Skouries Cu/Au Project, Greece

NI 43-101 Technical Report

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Prepared for

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1 SUMMARY

1.1 Executive Summary

1.1.1 Introduction

European Goldfields Limited ("EGL" or the "Company") and Hellas Gold SA ("HG") wish to revise the Mineral Resource and Mineral Reserve estimate disclosed in the National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") Technical Report authored by SRK ("2007 SRK Report") filed by EGL in May 2007 on the HG-owned Skouries copper-gold project in Greece (the "Project").

The 2007 SRK Report was based on a combined open pit and underground mine using a sub level caving ("SLC") mining operation whereas this update considers an open pit and underground mine using a Sub Level Open Stoping ("SLOS") method with backfill.

URS/Scott Wilson Ltd (Scott Wilson) has been retained by EGL to undertake an updated mining study on the Project. Scott Wilson understands that EGL has conducted previous work including definitive feasibility study level work on the open pit and underground mine and basic engineering work on the plant and infrastructure. The work conducted by Scott Wilson in this revised report includes a re-classification of the Mineral Resources and Mineral Reserves (though not the Mineral Resource model) and pre-feasibility level engineering on the underground mine to incorporate the SLOS underground mining method and also updating of the open pit schedule and costs (the "Technical Report"). EGL's subsidiary HG has submitted an environmental impact study ("EIS") that includes the 2007 SRK Report open pit design, therefore the SRK open pit design remains as described in the previous work for this study and this EIS has now been approved by the relevant ministries in Greece.

The following reports have been used for the basis of this Technical Report.

- Skouries Project Geological Resources and Mining Reserves, December 1998 (part of the 1998 feasibility study by TVX), TVX Gold Incorporated (1998 Mineral Resource Model)
- Technical Report on the Skouries Project, May 2007, SRK Consulting Ltd. ("2007 SRK Report")
- Skouries Project, Sub Level Open Stoping Underground Mine Design, April 2007, by Diogo Caupers. ("SLOS Study")
- 2007 Cost and Definition study prepared by Aker Kvaerner, (subsequently Aker Solutions and now Jacobs Engineering Group), in 2007
- Open Pit Mining Study, November 2008, URS Scott Wilson Ltd
- Skouries Cu/Au Project, Mining Method Option Study, August 2010, URS Scott Wilson Ltd

1.1.2 Property Description and Ownership

The Skouries copper-gold deposit is situated within the Serbo-Macedonian Massif, about 100 km by road from Thessaloniki, the second largest city in Greece. The deposit is located on a ridge with a flat crest and forms the catchment boundary between two river systems, the average elevation is approximately 620masl. The Project is located within concession numbers OP03, OP04, OP20, OP38, OP39, OP40, OP48, OP57, which have a combined area of 55.1 km^2. The concessions have been granted until 26th March 2026.

The deposit is 100% owned by HG, who is in turn 95% owned by EGL.

1.1.3 Geology and Mineralisation

The Skouries deposit is hosted predominately in a near-vertical pipe-shaped porphyry stock that has a surface expression of approximately 200 m in diameter. The central portion of the deposit contains two high-grade areas, one near the surface and a second approximately 350 m from surface. Below an elevation of 110masl (some 700 m below surface), the core of the porphyry is low-grade however high-grade zones exist around its perimeter. Below 200masl, the deposit starts to spread into the surrounding host rock and may be associated with porphyry branches or dykes which have developed from the central core. This portion of the deposit contains lower metal grades. Drilling to date shows the porphyry is open at depth, with ore grade mineralisation to a depth in excess of 800m.

1.1.4 Mineral Resource and Mineral Reserve Estimates

Scott Wilson has reviewed and revised the Mineral Resource model and defined Mineral Reserve estimates to report the current open pit and underground Mineral Resource and Mineral Reserve estimates.

The Mineral Resource NSR model is based on higher Mineral Resource related metal price assumptions. The Mineral Resource and Mineral Reserve estimates conform to the Canadian Institute of Mining's ("CIM") 'Definitions Standards for Mineral Resources and Mineral Reserves,' as prepared by the CIM Committee on Mineral Resources and Mineral Reserves on November 27, 2010 ("CIM"). In Scott Wilson's opinion, the Mineral Resource and Mineral Reserves classification results for the Skouries deposit discussed below are also in accordance with the Australasian Code ("JORC, 2004"). The open pit and underground Mineral Resource estimates are summarized in Table 1-1 .

Table 1-1 : Skouries Deposit Mineral Resources (as of 14th July 2011)

Location	Category	Metric Tonnes	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz)	Contained Cu (metric tonnes)
Open Pit	Measured	15,333,000	1.05	0.59	516,000	91,000
	Indicated	49,747,000	0.44	0.34	708,000	171,000
	Inferred	72,307,000	0.15	0.17	350,000	120,000
Underground	Measured	24,147,000	1.36	0.73	1,055,000	175,000
	Indicated	157,123,000	0.61	0.49	3,067,000	768,000
	Inferred	43,470,000	0.34	0.39	477,000	167,000
Total Measured and Indicated		246,350,000	0.67	0.49	5,346,000	1,205,000
Total Inferred		115,777,000	0.22	0.25	828,000	288,000

1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources were estimated using an Au price of US$1,200 per ounce and a Cu price of US$3.50/lb.
3. Mineral Resources were estimated using a NSR discard cut-off value of €4.29/t for the open pit and a breakeven NSR cut-off value of €19.43/t for the underground.
4. The values for tonnages, grades and contained ounces have been rounded.

Measured and Indicated Mineral Resources contained within the Skouries deposit that have been converted into Proven and Probable Mineral Reserves are estimated as of January 2011, in accordance with the requirements of NI 43-101 and the CIM definitions. As discussed, the Mineral Resource and Mineral Reserve estimates also conform to JORC. The Mineral Reserves have been used for the SLOS life of mine plan and economic analysis. Mineral Reserves as shown in Table 15-1 have been estimated assuming an open pit mining method and SLOS method with backfill for the underground operation. The Mineral Resources are inclusive of the Mineral Reserves.

The stope sizes, layouts and sequencing are described in Section 16. Mineral Reserves for the open pit and underground sections are shown in Table 1-2.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-2 : Skouries Deposit Mineral Reserves (as of 14th July 2011)

Location	Category	Metric Tonnes	Grade Au (g/t)	Grade Cu (%)	Containe d Au (oz)	Contained Cu (metric tonnes)
Open Pit	Proven	15,166,000	1.06	0.60%	516,000	91,000
	Probable	31,816,000	0.52	0.37%	530,000	119,000
Underground	Proven	19,278,000	1.40	0.74%	866,000	143,000
	Probable	72,102,000	0.72	0.53%	1,678,000	385,000
Total Proven and Probable		138,362,000	0.81	0.53%	3,590,000	736,000

Notes:

1. CIM definitions were followed for Mineral Reserves.
2. Mineral Reserves were estimated using an Au price of US$1,000 per ounce and a Cu price of US$2.50 per lb.
3. Mineral Reserves were estimated for the open pit using a NSR cut-off value of €4.29/t and the underground using a NSR cut-off value of €19.43/t.
4. The values for tonnages, grades and contained ounces have been rounded.
5. Open Pit dilution averages 0% and extraction is estimated to be 100%.
6. Underground dilution averages 3.5% and extraction is estimated at 95%

1.1.5 Status of Exploration and Development

EGL has not undertaken any exploration drilling at Skouries. The historical owners, TVX Gold Incorporated ("TVX"), undertook 72,232 m of drilling in three phases during 1996, 1997 and 1998. In 1998, TVX also commenced an exploration adit which was suspended during January 1999 after 689 m of development had been completed. It was flooded in October 2000.

Skouries is a large low grade deposit, with Project economics heavily dependent on the production rate and production costs. A Bankable Feasibility Study ("BFS") was prepared in 2006 by EGL based on an open pit operation to a depth of 240 m below surface, followed by an underground mine accessed by a vertical shaft and surface access ramp. The selected underground mining method was SLC at a production rate of 7.0 million metric tonnes per annum. The 2007 SRK Report updated the costs and prices of the BFS.

The SLOS Study was prepared in 2007 by Diogo Caupers at the request of HG, which looked at the possible use of 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 with the objectives of minimising the amount of surface tailings disposal and reducing the potential subsidence area and so to minimise the overall environmental impact of the Project.

EGL has recently concentrated on updating the mining method and mineral processing and metallurgical testing as described below.

1.1.6 Mining

Initial production will come from an open pit operation. The underground mine will consist of the orebody below the base of the open pit at 420 m level (240 m below surface), followed by a SLOS mining method commencing below the 30 m crown pillar down to the -105 masl level.

The deposit will be accessed from surface by a service decline ("Ramp") and a production shaft. A number of production levels will be developed from the access ramp. Production levels will be vertically spaced from each other at 25 m intervals

Each production level will have an auxiliary and permanent ventilation system to provide adequate amounts of fresh air for the safe and efficient execution of mining activities.

To allow a smooth transfer of production from open pit to underground without a production gap, the mine accesses, ore handling, crushing and hoisting facilities and dewatering systems will be developed and equipped prior to the start of production.

A conventional open pit mining operation, utilising an owner operator scenario, with a production rate of 8.0 Mtpa run-of-mine ore is envisaged for the Project. The mining method will consist of drilling, blasting, loading and hauling of ore and waste materials for processing and waste disposal. The primary mining equipment fleet will consist of conventional diesel powered open pit mining equipment, including;

- 2 No. 13 m^3 class 180 t hydraulic excavators.
- 11 No. 90 t to 100 t class off-road haulage trucks.
- 2 No. DTH 140 mm to 165 mm blast hole drilling rigs capable of a single pass of 12 m.

The open pit is circular in shape intersecting the natural ground surface at the 620 m and 670 m elevation points, resulting in the highest open pit face of approximately 250 m in height with the open pit floor at the 420 m elevation. Overall open pit slope angles vary from between 40º and 44º.

The open pit mine production schedule has been developed on a planned annual ore production rate of 8.0 Mtpa. An open pit mining operation of 345 day per year consisting of three, eight hour shifts operating 7 days a week is envisaged, resulting in a daily average ore mining rate of 23,200 tpd.

Table 1-3 : Open Pit Production Schedule Table

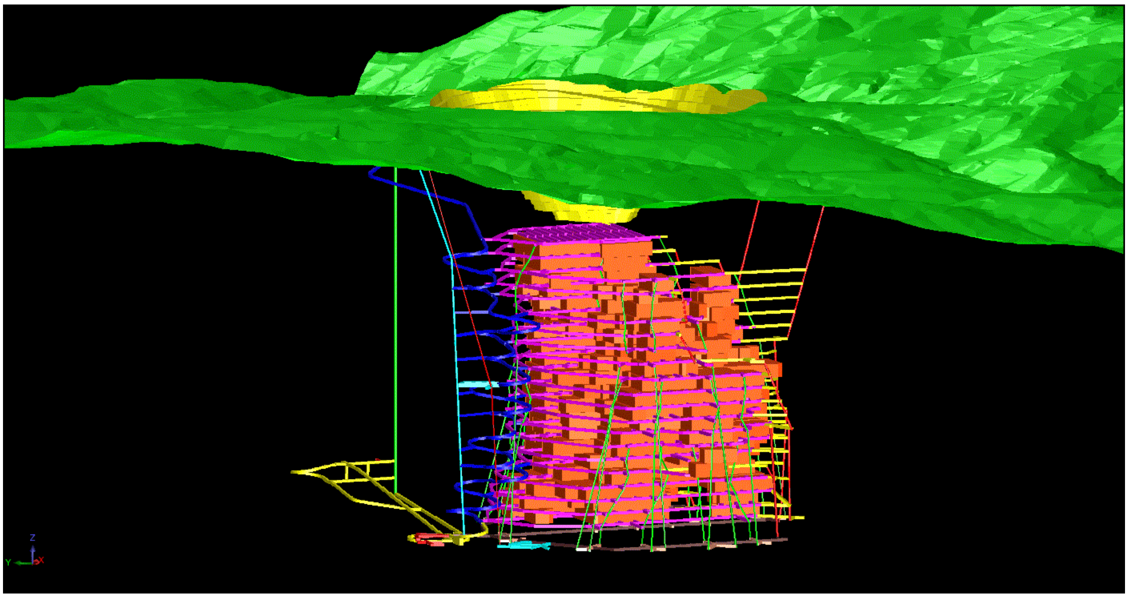
Year	-1	1	2	3	4	5	6	Total
Ore (t 000s)	2,018	6,023	8,052	8,030	8,030	8,030	6,799	46,982
Grade Au g/t	0.47	0.69	0.75	0.66	0.66	0.65	0.81	0.69
Grade Cu%	0.28	0.38	0.51	0.44	0.42	0.43	0.51	0.44
Waste (t 000s)	6,012	7,531	8,624	6,340	3,786	2,251	458	35,001

It is intended that some of the excess tailings from the underground mining operation will be mixed with cement and disposed of in the open pit as engineered backfill. This means that mining of the open pit has to be complete before the option of diverting the disposal of tailings away from the Tailings Management Facility ("TMF") valley site and in to the open pit can be used.

Underground production commences in Year 7 with pre-production of the underground mine commencing in Year 4. Production averages 4.4 Mtpa over most of the underground mine life. The mine design is based on three operational levels that will be mined simultaneously, accessed by a main ramp, positioned close to the orebody. The base level for each mining horizon will be linked to an exhaust vent raise, creating a main return airway. Intermediate ventilation raises will link each level with the closest main return airway drift. A main intake ventilation raise, close to the main ramp will be one of the three main air intakes (shaft, main ramp and intake ventilation raise).

Figure 1-1 shows the underground capital development for the mine in relation to the open pit.

Figure 1-1 : Underground Development Layout



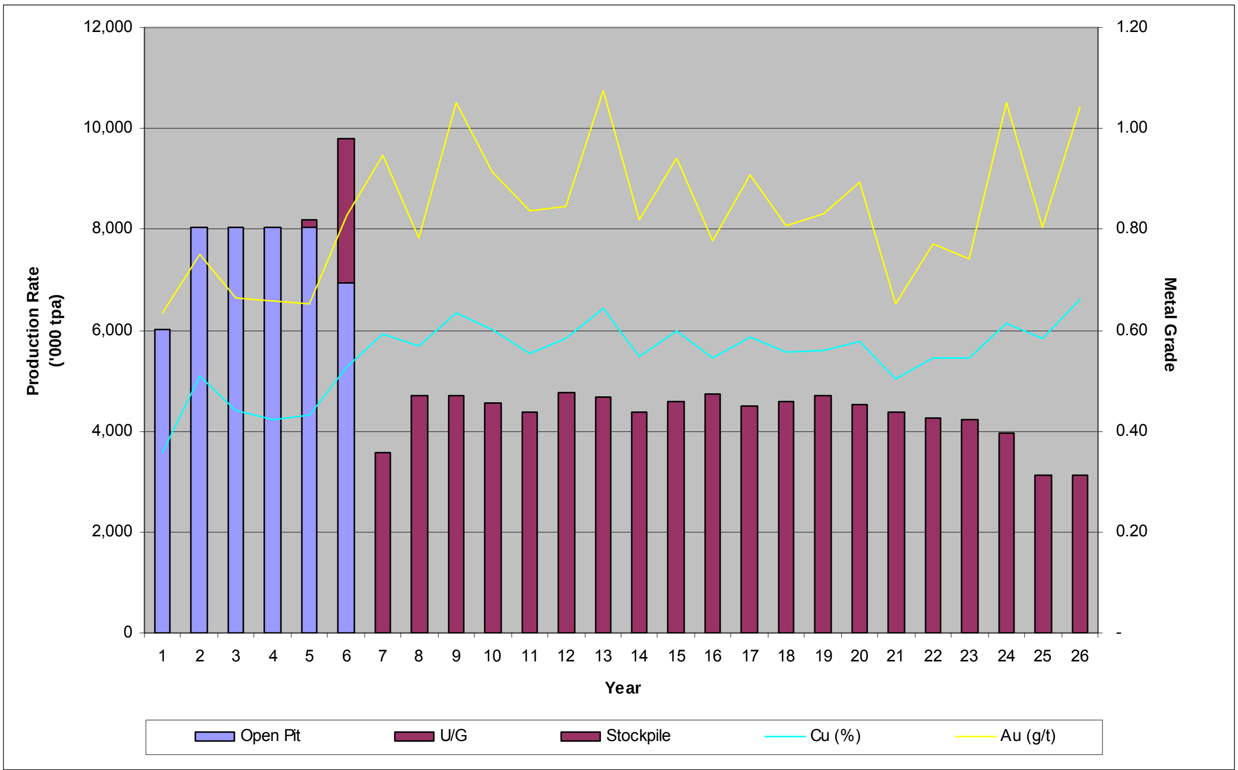
Based on the available geotechnical data the primary and secondary stope dimensions were estimated as follows:

- Width – 15 m;
- Height – 25 m;
- Length – nominal 80 m to 100 m.
- Average number of 64 stopes ranging from a minimum of 41 stopes to a maximum of 75 stopes per level.

All mining horizons are arranged from the bottom of the open pit (420 masl level, a 30 m thick crown pillar from pit floor to roof of stope, incorporating 5 m high drilling drives in the pillar, is left un-mined) down to the – 105 masl level (the limit of Indicated Mineral Resources).

The production rates, showing the ROM metal grades for the open pit and underground mine are shown below in Figure 1-2 .

Figure 1-2 : Open Pit and Underground Production Schedule



The ore produced from the development in Years 4, 5 and 6 shall be stockpiled on surface and fed into the plant in Years 6 and 7 to maintain as near to the 8 Mtpa plant capacity.

1.1.7 Mineral Processing

The process plant and infrastructure design of the Project has been based on extensive testwork carried out on samples that were representative of the resource. Technical information was provided by several specialist consultants, world class metallurgical testing facilities and international Engineering groups.

Outotec of Finland ("Outotec"), have completed an Engineering Study for the project which included the supply of equipment within their manufacturing range, grinding mills, the flotation equipment, the paste thickeners and the plant control system. In parallel with this the Athens based Engineering contractor, ENOIA, completed a Basic Engineering Study and have started Detailed Engineering including aspects of the plant and infrastructure outside of Outotec's scope. ENOIA will provide contract services and controls for all estimate areas of the Project working under the direction of HG.

The layout of the plant has been optimised over time incorporating many improvements which have resulted in capital cost reductions. Confirmatory geotechnical assessment is scheduled.

The Process Plant is of conventional design comprising surface ore reception facilities and primary crusher, coarse ore stockpile, SAG and Ball Mill grinding, gold gravity circuit, rougher, cleaning and scavenger flotation stages, filtration and paste thickening of the tailings for disposal. In addition, the infrastructure facilities include, the administration block, the workshops, fuel station and welfare facilities as well as power, water and other services. The design will take into account the ore delivery system from the underground phase of mining.

The TMF has been designed by Omikron Kappa Consulting Limited, ("Omikron"), which incorporated earlier engineering work by Golder Associates..

Two tailings dams are planned to be constructed in Karatza Lakkos and Lotsaniko stream valleys and will store the tailings produced by the processing of Skouries open pit mining ore, while the waste produced by the open pit mining should be used for their construction.

Flotation tailings generated from the underground mining phase will be returned to the mine as backfill which is a requirement of the SLOS method selected.

The Project entails open pit mining for the first 6 years. The feed rate will be 8 Mtpa for Years 1 to 5 and in Year 6 the remaining 7.0 Mt open pit ore will be processed. From Year 7 the plant will be supplied from the underground mining operation with 6.4 Mt and thereafter at between 3.1 Mtpa and 4.8 Mtpa for at least the next 20 years.

During years 1-6 the plant will be treating the softer open pit ore and the selected mills will accommodate the required tonnage.

The process plant design has assumed a nominal throughput for the harder underground material of approximately 881 tph. The plant design exceeds the output from the underground mining and plant operations will therefore be based on campaign treatment of the ore so that the plant will operate at optimum efficiency as well as allowing higher throughput if an alternative more productive extraction approach is adopted or additional ore streams are identified by exploration of the Company's nearby porphyry exploration targets.

Initially, for the first year of operation, plant feed will comprise oxidised ore arising from the open pit. As the near surface oxidised ore is depleted there will be a transition to sulphide ore and for the next five years sulphide ore will be mined from the pit. Thereafter the mining operation will move underground where sulphide ore will be extracted. The primary products from the process plant will be a high quality gold-copper concentrate and gold doré ingots which are estimated to contain approximately 30% of the gold contained in the ore feed.

Two design cases were evaluated by Outotec; the first being the nominal design case processing underground sulphide ore, while the second case reflects the higher tailings and concentrate production scenario:

Table 1-4: Evaluated Design Cases for Plant Study

	NOMINAL DESIGN (CASE 1)	DESIGN CASE 2
Concentrator Fresh Feed Rate (t/h) & Design Factor (X)	881 (1.00)	1,013 (1.15)
Cu in feed (%)	0.54	0.54
Au in feed (g/t)	0.83	0.83
Basis	Life of Mine averages	Life of Mine averages
% Cu in rougher conc.	5	5
% Cu overall recovery	91.1	91.1
% Cu in final conc.	26	26
% Au recovered by gravity	30	30
% Au overall recovery	84.0	84.0
SAG pebble circulating load, %	0 – 25	50

The Process Plant configuration is well established worldwide for treating porphyry copper ores and as such offers a well proven design using conventional equipment. This coupled with the straight forward ore metallurgy and design margins will ensure that the Plant will be a robust, low risk processing solution to the treatment of the Skouries ore throughout the life of the project. There are many plants operating successfully in the world using the Skouries design.

1.1.8 Economic Analysis

A pre-tax cash flow model was generated for the Project using economic estimates of the Project capital expenditure requirements and annual operating costs, as estimated in this Technical Report for the life of mine production schedules.

Costs have been estimated to an overall accuracy of +/-30%.

The base case cash flow is based on the following parameters:

Physicals

- 345 production days per year;
- 23,200 t of ore per day mining from open pits (8.0 Mtpa);
- 13,000 t of ore per day mining from underground (4.5 to 4.8 Mtpa);
- Average process recovery of 84% of gold and 90% of copper;
- Net smelter return of 97% for gold and 87% for Copper;
- Metal prices for Mineral Reserves of US$1,000/oz Gold and US$2.50/lb Copper;
- Mine life of 27 years based on the Mineral Reserve estimate.

- Under the terms of the existing mining licences, royalty is not payable for Skouries and taxes in Greece are currently 25%, reducing to 20% by 2014. Depreciation in Greece is applicable at the rate of 15% per annum.

A cash flow model using a gold and copper price of US$1,000 per ounce and US$2.50 per pound respectively and a discounted cashflow rate ("DCF") of 5% was prepared in order to examine the Project economics and the Net Present Value ("NPV"). All quoted dollar values are United States dollars unless stated otherwise.

The following key dollar inputs were used:

Table 1-5 Key Cashflow Parameters (Dollars)

	Units	Life of Mine	Average
ROM production	kt	**138,362**	**5,125**
ROM production gold grade	g/t	**0.81**	**0.81**
ROM production copper grade	%	**0.53%**	**0.53%**
Au Contained in Concentrate	k oz	**2,925**	**108**
Cu Contained in Concentrate	kt	**646**	**24**
Operating Costs	US $	**2,763.5**	**102.4**
Capital Costs	US $	**615.0**	**23**
Net Revenue (1)	US $	**6,144.5**	**228**
Undiscounted Operating Income	US $	**3,381.0**	**125**
Discounted cashflow @5%	US $	**1,399.3**	**52**

Note: (1) Revenue after deducting payability, TC/RC's, transport, deductions and marketing costs.

Using these parameters in a discounted cash flow model the Project gives a pre tax internal rate return of greater than 35% which meets the EGL company criteria for a viable project.

A sensitivity analysis was carried out to model potential fluctuations of key input parameters from the base case cash flow model.

The following parameters were evaluated over a range of a 20% increase to a 20% reduction to observe the impact on the Project's NPV:

- gold price;
- gold grade;
- copper price;
- copper grade;
- capital expenditure;
- operating expenditure;

- processing plant gold recovery was evaluated over a range of a 4% increase to a 4% decrease;
- processing plant copper recovery was evaluated over a range of a 4% increase to a 4% decrease.

The Project economics are most sensitive to copper grade and operating cost. Processing Plant recoveries do not significantly affect economics due to fluctuations within the ranges evaluated.

1.1.9 Conclusions

The authors of the report conclude that the level of data adequacy is considered sufficient for reporting Mineral Reserves, but further work is required particularly on the underground design to reach the level of a full and accurate feasibility study. Based upon the assumptions in this Technical Report and the work carried out, the authors of this Technical Report are of the view that the Project can be developed into a viable mining operation.

The authors of the report concludes that the Technical Report on the Skouries Project has met its objectives in determining the viability of the Skouries project based upon the work undertaken and assumptions made.

For this study the authors of the report have undertaken a high level review of the 1998 Mineral Resource Model, which led to the following changes:

- Updated the drill hole database with ten TVX holes (SOP-89 to 98) drilled from June to August 1998. These holes mostly targeted the southwest perimeter of the main body of mineralization.
- Updated the 5 m gold and copper composites and interpolated new gold (2010_Au) and copper (2010_Cu) block models using the same interpolation parameters as in 1998.
- Created a new Mineral Resource classification model based on continuous 3D solids.
- Created a solid to sterilize over-extrapolated blocks. The solid is mostly located along the north, northeast, and east flanks in areas with no bounding perimeter drill holes.

Moderate differences in the Mineral Resource model are mostly due to sterilizing over-extrapolated blocks and adding data for the ten drill holes. The new classification model has resulted in a significant decrease in the amount of Measured Mineral Resource material, however, the total Measured and Indicated Mineral Resources has not changed significantly.

1.1.10 Recommendations

The authors recommend that the EGL and HG management team continues with its planned program of progressing the Skouries Project which is outlined below:

Planned works once environmental permits have been approved include:

- Detailed geotechnical drilling of load bearing areas of the mill site,
- Geotechnical drilling of the embankment sites of the tailings facilities,
- Geotechnical drilling around the pit, shaft and portal sites,
- Drilling of Inferred Mineral Resources within the open pit area

- Develop basic engineering for the underground mine,
- Issue of the contractor ITB, which has been prepared, for the plant and infrastructure construction.
- Develop the Project team

Geotechnical investigation is required to confirm the geomechanical design parameters adopted in the underground mine design and confirm the location of the shaft.

In line with EU directives, and to ensure that they remain appropriate to the operations, EIS approvals in Greece are refreshed every five years to reflect prevailing conditions such as new Mineral Resources and Mineral Reserves, metal prices, employment levels, etc. Accordingly, this presents an opportunity, at that time, for EGL to re-assess the extraction approach, so as to ensure that in the context of the then prevailing conditions, the best available techniques are applied for the remainder of the mine life.

2 INTRODUCTION

This Technical Report has been prepared by Patrick Forward, Antony Francis and Scott Wilson for EGL. The information, conclusions, opinions and estimates contained herein are based on:

- Information available to Scott Wilson at the time of preparation of this Technical Report;
- Assumptions, conditions and qualifications as set forth in this Technical Report; and
- Data, reports and other information supplied by EGL, HG, Scott Wilson and other third party sources.

EGL and HG wish to revise the Mineral Resource and Mineral Reserve estimate disclosed in the 2007 SRK Report on the HG owned Skouries copper-gold project in Greece.

The 2007 SRK Report was based on a combined open pit and underground mine using a SLC mining operation.

This update considers an open pit and underground mine using a SLOS method with backfill.

Scott Wilson has been retained by EGL to undertake a mining study on the Project. Scott Wilson understands that EGL has conducted previous work including definitive feasibility study level work on the open pit and underground mine and basic engineering work on the plant and infrastructure. The work conducted by Scott Wilson includes a re-classification of the Mineral Resources and Mineral Reserves (though not the Mineral Resource model) and pre-feasibility level engineering on the underground mine to incorporate the SLOS underground mining method and also updating of the open pit schedule and costs. EGL's subsidiary HG has submitted an EIS that includes the 2007 SRK Report open pit design, therefore the SRK open pit design remains as described in the previous work for this study.

A further study was prepared in 2007 by Diogo Caupers at the request of HG, which looked at the possible use of Sub-Level Open Stoping "SLOS") with tailings backfill. The objective of the SLOS Study was to review the possibility of mining the deposit by the SLOS method as an alternative method to SLC, using tailings as backfill with the objectives of minimising the amount of surface tailings disposal and reducing the potential subsidence area and so to minimise the overall environmental impact of the Project.

For this current study, Scott Wilson has been appointed by EGL to assist EGL to update the Mineral Resource and Mineral Reserve estimates based on an open pit followed by a SLOS underground mining method. The study also uses current cost estimates and economic factors.

2.1 Sources of Information

Site visits were carried out by Patrick Forward, BSc, MAusIMM, Vice President Projects & Exploration, European Goldfields Limited, David JF Smith, C.Eng.IMMM, UK Director of Operations – Mining, URS Scott Wilson, Antony Francis, BSc, FIMMM, Senior Metallurgist, European Goldfields Limited. It should noted that despite David JF Smith not visiting the site since 2006 it is noted that the site is a Greenfield location and no works have been completed since that time.

Detailed discussions were held with personnel from EGL, both on site and in subsequent meetings and telephone calls.

Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 18, 19, 20, 21, 23, 24, 25, 26 and 27 of the report were prepared by Mr. Forward. Sections 13 and 17 of the report was prepared by Mr. Francis. Sections 15, and 16 of the report were prepared by Mr Smith. The documentation reviewed, and other sources of information, are listed at the end of this Technical Report in Section 27 References.

2.1.1 List of Abbreviations

Units of measurement used in this Technical Report conform to the SI (metric) system. All currency in this Technical Report is in Euros ("€") unless otherwise noted.

μμ	micron
°C	degree Celsius
°F	degree Fahrenheit
μγ	microgram
A	ampere
a	annum
AAS	atomic absorption spectrometry
Au	gold
bbl	barrels
Btu	British thermal units
C$	Canadian dollars
cal	calorie
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
Cr-Mo	chromium-molybdenum
Cu	copper
d	day
dia.	diameter
dmt	dry metric tonne
dwt	dead-weight tonne
ft	foot
ft/s	foot per second
ft^2	square foot
ft^3	cubic foot
g	gram
G	giga (billion)
Gal	Imperial gallon
g/L	gram per litre
g/t	gram per tonne
gpm	Imperial gallons per minute
g/cc	grams per cubic centimetre
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
hr	hour
ha	hectare
hp	horsepower
in	inch
in^2	square inch
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometre
km/h	kilometre per hour
km^2	square kilometre
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour

L	litre
LHD	load haul dumps
L/s	litres per second
m	metre
M	mega (million)
m^2	square metre
m^3	cubic metre
min	minute
masl	metres above sea level
mm	millimetre
Mm^3	cubic millimetre
mph	miles per hour
Mt	million tonnes
mtpa	million tonnes per annum
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
m^3/h	cubic metres per hour
opt, oz/st	ounce per short tonne
oz	Troy ounce (31.1035g)
oz/dmt	ounce per dry metric tonne
PGMs	platinum group metals
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
s	second
SAG	semi autogenous grinding
si	Le Systeme Internationale d'Unites adopted by the Conference General de Poids et Mesures, which is the international authority on the metric system
st	short tonne
STD	standard deviation
stpa	short tonne per year
stpd	short tonne per day
Tonne (t)	metric tonne
tpa	metric tonne per year
tpd	metric tonne per day
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
W	watt
wmt	wet metric tonne
wt	weight
w/w-%	weight for weight
yd^3	cubic yard
yr	year

3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by Patrick Forward and Antony Francis of EGL and Scott Wilson for EGL. For the purpose of this Technical Report, following reports are the main sources of other expert information:

- Skouries Project Geological Resources and Mining Reserves, December 1998 (part of the 1998 feasibility study by TVX), TVX Gold Incorporated (1998 Mineral Resource Model)
- Skouries Project Feasibility Study by TVX and Kvaerner Metals for TVX Hellas, September 1998.
- November 2008 – Skouries Open Pit Mining by Study Scott Wilson and Steve Nicol
- Skouries Project, Sub Level Open Stoping Underground Mine Design, April 2007, Diogo Caupers (“SLOS Study”)
- 2007 Cost and Definition study prepared by Aker Kvaerner, (subsequently Aker Solutions and now Jacobs Engineering Group), in 2007
- Open Pit Mining Study, November 2008, URS Scott Wilson Ltd
- Technical Report on the Skouries Project, May 2007, SRK Consulting Ltd. (“2007 SRK Report”)
- Skouries Project Basic Design Package for Hellas Gold S.A. by ENOIA S.A., Job No. EN-515, June 2009.
- Final Issue for Basic Engineering Skouries Concentrator Plant for Hellas Gold S.A. by Outotec O.Y.,Project No. DQ070049, March 2009.

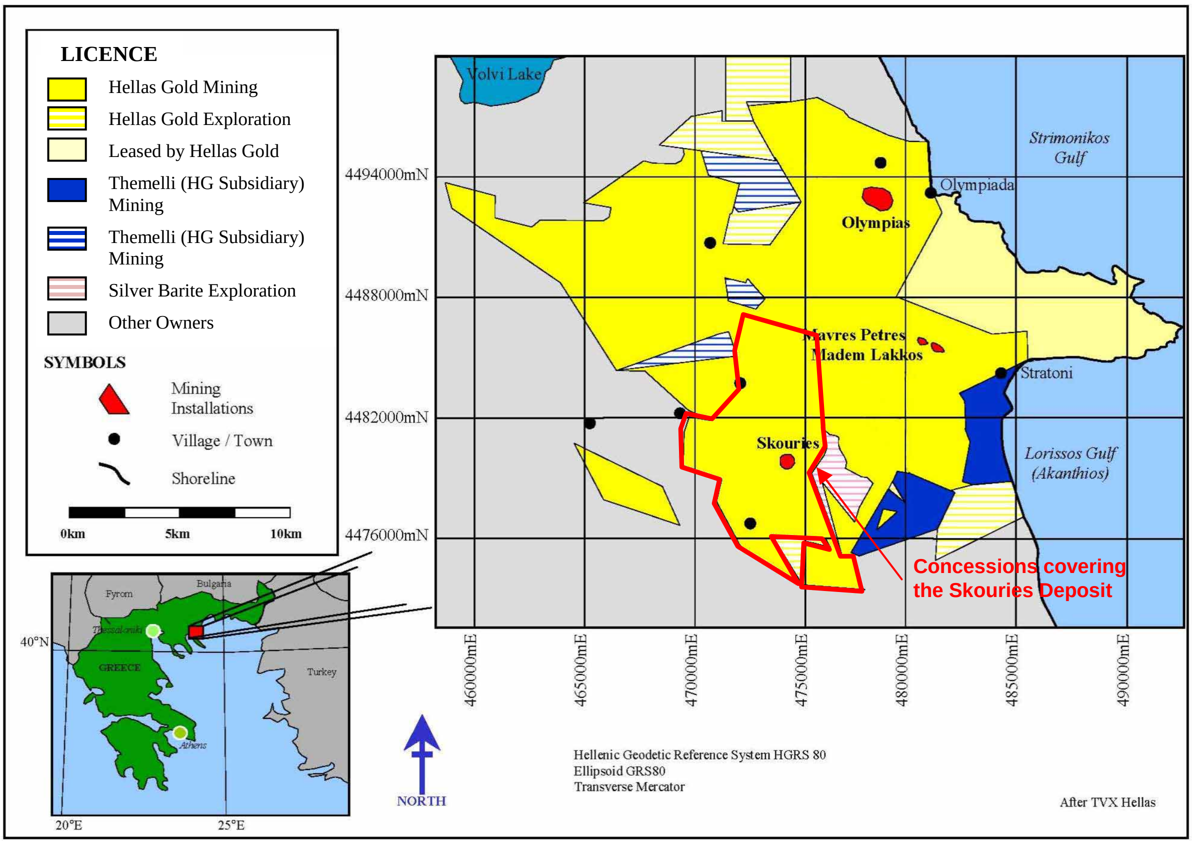
Scott Wilson has not researched property title or mineral rights for the Skouries deposit and expresses no opinion as to the ownership status of the property. Patrick Forward has reviewed the mineral right titles pertaining to the Skouries Project and verified that they are as stated in this report.

Except for the purposes legislated under provincial securities laws, any use of this Technical Report by any third party is at that party’s sole risk.

4 PROPERTY LOCATION AND DESCRIPTION

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, approximately 100 km east of Thessaloniki (Figure 4-1). The area is centred on co-ordinates 474000E and 4488000N of the Hellenic Geodetic Reference System HGRS '80, Ellipsoid GRS80, (approximately Latitude 40° 36' and Longitude 23°50'). The concessions include the Olympias Mine, which is currently on care and maintenance, Madem Lakkos and Mavres Petres Mines (collectively known as "Stratoni") which are currently in production and the Skouries copper-gold porphyry deposit. All three of these projects are collectively referred to as the Kassandra Mines Mineral Deposits Project.

Figure 4-1 : Property Location



4.1 Project Ownership

The Project was acquired from the Greek state in 2004 by HG; EGL currently holds a controlling interest of 95% in HG.

4.2 Land Tenure

The Project is located within concession numbers OP03, OP04, OP20, OP38, OP39, OP40, OP48, OP57 which have a combined area of 55.1 km^2. At the time of writing this report, EGL currently owns part of the surface rights over these concessions, amounting to approximately half the rights required. The remaining surface rights will be acquired during project development and the cost of this is covered in the owners capital costs estimate. The concessions have been granted until 26th March 2026. The concessions are conditionally renewable for a further two consecutive periods of 25 years each. No royalty is payable on future production.

The Greek Government co-ordinated by the Ministry of Environment, Energy and Climate Change has approved the EIS in respect of the development of the Project. This is the final key permit required for advancing the Project.

In addition the approved EIS for the projects, in order for production to commence construction and operating permits via submission of a technical study to the Greek Authorities are required. There are no known environmental liabilities attached to the property and there are no expenditure commitments

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

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 an airport with domestic and international scheduled flights. The Project area is readily accessible by car and bus, with regular bus schedules. The road network in the area is among the best in northern Greece and a major highway extends east from Thessaloniki to approximately 25 km north of the property.

The Project is situated approximately 11 km southwest of the Stratoni Port loading facility, 11 km south of the town of Palaeohori and 3 km northeast of the village of Megali Panagia. It lies on the southern edge of a gentle plateau with average elevation of 620 m. The highest point near the project is 685 masl. Steep valleys that drain towards the east and south dissect the edge of the plateau. Access from a national road is via 5 km of good gravel road.

The area is heavily wooded with oak, beech and pine being the principal species, whilst inland there are vineyards and fertile farmlands. The main farming products 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, whilst the highest temperatures occur during summer months ranging between 23°C and 34°C. Temperatures below 0°C are limited to the mountainous areas.

The area is well served by main power supplies via the Public Power Corporation ("PPC") and main power lines come to within four kilometres of the site. Communications are good; broadband is available and HG also has a back-up microwave phone link 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 metres below surface around the deposit.

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

The total land take of the Project is 211.8 ha of contiguous land. HG already owns some 12% of this area. A further 83% of the land comprises private and public forestry which will be leased. The remaining 5% will be purchased as needed during the construction period. This land take relates to the site layout diagrams given in this report for the mining, plant, waste dump, access and internal roads, tailings facilities and other associated infrastructure.

There are currently no facilities at the Skouries site.

6 HISTORY

There is a long history of mining in the area. Ancient mining reached a peak during the time of Phillip II and Alexander the Great, at which time silver and gold financed their conquests of the then known world 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 Turkish periods. Ancient mining is less well documented at Skouries.

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 below.

TVX Gold Incorporated ("TVX") began a drilling programme in August 1996 to confirm the deposit and to explore it at depth. A subsequent infill drilling programme 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 by NCL Ingenieria y Construccion S.A, SRK Consulting Cardiff and Kvaerner Metals Stockton and Kvaerner Metals Toronto in September 1998 with an updated EIS in February 1999.

Table 6-1 : Historical Mineral Resource, TVX/Kvaerner 1998 (Audited and Classified by EGL, 2007)

Category	Metric Tonnes	g/t Au	% Cu
Mineral Resources			
Measured	180.4	0.83	0.55
Indicated	10.8	0.61	0.47
Inferred	14.8	0.60	0.45
Total Measured and Indicated	191.2	0.82	0.55

Note: This resource statement is not being treated as current
Total may not add up due to rounding

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, 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 BFS was prepared in 2006 by EGL based on an open pit operation to a depth of 240 m deep followed by an underground mine accessed by a vertical shaft and surface access ramp. The selected underground mining method was SLC at a production rate of 7.0 Mtpa.

In 2007, SRK undertook 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 and produced the 2007 SRK Report.

Table 6-2 : NI 43-101 Mineral Reserves, SRK 2007

Category	Metric Tonnes	g/t Au	% Cu
Open Pit Mineral Reserves			
Proven	42,500,000	0.71	0.46
Probable	9,700,00	0.60	0.39
Sub-Total	**52,200,000**	**0.69**	**0.45**
Underground Mineral Reserves			
Proven SLC	32,400,000	1.07	0.62
Proven development	2,600,000	1.16	0.66
Probable SLC	55,100,000	0.81	0.57
Probable development	3,900,000	0.90	0.62
Sub-Total	**94,000,000**	**0.91**	**0.59**
All Sources			
Proven	77,500,000	0.87	0.54
Probable	68,700,000	0.78	0.55
Total	**146,200,000**	**0.83**	**0.54**

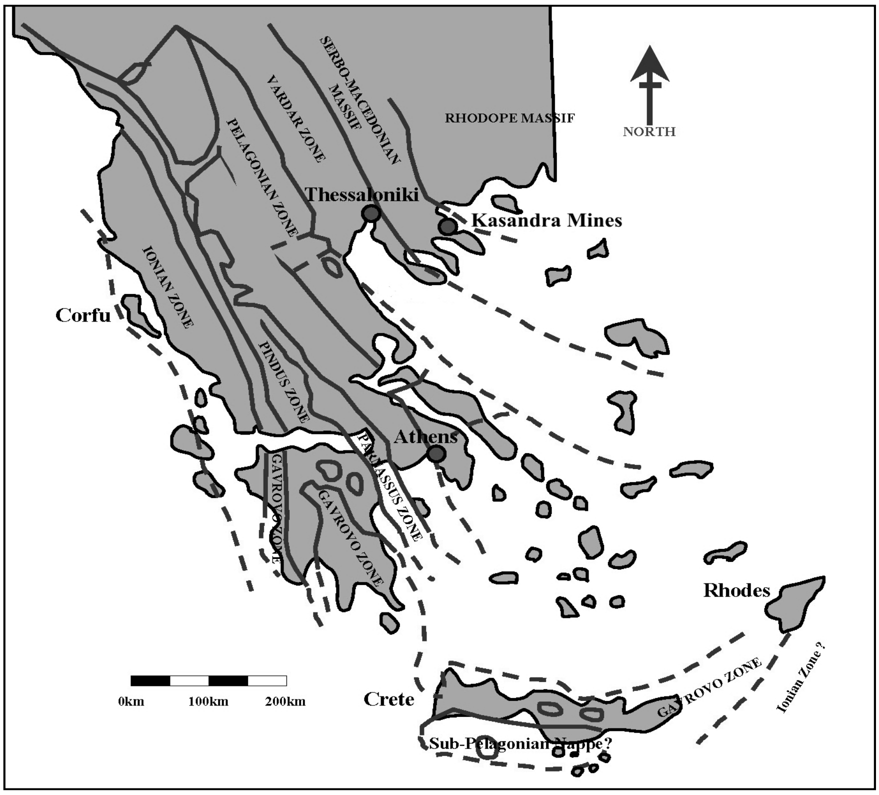
A further study was prepared in 2007 by Diogo Caupers at the request of HG, which looked at the possible use of SLOS with tailings backfill. The objective of the study was to review the possibility of mining the deposit by the SLOS method as an alternative method to SLC, using tailings as backfill to minimise the amount of surface tailings disposal and to reduce the potential subsidence area and so minimise the overall environmental impact of the Project.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The principal mountain ranges of mainland Greece form part of the Dinarotauric arc, a branch of the Alpine orogenic system. The region comprises remnant fragments of the Aegeis landmass. The arc can be subdivided into a series of northwest-trending, linear zones broadly coincident with the main mountain ranges. The zones represent successive subduction episodes resulting from the northeast movement of the African plate in the Jurassic to Eocene period and form distinctive structural units separated by thrusts or transitional zones. In Northern Greece, the Rhodope and Serbo-Macedonian massifs represent the backland beneath which the African plate was subducted. The massif formed an emergent area during the Alpine Orogeny and comprises a complex metamorphic terrain, previously affected by the Variscan Orogeny and earlier events; the Serbo-Macedonian massif comprises schists that are often mineralised and intruded by Variscan granites and hosts the Kassandra mining area which comprises the Skouries gold-copper porphyry and the Stratoni and Olympias massive sulphide deposits. Successive subduction events were accompanied by volcanism and arc-type plutonic igneous intrusions. The Vardar zone comprises a complex belt with ophiolites, rhyolites, limestone and flysch, intruded by Eocene granitic rocks and lies immediately southwest of the Serb-Macedonian massif. Figure 7-1 is a tectonic interpretation of Greece.

Figure 7-1 : Tectonic Zones of Greece



7.2 Local and Property Geology

The general geology of the concession area is shown in Figure 7-2. The deposit is located within the Serbo-Macedonian Massif, which comprises strongly tectonised and metamorphosed Palaeozoic rocks. The massif is locally subdivided into two northwest-trending lithostratigraphic-tectonic units, namely the Vertiskos Formation to the west, which includes amphibolite gneiss flanking biotite schists and interbedded amphibolites, and to the east the underlying Kerdilla Formation, consisting of granitised and migmatised mica gneiss with amphibolite and marble horizons. The units have been intruded by Oligocene sub-alkaline porphyry stocks including the body that hosts the Skouries copper-gold deposit, and are separated by the arcuate Stratoni Fault. Foliated leucocratic migmatites, locally termed pegmatites, occur within the Kerdilla Formation.

The Skouries deposit is a typical sub-alkaline copper-porphyry deposit, forming a near-vertical pipe intruded into amphibolite and biotite schist country rock. The alteration zones at Skouries are restricted in extent in contrast to well developed concentric zones typical of high level porphyries.

The deposit occurs within an elliptical pipe of coarse-grained porphyritic syenite, part of a suite of Oligocene porphyry stocks that intrude the Kerdilla and Vertiskos Formations along a northwest trending belt, Figure 7-3.

Figure 7-2 : General Geology of Concession Area

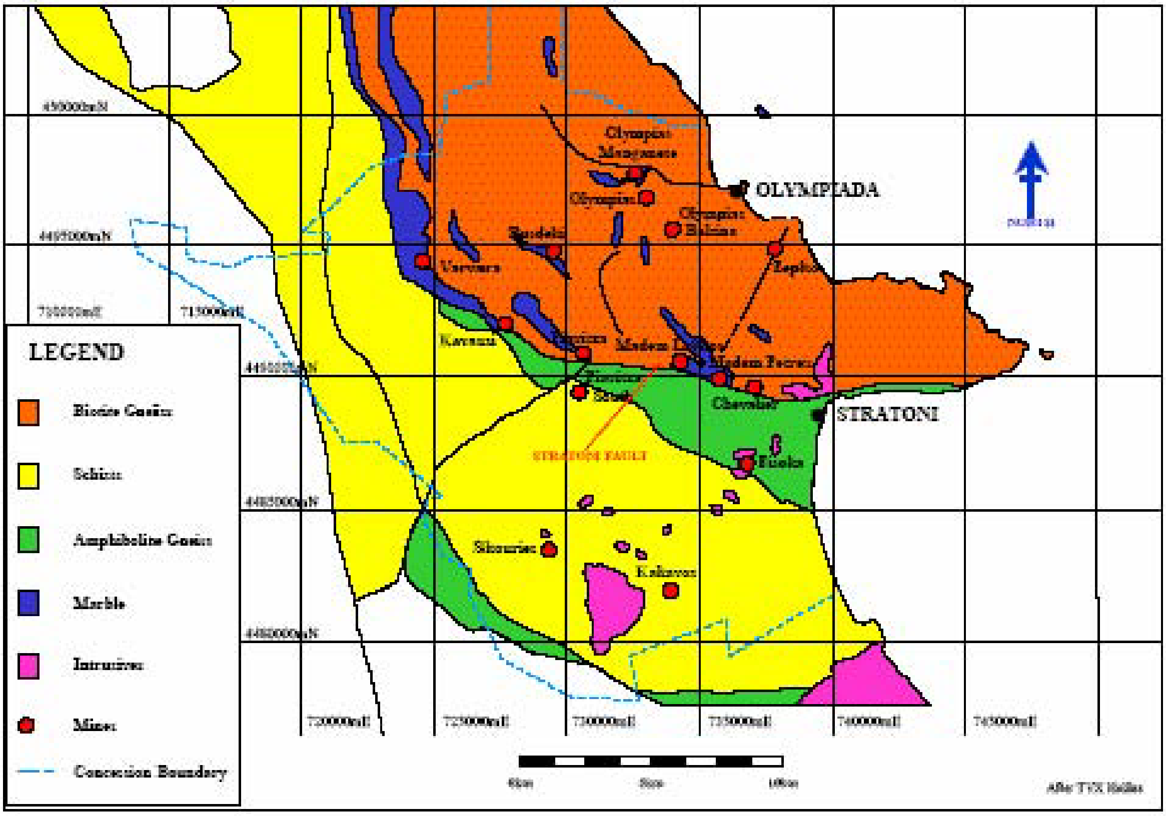
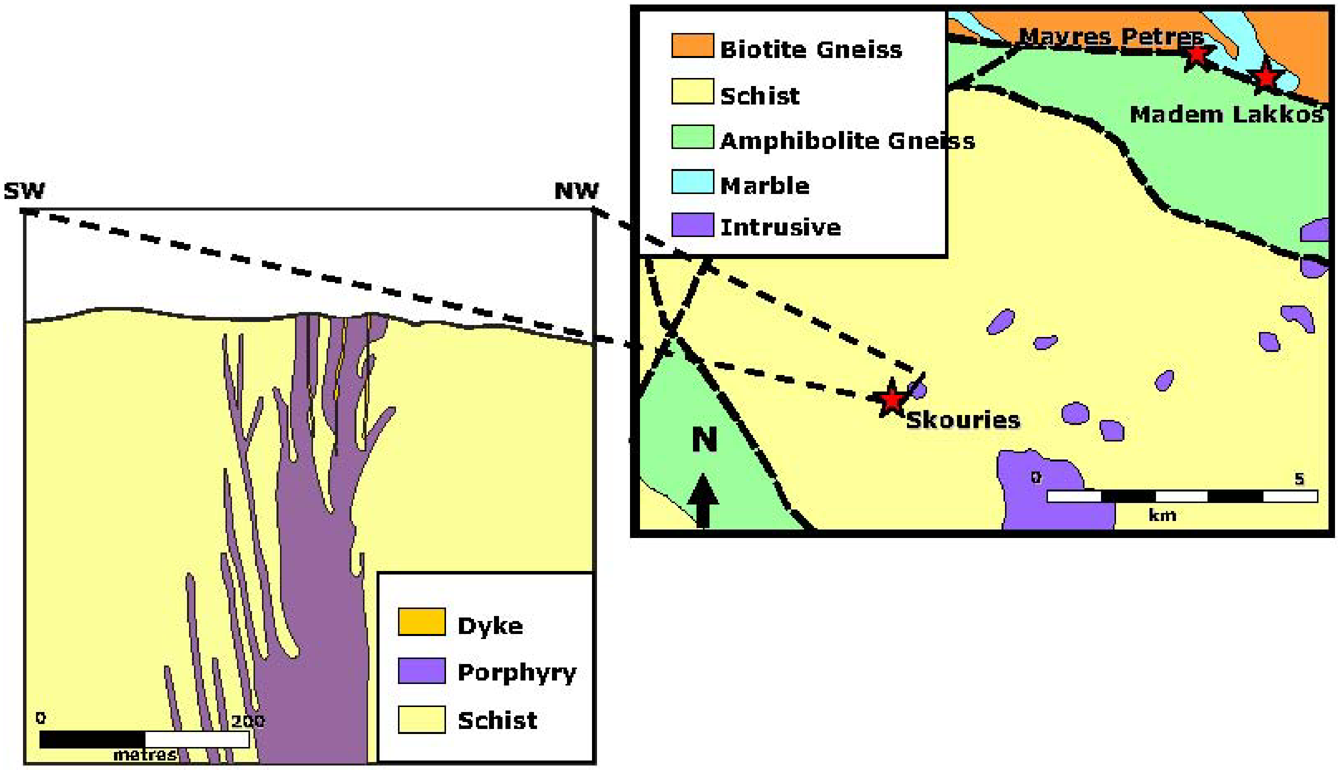


Figure 7-3 : Geological Plan and Cross Section of the Skouries Deposit



7.3 Mineralisation

Mineralisation is disseminated in nature and typical of a porphyry and it is subvertical in orientation. Mineralisation within the potassic zone primarily comprises chalcopyrite veinlets with subordinate bornite (0.1 to 5 mm thick) and disseminated chalcopyrite and bornite. Variable amounts of digenite, chalcocite, covellite, molybdenite (not in economic quantities) and pyrite occur together with rare galena and sphalerite. Magnetite occurs both as disseminations and in quartz veinlets. The propylitic zone contains less than 1% disseminated pyrite and rare chalcocite.

Gold mineralisation occurs as native gold associated with gangue minerals and ranges in size from a few microns to 160 μm. It also occurs as blebs within sulphides, particularly in bornite and chalcocite. Gold correlates strongly with copper. Interestingly, palladium was discovered to occur in the drillholes during testwork and could add value to the deposit.

An oxide zone occurs from surface to 30 to 50 m depth and includes malachite, cuprite, secondary chalcocite and minor azurite, covellite, digenite and native copper. The main porphyry deposit comprises two high-grade areas, one near surface and a second below 350 m depth.

8 DEPOSIT TYPE

The Skouries porphyry comprises a vertical pipe like intrusive which measures some 250 m by 150 m at surface and has been traced to a vertical depth of more than 800 m. Modelling of the deposit is based on this pipe like orientation. Several parallel dykes of similar composition occur to the south of the main porphyry, have widths up to 10 m along strike, lengths of up to 90 m and are interpreted to represent apophyses from the main body. They have pervasively mineralised the host schist and almost double the extent of the mineralised zone below 300 m depth. Both the porphyry and the schist have been intruded by a series of thin (< 5 m) sterile dykes.

The deposit is contained within concentric alteration zones comprising an inner potassic zone, within and surrounding the pipe. The alteration consists of K-feldspar and biotite with quartz and abundant magnetite. Stockwork quartz veinlets are well developed within the zone. An outer propylitic alteration zone affects the host schists and comprises chlorite, epidote, albite and calcite. Weak phyllic and argillic alteration is confined to vein haloes and faulting.

9 EXPLORATION

EGL has not undertaken any exploration drilling at Skouries.

The historical owners, TVX, undertook 72,232 m of drilling in three phases during 1996, 1997 and 1998. In 1998, TVX also commenced an exploration adit which was suspended during January 1999 after 689 m of development had been completed. It was flooded in October 2000.

10 DRILLING

10.1 Drill Hole Data

A total of 72,232 m of core drilled during the 1996-1998 TVX drilling campaign were available for Mineral Resource estimation. During this period TVX drilled a total of 111 long surface diamond drill holes using NQ size (47.6 mm core diameter). Holes reached to 800 m depths. Hole deviation was measured by Sperry Sun every 50 m depth. Drill runs were 3 m. Collar co-ordinates were surveyed using total station DMT-410 in the pit area.

10.2 Core Recovery Statistics

TVX drilling recoveries were logged in metres, within distinctive geotechnical units. Based on this data, recoveries were calculated for each sampled interval.

A total of 31,722 intervals have recovery records, of which 31,513 intervals correspond to porphyry and schist and 209 to sterile dykes.

A summary of a basic statistical analysis is shown in Table 10-1.

Table 10-1 : Core Recovery Statistics

Rock Code	Description	Number Intervals	Minimum Recovery (%)	Mean Recovery (%)	Maximum Recovery (%)	STD (%)	% of Total Population
1	All Porphyry	7,618	10.6	93.4	100	9.4	24.0
2	Schist	23,895	5.7	89.8	100	12.7	75.3
30	Sterile Dyke	209	36.7	92.3	100	10.1	0.7
Total		**31,722**	**5.7**	**90.7**	**100**		**100**

Source: SRK (2007)

Samples contained within the porphyry had an average recovery of 93.4% and represent approximately 24% of the entire sample population.

Average recovery within the schist was slightly lower at 89.8%, representing 75.3% of the population.

The average core recovery at Skouries was 90.7%, which was deemed to be acceptable by SRK in 2007 and with which Scott Wilson concurs.

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

The (TVX diamond) drilling grid pattern used was 50 m by 50 m. Holes were drilled at an angle of some 60° to the pipe but given the disseminated nature of the porphyry type mineralisation it would be misleading to convert intercepts to true widths on this basis.

After geological and geotechnical logging, diamond drill holes were split lengthwise using a diamond saw. One half was stored for future reference and the other half was sampled at regular 2 m intervals and sent for sample preparation and assaying. Each sample was given an individual sample number and rock type code.

Sampling was carried out on 2 m intervals and across geological boundaries which is viewed by EGL as representative given the disseminated nature of the mineralisation. Drill hole spacing is on a nominal 50 m grid, which is in the opinion of the authors of this Technical Report more than sufficient sample support for the disseminated nature of the deposit mineralisation.

The equipment used for sample preparation is as follows:

- Rhino jaw crusher (capable of reducing pieces of core to 2 mm).
- LM-2 ring pulveriser (capable of pulverising 1,500 g of material to 95% below 150 mesh).
- Riffle splitters, scale, trays.

Drill holes SK-08 to SK-30 (15,501 m) and SOP-1 to SOP-33 (14,932 m) were prepared at three different laboratories: I.G.M.E (the Greek Geological Survey) at Xanthi, I.G.M.E at Athens (both ISO accredited) and TVX at Stratoni (at the time ISO 9002 accredited), the latter by TVX personnel.

Drill holes SOP-34 to SOP-39 (3,045 m) were prepared at the Stratoni laboratory by TVX personnel (at the time ISO 9002 accredited).

Drill holes SOP-40 and onwards were prepared at the Skouries sample preparation laboratory located at Madem Lakkos by TVX personnel (at the time ISO 9002 accredited).

Screw top plastic bottles rather than envelopes or plastic bags were used for storing and shipping the samples.

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

In the opinion of the author the sampling protocol met industry standards and ensured that no bias was introduced to samples at the preparation stage.

11.1 Quality Control and Quality Assurance

The Quality Control and Quality Assurance ("QA"/"QC") procedures and outcomes as implemented by TVX are summarised below.

The QC system used specified duplicate assays by a different laboratory and "blind" coarse reject checks. The QC system consisted of:-

- 5% of pulps in ore were submitted to a different laboratory. At an early stage, SGS in France was used for the check assaying programme. Later, the Chemex Laboratory in Vancouver Canada was used (certified under ISO 9001). The purpose of this analysis is to detect any biases between laboratories as well as to calculate the assaying error and see if it is within industry standards.

- 5% of coarse rejects of samples in ore were submitted, under a different name, to the routine sample preparation laboratory for splitting of a second sub-sample to be pulverised and re-assayed by the main laboratory (ALS-Geolab certified under ISO 9001). The aim of this work is to validate the complete sample preparation and assaying procedure as well as to calculate the total error involved.

A size analysis of coarse rejects was done periodically to ensure that the first sample split is done when samples are below 2 mm, which ensures that the total sample preparation error is maintained within acceptable industry standards.

Check assays for a large number of samples from Skouries drill holes SK-8 and SK-10 were done at the SGS laboratory in Carcassonne, France (certified under ISO 9001).

The following statistical analyses were carried out by EGL:

- Standard statistics for each variable, their differences and relative error variances.

- To obtain the global relative error variance, the relative variances of the pairs were pooled.

- Relative Standard Error = SQRT (Pooled Relative Variance).

- Student's T test to assess any bias between the mean values of the original and duplicate results.

- Regression analysis between the original and duplicate results.

In the opinion of the authors of this Technical Report all the above tests gave acceptable repeatability of results.

11.1.1 Check Assays and Internal Checks

Check assays were done by ALS-Geolab in Santiago Chile as the main laboratory and Chemex laboratory in Vancouver, Canada was used as the control laboratory.

Comparisons were made for total copper and gold. Soluble copper was not included since laboratories use slightly different analytical methods and results are seldom comparable.

In order to study the variability of assays within the main laboratory, statistical comparisons were performed for ALS-Geolab internal check assays as well as with Chemex with and without cut-offs.

In general, it was found that both the internal and independent lab copper check assays were well within industry standards, while gold check assays show a larger scatter. These results are as expected, since the presence of coarse gold was proven by mineralogical studies.

11.1.2 Coarse Gold Analysis

As coarse gold is of considerable importance to the project it was necessary to undertake an investigation to obtain reproducible results. The analysis was done using data collected up to October, 1997.

At ALS-Geolab and Chemex, gold is normally determined by fire assay over 50 g of sample using AAS for the final reading. ALS-Geolab did an internal check assay every tenth sample. These results were also reported to the client. It should be noted that these are not blind checks and if discrepancies arise, samples are re-assayed and these "corrected" values are reported.

As discrepancies in gold check assays were observed by TVX, they decided to further investigate the Skouries coarse gold. A set of 37 drill hole samples were chosen from a project area known to have coarse gold. This area had been identified previously as a result of mineralogical study. Care was taken to obtain high and low grade samples both from porphyry and schist.

As a result of the coarse gold study, the following conclusions were made by TVX:-

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

11.1.3 Coarse Reject Checks

At the beginning of the exploration programme, samples were prepared at three different laboratories. Later, they were prepared at the specially built sample preparation laboratory in Madem Lakkos and sent to ALS-Geolab for analyses. For the coarse reject checks, care was taken to include samples that had been prepared by all laboratories.

Coarse reject checks consist of:-

- Mixing thoroughly the crushed material (below 10 mesh) left over from the sample preparation
- Splitting and pulverising a 1 kg sub-sample
- Sending a portion (200 g) to ALS-Geolab for gold and total copper analysis.

Coarse rejects comparisons showed larger errors (scatter) than check assays since these included sample preparation errors as well as assaying errors.

The following conclusions were drawn by TVX from the results of quality control procedures in sample preparation and assaying:

- The sample preparation protocol was adequate for both gold and copper.

- The large scatter of gold check assays and coarse rejects is due to the presence of coarse gold.

Following a study on lab bias and sample preparation Kvaerner Metals concluded that the assay results for the Skouries deposit are within acceptable error limits. The authors of this Technical Report have reviewed this data and concur with this opinion.

12 DATA VERIFICATION

Since the drilling data included in this Technical Report is solely that of the TVX drilling campaign, the results of which were fully reported in the 2007 SRK Report, the authors of this Technical Report have relied on the data verification procedures applied by TVX in 1998 and SRK in the 2007 SRK Report, neither of which identified any limitations to such data verification. Both TVX and SRK concluded that the sample preparation, security and analytical procedures were adequate and the authors of this Technical Report have the same opinion.

Metallurgical testwork for the project was carried out by Lakefield in the reports detailed in Section 27, References. Antony Francis has reviewed the independent testwork reports and it is his opinion that the testwork was carried out on representative samples and in a manner that did not introduce any bias to the results.

13 MINERAL PROCESSING AND METALLURGY TESTING

13.1 Metallurgical Testwork

Metallurgical testwork and studies have been performed at Lakefield Research, Canada on composites selected from core samples of the major rock types covering mineralogy, grinding and flotation. Based upon this information, the criterion for process plant and infrastructure design has been established. Additional testwork has been completed by Outotec in 2007, mostly at their laboratories in Pori, Finland to give additional design confidence. This includes flash flotation, gravity gold recovery, concentrate settling & filtration.

Mineralisation of the sulphide ore primarily comprises chalcopyrite veinlets with subordinate bornite and disseminated chalcopyrite and bornite. Variable amounts of digenite, chalcocite, covellite, molybdenite and minor pyrite occur together with rare galena and sphalerite. Magnetite occurs both as disseminations and in quartz veinlets. Gold mineralisation occurs as native gold associated with gangue minerals and ranges in size from a few microns to 160 μm. It also occurs as blebs within sulphides, particularly in bornite and chalcocite. It correlates strongly with copper. Palladium was discovered to occur during testing which could add value to the ore. The oxide zone occurs from surface to 30 m to 50 m depths and occasionally deeper consisting mainly of malachite with cuprite, secondary chalcocite and minor azurite, covellite, digenite and native copper.

The bench scale grinding testwork considered ore types and boundaries, hardness maps, grade versus hardness and mine plan/schedules to determine the grinding mills design. The mill selection is based on the recommendations by an independent consultant and discussions with mill vendors.

Extensive flotation investigatory work has been 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 types investigated during this phase of 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 analysis, CuS and sulphuric acid analysis, CuL. The degree of oxidation was defined by the ratio of these 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”).

Based on the analyses of the cores used to form the various ore composite samples used in the testwork their varying degrees of oxidation may be expressed as follows.

Table 13-1 : Degree of Oxidation of Ore

Classification	CuS/CuT	CuL/CuT
HO	0.50	0.77
MO	0.17	0.38
LO	0.05	0.17
HS	0.05	0.05

Following the establishment of the best flotation conditions, one locked cycle test was performed on a representative sample of each of the three oxide ore types. In all tests the flotation stages applied mimicked the same flow sheets 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 have significantly lower copper and gold recoveries compared to the sulphide mineralization; these are estimated to be ~52.4% and 71.5% respectively. These can be compared to the life of mine ("LOM") average recoveries of ~91% for copper and ~84% for gold. The testwork has shown that the oxide copper minerals and the associated gold can be recovered by conventional sulphidiser activated flotation. Interestingly, mineralogical investigations indicated that the oxide ore copper losses were mainly due to very fine sulphides locked in gangue rather than non floating oxide minerals. It will not be cost effective to grind to the fineness required to liberate these locked 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 and also causes difficulties in the thickening and flotation unit operations.

The results of all the locked cycle flotation tests for both sulphide and oxides were evaluated to establish a relationship between predicted recovery and head grade for both copper and gold. This led to the development by Aker Kvaerner of equations to predict the average recoveries of copper and gold to final concentrate as a function of the ore head grade. Although largely based on flotation data produced from test work at Lakefield Research, these recovery equations are for the process plant as currently proposed, i.e. including any gold recovered by the gravity gold 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 CuL:CuT, 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:

Recovery (Cu) = 99.41 – 56 x % oxide – 41 * 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)

Where % oxide or degree of oxidation = 100 x (CuL/CuT) as defined earlier.

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

These equations have been extrapolated for use in the project's mine planning and have been found to give realistic values for the copper and gold recoveries.

Bench scale gravity gold concentration tests carried out by South West Metallurgical confirm the viability to recover free gold from the primary grinding circuit. Clearly additional gravity gold will be recovered from the flotation concentrate regrind circuit.

The Outotec test work campaign of 2007 was focused on evaluating the installation of a Flash Flotation unit cell to treat the primary grinding circuit cyclone underflow. The objective is to recover gold and copper in coarse mineral particles before possible over grinding occurs. The test work did show the unit cell could recover the mineralised values as predicted but would probably not significantly impact on overall plant recoveries. Further, in the first year of production the plant feed will contain significant oxide mineralisation which is known not to respond to flotation as well as sulphide minerals. Therefore it was decided to retain the gravity gold recovery circuit as originally envisaged although space has been provided in the grinding plant lay out for a unit cell retrofit should this prove beneficial in the later years of operations.

The testwork also resulted in the number of concentrate cleaning stages being increased from two to three in order to provide the target concentrate grade of 26% copper during periods of low head grade. The testwork included chemical and mineralogical characterisation of the samples and provided further evidence of the presence of PGMs particularly palladium identified in the earlier studies.

14 MINERAL RESOURCE ESTIMATES

Scott Wilson has reviewed and revised the Mineral Resource model and defined Mineral Reserve estimates to report the current open pit and underground Mineral Resource and Mineral Reserve estimates. The Mineral Resource NSR model is based on higher Mineral Resource related metal price assumptions. The Mineral Resource and Mineral Reserve estimates conform to Canadian 'CIM Definitions Standards for Mineral Resources and Mineral Reserves,' as prepared by the CIM Committee on Mineral Resources and Mineral Reserves on November 27, 2010 (CIM). In Scott Wilson's opinion, the Mineral Resource and Mineral Reserves classification results for the Skouries deposit discussed below are also in accordance with the Australasian Code (JORC, 2004). The open pit and underground Mineral Resource estimates are summarized in Table 14-1.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-1: Skouries Deposit Mineral Resources (as of 14th July, 2011)

Location	Category	Metric Tonnes	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz)	Contained Cu (metric tonnes)
Open Pit	Measured	15,333,000	1.05	0.59	516,000	91,000
	Indicated	49,747,000	0.44	0.34	708,000	171,000
	Inferred	72,307,000	0.15	0.17	350,000	120,000
Underground	Measured	24,147,000	1.36	0.73	1,055,000	175,000
	Indicated	157,123,000	0.61	0.49	3,067,000	768,000
	Inferred	43,470,000	0.34	0.39	477,000	167,000
Total Measured and Indicated		246,350,000	0.67	0.49	5,346,000	1,205,000
Total Inferred		115,777,000	0.22	0.25	828,000	288,000

1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources were estimated using an Au price of US$1,200 per ounce and a Cu price of US$3.50/lb.
3. Mineral Resources were estimated using a NSR discard cut-off value of €4.29/t for the open pit and a breakeven NSR cut-off value of €19.43/t for the underground.
4. The values for tonnages, grades and contained ounces have been rounded.

For this study, Scott Wilson has made the following changes to the 1998 Mineral Resource Model:

- Updated the drill hole database with ten TVX holes (SOP-89 to 98) drilled from June to August 1998. These holes predominately target the southwest perimeter of the main body of mineralisation.
- Updated the 5 m gold and copper composites and interpolated new gold (2010_Au) and copper (2010_Cu) block models using the same interpolation parameters as in 1998.
- Created a new Mineral Resource classification model based on continuous 3D solids.

- Created a solid to sterilise over-extrapolated blocks. The solid is predominately located along the north, northeast and east flanks in areas with no bounding perimeter drill holes.

14.1 Drill Hole Database

The header, survey, lithology and assay data for the ten new drill holes have been added to the GEMS “DrillholeA” database. The new gold (Au) and total copper (CUT) assays have been imported directly into the ASSAYS_MOD table. The updated drill hole database is summarised in Table 14-2. The 5 m length composite table has also been updated for the new ten drill holes. The 10 m bench composites have been used to help build the Measured and Indicated Mineral Resource estimates and sterilisation solids.

The May 2007 composite extraction file data used in the 2007 SRK Report has been imported into point area databases “Aucomps” and ”CuComps” to preserve the final gold, copper and rock type values used in 1998; the composites for the new ten holes have been added to the new point areas.

Table 14-2 : Drill Hole Database Summary

Table Name	Number of Records
HOLE-ID	121
SURVEY	1,238
ASSAYS_MOD	34,864
ASSAYS_RAW	32,380
COMPOSITE	14,483
LITHO_SUM	1,883
RECOVERY	11,800
ZONE_SUM	193
10M_BENCH	6,987

A series of MS Access update queries have been run to assign simplified rock type codes to the “New Code” field in the LITHO_SUM table for holes SOP89 to 98 (Table 14-3). The “New Code” field has been used to populate the “Simp Lith” field in the ASSAYS_MOD and COMPOSITE tables.

Table 14-3 : Simplified Rock Type Codes

SIMP Code	New Code	Number of Intervals
0	2	8
1	1	43
3	30	17
4	2	124
7	2	6

SIMP Code	New Code	Number of Intervals
14	1	29
34	30	9
41	2	46
43	2	25
47	2	3
111	1	1
112	1	18
124	1	1
412	2	8
1112	1	1
1124	1	1
4112	2	5

14.2 Assay Statistics

There are 34,864 assay records including 26,520 records identified as schist (Simp_Code=2), 8,007 records identified as porphyry (Simp_Code=1) and 275 records related to post-mineralisation dykes (Simp_Code=30). A small number of records located at hole collars have no simplified rock codes. The porphyry samples average 1.11 g/t Au and 0.61% Cu. The schist samples average 0.37 g/t Au and 0.32% Cu (Table 14-4).

Table 14-4 : Assay Statistics

SIMP_CODE	N	Min Au (g/t)	Max Au (g/t)	Avg Au (g/t)	St Dev Au (g/t)
0	62	0.00	1.70	0.15	0.34
1	8,007	0.00	58.73	1.11	1.59
2	26,520	0.00	33.90	0.37	0.59
30	275	0.00	0.98	0.08	0.18

SIMP_CODE	N	Min CUT (%)	Max CUT (%)	Avg CUT (%)	St Dev CUT (%)
0	62	0.00	1.14	0.10	0.21
1	8,007	0.00	26.00	0.61	0.64
2	26,520	0.00	4.48	0.32	0.27
30	275	0.00	1.35	0.08	0.17

14.2.1 Assay Capping (Cutting)

There are very few high assays so capping has not been deemed necessary by Scott Wilson; however, a 20 m isotropic restricted search has been applied to composites over 8 g/t Au to rectify a local gold grade estimation problem observed in the schists.

14.2.2 Composite Statistics

The assays have been composited into five metre equal lengths starting at the drill hole collars. There are 14,256 composites in the new point area workspace which have been used to krige the new block model. The DrillholeA composite table has 14,483 records because it includes some dyke related composites that were removed from the 1998 composite extraction files. These 14,256 composites include 3,213 porphyry composites that average 1.11 g/t Au and 0.61% Cu and 11,002 schist composites that average 0.36 g/t Au and 0.31% Cu (Table 14-5). The assay and composite means are essentially the same.

Table 14-5 : Composite Statistics

ROCKTYPE	N	Min Au (g/t)	Max Au (g/t)	Avg Au (g/t)	St Dev (g/t Au)
1	3,213	0.00	25.68	1.11	1.26
2	11,002	0.00	13.94	0.36	0.47
30	41	0.00	0.97	0.34	0.23

ROCKTYPE	N	Min Cu (%)	Max Cu (%)	Avg Cu (%)	St Dev Cu (%)
1	3,213	0.00	10.68	0.61	0.54
2	11,002	0.00	3.42	0.31	0.24
30	41	0.00	0.77	0.32	0.18

14.2.3 Wireframes

The manual geological interpretation included the porphyry and the sterile dykes. The schist was not delimited due to the fact that it is the only other rock type, so everything that is not porphyry or sterile dyke corresponds to schist.

The porphyry (rock code 1 and 11) was manually modelled in sections every 50 m at 1:1000 scale. A total of 17 sections were interpreted on paper. Based on these 17 sections, 9 plans were plotted on 100 m level spacing and digitised

In order to have a better control of the final solid generation, a preliminary solid was constructed using the 17 sections and the 9 plans mentioned above. Plans were created every 20 m showing the intercepts of all digitised information located between levels -200 masl to 640 masl. These were then adjusted and modelled on-screen. Finally, a total of 43 plans were used and the 3-D solid for the porphyry was generated by extruding the 43 plans.

Solely for Mineral Resource estimation purposes, this solid was sub-divided in two sub-units, the main or central porphyry, and lateral porphyry branches. Both are mineralised separately from the schist.

14.2.4 Mineral Resource Cut-Off Grade

Mineral Resources were estimated using an Au price of US$1,200 per ounce and a Cu price of US$3.50/lb. These prices are higher than those used for the Mineral Reserve estimate given in Section 16 to reflect EGL's view on possible long term prices. The Mineral Resource NSR block model was used to report open pit Mineral Resources based on a discard cut-off value of €4.29/t and underground Mineral Resources based on a breakeven NSR cut-off value of €19.43/t. The underground cut-off reflects the Sub Level Open Stoping mining method and complies with the approved EIS. The open pit Mineral Resources are reported between surface and a Whittle shell that extends to the 430 m bench. The underground Mineral Resources are reported between the 390 m and -200 m elevations. No Mineral Resources have been reported in the crown pillar because it has been assumed for this study that it cannot be extracted. The SRK pit design extends down to the 420 m elevation. The crown pillar extends from the 420 m to 390 m elevations.

The Mineral Resource NSR equation is based on the Mineral Resource metal prices, the gold and copper variable grade recovery equations reported in the SRK 2007 Report, and the 0.97 and 0.886 factors developed for gold and copper, respectively, that reflect revenue after smelter treatment and other charges. The Mineral Reserve NSR equation is the same except the Mineral Reserve metal prices are used. The Mineral Resource NSR equation is provided below:

*Mineral Resource_NSR = ((Au ozs*0.97*882.35)+(Cu tonnes*0.886*5674))/Block tonnage*

Where,

*Au ozs= Au_2010*Plant recovery Au/31.1035*Block tonnage (gives ounces gold in block)*

*Cu tonnes= Cu_2010*Plant recovery Cu/100*Block tonnage (gives tonnes copper in block)*

And where,

*Plant recovery Au = 92.62 – 17.5 * % oxide – 22 * e^ (-1.2 * gold grade g/t)*

*Plant recovery Cu = 99.41 – 56 * % oxide – 41 * e^ (-338*copper grade %)*

14.2.5 Specific Gravity

This section deals with specific gravity determinations done in the field and checks carried out by other professional laboratories.

During the Skouries exploration campaigns a 20 cm to 30 cm piece of core was set aside for each 50 m of core approximately. These reference samples were kept for geomechanical testing and specific gravity measurements. Determinations were done using the traditional non waxed water immersion method. It is noted that the different rock types (porphyry, schist and dyke) are extremely competent and have hardly any pores or voids. Therefore all specific gravity determination methods (volumetric by water immersion, waxed or non waxed and geometric) should yield similar results.

A total of 483 samples were tested on site for specific gravity. Of these, 101 samples were porphyry and the remaining 382 were schist.

Quality control was done in two different ways:

Specific gravity determinations done at different times using the geometric method by two independent professional laboratories on different samples not measured previously by either TVX or HG.

Specific gravity determinations done by Lakefield Research Laboratories and CIMM Laboratories (CIMM) in Chile on 58 samples measured previously by either TVX or HG. Lakefield used a waxed volumetric method and CIMM used both waxed volumetric and water immersion. On a subset, CIMM also calculated specific gravity geometrically. Table 14-6 shows the summary statistical results.

Table 14-6 : S.G. Quality Control - Statistical Analysis

Lab	Method	Number	Min	Max	Mean	Std
TVX	Non Wax	57	2.00	2.86	2.67	0.13
LAKE	Wax	57	2.41	2.93	2.67	0.12
CIMM	Non Wax	57	2.15	2.84	2.61	0.13
CIMM	Wax	57	2.35	2.86	2.66	0.11
TVX	Non Wax	46	2.00	2.86	2.67	0.13
LAKE	Wax	46	2.41	2.93	2.67	0.12
CIMM	Non Wax	46	2.15	2.84	2.62	0.12
CIMM	Wax	46	2.35	2.86	2.67	0.10
CIMM	Geometric	46	2.41	2.88	2.68	0.10

All the comparisons involving Lakefield Research yield very poor pair-wise results. There does not seem to be a reasonable explanation for this.

As the specific gravity determinations of TVX, HG and CIMM gave reasonable comparisons, it was decided to use the TVX and HG measurements. It should be noted that these results are reliable but too few to build a three dimensional specific gravity model. Therefore mean values by rock type were used. These are:

Porphyry : 2.64 g/cc

Schist : 2.73 g/cc

14.2.6 Block Model

The lithological codes and block model parameters used in the block model for the Mineral Resource estimation are shown in Table 14-7 and Table 14-8, respectively.

Table 14-7 : Lithological Codes for Block Model

Rock Type	Code
Main porphyry	1
Porphyry branches	9
Schist	2
Dyke	30
Air	0

Table 14-8 : Rotated Block Model Parameters

Parameter	Value
Lower Left corner (E)	474,611.91
Lower Left corner (N)	4,479,261.5
Minimum Z	-210.00
Maximum Z	710.00
Model Rotation	40.67°
Number of Columns (E-W)	180
Number of Rows (N-S)	180
Number of Benches	92
Column Width	5.00
Row Width	5.00
Bench Height	10.00

The size of the blocks (5 m by 5 m by 10 m) was chosen because the expected bench height is 10 m and the drill hole spacing on the top levels of the deposit is a maximum 50 m by 50 m. Therefore the block size (in plan) is of the order of 1/10 of the drillhole spacing, which was considered reasonable.

The model was rotated so that the rows coincide with the direction of the sections defined and interpreted by the geologists and coincides with the strike direction of narrow dykes.

A rock type block model was defined by intersecting the 3D solids with the block model. Blocks that contained more than a certain proportion (by volume) of a specific rock type were assigned that particular code. This code was valid for the entire block. The proportions used were greater than 50% for porphyry and schist, and greater than 41.5% for the sterile dyke. These proportions were chosen so that the total volume of each rock type within the block model was similar to that of the three dimensional solids within the geology model. The total volume of the model containing sterile dyke is estimated as less than 5%.

14.2.7 Grade Variography

As part of the 1998 Mineral Resource modelling work, grade variography was carried out for each variable in each population. The calculations were done for the following directions: north-south, north-east, east-west, north-west, omnidirectional horizontal, vertical, and omnidirectional.

The step (lag) used was 5 m with a tolerance of 3 m, so that some data overlap would occur. This was considered necessary since some composite populations are very small. The tolerance angle used for directional variograms was 22.5°.

Variograms were calculated and theoretical models were fitted. Results from Kvaerner (1998) are shown in Table 14-9.

Table 14-9 : Project - Variograms

Rock Type	N Comp	Mean	Variance	Nugget (Co)	Type	Direction	Sill (C1)	Range (A1)	Sill (C2)	Range (A2)
Au Variograms										
Main Porphyry	2518	1.28	1.82	0.15	Exp	Omni	0.25	10	0.73	200
Schist	9794	0.38	0.24	0.15	Sph	Vert	0.30	15	0.40	300
Schist	9794	0.38	0.24	0.15	Sph	NW	0.40	100	0.65	300
Schist	9794	0.38	0.24	0.15	Sph	NE	0.35	40	0.40	300
CuT Variograms										
Main Porphyry	2518	0.67	0.28	0.15	Sph	Vert	0.20	20	0.90	225
Main Porphyry	2518	0.67	0.28	0.15	Sph	Omni-Hz	0.20	10	0.85	170
Schist	9794	0.33	0.06	0.15	Sph	Vert	0.20	15	0.52	300
Schist	9794	0.33	0.06	0.15	Sph	NW	0.20	50	0.75	250
Schist	9794	0.33	0.06	0.15	Sph	NE	0.20	40	0.50	250

The following conclusions can be made from the variography analysis:

- Nugget effects were fitted from the vertical variograms. These are between 5% and 18% of the total sill value.
- Generally, all variograms were very well behaved, especially for the large populations.
- For Au variograms, the effective ranges for main porphyry and schist were of the order of 150 m. No anisotropy was found within the porphyry, while a marked horizontal and vertical anisotropy was detected in the schist.
- For CuT variograms, the effective ranges for main porphyry were of the order of 150 m and for schist of the order of 200 m. A mild anisotropy was found within the main porphyry and a marked horizontal and vertical anisotropy was found in the schist.

14.2.8 Interpolation Validation

As the block model was estimated by ordinary kriging, it was appropriate to validate variograms and kriging plans. This was done in 1998 using the cross validation technique.

This technique consists in estimating, by ordinary point kriging, each composite location, using the correct population surrounding data and variogram model. The estimation is done ignoring the composite value being estimated so that, at the end of the process, the real composite value can be compared with the estimated value.

The cross validation procedure was applied to gold and total copper composite values of the entire porphyry, main porphyry and schist.

Cross validation results are presented in Table 14-10.

Table 14-10 : Cross Validation Results

		Composites		True Values		Estimated Values		
Rock Type	Element	Total	Estimated	Mean	Std	Mean	Std	R
All Porphyry	Au	3051	3047	1.16	1.28	1.17	0.99	0.74
Main Porphyry	Au	2518	2515	1.28	1.35	1.29	1.03	0.72
Schist	Au	9794	9792	0.38	0.49	0.38	0.40	0.74
All Porphyry	CuT	3051	3047	0.63	0.54	0.63	0.43	0.78
Main Porphyry	CuT	2518	2515	0.67	0.53	0.67	0.45	0.83
Schist	CuT	9794	9792	0.33	0.25	0.33	0.21	0.83

Considering the results presented above it was concluded that the kriging plan is acceptable.

14.2.9 Block Model Estimation and Validation

The 1998 ordinary kriging profiles for Au and Cu have been rebuilt in the new version of Gemcom (GEMS 6.2.4) and new gold (“Au_2010”) and copper (“Cu_2010”) models have been populated.

Models for gold and total copper were estimated using ordinary block kriging with a single search radius of 150 m. In all cases a block discretisation of 18 points was used (3 points in x-axis, 3 points in the y axis and 2 points in the z-axis).

The search radius is considered reasonable in view of the large ranges shown by the variograms. High grade values for copper were not considered important enough to restrict their search radii. Gold values above 8.00 g/t Au were restricted to a maximum search radius of 20 m instead of 150 m.

The porphyry branches are restricted to the southern area and their grades are low and similar to that of the schist that surrounds them. Consequently, both porphyry and schist composites were used to interpolate porphyry branch blocks. Porphyry composites could only interpolate porphyry blocks and schist composites could only interpolate schist blocks (Table 14-11).

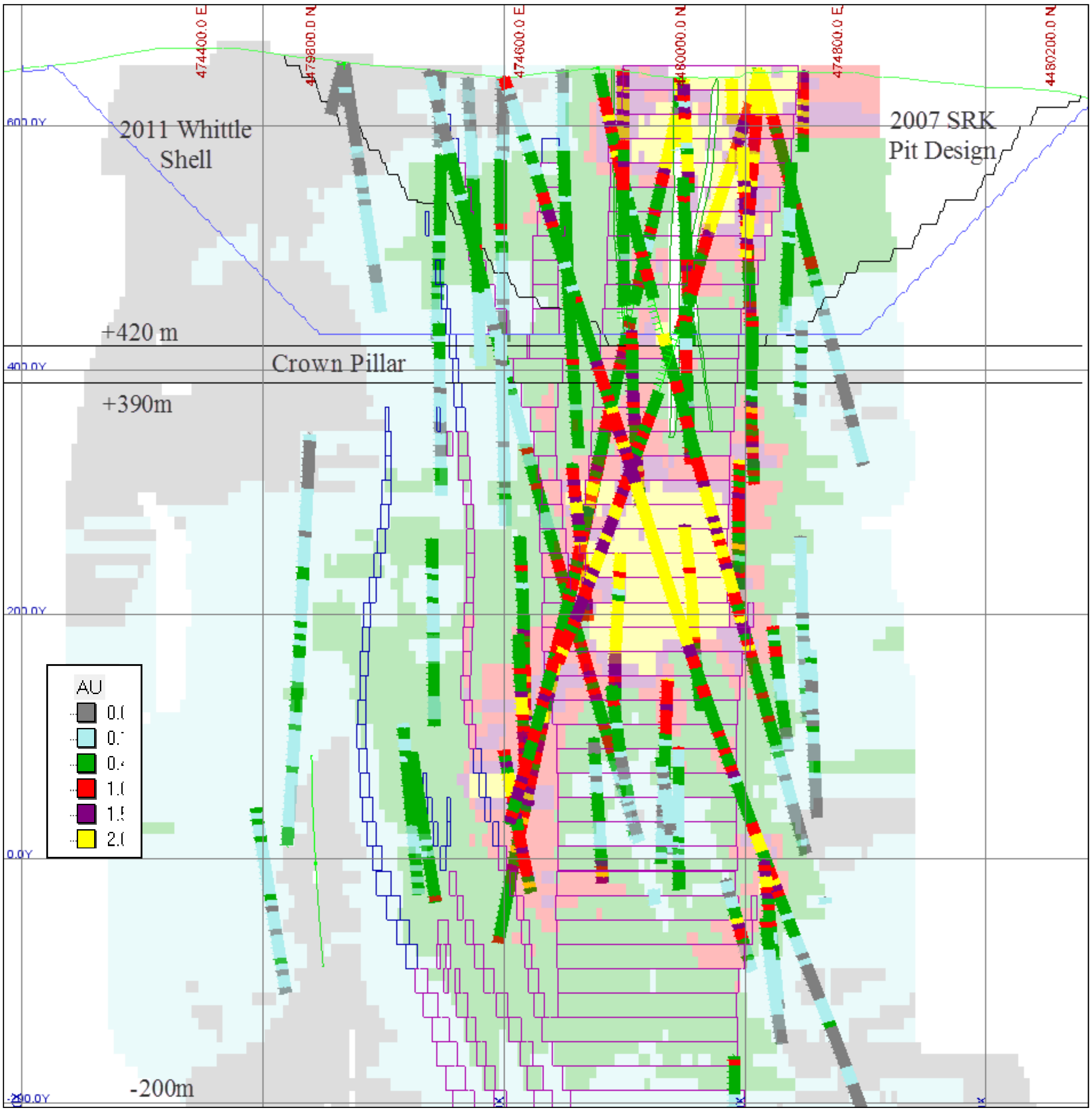
Table 14-11 : Criteria for Block Kriging by Rock Type

Block Rock Code	Rock Description	Composite Rock Code	Variogram Used
1	Main Porphyry	1	Main Porphyry
9	Porphyry Branches	1 + 2	Schist
2	Schist	2	Schist

The total copper and gold block models were validated graphically in sections and plans showing sample composites and block grades. The following comments are pertinent:

- There is a good correlation between composite data and block estimates for both total copper and gold. No problems in the estimation process were detected.
- Dyke blocks appear blank and can be seen in plans and sections.
- Grades are noticeably higher in the main porphyry and decrease towards the periphery where the lateral branches are located. This is especially true for gold.
- Section 10250 NW represents the core of the deposit. Two high grade zones, especially for gold values, are evident. The first one is located around the 600 m elevation and a deeper one is located between 300 m and 100 m elevation (Figure 14-1).

Figure 14-1 : Gold Assays and Blocks on Section 10,250NW



"Drift analysis" consists of dividing the deposit into horizontal slices, vertical north-south slices and vertical east-west slices in local co-ordinates (rotated). For each slice the mean composite grade is compared to the mean block grade. Thereafter, these trends are plotted in order to assess the smoothing caused by kriging.

This analysis was carried out in 1998 for the main porphyry only for measured plus Indicated Mineral Resource blocks. 50 m vertical and horizontal slices were used. In general, block and composite trends are similar, with the former being somewhat smoother than the latter. Usually, the block grade trends are slightly lower than the composite grade trends. This is due to the data clustering effect.

From the validation work performed to date, EGL believes that the block model represents satisfactorily the geology and grade distribution observed in the drill holes.

14.2.10 New Post-Interpolation Adjustments

There are sparse bounding perimeter drill holes located along the north, northeast and east flanks so a sterilisation solid has been manually built to override the interpolated grades in these areas.

14.2.11 New Mineral Resource Classification

The Mineral Resource classification is in accordance with the CIM definitions standards. In Scott Wilson's opinion, the Mineral Resource classification results for the Skouries deposit discussed are also in accordance with JORC.

Measured and Indicated Mineral Resource classification solids have also been manually built using 10 m gold bench composites as a guide to produce continuous bodies for each category.

With exception of the limits to the open pit which were defined in the recently approved EIS submitted by HG to the Greek authorities in July 2010, estimates were not affected by environmental, permitting, legal, title, taxation, socio-economic, marketing, political, mining, metallurgical infrastructure or other relevant factors other than those disclosed herein.

Most of the deposit is supported by 50 m spaced drill holes; however, some parts of the deposit are supported by even closer spaced drilling. The Measured Mineral Resource solid generally encompasses a central portion of the mineralisation where the drill holes are mostly spaced at less than 50 m apart. The overall average distance between composites and block centroids for Measured Mineral Resource is approximately 15 m. The Indicated Mineral Resource solid covers a broader area supported by approximately 50 m to 60 m spaced drill holes. The overall average distance between composites and block centroids for the Indicated Mineral Resource is approximately 25 m. All blocks that are not classified as Measured or Indicated Mineral Resource have been classified as Inferred Mineral Resource. The overall average distance between composites and block centroids for Inferred Mineral Resource is approximately 50 m.

The 1998 gold and copper omni-directional variograms have ranges of approximately 150 m to 200 m.

15 MINERAL RESERVES ESTIMATES

Measured and Indicated Mineral Resources contained within the Skouries deposit that have been converted into Proven and Probable Mineral Reserves are estimated as of July 2011, in accordance with the requirements of NI 43-101 and the CIM definitions. As discussed, the Mineral Resource and Mineral Reserve estimates also conform to JORC. The Mineral Reserves have been used for the SLOS life of mine plan and economic analysis. Mineral Reserves as shown in Table 15-1 have been estimated assuming an open pit mining method and SLOS method with backfill for the underground operation. The Mineral Resources are inclusive of the Mineral Reserves.

The stope sizes, layouts and sequencing are described in Section 16. Mineral Reserves for the open pit and underground sections are shown in Table 15-1.

Table 15-1: Skouries Deposit Mineral Reserves (as of 14th July 2011)

Location	Category	Metric Tonnes	Grade Au (g/t)	Grade Cu (%)	Contained Au (oz)	Contained Cu (metric tonnes)
Open Pit	Proven	15,166,000	1.06	0.6%	516,000	91,000
	Probable	31,816,000	0.52	0.4%	530,000	118,000
Underground	Proven	19,278,000	1.40	0.74%	866,000	143,000
	Probable	72,102,000	0.72	0.53%	1,678,000	385,000
Total Proven and Probable		138,362,000	0.81	0.53%	3,590,000	736,000

Notes:

1.CIM definitions were followed for Mineral Reserves.
2.Mineral Reserves were estimated using an Au price of US$1,000 per ounce and a Cu price of US$2.50 per lb.
3.Mineral Reserves were estimated for the open pit using a NSR cut-off value of €4.29/t and the underground using a NSR cut-off value of €19.43/t.
4.The values for tonnages, grades and contained ounces have been rounded.
5.Open Pit dilution averages 0% and extraction is estimated to be 100%.
6.Underground dilution averages 3.5% and extraction is estimated at 95%

15.1 Mineral Reserve Cut-Off Grade

Scott Wilson has built a NSR Mineral Reserve model to report the Mineral Reserves inclusive of the open pit and underground sections of the orebody.

The cut-off grade calculation for Mineral Reserves by Scott Wilson has used the same methodology and costs as those used to calculate the cut-off grades for Mineral Resources; however, a price of US$1,000 per ounce Au and US$2.50 per pound Cu has been applied to convert to Mineral Reserves.

Individual blocks were investigated within the block model through the application of formulae including metal grades, plant recoveries, NSR and mining costs to identify blocks that resulted in a positive value.

15.2 Mineral Resource to Mineral Reserve Conversion Methodology

At the instructions of EGL, Scott Wilson has not optimised the open pit design using current economic parameters. For reporting the in the 2007 SRK Report.

An updated block model has been produced by Scott Wilson for use in the underground mine design and mineral reserve estimation process. The block model has been divided into 25 m slices to represent the individual cuts in the sub-level open stoping method. A crown pillar of 30 m was left beneath the existing open pit and above the underground mining fronts.

Mineral Resources have been selected for conversion to Mineral Reserves by applying the SLOS mining layout to the Mineral Resource model. An NSR cut-off envelope and the location of the Measured and Indicated Mineral Resource blocks helped guide stope design on each level. Some sub-economic blocks have been included locally in the Mineral Reserve to allow for continuity.

Certain Mineral Resources have been excluded from the modelled slices for the following reasons:

- Too narrow or low-tonnage.
- Isolated from other blocks.
- En-echelon geometry preventing full extraction.
- NSR block values below the NSR cut-off and mineralization situated outside the NSR cut-off envelope.

Blocks just below the NSR cut-off and inside the stopes have been included as internal dilution. Scott Wilson has applied dilution from the hanging wall and footwall contacts in addition to paste fill dilution and an extraction percentage to the blocked out Mineral Resources to derive the tonnage and grade of Mineral Reserves for each lens. An extraction factor of 95% has been applied to account for ore losses in blasting and mucking.

15.3 Dilution

15.3.1 Open Pit

Scott Wilson has assumed 100% mining recovery and no additional dilution over and above the internal dilution reported within the Mineral Resource block model. The orebody is fairly large, regular and vertical, with mostly high grade material at the core with lower grade surrounding and waste on the perimeter. A few isolated waste blocks are located within the orebody, mostly one block size width, i.e. 5 m.

In practise, it can be expected that there will be some dilution on the ore/waste contacts but during mining it should be fairly limited due to the ore and waste zones being well defined. Therefore no additional dilution has been accounted for in this study

15.3.2 Underground

The SLOS method applied in this Technical Report uses ring drives driven from a spiral ramp located on the north-western edge of the orebody.

The mining method utilises a primary/secondary stoping sequence whereby the backfill floor thickness of 0.5 m and a sidewall thickness of 0.0 m is applied which results in a dilution of 1.3%. The secondary stope also has a backfill floor thickness of 0.5 m however a sidewall thickness of 0.5 m is now applied resulting in a dilution of 5.5%. Combining the primary and secondary stopes in defining the mine dilution results in an average mine dilution factor of 3.5%. Dilution is incorporated in the schedule at zero grade.

Control of external dilution from the primary stopes on the periphery of the ore body will play a role in dilution overall. The amount of dilution is strongly dependent on balancing a number of factors; namely, minimising dilution by controlled blasting and extraction, maximising extraction by mining as little of non economic material at the end of the stopes as possible, or maximising productivity by ensuring that the stope sides remain in competent ground and no geotechnical failures occur.

In Scott Wilson's opinion, external dilution is likely to be greater on the extremities of the orebody however, for the purposes of this Technical Report the Mineral Reserve conversion and production scheduling, only the total dilution is considered relevant.

In Scott Wilson's opinion, an average total dilution of 3.5% is at the low end of the range of industry experience with this mining method; however, it is reflective of the potential selectivity and grade control possible with this mining method and is considered appropriate for the estimation of Mineral Reserves in this Technical Report.

15.4 Extraction

A stope recovery factor of 95% was applied for the Mineral Reserve estimate, assuming a 5% loss as a result of production drilling short of the stope limits, blasting misfires and mucking losses.

16 MINING METHODS

16.1 General Mine Layout

Initial Production will come from an open pit operation. The underground mine will consist of the orebody below the base of the open pit at 420 m level (240 m below surface), followed by a SLOS mining method commencing below the 30 m crown pillar pillar and advancing to a final depth of -105 masl.

The deposit will be accessed from surface by a service decline (Ramp) and a production shaft. A number of production levels will be developed from the access ramp. Production levels will be vertically spaced from each other at 25 m intervals

Each production level will have an auxiliary and permanent ventilation system to provide adequate amounts of fresh air for the safe and efficient execution of mining activities.

To allow a smooth transfer of production from open pit to underground without a production gap, the mine accesses, ore handling, crushing and hoisting facilities and dewatering systems will be developed and equipped prior to the start of production and during the open pit operations.

16.2 Open Pit

A conventional open pit mining operation, utilising an owner operator scenario, with a production rate of 8.0 Mtpa run-of-mine ore is envisaged for the Project. The mining method will consist of drilling, blasting, loading and hauling of ore and waste materials for processing and waste disposal. Based on previous studies about 79% of all material will require drill and blast and approximately 21% will be mined utilising rip and dig methods. Nominally 5 m bench heights have been adopted which is suitable for the equipment selection and matches the Mineral Resource model individual block sizes. Drilling and blasting will be on 10 m bench heights and where possible mining will actually take place at a more suitable and productive 10 m bench height.

The primary mining equipment fleet will consist of conventional diesel powered open pit mining equipment, including;

- 2 No. 13 m^3 class 290 t hydraulic excavators.
- 11 No. 90 t to 100 t class off-road haulage trucks.
- 2 No. DTH 140 mm to 165 mm blast hole drilling rigs capable of a single pass of 12 m.

The above primary equipment will be supported by a fleet of auxiliary support equipment and mobile service equipment, which will be maintained in a fully equipped maintenance shop.

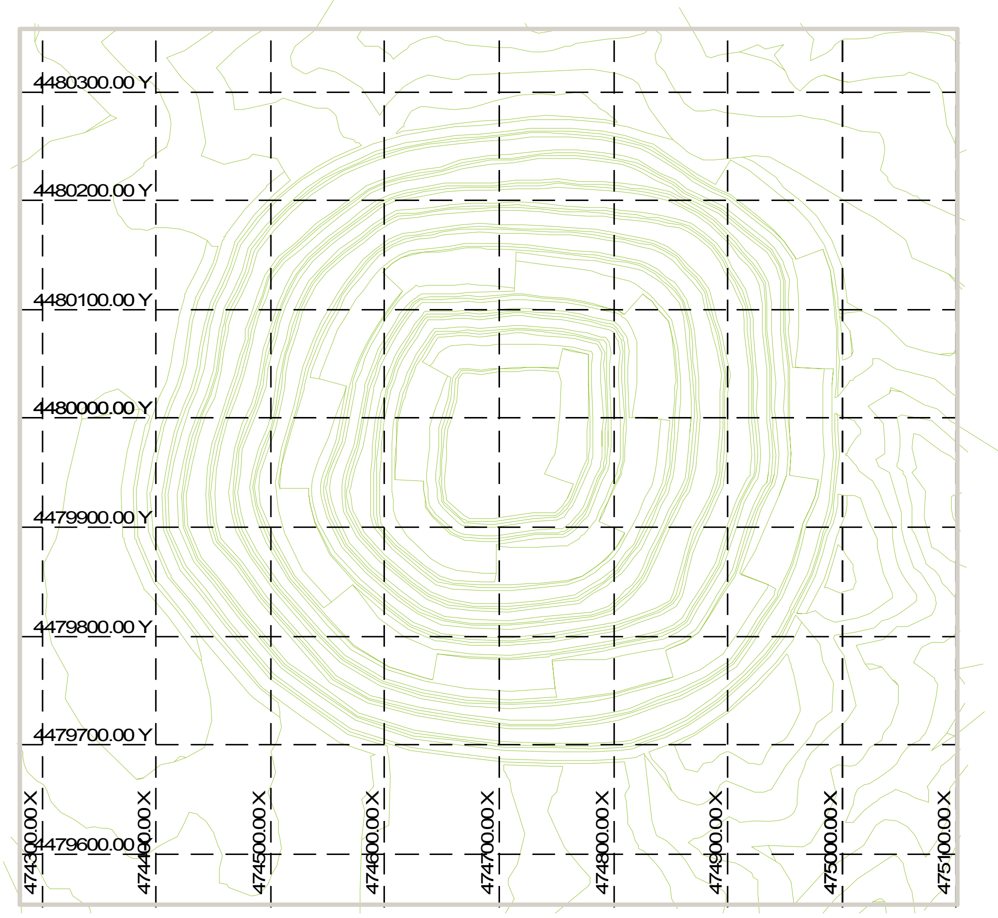
As previously mentioned open pit design was undertaken by SRK in the 2007 43-101 report and has not been changed for this study. The approved EIS includes the 2007 SRK open pit design, therefore the open pit design will remain fixed for this study as shown in Figure 16-1.

SRK undertook pit optimisation utilising Whittle software to determine an optimum pit shell based on cost and economic factors. The Whittle pit shell was then used to guide the final open pit design.

The open pit is circular in shape intersecting the natural ground surface at the 620 m and 670 m elevation points, resulting in the highest open pit face of approximately 250 m in height with the open pit floor at the 420 m elevation. Overall open pit slope angles vary from between 40º and 44º.

The internal open pit haulage ramp is designed for two way traffic with a design width of 25 m and a 1 in 10 gradient, i.e. 10%.

Figure 16-1 : Open Pit Design



16.2.1 Geotechnics

A geotechnical study was undertaken by SRK during the 2007 open pit design, this section is taken from the 2007 SRK Report on the Project.

The geotechnical conditions at Skouries are typified by:

- The porphyry is generally competent with rock qualities between Fair and Very Good which has good indications for open pit slopes and open stope stability but adverse indications for primary fragmentation. Some localised areas have poor rock qualities.
- The rock quality of the schist is generally Fair to Extremely Poor which has adverse implications for open pit slope angles.
- Groundwater is present in sub-vertical fractures, requiring dewatering and drainage measures.

The rock mass rating ("RMR") model developed for the Skouries Project Feasibility Study by TVX and Kvaerner Metals for TVX Hellas, dated September 1998 was used and provides coverage for most of a 300 m deep pit except near surface where the values at the margins were extrapolated. The Haines and Terbrugge slope angle design methodology was used to determine preliminary slope angles. SRK determined design sectors of similar RMR conditions. The slope angles relating to the RMR values (for a Factor of Safety of 1.3) are given in Table 16-1 and shown diagrammatically in Figure 16-2.

Figure 16-2 : Inter Ramp Slope Angles

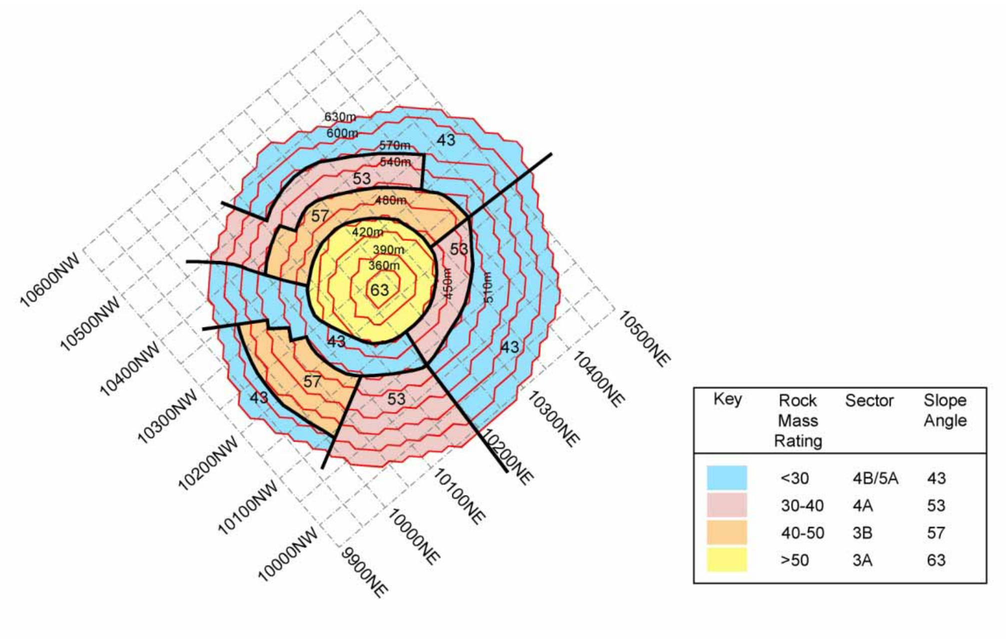


Table 16-1 : Slope Design Criteria (After Haines and Terbrugge)

RMR	<30	30-40	40-50	>50
Class	Very Poor	Poor	Fair	Good
Angle (º)	43	53	57	63
Berm width (m)	8.5 m	6.0 m	5.2 m	4.0 m

16.2.2 Production Schedule

The open pit mine production schedule has been developed on a planned annual ore production rate of 8.0 Mtpa. An open pit mining operation of 345 day per year consisting of three, eight hour shifts operating 7 days a week is envisaged, resulting in a daily average ore mining rate of 23,200 tpd.

A total of 47.1 Mt of ore with an average head grade of 0.69 g/t Au and 0.44% Cu will be mined over a 7 year mine life. The average stripping ratio of 0.74:1 equates to approximately 35 Mt of waste being produced. The Mineral Reserve estimate, and subsequently the production schedule, has been derived from an NSR calculation based on the parameters laid out below in Table 16-2:

Table 16-2 : Marginal Cut-off Grade

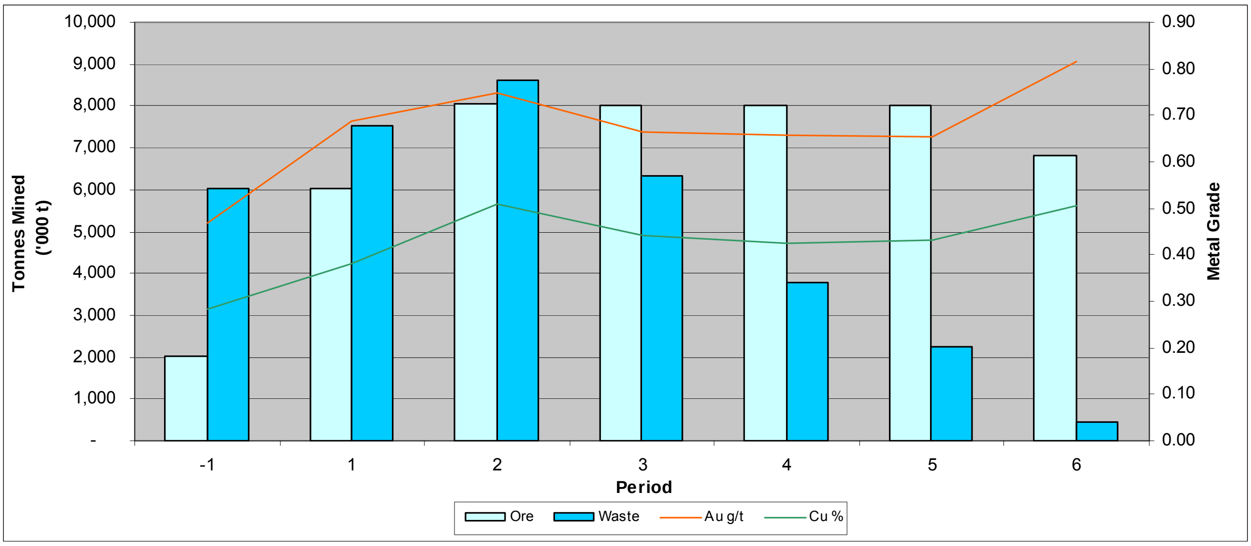
Parameters	Inputs
Gold NSR	97%
Gold Price	US$1,000/oz
Copper NSR	87%*
Copper Price	US$2.50/lb
Total Operating Cost	€20.76/t

*Using terms given below in Section 19

Table 16-3 : Open Pit Production Schedule Table

Year	-1	1	2	3	4	5	6	Total
Ore (t 000s)	2,018	6,023	8,052	8,030	8,030	8,030	6,799	46,982
Grade Au g/t	0.47	0.69	0.75	0.66	0.66	0.65	0.81	0.69
Grade Cu%	0.28	0.38	0.51	0.44	0.42	0.43	0.51	0.44
Waste (t 000s)	6,012	7,531	8,624	6,340	3,786	2,251	458	35,001

Figure 16-3 : Open Pit Production Schedule Graph



16.3 Underground Mine

16.3.1 Underground Development

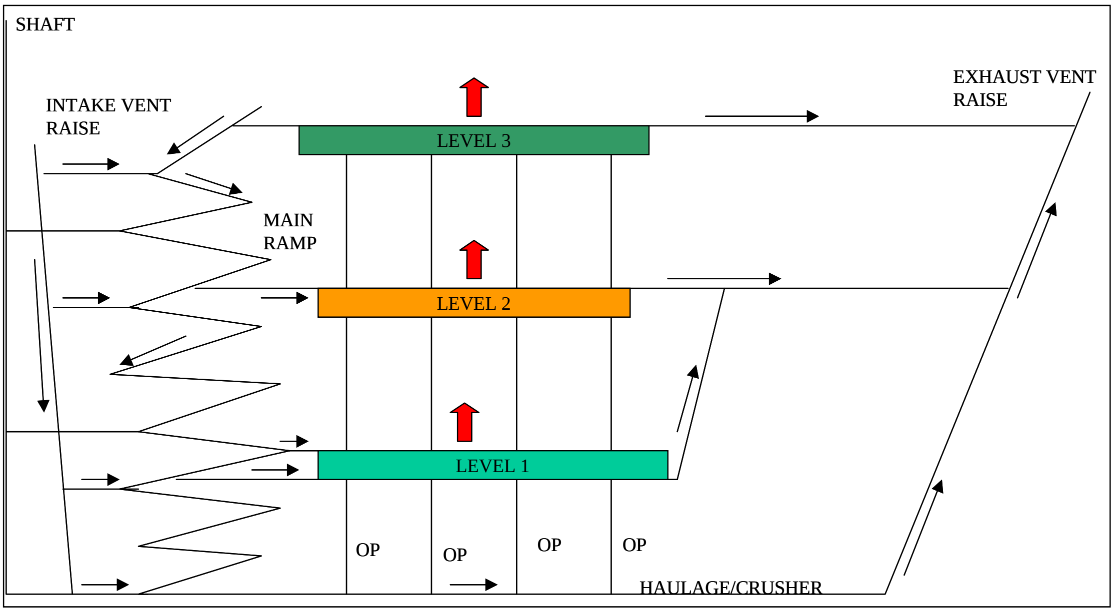
At Skouries, it is intended that some of the excess tailings from the underground mining operation will be mixed with cement and disposed of in the open pit as engineered stabilised fill. This means that mining of the open pit has to be complete before underground stope mining can start.

The production of ore from the open pit will be 8 Mtpa, resulting in a pit life of six years. Underground production commences in Year 7 with pre-production of the underground mine commencing in Year 4. Production averages 4.4 Mtpa over most of the underground mine life.

The mine design and schedule are based on three operational levels that will be mined simultaneously, accessed by a main ramp, positioned close to the orebody. The base level for each mining horizon will be linked to an exhaust vent raise, creating a main return airway. Intermediate ventilation raises will link each level with the closest main return airway drift. A main intake ventilation raise, close to the main ramp will be one of the three main air intakes (shaft, main ramp and intake ventilation raise).

Ore will be mucked from the stopes by LHD. Ore will be dropped via ore passes to a haulage level (-130 masl), where loaders will charge 50 t trucks that transport the ore to the main crusher. After crushing, the ore will be transferred to silos by conveyor where a skip loading facility will allow the transport of the ore to surface by a shaft

Figure 16-4 : SLOS Schematic Infrastructure

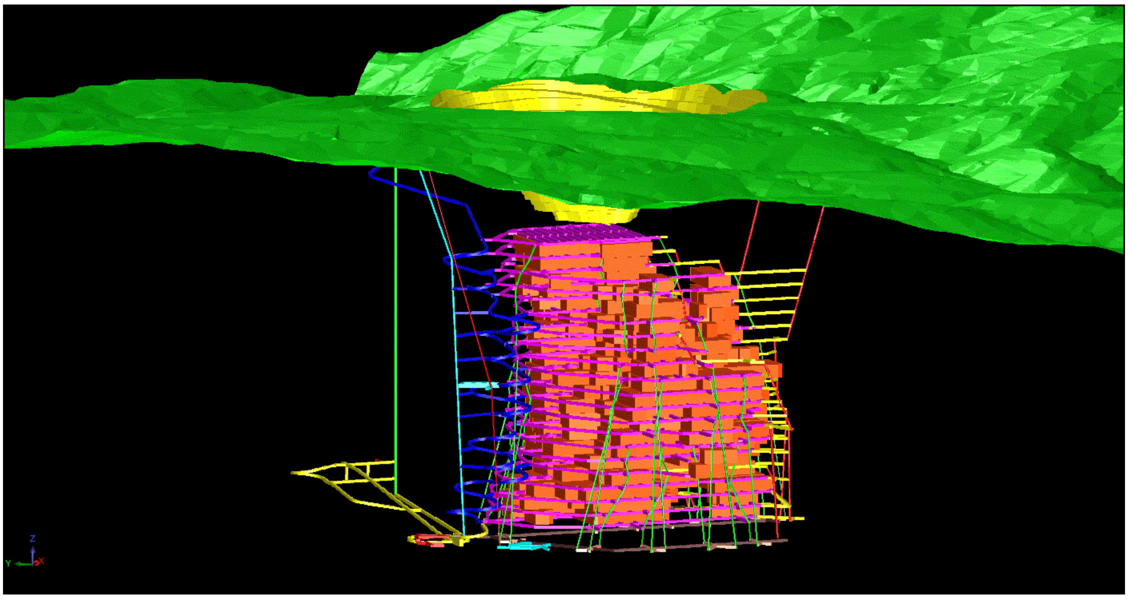


A classical sump / thickener / clear water bays / pump station arrangement has been included in the capital costs. An intermediate pump arrangement will pump the water from the thickener to the main pump station near to the distribution level of the shaft.

The main crusher will receive the ore from the trucks, and with the help of a conveyor, will transport the ore up to the ore storage silos near the shaft. Skip loading level, distribution drift and a main pump station will be accessed by a ramp driven from the shaft bottom drainage drift.

Figure 16-5 shows the underground capital development for the mine in relation to the open pit.

Figure 16-5 : Underground Development Layout



16.3.2 Stope level design

The SLOS mining method was used assuming the following mining philosophy:

- Each mining level is divided in mining panels;
- Each mining panel will have a length approximately equal to the Northwest/Southeast length of the orebody but with a maximum of approximately 380 m. The maximum overall width of the orebody on any level is approximately 440 m with a maximum mining panel (between the unrecoverable pillars) width of approximately 80 m to 100 m (Figure 16-6);

Figure 16-6 : Plan of 140 m Level Layout

- 25 m to 30 m wide unrecoverable pillars will be left between each mining panel. The pillars will accommodate the development of the accesses to the stopes and ore passes positioned near the stope entrances in order to decrease haulage distance;
- The pillars between each mining panel will have access drifts to the stopes. In order not to duplicate the development, the upper drift (drilling level) of a mining stope will be reused as an extraction drift on the level above of the same mining stope;
- Each stope access drift (on drilling and extraction levels) will be connected to a ventilation raise at the extremity of the orebody, that in turn will be connected to a main return airway drift;
- The upper return airway drift will be connected with main ventilation raises exhausting return air to surface. These ventilation raises will be equipped with exhaust fans, in order to ventilate the mine.

Based on the geotechnical available data the primary and secondary stope dimensions were estimated as follows:

- Width – 15 m;
- Height – 25 m;

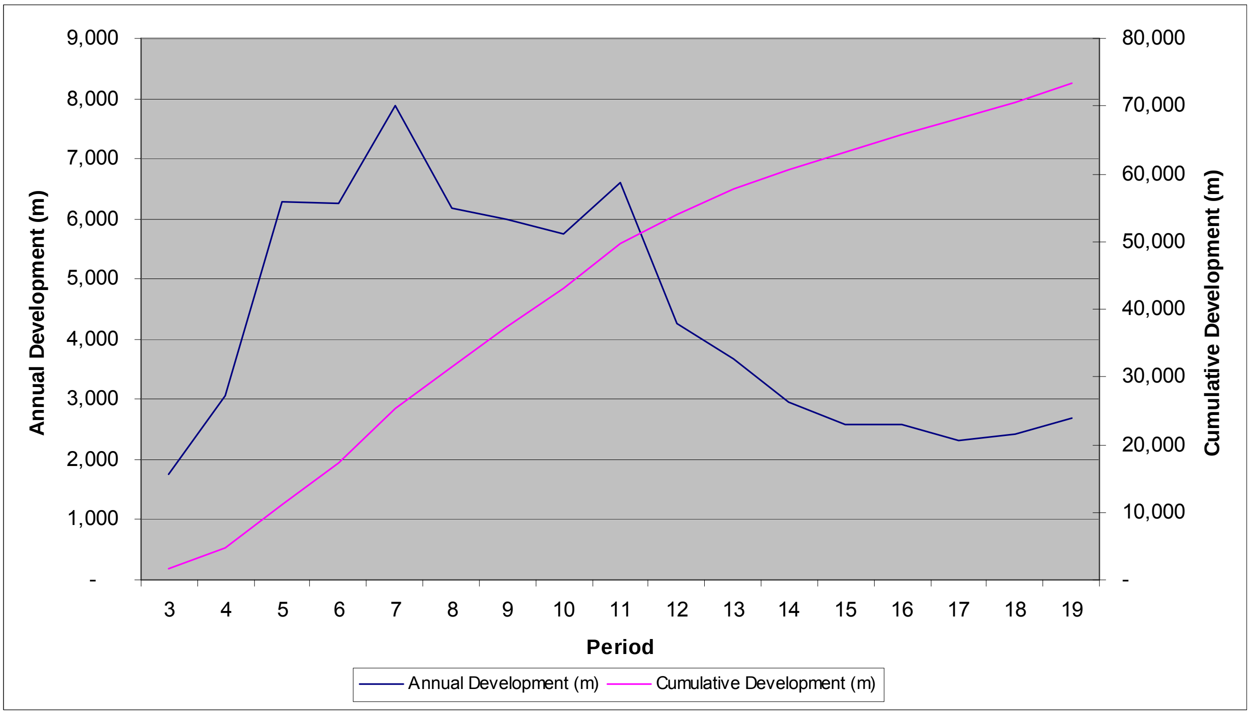
- Length – nominal 80 m to 100 m.
- Average number of 64 stopes ranging from a minimum of 41 stopes to a maximum of 75 stopes per level.

All mining horizons are arranged from the bottom of the open pit (420 masl level, a 30 m thick crown pillar from pit floor to roof of stope, incorporating 5 m high drilling drives in the pillar, is left un-mined) down to the – 105 masl level (the limit of Indicated Mineral Resources).

16.3.3 Production and Waste Development Schedules

Figure 16-7 presents the waste development.

Figure 16-7 : Waste Development



16.3.4 Underground production schedule

The production rates, showing the ROM metal grades for the open pit and underground mine are shown below in Figure 16-8 and Figure 16-10.

Figure 16-9 : Underground Production by Level

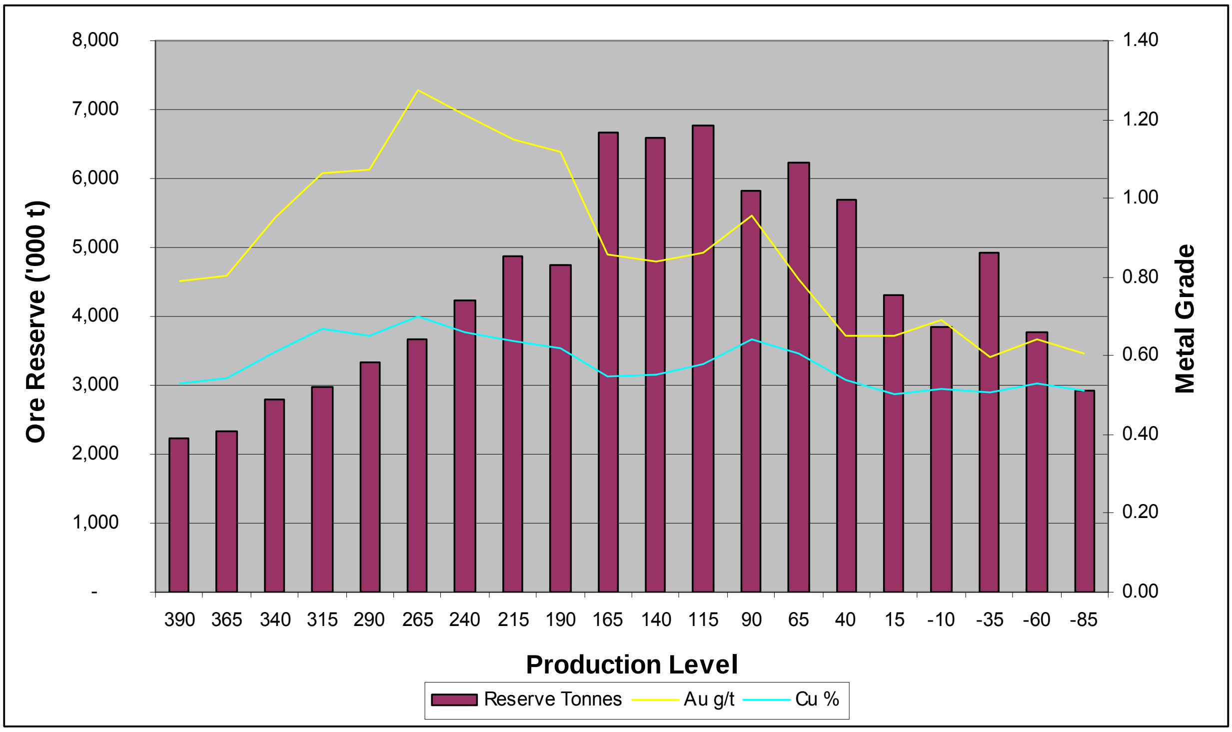
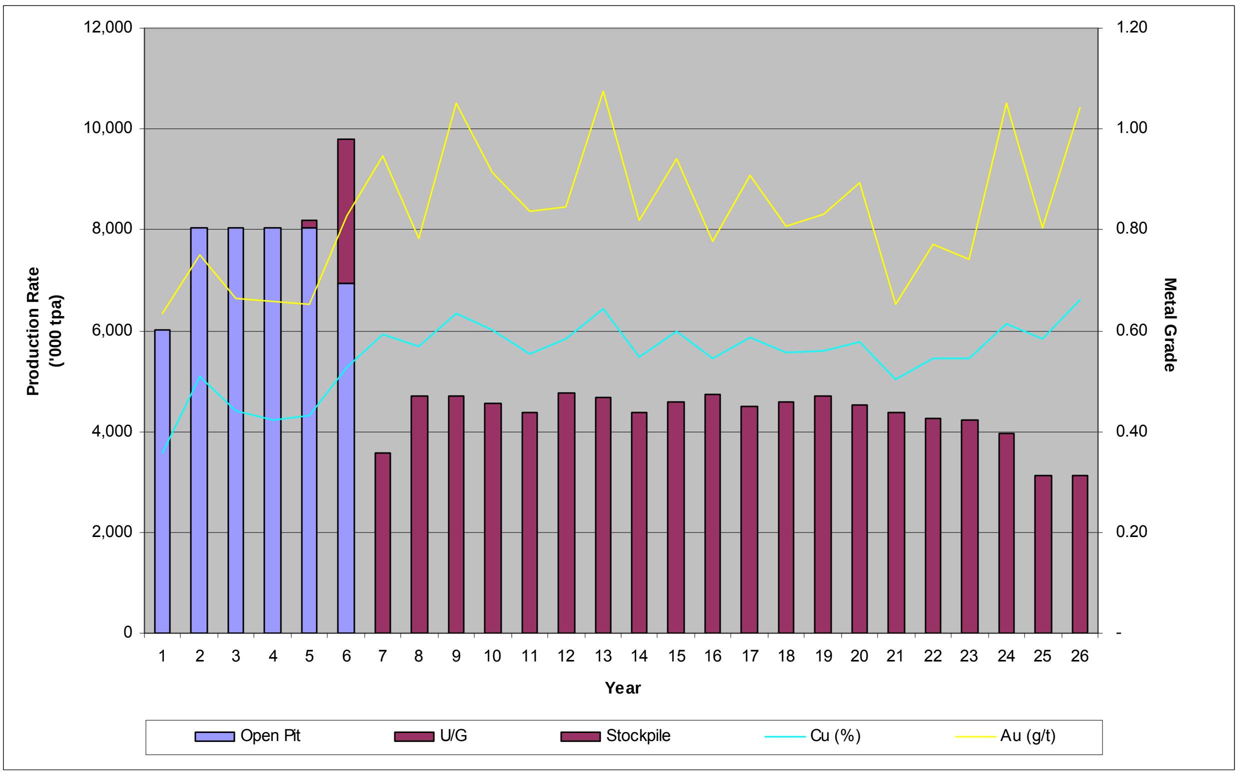
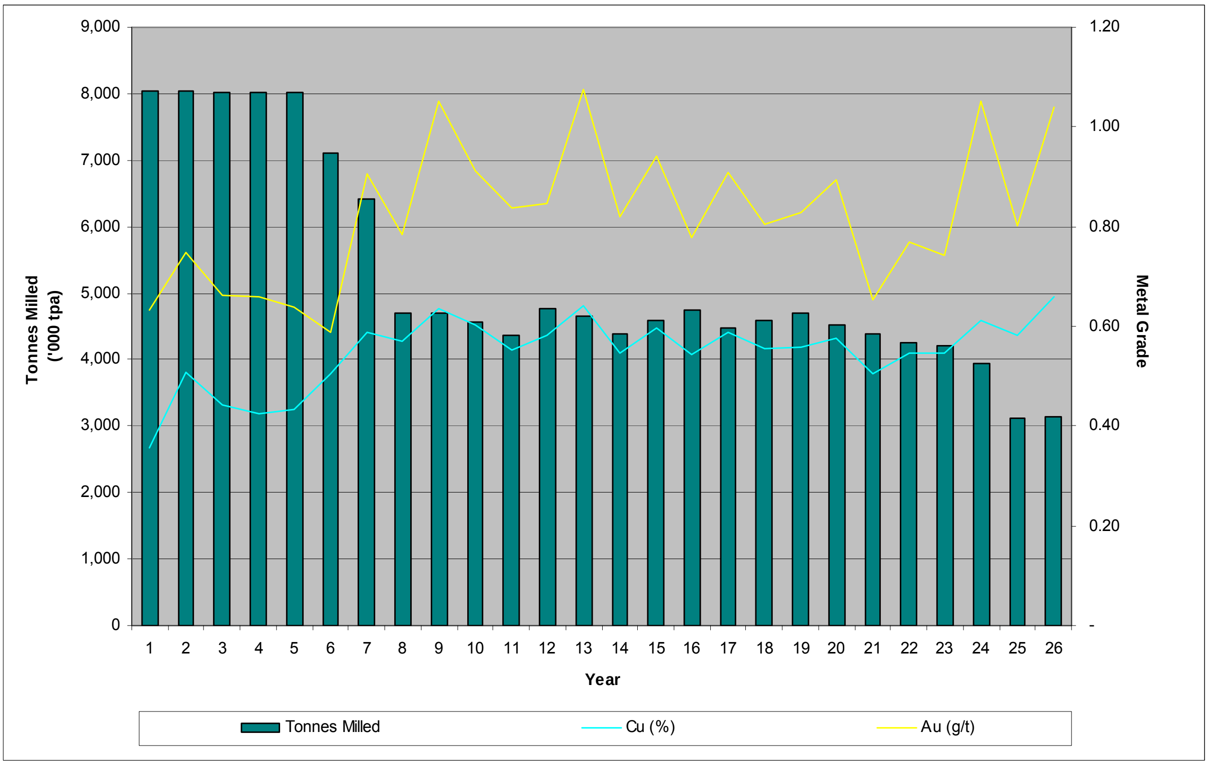


Figure 16-10 : Open Pit and Underground Production Schedule



The ore produced from the pre-production mine development in Years 4, 5 and 6 will be stockpiled on surface and fed into the plant in Years 6 and 7 to maintain as near to the 8Mtpa plant capacity as shown below in Figure 16-11

Figure 16-11 : Skouries Mill Schedule



17 RECOVERY METHODS

17.1 Process Plant Engineering Overview

The process plant and infrastructure design of the Project has been advanced from the 2007 Cost and Definition study prepared by Aker Kvaerner, (subsequently Aker Solutions and now Jacobs Engineering Group). Technical information was provided by several specialist consultants, world class metallurgical testing facilities and HG. The well known technology, engineering and equipment provider, Outotec of Finland ("Outotec"), have undertaken the Basic Engineering phase of the project, their scope of work has included the supply of equipment within their manufacturing range, grinding mills, the flotation kit, the paste thickeners and the plant control system. In parallel with this the Athens based Engineering contractor, ENOIA has completed a Basic Engineering Study and started the Detailed Engineering Study and is responsible for the process plant and infrastructure outside of Outotec's scope. ENOIA has also been appointed to provide contract services and controls for all estimate areas of the Project working under the direction of HG.

The layout of the plant has developed from the Cost and Definition study and has incorporated many improvements which have resulted in a reduction in the cut and fill requirement and therefore capital cost.

The scope of the Process plant engineering includes the surface ore reception facilities and primary crusher, coarse ore stockpile, grinding, gold gravity circuit, flotation, filtration and paste thickening. In addition, the infrastructure facilities include, the administration block, the workshops, fuel station and welfare facilities as well as power, water and other services.

The Project entails Open Pit mining for the first 6 years. The feed rate will be 8 Mtpa for Years 1 to 5 and in Year 6 the remaining 7.0 Mt open pit ore will be processed. From Year 7 the plant will be supplied from the underground mining operation with 6.4 Mt and thereafter at between 3.1 Mtpa and 4.8 Mtpa for at least the next 20 years.

During years 1-6 the plant will be treating the softer open pit ore and the selected mills will accommodate the required tonnage.

The process plant design has assumed a nominal throughput for the harder underground material of approximately 881 tph. The plant design exceeds the output from the underground mining and plant operations will therefore be based on campaign treatment of the ore so that the plant will operate at optimum efficiency as well as allowing higher throughput if an alternative more productive extraction approach is adopted or additional ore streams are identified by exploration of the Company's nearby porphyry exploration targets.

Initially, for the first year of operation, plant feed will comprise oxidised ore arising from the open pit. As the near surface oxidised ore is depleted there will be a transition to sulphide ore and for the next five years sulphide ore will be mined from the pit. Thereafter the mining operation will move underground where sulphide ore will be extracted. The primary products from the process plant will be a high quality gold-copper concentrate and gold doré ingots which are estimated to contain approximately 30% of the gold contained in the ore feed.

Two design cases were evaluated by Outotec; the first being the nominal design case processing underground sulphide ore, while the second case reflects the higher tailings and concentrate production scenario:

Table 17-1: Evaluated Design Cases for Plant Study

	NOMINAL DESIGN (CASE 1)	DESIGN CASE 2
Concentrator Fresh Feed Rate (t/h) & Design Factor (X)	881 (1.00)	1,013 (1.15)
Cu in feed (%)	0.54	0.54
Au in feed (g/t)	0.83	0.83
Basis	Life of Mine averages	Life of Mine averages
% Cu in rougher conc.	5	5
% Cu overall recovery	91.1	91.1
% Cu in final conc.	26	26
% Au recovered by gravity	30	30
% Au overall recovery	84.0	84.0
SAG pebble circulating load, %	0 – 25	50

The design of the process plant is based around a conventional flow sheet for the treatment of porphyry copper ores and as such offers a well proven design using conventional equipment. This coupled with the straight forward ore metallurgy and design margins will ensure that it offers a robust, low risk processing solution to the treatment of the Skouries ore. There are many plants operating successfully in the world using the Skouries design. The Alumbrera mine in Argentina is a recent example which treats similar copper/ gold ore and has proved to be a success.

17.2 Process Description

The majority of the information in this section of the Technical Report is founded on the Outotec Skouries Basic Engineering Study and work by ENOIA of Athens.

The Skouries Concentrator Process Flow Sheet is outlined in Figure 17-1 shown at the end of this Section.

The process plant design currently provides for a nominal 8 Mtpa throughput for the softer Open Pit ore and up to 4.8 Mtpa for the underground ore. The unit operations comprise gyratory primary crushing, the coarse ore stockpile and reclaim system, single-stream SAG/Ball Mill grinding and cyclone classification, free gold recovery by gravity separation and production of doré, flotation, tailing thickening and disposal and copper/gold concentrates thickening and filtration. The plant is of conventional design and comprises equipment of the type and size that has been proven in plants processing mineral ores elsewhere in the industry and throughout the world.

The design is conservative with the inclusion of a pebble crushing circuit, coarse and fine free gold recovery circuits and a doré production facility.

An overall plan of the plant layout is shown in Figure 17-2 : Plot Plan of the Skouries Process Plant.

17.3 Coarse Ore Stockpile

The primary crushed ore from the open pit crushing station and at a later stage from the underground mine will be conveyed and then discharged onto a covered conical stockpile. The stockpile will have a live capacity of 21,000 t, about 33% of total volume, equivalent to 24 hours of nominal mine production. The total stockpile storage capacity will be approximately 63,400 t, equivalent to three days of mine production. The stockpile level is controlled with microwave radar. Primary crushed ore from the open pit will be transported to the stockpile by a belt conveyor. The conveyor is equipped with a belt scale and an over-band magnet to remove any scrap metal before entering the stockpile.

When required, a bulldozer from the open pit mine can reclaim the dead sections of the stockpile.

17.4 Ore Feed to the Grinding Circuit

Material will be reclaimed from the coarse ore stockpile at up to the maximum concentrator design rate using three variable speed apron feeders, (two duty one standby). These are located beneath the stockpile in a tunnel and will each discharge onto a belt conveyor which feeds the primary grinding circuit SAG mill. The reclaim feeder positions and layout has been designed to maximize the stockpile live capacity.

Automatic dust suppression sprays will operate with each of the apron feeders. Spray water will be collected on the inclined reclaim tunnel floor where it flows to the discharge end and is collected in a floor sump and transferred by pump to the site drainage system via a solids settling tank.

All belt conveyors will have following controls:

- Blockage detector for discharge chutes (alarm and then belt trip out)
- Speed controller for belt slippage (interlocks)
- Four alignment switches for belt position (alarm followed by belt trip)

17.4.1 SAG- and Ball Mills

The grinding circuit is designed to produce primary ground product with the required P80 of 120 μm. The liberated mineral particles are produced by a two-stage wet grinding circuit comprising the SAG Mill and the Ball Mill. The fine slurry product is directed to flotation with a bleed from the cyclone underflow stream to the gravity circuit. The grinding circuit feed size has a nominal F80 of 150 mm and F100 of 300 mm.

The transfer size between the SAG- and Ball mill is in the range 2.0 mm to 3.6 mm.

The SAG mill diameter is 9.75 m (32 ft) with an effective grinding length of 4.57 m (15 ft). The mill has a grate in the discharge end liners and is installed with twin drive powered by two asymmetrically installed 4.8 MW motors. This is to provide easy access to the shell for liner changes and maintenance. The motors are variable speed to provide operational flexibility and to accommodate variations in ore grinding characteristics and plant feed rates.

The design/maximum SAG mill ball grinding media charge will be 20%. The grinding media is balls of 125 mm diameter made from high quality forged steel to resist breakage and distortion. The grinding media is charged into the mill feed chute manually using hoisted kibbles. The kibbles are filled by a front-end loader. The SAG mill will be lined with high specification Cr-Mo alloy cast steel liners.

The ball mill diameter is 7.01 m (23 ft) with an effective grinding length 9.75 m (32.5 ft). The mill is the overflow type and is driven by twin pinion drives powered by two 4.8 MW slip ring motors.

The maximum ball mill grinding media charge will be up to 35% of the mill volume. The grinding media is charged manually using hoisted kibbles and will depend on the mill's power draw. Forged steel balls 60 mm in diameter are used. The ball mill will be fitted with steel liners with a similar specification to those installed in the SAG mill with an option of having rubber liners as an alternative retrofit.

Both mills will include trunnion bearing and drive gear lubrication, protection- and cooling systems. All important mill parameters will be measured and used in the mill control strategy, where relevant. These will include mill motor power draw, SAG mill weight, fresh and total SAG mill feed rate, mill speed, lubricant flow, pressure & temperature. The mill pinions temperatures will be measured by an array of infra-red detectors for the early detection of drive misalignment.

Two liner handler machines will be used for relining the mills. An overhead crane serves the primary grinding area for maintenance purposes. The grinding area plant floor will be inclined towards two floor sumps. These sumps are provided with pumps which return spillages to grinding discharge sump.

17.4.2 Primary Grinding Circuit Operation

Two multi-idler type belt weightometers installed on the SAG feed conveyor measure the fresh feed and the total feed to the SAG mill. The difference between belt scale figures is the circulating pebble load to cone crusher circuit.

The SAG mill will operate with a solids content of about 72 w/w-%. Water fed to the SAG mill is controlled at the required ratio to the combined ore feed. The SAG mill product will flow through the discharge trunnion onto a vibrating polyurethane wash screen having a total area 2.9 m x 9 m with square openings of 12 mm x 12 mm. Screen undersize will flow to the common discharge sump with the ball mill discharge.

Two pumps, 1 duty, 1 stand-by, will pump the products of both mills to a hydro cyclone cluster comprising nine 650 mm diameter cyclones, 1 stand-by, for particle size classification. The cyclone overflow slurry will have a target solids content of 35% w/w and particle size P80 is 120 μm and will flow by gravity to the flotation section of the plant. An automatic pump chamber flushing system will be installed for both grinding circuit pumps.

The SAG screen oversize pebbles, >12 mm, will fall onto a conveyor system which transports pebbles into a 100 t capacity surge bin. The surge bin is necessary to compensate for changes in ore characteristics and consequential pebble generation rates before feeding to a cone crusher, there will be one running with one standby unit. The bin is equipped with an overspill chute, which discharges pebbles on to the ground in case excess pebbles are generated.

Vibrating feeder at the outlet of the pebble surge bin will ensure that the downstream cone crusher will be choke fed. This will prevent premature liner failure due to pebble channelling. The first pebble conveyor will be equipped with an over band tramp metal magnet above its discharge end. The magnet will continuously remove magnetic tramp steel into a transportable collection bin. A metal detector installed on the second conveyor will control the flow of non-magnetic tramp metal away from the pebble bin and crusher to a reclaim bin on the ground.

The cone crusher will have dedicated feed controls, lubrication and protection systems, which will be connected to the plant's main control system. The crushed product, nominally with a P80 of 16 mm, will be transported back onto SAG mill feed conveyor. In case of pebble crusher outage, the pebble bin will overflow directly onto the crusher product discharge conveyor.

The combined SAG mill screen undersize and ball mill discharge will be collected in the common sump and pumped by a variable speed pump to the hydro cyclone cluster for classification. The sump and pumps are located below floor level in a special open top chamber having raised walls high enough above the floor level to prevent flooding from the plant's floor. This chamber is equipped with a large drainage channel to the outside of the plant building to prevent flooding in the event of a catastrophic failure of the pumps or pipe lines and reducing the likelihood of burning out the electric motors driving the pumps.

Approximately 20 % of the cyclone underflow will be directed from a splitter box over a 2 mm aperture guard screen. The undersize of the screen will be pumped to the primary gold gravity circuit. The screen oversize along with the remaining 80 % of the cyclone underflow will fall directly into the ball mill feed chute.

The main plant control room will be located at an elevated level adjacent to the concentrator grinding and flotation sections to provide a panoramic view of operations and ready access into the plant.

17.4.3 Primary Grinding Circuit Control

The SAG and ball mill grinding circuit control will be implemented by means of a Siemens PCS7 central control system.

The feed rate to the SAG mill circuit is controlled by adjusting the operation of the reclaim feeders under the ore stockpile. The speed of the conveyor transporting the reclaimed ore to the mill is adjusted to ensure the conveyor is optimally loaded. The vibrating screen to process SAG mill discharge is sized so that up to around.40% – 50% of the mill product can be returned to the SAG mill after being passed through the pebble crusher circuit.

SAG mill operation is optimised by adjusting the mill speed to match both the feed rate and the ore hardness in combination with adjusting the charge level in the mill.

The energy absorbed by the SAG and ball mills will be measured by accurate mill power meters. The SAG mill charge level will be measured by monitoring the backpressure on its central lubrication system and through the mill charge analyzer device Millsense® which is connected to the plant control system.

To achieve optimum classification results as mill circuit feed and circulating load rates vary, hydro cyclone performance will be adjusted by controlling the cyclone feed density and feed pressure. The feed density is controlled by adjusting grinding sump process water addition and the feed pressure by varying the number of cyclones in operation by operation of their actuated feed valves. The grinding circuit will be installed with an on-stream particle size analyzer, PSI 300® to monitor the fineness of hydro cyclone overflow. The particle size sampler is installed in the cyclone overflow pipe.

The concentrator control system will include a grinding circuit expert control system ACT®, Advanced Control Tool which will maximize the amount of fresh feed to the grinding plant while maintaining the required product quality.

Space is provided for a flash flotation Skim Air® SK-2400 cell for possible later installation allowing the option of treating cyclone underflow during sulphide ore treatment which should reduce feed to the primary gravity circuit.

17.5 Flotation

The froth flotation circuit comprises a roughing flotation stage followed by three concentrate cleaning stages and a first stage cleaner tailings scavenger stage. Flotation cells are mechanically agitated, forced air flotation cells, which represent the latest proven technology used by Outotec. Rougher and 1st cleaner-scavenger concentrates are reground before being further upgraded in the cleaner circuit.

The cells are connected in series with two consecutive cells forming a step having the same slurry level. The level of each step is controlled by dart valves located at the outlet of each pair of cells. The level is monitored with a radar level transmitter.

Flotation air is produced by two multistage centrifugal air blowers, one duty and one standby. The airflow to each rougher cell is individually controlled, for the other flotation stages the control is for the pair of cells together. The air feed control is carried out by means of thermal air flow meters and butterfly control valves.

Flotation circuit pumps are horizontal type wear resistant variable speed slurry pumps. Pump speeds and hence capacities are controlled according to level measurements in the respective pump suction sumps. For each duty there are two pumps, one on standby, equipped with a flushing system to clean the standby pump and piping of slurry.

Copper and gold concentrations in important flotation process streams as well as the slurry density are analyzed by the Courier 6SL on-stream analyzer.

Flotation area floor sumps will collect any spillage which will be pumped by vertical sump pumps to appropriate points in the process.

17.5.1 Rougher Flotation

The primary cyclone overflow will flow by gravity through a launder sampler into an agitated conditioning tank at the beginning of the flotation circuit. There the slurry will be conditioned with a primary dose of flotation reagents. The sampler will serve both on-line analyzers, particle size and Cu grade. The pH and redox potential of the rougher flotation feed is continuously monitored.

Conditioned slurry feed will overflow from the conditioner into the feed box of rougher flotation bank. The bank will comprise of 10 floor-mounted Outotec TC-160®) Tank Cells. The Tank Cells each have a live volume of 160 m^3 and are arranged in a zigzag configuration, stepwise, providing controlled tails gravity flow between adjacent pairs of cells. Additional reagent dosing is provided to different cells as required by the process measurement and control system.

Rougher concentrate will flow by gravity in open launders through a sampler to the regrind mill discharge sump. The tailings are discharged by gravity through a dedicated sampler into the tailings receipt sump and from there, combined with other streams, primarily the scavenger cleaner tailings through another sampler to the tailings thickeners distribution box.

17.5.2 Regrinding

Rougher and cleaner-scavenger concentrates flow to the regrind mill discharge sump. Regrinding will be carried out by a ball mill 4.60 m, (15.1 ft), in diameter and with a 7.00 m, (23 ft), effective grinding length. The regrind mill has rubber liners and grinding media will be forged steel balls, 25 mm diameter. The mill will be driven by a single 2,25 MW slip ring motor and will include full lubrication, drive protection- and cooling systems.

All important mill parameters will be measured and where relevant, used in the mill control strategy. These will include mill motor power draw, discharge pump box water addition, mill charge level, together with lubricant flows, pressures and temperatures. Grinding media will be charged manually to the mill feed chute using a hoisted ball kibble depending on the mill's power draw.

The mill operates in circuit with a cluster of 12 hydro cyclones (10 duty, 2 standby). Regrind circuit operation will be optimized by the plant's central control system. The cyclone operation is controlled by adjusting the cyclone cluster feed density, slurry volumetric flow and inlet pressure by reducing the number of cyclones on-line in the cluster at low tonnages and reduced mill circulating loads.

Regrind cyclone underflow will be divided into two separate flows, a 20 to 25% proportion will flow to a 2 mm aperture guard screen located at the beginning of the fine gold gravity recovery circuit. Screen oversize along with the remaining 75 to 80% of cyclone underflow will fall directly into the regrind mill feed chute. After further processing the tailing from the fine Au gravity circuit will be returned to the regrind mill feed sump. Cyclone underflow will be divided with actuated dart valves from the cyclone splitter box.

In addition to the rougher and 1st cleaner-scavenger concentrates the regrind gravity gold circuit tailings plus regrind mill discharge will be collected in the regrind mill sump and pumped to the regrind cyclone cluster by pumps, one running with one on standby.

Space is again provided for the possible retrofitting of a flash flotation unit, the Outotec SkimAir® SK-240 cell to reduce the mass flow and to upgrade the feed to the regrind gravity circuit.

To achieve a correct pH level, lime solution will be added to the regrind mill discharge sump. pH and redox potential in the cleaner flotation feed will be continuously monitored and the addition rate of lime controlled through the plant's central control system.

The regrind circuit product target size will be a P80 is 34 μm. Control of this size is achieved using a Particle Size Indicator and this will ensure optimum liberation of locked gangue minerals prior to further concentrate upgrading in the downstream flotation cleaning stages. Cyclone overflow will be sampled which will be used for the on-line particle size- and on stream copper grade measurement.

The regrind mill cyclone overflow product will flow by gravity to an agitated conditioning tank located at the beginning of the flotation cleaning circuit. Here further flotation reagents will be added to chemically condition the fresh mineral particle surfaces generated in the regrind mill.

Overflow from the Conditioning tank will be directed to the First Cleaner flotation Tank Cell.

17.5.3 First Cleaner Stage

The first cleaner stage comprises of four 50 m^3 TC-50® flotation Tank Cells arranged in a 2+2 zigzag configuration. The flotation cells will be installed in a stepwise fashion to facilitate gravity flow. First cleaner flotation feed slurry pH and redox potential will be continuously monitored.

The first cleaner concentrate will be directed to a pump sump where it is combined with the tailings from the third cleaner and directed by a pump, one running with one on standby to the second cleaner feed box. The 1st cleaner concentrate will be sampled for on-line Cu analysis. The first cleaner flotation bank tailings will flow by gravity to the 1st cleaner-scavenger feed box and this stream will also be sampled for on-line Cu analysis.

17.5.4 Cleaner-Scavenger Stage

The cleaner-scavenger flotation bank will comprise six 50 m^3 Outotec TC-50® flotation Tank Cells arranged in a 2+2+2 zigzag configuration. The redox potential of the feed to the cleaner-scavenger flotation is continuously monitored.

Concentrate from this flotation stage will be returned by gravity to the regrind mill discharge sump and will be sampled for on-line Cu grade analysis. The cleaner-scavenger tailings will be discharged into the combined concentrator tailings sump and will also be sampled for on-line Cu grade analysis.

Second Cleaner Stage

The second cleaner bank comprises five 10 m^3 flotation Tank Cells arranged in a 2+2+1 bank configuration. The redox potential of the second stage cleaner flotation feed is continuously monitored.

Second stage cleaner flotation bank tailings will flow by gravity to the first cleaning stage conditioning tank and will be sampled for on-line Cu grade analyses.

The second stage cleaner concentrate will be directed by pump, one running, one standby, to the third cleaner flotation bank feed box and the stream sampled for on-line Cu analysis.

17.5.5 Third Cleaner Stage

The third cleaner bank will comprise four 10 m^3 flotation Tank Cells arranged in 2+2 bank configuration. Flotation stage tailings will flow by gravity to the first cleaner concentrate pump box.

Final concentrate produced by the 3rd cleaner stage will flow by gravity to a sump box from where it will be pumped by one running and one standby pump to the final concentrate thickener feed box. The final concentrate will be sampled for on-line Cu grade analysis.

All critical process streams within the flotation circuit will be continually sampled and the sample material will be transferred to the concentrator on-line elemental grade analyzer and for some streams also to the on-line particle size analyzer before being returned to suitable locations within the processing circuit. These on-line analysis systems provide real-time metallurgical and particle size analysis. Circuit recovery data will be calculated and displayed in the concentrator’s central control room by means of the computer based Process measurement and control system.

17.6 Gold Recovery Circuits

The gold gravity circuit will be accommodated in a tower arrangement to maximize the use of gravity flow between successive processing stages. For security purposes the gold gravity circuit will be located in a steel framed building with block and/or security mesh infill. Access to this area will be strictly controlled and monitored. CCTV cameras will be installed on each floor for remote surveillance from display screens and video recording equipment located in both the plant central control room and the Gold Room Security Office.

17.6.1 Coarse & Fine Gold Primary Gravity Concentrating Systems

The partially diluted undersize slurry from each of the 2 mm guard screens at the beginning of the coarse and fine gold gravity recovery circuits will be separately collected in their respective pump boxes and transferred by dedicated pumps, the first one to a primary splitter box and the latter to a regrind diverter box.

Coarse ore slurry from the primary grinding circuit will be equally distributed across three fully automated water flushed centrifugal type gravity concentrators. Tailings from these machines will return continuously by gravity back to the combined SAG and ball mill sump.

Each of the above gravity concentrators will automatically be taken off-line in turn at controlled and sequenced intervals while their accumulated concentrate products are back-flushed and discharged by gravity to the head of a secondary coarse gold gravity concentration circuit.

Fine slurry from the concentrate regrind circuit will be similarly treated in a single water-flushed fully automated centrifugal type gravity concentrator. Tailings from this unit will return by gravity to the regrind mill sump.

Concentrate will be periodically back-flushed from the fine gold gravity concentrator and will gravitate to the secondary fine gold gravity concentration circuit feed system.

17.6.2 Secondary Coarse & Fine Gold Gravity Concentration Systems

Primary gravity concentrates will be upgraded separately from the regrind material given the differences in both their size distributions and mineralogical constitution. These differences dictate the need for different process conditions to optimize gold recovery and grade. Each of the separate process streams will be operated continuously although some provision will be made for intermediate storage of primary concentrates to allow for maintenance of the secondary gravity concentration devices.

The final concentrates and tailings from each of the separate secondary gravity circuits are combined prior to their final treatment and transfer.

Diluted batch flows of coarse primary gravity gold concentrate will be collected in a spiral densifer from where the clarified overflow is directed to the combined secondary gravity circuit tailings sump.

The densified solids will be reclaimed at a controlled rate and discharged continuously over a 500 mm x 1,500 mm vibrating grit removal screen fitted with a 500 μm aperture deck. A regulated flow of screening spray water will ensure that the nominally gold barren screen oversize is washed before being delivered by chute to the combined secondary gravity circuit tailings sump. The grit screen undersize will be diluted by the screen sprays to an optimum solids concentration of about 25% by wt prior to final processing. Prior to flowing by gravity into an agitated holding tank the screen undersize will first pass through a wet drum magnetic separator where stray steel grinding media particles will be removed and directed to a pump sump.

The coarse gravity concentrate will be continuously pumped by one on duty with one standby pump from the agitated holding tank to a head box. The majority of the head box feed will return as overflow back to the agitated holding tank while a bleed will be continuously discharged into the feed box of a coarse gold shaking table where the concentrate will undergo a final stage of upgrading.

Dilute batch flows of fine gold primary gravity concentrate will be collected in a decant vessel where it will be held long enough for the solids to settle. A time-sequenced automatic top up water decant valve will add controlled flows of clarified process water into the combined secondary gravity circuit tailings sump. Once partially decanted and now at an optimum solids concentration, the remaining primary fine Au gravity concentrate will be discharged automatically into the agitated holding tank below.

Fine gravity concentrate will be continuously pumped by one on duty plus one standby pump from the respective agitated holding tank onto a 500 mm x 1,500 mm vibrating grit removal screen fitted with a 500 μm aperture deck. Nominally gold barren screen oversize will be discharged through a chute to the combined secondary gravity circuit tailings sump. The undersize from the grit screen will flow through a dedicated wet drum magnetic separator where stray steel grinding media particles will be removed. Underflow from the magnetic separator will be collected in the head box from where most of the flow will return back to the agitated holding tank by gravity. A bleed flow will be continuously discharged from the feed box to the fine gold shaking table where the concentrate will undergo a final stage of upgrading.

Tailings from both the coarse and fine gold shaking tables will be collected and fed through a small semi-automatic water flushed 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 where the combined gravity tailings will be pumped by one on duty with a standby pump into the SAG and ball mill discharge sump. The concentrate from this stage will be directed to the fine gravity shaking table feed holding tank.

The final gold gravity concentrate from both shaking tables will be discharged into a sealed concentrate storage box. Solids will settle out while free water will overflow into a launder, which will be directed to the secondary gravity tailings sump.

The partially filled concentrate storage box will be transferred by an electric overhead runway beam hoist into the adjacent Gold Room once a day. A second concentrate storage box will cycle with the first one.

17.6.3 Gold Gravity Concentration, Calcining & Smelting

The Gold Room will be constructed of reinforced concrete with a cast concrete floor and ceiling. It will be provided with a forced air ventilation system. Personnel access will be restricted and through a separate secure entry system incorporating personnel search facilities. An emergency means of egress plus a set of vehicle reversing doors for loading bullion transit vehicles will be provided together with a further set of security CCTV cameras which will be remotely linked to screens and video recording equipment in the main plant control room and the Gold Room Security Guard Office.

Final gravity concentrate delivered into the Gold Room will be manually decanted and transferred into stainless steel trays before being loaded into one of two drying and calcining oven shelves. Any sulphide fumes generated from the ovens will be collected in a waste fume stream and passed through a wet scrubber before being vented into the atmosphere.

Pre-dried and calcined material removed from the ovens will be temporarily stored on cooling racks located within the Gold Room extra secure Bullion Store which has its own safe vault door system.

Nominally once per week the cooled calcine will be blended with fluxes and transferred into an oil-fired smelting furnace. The resultant smelt will be poured over a series of 10 kg capacity cascade bullion moulds terminating in a slag pot. After cooling, cleaning, weighing and sampling the resultant doré gold bars will be transferred into the Gold Room Bullion Store to await secure shipment offsite to market. Smelting slag will be crushed within the Gold Room before being returned manually and tipped onto the SAG Mill Feed Conveyor for re-processing within the concentrator.

17.7 Flotation Reagents

The reagents used in the flotation circuit for the Skouries sulphide ores will be sodium isopropyl xanthate as the primary collector, Aeropromoter MX-5010, or an alternative thionocarbamate equivalent, as the promoter, methyl isobutyl carbinol as the main frother and Dowfroth 250 as auxiliary frother.

An additional reagent mixing facility is provided to act as a spare system in the event of maintenance needs and for evaluating new and/or additional reagents.

During the treatment of oxidized ores from the open pit, a sulphidiser will also be used. Continuous Eh measurement to control the correct stepwise dosage of the sulphidiser will be provided.

Milk of lime will be used as the pH modifier in the cleaner flotation and concentrate regrind circuits. Provision for lime addition in the rougher circuit has also been made should it be required when processing any acidic oxide ore blends. Lime storage and milk of lime preparation and distribution takes place in a dedicated building.

A separate building will house flotation collector, promoter and frother reagent mixing and transfer systems. The flocculant preparation system for the concentrate thickener is also located in this building. Drums and reagent bulk bags will be lifted onto a charging floor above the reagent mixing tanks by a travelling electric hoist. Sufficient drum and bulk bag storage will be provided for approximately one to two days consumption. An adjacent central reagent store will contain a further 60 days´ supply of all plant reagents.

The reagent building will be fully ventilated and provided with a safety shower. Reagent spillage within the separately bounded reagent preparation hall zones will be minimized by the selection of “leak free” reagent transfer and circulating pumps. Spillage and wash-down water flows within this area of the plant will be transferred by dedicated spillage pumps into the tailings thickener feed system.

Sulphidiser is stored and prepared in a separate temporary facility, equipped with an electric hoist plus a bulk bag handling system, a ventilation system and spillage control facilities.

All waste reagent drums will be crushed on site to facilitate ease of disposal in a responsible and environmentally sound manner.

Chemical addition points and dosing rates will be determined from the plant process measurement and control system to achieve optimised metallurgical performance and hence production to accommodate any variance in ore characteristics.

17.8 Flocculant

Polyelectrolyte flocculant solution will be added to the concentrate and tailings thickeners and to the mine water clarifier to aid the settling rate of solids and maintain clarity of their respective overflows. The flocculant solutions are prepared in two different systems, located also in different sections of the plant. One system serves the concentrate thickener and is located in the reagent building adjacent to the main concentrator building. The other system is located at the tailings handling area and serves the tailings thickeners and the mine water clarifier,

Flocculant for the concentrate thickener is prepared in a fully automatic compact mixing and storing unit and prepared once per day. The mixing and storing solution concentration is 0.5%. When the storage tank reaches the low level, the ready mixed solution flows to the storage tank by way of gravity and then dosed to the thickener as required.

The flocculant for the three tailings paste thickeners is prepared by a fully automatic continuous mixing process. The flocculant is delivered in 800 kg bulk bags and each bag represents more than one day's normal consumption. The mixing tank capacity will be a minimum of two bulk bags and therefore the system does not need attention during weekends. The mixing and storing solution concentration is 0.5% and the mixing takes place by continuously feeding the powder at a set rate to a set water flow to the mixing tank. The mixed solution is pumped continuously to the storage tank. The storage tank has a live holding capacity of 14 hours. The actual pumping rate to the storage tank is controlled by the variation of solution level in the storage tank and the set points to the flocculant and water feed to the mixing tank are controlled according to the fluid level in the mixing tank. The solution volume in the mixing tank must remain high to ensure proper mixing time. The 0.5% solution is continuously pumped to the feed well of each tailings thickener by dedicated variable capacity positive displacement pumps, three will be actively on duty with one on standby. After the dosing pumps the solutions will be diluted to around 0.1% wt with fresh water using flash in-line mixers.

The flocculant preparation and dosing equipment for the mine water clarifier is located in the same building. The dosing rate is low and therefore the solution will be prepared once a week to the final concentration of 0.1%.

The Tailings flocculant preparation facility is located in the tailings handling area remote from the general reagent store, therefore storage for 60 days consumption, i.e. about 45 bulk bags will be provided.

17.9 Thickening

17.9.1 Concentrate Thickening

Final concentrate slurry from the 3rd cleaner flotation stage will flow by gravity to a pump sump from where it will be pumped to the concentrate thickener feed box which also serves for de-aeration.

The concentrate will then flow to the centrally located thickener feed well where the flocculant solution is mixed in. The thickener has been sized at 12 m in diameter with a 2.4 m wall height and is equipped with a centrally driven rotating rake mechanism. The mechanism includes high torque alarms plus an automatic rake raise and lowering system to prevent damage to the drive system and reduce the danger of "bogging" or "rafting" the rakes.

The thickener underflow cone is provided with a pressure transmitter which indirectly measures the mass inside the thickener tank. Thickener underflow slurry density will be measured with density element in the underflow pipe after the pumps; either of these measurements can be chosen to control the capacity of the underflow pumps to maintain the required solids content of about 60% by weight.

The thickener bed level is measured by a mechanical float and flocculant addition rate is controlled according to this measurement to maintain a steady bed level. All instrumentation will be connected in to the plant’s central control system.

The underflow pumps, one operating with one standby pump, the thickened concentrate into the agitated filter feed storage tank which will serve as a feed buffer for the downstream concentrate filtration unit. In case of a short shut down of the concentrator process, these pumps are used to circulate the slurry back to the inlet of the thickener.

Concentrate thickener overflow will be collected in a circumferential launder and will flow by gravity into a holding tank adjacent to the thickener from where it will be pumped by one running with one standby pump, into the tailings receipt sump for return to the process.

17.9.2 Tailings Thickening

Combined flotation tailings from the rougher and cleaner scavenger sections plus concentrate thickener overflow water are collected in the tailings receipt sump located inside of the concentrator building. The slurry then flows by gravity to the tailings thickeners distribution box located on the ground above the thickener units. Three adjustable dart valves are located in this distribution box and will split the flow equally to each of the three tailings paste thickeners.

The paste thickeners will each have a diameter of 20 m and have an 8 m wall height. The rake mechanisms will be raiseable, be centrally driven and be installed with stall and high load protection functions.

The thickened product will have a nominal 70% wt solids content. This underflow is then pumped to the tailings management distribution sump. The underflow pumps are also used for re-circulation of the slurry when needed.

Clarified water overflow from each of the tailings paste thickeners will be collected and directed to the concentrator’s central process water storage tank for re-circulation back to process.

An emergency catchment pond is located to cater for the tailings thickeners at a lower elevation. This pond will be used to hold thickened paste discharge if the thickening plant needs to be shut down for a prolonged period of time or in the event of a thickener failure. Any excess spillage within the thickener area and overflow from the thickener feed distribution box will also gravitate to the emergency tailings pond. The pond will be cleaned out after any spillage events so as to provide the full storage capacity

17.10 Mine water Clarifier

Water from the mine water collection pond will be treated in the mine water clarifier. The settled solids will be transported to the tailings thickener feed box on a batch basis as required by the underflow density and the clarified overflow water will be used as process water or fresh water according to the composition.

17.11 Concentrate Filtering, Storage And Loading

Final concentrate slurry will be dewatered to meet the required Transportable Moisture Limit.

Thickened final concentrate will be stored in the mechanically agitated buffer tank located at the head of the concentrate filtration circuit. This tank will offer 12 hours´ holding capacity of filter plant feed based on the maximum design rate of concentrate production.

The buffer tank will provide feedstock to a single fully automatic pressure filter located on a raised platform. Thickened concentrate slurry will be pumped into the filter.

Space has been reserved for a second parallel filter press.

The filter will produce a cake with the required Transportable Moisture Limit for the shippers. This will be as low as possible to reduce transport costs.

The filter press will operate in a semi-continuous mode with a combination of hydraulic and pneumatic pressure with compressed air filter cake drying. The operation will be fully automated. The filter package will include automatic cloth washing and feed pipe flushing between each successive cycle.

The filtrate which will contain drying air will flow to a de-aerating chamber and filtrate holding tank. The filtrate will then be pumped by one running with one standby pump back to the concentrate thickener.

The filter cake will discharge from the filter press and fall through hinged drip collection tray doors into the concentrate storage area below.

The concentrate will be stored in a fully enclosed shed provided with air inlet louvers and a dust and fume extraction system to maintain a safe and environmentally acceptable working environment. The concentrate shed will offer approximately 3,500 t of cake storage. This is equivalent to a minimum of 5 days´ production at the maximum design concentrate production rate, 9 days at the nominal production rate.

A front loader will be provided to move any concentrate away from the immediate filter discharge zones and will also be used to load the concentrate haulage trucks when required.

Prior to leaving the site the loaded concentrate haulage trucks will pass through a wheel wash system. A dedicated sump pump will pump the wheel washings back to the concentrate thickener.

The empty and loaded trucks will be weighed over the weighbridge as part of the metallurgical accounting procedures.

17.12 Tailings Management Facility and Tailings Water Reclamation

The design and selection of the proposed tailings paste distribution system and TMF was carried out by Golder Associates. Vendors undertook the testwork to specify the paste thickeners. Further details of this system are given in the Golder Associates reports that form part of the overall Skouries study and are summarised in the next section of this document.

The paste tailings, with a nominal solids content of ~70% by weight will be delivered to the tailings management facility “TMF” through pipes and will consolidate to a final deposited concentration of approximately 80% wt solids. Water seepage from the paste plus rain water run-off and TMF catchment water less seepage and evaporation losses will be collected at the lowest point of the TMF surface and transferred to the Concentrator. Reclaim rates will be maximised within the limits of the concentrator's requirement for process make-up water so that the free liquid level within the TMF is suitably maintained to cater for sudden in-flows from seasonal storm events. Any overflow from the reclaim water tank will return to the TMF.

In the unlikely event that the TMF becomes filled with storm water an emergency overflow spill way will be provided as part of its construction to direct any excess flows into the nearest local water course.

A seepage collection pond and water return pump will be located below the TMF dam wall and will return collected seepage from the dam wall directly back into the TMF.

Reclaimed TMF water will be transferred overland back to the concentrator via three pump stations. The final pumping station will deliver the reclaim water back into the settling pond and then to the Mine Water Clarifier. The flow split of water returning to the process site will be determined by the instantaneous demands of each tank's level control systems.

17.13 Mine Water Storage and Treatment

Over the life of the project mine water will be generated sequentially from four different locations. Namely;

a) The Open Pit Mine, nominally during project years 1 to 7.

b) The existing Exploration Adit, nominally during project years 1 to ~3.

c) The new Development Adit, nominally during project years 2 to 6.

d) The shaft column of the Underground Mine, nominally after project year 6.

The overall Skouries water reticulation is shown schematically in Figure 17-3 : Schematic of Skouries Return Water Pumping.

Suspended solids settling facilities are provided by the mining operations but the levels are still expected to be variable and nominally too high for direct discharge to the environment or for use as fresh make-up water for the concentrator.

With the possible exception of water from the existing Exploration Adit, all flows of mine water will be pumped into one of two settling and storage ponds. Nominally each pond will hold 24 hours capacity of the maximum predicted rate of mine water generation over the life of the mine. One pond will be on line while the second will be cleaned of solids and be ready for feed when the settled solids in the duty pond need emptying.

The mine water from the existing Exploration Adit is expected to remain low in suspended solids for the majority of the time. Mine water from this source will be pumped into one of two existing solids settling lagoons located at the mouth of the adit. Normally one will be maintained on line while the other is being cleaned out of settled solids. Overflow water will flow into a collection chamber and depending on its turbidity and on demands from the concentrator this water will either be released directly into one of the local water courses, or pumped into the central mine water settling and storage pond.

Flows exiting the central mine water settling pond will be directed into a central discharge chamber. Facilities will be provided to release either part or all of it by gravity into one of the local water courses depending on its turbidity and on demands from the concentrator.

In the more likely event that its suspended solids content is still too high to meet the environmental release standards it will be pumped into the feed box of a reactor clarifier where it will be dosed with flocculant solution to aid the recovery of any remaining suspended slimes particles. Clarifier underflow solids will be directed to the Concentrator's paste thickeners once sufficient solids density is developed.

Overflow water will be collected in a holding tank adjacent to the clarifier and transferred to the plant site where it will normally be used as level controlled make-up for the Fresh Water Tank. In the event that the flow of reclaim water from the TMF is insufficient for the plant's needs or has been temporarily interrupted, clarified mine water will also be added automatically, via back-up level controls, into the Process Water Tank.

Any clarified mine water surplus to the concentrator's requirements will be discharged by gravity into one of the local water courses.

Spillage arising from the bunded Mine Water Clarifier area will be pumped back into the clarifier's feed box.

17.14 Process and Fresh Water Management and Storage Systems

The Concentrator is served by the following water supplies:

a) Process Water - This will be held in a central storage tank with a live storage capacity of 6,100 m^3 (nominally equivalent to 3 hours of plant requirements at normal operating capacity). Process water will be returned for use to the concentrator's process water network. This will supply most of the plant's dilution and make-up waters. The process water tank will be supplied with feeds comprising:

- Tailings thickener overflows. This will be the primary source of fresh make-up water delivered into the tank.
- TMF reclaim water.
- Clarified mine water. This will act as a secondary or back-up source of make-up into the tank.
- Fresh borehole water. This will act as a tertiary or emergency only back-up source of make-up water into the tank. It will only be called on in the unlikely event that both the reclaim and clarified mine water flows are insufficient to meet the concentrators requirements.

The process water tank will be furnished with a sloping concrete base plus a central sump with a draw off pipe. With the help of the pipe, settled slimes can (if a need ever dictates) be drawn off via a suction road tanker or a portable pumping system. This would only ever be done as a batch cleaning exercise. The provision of this slimes draw off facility should minimize any requirements to carry out full tank drain and clean outs which would necessitate the shutdown of the concentrator.

b) **Fresh (Utility) and Fire Fighting Water** – There are two separate tanks each with a volume of 850 m^3. Utility water will service the following sets of duty and standby pumps all of which require pressurized and solids free fresh water supplies to meet their various process duties:

- Fresh water pumps, 2 running, 1 standby - Primarily supplying reagent preparation demands plus filter cloth washing and dust suppression spray duties.
- Gravity circuit fresh water pumps, 1 running, 1 standby – Providing the fluidization and back flushing water for all of the plant's centrifugal gravity concentrators.
- Gland water pumps, 1 running, 1 standby - Providing all process pump gland water requirements.

Make up to the fresh water tank will be supplied from borehole water.

17.15 Plant Infrastructure and Utilities

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

The total power draw of the plant has been estimated to be 45 MVA and for the Mine estimated at 20 MVA, giving a total of 65 MVA. Therefore using a Diversity Factor of 0.75 the Greek power authority, PPC, will be required to supply 50 MVA of power to the site.

A single 60 kVA UPS system providing 30 minute back-up will be installed in the main control room for control system supplies, essential instrumentation and monitoring purposes. This will be supplied from the essential services LV switchboard.

An emergency diesel generator rated at 700 kVA will be also connected to the essential services LV switchboard to supply critical drives and equipment in the event of power failure. This includes the emergency lighting.

An additional 700 kVA emergency diesel generator will be provided at the Paste Thickeners area for paste recycling in the case of power failure.

The process plant and site infrastructure have been located on a site that provides the best balance between geotechnical constraints, (although more geotechnical work is required), 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 the major capital equipment units. The most important characteristic of the site is the relatively flat terrain at an elevation and proximity to the shaft that does not give rise to any ore transport issues.

A computer generated 3D view of the plant and associated infrastructure is shown in Figure 17-4.

17.16 Conclusions – Process Plant and Associated Infrastructure

The Project process plant design has been based on extensive metallurgical testwork. The size and type of equipment that has been selected is well proven in the industry and presents minimal risk.

The plant will process the copper/gold ore grading 0.53% copper and 0.81 g/t gold and achieve average recoveries of 90% and 84% respectively.

The project area is served by good infrastructure for power and transport including HG's port for the export of concentrates. There is a pool of skilled labour available.

The Project has progressed to detailed engineering, major long lead process items including the SAG and Ball Mills have been procured and activities to obtain project finance are at an advanced stage.

Figure 17-1 : Schematic of the Skouries Concentrator Flow Sheet

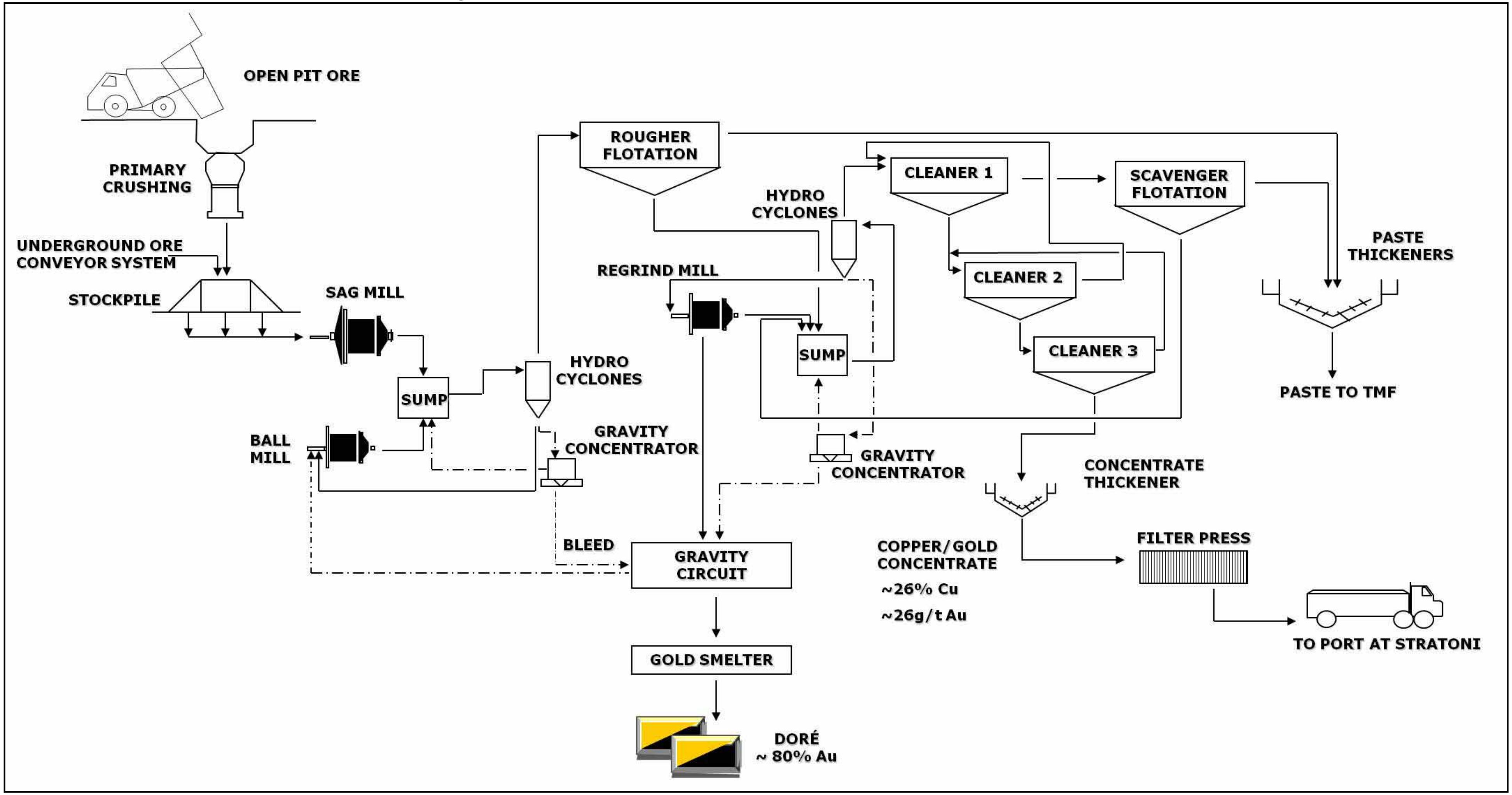


Figure 17-2 : Plot Plan of the Skouries Process Plant.

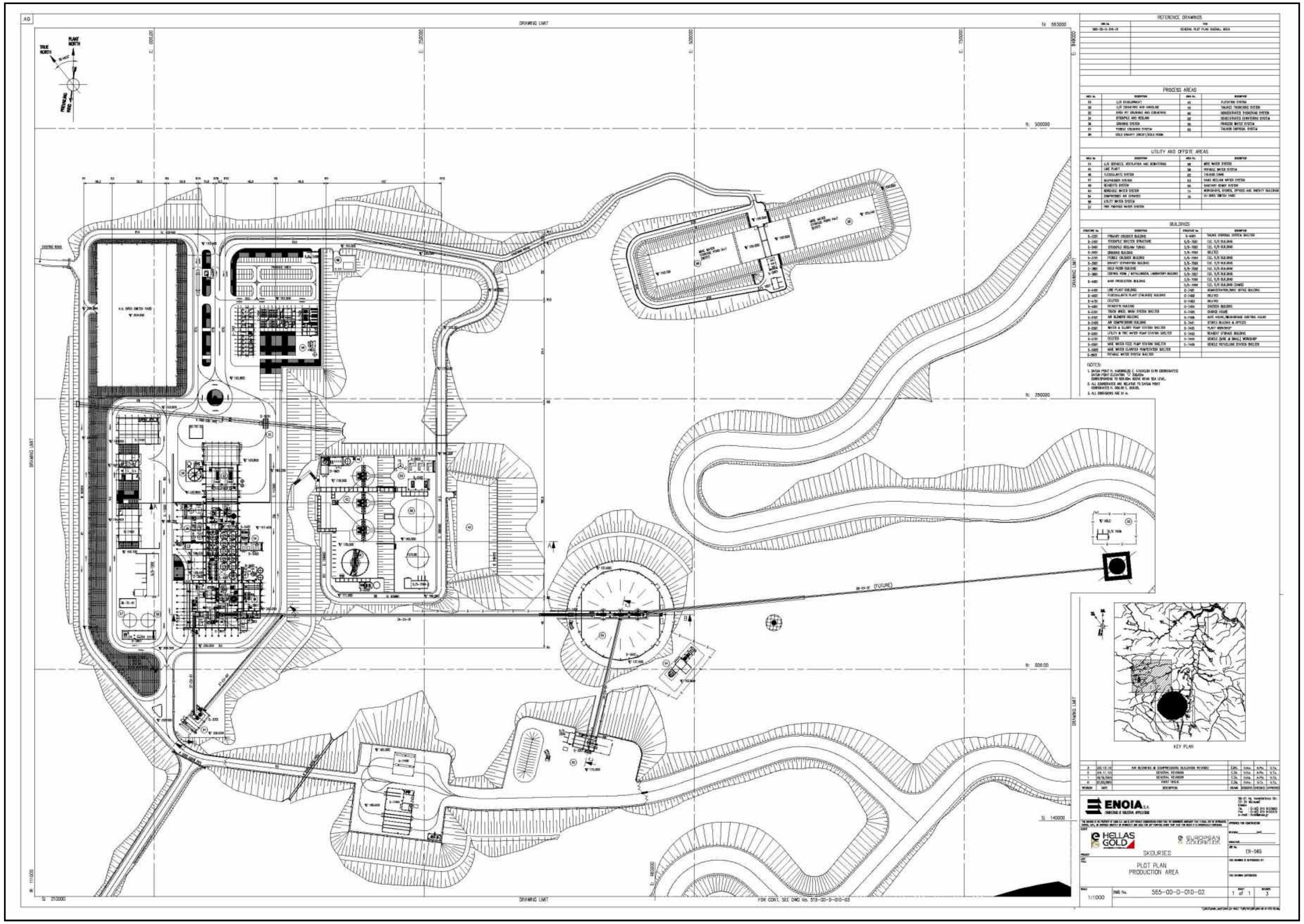


Figure 17-3 : Schematic of Skouries Return Water Pumping

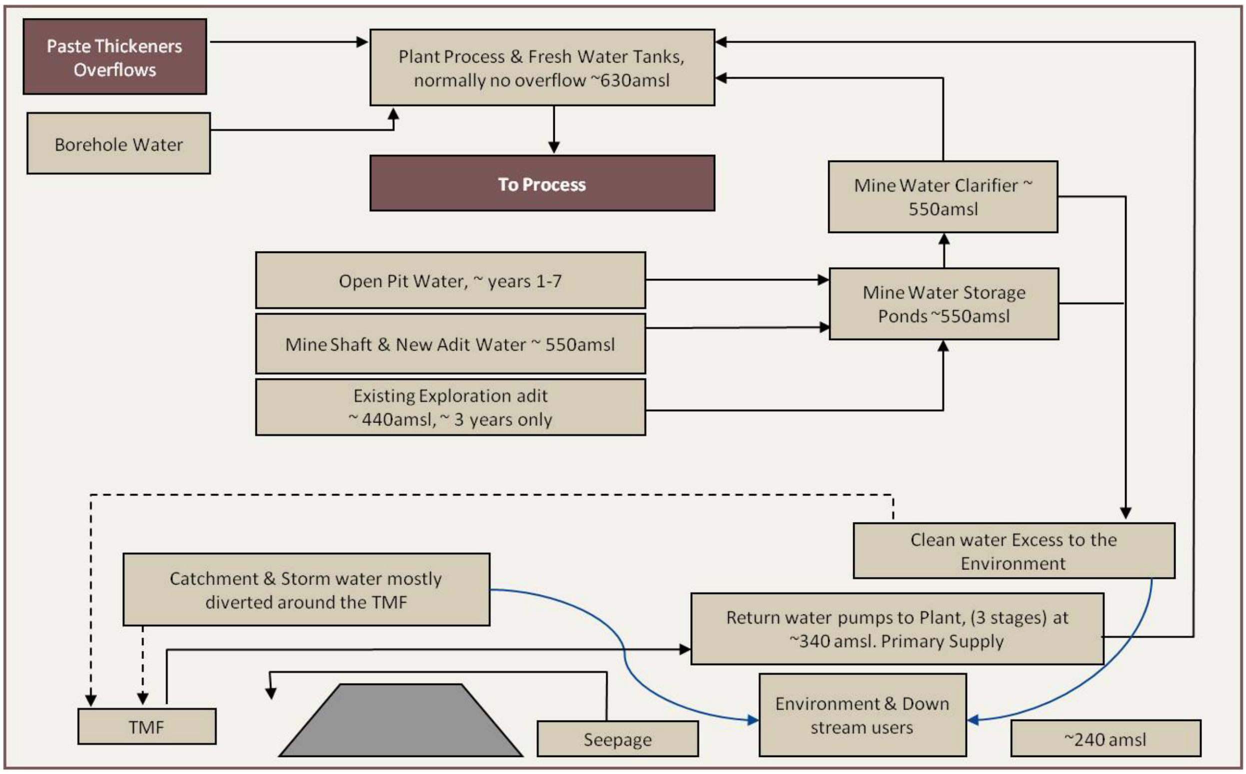


Figure 17-4 : Computer generated 3D view of the Skouries Concentrator and associated buildings.

18 PROJECT INFRASTRUCTURE

18.1 Waste Management

The principal waste streams generated from the mining and mineral processing operations are the overburden waste from the open pit, the underground development in waste (primarily the shaft, ramp and most of the ring drives) and the tailings from the recovery process. Effective strategies for management of this waste are summarised below.

18.1.1 Tailings Management Facility

The TMF has been designed by Omikron Kappa Consulting Limited (Omikron).

Two tailings dams are planned to be constructed in Karatza Lakkos and Lotsaniko stream valleys and will facilitate the storage of the tailings produced by the processing of Skouries open pit mining ore, while the waste produced by the open pit mining should be used for their construction.

The general design criteria of the tailings dam are:

- The two tailings dams will be designed in such way as to keep the dam footprints outside the Karolakkas stream delineation.
- The two tailings dams will be designed for the deposition of the waste and tailings resulting from the open pit mining.
- The waste and tailings mass balance should conform to the Pre-Design of Skouries Open Pit Mine.
- The tailings should be deposited in the form of paste with reduced water content. The paste thickened tailings is considered to have solids in discharge 72% by weight.
- Omikron states that the tailings do not contain dangerous substances and they do not have the potential to generate acid. From a geochemical point of view, the tailings are considered inert. The permeability of the thickened tailings is very low and therefore possible downwards seepage to the bedrock is limited. The sealing of the deposition area with special liners is not required, neither is the application of impermeable layers in the first meters of deposition.
- For environmental protection reasons, as well as, proper water management and reuse of the surface run-off waters accumulated in the surface of the deposition area for the needs of the process plant, the design provides for the water management within the deposition area and for the water recirculation to the process plant.

The produced tailings is envisaged to comprise 98.03% of the mined ore. By considering the above, the annual waste and tailings production by the open pit mining development for eight (8) years in total, is presented in the following table.

Table 18-1 : Total waste and tailings quantities from Skouries open pit mine

	Material volume (m^3)	Material mass (t)
Waste	12,832,742	35,001,413
Ore	17,428,497	47,134,094
Tailings (98.03% of mined ore)	30,803,702	46,205,553

Volume calculated with density of 1.5 t/m^3

The waste material used for the construction of the two dams is 13.34 Mm^3 while the minimum required storage capacity of the deposition areas should be 43.73 Mm^3. According to the pre-design arrangement of the TMF, the two dams are constructed with a final crest at +415 masl, while the final deposition surface of the tailings is at +410 masl. The final deposition surface of the tailings has gradient 6° to the horizontal, while in the vicinity of the dams' wall the tailings will be deposited almost horizontal to allow for flood waters storage before the operation of the spillway channels. There is an excess storage capacity of approximately 215,000 m^3 of tailings in relation to the tailings production requirements.

The access to the TMF will be achieved through the main access road designed for that purpose. The worksite will be established in the vicinity of the dams' downstream slope toe. The crusher for the mine waste from the open pit will be installed near the open pit in order to allow flexible transport of the aggregate by normal road trucks and minimise road sizes. The transportation of the tailings is designed through a network of pipes and pumps. The distribution to Karatza Lakkos tailings facility is proposed to be implemented from two locations at the northwest and southwest of the tailings basin. The distribution to Lotsaniko tailings facility is proposed to be implemented from three locations at the northern section of the tailings basin. The removed top soil from the deposition areas is proposed to be stored in the three designated dump areas.

The final surface of the tailings will be shaped during the latter stages of the filling process in each basin, with the application of proper layers with vertical thickness of 0.3 m. Thereinafter; a vegetation cover will be applied in the final surface of the facilities in order to be gradually attributed to the surrounding environment. In addition, a top soil cover will be applied in the downstream slope of the dams.

18.2 Road Network

The road network has been designed by Omikron which include.

- ***Main Access Road*****:** Asphalt road, 7.5 m wide and about 7.2 km long, used as access to the area of the facilities from the County Road Network. The alignment is based on an existing forest earth road.
- ***Road 1*****:** Gravel road, 25.0 m wide and 0.45 km long, used as connection of the Process Plant with the exit of the open pit ramp, through Road 2.
- ***Road 2*****:** Gravel road, 25.0 m wide and 1.5 km long, used as connection of the Main Access Road with the exit of the open pit ramp.
- ***Road 3*****:** Gravel road, 12.0 m wide and 9.2 km long, used as connection of the open pit through Road 2 with the toe of the dams.

In addition to the above, two more roads were considered as alternative solutions, for the connection of the open pit to the TMF:

- ***Road 4*****:** Gravel road, 12.0 m wide and 8.3 km long, used as connection of the open pit through Road 2 with the toe of the dams (Alternative solution I).

- ***Road 5*****:** Gravel road, 12.0 m wide and 1.3 km long, used as connection of Road 3 with the crest of the dams (Alternative solution II).

The whole project was designed considering the general balance of the earthworks (cuts and fills to be approximately the same if possible).

18.2.1.1 Main Access Road

The Main Access Road will connect the process plant and mining area with the National road network. The design considers use from regular passenger vehicles to regular trucks. It is also mentioned that this road will be used for the transportation of the main Process Plant components and the required equipment for the open pit mining.

The horizontal alignment and longitudinal section of the Main Access Road is based on an existing forest earth road which is upgraded to asphalt road with better overall horizontal characteristics. The design speed is 40 km/h and the total width for traffic is 7.5 m (bidirectional with 3.75 m of lane width). A concrete triangular drainage ditch of 1.40 m width for cut sections is also considered for the road drainage and a hard shoulder 1.25 m wide for fill sections.

The main design characteristics of the Main Access Road are:

- Minimum turn radius 40 m on the axis.

- Maximum longitudinal slope used 10.0% at the beginning of the road.

- Super elevation of 2.5% used in the straights and maximum of 5% in the turns.

- General slope gradient for open cut sections 2:1 and general slope gradient for fill sections 2:3.

- Triangular concrete drainage ditch used to collect the surface water of the road.

- Gravel hard shoulders of 1.25 m at the embankments.

- Finally the reconstruction of this road requires mostly earthwork operations and the construction of approximately 10 drainage culverts.

18.2.1.2 Internal Road Network

The internal road network in the wider area of the open pit and the associated facilities consists of Road 1 to Road 3 and the alternative solutions elaborated. These roads are designed for use mainly from heavy-duty mining equipment with reduced speed of up to 40 km/h. Roads 1 and 2 have 25 m width for traffic, are bi-directional and with gravel pavement. An earth triangular drainage ditch is foreseen in the open cut sections and a hard shoulder 1.25 m wide in the fill sections.

The main characteristics of the roads of the internal network are:

- Minimum turn radius 40 m on the axis.

- Maximum longitudinal slope used 10.0% in limited sections and basic longitudinal slope 6.5%.

- Super elevation of 2.5% used in the straights and maximum of 5% in the turns.

- Triangular earth drainage ditches used to collect the surface water of the pavement.
- Gravel hard shoulders of 1.25 m at the embankments.
- Finally the construction of these roads requires a lot of earthwork operations as well as, retaining walls in specific sections as shown in the relevant drawings.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

The prices used for the Mineral Reserve Estimate and subsequent economic analysis are based on a detailed review of long term rates applied by peer group review and a review of current rates used by mining investment analysts and bank finance terms. No hedging or forward selling has been used in this Technical Report and EGL do not intend to hedge gold production. In the opinion of the authors of this Technical Report the rates applied meet with industry norms and are applicable to the Project. Anticipated contracts and terms for sale of concentrates and gold doré are given below.

19.2 Contracts

This Technical Report assumes owner operation of the mine and plant at Skouries, though contract mining has not been ruled out at this stage. Construction of the plant and infrastructure is anticipated to be an EPCM style contract and at the time of writing the bidding document technical content has been finalised ready for issue and negotiation of commercial terms.

During operation the following contracts will be likely with local contactors:

- Transport of waste rock from the open pit to waste facilities and the TMF embankments.
- Transport of concentrate from the plant to Stratoni port or to Thessaloniki for onward transport by rail or ship.

No off take agreements have been signed by EGL and HG with potential concentrate off takers at the time of preparation of this Technical Report. However, EGL has advised the authors of this report that it is in early-stage negotiations with several off-takers in the form of copper smelters in Europe and globally. Analyses of the Skouries concentrate produced during the TVX 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. Summary terms are given below based on initial talks with major off-takers:

19.2.1 Copper terms

- Treatment Costs US$56/t of concentrate.
- Refining Costs US$0.056/lb Cu.
- No price participation.

These terms were used in the calculation of the reserve cut-off NSR's. In the economic analysis described in Section 22 of this report more conservative figures of US$70 for the treatment costs and US$0.07/lb for the refining cost were applied.

19.2.2 Gold terms

- Refining charge US$5/oz Au.
- Gold payable in copper concentrate 97%.
- Gold deduction 1 g/t Au.
- Gold payable in doré (from gravity circuit) 99.9%.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The EIS for the Skouries Mine is covered under the Kassandra Mines Mineral Deposits Project which involves an area of 26,400 ha, in north-eastern Chalkidiki (Macedonia Region). The Kassandra Mines Mineral Deposit Project will be implemented by HG, which obtained the mining rights by a contract included in the National Law No. 3220/2004.

The proposed Kassandra Mines Mineral Deposit Project includes the following sub-projects pertaining to Skouries:

- Mining Facilities of Skouries, new beneficiation plant and TMF.
- Port facilities in Stratoni.

''ENVECO S.A., Environmental Protection, Management and Economy S.A.'', under HG's management, has authored the full EIS.

20.1 Legislative framework for the preparation of the Environmental Impact Study

The full EIS was prepared principally by the application of:

- Law 1650/86 - 'The Protection of the Environment from Projects and Activities', as amended by Law 3010/2002 . This incorporates the EU directive on Environmental Impact Assessment (85/337/EEC as amended).
- Law JMD107017/06 which is the Greek implementation of the SEA Directive 2001/42/EC.

The EIS was submitted in July 2010 and has recently been approved. The EIS covers all environmental issues for the Project.

20.2 Landscape

Potential Impacts

The EIS considers the potential impact on earth morphology and landscape of the area could relate to:

- the Skouries open pit,
- the tailings dams,
- the construction of the process mill and the metallurgical plants,
- the necessary infrastructure for the project's operation (road construction for goods' transportation, etc)

20.3 Non Biotic Characteristics

20.3.1 Climatic and Meteorological Characteristics

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 of 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.

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

20.3.2 Morphology

Mount Holomontas Is the name given to the general Skouries study area. Despite long-lasting human activity in the area, the landscape does not show signs of intense deterioration. 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.

20.3.3 Soil

Physical properties

Soils in the study area are deep (>30 cm) with weak man-made influences and no signs of erosion. The bulk density of the soil is low, showing a slight increase as the depth increases. Low bulk density will be an important factor for successful management of soil extracted for the project and its use to rehabilitate the area, develop vegetation and lower erosion risks. The majority of the soils in the study area have a loamy-sandy texture.

Chemical properties - Soil quality

Sampling and analysis of soils provided the following conclusions for the Project:

As, Cd, Ni, Sb and Zn seem to exceed moderate and high risk limits. Concentrations of Cd and As in the whole area are out of limits. Concentrations of the remaining metals are within limits in some areas, but it is estimated that they exceed “no risk limits” for the whole area. Bearing in mind that 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.

20.4 Natural environment

The natural environment of Chalkidiki district shows a significant diversity, mainly attributed to its complex geomorphology. Two baseline studies have been performed relating to Skouries, one in 1998 and one in 2010. The combination of these studies clarified the ecological characteristics and the baseline conditions of the wider area.

20.4.1 Ecologic Characteristics - Main Ecosystems

Forest And Bush Ecosystems

The forest ecosystems cover almost entirely the Project area, showing very good tree growth, vegetation diversity and high density. 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.

Agro-Ecosystems

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.

Natural Rivers and Riverside Ecosystems

Significant rivers are:

- Asprolakkas, belonging to Asprolakkas basin which encompasses the whole of Skouries study area and part of Stratoni study area.

- Kokinolakkos.

Streams belonging to smaller hydrologic basins include Megali Panagia and Kerasia. All these rivers and streams, azonic, high and dense vegetation is present.

20.4.2 Flora and Fauna

From the extensive baseline studies carried out over the project area the species of flora and fauna are well known. Impacts of the project are judged to be temporary and not significant because species can migrate to similar habitats surrounding the area of activity during the life of mine and have a high potential to re-occupy the area after rehabilitation. The Company is developing a biodiversity management plan in order to monitor and assist in these natural processes.

20.4.3 Historical and cultural characteristics

During the Preliminary EIS, inspectorates of the Ministry of Culture performed archaeological investigations at all three Kassandra Mines Mineral Deposit locations including Skouries. Following site investigation of the Skouries area no archaeological issues were shown that could be impacted by the project

20.4.4 Economic and Social Environment

Demographics

During the period 1971-1991, the population of the study area was constantly increasing. In the period 1991-2001 there was a slight reduction (0.2%), partly due to reduction in mining activities between 1992 and 1996. The population reduction is most evident in the mountainous settlements. The lack of big cities, the area’s proximity to Thessaloniki and its geomorphology, are factors contributing to the area’s slow 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 Prefecture. Unemployment reaches 12.8% whereas the Prefecture’s unemployment is 10.8%. The majority of employees of the Secondary Production Sector are occupied in mining activities.

Social Requirements

There are no social obligations attached specifically to the Project at the time of issue of this report. However EGL 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.

20.4.5 Infrastructure

Water supply

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.

Sewage disposal

The majority of boroughs lack wastewater management plants and a substantial quantity of wastewater ends up in the rivers and streams of the study area.

Solid waste management

In Chalkidiki Prefecture there is a landfill in operation (in Kassandra Municipality). No solid waste treatment or recycling programme is currently in place in the study area.

Social infrastructure

Chalkidiki's medical care is mainly served by Poligiros General Hospital and 5 Health Centres. One of them (Health Centre of Palaeohori) is situated within the study area. Sports infrastructure is reasonable and Cultural associations are only present on a local level.

20.4.6 Aquatic environment

HG runs a monitoring program of surface, ground and marine water. Additional information comes from environmental baseline studies (Enveco S.A. 2005) and additional sampling and measurements performed for the purposes of the full EIS.

20.4.7 Air

Air quality

In order to assess air quality in relation to heavy metals concentration, a network of three samplers-analyzers was installed to establish base line levels.

20.4.8 Noise and vibration

From measurements performed in Skouries and given the nature of the area (almost entirely covered by forests), it is concluded that the acoustic environment is not impacted.

20.4.9 Man-made pressure on the environment

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 5 uncontrolled solid waste disposal sites but no regulated landfill site which could also impact water quality.

20.5 Public Consultation and Disclosure

20.5.1 Stakeholder Engagement Plan

The Stakeholder Engagement Plan ("SEP") is being developed by HG and the management of EGL 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.

20.5.2 Consultation meetings

Following the submission of the EIS to the Ministry of Environment, Energy and Climate Change, the Ministry forwarded it to the Prefectural Council after a period of consideration. This allowed the public consultation process to start. Two consultation meetings have taken place to date.

20.6 Potential Impacts - Hazards - Risks and Mitigation

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:

Table 20-1 : Phases of Project for Considering Impacts

Cash-flow για 33χρόνια

Project Phases
Development phase (years 0-4)
Operational phase A (years 4-7)
Operational phase B (years 7-13)
Operational phase C (years 13-30)
Rehabilitation phase (years 30-33)

On the basis of the impact analysis various mitigation plans have been formulated and continue to be developed with all relevant stakeholders. Now that the EIS has been approved by the Greek authorities the plans can be further developed to include clauses in the approval.

The results for the study are as follows:

20.6.1 Morphological Impacts

The open pit
The impacts will be major but temporary, as after rehabilitation the open pit will have returned to its previous state.

The TMFs
Parts of the existing thalwegs of Karatza Lakkos and Lotsaniko streams will be converted to plains with a total surface area of 126.9 ha. The impacts will be major, permanent and irreversible.

The Beneficiation Plant
Morphological changes will be needed to create a 12.62 ha surface on an altitude of +620m. During rehabilitation, main and ancillary facilities will be removed and soil will be added in order to resemble the existing morphology. Therefore, the impacts are moderate, permanent and irreversible.

The Road Network
Total land take will be 79.1 ha. Big amounts of excavation will be needed, so the impacts will be major, permanent and irreversible.

Mitigation
The proposed measures to be taken are:
- Shaping TMF slopes in such a way to ensure that soil can be deposited and trees planted during rehabilitation
- Monitoring rehabilitation for 5 years to ensure success

20.6.2 Visual Impacts

Potential impacts are associated with the construction of:
- The plant
- The TMFs

Only the TMFs will be visually disturbing from just one location in the village of Stagira 5.6 kilometres to the north of the project area and only during the operational phase. Other visual impacts occur only at the Project site itself and are minor, temporary and fully reversible.

Mitigation
- Painting external surfaces of existing and new buildings in colours fitting the surrounding environment
- Use of appropriate lighting at night time to minimise the distance from which the plants will be visible
- Planting around the plant facilities to minimise visual impacts
- Monitoring rehabilitation for 5 years to ensure success

20.6.3 Climatic and Bioclimatic impacts

For main and ancillary facilities, a total of 249 ha will be deforested. This might cause a slight drop in humidity levels locally. Small changes will occur to the microclimate of the Project's footprint, without affecting the wider study area. Impacts are categorized as negligible, temporary and fully reversible.

20.6.4 Geological Impacts

Impacts on geology are the extraction of schist waste rock that surrounds the ore, drilling in the area of the main reserves and mechanical cutting of weathered formations of the amphibolite - biotite schist. The impacts will be minor, permanent and irreversible.

20.6.5 Soil Impacts

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

Mitigation

Main mitigation measures, which have already been integrated into the Project, are:

- Using backfill to minimise ARD
- Separate storage and maintenance of natural soil
- Minimising waste of any type
- Maximising recycling and reuse
- Minimising land take
- Extensive environmental monitoring

20.6.6 Flora and Fauna Impacts

Some of the ecosystems that will be deforested here have a relatively higher ecological value. The impacts in the area are therefore locally major, permanent and irreversible. However, these flora species are also found in other parts of the wider area.

During the development phase and the operational phase, noise levels will be significant. Dust emissions are below limits. Wild mammals will be disturbed and will move away, as will birds that nest in dense vegetation. Surface and groundwater will not be significantly impacted; water fauna species will not be disturbed from a water quality point of view. The impacts here will be major, permanent and partially reversible as rehabilitation will allow these fauna to re-populate the area.

20.6.7 Land Use Impacts

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.

20.6.8 Historical and Cultural Impacts

No historical or cultural monuments will be occupied by the Project.

20.6.9 Social and Economic Environment

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.

20.6.10 Technical Infrastructure

For all phases, the impacts will be negligible, temporary and fully reversible.

20.6.11 Aquatic Environmental Impacts

Surface water

Impacts to surface water are associated to rainwater falling on the occupied area; this water will be stored and used as industrial water; it will not be contributing to stream flow. The TMF's 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, partially temporary and partially permanent; it can, however, be subject to remediation measures.

After the rehabilitation phase surface water will again contribute as it currently does downstream.

Groundwater
During the development and operational phases, groundwater will be pumped to lower the water table for the mining needs (approx. 480 m3/h). This water will be used to supplement the water lost in the form of moisture in the tailings. The advantage of this is that water will not come in contact with the ore body; it will therefore remain unmineralised and there will be no treatment needs before disposal.

The construction of the TMF's will not affect groundwater flow. 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.

Coastal water
The impacts on coastal water will be negligible during the development, operational and rehabilitation phases.

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

20.6.12 Atmospheric Environment

The simulation model for air contaminant dispersion was designed on a grid that was 3km high and covering a surface area of 22 km x 23 km. This grid covers the facilities of Skouries as well as EGL's other operations and proposed operations in the region.

In the immediate study area, there will be fluctuating emissions of atmospheric pollutants, depending on the stage of implementation of each project. However, the model shows that the highest emissions will be recorded when the open pit and the two TMF's in Skouries will be in operation.

The study ''Estimation of particulate matter and pollutants dispersion'' gave the following results:

- CO levels are extremely low and reach or exceed slightly 1 mg/m^3 in Skouries due to machinery use.
- Note that there is no limit for NOx except for NO^2 which is a fraction of NOx ranging from 10% to 90% depending on use and location. In any case, the concentrations resulted from the calculations are extremely low.
- Concentrations of particulates estimated by the model do not exceed 20 μg/m^3 (PM10) and 10 μg/m^3 (PM2.5).
- SO^2 concentrations were not significant, reaching a few μg/m^3 mainly around the sulphuric acid plant. EU limits for SO^2 is 200 μg/m^3.
- Volatile Organic Compounds reach 40 μg/m^3 in Skouries, with no limits set by European legislation.

Cumulative impacts
No limits are exceeded for particulate and air pollutants. No problem arising from heavy metals concentrations is predicted.

Conclusively, the impacts from the Project will be moderate, temporary and partially reversible.

Mitigation
The following mitigation measures are proposed:

- Application of dust collection systems for dry drills
- Blasting in Skouries should take meteorological conditions into account and be avoided under adverse conditions
- Sheeted trucks will be used for ore transportation and roads will be wetted. All conveyor belts will be covered
- Installing dust filters at crushing stations (both underground and overground), milling stations and cement silos
- Hydraulic transport of backfill material without intermediate storage
- Storage of concentrates in areas surrounded by 3 walls and roofed
- Machinery servicing on schedule to avoid unnecessary emissions
- Maintenance of fuel tanks to minimise losses in the form of vapour

20.6.13 Acoustic Environmental Impacts

As far as noise levels in Skouries open pit are concerned, the nuisance will be there, but it will be of limited duration and is not expected to significantly impact the acoustic environment of the area.

From a vibrations point of view, in all cases, the maximum ground vibration velocities will be within limits.

20.7 Closure

20.7.1 Aims of Closure

General principles of closure and rehabilitation:

- Minimise the negative consequences of closure.
- Maximise the positive benefits of closure.
- Minimise the likelihood that closure goals are not met.
- Maximise the likelihood that opportunities for lasting benefits are captured.
- Integrating closure into the planning, engineering, construction and operational phases.
- Rehabilitation and stabilization of impacted areas as soon as possible.
- Looking out for possibilities to reuse/recycle resources during operation and closure.
- Making sure that rehabilitation is in line with goals for safety, health and environment.

20.7.2 Closure Plan - Underground

Voids created from the extraction process will be hydraulically filled with a mixture of tailings and cement. When exploitation of the Mineral Reserves is over, the only pending arrangements for full closure are the removal of mining (mobile mining equipment, fans, etc.) and mechanical (industrial water network, electrical installations, etc.) equipment from the access works, backfilling access works where appropriate and the rehabilitation of the area around the entrances of the main access tunnels.

20.7.3 Skouries Open Pit

For the early mine years the open pit at Skouries will be extracting ore and waste rock, with tailings going to the TMF at Karatza Lakkos and Lotsaniko. For the following 20 years during underground operations, the pit will operate as a TMF for the excess tailings (cemented to a geotechnically stable state as necessary) that are not used for underground backfill.

20.7.4 Environmental Rehabilitation

Rehabilitation works will cover the following areas; soil management, choosing flora species, artificial planting technique - planting tests, establishing a nursery and a seed storage facility, mixing of planted species, establishing planting season, additional planting, maintenance of new plants.

20.7.5 Environmental Costs and Guarantees

EGL will ensure full compliance with EU legislation embodied in the Mine Waste Directive 2006/21/EC, in particular Article 14, which requires “a financial guarantee or equivalent so that all obligations under the issued permit are discharged, and there are funds readily available at any given time for the rehabilitation of the land affected by the waste facility.”

EGL’s support will in the first instance take the form of a full and unconditional corporate guarantee of HG’s reclamation and closure costs liabilities from EGL.

Further, EGL’s corporate guarantee will be fully supported by a surety bond provided by an internationally recognised, AAA-rated insurance company. A surety bond is a guarantee by the insurer to fund the reclamation and closure liabilities of HG’s mining operations if EGL is unable to fund these obligations. The Greek State would be a named party to the surety bond and this guarantee will be in the amount of €16.2 million to cover the Project. This is comprised of

- €475,180 per year in years 4 to 9 inclusive relates to the TMF closures and was estimated as part of the Enoia capital estimate in their Skouries Project Basic Design Package.
- €8M for retrenchment over years 25, 26, and 27 in slices of €2M, €2M and €4M respectively as production winds down
- €5M in year 27 for decommissioning.

Environmental monitoring is included in the operating costs based on costs incurred at EGL’s nearby operations factored to reflect the scale of the Skouries project.

21 CAPITAL AND OPERATING COST ESTIMATES

Operating costs have been estimated by HG and verified by the authors of this Technical Report. Estimations are based on local quotes and from costs experienced by HG at its current operation and adjusted for the throughput and proposed shift system.

Capital equipment costs for the plant were derived from detailed quotes from equipment suppliers acquired during basic engineering studies by Enoia and Outotec.

Capital costs for mining equipment are estimated by Scott Wilson from experience with recent projects with a contingency of 15%. The capital development costs have been estimated by HG from first principals calculations using costs derived from its current operations and local quotes and have been benchmarked against a comparable European operation.

The overall accuracy of the cost estimates for the underground mine are considered to be +/- 30%, which is an appropriate level for a pre-feasibility study.

21.1 Capital Cost

Table 21-1 : Total Project Capital Expenditure (Euros)

Activity	Pre-Production (‘000 €)	Production (‘000 €)
Pre-strip	9,799	0
Open Pit Mine	35,200	6,000
Underground Mine - SLOS	0	144,123
Enoia Capex Estimate 2011	149,626	0
Backfill distribution Plant	0	3,700
Roads	12,000	0
Tailings dam	3,897	21,103
Reclamation & Closure	0	15,851
Owners cost	10,388	0
Subtotal	**220,910**	**190,777**
Sustaining Capex		
Open Pit Mine	1,069	1,273
Underground Mine		35,794
Processing	327	6,876
Tailings dam	7	665
Roads	60	960
Subtotal	**1,463**	**45,569**
TOTAL CAPITAL COST	**222.373**	**236,346**

The plant and associated infrastructure capital estimates were developed by Enoia using established and accepted procedures.

21.2 Operating Cost

Skouries Mine Total Operating Cost The life of mine

Table 21-2 : Skouries Mine Key Operating Cost (Euros)

Activity	€/t	Unit
Open Pit	1.63	€/t moved
Underground Mining	12.49	€/t milled
Plant	3.31 to 4.45	€/t milled
Tailings	0.38	€/t milled
G&A	0.05	€/t milled

The variable plant operating cost estimates for the Open Pit phase of the project were developed using established methodology by Enoia and are dependent on the quantity of oxide present (only applicable in year one of the open pit) and the throughput. The same procedure was utilised for processing the ore from the open pit but have been adjusted by EGL for the reduced tonnage when the operation moves underground.

22 ECONOMIC ANALYSIS

A pre-tax cash flow model was generated for the Project using the Resources and Reserve estimates detailed in this Technical Report together with economic estimates of the Project capital expenditure requirements and annual operating costs for the life of mine production schedules as given above in Section 21.

Base Case

The base case cash flow is based on the following parameters:

Physical inputs based on the mine and plant design detailed in this Technical Report:

- 345 production days per year;
- 23,200 t of ore per day mining from open pits (8.0 Mtpa);
- 13,000 t of ore per day mining from underground (4.5 to 4.8 Mtpa);
- Average process recovery of 84% of gold and 90% of copper;
- NSR of 97% for gold and 87% for Copper;
- Metal prices for Mineral Reserves US$1,000/oz Gold and US$2.50/lb Copper;
- No Inferred Resources were used in the Economic Analysis.
- Mine life of 27 years based on the Mineral Reserve estimate.

Costs

A yearly cash flow model using a gold and copper price of US$1,000 per ounce of gold and US$2.50 per pound respectively, which includes selling costs for gold bullion and of copper concentrate and a discount rate of 5% was prepared in order to examine the Project economics and the NPV.

Under the terms of the existing mining licences, royalty is not payable for Skouries and taxes in Greece are currently 25%, reducing to 20% by 2014. Depreciation in Greece is applicable at the rate of 15% per annum.

Table 22-1 below summarises the key operational parameters of the model in dollars.

Table 22-1 Key Cash flow Parameters (Dollars)

	Units	Life of Mine	Average
ROM production	kt	**138,362**	**5,125**
ROM production gold grade	g/t	**0.81**	**0.81**
ROM production copper grade	%	**0.53%**	**0.53%**
Au Contained in Concentrate	k oz	**2,925**	**108**
Cu Contained in Concentrate	kt	**646**	**24**
Operating Costs	US $	**2,763.5**	**102**
Capital Costs	US $	**615**	**23**
Net Revenue (1)	US $	**6,144.5**	**228**
Undiscounted Operating Income	US $	**3,381.0**	**125**
Discounted cashflow @5%	US $	**1,399.3**	**52**

Note: (1) Revenue after deducting payability, TC/RC's, transport, deductions and marketing costs.

Cash Flow Analysis

Using these parameters in a discounted cash flow the Project gives a pre tax internal rate of return of greater than 35% which meets EGL's criteria for a viable project. Corporate tax has been applied to the cashflow model and EGL advise that no royalties apply under HG's contract with the State. Payback is less than two years from the commencement of open pit production.

Sensitivity Analysis

A sensitivity analysis was carried out to model potential fluctuations of key input parameters from the base case cash flow model.

The following parameters were evaluated over a range of a 20% increase to a 20% reduction to observe the impact on the Project's NPV:

- gold price;
- gold grade;
- copper price;
- copper grade;
- capital expenditure;
- operating expenditure;
- processing plant gold recovery was evaluated over a range of a 4% increase to a 4% decrease;
- processing plant copper recovery was evaluated over a range of a 4% increase to a 4% decrease.

The Project economics are most sensitive to copper grade, and operating cost. Processing Plant recoveries do not significantly affect economics due to fluctuations within the ranges evaluated.

23 ADJACENT PROPERTIES

Not applicable.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Manpower

Total manpower anticipated for the Skouries mine is outlined below in Table 24-1.

Table 24-1 : Skouries Manpower Schedule

Area	Total manpower
Open Pit	106
Underground	220
Process Plant	101
General and Administration	41

The Skouries Mine envisages that a number of operators from the open pit will transfer to the underground operations on the cessation of the open pit operations.

25 INTERPRETATION AND CONCLUSIONS

The authors of this Technical Report conclude that the level of data adequacy is considered sufficient for reporting Mineral Resources and Mineral Reserves, but further work is required particularly on the underground design to reach the level of a full and accurate revision of the 1998 Mineral Resource Model. Based upon the assumptions in this Technical Report and the work carried out by Scott Wilson and other contributors, the authors of this Technical Report are of the opinion that the Project can be developed into a viable mining operation.

Moderate differences in the Mineral Resource model are mostly due to sterilising over-extrapolated blocks and adding data for the ten drill holes. The new classification model has resulted in a significant decrease in the amount of Measured Mineral Resource material, however, the total Measured and Indicated Mineral Resource has not changed significantly.

25.1 Risks

The following are factors that may affect the validity of Mineral Resources and Mineral Reserves:

Foreign Country Risk

If there were any change in the economic, legal or political framework in Greece, or other circumstances arising, which materially reduce or suspend the EGL's operations, the results of the proposed Skouries operation and financial condition will be materially negatively affected.

Technical Risks

Mineral Resource and Mineral Reserve figures presented herein are based upon estimates made by EGL personnel and independent consultants. These estimates are imprecise and depend upon geological interpretation and statistical inferences drawn from drilling and sampling analyses, which may prove to be inaccurate, and may require adjustments or downward revisions based upon further development or exploration work. There can be no assurance that these estimated Mineral Reserves, Mineral Resources or other mineralisation figures will be accurate, or that this mineralisation could be mined or processed profitably.

Capital and Operating Cost Risks

EGL's forecasts of costs are based on a set of assumptions current as at the date of completion of this Technical Report. The realised operating and capital costs achieved on the Project may differ substantially from the forecasts owing to factors outside the control of EGL, including currency fluctuations, supply and demand factors for the equipment and supplies, global commodity prices, transport and logistics costs and competition for human resources. Though EGL incorporates a level of contingency in its assumptions, these may not be adequate depending on market conditions.

Mineral and Commodity Prices

The Project's profitability and long-term viability depend, in some part, upon the market price of gold and copper. The market price of gold and copper is volatile and is impacted by numerous factors beyond EGL's control, including: expectations with respect to the rate of inflation, the relative strength of the U.S. dollar and certain other currencies, interest rates, global or regional political or economic conditions, supply and demand for jewellery and industrial products containing metals, costs of substitutes, changes in global or regional investment or consumption patterns, and sales by central banks and other holders, speculators and producers of gold and copper in response to any of the above factors.

EGL's long term financial performance is dependent upon the market price of gold and other metals.

Currency Fluctuations

Gold and other metals are sold throughout the world principally in U.S. dollars. Further, the capital markets in which EGL expects to have access to for financing (debt and equity), are predominantly denominated in United States dollars. The Project's capital and operating costs are incurred principally in Euros. EGL does not currently use any derivative products to manage or mitigate any foreign exchange exposure. As a result, any significant and sustained appreciation of the Euro or other currencies against the U.S. dollar may materially increase EGL's costs and reduce revenues.

Financing Risks

Development of the Project by EGL will be dependent upon its ability to obtain financing through joint ventures, equity or debt financing or other means, and although EGL has been successful in the past in obtaining financing through the sale of equity securities and agreeing terms with banks, there can be no assurance that EGL will be able to obtain adequate financing in the future or that the terms of such financing will be favourable. Failure to cover capital expenditure or obtain additional financing could result in delay or indefinite postponement of development of the Project.

26 RECOMMENDATIONS

The authors recommend that the EGL and HG management team continues with its planned program of progressing the Project which is outlined below:

- Planned works once environmental permits have been approved include:
- Detailed geotechnical drilling of load bearing areas of the mill site,
- Geotechnical drilling of the embankment sites of the tailings facilities,
- Geotechnical drilling around the pit, shaft and portal sites,
- Drilling of Inferred Mineral Resources within the open pit area
- Continuing basic engineering on the underground,
- Issue of the contractor ITB, which has been prepared, for the plant and infrastructure construction.
- Develop the Project team

Geotechnical investigation is required to confirm the geomechanical design parameters adopted in the underground mine design.

In line with EU directives, and to ensure that they remain appropriate to the operations, EIS approvals in Greece are refreshed every five years to reflect prevailing conditions such as new Mineral Resources, metal prices, employment levels, etc. Accordingly, this presents an opportunity, at that time, for EGL to re-assess the extraction approach, so as to ensure that in the context of the then prevailing conditions, the best available techniques are applied for the remainder of the mine life.'

27 REFERENCES

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28 SIGNATURE PAGE

This report titled "Skouries Cu/Au Project, Greece NI 43-101 Technical Report " prepared for EGL and dated 14 July 2011, was prepared and signed by the following authors:

Dated at London	Patrick William Forward, BSc, FIMMM
14 July 2011	General Manager, Exploration
	European Goldfields (Services) Limited

Dated at London	Antony Francis, B.Sc, FIMMM
14 July 2011	Senior Metallurgist
	European Goldfields (Services) Limited

Dated at Chesterfield	David J.F. Smith, C. Eng.
14 July 2011	UK Head of Operations, Mining
	Scott Wilson Ltd.

29 CERTIFICATE OF QUALIFICATIONS

Patrick Forward

I, Patrick William Forward, BSc, FIMMM, as an author of this report entitled "Skouries Cu/Au Project, Greece, NI 43-101 Technical Report", prepared for European Goldfields Limited and dated 14 July 2011, do hereby certify that:

1. I am a graduate of the Imperial College of Science and Technology, London and hold a B.Sc. honours degree Mining Geology (1989).

2. I am presently employed as General Manager, Exploration of European Goldfields (Services) Limited of 11 Berkeley Street, Level 3, London, England W1J 8DS. My residential address is of 11 Kirkley Road, London, SW19 3AZ

3. I have been employed in my profession since graduation and with European Goldfields (Services) Limited since October 2004.

4. I have fifteen years' experience in all aspects of precious and base metal exploration.

5. I am a Fellow of Institute of Materials, Minerals and Mining (Membership 454049) and a Member of the Australasian Institute of Mining and Metallurgy (Membership 225134) and a "qualified person" for the purposes of Canadian National Instrument 43-101 (the "Instrument").

6. I am responsible for the contents of sections 1 to 12, 14, and 18 to 27 of the report (the "Report") dated 14 July 2011, prepared for European Goldfields Limited (the "Issuer") entitled "Skouries Cu/Au Project, Greece, NI 43-101 Technical Report".

7. The information contained in the Report was obtained from regular visits to the properties that are subject of the Report between November 2004 and February 2011, discussions and data reviews with Hellas Gold SA personnel and detailed discussions with other consultants to European Goldfields Limited. My most recent visit to site was 28th of February 2011 for a duration of five days.

8. There have been no limitations imposed upon my access to persons, information, data or documents that I consider relevant to the subject matter of this report.

9. As of the date hereof, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

10. I am not independent of the Issuer applying the tests set out in section 1.5 of the Instrument as I am an employee of European Goldfields (Services) Limited, an affiliated entity of the Issuer.

11. I have not had any prior involvement with the properties that are subject of the Report.

12. I have read the Instrument and Form 43-101 F1, and the Report has been prepared in compliance with the Instrument and Form 43-101 F1.

DATED at London, England, this 14th day of July, 2011.

PW [signature]

Mr P. W. Forward

Antony Francis

I, Antony Francis, B.Sc, FIMMM, as an author of this report entitled "Skouries Cu/Au Project, Greece, NI 43-101 Technical Report", prepared for European Goldfields Limited and dated 14 July 2011, do hereby certify that:

1. I am Senior Metallurgist at European Goldfields (Services) Limited of Level 3, 11 Berkeley Street, London, UK, W1J 8DS. My residential address is 8, Coulter Road, Kingsnorth, Ashford, Kent, TN23 3JQ, UK.

2. I graduated with a B.Sc. Hons, (Eng) Met degree from the Royal School of Mines, Imperial College of Science, Technology & Medicine, London University in 1971.

3. I am a Fellow of the Institute of Materials, Minerals and Mining.

4. I have practiced my profession for a total of 38 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that I am a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 13 and 17 of the technical report entitled "Skouries Cu/Au Project, Greece, NI 43-101 Technical Report" and dated 14 July 2011 (the "Technical Report") relating to the Skouries property. I have visited the Skouries property on regular occasions during 2005, 2006, 2007, 2008, 2009, 2010 and 2011 for an approximate total of 40 days with the most recent visit being from the 6th of July 2011 for a duration of three days.

7. I have had no prior involvement with the property that is the subject of the Technical Report. I am not aware of any limitations imposed upon my access to persons, information, data or documents that I consider relevant to the subject matter of the Technical Report.

8. To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am not independent of European Goldfields Limited and Hellas Gold SA pursuant to section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I own securities of European Goldfields Limited in the form of shares, and as such I have an indirect interest in the Skouries property.

Dated at London, England, this 14th day of July 2011.

A. D. Francis.

Antony Francis

David J.F. Smith

I, David J.F. Smith, MIMMM, C.Eng, as an author of this report entitled "Skouries Cu/Au Project, Greece, NI 43-101 Technical Report", prepared for European Goldfields Limited and dated 14 July 2011, do hereby certify that:

1. I am UK Director of Operations - Mining with Scott Wilson Ltd. of Royal Court, Basil Close, Chesterfield, Derbyshire, S41 7SL, United Kingdom. My residential address is 9 Main St, Ulley, Sheffield, S26 3YD, UK.

2. I am a graduate of University of Newcastle Upon Tyne in 1978 with a BSc (Mining Engineering).

3. I am a member of the Institute of Materials, Minerals and Mining.

4. I am registered as a Professional Engineer in the Country of United Kingdom (Reg. # 43860). I have worked as a mining engineer for a total of 30 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Review and report on numerous exploration and mining projects around the world.
   - Carried out mining engineering studies for mining projects in a number of countries.
   - Technical Manager with a major international mining contractor.
   - Mine operations engineer in South Africa, Ireland, Iran and UK.

5. I have read the definition of "qualified person" set out in National Instrument 43 101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43 101.

6. I visited the project site in October 2006 and the project offices in October 2010.

7. I am responsible for the preparation of Sections 15 and 16 of the Technical Report.

8. I am independent of the Issuer applying the test set out in Section 1.5 of National Instrument 43-101.

9. My prior involvement with the property that is the subject of the Technical Report was in 2006 and 2007 as part of the team that prepared the Bankable Feasibility Study in 2007.

10. I have read National Instrument 43-101, and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

11. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Chesterfield, England, this 14th day of July, 2011.

David J.F. Smith, C.Eng.