Eldorado Gold Corporation

Technical Report

Perama Hill Project
Thrace, Greece

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ABBREVIATIONS, ACRONYMS, AND UNITS OF MEASURE

ABBREVIATIONS AND ACRONYMS

Acid Rock Drainage	ARD
Acid-Base Accounting	ABA
Anglo American Research Laboratories	AARL
Australia Consolidated Minerals	ACM
Australian Laboratory Services	ALS
Ball Mill Work Index	BWI
Bulk Leachable Extractable Gold	BLEG
Bureau de Rechérche Géologiques et Miniérs	BRGM
Carbon In Leach	CIL
Cutoff Grade	COG
Eldorado Gold Corporation	Eldorado Gold
Environmental Terms of Reference	ETR
Front-End Loader	FEL
Golder Associates (UK)	Golder
Impact Work Index	IWi

Inmet Mining Corporation	Inmet
Joint Ministerial Decision	JMD
Life-of-Mine	LOM
Mining Titles	MT
Ministry of Development	MOD
Ministry of Environment	MOE
Net Acid Generation	NAG
Normandy LaSource	NLS
Normandy Mining Limited	Normandy
Preliminary EIA	PEIA
Quality Control/Quality Assurance	QC/QA
Radial Drilling Europe	Radial
Reverse Circulation	RC
Reverse Osmosis	RO
Rock Quality Designation	RQD
Rod Mill Work Index	RWi
Roscoe Postle Associates Inc.	RPA
Run-Of-Mine	ROM
Scott Wilson Ltd.	Scott Wilson
Specific Gravity	SG
Tailings Management Facility	TMF
Technical University of Athens	NTUA
Thracean Gold Mining	TGM
Toxicity Characteristic Leaching Procedure	TCLP

UNITS OF MEASURE

Annum (year)	a
Centimetre	cm
Cubic metre	m^3
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Degree	°
Degrees Celsius	°C
Gram	g
Grams per litre	g/L
Grams per millilitre	g/mL

Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Kilo (thousand)	k
Kilogram	kg
Kilograms per hour	kg/h
Kilometre	km
Kilometres per hour	km/h
Less than	<
Litre	L
Megawatt	MW
Metre	m
Metres above sea level	masl
Metric ton (tonne)	t
Micrometre (micron)	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million tonnes	Mt
Million tonnes per year (annum)	Mt/a
Minute (plane angle)	'
Minute (time)	min
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Percent	%
Second (plane angle)	"
Specific gravity	SG
Square centimetre	cm^2
Square kilometre	km^2
Square metre	m^2

Thousand tonnes	kt
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Year (annum)	a

SECTION 1 • SUMMARY

1.1 INTRODUCTION AND PROPERTY DESCRIPTION

Eldorado Gold Corporation (Eldorado or Eldorado Gold), an international gold mining company based in Vancouver, British Columbia, owns the Perama Hill Gold Project in Thrace, Northern Greece through its wholly owned Greek subsidiary Thracean Gold Mine ("TGM"). Eldorado has prepared this Technical Report of the Perama Hill Gold Project to support a first time disclosure of National Instrument 43-101 compliant mineral reserves.

Information and data for this report were obtained from Eldorado and engineering studies carried out by Aker Solutions E&C Limited (Aker). The work entailed review of pertinent geological, mining, process and metallurgical data in sufficient detail to support the preparation of this Technical Report. The Qualified Persons responsible for preparing this Technical Report are Richard Miller, P.Eng. and Stephen Juras, Ph.D., P.Geo., both employees of Eldorado and Peter Perkins, MIMMM, MSAIMM, an employee of Aker.

The Perama Hill Gold Project comprises two mining titles (MT) granted in December 1999 and covering an area of 1,897.5 ha. The project is located within a rural area 25 km west-northwest of the city of Alexandroupolis and 20 km south of Sapes in the eastern Thrace region of northern Greece. Present access to the project site is via farm tracks from the Perama village, situated at the end of a 4 km long asphalt road from the main Thessaloniki/Alexandroupolis Highway. Future access to the site will be via the Egnatia highway service road through a new road of 7 km.

Currently, the Project is in the process of obtaining the third license of five required to permit a mining project in Greece, the Environmental Permit Licence. The Preliminary Environmental Impact Assessment Study, (PEIA), a pre-cursor to the final Environmental Impact Assessment (EIA), was submitted in 2009. Evaluation of the PEIA is ongoing and expect later in 2010.

The topography in the immediate area consists of rolling hills ranging from 250 masl to 300 masl. Vegetation is sparse comprising small oak trees and Mediterranean thorn and scrub bushes. A stand of black pines occurs to the north of the project and is part of the Black Pine Forest. Climate of the wider project area is characterized as Mediterranean with dry and warm summers and mild winters. The mean annual rainfall is 557 mm.

1.2 GEOLOGY AND MINERALIZATION

The Perama Hill gold deposit is a stratabound, sediment hosted deposit of Eocene to Oligocene age, located at the eastern edge of the Maronia Graben at the contact point with a north-northeast trending eastern graben fault. The deposit is hosted by a felsic volcanic sandstone overlying a package of andesitic volcanic breccias. A conglomerate unit ranging up to several metres thick marks the transition from andesitic breccia to sandstone. The sedimentary units displays gentle westward to near horizontal average dips.

The tectonic events comprise two main phases. A first phase of extension and deposition of the volcano-sedimentary formation in the Maronia basin was followed by a second phase of transpression with sinistral movement and uplifting. The mineralization is believed to be contemporaneous with this second phase. The dominant structure is the 200°/-60°W East Graben Fault, juxtaposing the Cenozoic sequence described above with strongly deformed and drag folded non-mineralised Mesozoic meta-volcanics.

The deposit extends 750 m in a north-south direction and up to 300 m in an east-west direction and the thickness varies from 15 m to 20 m at the flanks and up to 125 m at the centre. The eastern margin of the deposit is marked by the East Graben Fault. Fine grained gold mineralization occurs in all three units though 80% of the gold is hosted by the sandstones. Mineralised sandstones display pervasive silicification and the development of cavities and voids filled with barite or kaolinite. Gold is associated with very fine-grained pyrite and telluride minerals.

The deposit has been deeply oxidised. In general, the oxidation limit roughly equates with the sandstone/volcanic breccia contact, allowing a thickness of more than 100 m of oxide mineralization in the middle of the deposit. This limit diminishes laterally to 15 m to 20 m on the deposit margins.

The Perama Hill deposit is described as an initial high sulphidation epithermal system overprinted by typical low sulphidation banded quartz-chalcedony, barite, pyrite stockwork veins and veinlets.

1.3 DRILLING, SAMPLING, AND ANALYSES

Drilling campaigns on the Perama Hill property occurred from 1996 to 1998 and in 2000. Holes drilled for resource estimation data totalled 9,093 m in 75 diamond drill holes and 9,116 m in 137 reverse circulation (RC) holes. Collars of all holes were surveyed at the end of each drilling phase. Only some of the deeper diamond holes were surveyed for downhole deviation, but as almost all the drill holes were less than 100 m long, deviation was not expected to be a significant factor.

The core and reverse circulation chip samples were transported to the TGM sample preparation facility located near the town of Sappes. At the preparation site, the core and chips were logged for geology and geotechnical information, and then photographed (core only). The reverse circulation samples were collected in 1 m intervals and split to get a 1 kg sample. The sample was further split in half to produce a duplicate. The core was marked in 1 m intervals and sample tickets were placed at the end of each interval. The core was then cut in half with a diamond rock saw. One half of this material was returned to the core box.

Samples were sent to the SGS-FILAB laboratory in France for analysis. At the SGS-FILAB laboratory, samples were pulverised with a ring mill to 90% less than 150 mesh. All samples in the database were fire assayed for gold using a 30 g sample size. Samples were also sent for 32 element ICP analysis.

Samples were controlled by a quality assurance – quality control program. The program involved insertion of blank and standard samples into the sample stream and the preparation of duplicate samples for analysis in a later sample batch. In addition, at the end of each phase pulp samples were forwarded to Australian Laboratory Services, of Perth, Australia (ALS) for checking of gold assays. Blank results demonstrated that overall the sample preparation was free of contamination. Standard and umpire assay results showed an approximate 5% low bias in samples from the core drilling programs. RC samples were shown to be in control. Duplicates agree within ±20% for the majority of samples.

Eldorado concludes that all the assay values that support the Perama Hill Mineral Resource estimates were in control. The effect of the low bias observed during parts of the diamond drill campaign is believed to have minimal impact on the Mineral Resource estimates. In places, the gold grade estimate may be slightly understated.

Data used for the resource estimation was verified by hard copy assay certificate checks to the digital database. The database was found to be valid and suitable for supporting resource estimation work.

1.4 METALLURGICAL TESTWORK AND PROCESSING

Metallurgical test work, including recovery and comminution studies on composite drill core samples have been undertaken on the hard, soft and a composite representative ore material. The results indicate that the material is all non-refractory and a standard circuit can be used for gold and silver extraction.

Based on the comminution, leach optimization and variability testwork performed, Aker designed a comminution circuit consisting of a three stage crushing circuit followed by a single stage ball mill, operating in closed circuit with hydro cyclones. A conventional CIL (carbon in leach) processing facility was allowed for with an annual throughput of 1.25 Mt/a of oxide ore containing gold and silver.

The comminution circuit will be designed to produce a product with 80% passing 75 µm. This material will then be thickened in a high-rate thickener before pre-aeration and then the CIL circuit. The CIL tailings will be detoxified to remove any remaining cyanide by the INCO process and carbon will be removed from CIL and the gold extracted by a split stream AARL (Anglo American Research Laboratories) elution process.

The tailings from the processing facility, after any remaining cyanide is removed, will be thickened and then filtered to remove any excess water. This material will then be transported by truck ready for placement in the lined tailings storage facility.

The Tailings Management Facility (TMF) is planned to be located within close proximity of the open pit and will ultimately comprise a structural fill embankment used in conjunction with filtered tailings for safely confining the Life of Mine (LOM) process waste. The design of the water supply network is based on a fresh water dam facility, designed to intercept surface runoff (rainfall) for discharge to the process plant, together with supernatant water reclaimed from the tailings management facility pond.

1.5 MINERAL RESOURCES

The Perama Hill mineral resources were modelled and estimated by Roscoe Postle Associates Inc. (RPA). Eldorado has reviewed the work in sufficient detail to assume responsibility for the Perama Hill resource estimates. Additionally, Eldorado has reclassified the Perama Hill mineral resource (added a component of Measured Mineral Resources).

3-D geologic models were created for the oxide mineralization, a small sulphide zone inlier, the East Graben Fault and the oxide-sulphide surface. A mineralized domain was also constructed within the oxide-mineralized zone. This domain, made using a 0.6 g/t Au cutoff grade, preserves the mineralization continuity and contains all potentially economic mineralization. A top cut of 30 g/t Au was implemented to assay data prior to compositing. The assays were composited into 2 m down-hole composites.

3D block models were made utilizing commercial mine planning software (Gemcom®). Block model cell size was 10 m east by 10 m north by 5 m high. Gold grade modelling consisted of interpolation by ordinary kriging. Silver grades were interpolated by inverse distance squared. Reasonableness of grade interpolation was reviewed by visual inspection of sections and plans displaying block model grades, drill hole composites, and geology. Good agreement was observed. Comparison of grades estimated by different interpolation methods showed little difference at the 1.0 g/t Au cutoff grade. Also, the distribution of modeled and assay gold data were compared on scatter plots by elevation and northings. General agreement was observed between the trends. Eldorado Gold concludes that the Perama Hill block model is valid and appropriate for supporting the Mineral Resource estimate.

The mineral resources of Perama Hill were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization satisfies sufficient criteria to be classified into Measured, Indicated, and Inferred mineral resource categories. Almost all of the Perama Hill mineral resources are classified either as Measured or Indicated mineral resources. Good geologic and grade continuity occurs in areas covered by the primary interpolation pass (40 m x 25 m x 25 m). Block covered by this pass were deemed to satisfy the requirements for Indicated mineral resources. A subset of model blocks covered by the primary interpolation pass, clustered around a more densely drilled portion of the deposit and contained within the current open pit design, was deemed to satisfy the requirement for Measured mineral resources. Blocks on this zone were approximately within 20 m to composites from at least two drill holes.

The Perama Hill mineral resources as of December 31, 2009, are shown in Table 1-1. These are reported at 1.0 g/t gold cutoff.

Table 1-1: Perama Hill Mineral Resources

Mineral Resource Category	Tonnes	Gold (g/t)	Contained Ounces
Measured	3,020,000	4.34	421,000
Indicated	8,690,000	3.37	942,000
Measured + Indicated	11,710,000	3.62	1,363,000
Inferred	333,000	2.58	27,000

1.6 MINERAL RESERVES

The Perama Hill mineral reserve estimates have been derived from the measured and indicated mineral resources occurring in the oxide portion of the deposit. Defined as the economically mineable part of the resource, the Perama Hill mineral reserves represent the portion of the mineral resources above the economic cutoff grade within the final pit design. Special consideration was applied to constrain the optimization shells from including any material that was within 500m of the Perama village.

The mineral reserves used a gold price of $825/oz, a 97% mining recovery and 3% mining dilution factors. An overall slope angle in the range of 32 degrees to 37.5 degrees was used on the East wall and 37.5 degrees elsewhere to derive the pit shell. The final pit design is approximately 630 m long in the north-south direction and up to 340 m in width. It extends from the top of Perama Hill at 248 m elevation to the pit floor at 125 m elevation. The highest elevation exposed on the final pit design is at 238 m elevation.

The mineral reserves of the Perama Hill Project were classified using logic consistent with the CIM definitions referred to in NI 43-101. The Measured and Indicated mineral resources contained within the final pit design satisfied sufficient criteria to be respectively converted to Proven and Probable mineral reserves. The cutoff grade (COG) assigned to the in-pit indicated oxide resources in order to estimate mineable reserves is 1.00 g/t Au. Table 1-2 shows a summary of the Perama Hill mineral reserves, as of December 31, 2009.

Table 1-2: Perama Hill Mineral Reserves

	Tonnes (Mt)	Gold (g/t)	Silver (g/t)
Proven Reserves	2.455	4.48	3.20
Probable Reserves	6.923	2.75	3.94
Proven and Probable Reserves	9.378	3.20	3.75
Contained Gold Reserves	0.966 Moz		
Contained Silver Reserves	1.129 Moz		

1.7 MINING OPERATIONS

The Perama Hill project has an orebody that is amenable to conventional open pit mining. The rate of production is 1.2 Mt of ore per year and an additional 350,000 tonnes of waste and low-grade material.

Main mine equipment include six 33 tonne trucks and two matching backhoes. A front-end loader is used on the ore stockpile at the crusher.

The pit will operate one 8-hour shift 5 d/wk and the feed to the plant from the ROM stockpile will be on 16 h/d for 6 d/wk. The reserves in the pit will allow for an 8-year mine operation.

1.8 OPERATING AND CAPITAL COSTS

Operating costs have been worked out in detail and the unit costs (in US$/t milled) of the major components are: $3.93 mining, $16.16 plant and $4.91 G&A, totalling $25.00. Capital costs (in US$ '000) have also been worked out in detail and total $187,369. Major components are:

- Equipment $53.383
- Installation $63,174
- Direct $7,348
- Indirect $40,173
- Sustaining $23,291

1.9 PAYBACK AND MINE LIFE

The payback period for the base case is 2.5 years. This period is based on the annual un-discounted cash flows. The mine will operate for 8 years from initial commissioning.

It is anticipated that the permitting process will be completed by early 2011 and that construction can commence shortly thereafter and that the mine will be ready for commercial production early 2014.

SECTION 2 • INTRODUCTION AND TERMS OF REFERENCE

2.1 TERMS OF REFERENCE

Eldorado Gold Corporation ("Eldorado"), an international gold mining company based in Vancouver, British Columbia, owns the Perama Hill Gold Project in Thrace, Northern Greece through its wholly owned Greek subsidiary Thracean Gold Mine ("TGM"). Eldorado has prepared this Technical Report of the Perama Hill Gold Project to support a first time disclosure of National Instrument 43-101 compliant mineral reserves.

Information and data for this report were obtained from Eldorado and engineering studies carried out by Aker Solutions E&C Limited. The work entailed review of pertinent geological, mining, process and metallurgical data in sufficient detail to support the preparation of this Technical Report.

Aker Solutions E&C Limited ("Aker") was retained by TGM in 2008-2009 to prepare an update of the project to reflect changes in the process design, tailings management facilities, capital and operating costs. Eldorado revised the mine plan in August 2009 upon which an updated economic evaluation was made and mineral reserves estimated.

The Qualified Persons responsible for preparing this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects ("NI 43-101") and in compliance with 43-101F1 (the "Technical Report") are Richard Miller, P.Eng. and Stephen Juras, Ph.D., P.Geo., both employees of Eldorado and Peter Perkins, MIMMM, MSAIMM, an employee of Aker.

Mr. Miller, Manager, Mining for Eldorado, was responsible for the preparation of the sections in this report that dealt with mineral reserves estimation, mine operations and related costs. He visited the Perama Hill Gold Project on September 18 to 19, 2009.

Dr. Juras, Director, Technical Services for Eldorado, was responsible for the preparation of the sections in this report that concerned geological information, sample preparation and analyses and mineral resource estimation. He visited the Perama Hill Gold Project on September 18 to 19, 2009.

Mr. Perkins, a Technical Manager for Aker, was responsible for the preparation of the sections in this report that dealt with metallurgy and process operations and related costs. He has visited the Perama Hill Gold Project in 2000.

All monetary amounts are expressed in United States of America Dollars (US$) unless otherwise stated.

The term "ore" is used for convenience throughout the report to denote the portion of the Measured and Indicated Mineral Resources that have been converted to Proven and Probable Mineral Reserves.

SECTION 3 • RELIANCE ON OTHER EXPERTS

Eldorado Gold has prepared this document with input from TGM staff. Third party experts have supplied additional work and information and the authors of this document have reasonable reliance on that information as coming from technical experts. This report therefore relies inherently on the conclusions and recommendations of the following third party consultants:

Eldorado Gold has used the information from the above-mentioned works understanding that it was prepared by Qualified Persons.

Roscoe Postle Associates (now part of the Scott Wilson Mining Group, and referred to in the report as "Scott Wilson")

- prepared the current geological and block model which has been described in the 2004 Technical Report titled "Report on Perama Hill Gold deposit mineral resource estimate, Greece" (Section 17).

Golder Associates (UK)

- contributed to the design and costing of the dry stack tailings management facility and provided guidance for a geotechnical assessment of the pit slope angles in their 1999 report, "Feasibility Level Geotechnical Investigations Perama Hill Open Pit, Alexandropoulos, Greece. (Sections 17 and 19)

Aker Solutions E&C Limited

- updated metallurgy and process plant work and costs to reflect changes in the process design, tailings management facilities, capital and operating costs. (Sections 16 and 19)

Analytical Solutions Ltd.

- reliance on report, "Review of Quality Control data, Perama Hill Project, Greece and August 1997 to December 1998. (Sections 12, 13, and 14).

SRK Consulting (UK)

- Mine closure and Rehabilitation report for the project. (Section 18)

Hypodomi 3E Ltd.

- Blasting Vibrations Study at Perama Hill Open Pit Mine, May 1999. (Sections 17 and 19)

SECTION 4 • PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY DESCRIPTION AND LOCATION

The Perama Hill gold deposit is located 30 km west-northwest of the city of Alexandroupolis and 20 km south of Sapes in the eastern Thrace region of northern Greece, as shown in Figure 4-1. The UTM coordinates (Zone 35 ED 50) for the centre of the property are approximately 385,070E and 4,529,600N. The Perama Hill Property comprises two mining titles (MT) granted in December 1999 and cover an area of 1,897.5 ha. These MTs expire in 2049. The Project straddles two mining titles (Mining Titles 54 and 55), which are located in the Evros Prefecture Figure 4-2.

Figure 4-1: Perama Hill Property Location Map

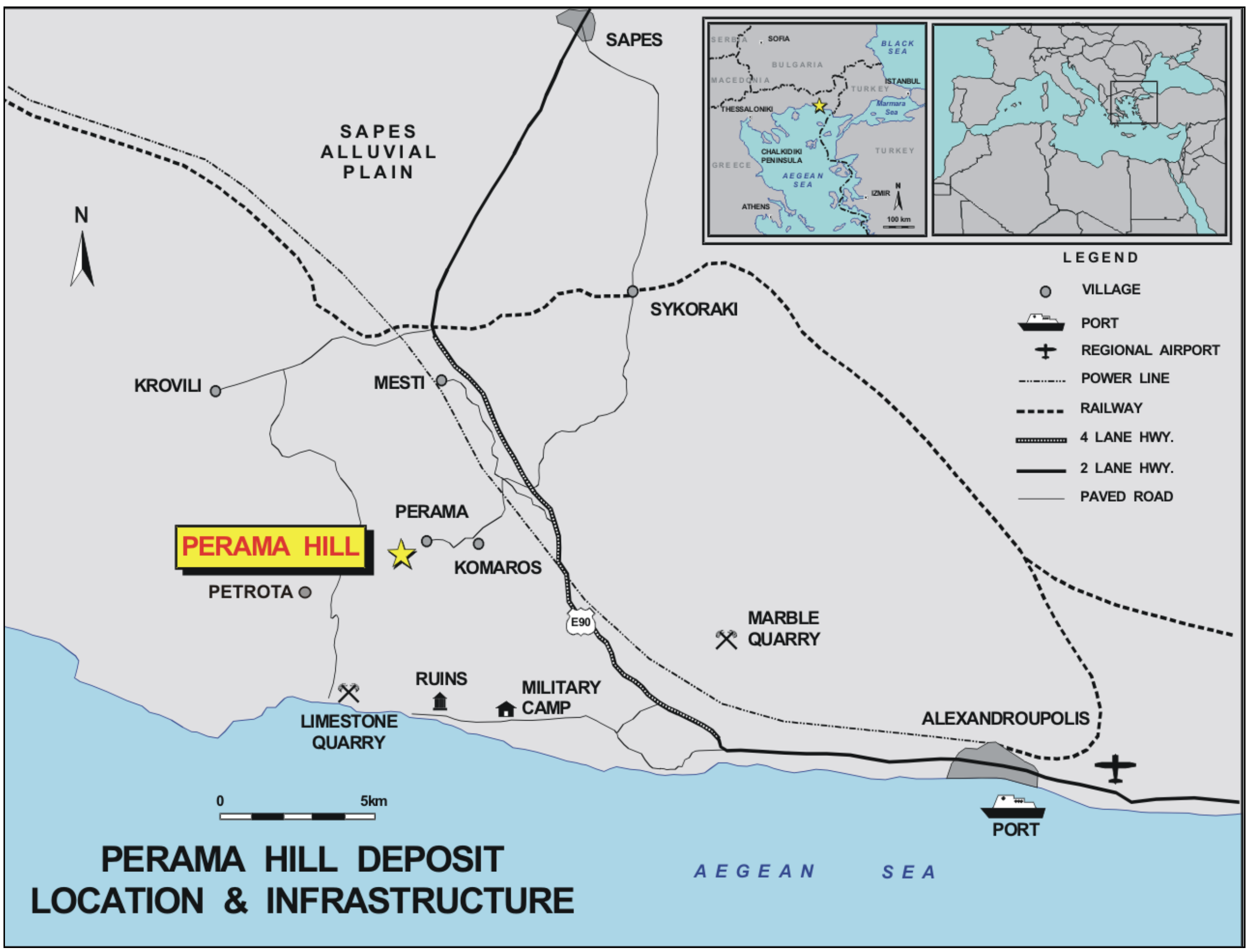
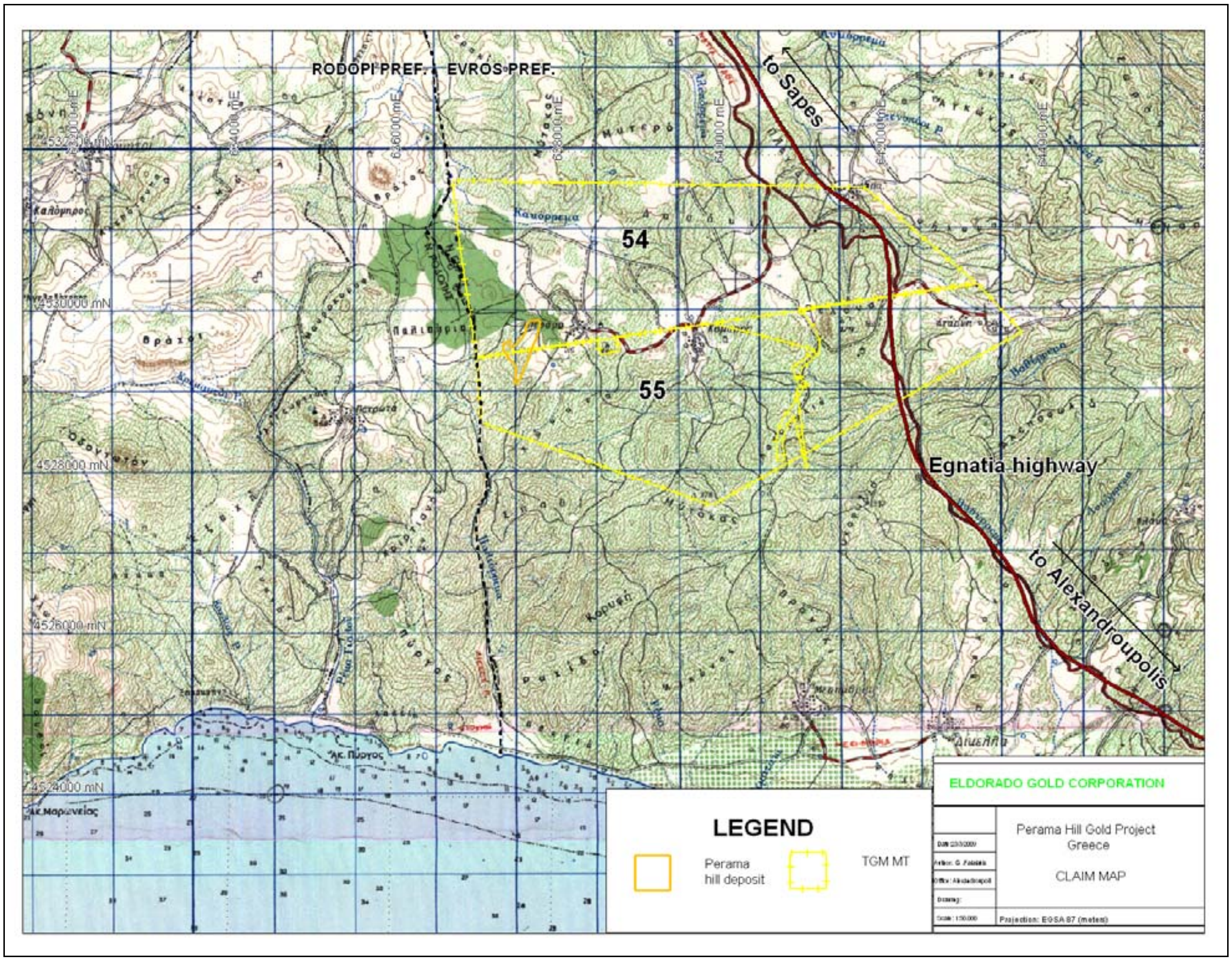


Figure 4-2: Perama Hill Property Claim Map



4.2 PERMITTING

Table 4-1 lists the five licenses required to permit the Perama Hill Gold Project. Currently, the Project is in the process of obtaining the third license, the Environmental Permit Licence. The Preliminary Environmental Impact Assessment Study, (PEIA) was submitted in 2009. Detailed descriptions of each license and their procedures are in Appendix A.

Table 4-1: Required Licenses for the Perama Gold Project

	License	Duration	Status	Submission Reg. No.	Decision Reg. No
1	Mining Exploration License	3 years with 1 year renewable possible	Completed		
2	Mining Title Acquisition	50 years with possible extension of 25 years	Completed		Presidential Decree FEK 2182/B/20.12.1999
3	Environmental Permits License	Renewable every 5-10 years, or with modification of mine exploitation or process plant	PEEE (3.1)	PEIA Study Submission: MOE 145829/30.10.2009	
4	Mine Operation License	Renewable with modification of mine exploitation or process plant			
5	Construction & Operation Licenses	Renewable every 3-5 years, or with modification of installations or machinery foundation and equipment			

Note: MOD = Ministry of Development, MOE = Ministry of Environment, JMD = Joint Ministerial Decision

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

Perama Hill is in the Evros Prefecture in Thrace, Greece. Access to the property is by driving from the Evros Prefecture capital, Alexandroupolis for 24 km west on the four lane highway E90 to a junction signed for Perama. A 6 km paved road ends at Perama village. Numerous farm access roads and tracks provide access to the Perama Hill deposit located approximately 1 km southwest of the village. The Perama village has a population of 120. The northeastern part of the open pit is located no closer than 0.5 km from the outer edge of Perama village. Other nearby villages are Komaros in Evros with 200 inhabitants and Petrota in Rodopi with 15 inhabitants. There are regular flights from Athens to Alexandroupolis, which has a well-developed seaport on the Aegean Sea with easy access to all sea routes.

The topography is dominated by a hilly terrain of ridges and moderately steep-sided valleys, varying in altitude between 250 masl and 300 masl with a general alignment that follows a northwest-southeast orientation. Perama Hill forms a gentle topographic high of 100 m rising above the broad Paleorema creek valley. The deposit is situated 5 km north of the Aegean Sea and is sparsely vegetated with small oak trees and Mediterranean thorn and scrub bushes. A stand of black pines occurs on the north and northwest slopes and is part of the Black Pine Forest. The east slopes and a portion of the top of the hill are cultivated for sheep grazing. The remainder of the hill and the Paleorema drainage are used only for occasional goat grazing.

The climate of the wider project area is characterized as Mediterranean with dry and warm summers and mild winters. The mean annual air temperature is approximately 15^{o}C with the minimum monthly average to appear in January (5°C) and the maximum monthly average in July (25.8°C). The mean annual rainfall is 557 mm with a range of between 25 mm and 867 mm per annum having been recorded. The mean annual pan evaporation is quite high (1,659.8 mm) and varies between 21.2 mm in January and 290.8 mm in August. The prevailing winds in the area come from the northeast, are of low to moderate intensity (2 to 4 Beaufort) while the percentage of calmness is quite high (~30%).

Figure 5-1: Looking NE at Perama Village from Perama Hill in December 2003

SECTION 6 • HISTORY

6.1 HISTORY OF OWNERSHIP

In 1989, the 67/33 Eurogold joint venture between Australia Consolidated Minerals (ACM) and Inmet Mining Corporation (Inmet) started exploration work in Greece. ACM had a 67% interest, which was acquired by Normandy Mining Limited (Normandy) in 1991. In 1994, a joint venture between Eurogold and S&B was formed. In 1999-2000, TGM incorporated as the joint venture and Inmet sold out of the Eurogold joint venture, leaving Normandy with 67%. S&B offered 13% of its shares to Normandy to bring it up to the final equity interests of 80% Normandy and 20% S&B. Newmont acquired its 80% interest when it acquired Normandy in 2002. Personnel from Normandy LaSource (NLS), previously Bureau de Rechérche Géologiques et Miniérs (BRGM), based in Orléans, France and Normandy subsidiary supervised some of the Perama Hill geological work. In November 2003 Frontier Pacific, a Canadian mining company based in Vancouver acquired all outstanding shares of TGM. In July 2008, Eldorado Gold Corporation acquired 100% of Frontier Pacific and thereby 100% of TGM.

The following is a summary of previous work from 1993 up to the end of 1998 and is taken from McAlister et al. (1999).

- Maronia graben area evaluated by Newcrest in 1993 utilizing the Bulk Leachable Extractable Gold (BLEG) technique (47 samples) and 27 rock samples. No anomalies were encountered, and they concluded that there was no significant gold and other element anomalies associated with the volcanics despite the presence of siliceous alteration and locally sericite-clay alteration.
- Perama South prospect was discovered by TGM in 1994 as a result of 1993-1994 BLEG sampling. Follow-up from 1993, 9 ppb Au BLEG returned two further BLEGs >9 ppb. An anomalous area of gold mineralization occurring along the foliation in calcium schists, measuring about 1.5 km x 1.5 km, was defined by –80 mesh sampling, rock chipping and mapping.
- From late 1994, the general area was recognized as having high-level epithermal gold potential. The small hills to the north of Perama Hill were identified as zones of interest, however, work continued mainly at Perama South.
- In August 1995, the stream to the south of Perama Hill was investigated further to a 22 ppb -80 mesh stream anomaly. This led to the “Discovery Outcrop.” Initial channels across the stockwork zone yielded 1.88 ppm Au over 48 m. Follow-up channel sampling continued until August 1996. Channel samples included 9.9 ppm Au over 53.4 m, 17.6 ppm Au over 20.1 m and 12.7 ppm Au over 17 m. Grid soils (50 m x 25 m spacing) outlined two northwest trending broad linear robust zones of >1 ppm Au anomalies covering an area of 12 ha with individual values of up to 12.5 ppm Au.
- Licence MEL 563 (now MT 55) granted in August 1996 covering Main Zone soil anomaly and part of the Perama South prospect.

- Reverse circulation drilling commenced in September and October 1996 for 18 holes at nominal 50 m depths (PC-01 to PC-18). Essentially all holes were mineralized over their full lengths.
- Diamond twin drilling in August and September 1997 by a local Greek contractor confirmed tenor of reverse circulation results (PD-01 to PD-06).
- Commencement of 100 m x 100 m grid diamond drilling in October 1997 by the drilling contractor Radial (PD-07 to PD-33).
- Initiation of reverse circulation drill program by the drilling contractor Radial Drilling Europe, with a UDR 650 drill rig in April 1998 (PC 19-24). Continued twin drilling for a total of 9 twinned holes to May 1998.
- Commencement of 50 m x 50 m grid drilling in March 1998 (PD-34 to PD-75 and PC-25 to PC-35).
- Public announcement of the project was made on April 16, 1998.
- Initiation of base line studies and environmental procedures.
- Sterilisation and geotechnical drilling in October 1998.
- Reverse circulation drilling (PC-36 to PC-102) on a 25 m x 25 m grid to cover roughly 60% of the deposit was completed by September 1998.
- Up to October 1998, a total of 16,342 m (9,093 m of diamond drilling and 7,249 m of reverse circulation drilling) in 177 holes were drilled for resource evaluation purposes. A further 1,244 m in 33 holes were drilled for geotechnical and condemnation purposes.

6.2 HISTORICAL RESOURCE AND RESERVE ESTIMATES

In March 1998, NLS used the 100 m x 100 m drilling results and a sectional polygonal method to estimate a global unclassified historical resource of approximately 11.1 Mt averaging 3.59 g/t Au. NLS applied a 0.65 g/t Au cutoff grade and high assays were not cut. The maximum sectional influence was 50 m.

In July 1998, NLS used the 50 m x 50 m drilling results and a sectional polygonal method to estimate a global unclassified historical resource of approximately 10.9 Mt averaging 3.83 g/t Au. NLS applied a 0.65 g/t Au cut-off grade and high assays were not cut. The maximum sectional influence was 37.5 m.

In 1999, NLS used commercial mine planning software to construct a block model. The NLS oxide mineralization wireframe model was loosely constrained at a 0.65 g/t Au cutoff and included a minor amount of sulphide mineralization. High assays were not cut. At a 1.0 g/t Au block cut-off grade, the 1999 NLS historical resource estimate totalled 10.69 Mt averaging 3.57 g/t Au and 7.05 g/t Ag and contained 38.2 tonnes (1.22Moz) of gold.

Kvaerner adopted the 1999 NLS historical resource estimate for the feasibility study completed in 2000 and used it to estimate the historical reserves based on a US$300/oz gold price. At a

1.45 g/t Au block cutoff grade, Kvaerner estimated historical proven and probable reserves totalling 11.0 Mt of oxide ore averaging 3.71 g/t Au and 8.3 g/t Ag and containing 1.3 Moz of gold.

6.3 2000 REVERSE CIRCULATION DRILLING

In the spring of 2000, TGM drilled 35 reverse circulation drill holes totaling 1,866 m. Some 24 drill holes (PR103 to PR126) were 40 m to 50 m long vertical holes drilled on a very close-spaced pattern (5 m to 10 m apart) for geostatistical purposes. Most of the other holes were drilled to better define the mineralization along the eastern margin.

SECTION 7 • GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Perama Hill area is located on the eastern margin of an interpreted north-south trending graben, which developed during the early Tertiary as a response to northward subduction of the African plate beneath the European continent. East-west oblique extension produced volcano-sedimentary basins, which developed and filled during the Eocene and Oligocene.

The graben ranges from 5 km to 15 km wide (east-west) and extends to the north for 15 km where it becomes covered by the Sappes alluvial plain. The graben fill consists of intermediate to felsic ignimbrites and epiclastics, andesitic hyaloclastites, debris flow breccias and epiclastics, intercalated with fossiliferrous limestones, marls and sandstones. The regional formation has a shallow north dip. Locally, mafic domes and dikes crosscut the formation and the entire volcano-sedimentary package rests on a gabbroic sequence interpreted as being of Paleocene age.

Although no evidence of any other intrusive activity has been observed within the graben, a quartz monzonite stock intruded by a rhyolite porphyry is exposed at Maronia a few km southwest of the mineralized zones near the west graben margin, and a similar one occurs on the east graben margin 15 km northeast of Perama Hill.

The west and east horst blocks consist of a complex package of lower to middle greenschist facies meta-volcanics and meta-sediments of various Mesozoic ages, which occur outside the graben and are exposed throughout the southern Rhodopi area. This package includes quartz-feldspar porphyry sills or flows, bituminous limestones, marbles and quartzites. Some of these units also host gold mineralization at the Perama South prospect.

7.2 LOCAL AND PROPERTY GEOLOGY

Perama Hill is underlain by a Cenozoic (late Eocene-Early Oligocene, 25 Ma to 35 Ma) age sequence of graben-fill volcanics and sediments in fault contact with Mesozoic meta-volcanics of the regionally extensive Peri-Rhodope zone.

The deposit is hosted by a felsic (>80% quartz) volcanic sandstone overlying a package of andesitic volcanic breccias. The sandstone is thick bedded to massive and contains features such as crossbedding, wood fragments, and fossil tubes. The andesites are composed of dominantly in-situ and reworked hyaloclastite, with intercalated thin to medium beds of andesite bearing conglomerates and finely bedded reworked tuff or ash.

A conglomerate unit ranging up to several metres thick and containing occasional andesite volcanic clasts marks the transition from andesitic breccia to felsic sandstone. In the northern part of the deposit, the sedimentary sequence has been eroded away possibly due to the topographic expression of thinner sandstone sedimentation and/or through uplift. This suggests a period of erosion prior to the deposition of sandstone. Mineralization in this area is mainly hosted by volcanic breccias.

In the south central part of the deposit, the volcanic sandstone can reach thicknesses of 120 m. The average dip of the sandstone is 15° to 20° west. Cross-sections indicate the sandstone was deposited in a paleo-depression, the axis of which is oriented north-northeast and sub parallel to the East Graben Fault. The sandstone dips are more horizontal in the northern part.

The eastern margin of the deposit is marked by the north-northeast trending East Graben Fault, which juxtaposes the volcano-sedimentary package against strongly deformed quartz-muscovite-chlorite schists of the lower greenschist facies series of the Peri-Rhodope zone. The local geology is shown in Table 7-1.

Mineralization occurs in all three Tertiary units, namely the andesitic breccia, conglomerates, and felsic sandstone and in some cases achieves grades as high as 30 g/t Au in each unit.

7.3 STRUCTURE

The tectonic events comprise two main phases. A first phase of extension and deposition of the volcano-sedimentary formation in the Maronia basin was followed by a second phase of transpression with sinistral movement and uplifting. The mineralization is believed to be contemporaneous with this second phase.

The dominant structure is the 200°/-60°W East Graben Fault, juxtaposing the Cenozoic sequence described above with strongly deformed and drag folded non-mineralized Mesozoic meta-volcanics. Three secondary fault trends are also recognised:

1. 160° to 180°/-75°W to -75°E. This trend is probably transpressive and one structure, the North Fault, has been identified in the north of the deposit. It trends 172°/-75°W with an apparent 30 m normal displacement with a sinistral sense of movement.

2. Another group limited to the north of the deposit, are dextral strike slip faults oriented 045° to 060°/-45° to -75°SE.

3. The stockwork zone at the discovery outcrop, to the south of Perama Hill, displays a dominant trend of 310° to 330°/-75°NE. This trend is represented by rock jointing and quartz-barite filled veins and has been observed throughout the Main Zone. It is considered to have played a major role in controlling hydrothermal fluid movement.

The structures present at Perama Hill are consistent with an interpreted strike slip regime with sinistral oblique slip on the main graben fault. The property geology is shown in Figure 7-2.

Figure 7-1: Perama Hill Gold Project, Local Geology

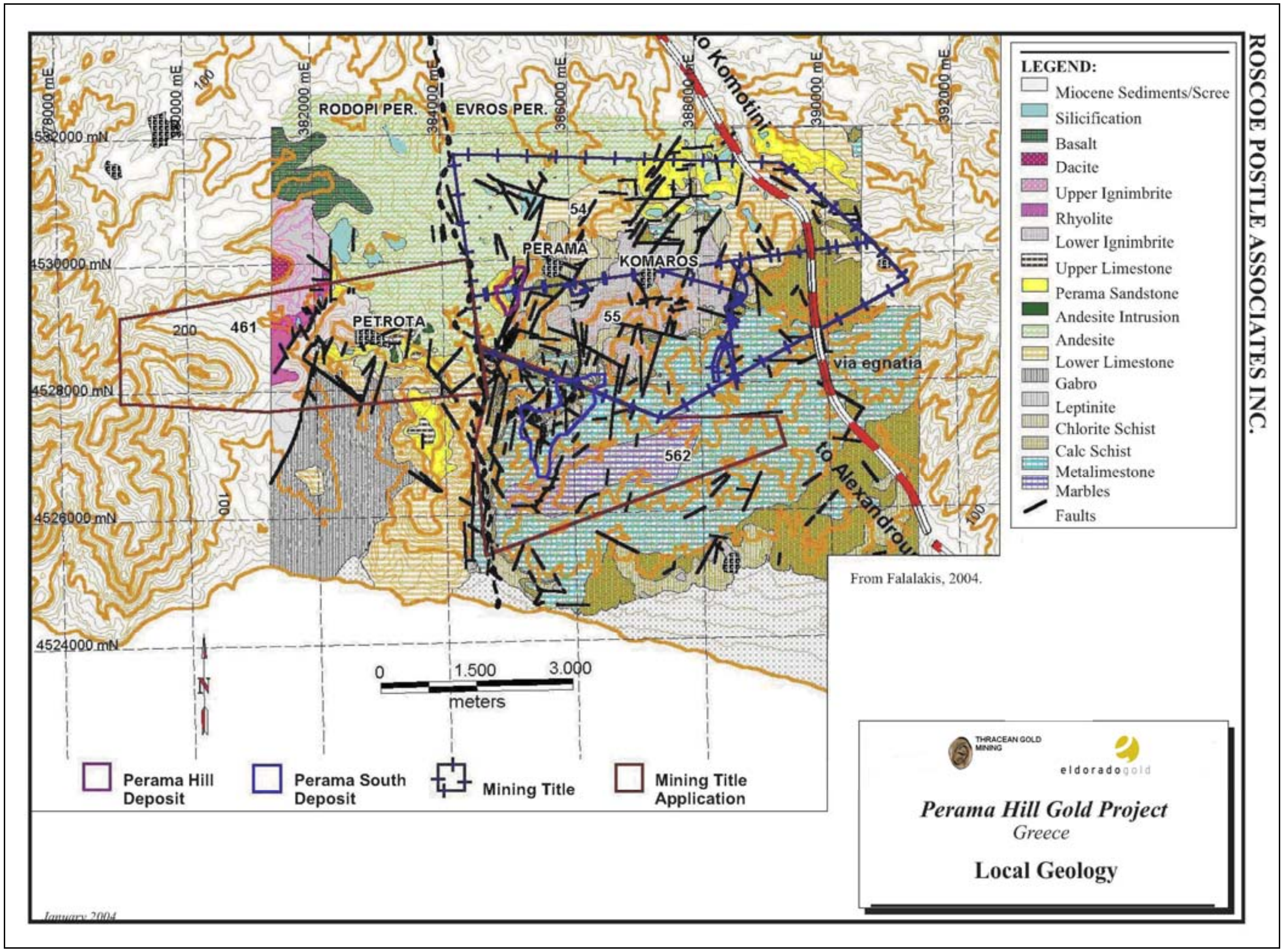
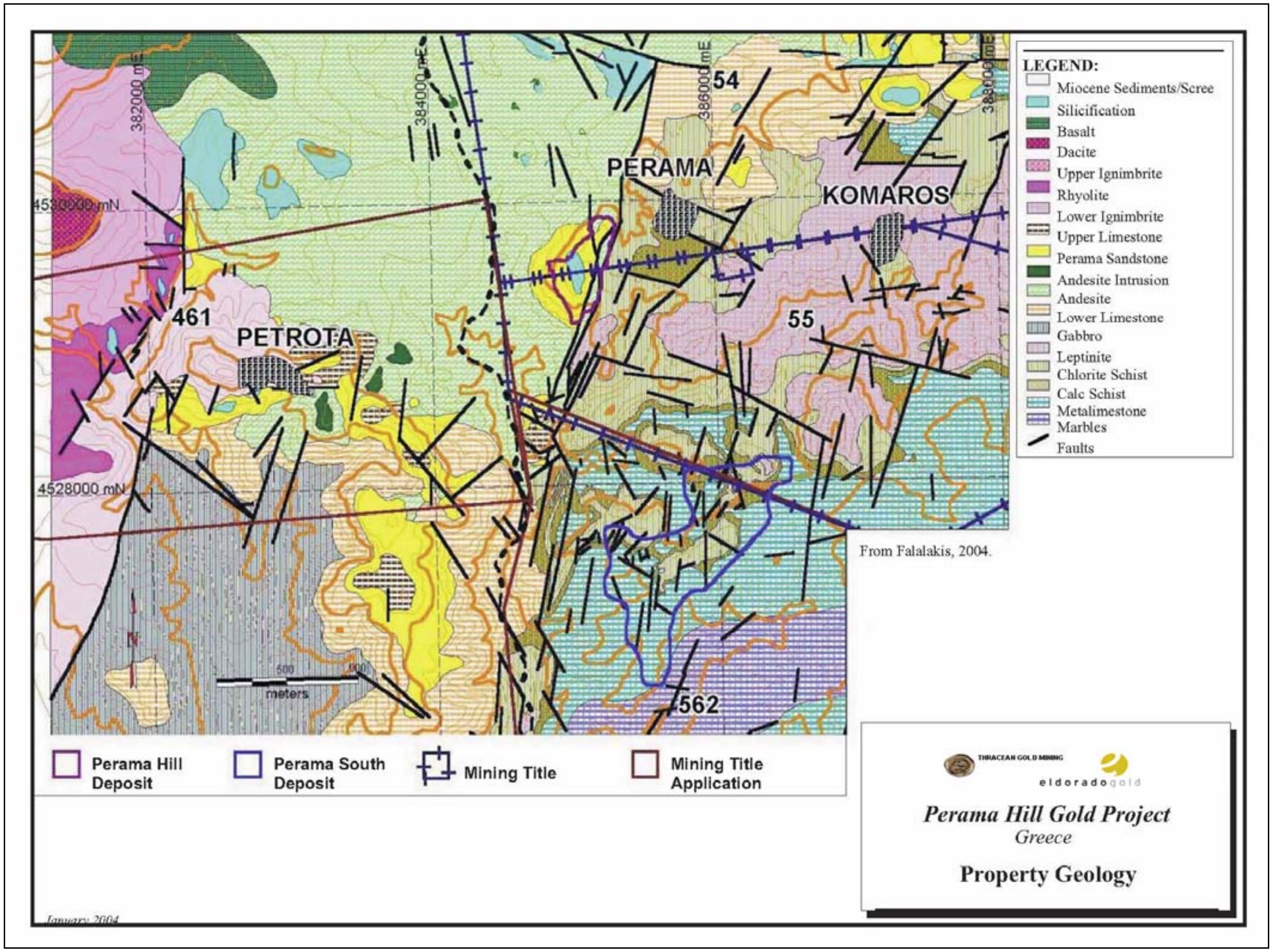


Figure 7-2: Perama Hill Gold Project, Property Geology



SECTION 8 • DEPOSIT TYPE

The Perama Hill gold deposit is a stratabound, sediment hosted deposit of Eocene to Oligocene age, located at the eastern edge of the Maronia Graben at the contact point with a north-northeast trending eastern graben fault. The Perama Hill deposit is described as an initial high sulphidation epithermal system overprinted by typical low sulphidation banded quartz-chalcedony, barite, pyrite stockwork veins, and veinlets. Two key controlling factors on the mineralization are the porosity and rock competency contrasts between the volcanic sandstone and underlying andesite package. The downhole multi-element data exhibit two distinct geochemical signatures:

1. A reduced facies in the andesite package with copper sulphides and sulphosalts, gold and silver tellurides and a Au, Ag, Cu, Pb, As, Bi, and Sn signature.

2. An oxidized facies in the sandstones of quartz-barite mineralization with an Au, Ag, Ba, Pb signature without Cu.

The signatures correspond to two different styles of mineralization: structurally controlled vein type mineralization in the andesite package and lithologically dependent stratabound mineralization in the volcanic sandstones (McAlister, 1999).

SECTION 9 • MINERALIZATION

The deposit extends 750 m in a north-south direction and up to 300 m in an east-west direction and the thickness varies from 15 m to 20 m at the flanks and up to 125 m at the centre. The gold mineralization has been fed into the system by structurally controlled feeders inside andesitic volcanic breccia and disseminated into overlying porous sandstone units. The central part of the deposit is mushroom-shaped. Eighty percent of the gold is hosted by sandstones and the rest is associated with andesitic volcanic breccia and conglomerates.

Mineralogical studies from the Perama Hill gold deposit indicate that gold mineralization is very uniform throughout the deposit and the gold grain size is small, generally less than 2 µm.

Gold mineralization occurred during the latest stage of silicification and gold bearing siliceous fluids were introduced after structural preparation by hydraulic fracturing and veining of the same hardened silicified rock. Gold is identified in the sulphide mineralization in association with very fine pyrite containing sub-micronic gold and gold silver telluride. Enargite and stannite group minerals (Cu, As, Sb sulphides) were also introduced in this event. Galena and sphalerite are also present in the sulphide ore but do not appear to correlate with gold.

In the oxidized portion of the Perama Hill deposit, gold recoveries average about 90% in standard cyanidation bottle roll tests. The gold occurs as micron size particles in association with quartz, clay and hematite. The sulphide mineralization is refractory with recoveries in the range of 2.5% to 16.5% gold in standard cyanidation bottle roll tests. The gold in the sulphide is locked in pyrite crystal lattices.

Mineralization in the sandstones accounts for more than 80% of the oxide deposit. Hydrothermal fluids, which travelled upwards through structurally controlled conduits in the andesite breccia were able to disseminate and travel laterally through a finely developed network of fracture systems developed in the porous sandstones.

This has provided for good grade continuity in the Perama Hill sandstone unit. A paleo water table is implied by the high barite content in the sandstones and interpreted high fluid pressure occurring at the contact with the volcanic breccia. Clay-barite ± quartz-pyrite brecciation with a significant increase of gold grade observed at this contact also supports this hypothesis.

Mineralization in the volcanic breccia unit is confined to the sulphide facies in the central part and the oxidized uplifted panel in the northern part of Perama Hill. Due to less supergene leaching in these less porous, quartz-free and strongly clay altered rocks, metals like Cu and Pb can be identified in outcrop in association with clay and strongly developed limonite minerals. Heterogeneity of the rock, allowing fluid circulation at the contact between clasts and matrix under hydraulic pressure, was a favourable factor.

9.1 ALTERATION

An initial stage of regionally extensive argillic alteration consisting of illite-smectite was followed locally by a silica-pyrite advanced argillic phase. Mineralization is characterised by intense silicification controlled by both lithology and structure. Mineralized sandstones display pervasive silicification and the development of cavities and voids filled with barite or kaolinite. The barite forms in small clusters of distinctively bladed crystals in the sandstones. The andesite breccias are characterised by clay altered devitrified clasts surrounded by a pervasively silicified matrix.

Supergene alteration has affected the upper sandstone formation resulting in a porous sandy texture. Gold grades are systematically higher in this material.

9.2 OXIDATION

Extensive development of Liesegang bands in the Perama Hill sandstones suggests a high degree of porosity and the possibility of supergene oxidation. Hypogene oxidation must also be considered to account for the deep level of oxidation in the deposit, despite the absence of specularite. It should be noted that the major East Graben Fault on the deposit's eastern margin does not display deep oxidation. In the volcanic breccia, the oxidation limit rarely exceeds 30 m except in highly faulted zones, possibly due to low porosity.

In general, the oxidation limit roughly equates with the sandstone/volcanic breccia contact, allowing a thickness of more than 100 m of non-refractory oxide mineralization in the middle of the deposit. This limit diminishes laterally to 15 m to 20 m on the deposit margins.

SECTION 10 • EXPLORATION

The exploration work to date is summarized under the Historical Work and Drilling Sections. No further exploration work has been done on the Perama Hill Property since 2000.

SECTION 11 • DRILLING

TGM carried out a number of drilling campaigns on the Perama Hill Property from 1996 to 1998 and again in 2000.

11.1 1996 TO 1998 DRILLING

Core drilling at Perama Hill was contracted to Radial Drilling Europe (Radial) a Greek registered division of Radial Drilling Ltd. of Townsville, Australia. This followed an initial program with Diadrill of Athens (PD-01 to PD-6) where some recovery problems and time delays were experienced. Radial utilized a track-mounted LY44 and a UDR650 for most of the core work. Each rig was staffed with one expatriate driller and two helpers hired from the Perama village, drilling was conducted on a two 12 hour shift basis. Radial was based in Sappes, Greece, prior to this program.

Core sizes of PQ and HQ were maintained throughout the program. Efforts were made to drill the oxide mineralization with PQ maximizing recoveries and reduce to HQ at the sulphide mineralization boundary. The core was collected at the drill site each morning by TGM staff for transport to the core preparation facility at Sappes.

Reverse circulation (RC) drilling was undertaken by Eurofor Drilling Ltd. of Belgium as well as by Radial. Samples were split on site with a field splitter by TGM staff with assistants provided by Perama village. The splitter was systematically cleaned with compressed air to avoid contamination.

At the conclusion of the 1996 to 1998 resource drilling a sterilisation programme was undertaken in the area of the proposed plant site, waste dumps, and tailings dam. This work involved 838 m of RC drilling in 25 holes. A further 6 diamond drill core holes were drilled in the proposed pit area for geotechnical purposes as well as three core holes in the proposed tailings dam area. Finally, one hole was drilled for metallurgical studies. In 1998 the drill hole accesses and drill sites were rehabilitated. This work included seeding grass.

11.2 2000 DRILLING

In April and May 2000 TGM drilled 24 close-spaced RC holes (PR-103 to PR-126) totalling 1,118 m to assess grade variability and 11 RC holes (PR-127 to PR-137) totalling 748 m to better define the mineralization along the eastern margin. Radial was the drilling contractor.

Table 11-1 shows a summary of all project-drilling to-date and Table 11-2 the resource evaluation drilling.

Table 11-1: Project Drilling Summary

Purpose	Year	Holes	Type	Metres
Resource Drilling	1996	18	RC	893.8
Resource Drilling	1997	11	DDH	1,421.4
Resource Drilling	1998	64	DDH	7,671.7
Resource Drilling	1998	84	RC	6,356.0
Resource Drilling	2000	35	RC	1,866.0
Resource Drilling Total		**212**		**18,208.9**
Sterilization	1998	25	RC	838.0
Geotechnical (Pit Walls)	1998	6	DDH	321.9
Geotechnical (Tailings Dam)	1998	3	DDH	90.0
Metallurgy	1998	1	DDH	117.2
Engineering Studies Total		**35**	**-**	**1,367.0**
Perama Hill Project Total		**247**	**-**	**19,575.9**

Table 11-2: Resource Evaluation Drilling Summary

Time Period	Contractor	Hole Series	Type	Holes	Metres	Samples	QC Samples
Exploration Drilling							
02/09/96 to 12/10/96	IGME	PR-01 to PR-18	RC	18	894	449	10
Confirmation Drilling							
16/08/97 to 14/11/97	DIADRILL	PD-01 to PD-06	DDH	6	752	747	51
100 m x 100 m Drilling							
14/11/97 to 19/03/98	RADIAL	PD-07 to PD-33	DDH	27	3,216	3,207	469
Twin Drilling Program							
21/04/98 to 23/04/98	RADIAL	PR-19 to PR-24	RC	6	607	607	87
50 m x 50 m Drilling							
19/03/98 to 18/06/98	RADIAL	PD-34 to PD-75, PG-04	DDH	43	5,200	5,056	735
	RADIAL	PR-25 to PR-35	RC	11	613	610	90
25 m x 25 m Drilling							
27/7/98 to 13/10/98	EUROFOR	PR-36 to PR-102	RC	67	5,136	5,136	787
	RADIAL						
Geostatistical & East Margin Drilling							
Spring 2000	RADIAL	PR-103 to PR-137	RC	35	1,866	1,865	77
Total Resource Drilling							
1996 to 2000		PD-01 to PD-75, PG-04	DDH	76	9,168	9,010	1,255
		PR-01 TO PR-137	RC	137	9,116	8,667	1,051
				213	**18,284**	**17,677**	**2,306**

SECTION 12 • SAMPLING METHOD AND APPROACH

12.1 SURVEYING OF DRILL HOLES

Downhole deviation (azimuth and inclination) measurements were taken at the bottom of 13 vertical and 2 inclined diamond drill holes in the series from PD-30 to PD-69 using an Ausmine camera. No significant deviation was observed. Flattening ranged from approximately 0.5° to less than 1° per 100 m in the vertical holes and the two inclined holes PD-39 and PD-65, 128 m and 90 m long respectively, essentially showed no change in dip and minor to moderate changes in azimuth. No deviation measurements were made for the reverse circulation holes. Collars of all reverse circulation and diamond drill holes were surveyed at the end of each drilling phase. In Scott Wilson's opinion, the portion of the drill holes that support the oxide resource are generally too short to deviate significantly and Scott Wilson believes that the drill hole sample location data are reasonable. Also, no significant deviation would be expected in the first hundred metres or so because most of the holes are vertical and large diameter drilling equipment was used. Eldorado Gold supports Scott Wilson's reasoning and opinion that hole deviation is essentially a non-issue at Perama Hill.

A NLS surveyor assisted by TGM staff using a TOPCON GTS 701 conducted an initial topographic survey of the project area during March and April 1998. A total of 95 ha were surveyed including the deposit, tailings dam area and the future plant site. A Hatt projection co-ordinate system was used and based on one geodesic point (Perama) and five previously existing points. The survey grid used a 20 m x 20 m spacing and an accuracy of approximately 0.25 m was obtained in both the horizontal and vertical co-ordinates. The contour interval was 2 m. Compilation of data was done on Sitework and Microstation software.

The second survey of the project site for the areas not covered during the April 1998 survey and a Greek registered surveyor completed notably the tailings dam facility area in June 1999. The NLS surveyor using Microstation conducted the merging of both surveys to complete a final topographic map of the project area.

12.2 CORE LOGGING AND SAMPLING

TGM staff collected diamond drill core at the rig and transported to the sample preparation facility located near the town of Sappes approximately 15 km north of the site. The core was washed and the hole number, box numbers and intervals were marked. This information was recorded on a Core Box Meterage Record form.

The core was photographed on a two box vertical stand and the prints were mounted in groups of three per page in the core photo albums, which are stored at the Alexandroupolis office.

Following preparation, the core was logged for geotechnical information including percent recovery, rock quality designation (RQD), joint frequency and condition, degree of breakage, weathering/alteration, and hardness. The information was stored in hard copy in the Alexandroupolis office and digitally.

The core was then logged for geology. The geological logs record lithology, degree of oxidation, grain size, observed quartz, sulphide and alteration minerals as well as a brief description of important features. This information was also entered in the digital database and TGM produced graphical or strip logs for all of the diamond drill holes and all of the RC holes from PR-25 to PR-137.

Reverse circulation chip samples were logged in a similar manner as the core and stored at the Sappes sample preparation facility. Reverse circulation samples were collected in 1 m intervals and split to get a 1 kg sample. The sample was further split in half to produce a duplicate.

The core was marked in 1 m intervals and sample tickets were placed at the end of each interval. The core was then delivered for sample preparation where it was cut in half with a diamond rock saw, lengthwise. In some cases, very soft material was sampled with a spoon. One half of this material was returned to the core box and the other half put in pans and dried overnight in the sample preparation oven at around 110°C.

After logging and sampling, the core boxes were stored at the sample preparation facility on core racks (Figure 12-1) and representative RC chips were also archived (Figure 12-2).

Figure 12-1: HQ Diameter Oxide Sandstone Drill Core

Figure 12-2: RC Chip Archive

12.3 CORE RECOVERY

The average recovery for the entire diamond core data set within the mineralized envelope is 95%. Individual drill holes range from 83% to 100%. Within the data set, 9% of the samples have recoveries of less than 80%.

The TGM recovery figures for the reverse circulation holes should only be considered as indicative rather than actual values. The weight of each individual sample was calculated using the average density of the deposit but there is significant variation in density along the length of a hole. The density measurements for oxide sandstone range from 1.58 g/cm³ to 2.63 g/cm³. The average recovery for the entire reverse circulation data set (excluding the initial exploration holes PR-01 to PR-18) within the mineralized envelope is 94% with individual drill hole recovery ranging from 50% to 107%. Recoveries for the initial exploration holes (PR-01 to PR-18) were not calculated but estimated by the drill geologist. All holes except one drill hole (PR-15) was qualified as "Good" with an estimated recovery of between 80% and 100%. Recovery for hole PR-15 was estimated at between 50% and 80%.

Scott Wilson recommends that the RC recovery data be factored using local density data in the future to generate more accurate RC recovery values. The RC recoveries show significantly more variability and some localised intervals, particularly near surface, have moderate to poor recoveries. Scott Wilson concludes that overall the diamond drilling and RC drilling recoveries are reasonable. Eldorado Gold concurs with Scott Wilson's conclusion and recommendation on RC data.

The assays for holes PD-01 and holes PR-54 are excluded in the mineral resource drill hole databases because of low recoveries, 77% overall for hole PD-01 and 73% for hole PR-54.

12.4 SIGNIFICANT MINERALIZED INTERCEPTS

Table 12-1 lists significant mineralized intercepts in the oxide zone of the Perama Hill deposit. Only those intercepts averaging 2.0 g/t Au or greater are shown. Assayed lengths approximate true width of deposit. Coordinates represent mid-point of intersection. A "-1" value denotes an unassayed sliver interval only partially assayed for silver.

Table 12-1: Oxide Zone Intercepts

DH-ID	Easting	Northing	Elevation	Length	Au g/t	Ag g/t
PD03	14033.1	17389.8	195.6	117.8	4.28	3
PD04	14054.4	17293.9	186	96	8.15	14
PD05	14124.4	17292.6	191.3	33	3.62	11
PD06	14145.6	17578	179.6	92	5.01	27
PD07	14201.6	17576.8	200.2	38	7.53	19
PD10	14086.3	17485.3	140.6	77	3.50	4
PD11	14229	17633.8	149	32	2.26	5
PD13	14070.7	17589.2	129.5	24	4.41	2
PD15	13979.2	17400.7	138.2	73	3.23	2
PD17	14263.7	17698.1	210.9	8	2.98	20
PD27	14184.7	17844.2	153.4	17	2.84	5
PD29	14255	17827.9	156	27	3.58	12
PD30	14131.3	17486.2	179	48	2.17	-1
PD31	14070	17187	146.4	13	3.40	15
PD35	14190	17630.5	161.5	30	3.13	-1
PD39	14105.5	17578.8	168.8	27	5.09	25
PD43	14079.8	17337.7	177.3	68	3.74	5
PD45	13991.3	17440.9	135.4	108	4.20	6
PD47	14031.3	17438	158.6	124.3	3.99	-1
PD48	14078.3	17542.1	158.1	55.1	2.97	10
PD49	14032.7	17491.6	159.8	92	2.42	-1
PD50	14128.8	17634.8	148.4	63	6.14	30
PD51	14032.3	17541.9	168.2	63	2.01	1
PD55	14225.4	17691.9	182.7	24	2.98	-1
PD58	14081	17441.2	163.8	89	8.27	-1
PD62	14120.4	17332.3	208.3	43	6.49	-1
PD64	14030.8	17338.8	161.1	88	5.96	7
PD66	14039.2	17314.4	149.6	90.5	6.96	-1
PD67	13928	17389.6	161.3	60	2.13	2
PD68	14052.3	17270.8	150.4	72.4	3.37	-1
PD69	13928.2	17438.5	146.8	53	4.25	4
PD70	13925.8	17490.5	167.2	35	3.07	-1
PD71	14179.9	17584.9	176.9	64	7.63	-1

DH-ID	Easting	Northing	Elevation	Length	Au g/t	Ag g/t
PD73	14084.8	17293.5	164.1	57	3.69	-1
PD74	14062.9	17387.4	146.7	81	8.89	-1
PG04	14148	17339.2	209.8	14	3.48	-1
PR01	14073.6	17292.1	215.8	44	4.69	10
PR02	14038.8	17273	220	50	2.83	6
PR03	14028.6	17320.4	219.7	45.29	4.26	2
PR05	14051.7	17390.7	226.9	45.5	10.73	8
PR06	14018.4	17395.5	228	37	7.15	5
PR07	13986	17400.9	216.3	52	4.72	3
PR08	14070.9	17388.9	205	91	2.85	4
PR09	14058.1	17339.4	222.3	48	3.72	3
PR10	14037.7	17441.9	221.1	52	6.29	6
PR100	14014.1	17318.7	185.9	80	2.63	-1
PR101	14070.7	17315.8	197.2	79	5.42	-1
PR102	14225	17745	192.5	43	7.70	72
PR11	14015	17446.8	218.4	51	6.74	5
PR12	14059.6	17438.3	222.4	52	7.00	3
PR13	14038.6	17487.6	218.9	38	4.39	5
PR14	14016.5	17491.7	213.2	45.29	4.85	10
PR15	14166.1	17573.7	215.1	51	10.02	53
PR16	14221.5	17784.5	192.2	48	3.89	19
PR18	14251.2	17690.9	206.6	16	7.87	44
PR19	14084.8	17485.3	188.1	79	2.77	-1
PR20	13979.2	17400.7	181.1	73	3.72	-1
PR21	14113	17396	206.8	59	2.18	-1
PR22	14192.4	17631	176.1	31	2.76	6
PR23	14078.2	17339	194.4	70	3.76	-1
PR29	13858.9	17500.4	173.8	8	2.26	-1
PR32	14094.7	17777.4	162.8	11	2.36	-1
PR33	13889.3	17541.1	165.6	11	2.86	0
PR36	14031	17390	209.2	70	3.76	-1
PR51	13996.6	17245.7	181.7	41	2.59	3
PR53	14030	17415	183.9	119	4.14	-1
PR56	14103.3	17540.3	198.7	53	4.59	-1
PR57	14150.1	17554.3	205.8	44	3.19	-1
PR58	14151.5	17578.4	209.1	50	6.28	-1
PR59	14184.7	17608.8	202.9	34	8.55	-1
PR60	14151.9	17611.6	193.6	54	7.43	-1
PR61	14131.8	17612.8	189.9	34	4.47	-1
PR62	14158.3	17636.9	190	26	7.27	-1
PR63	14101.7	17615.5	186.4	26	2.97	-1

DH-ID	Easting	Northing	Elevation	Length	Au g/t	Ag g/t
PR64	14109.8	17634.9	183.8	29	4.01	-1
PR66	14153.4	17660.6	190.7	31	10.59	-1
PR67	14126.9	17661.1	181.6	48	2.48	-1
PR68	14047.3	17576.4	184	13	2.08	-1
PR70	14054.9	17540.5	191.6	66	2.27	-1
PR71	14104.8	17490.4	205.7	70	3.66	4
PR72	14130.3	17466.9	210.2	48	2.20	-1
PR73	14104.4	17465.1	202.5	61	3.81	-1
PR74	14105.1	17439.8	203.5	60	6.78	2
PR75	14089.7	17385.7	204.4	71	2.57	-1
PR76	14080	17365	202.3	70	3.73	-1
PR80	14004.5	17216.2	189.2	25	3.09	-1
PR83	13977.9	17366.2	185.5	69	2.98	-1
PR84	13980.4	17415.8	191.9	84	4.21	-1
PR85	14054.1	17490.4	180.7	88	2.68	4
PR86	14009.2	17464	176.7	119	3.82	-1
PR87	14030.1	17462.8	172.9	124	3.57	-1
PR88	14055	17466	185.8	102	4.46	-1
PR89	14083	17466.2	182.3	88	3.16	-1
PR90	14004.8	17439.9	176.5	117	2.67	-1
PR91	14054.9	17438.2	184	109	3.95	-1
PR92	14012.6	17419.9	175	131	3.23	-1
PR93	14055	17415	185.5	108	2.37	-1
PR94	14077.6	17415.6	199.9	84	3.67	-1
PR95	14030.3	17388.9	184	100	2.97	-1
PR96	14004.8	17389.3	190.7	97	3.08	-1
PR97	14005.4	17365.2	182.2	84	2.54	-1
PR98	14030.6	17366.5	187.8	89	4.05	-1
PR99	14055.7	17364.4	191	90	6.73	-1

SECTION 13 • SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 SAMPLE PREPARATION AND ANALYSIS

After oven drying, the samples were crushed using a Rhino Jaw Crusher set at <2 mm ensuring that 90% of the crushed material passed through -10 mesh (2 mm) size.

The samples were then split using a Jones riffle splitter to produce two 250 g samples. One sample was sent for analysis, and the second (the duplicate split) was retained at the sample preparation laboratory for future reference. The remaining rejects were stored indoors.

Samples were grouped in batches and sent for analysis to SGS-FILAB in Carcassonne, France. At the SGS-FILAB laboratory, samples were pulverised with a ring mill to 90% less than 150 mesh (106 µm). All samples in the database were fire assayed for gold using a 30 g sample size. The detection limit on this procedure was 0.01 g/t Au.

Following the gold analysis, a 10 g portion of the sample was forwarded to SGS-FILAB, Dijon, France for ICP multi-element analysis. A sample size of 350 mg was then subjected to a 32 element ICP scan. This included the following elements; Ag, As, Sb, Cu, Pb, Zn, Li, Be, Na, Mg, Al, P, K, Ca, Fe, Ti, V, Cr, Mn, Co, Ni, Sr, Y, Zr, Mo, Cd, Sn, Ba, La, W, Bi and Tl. The ICP scan was done on all samples from holes PD-01 to PD-29 and on reverse circulation holes PR-01 to PR-18.

13.2 ASSAY QUALITY CONTROL AND QUALITY ASSURANCE

A quality control/quality assurance (QC/QA) program designed by Lynda Bloom of Analytical Solutions Ltd., Toronto, Canada was implemented in August 1997 at the beginning of the core drilling program. This involved the insertion of blank and standard samples in the sample stream and the preparation of duplicate samples for analysis in a later sample batch. In addition, at the end of each phase of the program SGS-FILAB was instructed to forward pulps to Australian Laboratory Services, of Perth, Australia (ALS) for checking of gold assays. Results of quality control are summarized below.

13.2.1 BLANKS

Blank samples consisting of barren quartzite and afterwards of quartz carbonate material were submitted at a rate of 2% of the total samples primarily to detect any gold contamination that may develop on equipment at the Sappes or SGS-FILAB sample preparation facilities. An analysis of this data concluded that no systematic contamination has occurred at both preparation facilities. Slightly higher blank values were obtained during the final phase of the RC drill programmes, which is likely attributed to the change in blank rock types.

13.2.2 STANDARDS

Two sets of control standards, produced using Perama Hill material, were used. A first series of seven oxide standards was prepared from the 1996 RC program reject material. The grades of these standards ranged from 0.60 g/t to 11.97 g/t Au. A second batch of twelve standards from the 1998 RC samples was prepared for the later in-fill work. Grades here ranged from 0.43 g/t to 5.81 g/t Au. These standards were submitted at the rate of 2 percent of the total samples and used to monitor the accuracy of the laboratory.

Overall the standards used of the course of the 1997 to 1998 work showed good reproducibility. However, results covering the diamond drill campaigns show an approximate low bias of 5 to 10 percent. Overall Eldorado Gold believes that the effect of this analytical bias on the Perama Hill project mineral resource estimate to be minimal. Locally the model may be slightly conservative relative to gold grades. The true magnitude of this bias on the resource model will be addressed by planned pre-production drilling programs.

13.2.3 DUPLICATES

Duplicates were submitted at rate of 10% of the total samples (15% crushed duplicate samples). The duplicate samples were renumbered and usually shipped in the following batch to increase the likelihood that a different assaying team performed the analysis. Duplicate assays showed good reproducibility, with the majority (80%) agreeing within ±20%.

13.2.4 UMPIRE ASSAYS

Approximately 5% of the database was submitted to ALS, Perth, Australia for external check analysis. These samples were pulps forwarded by SGS-FILAB in four separate batches from October 1997 through December 1998. Ten percent to 15% commercial standards were also submitted as part of this program. ALS umpire assays confirmed that SGS-FILAB assays had a minor negative bias as discussed in the section on Standards.

13.2.5 RE-ASSAYS

A program of re-assay to correct sample mix-ups and other laboratory errors on samples recommended by Analytical Solutions Ltd. was conducted. The last results were received by mid-December 1998 and corrections were made to the database prior to the resource estimation.

13.2.6 QC/QA CONCLUSIONS

Eldorado Gold concludes that all the assay values that support the Perama Hill Mineral Resource estimates were in control. The effect of the low bias observed during parts of the diamond drill campaign is believed to have minimal impact on the Mineral Resource estimates. In places, the gold grade estimate may be slightly understated.

13.3 DENSITY MEASUREMENTS

The density data file has 760 measurements, which includes 15 wet densities and 33 duplicates. Two hundred and thirty three of the measurements were first coated by paraffin wax before measuring by immersion. TGM compared the results from both methods carried out at the sample preparation facility in Sappes with external duplicate cross-checks at the National Technical University of Athens (NTUA). TGM found that the volumetric density data correlated very well with NTUA results. TGM concluded that the density results were satisfactory for resource and reserve estimation work. Eldorado Gold concurs with this conclusion.

Eldorado Gold concurs with this conclusion. Density data for the main rock type at the Perama Hill deposit are shown in Table 13-1.

Table 13-1: Density Data

Rock Type Description	Density Tests	Average (g/cm^3)	Minimum (g/cm^3)	Maximum (g/cm^3)
Oxide Mineralization	258	2.21	1.42	2.74
Oxide Waste	99	1.95	1.43	2.54
Sulphide Waste	319	2.29	1.58	2.96
Mesozoic Metavolcanics	36	2.51	1.81	2.69

SECTION 14 • DATA VERIFICATION

Scott Wilson carried out a database verification program and found no significant errors. Hard copy assay certificates are available at the TGM Alexandroupolis office for holes PD-01 to PD-29 and Scott Wilson also received digital assay certificates for the SGS-FILAB assays. While at the site in December 2003, Scott Wilson verified a number of high gold values in holes PD-07 and PD-10 in Gemcom® with the assay certificates and found no errors. Scott Wilson also checked 1,027 assays, representing approximately 6% of the assays in Gemcom®, with digital assay certificates and found no errors.

It is Scott Wilson's opinion that the Perama Hill Gemcom® database is valid and suitable for supporting resource estimation work. Having conducted similar reviews, Eldorado Gold concurs with Scott Wilson's opinion.

A twin-hole program, consisting of nine core holes (749 m) drilled adjacent to reverse circulation holes, was implemented to validate reverse circulation sampling. The upper part of core holes PD-03, PD-04, and PD-06 twinned exploration holes PR-05, PR-01, and PR-15, respectively. TGM found that the results helped confirm the exploration RC work, but were only partly useful since the exploration RC holes used 2 m sample intervals and some of the 1 m long core samples had low recoveries. A second phase of six holes was initiated in April 1998. Holes for the second phase were selected on the basis of their good recovery and the presence of a range of grades, lithologies and rock conditions. Table 14-1 lists the distances between the twin program drill holes.

Table 14-1: Twin Hole Program Distance between Holes

Pair	DDH	RC	Depth of Hole (m)	Distance between Holes (m)
1	PD03	PR05	45.5	2.0
2	PD04	PR01	44	2.0
3	PD06	PR15	52	4.0
4	PD10	PR19	105	1.8
5	PD15	PR20	105	1.5
6	PD32	PR21	90	4.8
7	PD35	PR22	105	2.4
8	PD43	PR23	97	2.0
9	PD19	PR24	105	2.1

Generally, there appears to be a good correlation between the twin holes with replication of the position of the assay peaks and their amplitude along the drill hole traces. The overall average gold grade based on the twinned reverse circulation and diamond drill holes are very similar if pairs 3 and 6 are excluded due to the excessive distance between these holes. In Scott Wilson's opinion the overall 2.5% difference is insignificant and Scott Wilson believes that both the core and RC samples should be used to support the resource estimate (Table 14-2).

Eldorado supports the conclusions reached by Scott Wilson.

Table 14-2: Twin Hole Program Results

Twin Hole Pair	DDH Average (g/t Au)	RC Average (g/t Au)	Depth (m)	Differences (%)
1	8.44	10.47	45.5	19.4
2	5.15	4.69	44.0	-8.9
4	3.95	3.41	105.0	-13.0
5	3.40	3.77	105.0	9.8
7	1.84	2.02	105.0	3.9
8	5.03	4.68	97.0	-6.9
9	1.03	1.20	105.0	14.0
Total	**3.58**	**3.67**	**606.5**	**2.5**
Excluded Results				
3	7.32	10.94	52.0	33.0
6	1.19	1.51	90.0	21.0

SECTION 15 • ADJACENT PROPERTIES

There are no properties adjacent to the Perama Hill project site.

SECTION 16 • MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 INTRODUCTION

The testwork results summarized in Sections 16.4 to 16.13, detail the relevant metallurgical testwork programs undertaken from 1993 to 2007 and form the basis for the process design parameters and criteria listed in Table 16-1 and Table 16-2 of Section 16.2. The flowsheets are attached as Appendix D.

16.2 PROCESS DESIGN PARAMETER

The process plant, comprising crushing, milling, CIL, elution and INCO SO_2/Air cyanide detoxification is designed to treat 1,250,000 t/a of the oxidised ore, at an availability of 90% over a eight year mine life. The crushing plant operates for 16 h/d 7 d/wk. The remainder of the plant operates for 24 h/d 7 d/wk.

16.3 PROCESS DESIGN CRITERIA

The process design criteria summary for the Perama Hill process is shown in Table 16-1. The design parameters selected have been based on metallurgical testwork results which are detailed in subsequent pages of this section.

Note that subsequent changes to the open pit design and mineral reserve have caused a slight reduction in gold and silver grades and doré production in comparison with the original design criteria.

Table 16-1: Process Design Criteria

	Unit	Amount
Availability / Utilization		
Annual Processing Rate	t/y	1,250,000
Daily Processing Rate	t/calendar day	3,400
Crusher Plant Operating Time	%	51.7
Crusher Processing Rate	t/operating hour	276
Grinding Plant Operating Time	%	90
Grinding Processing Rate	t/operating hour	159
Ore Properties		
Head Grade (LOM Average)	g	
Gold	g/t	3.7
Silver	g/t	8.3
Bond Abrasion Index	-	0.05 – 0.65

	Unit	Amount
UCS Index	MPa	18 – 44
Ball Mill Work Index	kWh/t	14.8 – 19.3
SG	t/m^3	2.68
Precious Metal Recoveries		
Gold	%	90
Silver	%	60
Annual Doré Production		
Gold in Doré	kg/a	4,162
	oz/a	133,800
Silver in Doré	kg/a	6,225
	oz/a	200,100

16.4 METALLURGICAL TESTWORK

The following is a chronological summary of the testwork reports reviewed in order to derive the process design criteria required for the Perama Hill treatment facilities. The list also includes pre-feasibility reports, as well as review reports, issued since 1998:

- 1998; testwork was carried out by BRGM. This included mineralogy, work index determination and cyanidation tests
- 1998; Pre-Feasibility study of the Perama Hill Gold Ore Processing: GBM Limited (July, 1998)
- 1999; testwork by:
  - CSMA Minerals Ltd., UK; Laboratory Testworks (sic) on Samples from Perama Hill Gold Deposit (Jul. 1999)
  - Perama Hill Gold Project Ore Characterization and Grinding Circuit Design – Orway Mineral Consultants
  - INCO: Perama Hill Project – Laboratory Evaluation of the INCO SO_2 / air Cyanide Destruction Process for Slurry treatment (Mar. 1999)
  - 1999; Summary Report of Metallurgical Testwork undertaken by CSMA Minerals, Orway Mineral Consultants and INCO technical Services – Andrew Sarosi
  - 1999; pre-approval of Site Location Study (PASL) by Enveco Echmes based on the GBM pre-feasibility study and the 1999 testwork
- 2000; optimization study testwork by:
  - The Recovery of Gold & Silver from Perama Hill Project Samples: Lakefield Research Limited (Canada) (Sep. 2000 & Nov, 2000) Includes Grindability, flotation, diagnostic

leach, Cyanidation & CIL, rheology, settling tests, carbon isotherms, environmental tests (ABA, NAG and TCLP)

- Test Report Outukumpu Mintec (Apr. 2000 & Oct.r 2000). High Rate thickening tests
- INCO. Laboratory evaluation of the INCO method (SO_2 + air) applied for the destruction of the CN^- ions in the Tailings (Jul. 2000 & Dec. 2000)
- Optimization Study of the Perama Project by Kvaerner (name changed to Aker Solutions) (Jul. 2000)
- Dec. 2000; Optimization Study – Second Edition by Kvaerner (name changed to Aker Solutions)

- 2007; copper analyses and copper distribution evaluation by Roscoe Postle.

16.5 TESTWORK PROGRAM COMPONENTS

The following testwork components were selected from the historical studies and were used in the design of the flowsheet of the Perama Hill plant. The main processes are the following:

- Head analysis and specific gravity (SG) determinations
- Mineralogical examination
- Comminution testwork
- Thickening – static tests
- Cyanide leaching
- Pressure and vacuum filtration test; pulp viscosity.

16.6 HEAD ANALYSIS AND SPECIFIC GRAVITY DETERMINATIONS

OMC received five samples, one for each ore type. Of these ore Types 2 and 5 together comprised 74% of the orebody with the sample for Type 2 being the hardest of the five and that for Type 5 the second softest. The sample of ore Type 2 comprised sections taken from a single hole. Hardness indicators showed that there was some variation in hardness within the hole.

Table 16-2 outlines the ore types as indicated in the table above. As Types 2 and 5 represent the largest portion of the ore, they were subjected to most of the tests.

Table 16-2: Ore Types and Distribution

Ore Type	Description	In Deposit		
		Grade Gold (g/t)	Weight (%)	Gold Distribution
1 (soft)	Highly weathered sandstone	10.0	7	19
2 (Hard)	Hard silicified sandstone	5.8	28	43
3 (Hard)	Breccia/sandstone contact	8.5	8	16
4 (Soft)	Breccia	1.5	11	5
5 (Soft)	Silicified sandstone	1.5	46	17
	Total	**3.9**	**100**	**100**

Samples of the five ore types identified for testwork were analysed to establish head grade values. The analyses are summarized in Table 16-3.

Table 16-3: Prefeasibility Testwork Head Grades

Sample	Gold (g/t)	Silver (g/t)	Copper (%)	Iron (%)	Sulphur (%)	Lead (%)	Zinc (%)
Composites of Ore Types 2 and 5:							
Composite 2	7.09	8.93	0.0164	1.63	0.36	0.006	0.086
Composite 5	3.57	2.42	0.0041	2.08	0.13	0.008	0.076
Ore type leach variability samples:							
OT 1	9.88	1.95	-	-	-	-	-
OT 2	7.45	17.6	-	-	-	-	-
OT 3	8.28	38.5	-	-	-	-	-
OT 4	3.61	29.1	-	-	-	-	-
OT 5	3.72	1.97	-	-	-	-	-
Column Leach Testwork Samples:							
CL 1	9.66	<0.5	-	-	-	-	-
CL 2	5.33	8.93	-	-	-	-	-
CL 4	2.87	5.88	-	-	-	-	-
CL 5	1.98	2.23	-	-	-	-	-
INCO Detox Samples:							
ID 1	25	4.6	-	-	-	-	-
ID 2	8.7	15	-	-	-	-	-
ID 3	23	42	-	-	-	-	-
ID 4	3.5	17	-	-	-	-	-
ID 5	3.4	8.6	-	-	-	-	-

Elemental analysis was also completed on the Hard, Soft and High Copper Composites that were analysed by SGS/Lakefield. The results are shown in Table 16-4.

Table 16-4: Elemental Analysis Results

		Testwork Sample		
		Hard	**Soft**	**High Cu**
Au	g/t	5.49	3.25	7.66
Ag	g/t	14.5	2.3	54.5
Cu	%	0.013	0.004	0.70
Cu (NaCN soluble)	%	0.004	<0.001	0.15
Zn	%	0.011	0.005	0.001
Ni	%	<0.001	<0.001	<0.001
Co	%	<0.02	0.02	<0.02
As	%	0.023	0.034	0.037
Sb	%	<0.001	<0.001	<0.001
Hg	ppm	4.2	4.7	6.7
S	%	0.48	0.15	0.34
Semi-quantitative scan:				
Be	ppm	<2	<2	-
Ca	ppm	340	360	-
Cd	ppm	<5	<5	-
Co	ppm	<20	<20	-
Cr	ppm	69	89	-
Cu	ppm	110	25	-
Fe	ppm	16000	23000	-
K	ppm	1100	4200	-
La	ppm	<50	<50	-
Mg	ppm	210	830	-
Mn	ppm	32	29	-
Mo	ppm	<10	<10	-
Na	ppm	240	480	-
Ni	ppm	13	6.9	-
P	ppm	210	350	-
Pb	ppm	700	750	-
Se	ppm	<50	<50	-
Te	ppm	<20	<20	-
Y	ppm	<5	<5	-
Zn	ppm	92	44	-

16.7 MINERALOGICAL EXAMINATION

Mineralogical investigations were carried out by both BRGM and CSMA Minerals on the ore samples received by them for testwork. Both groups found that the gold was present as sub-microscopic particles ranging from 0.2 µm to 2 µm. No coarse free gold was found in any samples.

The principal gangue minerals present were quartz (30% to 70%), barite (10% to 20%) and iron oxides (15% to 40%). Clay minerals were noted up to 15% in two of the minor ore types. Trace minerals noted included pyrite, rutile, zircon, calcite, galena, tourmaline, chalcopyrite and in one minor ore type, bismuthinite.

The oxide ore zone of the Perama Hill deposit is predominantly a silicified sandstone of varying hardness, which has both breccia and conglomerate components. This material has been classified into five principal ore types for which the grade and weighting have been calculated for the reserves.

16.8 COMMINUTION TESTWORK

16.8.1 PRE-FEASIBILITY TESTWORK

OMC performed the supervision and the interpretation of extensive comminution testwork on the various ore-type samples generated for the detailed testwork programme. The tests were conducted by Amdel Mineral Services Laboratory in Australia. This work is reported in Perama Hill Gold Project Ore Characterization and Grinding circuit Design – Orway Mineral Consultants and is summarized in Summary Report of Metallurgical Testwork undertaken by CSMA Minerals, Orway Mineral Consultants and INCO Technical Services – Andrew Sarosi.

A full grind characterization was carried out on samples Types 2 and 5 to determine the following:

- SAG mill sizing SAG mill Impact Work Index (IWi) by size fraction and JKTech drop weight tests
- Rod mill work index (RWi)
- Ball mill work index (BWi) at 106 and 150 µm screen size
- Abrasion index (Ai)
- UCS.

Work on samples of ore Types 1, 3, and 4 were limited to abrasion indices, RWi and BWi at 150 µm.

Table 16-5 shows the OMC's results.

Table 16-5: OMC Comminution Characterization Results

Parameter	Units	Samples of Ore Types				
		2	5	1	3	4
UCS						
Average	MPa	18	44	-	-	-
Range	MPa	11-24	4 – 123	-	-	-
Mode of Failure		Shear	Axial/Shear	-	-	-
Abrasion Index – Ai		0.648	0.120	0.045	0.165	0.048
Rod Mill Work Index						
P_{80}	µm	812	703	528	823	846
Wi	kWh/t	19.1	10.2	6.5	15.0	13.1
Ball Mill Work Index – 150 Closing Screen						
P_{80}	µm	119	116	118	113	105
Wi	kWh/t	17.8	12.1	12.3	17.3	13.9
RWi:BWi – Coarse		1.07	0.84	0.53	0.87	0.94
Ball Mill Work Index – 106 Closing Screen						
P_{80}	µm	80	76	-	-	-
Wi	kWh/t	18.3	12.6	-	-	-
RWi:BWi – Fine		1.04	0.81	-	-	-
Crush Impact Work Index*						
19 x 25 mm	kWh/t	9.8	6.5	-	-	-
25 x 38 mm	kWh/t	11.7	5.9	-	-	-
38 x 51 mm	kWh/t	16.9	6.7	-	-	-
51 x 76 mm	kWh/t	26.0	8.1	-	-	-
76 x 102 mm	kWh/t	37.1	16.8	-	-	-
SG		2.68	2.66	-	-	-
Drop Weight Results						
A		62.8	65.5	-	-	-
b		0.94	5.37	-	-	-
t_a		0.78	3.26	-	-	-

Note: * The values for the impact data are those of the 85^{th} percentile specimens

Bond ball mill work indices were determined on each of the five selected hard ore samples and the hard and soft composites. Summarized Table 16-6 shows a summary of the results.

Table 16-6: Bond Work Indices

Sample	Bond Ball Mill Wi kWh/t at 106 µm
Hard Composite	19.3
Soft Composite	14.8
Selected hard core:	
PD14	20.0
PD49	16.7
PD60	20.7
PD64	18.3
PD68	19.5
For comparison, using values based on OMC samples:	
Hard ore	18.2
Soft ore	12.8

Based on the above results, for mill selection evaluation, the following Bond work index values were used for the Optimization Study.

- Hard Ore 20 kWh/t
- Soft Ore 14.8 kWh/t

A Comminution circuit comprising SAG and Ball Milling was envisaged in the work performed by OMC. However, in view of the relatively low tonnage throughput and the process risk caused by the very wide range of ore hardness and competency in the orebody, during the OptimizationOptimization Studies of 2000 Aker Solutions were requested to incorporate conventional three stage crushing and single stage ball milling

16.9 LEACHING TESTWORK

16.9.1 DIAGNOSTIC LEACH

A diagnostic leach was done by SGS/Lakefield Research to improve understanding of the reason for the consistent loss of about 10% of the gold in cyanidation.

Previous mineralogical investigations had failed to identify the mode of occurrence of this lost gold. The diagnostic leach was performed at 80% -75 µm and comprised:

- Cyanidation at 1,000 ppm NaCN
- Aqua Regia leach on the cyanidation tails.

Summarized Table 16-7 shows a summary of the results.

Table 16-7: Diagnostic Leach Results

	% Distribution Au	
	Soft Ore	**Hard Ore**
Initial Cyanidation	90.7	85.9
Aqua Regia extracted	7.0	11.1
Aqua Regia Residue	2.2	3.0
Feed	100.0	100.0

The bulk of the gold in the cyanidation residue is encapsulated in, or coated with, acid soluble material such as iron oxides or sulphides.

16.9.2 PRE-FEASIBILITY TESTWORK

The preliminary leaching testwork programme was carried out by BRGM on samples of drill core representing the different ore types. Work focussed on gold recovery in cyanidation bottle roll tests at different grind sizes, various residence times and a study on the effect of using sea water in place of fresh water.

This work demonstrated very rapid leaching for all ore types with recovery of over 95% of the recoverable gold being achieved in four to six hours. Gold recoveries of around 90% were achieved on all ore types using carbon-in-leach (CIL). Sea water had little effect on gold recovery.

CSMA also undertook metallurgical testwork. A number of tests were completed at various grind sizes to determine the effect on recovery. P_{80}'s of 53, 63, 75, 106, 120, and 150 μm were tested. Lower recoveries were noted at the coarser grind sizes and all subsequent testwork was completed at a P_{80} of 75 μm.

CIL performance was also evaluated against CIP recovery levels. CIL showed a slightly higher recovery of around 1%.

Leach kinetics were tested by leaching composites for periods of up to 36 hours at 45% solids. The test data indicated very rapid leaching with maximum dissolution achieved in the first 8 hours. Recoveries of 95% of recoverable gold and 60% of the silver were reported in the same leaching time.

A series of tests were carried out on composites Types 2 and 5 with cyanide concentrations of 0.075%, 0.050% and 0.025% NaCN. These were compared to the results previously obtained for 0.1% NaCN. Generally, lowering the cyanide strength resulted in lower recoveries. However, increasing the cyanide strength was questionable in terms of economic justification. Therefore, a cyanide solution strength of 0.075% was used in subsequent tests.

16.9.3 OPTIMIZATION STUDY TESTWORK

The main objectives of the Optimization Study cyanidation testwork were to confirm the following:

- reagent consumptions and in particular to perform tests with much lower residual cyanide levels
- cyanidation at 50% solids instead of the 45% solids used previously
- need for, and residence time of pre-aeration
- CIL residence time required
- impact of copper and mercury.

The testwork included the use of oxygen in the leach throughout. All tests were done using the brackish water from Perama borehole W2R as it is established that this water would comprise the principal source of makeup water for the project.

Leach tests were performed on the hard, soft and blend composites at varying initial cyanide concentrations. In all tests, the cyanide concentration was allowed to decrease naturally in the later stage of the leach towards a target residual cyanide concentration of 200 ppm NaCN.

All tests were performed with a grind of 80% -75 µm, with lime added to the mill to reach a target pH of 10.0. In all tests, oxygen was used to give a target dissolved oxygen concentration of 10 ppm O_2. The cyanidation results are summarized in Table 16-8.

Reduction of the initial cyanide concentration from 700 ppm to 300 ppm NaCN in leach tests without carbon decreased the gold extraction after 8 hours by 5% to 8%, but had no effect on the extraction after 24 hours on either hard or soft ores. The hard ore does consume more cyanide than the soft ore; this is possibly due to the higher copper and sulphur grades in the hard ore.

It is concluded that the cyanide consumption in the CIL circuit should be about 0.45 kg/t to 0.50 kg/t on a blend. To obtain the cyanide addition rate, which with a cyanide detoxification plant in the circuit corresponds to the overall cyanide consumption, the residual cyanide must be added to the consumption in the CIL circuit when cyanide is not recovered from the CIL tail.

For this study, residual cyanide of 100 ppm NaCN has been assumed for operating costs as a result of these tests and an initial cyanide concentration of 300 ppm NaCN. The cyanide addition rate corresponding to a consumption of 0.50 kg/t and a residual of 100 ppm is 0.60 kg/t NaCN at a pulp density of 50%. If high copper ores were processed, this consumption would increase.

The lime consumption was reasonably consistent in the tests without pre-aeration at about 1.2 kg/t. For operating costs, a consumption of 1.5 kg CaO/t has been selected.

Subsequent testwork confirmed the rapid kinetics found in the pre-feasibility work.

The recoveries after 24 hours at 300 ppm NaCN initial cyanide concentration are shown in Table 16-8.

Table 16-8: 24 Hour Leach Recoveries at 300 ppm NaCN initial Concentration, Grind Size P_{80} passing 75 µm and 50% Pulp Density

	Extraction %	
	Au	**Ag**
Hard Ore Leach	85.0	63.4
Soft Ore Leach	90.1	76.4
Calculated Blend Leach	88.1	71.2
Measured Blend Leach	88.5	63.5
Blend CIL Average	90.3	62.6

For gold, the leach extraction from the blend test agreed with the figure calculated from the extractions on hard and soft ores and the 40% hard ore blend composition. Based on the blend CIL test results, the project design criteria of 90% Au and 60% Ag extraction is reasonable.

16.10 OPTIMIZATION STUDY CARBON ADSORPTION TESTWORK

To provide data to assist in estimating the design carbon loading value for periods with high ratios of silver to gold, a carbon adsorption isotherm test was conducted. This test was designed to determine the effect of silver at the planned highest grades on carbon adsorption.

The isotherm results showed the following:

- As carbon loading increases, the Ag/Au ratio on carbon decreases below the ratio in the leach pregnant solution, showing that silver is being displaced from the carbon by gold. The selection of the maximum carbon loading is therefore determined by the economics of elution circuit size and the value of silver loss.
- An evaluation was done that indicated that the 7 tonne carbon batch size selected for the Optimization Study should be retained, corresponding to a maximum carbon loading on high grade feeds of 8,000 g/t Au+Ag.
- The copper grade on carbon decreases rapidly as the gold and silver carbon loading increases up to 6,000 to 7,000 g/t Au + Ag. At very low carbon loadings, over 50% of the cyanide soluble copper adsorbed, giving over 50% Cu in the total metal on the carbon. At a carbon loading of 7,000 g/t Au+Ag, this had dropped to 10% Cu and only 5% of the cyanide soluble copper adsorbed.
- Approximately 10% of the 4 g/t mercury in the ore dissolved in the leach. The mercury loading on carbon increased as the gold and silver adsorption increased. At a carbon loading of 7,000 g/t Au+Ag, there was 117 g/t Hg on the carbon, representing about 7% of the mercury in the ore.

16.11 SETTLING TESTS

16.11.1 OPTIMIZATION STUDY TESTWORK

Settling tests were completed by both Outotec and SGS/Lakefield. High Rate tests were completed by Outotec on hard ore, soft ore and a blend of 60% hard and 40% soft. A subsequent phase of tests was completed on tailings blends comprising soft ore at 100%, 85% and 70%. SGS/Lakefield completed tests on conventional settling on 100% hard and 100% soft ore. Percol or Magnafloc 156 flocculant was used throughout following a series of flocculant screening tests.

The High Rate thickening tests concluded that to obtain a 50% to 55% solids underflow on ore blends containing at least 15% hard ore, a high rate thickener diameter of 21 m and a flocculant dosage of 35 g/t should be used for design for a daily ore throughput of 3,400 tonne.

The conventional tests concluded that a thickener diameter of 40 m would be required at similar flocculant dosages. If the flocculant dosage was decreased to 10 g/t the thickener unit loading rate would decrease and a larger thickener diameter of 55 m would be required.

A 21 m high rate thickener should be selected at a flocculant addition rate of 35 g/t Magnafloc 155. Underflow pulp density should vary from 50% to 55% solids and flocculant addition rate from 17 to 35 g/t depending on the ore blend being processed and thickener solids feed rate.

Overflow is likely to be cloudy and should not be used for applications where the presence of fine solids will be a problem.

16.12 CYANIDE DETOXIFICATION TESTS

16.12.1 PRE-FEASIBILITY TESTWORK

INCO evaluated the SO_2/Air process for the destruction of cyanide in the leach tailings from Perama Hill. The first phase of the INCO work was to produce representative tailings slurry for the cyanide destruction tests. High cyanide consumption levels were noted in some early tests and INCO recommended an extended pre-aeration period to convert the iron (II) to iron (III) and precipitate it out as ferric hydroxide prior to contact with cyanide.

INCO established that for cyanide destruction a single stage continuous flow treatment was adequate with a laboratory contact time of 60 minutes. An SO_2 dosage of about 3.5 g/g CN_{WAD} achieved good removal of cyanide and residual treatment. Because there was sufficient copper present in the cyanidation tails the process did not require copper addition. With fresh water a residual CN_{WAD} level of 0.2 ppm was achieved, but in only one test and on ageing for 27 days this dropped further to 0.1 ppm.

16.12.2 OPTIMIZATION STUDY TESTWORK

INCO completed another round of detoxification tests using the residue from the SGS/Lakefield CIL tests. Seventeen tests were completed during 2000 evaluating detoxification performance on cyanide tailings slurries generated with and without pre-aeration.

The tests indicated that a level of <1 ppm CN_{TOT} in solution for freshly treated residues is achievable and that adequate pre-aeration can prevent the re-dissolution of Fe and consequent increase in CN_{TOT} with ageing. In practice, there may be sufficient pre-aeration within the grinding circuit to prevent the re-dissolution of Fe. However, a dedicated pre-aeration tank has been provided.

The INCO product from the reactor should contain:

- CN_{TOT} <1.0 mg/L
- CN_{WAD} <0.5 mg/L
- SCN^- approx 10 mg/L
- OCN^- approx 240 mg/L
- $NH_3 + NH_4^+$ approx 10 mg/L

It should be noted that these are single pass solutions. In practise, where process water (or tailings water) is recycled, CN_{total}, SCN^- and OCN^- are expected to rise.

Copper sulphate addition is required to be added to the process. A mixing and dosing facility to dose to nominally 50 mg/L Cu^{2+} in solution is provided for pH control. The testwork used HCl equivalent to about 460 kg/d in the plant. For the Optimization Study it was assumed that there would be sufficient spent acid from the carbon acid wash to avoid the need for further acid addition. However, this cannot be guaranteed so in this study a sulphuric acid dosing facility has been provided.

For the Optimization Study, oxygen was initially specified for the Detox reactor instead of air. Oxygen was used instead of air only in a single test with inconclusive results. For this study air has been specified for the Detox to eliminate the shipment requirements for oxygen to site.

All the INCO testwork reports were provided to CyPlus, who are now the owners of the INCO technology, for them to establish the detoxification plant design criteria from the testwork and to specify the critical equipment. CyPlus have specified a single stage reactor with a residence time of 60 minutes at pH8. Two reactors in parallel were specified to satisfy the ETR requirement for double capacity. The intention was that, under normal operation with both reactors in operation, the effluent will be <1 mg/L CN_{WAD} whereas if only one reactor is in operation the effluent will be <10 mg/L CN_{WAD}

16.13 ENVIRONMENTAL TESTS ON INCO RESIDUES

The solids from the treated slurries from the July 2000 INCO tests were submitted to SGS/Lakefield for Toxicity Characteristic Leaching Procedure (TCLP), Acid-Base Accounting (ABA) and Net Acid Generation (NAG) analyses. These tests all indicate that while there is total sulphur present it occurs exclusively as sulphate. The sulphide S was less than the detection limit and as it is the presence of sulphide that will generate acid upon oxidation it was confirmed that the material has no potential to generate acid.

16.14 PROCESS DESCRIPTION

16.14.1 SUMMARY

The Perama Hill process plant will be designed to treat 1,250,000 t/a of conventional oxide ore cap containing gold and silver bearing ore at a treatment rate of 3,400 tonnes per calendar day at an overall plant availability of 90%. The processing facilities include three stage crushing, followed by a single stage ball mill with a classification step to produce 80% passing 75 µm product size.

Milled ore will be thickened in a high rate thickener prior to pre-aeration and gold/silver dissolution in the CIL circuit. Hydrated lime will be added in the milling circuit. Precious metal dissolution will be by cyanide. Oxygen will be injected into the pre-aeration tank and leaching tanks to enhance the leach kinetics. The CIL circuit will consist of six tanks with a total retention time of 20 hours.

The CIL tailings will be detoxified to remove any remaining cyanide by the INCO SO_2 Process. The Environmental Terms of Reference (ETR) requires a CN_{WAD} of < 10 mg/L. Studies indicate a target concentration of <1 mg/L CN_{WAD} should be attainable. Detoxified tailings will be thickened, pressure filtered and dry-stacked.

Carbon will be extracted from the CIL circuit and acid treated for calcium removal prior to elution on a daily basis. The gold and silver will be recovered by a split Anglo American Research Laboratories (AARL) elution system. Carbon will be reactivated and returned to the last stage CIL tank. Pregnant solution will be fed through the electrowinning circuit to recover the gold and silver metals. The resulting electrowinning sludge will be washed, filtered, dried and smelted to produce Doré metal. The barren solution will be pumped to the leaching circuit.

16.14.2 CRUSHING PLANT

The crushing plant will be designed to treat ore at a rate of 350 t/h to a product size of 100% passing 150 mm through a jaw crusher for a period of 16 h/d.

The ROM bin, with a capacity of 150 tonnes, will be at least one and a half times the truck capacity. The ROM bin will be equipped with a static grizzly and a permanently installed hydraulic rock breaker. Material will be discharged from the bin by a vibrating grizzly feeder. Minus 100 mm material scalped out by the grizzly will discharge onto the sacrificial collection conveyor. Oversize will pass to the primary crusher.

The primary jaw crusher, at a closed side setting of 150 mm will produce a product with a P_{80} of less than 150 mm. The product size will be finer when treating the softer material. Crushed product will combine with scalped grizzly undersize material on the sacrificial conveyor for transfer via the main conveyor to the crushing plant.

The crushing plant will be a conventional two stage crushing circuit comprising an open circuit secondary screen plus cone crusher followed by a closed circuit tertiary screen and cone crusher. The final product will be the tertiary screen undersize with a P_{80} of minus 8 mm. To ensure optimum choke feeding for the crushers feed surge bins of 15 minute holding capacity plus vibrating pan feeders will be provided at each crushing stage.

The crushed ore bin will have a live capacity of approximately 4,000 tonnes. The storage bin will be furnished with two belt feeders discharging onto the mill feed conveyor.

Dust extraction will be installed as required to comply with the ETR limit for dust emissions of 30 mg/Nm^3. Dust capture will be with cyclones and bag filters. In addition, dust suppression by water sprays will be employed at the discharge point into the fine ore silo.

16.14.3 MILLING

The milling circuit will consist of a single ball mill. A preliminary mill size of 4.88 m diameter x 7.62 m EGL is proposed powered by a 3.5 MW motor. The range of work Indices is high due to the variable nature of the ore. The hard material has a Work Index of 20 kWh/t and the soft material has a Work Index of 15 kWh/t. To handle this variation a variable speed drive is proposed that will allow the mill to operate between 60% to 80% of its critical speed, Nc. Because of the increase in ore hardness as mining progresses the ball charge loading will increase from 27% v/v in the early years to 35% v/v for the harder ore in later years.

Blending of ore feed may be required to ensure consistent mill throughput and grind size. Ore from different mine locations will be combined in a controlled manner ahead of the primary crusher and ROM stockpile. Equally, there is potential for an increase in feed throughput if the ore is slightly softer.

16.14.4 THICKENING AND TRASH SCREEN

A 21 m high rate thickener will thicken the mill discharge prior to CIL. The installation of a thickener between milling and leaching will enable the pulp density within the milling circuit to be optimised for milling efficiency and the pulp density of the CIL to be maximised to reduce reagent costs and minimise tankage volume for the required residence time.

A vibrating trash screen rather than a linear screen will be installed ahead of the thickener. The screen selection will minimise capital cost. As the mine will be open pit the amount of wood fibre should be minimal with no requirement for a specialist linear screen.

16.14.5 CYANIDATION CIRCUIT

The CIL circuit residence time will be 20 hours. It will consist of 6 CIL tanks with a single pre-aeration tank of the same volume as the leach tanks. The carbon concentration will be 10 g/L to 15 g/L with a projected carbon loading of up to 8,000 g/t Au + Ag.

Oxygen will be injected into the slurry to enhance the leach kinetics. Oxygen injection will be into the pre-aeration tank and into any number of CIL Tanks 1 to 4. Injection will be via a Slamjet sparger.

Carbon will be advanced along the tank train for 20 h/d using vertical centrifugal pumps. Loaded carbon pumping to elution will operate for 10 h/d.

The flow of pulp between CIL tanks will be by Kemix pumped type MPS (P) interstage screens. These eliminate hydraulic drops between tanks allowing all tanks to be mounted at the same elevation. The tanks will be connected by launders, which will include bypass arrangements so that any single tank can be isolated for maintenance.

Under normal operations one batch of carbon will be removed per day. During mining peak ore grades, up to two carbon batches per day will be needed and the carbon concentration in the leach slurry can be increased to 20 g/L.

16.14.6 ELUTION AND PRECIOUS METAL RECOVERY

The fully automated elution circuit will treat a 7 tonne carbon batch size with a combined gold and silver loading of 8,000 g/t. This selection will be consistent with a maximum of two elutions per day, 6 d/wk while mining peak ore grades during Years 2 and 3 and up to one elution per day otherwise.

A mercury retort will be available as there is a possible presence of up to 120 g/t mercury on the final loaded carbon.

A split AARL elution system was specified for the Optimization Study by TGM. A conventional dilute acid wash stage will precede elution to remove calcium and magnesium.

The makeup water quality to the project will vary from good quality rain water collected in the Saddle Dams to poor quality high chloride water from borehole W2R. The elution circuit will require a reliable supply of low chloride water which will be produced by a water treatment plant capable of treating the W2R water.

The elution column will be operated at a normal stripping temperature of 115 to 120°C. However, to facilitate the elution of silver during periods of high silver to gold ratios the circuit will be designed to operate at up to 130°C. Maximum stripping times (including column filling and emptying) will be in the order of 8 hours.

A Reverse Osmosis (RO) based water treatment plant is being considered to minimise dissolved solids and especially chlorides originating from W2R borehole water entering both the strip solution and contaminating the surface of the eluted carbon.

The elution circuit will regenerate all of the activated carbon from each stripping cycle. The regeneration kiln will be an LPG fired rotary kiln with a variable speed dewatering screw feeder and a heat recovery arrangement that pre-heats the carbon with off gas. The 440 kg/h carbon kiln will treat up to 67 t/wk of carbon and a 1.5 Bed Volume hopper will be provided ahead of the kiln for surge capacity. An eluted carbon by-pass will be operated in emergency only.

Loaded carbon will be pumped from the CIL via the loaded carbon screen into a loaded carbon tank adjacent to the CIL circuit. From there the carbon will be pumped in batches into the acid wash column.

The loaded carbon tank will enable loaded carbon to be pumped while a previous batch is still being acid washed.

The electrowinning circuit will operate with a recirculating flow from the pregnant solution tank. The last cycle will be in open circuit to return the barren electrolyte to the CIL. This philosophy can be reviewed at the detailed engineering stage.

The hot pregnant solution will be pumped to the head of an electrowinning train consisting nominally of two 150 ft^3 sludging cells in parallel incorporating 60 cathodes each 0.84 m^2.

Filtered precious metal sludge will be dried and calcined in a mercury retort, mixed with crushed (recycled) smelting slag and fluxes and smelted in an induction furnace. Smelting will be performed on 2 d/wk.

The ETR requires that bag filters are installed not only on the furnace off gas but also on the electrowinning cells vent gas. Aker Solutions have replaced the latter with a scrubber because there will be no dust in that stream but there will be traces of ammonia.

A pressurised vent system will be provided for the Goldroom to prevent build-up of any mercury vapour. The retort will be installed within a separate room within the Goldroom with its own extraction fan.

16.14.7 CYANIDE DETOXIFICATION

The INCO SO_2/air process has been specified by the ETR. Testwork has been completed that has shown that this process can readily achieve the ETR specified target of <10 mg/L CN_{WAD} and should also achieve under normal conditions <1 mg/L CN_{WAD}. The ETR also specifies that the capacity of the cyanide destruction unit must be double what is required and must have a back-up unit for lime addition.

The plant will consist of two reactors in parallel with a residence time of 60 minutes. Sulphuric acid will be added for pH adjustment. Spent acid can also be added from the carbon circuit acid wash. Copper sulphate will also be added as required.

A milk of lime addition facility will ensure that pH can be adjusted upwards if required. The reactors will be aerated by dedicated air compressors. CN_{WAD} will be analysed by an online analyser at two points, one on the combined feed tank and one on the combined discharge. Slurry pH will also be measured and used for control of lime and acid addition.

16.15 TAILINGS THICKENING AND FILTRATION

Slurry exiting the final detoxification facility will be gravity fed to the final tails sump, from where it will be pumped to the thickening and filtration section.

The detoxified tailings will be thickened in a tailings thickener, and finally dewatered in pressure filters to produce a low-moisture filtered product. This material will be dry-stacked in a lined tailings dam.

SECTION 17 • MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 MINERAL RESOURCES

The Perama Hill mineral resources were modelled and estimated by Roscoe Postle Associates Inc. (RPA). That work is described in a 2004 Technical Report, titled "Reports on the Perama Hill Gold Deposit Mineral Resource Estimates, Greece." Eldorado Gold has reviewed the work in sufficient detail to assume responsibility for the Perama Hill estimates. Additionally, Eldorado Gold has reclassified the Perama Hill mineral resource (added a component of Measured Mineral Resources). The work was done under the direction of Dr. Stephen Juras, P.Geo.

17.1.1 GEOLOGICAL MODELS

3D shapes were constructed for the Perama Hill oxide mineralization, a small inclusion of sulphide mineralization, the oxide-sulphide surface, and the East Graben Fault. The oxide and sulphide-mineralization wireframe-models were clipped at the topographic and oxide-sulphide surfaces.

Eldorado Gold checked these shapes for interpretational consistency on section and plan, and found them to have been properly constructed.

The geological interpretation is shown on the 215 and 180 Levels in Figure 17-1 and Figure 17-2 and on cross sections in Figure 17-3 to Figure 17-5.

17.1.2 DATA ANALYSIS

The Perama Hill deposits oxide database comprises 8,369 assays in 194 holes. The statistical properties of this data are summarized in Table 17-1. The resource assays average 1.05 m in length, 93% sample recovery, 4.33 g/t Au, (4.20 g/t Au when high assays are cut to 30 g/t Au), 6.46 g/t Ag, 376 ppm As, 9 ppm Sb, 137 ppm Cu, 576 ppm Pb, 57 ppm Zn, and 1,022 ppm Ba.

The multi-element data are not available for all of the assays therefore the multi-element averages in Table 17-1 may not be representative of the global oxide mineralization resource.

Silver assays are available for approximately half of the resource assays, which should be sufficient for a reasonable global silver grade estimate for the oxide mineralization resources. The silver coefficient of variation of 2.6 is higher than the 1.4 value for the gold resource assays interpolated silver grades may be somewhat less reliable than the gold grades locally. Since silver does not contribute significantly to the value of the resource and there are sufficient silver assays available.

Figure 17-1: 180 Level Plan

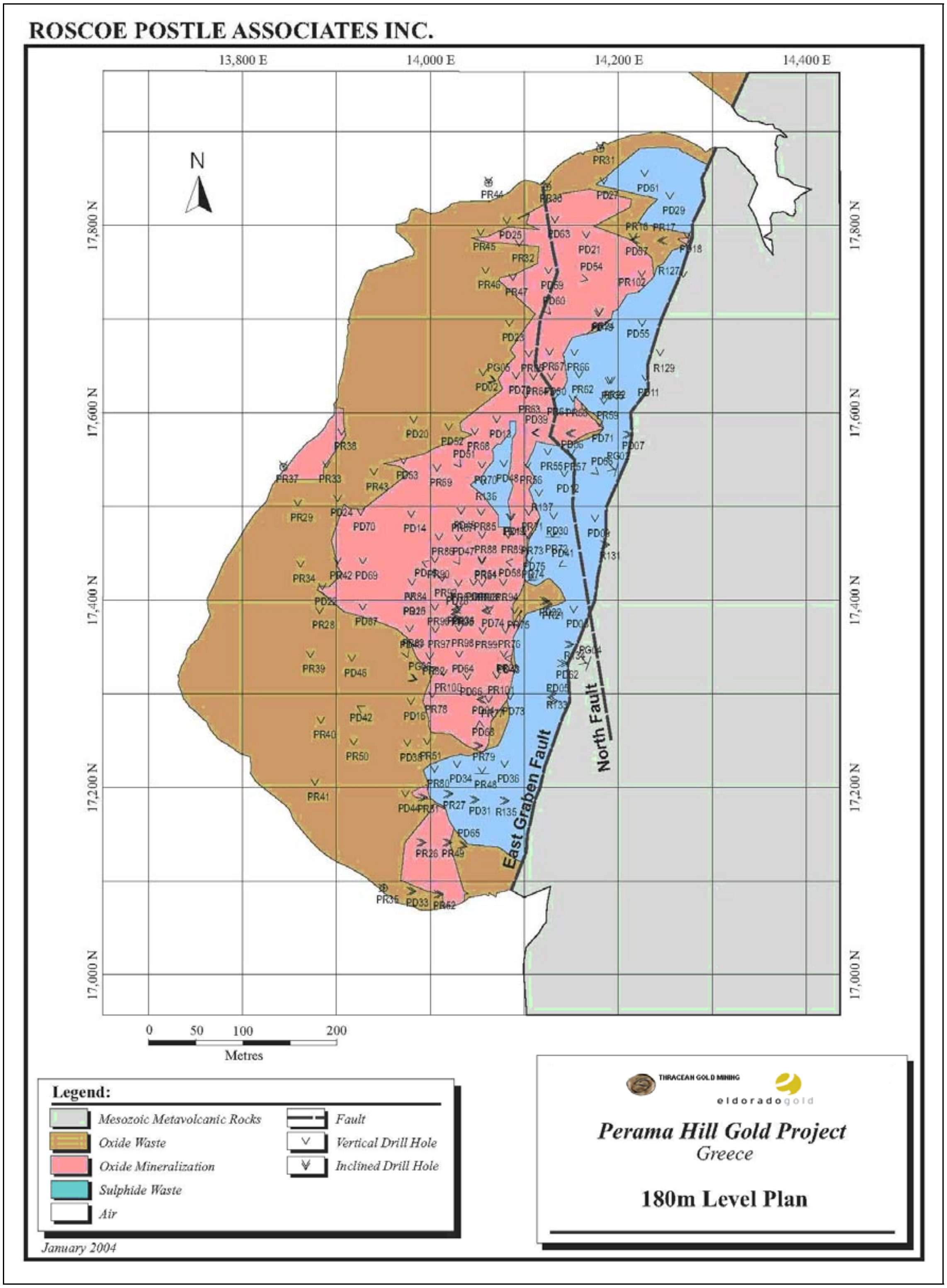


Figure 17-2: Section 17,300N

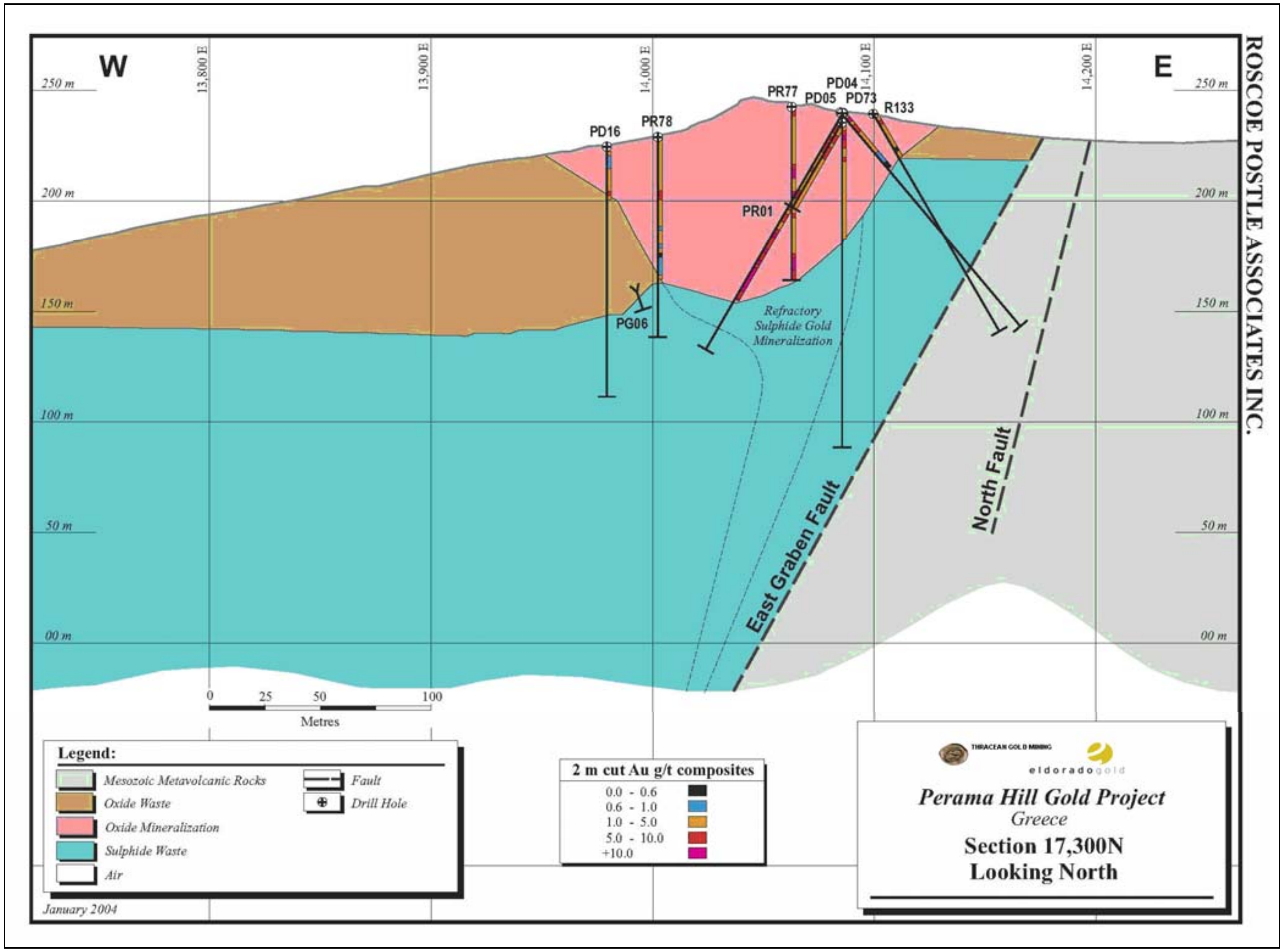


Figure 17-3: Section 17,450N

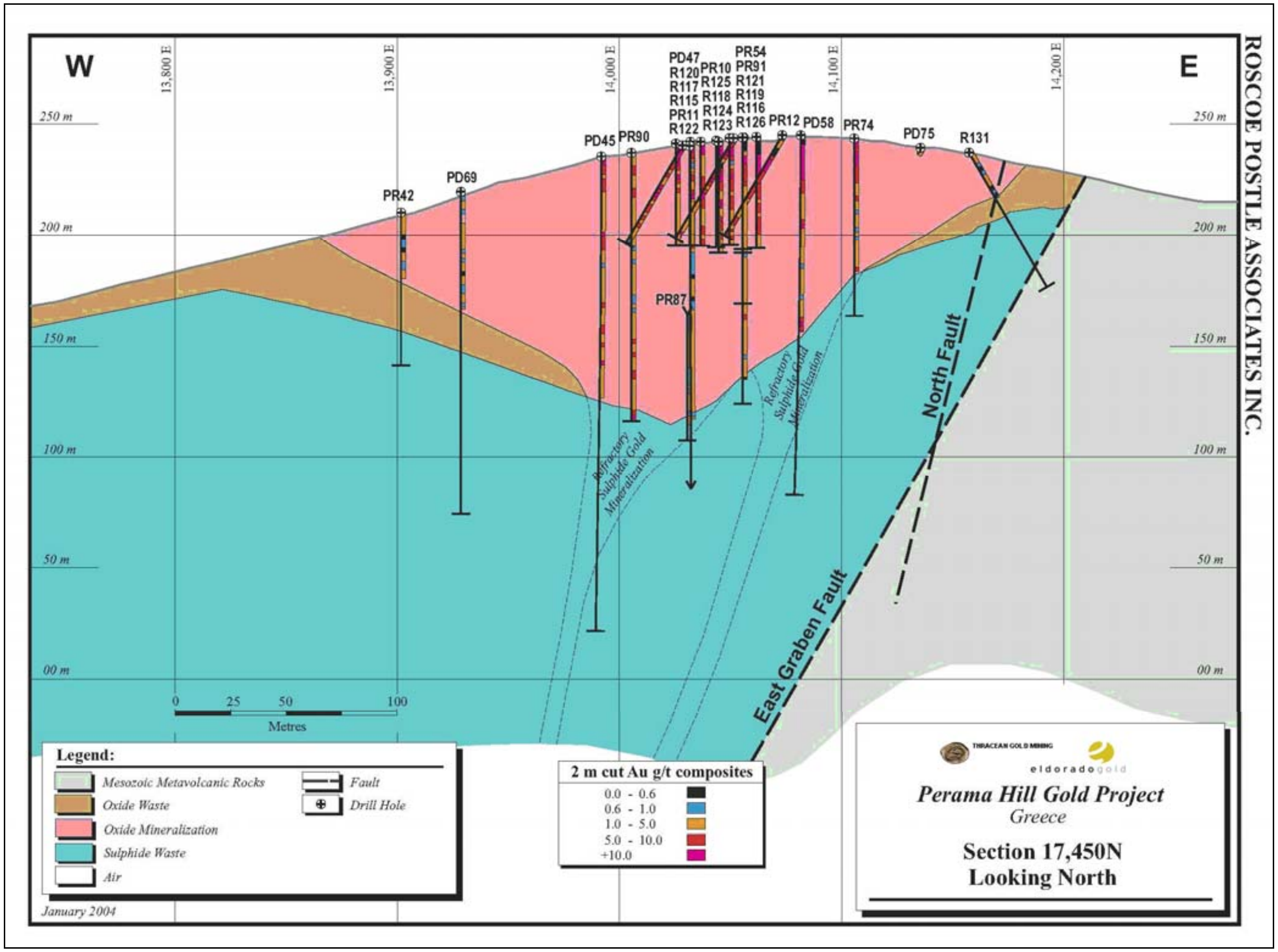


Figure 17-4: Section 17,550N

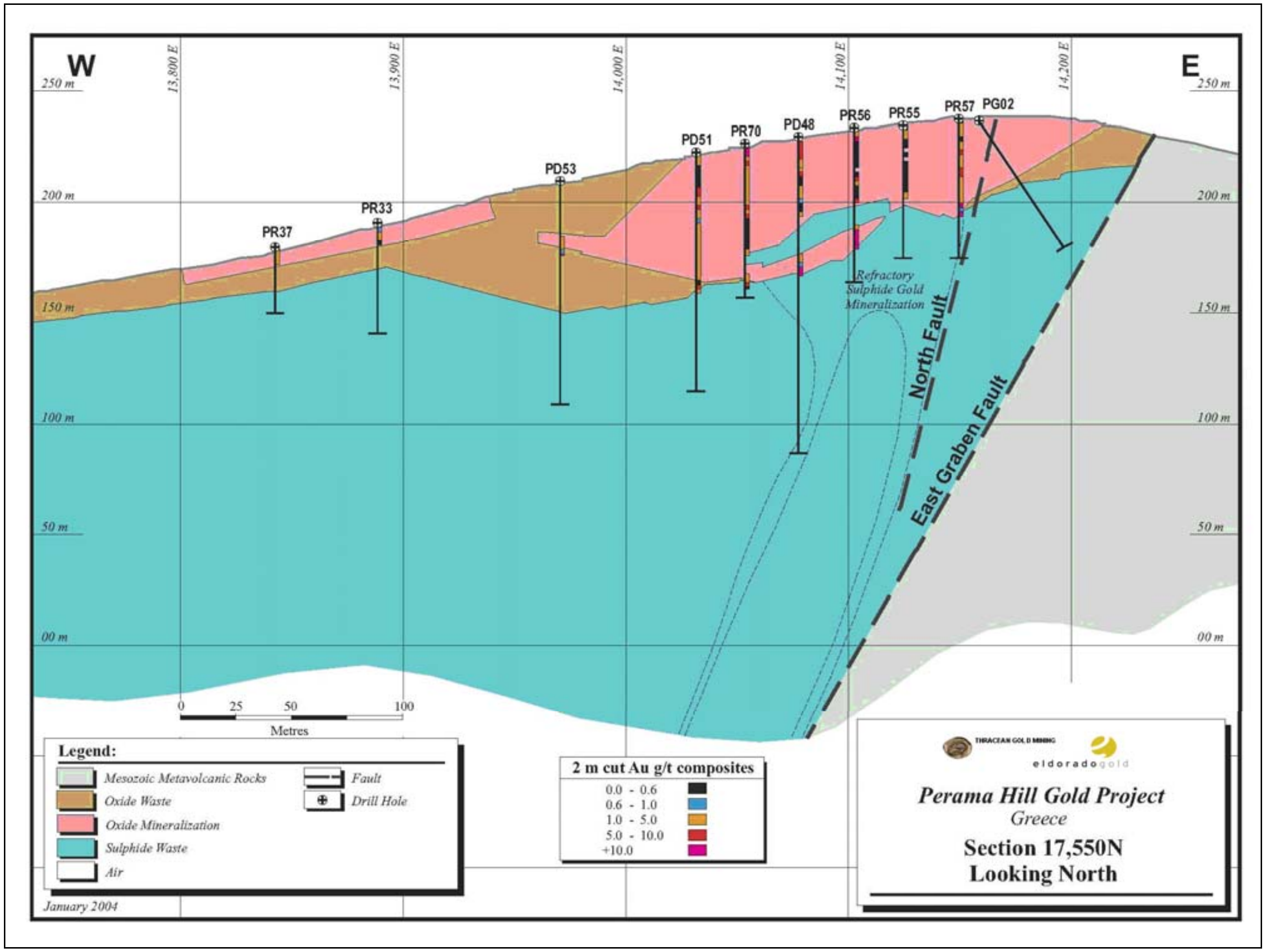


Figure 17-5: Section 17,775N

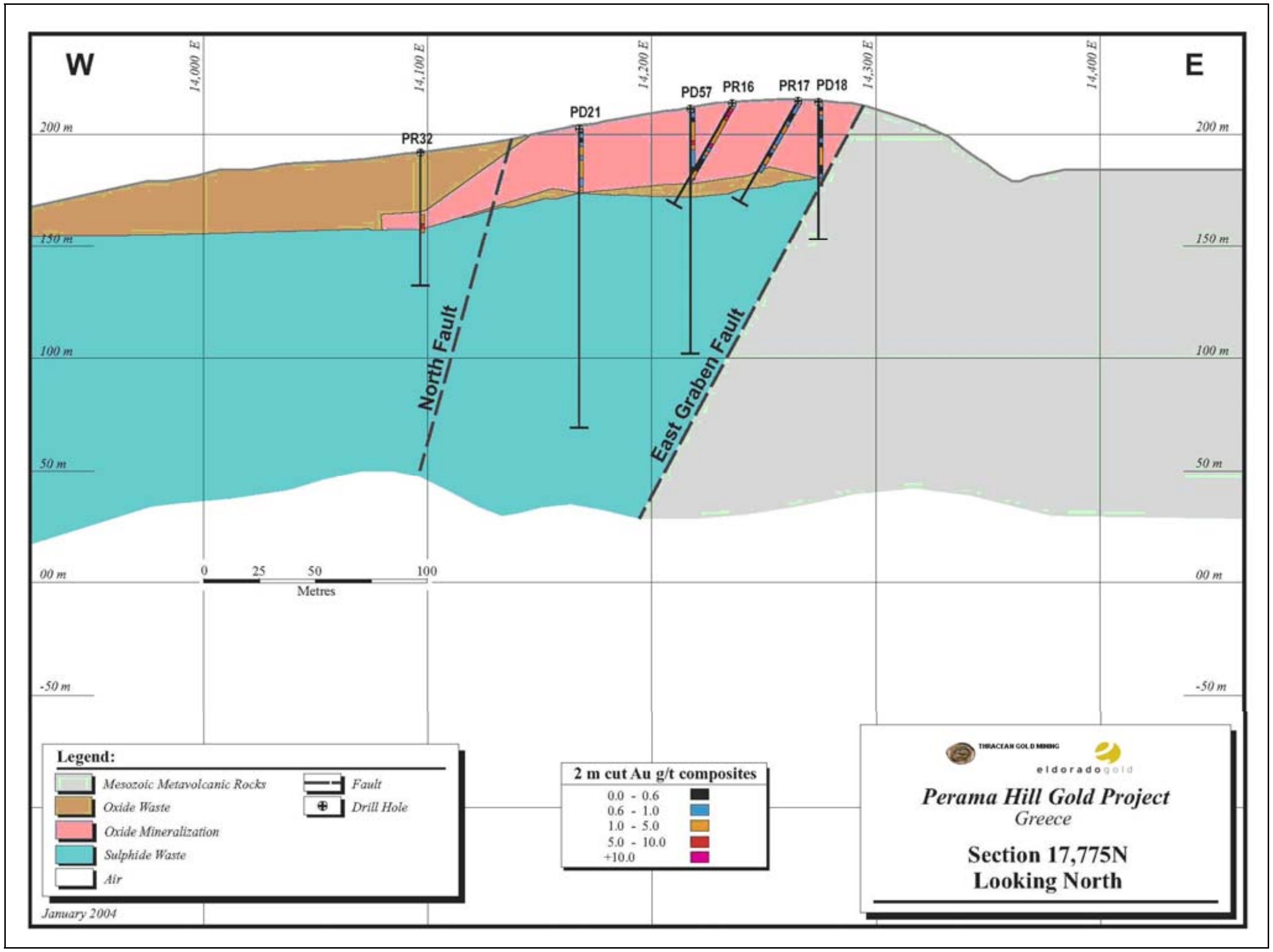


Table 17-1: Resource Assay Descriptive Statistics

Description	Length (m)	Recovery %)	Au (g/t)	Cut Au* (g/t)	Ag (g/t)	As (ppm)	SB (ppm)	Cu (ppm)	Pb (ppm)	Zn (ppm)	Ba (ppm)
Mean	1.05	93	4.33	4.20	6.46	376	9	137	576	57	1,022
Standard Error	0.00	0	0.07	0.06	0.27	6	0	16	21	3	19
Median	1.00	100	2.41	2.41	1.60	308	4	16	234	22	798
Mode	1.00	100	1.87	30.00	0.25	270	3	8	2	2	1
Standard Deviation	0.22	16	6.12	5.05	16.82	307	16	825	1,097	181	994
Sample Variance	0.05	258	37.49	25.47	282.99	94,297	267	680,097	1,203,578	32,796	988,983
Coefficient of Variation	0.21	0.17	1.42	1.20	2.60	0.82	1.80	6.03	1.90	3.18	0.97
Kurtosis	15.27	-1	92.61	8.55	62.05	41	259	281	59	391	33
Skewness	4.10	140	6.41	2.69	6.73	4	12	15	6	17	4
Range	2.50	4	164.00	30.00	280.00	5,401	456	22,100	17,418	4,860	9,999
Minimum	0.50	144	0.00	0.00	0.00	7	0	1	1	1	1
Maximum	3.00	739802	164.00	30.00	280.00	5,408	456	22,100	17,418	4,860	10,000
Count	8,369	7,932	8,369	8,369	3,817	2,719	2,719	2,719	2,719	2,719	2,719
Confidence Level (95.0%)	0.00	0	0.13	0.11	0.53	12	1	31	41	7	37

Note: * High Assays cut to 30 g/t Au

The assays situated within the oxide mineralization wireframe model were composited to 2.0 m lengths starting at the first oxide mineralization wireframe boundary from the collar and resetting at each new oxide mineralization wireframe boundary. Shorter composites generated adjacent to wireframe boundaries were not excluded. (Only 2.4% of the composites had lengths less than 2.0 m) The 4,462 composites average 4.23 g/t Au (4.36 g/t Au uncut), 6.64 g/t Ag and 1.98 m in length.

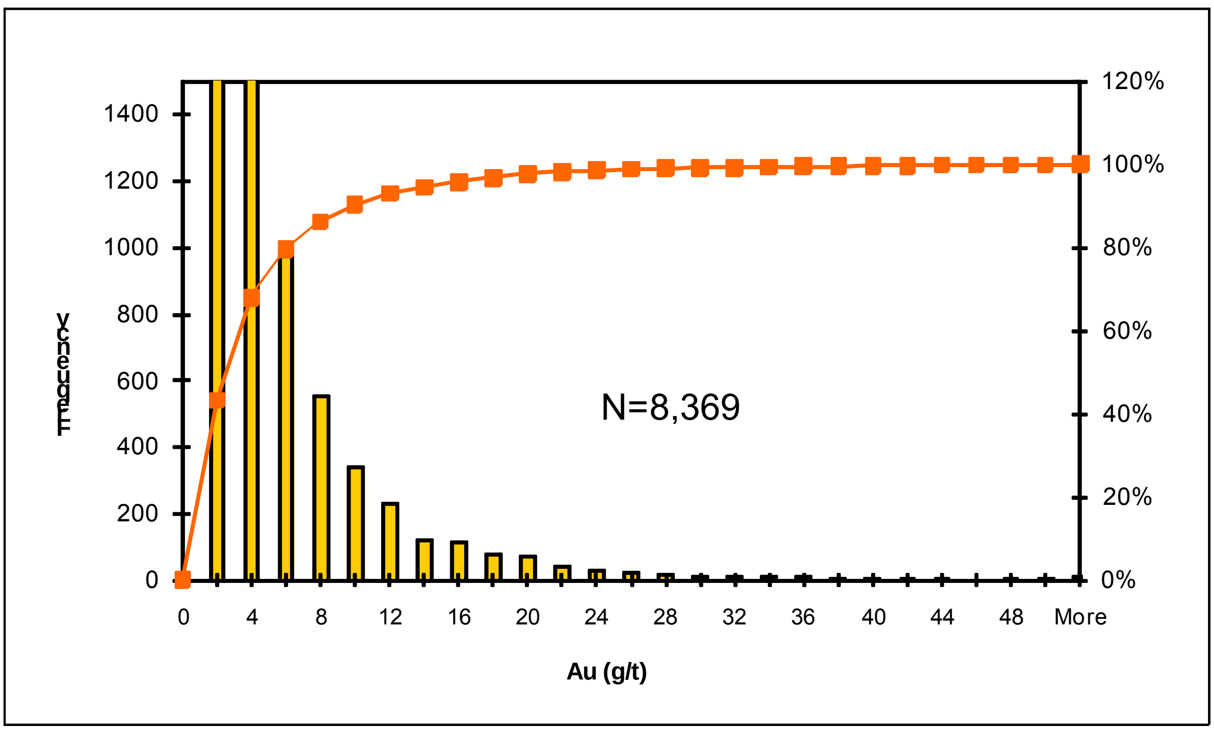
Mineralized Domain

A 0.6 g/t Au cutoff grade was chosen to define the mineralized envelope and to construct a 3-D wireframe model of the mineralization. This cutoff grade preserved the mineralization continuity and ensured that all potentially economic mineralization was included within the wireframe models. Some exceptions were made to preserve zone continuity. A minimum thickness of approximately 5 m was used to define the mineralization envelopes. The minimum thickness was not applied to oxide mineralization located at surface.

17.1.3 EVALUATION OF EXTREME GRADES

Extreme grades were examined by histogram (Figure 17-6) and cumulative frequency (on a log probability plot), A value of 30 g/t Au was chosen as a cutting level (Scott Wilson, 2004). Eldorado concurs with this selection.

Figure 17-6: Gold Resource Assays Histogram



All high assays were cut to 30 g/t Au before compositing to 2 m lengths. Block models were constructed based on uncut and cut gold assays.

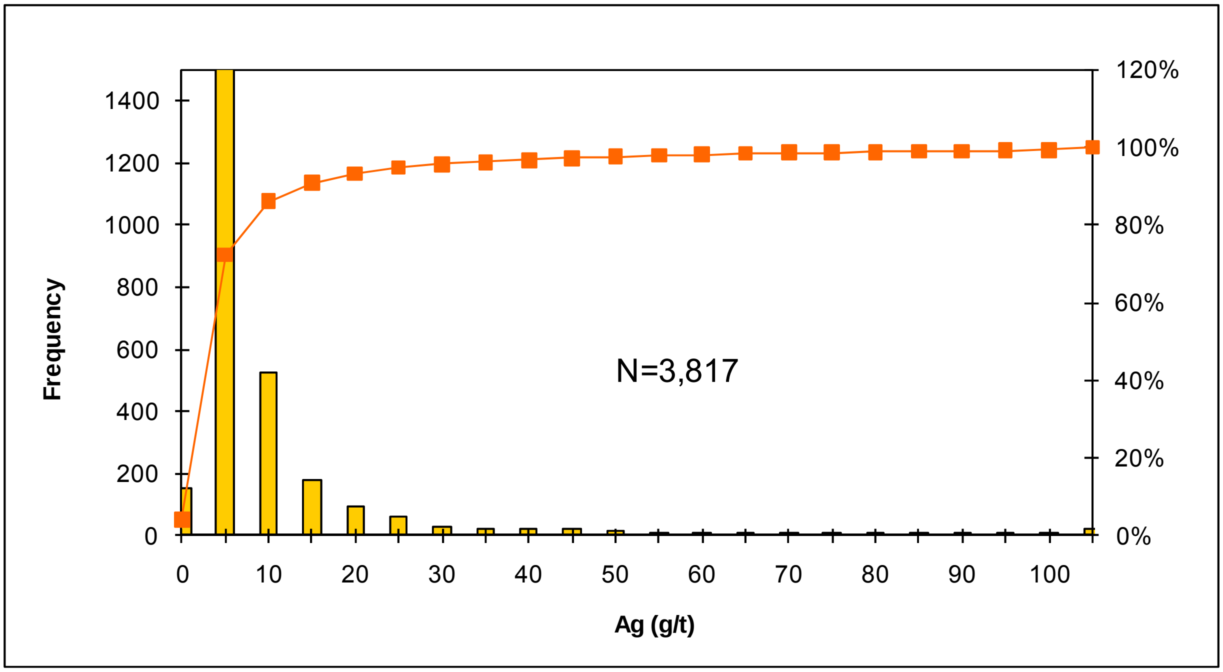
Scott Wilson (2004) also examined the amount of gold metal contained in each percentile in the top decile of the resource assays on a cut and uncut basis (Table 17-2). Cutting the high resource assays to 30 g/t Au reduces the contained gold metal in the top decile from 41.4% to 39.7% and in the top percentile from 9.9% to 7.3%. As a general rule, the top decile should not contain more than approximately 40% of the contained metal and the top percentile should not contain more than approximately 10%.

Scott Wilson's interpretation of the frequency distribution of the Perama Hill silver assays Figure 17-7 suggests that the likely range of possible silver cutting levels extends from approximately 50 g/t Ag to 100 g/t Ag. Cutting high silver assays to 100 g/t Ag or 50 g/t Ag reduces the average silver assay grade by approximately 5% or 16%, respectively. Silver values were not cut for the grade block model.

Table 17-2: Gold Metal Contained in Top Ten Resource Assay Percentiles

Percentile	Gold (g/t)	Percent of Total Gold Metal in Percentile	
		Uncut	Cut to 30 g/t Au
90	9.9	2.2%	2.2%
91	10.4	2.3%	2.4%
92	11.2	2.8%	2.9%
93	12.0	3.1%	3.2%
94	13.4	3.0%	3.1%
95	14.6	3.7%	3.8%
96	16.3	4.1%	4.3%
97	18.5	4.6%	4.8%
98	21.2	5.6%	5.8%
99	28.6	9.9%	7.3%
Total Gold Metal in Top Decile		**41.4%**	**39.7%**

Figure 17-7: Silver Resource Assays Histogram



17.1.4 VARIOGRAPHY

Variograms were calculated using the cut gold 2 m, 3 m, and 5 m composite data.

Variography, a continuation of data analysis, is the study of the spatial variability of an attribute. Variograms were calculated for gold in the oxide mineralized zone using the cut composite data (Scott Wilson, 2004). A series of variograms were built to test a number of strike directions, dips, and plunges. The longest ranges were observed in the north-south, horizontal dip variograms, termed the along "strike variograms." The "across strike" variograms was represented by the east-west, horizontal dipping variograms. The down hole variograms was also used, largely to set the nugget effect (Co).

The chosen variogram orientation closely matched the attitude of the Perama Hill deposit. The deposit strikes parallel to the East Graben Fault at N15°E and displays variable dips from sub-horizontal to gently westward. Also, the mineralization feeder structures at the base of the deposit strike approximately north-south and dip steeply to the west.

The selected variograms were modelled using a two structure, nested spherical model (Table 17-3). It consists of a nugget effect (Cu), two-nested structure variance contributions (C_1 and C_2) and ranges for each structure (R_1 and R_2).

Table 17-3: Variogram Modelling Parameters

Direction	C0	R1	C1	R2	C2	Lag	Spread
Down Hole (360°/-90°)	4	8 m	9.0	25 m	4.0	2 m	5°
Along Strike (360°/-00°)	4	16 m	8.0	41 m	8.5	16 m	5°
Across Strike (270°/-00°)	4	1 m	7.0	23 m	13.0	5 m	5°

17.1.5 MODEL SET-UP AND ESTIMATION

3D block models were made utilizing commercial mine planning software (Gemcom®). A block model comprising 256,000 blocks was built. The blocks are 10 m x 10 m x 5 m high in size and the model has 80 columns oriented at local grid north-south, 100 rows, and 32 levels. The model origin is at local grid 13,600E, 17,000N, and 260 m elevation and the model has not been rotated. The information for each block in the model includes:

- interpolated Au, cut Au, and Ag grades related to blocks that contain at least 1% oxide mineralization
- the percentage of oxide mineralization wireframe model that is in each block
- the percentage of waste rock that is in each block
- interpolated densities for mineralized blocks
- default densities for waste rocks (1.95 g/cm³ for oxide waste, 2.29 g/cm³ for sulphide waste, and 2.51 g/cm³ for metavolcanic waste)
- indicated and inferred resource classification identifiers for mineralization blocks
- the distance to the closest composite used to interpolate oxide mineralization block grades.

A search ellipsoid using a minimum of two and a maximum of twelve composites was used to interpolate gold, cut gold and silver grades into blocks using a three-pass process. The first pass used a search ellipsoid with a 40 m radius oriented at 360°/-00°, a 25 m radius oriented at 360°/-90°, and a 25 m radius orientated at 270°/-00°. The search radii in the second pass were twice the first pass. The search radii in the third pass were thrice the first pass.

Gold grade modelling consisted of interpolation by ordinary kriging. Silver grades were interpolated by inverse distance squared. Grades were also interpolated using inverse distance squared to interpolate silver values. Scott Wilson also used inverse distance and inverse distance cubed for comparison and validation purposes. The inverse distance method should simulate ordinary kriging with a high relative nugget effect and the inverse distance cubed should simulate ordinary kriging with a low nugget effect.

The 258 oxide-mineralization density-tests were used to interpolate the mineralization density block model. A search ellipsoid using a minimum of two and a maximum of twelve density values was used to interpolate density values into blocks using a two pass process. The first pass used a search ellipsoid with a 80 m radius oriented at 360°/-00°, a 50 m radius oriented at 360°/-90°, and a

50 m radius orientated at 270°/-00°. The search radii in the second pass were thrice the first pass and the minimum was decreased to one density value from two in the first pass. Inverse distance squared was used to interpolate the density values.

17.1.6 VALIDATION

The Perama Hill resource block model was validated by:

- visual inspection and comparison of block grades with composite and assay grades
- comparison of grades estimated using different interpolation methods
- statistical comparison of resource assay and block grade distributions
- comparison of assay and block grades by section and elevation ranges.

Grade interpolation was examined relative to drill hole composite values by inspection. The checks showed good agreement between drill hole composite values and model cell values, particularly in well drilled areas. Models were also clocked for proper coding in both section and plan. Coding was found to be properly done. The gold grade kriged estimates were checked against estimates generated using inverse distance squared and inverse distance cubed. The four methods give very similar cut gold grade estimates at the 1.0 g/t Au cutoff grade with differences at the second decimal place (Figure 17-8). This provides additional confidence that the gold and cut gold grade estimates are reliable and relatively insensitive to the relative nugget effect used in the ordinary kriging profile.

Lastly the Scott Wilson comparison of the distribution of cut gold values on scatter plots by elevation and by local grid northings for the drill hole and block data found that the block model is generally successful in following the spatial fluctuations in the assay drill hole values.

The cut gold block model is shown in the 3-D perspective looking down to the north in Figure 17-9, which also shows 5 m spaced topographic contours and the East Graben Fault.

The density block model shows a pattern of north-south bands with higher density centres that may correspond to the feeder structures (Figure 17-10).

Figure 17-8: Tonnage and Grade Results for Different Interpolation Methods

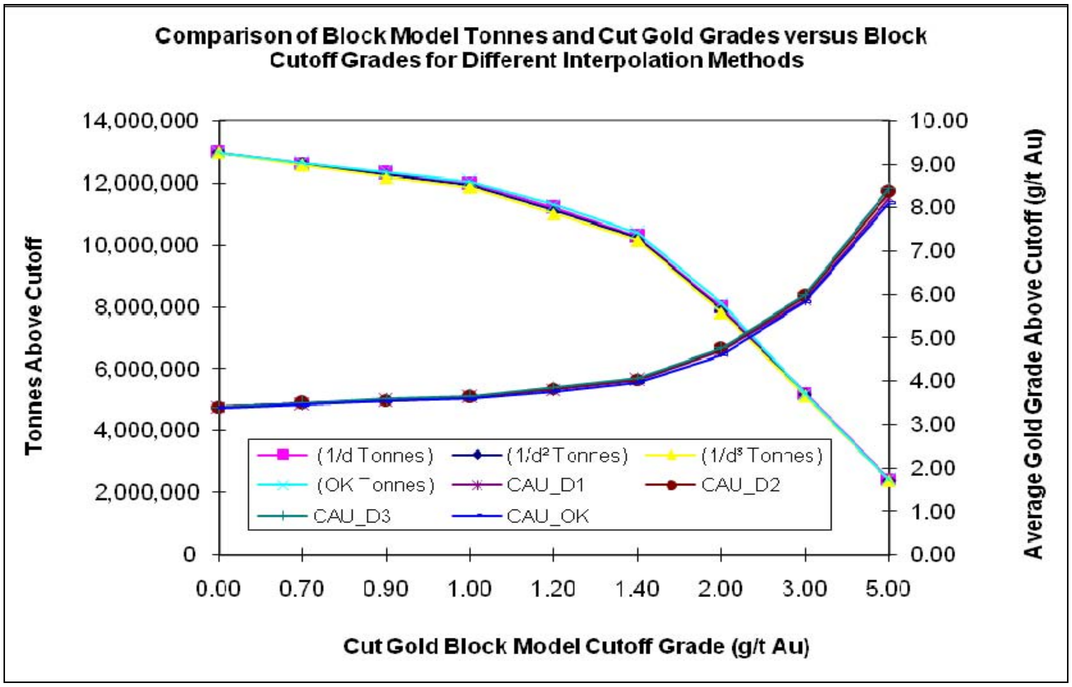


Figure 17-9: Perama Hill Cut Gold Block Model

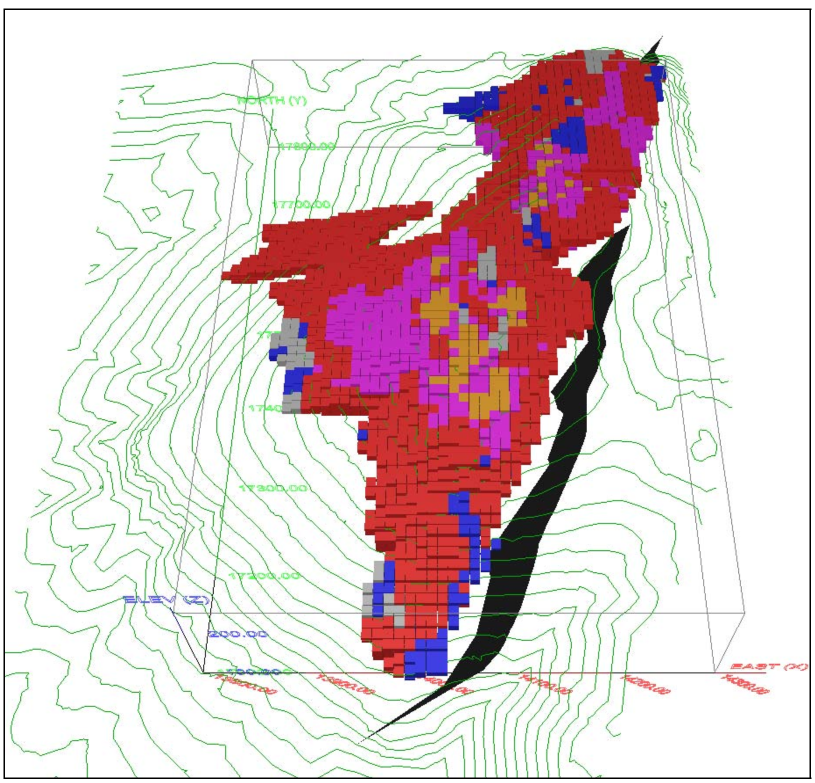
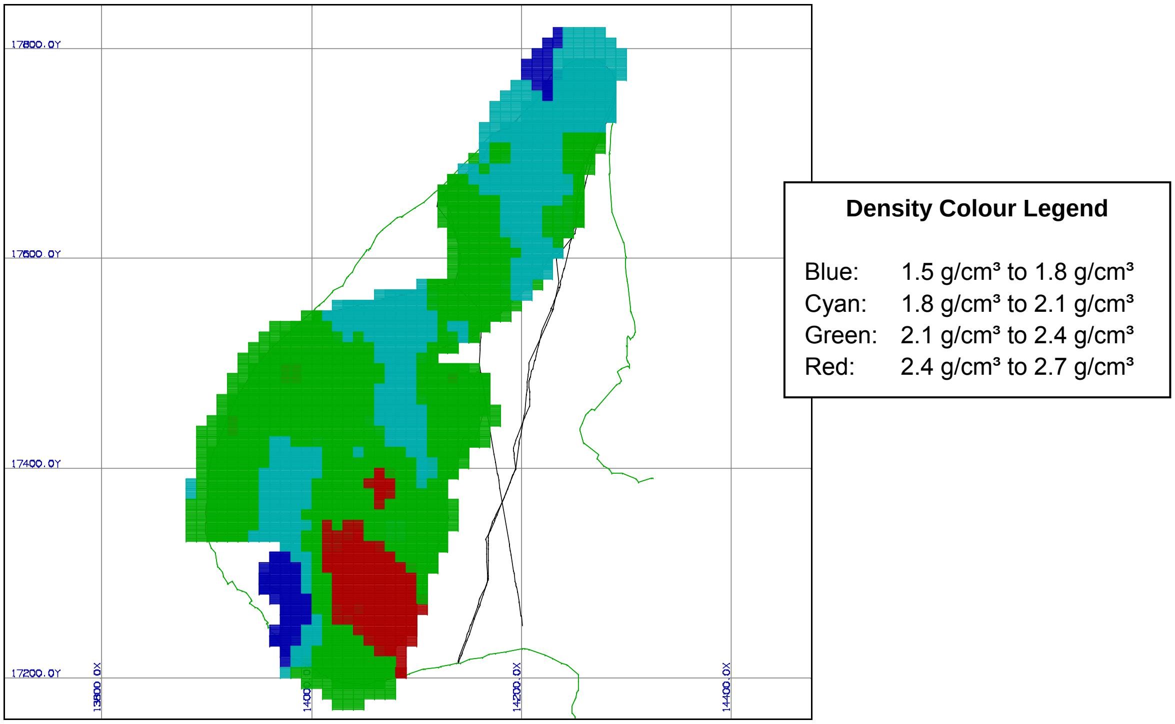


Figure 17-10: Perama Hill Density Block Model at the 215 m Elevation



In conclusion, Eldorado Gold considers that the Perama Hill block model is valid and appropriate for supporting the Mineral Resource estimate.

17.1.7 CLASSIFICATION AND SUMMARY

The mineral resources of Perama Hill were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization satisfies sufficient criteria to be classified into Measured, Indicated, and Inferred mineral resource categories. Almost all of the Perama Hill mineral resources are classified either as Measured or Indicated mineral resources.

Inspection of the Perama Hill model and drill hole data on plans and sections, combined with the spatial statistical work, showed good geologic and grade continuity in areas covered by the primary interpolation pass (40 m x 25 m x 25 m). Block covered by this pass were deemed to satisfy the requirements for Indicated mineral resources.

A subset of model blocks covered by the primary interpolation pass, clustered around a more densely drilled portion of the deposit and contained within the current open pit design (see Section 17.2), was deemed to satisfy the requirement for Measured mineral resources. Blocks on this zone were approximately within 20 m to composites from at least two drill holes.

The remaining model blocks not meeting the criteria for either Measured or Indicated mineral resources were assigned as Inferred mineral resources.

The Perama Hill mineral resources as of December 31, 2009, are shown in Table 17-4. This are reported at 1.0 g/t gold cutoff.

Table 17-4: Perama Hill Mineral Resources

Mineral Resource Category	Tonnes	Gold (g/t)	Contained Ounces
Measured	3,020,000	4.34	421,000
Indicated	8,690,000	3.37	942,000
Measured + Indicated	11,710,000	3.62	1,363,000
Inferred	333,000	2.58	27,000

17.2 MINERAL RESERVES

The Perama Hill mineral reserve estimates have been derived from the measured and indicated mineral resources occurring in the oxide portion of the deposit. Defined as the economically mineable part of the resource, the Perama Hill mineral reserves represent the portion of the mineral resources above the economic cutoff grade within the final pit design. The mineral reserve estimates have been appropriately adjusted for dilution and mining recovery. The mineral reserve estimates for the Perama Hill project were calculated under the direction of Richard Miller, P. Eng.

17.2.1 PIT OPTIMIZATION

Methodology

The first step in the process of evaluating the economic potential of the deposit was to undertake an open pit optimization analysis to assess the ore and waste tonnages, grades and extent of economically mineable resources. Special consideration was applied to constrain the optimization shells from including any material that was within 500m of the Perama village. Figure 17-11 graphically shows the area affected by this constraint. This analysis was carried out utilising the commercially available Lerchs-Grossman analysis software, Whittle 4X.

Table 17-5 shows a summary of the key operational and economic parameters utilized.

Sensitivity Analysis

The gold selling price for the whittle pit optimization and for the cutoff grade estimate used in the final mine production schedule was $825/oz. It can be seen from Figure 17-12 that ore tonnages are not significantly sensitive to the gold price ranging from $600/oz to $1,000/oz.

Figure 17-11: 3D View of Pit showing Zone of Mineralisation Excluded from Reserves

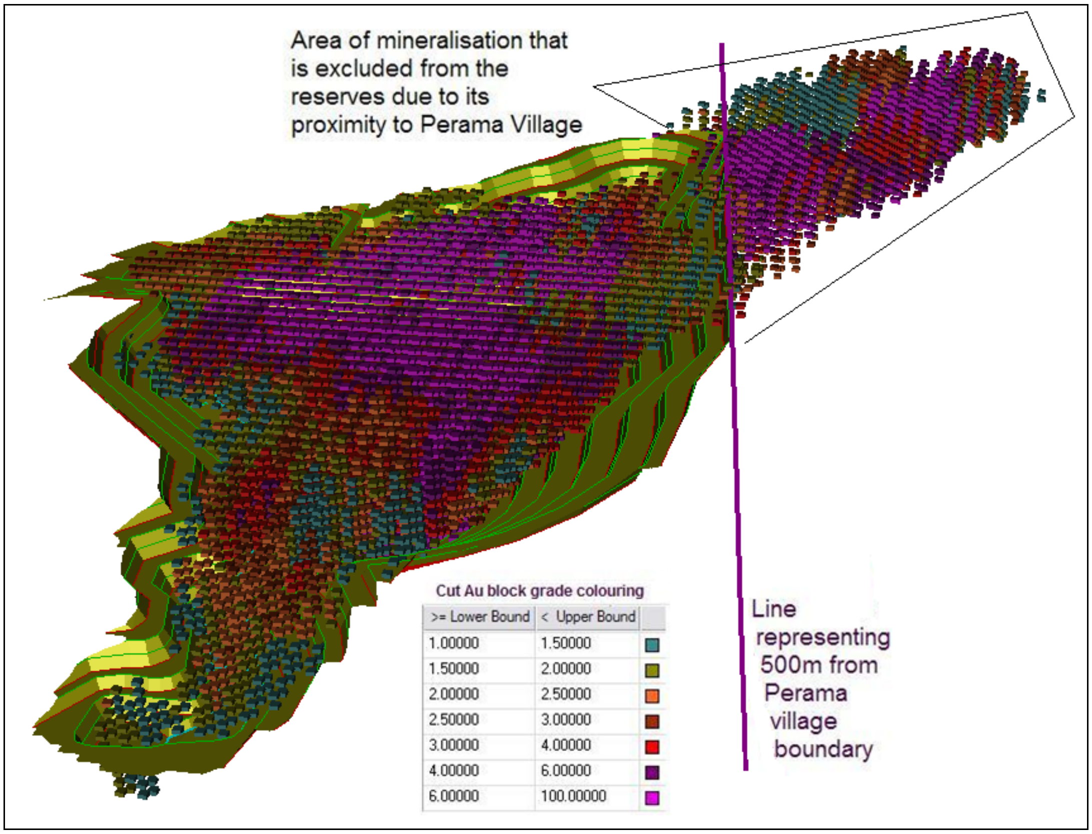
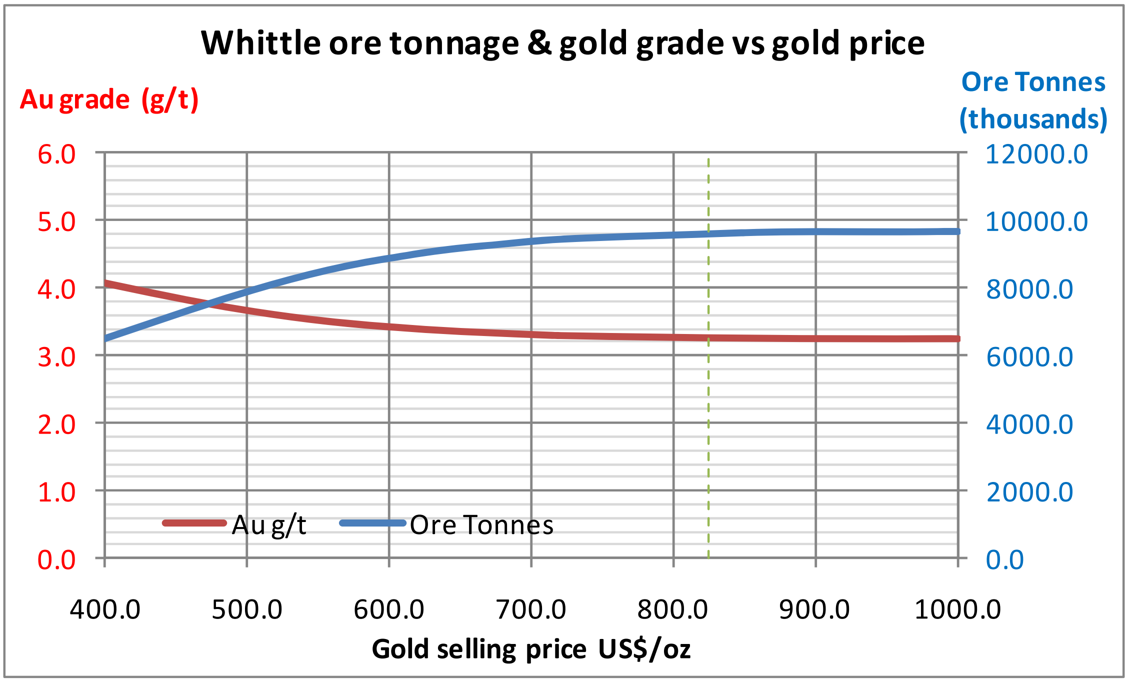


Table 17-5: Whittle Input Parameters

Gold Price	$825/oz
Silver Price	$12.08/oz
Mining Recovery	97%
Mining Dilution	3%
Mining Dilution Grade	0 g/t
Au Processing Recovery	90%
Ag Processing Recovery	60%
Blocks that are Included as being of economical benefit	Rocktype = Oxide ; and Classification = Indicated or Measured
Blocks that are Excluded from being of economical benefit	Rocktype = Sulphide ; and / or Classification = Inferred
Marginal Cutoff Grade	1.00 g/t combined Au Eq.
Mining Cost	Averaging $2.96/t mined using a base cost of $2.92/t and with cost adjustment factors for depth and sulphide rock
Process Cost (including G&A costs)	$20.10/t processed
Pit Wall Slope Angles	East wall along fault zone: 32°, all other walls 37.5°

Figure 17-12: Variation in Pit Tonnage and Gold Grade with Gold Price



Slope Angle

An overall slope angle in the range of 32 degrees to 37.5 degrees was used on the East wall and 37.5 degrees elsewhere to derive the pit shell. The southern part of the east wall, which is close to a major fault has been flattened to 32 degrees.

Silver Content

Silver was included in the Whittle optimization. Silver and gold were combined into gold equivalent grades for the optimization based on the selling prices and metallurgical recoveries mentioned in Table 17-6.

Operating Costs

Based on an initial assessment of costs and tonnage a base cost of $2.92/t mined was used for an oxide block on surface. Cost adjustment factors were used to increase the cost of deeper blocks and sulphide blocks and included principal cost centres of repair and maintenance (R&M), fuel, lubes, tires, wear parts, blasting, and labour. Sulphide waste material also required a higher reclamation cost. Overall the Whittle results came out with a weighted average mining cost of $2.96/t. The value $20.10/t processed is inclusive of all process and G&A operating costs.

17.2.2 PIT DESIGN

Bench- Berm Configurations

The open pit has been designed using 5 m high benches, which matches the individual blocks within the block model and is within the range of the selected mining equipment. Drilling and blasting is also based on 5 m benches to suit the design. Most of the final walls are double benched leaving 10 m high faces. All faces are planned at 60 degree face angles.

The open pit has been designed to meet the geotechnical criteria described earlier. A maximum overall angle utilized for the east wall varies from 32 degrees to 37.5 degrees, 37.5 degrees is also used for the south, west and north walls. Overall angles are shallower in parts of the pit due to the lie of the orebody.

Haul Road and Ramp Design Criteria

All surface haul roads and in-pit ramps are designed to be suitable for the operation of the widest mining equipment being used. In general both surface haul roads and in-pit ramps have been designed 16 m wide for two way traffic with a maximum gradient of 10%, which includes provision for a ditch and safety berm. The in-pit haul ramp from the last four benches to the pit floor is 10 m wide for single way traffic.

The in-pit ramps were exclusively placed on the west wall and this, along with the flat-lying contact of ore-grade mineralisation, has meant that the east wall will often be mined at a flatter angle than the limiting overall slope angle of 37.5 degrees.

Ultimate Pit Design

The ultimate pit was designed utilising the GEMCOM GEMS software. The final pit design is approximately 630 m long in the north-south direction and up to 340 m in width. It extends from the top of Perama Hill at 248 m elevation to the pit floor at 125 m elevation. The highest elevation exposed on the final pit design is at 238 m elevation.

Due to the surface topography and pit shape, the upper benches of the pit will not require permanent ramps since access roads can be constructed utilising the natural topography. The permanent pit highwall ramp begins at the 175 m elevation level. It has a minimum width of 16 m and at a gradient that does not exceed -10%. The top of the ramp is on the northern part of the west wall.

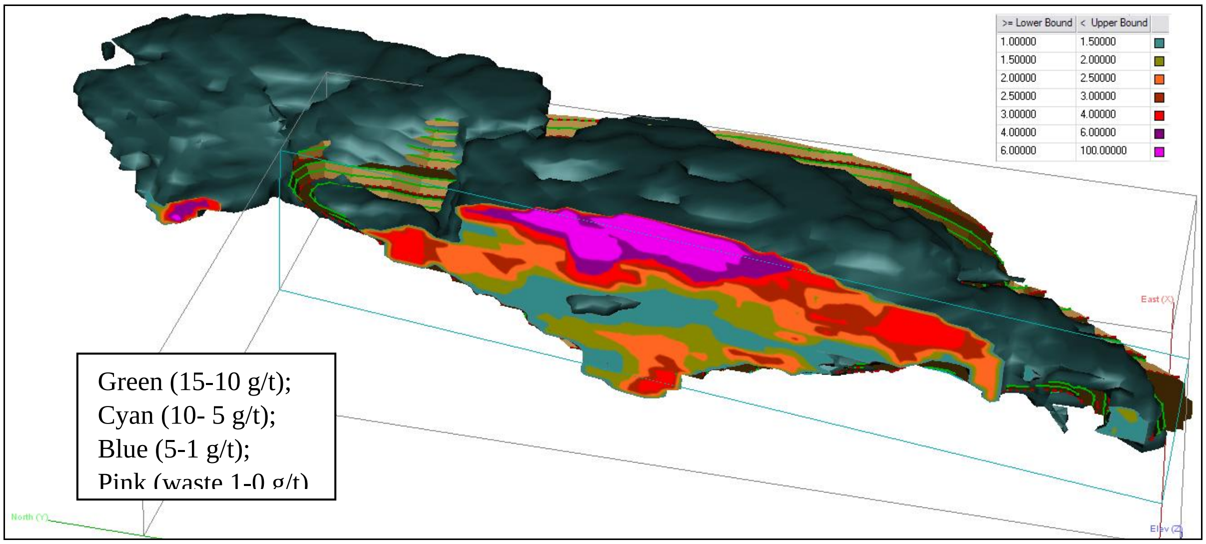
17.2.3 MINING DILUTION, RECOVERY AND GOLD GRADES

The orebody is very continuous and tabular in nature with little cover on surface. Consequently, a 97% mining recovery estimate as been made to account for minor ore loss at contact points and ore reporting to the waste dump.

With the very low stripping ratio, low powder factors, and the continuous tabular nature of the orebody, the dilution factor used was 3%. The grade of the diluting material has conservatively been taken as 0 g/t for both gold and silver.

Gold grades are highest near the surface of the pit and gradually reduce with depth although there is a slight increase of gold grade at the lowermost benches. Figure 17-13 shows the variation of grades through a longitudinal section of the deepest part of the orebody.

Figure 17-13: Longitudinal Section showing High Grade Zones



17.3 OPEN PIT CUTOFF GRADE

The cutoff grade (COG) assigned to the in-pit indicated oxide resources in order to estimate mineable reserves is 1.00 g/t Au.

17.3.1 CUTOFF GRADE DETERMINATION

The reserve cutoff grade signifies the minimum grade for a tonne of ore to break even after considering the total production costs for mining, processing, general and administration to process the same tonne of ore.

The selling cost for gold was calculated to be $8.00/oz/oz for the optimization study. For reserve cutoff grade estimates a fixed gold price of $825/oz was used. Earlier sensitivity investigations have shown that the reserve is not very sensitive to gold price fluctuations above and below this level. A silver price of $12.08/oz was also used for the optimization study. Because both metals contribute to the revenue of a block a weighted Au equivalent grade was calculated using the respective metallurgical recoveries of each metal. The recovery and cost parameters are shown in Table 17-6.

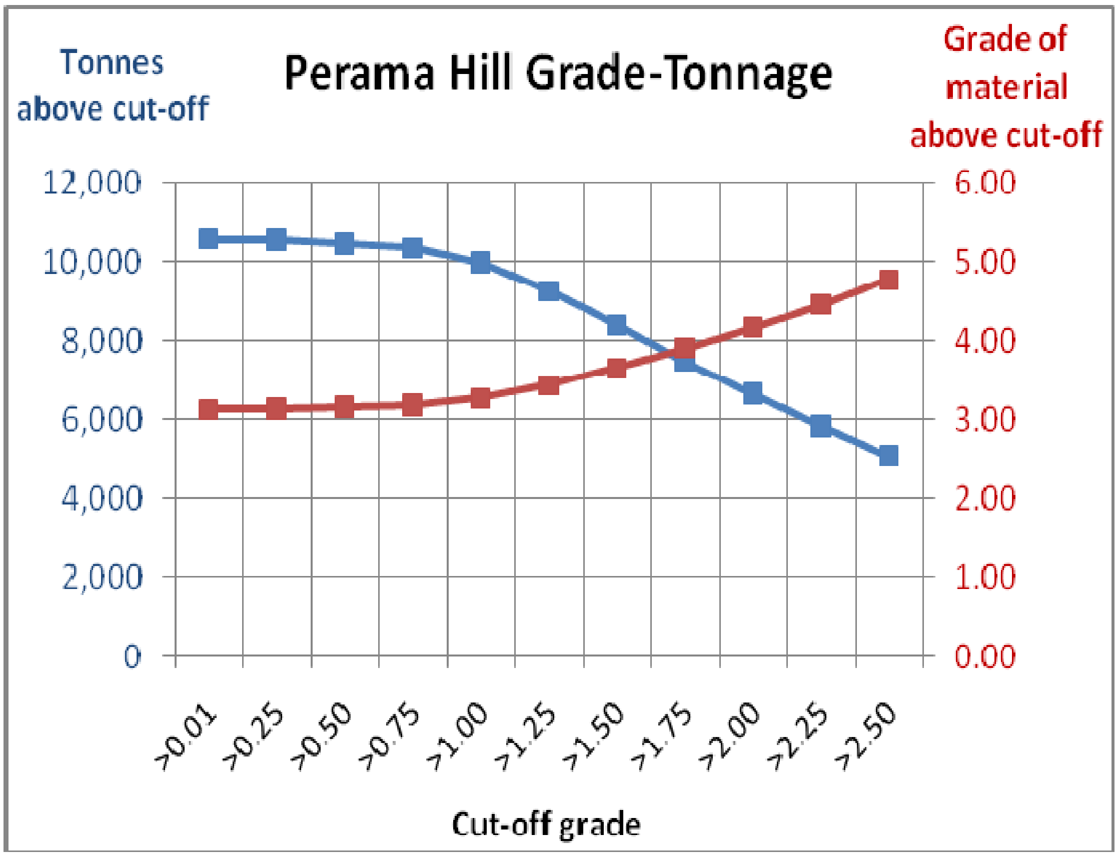
The total processing, general and administration costs are $20.10/t processed. This value is inclusive of all process and G&A operating costs.

For this study, mining costs have been excluded from the cutoff grade calculation. Any block within the final pit shell is to be mined whether it will be handled as ore or waste. Therefore, the cutoff grade calculation does not need to consider the cost of mining a block of ore other than the differential cost between handling it as ore or waste. With both the crusher and the rock dumps being in close proximity to the exit of the pit the differential in cost is not significant.

The breakeven cutoff grade was calculated to be slightly below 1.0 g/t AuEq. . Therefore, a value of 1.00 g/t AuEq. was chosen. This was evaluated against a gold only cutoff grade for reporting global mineral reserves and found to be virtually identical to a 1.00 g/t Au (only) cutoff grade.

A sensitivity analysis for cutoff grade variation, ore tonnages, and average gold equivalent grades is shown in Figure 17-14.

Figure 17-14: Sensitivity Analysis



17.3.2 DENSITY

In situ dry densities are based on diamond drill core testing, which have been interpolated in each block of the 3D block model. An analysis of tonnages and volumes within the ultimate pit shows weighted average dry densities for the following material mined:

- Soft Oxide Ore: 2.13 t/m^3

- Medium Oxide Ore: 2.13 t/m^3
- Hard Oxide Ore: 2.30 t/m^3
- Overall Oxide Ore: 2.18t /m^3
- Oxide Waste: 1.95 t/m^3
- Sulphide Waste: 2.29 t/m^3
- MT Waste: 2.51 t/m^3
- Overall Waste: 2.03 t/m^3
- Overall: 2.14 t/m^3

17.3.3 MINERAL RESERVE CLASSIFICATION AND SUMMARY

The mineral reserves of the Perama Hill Project were classified using logic consistent with the CIM definitions referred to in NI 43-101. The Measured and Indicated mineral resources contained within the final pit design satisfied sufficient criteria to be respectively converted to Proven and Probable mineral reserves.

Table 17-7 shows a summary of the Perama Hill mineral reserves, as of December 31, 2009. The reserves are based on a cutoff grade of 1.00 g/t Au, an estimated 3% dilution factor and 97% mining recovery.

The total waste quantity of 2.742 Mt includes 319,000 tonnes of mineralized low-grade oxide at an average grade of 0.87 g/t Au and 140,000 tonnes of mineralized sulphide material at an average grade of 3.71 g/t Au. This sulphide material will not be processed and is therefore considered as waste.

Inferred mineral oxide resources above the cutoff grade and within the pit limits total 0.407 Mt at an average grade of 2.39 g/t Au. This resource has been treated as waste material and has not been included in the mine production schedule. However, this material does represent an opportunity and would warrant an in-fill drill program to convert into potential Indicated mineral resources.

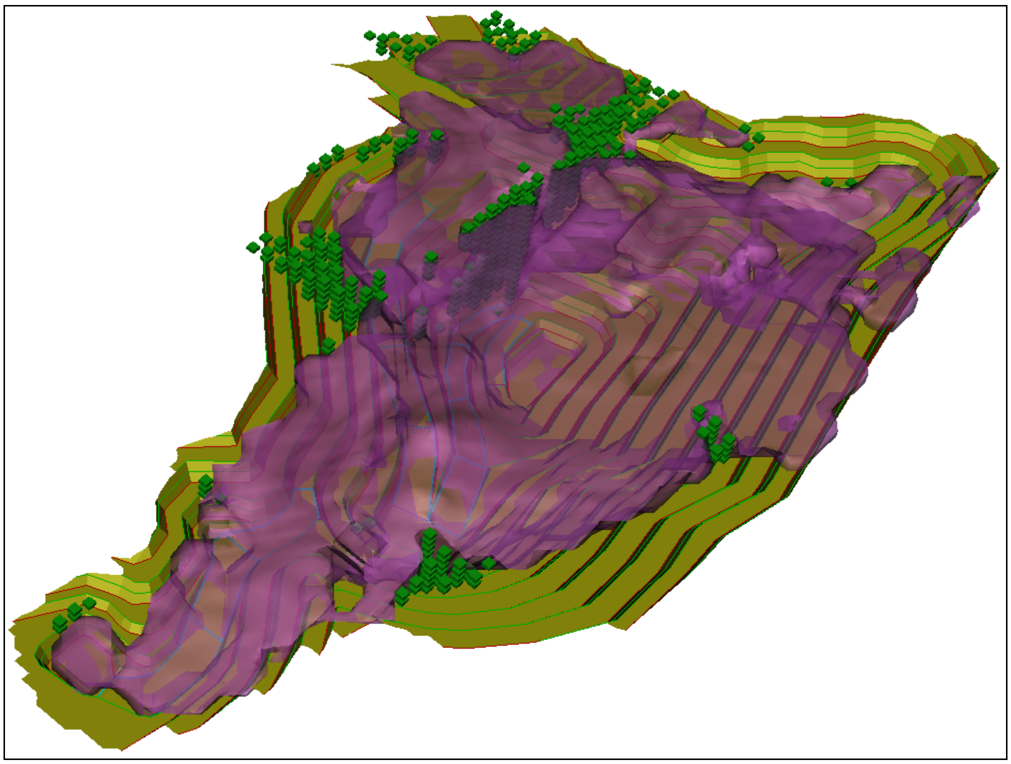
Figure 17-15 shows the inferred resource blocks (in green) within the final pit design and orebody (purple).

Table 17-6: Perama Hill Mineral Reserves

	Tonnes (Mt)	**Gold (g/t)**	**Silver (g/t)**
Ore (>1.00 g/t Au)			
Proven Reserves	2.455	4.48	3.20
Probable Reserves	6.923	2.75	3.94
Proven and Probable Reserves	9.378	3.20	3.75
Contained Gold Reserves	0.966 Moz		

	Tonnes (Mt)	Gold (g/t)	Silver (g/t)
Contained Silver Reserves	1.129 Moz		
Waste			
Inferred Oxide Resource	0.407	2.39	2.80
Sub-grade Oxide Waste	0.319	0.87	2.20
Unmineralized Oxide Waste	1.876	-	-
Sulphide Waste	0.140	3.71	10.23
Waste Subtotal	2.742	-	-
All Material	*12,120*	-	-
Strip Ratio	0.29:1	-	-

Figure 17-15: Inferred Blocks within the Final Pit Design and Location within the Orebody



SECTION 18 • OTHER RELEVANT DATA AND INFORMATION

18.1 INTRODUCTION

A closure plan has been prepared in compliance with the Environmental Policy adopted by Eldorado. It will conform to the requirements of the Greek authorities, and to best international practice. SRK Consulting (UK) Ltd, of Cardiff, has prepared the Mine Closure and Rehabilitation report for TGM.

The main points of the Closure Plan are given in the following section.

18.2 CONCEPTUAL CLOSURE PLAN

18.2.1 INTRODUCTION

The area around the Perama Hill gold deposit is rural, with sheep and goat farming and the cultivation of tobacco, cotton, and barley. The mine site is set amongst rolling hills with natural Maquis vegetation and forested areas of oak and black pine.

The orebody consists of oxidised deposits to a depth of 100 m, overlying deeper sulphide mineralization, which will not be exploited because of poor gold recoveries. The mine has an approximate life of eight years and is being developed as an open-pit. Following crushing and milling, gold recovery is by a conventional carbon in leach circuit, followed by electro winning and smelting. The tailings pass through an INCO cyanide detoxification plant before being deposited on a surface impoundment downstream of the plant.

In addition to the government requirements, the Corporate Environmental Policy under which TGM will operate requires that the company be environmentally responsible, implementing plans that protect the health and safety of employees and people living in the areas where the company operates. The Policy states that mine closure must satisfy good international practice, and that all costs associated with meeting present and future environmental operations and closure rehabilitation obligations are included in the project budgets.

18.2.2 CLOSURE PLAN KEY FEATURES

The closure activities and costs in the Closure Plan have been broken down into five main component areas:

1. Open-pit mine
2. Process plant and infrastructure
3. Waste rock dumps
4. Tailings management facility
5. Site rehabilitation.

For each component area, the main decommissioning and closure issues are highlighted along with the required performance criteria, the closure actions, and the provisions for the demonstration of closure performance. The issues, closure objectives, potential and actual closure actions, and performance criteria for each of the five areas are summarized in Table 18-5 the decommissioning and closure costs are summarized in

The key features of the closure and rehabilitation plan comprise:

- community consultation and consultation with the authorities on final land-use
- demolition and removal of process plant and infrastructure, and the revegetation and restoration of the area to its previous state
- partial backfilling and re-profiling of the open pit, involving the removal of all waste dumps outside the pit, the installation of drainage facilities and revegetation
- capping of the tailings impoundment and the restoration of vegetation
- maintenance of the saddle dams and associated drainage system
- continued monitoring and maintenance prior to final site clearance.

The plan anticipates an estimated decommissioning and closure period of some 18 months for all site infrastructures, the pit and the tailings impoundment, and an overall five years for site rehabilitation and environmental compliance monitoring.

Management aspects and costs of health and safety, safety systems and working procedures, decommissioning personnel and security have been addressed in the plan.

The closure costs cover the entire period, from the cessation of mining to the final site disposal, following rehabilitation.

Within the closure plan report the current environmental conditions and mining operations are described, with particular attention to sensitive environmental resources and those aspects of the operation, which will be important at closure. The plan then addresses the following principal components:

- open pit
- process plant and general site infrastructure
- waste rock dumps
- tailings disposal system
- site rehabilitation and revegetation

site water management.

Table 18-1: Open Pit Closure Issues, Performance Criteria, Closure Actions and Demonstration of Performance

Closure Issue	Performance Criteria	Closure Actions	Demonstration of Performance
Safe removal of pit infrastructure and mobile plant	All equipment with significant asset value to be removed and sold	A secure area will be established to store the removed equipment which will be cleaned of any contamination	All valued assets will be catalogued part of the asset management system
Illegal access	Prevent access to the top of pit walls and base of pit	Provide suitable safety fencing, warning signs, berms or other barriers as appropriate	All safety requirements are implemented and access prevented
Stability of pit slopes	Failure of pit walls will not jeopardise safety for local community or expose sulphide wall rock or deposited waste	Measures to stabilise the slopes, e.g. scaling, flattening, re-profiling, buttressing toe with backfill, improving drainage as required, combined with backfill to cover exposed sulphide rock	Slopes are stable and will not fail, especially where backfill covers sulphide material
Erosion of pit walls	Surface of pit walls stabilised to restrict surface erosion	Measures to stabilise slopes together with revegetation and installation of surface drainage to minimize surface flows	No visible signs of surface erosion
Pit flooding and water chemistry	Minimal entry of water into pit; water collecting in pit meets Greek surface water standards	Perimeter catch drains, inert cover to exposed sulphide in pit walls, wetland in pit as part of surface water management	Drainage system effective, quality of water draining from pit meets Greek surface water standards

Table 18-2: Process Plant and Infrastructure Closure Issues, Criteria Actions and Demonstration of Performance Area

Closure Issue	Performance Criteria	Closure Actions	Demonstration of Performance
Safe decommissioning of the process circuit and infrastructure	All decommissioning activities to be undertaken in a safe and environmentally friendly manner	As processing is completed, equipment will be sequentially cleaned and isolated prior to removal of valued assets	All valued assets will be catalogued
Safe demolition of the process and infrastructure areas	All activities to be undertaken in a safe and environmentally sound manner following prescribed procedures and programme	Removal and storage of valued assets; salvage moved to prescribed areas; buildings and infrastructure removed and decontaminated	Prescribed procedures and programme followed; business plan objectives met; site left in condition suitable for rehabilitation
Asset management	Valued assets recorded; appropriate procedures in place for storage and disposal	Assets removed and catalogued; sale arranged	Documentation in place recording removal, storage and sale; business plan objectives met
Achievement of business plan	Monitoring programme to record budget and time scale	Follow business plan	Business plan objectives achieved

Table 18-3: Waste Rock Dumps Closure Issues, Criteria, Actions and Demonstration of Performance

Closure Issue	Performance Criteria	Closure Actions	Demonstration of Performance
Safe decommissioning of the process circuit and infrastructure	All decommissioning activities to be undertaken in a safe and environmentally friendly manner	As processing is completed, equipment will be sequentially cleaned and isolated prior to removal of valued assets	All valued assets will be catalogued
Safe demolition of the process and infrastructure areas	All activities to be undertaken in a safe and environmentally sound manner following prescribed procedures and programme	Removal and storage of valued assets; salvage moved to prescribed areas; buildings and infrastructure removed and decontaminated	Prescribed procedures and programme followed; business plan objectives met; site left in condition suitable for rehabilitation
Asset management	Valued assets recorded; appropriate procedures in place for storage and disposal	Assets removed and catalogued; sale arranged	Documentation in place recording removal, storage and sale; business plan objectives met
Achievement of business plan	Monitoring programme to record budget and time scale	Follow business plan	Business plan objectives achieved

Table 18-4: TMF Closure Issues, Criteria, Actions and Demonstration of Performance

Closure Issue	Performance Criteria	Closure Actions	Demonstration of Performance
Safe decommissioning of the process circuit and infrastructure	All decommissioning activities to be undertaken in a safe and environmentally friendly manner	As processing is completed, equipment will be sequentially cleaned and isolated prior to removal of valued assets	All valued assets will be catalogued
Safe demolition of the process and infrastructure areas	All activities to be undertaken in a safe and environmentally sound manner following prescribed procedures and programme	Removal and storage of valued assets; salvage moved to prescribed areas; buildings and infrastructure removed and decontaminated	Prescribed procedures and programme followed; business plan objectives met; site left in condition suitable for rehabilitation
Asset management	Valued assets recorded; appropriate procedures in place for storage and disposal	Assets removed and catalogued; sale arranged	Documentation in place recording removal, storage and sale; business plan objectives met
Achievement of business plan	Monitoring programme to record budget and time scale	Follow business plan	Business plan objectives achieved

Table 18-5: Site Rehabilitation Closure Issues, Criteria, Actions and Demonstration of Performance

Closure Issue	Performance Criteria	Closure Actions	Demonstration of Performance
Contaminated land	Final land surface is not contaminated with oils, metals or process reagents	All contaminated land treated or removed for disposal in hazardous tip	Surface soils meet ANZECC, Dutch, VPR or UK standards
Final land use	Return site to beneficial land uses of agriculture and wildlife conservation	Flatter areas rehabilitated to grassland and steeper areas revegetated with indigenous species, oak and black pine	Successful establishment of vegetation after 2-3 years
Surface water management	Site profile avoids accumulation of surface water and minimises soil erosion; discharges meet Greek surface water standards	Design and execution of surface drainage plan for whole site	Drainage effective, no surface erosion, surface water quality meets Greek standards

Each of these components is dealt with by addressing and resolving relevant aspects, as follows:

- **Key issues** – what are the significant issues that need to be addressed at closure?
- **Performance Criteria** – what criteria should be used to show that decommissioning and closure will be accomplished effectively?
- **Investigations and testwork** – is additional information and data required, and what programmes are required to further investigate these criteria?
- **Closure options** – what alternative approaches to closure are viable or achievable?
- **Closure actions** – what closure actions are recommended?
- **Demonstration of** performance – how can TGM show that the closure actions will have met the proposed performance criteria?

The plan addresses the management of the decommissioning and closure of the mine, the required monitoring programmes, and the costs which will be expected to be incurred. The cost estimates are based on the use of mine personnel and equipment wherever possible, but include local contractor rates where applicable.

Table 18-6: Closure Costs

Description	Cost (US$ Million)
Administration	0.21
Rollover	0.84
Power	0.21
Process plant/infrastructure	1.60
Tailings capping	3.75
Open pit re-shaping/hydrology	3.33
Environmental/rehabilitation	3.29
Total	**13.24**

SECTION 19 • ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

19.1 MINE PLAN AND PRODUCTION

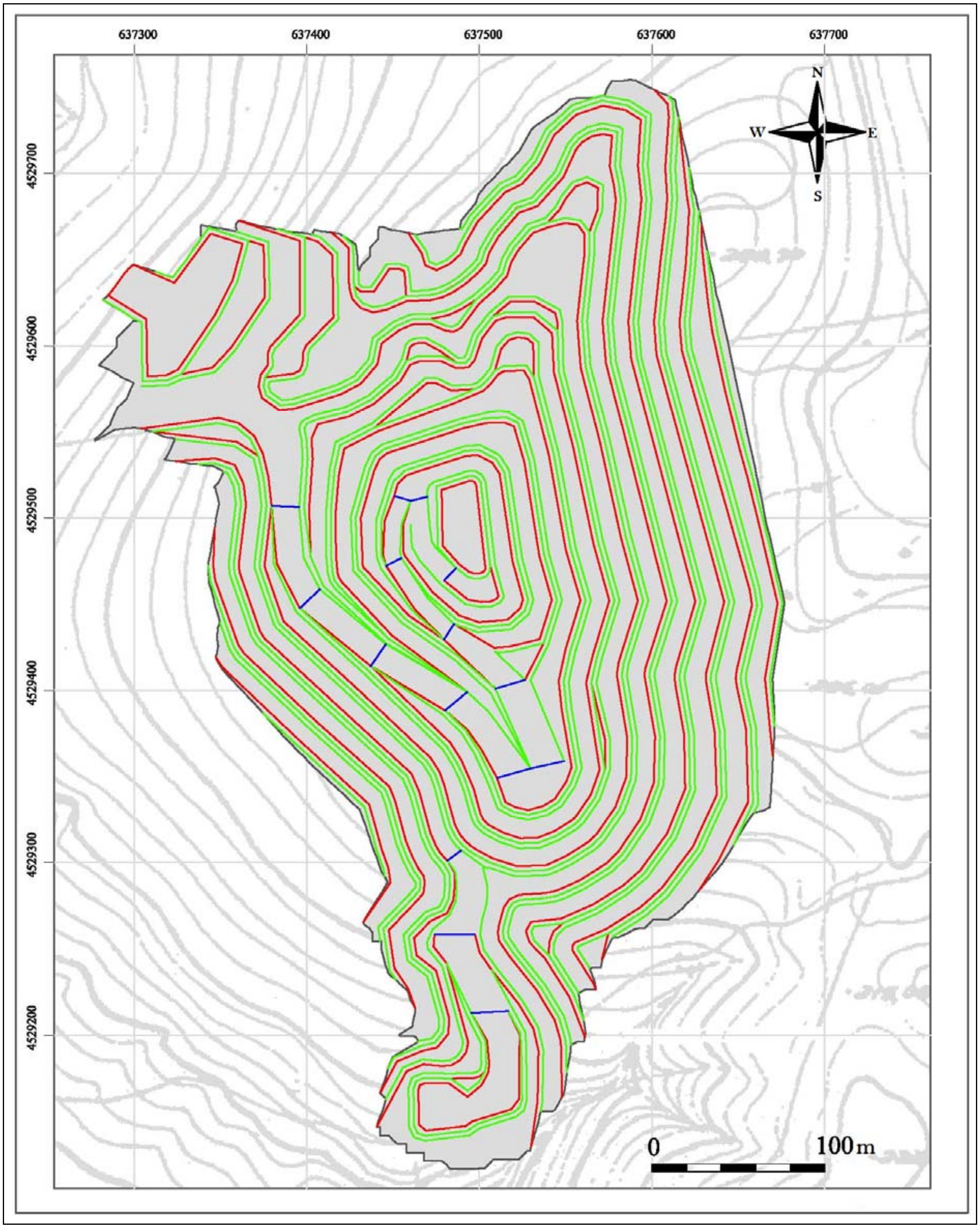
19.1.1 INTRODUCTION

The Perama Hill Deposit is located in Thrace, Northern Greece, within approximately 1 km of Perama Village. It consists of sediment hosted epithermal gold-silver mineralization. Of economic interest for this Study are the oxidised resources, which range from surface at 248 masl to depths of 125 masl. A conventional open pit mining operation is proposed utilizing well proven techniques and mine equipment.

19.1.2 PIT DEVELOPMENT

The ultimate pit dimensions are 630 m long from north to south and range from 100 m to 340 m wide, has been designed to economically extract the oxide resources that are convertible to ore reserves. See Figure 19-1. A total of 25 five-metre high benches will be developed. Each temporary and final bench face will have batter slope angles of 60 degrees. An overall angle varying from 32 to 37.5 degrees is utilized for the east wall and a maximum of 37.5 degrees for the south, west and north walls. However due to the lie of the orebody and the positioning of the ramp localised areas can be flatter than the design maximum overall slope angles. The highest pit wall lies 110 m high and is benched in 10m steps with face angles of 60 degrees and berms widths varying from 9 m to 12 m. Five metre benches are utilized in some places to better conform to the lie of the orebody.

Figure 19-1: Final Pit Configuration

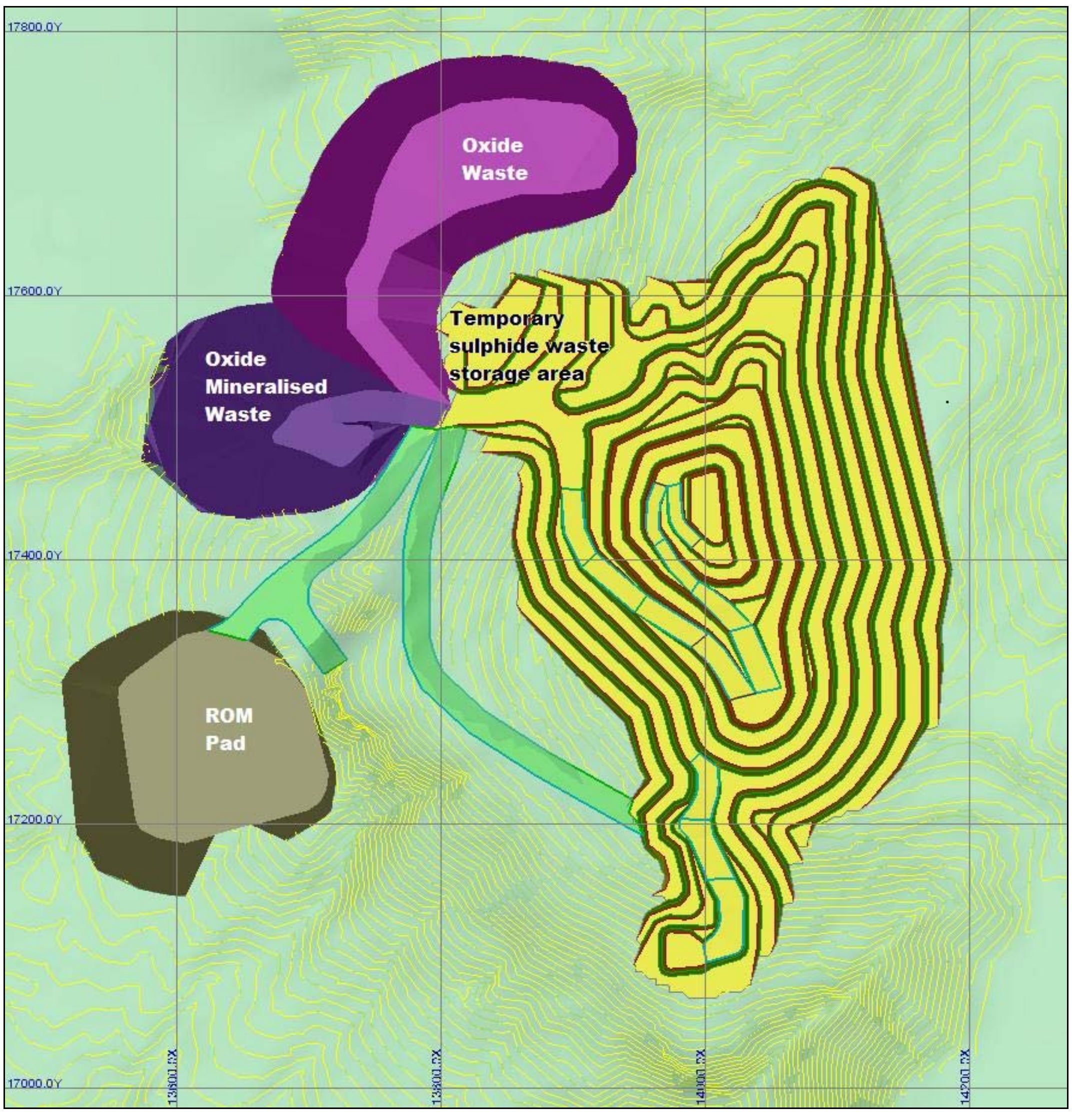


The upper benches of the open pit will be mined via a temporary ramp on the west side of the pit that joins with temporary ramps within the southern part of the pit. The benches below the 175 m elevation require the inclusion of a permanent 16 m wide two way in-pit haulage ramp along the west side of the pit. A 10 m single lane haulage ramp continues to the pit floor from the 145 m to 125 m elevation levels.

19.1.3 WASTE DISPOSAL

Two types of waste material will be produced during the life-of-mine. These are oxide and sulphide. Low grade oxide rock below cutoff grade will be segregated and stored separately within the oxide dump area. Sulphide waste will be temporarily stored in a separate waste dump, which is entirely within the development area of the pit itself. These planned areas are shown in Figure 19-2.

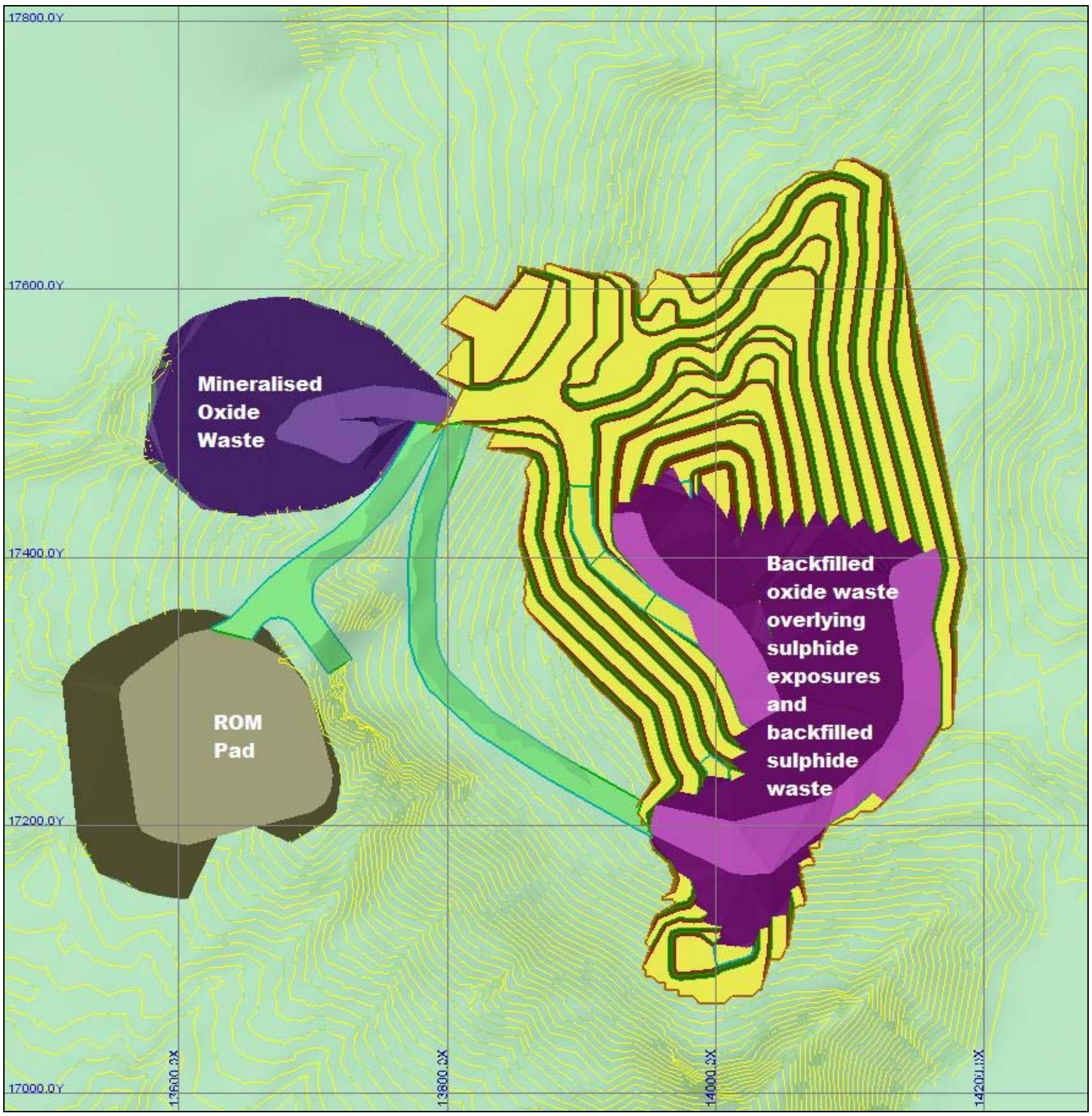
Figure 19-2: Final Pit Design showing Locations of the Waste Dumps and ROM Pad



Of the total waste rock 2.602 Mt, approximately 95% (2.712 Mt) consists of uneconomic oxide waste. The remaining 0.140 Mt (5% of the total waste) consists of potentially acid generating sulphide rock.

As part of closure at the end of the open pit life, the sulphide waste will be backfilled into the deeper parts of the open pit as shown in Figure 19-3. The oxide waste will be utilized during the operating period for tailings embankment fill and will also be used at the end of the operation to cover the backfilled sulphide waste and any remaining sulphide rock exposures in the open pit walls. The small quantity of remaining oxide waste not used for embankments or sulphide rock covering will be reclaimed by contouring and vegetating.

Figure 19-3: Final Pit Backfilled with Sulphide Waste and Final Oxide Cover



Over the mine life, approximately 58% of the oxide waste will be used in the construction of the tailings storage facility (TSF) embankment.

19.1.4 MINING SCHEDULE

A total of 9.378 Mt of ore with an average diluted gold grade of 3.20 g/t will be mined over an eight year life-of-mine.

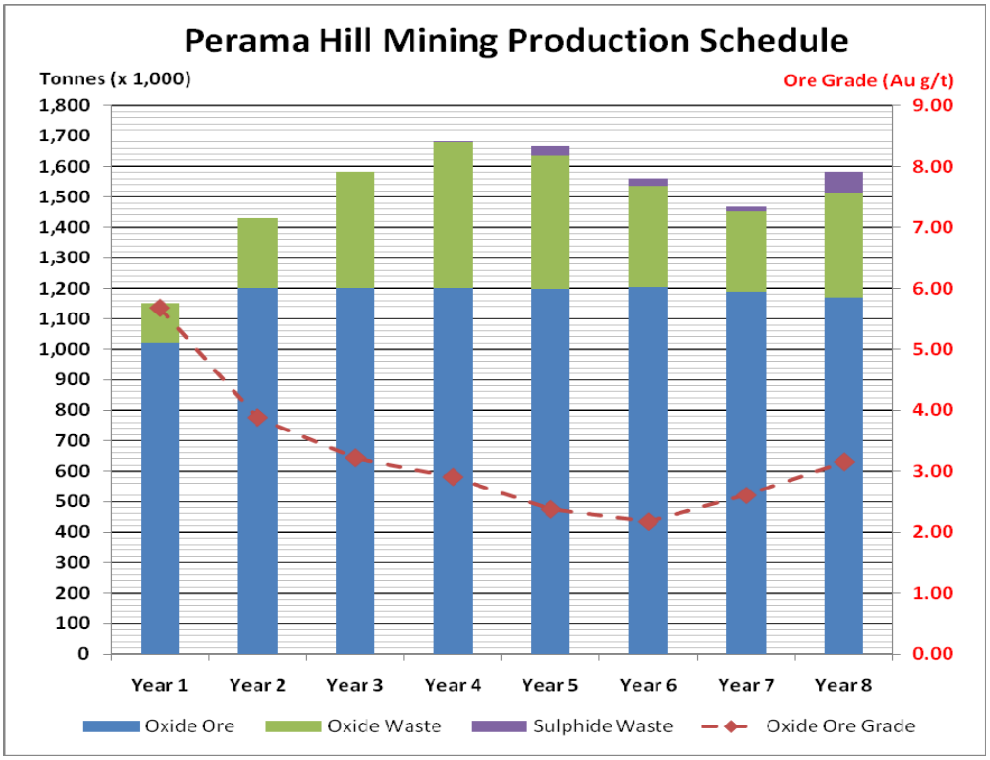
The mining production schedule is based upon the process plant design capacity and start up requirements. For Year 1, ore to ROM pad will be 1.0 Mt, for subsequent years production will increase to approximately 1.2 Mt/a. Table 19-1 and Figure 19-4 show the ore and waste schedule by year.

Table 19-1: Pit Production and Grade

Period	Oxide			Sulphide Waste Tonnage (t x '000)	Total Oxide & Sulphide Ore and Waste Tonnage (t x '000)	Strip Ratio W : O
	Ore		Waste Tonnage (t x '000)			
	Tonnage (t x '000)	Au Grade (g/t)				
Year 1	1,020	5.68	131	0	1,151	0.13 : 1
Year 2	1,201	3.88	228	0	1,429	0.19 : 1
Year 3	1,200	3.22	381	0	1,581	0.32 : 1
Year 4	1,200	2.91	479	3	1,682	0.40 : 1
Year 5	1,197	2.37	439	29	1,666	0.39 : 1
Year 6	1,203	2.17	335	24	1,562	0.30 : 1
Year 7	1,189	2.60	262	17	1,468	0.23 : 1
Year 8	1,169	3.15	346	67	1,581	0.35 : 1
Total	**9,378**	**3.20**	**2,602**	**140**	**12,120**	**0.29 : 1**

Due to the low stripping ratio of waste to ore and the near surface positioning of the orebody, and a net cut requirement in the construction phase, a pre-production period for waste stripping is not required. The production schedule incorporates the early completion of part of the north-western section of the mine by the end of Year 2 for the temporary sulphide waste dump.

Table 19-4: Production and Grade



19.1.5 MINING METHOD AND EQUIPMENT

Using fracture spacing and hardness models a matrix of the two was made to identify the degree of blasting or ripping that will be required. The material is generally soft and fractured. Although a large percentage of the material is expected to be ripped and/or excavated without any need for ripping, a conservative 100% drilling and blasting mining method was selected for equipment requirements and costing. Light and medium blasting will be employed for most of the material with an average powder factor of 0.12 kg/t, hard blasting, which is a fraction of about 1% of the total material would use a powder factor of 0.20 kg/t.

The selected excavating equipment will be two Caterpillar 385C hydraulic backhoes each with 5.5 m^3 bucket capacity loading into a fleet of six Mercedes Actros 4150 8x4 dump trucks each with a payload of 33 tonnes.

For blast hole drilling the Atlas Copco ROC F6 rock drill has been selected which is capable of meeting production targets and will also be used to provide grade control data.

A Caterpillar 980H front end loader (FEL) with a 5.0 m^3 capacity bucket will be used for ROM stockpile duties.

Equipment required to maintain haul roads, waste dumps and for in-pit duties include a Caterpillar D8T dozer, a Caterpillar 14H grader, and a 19,000 litre water bowser for dust suppression.

Equipment requirements also include a number of smaller service and utility vehicles, together with pit pumps and tower lights.

The mine will operate for five days per week, eight hours per day, while the crushing plant for 7 days per week, 16 hours per day. The surge capacity will be provided by a ROM ore stockpile yard. This stockpile area will also provide the opportunity for blending of the different ore types.

19.1.6 MINE PERSONNEL

The open pit personnel required to achieve the proposed production schedule averages 42 employees over the eight year open pit mine life, the breakdown of which includes 11 staff, 21 operating and 10 maintenance employees. Annual personnel levels vary between 38 and 42 employees over the mine life due to variations in production requirements and the truck cycle times.

19.1.7 MINE COSTS

Open pit capital expenditure for mine equipment and the construction of surface haul roads, ROM pad and the explosives magazine to achieve the production target totals $9.70 million over the mine life based on an owner operated mine. This capital cost includes $9.41 million for pre-production expenditure of which $4.23 million is for mine production equipment. All required mobile equipment for the mine will be purchased in the pre-production period. Years 1 to 8 will have a total of $0.29 million for deferred capital costs. Other infrastructure costs for mine shops, offices, warehouse, and a fuelling station are estimated in other chapters of this study.

Over the mine life the unit mine operating costs, excluding contingency, have been calculated at an average of $3.93/t of ore mined or $2.96/t for all material mined including waste. The latter figure varies from a high of $3.67/t during the final year when mining is deep and equipment is less productive to a low of $2.77/t in Year 4. The following Table 19-5-3 shows the operating cost breakdown by principal mining activities;

Table 19-2: Summary of Operating Costs by Activity

Activity	(US$/t mined)
Drilling	0.08
Blasting	0.16
Loading	0.33
Hauling	0.42
Ancillary	0.24
Labour	1.65
General	0.08
Total	**2.96**

19.2 GEOTECHNICAL CHARACTERISTICS

Golder Associates (UK) provided geotechnical input for the Perama Hill Project. This work is summarized in this section.

19.2.1 SLOPE STABILITY

The open pit geotechnical drilling programme consisted of 6 inclined boreholes which provided data to assess appropriate pit slope angles to utilise for the open pit design based on slope stability analysis.

Information from the geotechnical drilling and laboratory testing identified the existence of two dominating lithological units.

1. An overlying quartz-rich (>80% quartz) felsic volcanic sandstone that constitutes the west wall of the pit and is defined as:
    - typically extremely weak to moderately weak
    - up to 130 m thick in south central part of the deposit
    - thickly bedded to massive
2. An underlying completely argillised andesitic volcanic breccias that constitutes the east wall of the pit and is defined as:
    - typically extremely weak to moderately weak
    - consisting of regions that are completely altered to stiff clays.

The geotechnical model that was developed to analyse pit slope stability consisted of oxide and sulphide sandstone, conglomerates and breccias and metamorphic schists. At depth the breccia-schist faulted contact has been altered to graphite schist.

A stability analysis was conducted on interpreted likely geotechnical conditions. Factors of safety were used in the stability analysis for the proposed east and west pit wall slopes. A mean factor of safety of 1.2 was attributed to both east and west walls. This figure appears to be appropriate with the life-of-mine proposed to be 8 years. During the analysis factors of safety ranged for 1.15 to 1.5 for the west wall and 1.2 to 1.6 for the east wall. The maximum height of a slope was identified using a lower bounding factor of safety of 1.0 and maximum batter slope angles were identified for each wall. For each wall potentially different types and size of failure mechanisms were encountered resulting in differing overall slope angles and maximum batter slope angles defined. The recommendation for overall pit slope angles and maximum batter slope angles for east and west pit walls are 30 and 37.5 degrees respectively.

That geotechnical study was based on a previous project study, which included all resources being available for mining. Now the pit is being confined by a 500m wide zone from the Perama village boundary with the new pit design being truncated on the east side. This new design places the east wall well away from a fault zone that defined the eastern edge of the previous pit design. A review of the core and the geotechnical work concluded that where the east wall is near the fault zone (in the southern section), a 32 degree limiting overall slope angle is appropriate. Where the east wall becomes 100 m from the fault, it takes on the global 37.5 degree limiting overall slope angle.

19.2.2 GROUND WATER

Exploration drilling indicated that the entire oxide resource is located above the water table, pit highwalls should be dry. The open pit slope stability analysis assumed dry conditions therefore a programme of ground water monitoring will be carried out during operations, especially immediately after rainfall events to confirm dry conditions.

19.3 RIP AND BLAST CHARACTERISTICS

For the purpose of the open pit design, production and process planning an evaluation of the rip and blast requirements was made. This was done by developing a rippability and hardness model based on 3D interpretations of drill hole data.

The primary inputs in determining rippability for ore and waste rock are hardness and fracture interval spacing. Block models were developed for these two variables, and conditional manipulations were employed to categorise the material according to rippability criteria.

The main conclusions are that the majority of the open pit can be ripped and approximately 7% to 25% of the material within the open pit may potentially require blasting. In considering the productivity of digging equipment and the crusher, the mine plan allows for 60% blasting. For contingency purposes, though, the costs have been evaluated at 100% blasting.

19.3.1 BLAST VIBRATION ANALYSIS

The close proximity of the village of Perama to the proposed open pit area has necessitated an analysis of the possible blast vibration damage and air over pressure effects. Until actual production blast monitoring data is available, Hypodomi 3E Ltd. have recommended utilising a minimum scaled distance of 27 m/(kg explosives)½ as an initial design criteria for production blasting, based on field tests, conservative assumptions, and a contingency factor.

Hypodomi 3E also concludes that it will be possible to control air blasts such that no problems are caused to the residents and/or the houses of the village. To mitigate against the effects of ground vibration and air overpressure a Nonel system will be employed.

19.4 MINE PRODUCTION SCHEDULE

The mine production schedule was produced using the Gemcom GEMS program which links the block model and ultimate pit design with the required production criteria. A mine production schedule based on 5m high benches has been generated indicating quarterly ore and waste tonnages and ore grades for the first two years and annually thereafter.

Figures 19-5 and 19-6 present production ore tonnes and average grade distribution per 5 m bench intervals.

Figure 19-5: Production by Scheduled Bench

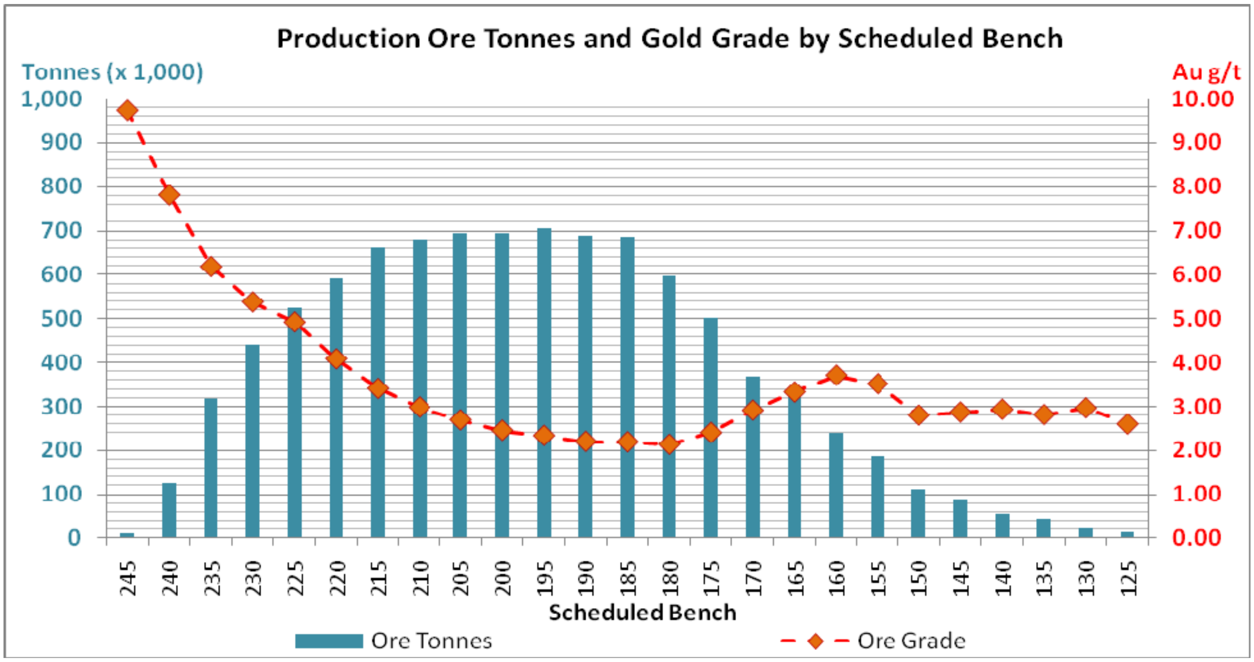
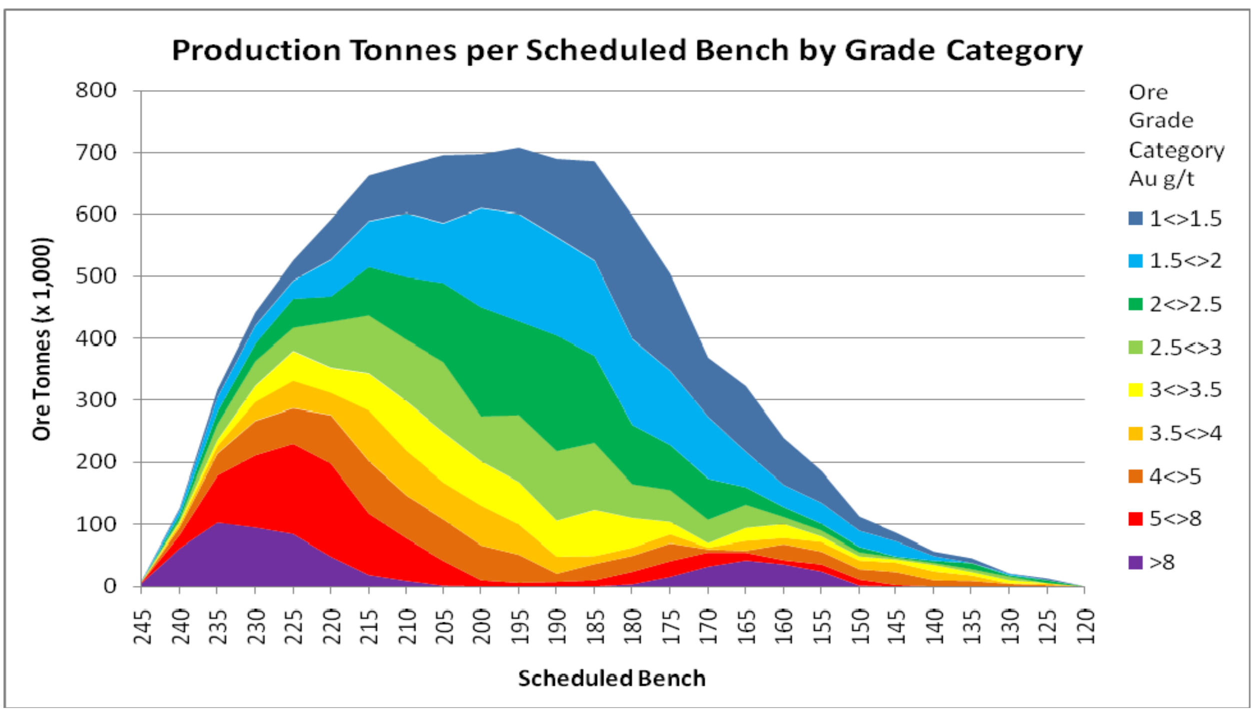


Figure 19-6: Production by Grade Category



With elevated gold grades near surface there is no need to have a starter pit followed by push backs. Mining on a simple top down approach achieves a near optimal NPV, has a constant equipment requirements, and allows for the progressively smooth increase of hard ore over time ensuring that the plant feed will always be less than 100% hard ore. The initial two years will see simultaneous mining on multiple benches as the temporary sulphide waste storage area is excavated and easy access ramps are cut. Thereafter it becomes a simple top down approach to excavating the pit to final depth.

As much as possible each bench will be excavated first from the west to reduce the noise in the direction of Perama village.

The mine is scheduled to operate eight hours per day for 5 days per week (Monday to Friday). The crusher will operate two 8 hour shifts per day for 7 days per week and will be fed with ore by a wheel loader from the adjacent run of mine (ROM) stockpiles. The loader will provide a homogeneous feed to the plant, blended by the required hardness ratios for processing.

Plant feed grade ranges from a high of 5.68 g/t Au during the first year to a low of 2.17 g/t Au in Year 6. These head grades result in contained gold in the plant feed varying from 183,000 oz in Year 1 to 84,000 oz in Year 6. Table 19-3 shows the crusher feed schedule by year.

Table 19-3: Crusher Feed Schedule

	Oxide Ore Feed to the Crusher							
	Tonnage	Au		Ag		Ore Material Type		
Period	(t x '000)	Grade (g/t)	Ounces (oz x '000)	Grade (g/t)	Ounces (oz x '000)	% Soft	% Med	% Hard
Year 1	1,000	5.68	183	3.64	117	24	50	26
Year 2	1,200	3.92	151	2.78	107	36	42	22
Year 3	1,200	3.23	125	2.66	102	32	37	30
Year 4	1,200	2.91	112	3.32	128	35	31	34
Year 5	1,200	2.38	92	3.97	153	29	32	38
Year 6	1,200	2.17	84	3.41	131	29	31	40
Year 7	1,200	2.60	100	4.68	180	22	34	45
Year 8	1,178	3.15	119	5.53	209	10	38	52
Total	**9,378**	**3.20**	**966**	**3.75**	**1129**	**27**	**37**	**36**

19.4.1 ANNUAL MINE PLANS

Mining in the first two years will be conducted in two locations. The northwest part of the pit will be mined to provide an in-pit area that will be used for temporary storage of sulphide waste from Year 4 to Year 8. The main part of mining however will be taking high grade ore from the top of the pit. All work following this will quite simply continue in 5 m vertical benches sequences from the 225m bench downwards.

19.5 MINE INFRASTRUCTURE

19.5.1 SURFACE HAUL ROADS AND RUN OF MINE (ROM) PAD

All haul roads and the ROM pad will be constructed from mine waste and cut material from process plant area. The ROM pad is designed for separate storage of hard, medium and soft ore.

The design for sizing of the haul roads and ROM pad is based on the following design parameters:

- BCM of ore in situ – 2.21 t/m^3
- bulk density of stacked ore – 1.53 t/m^3
- angle of repose of active loading face – 45°
- normal angle of repose – 36°
- minimum road reserve width – 20 m
- maximum road gradient – 10%
- minimum cross fall to road towards drainage ditch – 2%

The design of the ROM pad area is based on the following stockpile operating assumptions:

- minimum turning radius for front end loaders – 15 m
- active loading from only one stockpile at a time
- allowance for direct tipping from truck onto grizzly
- minimum distance from stockpile active face to grizzly structure – 25 m
- operational access way provided around stockpiles a minimum of 20 m wide.

The capital cost estimate for construction of the haul road, ROM pad civil and structural works is based on quantities measured from the layout and detail drawings, and on estimated unit rates for the main construction activities identified. A 3-dimensional computer model using available digitised ground mapping was created to determine the bulk earthworks (cut and fill) volumes for the ROM pad.

19.5.2 EXPLOSIVE AND ACCESSORIES MAGAZINES

The safe storage of mining explosives and ancillary materials involves construction of the earthworks platforms, buildings for storage of ANFO, detonators other ancillary materials and a guard hut at the entrance to the areas. The buildings will be surrounded by a 2 m high fence and pole mounted area lights for security of the explosives storage area.

The design for sizing the ANFO storage magazine, positioning of this building relative to the detonator store, selection of materials and type of construction are all based on statutory requirements. The capital costs for the magazine complex is $161,450.

19.5.3 WORKSHOP AND LAY DOWN AREAS

Facilities needed for the mine equipment maintenance include:

- change house space for 50 operator, maintenance, and engineering staff and hourly employees
- a crane equipped mine workshop with at least 2 truck repair/PM/welding bays, and 3 service equipment bays with an adjacent parts warehouse. An outdoor concrete wash pad with steam or pressure hot water cleaner is needed for cleaning equipment prior to entering the shop. A concrete pad for changing large vehicle tyres is also required
- an outdoor lay down area for large parts, and truck and loader tyres
- a parking area for mobile equipment not in use or awaiting maintenance, and for parking equipment during extended mine shutdowns such as weekends. During the week, it is recommended that mining equipment be left at the work face at the end of the shift and employees transported between change house and working area by van (included in mine support equipment).

19.5.4 FUEL DEPOT

All mine equipment will be diesel operated. Trucks will be fuelled at the fuelling depot, which is located at the plant site. The fuelling storage area will also include drum storage of lubricants and hydraulic oils.

A fuel storage capacity of 25,000 L will cover the peak requirement periods based on deliveries twice weekly.

A small fuel-lubrication truck is included for fuelling the mining fleet during operations.

19.5.5 DUST CONTROL

A 19,000-L water truck is included for dust suppression for both surface haulage roads, in-pit ramps and for the production drill rig. A water discharge pipe in the yard area will be supplied to fill the water truck.

19.5.6 PIT DEWATERING

The water table is expected to be located below the pit floor elevation level therefore pumping of groundwater is not to be expected. With the open pit situated on Perama Hill, surface water runoff will be managed principally by natural drainage away from the pit with short diversion channels along the pit perimeter where needed. The lower benches in the pit will be below the surrounding topography so some pumping is anticipated beginning in Year 6 when the southern part of the pit is reduced below the surrounding topography. An in-pit pump in temporary sumps within the pit floor will discharge water that results from direct precipitation and minor inflows from below the diversion channels via a pipeline to the surface diversion channel.

A high capacity diesel pump and pipeline is included in the mine capital cost allocations.

As the bottom of the pit is well above the water table no groundwater seepage is expected.

19.5.7 SURFACE WATER DIVERSION CHANNEL

A surface water diversion channel has been provided around the perimeter of the open pit and waste dump areas to contain surface water run-off and in-pit water. Mining and storage of sulphide material is a potential for acid rock drainage (ARD) and other contaminants from the site during operations. Any contact water will be diverted to collector sump and treated prior to being added to process water. Non-contact water will undergo sedimentation control.

19.6 WASTE DUMP DESIGN

Two types of waste material will be produced during the life-of-mine, oxide and sulphide. Low grade mineralized oxide rock, which is below cutoff grade but may have future potential will be segregated during grade control and stored separately within the oxide dump area. The precise location of this material will be known and easily reclaimable in case it becomes economic to treat it. Sulphide waste will be stored in a separate temporary waste dump, which will be positioned inside the footprint of the open pit.

The maximum waste rock quantities generated over the life-of-mine are shown in Table 19-4. These quantities have been derived from the mine production schedule.

Table 19-4: Waste Rock Schedule

Year	1	2	3	4	5	6	7	8	Total
Oxide Waste (t)	119,074	214,467	369,407	449,514	419,566	291,549	178,482	241,119	2,283,178
Mineralized Oxide (t)	12,183	13,994	12,070	29,165	19,767	43,039	83,903	104,418	318,540
Au grade of Mineralized Oxide (g/t)	0.84	0.84	0.88	0.84	0.87	0.87	0.90	0.87	0.87
Sulphide Waste (t)	-	-	-	2,987	29,239	24,204	16,542	66,931	139,902
Au grade of Sulphide Waste (g/t)	-	-	-	1.68	1.94	2.39	3.81	5.02	3.71
Total (t)	**131,257**	**228,461**	**381,477**	**481,666**	**468,572**	**358,791**	**278,927**	**412,468**	**2,741,620**

19.7 MINE OPERATION

Provision for 100% drill and blast mining method has been ensured although ripping and free-dig excavation will form part of the mining. Production drilling will also provides samples for grade control purposes. Hydraulic excavators and highway tipper trucks make up the production fleet. The bench height is planned at 5 m.

A CAT 980H with a 5.0 m^3 bucket Front End Loader (FEL) will be employed to feed the primary crusher from the ROM stockpiles. To optimise plant performance, the crusher will be supplied with a blend of ore from hard, medium and soft ore from the ROM stockpiles and directly from the pit when suitable.

In pit loading will be carried out with two Caterpillar 385C L excavators with 5.5 m^3 buckets. Based on the excavator productivity and expected availability, the amount of scheduled hours per year will range from 2,078 in Year 1 to a high of 3,373 in Year 8, and averaging 2,827 hours per year for the full eight years. Therefore a fleet consisting of two excavators will be required with an average utilisation of available scheduled hours being 78%.

For production haulage, six Mercedes Actros 4150-AK 8x4 rear dump trucks with a nominal capacity of 22 cubic metre box and 33 tonnes payload dump trucks are required for the life of the operation.

19.7.1 WASTE DUMPS AND HAUL ROADS

Haulage trucks will end dump waste material at the toe of the dump for the first lift and a dozer will subsequently level the waste material as required to maintain dump profiles. Access roads to the dump will be developed from the main haul road during the life-of-mine as and when required. A Caterpillar D8 dozer is included for waste dump, stockpile and in-pit duties. A CAT 14M grader will maintain surface and in-pit roads.

Sulphide waste will be encountered in Years 5 to 8. Such waste will be stored temporarily in the temporary sulphide waste dump area, which is a 1.1 ha part of the upper benches of the open pit that will be fully mined out by the end of Year 2. At the end of the mine life the sulphide waste will be transported back into the open pit to the lower benches. Once that is all complete it will be capped with oxide waste. The oxide waste capping in the open pit will also extend to any pit walls that have exposed in situ sulphide rock. The amount of oxide waste coming out of the pit will be sufficient for the purpose of tailings embankment and capping of sulphide rock/exposures in the open pit.

Approximately 6.7 ha will be required for oxide waste storage. This includes the uneconomic but mineralized oxide waste. The mineralized waste and any remaining oxide waste will be contoured and topsoiled to promote revegetation.

19.7.2 GRADE CONTROL

During mining operations, all production blast hole cuttings will be sampled and assayed on 5 m intervals. Operating cost estimates include utilising the production drill rig for grade control drilling.

The ore above cutoff grade in the pit will be visibly marked up for the operator to distinguish between ore and waste, a geologist will also be present to ensure mineralized ore is excavated and not designated as waste or left in the pit.

A minimum of three ore types and three waste types have been identified, and destinations specified, including:

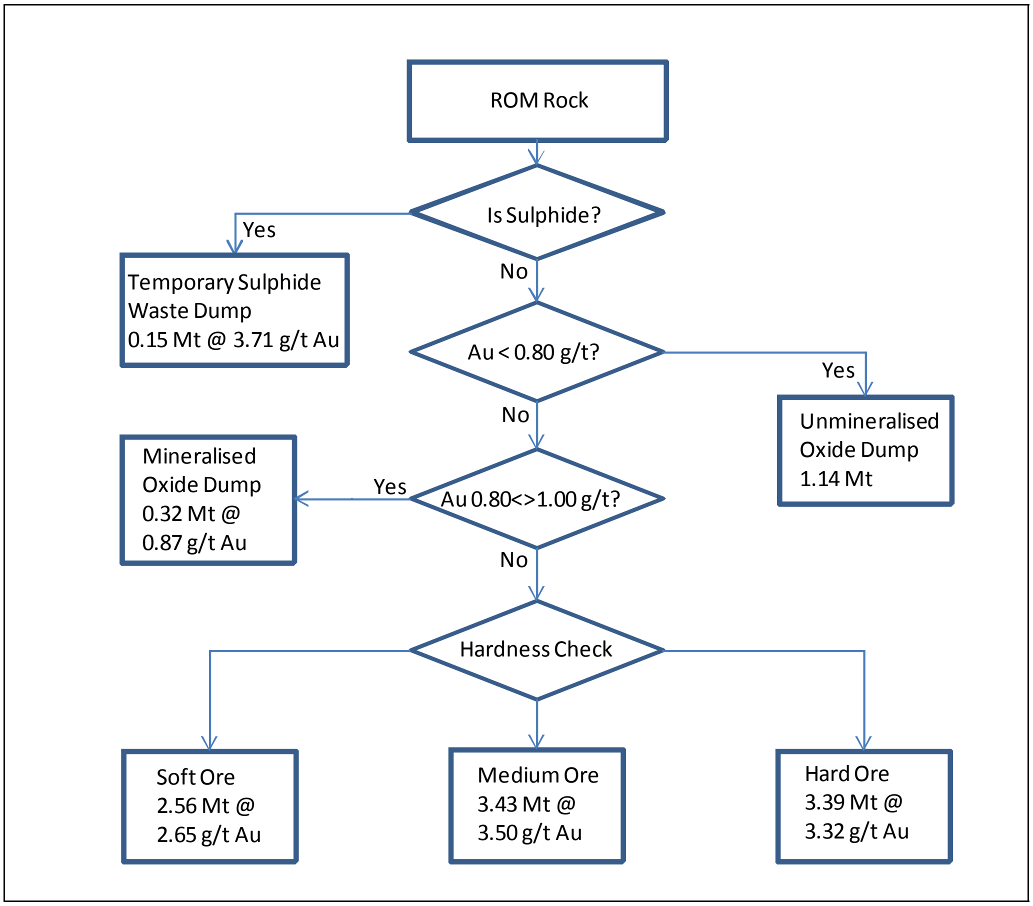
Oxide mineralisation above mill feed cutoff grade is identified as soft, medium and hard ore. This ore will be hauled to the jaw crusher bin or stockpiled on one of three ROM stockpile locations segregated by hardness. Blending of the crusher feed by hardness will optimise mill performance. Figure 19-7 shows the annual expected tonnages by hardness group.

Oxide waste containing gold mineralisation below 1.00 g/t (diluted) but above the marginal cutoff grade of 0.80 g/t has been quantified and will be identified during mining so that it can be segregated into a separate area of the oxide waste dump. A total of 319,000 tonnes at a grade of 0.87 g/t is expected to be in this category. Most of this will be mined in the final two years of the mine life. This material is considered to have no economic value.

Oxide waste below 0.80 g/t will be placed on the main oxide waste dump or, in some instances, will be hauled directly to the tailings embankment.

Sulphide waste rock will also be identified so that is can be disposed of separately in the temporary waste dump. The sulphide rock will only be encountered in the mining period of Years 5 to 8.

Figure 19-7: Mined Material Flow sheet



Assaying will be performed at the process plant assay laboratory on site. An average of forty eight samples will be taken daily which equates to approximately 12,000 samples per annum.

19.8 PROJECT INFRASTRUCTURE

19.8.1 SERVICES

The city of Alexandropolis is a major hub for agricultural and industrial activity in the area. Most major services will be supplied by local businesses in Alexandropolis. An industrial based workforce does exist in the area to provide skilled labour. Training of local villagers will provide a pool of equipment operators and plant operators.

19.8.2 BUILDINGS

Site infrastructure will be made up of the following facilities:

Truck Shop..	Mobile equipment maintenance and warehousing.
Office Buildings	Technical and administrative services.
Explosive Magazine	Explosive storage.
Crushing Plant..	Run of mine ore crushing.
Process Plant	Flotation and cyanidation plant.
Visitor Centre	Public relations and training centre.
Analytical Laboratory...	Sample preparation and analysis.
Canteen..	Food services.

In general these buildings will be of steel frame on concrete footings.

19.8.3 POWER

Power supply for the plant site operations will be provided at 20 kV. Power will be obtained from the national grid via a 15 km dedicated power-line, and then stepped down for distribution on the plant site at 6.0KV.

19.8.4 WATER

Water for general use will be supplied from a borehole drilled within the plant site area. Process water will be recycled from the filtration plant back into the recovery process. Contact and non-contact water on the plant site will be segregated to allow non-contact water to return to the natural water course.

19.8.5 SEWAGE

A biological treatment plant or RBC will be installed at the site. Capacity of the plan will be in excess of the total planned workforce.

19.9 PROCESS OPERATIONS

Process operations have been discussed in detail in Sections 16.14 and 16.15.

19.10 RECOVERABILITY

Perama Hill is a project that has no past operating results. The expected metallurgical recovery of gold and silver is 90% and 60%, respectively. Section 16 of this document provides the details of past testwork results.

19.11 ENVIRONMENTAL CONSIDERATIONS

A pre-cursor to the EIA, the Preliminary EIA (PEIA), was submitted to the Ministry of Environment in 2009. The PEIA details the description of the environment, description of the Project, and an evaluation and assessment of environmental impacts. These impacts include landscape and visual, soil, land cover, surface water and ground water.

A study done by Hypodomi 3E Ltd has identified the impact such as dust, noise and vibration, that blasting in the open pit can have on the local community. As a result blasting practices will follow the recommendations of that study.

19.12 MARKETS

The expected saleable products of the Perama Hill Project are limited to gold and silver in doré bars with gold representing nearly 99% of the total value of sales.

19.12.1 SUPPLY-DEMAND BALANCE

Global gold mine production in 2009 increased 6.0% from 2008 production and central banks' net sales of gold decreased by 89.8% in 2009. Central banks in the Washington Agreement reduced their selling activity, well below the 500 tonne annual threshold they had agreed to for 2009. Central banks outside of the Washington Agreement were net purchasers of gold in 2009. Recycled gold increased by 26.6% over 2008 levels, supplying over 1,500 tonnes of gold to the market. High gold prices are encouraging people to sell their unwanted jewellery and other items made of precious metals. Overall, the limited supply of gold to the market has been a positive influence on the price of gold, as mine supply has been flat to falling slightly, and central banks have reduced their selling activity.

Gold prices rose significantly in many of the traditional gold market currencies such as the Indian rupee and many Far East currencies. Jewellery and fabrication demand was 19.4% lower in 2009 compared with 2008 largely as a result of higher prices. While bar hoarding and producer dehedging were also lower during 2009, investment demand was over 480% higher with 1,375 tonnes of demand. Overall demand was 6.6% higher than in 2008. If the US dollar continues to be weak and gold prices remain high, fabrication and jewellery demand are not expected to be strong in the coming year. Investment demand and bar hoarding will have to increase to keep the market balanced.

19.12.2 PRICE

The price of gold is the largest single factor in determining profitability and cash flow from operations, therefore, the financial performance of the project has been, and is expected to continue to be, closely linked to the price of gold.

Historically, the price of gold has been subject to volatile price movements over short periods of time and is affected by numerous macroeconomic and industry factors that are beyond the Company's control. Major influences on the gold price include currency exchange rate fluctuations and the relative strength of the US dollar, the supply of, and demand for gold and macroeconomic factors such as the level of interest rates and inflation expectations. During 2009, the price of gold hit a new all time high of approximately $1,226/oz. The low price for the year was $802/oz. The average price for the year based on the London PM Fix was $972/oz, a $100 increase over the 2008 average price of $872/oz. The major influences on the gold price during 2009 were continuing strong investment demand in physical gold bars as well as gold linked instruments, further producer de-hedging, the global financial crisis that continues to unfold, and declining supply from central banks.

19.13 CONTRACTS

There are no current contracts for the Perama Hill Project.

19.14 TAXES

Current Greek regulations include a corporate tax of 25% and a NSR at 2.5% of gold and silver production. These factors have been included in the economic analysis.

19.15 ECONOMIC ANALYSIS

The economic evaluation indicates an after tax internal rate of return (IRR) of 23% and an after tax net present value (NPV) of US$111 million at a discount rate of 5.0%.

19.15.1 OPERATING COSTS

Operating costs have been worked out in detail and the unit costs (in US$/t milled) of the major components are summarized in Table 19-5.

Table 19-5: Operating Cost by Component

Costs US$/Tonne Milled:	
Mining Costs	
Drilling	0.11
Blasting	0.21
Loading	0.43
Hauling	0.54
Rehandle	0.08
Ancillary	0.31
Labour	2.14
General	0.10
Total opex	**3.93**
Plant Costs	
Salaries and Wages	3.20
Consumables	6.05
Power	3.60
Tailings	1.88
Maintenance Supplies	1.43
Total	**16.16**
General & Admin Costs	
Salaries and Wages	2.55
Power	
General Supplies	0.05
Vehicles	0.04
Accommodation	0.20
Community Support	0.83
Business Insurances	0.83
Services of part time doctor	0.17
Expatriate Differential	0.24
Total	**4.91**
Total Operating Costs	**25.00**

19.15.2 CAPITAL COSTS

Capital costs have been worked out in detail and a summary of the major components are shown in Table 19-6.

Table 19-6: Capital Cost Summary

	Cost (US$ x '000)
Equipment	53,383
Installation	63,174
Direct	7,348
Indirect	40,173
Total Pre Production	**164,078**
Sustaining	23,291
Total Capital	**187,369**

19.15.3 CASH FLOW FORECAST

The cash flow analysis for the Perama Hill Project is presented in Table 19-7.

19.15.4 SENSITIVITY ANALYSIS

Sensitivities to metal price, capital cost, and operating cost on the IRR and NPV were considered on the post-tax base case model. Figure 19-8 shows the sensitivity trends for Gold Price, Operating Costs, and Capex. The gold grade, gold price and gold metallurgical recovery sensitivity trends were all equivalent. There was very little NPV sensitivity for the respective silver trends.

The project is most sensitive to Gold price. The project NPV is slightly more sensitive to capital cost than to operating cost.

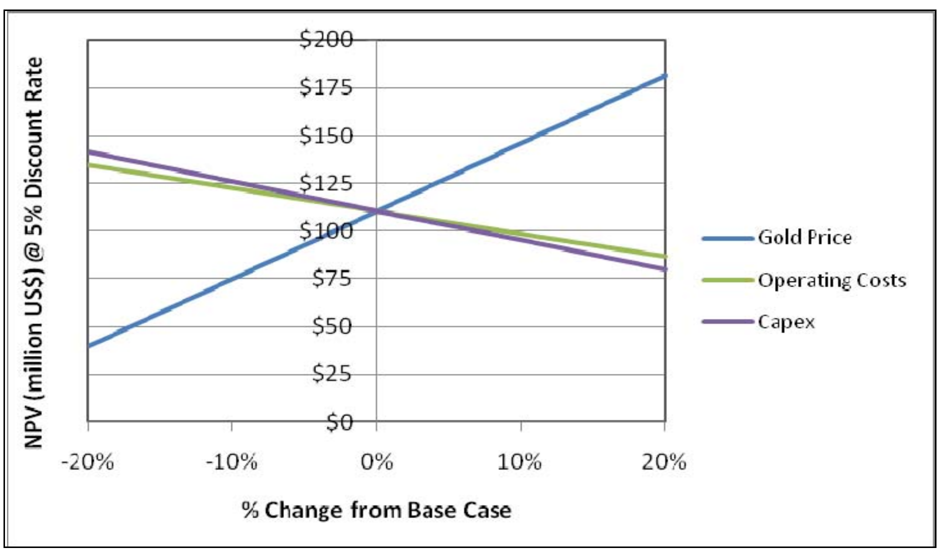
19.15.5 PAYBACK

The payback period for the base case is 2.5 years. This is the time required after revenue is first received in Year 1 to achieve break-even cumulative cash flow. The payback period is based on the annual un-discounted cash flows.

Table 19-7: Cash Flow Forecast

			Pre Production		Production								
OPERATIONS			Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Ore mined		000's tonnes			1,020	1,201	1,200	1,200	1,197	1,203	1,189	1,169	9,378
Au grade		g/t			5.68	3.88	3.22	2.91	2.37	2.17	2.60	3.15	3.20
Ag grade		g/t			3.66	2.74	2.66	3.33	3.99	3.40	4.69	5.55	3.75
Waste mined		000's tonnes			131.3	228.5	381.5	481.7	468.6	358.8	278.9	412.5	2,742
Total material		000's tonnes			1,151	1,429	1,581	1,682	1,666	1,562	1,468	1,581	12,120
Strip ratio					0.13	0.19	0.32	0.40	0.39	0.30	0.23	0.35	0.29
METAL PRODUCED													
Gold		000's ozs			167.8	134.9	111.8	101.0	82.2	75.6	89.5	106.7	869.4
Silver		000's ozs			72.0	63.6	61.6	77.1	92.0	78.8	107.6	125.0	677.7
OPERATING COSTS													
Mining		US$ 000			4,392	5,571	4,839	4,905	4,886	4,776	4,862	4,554	37,743
Processing		US$ 000			19,238	19,238	19,861	19,790	20,176	20,117	20,183	19,608	159,067
G&A		US$ 000			5,898	5,898	5,898	5,898	5,898	5,898	5,898	5,898	47,182
Dore transport and refining		US$ 000			1,342	1,079	894	808	657	605	716	853	6,955
Silver credit		US$ 000			(990)	(874)	(847)	(1,059)	(1,266)	(1,084)	(1,479)	(1,719)	(9,319)
Total		US$ 000			29,880	30,913	30,645	30,340	30,352	30,312	30,179	29,193	241,815
Cash Cost		US$/oz			178	229	274	301	369	401	337	274	278
EXPENDITURES													
Capex		US$ 000	82,039	82,039	250	200	5,600	200	630	5,500	100	10,811	187,369
Property payment		US$ 000			3,000								3,000
Working capital		US$ 000			3,000							(3,000)	-
PROFIT AND LOSS													
		US$ 000											
Gold Revenue					138,407	111,316	92,236	83,295	67,779	62,370	73,875	87,987	717,265
Operating cost					(29,880)	(30,913)	(30,645)	(30,340)	(30,352)	(30,312)	(30,179)	(29,193)	(241,815)
NSR (Au + Ag)	2.50%				(3,485)	(2,805)	(2,327)	(2,109)	(1,726)	(1,586)	(1,884)	(2,243)	(18,165)
Income before D&A					105,042	77,598	59,264	50,845	35,701	30,472	41,812	56,551	457,285
Depreciation & Amortization (straight line)					(23,421)	(23,421)	(23,421)	(23,421)	(23,421)	(23,421)	(23,421)	(23,421)	(187,369)
Income before tax					81,621	54,177	35,842	27,424	12,280	7,051	18,391	33,130	269,916
Income tax	25%				(20,405)	(13,544)	(8,961)	(6,856)	(3,070)	(1,763)	(4,598)	(8,283)	(67,479)
Net Income					61,216	40,633	26,882	20,568	9,210	5,288	13,793	24,848	202,437
Free Cash Flow After Tax		**US$ 000**	**(82,039)**	**(82,039)**	**78,387**	**63,854**	**44,703**	**43,789**	**32,001**	**23,209**	**37,114**	**40,458**	**199,437**

Figure 19-8: NPV Sensitivity Analysis



19.16 MINE LIFE

The mine will operate for 8 years from initial commissioning.

19.17 CONCLUSIONS AND RECOMMENDATIONS

- an ore body located at surface with a low stripping ratio amenable to open pit mining
- a proposed mining equipment fleet and mining method currently in use around the world at many mines and quarries
- the opportunity to mine high grade ore during the early years of the life-of-mine
- favourable climate and an easily accessible location
- regional infrastructure and population base capable of supplying the necessary services and personnel
- it is recommended that trial mining is carried out to verify whether or not ripping can be carried out successfully
- pit wall stability is based upon dry conditions therefore ground water monitoring should be carried out to confirm that the water table is below the pit floor level
- the project has attractive NPV and IRR with relatively low initial capital costs.

SECTION 20 • CONCLUSIONS, RISKS, OPPORTUNITIES, AND RECOMMENDATIONS

20.1 CONCLUSIONS

The geology of the Perama Hill gold deposit is well understood. The deposit is considered to be a stratabound, sediment hosted deposit formed by a high sulphidation epithermal system overprinted by a low sulphidation system.

A comprehensive QAQC program was employed for gold analyses on drill core and RC samples. Eldorado Gold concludes that all the assay values that support the Perama Hill Mineral Resource estimates were in control. The effect of a low analytical bias observed during parts of the diamond drill campaign is believed to have minimal impact on the Mineral Resource estimates.

There is potential to increase the Perama Hill Deposit Measured and Indicated mineral resources by as much as 5% to 10% by drilling more drill holes that target the areas with Inferred Resources and the areas along the western and eastern margins that are not completely closed off yet.

A small conventional open pit mine is proposed for Perama Hill. The open pit has an easily mined resource located at surface, entirely above the water table and with a low strip ratio. It is situated in an easily accessible location with a favourable climate and a regional infrastructure and population base capable of supplying the necessary services and personnel. From a mining perspective, the operation is expected to be straightforward and productive.

Sufficient testwork has been performed, by competent laboratories, to establish that the Perama Hill orebody can be processed by a conventional process plant, comprising crushing, milling, CIL, elution, and INCO cyanide detoxification, similar to many other gold oxide ore plants around the world. Confirmation was given that the gold and silver leach rapidly in cyanide, giving recoveries of about 90% Au and 60% Ag.

A pre-cursor to the EIA, the Preliminary EIA (PEIA), was submitted to the Ministry of Environment in 2009. The PEIA details the description of the environment, description of the Project, and an evaluation and assessment of environmental impacts. A decision is expected in 2010.

The estimates of the capital and operating costs of the project have been based on a high level of engineering in all areas of the operation. Using the cost estimates and ensuring that adequate contingencies are applied to all areas of the cost build-up, this project offers the potential of a robust return on investment to the participants.

The economic evaluation indicates an after tax internal rate of return (IRR) of 23% and an after tax net present value (NPV) of US$111 million at a discount rate of 5.0%.

20.2 RECOMMENDATIONS

Further evaluation of the nearby Perama South deposit should be conducted to assess its potential of adding mine life to the Perama Hill Project.

To verify rock excavatability forecasts it is strongly recommended that extensive testing and field trials be conducted. Seismic velocity testing as advocated by Caterpillar should be considered.

A ground investigation for the waste dump foundation is recommended to include geotechnical testwork of foundation material and oxide waste material.

Pit wall stability is based upon dry conditions therefore ground water monitoring should be carried out to confirm that the water table is below the pit floor level.

A review of a SAG/Ball Mill circuit as an alternative to three stage crushing schedule should be carried out prior to initiation of process engineering as it may reduce the capital cost for the project.

The law in Greece requires that a procedure of licensing abstraction boreholes must be followed. This can be time consuming. Any additional borehole sites should be selected as soon as possible and the necessary permits obtained.

A detailed review of the seismic stability of the TMF is recommended. Subsurface boreholes and borrow material investigation will be required to substantiate both the detail design and cost estimates for earthworks and construction.

Some sulphide waste rock is mined which may be acid generating will be mined during later stages of operation. This waste rock will be segregated in one area of the ultimate pit so that run-off can be controlled and directed to the Neutralization Plant.

The time taken for obtaining the necessary permits is a major risk in this project. The time allocated within the schedule is reasonable as far as present information dictates, but delays do have an impact on the project. In order to mitigate the potential risk of delay to the issue of the relevant permits caused by inadequate information, it is recommended that early stage engineering activities be undertaken to support the permit applications.

SECTION 21 • REFERENCES

Mc Alister, M., Hammond, J.M., Normand, D. & Kampasakalis, M., 1999. Discovery case history for the Perama Hill gold deposit, Greece. In: New Generatino Gold Mines Conference in Perth, West. Aust., 22-23rd Nov 1999.

Scott Wilson, 2004. Report on Perama Hill Gold deposit mineral resource estimate, Greece. National Instrument 43-101 Technical Report May 2004

SECTION 22 • SIGNATURE PAGE AND DATE

The effective date of this report entitled "Technical Report for the Perama Hill Project, Greece" is 28 January 2010. It has been prepared for Eldorado Gold Corporation by S. Juras, P.Geo., Richard Miller, P.Eng., and Peter Perkins, BSc, MIMMM, MSAIMM, each of whom are qualified persons as defined by NI 43-101.

Signed the 15th day of March 2010.

SIGNED

"Richard Miller"	*"Stephen Juras"*	*"Peter Perkins"*
Richard Miller, P.Eng Manager, Mining Eldorado Gold Corp.	Stephen Juras, PhD, P.Geo Director, Technical Services Eldorado Gold Corp.	Peter J. Perkins, BSc, MIMMM, MSAIMM Technical Manager Acker Solutions E&C Ltd.

SECTION 23 • CERTIFICATES AND CONSENTS

CERTIFICATE OF QUALIFIED PERSON

Stephen J. Juras, P.Geo
1188 Bentall 5, 550 Burrard St.
Vancouver, BC
Tel: (604) 601-6658
Fax: (604) 687-4026
stevej@eldoradogold.com

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

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

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

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

I was responsible for reviewing matters related to the geological data, the mineral resource estimation and classification work for the Perama Hill Project in Greece. The report entitled *Technical Report for the Perama Hill Project, Thrace, Greece, ,* with an effective date of January 2010, was prepared under my supervision (sections 7, 8, 9, 10, 11, 12, 13, 14, 15 and 17,sub-section 1). I visited the project on September 18 to 19, 2009.

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

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.4 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the sections for which I am responsible in this report entitled, *Technical Report for the Perama Hill Project, Thrace, Greece,* with an effective date of January 2010, has been prepared in compliance with same.

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

Dated at Vancouver, British Columbia, this 19th day of March 2010.

"Stephen Juras"

____

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

CERTIFICATE OF QUALIFIED PERSON

Richard Miller, P.Eng
1188 Bentall 5, 550 Burrard St.
Vancouver, BC
Tel: (604) 601-6671
Fax: (604) 687-4026
richardm@eldoradogold.com

I, Richard Miller, am a Professional Engineer, employed as Manager, Mine Engineering, of Eldorado Gold Corporation and residing at 832 Victoria Drive in the City of Port Coquitlam in the Province of British Columbia.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I graduated from the University of British Columbia with a Bachelor of Applied Science degree through the department of Mining and Mineral Process Engineering in 1987.

I have practiced my profession continuously since 1987 and have worked at copper, diamond and gold mines in Canada, South Africa, Namibia, Guinea and Turkey in the capacities of Mining Engineer, Project Manager and Mine Manager covering planning, surveying, production, contract management, department head and global manager covering operations in Turkey, Brazil and China. I have also consulted to mining related companies in Canada, Dominican Republic, Burkina Faso, Serbia and Russia.

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

I was responsible for reviewing matters related to the mining operations and directing the mineral reserve estimation work for the Perama Hill Project in Greece. The report entitled *Technical Report for the Perama Hill Project, Thrace, Greece,* with an effective date of January 2010, was prepared under my supervision (sections 17 sub-section 2, 18 and 19). I visited the project on September 18 to 19, 2009.

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

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.4 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101FI and the sections for which I am responsible in this report entitled, *Technical Report for the Perama Hill Project, Thrace, Greece,* with an effective date of January 2010, has been prepared in compliance with same.

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

Dated at Vancouver, British Columbia, this 19th day of March, 2010.

"Richard Miller"

____

Richard Miller, P.Eng

Date: 19 March 2010

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Fax: +44 (0)1642 334029
e-mail: peter.perkins@akersolutions.com

CERTIFICATE OF QUALIFIED PERSON

I, Peter Perkins, do hereby certify that:

1. I am a Technical Manager of Aker Solutions E&C Ltd, Phoenix House, 3 Surtees Way, Surtees Business Park, Stockton on Tees, TS18 3HR, United Kingdom.
2. I was responsible for Chapter VI, “Metallurgy and Treatment Plant” in both the Perama Hill Gold Project Optimization Study Report, December 2000 and subsequently the Feasibility Study Report, November 2008, from which the Mineral Processing and Metallurgical Testing section of the Eldorado Gold Corporation Technical Report, Perama Hill Project Thrace, Greece, dated 19 March 2010 is derived. I have reviewed this section of this Technical Report and accept responsibility for the contents thereof.
3. I graduated with a BSc (Eng) degree in Mineral Technology from the Royal School of Mines, Imperial College, London in 1968.
4. I am a current professional member of the Institute of Materials, Minerals and Mining (IMMM), a member of the Southern African Institute of Mining and Metallurgy (SAIMM), a Chartered Engineer (UK) and a Professional Engineer (South Africa).
5. I have practiced my profession continuously since 1968. Since 1986 I have been employed by Aker Solutions (previously Davy McKee, Davy International, Kvaerner and Aker Kvaerner) continuously on a variety of mineral projects including many as Lead Process Engineer for Feasibility Studies or Design and Construction projects for gold deposits.
6. 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, professional membership of IMMM and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
7. I visited the Perama Hill site during the execution of the December 2000 Optimization Study.
8. I am independent of Eldorado Gold Corporation in accordance with section 1.4 of NI 43-101.

9. I have had prior involvement with the property that is the subject of the Technical Report in that I have previously contributed to a number of feasibility studies on the Perama Hill Project on behalf of Aker Solutions (previously named Kvaerner E&C and subsequently Aker Kvaerner).
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. As of the date of this certificate, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Stockton on Tees, UK, this 19th day of March 2010.

"Peter Perkins"

____

Peter J. Perkins, BSc (Eng), ARSM,
CEng, PrEng, MIMMM, MSAIMM.

APPENDIX A

REQUIRED LICENSES AND PROCEDURES FOR THE PERAMA GOLD PROJECT

	License	Duration	Status	Submission Reg. No.	Decision Reg. No
1.0	Mining Exploration License	3 years with 1 year renewable possible	COMPLETED		
2.0	Mining Title Acquisition	50 years with possible extension of 25 years	COMPLETED		Presidential Decree FEK 2182/B/20.12.1999
3.0	Environmental Permits License	Renewable every 5-10 years, or with modification of mine exploitation or process plant	PEEE (3.1)	PEIA Study Submission: MOE 145829/30.10.2009	
4.0	Mine Operation License	Renewable with modification of mine exploitation or process plant			
5.0	Construction & Operation Licenses	Renewable every 3-5 years, or with modification of installations or machinery foundation and equipment			

1.0 Mining Exploration License Acquisition

1.1 Procedure

The Mining Exploration License (MEL) (max. 10,000 stremmata or 1000ha) is obtained following the submission of the relevant application (the procedure is described in article 20 of the Mining Code and the necessary documents are specified in article 21 of the Mining Code) submitted to the competent Prefecture.

Granting a MEL provides an entity with the right to perform exploration on a specifically defined area, and subject to its results, as analysed in the Feasibility Study 2.2.2, the Project Owner request the conveyance of the respected site for exploitation.

The duration of the MEL, issued by the Prefecture, is 3 years. Extension of the duration of the MEL is not allowed. If exploration results, as presented in the Feasibility Study (2.2.2), are not satisfactory or convincing enough, then the Minister of Environment can grant a one-year prolongation period for the completion of the file.

1.2 Documents

Section			
1.2.1	Ownership Land rights		
1.2.2	Land Use		

1.3 Studies

	Submitted Documents	Status	Reg. No.
1.3.1	Request	SUBMITTED	
1.3.2	Environmental Impact Assessment Study for exploration	SUBMITTED	
	Mining Exploration License Acquisition	APPROVED	

2.0 Mining Title Acquisition

2.1 Procedure

Application for a Mining Title (MT) is initiated after completion of the exploration phase, provided that exploration results prove the existence of a deposit that can be developed. The procedure includes the following:

Drafting of a Feasibility study proving that the Project Owner intends to exploit the reserves established during the previous Stage of MEL

Following completion of Mineral Exploration and provided that sufficient proof is given that a deposit that can be developed exists, a request (Mining Title Application with supporting documents such as results of exploration and economics) is submitted to the relevant Prefect to grant a Mining Title (as per art. 45 & 47 of the Mining Code).

The Prefecture transmits the file to the Department in charge of Mines at the Ministry of Environment, Energy and Climate Change (MOE)

The Ministry makes a request for advice/audit from IGME (Institute of Geological and Mineral Exploration)

The Ministry, on the basis of the published results and the audit by IGME, will judge whether an exploitable deposit has been put into evidence and whether the applicant will proceed to a rational and complete exploitation of the mineral wealth to the benefit of the Greek economy. If its opinion is positive, the file is returned to the Prefecture for publication of the application. If the exploration results are not satisfactory or convincing enough, a one-year prolongation period is granted and the file is then returned to the Prefecture.

The Prefect, after the return of the file, issues an announcement (proclamation) which is also published in the Government Gazette (F.E.K.). If there are no claims after 40 days contesting ownership of the Mineral Rights, the file is sent back to the Ministry of Environment, Energy and Climate Change (MOE)

The Minister issues a Presidential Decree to be signed by the President of the Hellenic Democracy, which is subsequently published in the Government Gazette (F.E.K.)

The required time from the day of submission of the request to the day of publication of the Presidential Enactment of the MT, is 10-12 months.

The duration of the MT is set to 50 years, with a possible extension of 25 years

2.2 Studies

	Submitted Documents	**Status**	**Reg. No.**
2.2.1	Request	SUBMITTED	
2.2.2	Feasibility Study with final results of Exploration	SUBMITTED	
	Mining Title Acquisition Decision	APROVED	Presidential Decree FEK 2182/B/20.12.1999

2.2.2 Feasibility Study with final results of Exploration

Feasibility Study must include the following:

Extended Summary

General Information (Company details, project owner, brief description of the company, general project details, legal framework, study team)

Description of the area of the Mining Exploration Licence. Determination of the location of mining exploration, as well as way of access to them.

Geology. Geological and Mineral Study of the explored areas, supported by the necessary geological sections.

The Exploration Program. Description of the performed studies, research and relevant expenses.

Calculation of Mining Reserves. Description of the type and the size of the explored ore, as well as the calculated and possible deposits.

Mine Design. Preliminary feasibility study of exploiting the explored ore as well as the suggestions for the development prospects of the mining site.

Process Plant. The proposed installations to be constructed, works and the relevant expenses for the commencement of the mine operation and its exploitation.

Economical Data. Market characteristics, expenses, operating costs, profitability of the investment.

Environment and Impacts. Natural and human environment. Impacts from teh implementation of the project

3.0 Environmental Permits

		Status	
3.1	Preliminary Environmental Estimation and Evaluation (PEEE)	PENDING	
3.2	Issuance of Environmental Terms		
3.3	Intervention License for exploitation from Forestry (required only in Forest Lands)		

3.1 Preliminary Environmental Estimation and Evaluation (PEEE)

3.1.1 Procedure

Although location of the future mine cannot be changed and approval of its location is not requested, location of other infrastructure pursuant to the mine activities may conflict with land uses and the overall spatial planning of the area. It is necessary to obtain a pre-approval for the site location of the plant installations, waste disposal and tailings disposal areas, according to the following procedure:

Submission of the documents (3.1.2) and Studies to the Ministry of Environment, Energy and Climate Change, Directorate of Environmental Planning (MOE/DEP), according to JMD 11014/703/104/03

The advice of a number of administrative bodies, subject to the special conditions and characteristics of the project and the broader project area is requested.

Following the new Greek decentralisation law, the advice of the Spatial Planning Committee of the local Prefecture, where the project is located is also requested.

Final advice from other environmental Departments of MOE.

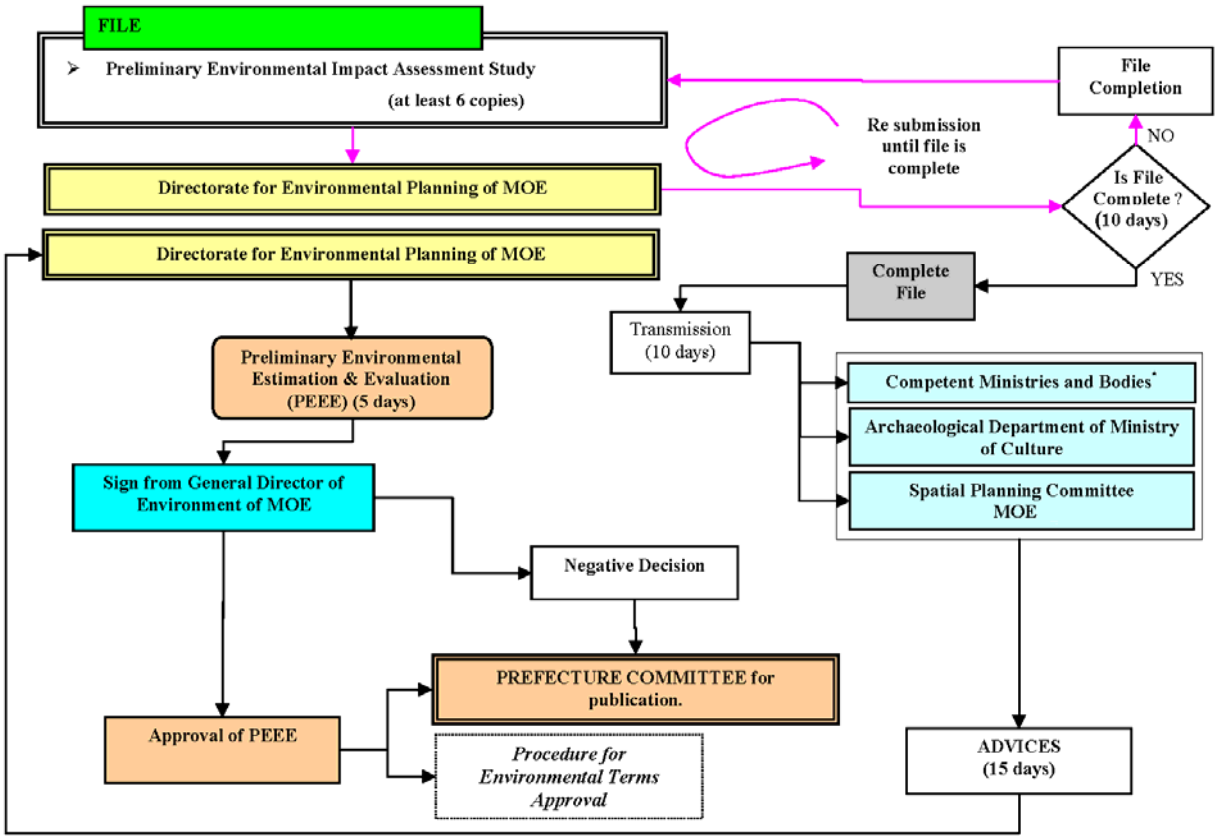
Directorate for Environmental Planning, on the basis of the expert opinions of the different Administrative Bodies, issues the approval and sets up environmental spatial planning terms.

The time needed for spatial planning is estimated to be 3-4 months.

3.1.2 Studies

	Submitted Documents	**Status**	**Reg. No.**
3.1.2.1	Request	SUBMITTED	MOE, 145829/30.10.2009
3.1.2.2	Preliminary Environmental Impact Assessment Study (PEIA)	SUBMITTED	MOE, 145829/30.10.2009
	Preliminary Environmental Estimation and Evaluation (PEEE) Decision	PENDING	

Procedure of Preliminary Environmental Estimation & Evaluation (PEEE) for activities of Subcategory 1 of Category A JMD 11014/703/Φ104/03 (FEK 332/B/20-3-2003) for the Perama Project.



* Ministry of Culture
Ministry of Hygiene
Ministry of Rural Development and Food

3.1.2.2 Preliminary Environmental Impact Assessment Study (PEIA)

The request Preliminary Environmental Impact Assessment Study (PEIA), is elaborated according to JMD 11014/703/Φ104/03 and JMD 69269/90, which must at least include the following:

- Owner of the project
- Description of the existing environmental status
- Project description
- Potential Impacts to the environment, stemming from the project operation
- Closure of the Project and Site rehabilitation.

- Alternative site locations and methods (including the zero option, i.e. no project implementation)
- Mitigation measures.
- Appendices with Maps and Certificates

3.3 Issuance of Environmental Terms

3.2.1 Procedure

The procedure for the Issuance of Environmental Terms is provided by L. 1650/86 as amended by L. 3010/2002 and JMD 11014/703/104/03, and includes the submission of the Environmental Impact Assessment Study (EIA).

The Environmental Impact Assessment Study (EIA) is submitted to the Department in charge of Ministry of Environment, Energy and Climate Change (MOE) and it is approved through a Joint Ministerial Decree (JMD) of the MOE and other Administrative Bodies, subject to the special conditions and characteristics of the project and the broader project area.

The time for its approval is from 5 to 12 months (depending on local Community support, the way the EIA addresses the environmental terms of the PEEE and the complexity of the Project).

The following procedure is necessary for its approval:

The study (3.2.2.2) is submitted to MOE.

All administrative bodies already consulted for the PEEE will be asked for their opinion.

It is sent to the Prefecture Council within 15 days. This period is set according to the existing regulations. In reality, the period is longer.

The Prefecture council(s) of the relevant Prefecture(s) where the mine and installation belong invite(s) individuals, local authorities, and other public interest groups to express their opinion. The invitation is set-up on notice boards and is announced in the local press for a period of 15 days.

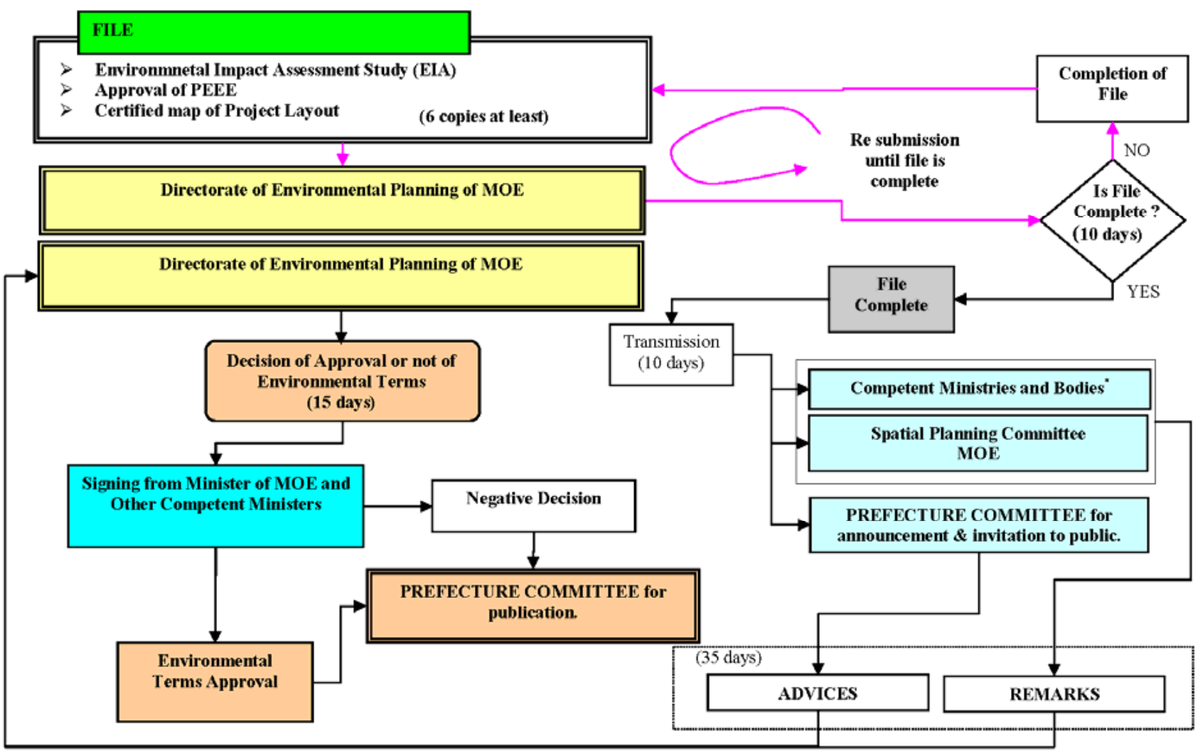
The Prefecture Council(s) (in the presence of all interested parties), after the expression of an opinion, pronounce(s) and send(s) its (their) decision to MOE within 15 days.

The MOE issues the JMD, which is also signed by other relevant Ministries.

3.2.2 Studies

	Submitted Documents	Status	Reg. No.
3.2.2.1	Request		
3.2.2.2	Environmental Impact Assessment Study (EIA)		
	Issuance of Environmental Terms Decision		

Procedure for Environmental Terms Approval for activities of Subcategory 1 of a Category
JMD 11014/703/Φ104/03 (FEK 332/B/20-3-2003) for the Perama Project



* Ministry of Culture
Ministry of Hygiene
Ministry of Rural Development and Food

3.2.2.2 Environmental Impact Assessment Study (PEIA)

The Environmental Impact Assessment Study (PEIA), is accompanied with a certified map of the Project area showing the Project and alternative locations, and the Preliminary Environmental Estimation and Evaluation approval.

The contents of EIA Study are described in JMD 11014/703/Φ104/03 and JMD 69269/90, and include the following:

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According to MD 39624/2209/E103/2009 (FEK 2076/B/25.9.09), issued for the implementation of 2006/21/EC Directive, the EIA Study must also include, a Management Plan of waste from Extractive Industries (Appendix III). The contents of the Extractive Wastes Management Plant, as well as other plans required in case of Category A Facilities, are fully described in MD 39624/2209/E103/2009 (FEK 2076/B/25.9.09).

According to the Directive 2006/21/EC on the Management of Waste from Extractive Industries, no waste facility shall be allowed to operate without a permit granted by the competent authority. The permit shall contain the elements specified in the following paragraph.

(a) the identity of the operator;

(b) the proposed location of the waste facility, including any possible alternative locations;

(c) the waste management plan pursuant to Article 5;

(d) adequate arrangements by way of a financial guarantee or equivalent, as required under Article 14;

(e) the information provided by the operator in accordance with Article 5 of Directive 85/337/EEC if an environmental impact assessment is required under that Directive.

3.3 Forestry Intervention License

A Forestry intervention license (according to law 998/79 as amended) is necessary before beginning exploitation, when the operation takes place in forests. The request is submitted to the relevant Forest Directorate and is issued according to the following procedure:

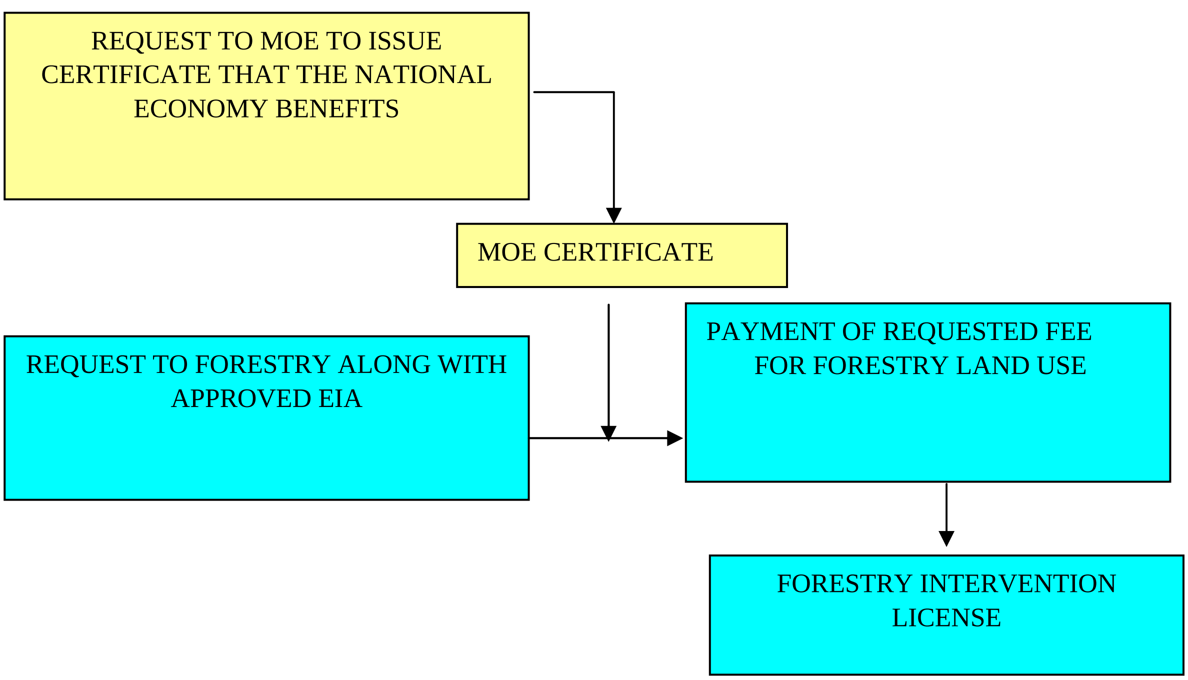
A request is submitted along with the approved environmental study.

A certificate granted by the MOE Directorate of Natural Resources certifying that the exploitation of the mine is particularly advantageous for the national economy is produced (it is granted within 10-12 days following the relevant request),

The necessary fee is paid for the use of the contemplated public forestry land

An intervention License is granted.

FORESTRY INTERVENTION LICENSE INTO FOREST LANDS



4.0 Mine Operation License

The Technical Study is submitted to and approved by the Minister of Development. The required time for the approval is 2-3 months. The Technical Study may be submitted along with the EIA Study.

	Submitted Documents	Competent Authority	Legislation	Status	Submission Reg. No. /Decision
4.1	Technical Study	MOE, Directorate of Natural Resourses	Article 97 of the Code of Mining and Quarrying Works (RMQW)		

5.0 Construction and Operation Licenses

Once the Environmental Terms are issued, based on the prevailing legislation, the following studies need to be available and approved for the implementation of the Mining Project. Moreover, additional studies may be asked by the Mine Project Owner (e.g. environmental monitoring, environmental management system etc). in compliance with the respective environmental terms.

	Submitted Documents	Competent Authority	Legislation	Status	Submission Reg. No. /Decision
5.1	Risk Assessment Study, according to SEVESO Directive (if required)	Fire Brigade of Competent Authority and MOE	Article 99 of RMQW		
5.2	Geotechnical Study for Dam Construction	MOE	Article 99 of RMQW		
5.3	Solid Wastes and Liquid Effluents Study	MOE	RMQW and JMD Ε1β/221/65, as amended with MD Γ1/17831/71, MD Γ4/1305/74 and JMD 69728/824/96.		
5.4	Fire Safety Study	Regional Fire Brigade	J.M.D 97/Φ1/4817/1990		
5.5	Power and Energy Study				

	Submitted Documents	Competent Authority	Legislation	Status	Submission Reg. No. / Decision
5.6	Specific Installations Study	Urban Planning Office of the local Prefecture			
5.7	Specific Installations Study	Regional Department of Development of local Prefecture			
5.8	Buildings Architectural Study	Urban Planning Office of the local Prefecture			
5.9	Buildings Structural Design Study				
5.10	Heat Insulation Study	Regional Department of Development of local Prefecture			
5.11	High Voltage Current Study	Urban Planning Office of the local Prefecture			
5.12	Water Supply Study	Regional Department of Development of local Prefecture			
5.13	Sewerage Study				
5.14	PPC Substation Study	Urban Planning Office of local Prefecture			
5.15	Timetable Schedule, for the construction of the industrial unit.				
5.16	Tender Documents, for the construction of the industrial unit.				

	Submitted Documents	Competent Authority	Legislation	Status	Submission Reg. No. / Decision
5.17	Health & Safety Document	Regional Department of Development of local Prefecture			
5.18	Operation License	Regional Department of Development of local Prefecture			