park there to be fuelled from the storage tanks. There is an oil / water separator and sump in the bunded area to reduce oil content in collected water. All contaminated water will be collected by a suction tanker and taken for treatment off-site. Similarly, separated oil emulsion will be taken away for treatment and recovery.

The underground mine fuel delivery system consists of two main parts – surface and underground. The surface system includes storage tanks that receive diesel fuel from site delivery tanker trucks, and the pumps and control systems to pump the fuel, via a buried fuel line, to the transfer tank at the head of the borehole that connects to the underground fuel bay on 170 Level of the mine.

18.9 Communications system

The communication system comprises fiber optic backbone and wireless communications, and tele-remote production operations. Fixed plant will be automated, and ventilation controls and fans will be remotely controlled from the surface control room. Wireless communication infrastructure will be established throughout the mine levels and for manually driven vehicles up the twin ramps.

The fixed plant automation design concept provides a central control base on surface, with modular expansion as the mine grows and develops via a series of PLC control systems, deployed based on process type and geographical zone for ventilation and dewatering infrastructure.

In parallel with the installation of the fiber optic hard-wired network, a fiber optic wireless network will be installed throughout the mine to enable Wi-Fi wireless tracking and tagging of personnel and vehicles via wireless nodes; VoIP wireless voice and text communications with personnel and vehicles and video cameras, gas and vibration monitoring will also be enabled via the fiber optic wireless network.

18.10 Water supply

Drinking water comes from boreholes, springs and a reservoir. The larger part of the local water supply network was reconstructed during the 1990s and serves almost the entire population of the area. At Skouries, a storage tank (1,000 m^3) is used for storage of utility / fresh water. The water comes mainly from the boreholes during the filling of the tank and then from the mine water clarifier. A storage tank (25 m^3) is used for storage of potable water. The water comes from the boreholes and is distributed via pumps and a main 100 mm pipeline. Maximized reuse of contact and non-contact water facilitates provision of a continuous supply for mine processing operations. A storage tank (850 m^3) is used for storage of firefighting water. The water is distributed where required via a main 254 mm pipeline.

18.11 Buildings

Site infrastructure is made up of the following facilities:

- A three-storey Mine Services Building (change house, production wickets / Technical Services)
- Surface warehouse
- Surface maintenance workshop
- Surface shotcrete plant
- Surface fuel storage facility
- Surface Central Control Room
- A concrete tower headframe (Phase 2)
- Electrical substation
- Paste backfill plant

18.12 Paste backfill plant

A paste backfill plant and distribution system has been designed for Skouries underground Phase 1 and Phase 2. The Skouries paste fill system will combine moist pressure-filtered tailings cake, thickened tailings slurry, and cementitious binder to produce 200 m^3/h (Phase 1) and 400 m^3/h (Phase 2) of paste fill at an average 70.5% solids by weight in a plant located on the eastern rim of the open pit. For further detail refer to Section 16.

19 Market studies and contracts

19.1 Markets

The Skouries process plant will produce a gold-copper concentrate that will be marketable to many upstream smelters and refiners. The Skouries Gold Project is one part of the Kassandra mines, which includes the nearby Olympias mine and the Mavres Petraes (Stratoni) mines. These three mines are permitted under the same environmental permit.

This report considers that all concentrates over the life of the Skouries Gold Project will be sold at competitive market rates to third parties.

19.2 Contracts

19.2.1 Construction contracts

Construction of the Skouries Gold Project commenced in 2012 and was placed into care and maintenance in 2017. The restart of the project will be executed using a standard engineering, procurement, and construction management (EPCM) methodology. Construction contracts are being tendered and awarded to qualified contractors by the Owner's construction management team.

19.2.2 Mining contracts

This technical report assumes several contracts will be in place for open pit and underground mining. In both cases, contractors will assist in implementing the initial portions of the development, while allowing for a transition to owner-operated mining as the project matures.

The open pit operation is designed to include a contractor for drilling, loading and hauling materials from the open pit and other excavation areas to the IEWMF to be placed on the KL Embankment or other construction embankments. This contractor will use small 60-tonne haulage trucks capable of navigating the steep terrain into the KL Valley. The contractor will continue to provide and operate the drills, major loading equipment and the ore and waste trucks through the end of Year 1. Beginning Year 2, only the waste trucks will continue to be on a contract and will be through the end of mine life in Year 9.

Four significant underground contracts support Phase 1 and Phase 2 operations described in the following subsections. These are considered to be reasonable by the mining QP.

19.2.2.1 Underground mining – preliminary works

The Preliminary Works Contract will cover development, mine services and material handling required to access and complete the test stoping program. Mine development will be halted during test stoping, and will resume in Year 1 under Contract A.

19.2.2.2 Underground mining - Contract A

Contract A is a development-only contract under which the contractor will provide all underground equipment, labour, supervision and training whilst Hellas will provide technical services and contract management. The contractor is assumed to charge Hellas for labour, supervision, equipment, materials handling, administration fees, profit, and mobilizations. Costs were built up from first principles. Contract A is envisioned to span 33 months of the project, beginning after development resumes in Year 1.

19.2.2.3 Underground mining - Contract B

Contract B supports the transition to production activities and owner-mining, including the initial training support for the owner-operated teams and will make use of the Owner's equipment, which will be commissioned prior to the start of the contract. The contract includes a number of expatriate and Greek trainers, Greek translators, and specialized vendor representatives to achieve training objectives. Contract B is scheduled to run for 12 months starting in Year 2.

19.2.2.4 Underground mining – Materials Handling System

The Phase 2 Material Handling System will be realized in Phase 1 through an EPCM contract with an experienced shaft sinking contractor. The scope includes engineering, construction of the shaft headframe, shaft sinking and equipping, and the excavation and construction of related infrastructure (ore bins, crusher, conveyor, loading pocket). From engineering to commissioning the project is scheduled over 5 years.

19.2.3 Concentrate sales contracts

No off-take agreements have been signed at the time of writing; however, several indicative non-binding, proposed term sheets have been received from European and global copper smelters. An analysis of the Skouries concentrate produced during the testwork campaigns indicates it is generally clean and will not incur any major penalties. It was also noted that the concentrates carry a palladium credit that has not been factored into the financial evaluation. A summary of the proposed term sheets is given below based on initial term sheets with major off-takers.

19.2.3.1 Copper terms

- Treatment costs: US$82.5 per dry tonne of concentrate.
- Refining costs: US$0.0825 per pound of refined copper.
- No price participation.
- Average copper payability of 96.2% assuming a minimum 26% concentrate grade.

The current custom smelting market indicates that the TC / RCs would be well below current assumptions, however these assumptions will be reassessed once Skouries is in production. The QP confirms these are the assumptions used for the economic analysis.

19.2.3.2 Gold terms

- Refining charge: US$6 per oz of gold produced.
- Gold payable in copper concentrate: 97.5%.

19.2.3.3 Transport costs

- Base case is transport of concentrate by truck to European smelters.
- Costs for concentrate transport range between US$14 and US$24 per tonne.

20 Environmental studies, permitting, and social or community impact

20.1 Environmental impact study

The EIS for the Kassandra Mines Mineral Deposits Project (Kassandra Project) includes an area of 26,400 ha, in north-eastern Halkidiki (Macedonia Region). The Kassandra Project includes the Skouries, Olympias and Stratoni sites. The Skouries Project covers approximately 255 ha of the Kassandra Project.

The EIS considers the potential impact on the local and regional environment as it relates to:

- Open pit and underground workings.
- Tailings impoundment.
- Process plant.
- Infrastructure necessary for the Project's operation.

ENVECO S.A. (Environmental Protection, Management and Economy consultants) under Hellas Gold's management, has authored the full EIS. The EIS was submitted in August 2010 and approved in July 2011. The EIS covers all environmental issues for the Project.

For the preparation of the full EIS, standards and directives required by the national and European Community legislation in force were used.

The full EIS was prepared principally by the application of:

- Laws 1650/86 (OGG 160A/18-10-86) for the Protection of the Environment, as this was amended by Law 3010/2002 (OGG 1427A/22-4-2002).
- Law 998/79 (OGG 289/29-12-1979) on the Protection of forests and in general forested areas of the Country.
- JMD 107017/06, which is the Greek implementation of the SEA Directive 2001/42/EC.
- Law 4014 (Government Gazette 209A/21.10.2011) "Environmental licensing of projects and activities, arbitrary regulation in connection with the creation of an environmental balance and other provisions of the Ministry of Environment", as amended by Article 2 of Law 4685/2020 (Government Gazette 92/7.5.2020) "Modernization of environmental legislation, incorporation into Greek legislation of Directives 2018/844 and 2019/692 of the European Parliament and of the Council and other provisions".
- Ministerial Decision. 37674/2016 (Government Gazette 2471B/10-8-2016) "Amendment and codification of the ministerial decision 1958/2012 - Classification of public and private projects and activities in categories and subcategories according to article 1 paragraph 4 of Law 4014/21.9.11 (Government Gazette 209/A/2011) as it has been amended and is in force".
- Ministerial Decision. 170225/2014, Specialization of the contents of the files of environmental licensing of projects and activities of Category AD of the decision of the Minister of Environment, Energy and Climate Change with no. 1958/2012 (21/B) as in force, according to article 11 of law 4014/2011 (209/A), as well as any other relevant details.

20.2 Baseline conditions

20.2.1 Introduction

Two baseline studies have been performed relating to Skouries, one in 1998 and one in 2010. The combination of these studies defines the ecological baseline conditions of the study area. A third baseline study was carried out in 2014 and a fourth study was carried out in 2017. Since the ecological studies reoccur every three years, a fifth study is currently ongoing covering 2021. For the purpose of defining baseline conditions the following definitions apply:

- Project footprint:
  - All surfaces taken by elements of the Project, as well as existing or future roads that will be used by the Project.
- Immediate Study Area:
  - A zone of 3 km around the Project footprint and 500 m on either side of any access roads within the Project footprint.
- Wider Study Area:
  - All municipalities that encompass the immediate study area, i.e., ex-municipalities of Arnaia, Megali Panagia and Stagira-Akanthos and current municipality of Aristotle.

The natural environment of Halkidiki district shows a significant diversity, mainly attributed to its complex geomorphology, as further described below.

20.2.2 Climatic and seismic

The climate varies between the continental climate of Central Europe and the Mediterranean climate. The biggest part of this area belongs to the weak mid-Mediterranean bioclimatic type.

The average annual precipitation determined from average monthly precipitation as recorded by Olympias (2006-2021) and Stratoni stations (2009-2021) is 728.1 mm and 628.4 mm respectively, and the average annual temperature is 16 degrees Celsius. Relative humidity has a mean of 76% and the evaporation mean is 564 mm.

No specific dominant wind direction is identified at the Skouries Station; however, 12.7% of winds blow east-north-east and 10.4% of winds blow north-east. North-north-west is the least common wind direction for Skouries, with it blowing in this direction less than 1% of the time. Wind speeds range from less than 0.5 m/s (Category 1) to 5.5 m/s to 7.9 m/s (Category 4). The dominant wind speed is 1.5 m/s to 3.3 m/s (Category 2), with 65.1% of wind being between these speeds.

The area under study is classified to Seismic Zone II.

20.2.3 Morphology

Around the higher ground that hosts the Skouries deposit is a sub-mountainous area with dense vegetation where the nearest settlements are situated, namely Megali Panagia, Palaeohori, and Neohori. Despite long-lasting human activity in the area, the landscape in the immediate study area does not show signs of intense deterioration.

20.2.4 Soil

Soils in the immediate study area are deep (>30 cm) with weak man-made influences and no signs of erosion. The soil is characterized as low bulk density with a loamy-sandy texture. The soil composition is considered excellent for use for rehabilitation of disturbed areas and providing a good basis for revegetation.

Generally, there is a lack of regulated limits by the Greek legislation however, due to the soil geochemical background, elevated concentrations of Pb, Zn, Cu, As, Cd, Mn can be found.

Sampling and analysis of soils over the wider study occurs annually. Areas suggest natural concentrations within the boundaries of the geochemical background of the area. As there has been no significant historic mining at the Skouries site, the presence of elevated concentrations of some metals suggests natural pollution that can be used as a baseline condition.

20.2.5 Flora and fauna

From the extensive baseline studies carried out over the Project area, the species of flora and fauna are well known.

The forest ecosystems cover almost the entirety of the immediate Project area, showing high density tree growth and vegetation diversity. Because of the size of the area and the non-intensive man-made pressure, forest ecosystems are ideal for sustaining fauna species. The potential regeneration of the forest ecosystems is considered to be very good after any anthropogenic pressure, which will greatly assist with rehabilitation.

20.2.6 Aquatic

The most significant watercourse in the immediate area is Asprolakkas, belonging to the Asprolakkas basin that encompasses the whole of the Skouries study area. All watercourses in the Project area exhibit dense vegetation. Analysis of watercourses in the Project area show only an exceedance of Se.

Analysis of groundwater in the area shows a natural exceedance of Sb, Pb, Mn, Fe, and As.

The main pressures on water quality within the wider region come from uncontrolled solid waste disposal sites, municipal wastewater treatment plants, and historic mine water treatment. There are no historic mine discharges in the immediate Skouries area. Secondary pollution sources are agricultural discharges (P and N loads) and stock breeding waste. In the study area, there are five uncontrolled solid waste disposal sites but no regulated landfill site that could also impact water quality.

Drinking water comes from boreholes, springs and a reservoir. The larger part of the water supply network was reconstructed during the 1990s and serves almost the entire population of the area.

20.3 Permitting

In July 2011, the MOE formally approved the EIS submitted by Hellas Gold for the three Kassandra Mines mine sites, being Olympias, Skouries and Stratoni, which involves an area of 26,400 ha, in northeastern Halkidiki (Macedonia Region). This EIS that covers all environmental matters for the Kassandra Mines was expected to expire in July 2021. However, due to the new environmental law (4685/2020), it will be extended for five years and then another four years because of Eldorado's ISO certificated Environmental Management System. This means it will now expire in 2030, provided all approvals from the MOE are received. A new EIA was submitted in 2021 in accordance with the new Business Plan.

For construction to commence and continue in a timely manner as well as production to commence, according to the mining law, a submission of a technical study is required. This was submitted and, in early 2012, the technical study was approved by the MOE. This study, as required from its approval terms, was supplemented by specific technical studies for the flotation plant, approved by MOE on 12 April 2013, and for the Karatzas & Lotsaniko TMF, approved by the MOE on

17 September 2014. In addition, information relating to the “auxiliary temporary facilities” was submitted and finally approved by the MOE on 16 January 2014.

An installation permit for the flotation plant was granted on 13 May 2013; this was extended twice, on 9 December 2014 and on 11 November 2016.

An installation permit for the auxiliary temporary facilities was granted on 24 March 2016. This was recently incorporated into the technical study by the MOE approval decision on 3 September 2019 in compliance with the new mining law (4442/16 & 4512/2018).

An updated technical study covering amended aspects of the process plant and associated infrastructure was submitted to the MOE in December 2015, and this was approved in May 2016. Subsequently, an updated specific technical study for the flotation plant was submitted to the MOE and approved on 11 November 2016. An update of the installation permit for the flotation plant was submitted by August 2016 and this was approved on 3 September 2019.

Permitting activities took place with the granting of the Skouries Process Plant building permit in February 2016. For this building permit, two minor updates took place in October and December 2019. This building permit allows buildings to be constructed over the process plant. In compliance with the revised technical study and the relevant installation permit, an update of the existing building permit and a new one for the other miscellaneous buildings were also required.

The relevant studies were submitted by 5 May 2020, and the first stage of the building permit granted by 15 June 2020. The second stage of this permit is in progress with uploading of supporting drawings and reports.

For dry-stack tailings in the TMF, an EIA modification folder was submitted by 12 October 2020 and granted 29 April 2021. A revised technical study, installation permit and the associated building permit are still required for the TMF and the respective filtration plant.

The Skouries Project was moved towards care and maintenance in late 2017 due to the non-issuance of permits. These permits included the Skouries electro-mechanical installation permit for the revised Technical Study. This was finally approved in September 2019. Another important pending issue that led to the decision for care and maintenance was the approval decision regarding the antiquities in the open pit. While this is not strictly a permit, it was required in order for the operations to commence. This approval decision was finally issued on 27 November 2019.

The NI 43-101 Technical Report published in March 2018 changed the view over the tailings deposition method to filtered tailings. This also required a modification to the existing permits. A relevant study was submitted to the mining department of the MOE in March 2018, but returned in August 2018, and in consequence of that, and in order to comply with current environmental legislation, a modification folder of the EIA was submitted on 12 October 2020.

Given the changes in the engineering, additional technical studies and permits are required to be granted in a timely manner to allow construction to move forward. Construction is expected to take approximately 2.5 years from commencement. Since 2012, the MOE and other agencies have not fulfilled their legislated permitting and licensing obligations. During 2015, the MOE revoked certain permits of Hellas Gold. This action was canceled by the Council of State but it has caused a negative impact on the schedule and budget to develop the asset.

20.4 Social and infrastructure

20.4.1 Economic and social environment

20.4.1.1 Demographics

The population of the study area has been decreasing, partly due to reduction in mining activities and lack of development. Another important characteristic is that the population is getting older, as the proportion of middle aged and elderly people is higher than those of Halkidiki and Greece.

20.4.1.2 Employment - unemployment

The study area lacks in development compared to the rest of the Halkidiki prefecture. Unemployment rates in the region have been increasing.

20.4.2 Land use

Agricultural activity in the wider area is of a small scale. Wheat represents the majority of the cultivation and grasslands are found mainly between residential areas and forests. Forest systems create the necessary conditions for the development of apiculture, which is of traditional significance to the area.

20.4.3 Sewage disposal

The majority of the areas lack wastewater management plants, and a substantial quantity of wastewater ends up in the rivers and streams of the study area. In Olympiada, as well as in Stratoni / Stratoniki, the construction of a state-of-the-art biological station as well as a sewerage system has been completed, while the construction of the project "Collection, transport, treatment and disposal of Arnaia - Paleochori Aristotelis Municipality" is in progress. In the future, the same project will be able to integrate the wastewater management of the settlement of Neochori, once the construction of a sewerage network is completed.

20.4.4 Solid waste management

In the Chalkidiki prefecture, there is a landfill in operation (in Kassandra municipality). In addition, there is a recycling program currently in place in the study area (municipality of Aristoteles).

20.4.5 Social infrastructure

Chalkidiki's medical care is mainly served by the Polygyros General Hospital and five health centres. One of them (Health Centre of Palaeohori) is situated within the study area. Sports infrastructure is reasonable and cultural associations are only present on a local level.

20.4.6 Historical and cultural

The Ministry of Culture has performed archaeological investigations and identified two archaeological sites on the Skouries Project: in the centre of the open pit and in the area of the mine water storage ponds in the mill area. The archaeological site located in the centre of the pit was relocated in 2020 to a location adjacent to the second archaeological site near the mine water ponds in the mill area.

20.5 Environmental impacts and mitigation

The aim of this Project is to have a positive balance of benefits versus environmental impacts, being in line with the principles of sustainable development. To reach this objective:

- Modern environmental protection technology will be used.
- The natural environment land take will be minimized.

- Project design will be implemented so as to minimize impacts on the environment and, at the same time, to mitigate past impacts.

20.5.1 Impact analysis

An impact analysis has been undertaken on each environmental parameter, using three basic criteria:

- Sensitivity / Susceptibility: Impacts are classified as: Negligible, Low, Medium, and High.
- Vulnerability / Value of the Recipient: Impacts are classified as: Negligible, Low, Medium, and High.
- Impact Size: Impacts are classified as: Negligible, Low, Medium, and High.

In considering impacts, the phases of the Project have been grouped as follows:

- Development Phase
- Operational Phases
- Rehabilitation Phase

With these criteria and evaluation of the importance of impacts, changes in man-made pressures on the environment were determined, with the impacts classified as: Insignificant, Low, Medium, and High.

The scale of the change in man-made impacts was classified into the following classifications:

- Nature of impacts
- Impact intensity
- Complexity of impacts
- Geographical reference level
- Likelihood of occurrence
- Temporal aspect – duration of impacts
- Ability to address through physical process (reversible, partially reversible, irreversible)
- Ability to address through artificial means (addressable, partially addressable, non-addressable)

On the basis of the impact analysis, various mitigation plans have been formulated and continue to be developed with all relevant stakeholders within the certified environmental management plan in accordance with ISO 14001. Impacts and mitigation plans are summarized below.

The results of the impacts assessment(s) identified no known environmental issues which could materially impact the ability to extract mineral resources.

20.5.2 Impacts on the climate and bioclimatic features

The climate in the Skouries spatial unit and in the wider study area is characterized as a transition between the continental climate of Central Europe and the Mediterranean climate. The majority of the area belongs to the weak to strong Mediterranean type of bioclimate, while part of the area belongs to the sub-Mediterranean type. As far as the current state of the environment is concerned, impact evaluation of the condition of the climate and bioclimatic features of the Skouries spatial unit is considered to be Negligible in terms of their sensitivity / susceptibility and vulnerability / value.

As far as the impacts on climate and bioclimatic features due to the occupation of natural / forested areas, hot or cold gas emissions, and greenhouse gas emissions, an estimate was made of the impact per project operating phase within the spatial unit. It was found that the worst operating period for the project in terms of the impacts on climate and bioclimatic features in the spatial unit is the development phase and operating Phase 1, which is the phase of surface exploitation. When the surface mine ceases to operate, the impacts are limited. The data in the impact assessment is set out in relation to the worst operating period, namely the development phase and operating Phase 1. The evaluation of impacts on climate and bioclimatic features is shown in Table 20.1.

Table 20.1 Evaluation of impacts on climate and bioclimatic features

Item evaluated	Impacts from occupation of natural / forested areas	Impacts from the creation of bodies of water	Impacts due to hot or cold gas emissions	Effects of greenhouse gas emissions
Nature of impacts	Negative	Negative	Neutral	Negative
Impact intensity	Medium	Low	-	Medium
Impact complexity	Direct	Direct	-	Direct
Geographical reference level	Local	Local	-	Wider area
Likelihood of occurrence	Certain	Certain	-	Certain
Temporal nature	Long-term	Long-term	-	Long-term
Ability to address through physical processes	Irreversible	Irreversible	-	Irreversible
Ability to address through artificial means	Partially addressable	Partially addressable	-	Partially addressable
Impact Size	Medium	Low	Negligible	Medium
Importance of impacts	Insignificant	Insignificant	Insignificant	Insignificant

20.5.3 Impacts on the morphological and landscape features

The worst operating periods for the project in terms of impacts on landscape and morphological features are the development and operating phases. As far as morphological features are concerned, the potential impact is related to occupied land to site the new investment plan projects covering a total area of 255.3 ha, which will include the mine facilities, flotation plant facilities, electricity facilities, the Karatzas Lakkos IEWMF disposal facility, and the soil disposal facility.

There will be an increase in the occupation of surface facilities compared to the existing terrain. Extensive morphological changes are expected in the area where the pit is being excavated. A dam will be created for the IEWMF at the Karatzas Lakkos stream with a total area of 119.5 ha, which will be reclaimed once the open pit mining is completed. The development of platforms to site surface facilities is required.

As far as the landscape features are concerned, there is no village nearby from which the projects being examined will be visible. It should be noted that the area where the works are sited passes through the 'Halkidiki Landscape' Zone of International Value, which has been evaluated as important for its aesthetic and natural beauty, its representativeness, readability, and uniqueness. In addition, the National Value Zone 'Wine Routes' is also located within the area where the projects are sited and is important at a national level for its unaltered nature and the already-recognized features. The evaluation of impacts on the morphological and landscape features is shown in Table 20.2.

Table 20.2 Evaluation of impacts on the morphological and landscape features

Item evaluated	Impact on morphological features	Impact on landscape features
Nature of impacts	Negative	Negative
Impact intensity	Medium	Medium
Impact complexity	Direct	Direct
Geographical reference level	Local	Local
Likelihood of occurrence	Certain	Certain
Temporal nature	Long-term	Long-term
Ability to address through physical processes	Irreversible	Irreversible
Ability to address through artificial means	Partially addressable	Partially addressable
Impact Size	Medium	Medium
Importance of impacts	Medium	Medium

20.5.4 Impacts on the geological, tectonic and pedological features

The Skouries spatial unit is primarily structured in geological terms of metamorphic crystalline-schist Paleozoic rocks and newer igneous infiltrations from the Tertiary, and belongs to the Serbo-Macedonian mass and, in particular, to the overlying Vertiskos Formation. The geological features of the study area do not have any special attribute that could give some significant geological value. Consequently, the impact of the condition of the geological, features of the Olympias spatial unit is considered to be Negligible in terms of their sensitivity / susceptibility and vulnerability / value.

In terms of tectonics, the Mesozoic carbonate rocks found in the Vertiskos formation and in the area of Skouries have been tectonically placed in the area via alpine orogenesis. It is considered that their sensitivity is medium and the same applies to their resilience to any man-made interventions. The value of the tectonic features of the study area does not have any special attribute. Consequently, the impact of the condition of the tectonic features of the Olympias spatial unit is considered to be Medium in terms of their sensitivity / susceptibility and vulnerability / value.

In terms of soil characteristics, the Skouries area belongs to the semi-mountainous zone. The soil has a small thickness (typically 0 m to 2 m) and is formed by the mechanical, chemical and organic weathering of solid rocks in the terrain of the relevant area. The area has a high soil geochemical background in terms of potential polluting elements and in particular Pb, Zn, Cu, As, Cd, and Mn associated with the metalliferous deposits in the area. The soils have high natural background values for quite a few metals. Consequently, the impact of the condition of the pedological features of the Skouries spatial unit is considered to be Medium in terms of their sensitivity / susceptibility and vulnerability / value.

The proposed development of the Skouries mine is expected to remove around 147 Mt of ore. Given that the quantities of explosives used in each blast are in accordance with national and international regulations, explosives cannot cause large-scale earthquake vibrations that could potentially have an impact on structures. According to the special seismic study carried out in the area of the Karatzas Lakkos IEWMF, it is clear that the faults encountered are inactive and there is no active fault. The evaluation of impacts on the geological, tectonic and pedological features is shown in Table 20.3.

Table 20.3 Evaluation of impacts on the geological, tectonic and pedological features

Item evaluated	Impact on geological features	Impact on tectonic features	Impact on soil due to occupation	Impact on soil due to run-off	Impact on soil due to dust
Nature of impacts	Negative	Neutral	Negative	Negative	Negative
Impact intensity	Negligible	-	Medium	Low	Low
Impact complexity	Direct	-	Direct	Direct	Direct
Geographical reference level	Local	-	Local	Local	Local
Likelihood of occurrence	Certain	-	Certain	Certain	Certain
Temporal nature	Long-term	-	Long-term	Long-term	Long-term
Ability to address through physical processes	Irreversible	-	Irreversible	Irreversible	Partially reversible
Ability to address through artificial means	Partially addressable	-	Partially addressable	Partially addressable	Partially addressable
Impact Size	Negligible	Negligible	Medium	Low	Low
Importance of impacts	Insignificant	Insignificant	Medium	Medium	Medium

20.5.5 Impact on biodiversity

Skouries is a new mine, where the pre-existing state of the natural environment was characterized by a forest undisturbed by man-made interventions - a unique dense forest with old trees, primarily oak and beech, suitable for many species of fauna to nest in. There are a total of 160 species of plants in the area of which 17 are characterized as ecologically very important. Two species of birds, the Golden Eagle and Black Stork, belonging to the Endangered category in the Red Book of threatened animals in Greece, were noted in the area.

As far as the impacts on the vegetation and ecosystem categories are concerned, the worst operating period is the project development and the open pit operation phase, where the overall occupation will be around 255.3 ha. The majority of the vegetation which will be occupied by the facility is characterized as agricultural crop land (14.2%). The percentage of beech forest (6.1%) and oak forest (1.7%) to be occupied is very significant.

The impacts on birdlife and other species of fauna from the occupation of the natural environment, constant human presence, noise from extractive activities and frequent movements of heavy wheeled vehicles are considered very significant.

A special area for conservation named Mt. Holomontas is located in the wider Skouries area GR1270001. The project does not coincide with that special area for conservation and is not expected to harm the integrity and cohesion of Natura 2000 areas in terms of habitats and flora and fauna species. Despite that, since the presence of all large mammals and birds extends over long distances, they are very likely to be disturbed by mining activities and the sudden change in the natural ecosystem.

It should be noted that there are 112.39 ha of the Skouries – Stratoni area within the wildlife reserve K129 known as Skouries – Kasteli – Kakkavos. The Skouries – Stratoni road occupies 32.88 ha and the exploratory boreholes 0.2 ha. Moreover, 0.8 ha of wildlife reserve K 115 is occupied by exploratory boreholes. The impacts are considered Negative, although a very small part of the project coincides with the wildlife reserve area.

In the surrounding area there are three areas known as Areas of Outstanding Natural Beauty (AONBs) that fall into the wider study area for the overall project: the Stagira (Olympias) and Kapros island, the Stratoniki - Kipouristra Gorge and the Ierissos area. The AONBs examined are characterized as of extreme importance in terms of the quality of natural beauty and as vulnerable due to man-made pressures. No projects or spatial units are located in the said areas and the impacts are deemed Neutral. The evaluation of impacts on biodiversity is shown in Table 20.4.

Table 20.4 Evaluation of the impacts on biodiversity

Item evaluated	Impact on vegetation and habitats	Impact on fauna and birdlife	Impact on special areas for conservation	Impact on special protection areas	Impact on protected areas under Law 1650/1986	Impacts on wildlife reserves	Impact on World Heritage Sites	Impact on forests	Impacts on AONBs	Impact on important areas for birds	Impact on the marine environmen t
Condition of the Recipient	High	High	Medium	Low	Low	Medium	Negligible	High	Negligible	Negligible	Negligible
Nature of impacts	Negative	Negative	Neutral	Neutral	Neutral	Negative	Neutral	Negative	Neutral	Neutral	Neutral
Impact intensity	High	High	Negligible	Negligible	Negligible	Low	Negligible	Medium	Negligible	Negligible	Negligible
Impact complexity	Direct	Direct	-	-	-	Direct	-	Direct	-	-	-
Geographical reference level	Local	Local	-	-	-	Local	-	Local	-	-	-
Likelihood of occurrence	Certain	Certain	-	-	-	Not at all likely	-	Certain	-	-	-
Temporal nature	Long-term	Long-term	-	-	-	Short-term	-	Short-term	-	-	-
Ability to address through physical processes	Irreversible	Irreversible	-	-	-	Partially reversible	-	Fully reversible	-	-	-
Ability to address through artificial means	Partially addressable	Partially addressable	-	-	-	Totally addressable	-	Addressable	-	-	-
Impact Size	High	High	Negligible	Negligible	Negligible	Low	Negligible	Medium	Negligible	Negligible	Negligible
Importance of impacts	High	High	Insignificant	Insignificant	Insignificant	Medium	Insignificant	High	Insignificant	Insignificant	Insignificant

20.5.6 Impacts on the man-made environment

Skouries is located within the boundaries of the Municipality of Aristotelis and, in particular, within the Megali Panagia Municipal Unit. Taking this into account and that the project is compatible with the guidelines laid down in the Special Spatial Planning and Sustainable Development Framework for Industry, impact evaluation of the current state of spatial planning, land uses in Skouries is considered as Low.

As far as the structure and functions of the man-made environment are concerned, Skouries includes the village of Paleohori, while the village of Megali Panagia is located 3 km south-west of the immediate intervention area, which in administrative terms falls within the Megali Panagia Municipal Unit. Consequently, taking into account the major dependence of man-made environment operations on forestry and mining, the impact evaluation of the state of the structure and functions of the man-made environment of the Skouries spatial unit is considered as Medium.

No monuments of historical and cultural value have been demarcated and listed in the intervention area for Skouries. However, in the immediate area of Skouries, there is a section of the temporarily listed Siderokafsia archaeological site and part of the listed Horouda archaeological site and certain unlisted archaeological sites. Consequently, impact evaluation of the condition of the cultural heritage of the Skouries spatial unit is considered to be Medium.

The impact assessment is set out in summary form in Table 20.5.

Table 20.5 Evaluation of impacts on the man-made environment

Item evaluated	Impact on planning / land uses	Impact on the man-made environment due to project siting	Impact on the man-made environment due to project operation	Impact on cultural heritage
Nature of impacts	Negative	Neutral	Negative	Negative
Impact intensity	Medium	-	Low	Low
Impact complexity	Direct	-	Direct	Direct
Geographical reference level	Local	-	Local	Local
Likelihood of occurrence	Certain	-	Certain	Certain
Temporal nature	Long-term	-	Long-term	Long-term
Ability to address through physical processes	Irreversible	-	Irreversible	Irreversible
Ability to address through artificial means	Partially addressable	-	Partially addressable	Partially addressable
Impact size	Medium	Negligible	Low	Low
Importance of impacts	Medium	Insignificant	Medium	Medium

20.5.7 Impact on technical infrastructure - physical goods

During the development and operating period, use of the road network in the spatial unit is expected from heavy vehicles and private cars of project workers. The Skouries mining facilities will be served by the wastewater treatment facilities for the biological treatment plant and sewerage system, which will be built within the facilities. The mine's industrial water requirements are met by water pumped from the underground mines, while the Skouries flotation plant needs are primarily met by recirculation of the plant water. Drinking water is supplied by local boreholes. A 150 kV/MV substation will be built to meet the electricity needs. An assessment of the impact of the project on technical infrastructure and material goods during the adverse operating period is set out below in Table 20.6.

Table 20.6 Evaluation of technical infrastructure impact – physical goods

Item evaluated	Impact on land transport infrastructure	Impact on environmental infrastructure systems	Impact on water supply networks	Impact on energy networks
Nature of impacts	Negative	Neutral	Negative	Positive
Impact intensity	Medium	-	Medium	Low
Impact complexity	Direct	-	Direct	Direct
Geographical reference level	Wider area	-	Local	Local
Likelihood of occurrence	Certain	-	Certain	Certain
Temporal nature	Long-term	-	Long-term	Long-term
Ability to address through physical processes	Irreversible	-	Irreversible	Irreversible
Ability to address through artificial means	Partially addressable	-	Partially addressable	Not addressable
Impact Size	Medium	Negligible	Medium	Low
Importance of impacts	Medium	Insignificant	Medium	Medium

20.5.8 Impact on the atmospheric environment

As far as the current state of the environment is concerned, the atmospheric environment of the Skouries spatial unit has High sensitivity taking into account current levels of air quality, while the resilience of the atmospheric environment in the villages is estimated at Medium, primarily due to local meteorological conditions in the area. Lastly, the value of the air is also assessed as Medium, given that, in small villages, air pollutants are related to the quality of life of their residents, even though in general, there are no particularly sensitive recipients.

An assessment and evaluation of the impacts on the atmospheric environment the Skouries project was completed. It is estimated the impacts from air emissions on the air quality of the surrounding area ranges from Low to Medium. The limits for gaseous and particulate pollutants are not exceeded.

20.5.9 Impacts due to noise and vibration

The mine, process plant and other activities are not expected to impact significantly on the acoustic environment of the wider area and especially on that of the inhabited areas. The most significant impacts on the acoustic environment of the wider area are expected from materials transportation activities within and outside the mining perimeter by trucks. The operation of the open pit is not projected to affect the wider area. None of the projected values exceeds regulated noise limit values.

Noise levels in the immediate Project area will be affected during development and Phase 1 of operations due to the open pit mining activities, but this will be of limited duration and will not exceed regulated limits.

The current state of the acoustic environment in the Skouries project area has ‘Low’ sensitivity taking into account the fact that, with the exception of urban traffic on the road network, there does not appear to be high noise levels from other activities. The resilience of the acoustic environment is estimated at a High level, given the relatively limited traffic of vehicles on the road network. Lastly, the value of the acoustic environment is also assessed as Low, given that in small villages low noise levels are related to the quality of life of their residents. Consequently, impact evaluation of the state of the acoustic environment in the Skouries spatial unit is considered as Low in terms of its sensitivity / susceptibility and vulnerability / value.

During the development and operating phases of the mining facilities in the Skouries project, the level of expected impacts on the acoustic environment in the area is considered to be negligible. Likewise, the size of expected impacts on construction works from vibrations is considered to be negligible.

Monitoring programs of noise and vibration will be undertaken through all phases of the Project.

The evaluation of impacts on the acoustic environment and vibrations is shown in Table 20.7.

Table 20.7 Evaluation of impacts on the acoustic environment and vibrations

Item evaluated	**Acoustic environment**	**Vibrations**
Nature of impacts	Negative	Negative
Impact intensity	Negligible	Negligible
Impact complexity	Direct	Direct
Geographical reference level	Local level	Local level
Likelihood of occurrence	Certain	Certain
Temporal nature	Long-term	Long-term
Ability to address through physical means	Non-addressable.	Non-addressable.
Ability to address through artificial means	Not required	Addressable

20.5.10 Impacts on water

The Skouries project area includes the Asprolakkas river water system as well as the Lotsaniko, Karatzas Lakkos, Ekklisiastikos Mylos and Asprolakkas streams. The Asprolakkas stream has an unknown overall status with an unknown chemical status, good ecological status and flows all year round. It appears to meet the legislative limits for chemical status with the exception of concentrations of Pb (dissolved), which are considered to be associated with the general metalliferous deposits in the area. The Asprolakkas immediately after the confluence with the Lotsaniko stream has high As values, also associated with the general metalliferous deposits in the area. The Karatzas Lakkos, Lotsaniko and Ekklisiastikos Mylos streams also flow all year round with quality characteristics similar to those of the Asprolakkas.

The Skouries project is located within the Holomontas - Oreokastro fault ground water system.

The Holomontas - Oreokastro ground water system is characterized as having ''Good quantitative status''. The Skouries sub-system is characterized as having "poor qualitative status". Considering the natural hydrochemical conditions in the area, one can conclude that the Skouries area is a Low vulnerability zone.

Qualitative impacts on the surface bodies of water are the most unfavourable during the development and open pit operating period. During this period, the overall volume of run-off from the Asprolakkas sub-basin will be reduced by up to 6.1% on average. A major drop in the supply rate for the Karatzas stream by 76.2% is expected in the 3rd year of development and a 19.8% increase in the supply rate for the Lotsaniko stream is expected in the 3rd year of development.

During the project period, any water that comes into contact with the mine and process areas will not arrive in streams without treatment. Water will be suitably treated to comply with the legislative limits for both disposal and for the final body of water prior to disposal in the Asprolakkas. There will be no impact on quality characteristics from the disposal of treated water in the Asprolakkas stream.

Impacts on the quantitative characteristics of the ground water systems are expected from the pumping of the Skouries underground mine. Pre-drainage will take place in the open pit development zone. Pumping is expected to gradually increase. It has also been planned that part of the treated pumping water from the mine will be channelled to the underground aquifer via re-injection boreholes.

Simulation shows that pumping to drain the mine will create an ellipsoidal drop-level cone along a NW-SE axis, while the zone which demarcates the drop level by 2 m is around 3.0 km SE, 2.9 km NW, 2.7 km SW and 0.9 km E during the five year period of operation at the end of the development phase. During full development of the underground mine, it is estimated that total pumping will continue at a rate of around 4,920 m^3/d for the entire period. This pumping will increase the drop cones analyzed in the combined open pit / UG mine phase creating a larger ellipsoidal drop cone. During rehabilitation when the pumping out is stopped, the rise in the ground water level will commence. The simulation estimates that after the first year of rehabilitation, the ground water level would be around 10 m to 15 m below the initial level (2020). After five years, the level will be 2 m to 5 m below the initial level and, after nine years, the lowered cone will have been fully restored and rehabilitated.

Inspection showed that, due to the measures taken (treatment of pumped water for re-injection, disposal of extractive waste in specially sealed barriers, filling in of inactive tunnels), coupled with low aquifer vulnerability, no further degradation in the quality of ground water is expected.

The assessment of impacts on the waters in the Skouries spatial unit is shown in Table 20.8.

Table 20.8 Evaluation of the impacts on water

Item evaluated	Impact on rivers in the water system (quantitative)	Impact on rivers in the water system (qualitative)	Impact on coastal water system (qualitative)	Impact on ground water system (quantitative)	Impact on ground water systems
Condition of the Recipient	**Low**		**Medium**	**Low**	
Nature of impacts	Negative	Negative	Negative	Negative	Negative
Impact intensity	Medium	Low	Negligible	Low	Low
Impact complexity	Direct	Direct	Indirect	Direct	Direct
Geographical reference level	Wider area	Wider area	Study area	Study area	Study area
Likelihood of occurrence	Confirmed	Confirmed	potential	Confirmed	Confirmed
Temporal nature	Long-term	Long-term	Long-term	Long-term	Long-term
Ability to address through physical processes	Partially reversible	Partially reversible	Partially reversible	Partially reversible	Partially reversible
Ability to address through artificial means	Partially addressable	Partially addressable	Addressable	Partially addressable	Partially addressable
Impact Size	Medium	Medium	Negligible	Medium	Medium
Importance of impacts	Low	Low	Insignificant	Low	Low

20.6 Public consultation and disclosure

20.6.1 Stakeholder engagement plan

Hellas Gold has an obligation to hire 90% of the workforce locally. Other than the commitment to maximize local employment, there are no social obligations attached specifically to the Project. However, Hellas Gold has a policy of assisting local communities that are stakeholders in its projects and will continue with this policy. This has included various town improvement schemes such as street paving, lighting, sewerage and municipal facilities.

In addition, Hellas Gold has committed to ensure the smooth integration of the Project into the socio-economic environment of the local area, by adopting a policy of filling job positions on a preferential basis from the local population. Employees from the construction stage will be gradually incorporated into the production team.

The Stakeholder Engagement Plan (SEP) has been developed by Hellas Gold and the management of Eldorado Gold with the aim of providing a structure for communication and consultation with all identified stakeholders that could affect the Project and that are affected by it, taking into consideration Greek, European and international law and best practice. The SEP is part of a suite of documents covering social and environmental management (other documents include Human Resources Plan, Hazardous Materials Plan, Health Safety and Security Plan, Discharge and Emissions Plan and Community Development Plan) and is seen by Eldorado as an important tool for transparency and effective risk management.

20.7 Closure and reclamation

20.7.1 Overview

The fundamental criteria for Project closure and environmental rehabilitation include the following:

- The Project site must be handed back in a state that will not give rise to risks to the health and safety of people, the flora and fauna in the area, and to environmental safety in general.
- All remaining structures, including interventions, in the natural terrain of the Project site, must not generate any risk to public health, safety, or the environment in terms of geotechnical stability.
- All remaining materials must not generate a risk to public health or the environment for future users of the area.
- Environmental rehabilitation must lead towards a self-sustaining ecosystem typical of the area. The purpose of the rehabilitation program must be to meet future land needs in the area and rehabilitation must seek to re-create safe and stable biological conditions that encourage natural regeneration and the development of biodiversity.

The closure and environmental rehabilitation activities for Skouries mine relate to the following facilities:

- Open pit and underground mine
- IEWMF
- Process facilities and infrastructure

To meet the requirements of the reclamation program, decommissioning, closure and reclamation of the affected areas must be undertaken. In all cases, after the original ground is graded to match surrounding morphology and provide positive drainage, closure capping will take place. The closure capping will consist of a layer comprised of inert waste materials from operation of the deposit, 2.5 m thick, and a layer of topsoil, 0.5 m thick. After capping, the sites will be planted with species endemic so as not to undermine the vegetative physiognomy of the area. The plant species to be planted will be maintained until they are capable of growing without any care.

Topsoil collected during the development stage of the Project will be stockpiled separately, protected, and preserved for the LOM for use in rehabilitation. This topsoil will be enriched with suitable organic matter and inorganic fertilizers to ensure a stable structure for the soil layer and to assist in the development of new vegetation.

As much as possible, reclamation is to be completed progressively as areas are decommissioned; this will be especially beneficial to the IEWMF, which is decommissioned at the completion of Phase 1 due to the large surface area involved.

During reclamation, necessary measures will be taken to minimize the dust generated during earthworks, particularly when meteorological conditions favour the spread and carrying of dust over large distances.

20.7.2 Open pit mine

Once surface extraction of the deposit is completed at the end of Phase 1, with activities thereafter focusing on underground operations, backfilling will commence with tailings to restore the pit to its original morphological condition and to permit final rehabilitation of the site.

20.7.3 Underground mine

The site around the entrance of the central access tunnels to the underground mine will be fully rehabilitated.

When decommissioning the underground mine, mobile and fixed mining equipment will be dismantled and removed. Equipment parts that are merchantable will be cleaned and stored at predetermined storage areas. Items that are not merchantable will be sent for recycling.

20.7.4 Integrated extracted waste management facility

Prior to completion of Phase 1, progressive reclamation of the IEWMF will begin by capping and grading areas as the filtered tailings placement reaches final elevation. At completion of Phase 1, the stacking equipment will be decommissioned and removed, and will either be stored for use in the open pit backfilling or sold. All remaining structures and equipment will be removed except for equipment necessary for water management during the reclamation period.

20.7.5 Process facilities and infrastructure

At completion of the operational phase, process facilities and infrastructure will be decommissioned and removed. Assets with residual value will be removed, cleaned, and collected together in pre-determined locations where they can be safely stored until they can be sold. All stocks of treatment reagents that are not used will be sold or safely disposed of.

Following decommissioning and removal of the equipment, metal structures, reinforced concrete structures and foundations will be removed.

Some site roads must be retained and must continue to be used as forest roads or as firefighting roads, where the competent forestry services or fire brigade consider this necessary. The majority of the roadside slopes will be restored.

20.8 Environmental costs and guarantees

Hellas Gold has provided a €50.0M (US$57.5M) Letter of Guarantee to the MOE as security for the due and proper performance of rehabilitation works in relation to the mining and metallurgical facilities of the Kassandra Mines project and the removal, cleaning and rehabilitation of the old, disturbed areas from historical mining activity in the wider area of the project. Additionally, a Letter of Guarantee to the MOE, in the amount of €7.5M (US$8.6M), has been provided as security for the due and proper performance of the Kokkinolakkas TMF (total €57.5M).

The total progressive rehabilitation costs for the surface IEWMF in Phase 1 are estimated to be US$15.9M. Additionally, the closure of the open pit includes backfilling with tailings during Phase 2 of the operation; these costs are also included in operating costs.

Additional costs for dismantling the processing plant, ancillary buildings, powerlines, and roads are estimated to be the residual of the asset retirement obligation (ARO) left once progressive rehabilitation has been subtracted. The estimate used in the economic model is US$10.7M as the cost net of salvage value.

Five years after completion of the rehabilitation works and the change in land use of each individual facility in line with the master plan, the effectiveness of the guarantee will be examined by the External Auditing Committee, which will make a recommendation on any adjustment to the letter of guarantee that is needed.

21 Capital and operating costs

21.1 Project capital cost estimate

The total Project capital cost includes the remaining cost to complete the Project construction until commercial production is achieved (‘initial capital’), and subsequent sustaining capital costs spread out over the remaining 20 years of the mine life. Capital costs are summarized in Table 21.1. Sunk costs to the end of 2021 are excluded from the capital cost estimate.

Table 21.1 Capital cost estimate summary*

Area	Capital cost (US$ M)
Development capital (pre-production)	
Underground Phase 1 development	123
Open pit	99
Process and infrastructure	390
IEWMF and water management	158
Power Line	9
Owners Cost	66
Total pre-production development capital	**845**
D**evelopment capital (Phase 2 underground)**	**172**
Underground	569
Open pit	21
Process and infrastructure	190
IEWMF and water management	81
Sub-total sustaining capital	**861**
Ramp up period (costs net of production)	-19
Addback spares	5
Total sustaining capital	**847**
Total capital (development and sustaining capital)	**1,863**

Note: Totals may not add up due to rounding.
* Contingency included in values shown.

The total initial capital cost is shown in Table 21.2. This includes the cost to complete the project construction until commercial production of the mill.

Table 21.2 Initial capital cost summary – US$ million

Description	Underground	Open pit	Process plant and infrastructure	IEWMF and water management	150 kV powerline	Owner's cost	Total
Direct cost	99	87	279	122	8	-	595
Indirect cost	12	-	108	16	0	64	199
Contingency	16	13	49	25	1	5	109
Subtotal	**127**	**100**	**436**	**163**	**9**	**69**	**904**
Late changes	(4)	(1)	(46)	(5)	(0)	(3)	(59)
Total	**123**	**99**	**390**	**158**	**9**	**66**	**845**

Note: Totals may not add up due to rounding.

The accuracies of the cost estimates are consistent with the standards outlined by the Association for the Advancement of Cost Engineering (AACE). The cost estimate is a feasibility-level estimate categorized as AACE Class 3.

Direct costs were developed from a combination of budget quotes, material take-offs, existing contracts, Project-specific references, and historical benchmarks. Indirect and owners' costs were estimated using a combination of existing commitments, calculated project requirements, and historical benchmarks. Contingency was applied to each cost item in the estimate, based on the level of engineering definition and reliability of its unit rates.

The capital cost estimate does not include sunk costs.

The sustaining capital cost spread out over the 20 years of the mine life is included separately and summarized in Section 21.1.15.

21.1.1 Accuracy

This estimate is considered Class 3 AACE with an accuracy of -15% / +20%.

21.1.2 Basis of cost and currency

All costs and pricing in the estimate are expressed in Q3 2021 US dollars (US$). Relevant base conversion rates are shown in Table 21.3.

Table 21.3 Exchange rates

Currency code	Currency name	Exchange rate
US$	United States Dollar	US$1.00 = US$1.00
C$	Canadian Dollar	C$1.00 = US$0.80
€	Euro	€1.00 = US$1.20

As of late changes, the exchange rate was adjusted from US$1.2/EUR to 1.13 (2022), 1.15 (2023), and 1.18 (2024). This adjustment represents a reduction of US$24.6M in the overall capital cost estimate with the total reduction due to the late changes being US$59M.

21.1.3 Open pit and underground mining

Open pit and underground mining equipment quantities and mining development costs were determined through build-up of mine plans.

Mining Plus and its subcontractors, Minefill, and Cementation went through a quotation process for the capital cost estimate.

Table 21.4 Open pit and underground mining capital

Description	Open pit (US$M)	Underground (US$M)	Total (US$M)
Mining equipment	12.7	0.8	13.5
Mining development	74.3	63.7	138
Water management		3.8	3.8
Underground mine infrastructure		2.4	2.4
Power and communication		10.1	10.1
Pre-production operating		17.5	17.5
Geotechnical		0.4	0.4
Subtotal	**87.0**	**98.7**	**185.7**
Indirect cost		11.6	11.6
Contingency	13.1	16.5	29.6
Late changes (exchange rate adjustment)	-1.3	-3.7	-5
Total	**98.8**	**123.1**	**221.9**

Note: Totals may not add up due to rounding.

21.1.4 Process plant and infrastructure

Fluor prepared the capital cost estimate for the process plant and infrastructure scope. Material take-offs were prepared for the legacy areas already designed and for new areas. Budgetary quotations were obtained for the major equipment and bulk material to support the Class 3 estimate, which is summarized in Table 21.5.

Table 21.5 Process plant and infrastructure capital

Description	US$M
Overall site	1.7
Mine	30.7
Crushing	22.7
Process plant	165.6
Tailing handing	18.3
Infrastructure	24.5
Water management	3.0
Ancillary facilities	12.8
Total	**279.2**
Indirect cost	107.7
Contingency	49.0
Late changes (exchange rate adjustment)	(45.9)
Total	**390.0**

Note: Totals may not add up due to rounding.

21.1.5 Labour

Labour rates were derived from information provided from existing contracts and questionnaires filled out by contractors that worked before on the Skouries Project. The all-in crew labour rates include all direct and indirect costs associated with the contractors.

The project will be constructed utilizing a Construction Management approach. The EPCM contractor will be responsible for dividing the work up into construction work packages which will, in turn, be subcontracted to qualified construction contractors.

21.1.6 Labour productivity

A labour productivity factor is used to account for overall labour force efficiency. Non-productive events are estimated based on expected construction conditions.

21.1.7 Commodity pricing

In general, direct unit costs were based on contract rates, quotations from vendors and contractors. For some minor items, allowances were carried based on historical data. Table 21.6 outlines the primary source for unit costs, by major commodity.

Table 21.6 Primary source for unit costs

Commodity	Primary source
Major mobile equipment	Vendor quotations
Process & ancillary equipment	Major Equipment – Vendor Quotations Minor Equipment – Vendor Quotation or Consultant Historical
Earthworks	Contract Rates and historical information
Concrete	Contract Rates, Local Supplier Quotations
Steel	Contract Rates, Local Supplier Quotations
Piping	Recently projects information
Electrical & instrumentation	Major Equipment – Vendor Quotations Bulk Materials – Local Supplier Quotations

21.1.8 Material quantities

Quantities were based on detailed material take-offs and equipment lists, with some allowances for minor items. The material take-offs were calculated as neat quantities. The unit rates cost included waste and overbuy as a factor.

Growth allowances were applied by discipline to cover typical increases due to design refinement and improved definitions of the design due to engineering progress and elements outside of the material take-off (i.e., gussets, fasteners, steel touch-up paint, etc.). The growth allowances were applied in two different areas: Legacy Plant, which includes the scope already designed; and New Plant, which represents the scope to be designed.

21.1.9 Indirect cost estimate

Indirect costs were calculated based on the proposed construction execution in consideration of the overall project scope and schedule. Table 21.7 summarizes the basis of indirect costs.

Table 21.7 Basis of indirect costs

Area	Primary source
Construction Indirects	The detailed estimate was prepared for the temporary facilities, services, site logistics, construction support equipment, and tools costs.
	Freight and logistics costs were estimated based on quotations received from vendors for major packages and the rest of equipment and material was calculated as a percentage of the cost and divided into overseas and European freight.
	Vendor representatives' costs were calculated based on the number of days required and daily rates.
	An allowance for pre-commissioning craft support was included in the estimate.
Spares / First Fills	Capital and commissioning spares were calculated as a percentage of equipment supply cost based on historical data.
	First fills required for start-up were estimated based on consumption rates by the Process Design Criteria, with unit costs referenced from quotations.
EPCM	Engineering and Procurement cost is based on deliverables list and hours per deliverable. A staffing plan was developed to support the Construction Management costs, which were based on the construction execution plan, which assumes that multiple subcontractors will perform the work.

21.1.10 IEWMF and water management

The material take-offs scope associated with IEWMF and water management was developed by Golder.

In general, labour and productivity factors and pricing were estimated in a manner similar to that for the process plant and infrastructure scope.

A summarized cost is shown in Table 21.8.

Table 21.8 IEWMF and water management cost

Area	**US$ million**
Overall site	5.9
Mine	8.4
IEWMF	29.5
Infrastructure (WTP)	25.2
Water management	53.1
Subtotal	**122.1**
Indirect cost	16.2
Contingency	24.8
Late changes (exchange rate adjustment)	(4.9)
Total	**158.2**

Note: Totals may not add up due to rounding.

21.1.11 Powerline 150 kV

The 150 kV permanent power supply is based on analysis developed in 2017 and commitments already in place. Indirect cost was added to support the construction of the powerline (US$9M).

21.1.12 Owner's cost

Owner's costs included labour and G&A costs for the owner's team during the period of active construction applied over a specified period. An allowance for insurance, travel, land acquisition, and archaeological discoveries was also included.

Table 21.9 Owner's cost

Description	**US$ million**
Owners team stuffing plan	10.8
Site related services	5.4
Other costs (insurances, tax-duties, etc.)	10.8
Eng.- permits-EoR Fees	13.3
G&A allocation	23.2
Contingency	5.1
Subtotal	**68.6**
Late change (Exchange rate adjustment)	(2.7)
Total	**65.9**

Note: Totals may not add up due to rounding.

21.1.13 Contingency

Contingency accounts for unforeseen costs within the project scope and has been applied to the Capital cost estimate with input gathered from all parties involved. The overall Project contingency for initial investment is 14%.

Contingency for the initial investment capital cost estimate is summarized by category in Table 21.10.

Table 21.10 Initial capital contingency – US$M

Category	**Total without contingency**	**Contingency (%)**	**Contingency**	**Total with contingency***
Open pit	87.0	15	13.1	100.1
Underground	110.3	15	16.5	126.8
Process plant and infrastructure	386.9	13	49.0	435.9
IEWMF and water management	138.3	18	24.8	163.1
150 kV powerline	8.3	10	0.8	9.1
Owner's cost	63.5	8	5.1	68.6
Total	**794.3**	**14**	**109.2**	**903.6**

Note: Totals may not add up due to rounding.
*Excludes late changes of (US$59M) from original estimate of US$904M.

21.1.14 Exclusions

The following items are specifically excluded from the capital cost estimate:

- Force majeure
- Fluctuation of currency exchange rates
- Sunk costs
- Event contingency
- Working capital
- Sustaining capital
- Scope changes
- Closure costs
- Removal of contaminated waste / soil
- Technology obsolescence from previous equipment or instrumentation
- Costs associated with COVID-19 impacts on execution

21.1.15 Sustaining capital cost estimate

During the life of the project, there will be a requirement for further capital expenditure. This may take the form of additional mining fleet as required, and equipment and further purchase of conveyors and IEWMF and Water Management. The sustainable capital summary is presented in Table 21.11.

Table 21.11 Sustaining capital cost (US$M)

Year	Open pit	Underground	Process plant and infrastructure	IEWMF and water management	Sub-total	Ramp up period (costs net of production)	Addback spares	Total
-1	-	-	-	-	-	-18.9	-	-18.9
1	1.8	103.5	11.4	11.9	128.6	-	5.0	133.6
2	3.2	59.5	55.6	9.6	127.8	-	-	127.8
3	5.3	9.2	1.9	9.4	25.7	-	-	25.7
4	3.4	7.3	2.8	7.9	21.5	-	-	21.5
5	4.3	26.0	2.8	6.5	39.7	-	-	39.7
6	1.2	33.5	4.7	6.5	45.9	-	-	45.9
7	-	47.9	4.7	3.2	55.9	-	-	55.9
8	0.3	58.6	4.7	2.5	66.1	-	-	66.1
9	0.1	33.6	32.6	4.7	71.0	-	-	71.0
10	-	28.6	6.7	0.8	36.1	-	-	36.1
11	0.1	26.9	6.7	0.8	34.4	-	-	34.4
12	1.2	17.4	6.8	0.8	26.1	-	-	26.1
13	-	16.1	7.0	0.8	23.9	-	-	23.9
14	-	28.9	7.2	0.8	36.9	-	-	36.9
15	-	23.9	7.3	0.8	31.9	-	-	31.9
16	-	15.9	7.4	0.8	24.0	-	-	24.0
17	-	10.4	7.4	0.8	18.5	-	-	18.5
18	-	9.3	6.3	0.7	16.3	-	-	16.3
19	-	7.3	4.0	0.4	11.8	-	-	11.8
20	-	4.7	2.1	11.9	18.7	-	-	18.7
Total	**20.7**	**568.5**	**190.0**	**81.3**	**860.5**	**-18.9**	**5.0**	**846.6**

Note: Totals may not add up due to rounding.

There other sustaining costs include an addback spare of US$5M in Year 1 and a credit for the ramp up period (costs net of production) of US$19M in Year -1.

21.2 Operating costs

The operating cost (opex) estimate provides the projected LOM operating costs associated with mining, the process plant, tailings filtration plant, backfill plant, WTP, water systems, and G&A facilities. The operating cost includes all on-site costs from mining through to the production of copper concentrate, including tailings filtration, tailings compaction, and paste production.

The operating cost estimate has been developed on a year-by-year basis in accordance with Eldorado's envisaged operations and mine plan. The estimated total costs by cost centre and cost category are presented in Table 21.12; they are based on the nominal throughput of the plant. All unit costs are reported as US$/t of ore processed. Annual costs in US$ correspond to the sum of all the operating costs for each operating year. A €/US$ exchange rate of 1.2 was used for the preparation of the operating costs during the production years. The cost per tonne averages for the open pit and underground mining are calculated based on the tonnages mined for the production years of those phases. The non-mining cost centre expenditures are averaged based on the process plant ore throughput for the production years. The opex excludes cost associated with pre-production years.

Table 21.12 Total operating costs by cost centre and category

Cost centre	Production Years Total Cost (US$)	Production Years Cost per tonne of production ore (US$/t)
Open pit mining	244,815,387	4.24*
Underground mining	1,681,025,005	19.32*
Process plant	1,247,247,282	8.54
Tailings filtration plant	314,300,479	2.15
Backfill plant	27,506,378	0.19
Water system	20,007,884	0.14
G&A	409,139,670	2.80
Subtotal mining	**1,925,840,391**	**13.18**
Subtotal non-mining	**2,018,201,653**	**13.81**
Total	**3,944,042,045**	**26.99**

Note: Totals may not add up due to rounding.
*These averages were calculated according to the total tonnages mined for the years of commercial production (open pit production ore mined of 57,772,918 t, underground production ore mined of 87,008,294 t, total production ore mined of 144,781,212 t, total production ore processed of 146,115,060 t).

21.2.1 Basis of estimate

The Feasibility Study operating cost estimate is prepared to target an accuracy of -15%/+20% and is based on Q3 2021 pricing.

The execution strategy for project implementation is summarized as follows and forms the basis of the operating cost estimate:

- The project will be carried out in two phases, with 8 Mtpa nominal throughput from Years 1 through 13 and 6.5 Mtpa nominal throughput from Years 14 to 20.
- Concentrate transport will be carried out by a trucking system that is included in the owner's cost and is therefore excluded from the operating cost estimate.

21.2.1.1 Estimate team entities

The operating cost estimate is the result of a combined effort of Eldorado, Hellas Gold, Fluor, Mining Plus, MineFill, Cementation, and Golder. The overall operating cost model has been consolidated by Fluor with support from all parties.

Table 21.13 summarizes the major participants involved in developing the operating cost estimate and provides a brief description of their responsibilities and input.

Table 21.13 Operating cost participants

Participant	Scope	Comment
Eldorado Gold / Hellas Gold	Utility and labour rates, staffing, G&A, mining equipment	Evaluation of open pit and underground mining operating costs provided by Mining Plus for mine equipment, mobile equipment fuel consumption and associated open pit and underground costs. Input of currency exchange rates, labour rates and headcount for the process plant and tailings filtration plant, utility rates, operating costs for G&A and selected operational contract items, and capital costs for the legacy plant as input for maintenance consumable costs.
Fluor	Processing and plant area ancillaries, and tailings filtration plant	Evaluation of on-site processing and ancillary facilities at the processing plant site with respect to maintenance consumables, operating consumables, power, and costs for selected G&A and operational contract items based on input from Eldorado and Hellas Gold. Responsible for the development of the process design of the tailings filtration plant including input to capital costs and operating costs.
Mining Plus	Open pit and underground mining	Responsible for operating cost inputs associated with the open pit and underground mine with respect to maintenance consumables, operating and maintenance labour, power consumption, fuel consumption, operational contracts, and G&A. Evaluation of the underground portion of the backfill paste system including labour, power, operating and maintenance consumables. Consolidation of input from Cementation.
MineFill	Backfill plant	Responsible for operating cost inputs associated with the above-ground portion of the backfill paste plant. This includes paste plant operating and maintenance labour, maintenance consumables, and G&A.
Golder	WTP	Responsible for the evaluation of the WTP at site. Direct inputs to the operating cost estimate are labour requirements, power, operating and maintenance consumables, and G&A.
Golder	Water system	Responsible for the evaluation of plant wide water systems at site. Direct inputs to the operating cost estimate are labour requirements, power, and maintenance consumables.
Golder	Tailings compaction	Responsible for the operating costs associated with tailings compaction in the IEWMF and open pit. Inputs include labour rates and requirements, fuel for mobile equipment, operating and maintenance consumables, and operating contracts.

21.2.1.2 Estimating format and reporting

Operating costs are estimated by separate cost centres, with each cost centre broken down by cost category. Estimating operating costs by cost centre and category allows the costs to be summarized in manageable units or packages. Table 21.14 provides a list of the cost centres and cost categories applied in the operating cost estimate.

Table 21.14 Operating cost centres and categories

Cost centres	Cost categories (applicable to each cost centre)
Open pit mining	Power
Underground mining	Operating consumables
Process plant	Maintenance consumables
Tailings filtration plant	Operating labour
Backfill plant	Maintenance labour
Water system	Fuel
G&A at site	Operational contracts
	G&A

21.2.1.3 Base date and currency exchange rate

The currency of the estimate is the US$ with a €/US$ exchange rate of 1.2. The base date of the operating cost estimate is Q2 2021 for the Feasibility Study scope, which is the point at which the estimated price is valid for a given set of conditions. The base date also qualifies the estimate in terms of market price for equipment, material commodities, labour rates, and contractor margins. Any escalation beyond the base date is excluded from the operating cost estimate. The operating costs for the legacy plant are based on previously purchased costs and estimates.

The cost of cash and any finance costs is excluded from the estimate.

21.2.1.4 Utility rates

The utility rates used in the estimate were provided by Eldorado and are shown in Table 21.15.

Table 21.15 Utility rates

Utility	Unit	Rate
Electrical power rate	€/kWh	0.089
Diesel supply rate	€/L	0.98

21.2.1.5 Cost centres

Cost centres categorize the total project scope of work into manageable units or packages and provide a structure for performing and defining how cost data are summarized, reported, and controlled. Table 21.16 on the following page shows the work breakdown structure (WBS) distribution per cost centre. The operating cost estimate is divided into the following cost centres:

- Open Pit Mining: All mining unit operations and maintenance costs within the mine battery limits, including drilling, blasting, loading, hauling, auxiliary equipment, mine services, planning, and stockpile rehandling.
- Underground Mining: All mining unit operations and maintenance costs within the mine battery limits, including drilling, blasting, loading, hauling, auxiliary equipment, mine services, associated backfill plant costs and planning.
- Process Plant: Operating and maintenance costs for the process plant including the operating and maintenance costs associated with process unit operations. Key facilities include primary crushing, plant area conveyors / stockpile, stockpile rehandling, grinding, flotation, filtration, concentrate dewatering, concentrate storage and reclaim, rougher tailings classification, rougher tailings transport to paste plant, and tailings thickening and transport to filtration plant.
- Tailings filtration plant: Operating and maintenance costs for the tailings filtration, filtrate solution transport to process water ponds, tailings filter cake conveying, spreading system, and tailings compaction.
- Backfill plant: Operating and maintenance costs for the paste plant, filtrate solution transport to tailings thickeners, and paste delivery.
- Water System: Operating and maintenance costs for the site-wide water system including mine dewatering, water diversion, and tailings seepage and associated ancillaries.
- G&A: Direct and indirect costs associated with items for general, administrative, security, safety, and employee / community related expenses. Overhead costs associated with the operation of the entire facilities are collected under the G&A cost centre. Direct G&A costs that can be attributed to a specific cost centre are captured against that cost centre.

Table 21.16 WBS distribution per cost centre

WBS	WBS description	Cost centre
10/12/20	Open pit mining	Mining
22/24/27/28	Underground mining	Mining
26	Backfill plant	Backfill plant
32	Open pit crushing and conveying	Process plant
34	Stockpile and reclaim	Process plant
36	Grinding system and control room	Process plant
37	Pebble crushing system	Process plant
38	Gold gravity circuit / gold room	Process plant
40	Flotation system / regrinding	Process plant
42	Tailings thickening system	Process plant
44	Lime plant	Process plant
46	Flocculants system	Process plant
47	Sulphidizer system	Process plant
48	Reagents system	Process plant
50	Concentrates thickening system	Process plant
52	Concentrates dewatering system	Process plant
53	Borehole water system	Water system
54	Compressed air services	Process plant
55	Process water system	Process plant
56	Utility water system	Process plant
57	Firefighting water system	Process plant
58	Mine water system	Process plant
59	Potable water system	Process plant
61	Filtration plant	Tailings filtration plant
62	IEWMF	Tailings filtration plant
63/64/68	Site water management	Water system
65	Water treatment plant	Water system
66	Sanitary sewer system	Process plant
70	Substations	Process plant
74	Open pit / mill administration facilities	Process plant
75	High-voltage open switchyard	Process plant
76	Security gatehouse on south access	Process plant

21.2.1.6 Sources of input

The following items provided input to the operating cost estimate:

- Process design criteria – Requirements for operating consumables consumption rates.
- Mechanical equipment budgetary quotations – Pricing and consumption information for certain large-scale operating consumables, such as crusher and mill wear items.
- Reagent and consumables pricing – Information provided by Hellas Gold from current operating plants.
- Energy consumption (kWh) – Power consumption for all equipment within Fluor's scope is derived from the energy summary as per the Electrical Load List. Legacy equipment information was obtained from the previous study load list. Third-party scope power consumption was provided directly from the respective third parties.

- Capital cost estimate – Direct cost of all purchased mechanical and electrical equipment for estimating the operating cost of maintenance consumables within Fluor's scope. Equipment already purchased as part of the legacy plant was provided by Hellas Gold.
- Mining operating costs – This included salary and headcount as provided by Mining Plus with input from Cementation and MineFill.
- Labour rates and staffing – Staffing levels and salary information for the processing plant and tailings filtration plant were provided by Eldorado and Hellas Gold.
- Feasibility Study mine and production plan – These were provided by Mining Plus in coordination with Eldorado.
- Utility rates and fuel pricing – Eldorado Gold provided the applicable utility rates for power and fuel.
- Third-party consultant information – Input information as per third-party consultant scope.

21.2.1.7 Labour rates and staffing

Labour rates and staffing plan were provided by Hellas Gold. Annual labour costs were estimated from the staffing headcount and salary information provided by Hellas Gold.

21.2.1.8 Operating consumables

Materials consumed in the process plant are driven by unit consumption rates or are consumed at a fixed rate each year. Consumption rates for reagents are sourced primarily from process design criteria and are provided, for the most part, on a unit consumption of processed ore basis.

Consumption information for non-mining, large-scale consumables (e.g., crusher and mill wear parts and liners) are estimated using vendor information and are primarily expressed as US$ per year. The estimated consumption of grinding media is based on the consumption rates (kg/t milled) in the process design criteria.

Non-mining operating consumables include the following:

- Reagents
- Grinding media
- Crusher liners
- Crusher wear parts
- Mill and regrind mill liners
- Filter cloths

Pricing for operating consumables is obtained from recent quotations and Hellas Gold's historical database.

Operating consumables costs associated with open pit and underground mining were estimated by Mining Plus and are reported in the open pit and underground mining cost centres of the operating cost summary. Operating consumables used in the paste backfill plant are reported as part of the underground mining cost centre as they are considered part of underground operations.

Operating consumables costs associated with the WTP are estimated by Golder and are reported in the water system centre of the operating cost summary.

21.2.1.9 Maintenance consumables

Maintenance costs for plant equipment are derived by multiplying the capital cost by a factor. These costs are referred to as the maintenance consumables in the operating cost estimate and are defined as components of a piece of equipment that are expected to require regular replacement to keep the equipment in good working order. For the estimate, the following factors are applied:

- A 2.5% maintenance cost factor applied to mechanical equipment direct capital cost.
- A 3% maintenance cost factor applied to electrical, instrumentation and control system equipment direct capital cost.

Direct capital cost input information used in the non-mining operating cost model is estimated by Fluor as part of the Feasibility Study scope and equipment included as part of the legacy plant as provided by Hellas Gold. Maintenance consumables information for third-party consultant scopes of work is input from their respective operating cost submissions.

21.2.1.10 Power

The Electrical Load List provides energy consumption estimates summarized by WBS area for the Fluor scope (related equipment and legacy plant equipment). Details for each piece of mechanical equipment and the ancillary load are included in the power consumption summary. Through the estimated average usage for each piece of equipment, the total annual energy requirement in kilowatt-hours is determined. Tabulating the energy consumption provides the total energy requirement of the project. The power consumption information for third-party consultant scopes of work is input from their respective operating cost submissions. To determine the power costs for the project, utility rates provided by Eldorado were used.

21.2.1.11 Fuel

Fuel costs include the consumption of diesel, mainly for mobile equipment operation. Diesel fuel consumption is estimated for the number of vehicles and mobile equipment. The supply cost of diesel is defined by Eldorado and is used to determine fuel costs for the project.

Fuel consumption associated with mining was estimated by Mining Plus and is reported in the mining cost centre of the operating cost summary.

21.2.1.12 Operational contracts

Operational service contract costs are third party contract costs and include costs for items such as facilities support, inspection, equipment contracts, and maintenance that are necessary for the operation of the processing facilities. The input information was obtained from relevant third-party consultants and Mining Plus.

21.2.1.13 General & administrative

G&A expenses are defined as costs that are not covered under any of the above categories but must still be included. G&A expenses include costs such as taxes, auditing, security, safety, environmental, and community relations. These costs are provided by Eldorado.

21.2.1.14 Third-party consultants – mining input

Mining operating costs were estimated by Mining Plus and MineFill as per the WBS shown in Table 21.17. All mining operating costs are provided as a dollar cost input to the operating cost model with the exception of power consumption and fuel consumption, which are provided in base units of kilowatt-hours and litres, respectively. Fluor consolidated the relevant cost information into the operating cost model in consultation with Mining Plus.

Table 21.17 Mining WBS input

WBS	WBS description	Cost centre
10/12/20	Open pit mining	Open pit mining
22/24/27/28	Underground mining	Underground mining
26	Backfill plant (underground scope)	Underground mining

21.2.1.15 Third-party consultants – non-mining input

Third-party consultants provided estimates for non-mining operating costs related to their scope of work. Fluor consolidated the cost information into the operating cost model in consultation with the relevant third parties.

Table 21.18 provides a summary of third-party consultant information that was input to the operating cost estimate.

Table 21.18 Non-mining third-party consultant input

Participant	WBS specifics	Input category
Golder WTP	65 Water Treatment Plant	Headcount Power consumption Operating consumables Maintenance consumables
Golder Tailings Compaction	61 Filtration Plant	Headcount and labour rates Fuel consumption Maintenance consumables Operating contracts
Golder Water System	10 Open Pit 12 Open Pit Facilities 53 Borehole Water System 62 IEWMF 63 Dams Reclaim Water System 68 Off-Site Water Management: Reinjection Arrays, CSR / Pipeline Infrastructure, etc.	Headcount Power consumption Maintenance consumables
MineFill	26 Backfill Plant	Headcount and labour rates Maintenance consumables G&A

21.2.1.16 Mining operating costs

Open pit mining costs (see Table 21.19) were estimated from first principles by unit operation based on projected fleet requirements for an annual production schedule. Fleet requirements were calculated based on historical benchmarks of equipment productivities and haulage simulations. Labour requirements were developed to support the operation and maintenance of the fleet, and for the general operation of the mine. Equipment operating cost and fuel consumption were estimated from a combination of manufacturer data and consultant in-house data.

Underground mine operating costs (see Table 21.20) were calculated from first principles modelling of the consumables (ground support, explosives, services, cement, aggregates, fuel) and equipment required to meet the development and production schedule. The operating unit costs for mobile equipment and fuel consumption rates were largely obtained from manufacturers. Labour requirements were developed to support the operation and maintenance of the fleet and for the general operation of the underground mine.

Table 21.19 Open pit mining cost summary

Cost category	Production years total cost (US$)	Production years cost per tonne of open pit production ore mined (US$/t)
Power	-	-
Operating consumables	14,933,415	0.26
Maintenance consumables	57,009,378	0.99
Operating labour	69,257,573	1.20
Maintenance labour	20,153,087	0.35
Fuel	64,649,097	1.12
Operational contracts	-	-
G&A	18,812,836	0.33
Total	**244,815,386**	**4.24**

Note: Totals may not add up due to rounding.

Table 21.20 Underground mining cost summary

Cost category	Production years total cost (US$)	Production years cost per tonne of underground production ore mined (US$/t)
Power	230,373,399	2.65
Operating consumables	430,504,731	4.95
Maintenance consumables	283,723,888	3.26
Operating labour	480,665,019	5.52
Maintenance labour	111,514,201	1.28
Fuel	95,919,422	1.10
Operational contracts	5,350,392	0.06
G&A	42,973,952	0.49
Total	**1,681,025,005**	**19.32**

Note: Totals may not add up due to rounding.

21.2.1.17 Process plant facilities

The costs described in this section are those for the processing facilities at the plant site. The specific key areas included in this cost centre are as follows:

- Primary crushing
- Conveying to stockpile
- Coarse ore stockpile
- Grinding
- Flotation
- Concentrate dewatering and storage
- Concentrate handling and loadout
- Process plant reagents
- Rougher tailings pumping, classification, and paste tailings delivery to paste plant
- Tailings pumping and pipeline system

A summary of total operating costs excluding pre-production in the process plant facilities by cost category is provided in Table 21.21. Unit costs are presented in US$ per tonne of ore processed.

Table 21.21 Process plant operating cost summary

Cost category	Production years total cost (US$)	Production years cost per tonne of production ore processed (US$/t)
Power	613,494,411	4.20
Operating consumables	475,146,946	3.25
Maintenance consumables	62,300,393	0.43
Operating labour	51,557,334	0.35
Maintenance labour	44,748,199	0.31
Fuel	-	-
Operational contracts	-	-
G&A	-	-
Total	**1,247,247,282**	**8.54**

Note: Totals may not add up due to rounding.

21.2.2 Tailings filtration plant and dry stacking

The costs described in this section are for the tailings filtration plant. The specific key areas included in this cost centre are as follows:

- Tailings reception
- Tailings filtration
- Filtrate water management
- Tailings filter cake conveying
- Dry stacking

A summary of total operating costs excluding pre-production in the tailings filtration plant by cost category is provided in Table 21.22. Unit costs are presented in US$ per tonne of ore processed.

Table 21.22 Tailings filter plant and dry stacking operating cost summary

Cost category	Production years total cost (US$)	Production years cost per tonne of production ore processed (US$/t)
Power	131,849,751	0.90
Operating consumables	14,198,587	0.10
Maintenance consumables	36,683,370	0.25
Operating labour	81,226,297	0.56
Maintenance labour	17,562,910	0.12
Fuel	21,540,557	0.15
Operational contracts	11,239,008	0.08
G&A	-	-
Total	**314,300,479**	**2.15**

Note: Totals may not add up due to rounding.

21.2.3 Paste backfill plant

The costs described in this section are for the paste plant. The specific key areas included in this cost centre are as follows:

- Paste production
- Filtrate water discharge pump
- Filtrate water discharge piping

A summary of total operating costs excluding pre-production in the paste backfill plant by cost category is provided in Table 21.23. Unit costs are presented in US$ per tonne of ore processed.

Table 21.23 Paste backfill plant operating cost summary

Cost category	Production years total cost (US$)	Production years cost per tonne of production ore processed (US$/t)
Power	-	-
Operating consumables	-	-
Maintenance consumables	8,181,592	0.06
Operating labour	13,673,314	0.09
Maintenance labour	3,916,105	0.03
Fuel	-	-
Operational contracts	-	-
G&A	1,735,368	0.01
Total	**27,506,378**	**0.19**

Note: Totals may not add up due to rounding.

21.2.4 Water system

The costs described in this section include the following:

- Site water management system including ponds, basin drainage, reinjection wells and associated piping.
- Water treatment system.

A summary of total operating costs excluding pre-production in the water systems by cost category is provided in Table 21.24. Unit costs are presented in US$ per tonne of ore processed.

Table 21.24 Water system operating cost summary

Cost category	Production years total cost (US$)	Production years cost per tonne of production ore processed (US$/t)
Power	13,444,421	0.09
Operating consumables	887,854	0.01
Maintenance consumables	1,857,763	0.01
Operating labour	1,727,188	0.01
Maintenance labour	1,556,532	0.01
Fuel	-	-
Operational contracts	-	-
G&A	534,087	0.00
Total	**20,007,844**	**0.14**

Note: Totals may not add up due to rounding.

21.2.5 General & administration

G&A costs include costs not defined in any other categories and consist of both indirect and overhead costs. These costs have been estimated by Eldorado / Hellas Gold with input from Fluor as needed. Indirect costs are incurred only when the facility is in operation, while overhead costs are incurred regardless of whether the mine is operating.

Indirect costs include items such as:

- Consultant fees, including associated costs and allowances to consult with subject matter experts of related fields.
- Transportation costs, including associated lease costs for on- or off-site transportation.
- Communications, including network communications applications and technology such as tele-communications and radio systems on site.
- Analytical services contracts, including third-party assay laboratory and testing facilities for mining, quality assurance, umpire analysis, and equipment applications.
- Services contracts, including janitorial services, catering services, laundry services, cleaning, and maintenance services.
- Safety, environmental and security costs, including safety programs and gear required for all employees to ensure personal safety while performing duties, and including associated operating supply costs and security services provided on site.
- Other indirect costs, including equipment and tools such as computers and software, temporary facilities, and other general associated costs.

Overhead costs include items such as:

- Taxes, including property and facility.
- Insurance, including general liability and property insurance for the operating site.
- Rights of way and other land use fees.
- Community relations, including donations to public services, such as local schools and hospitals, and associated fees and costs for literature publications and maintaining public relations within the local community.
- Travel and entertainment costs, including lodging, transportation, and meal-related expenses.
- Financial, safety, and environmental audits.
- Legal fees.
- Third-party contracts, including facilities support, inspection, and maintenance.

A summary of estimated operating expenses relating to G&A services is presented in Table 21.25.

Table 21.25 G&A operating cost summary

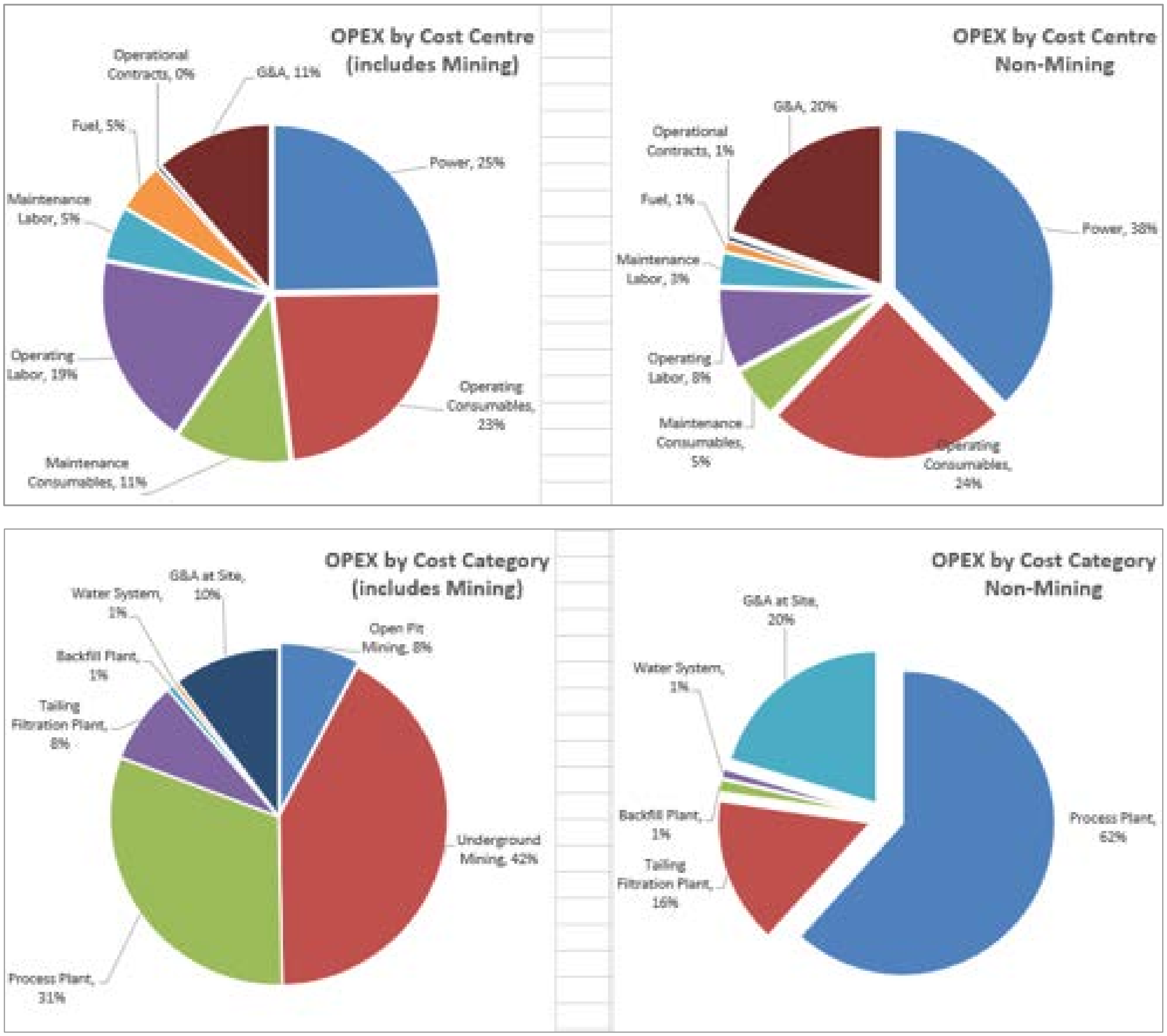
Cost category	Production years total cost (US$)	Production years cost per tonne of production ore processed (US$/t)
Power	-	-
Operating consumables	-	-
Maintenance consumables	-	-
Operating labour	18,724,136	0.13
Maintenance labour	-	-
Fuel	-	-
Operational contracts	-	-
G&A	390,415,534	2.67
Total	**409,139,670**	**2.80**

Note: Totals may not add up due to rounding.

21.2.6 Summary of operating costs

The estimated total operating costs by cost centre and cost category based on the total throughput of the plant are presented in Table 21.26 and Table 21.27. Unit costs are presented as total per tonne of ore processed. Figure 21.1 below shows the operating cost distribution.

Figure 21.1 Operating cost distribution



Source: Fluor 2022.

Table 21.26 Total production years operating cost summary

Cost category	Unit	Open pit mining	Underground mining	Process plant	Tailing filtration plant	Backfill plant	Water system	G&A at site	Total	Non-mining total
Power	US$	-	230,373,399	613,494,411	131,849,751	-	13,444,421	-	989,161,981	758,788,582
Operating consumables	US$	14,933,415	430,504,731	475,146,946	14,198,587	-	887,854	-	935,671,533	490,233,386
Maintenance consumables	US$	57,009,378	283,723,888	62,300,393	36,683,370	8,181,592	1,857,763	-	449,756,384	109,023,117
Operating labour	US$	69,257,573	480,665,019	51,557,334	81,226,297	13,673,314	1,727,188	18,724,136	716,830,861	166,908,268
Maintenance labour	US$	20,153,087	111,514,201	44,748,199	17,562,910	3,916,105	1,556,532	-	199,451,033	67,783,745
Fuel	US$	64,649,097	95,919,422	-	21,540,557	-	-	-	182,109,076	21,540,557
Operational contracts	US$	-	5,350,392	-	11,239,008	-	-	-	16,589,400	11,239,008
G&A	US$	18,812,836	42,973,952	-	-	1,735,368	534,087	390,415,534	454,471,777	392,684,989
Total	**US$**	**244,815,386**	**1,681,025,005**	**1,247,247,282**	**314,300,479**	**27,506,378**	**20,007,844**	**409,139,670**	**3,944,042,045**	**2,018,201,653**

Note: Totals may not add up due to rounding.

Table 21.27 Total production years operating cost per tonne of ore processed

Cost category	Unit	Open pit mining	Underground mining	Process plant	Tailing filtration plant	Backfill plant	Water system	G&A at site	Total	Non-mining total
Power	US$/t ore	-	2.65*	4.20	0.90	-	0.09	-	6.77	5.19
Operating consumables	US$/t ore	0.26*	4.95*	3.25*	0.10*	-	0.01	-	6.40	3.36
Maintenance consumables	US$/t ore	0.99*	3.26*	0.43	0.25	0.06	0.01	-	3.08	0.75
Operating labour	US$/t ore	1.20*	5.52*	0.35	0.56	0.09	0.01	0.13	4.91	1.14
Maintenance labour	US$/t ore	0.35*	1.28*	0.31	0.12	0.03	0.01	-	1.37	0.46
Fuel	US$/t ore	1.12*	1.10*	-	0.15	-	-	-	1.25	0.15
Operational contracts	US$/t ore	-	0.06*	-	0.08	-	-	-	0.11	0.08
G&A	US$/t ore	0.33*	0.49*	-	-	0.01	0.00	2.67	3.11	2.69
Total	**US$/t ore**	**4.24***	**19.32***	**8.54**	**2.15**	**0.19**	**0.14**	**2.80**	**26.99**	**13.81**

Note: Totals may not add up due to rounding.
*These averages were calculated according to the total tonnages mined for those years of production (open pit production ore mined of 57,772,918 t, underground production ore mined of 87,008,294 t, total production ore mined of 144,781,212 t, total production ore processed of 146,115,060 t).

22 Economic analysis

22.1 22.1 Cautionary statement

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to several known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- Mineral Resource and Mineral Reserve estimates.
- Assumed commodity prices and exchange rates.
- The proposed mine production plan.
- Projected mining and process recovery rates.
- Sustaining costs and proposed operating costs.
- Assumptions as to closure costs and closure requirements.
- Assumptions as to environmental, permitting and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what are estimated.
- Unrecognized environmental and social risks.
- Unanticipated reclamation expenses.
- Unexpected variations in quantity of mineralized material, grade, or recovery rate.
- Geotechnical or hydrogeological considerations during mining being different from what was assumed.
- Failure of mining methods to operate as anticipated.
- Failure of plant, equipment, or processes to operate as anticipated.
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis.
- Ability to maintain the social licence to operate.
- Accidents, labour disputes and other risks of the mining industry.
- Changes to interest rates.
- Changes to tax rates and incentive programs.

22.2 Methods, assumptions, and basis

The economic analysis is based on the Mineral Reserves as defined in Section 15, the mining methods and production schedule as expressed in Section 16, the recovery and processing methods as described in Section 17, and the capital and operating costs as outlined in Section 21.

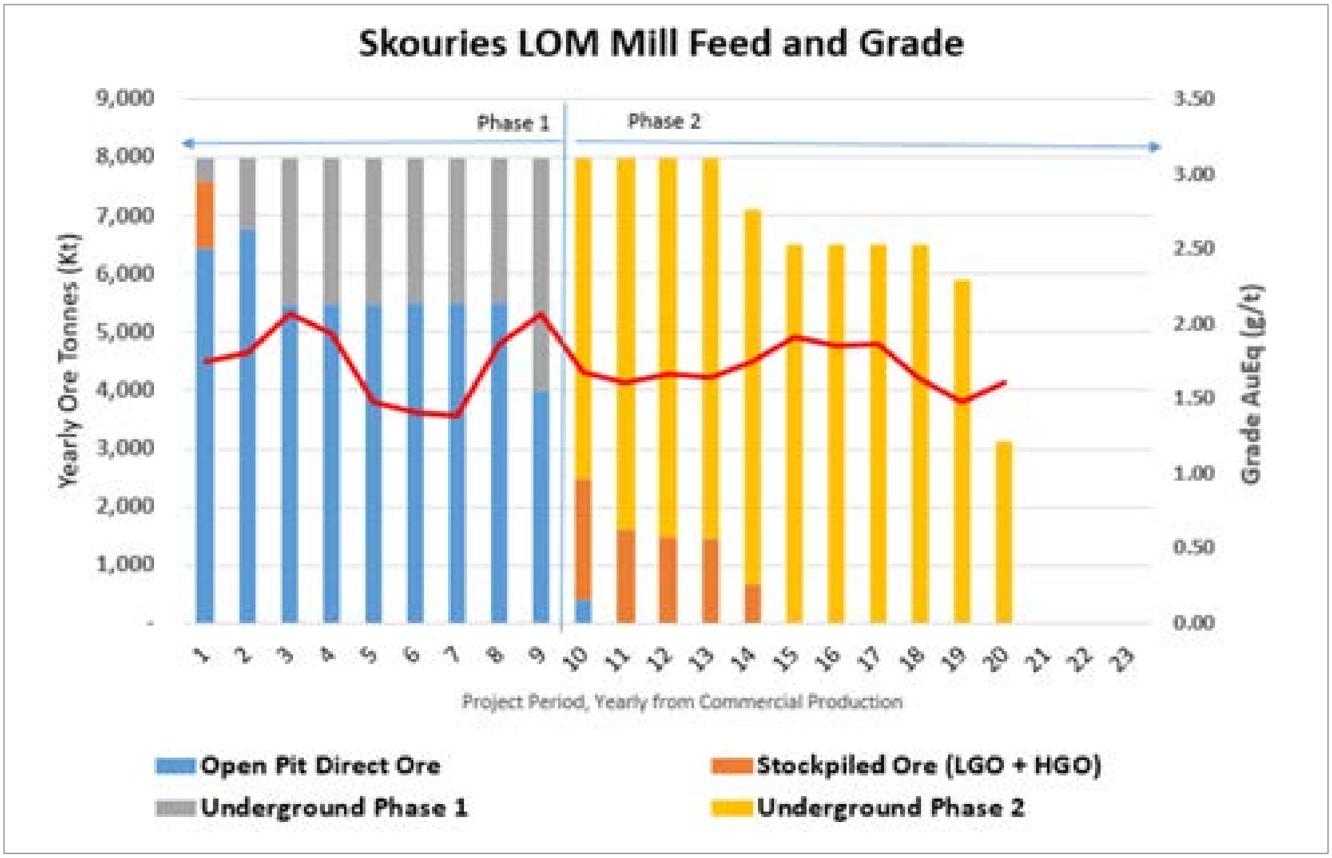
The Project case metal prices used in the economic model are US$1,500/oz Au and US$3.85/lb Cu. The economic model was also evaluated at the respective Mineral Reserve prices of US$1,300/oz and US$2.75/lb. The model makes use of a first principles build-up in Euros, with values then converted to US$. All reporting in this section is made in US$.

The model has been prepared on an annual basis for the LOM. The effective date of the estimate is assumed to be 1 January 2022. The remaining construction period is estimated to be 32 months to first ore including commissioning. The first year of commercial production is designated as Year 1. The LOM is 20 years from the start of commercial production until the depletion of Mineral Reserves.

22.3 Production schedule

A chart illustrating the mill production schedule is shown in Figure 22.1. The Project is configured in two phases, with Phase 1 considering largely open pit ore and deposition into the surface IEWMF; in Phase 2 the focus is on underground ore and tailings will be deposited into the mined out open pit.

Figure 22.1 Skouries LOM mill feed and gold equivalent grade



Source: MP 2022.

The Skouries mill will operate at the design capacity of 8.0 Mtpa over the first 13 years of the operation when both the underground and open pit ore are fed into the mill. After that, the throughput is reduced to an average of approximately 6.5 Mtpa for a further 6 years, as the capacity of the stand-alone Phase 2 underground mine becomes the limiting factor. In the last year of the operation the throughput is reduced to a rate of approximately 3 Mtpa as the underground mining sequence proceeds to the extraction of the final remnants of the Skouries ore.

22.4 Cash flows

The annual cash flow forecast is built from a first principles financial model. The model results are shown in Table 22.1 and Table 22.2, with the latter giving key details of the mine production schedule, operating and capital costs, mill production, offsite costs, and projected net cashflows over the project life and reported by Phase. Yearly numbers may not compute exactly to LOM totals due to rounding.

The after-tax cash flow analysis shows that the Skouries Project provides a robust return on the remaining capital to complete the Project scope and bring the Project into commercial production.

An internal rate of return (IRR) of 19% on an after-tax basis is achieved with the project case metal prices of US$1,500/oz Au and US$3.85/lb Cu. Using those metal prices, the net present value (NPV) of the Project is estimated to be US$1,273M using a discount rate of 5%, with a payback of the remaining capital expenditure achieved in 3.7 years from the start of commercial production. The NPV is calculated based on mid-year discounting starting in Year -3.

Table 22.1 Key economic results

Skouries project	**Unit**	**Value**
Total UG Ore	kt	87,519
Total OP Ore	kt	59,596
Total OP Waste	kt	59,224
Total Ore Milled	kt	147,115
Gold Grade	g/t	0.77
Copper Grade	%	0.50
Gold contained	koz	3,628
Copper contained	lbs M	1,629
Gold recovered and payable	%	81
Copper recovered and payable	%	87
Payable Gold produced	koz	2,949
Payable Copper produced	lbs M	1,411
Revenue split by commodity	Gold	44.9
Revenue split by commodity	Copper	55.1
Gross revenue	US$M	9,853
Capital costs	US$M	1,863
Operating costs (total)	US$M	3,944
Transport, treatment and refining costs	US$M	400
Royalties	US$M	193
Mine operating costs	US$/t	13.37
Processing costs	US$/t	10.82
G&A	US$/t	2.80
Operating costs (total)	US$/t	26.99
Payback period from start of Commercial Production	Years	3.7
Net pre-tax cash flow	US$M	3,393
Net post-tax cash flow	US$M	2,726
Post-tax NPV @5% discount rate	US$M	1,273
Post-tax NPV @8% discount rate	US$M	788
Post-tax IRR	%	19.0

Table 22.2 Skouries LOM production and cash flow forecast

			Project year and project phase												Project year and project phase											Closure
	Units	LOM totals	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
			Pre-production			Phase 1									Phase 2											
Production schedule																										
Open pit																										
Direct ore mined	kt	51,483	-	-	900	6,434	6,772	5,493	5,477	5,486	5,514	5,505	5,506	4,006	390	-	-	-	-	-	-	-	-	-	-	-
Au grade	g/t	0.61	0.00	0.00	0.78	0.88	0.79	0.74	0.69	0.29	0.36	0.40	0.51	0.69	0.89	-	-	-	-	-	-	-	-	-	-	-
Cu grade	%	0.42%	0.00%	0.00%	0.42%	0.56%	0.49%	0.45%	0.44%	0.28%	0.33%	0.33%	0.39%	0.47%	0.52%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Gold contained	koz	1,007	-	-	23	182	172	131	122	51	63	72	90	89	11	-	-	-	-	-	-	-	-	-	-	-
Cu contained	M lb	477	-	-	8	80	73	55	53	34	40	40	47	42	4	-	-	-	-	-	-	-	-	-	-	-
Deferred ore mined (HGO + LGO)	kt	8,114	-	60	863	1,967	1,853	1,373	1,998	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Au grade	g/t	0.32	0.00	0.53	1.22	0.25	0.21	0.19	0.19	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu grade	%	0.26%	0.00%	0.31%	0.57%	0.24%	0.22%	0.21%	0.21%	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%	0.00%	0.00%	0.00%	0.00%
Gold contained	koz	84	-	1	34	16	12	8	12	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu contained	M lb	46	-	0	11	10	9	6	9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total open pit waste	kt	59,224	5,690	8,809	5,942	7,733	7,075	7,456	6,903	4,950	2,952	1,347	303	66	-	-	-	-	-	-	-	-	-	-	-	-
Underground phase 1																										
Ore mined	kt	21,155	27	181	302	404	1,228	2,507	2,523	2,514	2,486	2,495	2,494	3,994	-	-	-	-	-	-	-	-	-	-	-	-
Au grade	g/t	1.30	0.46	0.68	0.36	0.45	1.13	1.61	1.47	1.47	1.12	1.01	1.62	1.24	-	-	-	-	-	-	-	-	-	-	-	-
Cu grade	%	0.68%	0.34%	0.51%	0.30%	0.36%	0.59%	0.78%	0.72%	0.74%	0.62%	0.59%	0.82%	0.68%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Gold contained	koz	886	0	4	3	6	45	130	120	118	90	81	130	159	-	-	-	-	-	-	-	-	-	-	-	-
Cu contained	M lb	318	0	2	2	3	16	43	40	41	34	32	45	60	-	-	-	-	-	-	-	-	-	-	-	-
Waste mined	kt	2,893	168	18	413	311	77	7	1	108	488	577	537	187	-	-	-	-	-	-	-	-	-	-	-	-
Total mined	kt	23,852	-	199	716	715	1,304	2,515	2,524	2,622	2,973	3,072	3,031	4,181	-	-	-	-	-	-	-	-	-	-	-	-
Underground phase 2																										
Ore mined	kt	66,364	-	-	-	-	-	-	-	-	-	-	-	-	5,505	6,375	6,500	6,531	6,443	6,491	6,503	6,496	6,496	5,909	3,115	-
Au grade	g/t	0.77	-	-	-	-	-	-	-	-	-	-	-	-	0.97	0.82	0.86	0.81	0.82	0.83	0.78	0.78	0.63	0.54	0.59	-
Cu grade	%	0.54%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.57%	0.54%	0.54%	0.54%	0.54%	0.56%	0.55%	0.56%	0.51%	0.47%	0.51%	0.00%
Gold contained	koz	1,652	-	-	-	-	-	-	-	-	-	-	-	-	172	169	179	171	170	173	163	163	132	102	59	-
Cu contained	M lb	787	-	-	-	-	-	-	-	-	-	-	-	-	69	76	78	78	77	80	79	80	73	62	35	-
Total waste tonnes	kt	122	-	-	-	-	-	-	-	-	-	-	-	-	49	50	12	5	4	-	-	0	2	-	-	-
Total tonnes	kt	66,486	-	-	-	-	-	-	-	-	-	-	-	-	5,554	6,425	6,512	6,536	6,447	6,491	6,503	6,496	6,498	5,909	3,115	-
Total mill feed																										
Open pit direct ore	kt	51,483	-	-	900	6,434	6,772	5,493	5,477	5,486	5,514	5,505	5,506	4,006	390	-	-	-	-	-	-	-	-	-	-	-
Stockpiled ore	kt	8,624	-	-	100	1,163	-	-	-	-	-	-	-	-	2,105	1,624	1,500	1,469	664	-	-	-	-	-	-	-
Underground phase 1	kt	20,644	-	-	-	404	1,228	2,507	2,523	2,514	2,486	2,495	2,494	3,994	-	-	-	-	-	-	-	-	-	-	-	-
Underground phase 2	kt	66,364	-	-	-	-	-	-	-	-	-	-	-	-	5,505	6,375	6,500	6,531	6,443	6,491	6,503	6,496	6,496	5,909	3,115	-
Ore milled	kt	147,115	-	-	1,000	8,000	8,000	8,000	8,000	8,000	8,000	8,000	8,000	8,000	8,000	7,999	8,000	8,000	7,107	6,491	6,503	6,496	6,496	5,909	3,115	-
Au grade	g/t	0.77	-	0.00	0.75	0.89	0.84	1.01	0.94	0.66	0.60	0.59	0.86	0.96	0.77	0.70	0.74	0.70	0.76	0.83	0.78	0.78	0.63	0.54	0.59	-
Cu grade	%	0.50%	-	0.00%	0.40%	0.55%	0.51%	0.55%	0.53%	0.43%	0.42%	0.41%	0.52%	0.57%	0.48%	0.47%	0.48%	0.49%	0.51%	0.56%	0.55%	0.56%	0.51%	0.47%	0.51%	0.00%
Mill feed grade AuEq	g/t	1.72	-	0.00	1.12	1.75	1.81	2.07	1.94	1.48	1.40	1.38	1.86	2.07	1.67	1.61	1.66	1.64	1.75	1.92	1.86	1.87	1.63	1.47	1.61	-
Gold contained	koz	3,628	-	-	24	228	217	261	241	169	153	153	220	248	197	180	190	181	175	173	163	163	132	102	59	-
Cu contained	M lb	1,629	-	-	9	97	89	98	93	76	74	73	92	101	84	84	85	86	80	80	79	80	73	62	35	-
Gold recovery total	%	83.3%	-	0%	66.9%	77.8%	84.2%	86.1%	85.5%	82.6%	81.9%	81.8%	84.8%	85.7%	83.6%	82.9%	83.3%	83.0%	83.7%	84.5%	84.0%	84.0%	82.3%	81.1%	81.8%	0.0%

	Units	LOM totals	Project year and project phase												Project year and project phase											Closure
			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
			Pre-production			Phase 1									Phase 2											
Operating costs																										
Mining costs	US$M	1,953.3	-	-	-	51.6	82.7	98.2	90.0	88.7	79.2	77.8	75.8	93.1	109.5	119.1	119.2	116.3	115.3	115.4	115.5	114.8	115.5	107.1	68.4	-
Processing costs	US$M	1,581.6	-	-	-	90.5	86.2	86.0	86.0	86.0	86.0	86.0	86.0	86.0	84.7	84.7	84.8	83.6	77.0	70.4	70.5	70.5	70.5	65.4	40.7	-
General + admin	US$M	409.1	-	-	-	25.9	25.3	19.6	19.5	19.3	18.5	18.3	18.7	18.4	18.7	18.3	18.3	18.4	18.3	18.6	18.6	22.6	24.7	24.7	24.7	-
Total operating costs	US$M	3,944.0	-	-	-	168.0	194.2	203.9	195.5	194.0	183.7	182.2	180.6	197.4	212.9	222.1	222.3	218.3	210.5	204.3	204.6	207.9	210.7	197.2	133.8	-
Total operating costs	$/t ore	26.99	-	-	-	21.00	24.28	25.49	24.44	24.25	22.97	22.77	22.57	24.68	26.61	27.77	27.78	27.28	29.62	31.48	31.47	32.00	32.43	33.37	42.95	-
Capital costs																										
Development capital (Pre-production)	US$M	844.6	179.0	443.8	221.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Development capital (Phase 2 underground	US$M	171.9	-	-	-	-	-	-	22.9	22.4	47.0	41.3	33.7	3.2	1.3	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital costs	US$M	846.6	-	-	-18.9	133.6	131.7	21.7	21.5	39.7	45.9	55.9	66.1	71.0	36.1	34.4	26.1	23.9	36.9	31.9	24.0	18.5	16.3	11.8	18.7	-
Total capital	US$M	1863.1	179.0	443.8	202.8	133.6	131.7	21.7	44.4	62.1	92.9	97.2	99.8	74.3	37.4	34.4	26.1	23.9	36.9	31.9	24.0	18.5	16.3	11.8	18.7	-
Metals produced and offsite costs																										
Concentrate production																										
Copper recovery to concentrate	%	90.0%	0.00%	0.00%	35.4%	70.0%	90.8%	93.1%	92.5%	89.8%	89.5%	89.2%	92.4%	93.5%	90.4%	90.5%	90.8%	90.8%	91.8%	93.3%	93.1%	93.2%	92.0%	91.1%	92.2%	0.0%
Gold recovered to concentrate	koz	3,024	-	-	16.1	177.2	182.9	224.6	206.3	139.8	125.4	124.9	186.8	212.5	164.9	149.2	158.1	150.3	146.1	145.7	136.5	136.7	108.6	83.0	48.5	-
Copper in concentrate	M lb	1,467	-	-	3.1	67.7	81.2	91.0	86.0	67.8	66.2	64.7	85.3	94.6	76.0	75.8	77.3	77.7	73.5	74.8	73.9	74.5	67.0	56.1	32.6	-
Dry tonnage of Concentrate	kt	2,560	-	-	5.5	118.2	141.8	158.8	150.1	118.3	115.5	113.0	148.9	165.1	132.6	132.2	134.9	135.6	128.2	130.6	128.9	130.1	116.8	97.9	56.8	-
Wet Tonnage of concentrate	kt	2,790	-	-	6.0	128.8	154.5	173.1	163.6	129.0	125.9	123.1	162.3	180.0	144.6	144.1	147.0	147.8	139.8	142.3	140.5	141.8	127.4	106.7	61.9	-
Payable copper	kt	640	-	-	1.37	29.55	35.44	39.71	37.52	29.58	28.88	28.24	37.23	41.29	33.16	33.06	33.73	33.89	32.06	32.64	32.23	32.52	29.21	24.46	14.20	-
Payable gold	koz	2949	-	-	15.68	172.77	178.31	218.95	201.15	136.29	122.29	121.75	182.09	207.20	160.80	145.45	154.11	146.57	142.45	142.09	133.09	133.31	105.88	80.96	47.33	-
Treatment charges	US$M	211	-	-	0.45	9.75	11.70	13.10	12.38	9.76	9.53	9.32	12.29	13.62	10.94	10.91	11.13	11.19	10.58	10.77	10.63	10.73	9.64	8.07	4.69	-
Transport charges	US$M	55	-	-	0.12	2.52	3.02	3.38	3.20	2.52	2.46	2.41	3.17	3.52	2.83	2.82	2.87	2.89	2.73	2.78	2.75	2.77	2.49	2.09	1.21	-
Cu refining fee	US$M	116	-	-	0.25	5.37	6.44	7.22	6.82	5.38	5.25	5.13	6.77	7.51	6.03	6.01	6.13	6.16	5.83	5.94	5.86	5.91	5.31	4.45	2.58	-
Au refining fee	US$M	18	-	-	0.10	1.06	1.10	1.35	1.24	0.84	0.75	0.75	1.12	1.28	0.99	0.90	0.95	0.90	0.88	0.87	0.82	0.82	0.65	0.50	0.29	-
Metals produced																										
Total payable gold produced	koz	2,949	-	-	15.7	172.8	178.3	219.0	201.2	136.3	122.3	121.8	182.1	207.2	160.8	145.5	154.1	146.6	142.4	142.1	133.1	133.3	105.9	81.0	47.3	-
Total payable copper produced	M lb	1,411	-	-	3.0	65.1	78.1	87.5	82.7	65.2	63.7	62.2	82.1	91.0	73.1	72.9	74.3	74.7	70.7	71.9	71.0	71.7	64.4	53.9	31.3	-
Total payable gold equivalent produced	koz	6,569	-	-	23.4	339.9	378.8	443.6	413.4	303.6	285.7	281.5	392.7	440.8	348.4	332.5	344.9	338.3	323.8	326.8	315.4	317.3	271.1	219.4	127.7	-
Annual revenues																										
Gold price	$/oz	1,500	1500	1500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	-
Copper price	$/lb	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	3.85	-
Gold revenue	US$M	4,423	-	-	23.5	259.2	267.5	328.4	301.7	204.4	183.4	182.6	273.1	310.8	241.2	218.2	231.2	219.9	213.7	213.1	199.6	200.0	158.8	121.4	71.0	-
Copper revenue	US$M	5,430	-	-	11.6	250.8	300.7	336.9	318.3	251.0	245.1	239.6	315.9	350.3	281.4	280.5	286.2	287.6	272.1	277.0	273.5	275.9	247.9	207.6	120.5	-
Total gross revenue	US$M	9,853	-	-	35.1	509.9	568.2	665.4	620.1	455.4	428.5	422.3	589.1	661.1	522.6	498.7	517.3	507.5	485.7	490.1	473.1	475.9	406.7	329.0	191.5	-
Royalties																										
Gold royalty	%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	2.20%	0.00%
Copper royalty	%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	1.65%	0.00%
Gold royalty	US$M	99.8	-	-	0.53	5.85	6.04	7.41	6.81	4.61	4.14	4.12	6.16	7.01	5.44	4.92	5.22	4.96	4.82	4.81	4.50	4.51	3.58	2.74	1.60	-
Copper royalty	US$M	93.2	-	-	0.20	4.30	5.16	5.78	5.46	4.31	4.21	4.11	5.42	6.01	4.83	4.81	4.91	4.94	4.67	4.75	4.69	4.73	4.25	3.56	2.07	-

			Project year and project phase												Project year and project phase											Closure
	Units	**LOM totals**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**
			Pre-production			**Phase 1**									**Phase 2**											
After-tax cash flows																										
Net revenue																										
Metal sales	US$M	9,853.2	-	-	35.1	509.9	568.2	665.4	620.1	455.4	428.5	422.3	589.1	661.1	522.6	498.7	517.3	507.5	485.7	490.1	473.1	475.9	406.7	329.0	191.5	-
Treatment, transport, and refining	US$M	-400.3	-	-	-0.9	-18.7	-22.3	-25.1	-23.6	-18.5	-18.0	-17.6	-23.4	-25.9	-20.8	-20.6	-21.1	-21.1	-20.0	-20.4	-20.1	-20.2	-18.1	-15.1	-8.8	-
Royalties	US$M	-193.0	-	-	-0.7	-10.2	-11.2	-13.2	-12.3	-8.9	-8.3	-8.2	-11.6	-13.0	-10.3	-9.7	-10.1	-9.9	-9.5	-9.6	-9.2	-9.2	-7.8	-6.3	-3.7	-
Net revenue	US$M	9,260.0	-	-	33.5	481.1	534.8	627.1	584.2	428.0	402.2	396.4	554.1	622.2	491.5	468.3	486.1	476.4	456.2	460.2	443.8	446.4	380.8	307.6	179.1	-
NSR	$/t ore	62.9	-	-	33.5	60.1	66.8	78.4	73.0	53.5	50.3	49.6	69.3	77.8	61.4	58.5	60.8	59.6	64.2	70.9	68.3	68.7	58.6	52.1	57.5	-
Earnings																										
Net revenue	US$M	9,226.5	-	-	-	481.1	534.8	627.1	584.2	428.0	402.2	396.4	554.1	622.2	491.5	468.3	486.1	476.4	456.2	460.2	443.8	446.4	380.8	307.6	179.1	-
Operating costs	US$M	-3,944.0	-	-	-	-168.0	-194.2	-203.9	-195.5	-194.0	-183.7	-182.2	-180.6	-197.4	-212.9	-222.1	-222.3	-218.3	-210.5	-204.3	-204.6	-207.9	-210.7	-197.2	-133.8	-
Depreciation	US$M	-2,249.2	-	-	-	-107.5	-117.6	-127.5	-129.2	-132.4	-137.0	-143.7	-150.6	-157.6	-162.8	-165.4	-167.8	-169.6	-171.3	21.8	-50.9	-42.9	-48.7	-45.9	-42.4	-
Taxes	US$M	-667.3	-	-	-	-45.2	-49.0	-65.1	-57.1	-22.3	-17.9	-15.5	-49.1	-58.8	-25.5	-17.8	-21.1	-19.5	-16.4	-61.1	-41.4	-43.0	-26.7	-14.2	-0.6	-
Net earnings	US$M	2,366.0	-	-	-	160.3	173.9	230.6	202.4	79.2	63.5	55.0	173.9	208.4	90.3	63.0	74.9	69.0	58.1	216.6	146.9	152.6	94.7	50.4	2.2	-
Cash flow																										
Net pre-tax cash flow	US$M	3,392.8	-179.0	-443.8	-202.8	179.5	208.8	401.5	344.2	171.9	125.6	117.1	273.8	350.5	225.4	211.8	237.8	234.3	208.9	223.9	215.2	220.0	153.8	98.7	26.6	-10.7
Net earnings	US$M	2,366.0	-	-	-	160.3	173.9	230.6	202.4	79.2	63.5	55.0	173.9	208.4	90.3	63.0	74.9	69.0	58.1	216.6	146.9	152.6	94.7	50.4	2.2	-
+ Depreciation	US$M	2,249.2	-	-	-	107.5	117.6	127.5	129.2	132.4	137.0	143.7	150.6	157.6	162.8	165.4	167.8	169.6	171.3	-21.8	50.9	42.9	48.7	45.9	42.4	-
- Capital	US$M	-1,863.1	-179.0	-443.8	-202.8	-133.6	-131.7	-21.7	-44.4	-62.1	-92.9	-97.2	-99.8	-74.3	-37.4	-34.4	-26.1	-23.9	-36.9	-31.9	-24.0	-18.5	-16.3	-11.8	-18.7	-
- Closure cost	US$M	-26.6	-	-	-	-	-	-	-	-	-	-	-	-	-15.9	-	-	-	-	-	-	-	-	-	-	-10.7
Net after tax cash flow	US$M	2,725.5	-179.0	-443.8	-202.8	134.2	159.7	336.4	287.2	149.6	107.6	101.6	224.7	291.7	199.9	194.0	216.6	214.8	192.5	162.9	173.8	177.0	127.1	84.5	26.0	-10.7
Payback	**years**	**3.7**																								
IRR	**%**	**19.0%**																								
NPV (5%)	**US$M**	**1,273.4**																								
NPV (8%)	**US$M**	**788.5**																								

22.5 Royalties and other fees

The Skouries Project is subject to a mineral production royalty regime that has a sliding scale NSR type of royalty payable to the Greek government.

The relevant projected royalty rates for copper and gold are shown in Table 22.3 and Table 22.4. The royalties are calculated on the payable metals that are produced by the site.

The royalty regime incorporates a sliding scale dependent on the metal price on the date of sale. For gold, the royalty ranges from 0% to 6.6% and, for copper, the royalty ranges from 0% to 2.75%.

For the Project case economics reported in this section, the gold royalty is assumed to be 2.2%, and the copper royalty is assumed to be 1.65%. In the case of the sensitivity analysis described in Section 22.10, the royalty corresponds to the metal prices used in the respective case.

Table 22.3 Gold royalty

Gold price (€/oz)		Gold price (US$/oz)		NSR royalty
From	To	From	To	
0	600	0	720	0.00%
600	900	720	1080	1.10%
900	1100	1080	1320	1.65%
1100	1300	1320	1560	2.20%
1300	1500	1560	1800	3.30%
1500	1650	1800	1980	4.40%
1650	1800	1980	2160	5.50%
1800	1800+	2160	2160+	6.60%

Table 22.4 Copper royalty

Copper price (€/tonne)		Copper price (US$/lbs)		NSR royalty
From	To	From	To	
0	5000	0	2.72	0.00%
5000	5800	2.72	3.16	0.55%
5800	6600	3.16	3.59	1.10%
6600	7400	3.59	4.03	1.65%
7400	8200	4.03	4.46	2.20%
8200	8200+	4.46	5.44	2.75%

22.6 Closure and salvage value

Closure costs are captured by the economic model in the form of an ARO that is offset by progressive rehabilitation as discussed in Section 20. The Phase 1 surface IEWMF operation includes progressive rehabilitation with re-grading of tailings, hauling and placing of waste rock and topsoil during Year 10 of the operation. These costs are accounted for in the financial model as operating costs since they will not be depreciated. The total progressive rehabilitation costs for the surface IEWMF in Phase 1 are estimated to be US$15.9M. Additionally, the closure of the open pit includes backfilling with tailings during Phase 2 of the operation; these costs are also included in operating costs.

Additional costs for dismantling the processing plant, ancillary buildings, powerlines, and roads are estimated to be the residual of the ARO left once progressive rehabilitation has been subtracted. The estimate used in the economic model is US$10.7M as the cost net of salvage value.

No allowance has been made in the financial model for the carrying costs of the ARO guarantee.

22.7 Taxation

Value added taxation (VAT) is redeemable in Greece for all operating and capital spending incurred on mining projects. As such the VAT component of any quotations or other costs used in this economic analysis have been removed. This implies that the VAT costs will be redeemed without delay and as they are incurred. There exists a risk that the responsible Greek authorities will delay the reimbursement of VAT, which could have a material effect on the timing of cashflows from the Project.

Corporate income tax rates in Greece are 22% of net earnings. Income from operations can be offset by operating costs and by depreciation of capitalized items according to a schedule of depreciation based on the type of asset. The depreciation schedule based on the type of asset is shown in Table 22.5.

Table 22.5 Depreciation rates for Greek corporate income tax

Type of asset	Depreciation rate
Land	0%
Buildings	4%
Mining Excavations	5%
Mining Equipment	10%
Mechanical Infrastructure	10%
Construction Expenses	Allocated to a direct cost
Other / Indirect Costs	Allocated to a direct cost
Capitalized Interest Payments	Allocated to a direct cost

22.8 Financing costs

Costs of financing the Project, such as interest on loans, are not included in the economic model. The Project is assumed to be funded by Hellas Gold, and any costs or charges relating to Eldorado's funding of the Hellas Gold subsidiary are beyond the scope of the analysis.

22.9 Third party interests

Hellas Gold is the 100% owner of the Kassandra mines, which includes the Skouries Project.

All data provided in this report is shown at the 100% ownership level.

22.10 Sensitivity analysis

The economic model was subjected to a sensitivity analysis to determine the effects of changing metal prices, operating costs and capital costs on the Project financial returns. The results of the sensitivity analysis are provided in Table 22.6 to Table 22.9.

A test of economic extraction for the Skouries Mineral Reserves is demonstrated by means of this sensitivity analysis. At the Mineral Reserve metal prices of US$1,300/oz Au and US$2.75/lb Cu the Project shows positive economics. The after-tax IRR is 9.8% and the NPV is estimated to be

US$354M using a 5% discount rate, with a calculated payback period of 8.1 years from start of Commercial Production.

The sensitivity analysis shows that the Project is most sensitive to the metal prices, followed by operating costs and then capital costs. The copper concentrate grade is the least sensitive. The sensitivity ranges shows that the Project is also robust when evaluated using lower metal price assumptions, or higher operating and capital costs. Positive cash flows and positive NPV are maintained at metal prices of US$1,125/oz Au and US$2.89/lb Cu (except for when the NPV is discounted at 8%), operating and capital cost increased by 25% individually, or concentrate grade reduced by 25%.

Table 22.6 Metal price sensitivity analysis

		Sensitivity ranges					
Parameters	**Units**	**Reserve case**	**-25%**	**-12.5%**	**Project case**	**+12.5%**	**+25%**
Gold price	US$/oz	1300.00	1,125.00	1,312.50	**1,500.00**	1,687.50	1,875.00
Copper price	US$/lb	2.75	2.89	3.37	**3.85**	4.33	4.81
Results (after tax)							
NPV 0%	US$M	1,104	834	1,818	**2,726**	3,596	4,451
NPV 5%	US$M	354	195	755	**1,273**	1,772	2,261
NPV 8%	US$M	105	-16	401	**788**	1,161	1,526
IRR%	%	9.8	7.7%	14.1%	**19.0%**	23.4%	27.3%
Payback period	years	8.1	8.8	5.3	**3.7**	3.1	2.7
Taxation	US$M	253	209	417	**667**	913	1,154
Royalties	US$M	87	79	120	**193**	308	444

Table 22.7 Capital cost sensitivity analysis

		Sensitivity ranges				
Parameter	**Units**	**-25%**	**-12.5%**	**Project case**	**+12.5%**	**+25%**
LOM capital costs	US$M	1,397	1,630	**1,863**	2,096	2,329
Results (after tax)						
NPV 0%	US$M	3,100	2,913	**2,726**	2,538	2,349
NPV 5%	US$M	1,578	1,426	**1,273**	1,121	968
NPV 8%	US$M	1,064	926	**788**	651	512
IRR	%	26.4	22.3	**19.0%**	16.3	14.1

Table 22.8 Operating cost sensitivity analysis

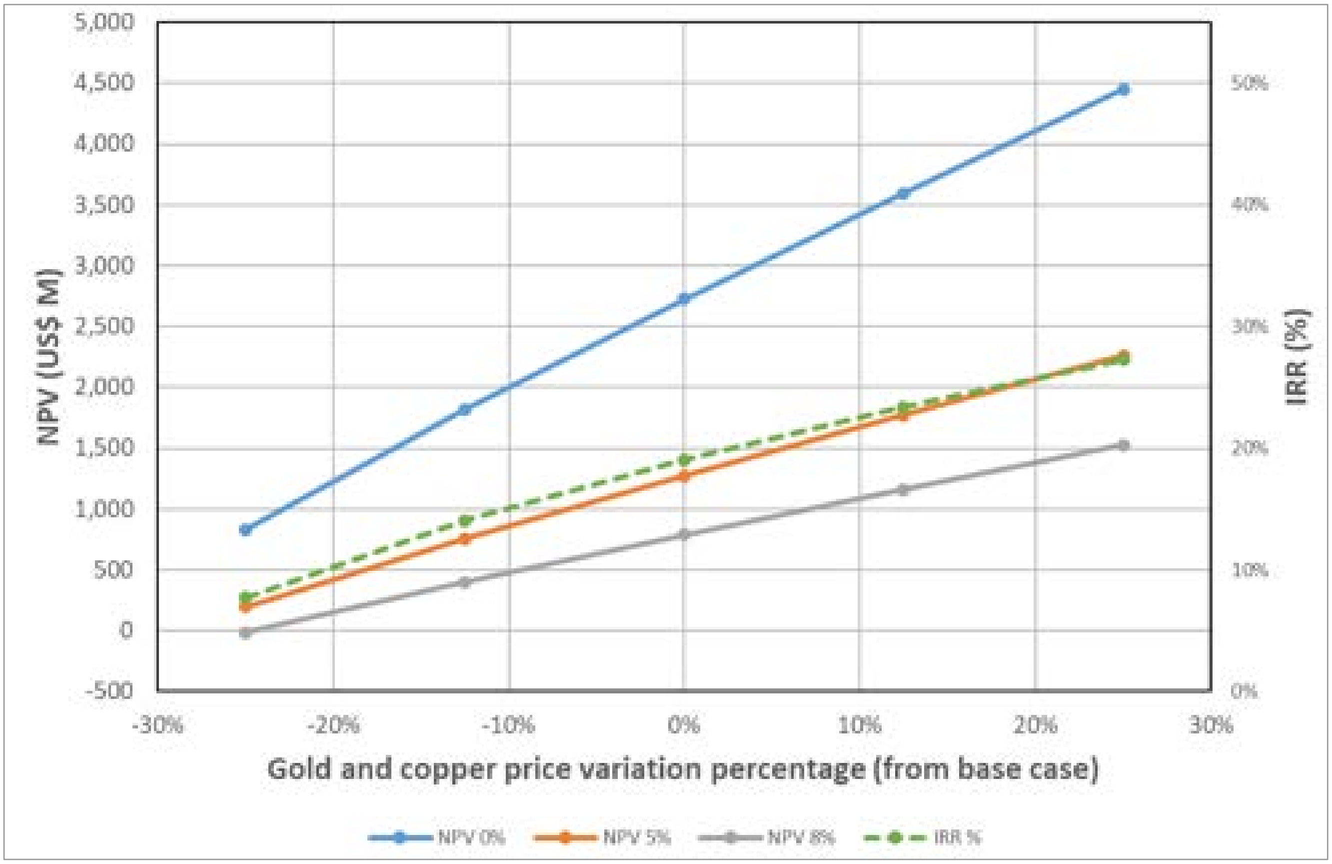
		Sensitivity ranges				
Parameter	**Units**	**-25%**	**-12.5%**	**Project case**	**+12.5%**	**+25%**
LOM operating costs	US$/t ore	20.24	23.62	**26.99**	30.36	33.74
Results (after tax)						
NPV 0%	US$M	3,495	3,110	**2,726**	2,338	1,950
NPV 5%	US$M	1,696	1,484	**1,273**	1,061	849
NPV 8%	US$M	1,097	943	**788**	634	478
IRR	%	22.4	20.8	**19.0**	17.2	15.3

Table 22.9 Concentrate grade sensitivity analysis

		Sensitivity ranges				
Parameter	**Units**	**-25%**	**-12.5%**	**Project case**	**+12.5%**	**+25%**
LOM operating costs	%Cu	19.5	22.75%	**26%**	29.25%	32.5%
Results (after tax)						
NPV 0%	US$M	2,601	2,672	**2,726**	2,767	2,800
NPV 5%	US$M	1,203	1,243	**1,273**	1,297	1,315
NPV 8%	US$M	736	766	**788**	806	820
IRR	%	18.4	18.8	**19.0**	19.2	19.4

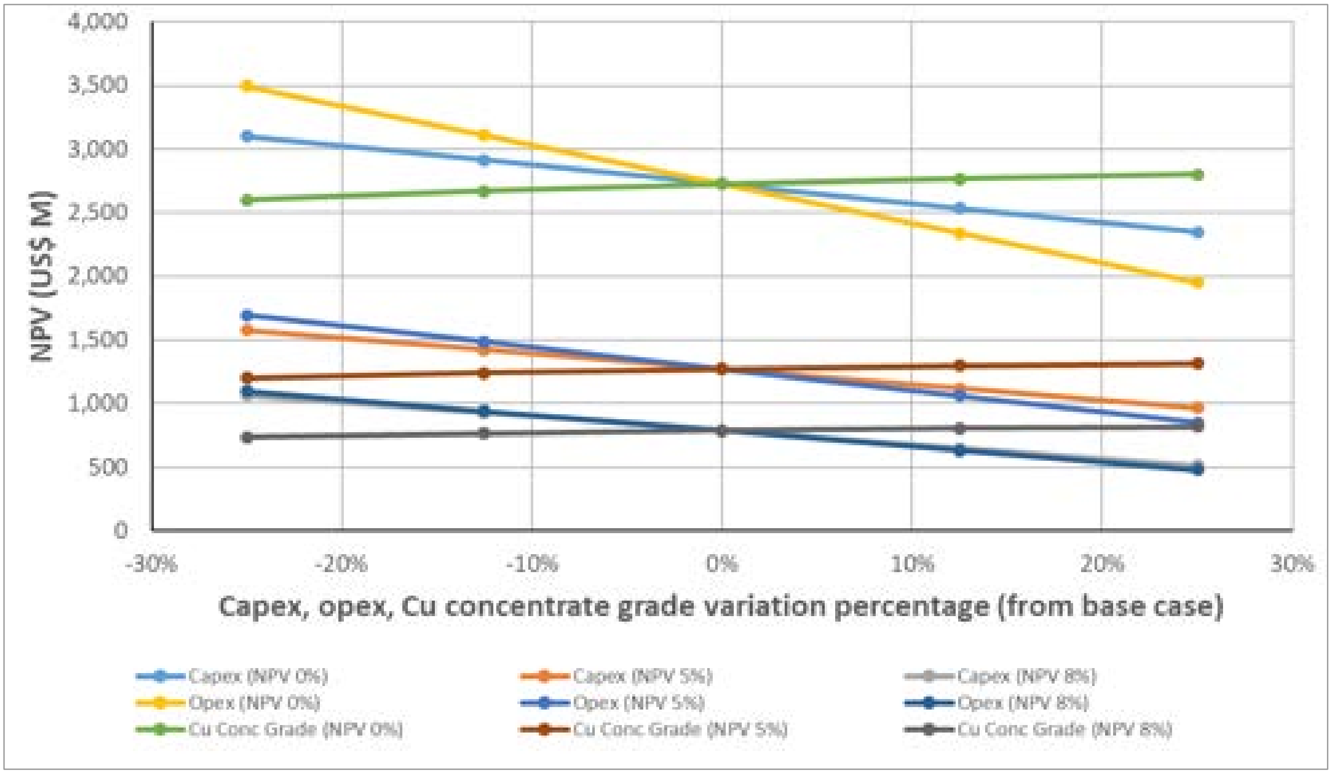
Note: Sensitivity plots for the metal price and the sensitivity to capital expenditure (capex), opex, and copper concentrate grade varied by ±25% are provided in Figure 22.2 to Figure 22.4.

Figure 22.2 Sensitivity plot for metal price analysis



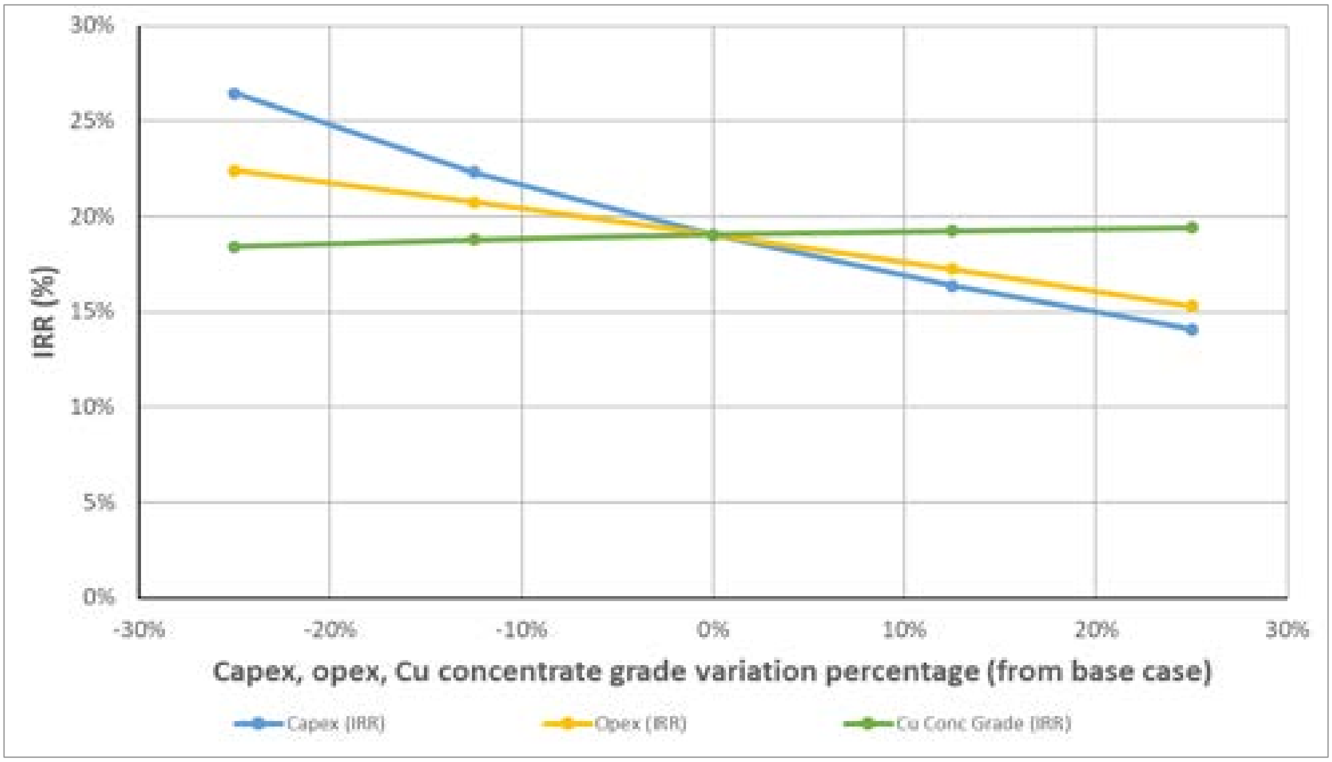
Source: AMC 2022.

Figure 22.3 NPV sensitivity plot for capital costs, operating costs, and copper concentrate grade



Source: AMC 2022.

Figure 22.4 IRR sensitivity plot for capital costs, operating costs, and copper concentrate grade



Source: AMC 2022.

23 Adjacent properties

The Property is located within the Kassandra Mines complex which is comprised of a group of Hellas Gold mining and exploration concessions, covering 317 km^2. The other properties within the complex are Olympias, which is a producing mine, and Stratoni, which is currently on care and maintenance. They are both in a different geological environment but will have some logistical ties to Skouries.

24 Other relevant data and information

24.1 Economics

The Skouries Project has been under construction since 2012. The capital costs incurred to the end of 2021 are sunk costs that have not been included in the capital cost estimate. These sunk costs are used in the economic evaluation described in Section 22, as they form a portion of depreciable assets used in estimating net earnings and tax payable.

The sunk capital costs are used in the estimate of depreciation in accordance with Greek tax law. An average depreciation rate of 7.1% has been estimated to apply to these costs over the LOM.

25 Interpretation and conclusions

It is concluded that the Skouries work completed to date, including exploration, site development and study work leading to current Mineral Resource and Mineral Reserve estimates, has demonstrated the strong technical and economic viability of the Project.

25.1 Permitting

Permitting of the Project is well advanced and while there was a period of delay in granting of permits which resulted in the Project being placed on care and maintenance in 2017, the outstanding permits are now in place.

An IA that was ratified in early 2021 amends the 2003 Transfer Agreement and provides a modernized legal and financial framework to allow for the advancement of Eldorado's investment in Skouries. After the 2019 Greek Parliamentary elections, when Eldorado initiated talks with the newly established government in regard to this IA, outstanding routine permits were released.

25.2 Mineral Resources

Resource information and data used in the preparation of the Mineral Resource estimates were obtained from historic TVX data. This was verified and supplemented by information from a surface diamond drill campaign undertaken by Eldorado and Hellas Gold. The Mineral Resource estimates are consistent with the CIM Definitions Standards (2014) referred to in NI 43-101. It is the opinion of the QPs that the information and analysis described in this report are sufficient for reporting Mineral Resources and hence Mineral Reserves.

Results of drilling indicate that the orebody is open at depth, with potential for Inferred Resources to be converted to Indicated Resources through further exploration. This is considered an opportunity for the Project, and further exploration at depth should be completed during operations.

25.3 Mineral Reserves

The Mineral Reserves at Skouries comprise an open pit and an underground component. The Mineral Reserves for the open pit deposit have been evaluated at a US$10.60/t NSR cut-off value. The cut-off used in the plan is US$10.62/t ore, this difference is negligible and does not affect the Mineral Reserve estimation. The Proven and Probable Mineral Reserves are 59.6 Mt with an average grade of 0.57 g/t Au and 0.40% Cu.

The underground contribution to Mineral Reserves has been evaluated at a diluted NSR cut-off of US$33.33/t, incorporating unplanned diluting material by weight of 5% in porphyry stopes and 5.5% in schist stopes that is assumed to carry no metal value, and assuming an overall mining recovery of 95%.

The underground Mineral Reserves have been estimated to be 87.5 Mt with an average grade of 0.90 g/t Au and 0.57% Cu. The Skouries project has included some material that is below the full breakeven cut-off values of US$37.49 and US$34.42 for Phase 1 and 2 respectively; this accounts for 4.1% of tonnes, 2.2% of Au and 2.9% of Cu, and is not considered material to the Mineral Reserves estimation.

25.4 Mining

The Skouries Project is designed as a two-phase mining operation. Phase 1 consists of a combined open pit and underground mine, operating over nine years. Phase 2 consists of mining from the underground mine for a further 11 years. The total LOM is 20 years.

Phase 1 total mill feed is 8.0 Mtpa, consisting of a nominal 5.5 Mtpa from the open pit mine combined with approximately 2.5 Mtpa from the underground mine. At the start of the mine life, during the initial two-year underground mine ramp-up period, the open pit feed rate is variable in order to maintain the 8.0 Mtpa mill feed. During Phase 1, 7.2 Mt of low-grade oxide ore is stockpiled to be rehandled for mill feed during Phase 2. Phase 1 is completed at the end of the open pit mine life in Year 9.

Phase 2 mine production, from Year 10 to the end of the LOM, is provided from the underground mine. Phase 2 mine development begins in Year 4 in order to allow a seamless ramp-up from the Phase 1 production of 2.5 Mtpa. During the first three years of Phase 2, the mill feed rate of 8.0 Mtpa is maintained by reclaiming low grade ore stockpiled during Phase 1, at a rate that balances the mill feed to 8.0 Mtpa through Year 12. From Year 13 on, Phase 2 mill feed is maintained at a nominal feed rate of 6.5 Mtpa, solely from underground mine production.

Open pit mining will be by a conventional truck-shovel operation. The mining sequence will consist of drilling, blasting, loading and hauling of ore and waste materials for processing and waste disposal. Direct feed ore (greater than US$18.0 NSR) from the open pit will be hauled to the Skouries processing plant by a fleet of 90 t trucks. During Phase 1, a portion of low grade ores will be hauled to the LGOS for future Phase 2 processing. Waste material will be hauled to an adjacent transfer point by 60 t trucks for the LOM.

SLOS has been confirmed as the most appropriate underground mining method for the Project and was incorporated into the EIS / JMD approval in 2011 and subsequent submission in 2021. Standard stope dimensions of 65 m high x 30 m long x 15 m wide for primary porphyry and all secondary stopes, and 65 m high x 20 m long x 15 m wide for primary stopes in schist have been designed. Production stopes will be backfilled with cemented paste fill. The stoping methodology is the same for both Phase 1 and Phase 2.

The higher mining rate in Phase 2 of the underground mine will be supported by an underground materials handling system that will supply crushed ore to a shaft, which will in turn feed the mill via conveyor. Waste will continue to be hauled by truck.

25.5 Metallurgical recovery

Significant metallurgical testwork and analysis has been completed to confirm the process designs and substantiate projected recoveries. Eldorado has reviewed and validated historical data obtained from EGL and completed additional confirmatory testwork. The QP has a high degree of confidence in the process design and the projected recoveries.

25.6 Infrastructure

The principal waste streams generated from the Project are overburden and waste rock from the open pit mining and underground development, and tailings from mineral processing operations. Overburden and waste rock will be stored on surface; tailings will be used underground as paste backfill, with excess being stored on surface. The Project mine plan and material balance have been developed such that the majority of overburden and waste rock are used for construction requirements. The remaining material will be placed in a waste rock dump located upstream from the IEWMF eliminating the need for a permanent waste rock dump.

The water within the Project site can be classified as contact water and non-contact water. Non-contact water is groundwater and surface water that is diverted around the mine facilities without being exposed to mine infrastructure. Contact water includes groundwater and surface water that has been exposed to mine infrastructure, as well as process water.

The general results of the SWWB show that excess contact water is expected to be generated, especially in Phase 1, and will need to be managed with on-site storage capacity and water treatment prior to discharge to the groundwater aquifer through reinjection wells. During Phase 2, reduced amounts of both non-contact and contact water are expected.

The main access road connects the process plant and mining area with the national road network. The major regional centre of Thessaloniki is approximately 80 km away and is accessed by highway EO 16. Thessaloniki has an international airport and one of Greece's largest seaports.

The Skouries Project site substation is fed from a new overhead 6 km long 150 kV transmission line connected to the national power grid. The high voltage substation constructed for the Skouries Project has a power capacity of 51 MW.

25.6.1 IEWMF Risk Assessment

GAL facilitated a qualitative risk assessment for the IEWMF, with participants from Hellas Gold / Eldorado, Fluor, and Golder Associates USA Inc (GAUSA). The purpose of the risk assessment was to identify major risks to the project related to the 2019 IEWMF design prepared by Omikron Kappa Consulting S.A. (OKC, 2019) to inform the 2021 feasibility-level design being prepared by GAUSA. The scope of the assessment was limited to risks associated with the 2019 IEWMF design and potential consequences to safety and health, environment, community, corporate reputation, and legal and financial standing.

The assessment resulted in several design modifications to mitigate risks. Actions to address residual risks and gaps are addressed in Section 26. A risk assessment of the 2021 IEWMF feasibility level-design is recommended to define residual project risks.

25.6.2 Site-Wide Water Management

Excess contact water is expected at the project site, especially during Phase 1. A management strategy involving temporary on-site storage and treatment and discharge is included in the design to mitigate risk associated with excess contact water. The excess contact water management strategy accounts for limiting the amount and duration of water accumulation on the IEWMF to reduce the overall risk of the IEWMF.

Some residual risks associated with excess contact water management remain and should be further evaluated during the next project design stage. Some actions and opportunities to mitigate risks associated with management of excess contact water are provided in Section 26.

25.7 Metal sales

The Skouries processing plant will produce a gold-copper concentrate that is expected to be marketable to a large number of downstream smelters, refineries, traders, and sales agents.

25.8 Capital and operating costs and financial modelling

The accuracies of the cost estimates are consistent with the standards outlined by the AACE. The cost estimate is a feasibility-level estimate categorized as AACE Class 3.

Direct costs were developed from a combination of budget quotes, material take-offs, existing contracts, Project-specific references, and historical benchmarks. Indirects and owners' costs were estimated using a combination of existing commitments, calculated project requirements, and historical benchmarks. Contingency was applied to each cost item in the estimate, based on the level of engineering definition and reliability of its unit rates.

The capital cost estimate does not include sunk costs. Total capital cost is estimated to be US$1,863 M and total operating cost over the LOM is estimated to be US$3,944M. Projected cash flows indicate an IRR of 19% on an after-tax basis with the project case metal prices of US$1,500/oz Au and US$3.85/lb Cu. Using those metal prices, the NPV of the Project is estimated to be US$1,273M using a discount rate of 5%, with a payback of the remaining capital expenditure achieved in 3.7 years from the start of commercial production.

25.9 Environmental

The EIS concluded that, during construction and operations, there are site specific impacts. In general, however, the impacts are considered reversible through the use of best practices during construction and operations, and appropriate decommissioning and reclamation at the end of the Project. In the wider study area, there are negligible impacts on the environment or surrounding villages.

26 Recommendations

Study work and site development and construction completed to date have provided a strong technical and economic basis for proceeding with full development of the Skouries Project. It is recommended to continue with that development while undertaking the work described below. The activities involve optimization and confirmatory work that will not affect the development schedule as presented. The work is to be led by Skouries operations personnel and is recommended to be undertaken as part of future design, construction and operation activities. Cost estimates for the recommended work items are provided below where applicable, with the total cost estimated at US$2.7M.

26.1 Geology

When in favourable underground locations, diamond drilling should be carried out to refine the knowledge of the deposit and assist in design and planning. In addition, results of drilling indicate that the orebody is open at depth, with potential for Inferred Resources to be converted to Indicated Resources through further exploration. This is considered an opportunity for the Project, and further exploration at depth should be completed during operations. Costs for future drilling will be accounted for as part of ongoing operations planning.

26.2 Mineral Resource estimation – QA/QC

The QP makes the following recommendations regarding current pass / fail criteria for CRMs:

- Assay batches with two consecutive CRMs outside two SDs should be re-run, regardless of the side of the mean on which they fall.
- As CRM data accumulates over time, results should be reviewed for biases in the data.
- For duplicate samples, these should constitute around 5% of the samples submitted to the laboratory.
- Include pulp duplicates in addition to coarse duplicates in future QA/QC programs. Also consider modifying the sample preparation protocol to try and minimize the effect of readily liberated gold grains throughout the sample preparation process.
- Include samples being sent to an external laboratory in any future QA/QC program.

These activities and associated costs will be part of normal operations.

26.3 Mining

The QP makes the following recommendations for the open pit mine:

- Perform geotechnical assessments on the Open Pit and Capping Rock Dump 1 areas up to an FS level to determine interaction risks.
- Rainfall impacts on pit and waste roads to be recorded during pre-production years to better inform maintenance costs and productivity differences.

The QP makes the following recommendations for the underground mine:

- In situ stress measurements to be taken once development has reached more competent rock in the general area in which test stoping has been proposed.
- Maintain dimensions of 15 m wide x 30 m long as the primary stope design basis until actual stope conditions are experienced and understood.
- Maintain recommended ground support design parameters, with ultimate proving to be achieved in the field.
- Perform confirmatory testwork on Skouries tailings for the backfill paste.

- Perform rheology testwork on tailings to verify the expected paste friction losses for the system.

The above activities and associated costs will be part of normal operations.

26.4 Engineering

Engineering for a large portion of the Project has been done at a detailed level. Further detailed engineering and procurement for the underground mine, filtration plant, IEWMF and water management is required for completion of Project development. It is recommended that these activities be undertaken in parallel with development and construction work on site so that Project implementation is not delayed.

The cost to complete this work is already included in the FS capital estimate.

26.4.1 IEWMF and ancillary facilities

Engineering and design for the IEWMF and ancillary facilities (WMP1, WMP2, LGO stockpile, cofferdam, and waste dump no. 1) in support of the Feasibility Study has been advanced to support this Technical Report; however, approximately 90% of the supporting documents have only been completed in draft format (with final completion pending Eldorado and third-party review), and the remainder are under development (with scheduled completion by February 2022).

Subsequent to the Feasibility Study, more in-depth third-party reviews and risk assessments are recommended, including a FMEA risk assessment of the feasibility-level design, a formal constructability review, safety in design review, and a review by a geotechnical or Independent Technical Review Board (ITRB).

The cost to complete this work is already included in the FS capital estimate.

26.4.2 Water management design and water balance

Engineering and design for the non-contact water diversion channels and IEWMF spillways and water balance modelling in support of the Feasibility Study have been advanced to support this technical report; however, approximately 90% of the supporting documents have only been completed in draft format (with final completion pending Eldorado and third-party review), and the remainder are under development (with scheduled completion by February 2022).

Excess contact water is expected at the project site, especially during Phase 1 of the project. A management strategy including temporary on-site storage, and treatment and discharge is included in the design to mitigate risk associated with excess contact water. However, management of excess contact water remains a potential risk for the project. The following should be evaluated during the next project stage to further reduce the risk associated with excess contact water:

- Opportunities to increase diversion of non-contact water groundwater as part of mine dewatering.
- Opportunities to increase on-site water storage capacity.
- Opportunities to optimize alignment of some water management structures, such as IEWMF spillways (e.g., move spillways to north side of the IEWMF).

The cost to complete this work is already included in the FS capital estimate.

26.4.3 Water treatment plant

The WTP concept, design and associated capital and operating cost estimates have been developed to a level that is consistent with the site water balance assessment and level of geochemical development for the mine-influenced water quality projections. The site water balance has direct impact on the hydraulic flexibility and sizing for most equipment. The inclusion of two treatment trains is directly related to the flow ranges that must be managed at the WTP. The robust multi-stage treatment system is based on the geochemical projections and will produce treated water that is compliant with water quality standards for both reinjection and surface water discharge. More than half of the unit processes included on the WTP PFD are included to remove only two contaminants (molybdenum and selenium). Geochemical testwork planned for 2022 may show that selenium and / or molybdenum do not require removal and then the system PFD can be revised and simplified.

There are other trade-off evaluations and optimizations that should be carried out to potentially reduce costs and improve operational efficiency. These include the following:

- VFDs have been used rather than control valves. A trade-off study should consider whether the cost savings in equipment relative to a control valve is warranted with the increased labour for installation versus the operational power cost savings.
- The selection of construction materials for tanks and pumps should be reviewed and a trade-off study completed to verify the optimal selection (plastic versus steel, carbon steel versus stainless steel or lined carbon steel).
- The general arrangement should be optimized with regard to location of the filter press loadout overhead doors and chemical delivery loadout in relation to the service road.
- It may be beneficial to locate the electrical room in a corner of the building to minimize the cost of bringing the power feed to the building. This must be balanced with other demands and costs within the building.
- Evaluation of tank and pump sizes should be made to provide common sizing and spare parts for optimization as the design progresses. Tank elevations can be adjusted to provide common elevations for access platforms and pipe supports.
- Modularization of certain equipment should be investigated to reduce on-site construction requirements and provide schedule advantages.

The cost to complete this work is already included in the FS capital estimate.

26.4.4 Pipeline and pumping systems

Engineering design of the pipeline and pumping systems has been completed and final material take-offs are being prepared to support the Feasibility Study. Existing topo discrepancies will need to be resolved during detailed design. The feasibility level designs have used what may be seen as a conservative approach assuming a worst-case scenario.

After finalizing the water balance modelling and tailings management strategy, the following can be further reviewed to optimize the pipeline and pumping scope at the detailed design stage:

- Selection of materials of construction for buffer tanks (steel versus concrete).
- Selection of pump types for specific services (vertical turbine pump versus submersible pump).
- Optimization of pipeline sizing, routing, and alignment.
- Optimization of electrical room design (use common electrical room to the fullest extent and modularize / fabricate electrical rooms off site to achieve on-site “plug-and-play” strategy).

The cost to complete this work is already included in the FS capital estimate.

26.5 Site investigations

The QPs recommend and support the work that Hellas Gold has indicated it will continue to complete on site during development and operations, including:

- Geotechnical investigations of the shaft and ventilation raises.
- Completion of exploration and studies aimed at increasing available Mineral Resources through expansion of the underground mine at depth.
- Optimization of development to enable test stoping to be completed as soon as possible.
- Optimization of the mine plan with respect to owner supplied and contractor fleet for waste haulage.
- Filtered tailings and LGO fills, in-place densities.

These investigations and studies are for the purposes of optimization and confirmation, and do not affect the production schedule as presented. The studies will be completed by the Skouries operations personnel and costs are included in the current cost estimate.

26.5.1 Geotechnical investigations

A geotechnical investigation program is planned to support characterization of foundation conditions for the IEWMF and ancillary facilities, including the filtration plant area. The proposed geotechnical investigation program includes 29 drillholes within the IEWMF footprint and 31 drillholes within the ancillary facilities. Piezometers will be installed in selected drillholes to better characterize phreatic conditions within the bedrock foundation. Samples will be collected and tested for geotechnical properties.

Estimated cost to complete this work is US$1.2M.

26.5.2 Tailings testing

A geotechnical testing program by Golder is planned on three tailings composites after additional filtration testing is conducted. The program will include the following:

- Standard index and classification testing.
- Permeability testing.
- Strength testing (critical state testing, consolidated drained and undrained triaxial tests, cyclic direct shear, and Bender element tests).
- Soil-water characteristics curve (SWCC) testing to determine the unsaturated characteristics of the tailings.

Mineralogical testing will be conducted by X-ray diffraction. This testing program will be conducted to help mitigate gaps in the pre-feasibility and feasibility studies in support of the detailed design. This program is scheduled for Q1-Q2 2022 and is estimated to cost approximately US$135,000.

26.5.3 Geochemical testing

A summary of the results of the gap analysis conducted for the Feasibility Study (GAL, 2021) is provided below:

- Additional geochemical characterization data (static and kinetic) is required for selected materials (filtered tailings, cemented paste backfill, oxide waste materials, and oxide ore to be stockpiled). This data can only be obtained through sampling and analyses of representative materials.
- Quantitative correlation is lacking between geological lithological units and the mine plan waste categories and the IEWMF design categories (zones of materials). This is a major data

gap and critical to answering questions about material spatial and compositional representativity and to develop geochemical source-terms that reflect the actual design of the waste and water facilities. This data gap can be addressed through professional consulting time and use of existing client resources (geologists, mine planner / mining engineer, block model interrogation, etc.).

- Additional samples of waste rock materials are required to improve spatial and compositional representativity of waste rock units. These data can only be obtained through sampling and analyses of core materials that are representative of these materials.

A proposed geochemical program has been planned to include the following:

- Step 1: Scope and conduct the quantitative correlation between geological lithological units, block model, mine plan waste categories, and waste facility material categories.
- Step 2: Develop / confirm and execute the geochemical characterization of additional geological materials (includes sampling plan, sampling, static and kinetic testing).
- In parallel with Step 2, develop geochemical source-terms using available geochemical data from Skouries, supplemented by relevant data from analogous sites (Olympias, Mavres Petres, Madem Lakkos) and using conservative assumptions. These source-terms can be used in the interim and updated once the proposed characterization program is completed.

The geochemical program is scheduled for Q1-Q3 2022 and is estimated to cost approximately US$300,000.

More geochemical characterization, data, and the development of detailed source-terms will not necessarily lead to potentially reduced treatment requirements, although it is reasonable to expect this, since conservative assumptions in the 2017 water quality predictions were generally used.

26.5.4 Treatability studies

Treatability studies are recommended to confirm chemical doses, residuals produced, and effluent quality. These treatability studies can be conducted in conjunction with the geochemical testing conducted by Hellas Gold operations personnel at Skouries or nearby operating mine site. Standard jar testing protocols should be used to confirm high-stage and low-stage treatment pH, lime dose, iron dose, acid dose, and sludge produced. Treated water quality can also be evaluated during this testing.

These activities and associated costs will be part of normal operations.

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28 QP Certificates

CERTIFICATE OF AUTHOR

I, John Morton Shannon, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as General Manager and Principal Geologist with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 This certificate applies to the Technical Report titled "Technical Report, Skouries Project, Greece" with an effective date of 22 January 2022 (the "Technical Report"), prepared for Eldorado Gold Corporation ("the Issuer").

3 I am a graduate of Trinity College Dublin in Dublin, Ireland (BA Mod Nat. Sci. in Geology in 1971). I am a member in good standing of the Engineers and Geoscientists British Columbia (Registration #32865) the Association of Professional Geoscientists of Ontario (Registration #0198). I have practiced my profession continuously since 1971, and have been involved in mineral exploration and mine geology for over 45 years since my graduation from university. This has involved working in Ireland, Zambia, Canada, and Papua New Guinea. My experience is principally in base metals and precious metals and have been Chief Geologist on two very large mines for major companies, with responsibility for all geological aspects of the operation.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Skouries Property on 28 May 2019 for one day.

5 I am responsible for Sections 2 - 12, 14, 23 and parts of 1, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 22 January 2022
Signing Date: 31 March 2022

Original signed and sealed by

John Morton Shannon, P.Geo.

General Manager / Principal Geologist

AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF AUTHOR

I, Gary Methven, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Principal Mining Engineer and Underground Manager with AMC Mining Consultants (Canada) Ltd. with an office located at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4.

2 This certificate applies to the Technical Report titled "Technical Report, Skouries Project, Greece" with an effective date of 22 January 2022 (the "Technical Report"), prepared for Eldorado Gold Corporation ("the Issuer").

3 I graduated from the University of Witwatersrand in Johannesburg, South Africa with a Bachelor of Science degree in Mining Engineering in 1993. I am a registered member in good standing with Engineers and Geoscientists British Columbia (License #180019), a member of Registered Professional Engineers of Queensland (License #06839), and a member of the Australian Institute of Mining and Metallurgy (CP). I have experience in narrow-vein gold deposits, flat and steeply dipping, bulk and selective mining methods for base metals, mine infrastructure, design and planning, mine production and financial evaluation, reserve estimation, technical reviews, feasibility and pre-feasibility studies, project and construction management, contracts management and cost estimation.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have not visited the Skouries Property.

5 I am responsible for Sections 20, 22, 24 and parts of 1, 15, 16, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 22 January 2022
Signing Date: 31 March 2022

Original signed and sealed by

Gary Methven, P.Eng.

Underground Manager / Principal Mining Engineer

AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF AUTHOR

I, John Battista, MAusIMM (CP), of Perth, Western Australia, do hereby certify that:

1 At the time of acting as QP for this project, I was employed as Principal Mining Consultant with Mining Plus UK Limited, with an office at 13-14 Orchard Street, Bristol BS1 5EH, United Kingdom.

2 This certificate applies to the Technical Report titled "Technical Report, Skouries Project, Greece" with an effective date of 22 January 2022 (the "Technical Report"), prepared for Eldorado Gold Corporation ("the Issuer").

3 I am a graduate of Western Australian School of Mines in Kalgoorlie, WA, Australia (Bachelor of Engineering degree in Mining Engineering in 1988). I am a member and Chartered Professional (Mining) in good standing of the Australian Institute of Mining and Metallurgy (Membership Number 105584). I have practiced my profession in mining continuously since 1989 and have worked on mining related precious and base metal projects in Oceania, North America, South America, Africa, Europe, and Asia.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Skouries Property from 15 – 16 September 2021 for 2 days.

5 I am responsible for parts of Sections 1, 15, 16, 25, and 26 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 22 January 2022
Signing Date: 31 March 2022

Original signed and sealed by

John Battista, MAusIMM (CP)
(formerly) Principal Mining Consultant
Mining Plus

CERTIFICATE OF AUTHOR

I, Mo Molavi, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Director / Mining Services Manager / Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 This certificate applies to the Technical Report titled "Technical Report, Skouries Project, Greece" with an effective date of 22 January 2022 (the "Technical Report"), prepared for Eldorado Gold Corporation ("the Issuer").

3 I am a graduate from Laurentian University in Sudbury, Canada (Bachelor of Engineering in 1979) and McGill University of Montreal, Canada (Master of Engineering in Rock Mechanics and Mining Methods in 1987). I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of Saskatchewan (License #5646), the Engineers and Geoscientists British Columbia (License #37594), and a Member of the Canadian Institute of Mining, Metallurgy and Petroleum. I have worked as a Mining Engineer for a total of 43 years since my graduation from university and have relevant experience in project management, feasibility studies, and technical report preparations for mining projects.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Skouries Property on 7 - 9 September 2016.

5 I am responsible for parts of Sections 1, 16, 18, 25, and 26 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 22 January 2022
Signing Date: 31 March 2022

Original signed and sealed by

Mo Molavi, P.Eng.
Director / Mining Services Manager / Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF AUTHOR

I, Dell Maeda, P.Eng., of Richmond, British Columbia, do hereby certify that:

1 I am currently employed as an Engineering Manager with Fluor Canada Ltd. with an office at 1075 W. Georgia St., Suite 700, Vancouver, British Columbia V6E 4M7.

2 This certificate applies to the Technical Report titled "Technical Report, Skouries Project, Greece" with an effective date of 22 January 2022 (the "Technical Report"), prepared for Eldorado Gold Corporation ("the Issuer").

3 I am a graduate of University of BC in Vancouver, Canada (Bachelor / Master of Applied Science in 1987). I am a member in good standing of the Association of Engineers and Geoscientitist British Columbia (License #17,470). I have experience in Materials and Mechanical Engineering, as well as Engineering Management and Project Management.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Skouries Property from 3 - 5 November 2021 for 3 days.

5 I am responsible for Section 21 and parts of 1, 18, 25, and 26 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 22 January 2022
Signing Date: 31 March 2022

Original signed and sealed by

Dell Maeda, P. Eng.
Engineering Manager
Fluor Canada Ltd.

CERTIFICATE OF AUTHOR

I, Richard Kiel, P.E., of Lakewook, Colorado, do hereby certify that:

1 I am currently employed as Director, Geological / Civil Engineer with Golder Associates USA Inc., with an office at Suite 200, 7245 W. Alaska Drive, Lakewood, Colorado 80226.

2 This certificate applies to the Technical Report titled “Technical Report, Skouries Project, Greece” with an effective date of 22 January 2022 (the “Technical Report”), prepared for Eldorado Gold Corporation (“the Issuer”).

3 I am a graduate of the South Dakota School of Mines and Technology with a B.S degree in Geological Engineering in 1979, and am a member of the Society for Mining, Metallurgy & Exploration (SME), and a registered professional civil engineer. My relevant experience after graduation and over the last 40 years for the purposes of the Technical Report include engineering, design, and construction support for mine waste facilities including tailings dams.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Skouries Property from 28 June to 1 July 2021 for a duration of four days.

5 I am responsible for Section 18.3 (exclusive of 18.3.1.1 and 18.3.1.2) and parts of 1, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 22 January 2022
Signing Date: 31 March 2022

Original signed and sealed by

Richard Kiel, P. E.
Director, Geological / Civil Engineer
Golder Associates USA Inc.

CERTIFICATE OF AUTHOR

I, Paolo Chiaramello, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Senior Water Resources Engineer with Golder Associates Ltd., with an office at Suite 200, 2920 Virtual Way, Vancouver, British Columbia V5M 0C4.

2 This certificate applies to the Technical Report titled "Technical Report, Skouries Project, Greece" with an effective date of 22 January 2022 (the "Technical Report"), prepared for Eldorado Gold Corporation ("the Issuer").

3 I am a graduate of Polytechnic of Turin (Italy) with an M.Eng. in Environmental Engineering degree, 2004. I'm a registered professional engineer in Italy (Province of Turin) and Canada (province of British Columbia, Northwest Territories and Nunavut). My relevant experience after graduation and over the last 18 years for the purposes of the Technical Report include water management planning and modeling and engineering design of water management structures.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have not visited the Skouries Property.

5 I am responsible for Section 18.4 and parts of 1, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property that is the subject of the Technical Report by working on water management planning and design during earlier stages of design of the Skouries Project.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 22 January 2022
Signing Date: 31 March 2022

Original signed and sealed by

Paolo Chiaramello, P. Eng.
Senior Water Resources Engineer
Golder Associates Ltd.

CERTIFICATE OF AUTHOR

I, Robert Chesher, FAusIMM (CP), RPEQ, MTMS, of Brisbane, Australia, do hereby certify that:

1 I am currently employed as a Technical Manager, Moscow / Principal Consultant with AMC Consultants Pty Ltd, with an office Level 21, 179 Turbot Street, Brisbane Qld 4000, Australia.

2 This certificate applies to the Technical Report titled "Technical Report, Skouries Project, Greece" with an effective date of 22 January 2022 (the "Technical Report"), prepared for Eldorado Gold Corporation ("the Issuer").

3 I am a graduate of University of Queensland in Saint Lucia, Australia (BA Science in Metallurgical in 1977). I am a Fellow in good standing of the Australian Institute of Mining and Metallurgy (AusIMM) and am accredited as a Chartered Professional of the AusIMM in the discipline of Metallurgy (License #311429). I am a Registered Professional Engineer of Queensland (RPEQ #24758). I have practiced my profession continuously since 1977. My expertise is in corporate and technical (metallurgical) consulting, focusing on operational and performance reviews, improvements, and optimization.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Skouries Property on 28 May 2019 for one day.

5 I am responsible for Sections 13, 17, 19 and parts of 1, 25, and 26 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 22 January 2022
Signing Date: 31 March 2022

Original signed and sealed by

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Technical Manager, Moscow / Principal Consultant
AMC Consultants Pty Ltd

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