Technical Report
Skouries Project
Greece

Centered on Latitude 40° 29'and Longitude 23°42'

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GLOSSARY

Units of Measure

Annum (year)	a
Billion	B
Centimeter	cm
Cubic centimeter	cm^3
Cubic meter	m^3
Day	d
Days per year (annum)	d/a
Degree	°
Degrees Celsius	°C
Dollar (American)	US$
Dollar (Canadian)	CAN$
Euro	€
Gallon	gal
Gram	g
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Horse Power	hp
Hour	h
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic meter	kg/m^3
Kilograms per hour	kg/h
Kilograms per square meter	kg/m^2
Kilometer	km
Kilometers per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilowatt hour	kWh
Kilowatt hours per year	kWh/a
Kilowatt	kW
Less than	<
Litre	L
Megawatt	MW
Meter	m
Meter above Sea Level	masl
Metric ton (tonne)	t
Microns	µm

Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimeter	mm
Million cubic meters	Mm^3
Million ounces	Moz
Million tonnes per Annum	Mtpa
Million tonnes	Mt
Million	M
Minute (time)	min
Month	mo
Newton	N
Ounce	oz
Parts per million	ppm
Pascal	Pa
Percent	%
Percent by Weight	wt%
Pound	lb
Quartz	Qz
Second (time)	sec
Square centimeter	cm^2
Square kilometer	km^2
Square meter	m^2
Thousand tonnes	kt
Three Dimensional	3D
Tonnes per day	t/d or tpd
Tonnes per hour	tph
Tonnes per year	tpa
Volt	V
Watt	W
Week	wk
Weight/weight	w/w

Abbreviations and Acronyms

Acid Base Accounting	ABA
Acid Neutralizing Capacity	ANC
Acidity or Alkalinity	pH
Aluminum	Al
Ammonium Nitrate/Fuel Oil	ANFO
Analytical Detection Limit	ADL
Antimony	Sb
Argon	Ar

Arsenic	As
Asset Retirement Obligation	ARO
Association for the Advancement of Cost Engineering	AACE
Atomic Adsorption	AA
Barium	Ba
Before Christ	BC
Biochemical Oxygen Demand	BOD
Cadmium	Cd
Carbon-in-leach	CIL
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Closure Capping Stockpile	CCS
Coefficient of Variance	CV
Counter-current decantation	CCD
Copper	Cu
Cyanide Amenability	CNA
Deep Cone thickeners	DCT
Distributed Control System	DCS
Semipure gold alloy	Doré
East	E
Eighty percent (80%) passing particle size, feed	F80
Eighty percent (80%) passing particle size, product	P80
Eldorado Gold Corporation	Eldorado
Engineering, Procurement, Construction Management	EPCM
Environmental Impact Assessment	EIA
Environmental Impact Statement	EIS
Environmental Management Plan	EMP
European Goldfields Ltd.	EGL
European Union	EU
Feasibility Study	FS
Flocculant	FLOC
Fresh Air Raise	FAR
Front End Loader	FEL
General and Administrative	G&A
Geological Strength Index	GSI
Ground-Engaging Tools	GET
Gold	Au
Gold Equivalent	Au Equiv
Gravity Recoverable Gold	GRG
High Density Polyethylene	HDPE
High Grade	HG
In Pit Tailings Management Facility	IPTMF
Inductively Coupled Plasma	ICP
Inflow Design Flood	IDF
Inner Diameter	ID

Integrated Waste Management Facility	IWMF
Intermediate Bulk Container	IBC
Internal Return Raise	IRAR
Internal Rate of Return	IRR
International Organization for Standardization	ISO
Iron	Fe
Joint Management Decision	JMD
KL	Karatza Lakkos
Lead	Pb
Lerchs-Grossman	L-G
Life-of-mine	LOM
Liquefied Propane Gas	LPG
Load Haul Dump	LHD
Low Grade	LG
Manganese	Mn
Measured & Indicated	M&I
Million Years	Ma
National Instrument 43-101	NI 43-101
Natural Gas	NG
Nearest Neighbour	NN
Net Present Value	NPV
Net Smelter Return	NSR
Nickel	Ni
North	N
North East	NE
North West	NW
Open Pit	OP
Ordinary Kriging	OK
Outer Diameter	OD
Oxide Ore Stockpile	OOS
Platinum Group Metals	PGM
Potassium	K
Prefeasibility Study	PFS
Probability Assisted Constrained Kriging	PACK
Programmable Logic Controllers	PLCs
Qualified Person(s)	QP(s)
Quality assurance	QA
Quality control	QC
Remote Mining Technology	RMT
Return Air Raise	RAR
Reverse Circulation	RC
Rock Mass Rating	RMR
Rock Quality Designation	RQD
Run of Mine	ROM

Scanning Electron Microscope	SEM
Selective Mining Unit	SMU
Selenium	Se
Semi-autogenous Grinding	SAG
Serbo-Macedonian Metallogenic Province	SMMP
Silicon	Si
Silver	Ag
Site Wide Water Balance	SWWB
Site Wide Water Management	SWWM
South	S
South East	SE
South Water Management Pond	SWMP
South West	SW
Specific Gravity	SG
Standard Reference Material	SRM
Static Secondary Ion Mass Spectrometry	SSIMS
Strategic Environmental Assessment	SEA
Strontium	Sn
Sub Level Caving	SLC
Sub Level Open Stoping	SLOS
Sulphide	S^{2-}
Sulphuric Acid	H_2SO_4
Tailings Management Facility	TMF
Technical Study	TS
Tellurium	Te
Total dissolved Solids	TDS
Total Suspended Solids	TSS
TVX Gold Inc.	TVX
Unconfined Compressive Strength	UCS
Underground	UG
Universal Transverse Mercador	UTM
Uranium	U
Value Added Tax	VAT
Variable Speed Drive	VSD
Waste Material Transfer Area	WMTA
Water Management Pond	WMP
Water Treatment Plant	WTP
West	W
Work Breakdown Structure	WBS
Zinc	Zn

SECTION • 1 SUMMARY

1.1 INTRODUCTION

Eldorado Gold Corporation (Eldorado) and its subsidiary Hellas Gold have prepared this technical report for the Skouries Project located in Halkidiki Peninsula in Northern Greece. The information and data included in this report were prepared in accordance with the requirements as defined in the National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects.

Eldorado is an international gold mining company based in Vancouver, British Columbia. Hellas Gold was acquired as part of the acquisition of European Goldfields Limited (EGL) completed in February 2012. The Skouries Project is an advance stage project with development work on process facilities and infrastructure started.

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

The previous technical report on the Skouries Project was filed by EGL in July 2011. This report updates the mineral resource and mineral reserve estimate and reflects updates to the open pit and underground mining production schedules, tailings and water management designs reflecting industry best practice, and updates the operating and capital cost estimates reflecting the changes to the scope of the Project.

The effective date of the report is January 1, 2018. As of the date of the report, the Project has been placed onto care and maintenance awaiting approval of several permits necessary to complete the development plans outlined in this report. Given the uncertain timeline relating to these permits, the project schedule, and all future dates reported herein, refer to nominal dates, with commercial production starting on day 1 of Year 1, and being preceded by the 2.25 years of remaining development that occurs in Years -1, -2 and the last quarter of Year -3.

Information and data contained in, or used in the preparation of mineral resource and mineral reserve updates was obtained from historic data obtained from EGL verified and supplemented by information from a surface diamond drill campaign undertaken in 2012 and 2013 by Eldorado and Hellas Gold. Metallurgical data and process designs were based on historic data obtained from EGL reviewed and validated by Eldorado. The open pit and underground mine designs and mining methods, tailings management and water management were designed from first principles to a pre-feasibility level.

When preparing reserves for any of its projects, Eldorado uses a consistent prevailing gold price methodology that is in line with the 2015 CIM Guidance on Commodity Pricing used in Resource and Reserve Estimation and Reporting. These are the lesser of the three-year moving average and the current spot price. These were set as of September 2017 for Eldorado's current mineral reserve work. For gold and copper, these were US$1,200/oz Au and US$2.50/lb Cu, respectively. All cut-off grade determinations, mine designs and economic tests of economic extraction used these pricing for the Skouries Project and the mineral reserves work discussed in this technical report. In order to demonstrate the potential economics of a project, Eldorado may elect to use metal pricing closer to the current prevailing spot price and then provide some sensitivity around this price (for the Skouries Project, metal prices used for this evaluation were US$1,300/oz Au and US$2.75/lb Cu). This analysis generally provides a better 'snapshot' of the project value at prevailing prices rather than limiting it to reserve prices that might vary somewhat from prevailing spot prices. Eldorado stresses that only material that satisfies the mineral reserve criteria is subjected to further economic assessments at varied metal pricing.

The qualified persons responsible for preparing this technical report as defined in National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects and in compliance with 43-101 F1 Technical Report, are Richard Alexander, P. Eng., Stephen Juras, Ph.D., P. Geo, Paul Skayman, FAusIMM, Colm Keogh, P. Eng., and John Nilsson, P. Eng..

Third party experts have supplied some information that was used for the development of the study. The qualified persons have reasonable reliance on the information provided by the following third party consultants, including Cementation Canada Inc., Golder Associates Ltd., Knight Piésold Ltd., Norwest Corporation and Hatch Ltd. in Canada and Jacobs Engineering Group Inc. in UK.

1.2 PROPERTY DESCRIPTION

The Project is situated at an elevation range of 350 to 620 m above mean sea level within the Kassandra Mines Complex found within the Chalkidiki Peninsula of Northern Greece (Figure 1-1). The Complex comprises of a group of mining and exploration concessions, including the Olympias mine, Madem Lakkos and Mavres Petres mines and the Skouries copper-gold porphyry deposit. The concessions cover approximately 317 km^2 and are situated 100 km east of the regional center of Thessaloniki.

The Project is located within concession numbers OP03, OP04, OP20, OP38, OP39, OP40, OP48 and OP57, which has a combined area of 55.1 km^2. Hellas Gold has been granted mining rights over these concessions until April 6, 2024. The concessions are conditionally renewable for a further two consecutive periods of 25 years each. Hellas Gold has ownership of a small portion of private land within the concessions. There are no known environmental liabilities attached to the property and there are no current environmental expenditure commitments.

1.3 PERMITTING

Permitting of the Project is well advanced with approval of the technical study, electromechanical installation permit and building permit completed thru 2012 to 2014. The site has been placed on care and maintenance as a result of non-issuance of routine permits. A revision to the permits are planned to be submitted in 2018 covering mine designs and the revised tailings management plans reflecting industry best practice.

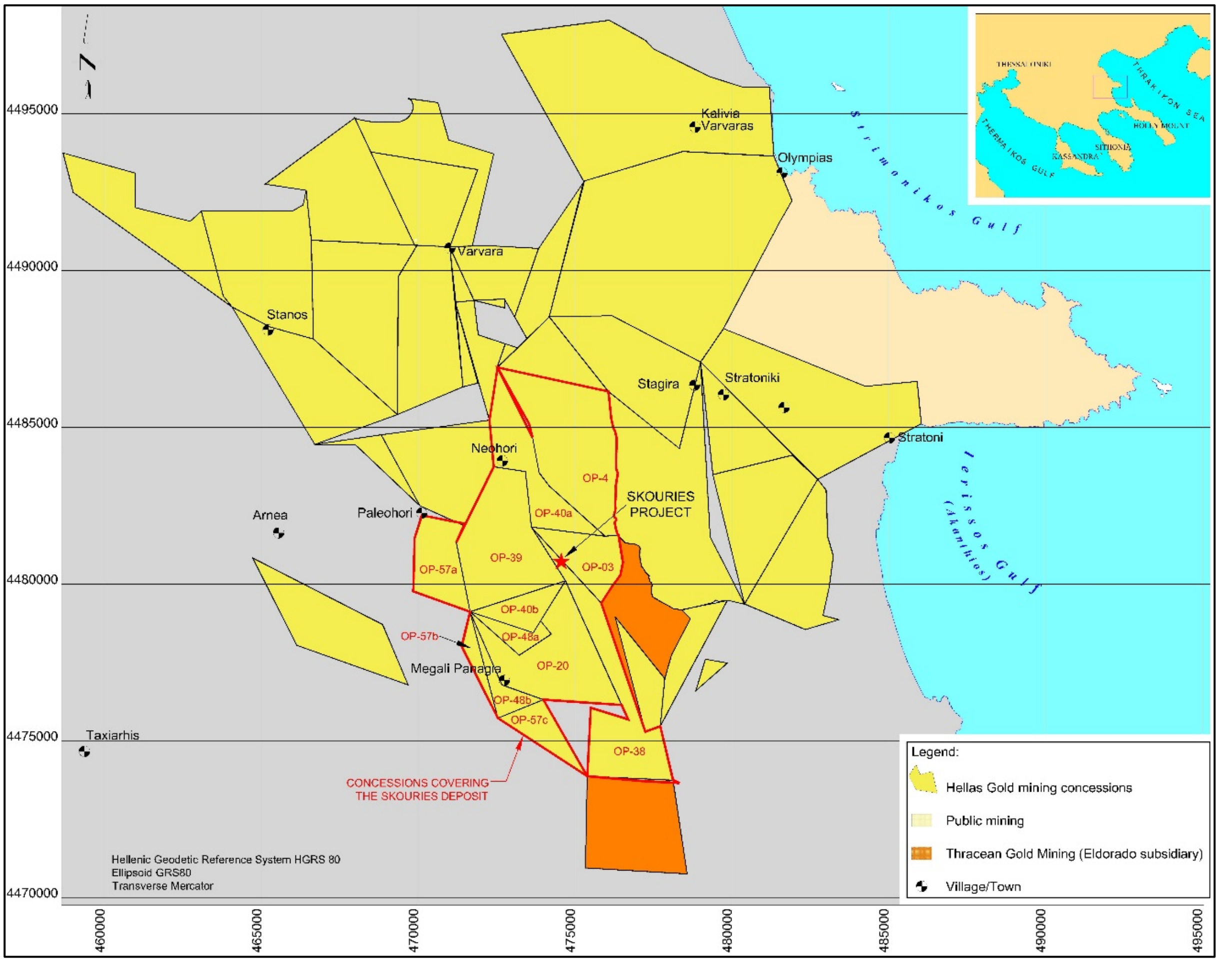


Figure 1-1: Property Location

1.4 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 and audited the TVX program and prepared a prefeasibility study in 2006. The prefeasibility study reflected an open pit operation followed by an underground mine using sublevel caving (SLC) underground mining methods at a production rate of 7 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 calculated from this work formed the basis for those quoted in EGL's final NI 43-101 technical report, effective July 14, 2011.

1.5 DEPOSIT GEOLOGY

The Skouries deposit is centered on a small pencil-porphyry stock that has a surface expression of approximately 200 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, and has a subvertical, pipe-like shape.

Ore has been tested to a depth of 920 m from surface and the results show the ore body is open at depth. Potassic alteration and copper-gold mineralization also extends into the country rock; approximately two thirds of the measured and indicated tonnes are hosted outside the porphyry with about a 50:50 split in gold-equivalent ounces.

1.5.1 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 2012-13 by Eldorado.

TVX drilled a total 72,232 m of core in 121 drillholes using NQ (47.6 mm) diameter core. Holes reached a maximum depth of 1,013 m. Eldorado Gold conducted two drill campaigns on the Skouries Project in 2012 – 2013, a 34-hole, infill program comprising 6,922 m and a10-hole, 6,617 m confirmation program.

1.5.2 Sample Preparation and Analysis

The majority of the 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 resource estimation. Confidence in those data is also supported by Eldorado's confirmation drill program.

1.6 METALLURGICAL TESTWORK

Metallurgical testwork and studies were performed by Lakefield Research, Canada to support the 2007 Cost and Definition study by Aker Kvaerner for EGL. Testwork was completed on composites selected from core samples of the major rock types covering mineralogy, grinding and flotation. Extensive flotation testwork was undertaken to enable metal recoveries to be correlated with the mine plan. Additional testwork was completed by Outotec in 2007, mostly at their laboratory in Pori, Finland to give additional design confidence.

Further supplementary testwork was undertaken by FLS Knelson in 2013 on gravity gold recovery and by Wardell Armstrong in 2015 and on flotation by Solvay (former Cytec) in 2016 and Bureau

Veritas Commodities Canada in 2017.

Initial preliminary bench scale gravity gold concentration tests were carried out by South West Metallurgical and further gold gravity concentration testwork was undertaken in 2013 by FLSmidth Knelson

1.7 MINERAL RESOURCES

The mineral resource estimate for the Skouries deposit was developed using assays and data from surface diamond drillholes. The resource estimate was made from a 3D block model based on initial outlines derived by a method of probability assisted constrained kriging (PACK). The block size for the Skouries model was selected based on mining selectivity considerations 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 are 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 g/t, applied to the assay data prior to compositing.

The assays were composited into 4 m fixed-length down-hole 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. Blocks and composites were matched on estimation domain. Reasonableness of the modeling was assessed by a detailed visual validation of sections and plans. Coding was found to be properly done. Grade interpolation was examined and the checks showed good agreement between drillhole composite values and model cell values. The block model estimates were also checked for global bias and local trends. Results show no problems with global bias in the estimates and the observed trends behave as predicted and show no significant trends of gold or copper in the estimates in Skouries model.

1.7.1 Mineral Resource Classification

The mineral resources of the Skouries deposit were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization of the Project satisfies sufficient criteria to be classified 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 predicting planned annual and quarterly production, contributed to the setup 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 is 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. Where the sample spacing was about 50 m or less, the confidence in the grade estimates and lithology contacts were the highest thus permissive to be classified as measured mineral resources. A three-hole rule was used where blocks containing an estimate resulting from three or more samples, all within 50 m and from different holes, were classified as measured mineral resources.

All remaining model blocks containing a gold grade estimate was assigned as Inferred mineral resources.

1.7.2 Mineral Resource Estimate

The Skouries mineral resources as of December 31, 2017 are shown in Table 1-1. The Skouries mineral resource is reported at a 0.2 g/t Au equivalent cutoff grade for open pit mineral resources and 0.60 g/t Au equivalent cutoff grade for underground mineral resources.

Only material within the January 2016 pit design was cast as open pit resources; all other material was considered as underground.

Table 1-1: Skouries Mineral Resources, as of December 31, 2017

Category	Resource (t x 1,000)	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz x 1,000)	Contained Cu (t x 1,000)
Open Pit Resources					
Measured	61,729	0.56	0.39	1,121	244
Indicated	27,734	0.21	0.22	189	61
Measured & Indicated	89,463	0.46	0.34	1,310	305
Inferred	3,744	0.12	0.15	15	6
Underground Resources					
Measured	38,289	1.15	0.63	1,413	240
Indicated	161,529	0.52	0.43	2,678	697
Measured & Indicated	199,818	0.64	0.47	4,091	937
Inferred	166,393	0.31	0.34	1,665	572
Total Resources					
Measured	100,018	0.79	0.48	2,534	484
Indicated	189,263	0.47	0.4	2,867	758
Measured & Indicated	289,281	0.58	0.43	5,401	1,242
Inferred	170,136	0.31	0.34	1,680	578

Au Equivalent cut-offs: Open Pit = 0.20 g/t ; Underground = 0.60 g/t

1.8 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 resource classification. 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 determination of reserves.

1.8.1 Open Pit Mineral Reserve Estimate

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 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 is also affected by a requirement to provide construction materials for the integrated waste management facility (IWMF).

The mineral reserves for the open pit deposit has been evaluated at an US$9.47/t NSR cutoff grade. The proven and probable mineral reserves are 59.5 Mt with an average grade of 0.56 g/t Au and 0.40% Cu.

1.8.2 Underground Mineral Reserve Estimate

The underground contribution to mineral reserves has been evaluated at a diluted NSR cut-off of US$33.33/t, incorporating 5% unplanned diluting material by weight that is assumed to carry no metal value, and assuming an overall mining recovery of 95%.

The mineral reserves for the underground deposit has been estimated to be 98.2 Mt with an average grade of 0.84 g/t Au and 0.55% Cu.

1.8.3 Combined Mineral Reserve Estimate

The combined mineral reserves for the Skouries Project, as of December 31, 2017, are stated in Table 1-2. These represent the weighted sum of the open pit mineral reserves and the underground reserves. The cut-offs for the mineral reserves are NSR based with US$9.47/t used in the open pit portion and US$33.33/t for the underground estimate.

Table 1-2: Skouries, Combined Mineral Reserves, as of December 31, 2017

Category	Ore (t x 1,000)	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz x 1,000)	Contained Cu (t x 1,000)
Proven	75,804	0.87	0.52	2,132	393
Probable	81,862	0.62	0.47	1,641	386
Proven & Probable	157,666	0.74	0.49	3,773	779

1.9 MINE PRODUCTION SCHEDULE

The Skouries Project is designed as a two phase mining operation. Phase 1 consists of a combined open pit and underground mine, operating over 10 years. Phase 2 consists of mining from the underground mine for further 13 years. The total life of mine (LOM) is 23 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 mill feed rate from the mining operation is shown in Figure 1-2.

Phase 1 total mill feed is 8.0 Mtpa, consisting of a nominal 5.5 Mtpa from the open pit mine combined with a 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 6.4 Mtpa in order to maintain the 8.0 Mtpa mill feed. During Phase 1 approximately 6.5 Mt of oxide ore is stockpiled to be rehandled for mill feed during Phase 2. Phase 1 is complete at the end of the open pit mine life in Year 10.

Phase 2 mine production, from Year 11 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 2.4 Mtpa. During the first three 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 of approximately 1.6 Mtpa. From Year 14 on, Phase 2 mill feed is maintained at a nominal feed rate of 6.2 Mtpa, solely from underground mine production.

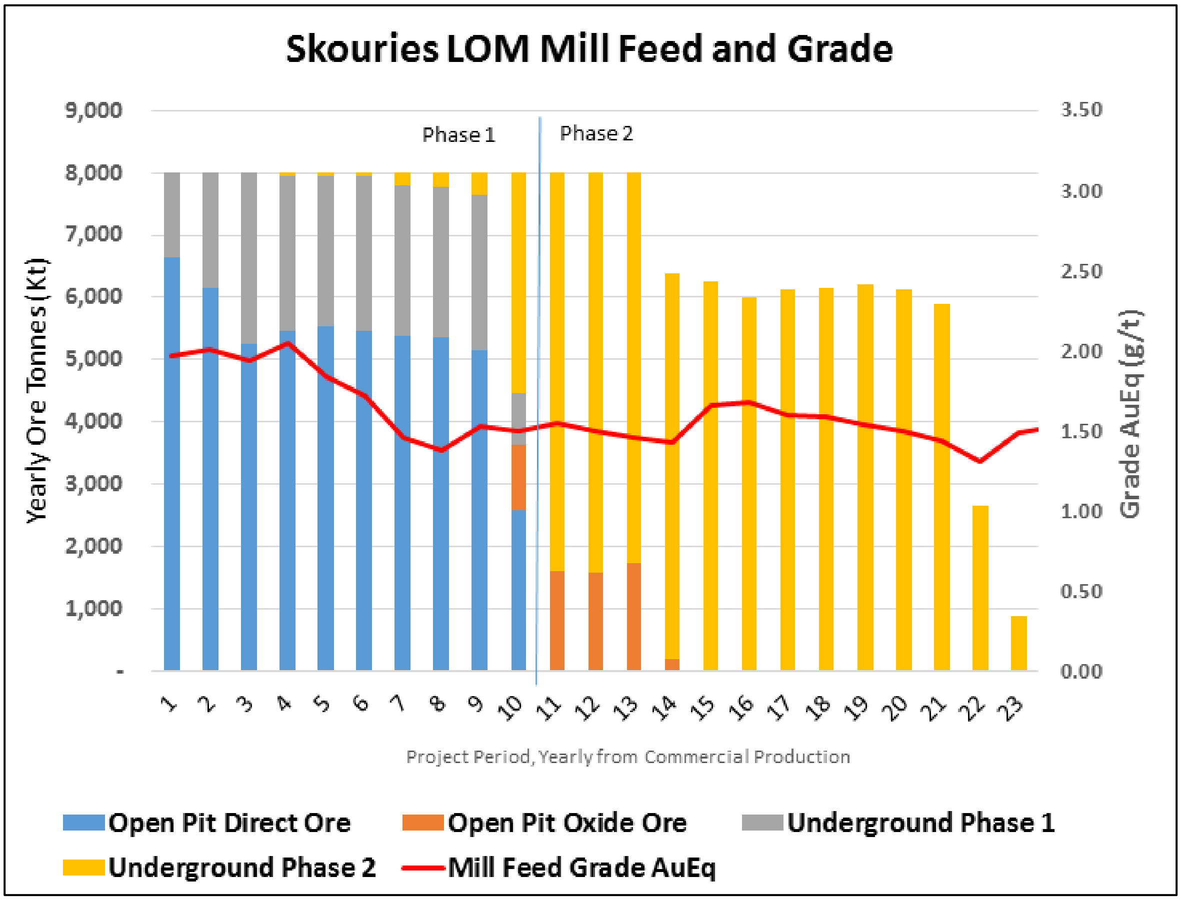


Figure 1-2: Skouries Life of Mine Ore Production Schedule

1.10 MINING METHODS

1.10.1 Open Pit

Open pit mining will be by 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. Based on the modelled rock types, approximately 17% of the mined material is amenable to free digging, this material will not require blasting. The final pit design is shown on Figure 1-3.

Direct feed ore from the open pit will be hauled to the Skouries processing plant by the owner operated fleet of 90 t trucks. During Phase 1 a portion of oxide ores will be hauled to the oxide ore stockpile (OOS) where it is planned to be re-handled during the Phase 2 of the Project. Waste material will be hauled to a transfer point adjacent to the OOS by the 90 t trucks where it will be re-handled by a fleet of smaller contractor trucks and placed in one of the material management structures within the IWMF. There is no separate waste rock dump, all waste material will be utilized for construction of structures within the IWMF.

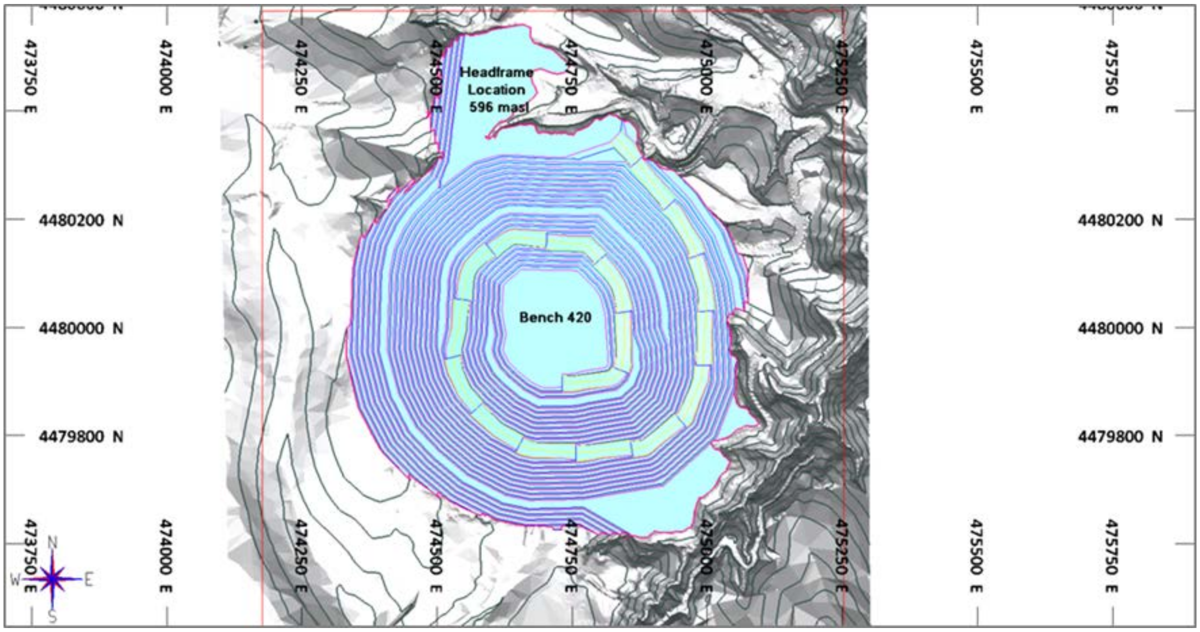


Figure 1-3: Final Pit Designs

The number of haulage units was determined by calculating cycle times in Talpac© using annual haul cycle profiles from MineSight©. Haul simulations were carried out based on the designated 90 t haul trucks and the smaller 60 t, 40 t and 15 t contractor trucks.

1.10.2 Underground Mining

The Skouries orebody that extends below the bottom of the open pit is amenable to a bulk underground mining methods and has been evaluated under several different design approaches since the late 1990's including block caving, SLC and SLOS. SLOS has been confirmed as the most appropriate underground mining method for the Project and was incorporated into the EIS/JMD approval in 2011.

The RQD % values from drill core indicate most of the Phase 1 stoping area to be in good ground and the Phase 2 stoping areas to be in a combination of good to fair ground. Standard stope dimensions of 65 m high x 30 m long x 15 m wide in porphyry, and 65 m high x 20 m long x 15 m wide for primary stope design in schist can be realized. Production stopes will be backfilled with cemented paste fill. The stoping methodology is the same for both Phase 1 and Phase 2. Two test stopes have been planned to provide proof of concept for the 60 m high x 15 m wide x 30 m long basic stoping units in both porphyry and potassic schist rock types.

All levels in both phases have similar designs. Peripheral development ring-drives will provide access to all sides of the orebody. Ore drives for stope extraction will traverse the orebody on 15 m centers, 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 center out.

Typical sublevel arrangement of underground mine is shown on Figure 1-4.

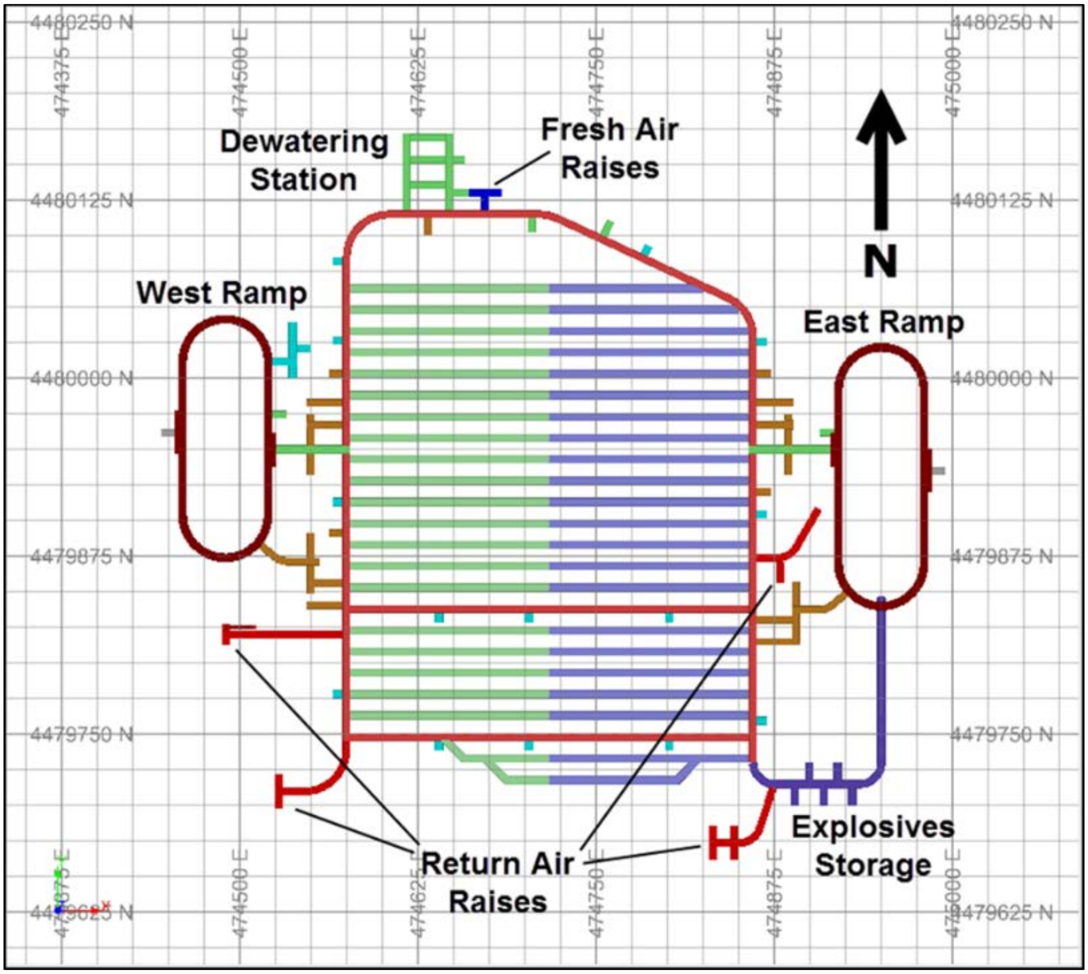


Figure 1-4: Typical Sublevel Arrangement (230 Level)

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.

1.10.3 Underground 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 loading bays via the dual ramp system.

The Phase 2 materials handling will involve shaft hoisting of ore to surface. Shaft hoisting is critical to enable a ramp up to full production of 6.4 Mtpa from the Phase 1 production of 2.5 Mtpa. In order to hoist the material by shaft underground crushing will be implemented. 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 at its core includes tele-remote operation of mechanized equipment by an operator located on surface or in a remote area of the underground mine.

1.11 RECOVERY METHODS

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 of operation until the depletion of reserves the plant will process ore extracted from the underground mine at a reduced tonnage of 6.2 Mtpa. During years 11 to 15 oxide ore is being rehandled to maintain mill feed at 8.0 Mtpa during this period.

The plant will process the copper/gold ore with a LOM average grade of 0.49% copper and 0.74 g/t gold. Expected LOM average recoveries are 87.9% for copper and 82.4% for gold, respectively. The mill will produce a doré which contains approximately 80% gold and 10% silver and 10% copper and a 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, 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 is presented in Figure 1-5.

The unit operations comprise of:

- Primary crushing and ore stockpile.
- Grinding and pebble crusher.
- Flotation and regrind.
- Gravity concentration and recovery.
- Concentrate and tailings thickening.
- Concentrate filtering storage and loadout.
- Tailings filtration.
- Reagents and services.

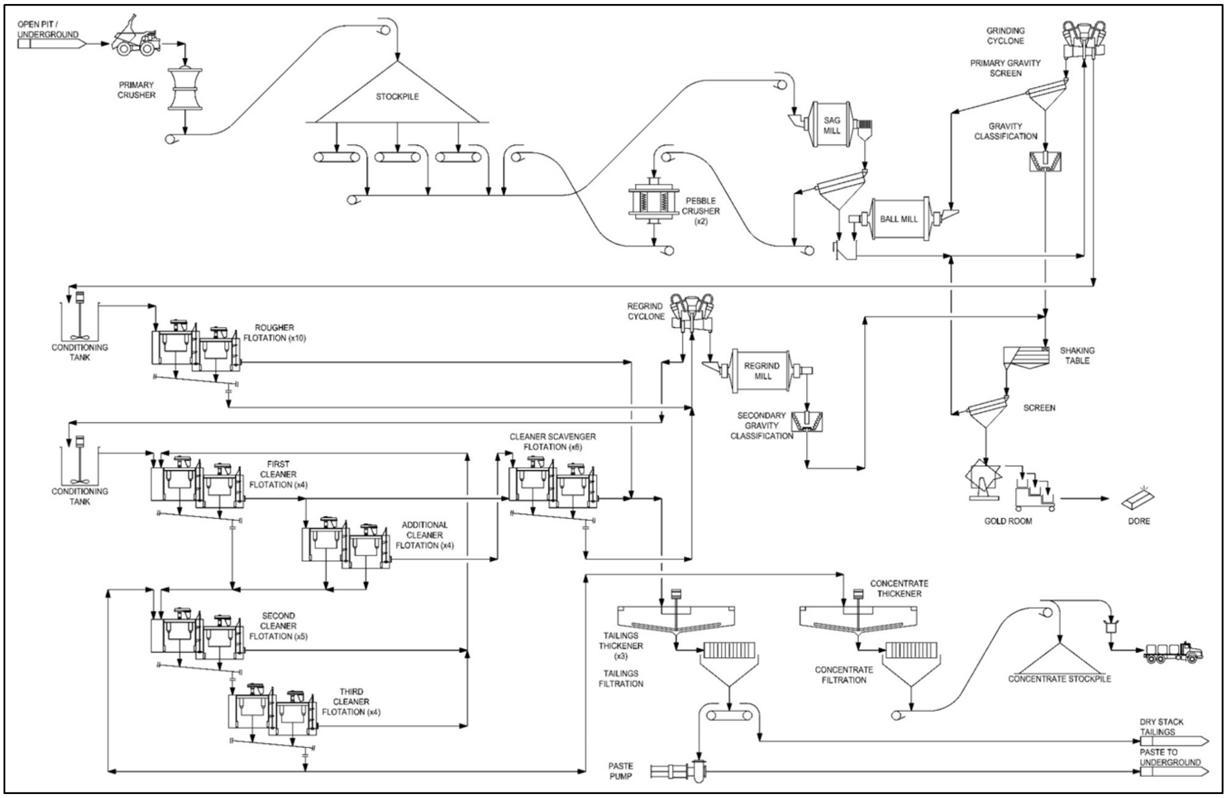


Figure 1-5: Simplified Process Flow Diagram

1.12 PROJECT INFRASTRUCTURE

1.12.1 Waste Management

The principal waste streams generated from the Project are the overburden and waste rock from the open pit mining and underground development and the tailings from the mineral processing operations. Overburden and waste rock will be stored on surface and tailings will be used underground as paste backfill with the remainder being stored on surface. The project mine plan and material balance has been developed such that overburden and waste rock is entirely used for construction requirements eliminating the need for a separate waste rock dump. The waste management plan has been developed to provide for surface storage of waste streams in the IWMF all within one watershed.

This compact mine plan utilizes industry best practice to minimize disturbance to the natural environment, including surface water and ground water impacts. The major waste management components are shown on Figure 1-6.

The Skouries Project will be mined in two operational phases. During Phase 1, over the first ten years of operation, the mine will be developed using both open pit and underground mining techniques. Overburden and waste rock from the Phase 1 mining activities will be used as a source of construction materials for the IWMF embankment, water management pond (WMP), south water management pond (SWMP), the OOS and closure capping stockpile (CCS) pads and site infrastructure. Tailings will be deposited underground as paste backfill with the balance stored above ground on surface as filtered tailings in the IWMF. During Phase 2, the mining will consist entirely of underground mining for an additional thirteen years. During this period, tailings will be deposited underground as paste backfill and on surface as filtered tailings in the open pit. The IWMF will be decommissioned and progressively reclaimed at the end of Phase 1.

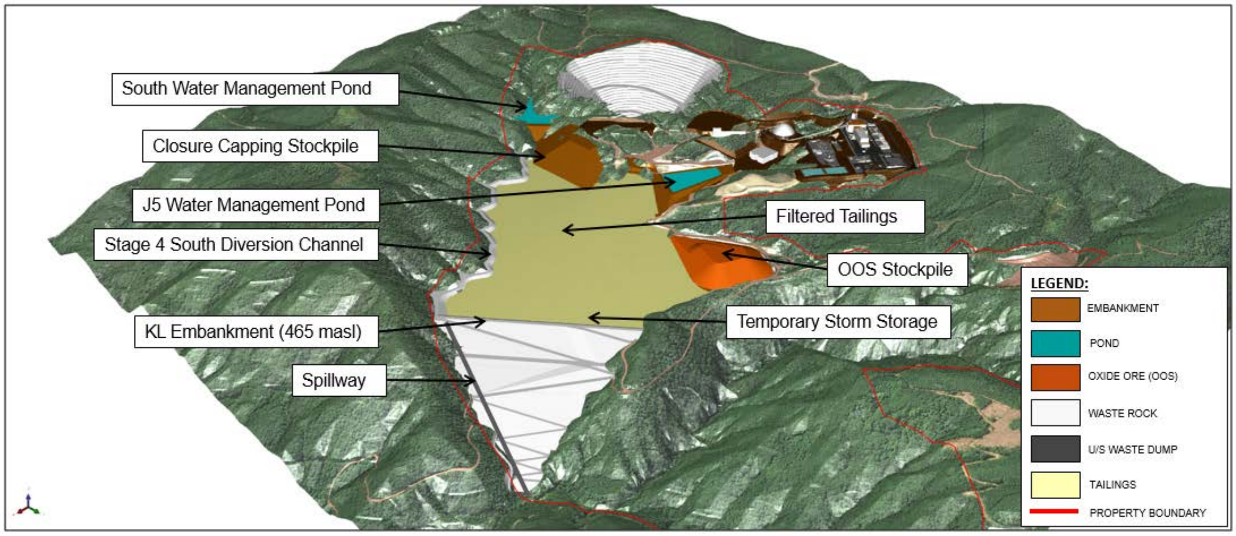


Figure 1-6: Phase 1 IWMF Site Layout (Year 10)

1.12.2 Water Management

The water within the Project site can be classified into two categories, 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 falls in the form of precipitation and has been exposed to mine infrastructure. A numerical groundwater model was developed for the Project utilizing site specific data from field investigations to estimate the dewatering rates for contact and non-contact water

1.12.2.1 Site Wide Water Balance

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

The general results of the SWWB show that excess contact water is expected to be generated during Phase 1, and will need to be managed with onsite storage capacity and potential treatment for use as irrigation. After that, during Phase 2 there will be limited amount of both non-contact and contact water expected.

1.12.2.2 Water Quality and Treatment

Water quality predictions were developed using the existing geochemical characterization dataset, site monitoring data, and the site wide water management plan.

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 irrigation purposes.

1.12.3 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 center 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 sea ports. 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 the highway and rail infrastructure.

1.12.4 Power Supply

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

1.13 MARKET STUDIES AND CONTRACTS

The Skouries process plant will produce two saleable products: gold-copper concentrate and poured gold doré bars. Both products are expected to be marketable to a large number of upstream smelter and refiners.

The gold doré bars will be sold to refineries, and transport will occur over land, or by air, by a security company capable of insuring and delivering the high value doré product.

In the future, the commissioning of the Olympias Phase III process facility and associated infrastructure including a port, flotation plant, smelter, and waste management facilities will allow for more cost effective shipment of concentrate for the Skouries concentrate. At 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 technical report, however several indicative non-binding proposed term sheets have been received from European and global copper smelters.

1.13.1 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. Construction contracts are being tendered and awarded to qualified contractors by the owners construction management team.

This technical report assumes development contractors will be utilized for the open pit and underground mine development.

1.14 ENVIRONMENTAL

The EIS for the Kassandra Mine Mineral Deposits Project (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.

1.14.1 Baseline Studies

Three baseline studies have been performed relating to Skouries to support the EIS, the combination of these comprehensive studies have 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 pressure from unregulated landfills and wastewater discharge.

1.14.2 Impact Assessment

1.14.2.1 Environmental Impacts

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 to the environment or surrounding villages.

Hellas Gold runs an extensive regional monitoring program covering monitoring of air, water, noise and vibration, these will continue through the life of mine and post closure.

1.14.2.2 Social and Cultural Impacts

There is currently high unemployment in the region 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 archeological 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 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 to do so. 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.

1.14.2.3 Closure and Reclamation

The closure and environmental rehabilitation program includes a comprehensive criteria for closure and reclamation of the Project. Plans have been developed including 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.

1.14.2.4 Environmental Bonds

Hellas Gold has provided the Mining & Industrial Minerals Directorate of the Ministry of the Environment, with a letter of guarantee for € 50 million in favor 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.

1.15 CAPITAL AND OPERATING COSTS

All cost estimates are presented in Q1 2018 US Dollars (US$). The accuracy of the capital and operating cost estimates is consistent with the standards outlined by the Association for the Advancement of Cost Engineering (AACE) to a prefeasibility-level estimate categorized as AACE Class 4.

1.15.1 Capital Costs

The total Project capital cost includes the investment cost from restart of construction up until commercial production of the mill, and then followed by sustaining capital costs spread out over the remaining 23 years of the mine life.

Capital costs are summarized in Table 1-3. Sunk costs are not included in the capital costs.

Capital costs were built from first principles. Quantities were based on material take offs. Labour rates are derived from existing contracts for the Skouries Project and benchmarked against actual contracts from the Olympias Phase II Project, the all-in crew labour rates include all direct and indirects costs associated with the contractors. A labour productivity factor is used to account for overall labour force efficiency. Equipment and commodity pricing were based on contract rates and quotations from contractors and suppliers with smaller items being estimated from database. The capital cost estimates include indirects, owners costs and contingency. Contingency was built up from analysis of each individual item, resulting in an overall Project contingencies for initial investment and sustaining capital cost estimates 14%, and 19% respectively.

Table 1-3: Capital Cost Summary

Area	Initial (US$ x 1,000)	Sustaining [1] (US$ x 1,000)
A - Overall Site	14,508	0
B - Open Pit Mine	66,694	22,975
B - Underground Mine	144,019	405,352
C - Stockpile and Materials Handling	11,801	596
D - Process Plant	134,833	41,400
E - Underground Backfill Plant	0	27,619
F - Integrated Waste Management Facility (IWMF)	22,446	21,948
G - In Pit Tailings	0	40,546
I - Water Management	15,248	3,486
H - Infrastructure	49,288	5,993
J - Ancillary Facilities	9,001	2,146
K - Off Site Infrastructure	4,541	0
P - Environmental	0	2,708
Direct	**472,379**	**574,769**
Indirects	**99,563**	**59,661**
Owners Cost	**30,340**	**0**
Contingency	**86,892**	**123,587**
Total Installed Cost	**689,174**	**758,016**

Note: [1] LOM mining and ore processing.

1.15.2 Operating Costs

Operating costs were calculated for each year of operation, totalling US$3.34 B for an average of US$21.32/t ore milled over life-of-mine. Life of mine (LOM) operating costs are summarized in Table 1-4.

Open pit and underground mining operating costs were estimated from first principles by unit operation, based on projected fleet requirements, consumables and manpower. Fleet requirements were calculated based on historical benchmarks of equipment productivities and haulage simulations. Equipment operating cost and fuel consumption were estimated from a combination of manufacturers' data and consultant's in-house data. Labour requirements were developed to support the operation and maintenance of the fleet, and for the general operation of the mine.

Process operating costs were based on estimated annual consumption of process reagents, major wear parts, and utilities. Budget quotations were obtained for supply of all significant consumables and utilities. Power consumption was calculated based on electrical load lists. Labour requirements were developed to support the operation and maintenance of the process facilities.

Table 1-4: Operating Costs

Category	LOM Average (US$/t ore)	LOM Expenditure (US$ x 1,000)
Open Pit Mining (US$/t of OP ore)	4.51	238,876
Underground Mining (US$/t of UG ore)	16.50	1,602,340
Total Mining (US$ /t of LOM ore)	**11.75**	**1,841,216**
Stockpile Rehandling	0.06	9,818
Processing Cost	6.73	1,055,007
Filter Plant	0.76	119,067
IWMF and Water Management	0.62	96,788
G&A	1.39	218,317
Operating Cost	**21.32**	**3,340,213**

Operating costs for the IWMF and water management and treatment costs were calculated based on operation and maintenance requirements of pumps, pipelines, and mobile equipment. Labour requirements for operation was calculated based on fleet sizing.

General and administrative costs were estimated based on a projected personnel list with salaries indicative of local standards, and annual allowances for general supplies. Salaries and wages were provided as full-burden rates by Hellas Gold and based on detailed personnel lists developed for each operating department.

1.16 ECONOMIC ANALYSIS

The economic analysis is based on the mineral reserves the production schedule, recoveries and the capital and operating costs.

The economic 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 21.2% on an after tax basis is achieved with the Project case metal prices of US$1,300/oz Au and US$2.75/lb Cu. Using those metal prices, the net present value (NPV) of the Project is estimated to be US$925 M using a discount rate of 5%, with a payback of the remaining capital achieved in 3.4 years from the start of commercial production.

1.16.1 Sensitivity Analysis

The economic model was subjected to a sensitivity analysis to determine the effects of changing metal prices, and changes in operating and capital cost assumptions, on the Project financial returns, as shown on Table 1-5 and Table 1-6.

The sensitivity analysis shows that when evaluated at the current reserve metal prices used in preparation of this report, namely US$1,200/oz Au and US$ 2.50/lb Cu, the after tax IRR is 17.4% and the NPV is improved to US$669 M using the same 5% discount rate and the payback period is 3.9 years.

The sensitivity analysis shows that the Project is also robust when evaluated using lower metal price assumptions, or higher operating and capital costs. The Project continues to develop net positive cash flows and positive net present value at metal prices of US$1,100/oz Au and US$2.3/lb Cu, or with operating and capital cost increased by 15% individually.

Table 1-5: Metal Price Sensitivity Analysis

		Sensitivity Ranges				
Parameters	**Units**	**-15%**	**-7.5%***	**Project Case**	**+7.5%**	**+15%**
Gold Price	US$/oz	1,100	1,200	**1,300**	1,400	1,500
Copper Price	US$/lb	2.30	2.50	**2.75**	3.00	3.20
Results (after Tax)						
NPV 0%	US$ M	994	1,418	**1,864**	2,332	2,700
NPV 5%	US$ M	421	668	**925**	1,197	1,411
NPV 8%	US$ M	221	407	**603**	809	972
IRR %	%	13.4	17.4	**21.2**	25.0	28.0
Payback period	yrs	4.7	3.9	**3.4**	3.0	2.7
Taxation	US$ M	348	510	**686**	873	1,022
Royalties	US$ M	49	54	**96**	104	178

*Reserve Case Metal Prices

Table 1-6: Capital and Operating Costs Sensitivity Analysis

		Sensitivity Ranges		
Parameter	**Units**	**-15%**	**Project Case**	**15%**
LOM Capex*	US$ M	1,237	**1,447**	1,657
Results (after Tax)				
NPV 0%	US$ M	2,021	**1,864**	1,706
NPV 5%	US$ M	1,056	**925**	795
NPV 8%	US$ M	722	**603**	484
IRR	%	26.0	**21.2**	17.5
LOM Operating Costs	US$/t ore	18.1	**21.3**	24.5
Results (after Tax)				
NPV 0%	US$ M	2,222	**1,864**	1,502
NPV 5%	US$ M	1,120	**925**	730
NPV 8%	US$ M	746	**603**	460
IRR	%	23.6	**21.2**	18.7

*Excluding Pre-production Metal Sales

1.17 OTHER RELEVANT DATA AND INFORMATION

The Skouries Project has been under construction since 2012 and the capital costs incurred to date 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.

Hellas Gold will submit revised technical studies for the mine, waste management and water management facilities. Following approval of the revised technical study Hellas Gold must apply for revised electrical mechanical installation permit and the construction permit.

1.18 INTERPRETATIONS AND CONCLUSIONS

It is concluded that the work completed in the prefeasibility study indicate that the exploration information, mineral resource and mineral reserve estimates and Project economics are sufficiently defined to show that the Project is technically and economically viable.

1.18.1 Mineral Resources and Mineral Reserves

It is the opinion of the qualified persons that the information and analysis provided in this report is considered sufficient for reporting mineral resources and mineral reserves.

Results of the investigations indicate that the ore body is open at depth. Eldorado considers this an opportunity to the Project that may result in increased resource and reserves being realized.

1.18.2 Recovery

Significant metallurgical testwork and analysis has been completed to confirm the process designs and substantiate stated recoveries. The qualified persons have a high degree of confidence in the process designs and do not associate any risks with the stated recoveries.

1.18.3 Capital and Operating Costs and Financial Model

The accuracy of the capital and operating cost estimates is consistent with the standards outlined by the Association for the Advancement of Cost Engineering (AACE). The cost estimates prepared for the study are categorized as AACE Class 4 estimates built off prefeasibility level designs completed based on first principles. The economic model has been built from first principles and includes all relevant data, the qualified persons have a high level of confidence in the stated economic performance of the Project.

The realized economic performance achieved on the Project may be affected by factors outside the control of Eldorado, including mineral prices and currency fluctuations. The economic model may be positively affected by shipment of concentrate to Olympias once commissioned. Additionally the inclusion of resources not included in the current reserve estimate at depth presents a potential benefit to the Project economics.

1.18.4 Permitting

Hellas Gold has obtained the critical permits required to proceed with the Project including the technical study, electromechanical installation permit and building permit. These permits will be updated and amended based on revisions to the mine and waste management reflecting Industry best practice. Delays in receiving approval of revisions to the technical study and permits could impact the development of the Project.

1.19 RECOMMENDATIONS

The prefeasibility study investigations have provided a technical and economic solution that forms the basis to proceed with development of the Project. It is recommended to continue with development of the Project while undertaking the following work.

The site is currently on care and maintenance. Prior to work recommencing on site the revised permits must be issued. The majority of the work supporting application of these permits has been completed so the cost to complete is negligible.

Engineering of the majority of the Project has been developed to a detailed engineering level. Detailed engineering and procurement of the mine, revised IWMF and water management will be completed in the first half of 2018. The budget to complete this design work is US$11.8 M.

Work that Hellas Gold operation will continue to complete on site during construction and operations include:

- Geotechnical investigations at the shaft and ventilation raises.

- Complete exploration and studies to increase resources at depth.
- Optimize development to enable test stoping to be completed as soon as possible.
- Optimize the mine plan with respect to owner supplied and contractor fleet for waste haulage.

These investigations and studies are optimizations and confirmatory work and do not affect the production schedule as presented. The studies are be completed by the Skouries operations personnel and costs are included in the current cost estimate.

SECTION • 2 INTRODUCTION

Eldorado Gold Corporation (Eldorado) and its subsidiary Hellas Gold have prepared this technical report for the Skouries Project located in Halkidiki Peninsula in Northern Greece. The information and data included in this report were prepared in accordance with the requirements as defined in the National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects.

Eldorado is an international gold mining company based in Vancouver, British Columbia. Hellas Gold was acquired as part of the acquisition of European Goldfields Limited (EGL) completed in February 2012. The Skouries Project is an advance stage Project with the Environmental Impact Statement (EIS) permit approved by the Greek Government in 2011 and development work started. The gold-copper porphyry deposit is to be mined using conventional open pit and underground mining techniques. The mineral process facilities will produce doré and a gold-copper concentrate.

The previous technical report on the Skouries Project was filed by EGL in July 2011. This report updates the mineral resource and mineral reserve estimates and reflects updates to the open pit and underground mining production schedules, tailings and water management designs reflecting industry best practice and the operating and capital cost estimates.

Information and data contained in, or used in the preparation of mineral resource and mineral reserve updates was obtained from historic data obtained from EGL verified and supplemented by information from a surface diamond drill campaign undertaken in 2012 and 2013 supervised by Eldorado and undertaken by Eldorado and Hellas Gold. Metallurgical data and process designs were based on historic data obtained from EGL reviewed and validated by Eldorado. The open pit and underground mine designs and mining methods, tailings management and water management were designed from first principles to a pre-feasibility level.

When preparing reserves for any of its projects, Eldorado uses a consistent prevailing gold price methodology that is in line with the 2015 CIM Guidance on Commodity Pricing used in Resource and Reserve Estimation and Reporting. These are the lesser of the three-year moving average and the current spot price. These were set as of September 2017 for Eldorado's current mineral reserve work. For gold and copper, these were US$1,200/oz Au and US$2.50/lb Cu, respectively. All cut-off grade determinations, mine designs and economic tests of economic extraction used these pricing for the Skouries Project and the mineral reserves work discussed in this technical report. In order to demonstrate the potential economics of a project, Eldorado may elect to use metal pricing closer to the current prevailing spot price and then provide some sensitivity around this price (for the Skouries Project, metal prices used for this evaluation were US$1,300/oz Au and US$2.75/lb Cu). This analysis (in Section 22 of this report) generally provides a better 'snapshot' of the project value at prevailing prices rather than limiting it to reserve prices, that might vary somewhat from prevailing spot prices. Eldorado stresses that only material that satisfies the mineral reserve criteria is subjected to further economic assessments at varied metal pricing.

The qualified persons responsible for preparing this technical report as defined in NI 43-101, Standards of Disclosure for Mineral Projects and in compliance with 43-101F1 Technical Report, are

Richard Alexander, P. Eng., Stephen Juras, Ph.D., P.Geo, Paul Skayman, FAusIMM, Colm Keogh, P.Eng. and John Nilsson, P.Eng.

Mr. Alexander, Project Director for the Company, was Project Manager responsible for overall preparation of the technical study and related costs. He most recently visited the Skouries Project site on April 18 to 20, 2016.

Dr. Juras, Director, Technical Services for the Company, was responsible for the preparation of the sections in this report concerned with geological information, sample preparation and analyses and mineral resource estimation. He most recently visited the Skouries Project site on November 12, 2017.

Mr. Skayman, Chief Operating Officer for the Company, was responsible for the preparation of the sections in this report that dealt with metallurgy and process operations. He most recently visited the Skouries Project site on July 14, 2017.

Colm Keogh, Manager, Mine Engineering (Underground) for the Company, was responsible for the preparation of the sections in this report that dealt with the underground mine mineral reserves estimation and mining methods. He most recently visited the Skouries Project site on November 12, 2017.

Mr. Nilsson, President of Nilsson Mine Services Ltd. (Nilsson), was responsible for the preparation of the sections in this report that dealt with the open pit mineral reserves estimation and mining methods. He most recently visited the Skouries Project Site on August 30 to 31, 2015.

Third party experts have supplied some information that was used for the development of the study. The qualified persons have reasonable reliance on the information provided by the following third party consultants, including Cementation Canada Inc., Golder Associates Ltd., Knight Piésold Ltd., Norwest Corporation and Hatch Ltd. in Canada and Jacobs Engineering Group Inc. in UK.

SECTION • 3 RELIANCE ON OTHER EXPERTS

Eldorado prepared this document with input from Hellas Gold. The qualified persons did not rely on a report, opinion or statement of another expert who is not a qualified person, concerning legal, political, environmental or tax matters relevant to the technical report.

SECTION • 4 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

The Project is located within the Kassandra Mines complex found within the Chalkidiki Peninsula of Northern Greece. The complex comprises 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, Madem Lakkos and Mavres Petres Mines (collectively known as ''Stratoni'') which are currently in production and the Skouries copper-gold porphyry deposit under development.

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

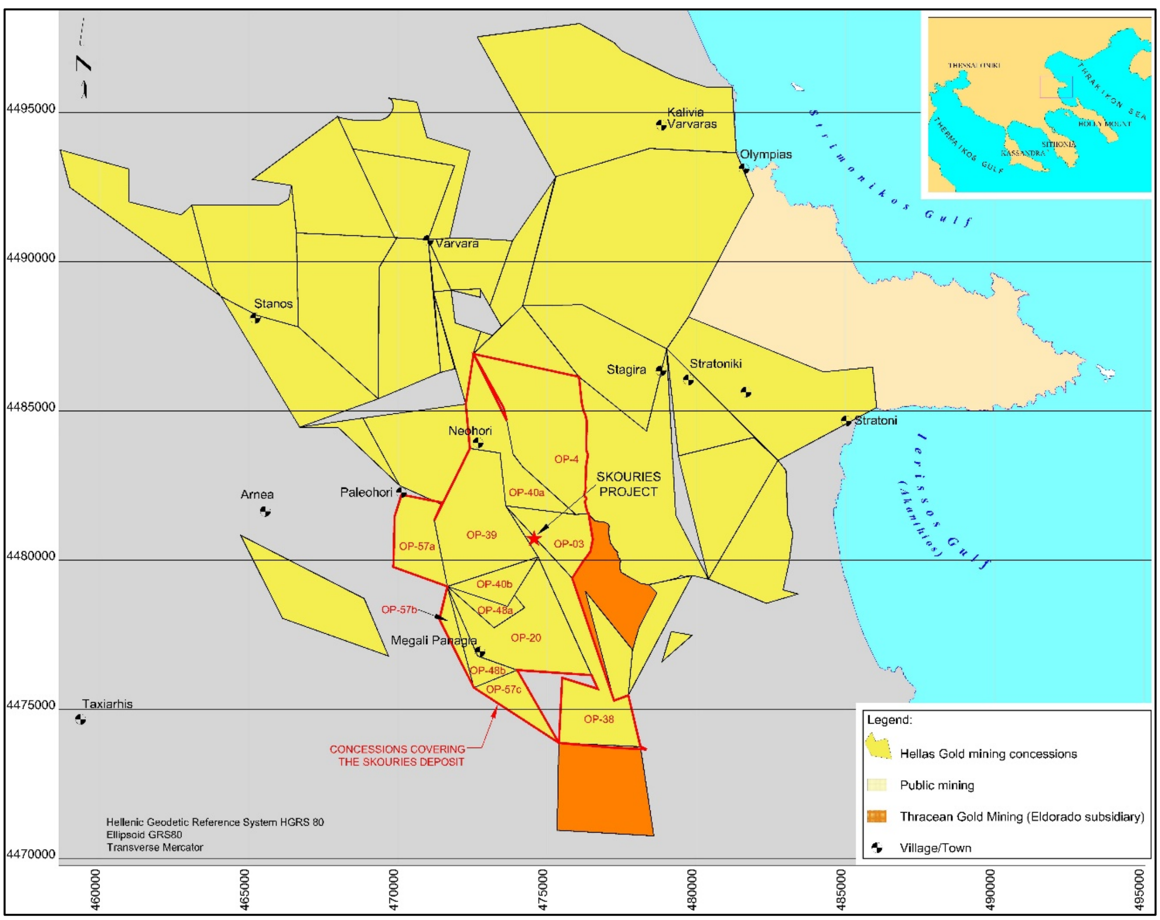


Figure 4-1: Property Location

4.2 LAND TENURE

The Project is located within concession numbers OP03, OP04, OP20, OP38, OP39, OP40, OP48 and OP57, which has a combined area of 55.1 km^2. Hellas Gold has been granted mining rights over these concessions until April 6, 2024. The concessions are conditionally renewable for a further two consecutive periods of 25 years each. Hellas Gold has ownership of a small portion of private land within the concessions. There are no known environmental liabilities attached to the property and there are no current environmental expenditure commitments.

4.3 PERMITTING

4.3.1 Technical Study

The technical study for the Project was approved in February 2012. A subsequent further approval for the detailed paste tailings deposit was obtained in September 2014 and a further approval covering general electromechanical modifications was approved in May 2016.

4.3.2 Electromechanical Installation Permit

The electromechanical installation permit was approved in May 2013. A further modification was requested in October 2016 and is still pending.

4.3.3 Building Permit

The building permit was approved in October 2014 (Stage 1) and February 2016 (Stage 2). After the approval of the updates to the electromechanical installation permit, an updated building permit will be requested for the filter building and any other ancillary building structures associated with revisions to the mine designs and tailings management.

4.3.4 Permitting Status

The Skouries Project, as of the effective date of this report, has been placed on care and maintenance as a result of non-issuance of routine installation permits.

Hellas Gold will be preparing and submitting amended technical studies for the underground mining, the process plant and the waste management facilities. The technical study revisions include the following:

- The IWMF as described in Section 18 of this report is a departure from the previously approved technical study which involved deposition of paste tailings in both Karatza Lakkos (KL) and Lotsaniko valleys. This change was made in order to apply best available technology, specifically the application of filtered tailings which reduces the disturbed footprint significantly by storing all the tailings and waste rock in the KL Valley.
- The underground mine as proposed in this report, requires one or several additional technical studies to be completed to realize the full scope of the proposed development. The existing underground development undertaken in 2013-2015, under a separate

technical study, can be re-started, and would allow for early development, including test stoping, which may form an important input into the full scope technical study to approve the underground mining plan.

Following the approval of the revised technical studies Hellas Gold will apply for revised electrical mechanical installation permits and construction permits.

SECTION • 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 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 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 agriculture of the region are wines, honey, olives and oil.

The Chalkidiki Peninsula climate is generally mild with limited 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°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.

5.2 LOCATION AND ACCESSIBILITY

The Project 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 Project area 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 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 7 km from 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 southwest of Stratoni, 11 km south of the town of Palaeohori and 3 km northeast of the village of Megali Panagia.

5.3 INFRASTRUCTURE AND LOCAL RESOURCES

The area is well served by main power supplied via the Public Power Corporation (PPC). A high voltage 150 kV overhead power line connected to the main grid, is currently under construction 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 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.

5.4 SURFACE RIGHTS

The total land take of the Project is 452.3 ha of contiguous land. Hellas Gold owns some 7% of this area. The concession of a further 92.7% of the land (comprises private and public forestry) is granted to Hellas Gold. The remaining 0.3% is in the process of expropriation. This land take 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.

SECTION • 6 HISTORY

6.1 ANCIENT HISTORY

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 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 MODERN HISTORY

6.2.1 1960 to 1990s

The Skouries deposit was initially drilled by Nippon Mining and Placer Development (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/Kvaerner (1996 to 2004)

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.

A mineral resource estimation was completed as part of a feasibility study initiated by TVX with SRK Consulting and Kvaerner Metals in September 1998 with an updated EIS in February 1999. A summary of the 1998 mineral resources is included in Table 6-1.

Table 6-1: Historical Mineral Resource, TVX/Kvaerner 1998

Category	Resource (Mt)	Grade Au (g/t)	Grade Cu (%)
Mineral Resources			
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

6.2.3 European Goldfields (2004 to 2012)

EGL acquired the property in 2004 and audited and reviewed the 1998 mineral resource statement; EGL concluded that the mineral resource was classified according to the definitions and guidelines of the Canadian Institute of Mining (CIM), Metallurgy and Petroleum respecting mineral resources and mineral reserves. The historical mineral resources were reported at a nominal 0.4 g/t Au cut off.

A feasibility study was prepared in 2006 by EGL 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 sublevel caving (SLC) at a production rate of 7.0 Mtpa.

In 2007, SRK was retained by EGL to undertake an engineering study applying open pit and SLC mining methods based on the 1998 TVX/2004 EGL geological model to update the mineral reserves based on higher metal selling prices. The 2007 NI 43-101 technical report mineral reserves are summarized inTable 6-2.

Table 6-2: EGL 2007 Historical Proven and Probable Mineral Reserves

Category	Ore (Mt)	Grade Au (g/t)	Grade Cu (%)
Open Pit Mineral Reserves			
Proven	42.5	0.71	0.46
Probable	9.7	0.6	0.39
Sub-Total	**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	0.9	0.62
Sub-Total	**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**

A later study investigated the possible use of sublevel 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 calculated from this work formed the basis for those quoted in EGL's final NI 43-101 technical report, effective July 14, 2011. These 2011 reserves are summarized on Table 6-3.

Table 6-3: EGL 2011 Historical Proven and Probable Mineral Reserves

Proven and Probable Mineral Reserves	Ore (Mt)	Grade Au (g/t)	Grade 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

SECTION • 7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Western Tethyan orogenic belt in southeast 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 Serbo-Macedonian 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 northwest-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, FYROM; and Tulare, Serbia) and carbonate replacement deposits (Olympias, Mavres Petres, Madem Lakos and Piavitsa, Greece), as well as the Plavica high sulfidation epithermal deposit (FYROM). The mineral deposits formed during post-collision extension and emplacement of intermediate to felsic magmas with high K calc-alkalic 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.

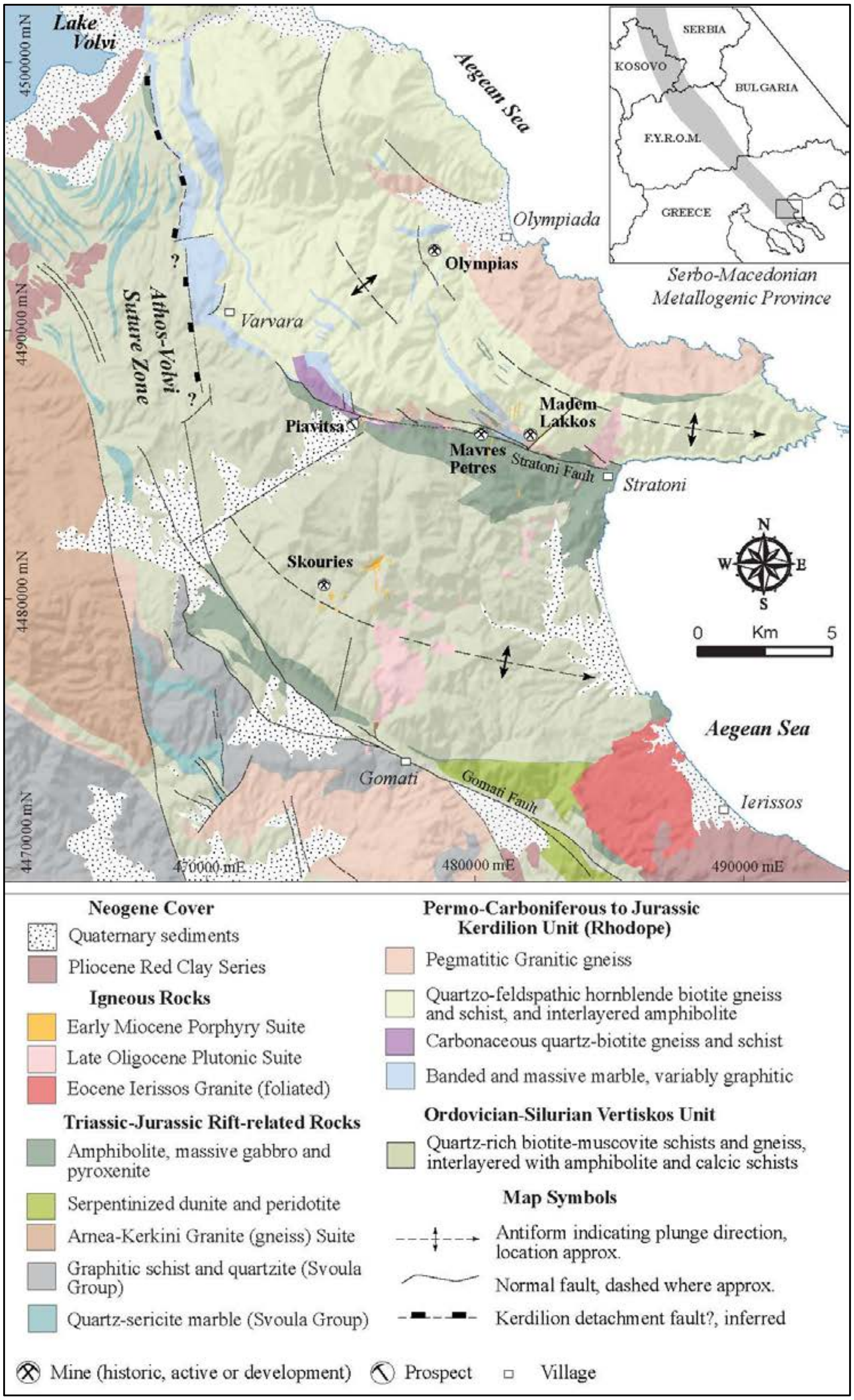


Figure 7-1: Geological Map of the Kassandra Mining District (modified from Siron et al., 2016)

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 164-134 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 are 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 a NNE-trending structural corridor defined by the alignment of igneous centers 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.

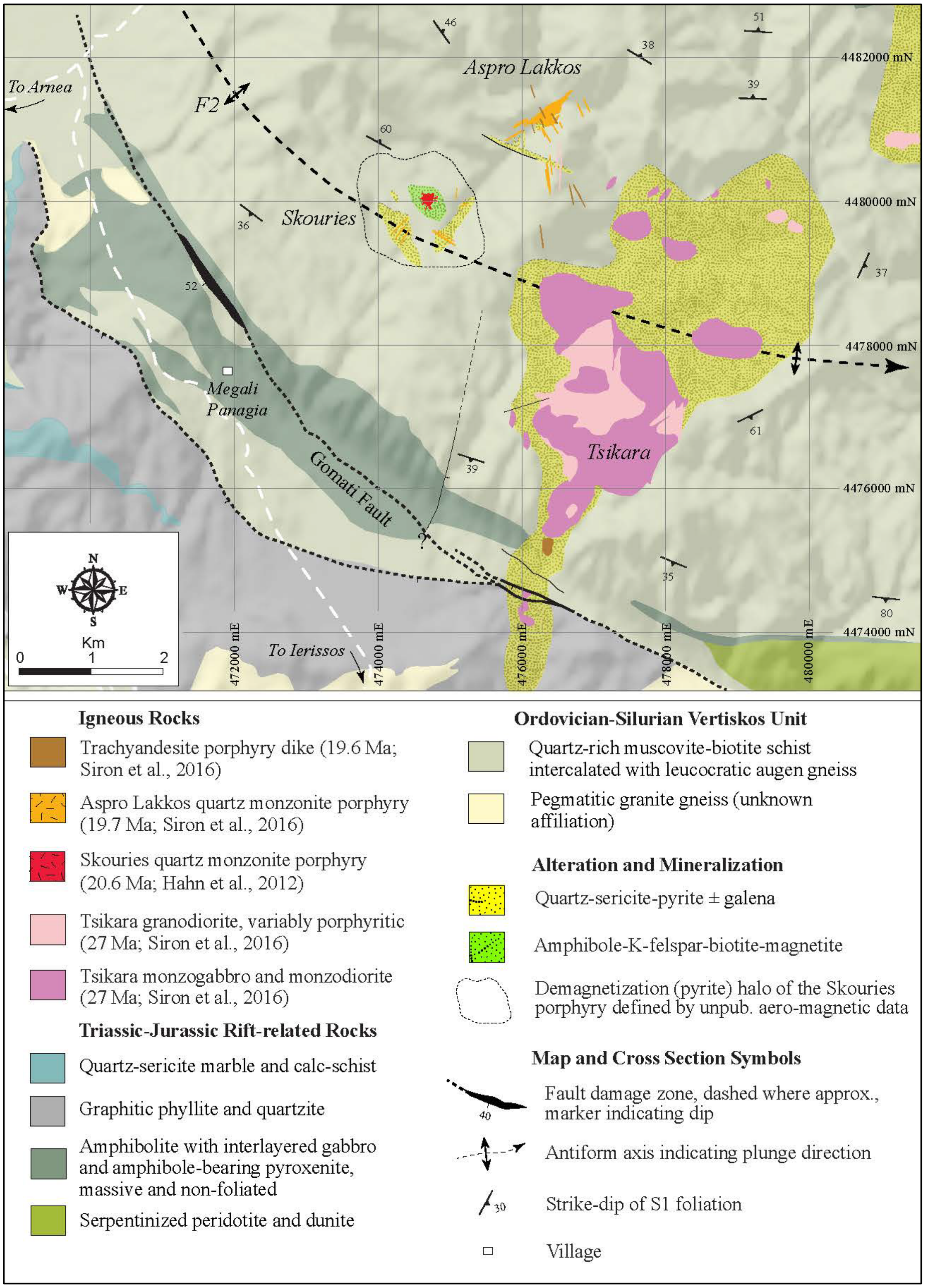


Figure 7-2: Geological Map of the Skouries deposit and surrounding area (modified from Siron et al., 2016)

SECTION • 8 DEPOSIT TYPES

8.1 DEPOSIT GEOLOGY

The Skouries deposit is centered 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 northeast dipping limb of a district-scale F2 antiform. Ore has been tested to a depth of 920 m from surface as shown on Figure 8-1. Surface exposures and drill data indicate that the porphyry stock has a subtle northeast 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 extends into the country rock; approximately two thirds of the measured and indicated tonnes are hosted outside the porphyry with about a 50:50 split in gold-equivalent ounces. The potassic alteration is characterized by K-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.
- Quartz-magnetite veinlets with chalcopyrite ± bornite.
- Quartz-biotite-chalcopyrite ± bornite-apatite-magnetite veinlets.
- Localized, late stage set of pyrite ± chalcopyrite-calcite-quartz veins.

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).

8.2 DEPOSIT MODEL

Skouries is typical of a gold-copper pencil porphyry; 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.

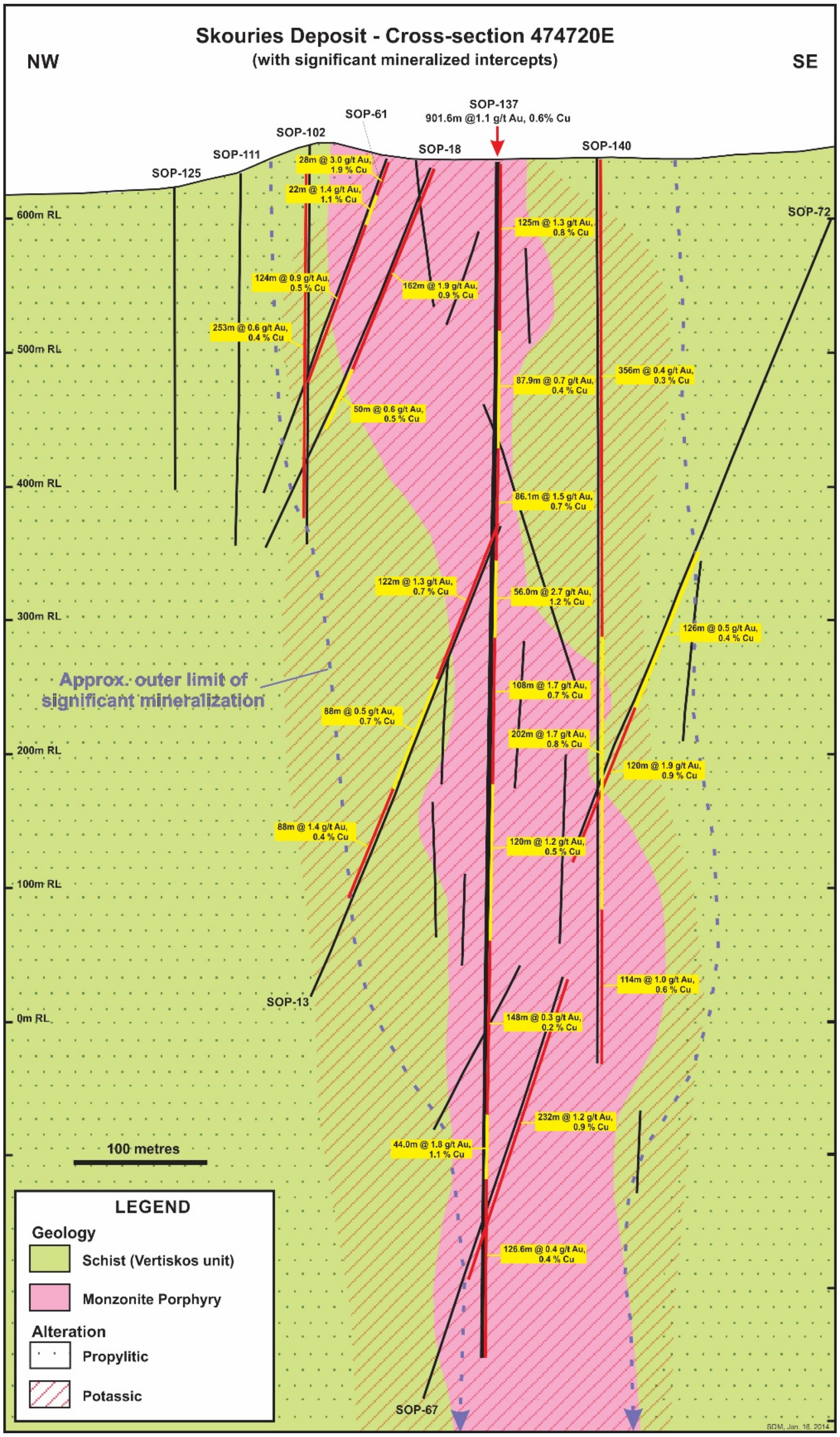


Figure 8-1: Geological Cross Section through the Skouries Porphyry Au-Cu Deposit

SECTION • 9 EXPLORATION

Eldorado has not undertaken any exploration works at the Skouries Project.

SECTION • 10 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 2012-13 by Eldorado. This data is summarized in Table 10-1. The location of these drillholes are shown on a collar plan map in Figure 10-1.

TVX drilled a total 72,232 m of core in 121 drillholes using NQ (47.6 mm) diameter core. Holes reached a maximum depth of 1,013 m. Hole deviation was measured by Sperry Sun every 50 m depth. All of the drillcore from this period was removed from site prior to Eldorado obtaining this Project through the acquisition of EGL.

Eldorado conducted two drill campaigns on the Skouries Project in 2012 - 2013: 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 category; 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 resource estimation.

Table 10-1: Summary of geological 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	1001
	SOP-01 to -98	Infill	Y	98	56,731.50	578.9	1013
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

The Eldorado drilling was done by wireline method with H-size (HQ, 63.5 mm nominal core diameter) and, less commonly, N-size (NQ, 47.6 mm nominal core diameter) equipment using up to four drill rigs. Down-hole surveys were taken every 5 m by gyro system survey tools. Drill hole collars were located respective to a property grid. Proposed hole collars and completed collars were surveyed by the Project survey team.

For the Eldorado drill campaign, standard logging and sampling conventions were used to capture information from the drill core. The core was logged in detail onto paper logging sheets, and the data were then entered into the Project database. The core was photographed before being sampled. Material was stored as stacked pallets in an organized “core farm”.

Core recovery at the Skouries Project was very good to excellent. Holes drilled mostly in schist had slight lower recovery than those drilled into the porphyry. The TVX historic recovery average was 91%. Eldorado’s pit infill drilling, mainly in schist units, averaged 91% whereas the deep confirmation drill holes that tested the bulk of the copper and gold mineralization of the deposit, averaged 96% core recovery. Overall, both the historic and recent Skouries drill programs and data capture were performed in a competent manner.

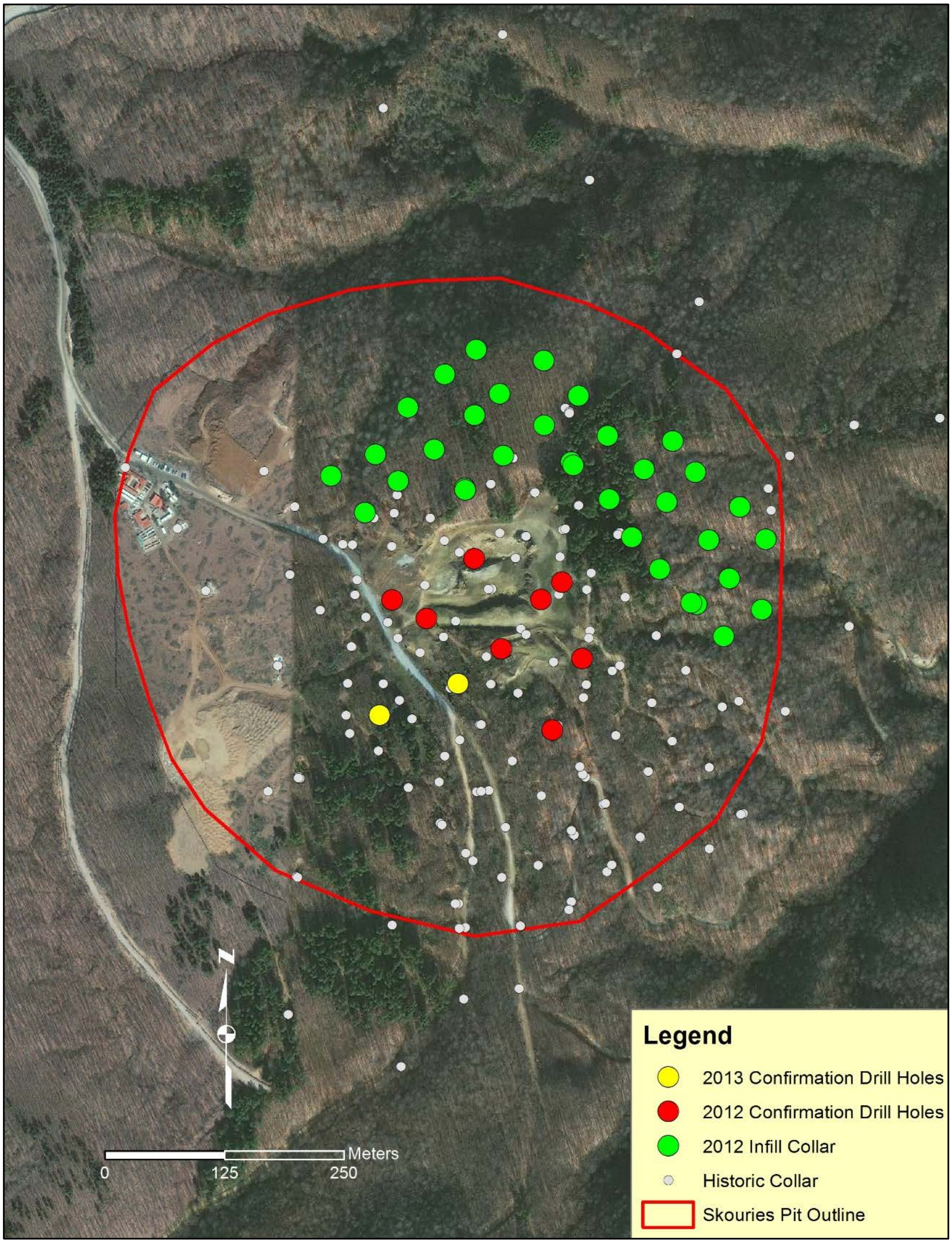


Figure 10-1: Skouries Project Drill Hole Collar Map

SECTION • 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 PRE-ELDORADO GOLD SAMPLING AND ASSAYING

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 are described in the previous technical report (Forward et al, 2011). Eldorado has reviewed these studies and data and agrees with the conclusions that the drill data are acceptable to be used for resource estimation. Confidence in those data is also supported by Eldorado's confirmation drill program. This section deals with Eldorado's sampling and quality assurance and quality control (QA/QC) work since acquiring the Skouries Project in 2012.

11.2 SAMPLING METHOD AND APPROACH

All diamond drilling in Skouries was done with wireline-equipped drilling rigs which produced drillcores of mostly HQ size. Some deep drillholes required a reduction to NQ rods to complete the drill hole. 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. Drillcore was later delivered to the logging site at the Stratoni mine area. Sample numbers were written on wooden core boxes allowing gaps in numbering sequence for control 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 diamond drillholes were sampled. Geology and geotechnical data were collected from the core and core is 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's Canakkale preparation facility in Northwest Turkey.

11.3 SAMPLE PREPARATION AND ASSAYING

The samples were crushed to 90% minus 3 mm and prepared according to the following protocol.

11.3.1 Entire Sample

- A 1 kg subsample was riffle split from the crushed minus 3 mm sample and pulverized to 90% minus 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 Standard Reference Material (SRM), a duplicate and 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 SRM was used to monitor accuracy of the assay results.

11.3.3 Sample Pulps

- The sample pulps were sent from the Çanakkale facility to AcmeLabs (now Bureau Veritas mineral laboratories) analytical laboratory in Vancouver, Canada. All samples were assayed for gold by 30 g fire assay with an AA finish, and assayed for copper by aqua regia digestion followed by ICP-MS or ICP-ES analysis.

11.4 QA/QC PROGRAM

Assay results were provided to Eldorado in electronic format and as paper certificates. Upon receipt of assay results, values for SRMs and field blanks were tabulated and compared to the establish SRM pass - fail criteria:

- Automatic batch failure if the SRM result is greater than the round-robin limit of three standard deviations.
- Automatic batch failure if two consecutive SRM results are greater than two standard deviations on the same side of the mean.
- Automatic batch failure if the field blank result is over 0.1 g/t Au.

If a batch fails, it was re-assayed until the contained control samples pass. 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.

11.4.1 Blank Sample Performance

Assay performance of field blanks is presented in Figure 11-1 for gold. The analytical detection limit (ADL) for gold is 0.01 g/t. The rejection threshold was chosen to equal 0.1 g/t. The results show no evidence of contamination.

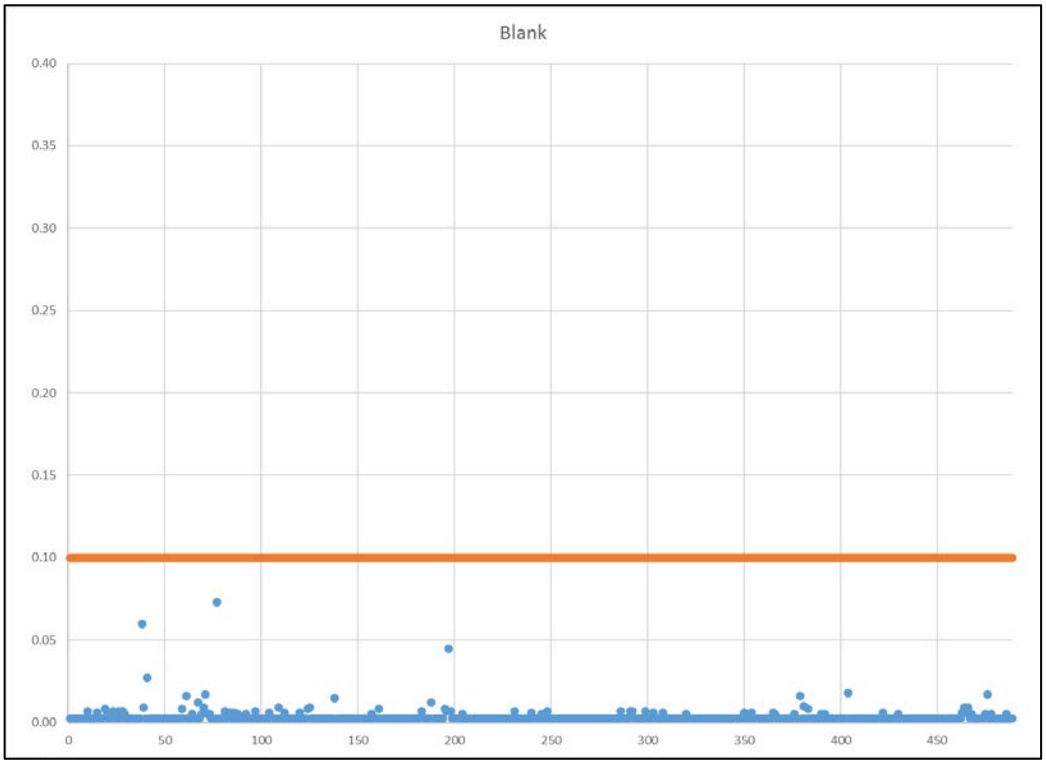


Figure 11-1: Skouries Project Blank Data, 2012-2103

11.4.2 Standard Reference Material Sample Performance

Eldorado Gold strictly monitored the performance of the SRM samples as the assay results arrived at site. Numerous SRM samples were used for both Au and Cu. Au SRM grade range covered values between 0.26 g/t to 16.0 g/t whereas the Cu SRM grade range covered values between 0.47% to 1.6%. Examples of SRM charts for both Au and Cu are shown in Figure 11-2 and Figure 11-3. All samples were given a “fail” flag as a default entry in the Project database. Each sample was re-assigned a date-based “pass” flag when assays have passed acceptance criteria.

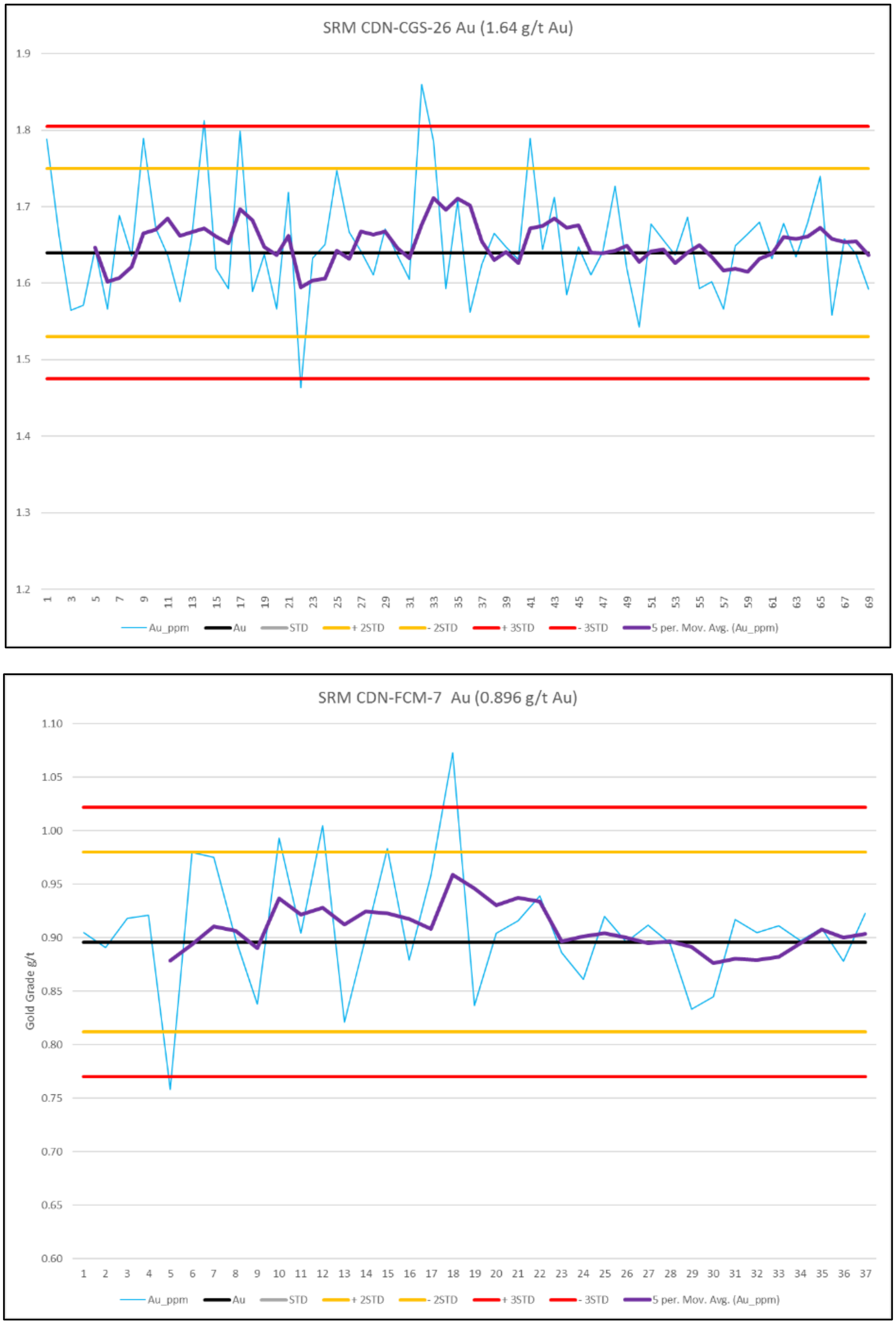
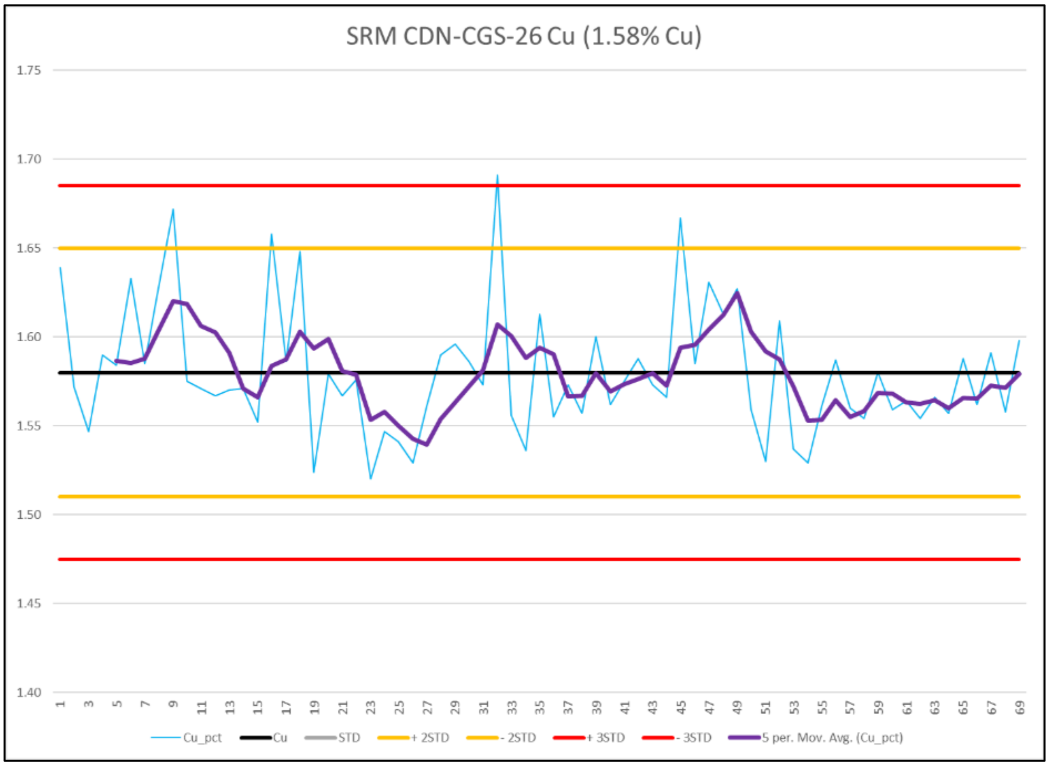


Figure 11-2: Gold Standard Reference Material (SRM) Charts, 2012-2103, Skouries Project



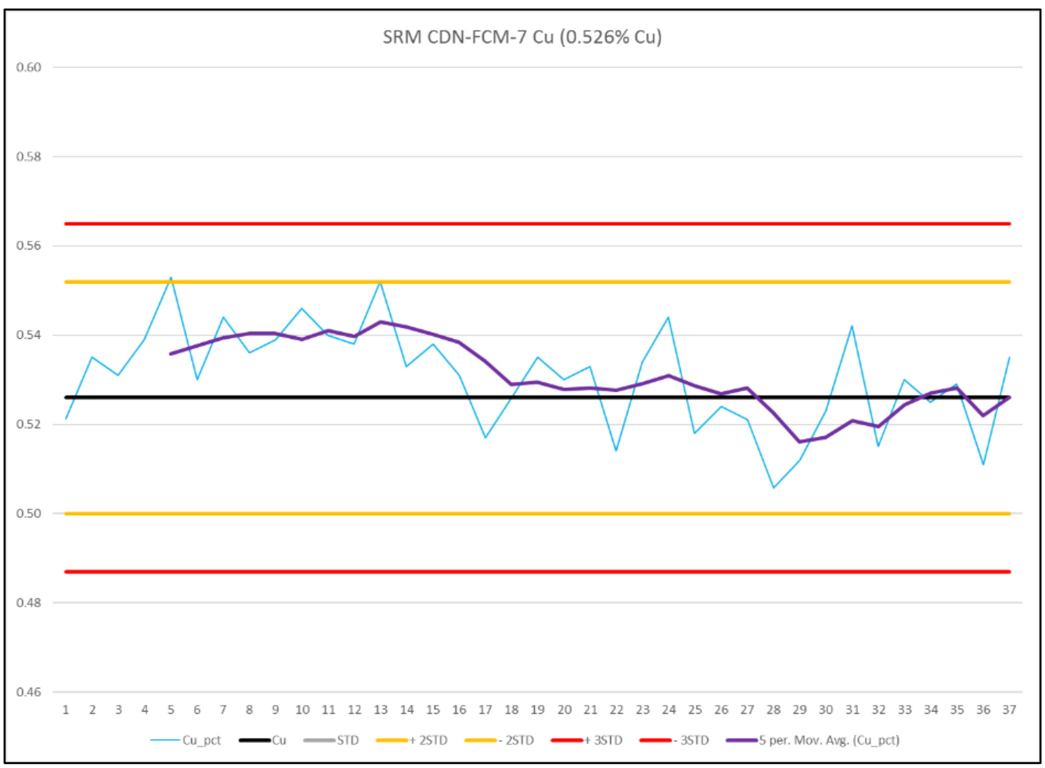


Figure 11-3: Copper Standard Reference Material (SRM) Charts, 2012-2103, Skouries Project

11.4.3 Duplicates Performance

Eldorado regularly submitted coarse reject duplicates to monitor analytical precision. Overall, this data showed acceptable precision. The duplicate data are shown in relative difference and percentile rank plots in Figure 11-4 and Figure 11-5 for gold and Figure 11-6 and Figure 11-7 for copper. Gold data shows a higher scatter than copper in the relative difference plots. This is quantified by the percentile rank plots. For the 90th percentile of the population as shown on the percentile rank plot, a maximum difference of 20% is recommended for the coarse reject duplicates because these duplicate types can be controlled by the subsampling protocol. Copper shows less than 3% difference in the coarse reject data whereas the gold hits the 20% difference threshold at the 80th percentile. At the 90th percentile gold displays 33% difference. The early 1990's campaign experienced a similar behavior for gold which they attributed to the presence of coarse gold. However Eldorado believes that the effect is not a coarse gold issue but one of readily liberated gold during the sample preparation work.

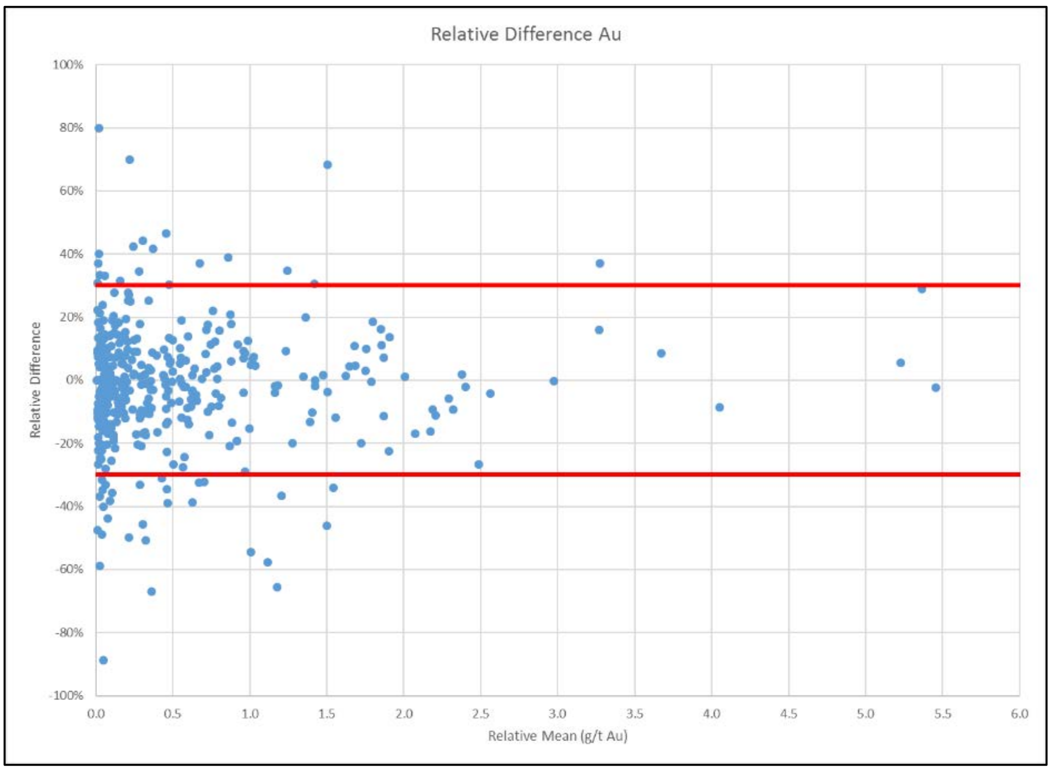


Figure 11-4: Relative Difference Plot of Gold Duplicate Data, Skouries Project, 2012-2013

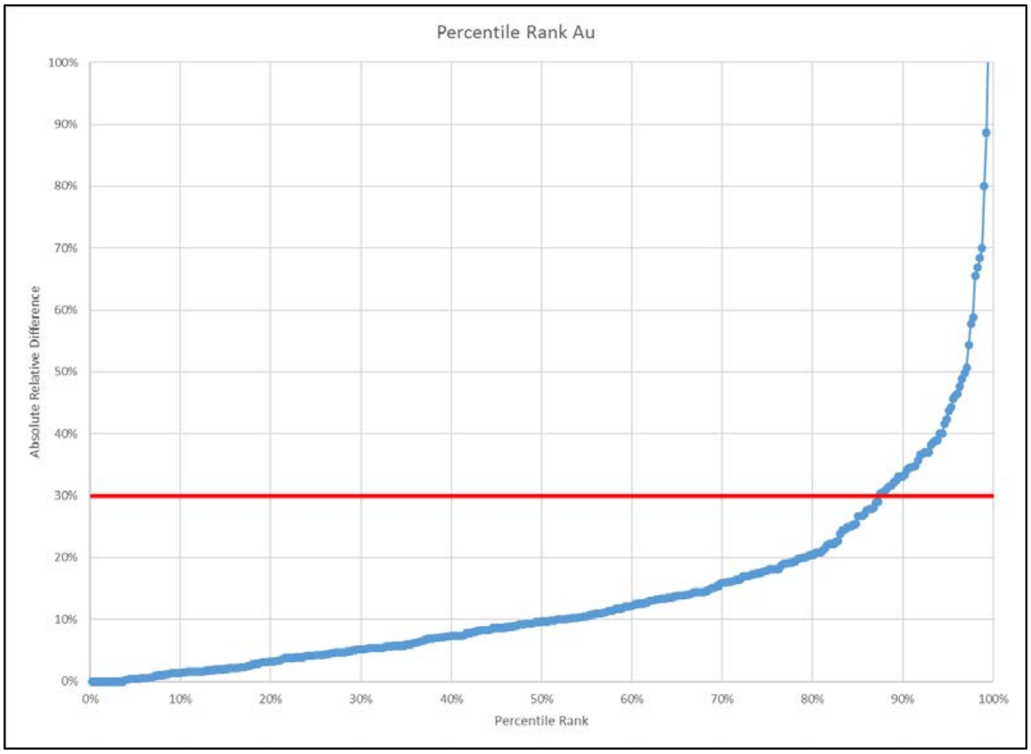


Figure 11-5: Percentile Rank Plot of Gold Duplicate Data, Skouries Project, 2012-2013

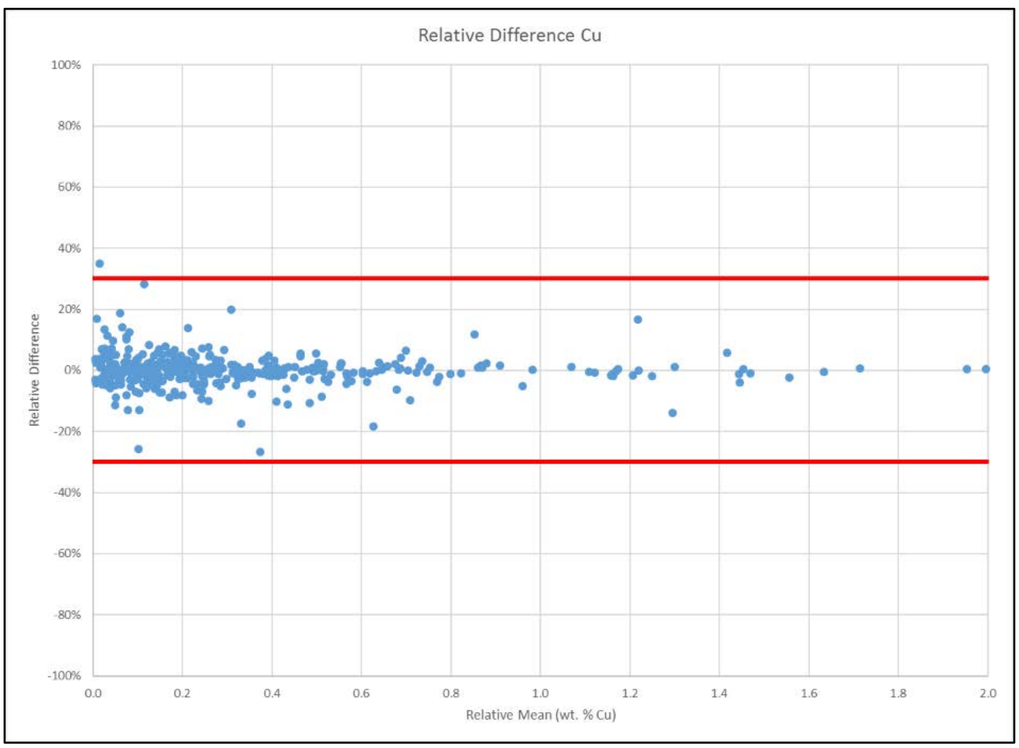


Figure 11-6: Relative Difference Plot of Copper Duplicate Data, Skouries Project, 2012-2013

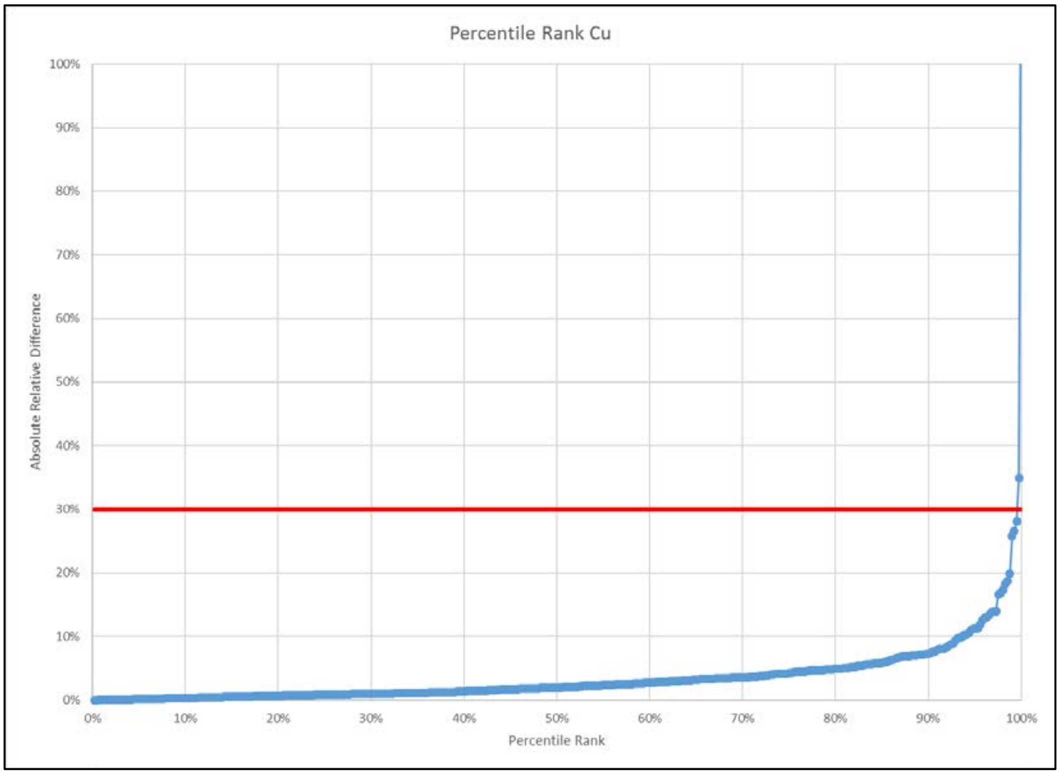
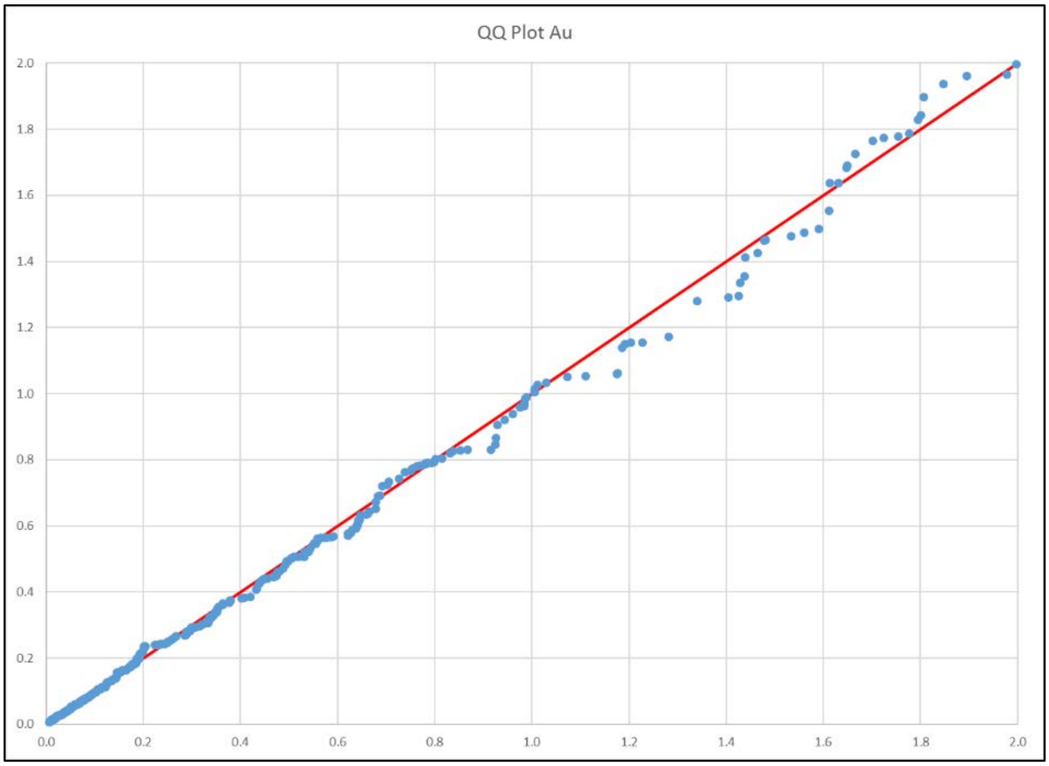


Figure 11-7: Percentile Rank Plot of Copper Duplicate Data, Skouries Project, 2012-2013

To ensure that the liberation issue with the gold mineralization at Skouries is more or less random, effects due to potential bias were investigated. First test was the pattern displayed in the relative difference plot (Figure 11-4). The data plot symmetrically about zero suggesting no bias in the assay process. The second test was on Quantile - Quantile (Q-Q) plots. If the distribution lies on or oscillates tightly about the 1:1 line, then the sample population is unbiased. This is the pattern observed for both gold and copper in Figure 11-8.



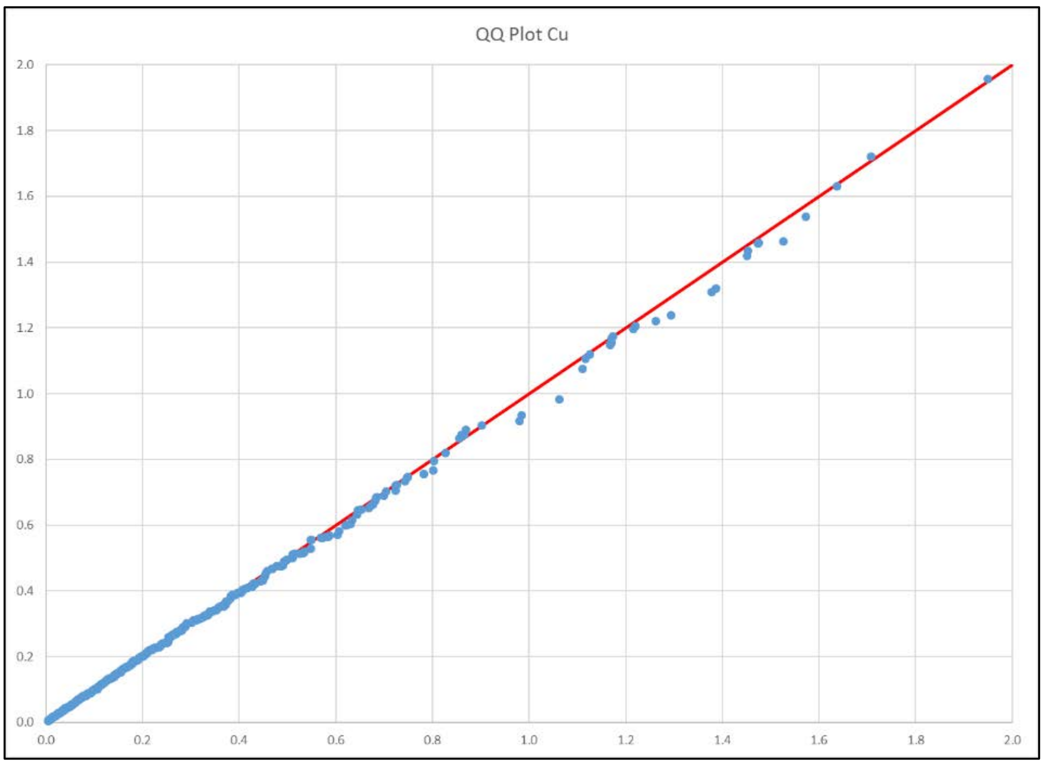


Figure 11-8: Q-Q Plots for Gold and Copper Duplicate Data, Skouries Project, 2012-2013

11.4.4 Specific Gravity Program

Samples taken for assay from core holes were measured for specific gravity and 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). The bulk density is calculated by W1 / (W1-W2).

11.5 CONCLUDING STATEMENT

In Eldorado's opinion, the QA/QC results demonstrate that the Skouries deposit assay database, particularly for new data obtained from 2012 to 2013, is sufficiently accurate and precise for resource estimation.

SECTION • 12 DATA VERIFICATION

Monitoring of the quality control samples showed all data were in control throughout the preparation and analytical processes. In Eldorado's opinion, the QA/QC results demonstrate that the Skouries Project assay and geologic database, particularly for new data obtained in 2012 and 2013, is sufficiently accurate and precise for resource estimation and grade control work.

Checks to the entire drillhole database were also undertaken. Checks were made to original assay certificates and survey data. Any discrepancies found were corrected and incorporated into the current resource database. Eldorado therefore concluded that the data supporting the Skouries Project resource work is sufficiently free of error to be adequate for estimation.

A program of confirmation drilling was completed in 2012 and 2013. These holes redrilled volumes of mineralization previously tested by the 1990s work from which no core remained. Eldorado compared the two data sets by re-estimating the Skouries mineral resource using the 1990s drillholes and 2012 infill drilling and then comparing the generated block model to the verification drill hole assay results. These comparisons are shown in Figure 12-1 for gold and Figure 12-2 for copper. The verification drillholes match the block model grade very well. Thus Eldorado was able to verify the results obtained from the 1996-98 drill campaign despite having none of that drillcore available.

Taken all together, these observations demonstrate that the data gathered and measured for the purposes of estimating the gold grades at the Skouries Project are verified.

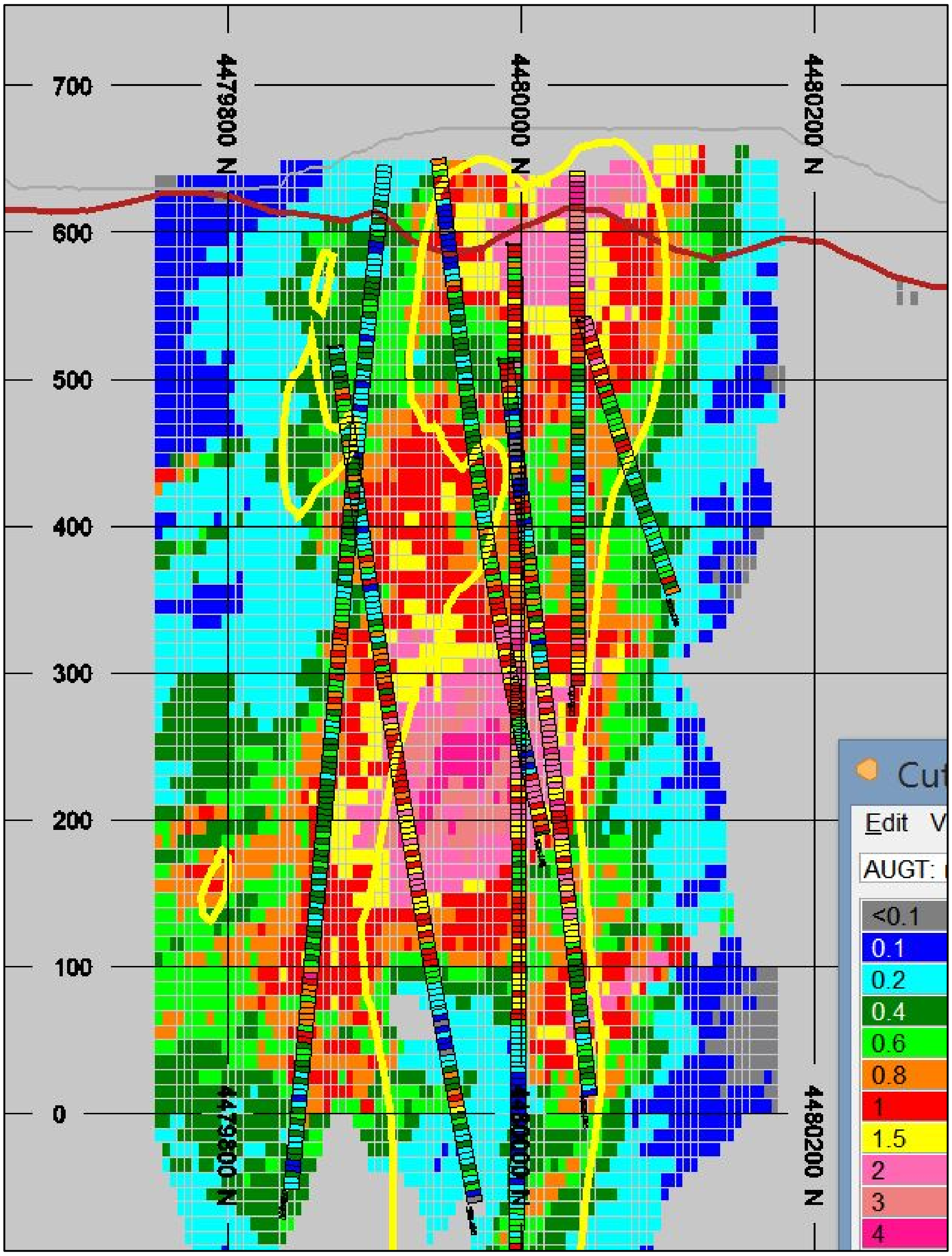


Figure 12-1: Comparison of Gold Grades for Verification Drillholes and Block Model

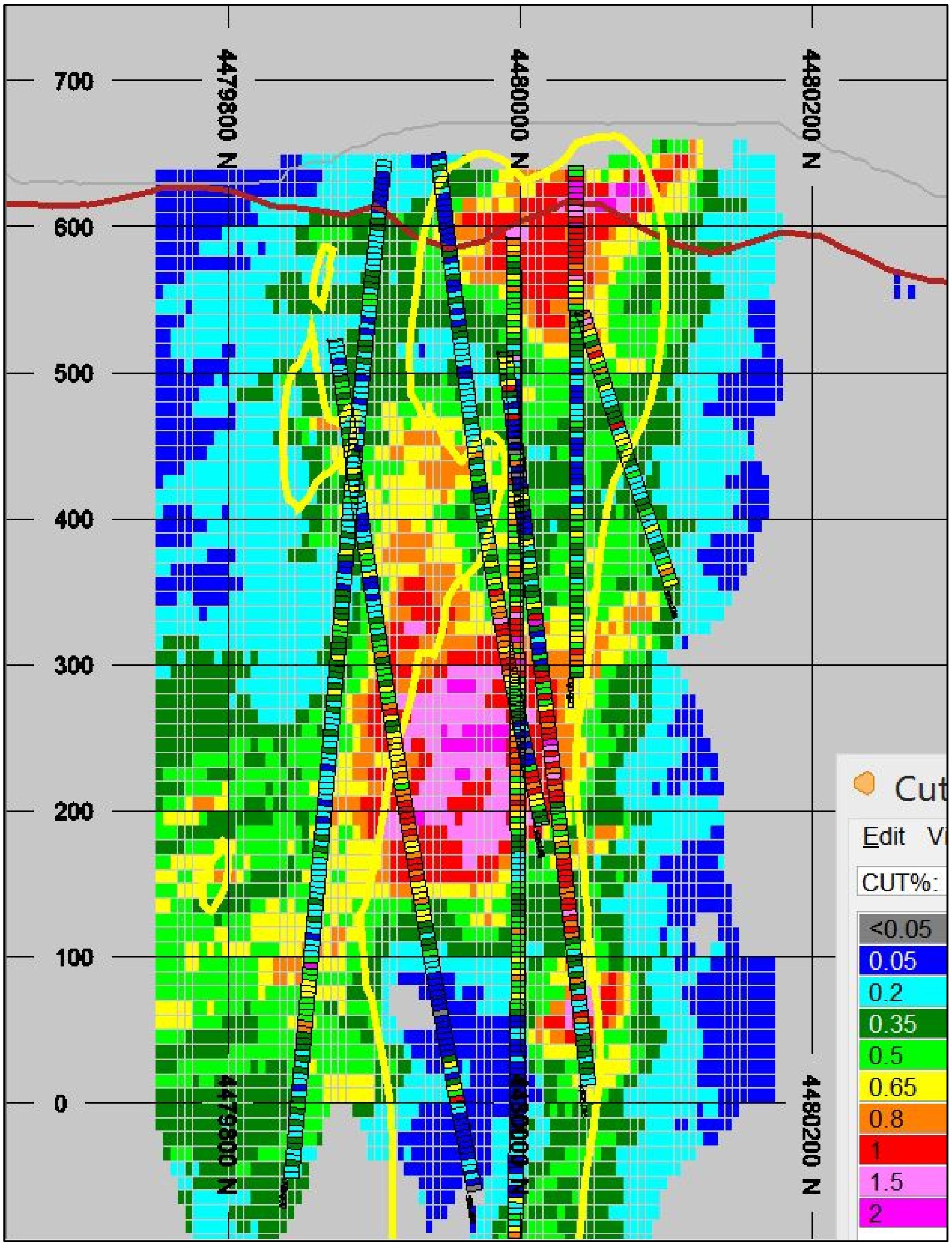


Figure 12-2: Comparison of Copper Grades for Verification Drillholes and Block Model

SECTION • 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 criterion for process plant and infrastructure design were established.

Additional testwork was completed by Outotec in 2007, mostly at their 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 (former Cytec) in 2016 and Bureau Veritas Commodities Canada in 2017 worked on selective flotation of copper from pyrite-rich ore.

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 to 160 µm. Gold also occurs as blebs within sulphides, particularly in bornite and chalcocite. It correlates strongly with copper. Palladium was discovered to occur during metallurgical testing which could add value to the ore. The oxide zone occurs from surface to 30 to 70 m depth and occasionally deeper consisting mainly of malachite, cuprite, secondary chalcocite and minor azurite, covellite, digenite and native copper.

The bench-scale grinding testwork considered ore types, boundaries, hardness maps, grade and mine plan to determine the grinding mill design. The mill selection is based on the results of the testwork and mill vendor recommendations.

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 were divided into three types, high oxide (HO), medium oxide (MO) and low oxide (LO) materials, 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 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 that material thought to be un-associated open pit sulphide 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 applied mimicked the same flowsheets as that used during the earlier testing of primary sulphide ores. Sodium hydrosulphide was used as a sulphidiser agent to float the oxide ores.

The oxide ores to be processed in the first year of operation and rehandled in Phase 2 have significantly lower copper and gold recoveries compared to the sulphide mineralization. These are estimated to be approximately 50% and 70% respectively. These can be compared to the life of mine (LOM) average recoveries of 87.2% for copper and 82.4% for gold when sulfide ore is floated. The testwork has shown that the oxide copper minerals and the associated gold can be recovered by conventional sulphidiser 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 will not be cost effective to grind to the fineness required to liberate these locked copper sulphides particularly as the observation relates to only the first year of operation. It is also well known that ultra-fine particle flotation of sulphides is not particularly successful in terms of both value recovery and concentrate grade. Ultra-fine particles also cause difficulties in the thickening and filtration operations.

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 for the process plant as currently proposed, i.e. including any gold recovered by the gravity concentration circuits. The equations developed are of the mathematical form:

- $y = a - bx - ce^{-dz}$

Where y represents copper or gold recovery; a, b, c and d are constants, x is the % oxide expressed as Cu_L:Cu_T, e is natural e and z is the respective copper or gold head grades.

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 have been further developed by SRK in their Mining Pre-feasibility Study of November 2005. Their current forms are given below:

- Copper Recovery (%) = 99.4 – 56.0 x % oxide – 41.0 x $e^{(-338 \text{ x Cu Head Grade \%})}$
- Gold Recovery (%) = 92.6 - 17.5 x % oxide – 22.0 x $e^{(-1.2 \text{ x Au Head Grade g/t})}$

Where “% oxide” or degree of oxidation = 100 x (Cu_L/Cu_T) as defined earlier. Cu_T is total copper, and Cu_L is sulfuric acid soluble copper.

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

These equations have been applied to the Project's mine planning and have been found to give reasonable values for the copper and gold recoveries.

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

Further gold gravity concentration testwork was undertaken in 2013 by FLSmidth Knelson and confirmed the applicability of centrifugal concentrators for gold recovery in both primary grinding circuit and regrind circuit. This testwork is the basis for the current gravity concentration circuit design.

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 did show that the unit cell flash flotation could recover the mineralised 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 unit cell can take place if this proves beneficial in the later years of operations.

The testwork also demonstrated that the number of flotation concentrate cleaning stages needs to be increased from two stages to three stages in order to achieve the targeted concentrate grade of 26% copper during periods when low grade ore is processed. Chemical analysis of the flotation concentrate samples during early studies provided further evidence of the presence of platinum group metals (PGM) particularly palladium.

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.

SECTION • 14 MINERAL RESOURCE ESTIMATES

The mineral resource estimate for the Skouries deposit used data from surface diamond drillholes. The resource estimate was made from a 3D block model created by utilizing commercial mine planning software. The block model cell size was 5 m east by 5 m north by 10 m high.

14.1 MINERALIZATION DOMAINS

As with many porphyry deposits, using only lithology or alteration based domains to constrain grade interpolation fails 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.

14.2 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 discover whether statistically distinct domains needed to be defined. The lithology categories porphyry and schist (includes all non-porphyry units) were investigated within the mineralized shell.

Descriptive statistics, histograms and cumulative probability plots, box plots and contact plots have been 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 was converted to 4 m downhole composites. The statistical properties from this analysis are summarized in Table 14-1.

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 variance (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 by local extreme grades.

Table 14-1: 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

14.3 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.

14.4 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 14-2 and Table 14-3.

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 14-2: 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) or exponential (EXP). 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.

Table 14-3: 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.

14.5 MODEL SETUP

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

The assays were composited into 4 m fixed-length down-hole 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 was 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 and 2.73 represented historical averages for the intrusive units (porphyry domain) and non-intrusive units (schist domain), respectively. More recent specific gravity (SG) tests, conducted as checks by Eldorado, yielded very similar averages.

A 30 to 70 m thick, near surface oxidation of sulphide minerals has occurred at Skouries. This surface has been modeled and used to tag the model into an oxide and sulphide zone. Specific sub-units within this unit, namely overburden and red clay, contain no appreciable metal values. Model grades in this sub-units where reset to zero after grade interpolation.

14.6 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 on 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 uninterpolated 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 2 and maximum of 3 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 4.

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 hard 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 number of composites were adjusted to incorporate an appropriate amount of grade smoothing.

14.7 VALIDATION

14.7.1 Visual Inspection

Eldorado completed a detailed visual validation of the Skouries 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 done. 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. The PACK shell is shown by the outer green outline; the Porphyry unit by the thick yellow line and the reddish brown line demarcates the oxide – sulphide contact and the bold white line represents the open pit design. Small model blocks denote the Inferred mineral resource.

14.7.2 Model Checks for Bias

The block model estimates were checked for global bias by comparing the average metal grades (with no cutoff) 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 cutoff grade is imposed and is a good basis for checking the performance of different estimation methods. Results, summarized in Table 14-4, show no problems with global bias in the estimates.

Table 14-4: 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

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 Skouries model.

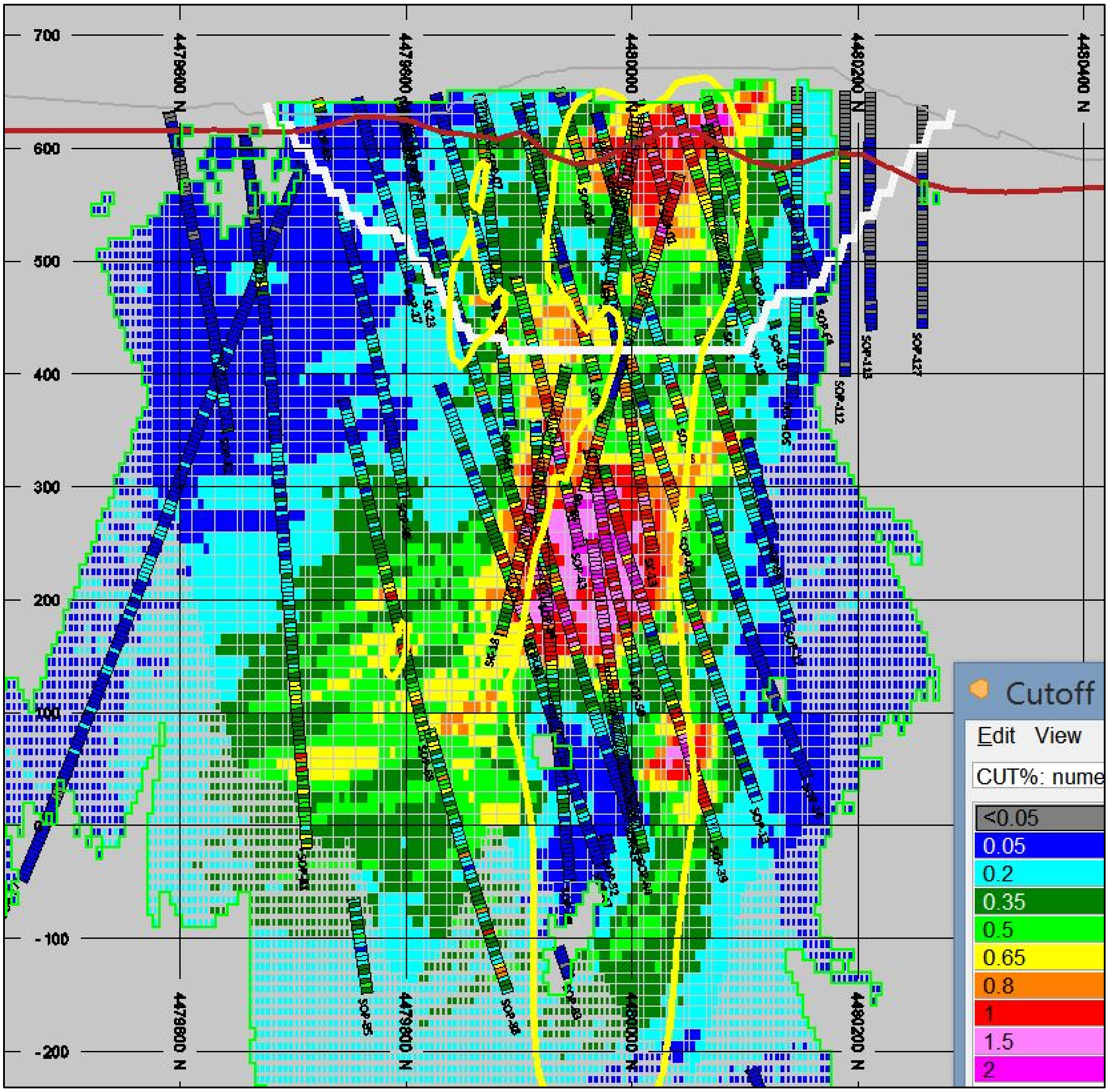


Figure 14-1: Skouries N-S section 474700E showing copper block model values and drillhole composites grades.

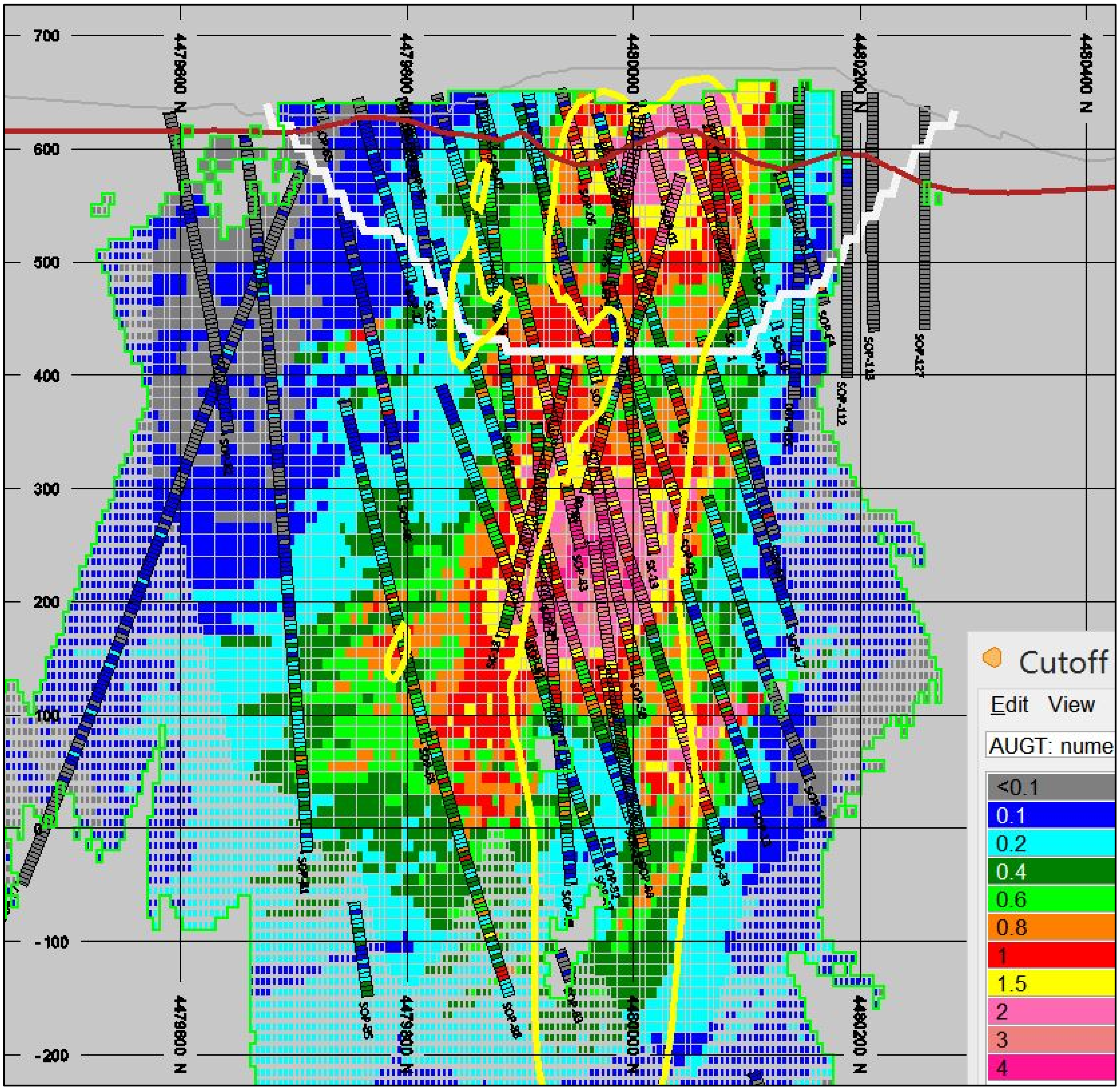


Figure 14-2: Skouries N-S section 474700E showing gold block model values and drillhole composites grades.

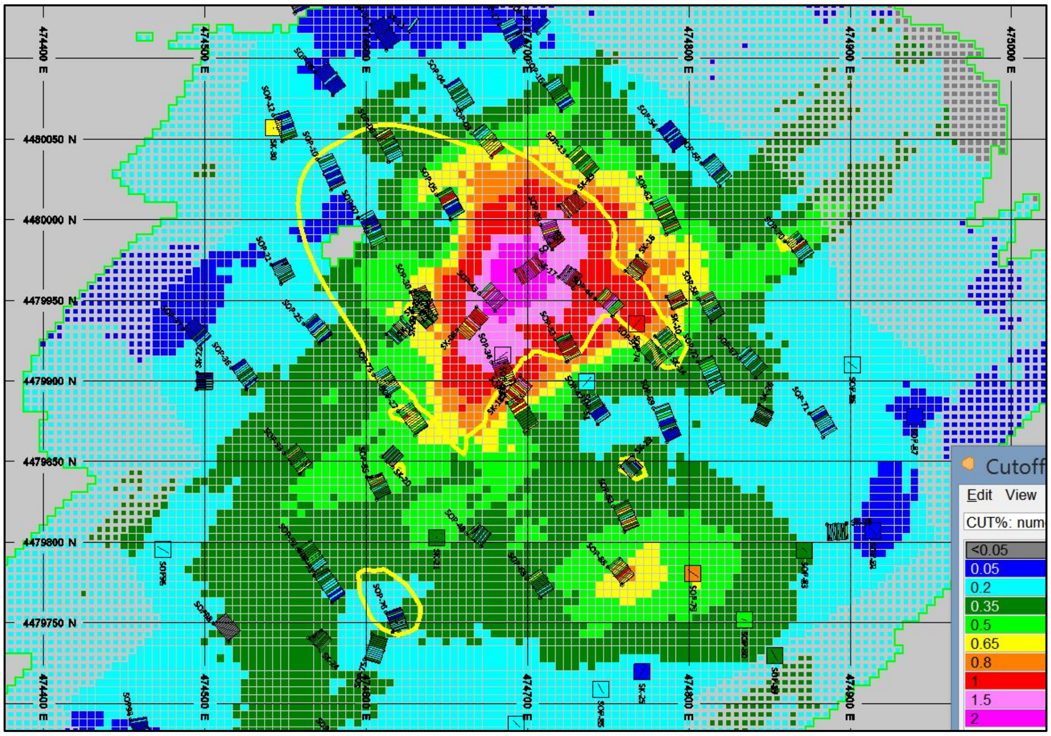


Figure 14-3: Skouries plan views at 245 m elevation showing copper block model values and drillhole composites grades.

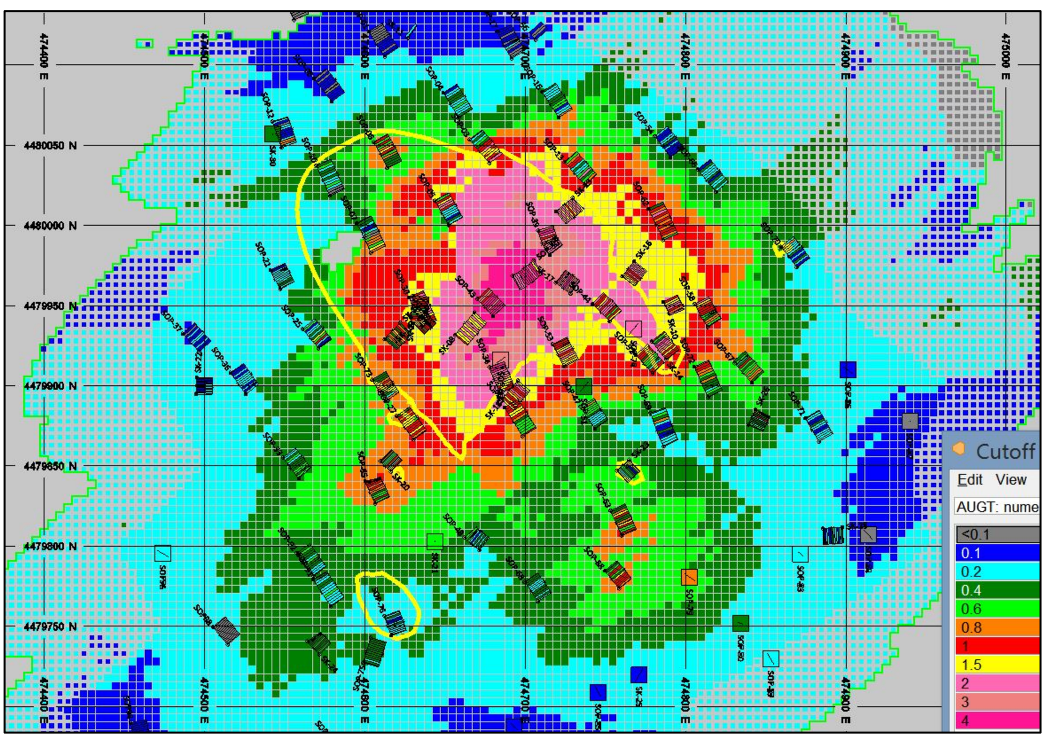


Figure 14-4: Skouries plan views at 245 m elevation showing and gold block model values and drillhole composites grades.

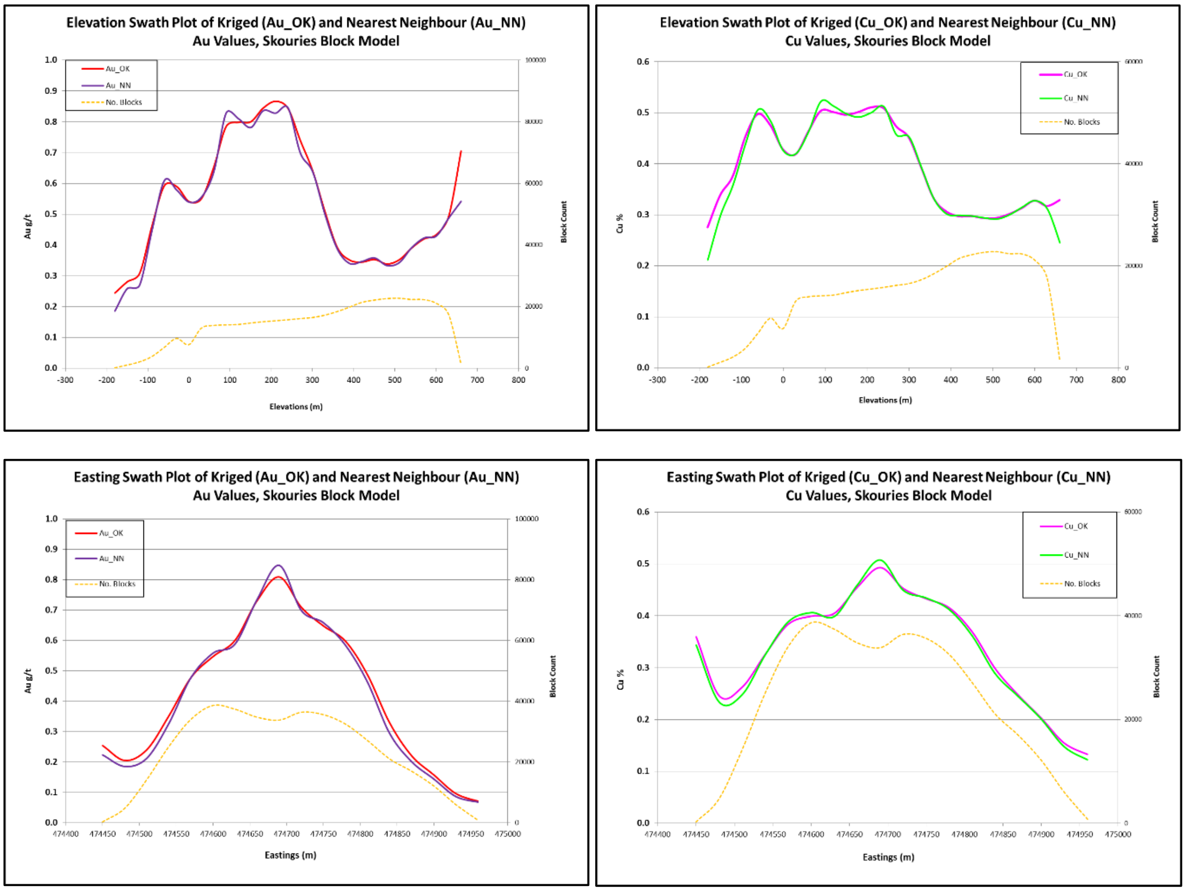


Figure 14-5: Swath Plots of Skouries Block Model Gold and Copper Values

14.8 MINERAL RESOURCE CLASSIFICATION

The mineral resources of the Skouries deposit were classified using the CIM definitions referred to in NI 43-101. The mineralization of the Project satisfies sufficient criteria to be classified 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 predicting planned annual and quarterly production, contributed to the setup 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 is 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. Where the sample spacing was about 50 m or less, the confidence in the grade estimates and lithology contacts were the highest thus permissive to be classified as measured mineral resources. A three-hole rule was used where blocks containing an estimate resulting from three or more samples, all within 50 m and from different holes, were classified as measured mineral resources.

All remaining model blocks containing a gold grade estimate was assigned as inferred mineral resources.

14.9 MINERAL RESOURCE SUMMARY

The Skouries mineral resources as of December 31, 2017 are shown in Table 14-5. The Skouries mineral resource is reported at a 0.2 g/t Au equivalent cutoff grade for open pit mineral resources and 0.60 g/t Au equivalent cutoff grade for underground mineral resources. The gold equivalent formula is: AuEqv = Au (g/t) + 1.6 * Cu (%).

Table 14-5: Skouries Mineral Resources, as of December 31, 2017

Category	Resource (t x 1,000)	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz x 1,000)	Contained Cu (t x 1,000)
Open Pit Resources					
Measured	61,729	0.56	0.39	1,121	244
Indicated	27,734	0.21	0.22	189	61
Measured & Indicated	89,463	0.46	0.34	1,310	305
Inferred	3,744	0.12	0.15	15	6
Underground Resources					
Measured	38,289	1.15	0.63	1,413	240
Indicated	161,529	0.52	0.43	2,678	697
Measured & Indicated	199,818	0.64	0.47	4,091	937
Inferred	166,393	0.31	0.34	1,665	572
Total Resources					
Measured	100,018	0.79	0.48	2,534	484
Indicated	189,263	0.47	0.4	2,867	758
Measured & Indicated	289,281	0.58	0.43	5,401	1,242
Inferred	170,136	0.31	0.34	1,680	578

Au Equivalent cut-offs: Open Pit = 0.20 g/t ; Underground = 0.60 g/t AuEqv = Au g/t + 1.6 * Cu%
Only material within the Jan 2016 pit design was cast as open pit resources; all other material was considered as underground

SECTION • 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 15.2.

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.

15.1.1 Open Pit Optimization

15.1.1.1 Introduction

The open pit optimization was carried out using Minesight® mine planning software. A series of unsmoothed pit shells were created using a Lerchs Grossman algorithm with revenue factors declining from unity. The unsmoothed pit shells were then used as a guide for developing a detailed design 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 is also affected by a requirement to provide construction materials for the integrated waste management facility (IWMF).

15.1.1.2 Economic Parameters Applied to Mine Design

Metal Prices

Base case pit optimization metal prices are as follows:

- Copper: US$2.50/lb
- Gold: US$1,200/oz

Smelter Terms and Offsite Costs

Copper will be recovered by flotation methods, and report to a single copper concentrate. Gold will also report to the copper concentrate, and in addition, gold will report to the gravity concentrate that will be refined at site and shipped as doré. 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 net smelter return 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 test block within the sulphide zone of the resource block model.

Table 15-1: Example of Net Smelter Return Calculation

Test Block NSR Calculation	Units	Test Block Values
Copper Head Grade	%	0.360
Gold Head Grade	g/t	0.576
Oxide Ore	%	-
Metallurgical Recovery		
Copper Recovery	%	87.27%
Gold Recovery	%	81.60%
Metal Pricing		
Copper Price Participation Level	US$/lb	$2.50
Copper Price Participation	%	0.0%
Copper Price Participation Cap	US$/lb	$0.00
Copper Price	US$/lb	$2.50
Price Participation	US$/lb	$0.00
Copper Price Realized Net of PP	US$/lb	$2.50
Gold Price	US$/ounce	$1,200.00
Copper Concentrate		
Copper Concentrate Grade		26.0%
Moisture Content	%	9.0%
Contained Copper	lb/dmt	573.20
Contained Gold	g/dmt	31.90
Payable Copper	lb/dmt	551.15
Payable Gold	g/dmt	30.85
Concentrate - Recovery Based	dmt/t ore	0.0121
Gross Value Concentrate	US$/dmt	**$2,562.03**
Dore		
Gold to Dore	%	18.0%
Gold Payment Dore	%	100.0%
Gold Deduction	%	99.7%
Gold Recovered	grams	0.085
Gold Payable	grams	0.084
	US$/t ore	**$3.25**
Copper Concentrate Handling		
Mine to Port Concentrate Freight Cost	US$/wmt	$50.00
Ocean Concentrate Freight Cost	US$/wmt	$25.00
Total Concentrate Handling	US$/wmt	$75.00
Total	US$/dmt	**$82.42**
	US$/t ore	**$1.00**
Copper Concentrate Treatment and Refining		
Deduction for Copper	unit	1.0%
Copper Payment	%	96.2%
Treatment Charges	US$/dmt	$82.50
Gold to Concentrate	%	82.0%
Gold Deduction - Adjustment	%	99.7%
Gold Payment Concentrate	%	97.0%
Copper Refining Cost	US$/payable lb	$0.0825
Gold Refining Cost	US$/payable oz	$6.00
Total Treatment and Refining	US$/dmt	**$128.16**
	US$/t ore	**$1.55**
Copper Net Smelter Return		
Net Smelter Return	US$/dmt	**$2,351.45**
	US$/payable lb Cu	$4.27
NSR Before Royalty - Including Dore	**NSR US$/t**	**$31.67**
Royalty Gold	%	1.50%
Royalty Copper	%	0.50%
Gold Value for Royalty Calculation	**NSR US$/t**	**$17.56**
Copper Value for Royalty Calculation	**NSR US$/t**	**$14.11**
Check	**NSR US$/t**	**$31.67**
Royalty Gold	**US$/t**	**$0.263**
Royalty Copper	**US$/t**	**$0.071**
NSR After Royalty	**NSR US$/t**	**$31.33**

Concentrate transportation costs were estimated at US$75.00/wmt. The concentrate is assumed to have a moisture content of 9.0%.

Treatment charges for copper concentrate were estimated to be US$82.50/dmt and refining charges were estimated to be US$0.0825/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 96.2% payability for copper in concentrate.

The NSR calculations allows 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 1.5% for gold and 0.5% for copper were also applied to the NSR model as described in Section 22.

Onsite Operating Costs and Increments

The onsite operating costs used for pit limit analysis include general and administration (G&A), processing, mining and tailings handling costs. The G&A, processing costs, and tailings costs were estimated to be US$1.37/t and US$7.07/t and US$1.03/t respectively. Preliminary operating costs for mining ore and waste were US$1.27/t and US$2.31/t respectively and reflected estimates for local contractor rates. An incremental haulage cost of US$0.046/t/bench was added for each 10 m bench below the open pit entrance at 620 masl.

15.1.1.3 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. A gravity circuit is also included to capture free gold as a concentrate to be refined to doré. Approximately 18% of the recoverable gold is estimated to report to the gravity concentrate.

Process Recovery

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

- Recovery (Cu) = 99.41 – 56 x % oxide – 41 x e^(-338 x Cu Head Grade %)
- Recovery (Au) = 92.62 – 17.5 x % oxide – 22 x e^(-1.2 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.

15.1.1.4 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 3-D model. The Minesight® mine planning block model limits and block dimensions are shown in Table 15-2.

Table 15-2: Block Model Limits

Parameters	Units	Values		
Model Limits 2015-2017		**Minimum**	**Maximum**	**Length**
Limits X	metres	473,877.06	475,552.06	1,675.00
Limits Y	metres	4,478,985.50	4,480,785.50	1,800.00
Limits Z	metres	640	860	1,500.00
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

Resource Class: 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.

Mining Recovery**:** Mining recovery is assumed to be 100%. No mining losses were applied to the ore reserves for the following reasons:

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

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 was applied in pit optimization.

15.1.1.5 Wall Slope 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 15-3.

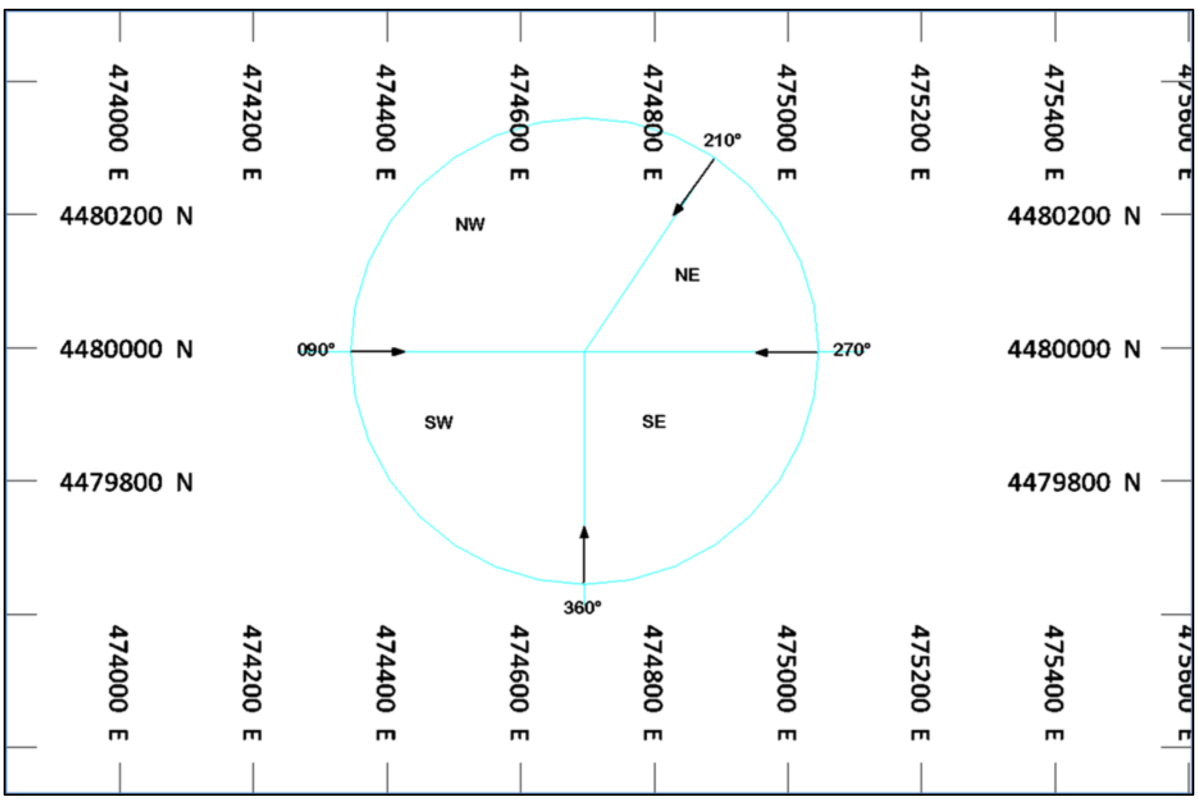


Figure 15-1: Slope Sector Orientation

Table 15-3: 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

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

15.1.1.6 Pit Limit Analysis

Unsmoothed pit limits were developed using a Minesight® variable slope Lerchs Grossman algorithm. The preliminary net minegate revenue and operating costs were used to estimate the value of each regular block in the model. A series of 30 nested pit limits were defined using revenue factors between 0.20 and 1.00. As noted in previous sections, the pits were constrained by an EIS boundary and the crown pillar elevation at 420 masl. The nested pit limits used to develop the pit design are shown in Figure 15-2 and Figure 15-3.

The resources within the unsmoothed, constrained Lerchs-Grossman pit shells are summarized in Table 15-4. The revenue factors that are shown were applied to the NSR values. The overall size of the pit is limited by the EIS constraints and the crown pillar location. The pit limits are relatively insensitive to metal prices within the ultimate constrained pit limits.

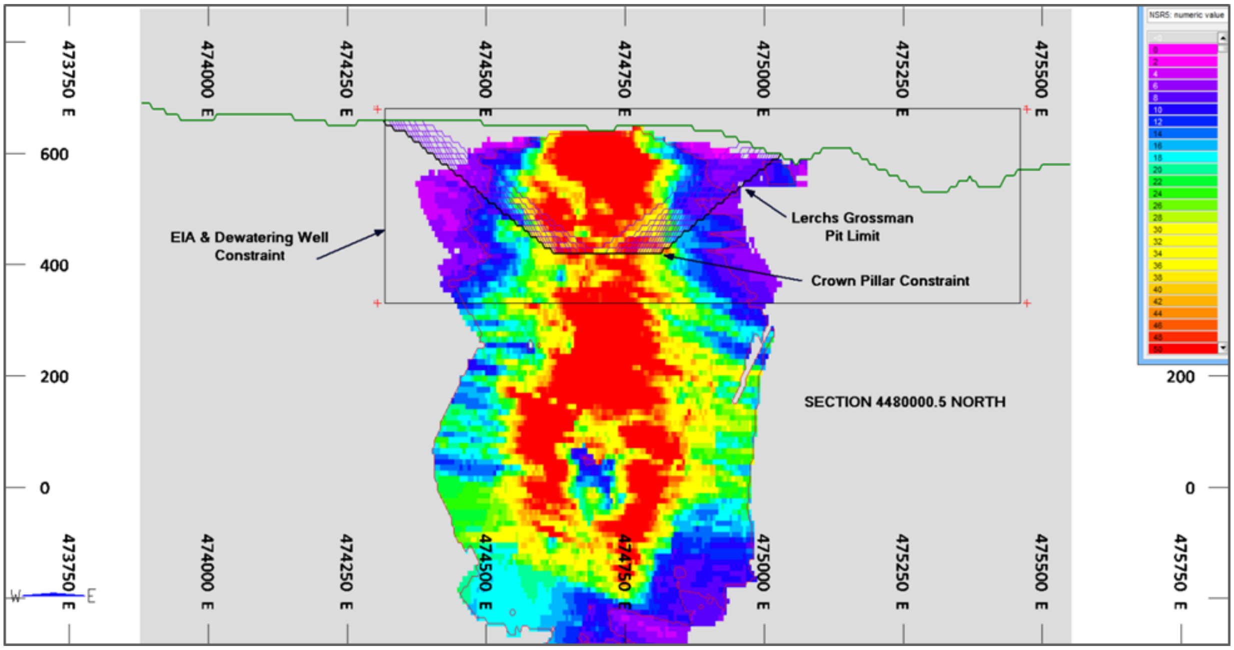


Figure 15-2: Cross Section NSR Lerchs Grossman Pit Limits

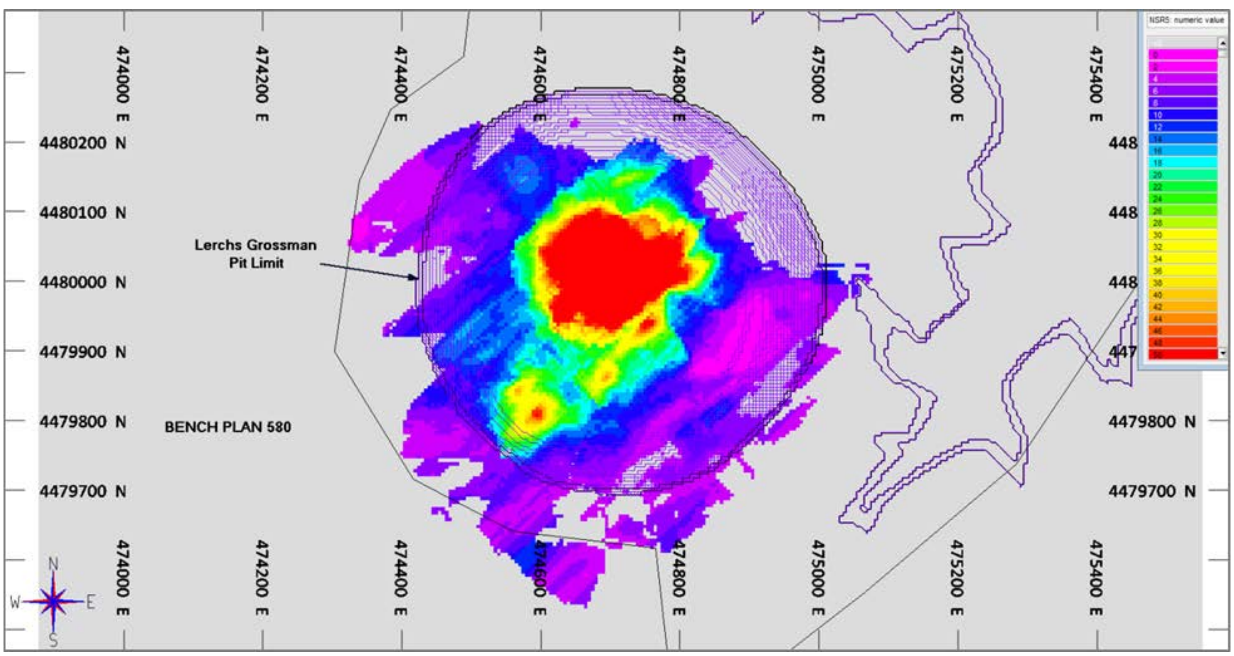


Figure 15-3: Bench Plan NSR Lerch Grossman Pit Limits

Table 15-4: Lerchs Grossman In-Pit Resources

	Revenue Factor	Insitu Ore (kBCMS)	ROM Ore (ktonnes)	Waste (ktonnes)	ROM S/R	Insitu NSR5	Grades CuCut	AuCut	Oxid%	CuRec	AuRec
PIT01	0.20	12,835.0	32,791.0	16,026.0	0.49	38.56	0.46	0.72	13.12	80.85	79.52
PIT02	0.23	14,962.0	38,492.0	20,127.0	0.52	36.96	0.44	0.68	11.19	81.58	79.47
PIT03	0.26	16,556.0	42,788.0	23,844.0	0.56	35.91	0.43	0.65	10.07	81.95	79.41
PIT04	0.28	17,312.0	44,834.0	25,951.0	0.58	35.48	0.43	0.64	9.61	82.09	79.37
PIT05	0.31	17,975.0	46,632.0	28,021.0	0.60	35.11	0.43	0.63	9.24	82.20	79.34
PIT06	0.34	18,529.0	48,134.0	29,941.0	0.62	34.80	0.42	0.63	8.95	82.27	79.31
PIT07	0.37	18,811.0	48,899.0	30,913.0	0.63	34.62	0.42	0.62	8.81	82.30	79.29
PIT08	0.39	19,134.0	49,777.0	32,224.0	0.65	34.44	0.42	0.62	8.66	82.33	79.27
PIT09	0.42	19,418.0	50,549.0	33,351.0	0.66	34.26	0.42	0.61	8.53	82.35	79.25
PIT10	0.45	19,598.0	51,039.0	34,089.0	0.67	34.14	0.42	0.61	8.44	82.37	79.23
PIT11	0.48	19,835.0	51,683.0	35,162.0	0.68	33.99	0.42	0.61	8.34	82.38	79.21
PIT12	0.50	19,903.0	51,869.0	35,468.0	0.68	33.95	0.41	0.61	8.31	82.39	79.20
PIT13	0.53	20,210.0	52,704.0	36,945.0	0.70	33.74	0.41	0.60	8.18	82.40	79.17
PIT14	0.56	20,246.0	52,801.0	37,129.0	0.70	33.72	0.41	0.60	8.16	82.41	79.17
PIT15	0.59	20,370.0	53,138.0	37,714.0	0.71	33.63	0.41	0.60	8.11	82.41	79.15
PIT16	0.61	20,421.0	53,279.0	38,000.0	0.71	33.60	0.41	0.60	8.09	82.42	79.15
PIT17	0.64	20,510.0	53,520.0	38,467.0	0.72	33.53	0.41	0.60	8.05	82.42	79.14
PIT18	0.67	20,604.0	53,777.0	38,988.0	0.72	33.47	0.41	0.60	8.02	82.42	79.13
PIT19	0.70	20,621.0	53,823.0	39,050.0	0.73	33.45	0.41	0.60	8.01	82.42	79.12
PIT20	0.72	20,647.0	53,894.0	39,199.0	0.73	33.43	0.41	0.60	8.00	82.42	79.12
PIT21	0.75	20,706.0	54,053.0	39,566.0	0.73	33.39	0.41	0.60	7.97	82.43	79.12
PIT22	0.78	20,757.0	54,193.0	39,862.0	0.74	33.35	0.41	0.59	7.95	82.43	79.11
PIT23	0.81	20,834.0	54,402.0	40,293.0	0.74	33.30	0.41	0.59	7.92	82.43	79.10
PIT24	0.83	20,880.0	54,529.0	40,568.0	0.74	33.26	0.41	0.59	7.90	82.43	79.09
PIT25	0.86	20,983.0	54,810.0	41,142.0	0.75	33.18	0.41	0.59	7.86	82.43	79.08
PIT26	0.89	21,028.0	54,931.0	41,452.0	0.75	33.14	0.41	0.59	7.85	82.43	79.07
PIT27	0.92	21,032.0	54,941.0	41,480.0	0.75	33.14	0.41	0.59	7.85	82.43	79.07
PIT28	0.94	21,043.0	54,971.0	41,531.0	0.76	33.13	0.41	0.59	7.84	82.43	79.07
PIT29	0.97	21,043.0	54,971.0	41,531.0	0.76	33.13	0.41	0.59	7.84	82.43	79.07
PIT30	1.00	21,145.0	55,250.0	42,226.0	0.76	33.05	0.41	0.59	7.80	82.43	79.06

15.1.2 Pit Design

The ultimate pit configuration is shown in Figure 15-4. 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 IWMF. The pit has also been expanded to the north at 596 masl to provide additional space surrounding the headframe location for the Phase 2 of the underground mine.

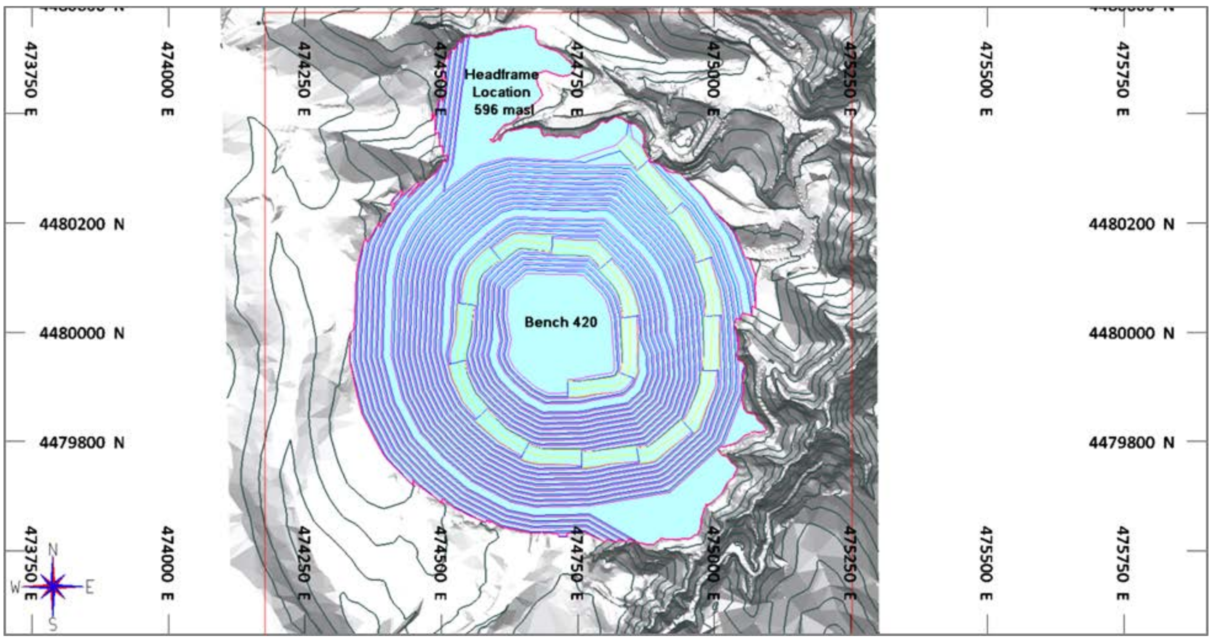


Figure 15-4: Final Pit Design

15.1.3 Open Pit Mineral Reserves

The mineral reserves for the deposit were estimated using a gold price of US$1,200/oz and copper price of US$2.50/lb. The mineral reserves are reported using a US$9.47/t NSR cutoff grade. The proven and probable mineral reserves are 59.5 Mt with an average grade of 0.56 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 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.

Table 15-5: Skouries Open Pit Mineral Reserves, as of December 31, 2017

STARTER PIT

Category	Ore (t x 1,000)	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz x 1,000)	Contained Cu (t x 1,000)
Proven	24,754	0.79	0.48	629	119
Probable	2,515	0.31	0.27	25	7
Proven & Probable	27,269	0.74	0.46	654	126

FINAL PIT

Category	Ore (t x 1,000)	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz x 1,000)	Contained Cu (t x 1.000)
Proven	23,563	0.48	0.38	364	89
Probable	8,636	0.22	0.23	61	20
Proven & Probable	32,199	0.41	0.34	425	10

TOTAL PIT

Category	Ore (t x 1,000)	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz x 1,000)	Contained Cu (t x 1,000)
Proven	48,317	0.64	0.43	993	208
Probable	11,151	0.24	0.24	86	27
Proven & Probable	59,468	0.56	0.40	1,079	235

15.2 UNDERGROUND MINERAL RESERVE ESTIMATE

Work completed by AMC in 2016 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.

The current underground design originated from mineral reserves estimated using a Cu price of US$3.00/lb. The change in the Cu price to US$2.50/lb has no impact to the Mineral Reserves developed at that time given that the margin on the lowest value ore has been demonstrated to remain positive against the backdrop of updated operating costs.

The underground contribution to mineral reserves has been evaluated at an NSR cut-off of US$33.33/t, incorporating 5% unplanned diluting material by weight that is assumed to carry no metal value, and assuming an overall mining recovery of 95%. The underground mineral reserves estimate, as of December 31, 2017, is presented in Table 15-6.

Table 15-6: Skouries Underground Mineral Reserves, as of December 31, 2017

Category	Ore (t x 1,000)	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz x 1,000)	Contained Cu (t x 1,000)
Proven	27,487	1.29	0.67	1,140	185
Probable	70,711	0.68	0.51	1,555	359
Proven & Probable	98,198	0.84	0.55	2,695	544

15.2.1 Cut-off Grade

The cut-off value supporting the estimation of underground mineral reserves is based on projected operating costs at a production rate of 6.0 Mtpa. The projected operating costs indicated that a net smelter return (NSR) cut-off of US$33.33/t would adequately cover all site costs on a breakeven basis while maintaining a cash margin on the lowest grade stopes. The US$33.33/t cut-off value reflects diluted material (mill feed), which corresponded to US$35/t in situ before 5% diluting material is added.

The Skouries underground mineral reserves also include weakly mineralized development material for which mining costs will have been sunk and only the marginal cost of milling (estimated at US$8.35/t) remains to recover saleable metal. This marginal material represents a relatively small fraction (approximately 2%) of the overall underground mineral reserves.

The NSR parameters on which the estimated mineral reserves are based are presented in Table 15-7.

Table 15-7: NSR Parameters used to estimate the Underground Mineral Reserves

Metal pricing		
Copper price	US$/lb	$2.50
Gold price	US$/ounce	$1,200
Recoveries		
Copper recovery	%	See formula*
Gold recovery	%	See formula**
Copper recovery cap	%	95
Gold recovery cap	%	90
Copper concentrate		
Copper concentrate grade	%	26.0
Moisture content	%	9.0
Doré		
Gold to doré	%	17.9
Gold payment doré	%	100
Gold deduction	%	100
Copper concentrate handling		
Mine to port concentrate freight cost	US$/wmt	$59.80
Ocean concentrate freight cost	US$/wmt	
Copper concentrate treatment and refining		
Treatment charges	US$/dmt	$97.35
Gold to concentrate	%	82.1
Gold deduction - adjustment	%	100
Gold payable concentrate	%	96.6
Copper payable	%	96.2
Copper refining cost	US$/payable lb	$0.097
Gold refining cost	US$/payable oz	$5.50
Royalties		
Gold royalty	%	2.0
Copper royalty	%	0.5

* Cu recovery (%) = 99.41 – 56 x oxide (%) – 41 x $e^{(-338 \text{ x Cu Grade (\%)})}$

** Au recovery (%) = 92.62 – 17.5 x oxide (%) – 22 x $e^{(-1.2 \text{ x Au Grade (g/t)})}$

15.2.2 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.

Backfill dilution is inevitable and inherent in mines that employ SLOS, and it has been estimated that 5% to 5.5% (by weight) will enter the mill feed. 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 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 within sills will expose paste backfill in the roof of the excavation if a stope above 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 dilution applied to primary and secondary stopes is reported in Table 15-8. The only dilution applied to primary stopes is on one end wall and this may be either waste or fill depending on the sequence; this dilution is estimated to be one metre in thickness. The secondary stopes have dilution applied to three sides, which is either waste or fill dilution on the end wall, and fill dilution on the sidewalls. The dilution applied to each of the three sides is estimated to be one metre in thickness. The overall dilution by weight was calculated as 5%, as shown in Table 15-8.

Table 15-8: Dilution Factor Assumptions

Description		Primary stopes	Secondary stopes
Stope volume (60 m x 30 m x 15 m)	m^3	27,000	27,000
Backwall dilution volume (1 m)	m^3	900	900
Sidewall dilution volume (right) (1 m)	m^3	0	1800
Sidewall dilution volume (left) (1 m)	m^3	0	1800
Floor dilution (0.2 m)	m^3	0	90
Total dilution tonnage (both)	t	7,686	
Dilution by weight	%	5%	

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.

15.3 MINERAL RESERVE SUMMARY

The combined mineral reserves for the Skouries Project, as of December 31, 2017, are stated in Table 15-9. These represented the weighted sum of the open pit mineral reserves (Table 15-5) and the underground reserves (Table 15-6). As previously mentioned the reporting cut-offs for the mineral reserves are NSR based with US$9.47/t used in the open pit portion and US$33.33/t for the underground estimate.

Table 15-9: Skouries, Combined Mineral Reserves, as of December 31, 2017

Category	Ore (t x 1,000)	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz x 1,000)	Contained Cu (t x 1,000)
Proven	75,804	0.87	0.52	2,132	393
Probable	81,862	0.62	0.47	1,641	386
Proven & Probable	157,666	0.74	0.49	3,773	779

SECTION • 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 10 years. Phase 2 consists of mining from the underground mine for further 13 years. The total life of mine (LOM) is 23 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 mill feed rate from the mining operation is shown in Figure 16-1.

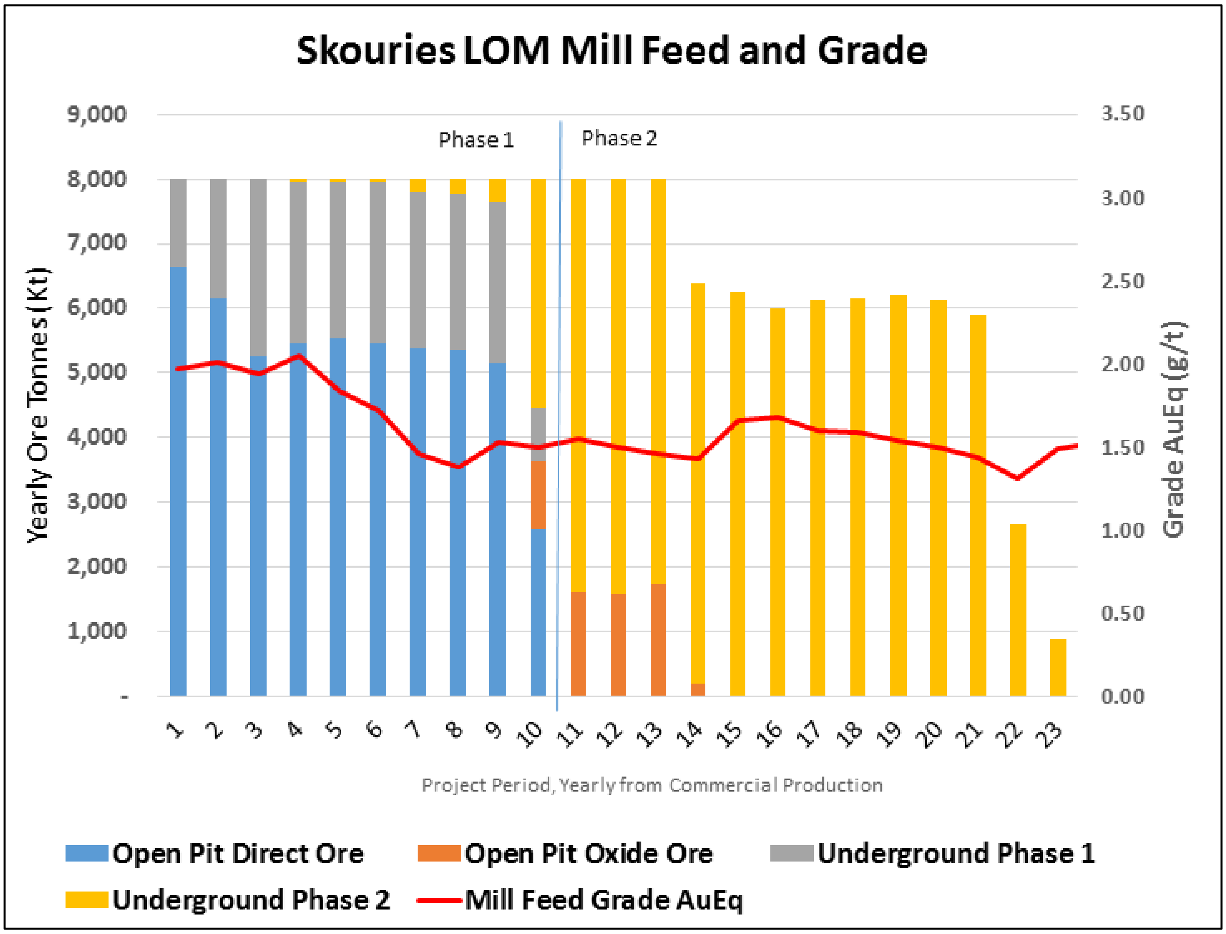


Figure 16-1: Skouries Life of Mine Ore Production Schedule

Phase 1 total mill feed 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 6.4 Mtpa in order to maintain the 8.0

Mtpa mill feed. During Phase 1 approximately 6.5 Mt of oxide ore is stockpiled to be rehandled for mill feed during Phase 2. Phase 1 is complete at the end of the open pit mine life in Year 10.

Phase 2 mine production, from Year 11 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 2.4 Mtpa. During the first three 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 of approximately 1.6 Mtpa. From Year 14 on, Phase 2 mill feed is maintained at a nominal feed rate of 6.2 Mtpa, solely from underground mine production.

16.2 OPEN PIT MINING METHODS

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 0.88:1. 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 oxide ores will be hauled directly to the plant, and an additional portion will be hauled to the oxide ore stockpile (OOS) where it will be re-handled during the Phase 2 of the Project. Waste material will be hauled to a transfer point adjacent to the OOS area where it will be re-handled by a fleet of smaller contractor trucks and placed in one of the material management structures within the IWMF. The structures internal to the IWMF are the J5, OOS pad, KL embankment, upstream (US) waste zone and the capping stockpile pad.

16.2.1 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 wastes which does not contain mineralization. All other waste and ore material types will be drilled and blasted using the specifications outlined in Table 16-1. 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 orezone 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 split the sample into an upper or lower bound, as the orebody is deemed to be continuous over the 10m sampled interval.

Table 16-1: 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
Borehole Diameter	216 mm
Hard Rock Powder Factor	0.25 kg/t
Weak Rock Powder Factor	0.18 kg/t
Explosive Type	70/30 Blended Emulsion/ANFO mix
Delay Type	Nonel

Wall control blasting will be used adjacent to a final and intermediate pit walls to prevent over breakage of the wall and to maintain its overall 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 presplit hole will be drilled at an approximately 70º angle parallel to the pit wall and will be drilled 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.

16.2.2 Open Pit Hauling

The existing site road network will be incorporated into the access development for the pit and material management structures within the IWMF. The access roads will be expanded into mine haul roads and subsequently widened as required by means of end-dumping with suitable waste rock from the pit and cutting of haul road embankment slopes.

The primary haul road widths are based on a 90 t size haul truck with a width of 25 m. Haul truck road widths for 40 t haul trucks range from 12 m to 15 m depending on ditching and berm requirements. Table 16-2 outlines the road widths for each truck class. Road grades are limited to 10% in-pit and ex-pit for the 90t trucks and approximately 12.5% for the contractor trucks. Runaway lanes will be constructed as required for safe operations.

Table 16-2: Haul Road Design Widths

Haul Truck Class/Size	Vehicle Width (m)	One-Way Traffic Road Width (m)	Two-Way Traffic Road Width (m)
15 t-Contractor	2.5	5.0	10
40 t-Contractor	3.4	7.0	14
60 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. This study included the designs of the haul road network from the open pit to the IWMF for every year of the operation. An example of this road network in Year 3 is shown in Figure 16-2.

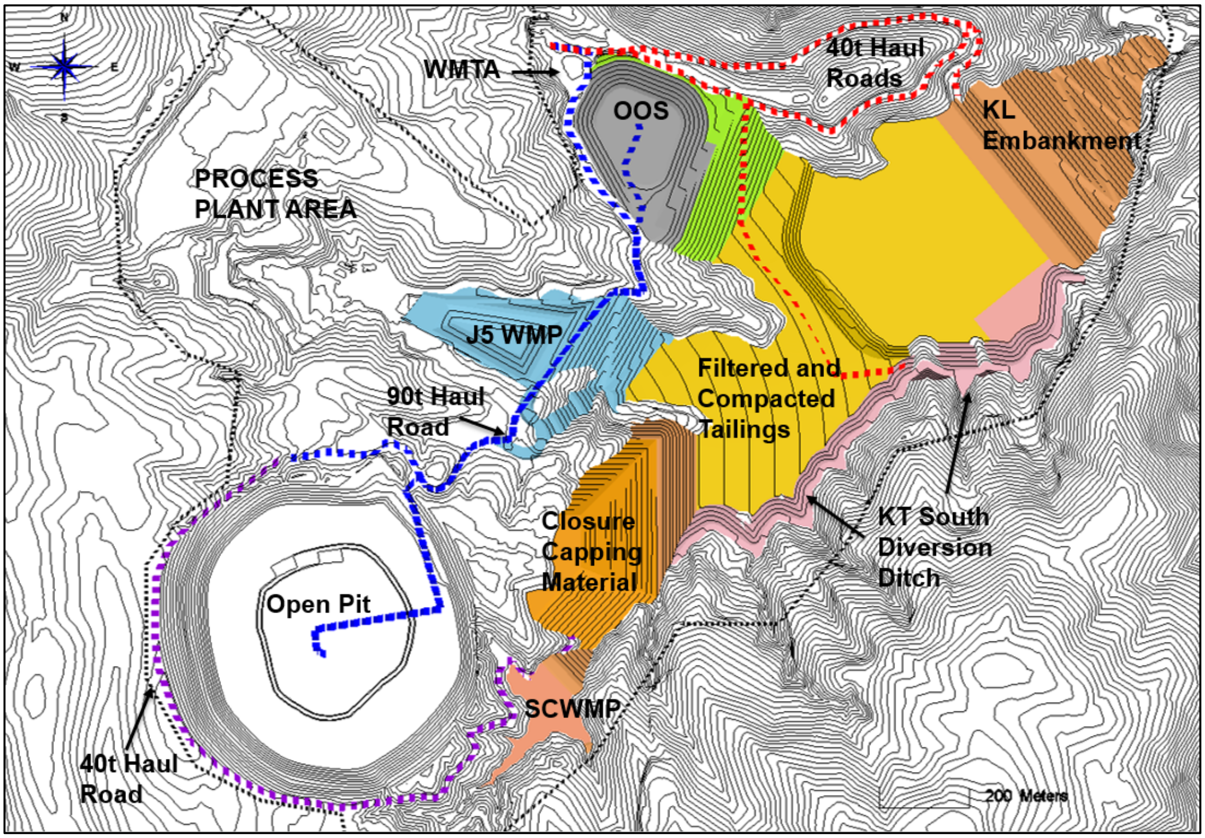


Figure 16-2: Haulage Routes in Year 3 of Open Pit

The primary mining equipment fleet will be supplemented by a contractor fleet that will haul waste material directly from the pit in the pre-production periods and from the waste material transfer area (WMTA) to the various material management structures in Years 1 to 10.

The number of haulage units was determined by calculating cycle times in Talpac© using annual haul cycle profiles from MineSight©. Haul simulations were carried out based on the designated 90 t haul trucks and the smaller 60 t, 40 t and 15 t contractor trucks. Rolling resistance was estimated to be 5% within 100 m of the loading face and 100 m of the dump, and 3% on all established roads. 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 by breaking data up to a maximum of 25 km/hr.

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. Table 16-3 lists the factors used, and subsequent truck tonnages used for determining the number of units required. The current study did not include carry back of clay materials as these are only a small (<10%) portion of the total material moved.

Table 16-3: Material Fill Factors and Adjusted Haul Truck Capacities

	Hard Rock (C)	Overburden (B1)	Weak Rock(B2)	Red Clay (A)
Material Fill Factor	1	0.7	0.85	0.65
15 t-Contractor	15 t	10.5 t	12.75 t	9.75 t
40 t-Contractor	40 t	28 t	34 t	26 t
60 t-Contractor	60 t	42 t	51 t	39 t
90 t-Primary Fleet	90 t	63 t	76.5 t	58.5 t

16.2.3 Open Pit Loading

The primary mining loading fleet will consist of conventional diesel hydraulic excavators and front end loaders with an 11-12 m^3 bucket. 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 IWMF 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 only 90 t haul trucks will be loaded in the open pit. This will simplify the operations, and keep the smaller off-road haul trucks away from the larger 90 t mining trucks.

16.3 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. The overall equipment hours per year varies from 5,000 to 5,300 hrs/yr depending on the type of equipment. Due to the approximately 10 year mine life and annual operating hours, it is estimated that no major equipment will need to be replaced due to exceeding its useful life in operating hours. The maximum number of units required in any given year of the Project are shown in Table 16-4. The fleet size generally will drop over time as the proportion of waste materials moved in the later years of the Project reduces due to the low stripping ratio of the open pit.

Table 16-4: Open Pit Owners Primary Mining Fleet

Equipment Type	Class/Size	Max number of units
Haul Trucks	90 t	11
Excavators	11 m^3	1
Front End Loaders	12 m^3	2
Blast Hole Drills	Rotary and ITH 114mm to 216mm	2

Support equipment requirements were calculated using an internal database and experience in similar conditions. Table 16-5 summarizes total support equipment requirements, which are generally fixed throughout the life of the Project. This equipment supports the primary mining fleet only from the open pit to the waste materials transfer area and the oxide ore stockpile. Additional support equipment will be used by the contractor for construction of the IWMF.

Table 16-5: Open Pit Owners Support Mining Fleet

Equipment Type	Class/Size	Total Quantity
Dozer	CAT D8	1
Wheel Dozer	CAT 834K	1
Grader	CAT 12M & 16M	2
Small Excavator	CAT 345	1
Water Truck	8000 gal & 2500 gal	2

16.4 OPEN PIT PRODUCTION SCHEDULE

The open pit mine production schedule has been developed on a planned average annual ore production rate of 5.5 Mtpa. The actual yearly rate varies according to the 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, resulting in a daily average ore mining rate of 15 ktpd. A total of 59.5 Mt of ore with an average head grade of 0.56 g/t Au and 0.40% Cu will be mined over a 10 year mine life with 53.9 Mt being direct feed ore, while a further 5.4 Mt are designated as stockpiled oxide ore. The average waste to ore stripping ratio of 0.88:1 equates to approximately 52.9 Mt of waste being produced.

16.4.1 Open Pit Production Schedule Year -2

Initial pit development starts using the 40 t contractor fleet mining waste in order to begin the construction of access roads, J5, and the OOS. Waste material mined directly from the pit will be hauled to the WMTA and then re-handled and loaded into 15 t contractor construction trucks. The 15 t trucks will then haul the material to the bottom of J5 and place it in lifts according to the geotechnical requirements. The initial material for the OOS base pad comes from a cut within the OOS footprint. This cut is mined using 15 t contractor trucks and the material gets hauled to the bottom of the OOS valley and placed in lifts. Some of the smaller mining equipment is commissioned along with road and pit support equipment.

Material is required to be re-handled via the WMTA due to the restrictions and viability of upgrading the existing haul roads to handle larger haul truck traffic. It was determined that this re-handling would limit haul road construction costs and more importantly minimize interactions between the smaller contractor fleet and the larger primary mining fleet. Direct placement has been considered wherever possible.

16.4.2 Open Pit Production Schedule - Year -1

The initial cut of the open pit continues to advance as material is primarily mined using the 40 t contractor fleet. Two of the 90 t primary fleet haul trucks are commissioned in the latter half of the year along with a primary loader and some additional support equipment.

Waste material placement at the end of Year -1 is shown on Figure 16-3. The build-up of J5 is completed by the end of the period using the 40 t fleet hauling directly out of the pit. The OOS is at the 440 m elevation, and material comes directly from the pit and the cut within the OOS footprint. All the material for the OOS is directly placed using the 40 t contractor fleet. Some initial oxide ore from the pit begins to be hauled and placed directly on to the OOS using the primary fleet of 90 t haul trucks. Construction of the KL embankment begins using the 15 t contractor trucks hauling material from the WMTA to the initial lifts of the embankment up to the 345 m elevation.

A section of the haul road for all material being hauled from the WMTA to the J5, the OOS, and the KL embankment is designed as a one way haul route. This was done to increase safety and to utilize existing roads, which minimized new disturbance. This section of haul road will be utilized for most of the mine life as it provides access to key waste material destinations.

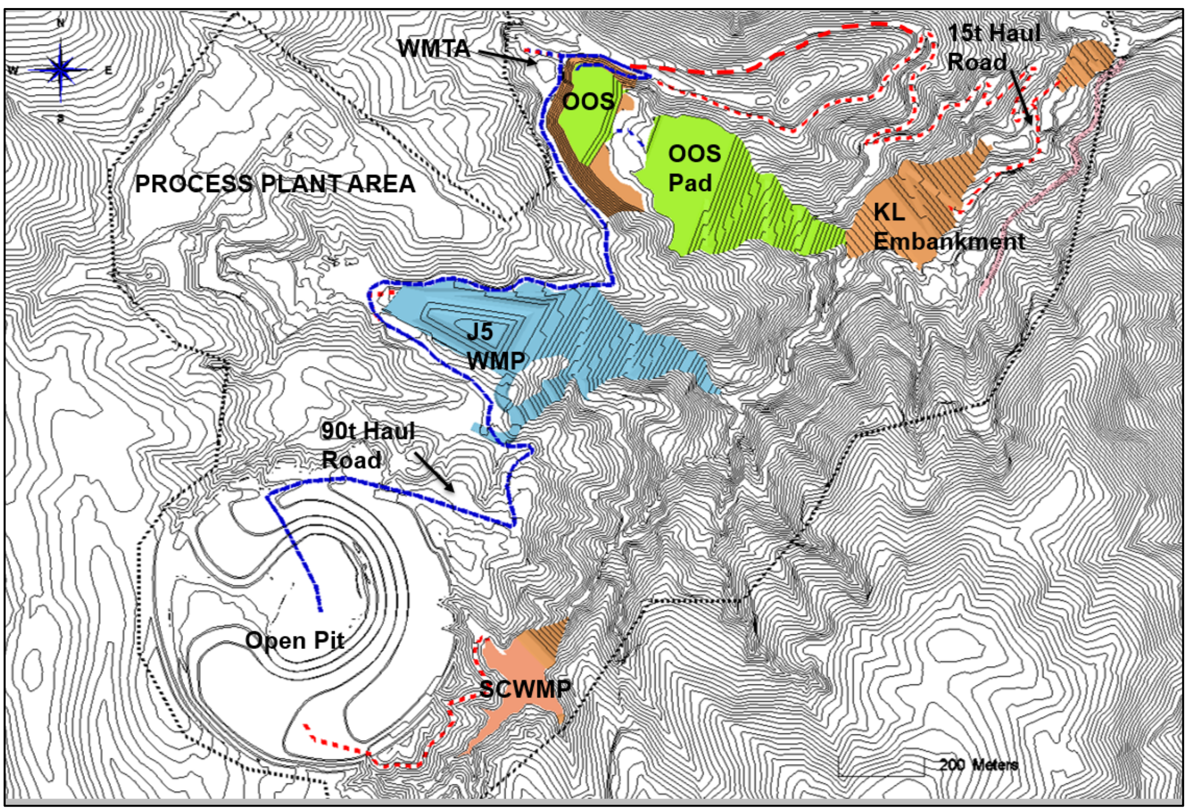


Figure 16-3: Skouries Project End of Year -1

16.4.3 Open Pit Production Schedule - Year 1

In Year 1 mining transitions into the production phase using the owners primary mining fleet as most of the mining equipment is commissioned, in addition to the Skouries processing plant. Waste is hauled using 90 t haul trucks from the open pit to the WMTA. Oxide ore is hauled directly to the OOS by the 90 t haul trucks. The OOS is completed to a height of 488 m.

The KL embankment is built up to an elevation of 360 m. Most of the material is hauled from the WMTA to the KL embankment and US waste zone using the 40 t contractor fleet, with some of the smaller initial lifts being placed using the 15 t contractor fleet. Filtered tailings material begins to be stacked in the valley upstream from the KL embankment using a series of portable conveyors and dozers.

16.4.4 Open Pit Mining Schedule - Years 2 to 3

In Years 2 and 3 there is an increase in the waste material mined in order to balance the material required for the tailings embankment construction. It is assumed that 60 t trucks will be used by the contractor to haul material from the WMTA to the tailings embankment to reduce the congestion and number of units required to move the material in those years. No 15 t contractor trucks will be used for the remainder of the mine life as most of the material management structures are now accessible to larger haul trucks. The 90 t haul trucks continue to be the primary fleet and haul ore directly to the mill while the oxide ore continues to be hauled directly from the pit to the OOS which reaches an elevation of 520 m.

In Year 2 a capping stockpile pad is built upstream of the dry tailings placement. A material transfer point will be established just outside of the pit crest in order to transfer material from the 90 t fleet to the 60 t contractor fleet. A transfer point at the pit edge was chosen to reduce road disturbance west of the open pit and to keep contractor trucks out of the open pit area. The waste material will be routed along the western edge of the pit boundary and down to the capping stockpile pad. In Year 3 this same haul route will be used to haul capping material to the stockpile pad. Waste material placement at the end of Year 3 is shown on Figure 16-2.

All waste from the pit is hauled either to the capping stockpile transfer area or the WMTA using the 90 t fleet.

The KT1 diversion ditch is built along the southern edge of the KL Valley and waste material from the WMTA is hauled using the existing KL embankment haul road. Waste will be placed in accordance to the geotechnical specifications.

The tailings embankment is built up to an elevation of 415 m using the 60 t contractor fleet and filtered tailings is stacked upstream at a crest elevation of approximately 433 m.

16.4.5 Open Pit Mining Schedule - Years 4 to 5

The amount of waste material mined begins to decrease and the contractor fleet returns to 40 t trucks for the remainder of the mine life. The 90 t haul trucks continue to haul ore material directly to the mill and the last of the oxide ore is placed in the OOS in Year 4. Waste material continues to be hauled from the pit to the WMTA.

The open pit depth is at approximate 500 m elevation while the pit ramp exit is at approximately 585 m elevation and so the distance and grade of the haul out of the pit begins to play a more significant role operationally.

The tailings embankment is built up to an elevation of 420 m using the 40 t contractor fleet hauling from the WMTA. There is also a diversion ditch extension that is built, and waste material is hauled to it along a route down the face of the OSS fill and across the placed filtered tailings. Filtered tailings is stacked to a crest elevation of approximately 448 m.

16.4.6 Open Pit Mining Schedule - Years 6 to 10

Waste material mined continues to decrease and 90 t haul trucks haul all ore material directly to the mill as the oxide ore stockpile has now reached capacity. Waste material continues to be hauled from the pit to the WMTA and from there it is hauled on to the KL embankment using the 40 t contractor fleet. The tailings embankment is competed in Year 6 at a height of 465 m.

The KT1 diversion ditch is extended in Year 7, and material is hauled from the WMTA down across the face of the OOS and across the placed filtered tailings.

The capping stockpile material is re-handled in Years 8, 9, and 10 and placed on top of the final tailings surface. The thickness of the capping is 2.5 m of inert materials as a part of the closure of the IWMF. Waste material placement at the end of Year 10 is shown on Figure 16-4.

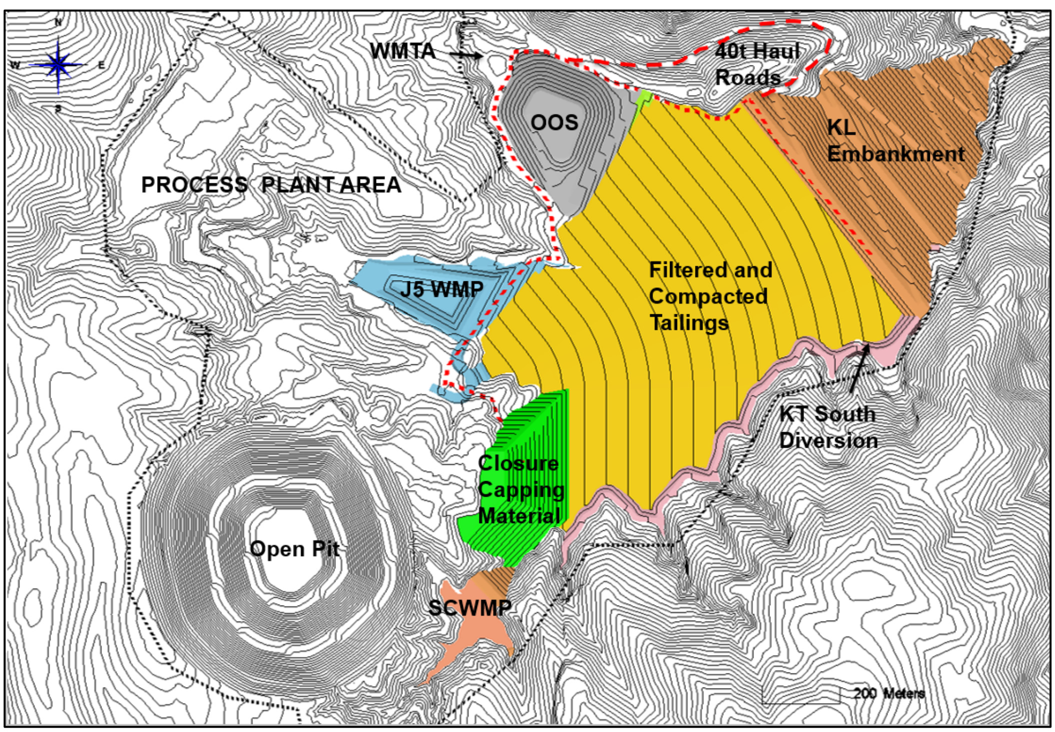


Figure 16-4: Skouries Project End of Year 10

16.5 OPEN PIT MINE INFRASTRUCTURE

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

16.5.1 Ancillary Facilities

The location of the facilities are shown on Figure 16-5. Surface ancillary facilities are well situated to be in close proximity 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.

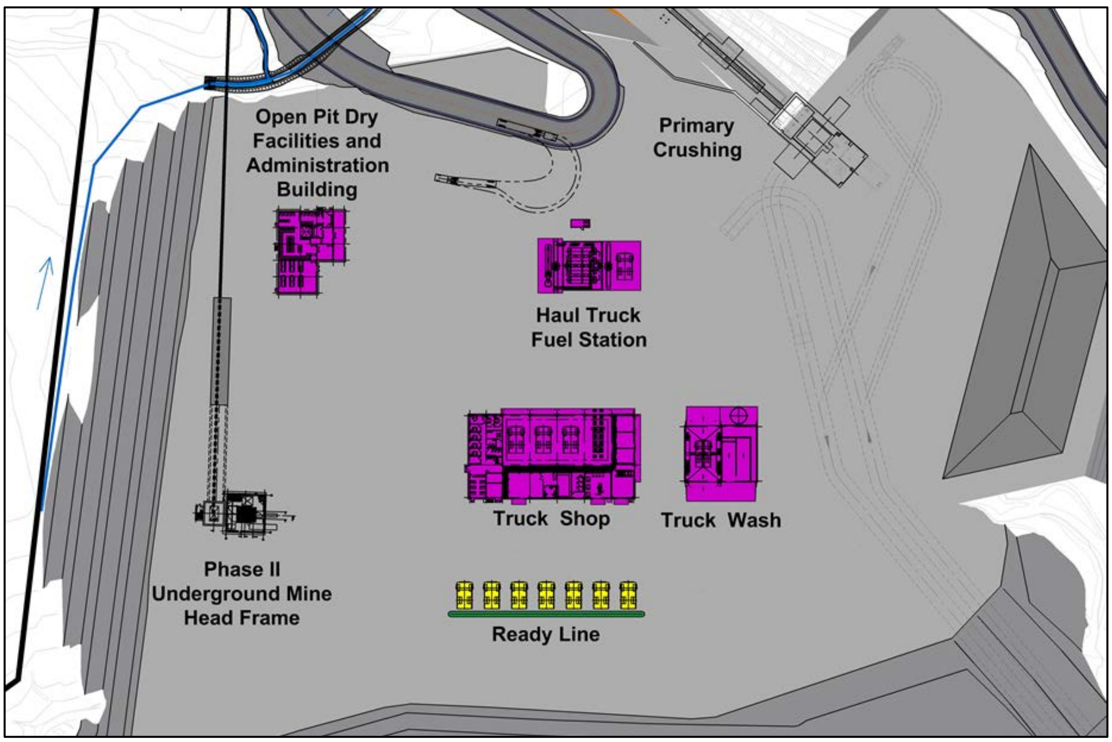


Figure 16-5: Open Pit Mine Infrastructure

16.6 UNDERGROUND MINING

The Skouries orebody that extends below the bottom of the open pit is amenable to a 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 sub-level open stoping. Sub-level open stoping (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, and the ability to backfill the depleted open pit. These characteristics have been incorporated into the EIS approval in 2011.

16.7 UNDERGROUND GEOTECHNICAL

16.7.1 In Situ Stress

It is recommended that planning proceed on the basis of the assumed stress field from the World Stress Map, but that measurements be taken once development has reached more competent rock in the general area in which test stoping has been proposed. Input parameters developed previously to assess the scenarios of major principal stress vertical and major principal stress horizontal is given below. These parameters were used for stability assessment. Table 16-6 provides input parameters used to assess major vertical and horizontal principal stress.

Table 16-6: Stress Input Parameters - Major Principal Stress Vertical (z is the depth in metres below surface level)

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 $\sigma 3$
$\sigma_3 = \sigma_h$	185°/00°	0.0087 z + 0.033	Uniaxial strain model

16.7.2 Stope Design Method

The majority of the stoping is considered to take place in reasonable quality rock mass. The geotechnical analysis has indicated that, for stoping in the porphry, a 60 m sublevel interval (65 m stope height) can largely be viable without significantly compromising stope wall stability if the length of the stope does not exceed 30 m. For primary stoping in schists, preliminary stope lengths have been set at 20 m. Stope widths have been limited to 15 m in consideration of stability of the stope back. Thus, the standard stope dimensions were set to 65 m high x 30 m long x 15 m wide in porphyry, and 65 m high x 20 m long x 15 m wide for primary stope design in schist material.

Table 16-7 provides a summary of rock mass rating (RMR) values used in previous studies together with the A, B, and C parameters determined during previous work.

The parameters shown are for the evaluation of the stope sidewall height, which determines the sublevel interval. The hydraulic radius of 10.3 is for a stope 65 m high (floor to floor) and with a distance along strike of 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. The potential impact of exposing schist sidewalls is an increase in dilution.

Table 16-7: Parameters used for the Stability Graph Method[1]

Stope walls	Hydraulic radius	RMR_{90}[2]	Q'	A	B	C	N'
SRK, 2015- representative porphyry upper bound	10.3	75	31.3	1.0	0.4	8.0	100.3
SRK, 2015- representative porphyry lower bound	10.3	65	10.3	1.0	0.4	8.0	33.0
SRK, 2015- excavation design - porphyry upper bound	10.3	70	18.0	1.0	0.4	8.0	57.6
SRK, 2015- excavation design - porphyry lower bound	10.3	55	3.4	1.0	0.4	8.0	10.9
AMC, 2016- excavation design - porphyry upper bound	10.3	75	31.3	1.0	0.4	8.0	100.3
AMC, 2016- excavation design - porphyry lower bound	10.3	60	5.9	1.0	0.4	8.0	18.9

1. AMC 2016 values were retained for this study.
2. Q' vales have been calculated using the relationship RMR = 9* Ln(Q') + 44.

Assessments were conducted using the NGI-Q stability graph as well as the stability graph method to assess appropriate stoping spans. This work concluded that porphyry stoping widths, without cable bolt support, of 15 m were achievable in ground of Good and Intermediate rock mass quality. Within poor rock mass ground conditions and without cable support, the stoping width would have to be reduced to the order of 7.5 m.

Additional work was conducted to assess both the stope spans and inter-level distances using these same techniques and assessing if cable bolt support was viable. The results of this work suggest that, for Skouries, there may be areas 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 drawpoints, and in secondary stoping where blast damage and mining deformations are likely to impact ground conditions.

16.7.3 Vertical Development Design

Ground conditions for vertical development using raise boring and Alimak were considered during the rock mass assessment. 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 45 m and limited to 1 m in diameter. Planning and costing proceeded on the basis of pre supporting to 40 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 shotcrete over the entire length of the raise. This will be the 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 constructed using an Alimak climber machine. Boreholes will be used as a conduit for services from level to level. Vertical development sizes used in the mine design are summarized in Table 16-8.

Table 16-8: Vertical Development Dimensions

Description	Size (m)
Return air raises to surface (raise bored)	3.5 diameter
Fresh air raises to surface (raise bored)	3.5 diameter
Internal return air raises (Alimak)	3.0 x 3.0
Internal fresh air raises (Alimak)	3.0 x 3.0
Ore bins (coarse and fine)	6.0 diameter
Shaft	8.2 m (7.6 m finished) diameter

16.7.4 Test Stoping

Two test stopes have been planned between the 350 Level and the 290 Level to provide proof of concept for the 60 m high x 15 m wide x 30 m long basic stoping units in both porphyry and potassic 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 test work and to corroborate key stoping parameters.

16.7.5 Development Ground Support Design

The primary rock failure mechanism anticipated at Skouries is progressive unravelling of the blocky rock mass up to a naturally stable shape (arch). The failure mechanism is likely to be predominantly structurally controlled rather than stress driven. Localized caving-type failure may also be anticipated in areas of sheared lithological contacts as well as in areas where the rock mass is affected by major faults.

The recommended rockbolt spacings and lengths for various drive dimensions and rock types are presented in Table 16-9.

Table 16-9: 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

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.

16.7.6 Development through Paste Fill

The proposed SLOS mining method involves development in pastefill to recover ore located in sills. 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 thick, sprayed floor-to-floor. In addition since the sill levels are known in advance (230, 110 , 10 Levels) a 10m thick zone of paste fill with increased cement content will be planned for the stopes which are located in areas where re-development are required.

16.7.7 Crown Pillar Stability Assessment

The three-dimensional non-linear finite difference numerical code FLAC3D was used to assess the crown pillar stability. This method incorporates inelastic material behavior (yielding and dilating rock mass).

The numerical analysis indicates that the intended 70 m thick crown pillar will remain stable after all underground stopes are extracted and the pit is filled with dewatered processing plant tailings. A factor of safety 1.3 was applied to the analysis. The modelling was completed under the assumption that the in situ major principal stress component is vertical (the base case in this study) an assumption which needs to be confirmed with site-specific stress measurements from underground.

16.8 SUBLEVEL DESIGN

All levels in both phases have similar designs (see Figure 16-6). Peripheral development (Ring-drives) will provide access to all sides of the orebody, and terminate at return air raise locations. Ore drives for stope extraction will traverse the orebody east to west on 15 m centers, 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 center out.

Level development priority is to complete the inter-level fresh air raises, 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 in this same location.

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.

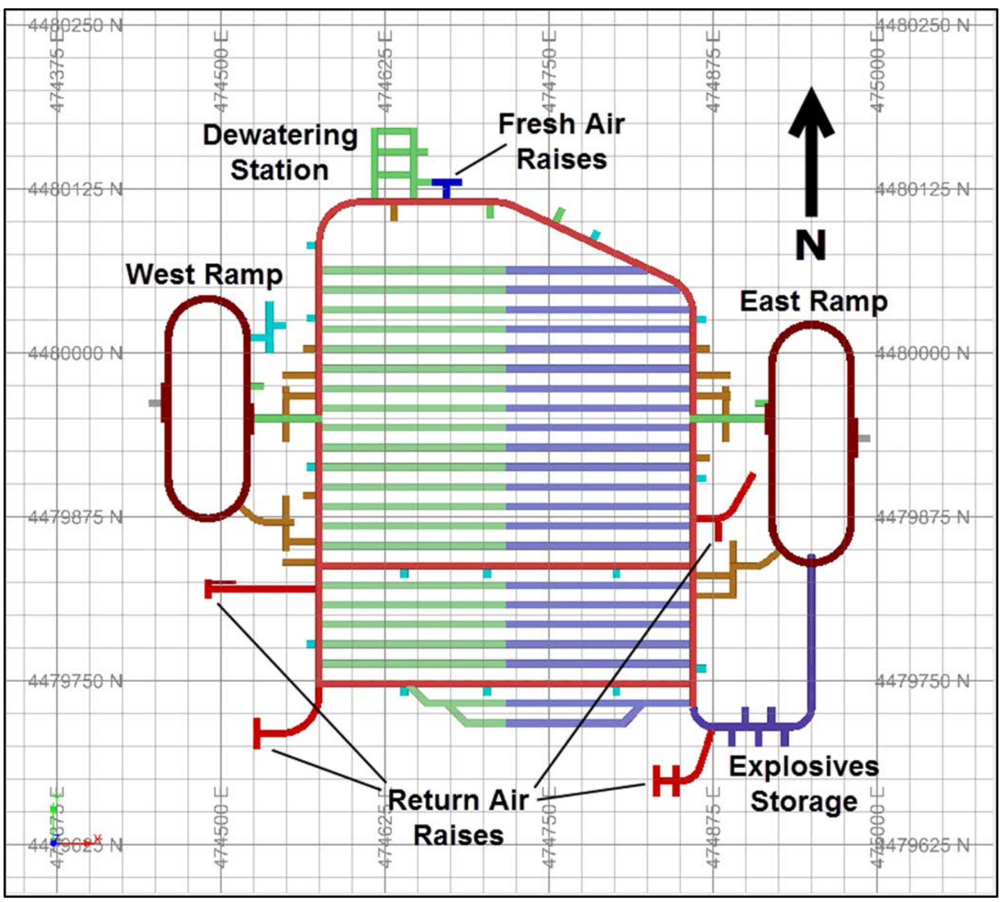


Figure 16-6: Typical Sublevel Arrangement (230 Level)

16.8.1 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-meter minimum clearance on each side. The ore drives (6 m wide) are designed to provide adequate room for efficient production drilling. Table 16-10 shows the standard lateral development dimensions.

Table 16-10: 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)	7.0	7.0
Large workshop development (Phase 2)	12.0	10.0

16.9 UNDERGROUND PRODUCTION STOPE 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.

16.9.1 Drill and Blast

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 drop raise blast is designed at 3.2 m x 2.4 m and uses eight down-holes that are 114.3 mm in diameter. Once the initial void has been created, adjacent ring blast holes are slashed towards the slot raise to create further void. Once the slot raise is fully developed, the main production rings will have a complete free face to blast towards.

Production drill rings will consist of down hole fans of 60 m. The total stope, including slot raise, will generally be opened in six blasts. Figure 16-7 provides a plan view of the standard stope design and the 1.2 m diameter slot raise.

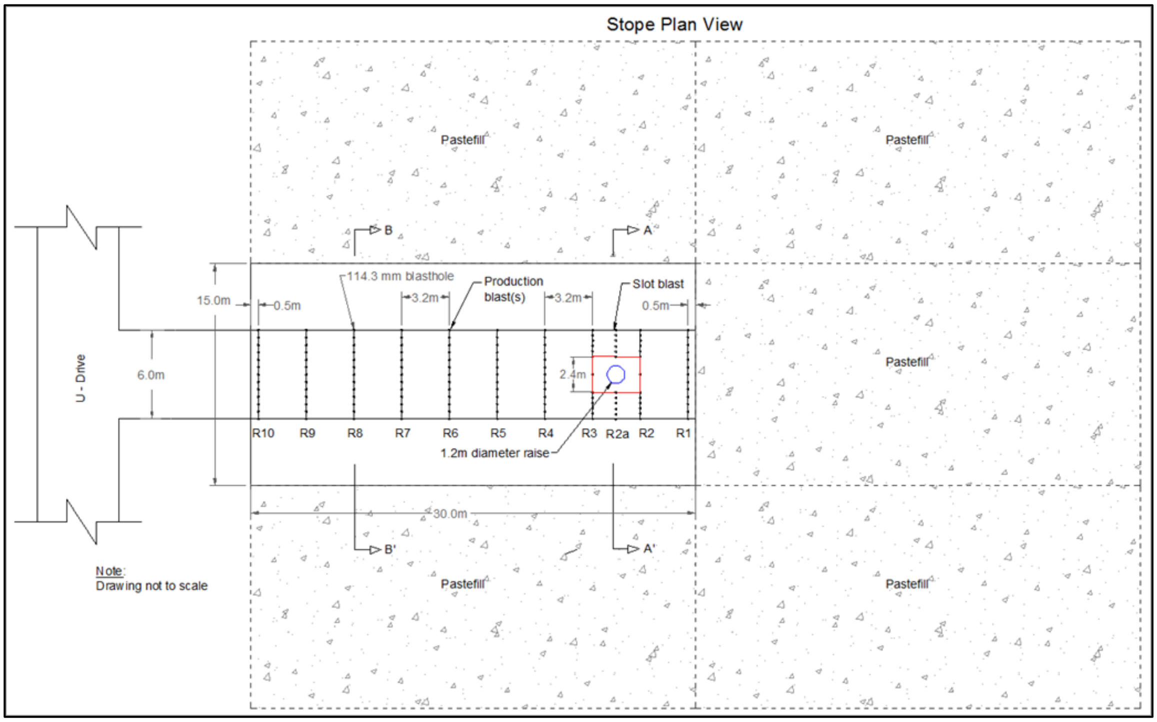


Figure 16-7: Standard Stope Design - Plan View

For opening up the slot area, the 60 m stope is divided into three lifts of approximately 20 m, and holes will require stemming to contain the top of the explosive column. Holes will also require plugs at the bottom where they break through to the void below.

Figure 16-8 provides a stope long section view of the general stope blasting sequence.

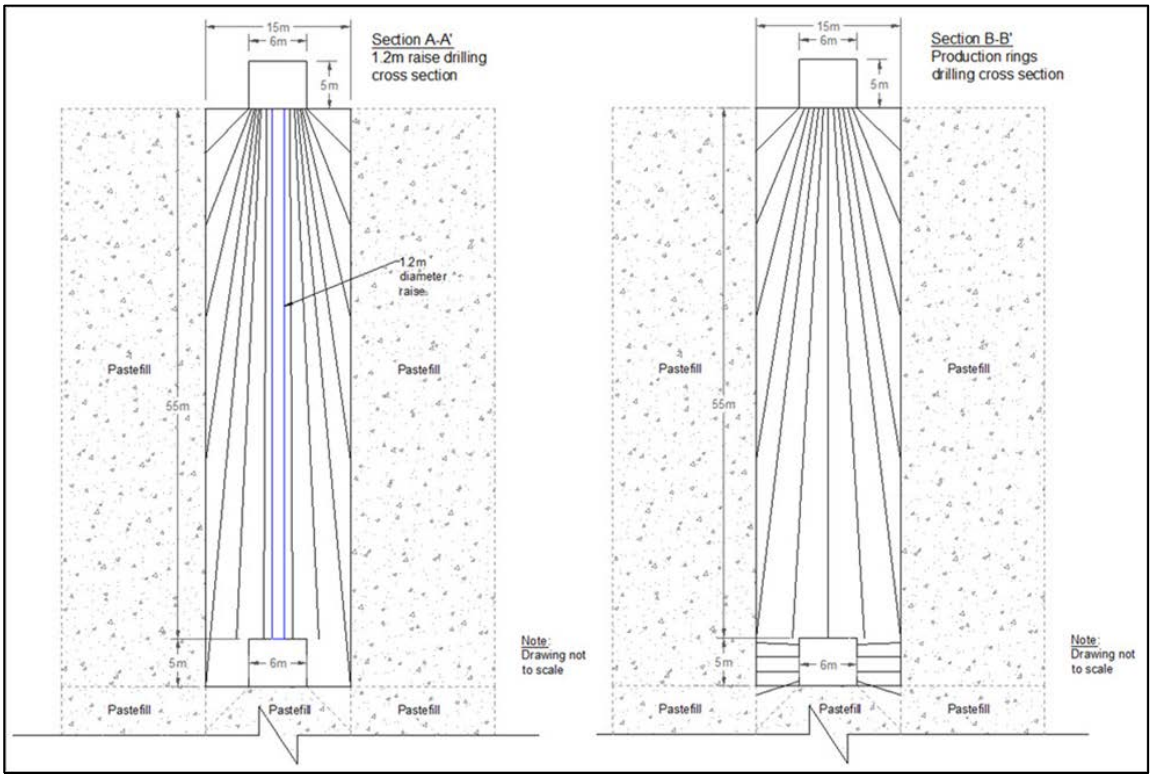


Figure 16-8: Standard Stope Design Drill Section (secondary stope shown)

16.10 UNDERGROUND MINING SCHEDULE

16.10.1 Stope Cycle Time and Mine Production Rate

Overall stope cycle times 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 each stope will produce approximately 1,030 tpd during the mining cycle (drilling, blasting, mucking, but not backfilling). A separate cycle time was determined for the 20 m long stopes projected to be required in schist material, resulting in an estimate of 935 tpd as shown on Table 16-11.

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 6.4 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 requirements is due to the fact that drilling of the next stope in the cycle can start whilst the former stope is being backfilled and cured. The higher numbers reflects no overlap in those activities whilst the lower number reflects a full overlap.

Table 16-11: 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	%	5.1%	5.1%
Undiluted tonnes	t	72,630	48,420
Ore development tonnes	t	2,421	2,421
In situ longhole tonnes	t	70,209	45,999
Stope tonnes recovered, including dilution	t	70,100	45,928
Slot raise drilling	m	1,653	1,653
Ring drilling	m	3,037	1,735
Number of blasts	each	6	6
Average load and haul rate	t/day	2,000	2,000
Volume to backfill	m3	26,960	17,973
Fill factor	%	97%	97%
Total fill required	m3	26,151	17,434
Plug volume - 10 m height	m3	4,500	3,000
Plug pouring rate	m3/day	4,500	3,000
Remainder pouring rate	m3/day	4,598	5,774
Curing time	days	14	14
Operational contingency for lost time	%	25%	25%
Cycle times with contingency			
Longhole drilling	days	19.3	15.4
Stope blasting	days	5.0	5.0
Stope production mucking	days	43.8	28.7
Ancillary Activities	days	4.4	4.4
Backfilling	days	29.6	26.9
Total cycle time	**days**	**102.1**	**80.3**
Total production rate - excluding fill	t/day	1,030	935
Total production rate - with fill	**t/day**	**687**	**572**

This study includes the implementation of Remote Mining Technology (RMT), which has an impact on the cycle times of stopes and the productivity of equipment. This technology at its core 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 times for employees to reach underground.

RMT implementation at the Skouries mine is anticipated in Year 1 for development drill jumbos, Year 2 for load haul dumps (LHD) and production drills, and Year 9 for production trucking. To assess productivity improvements to be taken into consideration, a revised cycle time for stopes was prepared and is shown in Table 16 12. This cycle takes into consideration all stope activities with longer durations of operation assumed on a daily basis for the LHDs, production drill rigs and, later, the trucks. The assumed productivity improvements are 18% for development drilling, 10% for 20 m stopes and 15% for 30 m stopes.

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.

Table 16-12: 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	%	5.1%	5.1%
Undiluted tonnes	t	72,630	48,420
Ore development tonnes	t	2,421	2,421
In situ longhole tonnes	t	70,209	45,999
Stope tonnes recovered, including dilution	t	70,100	45,928
Slot raise drilling	m	1,653	1,653
Ring drilling	m	3,037	1,735
Number of blasts	each	6	6
Average load and haul rate	t/day	2,533	2,533
Volume to backfill	m3	26,960	17,973
Fill factor	%	97%	97%
Total fill required	m3	26,151	17,434
Plug volume - 10 m height	m3	4,500	3,000
Plug pouring rate	m3/day	4,500	3,000
Remainder pouring rate	m3/day	5,774	5,774
Curing time	days	14	14
Operational contingency for lost time	%	25%	25%
Cycle times with contingency			
Longhole drilling	days	16.6	13.9
Stope blasting	days	5.0	5.0
Stope production mucking	days	34.6	22.7
Ancillary Activities	days	4.4	4.4
Backfilling	days	28.4	26.9
Total cycle time	**days**	**89.0**	**72.8**
Total production rate - excluding fill	t/day	1,247	1,105
Total production rate - with fill	**t/day**	**788**	**631**

16.11 UNDERGROUND MATERIALS HANDLING

16.11.1 Phase 1 Materials Handling

The material handling strategy for Phase 1 is based on truck haulage of ROM ore directly to surface from the loading bays via the dual ramp system as shown in Figure 16-9. 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 simultaneously with other equipment that also use the ramps. This volume of material must also account for development activities generated from the Phase 2, which must begin in Year 4.

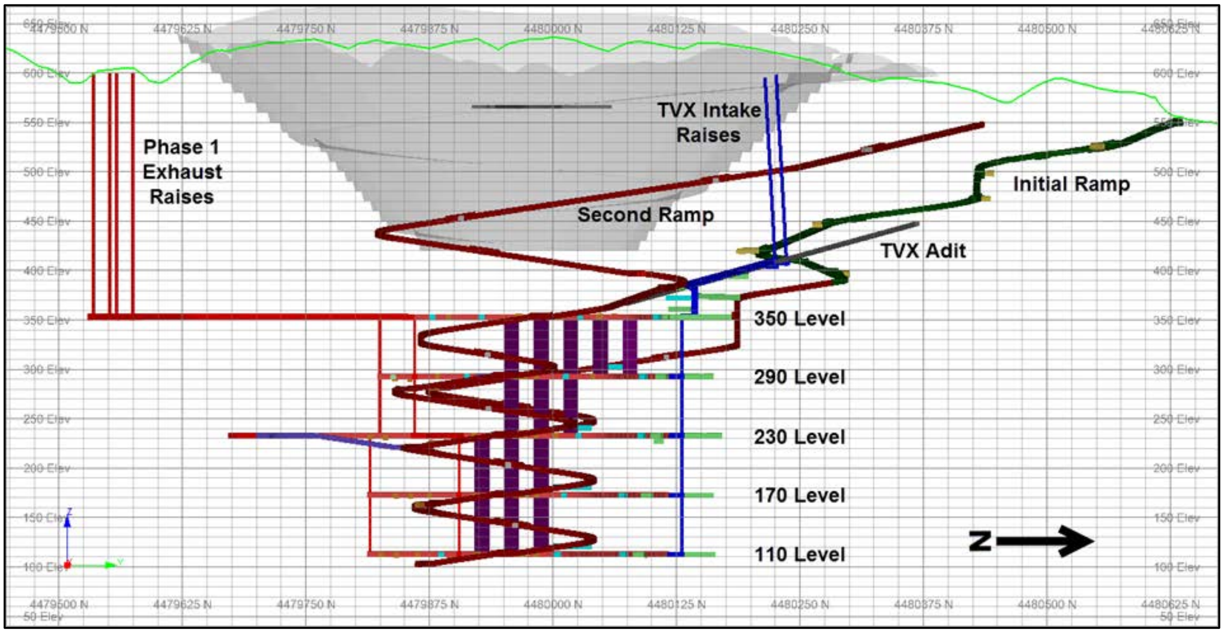


Figure 16-9: View looking West of General Access and Development Infrastructure for Phase 1

16.11.2 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 6.4 Mtpa from the Phase 1 production of 2.5 Mtpa. During Phase 2, all stope ore and possibly some development ore will be hoisted to surface via the shaft. Development waste will continue to be trucked to surface via the dual ramp system, but these quantities are expected to be minimal. The use of ore and waste passes within the underground mine is not planned and no provision is made to hoist waste rock to surface.

In order to hoist the ore to the surface ROM ore will have to be crushed underground. The materials handling will 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 16-10.

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 50 Level or directly to one of two crusher dump pocket on the -10 Level. The coarse ore bin dump pockets are each equipped with a grizzly and rock breaker, the crusher tipping points are equipped with a stationary rock breaker.

The gyratory crusher is fed from the central crusher apron feeder from the coarse ore bins or direct tipping by haul trucks. Prior to the commencement of Phase 2 production, a shaft will be sunk from surface to -100 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 minus 40 elevation and the shaft will be fitted with counterbalanced 31 t skips on cable rope guides and powered by a friction hoist.

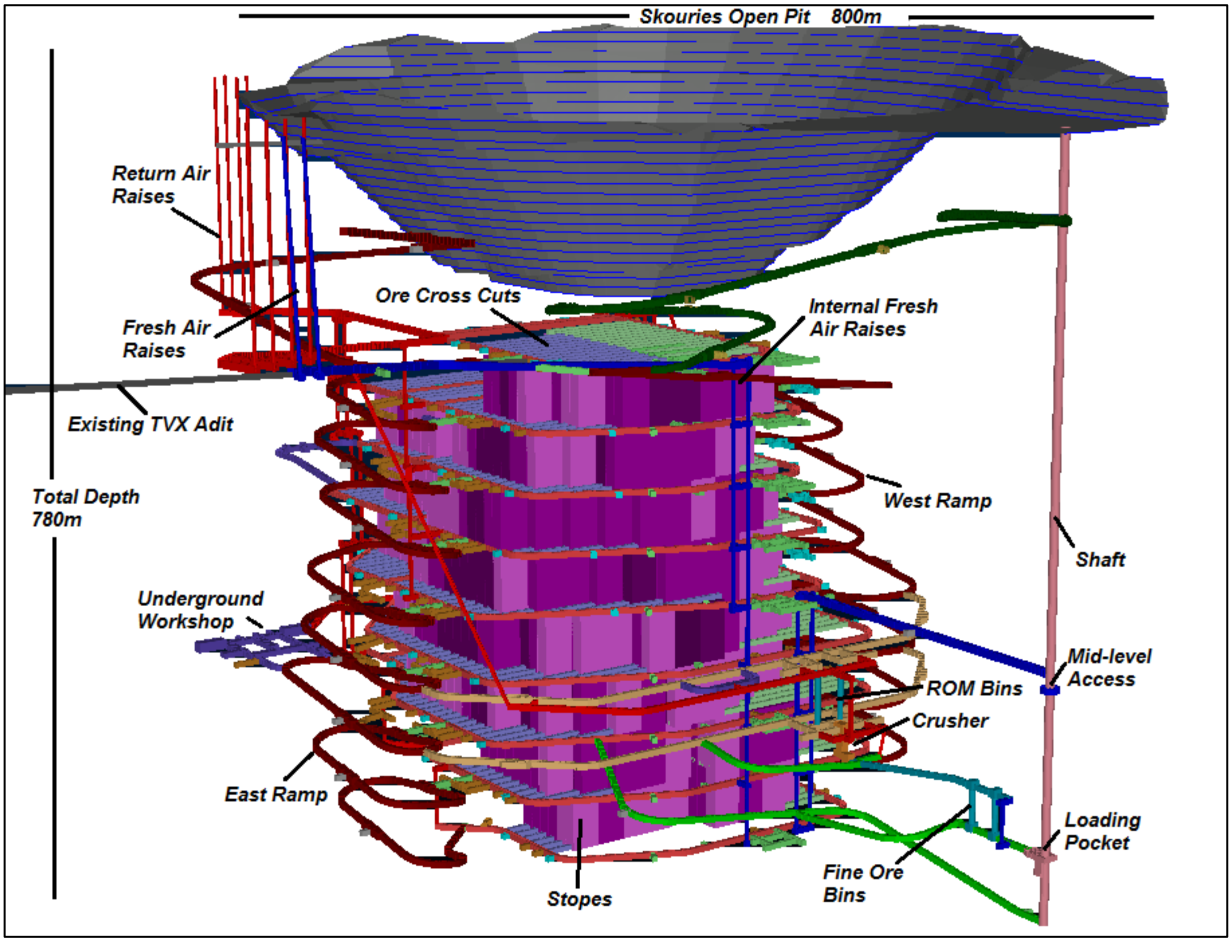


Figure 16-10: View looking South of General Access and Development Infrastructure for Phase 2

All stope ore extracted on -10 Level and below will be hauled directly to the crusher and all stope ore extracted on 50 Level and above will be hauled to the coarse ore storage bins on 50 Level from where it will subsequently be fed to the crusher.

The production hoist will be a multi-rope friction hoist and has been rated to have 25% excess hoisting capacity in line with all other fixed underground mine plant. The hoist will be operated in fully automated mode apart from maintenance or emergencies. The head frame has been located on a 30 m cut of overburden reducing the overburden depth to a manageable 30 to 35 m. At this depth it is feasible to install secant piles for 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 binhouse. Ore will transfer from the binhouse onto the surface stockpile feed conveyor which feeds the existing fine ore stockpile.

16.11.3 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 16-11.

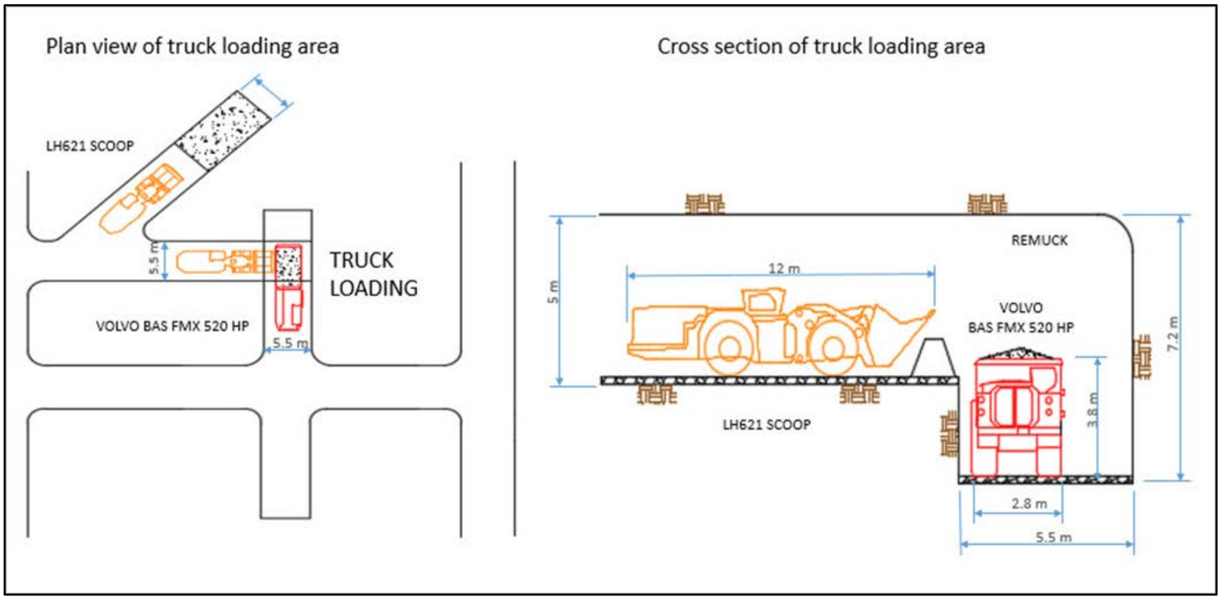


Figure 16-11: Representative plan and section of level re-muck and truck loading

16.11.4 Underground Access

Primary access to the underground workings will be through a dual ramp system from the surface. The initial west ramp is currently being developed and, at the time of this report, had advanced approximately 1,300 m at a -12.6% gradient to an elevation of 384.7 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 1990 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 fresh air raises 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 and remuck bays 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.

16.12 UNDERGROUND MINE INFRASTRUCTURE

Mine infrastructure including ancillary facilities and services have been fully developed 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.

16.12.1 Phase 1 Infrastructure

16.12.1.1 Ancillary Surface Facilities

The majority of the ancillary facilities for Phase 1 are located on surface, the location of the facilities are shown in Figure 16-12. Surface ancillary facilities include the production services building, the surface workshop and warehouse and surface fuel and shotcrete systems and power supply.

Underground ancillary facilities in Phase 1 are limited to satellite service bays and explosives storage as shown in Figure 16-13.

16.12.1.2 Mine Services

Mine services for Phase 1 include electrical distribution, compressed air, ventilation controls and service water.

16.12.1.3 Dewatering and Service Water

The mine dewatering 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 current hydro-geological modelling completed in 2017.

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 kept strictly separate from contact water. It is assumed that 70% of the borehole ground water inflow will be captured as non-contact water and the remaining 30% will report to the contact water sumps.

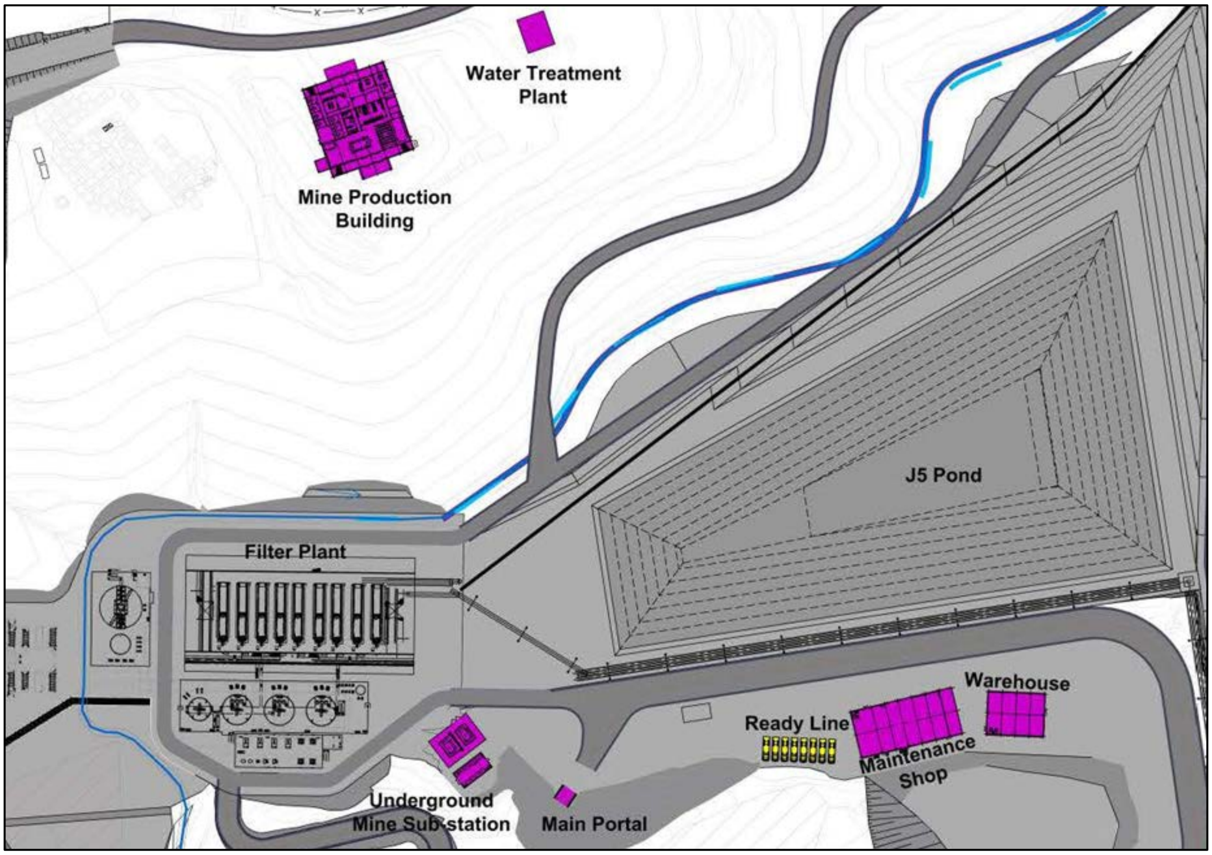


Figure 16-12: Phase 1 Mine Surface Facilities

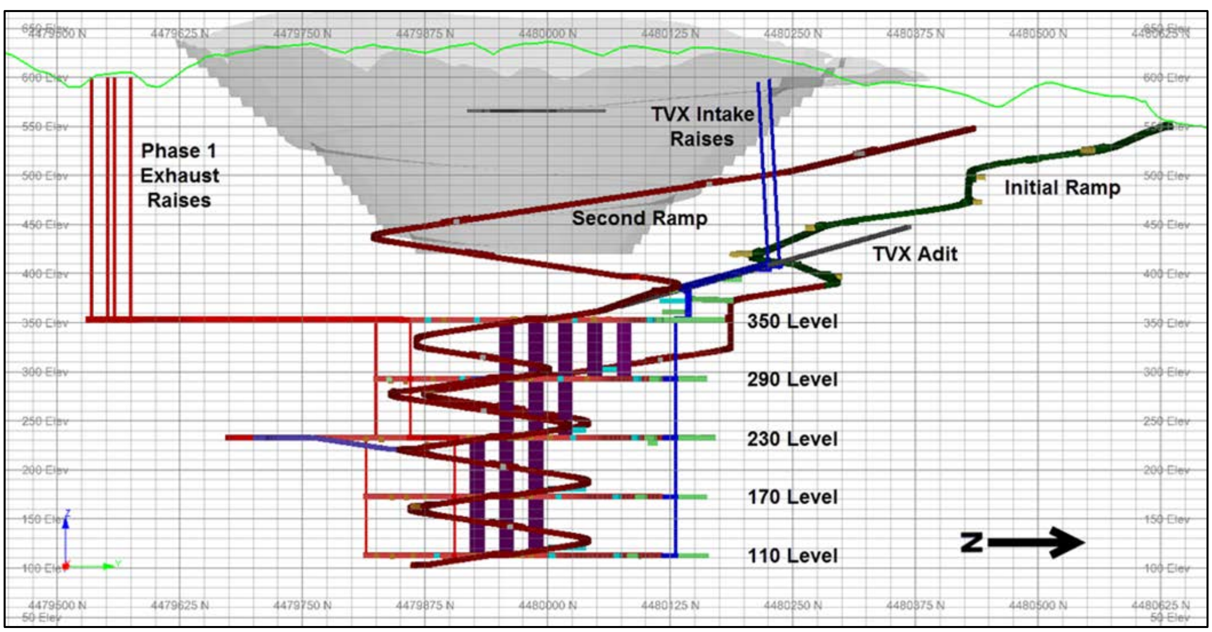


Figure 16-13: Underground Infrastructure for Phase 1

16.12.1.4 Power Controls and Communication

For the permanent power supply the entire underground mining electrical power will be fed from the 20 kV sub-station located at the filter plant. The power then is delivered to the 10 MW underground mine main sub-station located near the mine portal. The underground medium voltage distribution voltage will be 20 kV.

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 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.

16.12.2 Phase 2 Infrastructure

16.12.2.1 Ancillary Facilities

Since trucks will no longer be hauling ore to surface regularly in Phase 2, an underground workshop will be established on 50 Level south of the east ramp. An underground fuel bay will also be established on 50 Level, accessible from the main haulage drift near the tipping points for the coarse ore bins. This location for the fuel bay allows for convenient fuelling of mobile equipment along a main travel way. It also allows the fuel bay to be connected directly to a return airway.

The majority of Phase 2 ancillary facilities are located underground, the location of the facilities is shown on Figure 16-14. Underground facilities include the underground workshop, mine rescue and materials handling. Services development will continue with egress, dewatering, ventilation compressed air, power, 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.

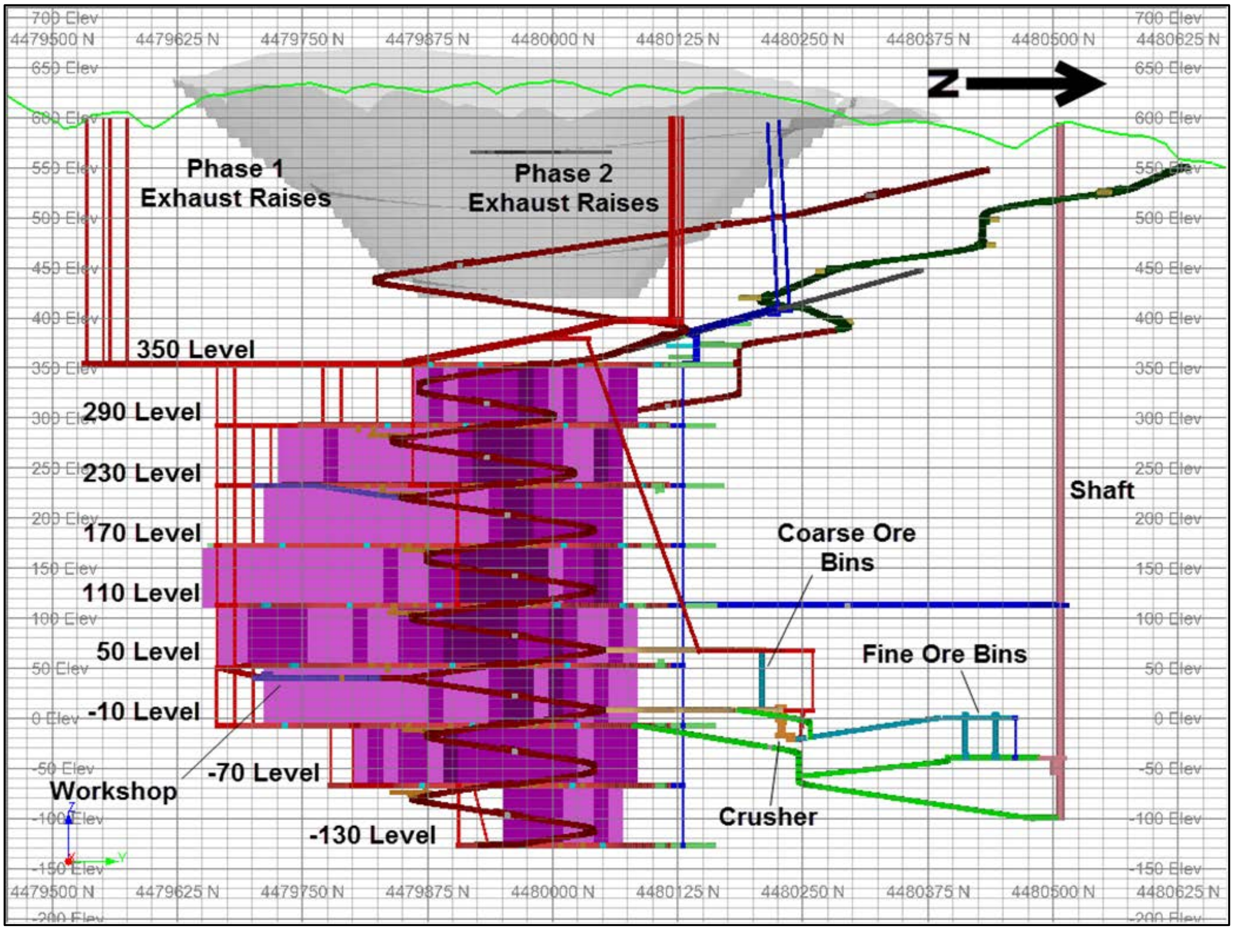


Figure 16-14: Phase 2 Mine Infrastructure

16.12.3 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.

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. These will be independent of the external reticulated compressed air supply. The remaining personnel working underground, namely the production, development, and service crews, are provided refuge by means of mobile self-sufficient rescue chambers.

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 to adit the surface.

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 several mine emergencies, including fire, serious accident, or injury.

The primary purposes of fire doors are 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 fuel bay and workshop areas.

16.13 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 return air raises (RAR). Fresh air is drawn into the mine through both the west and east ramp portals as well as the fresh air raises (FAR), 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 RARs. Auxiliary fans with ducting deliver air to work places not ventilated with flow-through ventilation.

16.13.1 Design Criteria and Basis for Design

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

- "Mining and Quarrying Operations Regulations", May 23, 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.

16.13.2 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 6.4 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.
- All internal FARs are excavated to 3 m by 3 m.
- The exhaust air from the workshop area travels via transfer drift to the 50 Level internal return air raise (IRAR). The powder magazine storage area is ventilated on 230 Level via internal return air raises. 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 regulators to achieve the estimated airflows in each working area of the mine.

16.13.3 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.

Based on the steady state production and development scenario, including air for infrastructure, the required total airflow allocation for is summarized in Table 16-13.

Table 16-13: Total Airflow Requirements

Activity / location	Phase 1 (m^3/s)	Phase 2 (m^3/s)
Production	153	459
Development	102	102
Haulage	95	213
Ancillary equipment and shop	125	207
Crusher	-	35
Sub total	475	1,016
Balancing inefficiencies (20%)	95	204
Total	**570**	**1,220**

16.13.4 Ventilation Strategy

The ventilation system for Skouries is designed as a “pull” or exhausting system. Fresh air enters at the portals and FARs and is exhausted to the RARs through to the surface. Surface-mounted exhaust fans provide the pressure differential to pull air through the Skouries underground mine.

Figure 16-15 shows the surface to 350 Level ventilation arrangement, which is representative of the ventilation strategy for Phase 1 and Phase 2.

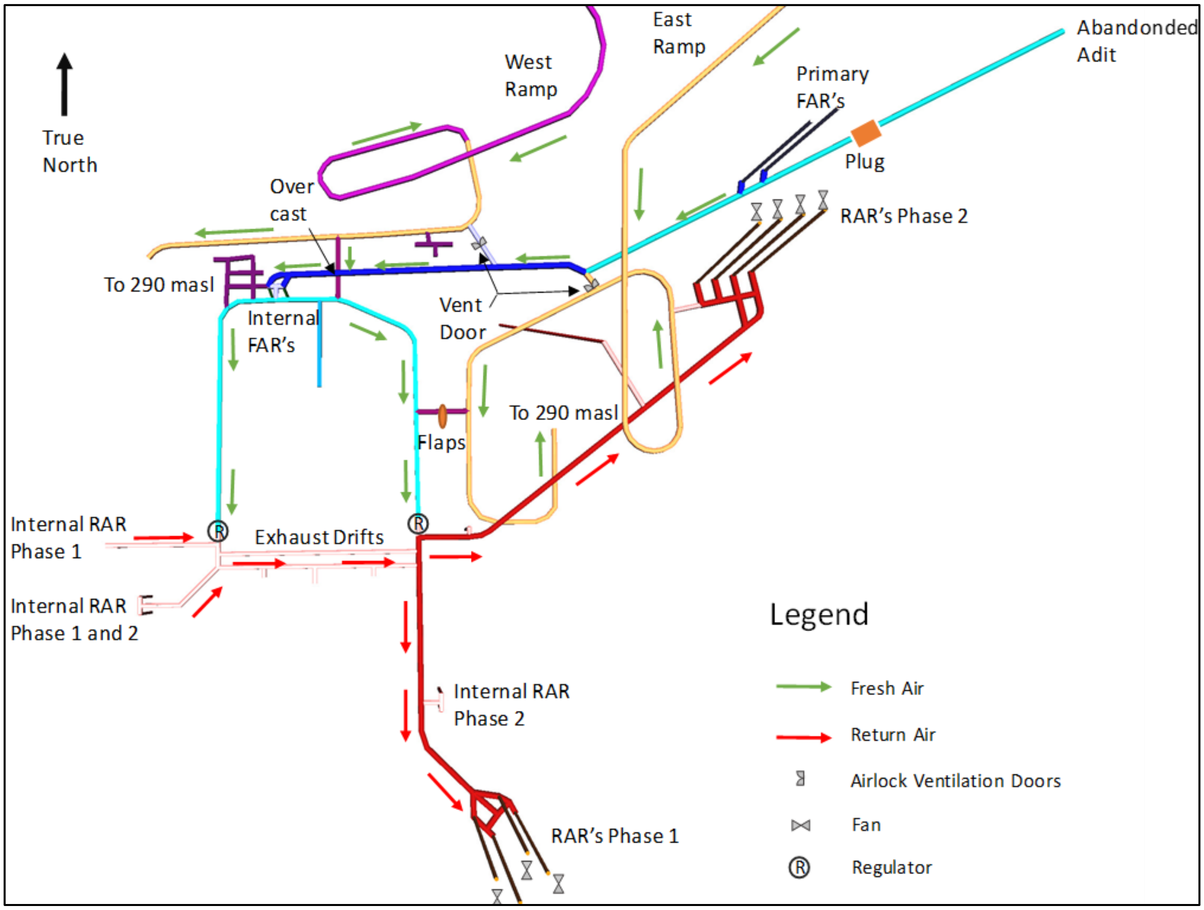


Figure 16-15: Surface to 350 Level Ventilation Arrangement

During normal conditions, fresh air will be delivered down the west and east ramps and through the primary 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 fresh air raise 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 fresh air raises connecting from 110 Level to the levels below. The air distributed through these IFAR's to U-Drives on each level will be regulated via secondary distribution fans as discussed.

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 return air raise 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 internal RARs on the east side of the deposit.

Figure 16-16 shows the final Phase 2 system looking from the southwest, which functions as follows:

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

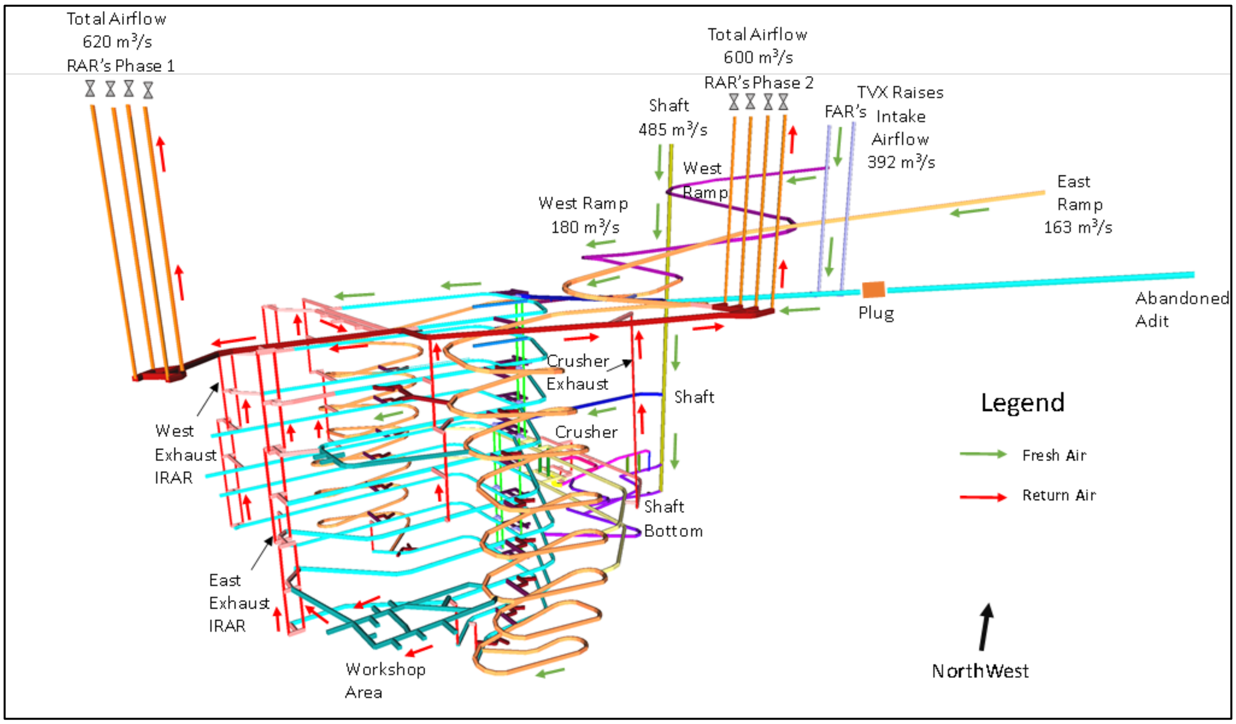


Figure 16-16: Phase 2 Steady State Ventilation System

16.14 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 pressure-filtered tailings cake, thickened tailings slurry, and cementitious binder to produce 200 m^3/hr (Phase 1) and 450 m^3/hr (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 pump assistance via boreholes and pipelines to the stopes. Apart from four 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 sublevel 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 50% to 55% and 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 monitoring for any risk of impending blockage. The paste fill reticulation system for Phase 1 is shown in Figure 16-17. In Phase 2, a third surface borehole will be drilled, and the internal boreholes extended down to -70 Level. Trunk delivery pipelines on each of the sublevels will be extended to stoping crosscuts to enable filling of Phase 2 stopes.

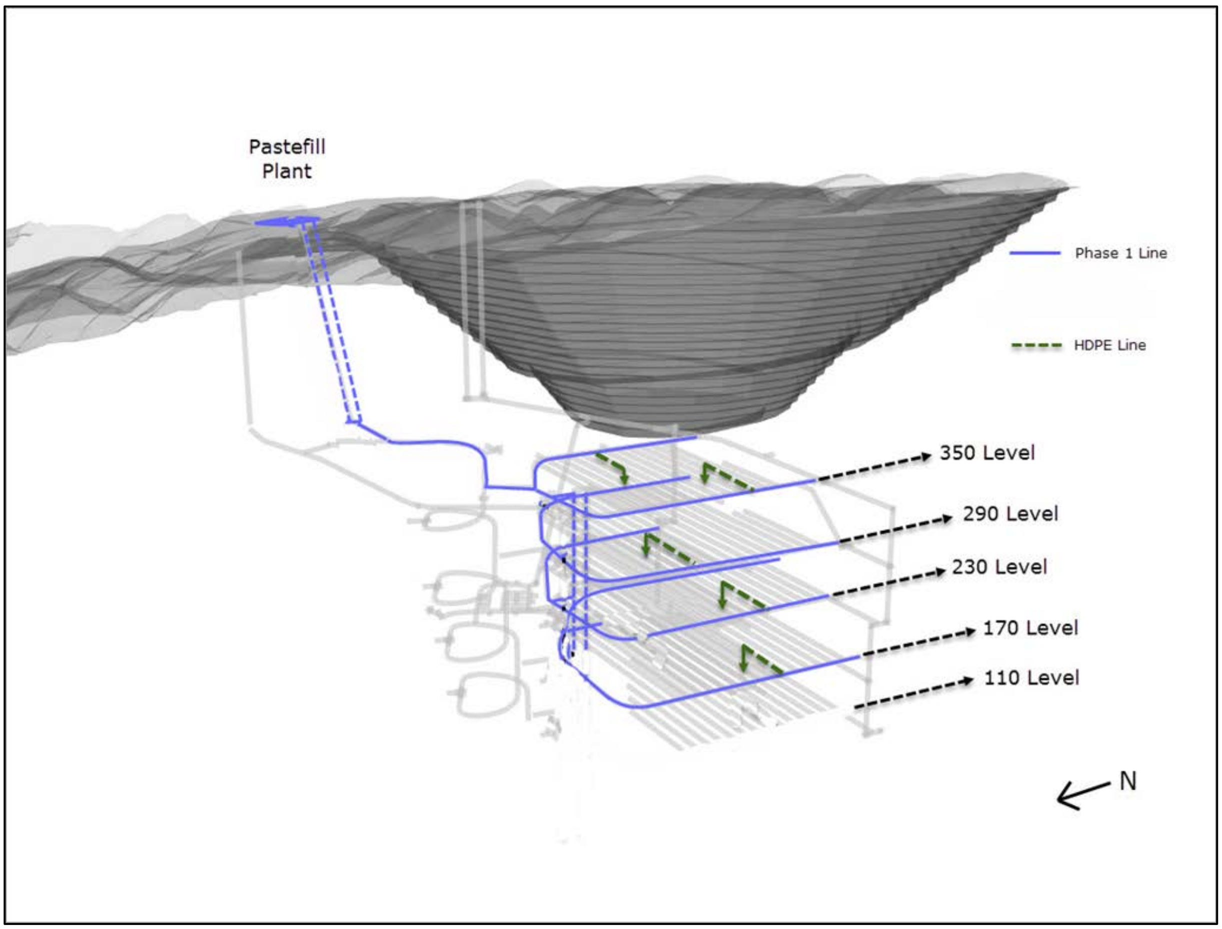


Figure 16-17: Phase 1 Isometric Schematic of Paste Reticulation

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 has applied vertical strength zoning 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, 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 applies to all stopes mined off 110 Level, which will have stopes extracted below in Phase 2, and all stopes mined on 230 Level in the initial mining sequence.

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 exposures for other stopes. The summary of unconfined compressive strength (UCS) and binder recipes is provided in Table 16-14.

Table 16-14: 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%

16.14.1 Paste Backfill Plant

All tailings from the Skouries mill will be thickened to 68.5% w/w in three deep cone thickeners (DCT) and then dewatered to 85% w/w in a series of pressure filters to produce a moist tailings cake that can be conveyed and stacked in the surface IWMF or, later, to the open pit. The pressure filter plant will be located close to the mine portal. When backfill is required, some of the tailings slurry will be pumped and cake conveyed to the paste fill plant located adjacent to the shotcrete plant on the edge of the open pit. The Phase 1 plant also makes provision for light cycloning of the tails to strip out the finest clay-sized fractions should this be required.

Figure 16-18 shows an isometric view of the Phase 1 and Phase 2 plants. Two large binder silos and agitated tailings slurry storage tank are shown, which are shared between the paste fill plants. In

Phase 2 the tailings delivery conveyor will be extended to the open pit for disposal and a second transfer point installed to supply the Phase 2 paste plant.

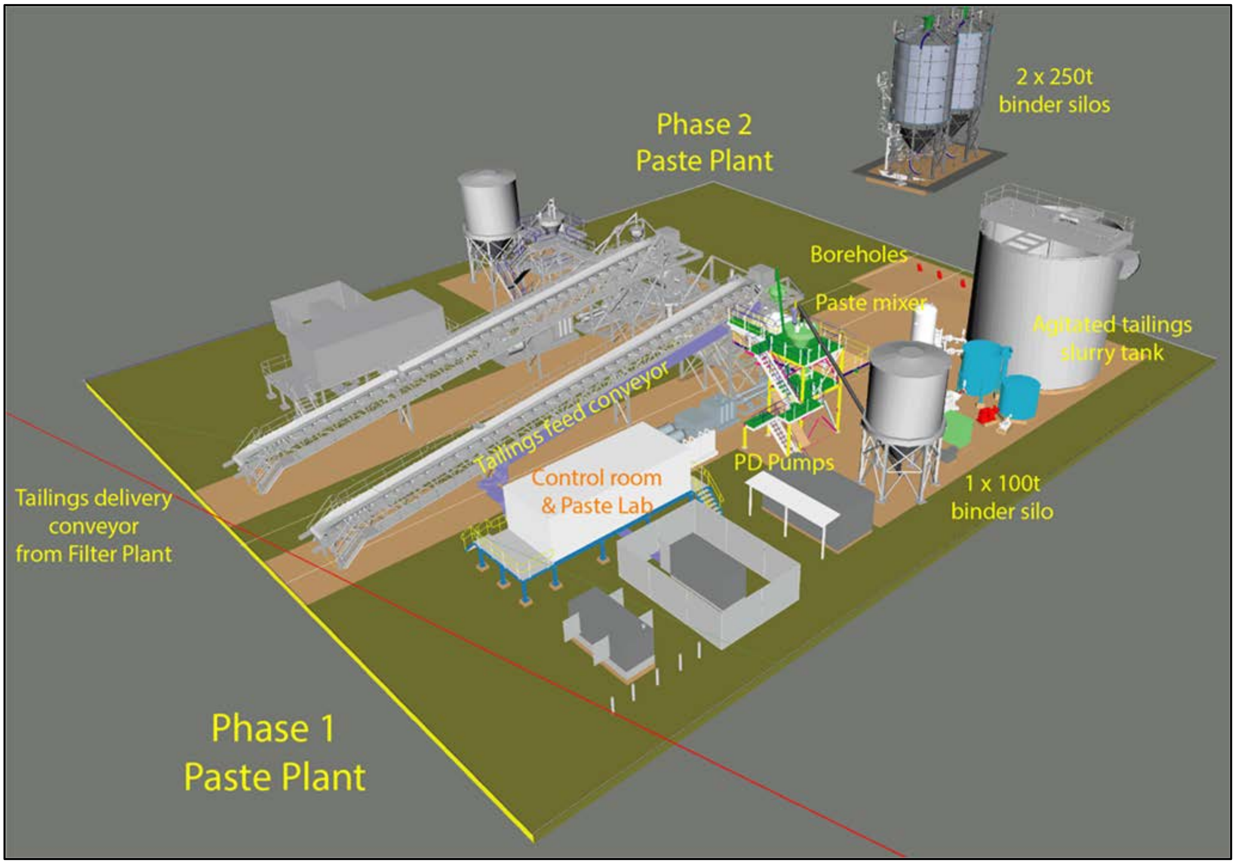


Figure 16-18: Isometric render View of the Phase 1 and 2 Paste Fill Plants and Binder Silos

The Phase 1 paste plant is designed to mix tailings filter cake, tailings slurry, and cement to produce 200 m^3/hr of paste (full capacity of 250 m^3/hr). The plant is a continuous mixing operation with paste delivered underground via a positive displacement pump. In Phase 2 a second plant identical to Phase 1 is envisaged that will be capable of 250 m^3/hr, resulting in a total paste plant throughput of 450 m^3/hr.

Tailing slurry at 68.5% solids by weight is pumped from the process plant via pipeline into an agitated storage head tank at the paste plant. A single 610 m^3 tank will provide for three hours storage buffer during Phase 2 and seven hours during Phase 1 operation. From the slurry tank, a single mixer slurry pump provided with variable speed drive delivers slurry to the vortex mixer where the slurry is pre-mixed with cement.

Cement will be delivered to site in bulk and pneumatically unloaded into one of two 320 m^3 (nominally 425 t) cement silos. The silos are equipped with a reverse pulse dust collector to eliminate dust emissions.

The paste rheology (slump) is controlled by producing a consistent paste solids content, which is controlled by the adjustment of trim water. The plant is operated to maintain a constant (maximized) paste pump discharge pressure for each stope.

All sampling from the mixer is via a sampling valve. Regular samples will be taken and measured for rheology (slump) properties and placed into curing cylinders for strength testing. A simple QC laboratory within the paste plant will include a temperature-controlled humidified curing chamber, a 50-kN-UCS strength tester, and a vane-shear rheometer. 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.

Paste fill from the paste mixer is transported underground using a hydraulic piston pump.

16.14.2 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 360 Level. The boreholes will be drilled 235 m long at 72º dip at NB 300 mm and will be cased to NB 200 mm using threaded oil-industry type hardened pipe. A third borehole has been included in the costing estimate as a contingency. The holes will be drilled first with HQ diamond drills and then reamed out to final diameter of 311 mm (12.25”). In Phase 2, one more borehole plus a contingency 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 sublevels; this will be repeated on each sub-level. The holes will be drilled at 311 mm NB and cased to 200 mm NB with butt-welded black steel Schedule 80 pipeline. In Phase 2, this borehole system will be extended down to 70 Level.

On each of the paste delivery levels, Schedule 80 black steel 200 mm NB pipelines will be installed around the perimeter drives. At each primary and secondary corridor, valves will be manually opened and pipes connected as required to ensure direct connection between the paste plant on surface and the crosscut for paste delivery into the target stope.

For each cross cut, 250 mm OD and 200 mm ID PN16 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 each HDPE pipe will be used twice, thereby halving the crosscut pipe quantities.

16.14.3 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 recipe requirements. A suitable location for the draw point fill barricade will be selected and prepared. 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 or fibrecrete barricade constructed in the drawpoint. The barricade will be located approximately one drive width 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/hr, up to 11 m of slurry head can be expected behind each barricade constructed at the drawpoint. This is equivalent to a horizontal pressure of 200 kPa at 1.87 t/m^3.

The paste will be poured continuously into each stope, in one long pour where possible. This has the benefit of continuous fill runs taking place over 5 to 7 days without the need to flush lines. Interruptions to supply of paste for up to one hour (and possibly two hours with experience) will be permitted without flushing but extended delays will require a flush cycle and restart of the paste fill line. At the typical plan area of each stope of 450 m^2, and filling at 200 m^3/h, the fill rise rate will be approximately 0.45 m/hr. Therefore, filling a 60 m high stope will require in the order of 133 hours. Initial stope and barricade pressures will be monitored to verify design assumptions, and adjustments to design, if required, will be made.

16.15 MOBILE EQUIPMENT REQUIREMENTS

Table 16-15 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 6.4 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: 49.4 t (BAS FMX 10 x 4).
- Payload: 40.0 t (Volvo A45G).
- Average haul speed: 10 km/hr loaded, 15 km/hr empty.
- Queue for loading time: 5 minutes.
- Loading time: 6 minutes (3 LHD passes per load).
- Dumping time: 2 minutes.

- Lost time for traffic conflicts: 10%.
- Engine hours accumulated through activities other than haulage: 15%.

Figure 16-19 provides a summary of the development and production equipment fleet by year.

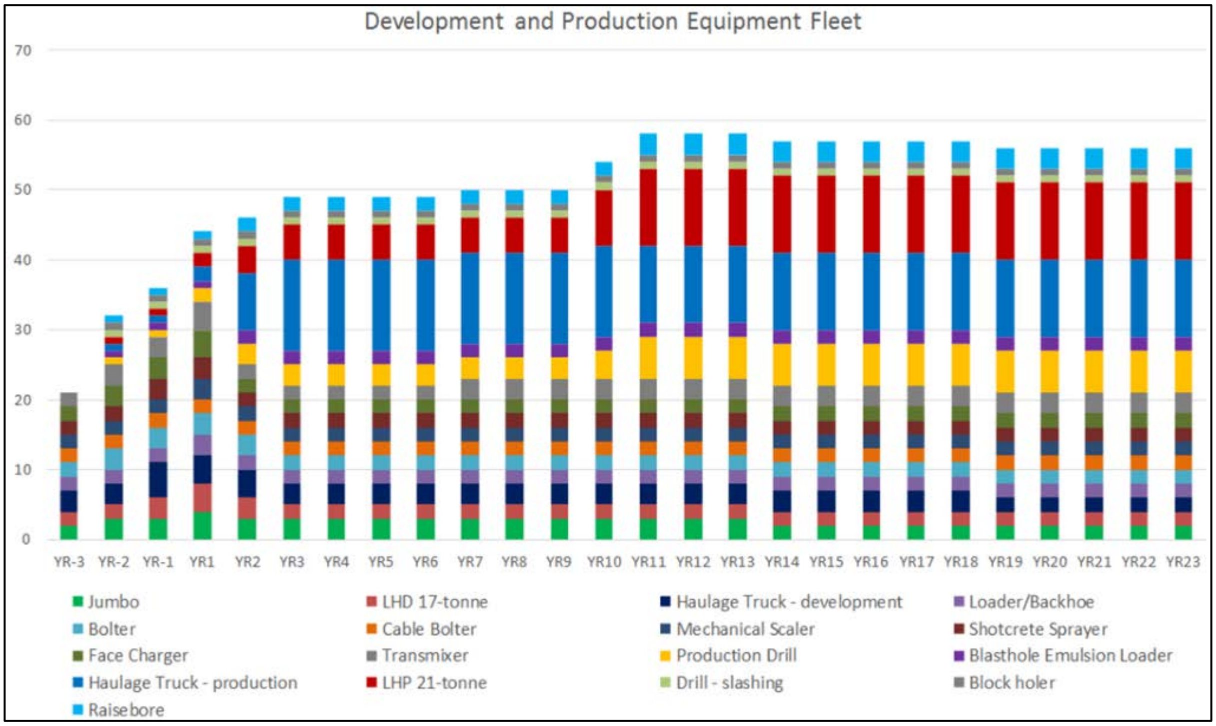


Figure 16-19: Development and Production Equipment Fleet (excluding service fleet)

Table 16-15: Mobile Equipment Requirements

Equipment type	Phase 1 peak	Phase 2 peak	Manufacturer	Model
Development				
2-Boom jumbo	3	3	Sandvik	DD422i
LHD 17-tonne	3	2	Sandvik	LH 517
Haulage truck - development	5	3	Volvo	A45G FS
Loader / backhoe	2	2	JCB	3X17
Bolter	3	2	Sandvik	DS411-C
Cable bolter	2	2	Sandvik	DS421
Mechanical scaler	2	2	Normet	Scamec 2000 M
Shotcrete sprayer	3	2	Normet	Spraymec LF050 VC
Face charger	3	2	Normet	Charmec LC605DEV
Transmixer	3	3	Volvo	FMX Concrete Mixer
Production				
Production drill - Longhole	3	6	Sandvik	ITH DU 422i
Blasthole emulsion loader	2	2	Normet	Charmec LC605
Haulage truck - production	13	13	Volvo	BAS FMX 520HP 10x4
LHD 21-tonne	5	11	Sandvik	LH 621
Drill - slashing	1	1	Sandvik	DL321
Block holer	1	1	Maclean	BH3
Raiseborer	2	3	Atlas Copco	34RH C QRS
Services				
Personnel carrier	4	5	Paus	Minca 18A
Scissor lift truck	2	2	Normet	UTILIFT MF 540
Service truck	1	1	Getman	A64
Boom truck	1	2	Getman	A64 Crane-S
Emulsion transport	1	2	Normet	Multimec MF 100
Explosives transport	1	2	Paus	Universal 50-3
Motor grader	2	2	Paus	PG10HA
Fuel truck	2	2	Paus	Universal 50-3
Pick-up truck	17	25	Toyota	Landcruiser HZJ79
Forklift	1	2	Komatsu	FD45T-10
Telehandler	3	3	Manitou	MT 1335 HA
Water truck	2	2	Volvo	FMX Water truck
Total underground	**87**	**108**		

SECTION • 17 RECOVERY METHODS

17.1 PROCESS DESIGN BASIS

The process plant and infrastructure design of the Project was originally prepared by Aker Kvaerner, (subsequently called Aker Solutions, and now called Jacobs) and presented in the 2007 Cost and Definition study. Subsequent to the Cost and Definition study Outotec was retained on a design-supply contract to complete the basic engineering of the Project process facilities. Outotec's scope of work included the supply of process equipment within their manufacturing range, including the semi-autogenous grinding (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, Golder Associates and Patersen & Cook completed the basic and detailed engineering of 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. The process plant general arrangement is shown in Figure 17-1.

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 a reduced tonnage of 6.4 Mtpa, while also processing rehandled 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 a 6.4 Mtpa rate.

The main design parameters of the process plant are indicated in Table 17-1. The plant will process the copper/gold ore grading 0.49% copper (LOM average) and 0.74 g/t gold (LOM average). Expected LOM average recoveries are 87.9% for copper and 82.4% for gold, respectively. The mill will produce a doré which contains approximately 80% gold and 10% silver and 10% copper and a copper 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.

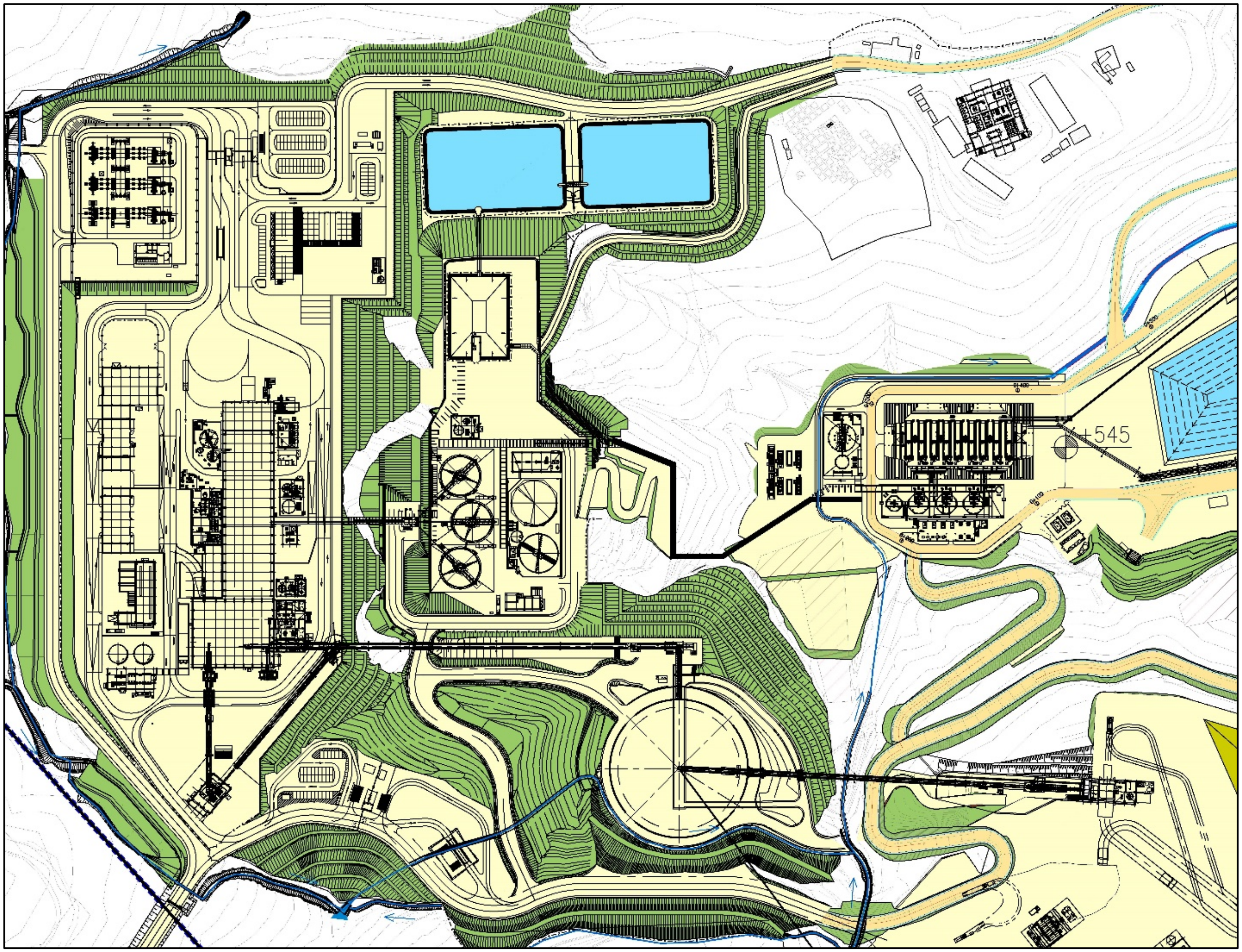


Figure 17-1: Plant Layout

Table 17-1: Basic Design Parameters of Skouries Process Plant

	Parameter	Design Basis
1	Life of Mine	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 t/d
5	Open pit primary crushing product particle size	80% passing 150 mm
6	Maximum daily ore treatment	24,000 t/d
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	0.74 g/t
11	LOM average copper grade in the mill feed	0.49%
12	LOM overall gold recovery	82.4%
13	Average gold recovered by gravity concentration	18.0 %
14	LOM copper recovery	87.9%
15	LOM average gold grade in flotation concentrate	27 g/t
16	Average copper grade in flotation concentrate	26%

17.2 PROCESS DESCRIPTION

The process plant design has been based on extensive metallurgical testwork. The design of the process is based on a conventional flowsheet for the treatment of porphyry copper ores and as such offers a well proven design. This coupled with the straightforward ore metallurgy and design margins will ensure that it 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 is presented in Figure 17-2. The unit operations comprise of the following:

- Primary crushing and ore stockpile.
- Grinding and pebble crusher.
- Flotation and regrind.
- Gravity concentration and recovery.
- Concentrate and tailings thickening.
- Concentrate filtering storage and loadout.
- Tailings filtration.
- Reagents and services.

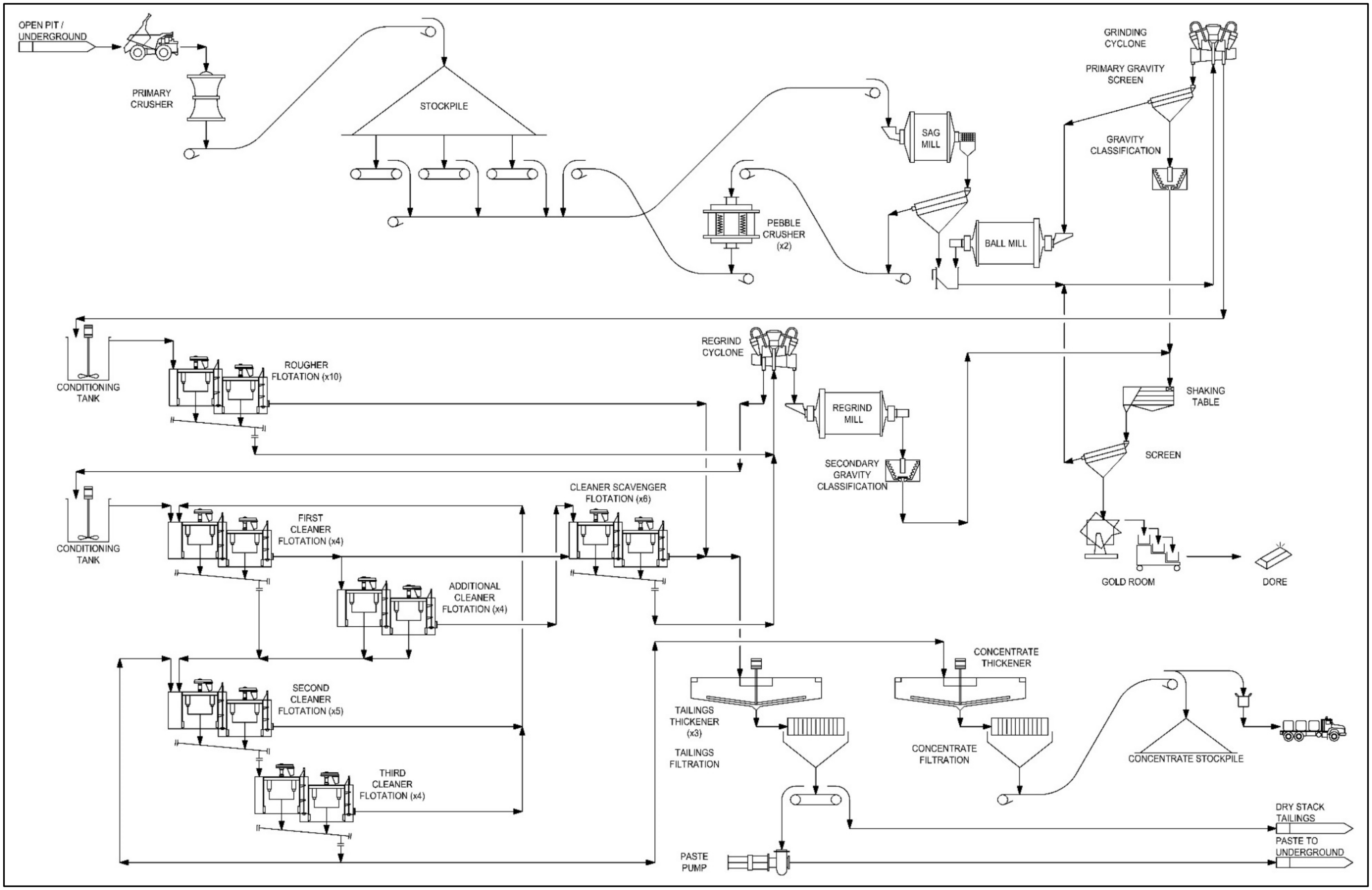


Figure 17-2: Simplified Process Flow Diagram

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 the underground, to the primary crushing station dump pocket. The ore is then crushed by the gyratory primary crusher. Then, 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 coarse ore stockpile is implemented using a belt conveyor system which is comprised of the primary crusher sacrificial conveyor and the primary crusher discharge conveyor. An overband magnet at the discharge of the sacrificial conveyor removes scrap metal before entering 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 production. The stockpile 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 capacity and each one is driven by a 75 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 equipped with blockage detector 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 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. A gravity concentration circuit is integrated in the primary grinding 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 and 3.6 mm.

The SAG mill has diameter of 9.75 m, effective grinding length 4.57 m and is driven by two 4.8 MW motors with VFD including the auxiliary lubrication circuits. The grinding media is balls of diameter 125 mm 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 one will be driven by a 450 kW motor.

The primary ball mill has diameter 7.01 m, effective grinding length 9.75 m and is driven by two 4.8 MW motors. The grinding media is balls of diameter 60 mm. 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, greater than 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 as well as with the primary gravity circuit tailings. In this sump, process water of controlled quantity is also added to ensure the wet grinding in the ball mill.

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 solid and particle size of typically 80% passing 120 µm comprises the feed of the flotation circuit. The cyclone underflow slurry is directed to the ball mill feed chute.

Approximately 20% to 30% w/w of the cyclone underflow is directed to the primary gold gravity circuit located in an adjacent building. The tailing of the primary gold gravity circuit is directed back to the SAG and ball mill sump.

The grinding circuit operation is controlled by an automated control system to ensure that the product size is 80% passing 120 µm at the maximum daily ore feed of 24,000 t and for all types of ore.

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 at 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

The flotation is carried out in six stages

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

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

The flotation circuit is fed from the ball mill 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. Then, it flows into the rougher flotation bank. This conditioning tank is also used to increase surge storage capacity.

The flotation cells are mechanically agitated 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 standby).

In order to control the flotation circuit operation and the quality of intermediate concentrates and tailing, appropriate automations, automatic sampling of the main streams and on-stream analyzer will be installed. 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, sulphidiser 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 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, sulphidiser 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 and are directed together to the tailings thickeners. Additional reagent dosing (SIPX or AP3418, MIBC or DF250 frother, promoter AP5540, sulphidiser if necessary, guar gum and lime) is provided to the cells in the 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, sulphidiser 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 (sulphidiser 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 100 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

The concentrate regrinding is carried out by the regrind ball mill. The regrind ball mill has diameter 4.60 m and effective grinding length 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 is forged steel ball typically with 25 mm diameter.

The regrind ball mill operates in closed circuit with a cluster of 14 hydrocyclones, 10 duty and 4 standby. The operation of the regrind circuit will be controlled by the central DCS control system 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 hydrocyclones cluster. The regrind cyclone cluster overflow will flow to the 1st cleaning circuit.

The regrind cyclone underflow feeds the regrind ball mill. A portion of cyclone underflow will be fed to a gravity concentration circuit, which is different from the one in the primary grinding circuit. The regrind gold gravity concentration tailings flow back to the regrind ball mill.

17.6 GRAVITY CONCENTRATION AND RECOVERY CIRCUIT

The gold gravity concentration circuit has been designed in a vertical arrangement to maximize the gravity flow among successive processing stages.

The gold gravity concentration circuit is comprised of centrifugal Knelson gravity concentrators. The principle of operation of the gravity concentrators is based on the difference between specific gravities of gold and other accompanying minerals, through which separation is accomplished. This circuit is comprised of three stages.

17.6.1 Primary Gold Gravity Concentration Circuit

A portion of slurry from cyclone underflow in the primary grinding circuit passes through two screens (with 2 mm opening) at the front end of the circuit and screen undersize slurry is equally distributed to two centrifugal gravity concentrators. The gravity tailings are recycled back to the SAG and ball mill sump. The produced concentrate from each of these two concentrators flows by gravity into a secondary gravity concentration circuit for upgrade.

The primary gold gravity concentrate slurry is collected in a settling tank for the coarse gold and a second settling tank for the fine gold, where sufficient time for solids settling is provided. Then the concentrate is fed to a shaking table where the final upgrade is carried out. A belt magnetic separator is installed above the shaking table for removal of any tramp iron and other magnetic particles.

The tailings from the above shaking table are collected and directed to a small semi-automatic centrifugal gravity concentrator. This will act as a final gold scavenging stage prior to the tailings being discharged into the combined secondary gravity circuit tailings sump. From this sump, the tailings are pumped to the SAG and ball mill sump. The concentrate produced from this process is directed to a settling tank and then to a second shaking table for final upgrade. Solids settle out while the overflow ends up to the secondary gravity separation tailings sump.

The final gravity concentrate from both shaking tables is collected into a sealed concentrate storage box.

17.6.2 Secondary Gold Gravity Concentration Circuit

The fine material produced from the regrinding circuit passes through the screen (with 2 mm opening) installed at the beginning of the circuit and feeds three centrifugal gravity concentrators in the first step and two gravity centrifugal concentrators in the second step. The regrind gravity tailings are directed back to the regrinding circuit. The concentrate is directed to the secondary gold gravity separation circuit for upgrade.

The gold storage cabinet is transferred by an electric overhead beam hoist into the adjacent gold room once a day.

17.6.3 Gold Room Drying, Calcining & Smelting

The gold room is constructed of reinforced concrete and is provided with dedicated flue gas exhaust and ventilation systems. In this room, the doré gold bars will be produced, stored and loaded in specially designed vehicles.

The final gravity concentrate delivered to the gold room is manually distributed in stainless steel trays, which are loaded into a drying/calcining oven. The dried and calcined material is temporarily stored on cooling racks within the gold room. The flue gases generated from the oven are collected and directed through a wet scrubber before being vented into atmosphere.

Approximately once per week the cooled dry product is blended with fluxes and transferred into an induction smelting furnace. The resultant smelt is poured over a series of 10 kg cascade doré gold molds which produces semi pure alloy of approximately 80% gold. The smelting slag is crushed manually within the Gold Room and tipped onto the SAG mill feed conveyor for re-processing within the concentrator.

17.7 THICKENING

17.7.1 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 for the acceleration of solids sedimentation rate. The solids concentration in the thickener underflow is typically 60% w/w. The underflow is pumped to filter presses for filtration.

17.7.2 Tailings Thickening

The flotation tailings are equally distributed to three hi-rate tailings thickeners of 26 m diameter. Flocculant solution is added for the acceleration of solids sedimentation rate. The solids concentration in the thickeners underflow is typically 55 to 65% w/w. During the underground mine development phase, the thickener underflow is filtered for the tailings disposal, whereas during the underground mine operation phase, in order to refill the gaps caused by the mining activity, a portion of the thickener underflow is pumped to the back-filling unit and the remaining portion to the tailing disposal. The overflow from the three thickeners is recirculated as process water in the plant.

17.8 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. In the building there is also a second filter press for spare capacity as well as concentrate bulk bag loading system.

The concentrate is transferred offsite via closed cargo trucks for transportation to the smelter.

17.9 TAILINGS FILTRATION

Tailings slurry, from the copper/gold processing facility, are pumped into a three-way distribution box which splits the slurry evenly into three streams. Each stream gravitates into one of three agitated tailings stock tanks which provide buffer storage ahead of the filtration stage. During normal operations slurry from these three tanks is pumped forward to seven of the nine filter presses. Of the remaining two filters, one is immediately available to provide standby capacity while the other is off line for planned maintenance.

Slurry from tailings stock tank No.1 is pumped to tailings filter presses No.1 to 3. Slurry from tailings stock tank No.2 is pumped to tailings filter presses No.4 to 6 and slurry from tailings stock tank No.3 is pumped to tailings filter presses No.7 to 9. The tailings are pumped from the stock tanks into the filter presses by centrifugal pumps until a pressure of 6 to 7 bar is reached. The filters produce a moist cake with a moisture content of approximately 15% w/w and the filtrate contains a small amount of very fine solids.

Closing of the filter press is carried out in two stages. The mobile plate head is initially closed by means of a pinion/rack mechanism. The press is then fully closed and sealed by four hydraulic jacks located on the fixed head.

Each filter is equipped with a cloth shaking and a cloth rinsing system to ensure good cake discharge and reliability. Cloth rinsing is applied after each cycle when the plate pack is open and cake discharge is complete. Rinsing water is collected on the swivel plates underneath the plate pack. The operation of the filter is controlled automatically by the PLC control.

The filters operate on a batch basis with a cycle time of nominally 12 to 13 minutes.

The filter cycle stages are as follows:

- Filter filling and filtration.
- Core wash and blow.
- Air drying of cake with membrane squeezing.
- Press opening for cake discharge.
- Cloth shaking and rinsing.
- Press closing.

Discharged filter cake drops through filter cake breakers onto reversible variable speed filter discharge conveyors, one per filter press. The discharge conveyor under each filter collects all the filter cake dropped from a single cycle. The belt operates at a higher speed during the cake drop to give an even distribution of cake along the whole length of the belt. When the drop finishes the belt slows down to discharge its load at a constant rate over the time period of the remainder of the filter cycle so that the belt is fully cleared just before the next cycle drop.

The discharge conveyors for filter press No. 1, 3, 5, 7 and 9 can deliver filter cake onto dedicated intermediate movable divert conveyors if filter cake is required to be conveyed to the backfill plant. In “Position 1” the movable divert conveyors are not operational. The filter discharge conveyors deliver cake directly onto the common filter cake collection conveyor for transfer to the IWMF overland conveying system.

For production of filter cake for transfer to the backfill plant the movable divert conveyors are winched to a new position “Position 2” to receive filter cake from the filter discharge conveyor. The divert conveyor transfers the cake onto the common filter cake backfill collection conveyor for transfer to the mine backfill plant overland conveyor system.

The discharge conveyors for filter press No. 2, 4, 6 and 8 deliver filter cake directly onto the common filter cake collection conveyor for transfer to the IWMF overland conveying system.

17.10 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 test work for the Project was based on) 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 action of each reagent is described in detail in the following paragraphs.

17.10.1 Chemical Reagents System

17.10.1.1 Sodium IsoPropyl Xanthate (SIPX)

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

The bags are transported by an electric hoist in the 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% wt concentration.

The mixed SIPX solution will be transferred by pump from the mixing tank into a storage tank, which provides a 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

17.10.1.2 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 a 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.

17.10.1.3 Frothers

Both MIBC and DF250 will be delivered in barrels of 200 L. On average, five barrels per day are pumped into a storage tank, which provides a 58 hours (for MIBC) and 71 hours (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.

17.10.1.4 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 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.

17.10.2 Sulphidiser System

The Sulphidiser, i.e., sodium hydrosulfide NaHS, is used to regenerate the surface of the non-sulfur copper minerals such as cuprite (Cu_2O), azurite ($2CuCO_3.Cu(OH)_2$), chrysocolla, ($CuSiO_3.2H_2O$), 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 sulfur-containing copper minerals. In both cases, sulphidiser (NaHS) acts as an activator and this process is known as “sulphidation”.

Sodium hydrosulfide 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 for the first two years of the plant operation, given the fact that the sulphidiser (NaHS) will be used only during the treatment of oxidized minerals from the open pit.

The sulphidiser 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.

17.10.3 Lime System

The hydrated lime is delivered with tanker trucks (carrying 20 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 7 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 % wt.

The 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, pH probes and control valves. Reagent dosing rates are set and controlled by the main control system in the plant.

17.11 FLOCCULANT SYSTEM

Polyelectrolyte flocculant solution is added to concentrate and tailing slurries to aid the settling rate of solids. The flocculant solutions are prepared in three different systems, located also in different sections of the plant. One system serves the concentrate thickener, the second system serves the tailings thickeners and the third one serves the mine water clarifier.

All flocculant solutions concentration will be 0.25%.

17.12 PROCESS AND FRESH WATER MANAGEMENT AND STORAGE SYSTEMS

17.12.1 Process Water System (Area 55)

A storage tank (6,100 m^3) is used for storage of process water. The water comes mainly from the boreholes as well as 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 (2 running / 1 stand-by) and the 610 mm distribution pipeline.

17.12.2 Utility Water System (Area 56)

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. The water is then distributed via pumps to the various locations such as pumps cooling systems and pumps gland seals, dust suppression systems, buildings etc.

17.12.3 Fire Fighting Water System (Area 57)

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 made using the firefighting pump station which is comprised of an electrical driven fire water pump, a diesel driven fire water pump and a fire water jockey pump.

17.12.4 Potable Water System (Area 59)

A storage tank (25 m^3) is used for storage of potable water. The water comes from the boreholes and is distributed where required via the necessary pumps and a main 100 mm pipeline.

17.13 PLANT INFRASTRUCTURE AND UTILITIES

The process plant will be supported by a comprehensive operational support complex comprising of maintenance facilities, stores, change-houses, administration offices, first aid, messing facilities, security, fencing, and process plant mobile equipment.

Total power consumption of the infrastructure, open pit and processing plant is 43.5 MW, in which the process plant consuming 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 is 45.8 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, essential instrumentation and monitoring purposes. These shall be supplied from essential services LV switchboards with diesel generator back up.

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 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, a 110 kVA at the primary crusher, a 1,700 kVA at the filter plant and a 450 kVA for the supply of potable water package and administration buildings.

The process plant and site infrastructure have been located on a site that provides the best balance between geotechnical constraints, and location 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.

SECTION • 18 PROJECT INFRASTRUCTURE

18.1 WASTE MANAGEMENT

The principal waste streams generated from the Project are the overburden and waste rock from the open pit mining and underground development and the tailings from the mineral processing operations. Overburden and waste rock will be stored on surface and tailings will be used underground as paste backfill with the remainder being stored on surface. Underground storage of paste backfill is discussed in Section 16. This section references surface storage methods. The main objectives of the waste management strategy is to provide secure containment of waste rock and tailings in economically and technically feasible configuration. The Project mine plan and material balance has been developed such that overburden and waste rock is entirely used for construction requirements eliminating the need for a separate waste rock dump.

The mine plan has been developed to provide for surface storage all within the KL watershed. In addition to waste rock and tailings storage the J5 water management pond (WMP), south water management pond (SWMP), OOS and closure capping stockpile (CCS) are all included within the KL Valley. This compact mine plan utilizes industry best practice to minimize disturbance to the natural environment, including surface water and ground water impacts. The major waste management components are shown on Figure 18-1.

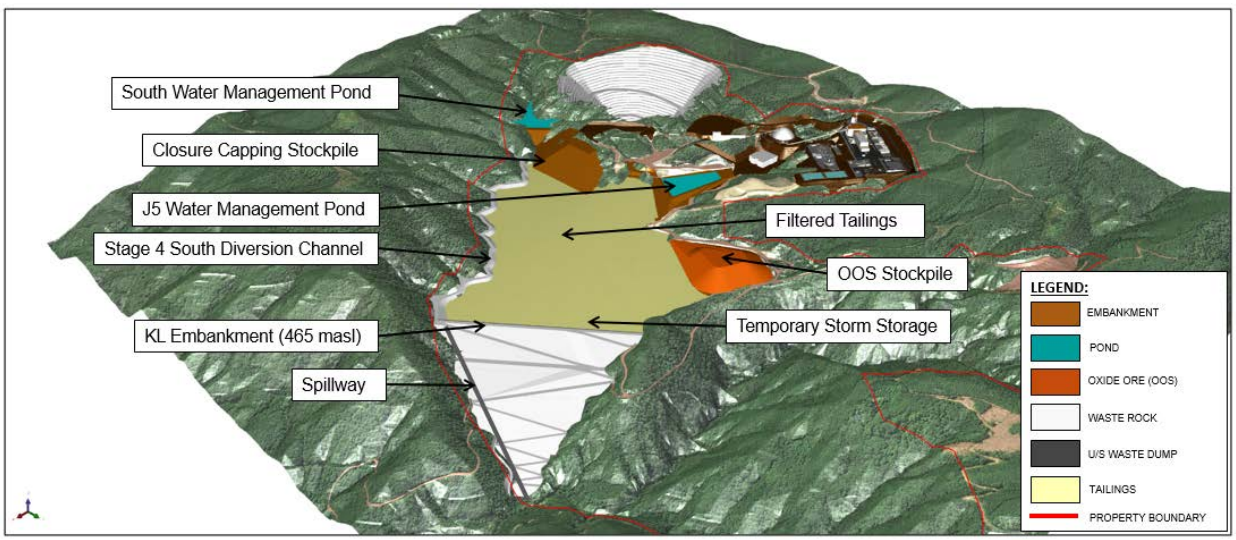


Figure 18-1: Phase 1 IWMF Site Layout (Year 10)

The Skouries Project will be mined in two operational phases. During Phase 1, over the first ten years of operation, the mine will be developed using both open pit and underground mining techniques. Overburden and waste rock from the Phase 1 mining activities will be used as a source of construction materials for the IWMF embankment, WMP, SWMP, the OOS and CCS pads and site infrastructure. Tailings will be deposited underground as paste backfill with the balance stored above ground on surface as filtered tailings in the IWMF. During Phase 2 the mining will consist entirely of underground mining for an additional thirteen years. During this period, tailings will be deposited underground as paste backfill and on surface as filtered tailings in the open pit. The IWMF will be decommissioned and progressively reclaimed at the end of Phase 1. Details of the waste management strategy for the two phases of mining follows.

18.1.1 Phase 1 – IWMF Tailings Disposal

Ore processing from the open pit and underground mining during Phase 1 will generate approximately 80 Mt of tailings, of which 12 Mt will be used underground as paste backfill and the balance of 68 Mt will be stored on surface as filtered tailings in the IWMF. Mining of the open pit will generate 52.9 Mt of overburden and waste rock and underground development will produce 2 Mt of waste rock. The overburden and waste rock produced will be used as a source of fill for construction.

Tailings will be thickened in deep cone thickeners at the process plant and pumped to the filter plant for filtration prior to placement in the IWMF or use underground as backfill. Filtered tailings will be transported to the IWMF via a conveyor system and placed in compacted lifts with stacking equipment, dozers, and compactors. The final tailings surfaces will have a maximum grade of nine percent towards the southeast; to facilitate drainage towards the spillway that will be located at the south side of the KL embankment.

A basin underdrainage system will extend throughout the IWMF basin to provide an outlet for groundwater seepage and water that drains from the tailings mass. The basin underdrain will terminate at two underdrain collection sumps located at the upstream toe of the IWMF embankment. Water that collects at the basin underdrain sumps will be pumped through sloping riser pipes and recycled to the WMP.

Use of waste materials has been balanced according to the scheduled availability of overburden and waste rock from the mining activities and the requirement of construction materials that is required each year. The overburden and waste rock materials will be primarily used in the construction of the IWMF components with the balance being used for construction of other site infrastructure. A description of the main components of the IWMF follows.

The final crest of the IWMF will be constructed to an elevation of 465 masl, the total estimated fill volume of the IWMF embankment is 33 Mt of fill. Fill for the embankment will be provided from the open pit overburden and waste rock material and placed to suit the scheduled volume of tailings that must be stored in each year. The embankment will be constructed with an upstream waste rock zone, a core zone of low permeability red clay fill and a hard rock buttress. The core zone will be keyed into the bedrock foundation to provide a seepage cut-off. Any seepage water that passes through this zone will be collected in the seepage collection sump located downstream of the IWMF embankment and will be pumped to the WMP.

The two site water management ponds will be constructed at the upper end of the KL Valley from approximately 6 Mt of compacted waste rock provided from the mining operation. The water management ponds are elevated by waste rock pads to keep them above the final elevation of filtered tailings for use throughout the life of mine. An estimated 3 Mt of overburden and waste rock will be used for construction of diversion ditches.

Approximately 5.4 Mt of oxide ore mined in Phase 1 will be placed in a stockpile located at the north edge of the IWMF. Oxide ore will be stored through Phase 1 and processed early in Phase 2 of the mine operation. The OOS will be placed on a fill platform constructed with approximately 3.5 Mt of compacted waste rock that elevates the OOS to keep it above the final tailings elevation of filtered tailings to allow the material to be reclaimed during Phase 2.

An estimated 3 Mt of closure capping material will be required for the reclamation of the IWMF to meet environmental commitments. The capping material will be stockpiled at the north west edge of the IWMF and adjacent to the KL south diversion system. The capping material will be placed on a fill platform constructed of 2 Mt of compacted waste rock that elevates the capping material above the final tailings elevation of filtered tailings to allow it to be reclaimed at the end of Phase 1.

The IWMF closure plan is shown on Figure 18-2. The general closure design strategy develops a long-term passive–care facility with the construction of a durable cover system. Waste rock sourced from the CCS will be placed over the entire tailings surface to a minimum thickness of 2.5 m. The waste rock will then be overlain by topsoil from stockpiles and revegetated. The surface will be graded to be similar to the natural surrounding terrain. Self-sustaining closure drainage systems will be designed to have the same characteristics as the pre-development natural drainage systems. Capping of the IWMF will be progressive starting in year seven of Phase 1 on areas where the tailings has reached final elevation.

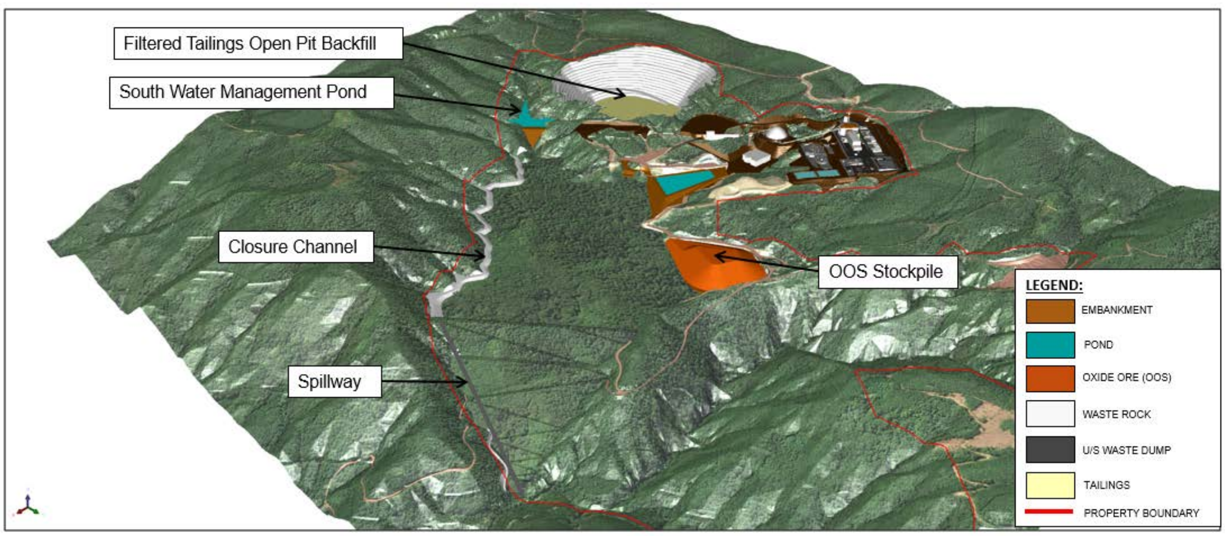


Figure 18-2: IWMF Closure Configuration

18.1.2 Phase 2 - In-Pit Tailings Disposal

Ore processing from the underground mining operations during Phase 2 will generate approximately 75 Mt of tailings, of which 35 Mt will be used underground as paste backfill and the balance of 40 Mt will be on surface in the mined out open pit. The underground mine development for Phase 2 is essentially completed during Phase 1 resulting in only a very 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.

Filtered tailings will be conveyed from the filter plant to the pit for disposal into the open pit by overland conveyor from the filter plant. Filtered tailings will be transported down into the base of the open pit using a series of overland conveyors along the haul road, deposited in the open pit, and spread and compacted in one meter thick lifts, progressing across the open pit from the haul ramp. Overland conveyors, grasshoppers and support equipment used during Phase 1 in the IWMF will be utilized in 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 which directs all inflows to the wet wells. Contact water will be managed in the J5 WMP and the SWMP during Phase 2.

A dense and well-drained tailings mass will be achieved by densification of the tailings by placing and compacting in thin lifts, collecting, removing and/or diverting groundwater, consolidation seepage and surface runoff to the open pit via a number of mitigation measures including the successful installation and operation of wet wells, drains, sumps and pumping systems.

The IWMF Phase 2 closure plan is shown on Figure 18-3. The open pit tailings surface will be closed and reclaimed to meet environmental commitments. Waste rock will be placed over the entire tailings surface to a minimum thickness of 2.5 m. The waste rock will then be overlain by topsoil from stockpiles and revegetated. Self-sustaining closure drainage grading and outlet system will be established to prevent ponding on the surface.

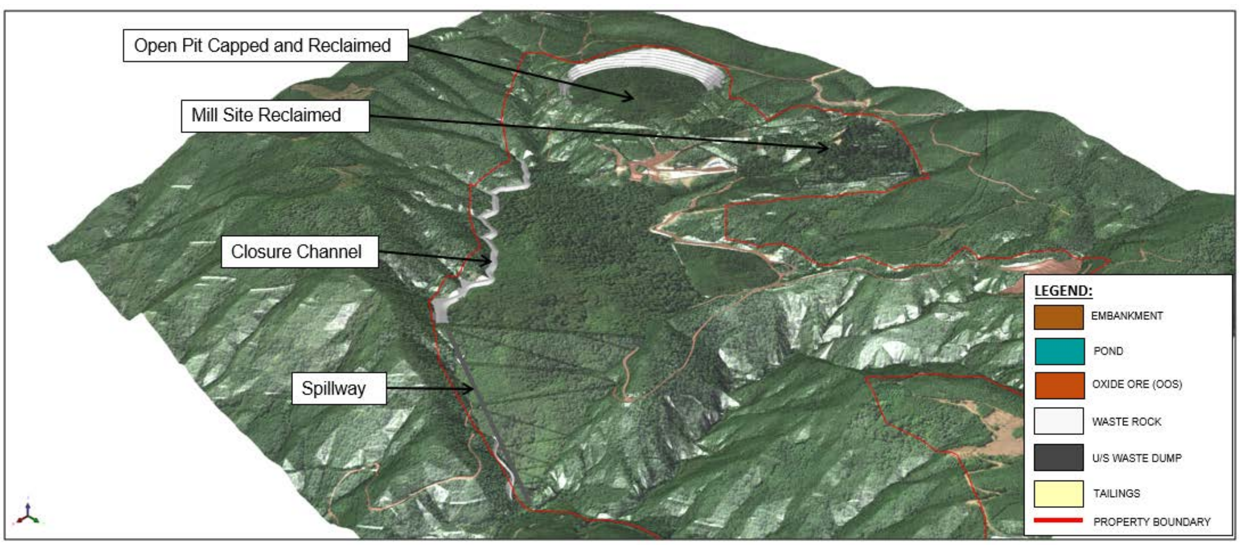


Figure 18-3: Phase 2 Final Closure Configuration

18.2 WATER MANAGEMENT

The water within the Project site can be classified into two categories, 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. A numerical groundwater model was developed for the Project utilizing site specific data from field investigations to estimate the dewatering rates for contact and non-contact water

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.
- 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.

18.2.1 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 KL south diversion channel, KT2 diversion system, and the SWMP diversion channel. The non-contact water collected by these systems will be conveyed downstream of the Project site to natural watercourses through engineered discharge structures for erosion control. The concrete lined diversion channels will be sized to safely convey peak flows.

The mine dewatering system will comprise of intercepting both open pit and underground water through a surface network of vertical dewatering wells around the open and lateral boreholes drilled radially from the underground mine levels. The main objective of this system is to intercept clean groundwater before it contacts the pit walls and underground development. The open pit perimeter vertical wells and underground lateral boreholes are designed to lower the groundwater level around the open pit and underground workings, resulting in depressurizing the area around the mine development. The non-contact water from the dewatering network will be re-injected in the bedrock aquifer through wells located in the upper areas of the Lotsaniko Valley and to supply process water.

18.2.2 Contact Water Management System

Contact water within the Project site will be collected at source, and centrally managed in the WMP. The main sources of contact water at the Project site include runoff and seepage from the IWMF, runoff and seepage to the open pit and site and underground dewatering. Contact water in the WMP will be mainly used for process water and dust suppression, with excess potentially treated and used for irrigation.

During the wetter months during Phase 1, contact water is expected to exceed Project water demand. Excess contact water during these months will be allowed to temporarily accumulate in the IWMF and/or will be temporary stored in the SWMP for use during the subsequent drier months. Any remaining surplus contact water will be transferred to a water treatment plant for treatment and used for irrigation. During Phase 2 the contact water will decrease significantly and essentially all contact water will be used in the process, so excess contact water will not be an issue. Disposal methods for this study have been developed in accordance with the requirements of the EIS, alternatives to irrigation are currently being investigated.

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. All transfer pipelines will comprise of high-density polyethylene (HDPE) pipes.

18.2.3 Site Wide Water Balance

A site wide water balance (SWWB) model was developed for the Project using 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 to be disposed of.
- Optimize water management infrastructure of the mine site including pumping rates and pond storage levels.

The general results of the SWWB show that excess non-contact and contact water is expected to be generated during Phase 1, and will need to be managed. After that, during Phase 2 due to reduction in underground mine dewatering rates and the reduction in surface contact water runoff due to reclamation of the IWMF, there will be limited amount of both non-contact and contact water expected.

18.2.4 Water Quality and Treatment

Water quality predictions were developed using the existing geochemical characterization dataset, site monitoring data, and the site wide water management plan. The objective of the water quality predictions was to provide input for optimization of current designs for the proposed Project.

The available results of static and kinetic geochemical testing of waste rock and tailings material, and the results of water quality monitoring from the current groundwater dewatering array at the Project site were used to develop water quality source terms for the main mine facilities. The source terms and the water balance were combined in the water quality model to predict water quality composition in the WMP at key stages of mine development. The water quality modeling results were compared to the most conservative values from the Maximum Allowable Values (MAVs) of the Chalkidiki Prefecture Decision/Prefect Order 96400/85 to identify treatment goals.

Surplus contact water above the overall Project water demand will be transferred to a centralized water treatment plant (WTP) for treatment. Water treatment will occur during the warmer months. The WTP is designed for total suspended solids (TSS), metals and selenium removal, and effluent polishing. Two identical treatment process trains will be installed for a combined treatment capacity of 120 m^3/hr and would consist of conventional treatment techniques.

18.3 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 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 from the intersection of the plantsite access road, accessed by a two lane highway. From Palaiochori the major regional center 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 sea ports. 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 the 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 meters of storage and specializes in the handling of wide range of bulk cargo. The recently expanded and modernized container port is the second largest container port in Greece and has large oil and gas terminal and one of the largest passenger terminals in the Aegean Sea.

The port of Thessaloniki will serve the Project during construction for receipt of freight for 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. The Skouries to Stratoni access road will be constructed as part of the Stratoni upgrade, the resulting 23 km road will significantly reduce the haul distance and result in reduced operating costs associated with transportation costs. The road will follow existing forest roads and be upgraded to a 7 m wide bi-directional asphalt paved road suitable for heavy vehicle transport.

18.4 POWER SUPPLY

The high voltage substation is located on the northwest 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.

Hellas Gold has 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 which 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.

SECTION • 19 MARKET STUDIES AND CONTRACTS

19.1 MARKETS

The Skouries process plant will produce two saleable products: gold-copper concentrates and poured gold doré bars. Both products are expected to be marketable to a large number of upstream smelter and refiners.

The Skouries 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 permit references the future Olympias Phase III construction of a gold metallurgy plant and associated infrastructure including a port, floatation plant, flash smelter, and waste management facilities at the Olympias mine. Once completed, a portion of the Skouries copper and gold concentrates will be utilized as feed into the flash smelter, while the remaining concentrate will continue to be sold into global markets via the new port.

At the effective date of this report, the proposed Olympias Phase III developments have not been engineered to a level where processes, recovery and associated capital and operating costs can be disclosed in the economic evaluation of processing Skouries concentrates in the Olympias mill and shipping of concentrates through the new port. Further test work, design and evaluations are ongoing. Additionally a schedule for the commissioning of the Olympias mill and port facilities 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.

The gold doré bars will be sold to refineries, and transport will occur over land, or by air, by a security company capable insuring and delivering the high value doré product.

19.2 CONTRACTS

19.2.1 Construction 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. Construction contracts are being tendered and awarded to qualified contractors by the owners construction management team.

19.2.2 Mining Contracts

This technical report assumes several contracts relating to 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 operate mining as the Project matures.

In the underground there are two main contracts which have been envisioned.

19.2.2.1 Contract A

Contract A, is a development only contract where the contractor will provide all underground equipment and manpower whist Hellas Gold gold will provide, consumables, materials, designs and oversight. The contractor is assumed to charge Hellas Gold for labour rates, equipment capital, materials handling, administration fees, profit and mobilizations. These costs were built up from first principles and benchmarked against globally competitive rates. Contract A is envisioned to span the first 30 months of the Project.

19.2.2.2 Contract B

Contract B, is designed primarily to provide 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 expat and Greek trainers, Greek translators, and specialized vendor reps to achieve the training goals of the contract. Contract B is scheduled to run for 12 months.

The open pit operation is also designed to include a contractor for haulage of materials from the WTMA into the KL basin to be placed on the embankment or upstream waste area. This contractor will use small 15 t, 40 t and/or 60 t haulage trucks capable of navigating the steep terrain into the KL Valley.

19.2.3 Concentrate Sales Contracts

No off-take agreements have been signed by Eldorado nor Hellas Gold with potential concentrate off takers at the time of preparation of this technical report, however several indicative non-binding proposed term sheets have been received from European and global copper smelters. Analyses of the Skouries concentrate produced during the testwork campaigns indicate it to be generally clean and will not incur any major penalties. It was also noted that the concentrates carry a palladium credit which has not been factored into the financial evaluation. A summary of the proposed term sheets are given below based on initial term sheets with major off-takers.

19.2.4 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 26% concentrate grade.

19.2.5 Gold Terms

- Refining charge US$6 per oz of gold produced.
- Gold payable in copper concentrate 97.5%
- Gold payable in doré (from gravity circuit) 99.7%.

19.2.6 Transport Costs

- Base case is transport of concentrate by truck to European Smelters.
- Costs for transport €17 to 29/ tonne.

SECTION • 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 Chalkidiki (Macedonia Region). The Kassandra Project includes the Skouries, Olympias and Stratoni sites. The Skouries Project covers approximately 250 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 S.A., 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. Hellas Gold plans to submit an EIS modification in 2018 covering incorporation of the revised tailings management plans.

20.2 LEGISLATIVE FRAMEWORK FOR THE PREPARATION OF THE ENVIRONMENTAL IMPACT STUDY

For the preparation of the full EIS, legislation, 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.

20.3 BASELINE CONDITIONS

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 to evaluate the impact of the Project development to the ecological characteristics of the wider area. 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 three 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, Panagia and Stagira-Akanthos and current municipality of Aristotle.

The natural environment of Chalkidiki district shows a significant diversity, mainly attributed to its complex geomorphology, as further described below.

20.3.1 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 mean annual precipitation as recorded by Olympias and Stratoni stations (1997-2002) is 764.7 mm and 884.5 mm respectively, and the mean annual temperature is 15.34°C and 12.21°C. Relative humidity is a mean of 76% and evaporation is 564 mm.

Low intensity winds (up to 3 Beauforts) are dominant; mainly NW winds (frequency 36.65%) followed by SE winds (frequency 31.50%).

The area under study is classified to Seismic Zone II.

20.3.2 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.

20.3.3 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.

Sampling and analysis of soils over the wider study areas suggest natural concentrations of As and Cd exceed regulated limits. Concentrations of Ni, Sb and Zn are within limits in some areas and are outside of limits in other areas. As there has been no significant historic mining at the Skouries site, the above described situation suggests natural pollution that can be used as a baseline condition.

20.3.4 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.

20.3.5 Aquatic

The most significant watercourse is in the study area is Asprolakkas, belonging to Asprolakkas basin which encompasses the whole of Skouries study area. All watercourses in the Project area exibit a 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 (5) uncontrolled solid waste disposal sites but no regulated landfill site which could also impact water quality.

Drinking water comes from boreholes, springs and a reservoir. The larger part of the water supply network has been reconstructed during the 1990's and serves almost the entire population of the area.

20.3.6 Social and Infrastructure

20.3.6.1 Economic and Social Environment

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 Chalkidiki and Greece.

Employment - Unemployment

The study area lacks in development compared to the rest of the Chalkidiki prefecture. Unemployment rates in the region have been increasing.

20.3.6.2 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.

20.3.6.3 Sewage Disposal

The majority of the mountains' boroughs lack wastewater management plants and a substantial quantity of wastewater ends up in the rivers and streams of the study area. A waste water collection and treatment system has completed recently for Stratoni – Stratoniki – Stagira.

20.3.6.4 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).

20.3.6.5 Social Infrastructure

Chalkidiki's medical care is mainly served by Polygyros General Hospital and five (5) health centers. 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.

20.3.7 Historical and Cultural

The Ministry of Culture has performed archaeological investigations and identified two archeological sites on the Skouries Project, in the center of the open pit and in the area of the mine water storage ponds in the mill area.

20.4 ENVIRONMENTAL IMPACTS AND MITIGATION

The aim of this Project is to have a positive balance of benefits vs. 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 in a way to minimize impacts on the environment and at the same time mitigating past impacts.

20.4.1 Impact Analysis

An impact analysis has been undertaken on each environmental parameter using three basic criteria:

- **Severity:** Impacts are classified as: Negligible, Insignificant, Minor, Moderate, Major, or Highly significant.
- **Duration**: Impacts are classified as: Permanent or Temporary.
- **Reversibility:** Impacts are classified as: Reversible (Partially or fully reversible) or Irreversible.

In considering impacts the phases of the Project have been grouped as follows:

- **Development Phase.**
- **Operational Phases.**
- **Rehabilitation Phase.**

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 (EMP) in accordance with ISO 14001. Impacts and mitigation are summarized below.

20.4.2 Climate

Deforestation of the site will potentially result in a minor drop in humidity levels in the microclimate of the Project's footprint, without affecting the wider study area. Impacts are considered as negligible and fully reversible as at the completion of the Project the area will be reforested with species compatible with the bio and micro climate.

A simulation model for air contaminant dispersion was modelled for all phases of the Project on a grid that was 3 km high and covering a surface area of 22 km x 23 km. This grid covers the facilities of Skouries. The modelling shows no limits are exceeded for particulate and air pollutants the impacts from the Project on air quality will be moderate, temporary and reversible. The highest impact on air quality will occur during the development phase and Phase 1 of operations due to the open pit mining and the construction of the IWMF embankment and associated infrastructure. However even during this period predicted particulates levels are extremely low.

Mitigation measures will include integration of dust suppression systems into the designs, and watering of IWMF and roads, covered conveyor belts along their entire length and covered surface stockpiles for crushed ore storage.

Concentrates transported from the flotation plant to port will be stored in closed areas under roof and transported in sealed / covered trucks.

20.4.3 Morphology

Impacts to the surface are due to the open pit, IWMF and process plant.

- **Open Pit**
  The open pit will cover approximately 40 ha. The impacts will be major, temporary and reversible as after rehabilitation the open pit will have returned to its previous state.

- **Integrated Waste Management Facility**
  The IWMF with a total surface area of 110 ha will have a significant impact on the morphology as the KL valley will be filled with tailings and are considered major, permanent and irreversible.

- **Process Plant and Infrastructure**
  Morphological changes as a result of construction of the facilities will disturb approximately 13 ha. The impacts are considered moderate, temporary and partially reversible as some roads will remain after reclamation.

After rehabilitation the disturbed areas will be returned to a morphology resembling surrounding terrain by proper grading during reclamation. Impacts are mitigated by backfilling and shaping the reclaimed areas and then reclaiming with top soil and revegetation. The open pit, integrated waste management facility is graded to reflect the surrounding morphology. After reclamation the area will have the same structure in terms of soil and subsoil as the current surface. Reclamation of the IWMF will commence progressively early in the Project in order to accelerate the rehabilitation process.

20.4.4 Visual Impacts

Potential impacts are associated with the construction of the process facilities and IWMF. It has been determined that only the IWMF will be visually disturbing from just one location in the village of Stagira 5.6 km to the north of the Project area and only during the operational phase. Due to distance from the village of Stagira and low horizontal visual coverage, the impacts are minor, temporary and fully reversible.

Efforts will be made to ensure that the paintwork on the outer surfaces of new facilities blends into the environment by using paint colors on the structures and planting trees endemic to the area around the edge of the facilities to ensure that it is visually isolated as much as possible during the operational phase.

Suitable lighting will be used, to minimize the distance from which any buildings and facilities are visible at night. The use of strong outdoor diffused lighting which does not promote safety or meet the operational needs of the Project and its facilities will be avoided.

20.4.5 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 predicted to affect the wider area. None of the predicted values exceeds regulated noise limit values.

Noise levels in the immediate Project area will be affected during development and the Phase 1 of operations due to the open pit mining activities, but it will be of limited duration and will not exceed regulated limits.

Based on the study the cumulative acoustic impacts are classified as minor, temporary and reversible.

From a vibrations point of view, in all cases, the maximum ground vibration velocities have been determined to be within acceptable limits.

Monitoring programs of noise and vibration will be undertaken thru all phases of the Project.

20.4.6 Soil

Impacts on soil are expected to be minor, partially reversible by taking rehabilitation measures but permanent.

The top soil from stripping of the open pit, IWMF process facility and infrastructure will be temporarily stored in deposition areas specified in the approved EIS in order to be used for later rehabilitation.

20.4.7 Flora and Fauna

Impacts of the flora in the Project footprint are judged to be major during development and operational phases but proper reclamation will allow the areas flora to recover over the long term. The potential regeneration of the forest ecosystems is considered to be very good after any anthropogenic pressure which will greatly assist with rehabilitation. This process is accelerated by using progressive reclamation as areas are decommissioned. The Company is developing a biodiversity management plan in order to monitor and assist in these natural processes.

During the development and operational phase wild animals will be disturbed and will move away, as will birds that nest in dense vegetation. The impacts on fauna are considered major however reversible as rehabilitation will allow these areas to be repopulated by fauna endemic to the wider area.

20.4.8 Aquatic

The cumulative impacts on the aquatic environment at and around the Project will be minor, permanent and partially reversible.

20.4.8.1 Surface Water

The main impacts on surface water will be rainfall that falls on developed areas and is considered as contact water. Surface contact water runoff from developed areas will be collected and used in the process facility, no discharge of contact water to the surrounding environment is realized. The impacts on the local river system are expected to be minor, since no significant ecological quality or hydromorphology characteristics will change.

The impacts on surface water in Skouries will be moderate however, be subject to remediation measures. After the rehabilitation phase surface water will again contribute as it currently does downstream.

An important part of management of water at the Skouries site will be collection and diversion of non-contact water from the watershed surrounding the Project to minimize the surface contact water. Ditches will be constructed around the facility in order to divert the non-contact water around the site and deliver it downstream.

Hellas Gold runs an extensive monitoring program of surface, ground and marine water. Monitoring will continue through the mine life and post closure. Surface contact water is reduced in Phase 2 of the Project as the IWMF is progressively reclaimed. Other areas are reclaimed as soon as they are decommissioned.

The impacts on coastal water, associated with the discharge of Asprolakkas stream will be negligible during the development, operational and rehabilitation phases.

20.4.8.2 Groundwater

During the development and operational phase groundwater is pumped from the open pit perimeter dewatering wells and horizontal boreholes in the underground mine. The dewatering wells and boreholes are in place in order to lower the water table around the mine. The advantage of the dewatering is that water will not come in contact with the ore body and it will therefore remain as non-contact water, there will be no treatment needed before disposal. Non-contact water will be either used to supplement process water makeup or reinjected in wells outside of the dewatering cone in the Lotsaniko basin aquifer to the through a number of injection boreholes. Emergency water storage has been provided for the wet season.

The construction of the IWMF will not affect groundwater. Tailings have zero acid forming capacity and leach tests for all elements gave results complying with regulations for disposal in inert material landfills. The impacts to the groundwater during operational phases are negative and minor.

During the rehabilitation phase, pumping to lower the water table will cease and groundwater will return to its natural levels. The impacts of this phase will be positive and permanent.

20.4.9 Social and Infrastructure

20.4.9.1 Social

The Project will have positive impacts on primary, secondary and tertiary production due to the effects the significant increase of employment will have to the average salary and consequently demand of commodities. The impacts will be positive, major and irreversible and due to the Project's timeline considered permanent.

20.4.9.2 Land Use

The Project will change the land use from natural uses to semi-natural/artificial uses during the operational phase. Therefore, the impacts are considered to be negative, significant but partially reversible.

20.4.10 Historical and Cultural

No historical or cultural monuments will be occupied by the Project, the impacts are negligible temporary and fully reversible. The site of archaeological interest located in the open pit is to be moved in an area that will be decided by the Central Archaeological Committee while the respective site in the mill area will not be impacted due to the redesign and relocation of the mill facilities.

During development and operational phases potential impacts to historic and cultural sites will be mitigated by having all excavation works monitored by independent archaeologists and in case of archaeological findings, excavations will be suspended and archaeological study initiated.

20.5 PUBLIC CONSULTATION AND DISCLOSURE

20.5.1 Stakeholder Engagement Plan

20.5.1.1 Social Requirements

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 to do so. 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 the Company as an important tool for transparency and effective risk management.

20.5.1.2 Closure and Reclamation

The fundamental criteria for Project closure and environmental rehabilitation include the following:

- The Project site must be handed back in a state which 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 which 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.
- Integrated waste management facility.
- 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 layer comprised of inert waste materials from operation of the deposit, 2.5 m thick, and a layer of top soil 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.

Top soil collected during the development stage of the Project will be stockpiled separately, protected and preserved for the life of mine for use in rehabilitation. This top soil 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 IWMF 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 favor the spread and carrying of dust over large distances.

20.5.1.3 Open Pit Mine

Once surface extraction of the deposit is completed at the end of Phase 1, which indicates that activities will thereafter focus 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.

20.5.1.4 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 which are merchantable will be cleaned and stored at predetermined storage areas. Items that are not merchantable will be sent for recycling.

20.5.1.5 Integrated Waste Management Facility

Prior to completion of Phase 1 progressive reclamation of the IWMF will begin by capping and grading areas as the filtered tailings 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 backfill or sold. All remaining structures and equipment will be removed except for equipment necessary for water management during the reclamation period.

20.5.1.6 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.

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 road side slopes will be restored.

20.5.2 Environmental Costs and Guarantees

Hellas Gold has provided the Mining & Industrial Minerals Directorate of the Ministry of the Environment, with a letter of guarantee for € 50 million in favor of the Greek State as an assurance that the funds necessary for rehabilitation projects will be available.

The letter of guarantee shall remain valid for such period as the terms and conditions of this decision are valid plus an additional 5 years while the level of the guarantee is adjusted every 5 years to match inflation and to reflect the progress of rehabilitation works. 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 referred to in env. term d1.39 which will make a recommendation on the adjustment to the letter of guarantee which is needed.

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

SECTION • 21 CAPITAL AND OPERATING COST

The currency exchange rates used are as per Q4 2017 market conditions. All costs are presented in US Dollars (US$) based on the exchange rates shown in Table 21-1.

Table 21-1: Exchange Rates

Currency Code	Currency Name	Exchange Rate
US$	United States Dollar	US$1.00 = US$1.00
CAN$	Canadian Dollar	CAN$1.00 = US$0.75
€	Euro	€1.00 = US$1.15

21.1 CAPITAL COSTS

The total Project capital cost includes the cost to complete the Project construction until commercial production of the mill, and then followed by sustaining capital costs spread out over the remaining 23 years of the mine life. Capital costs are summarized in Table 21-2. Sunk costs are not included in the capital costs.

Table 21-2: Capital Cost Summary

Area	Initial (US$ x 1,000)	Sustaining [1] (US$ x 1,000)
A - Overall Site	14,508	0
B - Open Pit Mine	66,694	22,975
B - Underground Mine	144,019	405,352
C - Stockpile and Materials Handling	11,801	596
D - Process Plant	134,833	41,400
E - Underground Backfill Plant	0	27,619
F - Integrated Waste Management Facility (IWMF)	22,446	21,948
G - In Pit Tailings	0	40,546
I - Water Management	15,248	3,486
H - Infrastructure	49,288	5,993
J - Ancillary Facilities	9,001	2,146
K - Off Site Infrastructure	4,541	0
P - Environmental	0	2,708
Direct	**472,379**	**574,769**
Indirects	**99,563**	**59,661**
Owners Cost	**30,340**	**0**
Contingency	**86,892**	**123,587**
Total Installed Cost	**689,174**	**758,016**

Note: [1] LOM mining and ore processing.

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 prefeasibility-level estimate categorized as AACE Class 4.

Direct costs were developed from a combination of firm quotes, 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 overall Project contingencies for initial and sustaining cost estimates are 14% and 19% respectively.

Construction has substantially progressed for the plant site earthworks, process plant, site infrastructure, and ancillary facilities. Where construction has progressed, the capital cost estimate details included for the "estimate to complete" forecast as of end of Q3 2017.

The capital cost estimate does not include for sunk costs and expresses capital costs from beginning of 2018. As a result, credit was subtracted for the value earned during Q4 2017.

21.1.1 Basis of Estimate

21.1.1.1 Accuracy

An analysis was conducted to confirm and support the initial capital cost accuracy statement. This was accomplished by determining the weighted average deviation of the overall estimate by grouping the costs into the AACE classifications as shown in Table 21-3.

Table 21-3: Initial Capital Cost Estimate Accuracy Analysis

AACE Classification	Total Initial Capital (US$ x 1,000)	Lower Limit [1]		Upper Limit [1]	
		(%)	(US$ x 1,000)	(%)	(US$ x 1,000)
Class 1	0	-5	0	10	0
Class 2	369,627	-10	332,664	15	425,071
Class 3	183,774	-15	156,208	20	220,529
Class 4	134,629	-20	107,703	30	175,017
Class 5	1,145	-30	801	50	1,717
Total	**689,174**	**-13**	**597,376**	**19**	**822,334**

Note: [1] As defined by AACE, 50% confidence interval with contingency included.

The capital cost estimate for the plant site earthworks, process plant, site infrastructure, and ancillary facilities were costed based on quantities derived from detailed engineering and committed contract rates. These areas were costed to an AACE Class 2 level of accuracy.

The pre-production segment of the underground mine was engineered to a feasibility study level and costed in accordance with AACE Class 3 requirements. The procurement of mobile equipment were classified as Class 2 as sufficient data is available to verify the confidence range for these items.

Tailings filtration, IWMF and water management areas were completed to a prefeasibility study level of detail.

21.1.1.2 Labour

Labour rates were derived from existing contracts for the Skouries Project and benchmarked against contracts from the Olympias Phase II Project. The all-in crew labour rates include all direct and indirects costs associated with the contractors.

A labour productivity factor is used to account for overall labour force efficiency. Non-productive events are estimated based on expected construction conditions. The overall productivity factor was calculated to be 1.67 based on expected non-productive time and local installation productivities.

21.1.1.3 Commodity Pricing

In general, direct unit costs were based on contract rates, committed prices to complete, quotations from contractors, and quotations from suppliers. For some minor items, allowances were carried based on historical data. Table 21-4 outlines the primary source for unit costs, by major commodity.

Table 21-4: Primary Source for Unit Costs

Commodity	Primary Source
Mobile Fleet Equipment	Major Equipment – Vendor Quotations Minor Equipment – Vendor Quotation or Consultant Historical Data
Process & Ancillary Equipment	Major Equipment – Vendor Quotations Minor Equipment – Vendor Quotation or Consultant Historical Committed Costs to Go
Earthworks	Contract Rates
Concrete	Contract Rates, Local/European Supplier Quotations
Steel	Contract Rates, Local/European Supplier Quotations
Platework	Local/European Supplier Quotations
Piping	Local/European Supplier Quotations
Electrical & Instrumentation	Major Equipment – Vendor Quotations Bulk Materials – Local/European Supplier Quotations Committed Costs to Go

21.1.1.4 Material Quantities

Quantities were generally based on detailed material take-offs and equipment lists, with some allowances for minor items. Table 21-5 summarizes the primary source of quantities by major area.

Table 21-5: Primary Source of Quantities

Area	Primary Source
A – Overall Site	Material Take-offs Balance of committed contracts
B – Underground Mine	Detailed Equipment List Detailed Mobile Equipment Schedule Material Take-offs Mine Plan/Schedule, Manpower Plan
B – Open Pit Mine	Detailed Mobile Equipment Schedule Mine Plan Manpower Plan
C – Crushing Plant	Detailed Equipment List Material Take-offs
D – Process Plant	Detailed Equipment List Material Take-offs Pre-feasibility level design of Filter Plant
E – Backfill Plant	Paste Plant Quotation by Vendor Lump-Sum Cost Estimate (Paste Plant Infrastructure)
F – Integrated Waste Management Facility (IWMF)	Material Take-offs
G – In-pit Tailings	Material Take-offs
H – Infrastructure	Detailed Equipment List Material Take-offs Lump-sum Cost Estimate (Shotcrete Plant) Lump-sum Cost Estimate (Surface Fuel Delivery)
J – Ancillaries	Detailed Equipment List Material Take-offs
K – Offsite Infrastructure	Lump-Sum Cost Estimate (Overhead Power Line)

21.1.1.5 Indirect Cost Estimate

Indirect costs were calculated based on a combination of remaining cost to complete of existing commitments, factored from direct costs, and based on historical experience. Table 21-6 summarizes the basis of indirect costs.

Table 21-6: Basis of Indirect Costs

Area	Primary Source
Q – Construction Indirects	Calculated allowances for temporary construction facilities, temporary construction services, site logistics, construction support equipment and tools. Some allowances derived from latest monthly costs at the Skouries site.
	Freight and logistics costs were calculated as a percentage of equipment and material supply cost, and divided into overseas and European freight.
	Vendor representatives' costs were calculated based on number of days required and daily rates.
	Miscellaneous refurbishment costs were included in the indirects for equipment and ongoing site maintenance.
R – 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 in accordance with the Process Design Criteria, with unit costs referenced primarily from recent quotations.
S – EPCM	EPCM costs were calculated based on balance of committed contracts, and EPCM estimates as provided by consultants.

21.1.1.6 Owner's Costs

Owner's costs included labour and general and administrative costs for the owners team during the period of active construction applied over a specified period of time. An allowance for insurance, travel, land acquisition, and archeological discoveries was also carried.

21.1.1.7 Contingency

Total contingency allocation was applied to individual line items, based on the overall subjective level of engineering and definition. The overall Project contingencies for initial investment and sustaining capital cost estimate are 14%, and 19% respectively.

Contingency for the initial investment capital cost estimate is summarized by category in Table 21-7.

Table 21-7: Initial Capital Contingency Classification

Category	Total Without Contingency (US$ x1,000)	% of Total Value	Total With Contingency (US$x1,000)	Contingency (%)
Allowance (50%)	763	0	1,145	50
Scoping Study (30%)	0	0	0	30
Preliminary/PFS (25%)	107,703	17	134,629	25
PFS/FS (20%)	0	0	0	30
Reference/FS (15%)	159,804	26	183,774	15
Fixed Quote/60% Eng (10%)	356,137	57	391,751	10
Q4 2017 Sunk Cost Credit	-22,124	0	-22,124	0
Total	**602,283**	**100**	**689,174**	**14**

21.2 OPERATING COSTS

The operating cost estimate was assessed to be within a prefeasibility accuracy level (AACE Class 4). Operating costs include allocations for the following.

- Open pit mining.
- Underground mining.
- Stockpile rehandling.
- Processing.
- Tailings filtration, materials handling, and stacking.
- Water management.
- General & administration.

Operating costs were calculated for each year of operation, totalling US$3.34 B for an average of US$21.32/t ore milled over life-of-mine, summarized in Table 21-8.

Open pit mining costs 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 manufacturers' data and consultant's in-house data.

The underground mine operating costs 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 21-8: Operating Costs

Category	LOM Average (US$/t ore)	LOM Expenditure (US$ x 1,000)
Open Pit Mining (US$/t of OP ore)	4.51	238,876
Underground Mining (US$/t of UG ore)	16.50	1,602,340
Total Mining (US$ /t of LOM ore)	**11.75**	**1,841,216**
Stockpile Rehandling	0.06	9,818
Processing Cost	6.73	1,055,007
Filter Plant	0.76	119,067
IWMF and Water Management	0.62	96,788
G&A	1.39	218,317
Operating Cost	**21.32**	**3,340,213**

Process operating costs were based on estimated annual consumption of process reagents, major wear parts, and utilities. Budget quotations were obtained for supply of all significant consumables and utilities. Power consumption was calculated based on electrical load lists.

Operating costs for the IWMFs were calculated based on operation and maintenance requirements of pumps, pipelines, and mobile equipment. Labour requirements for operation of mobile equipment and supervision was calculated based on fleet sizing. Water management costs were calculated based on operations and maintenance requirements for pumps, pipelines, and the water treatment plant. Power costs were calculated based on the operating power of these facilities.

General and administrative costs were estimated based on a projected personnel list with salaries indicative of local standards, and annual allowances for general supplies.

21.2.1 Basis of Estimate

21.2.1.1 Cost Basis

Underground mine operating costs were based on the production schedule, equipment operating costs from manufacturers, quotations on bulk from Greek primarily suppliers, and staffing plan.

The open pit and process operating costs were based on consumption inputs from the design criteria, load lists, updated quotations of consumables, and staffing plan.

General and Administrative (G&A) costs were estimated based on information from other Eldorado active operations in Greece. Kassandra costs were based on the 2018 Kassandra allocation budget. Mining, process, and G&A costs can be further categorized within salaries & wages, energy, consumables and repairs and maintenance. Table 21-9 provides a further breakdown of the operating cost.

Salaries and wages were provided as full-burden rates by Hellas Gold, and based on detailed personnel lists developed for each operating department. Energy costs for diesel and power supply were based on supplier quotations. Consumption was estimated dependant on operating conditions and detailed electrical load lists. Consumables and maintenance costs have been obtained from supplier quotations, relevant in-house data, or were factored.

Table 21-9: Operating Cost Breakdown

Category	US$/t ore
Open Pit Mining (US$/t of OP ore)	**4.51**
Salaries and Wages (Mechanics not included)	1.20
Power & Diesel	1.43
Mine Operating and Consumables	1.88
Underground Mining (US$/t of UG ore)	**16.50**
Salaries and Wages	5.00
Power & Diesel	2.89
Mine Operating and Consumables	8.60
Total Mining (US$/t of LOM ore)	**11.75**
Stockpile Rehandling	**0.06**
Power & Diesel	0.06
Process	**6.73**
Salaries and Wages	0.80
Power	2.89
Consumables and R&M	3.05
Tailings Filtration	**0.76**
Salaries and Wages	0.14
Power	0.19
Consumables and R&M	0.43
IWMF and Water Management	**0.62**
Salaries and Wages	0.22
Power	0.17
Consumables and R&M	0.23
G&A	**1.39**
G&A Mill - Salaries and Wages	0.41
Kassandra Allocation	0.99
Operating Cost	**21.32**

SECTION • 22 ECONOMIC ANALYSIS

22.1 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$1,300/oz Au and US$2.75/lb Cu (see Section 2). The economic model was also evaluated at the mineral reserve prices of US$1,200/oz and US$2.50/lb, The model makes use of a first principles build up in Euros and then converts to US$. All reporting in this section is made in US$.

The model has been prepared on a yearly life of mine basis. The effective date of the estimate is from the re-start of construction activities once the permitting issues as outlined in Section 24 have been resolved. The remaining construction period is estimated to be 2.25 years including commissioning, and the first year of commercial production in designated as Year 1. The LOM is 23 years from the start of commercial production until the depletion of economic mineral reserves.

22.2 PRODUCTION SCHEDULE

The production schedule is shown in Figure 22-1. The Project is configured in two phases with the distinction being the location of the tailings deposition. Phase 1 considers deposition into the surface IWMF, in Phase 2 the tailings will be deposited into the mined out open pit.

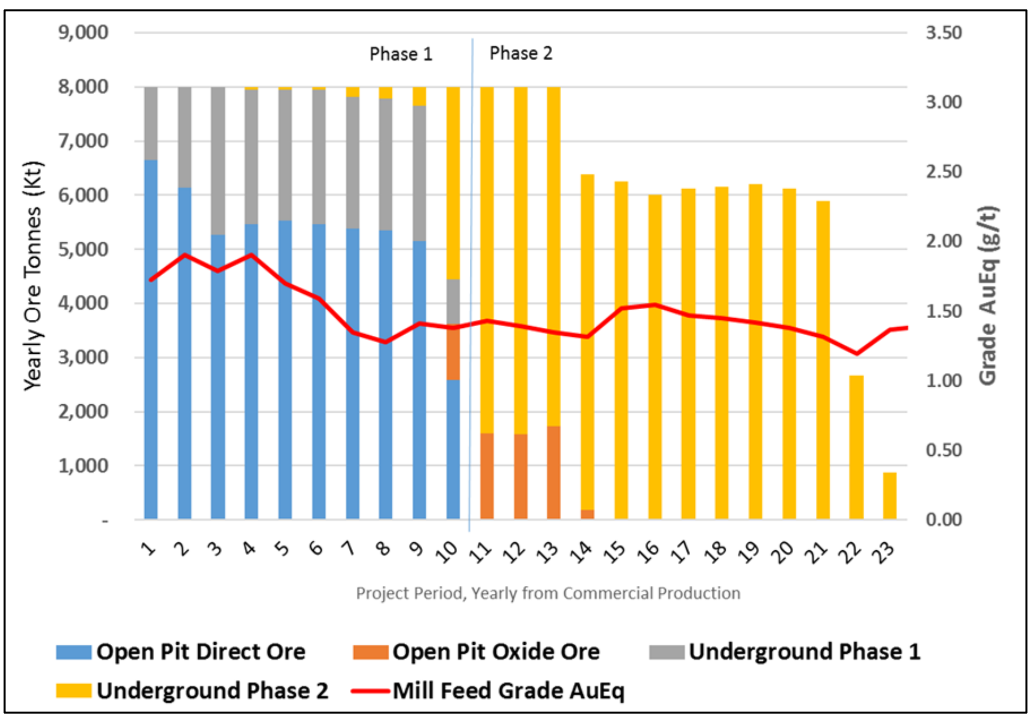


Figure 22-1: Skouries LOM Mill Feed and Grade

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 6.0-6.4 Mtpa for a further 8 years, as the capacity of the stand-alone Phase 2 underground mine becomes the limiting factor. Finally, in the last two years of the operation the throughput is again reduced to a rate of 2-3 Mtpa as the underground mining sequence proceeds to the extraction of the final remnants of the Skouries ore.

22.3 CASH FLOWS

The annual cash flow forecast is built from a first principles financial model, the model results are shown in Table 22-1 through to Table 22-10 which include the build details of the production schedule, the operating and capital costs, the mill production and offsite costs and the net cashflows over the project life and reported by Phase. Yearly numbers may not add to LOM totals due to rounding.

The after-tax cash flow analysis in Table 22-10 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 21.2% on an after tax basis is achieved with the project case metal prices of US$1,300/oz Au and US$2.75/lb Cu. Using those metal prices, the net present value (NPV) of the Project is estimated to be US$925 M using a discount rate of 5%, with a payback of the remaining capital achieved in 3.4 years from the start of commercial production.

Table 22-1: Skouries Phase 1 Production Schedule

SKOURIES PRODUCTION SCHEDULE	Units	LOM Totals	Phase 1 Totals	Project Year and Project Phase												
				-3	-2	-1	1	2	3	4	5	6	7	8	9	10
				Pre-Production			Phase 1									Transition
OPEN PIT																
Direct Ore Mined	**t x 1000**	**53,987**	**53,987**			**1,000**	**6,652**	**6,146**	**5,264**	**5,465**	**5,538**	**5,463**	**5,379**	**5,348**	**5,149**	**2,583**
Au Grade	g/t	0.61	0.61			1.19	1.06	0.74	0.67	0.58	0.49	0.30	0.35	0.46	0.50	0.84
Cu Grade	%	0.42%	0.42%			0.67%	0.62%	0.48%	0.42%	0.39%	0.35%	0.28%	0.31%	0.37%	0.38%	0.53%
Gold Contained	oz x 1000	1,059	1,059			38	227	147	113	103	86	52	61	80	82	70
Cu Contained	lb x 10^6	496	496			15	91	65	49	47	43	34	37	43	43	30
% Oxide	%	11%	11%			77%	69%	12%	1%	0%	0%	0%	0%	0%	0%	0%
Oxide Ore Stockpile	**t x 1000**	**5,482**	**5,482**			**510**	**2,470**	**935**	**1,400**							
Au Grade	g/t	0.12	0.12			0.13	0.12	0.11	0.12							
Cu Grade	%	0.17%	0.17%			0.18%	0.17%	0.16%	0.17%							
Gold Contained	oz x 1000	21	21			2	9	3	5							
Cu Contained	lb x 10^6	20	20			2	9	3	5							
% Oxide	%	51%	51%			78%	78%	21%	10%							
Total Open Pit Waste	**t x 1000**	**52,900**	**52,900**	**-**	**2,800**	**6,240**	**5,864**	**10,634**	**11,300**	**4,950**	**4,662**	**1,970**	**700**	**520**	**722**	
Red Clay Waste	t x 1000	4,585	4,585	-	1,028	1,181	570	1,251	554	-	-	-	-	-	-	
Overburden Waste	t x 1000	7,648	7,648	-	1,071	2,198	1,493	1,083	1,803	-	-	-	-	-	-	
Weak Rock Waste	t x 1000	15,385	15,385	-	643	2,359	3,450	3,866	4,982	85	-	-	-	-	-	
Hard Rock Waste	t x 1000	25,283	25,283	-	58	503	351	4,434	3,960	4,865	4,662	1,970	700	520	722	
Total Mined in Open Pit					**2,967**	**7,750**	**14,986**	**17,715**	**17,964**	**10,415**	**10,200**	**7,433**	**6,079**	**5,868**	**5,872**	**2,583**
UNDERGROUND PHASE 1																
Ore Mined	**t x 1000**	**22,163**	**22,163**		**314**	**750**	**936**	**1,854**	**2,736**	**2,490**	**2,417**	**2,479**	**2,431**	**2,435**	**2,502**	**820**
Au Grade	g/t	1.39	1.39		0.46	0.55	1.63	1.62	1.69	1.61	1.65	1.51	1.24	1.25	1.05	0.97
Cu Grade	%	0.73%	0.73%		0.34%	0.40%	0.86%	0.82%	0.86%	0.81%	0.82%	0.77%	0.66%	0.66%	0.59%	0.54%
Gold Contained	oz x 1000	993	993		5	13	49	96	148	129	128	121	97	98	85	26
Cu Contained	lb x 10^6	356	356		2	7	18	34	52	44	44	42	35	36	32	10
% Oxide	%	0%	0%		0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Waste Mined	t x 1000	821	821		268	281	190	3	1	-	-	-	-	-	-	-
Total Mined	t x 1000	22,984	22,984		582	1,031	1,126	1,857	2,737	2,490	2,417	2,479	2,431	2,435	2,502	820
UNDERGROUND PHASE 2																
Ore Mined	**t x 1000**	**76,035**	**4,457**							**45**	**46**	**58**	**191**	**217**	**348**	**3,553**
Au Grade	g/t	0.68	0.73							0.37	0.20	0.25	0.36	0.43	0.68	0.79
Cu Grade	%	0.50%	0.47%							0.36%	0.24%	0.30%	0.39%	0.40%	0.51%	0.48%
Gold Contained	oz x 1000	1,671	104							1	0	0	2	3	8	90
Cu Contained	lb x 10^6	836	46							0	0	0	2	2	4	38
% Oxide	%	0%	0%							0%	0%	0%	0%	0%	0%	0%
Total Waste Tonnes	t x 1000	1,865	1,320							100	215	183	194	346	270	12
Total Tonnes	t x 1000	77,900	5,777							144	261	241	385	563	618	3,565
OXIDE ORE STOCKPILE																
OOS balance beginning of period	t x 1000					480	1,741	3,798	4,734	6,133	6,133	6,133	6,133	6,133	6,133	6,133
Au grade beginning of period	g/t					0.36	0.38	0.16	0.15	0.14	0.14	0.14	0.14	0.14	0.14	0.14
Cu grade beginning of period	%					0.29%	0.31%	0.19%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%
%Oxide	%					35%	32%	65%	56%	46%	46%	46%	46%	46%	46%	46%
Additions to OOS during Period	t x 1000				480	1,261	2,470	935	1,400							
Au grade additions	g/t				0.36	0.38	0.12	0.11	0.12							
Cu grade additions	%				0.29%	0.31%	0.17%	0.16%	0.17%							
% Oxide	%				35%	31%	78%	21%	10%							
Subtraction from OOS during Period	t x 1000						412									1,044
Au grade subtraction	g/t						0.85									0.14
Cu grade subtraction	%						0.56%									0.18%
% Oxide	%						0%									46%
TOTAL MILL FEED																
Open Pit Direct Ore	t x 1000	53,987				1,000	6,652	6,146	5,264	5,465	5,538	5,463	5,379	5,348	5,149	2,583
Open Pit Oxide Ore	t x 1000	6,133														1,044
Underground Phase 1	t x 1000	21,512					1,348	1,854	2,736	2,490	2,417	2,479	2,431	2,435	2,502	820
Underground Phase 2	t x 1000	76,035								45	46	58	191	217	348	3,553
Ore Milled	**t x 1000**	**157,666**	**81,000**			**1,000**	**8,000**	**8,000**	**8,000**	**8,000**	**8,000**	**8,000**	**8,000**	**8,000**	**8,000**	**8,000**
Au Grade	g/t	0.74	0.83			1.19	1.13	0.96	1.02	0.90	0.83	0.67	0.62	0.70	0.68	0.73
Cu Grade	%	0.49%	0.50%			0.67%	0.64%	0.56%	0.57%	0.52%	0.49%	0.44%	0.42%	0.46%	0.45%	0.46%
Mill Feed Grade AuEq	**gEq/t**	**1.50**	**1.59**			**1.87**	**1.88**	**1.80**	**1.92**	**1.72**	**1.61**	**1.36**	**1.29**	**1.43**	**1.39**	**1.45**
Gold Contained	oz x 1000	3,774	2,161			38	292	247	261	232	215	173	160	180	174	189
Cu Contained	lb x 10^6	1,717	898			15	113	99	101	91	87	77	74	81	79	81
Gold Equivelent Contained	**oz x 1000**	**7,596**	**4,130**			**60**	**483**	**462**	**493**	**441**	**413**	**350**	**331**	**367**	**358**	**373**
% Oxide	%	5.7%	8.2%			77%	57%	9%	1%	0%	0%	0%	0%	0%	0%	6%

Table 22-2: Skouries Phase 1 Operating Cost Schedule

SKOURIES ANNUAL OPERATING COST	Units	LOM Totals	Phase 1 Totals	Project Year and Project Phase												
				-3	-2	-1	1	2	3	4	5	6	7	8	9	10
				Pre-Production			Phase 1									Transition
OPEN PIT MINING																
Drill and Blast	mUSD	15.1	15.1				1.7	2.5	2.4	1.9	1.8	1.2	1.0	0.9	1.0	0.8
Mine Rock Load and Haul	mUSD	153.2	153.2				23.3	25.8	25.6	17.3	17.2	12.2	8.2	7.7	8.0	7.9
Contractors Profit Costs	mUSD	10.8	10.8				1.7	2.4	2.4	1.2	1.2	0.5	—	0.2	0.3	1.0
Overhead	mUSD	38.7	38.7				4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	2.6	2.3
Operations support	mUSD	21.1	21.1				2.0	2.2	2.2	2.1	2.1	2.1	2.1	2.1	2.1	2.1
UNDERGROUND MINING																
Ore Cross-cut	mUSD	129.7	74.2				18.9	14.3	9.3	3.7	0.5	0.3	1.8	4.8	8.6	12.1
Development through PF	mUSD	25.0	2.5				—	0.4	0.2	0.4	0.6	0.3	0.1	0.3	—	0.0
SLOS 20m	mUSD	165.5	31.8				2.0	—	1.2	2.8	2.1	5.3	3.8	2.5	4.4	7.6
SLOS 30m	mUSD	324.0	118.1				3.5	10.3	14.7	12.9	13.7	10.9	12.1	13.4	11.7	15.0
Backfill	mUSD	327.2	88.6				2.5	7.9	9.3	9.0	9.0	9.3	9.1	9.1	9.2	14.3
Mine General (Total)	mUSD	461.3	156.2				16.2	13.4	13.8	13.8	14.0	14.0	14.1	15.5	17.7	23.8
Mine Maintenance (Total)	mUSD	169.6	48.3				3.7	4.2	4.6	4.6	4.6	4.6	4.6	4.6	4.6	8.0
STOCKPILE REHANDLE																
Oxide Ore Stockpile	mUSD	9.8	2.2				0.6	—	—	—	—	—	—	—	—	1.6
PROCESSING COSTS																
8.0 mpta Rate	mUSD	691.6	532.0				53.2	53.2	53.2	53.2	53.2	53.2	53.2	53.2	53.2	53.2
6.0 mtpa Rate	mUSD	363.4	0.0				—	—	—	—	—	—	—	—	—	—
FILTER PLANT COSTS																
Filter Plant Opex	mUSD	119.1	60.8				6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1
TAILINGS HANDLING																
IWMF																
Basin Drain Opex	mUSD	0.6	0.4				0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
KL Embankment Opex	mUSD	3.1	3.0				0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3
Tailings Handling	mUSD	40.6	40.5				4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0
IN_PIT TMF																
Open Pit Drainage and Reclaim System	mUSD	2.3	0.0													0.0
Tailings Handling	mUSD	42.3	0.3													0.3
WATER MANAGEMENT																
Site Pumping Costs	mUSD	4.0	4.0				0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
Water Treatment Plant	mUSD	3.9	3.9				0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
GENERAL+ADMIN							—	—	—	—	—	—	—	—	—	—
G+A Kassandra Allocation	mUSD	154.9	69.6				7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0
G+A Skouries Mill Admin	mUSD	63.5	28.5				2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9
TOTAL OPERATING COSTS	**mUSD**	**3340.21**	**1503.95**				**154.65**	**161.82**	**164.04**	**148.20**	**145.31**	**139.27**	**135.43**	**139.79**	**144.37**	**171.08**
	USD/ t ore	**21.32**	**18.57**				**19.33**	**20.23**	**20.50**	**18.52**	**18.16**	**17.41**	**16.93**	**17.47**	**18.05**	**21.38**

Table 22-3: Skouries Phase 1 Capital Cost Schedule

SKOURIES ANNUAL CAPITAL COSTS	Units	LOM Totals	Phase 1 Totals	Project Year and Project Phase												
				-3	-2	-1	1	2	3	4	5	6	7	8	9	10
				Pre-Production			Phase 1									Transition
DEVELOPMENT CAPITAL (Pre-production)																
Overall Site	mUSD	19.6	19.6	—	9.8	9.8										
Open Pit Mine	mUSD	67.5	67.5	1.7	21.2	44.5										
Underground Mine	mUSD	144.0	144.0	8.5	44.7	90.8										
Stockpile and Materials Handling	mUSD	11.9	11.9	—	6.0	6.0										
Process Plant	mUSD	68.5	68.5	—	34.3	34.3										
Filter Plant	mUSD	68.9	68.9	—	34.5	34.5										
Integrated Waste Management Facility (IWM	mUSD	24.3	24.3	2.2	5.8	16.3										
Water Management	mUSD	17.2	17.2	1.4	10.5	5.2										
Infrastructure	mUSD	50.6	50.6	3.7	23.8	23.1										
Ancillary Facilities	mUSD	9.2	9.2	—	4.6	4.6										
Off Site Infrastructure	mUSD	6.1	6.1	—	3.0	3.0										
Indirects	mUSD	60.6	60.6	1.8	26.6	32.2										
Spares/Warehouse Fills	mUSD	15.1	15.1	0.4	6.8	7.9										
EPCM	mUSD	27.1	27.1	0.7	12.2	14.2										
Owners Cost	mUSD	33.8	33.8	—	16.9	16.9										
Contingency	mUSD	86.9	86.9	3.4	35.2	48.3										
Commissioning Revenue minus Cost	mUSD	-42.0	-42.0	—	—	-42.0										
SUSTAINING CAPITAL COSTS																
Open Pit Mine	mUSD	23.0	23.0				13.1	1.7	1.9	1.6	1.9	1.0	0.7	0.6	0.4	—
Underground Mine	mUSD	405.4	275.2				59.1	8.1	3.3	6.0	20.5	21.7	36.9	48.4	40.1	31.0
Stockpile and Materials Handling	mUSD	0.6	0.6				—	—	—	—	—	—	—	—	0.6	—
Process Plant	mUSD	41.4	9.0				0.5	0.6	0.9	0.9	0.8	0.9	0.9	0.9	1.0	1.7
Backfill Plant	mUSD	27.6	27.6				12.1	3.2	—	—	—	—	—	—	12.3	—
Integrated Waste Management Facility (IWM	mUSD	21.9	21.8				4.5	4.0	2.7	3.4	1.6	1.5	1.5	2.4	0.1	0.1
In Pit Tailings	mUSD	40.5	11.4				—	—	—	—	—	—	—	—	0.7	10.7
Water Management	mUSD	3.5	3.5				0.1	—	1.3	—	1.2	—	—	1.0	—	—
Infrastructure	mUSD	6.0	6.0				6.0	—	—	—	—	—	—	—	—	—
Ancillary Facilities	mUSD	2.1	2.1				2.1	—	—	—	—	—	—	—	—	—
Environmental	mUSD	2.7	2.7				—	1.2	1.5	—	—	—	0.0	—	—	—
Indirects	mUSD	23.5	15.5				5.6	1.2	1.0	1.0	0.7	0.5	0.5	1.2	1.2	2.6
Spares/Warehouse Fills	mUSD	12.6	9.3				3.1	0.5	0.2	0.1	0.4	0.5	0.8	1.1	1.2	1.4
EPCM	mUSD	23.5	20.6				4.1	2.8	2.2	2.0	1.9	1.0	1.0	1.4	2.0	2.2
Contingency	mUSD	123.6	84.5				20.7	4.4	3.3	3.4	6.0	5.4	8.6	11.4	11.3	9.9
TOTAL CAPITAL	**mUSD**	**1405.2**	**1159.9**	**23.8**	**295.9**	**327.5**	**130.9**	**27.8**	**18.4**	**18.3**	**35.1**	**32.3**	**50.9**	**68.4**	**71.0**	**59.6**

Table 22-4: Skouries Phase 1 Metal Production and Off-site Cost Schedule

SKOURIES METALS PRODUCED AND OFFSITE COSTS	Units	LOM Totals	Phase 1 Totals	Project Year and Project Phase												
				-3	-2	-1	1	2	3	4	5	6	7	8	9	10
				Pre-Production			Phase 1									Transition
MILLING SCHEDULE																
Ore Milled	**$t \times 10^6$**	**157.67**	**81.00**			**1.00**	**8.00**	**8.00**	**8.00**	**8.00**	**8.00**	**8.00**	**8.00**	**8.00**	**8.00**	**8.00**
Grade Au	g/t	0.74	0.83			1.19	1.13	0.96	1.02	0.90	0.83	0.67	0.62	0.70	0.68	0.73
Gracde Cu	%	0.49%	0.50%			0.67%	0.64%	0.56%	0.57%	0.52%	0.49%	0.44%	0.42%	0.46%	0.45%	0.46%
Contained Au	oz x 1000	3774	2,161			38.40	291.62	247.25	261.48	231.79	214.73	173.15	159.70	180.41	174.17	188.76
Contained Cu	t x 1000	779	408			6.66	51.48	44.73	45.88	41.45	39.26	34.86	33.67	36.62	36.02	36.98
Oxide	%	5.7%	8.17%			77%	57%	9%	1%	0%	0%	0%	0%	0%	0%	6%
Gold Recovery Total	%	82.4%	82.90%			73.8%	77.0%	84.1%	86.0%	85.2%	84.5%	82.8%	82.2%	83.1%	82.9%	82.5%
CONCENTRATE PRODUCTION																
Copper Recovery to Concentrate	%	87.9%	87.15%			51.7%	62.8%	88.1%	93.0%	92.3%	91.6%	90.0%	89.5%	90.7%	90.5%	87.5%
Gold Recoverd to Concentrate	oz x 1000	2,530	1,467			23.2	184.1	170.5	184.4	161.9	148.9	117.6	107.6	123.0	118.3	127.6
Copper in Concentrate	t x 1000	682	352			3.4	32.3	39.4	42.7	38.3	36.0	31.4	30.1	33.2	32.6	32.3
Copper in Concentrate	$lb \times 10^6$	1,502	775			7.6	71.2	86.8	94.1	84.3	79.3	69.1	66.4	73.2	71.8	71.3
Dry Tonnage of Concentrate	t x 1000	2,621	1,353			13.2	124.3	151.5	164.2	147.1	138.3	120.7	115.9	127.7	125.3	124.4
Wet Tonnage of Concentrate	t x 1000	2,857	1,474			14.4	135.5	165.2	179.0	160.4	150.8	131.5	126.4	139.2	136.6	135.6
Mass Pull (con/ore)	%	1.66%	1.67%			1.32%	1.55%	1.89%	2.05%	1.84%	1.73%	1.51%	1.45%	1.60%	1.57%	1.55%
Payble Copper	t x 1000	656	338			3.31	31.09	37.90	41.07	36.80	34.60	30.18	28.99	31.94	31.34	31.11
Payable Gold	oz x 1000	2467	1,430			22.66	179.50	166.20	179.75	157.82	145.14	114.64	104.93	119.91	115.38	124.43
Treatement Charges	mUSD	216.27	112			1.09	10.26	12.50	13.55	12.14	11.41	9.95	9.56	10.54	10.34	10.26
Transport Charges	mUSD	55.86	29			0.28	2.65	3.23	3.50	3.14	2.95	2.57	2.47	2.72	2.67	2.65
Cu Refining Fee	mUSD	119.22	62			0.60	5.65	6.89	7.47	6.69	6.29	5.49	5.27	5.81	5.70	5.66
Au Refining Fee	mUSD	15.18	9			0.14	1.10	1.02	1.11	0.97	0.89	0.71	0.65	0.74	0.71	0.77
DORE PRODUCTION																
Payable Gold in Dore	oz x 1000	553.8	321			5.09	40.29	37.31	40.35	35.42	32.58	25.73	23.55	26.92	25.90	27.93
Dore Fee	mUSD	2.78	2			0.03	0.20	0.19	0.20	0.18	0.16	0.13	0.12	0.13	0.13	0.14
						-										
TC/RC OFFSITE	mUSD	409.32	212.36			2.14	19.86	23.83	25.82	23.11	21.71	18.85	18.07	19.94	19.55	19.48
						-										
METALS PRODUCED						-										
Total Payable Gold Produced	oz x 1000	3,021	1,751			27.7	219.8	203.5	220.1	193.2	177.7	140.4	128.5	146.8	141.3	152.4
Total Payable Copper Produced	lbs X 106	1,445	746			7.3	68.5	83.5	90.5	81.1	76.2	66.5	63.9	70.4	69.1	68.6
Total Payable Gold Eq Produced	ozEq x 1000	6,078	3,323			43.2	361.2	376.6	411.6	364.8	339.0	281.1	263.7	295.8	287.4	298.7
						-										
ANNUAL REVENUES																
Gold Price	UDS / oz					1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Copper Price	USD / lb					2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75
Gold Revenue	mUSD	3,927	2,269			36.1	281.1	259.9	286.1	251.2	231.0	182.5	167.0	190.9	183.7	199.7
Copper Revenue	mUSD	3,974	2,051			20.1	188.4	229.7	248.9	223.0	209.7	182.9	175.7	193.6	190.0	188.6
Total Gross Revenue	mUSD	7,901	4,320			56.2	469.6	489.6	535.0	474.2	440.7	365.4	342.8	384.5	373.6	388.3
						-										
ROYALITIES						-										
Gold Royalty						2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%
Copper Royalty						0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%
Gold Royalty	mUSD	78.19	45			0.72	5.60	5.17	5.70	5.00	4.60	3.63	3.33	3.80	3.66	3.98
Copper Royalty	mUSD	17.91	9			0.09	0.85	1.04	1.12	1.01	0.95	0.82	0.79	0.87	0.86	0.85

Table 22-5: Skouries Phase 1 After-tax Cashflow and Cash Cost Schedule

SKOURIES AFTER TAX CASH FLOWS	Units	LOM Totals	Phase 1 Totals	Project Year and Project Phase												
				-3	-2	-1	1	2	3	4	5	6	7	8	9	10
				Pre-Production			Phase 1									Transition
NET REVENUE																
Metal Sales	mUSD	7,901.4	4,319.9			56.2	469.6	489.6	535.0	474.2	440.7	365.4	342.8	384.5	373.6	388.3
Treatement, Transport and Refining	mUSD	-409.3	-212.3			-2.1	-19.8	-23.8	-25.8	-23.1	-21.7	-18.8	-18.1	-19.9	-19.5	-19.5
Royalties	mUSD	-96.1	-54.4			-0.8	-6.4	-6.2	-6.8	-6.0	-5.5	-4.5	-4.1	-4.7	-4.5	-4.8
Net Revenue	**mUSD**	**7,396.0**	**4,053.2**			**53.2**	**443.3**	**459.6**	**502.4**	**445.1**	**413.5**	**342.1**	**320.6**	**359.9**	**349.6**	**364.0**
Net Smelter Return	USD / t ore	46.9	50.0			53.2	55.4	57.5	62.8	55.6	51.7	42.8	40.1	45.0	43.7	45.5
CASH COSTS																
Cash Operating Cost (Cu as by-product)	USD / oz	-70	-184				-64.5	-220.5	-268.3	-267.6	-240.1	-176.7	-173.1	-230.8	-184.4	12.9
Cash Operating Cost (Cu as ozEq)	USD / ozEq	621	523				483.1	492.9	461.3	469.6	492.6	562.5	582.2	540.1	570.3	638.0
AISC (Cu as by -product)	USD / oz	215	146				570.8	-50.6	-153.8	-141.7	-11.2	85.1	255.4	267.0	350.1	432.4
AISC (Cu as ozEq)	USD / ozEq	701	584				526.3	545.3	522.6	514.0	542.4	621.0	642.2	604.1	633.6	761.1
DEPRECIATION MODEL																
Book Value of Assets at Begining of Period	mUSD			389.9	413.6	709.5	1,032.7	1,163.7	1,121.2	1,060.4	997.7	950.6	899.4	864.4	844.7	824.2
Capital Additions	mUSD			23.8	295.9	323.2	130.9	27.8	18.4	18.3	35.1	32.3	50.9	68.4	71.0	59.6
Depreciation	mUSD			0.0	0.0	0.0	70.2	79.1	81.0	82.3	83.5	85.9	88.1	91.6	96.2	101.0
EARNINGS																
Net Revenue	mUSD	7,342.8	4,000.0				443.3	459.6	502.4	445.1	413.5	342.1	320.6	359.9	349.6	364.0
Operating Costs	mUSD	-3,340.2	-1,504.0				-154.7	-161.8	-164.0	-148.2	-145.3	-139.3	-135.4	-139.8	-144.4	-171.1
Depreciation	mUSD	-1,664.1	-859.0				-70.2	-79.1	-81.0	-82.3	-83.5	-85.9	-88.1	-91.6	-96.2	-101.0
Taxes	mUSD	-686.3	-474.7				-63.3	-63.4	-74.6	-62.3	-53.5	-33.9	-28.1	-37.3	-31.6	-26.6
Net Earnings	**mUSD**	**1,652.2**	**1,162.3**				**155.1**	**155.3**	**182.7**	**152.4**	**131.1**	**83.0**	**68.9**	**91.3**	**77.4**	**65.2**
CASH FLOW																
Net Earnings	mUSD	1,652.2	1,162.3				155.1	155.3	182.7	152.4	131.1	83.0	68.9	91.3	77.4	65.2
+ Depreciation	mUSD	1,664.1	859.0				70.2	79.1	81.0	82.3	83.5	85.9	88.1	91.6	96.2	101.0
-Capital	mUSD	-1,400.9	-1,155.6	-23.8	-295.9	-323.2	-130.9	-27.8	-18.4	-18.3	-35.1	-32.3	-50.9	-68.4	-71.0	-59.6
Net After Tax CashFlow	**mUSD**	**1,863.8**	**865.6**	**-23.8**	**-295.9**	**-323.2**	**94.3**	**206.6**	**245.3**	**216.4**	**179.5**	**136.6**	**106.1**	**114.4**	**102.6**	**106.6**
Payback	**years**	**3.45**														
IRR	**%**	**21.25%**														
NPV (5%)	**mUSD**	**925.2**														
NPV (8%)	**mUSD**	**602.9**														

Table 22-6: Skouries Phase 2 Production Schedule

SKOURIES PRODUCTION SCHEDULE	Units	LOM Totals	Phase 2 Totals	Project Year and Project Phase												
				11	12	13	14	15	16	17	18	19	20	21	22	23
				Phase 2												
OPEN PIT																
Direct Ore Mined	**t x 1000**	**53,987**														
Au Grade	g/t	0.61														
Cu Grade	%	0.42%														
Gold Contained	oz x 1000	1,059														
Cu Contained	lb x 10^6	496														
% Oxide	%	11%														
Oxide Ore Stockpile	**t x 1000**	**5,482**														
Au Grade	g/t	0.12														
Cu Grade	%	0.17%														
Gold Contained	oz x 1000	21														
Cu Contained	lb x 10^6	20														
% Oxide	%	51%														
Total Open Pit Waste	**t x 1000**	**52,900**														
Red Clay Waste	t x 1000	4,585														
Overburden Waste	t x 1000	7,648														
Weak Rock Waste	t x 1000	15,385														
Hard Rock Waste	t x 1000	25,283														
Total Mined in Open Pit																
UNDERGROUND PHASE 1																
Ore Mined	**t x 1000**	**22,163**														
Au Grade	g/t	1.39														
Cu Grade	%	0.73%														
Gold Contained	oz x 1000	993														
Cu Contained	lb x 10^6	356														
% Oxide	%	0%														
Waste Mined	t x 1000	821														
Total Mined	t x 1000	22,984														
UNDERGROUND PHASE 2																
Ore Mined	**t x 1000**	**76,035**	**71,579**	**6,408**	**6,413**	**6,276**	**6,197**	**6,255**	**6,005**	**6,129**	**6,149**	**6,198**	**6,115**	**5,893**	**2,666**	**875**
Au Grade	g/t	0.68	0.68	0.86	0.81	0.76	0.73	0.72	0.66	0.64	0.64	0.61	0.57	0.50	0.59	0.57
Cu Grade	%	0.50%	0.50%	0.52%	0.52%	0.53%	0.52%	0.52%	0.51%	0.51%	0.49%	0.48%	0.47%	0.44%	0.48%	0.51%
Gold Contained	oz x 1000	1,671	1,567	178	166	154	145	145	128	127	126	122	113	95	51	16
Cu Contained	lb x 10^6	836	790	74	74	74	72	72	67	69	66	66	63	57	28	10
% Oxide	%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Total Waste Tonnes	t x 1000	1,865	545	2	36	219	143	87	25	9	20	4	-	-	-	-
Total Tonnes	t x 1000	77,900	72,123	6,410	6,449	6,495	6,340	6,342	6,030	6,138	6,169	6,202	6,115	5,893	2,666	875
OXIDE ORE STOCKPILE																
OOS balance beginning of period	t x 1000			5,089	3,497	1,910	186									
Au grade beginning of period	g/t			0.14	0.14	0.14	0.14									
Cu grade beginning of period	%			0.18%	0.18%	0.18%	0.18%									
%Oxide	%			46%	46%	46%	46%									
Additions to OOS during Period	t x 1000															
Au grade additions	g/t															
Cu grade additions	%															
% Oxide	%															
Subtraction from OOS during Period	t x 1000			1,592	1,587	1,724	186									
Au grade subtraction	g/t			0.14	0.14	0.14	0.14									
Cu grade subtraction	%			0.18%	0.18%	0.18%	0.18%									
% Oxide	%			46%	46%	46%	46%									
TOTAL MILL FEED																
Open Pit Direct Ore	t x 1000	53,987														
Open Pit Oxide Ore	t x 1000	6,133		1,592	1,587	1,724	186									
Underground Phase 1	t x 1000	21,512														
Underground Phase 2	t x 1000	76,035		6,408	6,413	6,276	6,197	6,255	6,005	6,129	6,149	6,198	6,115	5,893	2,666	875
Ore Milled	**t x 1000**	**157,666**	**76,666**	**8,000**	**8,000**	**8,000**	**6,383**	**6,255**	**6,005**	**6,129**	**6,149**	**6,198**	**6,115**	**5,893**	**2,666**	**875**
Au Grade	g/t	0.74	0.64	0.71	0.67	0.62	0.71	0.72	0.66	0.64	0.64	0.61	0.57	0.50	0.59	0.57
Cu Grade	%	0.49%	0.48%	0.45%	0.45%	0.46%	0.51%	0.52%	0.51%	0.51%	0.49%	0.48%	0.47%	0.44%	0.48%	0.51%
Mill Feed Grade AuEq	**gEq/t**	**1.50**	**1.41**	**1.41**	**1.37**	**1.33**	**1.53**	**1.56**	**1.48**	**1.46**	**1.43**	**1.39**	**1.33**	**1.21**	**1.38**	**1.40**
Gold Contained	oz x 1000	3,774	1,583	183	171	160	146	145	128	127	126	122	113	95	51	16
Cu Contained	lb x 10^6	1,717	811	80	80	81	72	72	67	69	66	66	63	57	28	10
Gold Equivelent Contained	**oz x 1000**	**7,596**	**3,466**	**362**	**352**	**343**	**315**	**313**	**286**	**289**	**282**	**277**	**261**	**229**	**118**	**39**
% Oxide	%	5.7%	3.0%	9%	9%	10%	1%	0%	0%	0%	0%	0%	0%	0%	0%	0%

Table 22-7: Skouries Phase 2 Operating Cost Schedule

SKOURIES ANNUAL OPERATING COST	Units	LOM Totals	Phase 2 Totals	Project Year and Project Phase												
				11	12	13	14	15	16	17	18	19	20	21	22	23
				Phase 2												
OPEN PIT MINING																
Drill and Blast	mUSD	15.1														
Mine Rock Load and Haul	mUSD	153.2														
Contractors Profit Costs	mUSD	10.8														
Overhead	mUSD	38.7														
Operations support	mUSD	21.1														
UNDERGROUND MINING																
Ore Cross-cut	mUSD	129.7	55.50	11.9	12.0	8.2	4.5	4.0	4.1	4.0	3.9	2.8	—	—	—	0.0
Development through PF	mUSD	25.0	22.57	2.5	2.0	2.3	1.5	0.4	1.9	2.7	3.0	2.8	1.6	1.6	0.2	—
SLOS 20m	mUSD	165.5	133.66	12.6	17.3	17.3	13.6	9.7	13.7	10.7	8.7	8.7	8.3	6.0	5.7	1.2
SLOS 30m	mUSD	324.0	205.88	16.7	12.3	12.2	15.8	19.7	15.0	18.3	20.3	20.6	20.8	22.2	9.0	3.1
Backfill	mUSD	327.2	238.59	21.0	21.2	20.7	20.9	20.9	20.2	20.5	20.3	20.4	20.3	19.6	9.5	3.0
Mine General (Total)	mUSD	461.3	305.09	24.9	25.0	25.1	24.9	25.6	25.5	25.5	25.5	25.5	25.4	23.9	22.5	5.7
Mine Maintenance (Total)	mUSD	169.6	121.27	10.3	10.3	10.2	10.2	10.2	10.0	10.1	10.1	10.2	10.1	10.0	7.5	2.0
STOCKPILE REHANDLE																
Oxide Ore Stockpile	mUSD	9.8	7.63	2.4	2.4	2.6	0.3	—	—	—	—	—	—	—	—	—
PROCESSING COSTS																
8.0 mpta Rate	mUSD	691.6	159.60	53.2	53.2	53.2	—	—	—	—	—	—	—	—	—	—
6.0 mtpa Rate	mUSD	363.4	363.41	—	—	—	44.0	43.2	41.4	42.3	42.4	42.8	42.2	40.7	18.4	6.0
FILTER PLANT COSTS																
Filter Plant Opex	mUSD	119.1	58.27	6.1	6.1	6.1	4.9	4.8	4.6	4.7	4.7	4.7	4.6	4.5	2.0	0.7
TAILINGS HANDLING																
IWMF			0.00													
Basin Drain Opex	mUSD	0.6	0.12	0.0	0.0	0.0										
KL Embankment Opex	mUSD	3.1	0.07	0.0	0.0	—										
Tailings Handling	mUSD	40.6	0.03	0.0	0.0	0.0										
IN_PIT TMF																
Open Pit Drainage and Reclaim System	mUSD	2.3	2.21	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.1	0.1
Tailings Handling	mUSD	42.3	39.55	2.9	2.9	2.9	2.9	2.9	2.9	2.9	3.2	3.2	3.3	3.2	3.2	3.2
WATER MANAGEMENT																
Site Pumping Costs	mUSD	4.0	0.00	—	—	—	—	—	—	—	—	—	—	—	—	—
Water Treatment Plant	mUSD	3.9	0.00	—	—	—	—	—	—	—	—	—	—	—	—	—
GENERAL+ADMIN				—	—	—	—	—	—	—	—	—	—	—	—	—
G+A Kassandra Allocation	mUSD	154.9	85.26	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	1.7
G+A Skouries Mill Admin	mUSD	63.5	34.94	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	0.7
TOTAL OPERATING COSTS	**mUSD**	**3340.21**	**1833.76**	**174.60**	**174.77**	**170.89**	**153.55**	**151.41**	**149.32**	**151.51**	**152.24**	**151.71**	**146.60**	**141.58**	**87.97**	**27.61**
	USD/ t ore	**21.32**	**23.92**	**21.83**	**21.85**	**21.36**	**24.06**	**24.20**	**24.87**	**24.72**	**24.76**	**24.48**	**23.97**	**24.03**	**32.99**	**31.56**

Table 22-8: Skouries Phase 2 Capital Cost Schedule

SKOURIES ANNUAL CAPITAL COSTS	Units	LOM Totals	Phase 2 Totals	Project Year and Project Phase												
				11	12	13	14	15	16	17	18	19	20	21	22	23
				Phase 2												
DEVELOPMENT CAPITAL (Pre-production)																
Overall Site	mUSD	19.6	—													
Open Pit Mine	mUSD	67.5	—													
Underground Mine	mUSD	144.0	—													
Stockpile and Materials Handling	mUSD	11.9	—													
Process Plant	mUSD	68.5	—													
Filter Plant	mUSD	68.9	—													
Integrated Waste Management Facility (IWM	mUSD	24.3	—													
Water Management	mUSD	17.2	—													
Infrastructure	mUSD	50.6	—													
Ancillary Facilities	mUSD	9.2	—													
Off Site Infrastructure	mUSD	6.1	—													
Indirects	mUSD	60.6	—													
Spares/Warehouse Fills	mUSD	15.1	—													
EPCM	mUSD	27.1	—													
Owners Cost	mUSD	33.8	—													
Contingency	mUSD	86.9	—													
Commissioning Revenue minus Cost	mUSD	-42.0	—													
SUSTAINING CAPITAL COSTS																
Open Pit Mine	mUSD	23.0	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Underground Mine	mUSD	405.4	130.1	16.1	10.8	13.2	14.3	21.7	13.3	3.4	11.2	13.8	2.3	8.4	1.3	0.3
Stockpile and Materials Handling	mUSD	0.6	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Process Plant	mUSD	41.4	32.4	2.9	2.9	2.8	2.8	2.8	2.7	2.8	2.8	2.8	2.8	2.7	1.2	0.4
Backfill Plant	mUSD	27.6	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Integrated Waste Management Facility (IWM	mUSD	21.9	0.2	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
In Pit Tailings	mUSD	40.5	29.1	2.1	0.3	22.1	0.2	0.8	0.1	0.1	0.1	1.4	0.8	0.1	0.1	0.8
Water Management	mUSD	3.5	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Infrastructure	mUSD	6.0	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Ancillary Facilities	mUSD	2.1	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Environmental	mUSD	2.7	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Indirects	mUSD	23.5	7.8	0.7	0.2	3.1	0.2	0.5	0.5	0.2	0.3	0.7	0.3	0.5	0.1	0.5
Spares/Warehouse Fills	mUSD	12.6	3.3	0.7	0.2	0.1	0.2	0.7	0.3	0.1	0.4	0.2	0.0	0.3	0.0	0.0
EPCM	mUSD	23.5	2.9	0.3	0.0	2.0	0.0	0.1	0.0	0.0	0.0	0.2	0.1	0.0	0.0	0.1
Contingency	mUSD	123.6	39.0	3.5	2.7	10.0	3.4	4.6	2.9	1.3	2.3	3.8	1.5	1.9	0.6	0.5
TOTAL CAPITAL	**mUSD**	**1405.2**	**244.8**	**26.3**	**17.1**	**53.4**	**21.1**	**31.2**	**19.9**	**7.9**	**17.1**	**23.0**	**7.8**	**13.9**	**3.5**	**2.6**

Table 22-9: Skouries Phase 2 Metal Production and Off-site Cost Schedule

SKOURIES METALS PRODUCED AND OFFSITE COSTS	Units	LOM Totals	Phase 2 Totals	Project Year and Project Phase												
				11	12	13	14	15	16	17	18	19	20	21	22	23
										Phase 2						
MILLING SCHEDULE																
Ore Milled	**t x 10^6**	**157.67**	**74.67**	**8.00**	**8.00**	**8.00**	**6.38**	**6.26**	**6.01**	**6.13**	**6.15**	**6.20**	**6.11**	**5.89**	**2.67**	**0.87**
Grade Au	g/t	0.74	0.66	0.71	0.67	0.62	0.71	0.72	0.66	0.64	0.64	0.61	0.57	0.50	0.59	0.57
Gracde Cu	%	0.49%	0.49%	0.45%	0.45%	0.46%	0.51%	0.52%	0.51%	0.51%	0.49%	0.48%	0.47%	0.44%	0.48%	0.51%
Contained Au	oz x 1000	3774	1,583	182.78	171.44	159.56	146.08	145.34	128.30	126.92	126.06	121.75	112.99	94.56	50.98	16.04
Contained Cu	t x 1000	779	368	36.39	36.38	36.65	32.82	32.52	30.40	31.09	30.12	29.85	28.55	25.71	12.87	4.45
Oxide	%	5.7%	3.13%	9%	9%	10%	1%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Gold Recovery Total	%	82.4%	84.07%	81.6%	81.1%	80.4%	83.0%	83.4%	82.7%	82.5%	82.4%	82.1%	81.6%	80.5%	81.8%	81.5%
CONCENTRATE PRODUCTION																
Copper Recovery to Concentrate	%	87.9%	91.97%	85.5%	85.5%	85.2%	91.5%	92.3%	92.0%	92.0%	91.6%	91.4%	91.0%	90.0%	91.4%	92.1%
Gold Recoverd to Concentrate	oz x 1000	2,530	1,063	122.4	114.1	105.2	99.4	99.4	87.0	85.8	85.2	81.9	75.6	62.4	34.2	10.7
Copper in Concentrate	t x 1000	682	330	31.1	31.1	31.2	30.0	30.0	28.0	28.6	27.6	27.3	26.0	23.1	11.8	4.1
Copper in Concentrate	lb x 10^6	1,502	727	68.6	68.6	68.8	66.2	66.2	61.6	63.0	60.8	60.1	57.2	51.0	25.9	9.0
Dry Tonnage of Concentrate	t x 1000	2,621	1,269	119.7	119.6	120.1	115.4	115.5	107.6	110.0	106.1	104.9	99.9	89.0	45.2	15.8
Wet Tonnage of Concentrate	t x 1000	2,857	1,383	130.4	130.4	130.9	125.8	125.9	117.2	119.9	115.6	114.3	108.9	97.0	49.3	17.2
Mass Pull (con/ore)	%	1.66%	1.70%	1.50%	1.50%	1.50%	1.81%	1.85%	1.79%	1.80%	1.73%	1.69%	1.63%	1.51%	1.70%	1.80%
Payble Copper	t x 1000	656	317	29.93	29.92	30.03	28.88	28.88	26.90	27.52	26.54	26.24	24.98	22.27	11.31	3.94
Payable Gold	oz x 1000	2467	1,037	119.31	111.22	102.61	96.96	96.89	84.84	83.68	83.03	79.87	73.69	60.88	33.36	10.46
Treatement Charges	mUSD	216.27	105	9.87	9.87	9.90	9.52	9.53	8.87	9.08	8.75	8.65	8.24	7.34	3.73	1.30
Transport Charges	mUSD	55.86	27	2.55	2.55	2.56	2.46	2.46	2.29	2.34	2.26	2.24	2.13	1.90	0.96	0.34
Cu Refining Fee	mUSD	119.22	58	5.44	5.44	5.46	5.25	5.25	4.89	5.00	4.83	4.77	4.54	4.05	2.06	0.72
Au Refining Fee	mUSD	15.18	6	0.73	0.68	0.63	0.60	0.60	0.52	0.51	0.51	0.49	0.45	0.37	0.21	0.06
DORE PRODUCTION																
Payable Gold in Dore	oz x 1000	553.8	233	26.78	24.96	23.03	21.76	21.75	19.04	18.78	18.64	17.93	16.54	13.67	7.49	2.35
Dore Fee	mUSD	2.78	1	0.13	0.13	0.12	0.11	0.11	0.10	0.09	0.09	0.09	0.08	0.07	0.04	0.01
TC/RC OFFSITE	mUSD	409.32	196.96	18.73	18.67	18.67	17.94	17.95	16.68	17.03	16.44	16.24	15.45	13.73	6.99	2.43
METALS PRODUCED																
Total Payable Gold Produced	oz x 1000	3,021	1,270	146.1	136.2	125.6	118.7	118.6	103.9	102.5	101.7	97.8	90.2	74.6	40.8	12.8
Total Payable Copper Produced	lbs X 106	1,445	699	66.0	65.9	66.2	63.6	63.7	59.3	60.7	58.5	57.8	55.1	49.1	24.9	8.7
Total Payable Gold Eq Produced	ozEq x 1000	6,078	2,755	287.5	277.5	267.7	253.6	253.3	229.3	230.8	225.4	220.1	206.7	178.4	93.6	31.2
ANNUAL REVENUES																
Gold Price	UDS / oz			1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Copper Price	USD / lb			2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75
Gold Revenue	mUSD	3,927	1,658	192.4	179.5	165.9	154.6	154.2	135.0	133.2	132.2	127.1	117.3	96.9	53.1	16.6
Copper Revenue	mUSD	3,974	1,923	181.4	181.4	182.0	175.0	175.1	163.1	166.8	160.8	159.0	151.4	135.0	68.6	23.9
Total Gross Revenue	mUSD	7,901	3,582	373.8	360.8	348.0	329.7	329.3	298.1	300.0	293.0	286.2	268.7	231.9	121.6	40.5
ROYALITIES																
Gold Royalty				2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%
Copper Royalty				0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%
Gold Royalty	mUSD	78.19	33	3.83	3.57	3.30	3.08	3.07	2.69	2.65	2.63	2.53	2.34	1.93	1.06	0.33
Copper Royalty	mUSD	17.91	9	0.82	0.82	0.82	0.79	0.79	0.74	0.75	0.73	0.72	0.68	0.61	0.31	0.11

Table 22-10: Skouries Phase 2 After-tax Cashflow and Cash Cost Schedule

SKOURIES AFTER TAX CASH FLOWS	Units	LOM Totals	Phase 2 Totals	Project Year and Project Phase												
				11	12	13	14	15	16	17	18	19	20	21	22	23
										Phase 2						
NET REVENUE																
Metal Sales	mUSD	7,901.4	3,581.5	373.8	360.8	348.0	329.7	329.3	298.1	300.0	293.0	286.2	268.7	231.9	121.6	40.5
Treatement, Transport and Refining	mUSD	-409.3	-197.0	-18.7	-18.7	-18.7	-17.9	-17.9	-16.7	-17.0	-16.4	-16.2	-15.4	-13.7	-7.0	-2.4
Royalties	mUSD	-96.1	-41.7	-4.6	-4.4	-4.1	-3.9	-3.9	-3.4	-3.4	-3.4	-3.2	-3.0	-2.5	-1.4	-0.4
Net Revenue	**mUSD**	**7,396.0**	**3,342.9**	**350.4**	**337.7**	**325.1**	**307.8**	**307.5**	**278.0**	**279.6**	**273.2**	**266.7**	**250.3**	**215.6**	**113.3**	**37.7**
Net Smelter Return	USD / t ore	46.9	44.8	43.8	42.2	40.6	48.2	49.2	46.3	45.6	44.4	43.0	40.9	36.6	42.5	43.1
CASH COSTS																
Cash Operating Cost (Cu as by-product)	USD / oz	-70	84	80.7	87.6	59.2	-29.6	-48.2	28.3	17.1	77.1	91.3	117.7	273.0	646.7	480.3
Cash Operating Cost (Cu as ozEq)	USD / ozEq	621	737	672.5	697.0	708.3	676.3	668.6	723.9	730.4	748.4	763.0	783.9	870.8	1,014.9	963.4
AISC (Cu as by -product)	USD / oz	215	309	289.6	243.2	510.1	180.1	247.3	252.7	127.1	278.5	360.1	237.8	494.1	764.7	718.0
AISC (Cu as ozEq)	USD / ozEq	701	841	780.0	774.4	923.3	774.7	807.0	825.6	779.2	839.2	882.4	836.3	963.2	1,066.4	1,061.0
DEPRECIATION MODEL																
Book Value of Assets at Begining of Period	mUSD			787.6	712.8	624.8	571.3	484.4	403.9	331.3	296.8	277.0	264.4	236.5	214.8	184.5
Capital Additions	mUSD			26.3	17.1	53.4	21.1	31.2	19.9	7.9	17.1	23.0	7.8	13.9	3.5	2.6
Depreciation	mUSD			105.1	106.9	108.0	111.7	92.5	42.4	36.9	35.7	35.6	35.6	33.8	32.2	28.6
EARNINGS																
Net Revenue	mUSD	7,342.8	3,342.9	350.4	337.7	325.1	307.8	307.5	278.0	279.6	273.2	266.7	250.3	215.6	113.3	37.7
Operating Costs	mUSD	-3,340.2	-1,833.8	-174.6	-174.8	-170.9	-153.6	-151.4	-149.3	-151.5	-152.2	-151.7	-146.6	-141.6	-88.0	-27.6
Depreciation	mUSD	-1,664.1	-805.1	-105.1	-106.9	-108.0	-111.7	-92.5	-42.4	-36.9	-35.7	-35.6	-35.6	-33.8	-32.2	-28.6
Taxes	mUSD	-686.3	-211.5	-20.5	-16.3	-13.4	-12.4	-18.5	-25.0	-26.4	-24.7	-23.0	-19.7	-11.7	0.0	0.0
Net Earnings	**mUSD**	**1,652.2**	**492.4**	**50.2**	**39.8**	**32.8**	**30.3**	**45.2**	**61.3**	**64.7**	**60.5**	**56.3**	**48.3**	**28.5**	**-6.9**	**-18.6**
CASH FLOW																
Net Earnings	mUSD	1,652.2	492.4	50.2	39.8	32.8	30.3	45.2	61.3	64.7	60.5	56.3	48.3	28.5	-6.9	-18.6
+ Depreciation	mUSD	1,664.1	805.1	105.1	106.9	108.0	111.7	92.5	42.4	36.9	35.7	35.6	35.6	33.8	32.2	28.6
-Capital	mUSD	-1,400.9	-244.8	-26.3	-17.1	-53.4	-21.1	-31.2	-19.9	-7.9	-17.1	-23.0	-7.8	-13.9	-3.5	-2.6
Net After Tax CashFlow	**mUSD**	**1,863.8**	**1,052.8**	**129.0**	**129.6**	**87.4**	**120.8**	**106.4**	**83.8**	**93.7**	**79.1**	**68.9**	**76.1**	**48.4**	**21.9**	**7.5**
Payback	**years**	**3.45**														
IRR	**%**	**21.25%**														
NPV (5%)	**mUSD**	**925.2**														
NPV (8%)	**mUSD**	**602.9**														

22.4 ROYALTIES AND OTHER FEES

The Skouries Project is subject to a mineral production royalty regime which has a sliding scale NSR type of royalty payable to the Greek government.

The relevant royalties for the copper and gold are shown in Table 22-11 and Table 22-12. The royalties are calculated on the payable metals that are produced by the site with deductions allowable for the downstream smelter costs including refining, and treatment. As such the royalty can be considered a net smelter returns royalty.

The royalty regime incorporates a sliding scale depending on the metal price on the date of sale. For gold, the royalty ranges from 0-6% and, for copper, the royalty ranges from 0-2.5%.

For the Project case economics reported in this section the gold royalty is 2.0%, and the copper royalty is 0.5%. In the case of the sensitivity analysis, the royalty corresponds to the metal prices used in the respective case.

Table 22-11: Gold Royalty

Gold Price (€/ oz)		NSR Royalty
From	To	
0	600	0 %
600	900	1.0 %
900	1100	1.5 %
1100	1300	2.0 %
1300	1500	3.0%
1500	1650	4.0%
1650	1800	5.0%
1800	1800+	6.0%

Table 22-12: Copper Royalty

Copper Price (€ / tonne)		NSR Royalty
From	To	
0	5000	0%
5000	5800	0.5%
5800	6600	1.0%
6600	7400	1.5%
7400	8200	2.0%
8200	8200+	2.5%

22.5 CLOSURE AND SALVAGE VALUE

Closure costs are captured by the economic model in the form of an asset retirement obligation (ARO) that is offset by progressive rehabilitation as discussed in Section 20. The Phase 1 surface IWMF operation includes progressive rehabilitation with re-grading of tailings, hauling and placing of waste rock and topsoil during Years 8 to 13 of the operation. These costs are accounted for in the financial model as opex since they will not be depreciated. The total progressive rehabilitation costs for the surface IWMF in Phase 1 are estimated to be US$15.8 M. Additionally, the closure of the open pit includes backfilling with Tailings during the Phase 2 of the operation, these costs are also included in opex.

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$29.6 M as the cost net of salvage value.

The financial model incorporates a period of three years post closure where costs totaling US$4 M relating to the Kassandra environmental department, laboratory and other associated admin costs will be incurred. The purpose is to develop the monitoring regime which will be put in place once the site is closed, and to monitor and manage the dismantling and rehabilitation of the site. After this rehabilitation is complete, the economic model includes a further 10 years of monitoring and laboratory sampling at a rate of US$1 M per year.

No allowance has been made in the financial model for the carrying costs of the ARO guarantee.

22.6 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 29% 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 22-13.

Table 22-13: 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

22.7 FINANCING COSTS

Cost 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.

22.8 THIRD PARTY INTERESTS

Hellas Gold is the 100% owner of the Kassandra mines, which includes the Skouries Project. Eldorado owns a 95% interest in Hellas Gold, with the remaining 5% held by Aktor Enterprises Ltd., a Greek construction firm.

All data provided in this report is shown at the 100% ownership level.

22.9 SENSITIVITY ANALYSIS

The economic model was subjected to a sensitivity analysis to determine the effects of changing metal prices, and changes in operating and capital assumption, on the Project financial returns. As shown on Table 22-14 and Table 22-15.

The test of economic extraction for the Skouries mineral reserves is demonstrated by means of this sensitivity analysis. At the mineral reserve metals prices of US$1,200/oz Au and US$2.50/lb Cu the Project shows strong positive economics. The after tax IRR is 17.4 % and the NPV is estimated to be US$669 M using the 5% discount rate, with a calculated payback period of 3.9 years.

The sensitivity analysis also shows that the Project is also robust when evaluated using lower metal price assumptions, or higher operating and capital costs. The Project continues to develop net positive cash flows and positive net present value at metal prices of US$1,100/oz Au and US$2.30/lb Cu, or with operating and capital cost increased by 15% individually.

Table 22-14: Metal Price Sensitivity Analysis

		Sensitivity Ranges				
Parameters	**Units**	**-15%**	**-7.5%***	**Project Case**	**+7.5%**	**+15%**
Gold Price	US$/oz	1,100	1,200	**1,300**	1,400	1,500
Copper Price	US$/lb	2.30	2.50	**2.75**	3.00	3.20
Results (after Tax)						
NPV 0%	US$ M	994	1,418	**1,864**	2,332	2,700
NPV 5%	US$ M	421	668	**925**	1,197	1,411
NPV 8%	US$ M	221	407	**603**	809	972
IRR %	%	13.4	17.4	**21.2**	25.0	28.0
Payback period	yrs	4.7	3.9	**3.4**	3.0	2.7
Taxation	US$ M	348	510	**686**	873	1,022
Royalties	US$ M	49	54	**96**	104	178

*Reserve Case Metal Prices

Table 22-15: Capex and Opex Sensitivity Analysis

		Sensitivity Ranges		
Parameter	**Units**	**-15%**	**Project Case**	**15%**
LOM Capex*	US$ M	1,237	**1,447**	1,657
Results (after Tax)				
NPV 0%	US$ M	2,021	**1,864**	1,706
NPV 5%	US$ M	1,056	**925**	795
NPV 8%	US$ M	722	**603**	484
IRR	%	26.0	**21.2**	17.5
LOM Operating Costs	US$/t ore	18.1	**21.3**	24.5
Results (after Tax)				
NPV 0%	US$ M	2,222	**1,864**	1,502
NPV 5%	US$ M	1,120	**925**	730
NPV 8%	US$ M	746	**603**	460
IRR	%	23.6	**21.2**	18.7

*Excluding Pre-production Metal Sales

SECTION • 23 ADJACENT PROPERTIES

Not applicable.

SECTION • 24 OTHER RELEVANT DATA AND INFORMATION

24.1 ECONOMICS

The Skouries Project has been under construction since 2012 and the capital costs incurred to date are sunk costs and are not included in the capital cost estimate. The sunk costs are used in the economic evaluation described in Section 22, as they form a portion of depreciable assets used to estimate net earnings and tax payable.

The sunk capital costs estimated up until the end of the 4rd Quarter of 2017 are estimated at US$434.5 M. These costs are used in the estimate of depreciation and in accordance with Greek tax law, an average deprecation rate of 6.8% has been estimated to apply to these costs. Costs included in the estimate are from restart of development, care and maintenance costs have not been included in the capital cost estimate of economics.

24.2 MANPOWER ESTIMATES

The total manpower estimates for the Skouries Project are shown in Figure 24-1.

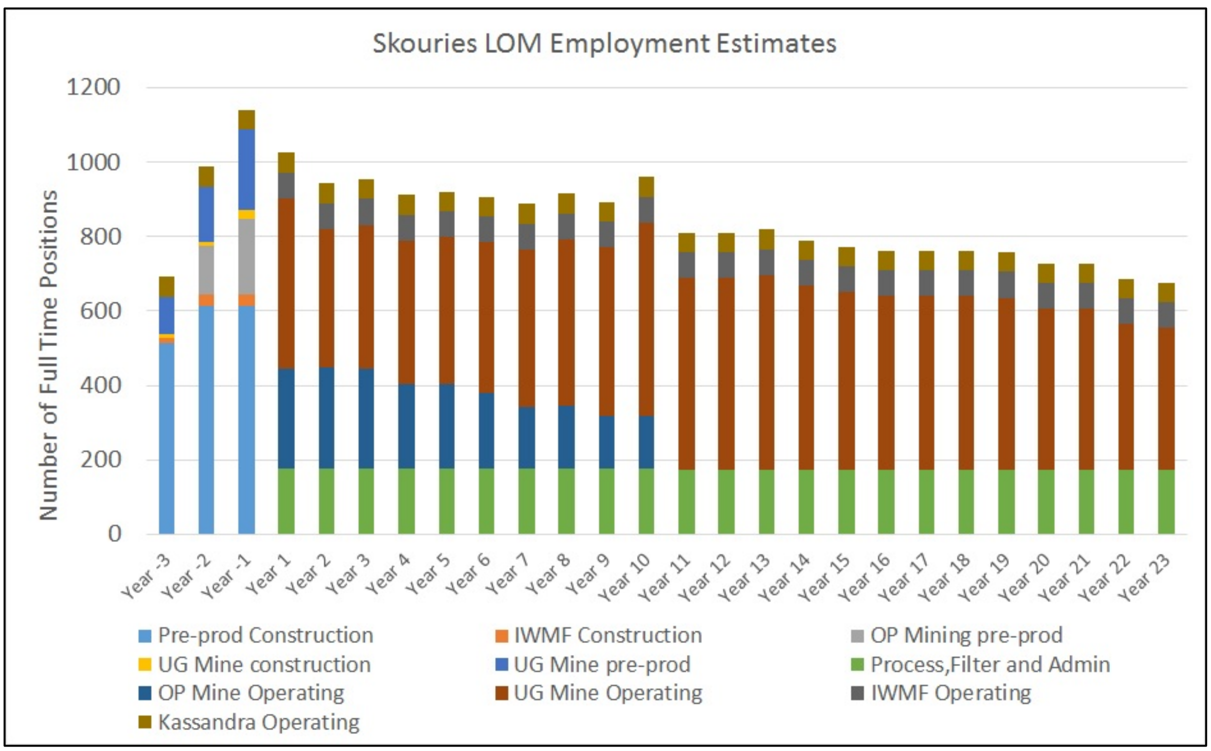


Figure 24-1: Skouries Project, Manpower Estimates

24.3 SCHEDULE

This report is based on the Skouries Project development restarting after receipt of required approvals and permits. The Gantt chart summary level schedule for development of the Project is shown in Figure 24-2.

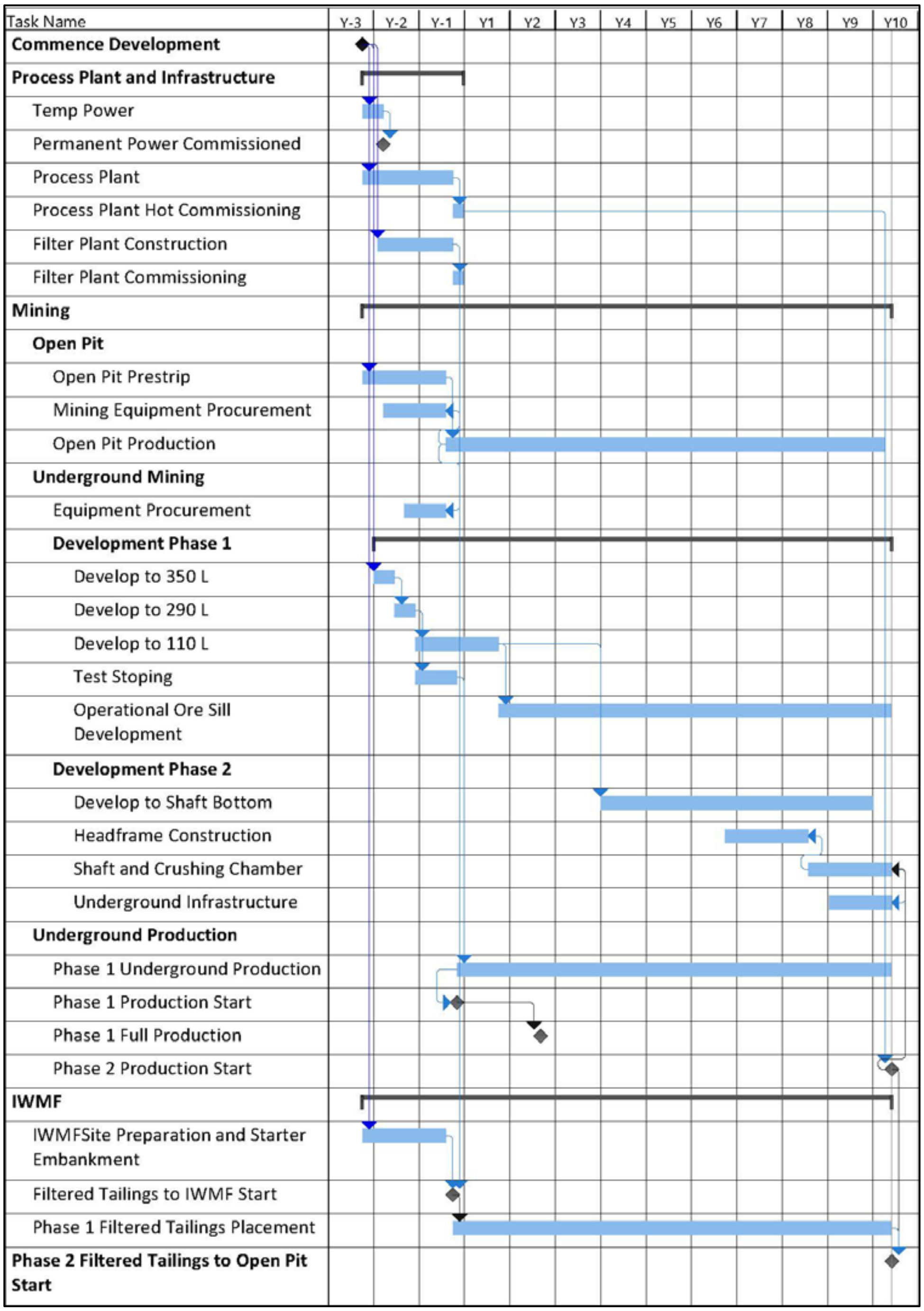


Figure 24-2: Skouries Project, Implementation Schedule

The effective date of the report is January 1, 2018. As of the date of the report, the Project has been placed onto care and maintenance awaiting approval of several permits necessary to complete the development plans outlined in this report. Given the uncertain timeline relating to these permits, the project schedule, and all future dates reported herein, refer to nominal dates, with commercial production starting on day 1 of Year 1, and being preceded by the 2.25 years of remaining development that occurs in Years -1, -2 and the last quarter of Year -3.

SECTION • 25 INTERPRETATION AND CONCLUSIONS

It is concluded that the work completed in the prefeasibility study indicate that the exploration information, mineral resource and mineral reserve estimates and Project economics are sufficiently defined to indicate the Project is technically and economically viable.

25.1 MINERAL RESOURCES AND MINERAL RESERVES

Information and data contained in, or used in the preparation of mineral resource and mineral reserve update was obtained from historic data obtained from EGL verified and supplemented by information from a surface diamond drill campaign undertaken by Eldorado and Hellas Gold. The mineral resource and mineral reserve are consistent with the CIM definitions referred to in NI 43-101. It is the opinion of the qualified persons that the information and analysis provided in this report is considered sufficient for reporting mineral resources and mineral reserves.

Results of drilling indicate the ore body is open at depth and indications are that inferred resources could be converted to indicated resources with further exploration. Eldorado considers this an opportunity to the Project, further exploration at depth should be completed during operations.

The open pit depth in the current design is being constrained at depth by the crown pillar, the resources associated with the crown pillar have not been included in the mine plan shown in this study.

25.2 RECOVERY

Significant metallurgical testwork and analysis has been completed to confirm the process designs and substantiate stated recoveries. Eldorado has reviewed and validated historic data obtained from EGL and completed additional confirmatory testwork. The qualified persons have a high degree of confidence in the process designs and the stated recoveries.

25.3 CAPITAL AND OPERATING COSTS AND FINANCIAL MODEL

The accuracy of the capital and operating cost estimates is consistent with the standards outlined by the AACE. The economic model has been built from first principles and includes all relevant data, the qualified persons have a high level of confidence in the stated economic performance of the Project.

Eldorado’s forecasts of costs are based on a set of assumptions current as at the date of completion of this technical report. The realized economic performance achieved on the Project may be affected by factors outside the control of Eldorado, including but not limited to mineral prices and currency fluctuations.

25.4 PERMITTING

Permitting of the Project is well advanced with approval of the technical study, electromechanical installation permit and building permit completed from 2012 through to 2014. There are outstanding routine permits that Eldorado expects will be granted in a timely manner. Delays in issuance of these permits could cause delays to the development of the Project.

Amended technical studies, electromechanical installation permits and building permits will be submitted by Hellas Gold. Delays in receiving approval of the amended technical studies will impact the ability of Eldorado to implement the Project.

SECTION • 26 RECOMMENDATIONS

The prefeasibility study investigations have provided a technical and economic solution that forms the basis to proceed with development of the Project. It is recommended to continue with development of the Project while undertaking the work described below.

26.1 PERMITTING

Prior to work recommencing on site Hellas Gold must obtain the revised technical study approval and permits. Eldorado expects that the revised permits will be received presently. The majority of the work supporting the preparation of these permits submission has been completed so the cost and schedule is negligible.

26.2 ENGINEERING

Engineering of the majority of the Project has been developed to a detailed engineering level. Detailed engineering and procurement of the underground mine, filtration plant, IWMF and water management will be completed to support the development of the Project. A budget of US$11.8 M to complete the engineering and procurement of these areas has been included in the capital cost estimate. Development work on site can commence in parallel with these activities so the implementation is not delayed pending completion of these activities.

26.3 SITE INVESTIGATIONS

Work that Hellas Gold Operations will continue to complete on site during development and operations include:

- Geotechnical investigations at the shaft and ventilation raises.
- Complete exploration and studies to increase resources by expanding the underground mine at depth.
- Optimize development to enable test stoping to be completed as soon as possible.
- Optimize the mine plan with respect to owner supplied and contractor fleet for waste haulage.

These investigations and studies are optimizations and confirmatory work and do not affect the development schedule as presented. The studies are be completed by the Skouries operations personnel and costs are included in the current cost estimate.

SECTION • 27 REFERENCES

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SECTION • 28 CERTIFICATES OF AUTHORS AND DATE AND SIGNATURE PAGE

Date and Signature Page

The effective date of this report entitled "Technical Report, Skouries Project, Greece" is January 1st, 2018. It has been prepared for Eldorado Gold Corporation by Richard Alexander, P. Eng., Stephen Juras, Ph.D., P. Geo, Paul Skayman, FAusIMM, Colm Keogh, P. Eng. and John Nilsson, P. Eng. , each of whom are qualified persons as defined by NI 43-101.

Signed the 15th day of March 2018.

"Signed and Sealed"

Richard Alexander

Richard Alexander, P. Eng.

"Signed and Sealed"

Stephen J. Juras

Stephen J. Juras, Ph.D., P. Geo.

"Signed"

Paul J. Skayman

Paul J. Skayman, FAusIMM

"Signed"

Colm Keogh

Colm Keogh, P. Eng.

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John Nilsson

John Nilsson, P. Eng.

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I, Richard Alexander, am a Professional Engineer, employed as Project Director, of Eldorado Gold Corporation and reside at 5922 Boundary Place in the City of Surrey in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Skouries Project, Greece*, with an effective date of January 1st, 2018.

I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia). I graduated from the University of Alberta with a Bachelor of Science degree in mechanical engineering in 1985.

I have practiced my profession continuously since 1985 and have worked in project management and engineering and construction management of mineral projects in South America, Central America, Canada, United States, Turkey, Romania, Greece, and the former Soviet Union.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Skouries Project on numerous occasions with my most recent visit occurring on April 18 to April 20, 2016.

I was responsible for coordinating the preparation of the technical report. I am responsible for the preparation or supervising the preparation of items 1, 2, 3, 4, 5, 6, 18, 19, 20, 21, 22, 23, 24, 25, 26, and 27 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the items for which I am responsible in this report entitled, *Technical Report, Skouries Project, Greece*, with an effective date of January 1st, 2018, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 15th day of March 2018.

"Signed and Sealed"

Richard Alexander

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I, Stephen J. Juras, am a Professional Geoscientist, employed as Director, Technical Services, of Eldorado Gold Corporation and reside at 9030 161 Street in the City of Surrey in the Province of British Columbia.

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I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia). I graduated from the University of Manitoba with a Bachelor of Science (Honours) degree in geology in 1978 and subsequently obtained a Master of Science degree in geology from the University of New Brunswick in 1981 and a Doctor of Philosophy degree in geology from the University of British Columbia in 1987.

I have practiced my profession continuously since 1987 and have been involved in: mineral exploration and mine geology on gold, copper, zinc and silver properties in Canada, United States, Brazil, China, Greece and Turkey; and ore control and resource modelling work on gold, copper, zinc, silver, platinum/palladium and industrial mineral properties in Canada, United States, Mongolia, China, Brazil, Turkey, Greece, Romania, Peru and Australia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Skouries Project on numerous occasions with my most recent visit occurring on November 12, 2017.

I was responsible for reviewing matters related to the geological data and directing the mineral resource estimation and classification work for the Skouries Project in Greece. I am responsible for the preparation or supervising the preparation of items 7, 8, 9, 10, 11, 12 and 14 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the items for which I am responsible in this report entitled, *Technical Report, Skouries Project, Greece*, with an effective date of January 1st, 2018, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 15th day of March 2018.

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I, Paul J. Skayman, am a Professional Extractive Metallurgist, employed as Chief Operating Officer, of Eldorado Gold Corporation and reside at 3749 West 39th Avenue in Dunbar, Vancouver in the Province of British Columbia.

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I am a fellow of the Australian Institute of Mining and Metallurgy. I graduated from the Murdoch University with a Bachelor of Science (Extractive Metallurgy) degree in 1987.

I have practiced my profession continuously since 1987 and have been involved in operation and management of gold extraction operations in Australia, Ghana, Tanzania, Guinea and China. This work has also included Feasibility Studies, Project Acquisition and Development / Construction of said projects.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Skouries Project on numerous occasions with my most recent visit occurring on July 14, 2017.

I was responsible for reviewing matters related to the metallurgical data for the Skouries Project in Greece. I am responsible for the preparation or supervising the preparation of Sections 13 and 17 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the items for which I am responsible in this report entitled, *Technical Report, Skouries Project, Greece*, with an effective date of January 1st, 2018, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 15th day of March 2018.

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I, Colm Keogh, am a Professional Engineer, employed as Manager, Mine Engineering (Underground) of Eldorado Gold Corporation and residing at 1107 Miller Road, Bowen Island in the Province of British Columbia.

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I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia). I graduated with a Bachelor of Applied Science degree (Mining) from the University of British Columbia in Vancouver, British Columbia in 1988.

I have practiced my profession continuously since 1988 in a range of operational, technical and consulting roles.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Skouries Project on numerous occasions with my most recent visit occurring on November 12, 2017.

I was responsible for matters relating to the underground mine design and production schedule for the Skouries Project in Greece. I am responsible for the preparation or supervising the preparation of Items 15.2, 15.3, 16.1, 16.6, 16.7, 16.8, 16.9, 16.10, 16.11, 16.12, 16.13, 16.14 and 16.15 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the items for which I am responsible in this report entitled, *Technical Report, Skouries Project, Greece*, with an effective date of January 1st, 2018, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 15th day of March 2018.

"Signed"

Colm Keogh

____

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I, John Nilsson, am a Professional Engineer, employed as President, of Nilsson Mine Services Ltd. and residing at 20263 Mountain Place in the city of Pitt Meadows in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Skouries Project, Greece*, with an effective date of January 1st, 2018.

I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia). I graduated from Queen's University with a Bachelor of Science degree in geology in 1977 and subsequently a Master of Science degree through the Department of Mining Engineering in 1990.

I have practiced my profession in geology and mining continuously since 1977 and have worked on mining related precious and base metal projects in North America, Central America, South America, Africa, Europe and Asia.

As a result of my experience and qualifications, I am a qualified person as defined in National Instrument 43-101.

I have visited the Skouries Project site on August 30 to August 31, 2015.

I was responsible for developing the mine plan for the Skouries Project in Greece. I am responsible for the preparation or supervising the preparation of Sections 15.1, 15.3, 16.1, 16.2, 16.3, 16.4 and 16.5 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the items for which I am responsible in this report entitled, *Technical Report, Skouries Project, Greece*, with an effective date of January 1st, 2018, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 15th day of March 2018.

"Signed and Sealed"

John Nilsson

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John Nilsson, P. Eng.