Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia

EUROMAX RESOURCES LTD.

Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia

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Original signed by Robert Davies Robert Davies, BSc, EurGeol, CGeol	Date	19th December 2014
Original signed by Arun Vathavooran Arun Vathavooran, PhD, CEng MIMMM SME	Date	19th December 2014
Original signed by Laszlo Bodi Laszlo Bodi, P.Eng.	Date	19th December 2014
Original signed by Doug Roy Doug Roy M.A.Sc., P. Eng	Date	19th December 2014
Original signed by David Patrick Dr David Patrick PhD, CEng, FIMMM, FAusIMM	Date	19th December 2014
Original signed by Daniel Leroux Daniel Leroux, M.Sc., P.Geo	Date	19th December 2014
Original signed by David Carter David Carter BSc, PhD, MICE, FIMMM, C Eng, FGS	Date	19th December 2014
Original signed by Gordon Antony Jackson Gordon Antony Jackson, BSc(Eng), FIMMM	Date	19th December 2014
Original signed by Gareth Digges La Touche Gareth Digges La Touche, BSc, MSc, FGS, CGeol, EurGeol	Date	19th December 2014

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GLOSSARY

UNITS OF MEASURE

Above mean sea level .... AMSL
Annum (year) .... a
Centimetre .... cm
Cubic metre m3
Degree .... °
Degrees Celsius .... °C
Dollar (American) .... US $
Equivalent .... equiv.
Gram .... g
Grams per tonne .... g/t
Greater than .... >
Kilo (1,000) .... k
Kilogram .... kg
Kilometre .... km
Kilo ounces .... koz
Kilo pounds .... klb
Kilo tonne .... kt
Kilovolt .... kV
KiloWatt .... kW
Less than .... <
Maximum .... max.
Metre .... m
Metric ton (tonne) .... t
Millimetre .... mm
Million .... M
Million tonnes .... Mt
Million years ago .... Ma
Minimum .... min.
Nano-Telsa .... nT
Number .... No.
Ounce .... oz
Parts per million .... ppm
Percent .... % or pct
Per ounce .... /oz
Per pound .... /lb
Per tonne .... /t
Per year .... /a
Plus .... +
Pound .... lb
Square kilometre km2
Square metre m2
Three Dimensional 3D
Tonne (1,000 kg) .... t
Tonnes per cubic metre t/m3

ABBREVIATIONS AND ACRONYMS

Acid Rock Drainage .... ARD
Albite .... Ab
Adularia .... Adul
Anhydrite .... An
Atomic Absorption Spectroscopy .... AAS
Bachelor of Applied Science .... BAppSc
Bachelor of Science .... BSc
Bismuth .... Bi
Cadastre Municipality .... CM
Canadian Institute of Mining, Metallurgy and Petroleum .... CIM
Carbonate .... Carb
Chalcopyrite .... Cpy
Chartered Geologist .... CGeol
Chartered Engineer .... C.Eng.
Chlorite .... Chl
Copper .... Cu
Diamond Core .... DC
Digital Elevation Model .... DEM
Drawing Exchange Format File .... .dxf
East .... E
Epidote .... Ep
Euromax Resources Ltd. .... Euromax
European Geologist .... EurGeol
European Engineer .... Eur.Ing.
Fellow of the Geological Society .... FGS
Fire Assay .... FA
Fresh Enriched Stockwork .... FES
Fresh Non-Stockwork .... FNS
Fresh Non-Enriched Stockwork .... FNES
Galena .... Ga
Gemcom Surpac TM (version 6.3) .... Surpac
General and Administration .... G&A
Global Positioning System .... GPS
Gold .... Au
Incorporated .... Inc.
Induced Polarisation .... IP
Inductively Coupled Plasma Optical Emission Spectroscopy .... ICP-OES Iron
Kaolinite .... Kaol
Lead .... Pb
Limited .... Ltd.
Lower Failure Limit .... LFL
Magnetite .... Mt
Member of the Australian Institute of Geoscientists .... MAIG
Member of the Institute of Materials, minerals and Mining .... MiMMM
Member of the Southern Africa Institute of Mining and Metallurgy .... MSAIMM
Microsoft Access TM .... Access
Microsoft Excel TM .... Excel
Molybdenum / Molybdenite .... Mo
Moose Mountain Technical Services .... Moose Mountain
Municipality of Bosilovo Census, 2002 .... The Census
National Instrument 43-101 .... NI 43-101
Net Present Value .... NPV
North .... N
Not Applicable .... n/a
Oxide Non-Stockwork .... ONS
Oxide Stockwork .... OS
Phelps Dodge Exploration .... PDX
Portable Document Format .... PDF
Potassium .... K
Potassium Feldspar .... KF
Prefeasibility Study .... PFS

Preliminary Economic Assessment .... PEA
Professional Engineer .... P.Eng.
Pyrite .... Py
Qualified Person .... QP
Quality Assurance / Quality Control .... QA/QC
Quartz .... Qz
Reference .... Ref.
Rock Quality Designation .... RQD
Sericite .... Ser
SGS United Kingdom Ltd. .... SGS
Silver .... Ag
Society of Mining, Metallurgy and Exploration .... SME
South .... S
Sphalerite .... Sp
Tetra Tech WEI Inc. .... Tetra Tech
Transitional Non-Stockwork .... TNS
Transitional Stockwork .... TS
United Kingdom .... UK
Universal Transverse Mercator .... UTM
Upper Failure Limit .... UFL
West .... W
With or without .... ±
Zinc .... Zn

1.0 SUMMARY

The following technical report comprises a Pre-Feasibility Study concerning the Ilovitza copper gold deposit which lies within the Ilovitza property in Eastern Macedonia owned by Euromax Resources Ltd ("Euromax").

Euromax is a public company incorporated in British Columbia, Canada. Euromax is listed on the TSX Venture Exchange.

Various authors have contributed to the Report. The preparation of this Summary was supervised by Dr David Patrick PhD, CEng, FIMMM, FAusIMM of A C A Howe International Limited, and Daniel Leroux, M.Sc., P.Geo of ACA Howe International Limited, both independent QPs as defined by NI 43-101. The Summary contents rely on the information supplied by the other QPs responsible for the individual sections this report.

This report has been prepared in accordance with National Instrument 43-101 (NI 43-101) and Form 43-101F1.

1.1 PROPERTY DESCRIPTION AND LOCATION

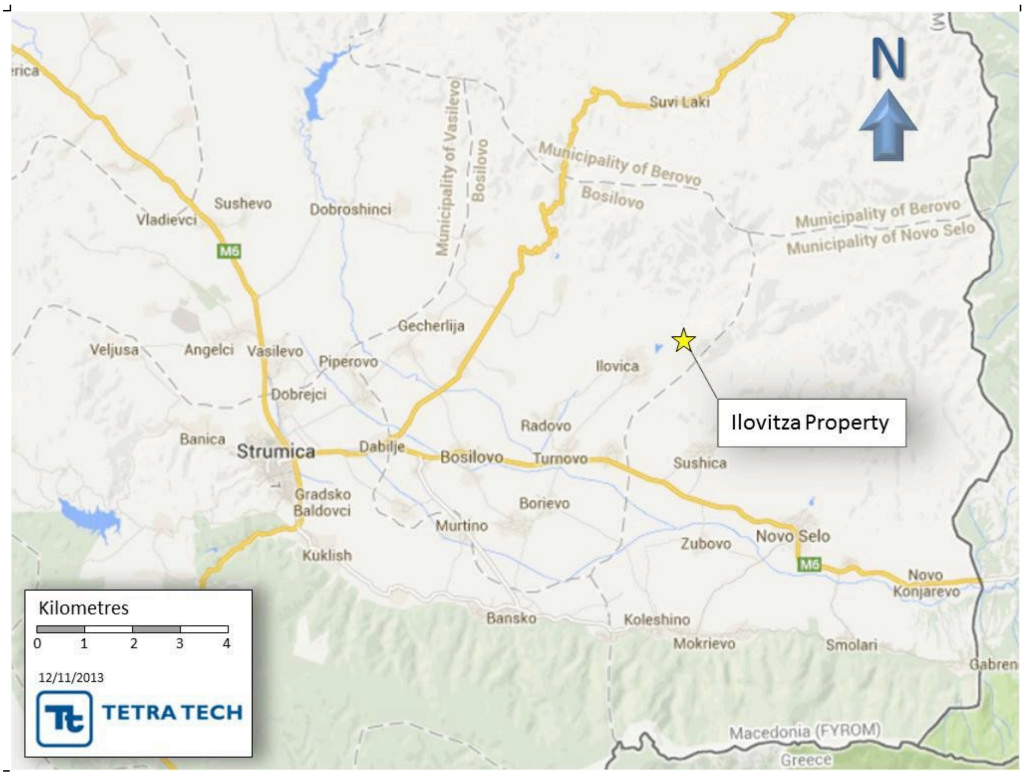
The property is located in the southeast of Macedonia, approximately 15 kilometres (km) to the west (W) of the border with Bulgaria, as shown in Figure1.1. The centre of the mineralised zone is located at coordinates 7654000E 4595500N UTM HKOGEL Projection.

Figure 1.1 Regional Location Map

Source: Tetra Tech

The project is within the municipality of Bosilovo, approximately 20 km to the east of the town of Strumica. See Figure 1.2.

Figure 1.2 Local Location Map



Source: Google

1.14.2 Property Details

A letter from Euromax's lawyer in Skopje, Macedonia: Mens Legis, dated 6th September 2013, states the following regarding title information:

"MENS LEGIS Law Firm is a legal counsel of Euromax Resources Ltd. Canada for Macedonia and a legal counsel of Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje) in Macedonia.

Euromax Resources (Macedonia) Ltd. (99.9%) and Euromax Resources (Macedonia) UK Ltd. (0.1%) are owners of the Macedonian Company Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje).

Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje) is the sole owner of the following mining concessions in Macedonia:

Concession for exploitation of mineral raw materials – ore of copper and gold on the locality "Village Ilovitza", municipality of Bosilovo, Macedonia (Concession agreement Ref. No. 24-6749/1 of 24.07.2012) known as Ilovitza 6. The concession is granted for period of 30 years. The concession area amounts 1.68 km^2.

Concession for detailed geological exploration of minerals – copper and gold on the location of Ilovitza, municipality of Bosilovo, Macedonia (based on the Concession agreement Ref. No. 04-02/11 of 21.02.2011) known as Ilovitza 11. The concession is granted for period of 4 years starting form 21.02.2011 and ending 21.02.2015. The concession area amounts to 3.27 km^2 and is listed in the Title Deeds No. 234, 235, and 566 in Cadastre Municipality (CM) of Shtuka, Title Deed No. 154 in CM Barbarevo and Title Deed No. 277 in CM of Susica.

According to the Decision of the Company for an increase of their core capital, admission of a new member with a new monetary contribution, change of the name of the Company and a change of the abbreviated name of the Company dated 19.12.2012 and the Agreement for establishment of the Company dated 03.01.2013, the firm "Company for production, trade and services PHELPS DODGE VARDAR DOOEL Skopje" is changed and states: "Company for production, trade and services EUROMAX RESOURCES DOO Skopje", and the abbreviated name is changed from "PHELPS DODGE VARDAR DOOEL Skopje" to "EUROMAX RESOURCES DOO Skopje".

The above mentioned changes are registered in the Central Registry of the Republic of Macedonia in accordance with the Resolution under reference number 30120130000235 dated 04.01.2013.

According to the due diligence MENS LEGIS has conducted in Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje) and in our understanding, Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje) has fulfilled all liabilities based on applicable fees and taxes related to the above concessions. We are not aware of any outstanding liabilities based on the concession agreements.

The above listed concessions in our opinion are in good standing and are not subject to any liens or encumbrances."

Note: % = percent Ref. = reference No. = number km^2 = square kilometre

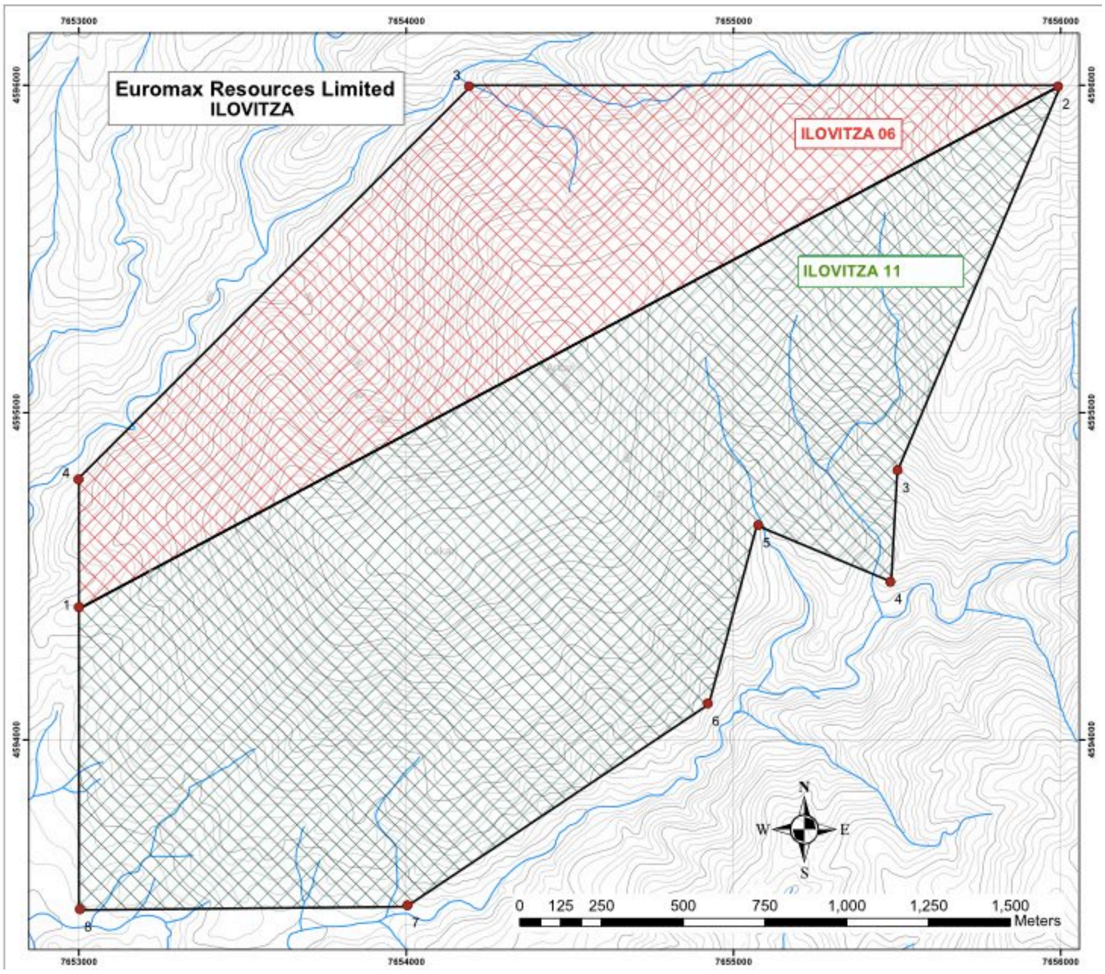
When the Ilovitza 6 exploitation concession was granted an Environmental Impact Assessment for the whole project was also approved. It should be noted that the documentation for conversion of the Ilovitza 11 exploration concession to an exploitation concession is in preparation. The documentation comprises a detailed geological report, a scoping study of the proposed operation and a cadastral report. It is expected that the process will be completed in the first half of 2015.

The property boundaries are map coordinates determined by paper staking, Table 1.1 and Figure 1.3.

Table 1.1 Ilovitza Concessions

Concession	Point No.	UTM HKOGEL Projection	
		Easting	Northing
Ilovitza 11	1	7653000	4594400
Ilovitza 11	2	7656000	4596000
Ilovitza 11	3	7655500	4594820
Ilovitza 11	4	7655482	4594483
Ilovitza 11	5	7655071	4594655
Ilovitza 11	6	7654925	4594109
Ilovitza 11	7	7654000	4593490
Ilovitza 11	8	7653000	4593480
Ilovitza 6	1	7653000	4594400
Ilovitza 6	2	7656000	4596000
Ilovitza 6	3	7654200	4596000
Ilovitza 6	4	7653000	4594800

Figure 1.3 Concession Boundary Map



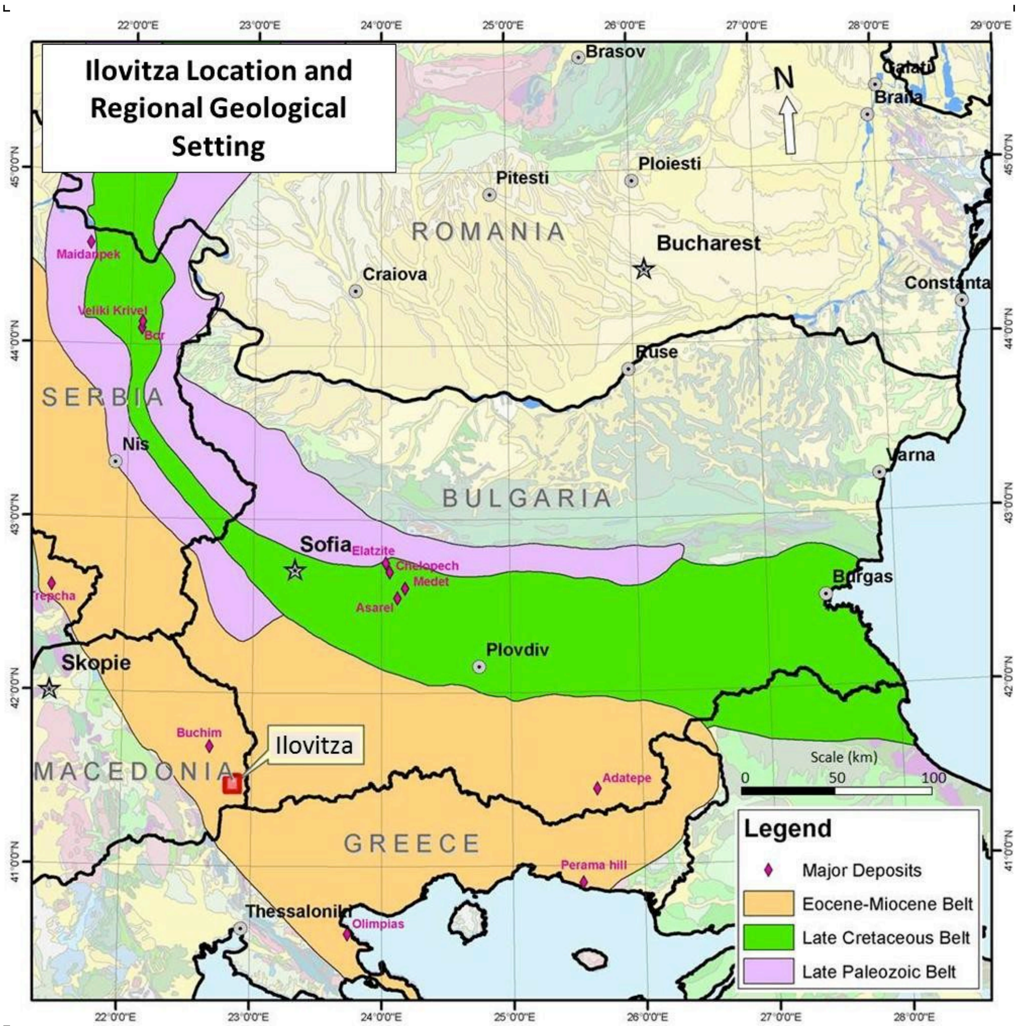
Source: Euromax

The project is situated on the western slopes of the Maleševske mountain range. The project area is part of Mount Ograzhden and ranges from 450 metres (m) above mean sea level (AMSL) to a maximum elevation of approximately 860 metres AMSL. The main valley where the village of Ilovitza is located is at approximately 260 metres AMSL.

1.2 Geology and Mineralisation

Ilovitza is a porphyry copper-gold deposit, located in a northwest-southeast striking Tertiary magmatic arc, that covers large areas of Central Romania, Serbia, Macedonia, Southern Bulgaria, Northern Greece and Eastern Turkey, see Figure 1.4.

Figure 1.4 Regional Geological Setting



Source: Euromax

The porphyry deposits in the region are in close spatial and temporal association with intermediate to felsic, medium to high potassium (K) calc-alkaline igneous rocks. The low sulphidation epithermal deposits are related to bimodal volcanic rocks.

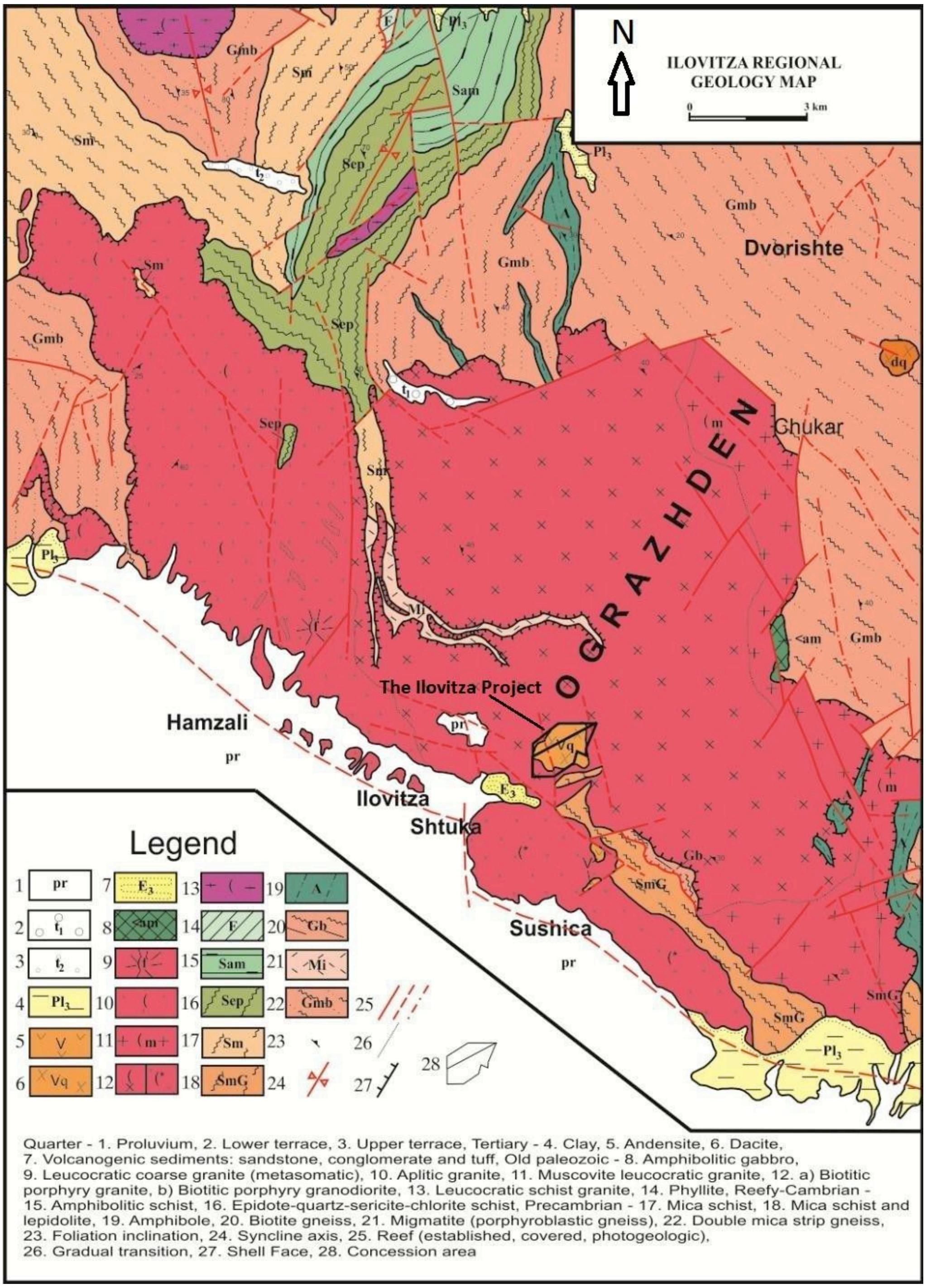
The Ilovitza deposit, which was emplaced circa 29 Million years ago (Ma), is an isolated porphyry copper-gold deposit located about 20 km west of the 33-38 Ma Osogovo-Besna-Kobila lead-zinc belt and about 30 km east of the 22-27 Ma Leche-Buchim-Chalkidiki copper-gold belt.

1.3 Property Geology

The Ilovitza porphyry system is about 1.5 km in diameter and is associated with a poorly exposed dacite-granodiorite plug, emplaced along the north-eastern border of the northwest-southeast elongate Strumitza graben, see Figure 1.5.

The exact location of the deposit is controlled by major north-south cross cutting faults and minor northwest-southeast faulting, parallel to the faulted border of the graben.

Figure 1.5 Local Geological Setting



Source: Euromax

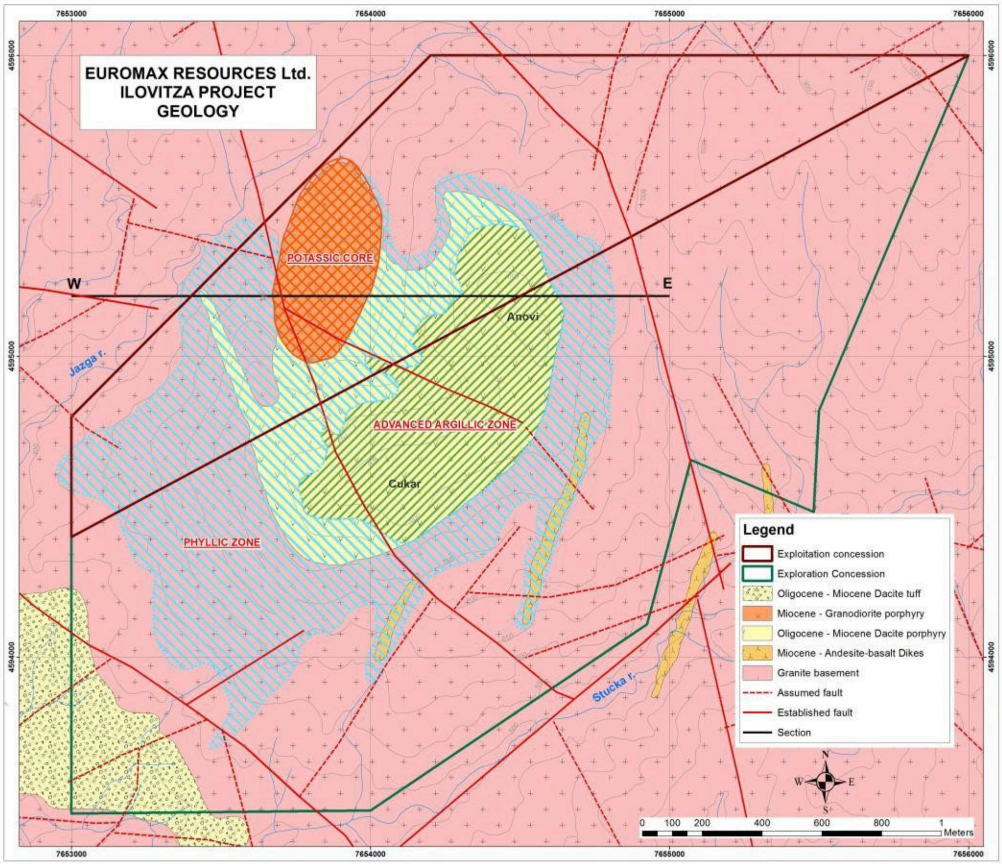
The Strumitza graben (shown in white on Figure 1.5) is a typical post-collision extension structure, about 30 km long and 10 km wide in size and up to more than 1 km in depth. The graben has been filled with terrigenous clastic sediments and felsic volcanic rocks over the last 40 million years.

At surface, the Ilovitza intrusive complex consists of a central dacitic breccia diatreme, approximately 1.3 km in diameter. The diatreme is intruded by at least one dacite and two granodiorite porphyry stocks that have generated several hydrothermal pulses, resulting in widespread multi-phase veining within a mineralised stockwork.

1.4 ALTERATION

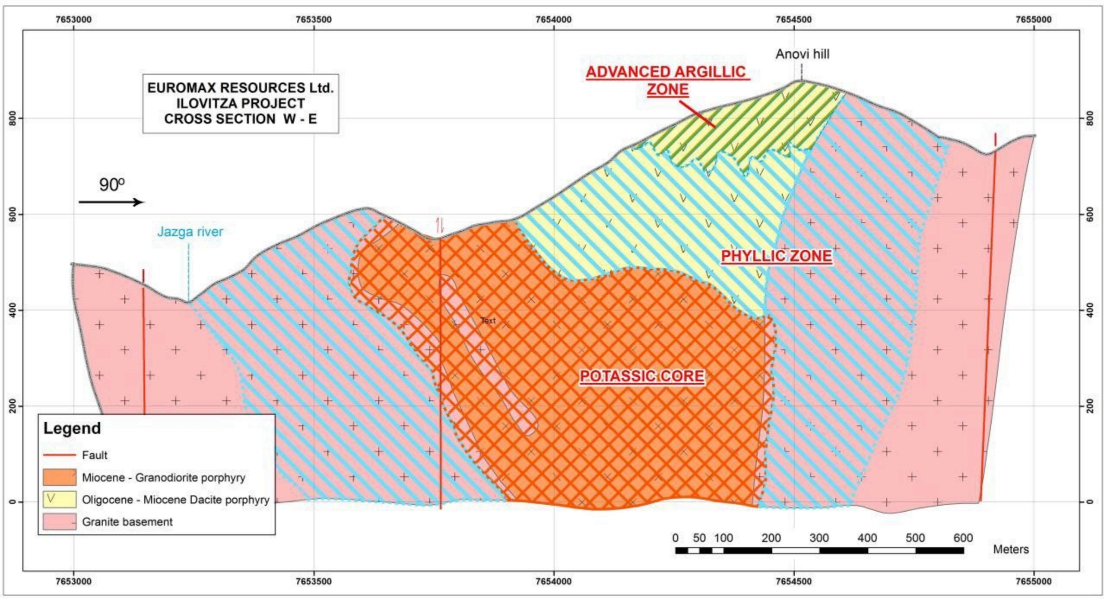
Alteration related to Tertiary magmatic activity at Ilovitza is variably present over an area of about 8 km² see Figure 1.6 and Figure 1.7. Pervasive alteration is largely confined to a roughly 1.5 km² area in and adjacent to the main intrusive complex. Smaller areas of pervasive and structurally-controlled alteration extend somewhat asymmetrically to the south and east of the intrusive complex.

Figure 1.6 Property Geology and Alteration Plan



Source: Euromax

Figure 1.7 Property Geology and Alteration (West-East Cross Section)



Source: Euromax

1.5 MINERALISATION

The main sulphide mineral at Ilovitza is chalcopyrite (Cpy), followed by pyrite (Py) and secondary copper sulphides such as chalcocite, covellite and bornite. molybdenite, galena and sphalerite (Sp) are present in minor amounts and occasional traces of sulphosalt minerals such as tetrahedrite-tennantite and tellurides of gold (Au) and silver (Ag) are observed.

High temperature oxide mineralisation such as magnetite (Mt), dominates at depth, associated with pyrrhotite and chalcopyrrhotite in what is interpreted as the core of the system.

Subsurface porphyry copper-gold mineralisation is expressed at surface by a limonitic, leached stockwork zone approximately 900 by 600 m in size, containing 0.08 to 0.70 parts per million (ppm) gold (Au), 50 to 450 ppm copper (Cu) and 10 to 128 ppm molybdenum (Mo).

Hypogene copper grades greater than (>) 0.15% are largely due to disseminated chalcopyrite, which appears largely confined to the western two-thirds of the stockwork zone.

A supergene-enriched zone ranging from 9 to 70 m in thickness and containing 0.25 to 0.69% Cu as chalcocite and covellite represents enrichment of about 1.5 to 3 times the hypogene grades.

1.6 EXPLORATION

The following exploration activities were completed between 2004 and 2013: Geological mapping, rock chip sampling, soil geochemistry sampling, Induced Polarisation (IP) / Resistivity and Magnetic geophysical surveys.

Mapping was completed on 1:2,000 and 1:5,000 scales and comprised observations with respect to petrology, style of alteration and mineralisation.

In total, three phases of soil sampling have been undertaken on the property, resulting in a total of 540 sampling points arranged on a 100 x 100 m grid covering an area of circa 5,000 square metres (m^2).

1.7 DRILLING

A total of 66 holes were drilled on the property over 9 campaigns between 2004 and 11th July 2013. Table 1.2 summarises the scope of the drilling campaigns completed on the property.

Table 1.2 Summary of Drilling Campaigns

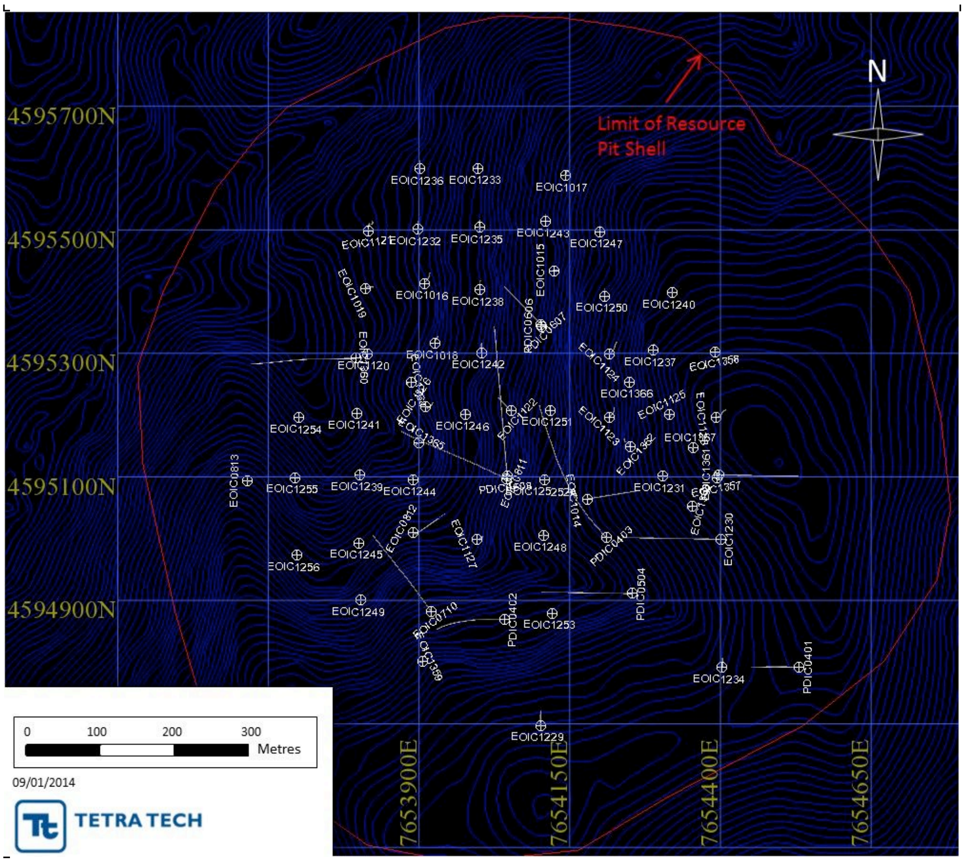
Year	Drilling Technique	No. of Holes	Total Length Drilled (m)	Company
2004	DC	3	1,178	PDX
2005	DC	1	385	PDX
2006	DC	3	1,238	PDX
2007	DC	2	999	Euromax
2008	DC	3	1,600	Euromax
2010	DC	6	3,016	Euromax
2011	DC	10	4,387	Euromax
2012	DC	28	12,081	Euromax
2013	DC	11	4,148	Euromax
Total	**DC**	**66**	**29,032**	-

Key

DC = Diamond Core PDX = Phelps Dodge Exploration

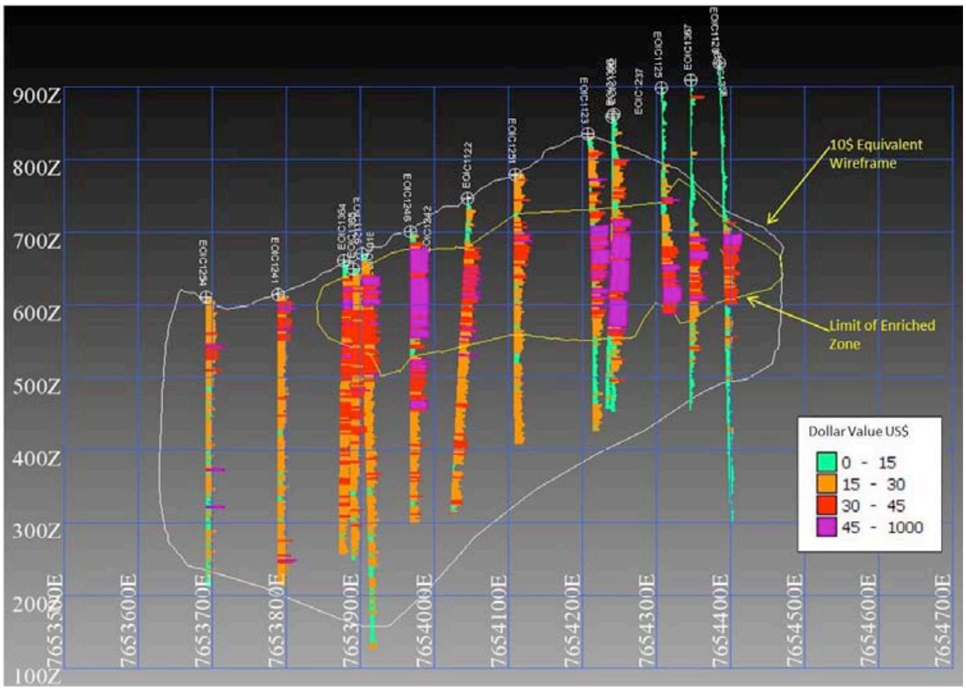
The drillholes are generally vertical or steeply dipping; with 53 of the 67 drillholes being vertical and the remainder being between 60° and 75° as shown on Figure 1.8. A typical west-east section isillustrated in Figure 1.9.

Figure 1.8 Drillhole Locations on the Ilovitza Property



Source: Euromax

Figure 1.9 Typical West-East Cross Section



Source: Tetra Tech Notes: West-east section taken at Y = 4595200

In Tetra Tech's opinion, the drilling and sample preparation, quality assurance / quality control (QA/QC) procedures and security provisions are acceptable and the data can be relied upon for resource estimation.

1.8 Metallurgical Testwork Review

The historical metallurgical investigations were conducted by two organisations: ITMNS in Belgrade, Serbia and SGS, UK (SGS). A high level summary of these investigations is presented in this report.

As part of the prefeasibility study, additional mineralogical and metallurgical testwork was carried out by SGS. They tested a composite sample made up of samples collected from dedicated metallurgical drill holes within the defined mineral resource.

The mineralogical investigations completed at SGS indicate that significant pyrite liberation occurs at a grind size of approximately P80 = 150 microns (µm). A significant proportion of the gold is locked in pyrite, and a pyrite concentration step would be beneficial to increase the overall gold recovery.

Tetra Tech analysed the metallurgical testwork results with the objective of identifying the optimal process design flowsheet. The metallurgical studies indicated that the ore is amenable to flotation and cyanidation and is efficiently processed with a flotation and Carbon in Leach flowsheet.

Based on the metallurgical testwork results, the life of mine recovery for Copper (Cu) and Gold (Au) were estimated at 84 percent (%) and 88% respectively.

1.9 Mineral Resource

This report has adopted the definition of Mineral Resource as outlined within the CIM Definition Standards on Mineral Resources and Mineral Reserves (CIM, 2010).

Tetra Tech has estimated the Mineral Resources for the project, with an effective date of 27th November 2013. The most recent data included in the estimate was received on 2nd October 2013.

Euromax provided geological and analytical data in Excel and Access database format. A topographic survey was provided in drawing exchange format file (.dxf) format and consisted of a satellite radar digital elevation model (DEM). Modelling and estimation has been completed using Geovia Surpac version 6.3.1.

Exploratory data analysis highlighted a number of statistically differentiated grade populations, which were interpreted to be controlled by the following:

- Level of hydrothermal alteration
- Oxidation state
- Supergene leaching and enrichment.

Wireframe models were used to isolate grade populations into domains for the purpose of sample selection and to constrain the grade interpolation.

Statistical and grade continuity analyses were completed to characterise the mineralisation and subsequently used to develop grade interpolation parameters. Grade estimation was completed using ordinary kriging. The search ellipsoid dimensions and orientations were chosen to reflect the continuity revealed by geostatistical studies and optimised using quantitative kriging neighbourhood analysis.

Estimates for silver and molybdenum were not made.

A Mineral Resource classification scheme consistent with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines (2010) was applied. The estimates are categorised in the Inferred, Indicated and Measured Mineral Resource categories, reported above a dollar equivalent cut-off grade that defines the Resource as potentially mineable by open pit mining methods.

Dollar equivalent cut-offs were calculated based upon spot metal prices as of 19th August 2013. The metal prices used are American Dollars (US $) 1,366 per ounce (/oz) Au and US $3.30 per pound (/lb) Cu.

The dollar equivalent is calculated using the following formula:

Dollar eq = [Au * Recovery * Price] + [Cu * Recovery * Price]

A pit optimisation was performed using the Lerchs & Grossman algorithm as implemented in Vulcan. The pit shell was generated to define blocks within the model that have reasonable prospects for economic extraction.

Resource grade / tonnage sensitivity tables were created based upon a range of dollar equivalent cut-offs for blocks within the overall Resource pit shell. A base case cut-off of US $16 per tonne (/t) was chosen for sulphide materials and US$8 /t for oxide materials.

For the purpose of Resource reporting, the transitional material has been grouped with either the oxidised or fresh material based upon the copper content. Where the transitional material has less than 0.2% copper, it is regarded as oxide and where greater than 0.2% it is considered as fresh. This approach reflects the fact that there would not be a separate process route for transitional material.

The Mineral Resource for fresh material is summarised in Table 1.3 and Table 1.4.

Table 1.3 Measured and Indicated Fresh Mineral Resource Based upon a Dollar Equivalent cut-off of $16 /t

Classification	Tonnage (Kt)	Grade		Contained Metal	
		Au (g/t)	Cu (%)	Au (Koz)	Cu (Klb)
Measured	18,440	0.34	0.22	200	88,677
Indicated	218,640	0.33	0.22	2,341	1,036,427
Total M + I	**237,080**	**0.33**	**0.22**	**2,541**	**1,125,104**

Notes: g/t = grams per tonne Koz = kilo ounces Klb = kilo pounds Kt = kilotonne

Table 1.4 Inferred Fresh Mineral Resource Based upon a Dollar Equivalent cut-off of $16 /t

Classification	Tonnage (Kt)	Grade		Contained Metal Tonnage (Kt)	
		Au (g/t)	Cu (%)	Au (Koz)	Cu (Klb)
Inferred	19,850	0.36	0.22	226	96,942

The oxide Mineral Resources within the constraining pit shell are summarised within Table 1.5 and Table 1.6.

Table 1.5 Measured and Indicated Oxide Mineral Resource Based upon a Dollar Equivalent cut-off of $8 /t

Classification	Tonnage (Kt)	Grade Au (g/t)	Contained Metal Au (Koz)
Measured	1,340	0.38	16
Indicated	34,540	0.33	365
Total	**35,880**	**0.33**	**381**

Table 1.6 Inferred Oxide Mineral Resource Based upon a Dollar Equivalent cut-off of $8/t

Classification	Tonnage (Kt)	Grade Au (g/t)	Contained Metal Au (Koz)
Inferred	6,750	0.25	55

Notes for Tables 1.3 to 1.6:

1. Dollar equivalent cut-offs are based upon the following calculation:
   *Dollar Eq = (Au * recovery *price) + (Cu * recovery *price)*
2. The following assumptions were adopted for the calculation of the dollar equivalent:
   - Au recovery in oxide of 86%
   - Cu recovery in oxide of 0%
   - Au recovery in mixed and fresh 65%
   - Cu recovery in mixed and fresh 85%
   - Recoveries based on previous test work are not viewed by Euromax as materially different from the final recoveries in this study and do not warrant re-reporting of the resource
   - Spot metal prices effective 19th August 2013 of US $1,366 /oz Au and US $3.30 /lb Cu.
3. Numbers may not add exactly due to rounding.
4. Tonnages calculated using the densities outlined in table 14.6.
5. Mineral Resources that are not mineral reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
6. Contained gold within this report is quoted in Troy ounces

A constraining pit shell has been applied to the 3D block model to ensure reasonable prospects of economic extraction for the above reported Resources. This does not represent a formal pit optimisation. The pit was generated using the Lerchs & Grossman algorithm as implemented in Vulcan.

1.10 Mineral Reserves

A mining plan and schedule were developed for mining the mineral resources that have been estimated for the project. An economic analysis of this proposed mining project was carried out and the results were positive. The preliminary mine plan is based on Measured and Indicated mineral resources. This report's preliminary feasibility level of detail requires that both the Measured and Indicated mineral resources be classified as a Probable mineral reserve as shown in Table 1.7. No Proven mineral reserves have been designated.

Table 1.7: Mineral reserves (diluted and recovered).

Probable Reserve, Oxide (Diluted and Recovered)	16 Million tonnes
Gold Grade	0.33 g/tonne
Gold Ounces	172,000

Primary/Transitional Probable Reserve (Diluted and Recovered)	209 Million tonnes
Gold Grade	0.34 g/tonne
Gold Ounces	2.28 Million
Copper Grade	0.20%
Copper Pounds	905 Million
Total Probable Reserve (Diluted and Recovered, Rounded)	225 Million Tonnes
Gold Grade	0.34 g/tonne
Gold Ounces (Rounded)	2.45 Million
Copper Grade	0.20%
Copper Pounds	905 Million

Notes:
1. Unplanned dilution equals 5% at diluting grades of 0.17 g/tonne gold and 0.05 % copper.
2. Mining losses = 5%.
3. Mineral reserves are a subset of mineral resources.

1.11 In-Pit Inferred Mineral Resources

Though the mine plan was based on Measured and Indicated mineral resources, Table 1.8 shows the Inferred mineral resources that occur within the planned pit. Figure 1.10 and Figure 1.11 illustrate where these blocks are located within the pit. The Inferred blocks are planned to be mined but are not considered to be part of the Mineral Reserve. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves.

With additional drilling, it is possible that these in-pit inferred mineral resources could be upgraded to higher mineral resource categories. However, there is no guarantee that this would occur.

For the purpose of mine scheduling, the in-pit Inferred mineral resource blocks were considered to be waste rock.

Table 1.8. In-pit Inferred mineral resources.

Oxide	
Tonnes (Millions)	2.14
In-Situ Ounces (000s)	19.7
In-Situ Gold Grade (g/tonne)	0.29
Primary/Transitional	
Tonnes (Millions)	15.34
In-Situ Ounces (000s)	166
In-Situ Gold Grade (g/tonne)	0.34
In-Situ Copper Pounds (Millions)	73.70
In-Situ Copper Grade	0.22%
Total Inferred (Rounded)	
Tonnes (Millions)	17.5
In-Situ Ounces (000s)	186
In-Situ Gold Grade (g/tonne)	0.33
In-Situ Copper Pounds (Millions)	73.7
In-Situ Copper Grade	0.22%

1.12 Mining Methods

Following a detailed geotechnical study, it was concluded that pit slope angles of 39° would provide long-term slope stability. Pit optimisation was carried out using the provisional parameters outlined in Table 1.9.

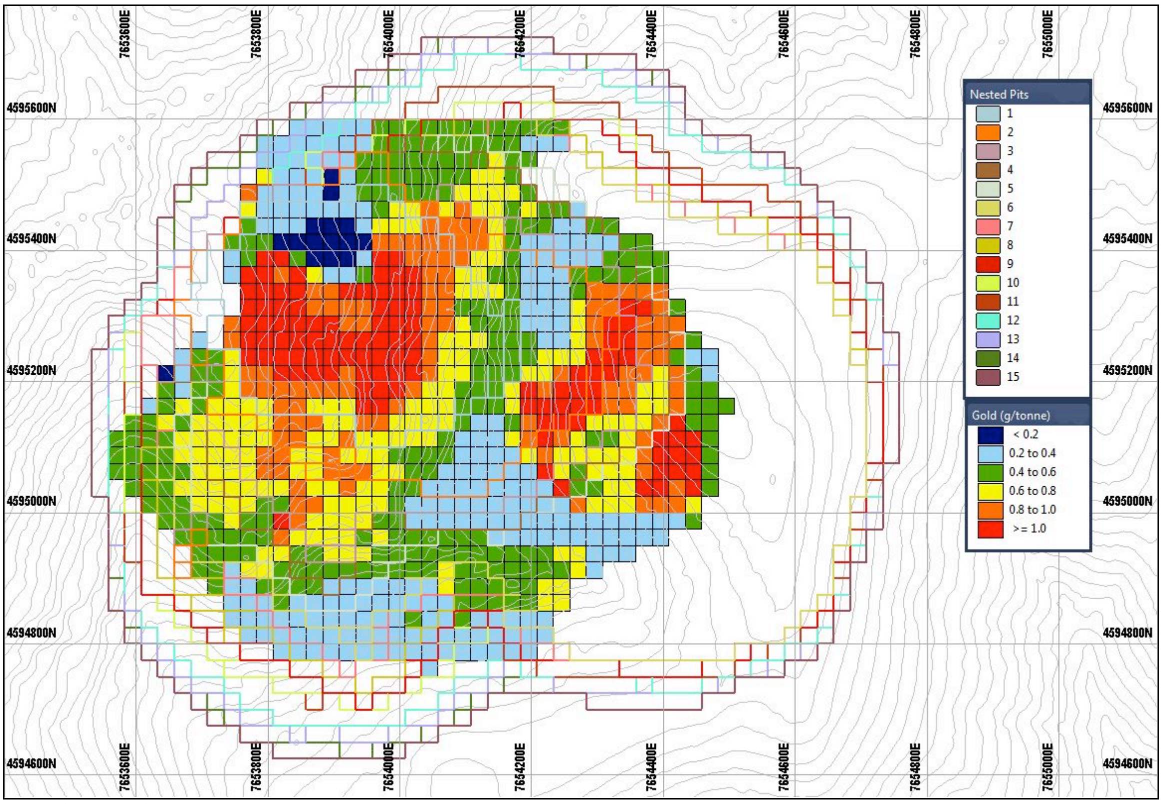
Table 1.9. Pit optimisation parameters.

Parameter	Value
Gold Price	$US 1250 per Ounce
Copper Price	$US 3.00 per Pound
Exchange Rate	$US 1 = $MKD 43.50
SG, Non-Mineralised Rock	2.45
Mining Cost, Ore & Waste	$1.80 per tonne
Mining Cost Escalation With Depth	$0.01 per metre below 400 m MSL
Mining Dilution	5%
Mining Losses	5%
Processing Cost	Oxide $6.24 per tonne Fresh $9.25 per tonne
Milling Rate	10 Million Tonnes per Year
Milling Recovery	
Oxide	90% Gold 0% Copper
Fresh	90% Gold 85% Copper
General & Administration	$1 per tonne
Pit Slope	39°
Reclamation Cost, Ore & Waste	$0.25 per tonne
Selling Costs	Gold $62.50 per Ounce Copper $0.15 per Pound

Note: Mill recoveries based on earlier test results but difference to final recoveries not viewed as material by Euromax.

Fifteen nested pits were outlined at various gold prices (Figure 1.10 and Figure 1.11) and four nested pits were selected for more detailed pit design and production scheduling.

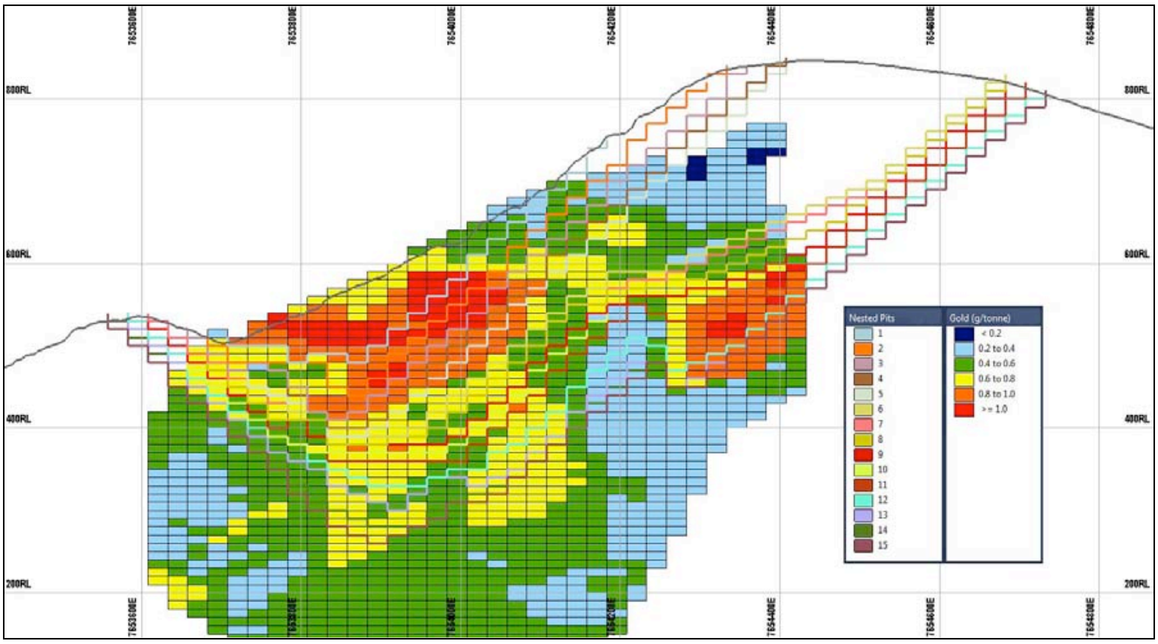
Figure 1.10 Plan view of nested pits.



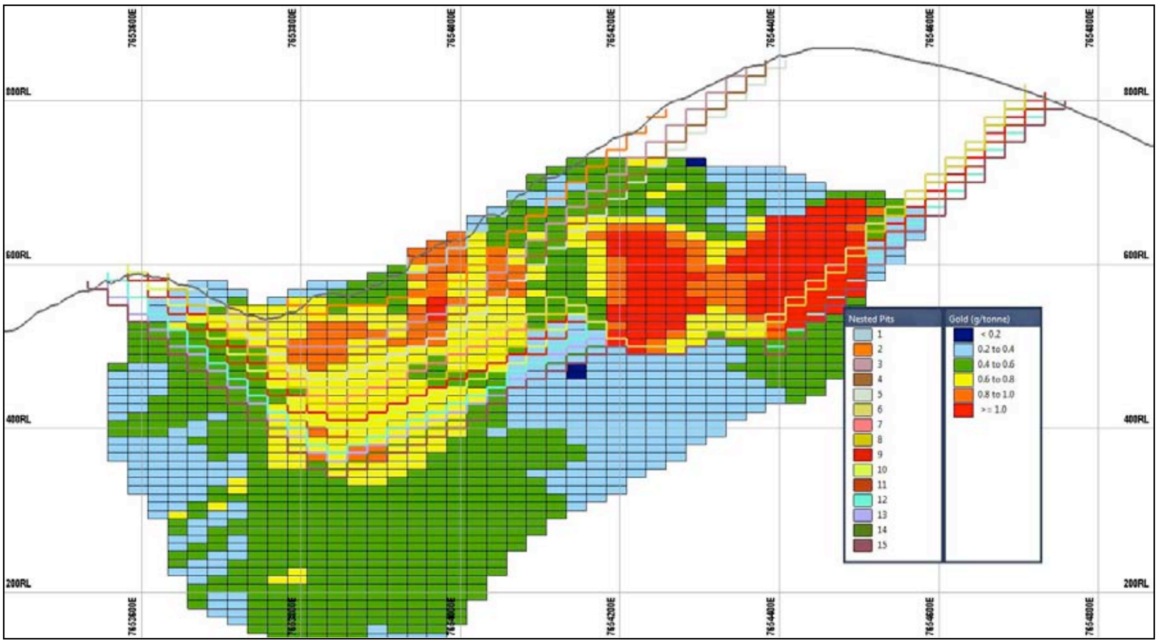
Source ACA Howe

Figure 1.11 Sections through optimum pit shells with block grades (Au equivalent), preliminary work

Section 4,595,285 North, Facing North



Section 4,595,145 North, Facing North



Source ACA Howe

Gold equivalency of one per cent copper equivalent to 1.55 g/tonne gold using prices of US$1,250/oz gold and US$3/lb copper and recoveries of 85% copper and 90% gold, was calculated. A diluting grade equal to the average grade of blocks within the block model and below a cut-off grade of 0.3 g/t gold equivalent were used, namely 0.17 g/t gold and 0.05% copper.

The *in situ* block cut-off grades were calculated for oxide and primary/transitional mill feed as 0.21 and 0.25 g/tonne of gold or gold-equivalent, respectively. Considering the need for some profit to be realised, those grades were increased to 0.23 and 0.27 g/tonne, respectively.

The oxide requires stockpiling and re-handling, so the *in situ* oxide cut-off was increased to 0.25 g/tonne. To account for a longer than initially-expected mill feed haul distance, the primary/transitional cut-off was increased to 0.30 g/tonne.

Based on the results of pit optimisation, the above cut-off grades were applied to a 10 million tonne a year schedule.

The waste stripping schedule was brought forward in order to provide sufficient material for constructing the tailings dam, with a total of 10 million tones of material being stripped prior to production.

The 21-year mine life of the resulting pit was subdivided into four phases, Table 1.10. The phases were designed to balance:

- early capital payback;
- operational constraints;
- overall profitability; and,
- a reasonable mine life.

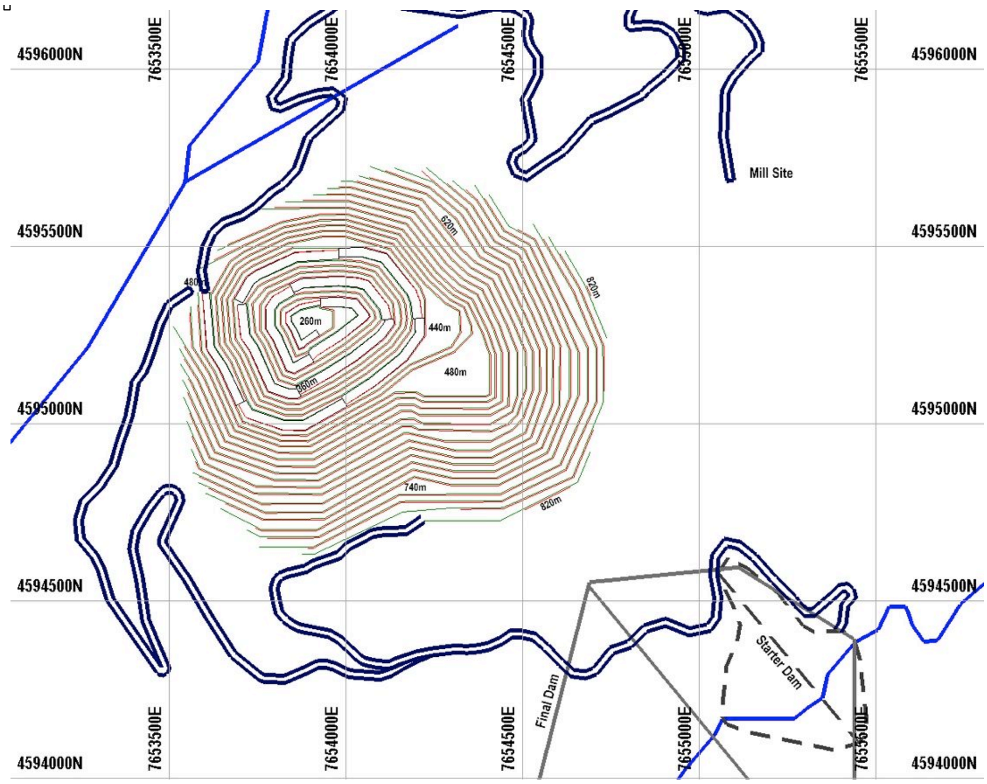
Table 1.10 Phases by year.

Phase	Nested Pit*	Year(s)	Bottom Elevation	Description
1	1	1-2	480 m	Starter Pit
2	2	2-3	440 m	Expansion of Starter Pit
3	6	3-9	400 m	Pushback and Deepening
4	15	9-21	260 m	Pushback and Deepening
Oxide Stockpile Milling		21-23		

Note: The detailed, de-optimised design *closely* follows the nested pit.

The final pit is illustrated in Figure 1.12.

Figure 1.12 Final pit (Phase 4).



Source ACA Howe

Euromax wish to use 90-tonne haul trucks. To determine the number of trucks that would be needed, haul roads were designed and cycle times for each phase were estimated based on the haul distances summarised in Table 1.11.

Table 1.11 Average ore haulage distance by Phase.

Phase	Average Ore Haulage Distance (m)	Average Distance Plus 25%*	Average Slope Gradient
1	1,200	1,500	-8%
2	1,100	1,400	-5%
3	1,600	2,000	-7%
4	1,000	1,300	0

** To account for curves - rounded up to the nearest 100.*

Based on production requirements and the average haul distances a mobile equipment fleet was selected (Table 1.12).

Table 1.12 Mobile equipment fleet.

Item	Description	Number
CAT 777 Truck	90 tonne Capacity	8 Preproduction 13 Years 1-6 11-12 Years 7-11 10 Years 12-21 5 Years 22-23
CAT 990 Loader	15 tonne Bucket Capacity	1 Pre-production 3 Years 1+
CAT 375 Excavator	75 tonne	1
CAT 345 Excavator	45 tonne	1
CAT D10 Bulldozer	430 kWatt (580 HP), Waste Pile	1
CAT D8 Bulldozer	300 kWatt (405 HP), Pit Work	2
CAT 770 Water Truck	35 tonne Capacity	1
CAT 24 Motor Grader	400 kWatt (530 HP), 7.3 m Blade	1
CAT 16 Motor Grader	220 kWatt (300 HP), 4.9 m Blade	1
Sandvik D75 Drill	Up to 280 mm Hole, Production	2
Sandvik DX800 Drill	75-125 mm Hole, Road Work / Secondary Blasting	1
CAT 825 Compactor	260 kWatt (350 HP), Dam Compaction	1
ANFO Prill Truck	20 tonne	1
Boom Truck	5 tonne	2
Lube/Fuel Truck		1
Man Bus		2

After the tailings dam embankment is complete, there would be 40 million tonnes of waste left over It was determined that trucking the remaining waste to the tailings pond to buttress the main tailings embankment was the optimal option for waste rock management.

A truck workshop was designed that would incorporate four bays and would also have space for an electrical and machine shop, a tyre shop, a tool crib, a warehouse, a dry (changing and showering facility), a welding shop, and office space. A Quantity Surveyor estimated the total construction cost at $US 9.7 million, exclusive of contingency and sales tax.

1.13 Recovery Methods

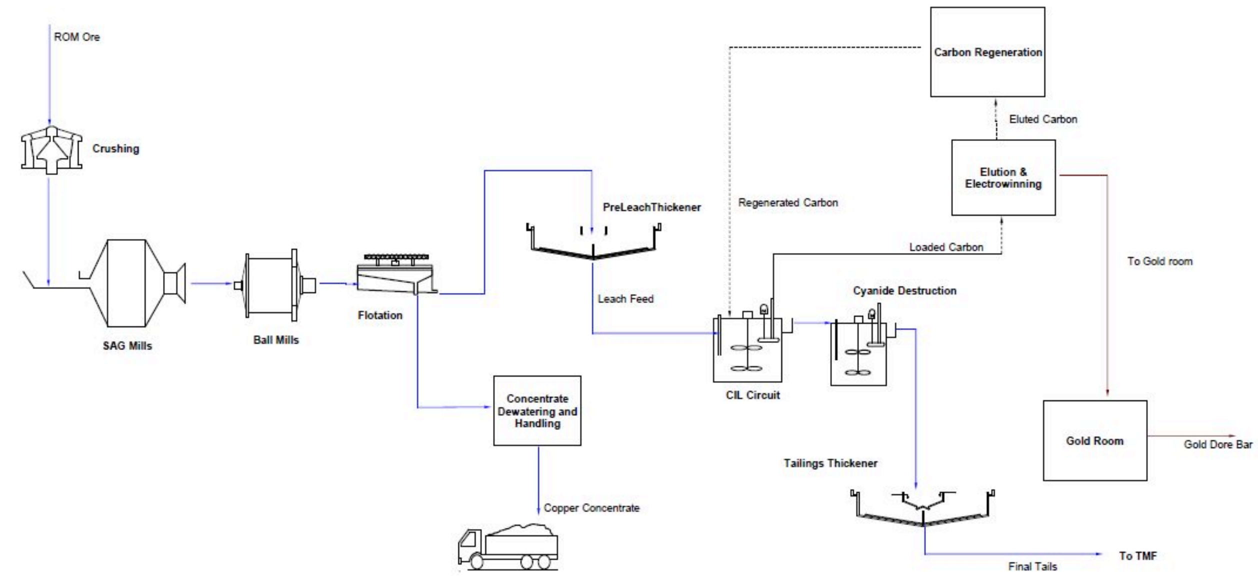
The process plant will be constructed for a 10 million tonnes per annum (Mt/a) capacity based on a flowsheet that produces a saleable copper concentrate and maximises the overall copper and gold recovery. The Ilovitza ore is derived from a porphyry copper gold deposit is moderately hard, and is amenable to both flotation and cyanidation. The process flowsheet has been developed based on the test work reported in Section 13.0 with the objective of producing a saleable copper concentrate and maximising the gold recovery.

The run of mine (ROM) ore will be crushed by a gyratory crusher and then ground in two-stages comprising semi autogenous grinding (SAG) mill and ball mill conventional milling circuit in order to produce slurry with an optimum size distribution for flotation and leaching. The ground slurry, with a particle size of P80 = 75 μm, is fed into flotation to produce a saleable copper concentrate. The copper concentrate at an expected copper grade of 24% is dewatered in the concentrate thickener and filter and shipped for smelting.

The flotation tails are fed into a pre-leach thickener and the thickener underflow will then be pumped through Carbon in Leach (CIL) tanks. Flotation tailings slurry will be leached in 16 CIL tanks which utilise cyanide leaching and recovery of the dissolved precious metals onto activated carbon. The carbon is then pressure stripped with a hot caustic solution to elute the precious metals into a pregnant solution which, in turn is treated by conventional electrowinning to produce a gold sludge that is suitable for direct smelting on site.

Tailings from the process plant will be pumped to the Tailings Management Facility (TMF). A flow diagram of the process route is given in Figure 1.10.

Figure 1.10 Block Flow Diagram of the Process Plant



Source Tetra Tech

The installed plant capital cost of the proposed process plant is estimated at US $249.6 Million ±25%.

The total process plant operating cost has been estimated based on the process design work and the reagent consumptions estimated based on the prefeasibility study test work results. The estimated process plant operating cost is US $6.50 per tonne (/t) within ±25%.

The overall copper and gold recoveries are estimated at 84% and 88% respectively.

1.14 INFRASTRUCTURE

1.14.1 EXISTING INFRASTRUCTURE

The property is well served by asphalt paved roads. The site is located in the hills approximately 3 km northeast of the village of Ilovitza. The village of Ilovitza is located 3 km to the north of road number M6, which is a two lane asphalt paved road that leaves Strumica to the east.

The site is well linked to the road network in southern Europe. The E-75 International Road passes south through central Europe and Macedonia to the Port of Thessaloniki. The E-75 passes some 50 km west of the site and is linked to the M6 Highway, which passes east to the border with Bulgaria and links with the town of Petrich (40 km away in south-western Bulgaria).

The Bulgarian rail network extends as far as Petrich, and links to the Pirdorp smelter at Chelopech, the Black Sea ports and the Mediterranean Port of Thessaloniki.

The Thessaloniki Port is located 145 km away from the property and is understood to be suitable for both bulk cargoes and container goods.

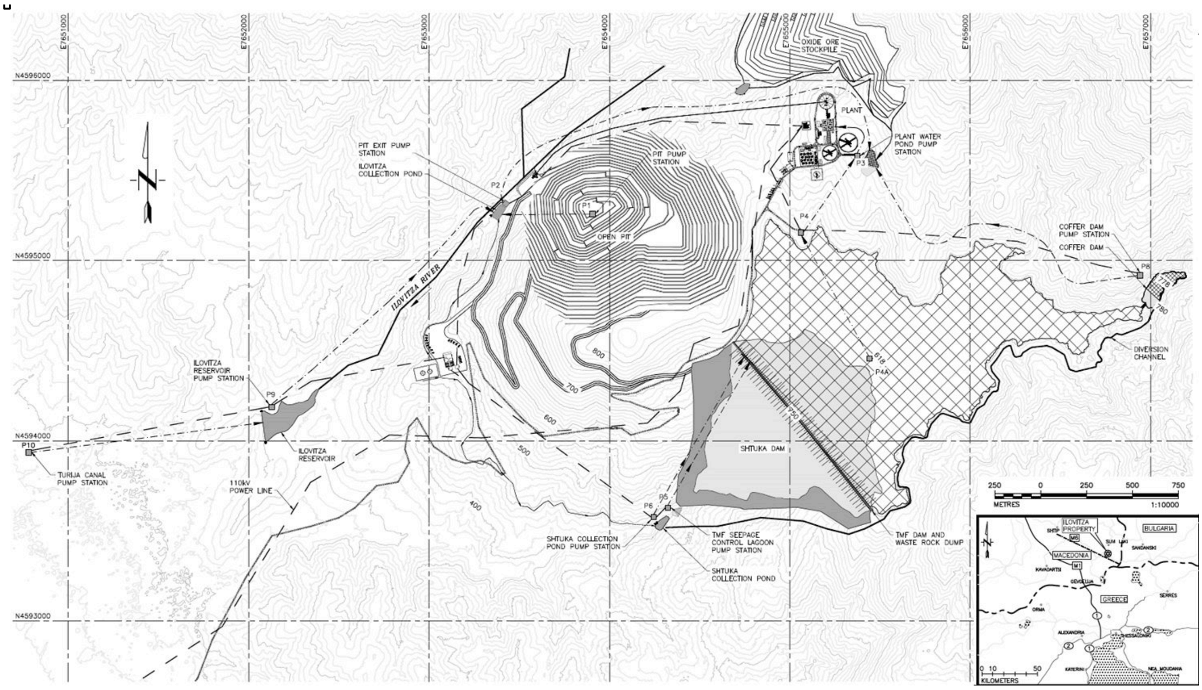
There is an existing water reservoir on the edge of the site, which is used for agricultural purposes. Hydrological and hydrogeological studies are ongoing on the property to establish local ground water conditions. There is a larger water reservoir at Turija, some 15 kilometres to the West-Northwest which links to the area via a canal.

Macedonia is connected to the European power grid via the National Grids of Bulgaria, Greece and Serbia. A 110 kilovolt (kV) power transmission line passes within 5 km of the site, with an existing substation near the town of Sushica, approximately 8 km from the site.

1.14.2 Site Infrastructure

The site layout is shown in Figure1.11.

Figure 1.11 Proposed Site Layout

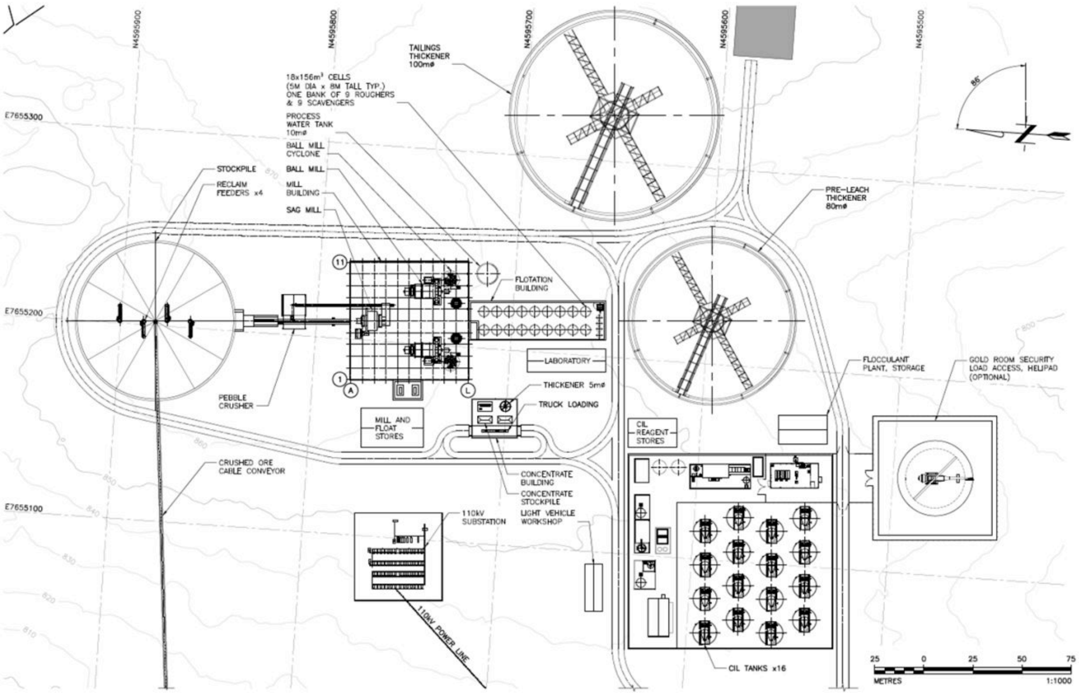


Source: Tetra Tech

The mine site and facilities are based in two main areas, an Upper site and Lower site. The Lower site has the Run of Mine (ROM) pad and primary crusher adjacent to the mine and pit exit around the 480 m elevation. The haul truck workshop and main fuel storage area are also adjacent around the 450 m elevation.

The remaining facilities are located on an upper site around the 850 m elevation. This upper site includes the crushed ore stockpile, process plant, with gold room and product dispatch, as well as the ancillary facilities including the administrative and social building, stores and workshops, Figure 1.12.

Figure 1.12 Upper Site Layout



Source: Tetra Tech

Ore delivered to the ROM pad / primary crusher is crushed and conveyed via a cable conveyor to the crushed ore stockpile at the upper site for processing. Following processing the resulting tailings are discharged to a tailings management facility (TMF), located in the Shtuka valley.

Waste rock from the mining process will be transported by haul tracks to the Shtuka valley for use in constructing the tailings dam with surplus waste rock dumped on the downstream face of the tailings dam. Oxide ore will be stored in a temporary stockpile close to the plant for processing at the end of the life of mine.

Water around the site will be managed, primarily to ensure that the environment is protected, while providing a secure water supply to the processing plant and maintaining clean water flows in the existing rivers/creeks.

The proposed open pit and site facilities are located as shown on Figure 1.11. The upper and lower sites will be connected to the existing highway M6 by a newly constructed paved road. The new intersection at M6 is presently proposed for construction between Turnovo and Sekirnik. A network of internal gravel covered roads will connect the site facilities.

A new power supply will also be constructed to support operations. This will include a 7.5 km high voltage transmission line from the existing 110 kV transmission line some 2.5 km southeast of the village of Ilovitza to the upper site substation. A medium and lower voltage distribution network will supply power from the main upper site substation to the other site facilities.

The mine area is in an active seismic zone and all facilities should be designed accordingly and form part of the engineering design requirements for future studies.

1.14.3 Waste Rock Dumps

At this stage all waste rock is proposed to be used for the construction of the tailings and water storage dams. Excess waste rock totalling some 50 Mt will be placed as a buttress on the downstream face of the TMF embankment. This has been modelled by the Faculty of Civil Engineering in Skopje.

1.14.4 Tailings Management Facility (TMF)

The Faculty of Civil Engineering in Skopje were commissioned to provide a preliminary design for a Tailings Management Facility using the following assumptions:

- That the embankment would be, where possible, constructed from waste rock as provided in the mining schedule.
- Use a downstream construction approach.
- Position the embankment in the Shtuka valley so as to maximise capacity for tailings storage.
- Assume thickened tailings of about 60 to 65% solids.
- Drainage would be with perimeter drainage channels.
- Utilise a spillway design and lagoon downstream to cope with any large precipitation events.
- Take into account the seismic conditions of the area.
- Use the known geotechnical features of the valley based on an earlier test pit programme supervised by the faculty.
- Use a single embankment.

Seven profiles were considered for the tailings embankment within the Shtuka valley. Final profiles for further consideration were selected on the basis of suitable storage volume and geotechnical characteristics.

The final design of the TMF is located on Figure 1.11.

1.14.5 Oxide Ore Temporary Stockpile

A design for the oxide stockpile has been developed by Euromax together with the Faculty of Natural and Technical Sciences at the University of Stip, Macedonia. The oxide dump was designed to accommodate the oxide material above cut-off grade mined throughout the mine life and to be processed once all the sulphide and transition material is exhausted. The total amount of oxide material that the stockpile is required to accommodate is 16.2 Mt.

1.15 Environmental and Social Studies

An Environmental Impact Statement (EIS), prepared by a Macedonian company, Rudplan DOOEL, based on the project Conceptual Study prepared by Phelps Dodge Vardar DOOEL (now known as Euromax Resources DOO,) was presented to the Macedonian Government in October 2011. Approval of the EIS was received from the Ministry of Environment and Physical Planning (MEPP) in November 2011. Euromax recognises that further baseline investigations and impact assessment are needed to reflect the refined definitions of the project scope achieved through this PFS study and ensure that the Ilovitza project is Equator Principle and IFC Performance Standard compliant so that there are no barriers to project financing and to ensure that the project is developed to the highest standards.

Additional environmental and social baseline studies have been commissioned from Golder Associates UK, together with further ground and surface water studies to be completed by Schlumberger Water Services, UK, Table 1.13.

Table 1.13 Continuing Environmental and Social Programme

Environmental Baseline	
Geology	Biodiversity and ecosystem services
Geomorphology and landscape	Climate
Soils and Land Capability	Air quality
Land Use	Noise
Agriculture and forestry	Traffic
Water Studies	
Geochemistry	Groundwater
Surface water	Water supply
Social Baseline	
Social and economic data	Archaeology and cultural heritage
Stakeholder Engagement	Visual assessment
Household survey	

On the basis of the current understanding of the project definition and the limited information available on environmental and social aspects, five potentially material issues have been identified. These are:

- Water supply;
- Geochemistry of the ore and waste rock;
- Community relations;
- Opportunities for local economic development; and
- Closure.

These issues will be addressed through development of a revised environmental and social impact assessment as the project advances. However, to date no fatal flaws of serious environmental or social liabilities have been uncovered.

1.16 Capital and Operating Costs

The Pre-Feasibility Study on the Ilovitza gold-copper project has defined operating and capital costs as detailed in this section.

All costs have been estimated in US dollars. Euro values have been converted to dollars at a long-term exchange rate of 1.4 dollars to the euro.

A summary of the total estimated capital costs is given in Table 1.14.

Table 1.14 Capital Cost Summary

Description (US$ million)	Initial Capex	Sustaining Capex
Mining Fleet (incl. conveyor)	34.8	128.0
Processing Plant	249.5	(in opex)
Owners costs	10.0	-
Infrastructure	103.8	30.6
Tailings (incl. pre-strip)	58.1	47.5
Reclamation (end of mine life)	-	30.0
Sub-total	**456.2**	**236.1**
Contingency (10%)	45.6	-
Total	**501.8**	**236.1**

Operating costs were calculated on a first principles basis.

A summary of the key operating costs is given in Table 1.15.

Table 1.15 Summary of Operating Costs

Mining - Average LOM cost (US$/t ore)	
Mining - Oxide (incl. rehandle cost)	1.96
Mining - Sulphide	1.72
Mining - Waste (excl. pre-strip)	1.59
Conveyor	0.10
Processing	
Oxide Processing	5.23
Sulphide Processing	6.50
Infrastructure opex	0.29
G&A	1.00

Economic Analysis

An economic evaluation of the project using discounted cashflow was prepared on a pre-tax and a post-tax basis. For the 23-year mine life, 225Mt total throughput, operating at 10 Mt/a, the PFS returns the following financial results:

- 18.6% Internal Rate of Return (IRR) pre tax, 16.5% IRR post-tax
- 6.3 years pre-tax payback, 6.8 years post-tax payback on $501.8 million initial capital
- US$675 million pre-tax Net Present Value (NPV) at a 5% discount value.
- US$558 million post-tax NPV at 5% discount value.

The base case copper and gold prices used in this analysis are $3/lb copper and $1,250/oz gold. These are estimated to be realistic long-term prices for the current market.

In the absence of letters of intent, the following off-site charges were assumed for the copper concentrate produced:

- 95.83% payable copper (based on copper grade of 24% for concentrate)
- 97.00% payable gold
- $75/dmt concentrate – copper treatment charge
- $0.075/payable lb copper – copper refining charge
- $5.00/toz gold – gold refining charge
- 0.1% net invoice value (NIV) – insurance losses and marketing
- $45.00/wmt concentrate – transport charge.

It was assumed that concentrate would have a copper grade of 24% and an average moisture content of 10%.

For doré production the following terms were applied:
- 99% payability gold
- $1.00/toz gold – gold refining charge
- $5.00/toz gold – insurance and transport charge

By-product silver revenues based on consistent payable silver credits in assays of concentrate produced during metallurgical testwork were considered in the financial evaluation.

Sensitivities to the following parameters were also examined.
- discount rate
- copper price
- gold price

- capital cost
- on-site operating costs

The results for changes in discount rate and metal price are presented in Table 1.16.

Table 1.16 Sensitivity of the Project NPV to Metal Price and Discount Rate

Gold (US$/oz)	Copper (US$/lb)	NPV @ 0% discount (US$M)	NPV @ 5% discount (US$M)	NPV @ 7.5% discount (US$M)	Pre-tax IRR (%)
1,100	2.50	757.5	284.3	146.4	11.4%
1,250	3.00	1,420.8	675.1	459.0	18.6%
1,400	3.50	2,084.0	1,066.0	771.6	24.9%

Clearly the project is sensitive to both changes in discount rate and metal prices but the project still does offer positive returns on investment at lower prices and rates.

The sensitivities to changes in copper prices are shown in Table 1.17.

Table 1.17 Project NPV Sensitivity to Copper Price

Copper Price US$/lb	$2.5	$2.75	$3.0	$3.25	$3.5
NPV US$M @ 5% discount	$463M	$569M	$675M	$781M	$887M

Sensitivities to a change only in gold prices are shown in Table 1.18

Table 1.18 Project NPV Sensitivity to Gold Price

Gold Price US$/oz	$1,100	$1,175	$1,250	$1,325	$1,400
NPV US$M @ 5% discount	$496M	$586M	$675M	$765M	$854M

The results indicate that the project is more sensitive to gold prices which cause the same magnitude of change in NPV for a much smaller swing in price as a percentage of the base case.

An analysis of sensitivity to operating and capital costs has also been completed. Changes in project NPV in response to variations in capital and operating costs are given in Table 1.19.

Table 1.19 Project NPV Sensitivity to Operating and Capital Costs

	-5%	-2%	0%	+2%	+5%
NPV US$M with changes in capex	$707M	$688M	$675M	$662M	$643M
NPV US$M with change in opex	$744M	$703M	$675M	$648M	$606M

The results indicate the project is more sensitive to changes in operating costs than to changes in capital but that the project remains viable within the ranges tested.

1.17 Interpretation and Conclusions

The current Pre-Feasibility Study has defined a mining project at the Euromax owned Ilovitza Gold-Copper project that justifies continuing development to Feasibility Study and Front End Engineering level. Overall the study complies with industry standard practices for PFS level and is considered to have an accuracy of plus or minus 15% or better. The scope of the project including preliminary mine design, mine schedule, process flow sheet and process plant design, waste management and tailings management facility have been assessed and viable solutions defined in each case.

Based on the supplied data and application of appropriate methods, an open pit mining project was outlined that could be profitably mined and based on the results of a positive economic analysis of the proposed mine, mineral reserves were identified.

Processing of the ore using a flow sheet comprising crushing, grinding by SAG and then ball mill, flotation of a copper concentrate and treatment of the flotation tailings for additional recovery of gold has been defined as a viable process route. The process route has had sufficient testwork carried out in order to establish process operating and capital costs to the required level of detail for a PFS level study. Costs have been established for a 10 million tonne per year operation. Oxide ore will be processed at the end of mine life once the Sulphide ore is exhausted. The process will produce a copper concentrate with payable gold credits and gold doré.

Existing infrastructure has been examined and the required additional infrastructure designed to a level appropriate for the study. This comprises a system of roads and power lines to connect to the local networks, a water pumping network to ensure sufficient make up water for the plant, a series of buildings and workshops to house the various parts of the project and accommodate the required support for this and a tailings and waste management facility in the Shtuka valley, adjacent to the mine. The footprint is as compact as possible and has the advantage of impacting only the drainage systems, which pass directly by the deposit.

No fatal flaws have been found with respect to environmental and social issues and the project remains within the parameters of the EIS approved in 2012.

Financial analysis of the parameters defined by the PFS demonstrate a viable project. The mine schedule taken forward to the financial model delivers higher than average grade in the first eight years. As with all bulk mining projects, the project is sensitive to changes in metal price but still returned a positive return within the range of sensitivities tested for metal price, discount rate, operating costs and capital costs.

1.18 Recommendations

The definition of a viable project from the positive results of this pre-feasibility study leads to the recommendation that the project is advanced to the next stages of feasibility study and front end engineering. Key aspects of this are as follows.

1.18.1 Infill Drilling

It is recommended that an infill-drilling programme is undertaken on the property in order to achieve the following:

- An increased understanding of the short-range variability of the grade continuity.
- Allow further Measured mineral resources to be defined in areas likely to be in the mine plan of the early years from which Proven mineral reserves might be derived
- Conversion of Inferred mineral resources that fall within the current reserve pit to a higher category
- Sufficient data to allow further segregation of mineralisation populations on the basis of lithology and alteration.
- Increased understanding of the structural geology and controls of the deposit.
- Additional material for metallurgical testwork.
- Additional geotechnical information.

1.18.2 Mining

More-detailed mine design, production scheduling, and equipment selection work should be carried out to support a definitive feasibility study. This will require further geotechnical drilling and modelling of ground water. It is recommended that sufficient Measured mineral resources be defined to support Proven mineral reserves for the first three to five years of the mine schedule, assuming the required engineering detail is also achieved.

1.18.3 METALLURGICAL TESTWORK AND MINERAL PROCESSING.

Further metallurgical testwork is required to support feasibility and front end engineering studies. This should include geo-metallurgy to investigate variability within the deposit, in particular the higher-grade areas which fall within the early years of the current mine plan.

Feasibility level design for the process plant should be advanced. Commissioning of a feasibility study and front end engineering and design from an engineering group and / or equipment supplier could streamline the process for this scale of plant. Integration of mine infrastructure studies is advisable to ensure the integrity of the study.

1.18.4 INFRASTRUCTURE

More detailed geotechnical investigations are required over the proposed sites for the mine infrastructure, in particular the plant site, tailings management facility, truck workshop and proposed road corridor. More detailed design of the tailings facility is required and this should be supported by an appropriate level of tailings testwork to investigate tailings rheology and any potential for acid drainage.

1.18.5 WATER

The current studies into water should be continued with the drilling of boreholes to investigate ground water levels, flows and quality and the continued monitoring of existing drill holes, wells and surface water. This will enable modelling of surface and underground water which will be vital for all the engineering aspects of the project as well as for environmental and social considerations.

1.18.6 ENVIRONMENTAL AND SOCIAL

The current baseline monitoring should continue in order to allow the impacts of the feasibility study and engineering to be correctly assessed in an updated environmental and social impact assessment. This assessment should include stakeholder engagement and aim to be equator principle and IFC performance standard compliant to ensure there are no barriers to financing construction.

1.18.7 FEASIBILITY STUDY AND FRONT END ENGINEERING BUDGET

A provisional budget for the Ilovitza project feasibility study and front end engineering has been prepared. The total estimated costs are US$11.7 million.

2.0 INTRODUCTION

The following report is a Prefeasibility Study written on behalf of Euromax Resources Ltd ("Euromax") concerning the Ilovitza copper gold deposit, which lies within the Ilovitza Property in southeastern Macedonia.

Euromax is a public company incorporated in British Columbia, Canada. Euromax is listed on the TSX Venture Exchange.

The maps and tables for this report were produced or derived from existing documents relating to the Property or produced specifically for the report as indicated.

Various authors have contributed to the report. Patrick Forward, BSc., FIMMM co-authored sections 1-6, 18.3.1.1-18.3.1.4, 18.7-18.9 and 19-27, however, Mr Forward is not an independent of the Company, and therefore does not take responsibility for these sections. A summary of the QPs responsible for each section of this report is provided in Table 2.1.

Table 2.1 Summary of QPs

1	Summary	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo
2	Introduction	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo
3	Reliance on Other experts	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo
4	Property Description Location	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo
5	Accessibility, Climate, Local resources, Infrastructure and Physiography	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo
6	History	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo
7	Geological Setting and Mineralisation	Tetra Tech	Robert Davies, B.Sc., EurGeol, CGeol
8	Deposit Types	Tetra Tech	Robert Davies, B.Sc., EurGeol, CGeol
9	Exploration	Tetra Tech	Robert Davies, B.Sc., EurGeol, CGeol
10	Drilling	Tetra Tech	Robert Davies, B.Sc., EurGeol, CGeol
11	Sample Preparation, Analyses and Security	Tetra Tech	Robert Davies, B.Sc., EurGeol, CGeol
12	Data Verification	Tetra Tech	Robert Davies, B.Sc., EurGeol, CGeol
13	Mineral Processing and Metallurgical Testing	Tetra Tech	Arun Vathavooran, PhD, CEng MIMMM SME
14	Mineral Resource Estimates	Tetra Tech	Robert Davies, B.Sc., EurGeol, CGeol
15	Mineral Reserve Estimates	ACA Howe Intl.	Doug Roy M.A.Sc., P. Eng
16	Mining Methods	ACA Howe Intl.	Doug Roy M.A.Sc., P. Eng

17	Recovery Methods	Tetra Tech	Arun Vathavooran, PhD, CEng MIMMM SME
18	Project Infrastructure		
18.1	Introduction	Tetra Tech	Laszlo Bodi, P.Eng.
18.2	Site Layout	Tetra Tech	Laszlo Bodi, P.Eng.
18.3	Site Facilities	Tetra Tech	Laszlo Bodi, P.Eng.
18.3.1	Lower Site	Tetra Tech	Laszlo Bodi, P.Eng.
18.3.1.1	ROM Pad and Primary Crusher	ACA Howe Intl.	Doug Roy M.A.Sc., P. Eng
18.3.1.2	Explosives Store	ACA Howe Intl.	Doug Roy M.A.Sc., P. Eng
18.3.1.3	Truck Maintenance Workshop and Stores	ACA Howe Intl.	Doug Roy M.A.Sc., P. Eng
18.3.1.4	Fuel Storage	ACA Howe Intl.	Doug Roy M.A.Sc., P. Eng
18.3.2	Upper Site	Tetra Tech	Laszlo Bodi, P.Eng.
18.3.3	Water Supply and Distribution	Tetra Tech	Laszlo Bodi, P.Eng.
18.3.4	Sewage and Water Treatment	Tetra Tech	Laszlo Bodi, P.Eng.
18.3.5	Remote Facilities	Tetra Tech	Laszlo Bodi, P.Eng.
18.3.6	Building List	Tetra Tech	Laszlo Bodi, P.Eng.
18.4	Roads	Tetra Tech	Laszlo Bodi, P.Eng.
18.5	Rail Connection	Tetra Tech	Laszlo Bodi, P.Eng.
18.6	Port Facilities	Tetra Tech	Laszlo Bodi, P.Eng.
18.7	Waste Rock Dumps	ACA Howe Intl.	Doug Roy M.A.Sc., P. Eng
18.8	Tailings Management Facility (TMF)	Phoenix Mining Consultants Ltd	David Carter BSc, PhD, MICE, FIMMM, C Eng, FGS
18.9	Oxide Ore Temporary Stockpile	ACA Howe Intl.	Doug Roy M.A.Sc., P. Eng
18.10	Water	Tetra Tech	Laszlo Bodi, P.Eng.
18.11	Dams	Tetra Tech	Laszlo Bodi, P.Eng.
18.12	Pipelines	Tetra Tech	Laszlo Bodi, P.Eng.
18.13	Power	Tetra Tech	Laszlo Bodi, P.Eng.
18.14	Capital and Operating Costs	Tetra Tech/ ACA Howe Intl.	Laszlo Bodi, P.Eng. Doug Roy M.A.Sc., P. Eng
19	Market Studies and Contracts	T.J. Metallurgical services	Gordon Antony Jackson, BSc(Eng), FIMMM
20	Environmental Studies, Permitting, and Social or Community Impact	Golder Associates	Gareth Digges La Touche, BSc., CGeol. EurGeol
21	Capital and Operating Costs	T.J. Metallurgical services	Gordon Antony Jackson, BSc(Eng), FIMMM
22	Economic Analysis	T.J. Metallurgical services	Gordon Antony Jackson, BSc(Eng), FIMMM
23	Adjacent Properties	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo
24	Other Relevant Data and Information	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo

25	Interpretation and Conclusions	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo
26	Recommendations	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo
27	References	ACA Howe Intl.	Dr David Patrick PhD, CEng, FIMMM, FAusIMM Daniel Leroux, M.Sc., P.Geo

Mr. Robert Davies of Tetra Tech conducted a property inspection between the 17th and 20th of June 2013. Mr. Davies is a B.Sc., EurGeol, CGeol and a QP as defined by NI 43-101.

Mr. Doug Roy of A C A Howe International Limited conducted a site visit between the 6th and 10th of April, 2014. Mr. Roy is a M.A.Sc., P. Eng and a QP as defined by NI 43-101.

3.0 RELIANCE ON OTHER EXPERTS

For legal aspects of the exploration licences, mining concessions, royalties and rights granted by the Government of Macedonia, along with environmental and political issues, the authors are relying on information provided by Euromax and its legal representatives, Mens Legis of Skopje, Macedonia. This is disclosed in Section 4.0.

In Section 18.8 Tailings Management Facility (TMF), work by the Faculty of Civil Engineering in Skopje, Macedonia has been used by the authors. This work was supervised and verified by the author of this section who is a QP as defined by NI 43-101.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

The property is located in the southeast of Macedonia, approximately 15 km to the west of the border with Bulgaria, Figure 4.1. The centre of the mineralised zone is located at 41.479° north, 22.836° east.

Figure 4.1 Regional Location Map

Source: Tetra Tech

The project is within the municipality of Bosilovo, approximately 20 km to the east of the city of Strumica, Figure 4.2.

Figure 4.2 Local Location Map

Source: Tetra Tech

4.2 PROPERTY DETAILS

A letter from Euromax's lawyer in Skopje, Macedonia: Mens Legis, dated 6th September 2013, states the following regarding title information:

"MENS LEGIS Law Firm is a legal counsel of Euromax Resources Ltd. Canada for Macedonia and a legal counsel of Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje) in Macedonia.

Euromax Resources (Macedonia) Ltd. (99.9%) and Euromax Resources (Macedonia) UK Ltd. (0.1%) are owners of the Macedonian Company Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje).

Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje) is the sole owner of the following mining concessions in Macedonia:

Concession for exploitation of mineral raw materials – ore of copper and gold on the locality "Village Ilovitza", municipality of Bosilovo, Macedonia (Concession agreement Ref. No. 24-6749/1 of 24.07.2012) known as Ilovitza 6. The concession is granted for period of 30 years. The concession area amounts 1.68 km^2.

Concession for detailed geological exploration of minerals – copper and gold on the location of Ilovitza, municipality of Bosilovo, Macedonia (based on the Concession agreement Ref. No. 04-02/11 of 21.02.2011) known as Ilovitza 11. The concession is granted for period of 4 years starting form 21.02.2011 and ending 21.02.2015. The concession area amounts to 3.27 km^2 and is listed in the Title Deeds No. 234, 235, and 566 in Cadastre Municipality (CM) of Shtuka, Title Deed No. 154 in CM Barbarevo and Title Deed No. 277 in CM of Susica.

According to the Decision of the Company for an increase of their core capital, admission of a new member with a new monetary contribution, change of the name of the Company and a change of the abbreviated name of the Company dated 19.12.2012 and the Agreement for establishment of the Company dated 03.01.2013, the firm "Company for production, trade and services PHELPS DODGE VARDAR DOOEL Skopje" is changed and states: "Company for production, trade and services EUROMAX RESOURCES DOO Skopje", and the abbreviated name is changed from "PHELPS DODGE VARDAR DOOEL Skopje" to "EUROMAX RESOURCES DOO Skopje".

The above mentioned changes are registered in the Central Registry of the Republic of Macedonia in accordance with the Resolution under reference number 30120130000235 dated 04.01.2013.

According to the due diligence MENS LEGIS has conducted in Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje) and in our understanding, Euromax Resources DOO Skopje (previously Phelps Dodge Vardar DOOEL Skopje) has fulfilled all liabilities based on applicable fees and taxes related to the above concessions. We are not aware of any outstanding liabilities based on the concession agreements.

The above listed concessions in our opinion are in good standing and are not subject to any liens or encumbrances."

When the Ilovitza 6 exploitation concession was granted an Environmental Impact Assessment for the whole project was also approved. It should be noted that the documentation for conversion of the Ilovitza 11 exploration concession to an exploitation concession is in preparation. The documentation comprises a detailed geological report, a scoping study of the proposed operation and a cadastral report. It is expected that the process will be completed in the first half of 2015.

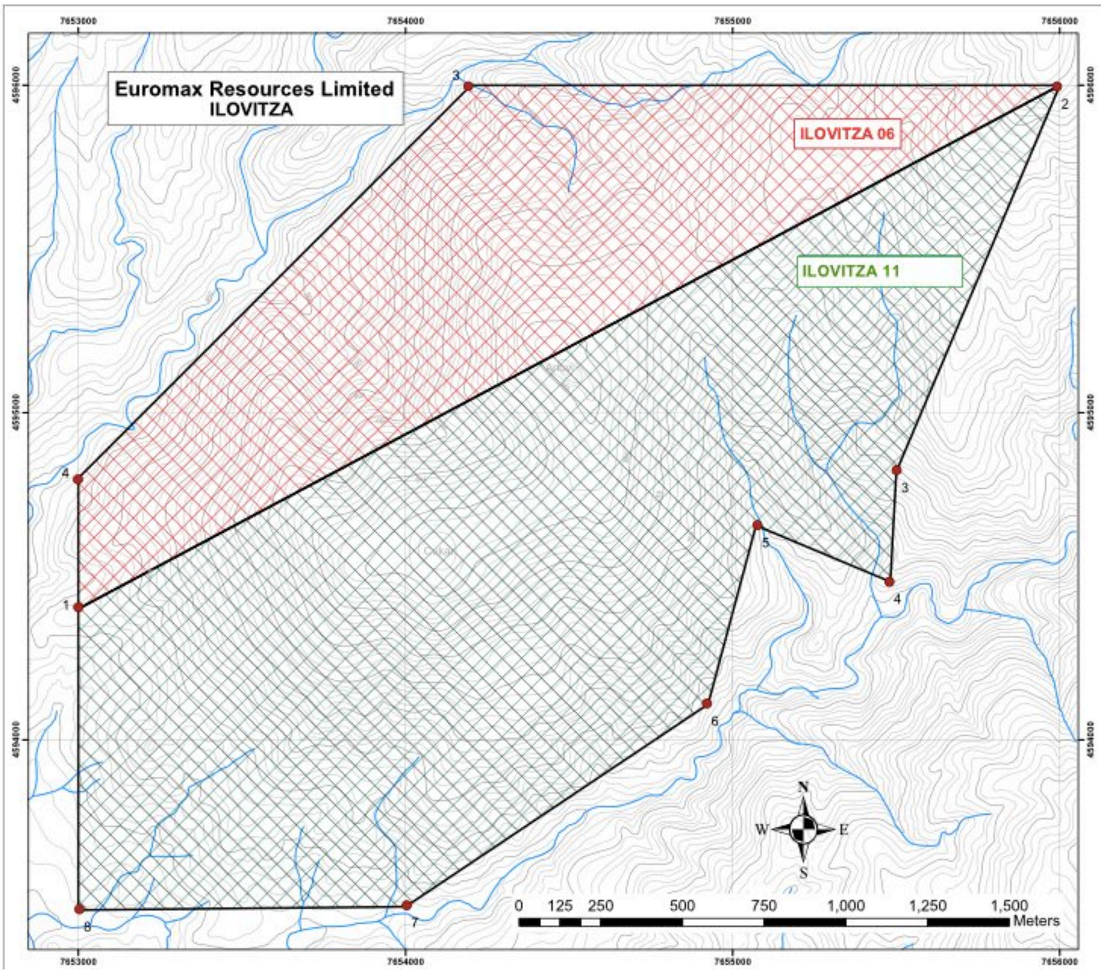
Royalties payable for metals produced from the exploitation concession are fixed at 2% calculated on a net smelter return basis through the national mining law.

The property boundaries and map coordinates are determined by paper staking and given in Table 4.1 and Figure 4.3 shows the concessions on a topographic map.

Table 4.1 Ilovitza Concessions

Concession	Point No.	UTM HKOGEL Projection	
		Easting	Northing
Ilovitza 11	1	7653000	4594400
Ilovitza 11	2	7656000	4596000
Ilovitza 11	3	7655500	4594820
Ilovitza 11	4	7655482	4594483
Ilovitza 11	5	7655071	4594655
Ilovitza 11	6	7654925	4594109
Ilovitza 11	7	7654000	4593490
Ilovitza 11	8	7653000	4593480
Ilovitza 6	1	7653000	4594400
Ilovitza 6	2	7656000	4596000
Ilovitza 6	3	7654200	4596000
Ilovitza 6	4	7653000	4594800

Figure 4.3 Concession Boundary Map



Source: Euromax

At this time there are no environmental liabilities identified on the property, nor are there any known material factors or risks that may affect access, title or the right or ability to perform work on the property.

Parts of this report relating to the legal aspects of the ownership of the mineral claims, rights granted by the Government of Macedonia, and environmental and political issues have been prepared or arranged by Euromax.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 SITE TOPOGRAPHY, ELEVATION AND VEGETATION

The project is situated on the western slopes of the Maleševske mountain range. The project area is part of Mount Ograzhden and ranges from 450 metres AMSL to a maximum elevation of approximately 860 metres AMSL. The main valley where the village of Ilovitza is located is at approximately 260 metres AMSL.

Figure 5.1 View of the Ilovitza Project Area Looking West

Source: Tetra Tech

The area comprises an undulating mountainous topography, with moderately rugged steep slopes up to 30° and generally rounded mountain tops, Figure 5.2. The Strumica valley floor is flat, Figure 5.3.

Figure 5.2 View of the Mountainous Topography Looking West from the Project Area

Source: Tetra Tech

Figure 5.3 View of Flat Valley Floor Looking East from the Project Area

Source: Tetra Tech

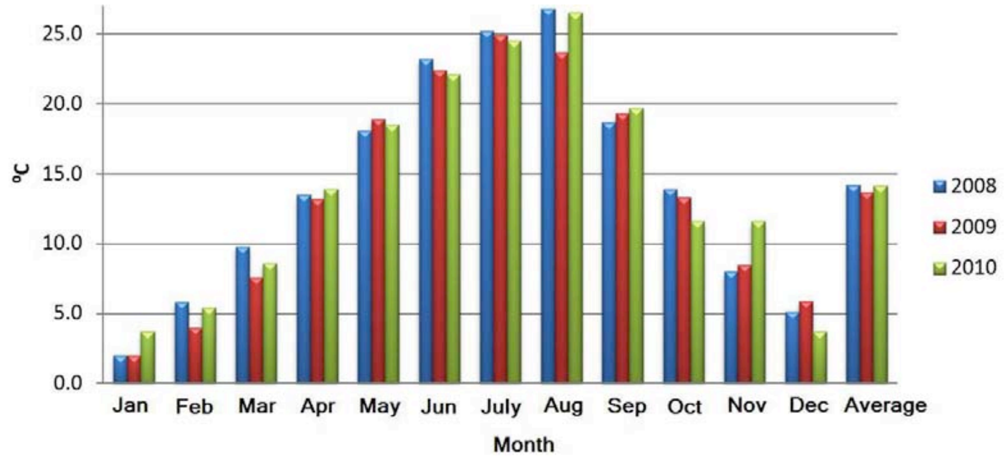
The mountains of the area are relatively unspoiled and contain a combination of Sub-Mediterranean Balkan forests and Balkan Mid-European forests. The forests are natural and include oak, beech, chestnut, black and white pine and walnut trees, with Sessile Oak forests dominating. Clearing of the forest for livestock pasture has occurred creating steppes on the less steep saddles and lower slopes. Where this has occurred, meadow grassland now predominates.

5.2 CLIMATE

Due to the location of the project, it is influenced by both Sub-Mediterranean and Eastern-Continental climatic systems. The region has long hot summers with high daily temperatures exceeding 40 degrees Celsius (°C) and low rainfall.

Winters are relatively cold with temperatures falling to -20°C for short periods. Figure 5.4 presents the average monthly air temperatures for 2008, 2009 and 2010 taken from the Strumica weather station.

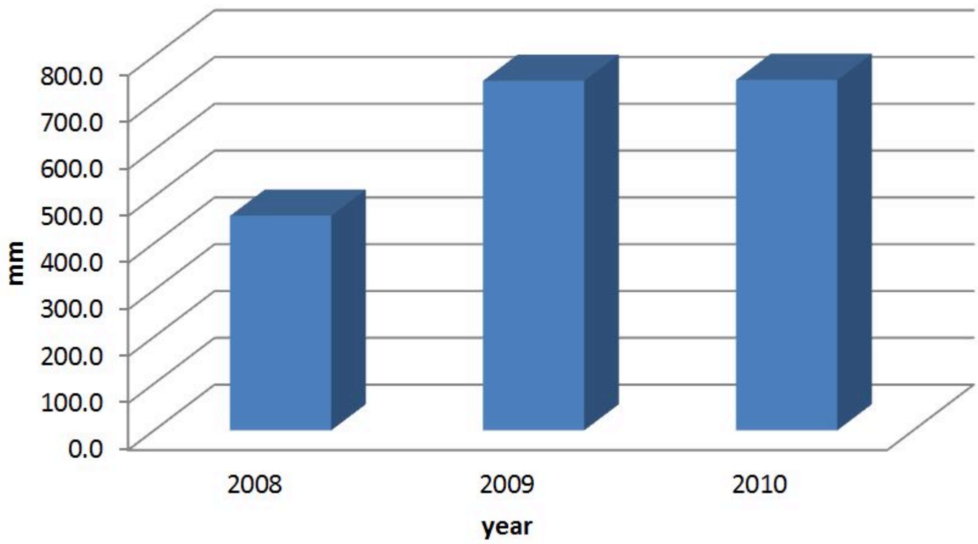
Figure 5.4 Average Air Monthly Temperatures



Source: Hydrometeorology Directorate, Strumica.

The annual average rainfall is approximately 600 millimetres (mm), Figure 5.5. The monthly precipitation fluctuations follow a typical Mediterranean regime, with October, November and December having the highest levels of precipitation and August and September the least.

Figure 5.5 Annual Rainfall at Strumica Weather Station 2008 to 2010



Source: Hydrometeorology Directorate, Strumica.

5.3 LOCAL POPULATION

The Maleševske mountain range in which the project is situated is largely unpopulated; with the vast majority of inhabitants residing within the Strumica valley to the west of the mountains.

The total population of the Strumica valley region is approximately 100,000 inhabitants. The villages of Shtuka and Ilovitza have 1,500 to 2,000 inhabitants each and they belong to the municipality of Bosilovo. Table 5.1 presents the statistics collected within the 2002 Census (the Census) regarding the municipality of Bosilovo.

Table 5.1 Population Statistics for Bosilovo Municipality

Total Population	Economically Active			Economically Non-active
	All	Employed	Unemployed	
5,741	4,208	3,190	1,018	1,533

Note: Economically non-active are defined by the Census as follows: Housewives (engaged on duties within the household in their own house), persons that are serving a sentence in a prison or are in military service, those permanently incapable of work, children under 15 years of age, students and pensioners.

The Census states that the majority of the population (70.7%) work in agriculture; 12.6% work in industry and 16.7% work in the services.

5.4 INFRASTRUCTURE

Infrastructure relating to the proposed Ilovitza project is described in detail in Section 18.0 of this report but is summarised below for completeness.

5.4.1 ROADS

The property is well served by asphalt paved roads. The site is located in the hills approximately 3 km northeast of the village of Ilovitza. The village of Ilovitza is located 3 km to the north of road number M6, which is a two lane asphalt paved road that leaves Strumica to the east. To reach the village of Ilovitza, a good quality 3 km single carriage asphalt paved road is taken from the M6.

Access to the interior of the property is by gravel roads that have been constructed to provide access for the drilling equipment. These tracks are generally cut into the weathered bed rock and are uneven and steep in places, but passable by four wheel drive vehicles.

The travelling distance and times from International Airports to Strumica (via car) are given below:

- Alexander the Great Airport, Skojpe: 1 hour 55 minutes, 151 km
- Thessaloniki International Airport, Greece: 1 hour 50 minutes, 146 km
- Sofia Airport, Bulgaria: 3 hours 18 minutes, 237 km.

The site is well linked to the road network in southern Europe. The E-75 International Road passes south through central Europe and Macedonia to the Port of Thessaloniki. The E-75 passes some 50 km west of the site and is linked to the M6 Highway, which passes to the south of Ilovitza, by Route 604 over the Belasic Mountains.

The M6 Highway also passes east to the border with Bulgaria and links with the town of Petrich (40 km away) and the Bulgarian road and rail networks.

5.4.2 RAIL

The railway infrastructure in the Republic of Macedonia has been in place since 1873, when the first railway track from Skopje to Thessaloniki in Greece was constructed. Today the railways network comprises about 925 km of single track standard gauge lines. The Macedonian railway network system is connected north-south with the railway network systems of Serbia and Greece. A further line is under construction to the north to Bulgaria. The railway transport system is managed by the publicly owned Macedonian Railways (Makedonski Železnici, MŽ) and at present, the only provider of railway services in the country.

The main station in Skopje is approximately 160 km from Ilovitza, although the railway follows the E-75 corridor, with loading sidings in Gevgelija, on the Greek border, approximately 70 km from the town of Ilovitza, Figure 5.6.

Figure 5.6 Macedonian Railway Network

Source: Wikipedia

Alternatively the Bulgarian rail network extends as far as Petrich, in south-western Bulgaria, approximately 45 km from site, Figure 5.7. Petrich is not shown on Figure 5.7, better to use Figure 18.15 from infrastructure sectionThis also links to the Pirdorp smelter at Chelopech, the Black Sea ports and the Mediterranean Port of Thessaloniki.

Figure 5.7 Bulgarian Railway Network

Source: Wikipedia

5.4.3 Port

The Thessaloniki Port is located 145 km away from the property and is understood to be suitable for both bulk cargoes and container goods.

5.4.4 Water Dams

There is an existing water reservoir on the edge of the site, which is used for agricultural purposes. Hydrological and hydrogeological studies are ongoing on the property to establish local ground water conditions. There is a larger water reservoir at Turija, some 15 kilometres to the West-Northwest which links to the area via a canal.

5.4.5 Power

Macedonia is connected to the European power grid via the National Grids of Bulgaria, Greece and Serbia as illustrated by Figure 5.8.

A 110 kilovolt (kV) power transmission line passes within 5 km of the site, with an existing substation near the town of Sushica, approximately 8 km from the site.
The available capacity of this line and substation are unknown at present.

Figure 5.8 Macedonian Power Distribution Network

Source: Macedonian Transmission System Operator

6.0 HISTORY

In 1958 T. Ivanovski and P. Chedomil mapped the south western slopes of the Ograzhen Mountain. They concluded that there were indicators of an ore body that included a deep sulphide deposit. (Ivanovski, T. and Chedomil, P. 1957).

Mineralisation was discovered in 1973 by the Skopje Bureau of Geology team, headed by D. Denkovski. Copper (Cu), lead (Pb) and zinc (Zn) mineralisation was linked to the dacite intrusions at Ilovitza. (Denkovski. D, 1971).

Prior to Euromax, the property was held and explored by PDX, who completed drilling campaigns between 2004 and 2006. Euromax has integrated the results of the PDX drilling into the drillhole database that has been provided to Tetra Tech.

Between 2004 and 2006, PDX completed the drilling shown in Table 6.1.

Table 6.1 Summary of Main Intercepts from the 2004 to 2006 Drilling Campaigns

Drillhole	Cu (%)	Au (g/t)	Mineralised Intercepts
PDIC-04-01	-	-	Very low values
PDIC-04-02	0.15	0.20	288 m (98 to 386 m) at a cut-off 0.1% Cu
PDIC-04-03	0.24	0.27	147.7 m (393 to 540.7 m) at a cut-off 0.1% Cu
PDIC-05-04	-	0.29	132 m (165 to 297 m) at a 0.1 g/t Au cut-off
PDIC-06-06	0.23	0.30	450 m (54 to 504 m) without a cut-off
PDIC-06-07	0.28	0.45	93 m (132 to 225 m) without a cut-off
PDIC-06-08	0.25	0.36	393 m (57 to 450 m) without a cut-off

Source: Euromax

In 2008 Euromax signed an option agreement with PDX to acquire 100% of the property.

6.1 Historical Mineral Resources

The maiden Mineral Resource for the property was estimated by Geoffrey S. Carter, Professional Engineer (P.Eng), of Broad Oak Associates, Toronto, on behalf of Euromax. The Resource was disclosed in a Technical Report, filed with the Canadian Securities Administrators, on the 12th August 2008. (Carter, 2008). The Resource was classified using CIM standards.

The historical Resource estimate was completed using polygonal estimation techniques, based upon eight holes drilled on approximately 200 m centres.

Mr. Carter estimated that there are 303 million tonnes (Mt) of Inferred sulphide Resources at 0.23% Cu, 0.32 g/t Au, 0.005% Mo at a 0.2% Cu cut-off, for a copper equivalency of 0.51%.

The copper equivalent values were calculated using total contained metal and 100% recoveries using: US $550 /oz for gold, US $20 /lb for molybdenum, and US $1.25 /lb for copper. The deposit was estimated to contain 1,560 million pounds of copper, 2.9 million ounces of gold, and 34.6 million pounds of molybdenum (Carter, 2008).

The 2008 Resource estimate was superseded by a Mineral Resource estimate prepared by Moose Mountain Technical Services (Moose Mountain), with an effective date of the 15th February 2012 (Bird and Morris 2012). The Moose Mountain Resource estimate was presented in a Technical Report released on the 17th May 2012 and re-stated in a PEA prepared by Tetra Tech and filed with the Canadian Securities Administrators on the 22nd March 2013.

Tetra Tech prepared a Resource update for the project with an effective date of the 26th of July 2013. The Resource estimate was disclosed within a Technical Report, filed with Canadian Securities Administrators on the 16th September 2013. The Tetra Tech Resource estimate has been discussed in more detail in section 14.12, where it is compared to the current estimated Resource.

Euromax is not treating the historical Mineral Resource estimates as current. The Resource estimates completed by Moose Mountain in 2012 and Tetra Tech in July 2013 have been entirely superseded by the estimated Resource disclosed within this report.

6.2 Historical Mineral Reserves

No Mineral Reserves have been estimated for the project. However, as part of the PEA of March 2013, Tetra Tech completed a preliminary optimisation using CAE Mining's NPV Scheduler software in order to estimate the conceptual mineable amount of the Resource contained within an optimised pit shell. The optimisation used the Resource model prepared by Moose Mountain, with an effective date of the 15th of February 2012.

The PEA was preliminary in nature and included Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves. There is no certainty that the preliminary economic assessment will be realised.

Only sulphide and mixed materials were used for the optimisation. The optimisation used the parameters outlined in Table 6.2.

Table 6.2 PEA Open Pit Optimisation Parameters

Parameter		Unit	Quantity
Metal Price	Copper	US$ /lb	3.02
	Gold	US$ /oz	1,264
Cost of Sales	% of metal price	%	10
Annual Production Rate Sulphides	-	Mt /a	8
Annual Discount Factor	-	%	7
Mining Cost	-	US$ /t	2.00
Mining Dilution	-	%	5
Mining Losses	-	%	95
Processing Cost	Sulphide – float	US$ /t	6.34
Metal Recovery – Sulphide	Copper	%	90
	Gold	%	83
General and Administration Costs	Included in selling costs	$US /t	-
Overall Slope Angle	-	degrees	45

Note: Mt /a = Million tonnes per annum

At the time of preparing the PEA, geotechnical drilling had not been undertaken so final wall angles of 45° were assumed. An assumed cut-off grade of 0.27 g/t Au was applied.

As a result of optimisation, a number of ultimate pit shells were produced. Each of the ultimate pit shells (final pit envelope) contains the maximum Resources for the given economic and technical criteria, based upon maximising Net Present Value (NPV). The NPV in these models consider operating cost, but not capital costs. The results of the optimisation runs are presented in Table 6.3.

The tonnages reported in Table 6.3 correspond to a pit shell with a maximum NPV adhering to the applied economical restrictions.

Table 6.3 PEA Pit Optimisation Results

Material		Tonnes (t)	Cu (t)	Cu Grade (%)	Au (oz)	Au Grade (g/t)
Sulphide	Indicated	11,782,040.00	28,174	0.24	145,225	0.38
	Inferred	131,275,520.00	295,031	0.22	1,792,383	0.42
Waste		401,206,080.00	-	-	-	-
Strip Ratio		2.80	-	-	-	-

Note: The PEA was preliminary in nature. It included Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves. There is no certainty that the PEA will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The results of the pit optimisation are deemed no longer relevant due to the Resource model having subsequently been significantly updated. Euromax has retained Tetra Tech as the main consultant to complete a PFS on the project.

The PFS will include the Mineral Resources disclosed within this document and maiden Mineral Reserves for the property. The PFS is due to be completed within the first quarter of 2014.

Euromax are not treating the pit optimisation study completed as part of the 2013 PEA as current.

6.3 HISTORICAL PRODUCTION

No historical production has occurred on the property.

7.0 GEOLOGICAL SETTING AND MINERALISATION

7.1 REGIONAL GEOLOGY

Ilovitza is a porphyry copper-gold deposit, located in a northwest-southeast striking Cenozoic magmatic arc, that covers large areas of Central Romania, Serbia, Macedonia, Southern Bulgaria, Northern Greece and Eastern Turkey, see Figure 7.1.

There are five widely recognised deposit types within the belt, of which the porphyry copper-gold, replacement lead-zinc and low sulphidation epithermal gold are economically the most significant.

The major Cenozoic metal deposits, from northwest to southeast are:

- Rosia Montana low sulphidation epithermal gold and Rosia Poeni porphyry copper-gold in Romania.
- Trepcha lead-zinc in Kosovo.
- Sasa lead-zinc & Ilovitza & Buchim porphyry copper-gold deposits in Macedonia.
- Olympias porphyry copper-gold and Perama Hill low sulphidation epithermal gold in Greece.
- Madan lead-zinc and Ada Tepe low sulphidation epithermal gold in Bulgaria, and Ovacik low sulphidation epithermal gold in Turkey.

Figure 7.1 Ilovitza Location and Regional Geological Setting

Source: Euromax

A series of significant isolated deposits formed over the last 40 million years within at least ten discrete metallogenic belts.

Individual metallogenic belts are typically tens to hundreds of kilometres long, dominated by a single deposit type and were active for periods of 5 to 10 million years. The metallogenic belts are generated under post-collision, moderate extensional tectonic conditions, predominantly in back arc settings, over previously thickened crust.

The porphyry deposits in the region are in close spatial and temporal association with intermediate to felsic, medium to high potassium calc-alkaline igneous rocks. The low sulphidation epithermal deposits are related to bimodal volcanic rocks.

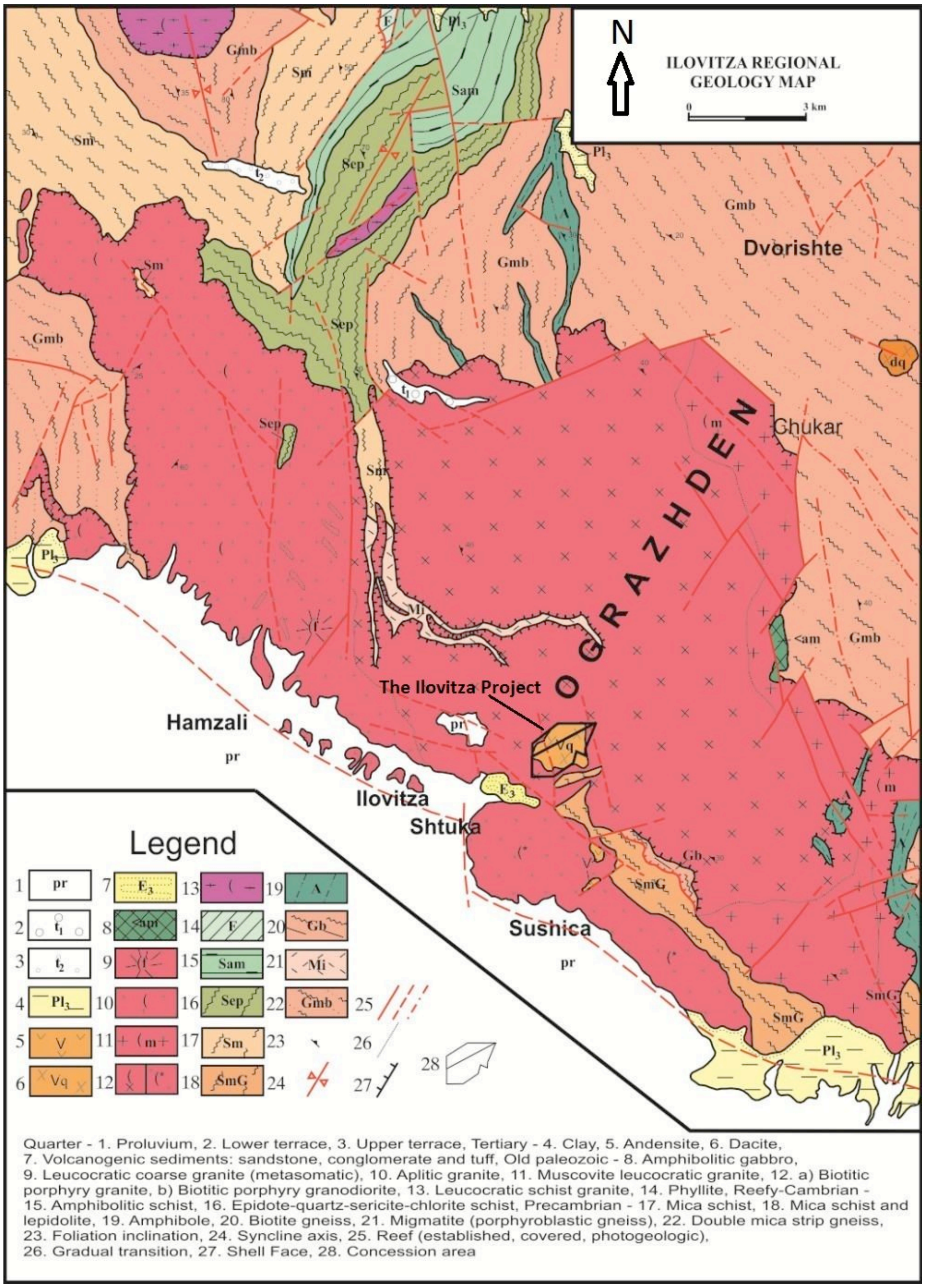
The Ilovitza deposit, which was emplaced circa 29 Ma, is an isolated porphyry copper-gold deposit located about 20 km west of the 33-38 Ma Osogovo-Besna-Kobila lead-zinc belt and about 30 km east of the 22-27 Ma Leche-Buchim-Chalkidiki copper-gold belt.

There is a well recorded, systematic thickening of the crust from southwest to northeast across the metallogenic belts, from 34 to 35 km in thickness beneath the Leche-Buchim-Chalkidiki belt, approximately 37 km in thickness below the Ilovitza deposit, 41 to 45 km in thickness below the Osogovo-Besna Kobila belt and greater than 50 km in thickness further northeast under the Rodope mountains.

7.2 PROPERTY GEOLOGY

The Ilovitza porphyry system is about 1.5 km in diameter and is associated with a poorly exposed dacite-granodiorite plug, emplaced along the north-eastern border of the northwest-southeast elongate Strumitza graben, see Figure 7.2. The exact location of the deposit is controlled by major north-south cross cutting faults and minor northwest- southeast faulting, parallel to the faulted border of the graben.

Figure 7.2 Local Geological Setting



Source: Euromax

The Strumitza graben (shown in white on Figure 7.2) is a typical post-collision extension structure, about 30 km by 10 km in size and up to more than 1 km in depth. The graben has been filled with terrigenous clastic sediments and felsic volcanic rocks over the last 40 million years.

At surface, the Ilovitza intrusive complex consists of a central dacitic breccia diatreme, approximately 1.3 km in diameter. The diatreme is intruded by at least one dacite and two granodiorite porphyry stocks that have generated several hydrothermal pulses, resulting in widespread multi-phase veining within a mineralised stockwork.

The Ilovitza porphyry is centred on a hill of more than 400 m of absolute relief, surrounded at lower elevations by numerous small dykes and irregular bodies of dacitic tuff and breccias and intermediate volcanic rocks.

The Ilovitza magmatic complex is emplaced into lower Palaeozoic granite. The granite is locally weakly foliated, coarsely porphyroblastic, and forms a roughly northwest-elongate body some 4 by 12 km in size, intruding Precambrian mica schist and gneiss. Portions of the main dacitic diatreme locally contain abundant xenoliths of basement granite near the lithological contact.

Numerous isolated outcrops of dacite porphyry elsewhere within the diatreme breccia commonly have vertical flow laminations but are too small to show as individual mapable units. Drilling demonstrates that dacite and granodiorite porphyries expand at shallow depths into a fairly continuous body.

7.3 ALTERATION

The phyllic alteration is elongated towards the south of the main porphyry and may represent an alteration overprint from a deeper as yet unidentified intrusive that would require deep drilling to confirm. The mineralisation originally identified in the area occurs to the south of the pervasively altered zone and comprised silica alunite and vuggy silica breccias indicative of high sulphidation epithermal veining and may represent high level mineralisation associated with this concealed intrusion. There is also indication from the IP survey of a deep seated feature with similar characteristics to the main porphyry body developing in the extreme south of the area.

Alteration related to Cenozoic magmatic activity at Ilovitza is variably present over an area of about 8 km^2 (see Figure 7.3 and Figure 7.4). Pervasive alteration is largely confined to a roughly 1.5 km^2 area in and adjacent to the main intrusive complex. Smaller areas of pervasive and structurally-controlled alteration extend somewhat asymmetrically to the south and east of the intrusive complex. Alteration has not been studied in detail, but visual observations document the following zones:

- **Distal:** Structurally-controlled silicification, and silica-or silica-alunite-sulphide/ Iron Oxide (FeOx) altered rocks (‘advanced argillic’), surrounded by narrow zones of clay alteration and bleaching, hosted in both fractured zones within basement granite, or within dykes of Cenozoic tuff-breccia. Such occurrences are present in zones of a few metres up to approximately 100 m in maximum dimension, and occur throughout the entire 8 km^2 altered area.

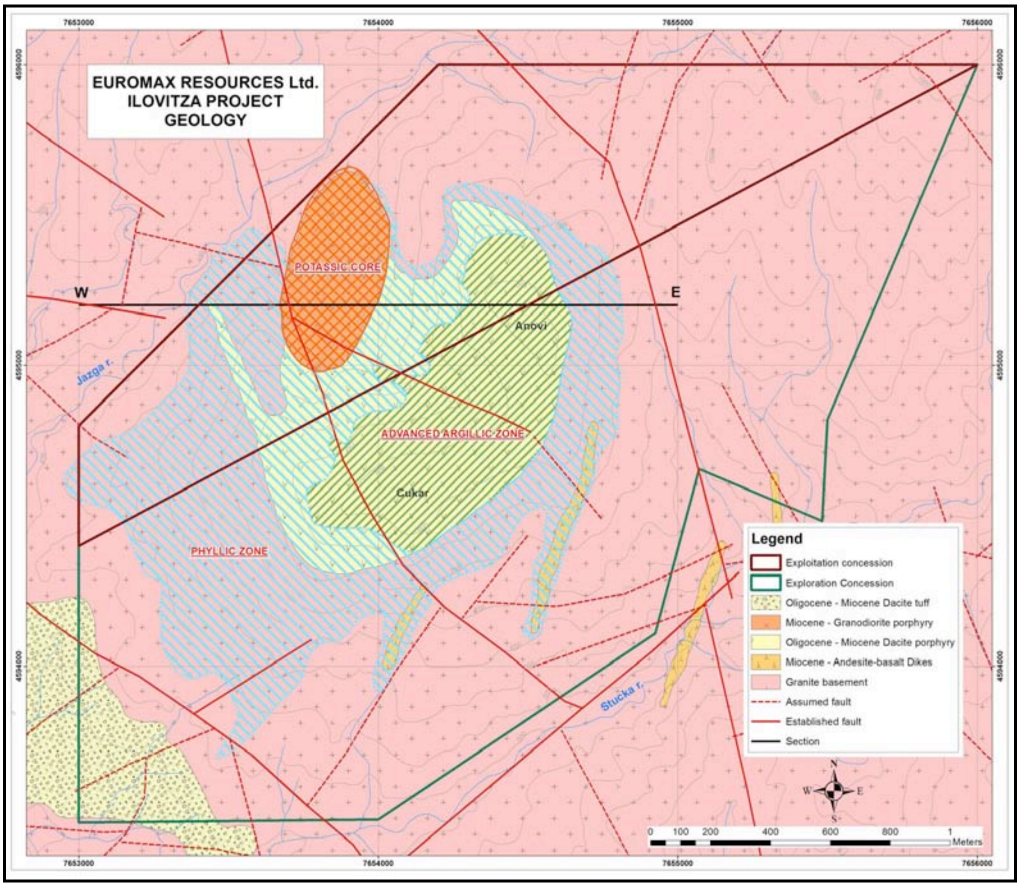
- **Proximal:** Pervasive quartz-sericite-clay-Iron Oxide (‘phyllic’) alteration, which contains larger bodies of quartz-alunite alteration, hosted in both basement granite and Cenozoic magmatic rocks.

- **Proximal stockwork:** Quartz-pyrite/ Iron Oxide alteration and intense clay-sericite alteration largely confined to Cenozoic dacitic breccia and dacite-granodiorite intrusive rocks.

- **Central**: Quartz-magnetite-sulphide/ Iron Oxide stockwork and dissemination, with matrix alteration of illite-sericite, chlorite ('intermediate argillic alteration) containing patches of residual secondary biotite and potassium feldspar, hosted in dacite-granodiorite porphyry, and minor andesite and latite-andesite porphyry dykes.
- **Supergene:** Sulphide oxidation, leaching, and argillisation, locally extending as much as 150 m below surface.

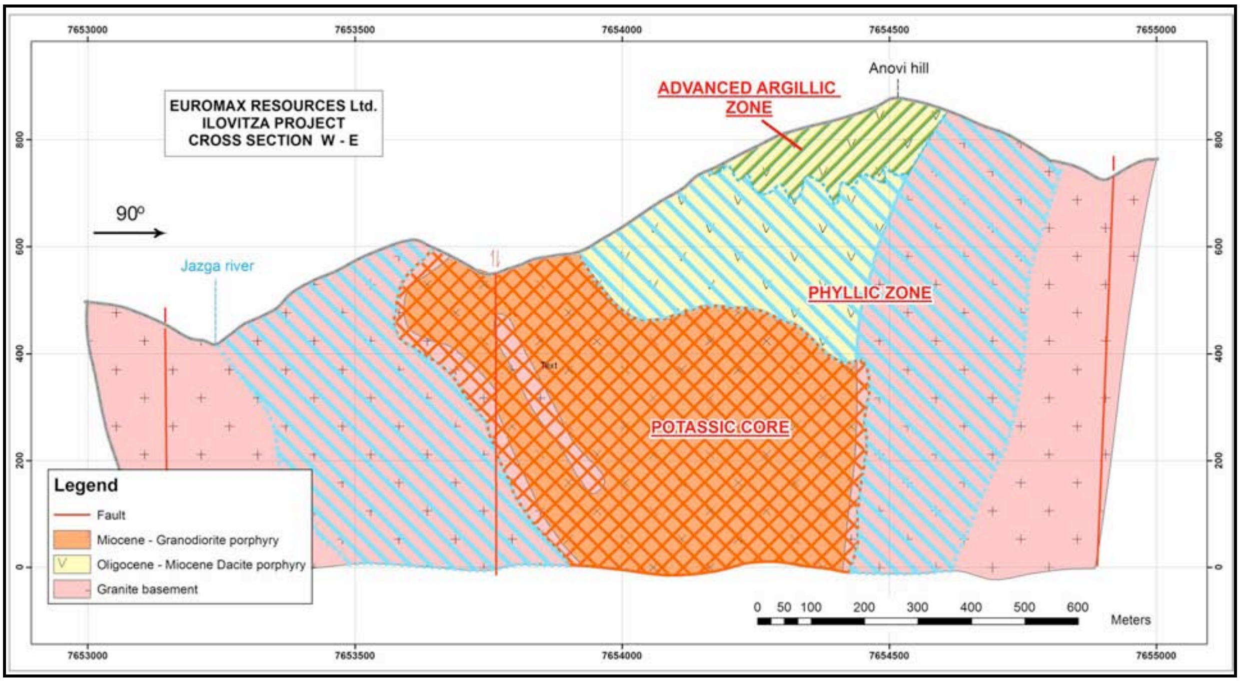
Distinct propyllitic alteration is absent at Ilovitza. Alunite is typically sugary to coarsely crystalline in texture and locally occurs as mono-mineral, coarsely-crystalline fibrous veins, breccia cement, and phenocryst replacement in pervasive silica-alunite altered dacite breccia. It appears to be hypogene in origin.

Figure 7.3 Property Geology and Alteration Plan



Source: Euromax

Figure 7.4 Property Geology and Alteration (West-East Cross Section)



Source: Euromax

7.4 MINERALISATION

The main sulphide mineral at Ilovitza is chalcopyrite, followed by pyrite and secondary copper sulphides such as chalcocite, covellite and bornite. Molybdenite, galena and sphalerite are present in minor amounts, and occasional traces of sulphosalt minerals such as tetrahedrite-tennantite and tellurides of gold and silver are observed.

High temperature oxide mineralisation such as magnetite, dominates at depth, associated with pyrrhotite and chalcopyrrhotite in what is interpreted as the core of the system.

A variety of iron hydroxide group minerals are largely developed within the oxidation and cementation zones. Very occasionally gold nuggets are observed at the base of the oxidation zone.

Ilovitza was known historically for minor lead-zinc (and minor copper) and gold occurrences (Cifliganec, 1993), confined to distal and peripheral silica-iron oxide and silica-alunite bodies outside of the pervasively-altered intrusive complex. Deep oxidation and leaching (up to 150 m) of the topographically-elevated intrusive complex obscured its sulphide content.

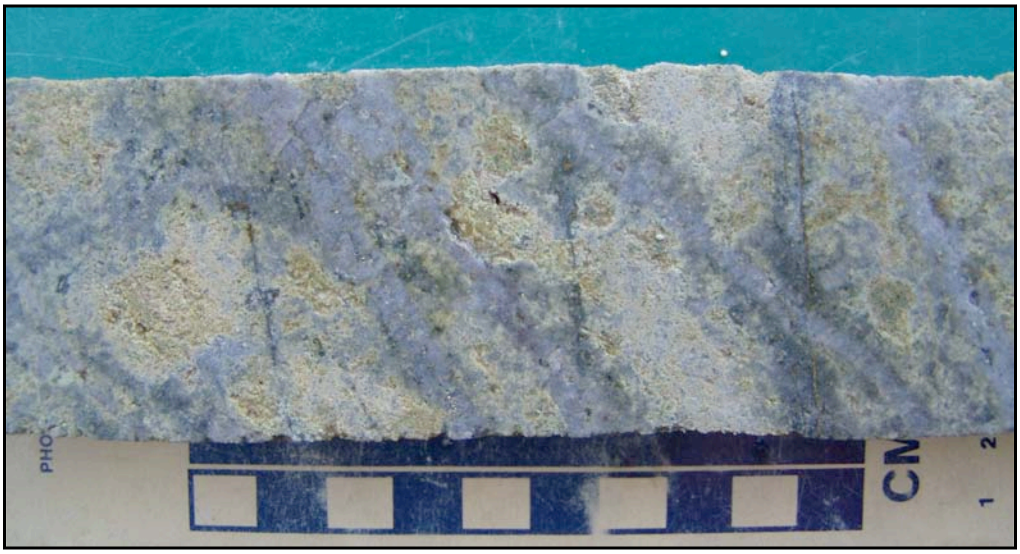
The only visible evidence of copper mineralisation at surface includes: Traces of enargite found in one ledge; very rare green copper oxides, and thin chalcocite coatings on sparse un-oxidised pyrite deposits exposed in a creek below the leached cap to the west of the porphyry. Since the start of detailed exploration on the property, copper mineralisation has become more obvious on surface, on newly bulldozed drill-roads.

Subsurface porphyry copper-gold mineralisation is expressed at surface by a limonitic, leached stockwork zone approximately 900 m by 600 m in size, containing 0.08 to 0.70 ppm Au, 50 to 450 ppm Cu and 10 to 128 ppm Mo.

At the highest elevations, central portions of this leached cap contain up to 50 to 100 quartz and limonite-quartz veins per metre, comprising up to 25% of the rock volume, within a sericitised and intensely (supergene) clay-altered matrix.

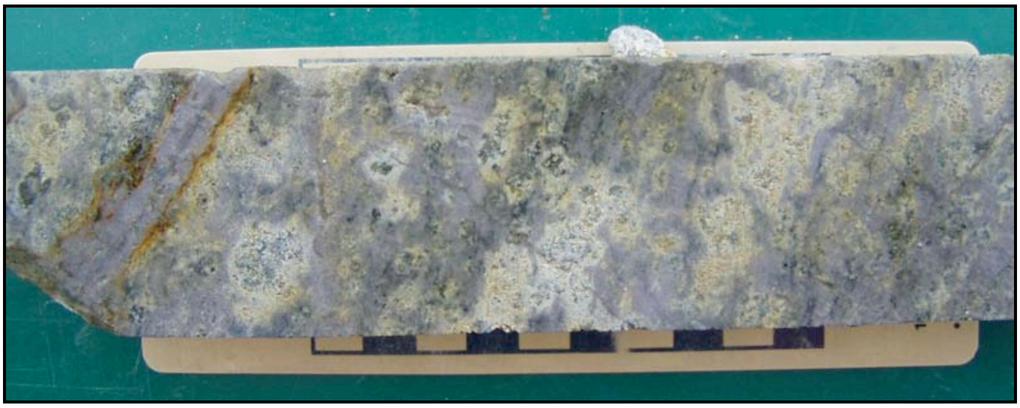
Quartz-dominant veinlets are largely devoid of sulphide cavities and have the texture of both discontinuous A-type (Figure 7.5) veins as well as linear, centre-line B-type veins (Figure 7.6).

Figure 7.5 A-Type Quartz Veins and Superimposed Late Stage Argillisation



Source: Gerhard Westra

Figure 7.6 B-Type Vein (Left side of core) Cutting Earlier Silicification

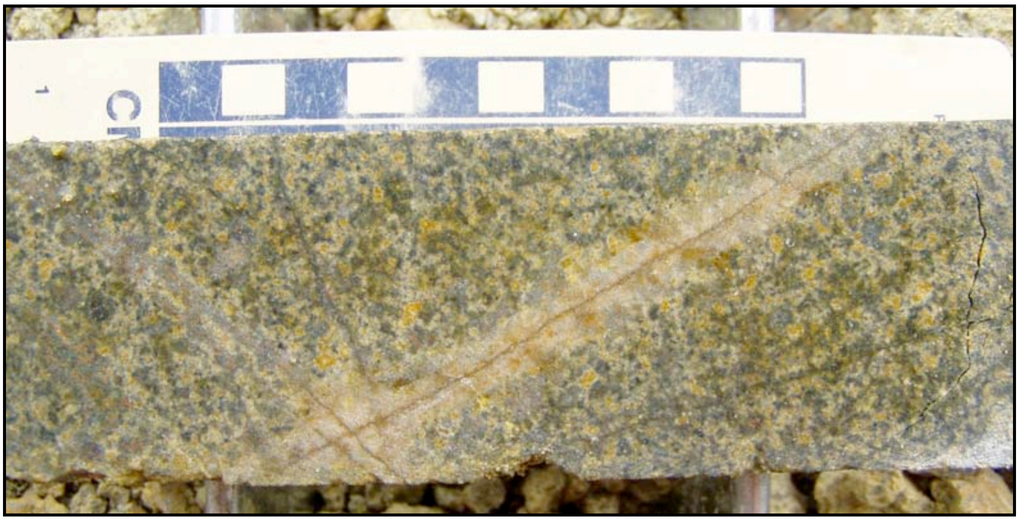


Source: Gerhard Westra

Small exposures at lower elevations (circa 550 m) on the western side of the stockwork zone contain 3 to 7% veinlets and disseminated magnetite, approximately 1% goethitic limonite, and lesser quartz-magnetite veinlets, within a silicified, chloritised matrix, likely representing intermediate argillic overprint of former K-silicate alteration.

Ground magnetic surveys clearly define the subsurface magnetite alteration as a roughly north northeast-elongated, 80 to 1600 nano-tesla (nT) magnetic high, roughly 800 m long and up to 300 m wide, which appears to plunge to the east and south. Peripheral portions of the overall stockwork zone are characterised by sparse, hairline pyrite fractures with quartz-sericite halos, corresponding to D-type veins, Figure 7.7.

Figure 7.7 Narrow D-Type Vein with Sericite Halo



Source: Gerhard Westra

Surface rock chip sampling, drilling, and to a lesser extent soil sampling define a large body containing 0.1 to 1 ppm Au coinciding with the zone of stockwork veining.

Hypogene copper grades greater than 0.15% are largely due to disseminated chalcopyrite, which appears largely confined to the western two-thirds of the stockwork zone. The hypogene copper mineralisation is characterised by the presence of magnetite (+martite), chlorite and a relict biotite-K feldspar groundmass. The pyrite and chalcopyrite are usually present in equal proportions. There is an increase in the proportion of pyrite in the eastern zone of the porphyry at higher elevations, along with more intense phyllic / argillic alteration, and an absence of magnetite.

A supergene-enriched zone ranging from 9 to 70 m in thickness and containing 0.25 to 0.69% Cu as chalcocite and covellite represents enrichment of about 1.5 to 3 times the hypogene grades.

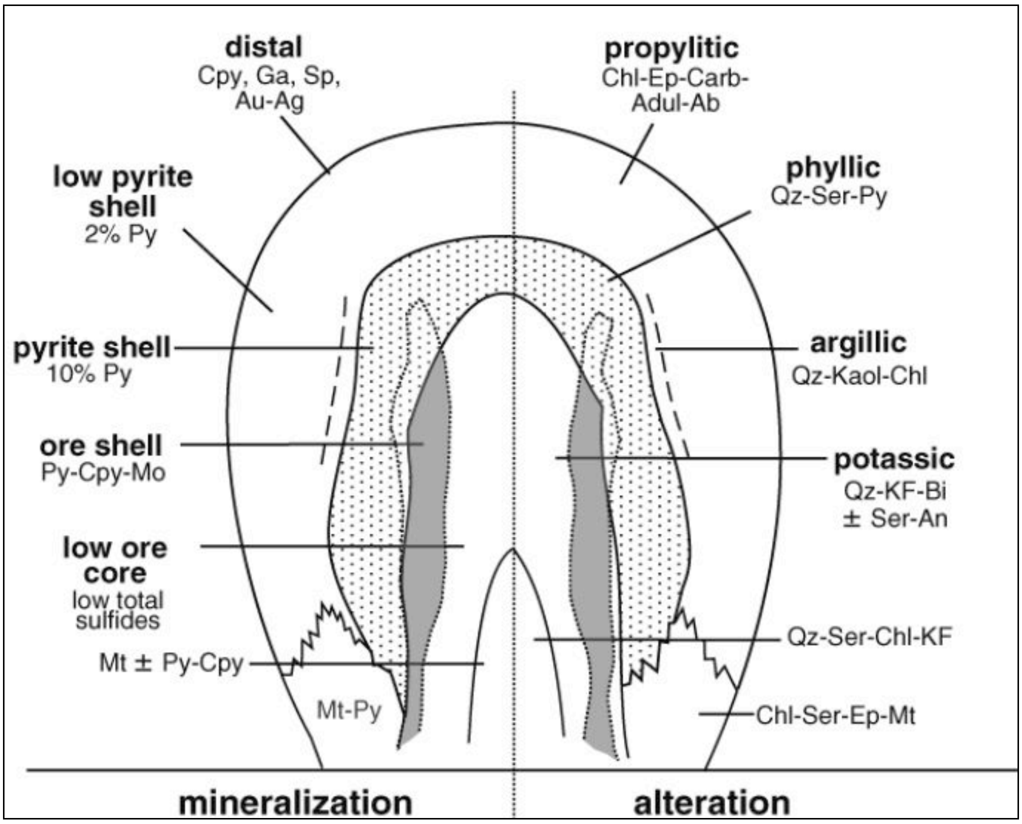
The leached cap generally contains approximately 150 ppm Cu. Molybdenum averages 20 to 80 ppm throughout the copper / gold mineralised zone and is present largely as molybdenite in quartz veinlets that lack regular distribution.

8.0 DEPOSIT TYPES

The mineral deposit type at Ilovitza is 'Alkaline Copper Gold Porphyry'. The deposit has characteristics which are typical for this deposit type.

The mineralisation is spatially, temporally and genetically associated with hydrothermal alteration of the intrusive bodies and host rocks. Figure 8.1 presents a simplified / idealised section through a porphyry deposit, showing the generally accepted model in relation to alteration and mineralisation.

Figure 8.1 Idealised Hydrothermal Alteration & Typical Mineralisation Associated with Porphyry Deposits



Source: Lowell & Guilbert 1970

Notes: Idealised alteration zoning pattern in the Lowell-Guilbert model of porphyry deposits (after Lowell & Guilbert 1970). At Ilovitza, supergene leaching and enrichment have overprinted the simple hydrothermal alteration model outlined in Figure 8.1.

Key

±	=	with or without	Ep	=	Epidote
Ab	=	Albite	Ga	=	Galena
Adul	=	Adularia	Kaol	=	Kaolinite
An	=	Anhydrite	KF	=	Potassium Feldspar
Bi	=	Bismuth	Mo	=	Molybdenite
Carb	=	Carbonate	Qz	=	Quartz
Chl	=	Chlorite	Ser	=	Sericite

As is typical of this deposit type, stockworks, veinlets and disseminations of pyrite, chalcopyrite, bornite and magnetite occur in large zones of mineralisation in and adjoining porphyritic intrusions of diorite composition.

Subsequent supergene leaching / enrichment and advanced argillic alteration have overprinted the typical hydrothermal alteration and mineralisation pattern in the upper portion of the deposit.

9.0 EXPLORATION

The following exploration activities have been undertaken on the property:

9.1 FIELD MAPPING AND ROCK CHIP SAMPLING

Detailed geological mapping was completed on 1:2,000 and 1:5,000 scales and comprised observations with respect to petrology, style of alteration and mineralisation.

Rock chip samples were collected from the outcrops which were identified as having potential to host mineralisation. The samples were submitted to the assaying laboratory described in Section 11.0. The laboratory completed multi-element analysis using inductively coupled plasma optical emission spectroscopy (ICP-OES). Outcrops and chip sample locations were delineated using a hand held Global Positioning System (GPS). In addition, all of the existing roads and trails were GPS tracked.

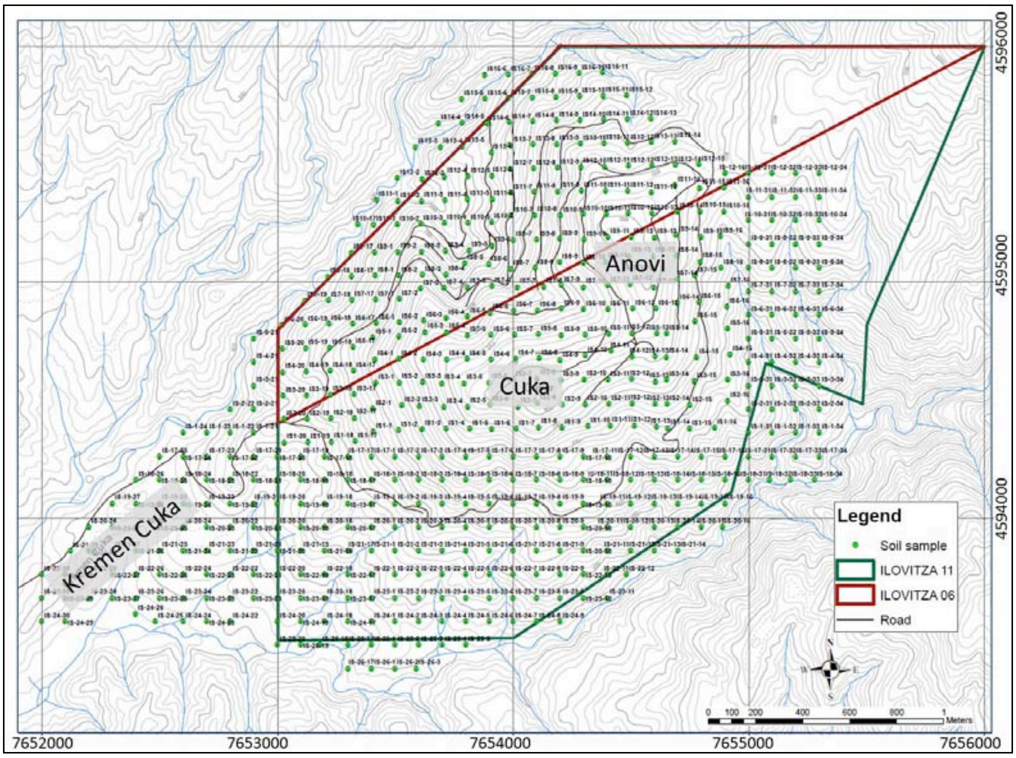
9.2 SOIL GEOCHEMISTRY SURVEY

In total, three phases of soil sampling have been undertaken on the property, resulting in a total of 540 sampling points arranged on a 100 x 100 m grid, see Table 9.1 and Figure 9.1.

Table 9.1 Soil Geochemistry Samples

Soil Sampling Campaign	No. of Sample Points	Company	Area
2004	146	PDX	Anovi & Cukar
2006	111	PDX	Anovi & Cukar
2007	283	PDX & Euromax	Kremen Cukar
Total	**540**	-	-

Figure 9.1 Soil Geochemistry Sampling Locations



Source: Euromax

The total area covered by the soil geochemistry sampling was approximately 5,000 m^2.

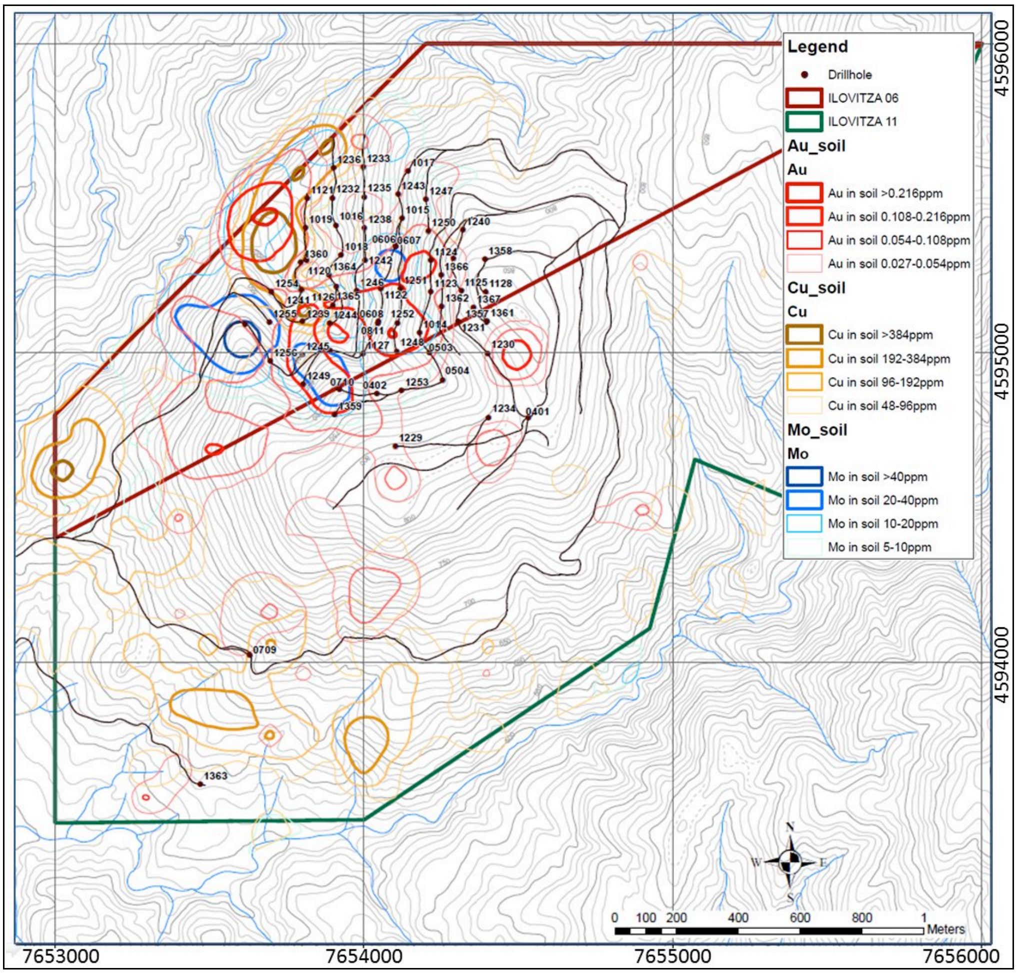
The soil sampling targeted the subsoil horizon, which is generally at a depth of 20 to 30 centimetres (cm) (the "B" horizon of the soil profile), as this unit generally contains the accumulated minerals. The soil surveys were completed by initially removing the humus topsoil layer with a spade, before taking a 2 to 3 kg sample of the subsoil. The remainder of the soil was restored to the sampling location and rehabilitation of disturbed areas was performed.

The samples were dried at room temperature and sieved using an 80# sieve (0.178 mm mesh sieve). Samples were split via quartering to produce two 150 gram (g) representative sub-samples. One of the 150 g sub-samples was sent to the laboratory to be assayed and a duplicate was retained. Pulverisation was completed by the testing laboratory.

Assaying was completed by Eurotest Control AD in Sofia (described further in Section 11.0). Assaying was completed using ICP analysis, with samples showing elevated gold, copper and molybdenum grades analysed using Atomic Absorption Spectroscopy (AAS).

Results of soil sampling over the property indicate significant copper anomalies (>200 ppm Cu) to the northwest, southwest and south of the mineralised intrusive, Figure 9.2. These anomalies are believed to represent down slope dispersion of the copper from the central area of mineralisation. In contrast, significant gold (>0.10 ppm) and to a lesser extent molybdenum (>20 ppm), show less down slope dispersion and more accurately delineate the underlying mineralisation.

Figure 9.2 Soil Geochemistry Anomalies



Source: Euromax

9.3 GEOPHYSICAL SURVEYS

9.3.1 MAGNETIC SURVEY

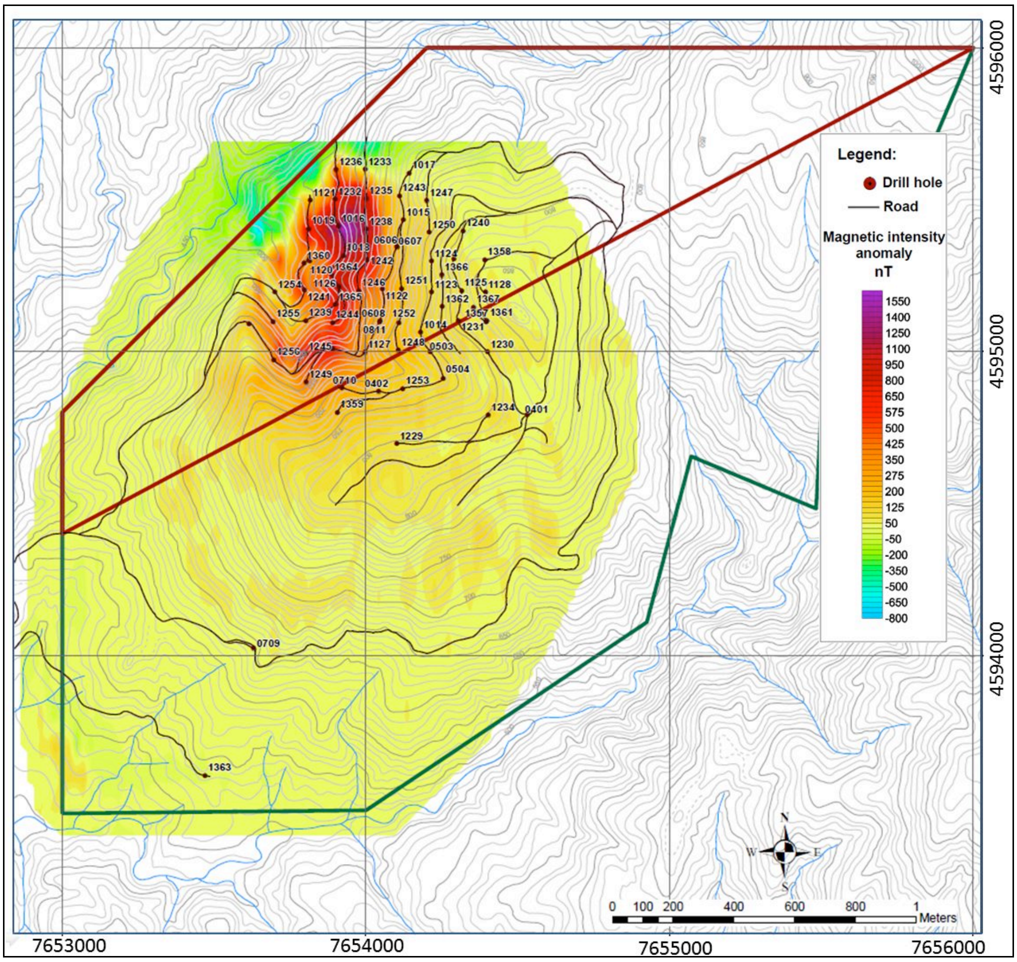
A total magnetic intensity survey was conducted by PDX on behalf of Euromax between 1st and 6th of April 2005. Twenty-four east-west lines spaced 100 m apart were surveyed with readings taken every 10 m. Three proton magnetometers, manufactured in Poland, were used. Their sensitivity was ± 0.1 nT. Two magnetometers were used for traversing while the third was used as a base station.

The aim of the survey was to outline the lateral and vertical extension of stockwork zones with secondary magnetite enrichment intersected in drill holes PDIC-04-03, PDIC-04-02 and PDIC-04-01. The stockworks comprise quartz-sulphide veinlets in illite-sericite-chlorite altered granodioritic host rocks.

Magnetic susceptibility measurements were taken at an average interval of about 10 cm on core from these holes using an electromagnetic inductance bridge. The sensitivity of the bridge was in the order of 0.0001 SI.

A high amplitude magnetic anomaly was outlined; the magnetic susceptibility measurements demonstrated that the only magnetic rocks in the area are the secondary magnetite enrichment stockwork zones that are the source of the magnetic anomaly. The data were modeled using Geosoft's 2D and 3D inversion programme "Potent". The magnetic models indicated that the magnetic stockwork zone trends north-northeast along an 800 m strike length and is approximately 300 m wide, though inherent ambiguities in the interpretation process may have underestimated the width of the body. Figure 9.3 shows the survey results.

Figure 9.3 Ground Magnetic Survey



Source: Euromax

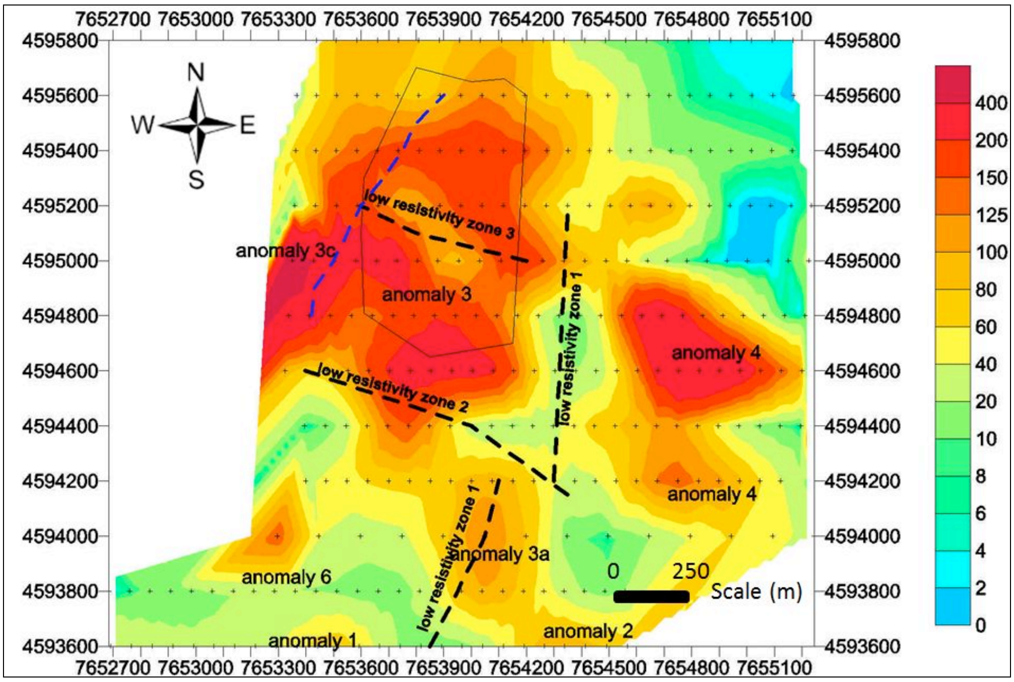
9.3.2 INDUCED POLARISATION / RESISTIVITY SURVEY

A high resolution pole dipole array survey was conducted by Euromax at Ilovitza in July-August 2008 using dipole lengths of 300 m and 150 m and n spacings of 0.5, 1, 1.5, 2, 2.5, 3, 3.5, 4, 4.5, 5 and 5.5 for the array with dipole length of 300 m and n= 1, 2, 3, 4, 5, 6, 7 for the 150 m dipole length. The raw data was processed using the Zonge Datpro software package and interpreted using the GEOTOMO software package. Simultaneous inversion modelling of data measured with different electrode configurations was conducted.

The IP/resistivity survey identified a number of intense IP anomalies, interpreted to be related to sulphide and magnetite mineralisation previously intersected in drill holes, Figure 9.4 and Figure 9.5. The resistivity models revealed the presence of linear, almost vertical low resistivity features, interpreted as fault zones. A specific feature of the resistivity models is the presence of almost horizontal low resistivity layers probably associated with more intensive fracturing of the rocks above silicified granodiorite. The most prominent IP anomaly coincided spatially with the magnetic stockwork zone defined previously by the magnetic survey and tested by several drill holes. Compared to typical low grade porphyry systems, the IP values measured at Ilovitza were significantly elevated. The high IP intervals correlated with high total sulphide values of up to 3 to 5%, though while the copper mineralisation in drillholes coincides with high sulphide concentrations it was not possible to distinguish between anomalies related to a barren pyrite halo and IP anomalies associated with porphyry copper mineralisation.

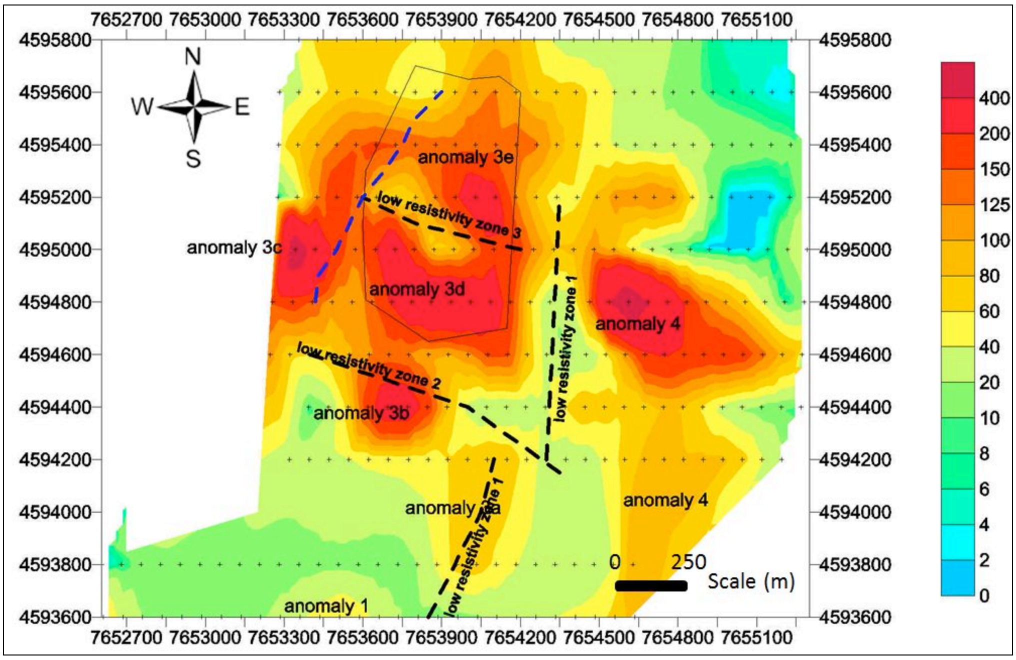
The IP survey indicated that the porphyry system could extend to a depth of at least 550 m from surface. The core of the hydrothermal system as defined by the IP and magnetic surveys trends north-northeast and extends to about 800 to 1200 m along strike. To the east the interpreted body is truncated by a north-south trending, near vertical resistivity low, most probably a large fault zone. Recent drilling suggests that this fault has a downthrow to the east and that further mineralisation occurs within dacitic rocks east of the fault. The western margin of the IP high extends beyond the bounding magnetic lineament and may represent a pyritic halo continuing beyond the area of magnetite mineralisation in the core of the system. The southern margin of the porphyry as interpreted from the IP coincides with a west-northwest-trending, near vertical low resistivity feature, interpreted as a fault. The latter appears to offset to the west-northwest the prominent north-south fault that bounds the eastern part of the main porphyry. A parallel west-northwest-trending vertical fault 600 m to the north is also interpreted to be truncated to the east by the north-trending fault. These interpreted faults are shown on Figure 9.4, Figure 9.5, Figure 9.6 and Figure 9.7.

Figure 9.4 2D IP Inversion Model on Level 350 m from Surface



Source: Euromax

Figure 9.5 **2D IP Inversion Model on Level 350 m from Surface**

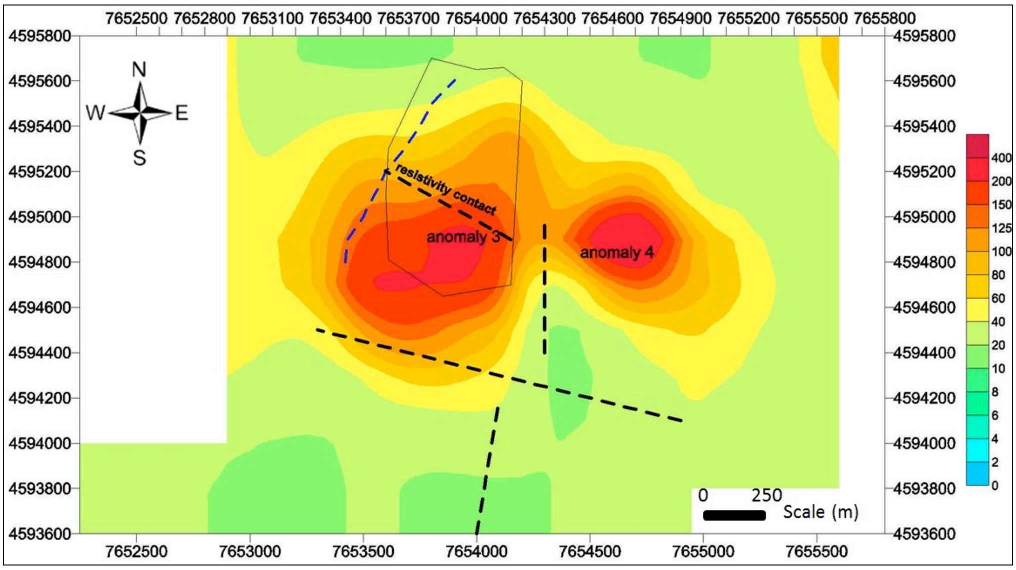


Source: Euromax

Several IP anomalies form a discontinuous annular zone around the interpreted core of the system, probably related to the pyrite halo.

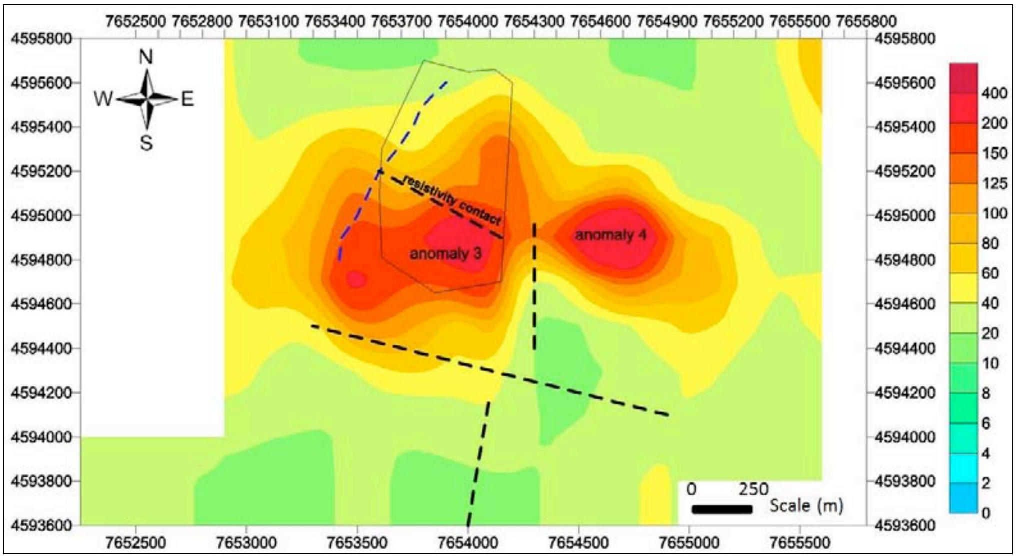
The resistivity models presented in Figure 9.6 and Figure 9.7 indicate the presence of near to horizontal low resistivity layers to the west of the core of the system interpreted to reflect the presence of intensive stockwork zones with copper mineralisation. Initial examination of the more recent drilling conducted on the property has not established conclusive lithological or mineralogical causes of the horizontal IP anomaly. The geological log of drill hole EOIC-07-10 indicates that, as suggested above, silicification increases in the high resistivity zone. However, it seems that this increase in silicification is not a major feature as chlorite-sericite is logged as the dominant alteration in this zone. The beginning of the horizontal low resistivity zone appears to coincide with the intercept of fresh, unweathered dacite, although a lithological change from dacite to granodiorite further down the hole does not seem to affect the resistivity model. In drill hole PDIC-04-02, magnetite is logged in the transition from low to high resistivity. A further observation is that the area of low resistivity correlates with low grade copper and gold in the Ilovitza block model and, in fact, grade appears to increase as the higher resistivity zone is intersected.

Figure 9.6 3D Resistivity Inversion Model on Level 350 m from Surface



Source: Euromax

Figure 9.7 3D Resistivity Inversion Model on Level 260 m from Surface



Source: Euromax

The enriched stockwork appears to be structurally controlled and, as mentioned above, downthrown to the east by a fault originating at surface at approximately 7654200mE on section 4595200mN.

It is concluded that additional drilling should be conducted to delineate the structures interpreted from the IP survey and to test IP anomalies beyond the core of the deposit both laterally and to depth. In addition, existing core should be relogged with specific attention to the contacts between low and high resistivity zones and faults. If possible, attempts should be made to orientate core in future drill programmes.

10.0 DRILLING

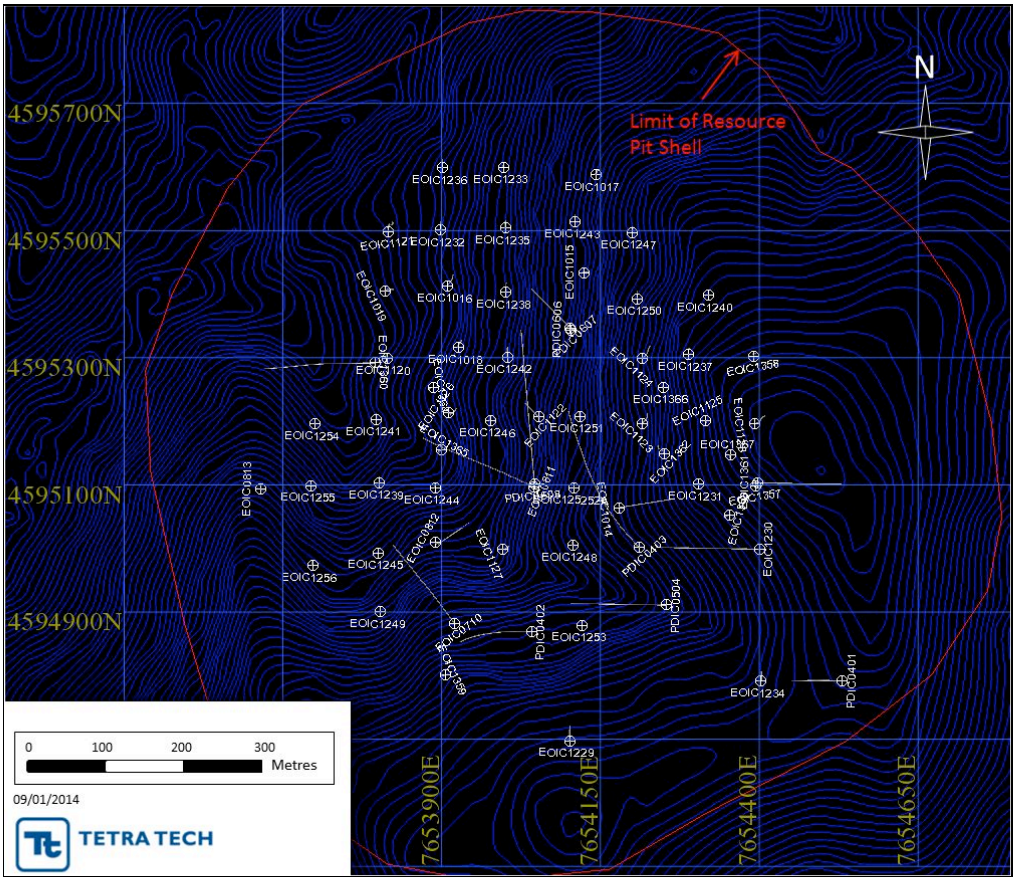
A total of 66 holes were drilled on the property over 9 campaigns between 2004 and 2nd October 2013. Table 10.1 summarises the scope of the drilling campaigns completed on the property.

Table 10.1 Summary of Drilling Campaigns

Year	Drilling Technique	No. of Holes	Total Length Drilled (m)	Company
2004	Diamond Core	3	1,178	PDX
2005	Diamond Core	1	385	PDX
2006	Diamond Core	3	1,238	PDX
2007	Diamond Core	2	999	Euromax
2008	Diamond Core	3	1,600	Euromax
2010	Diamond Core	6	3,016	Euromax
2011	Diamond Core	9	4,387	Euromax
2012	Diamond Core	28	12,081	Euromax
2013	Diamond Core	11	4,148	Euromax
Total	**Diamond**	**66**	**29,032**	**-**

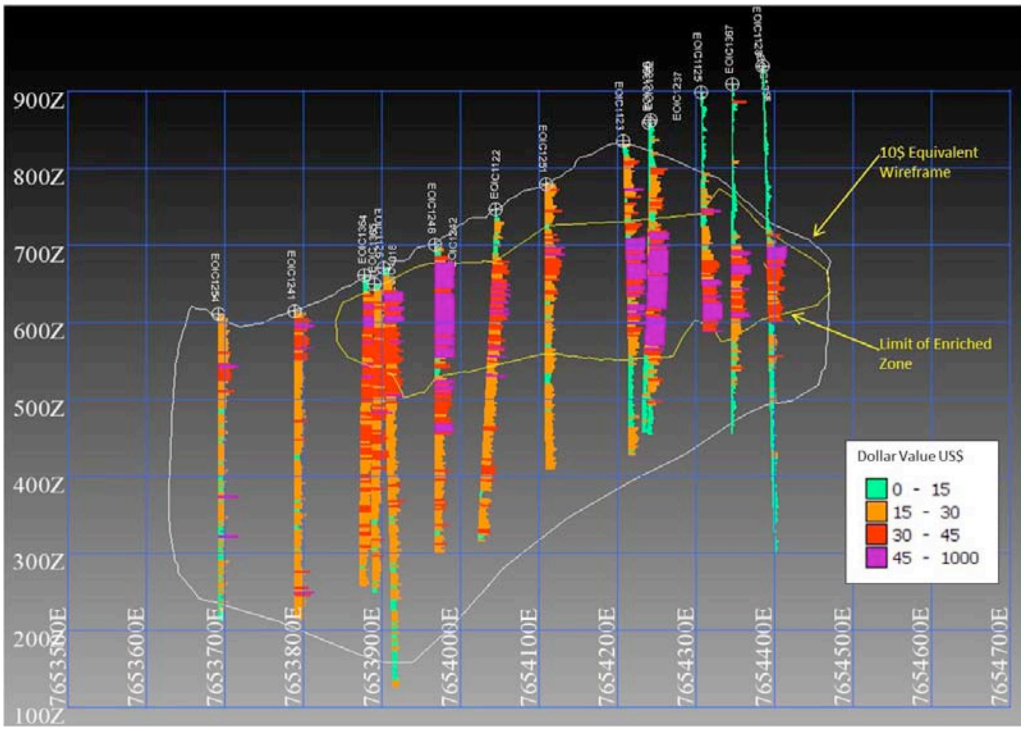
The drillholes are generally vertical or steeply dipping; with 53 of the 66 drillholes being vertical and the remainder being between 60 and 75°; the angled holes are drilled in a range of azimuths, from 110° through to 355°. The drillhole locations are illustrated in Figure 10.1, with a typical west-east section illustrated in Figure 10.2.

Figure 10.1 Drillhole Locations on the Ilovitza Property



Source: Tetra Tech

Figure 10.2 Typical West-East Cross Section



Source: Tetra Tech

Note: West-east section taken at Y = 4595200; dollar value derivation shown in Section 10.3

All of the holes were drilled using rotary diamond coring techniques. Drillholes were collared with PQ diameter (85 mm core) and then advanced with HQ (61.1 mm core) and then occasionally NQ (45.1 mm core) diameters. Occasionally, difficult ground conditions were encountered around the base of oxidation. Where this occurred, these sections were cemented and re-drilled.

A wireline system was used to hoist the core tube to surface to allow the drill core to be extracted. The drill core was placed into 1 m long aluminium core boxes, with the core laid out so that the order of drilling is retained. The core was placed with the top of the hole in the top left hand corner of the core box and the deepest core located in the bottom right hand corner of the core box.

10.1 Core Logging and Sampling

Once the core had been collected and transported to the logging facilities in Strumica, the core boxes were laid out on the logging racks for inspection by the exploration geologist.

Core blocks were checked for consistency and core boxes were marked with the drillhole number and interval depths. High quality digital photographs of the core were taken both wet and dry.

Logging included observations relating to lithology, alteration, mineralisation, structure, recovery and rock quality designation (RQD). Geologists initially logged the holes onto paper logging forms before entering the logs into Microsoft Excel™ (Excel) on a daily basis.

Drill core recovery is very good, generally >95%, throughout the deposit. Within the oxide zone the core is general highly fractured and as such the RQD is low, however the overall core recoveries remain high.

Once logged, the core was marked for cutting. Core was cut with a diamond blade circular table saw. Samples were taken over variable intervals based upon lithology and alteration observations, collected during detailed logging of the core. Where there are no pertinent changes in lithology or alteration, the samples have generally been collected on the basis of 3 m intervals.

Generally half of the core samples were taken and processed for analysis. Where density samples were taken, one quarter of the core was collected for density determination and one quarter was taken for assaying. Half core samples have been retained and are stored in the facilities at Strumica.

The sample preparation process is described in Section 11.3.

10.2 Collar And Downhole Surveys

10.2.1 Collar Surveys

In October 2011 Euromax engaged an independent company (DGU Geo Prem of Strumica) to complete a collar survey for drillholes 1 to 27. The survey was completed using a Topcon GPS (model reference: Hiper Pro RTK Base and Rover device). The measurements were collected using real time kinematics from a known trigonometric point with loaded parameters for the local site. The accuracy of measurements was 2 to 5 cm.

Collar surveys for drillholes 28 to 67 have been completed using a handheld GPS (Garmin GPS map 62s). The manufacturer specification states that the device accuracy is less than (<) 10 m. Holes will be surveyed as part of an overall site survey during the FS.

10.2.2 Downhole Surveys

Downhole surveys were completed by the drilling contractor as the drill string was extracted from the holes. Downhole surveys were completed using a digital survey instrument (JKH-R magnetic single shot inclinometer) with readings taken every 50 m. Generally the drillholes show very low deviation from the planned hole paths, deeper holes show up to 5° variance from design for both dip and azimuth.

10.3 Historic Drilling

Between 2004 and 2006 PDX drilled seven holes on the property totalling 2,801 m. A full QA/QC programme was undertaken, which included: Blanks, duplicates and standards, as discussed in Section 11.6.

The 2004 to 2006 drilling has also been validated by Euromax through the twinning of three of the holes drilled by PDX. The grades and core recoveries observed for the PDX drillholes are generally in line with those subsequently recorded within the Euromax drilling. Table 10.2 presents the significant intercepts (>0.7 g/t Au equivalent) associated with the 2004 to 2006 drilling campaigns. Gold equivalent based on 100% recovery and prices as follows: Au $1,400/oz, Cu $7,500/tonne, with a maximum 9 metres internal waste allowed.

Table 10.2 Significant Intercepts from 2004 to 2006 Campaigns

Drillhole No.	From (m)	To (m)	Length (m)	Au (g/t)	Cu (%)
PDIC 402	240.0	386.0	146.0	0.25	0.20
PDIC 403	417.0	537.0	120.0	0.29	0.26
PDIC 504	210.0	264.0	54.0	0.32	0.14
PDIC 606	54.0	405.0	351.0	0.36	0.25
PDIC 607	132.0	225.5	93.5	0.45	0.28
PDIC 608	57.0	450.0	393.0	0.36	0.25

10.4 Euromax Drilling 2007 to 2013

Between 2007 and 11th of July 2013, Euromax drilled a further 59 holes, with a total length of 25,733 m, Table 10.3 presents the significant intercepts (> 0.7 g/t Au equivalent).

Table 10.3 Notable Intercepts from 2007 to 2013 Campaigns

Drillhole No.	From (m)	To (m)	Length (m)	Au (g/t)	Cu (%)
EOIC 810	140.0	182.0	42.0	0.26	0.36
EOIC 811	265.0	323.0	58.0	0.35	0.25
EOIC 812	49.0	121.0	72.0	0.29	0.36
EOIC 814	105.0	153.0	48.0	0.35	0.49
and	207.0	235.0	28.0	0.50	0.47
EOIC 815	123.0	408.0	285.0	0.44	0.27
EOIC 1016	3.6	30.0	26.40	0.69	0.87
and	278.5	314.0	35.5	0.37	0.27
and	332.0	389.0	57.0	0.32	0.32
EOIC 1018	4.5	18.0	13.5	0.41	0.66
and	33.0	171.0	138.0	0.47	0.36

Drillhole No.	From (m)	To (m)	Length (m)	Au (g/t)	Cu (%)
and	195.0	232.2	37.2	0.35	0.29
and	253.0	277.0	24.0	0.35	0.28
and	308.0	332.0	24.0	0.40	0.25
EOIC 1019	109.0	120.5	11.5	0.45	0.26
and	205.5	227.0	21.5	0.43	0.23
EOIC 1020	13.1	61.0	47.9	0.44	0.45
and	77.8	117.0	39.2	0.34	0.30
EOIC 1021	254.0	269.0	15.0	0.37	0.26
and	472.0	514.1	42.1	0.59	0.56
EOIC 1022	66.5	192.9	126.4	0.47	0.26
and	334.0	355.0	21.0	0.37	0.25
EOIC 1023	126.0	254.0	128.0	0.55	0.34
EOIC 1024	127.0	151.0	24.0	0.32	0.29
EOIC 1025	216.0	312.4	96.4	0.55	0.29
EOIC 1026	29.0	167.0	138.0	0.48	0.30
EOIC 1027	279.0	297.0	18.0	0.26	0.30
EOIC 1028	235.0	334.4	99.4	0.57	0.23
EOIC 1230	117.0	192.1	75.1	0.38	0.30
EOIC 1231	161.0	258.0	97.0	0.33	0.28
EOIC 1233	38.5	50.0	11.5	0.44	0.26
and	253.0	271.0	18.0	0.56	0.16
EOIC 1235	7.0	38.0	31.0	0.26	0.31
and	349.4	398.0	48.6	0.34	0.28
EOIC 1237	270.9	370.0	99.1	0.52	0.25
EOIC 1238	18.0	75.0	57.0	0.46	0.29
and	87.5	120.0	32.5	0.50	0.38
EOIC 1239	10.0	96.0	86.0	0.39	0.28
and	129.0	184.0	55.0	0.45	0.28
EOIC 1241	2.4	64.0	61.6	0.35	0.30
EOIC 1242	40.0	111.0	71.0	0.66	0.38
EOIC 1244	34.0	107.0	73.0	0.32	0.30
and	173.0	268.0	95.0	0.42	0.28
EOIC 1245	47.0	68.0	21.0	0.19	0.54
and	149.0	170.0	21.0	0.46	0.29
EOIC 1246	15.0	248.0	233.0	0.64	0.32

Table continues…

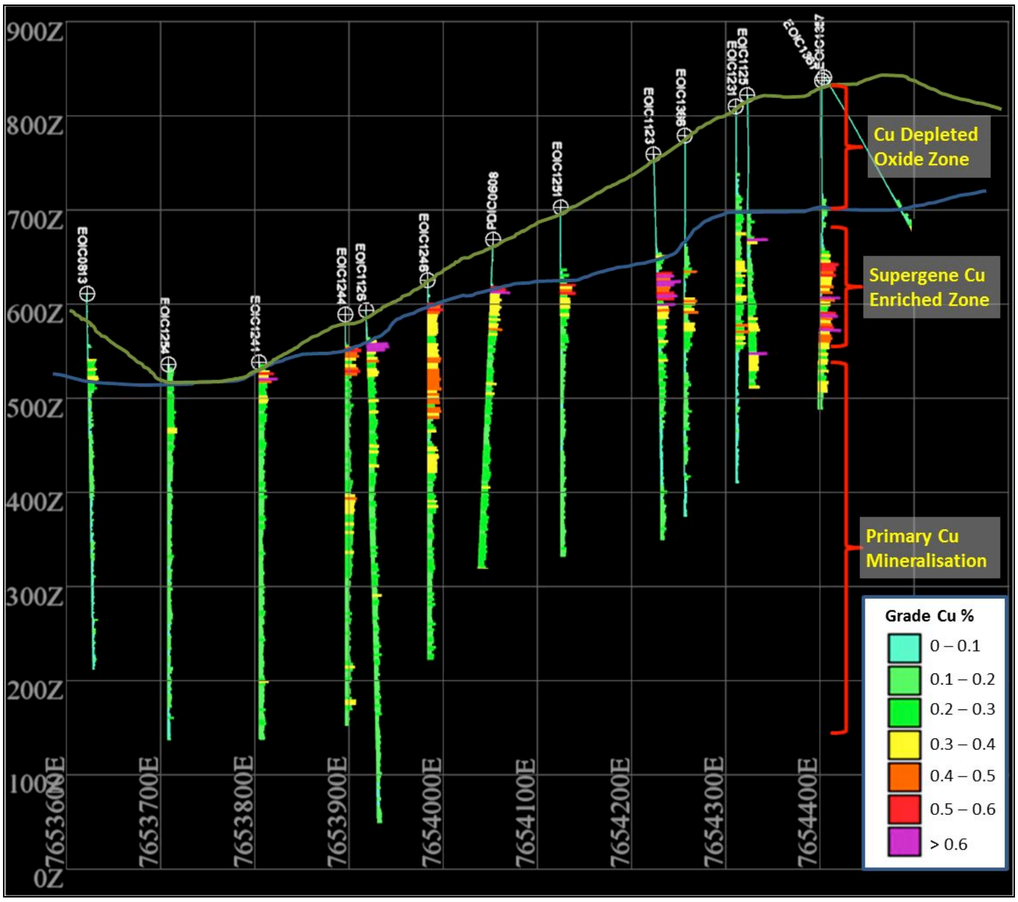
Drillhole No.	From (m)	To (m)	Length (m)	Au (g/t)	Cu (%)
EOIC 1251	81.5	126.5	45.0	0.36	0.33
EOIC 1357	181.5	338.0	156.5	0.99	0.37
EOIC 1360	0.0	62.5	62.5	0.31	0.44
EOIC 1361	180.0	203.0	23.0	0.35	0.34
EOIC 1362	143.0	305.3	162.3	0.90	0.35
EOIC 1364	26.0	240.0	214.0	0.45	0.28
EOIC 1365	57.0	177.0	120.0	0.39	0.25
EOIC 1366	172.5	228.0	55.5	0.45	0.29
EOIC 1367	214.0	305.0	91.0	0.52	0.23
EOIC 1368	157.0	272.0	115.0	0.78	0.28

Table continues…

The mineralisation is generally dispersed throughout the porphyry, but with a broadly horizontal higher grade zone associated with the leaching and supergene enrichment beneath the oxidised material, see Figure 10.3. Sectional interpretations have been developed which eliminate any apparent thickening of the enriched zone associated with the inclined drillholes. Fifty-three of the 66 drillholes are vertical and as such the higher grade mineralised intercepts observed in these holes are representative of the true thickness. Geostatistical analysis does not provide evidence of any gold nuggets within the supergene enriched zone.

The cross section presented in Figure 10.3 illustrates the interpretation of the drilling results in relation to copper depletion in the oxide materials and supergene enrichment beneath. The gold assays show a similar but less pronounced distribution.

Figure 10.3 East West Section – Copper Assays (%)



Source: Tetra Tech

Note: Section through Northing: 4595195

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Summary

Euromax undertakes the bulk of the sample preparation activities in its facilities in Strumica. The activities include: Sample selection, core cutting, sample drying and crushing. Milling of the samples is undertaken by the testing laboratories.

All assaying was completed by Eurotest Control AD (Eurotest), a laboratory with International Organisation for Standardisation (ISO) 9000 accreditation in Sofia, Bulgaria. Eurotest does not have any previous relationship with Euromax or its management.

11.2 General Sampling Methodology

The mineralised zones at Ilovitza have been sampled on the basis of the lithological and alteration observations collected during detailed logging of the core. Where there are no pertinent changes in lithology or alteration, the samples have generally been collected on the basis of 3 m intervals.

As of the 11th July 2013, the drillhole database for the project contained 9,522 samples, 8,917 of which are 3 m in length or less. The remaining 605 samples were between 3 and 5 m in length, with 556 being between 3 and 4 m in length. There is no correlation between grade and sample length.

11.3 Sample Preparation

After being logged, the sample intervals are marked and the core is sawn in half. The core is photographed after splitting. Tetra Tech was provided with the complete photographic database, which contains a high definition photograph of every box of drill core associated the project.

The following sample preparation procedure is followed:

- Half core samples are crushed to < 2 mm grain size.
- Two 200 g representative sub-samples are split from the whole via several stages of quartering.
- One 200 g sample is sent to Eurotest, where it is pulverised and assayed.
- The second 200 g sample is retained and stored at Strumica.
- Pulp rejects (100 to 150 g) are returned to Euromax and stored at Strumica.
- Coarse rejects of (4 to 10 kg bags) are stored at Strumica.

All samples are securely stored in the core storage facility prior to transport by Euromax personnel to the Eurotest Laboratory in Sofia, Bulgaria.

Figure 11.1 Core Cutting Equipment in Euromax Facilities at Strumica

Source: Tetra Tech

Samples of half core are placed into plastic bags and sealed with single use ties under the supervision of Euromax geologists. Bags are labelled with a unique sample number, as well as a sample interval and drillhole reference.

The samples are then dried in the drying units at Strumica, Figure 11.2.

Figure 11.2 Sample Drying Equipment in Euromax Facilities at Strumica

Source: Tetra Tech

Once fully dried, the samples are crushed to 2 mm using a jaw crusher, Figure 11.3.

Figure 11.3 Jaw Crusher Equipment at Euromax Facilities in Strumica

Source: Tetra Tech

Once crushed and split to 200 g, the samples are sent to the laboratory with associated chain of custody documentation. The duplicate sample is retained by Euromax and stored in Strumica, Figure 11.4.

Figure 11.4 Duplicate 200g Samples Stored at Strumica

Source: Tetra Tech

11.4 Confidentiality and Security of Data

The results of the laboratory analysis are strictly confidential and the sole property of Euromax. All information stored on computer systems is accessible via a password protected network drive, accessible only to a limited number of Euromax staff. One member of staff is responsible for the transfer of assay results to the drillhole database.

11.5 Assaying

All assaying was completed by Eurotest. Assaying is undertaken on a pulverised 30 g sub-sample of the 200 g sample sent to the laboratory. Gold is assayed by Fire assay with an Atomic Absorption spectroscopy (AAS) finish. Unusually high values are checked by metallic screen assay.

All other elements including copper, molybdenum and silver are assayed by Inductively Coupled Plasma Atomic Spectroscopy. Any over limit samples (> 10,000 ppm for copper and molybdenum and >10 ppm for silver) are re-assayed by AAS.

11.6 Quality Assurance / Quality Control Programme

QA/QC samples, including crush duplicates, standard reference materials (SRM), and blanks were inserted approximately every 20th sample. This ensured that at least one set of QA/QC samples were included in every batch of samples issued to the testing laboratory.

11.6.1 STANDARDS

The QA/QC programme included inserting SRM into the sample stream. A standard was included approximately every 20 samples. This represents approximately 5% of the samples issued to the laboratory.

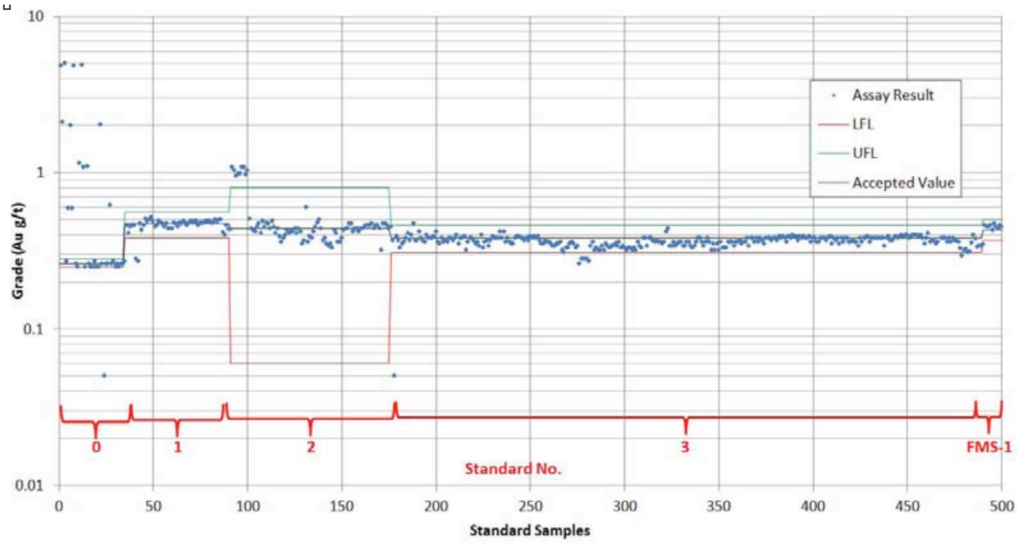
Table 11.1 **Standard Reference Material Conformance**

SRM	Au (ppm)						Cu (ppm)						Overall Failure %
	Accepted Value	UFL	LFL	Mean Result	Count	Failure Count	Accepted Value	UFL	LFL	Mean Result	Count	Failure Count	
0	0.26	0.28	0.25	0.26	21	0	-	-	-	-	-	-	-
1	0.47	0.56	0.38	0.46	63	2	174	2946	0	650	63	5	6.3
2	0.44	0.81	0.06	0.49	87	9	3090	7807	0	3865	87	9	10.3
3	0.38	0.46	0.31	0.36	337	3	2489	2794	2184	2494	337	3	2.2
FMS-1	0.43	0.50	0.37	0.45	34	4	5080	5152	5008	5052	34	1	7.9

Notes: UFL = upper failure limit LFL = lower failure limit

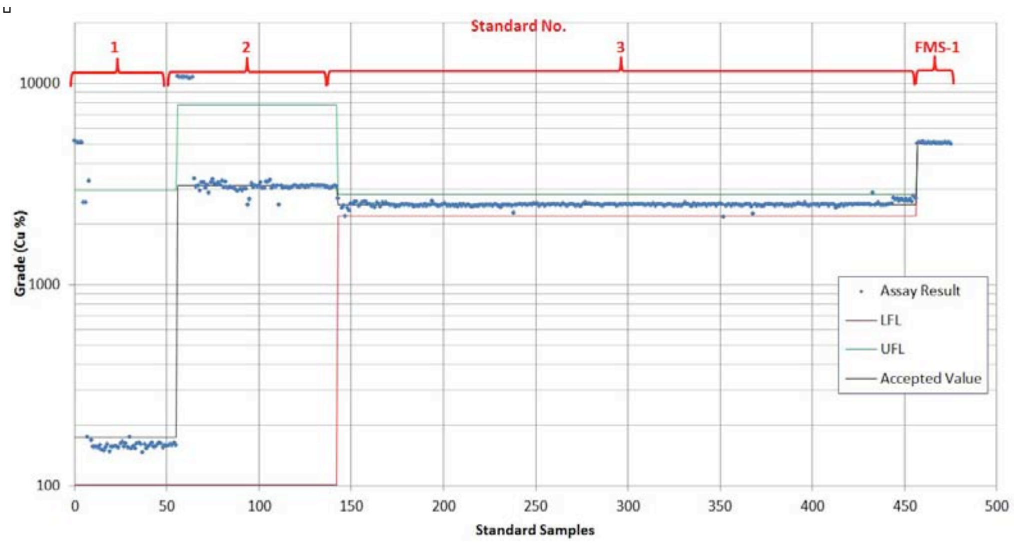
In total 12 different SRMs have been utilised. Seven of the standards were used less than 10 times and therefore statistical inference cannot be derived from this data, however visual inspection shows satisfactory performance. The five standards presented in Table 11.1 are those currently used at the project, with the exception of standard 3, which has been exhausted and was replaced with FMS-1. Figure 11.5 and Figure 11.6 illustrate the performance of the standards in relation to the accepted values.

Figure 11.5 Standard Performance Chart - Gold



Source: Tetra Tech

Figure 11.6 Standard Performance Chart - Copper



Source: Tetra Tech

Standards 0, 1, 2 and 3 were prepared from core from the Ilovitza drilling. The standards were independently prepared for Euromax by Eurotest control laboratory.

The compliance of the results in relation to the accepted values for the standards is generally satisfactory. It was noted that all of the failures associated with standard 2, relate only to drillholes EOIC1014 and EOIC1015. The failures showed very consistent results which were significantly higher than the accepted value for the reference material. The systematic error is not reflected in the assay results for the drillholes. Given the consistency of the results associated with the standards and the fact that the systematic error is not reflected in the main assay results, it has been assumed that the incorrect SRM was submitted with this batch of samples.

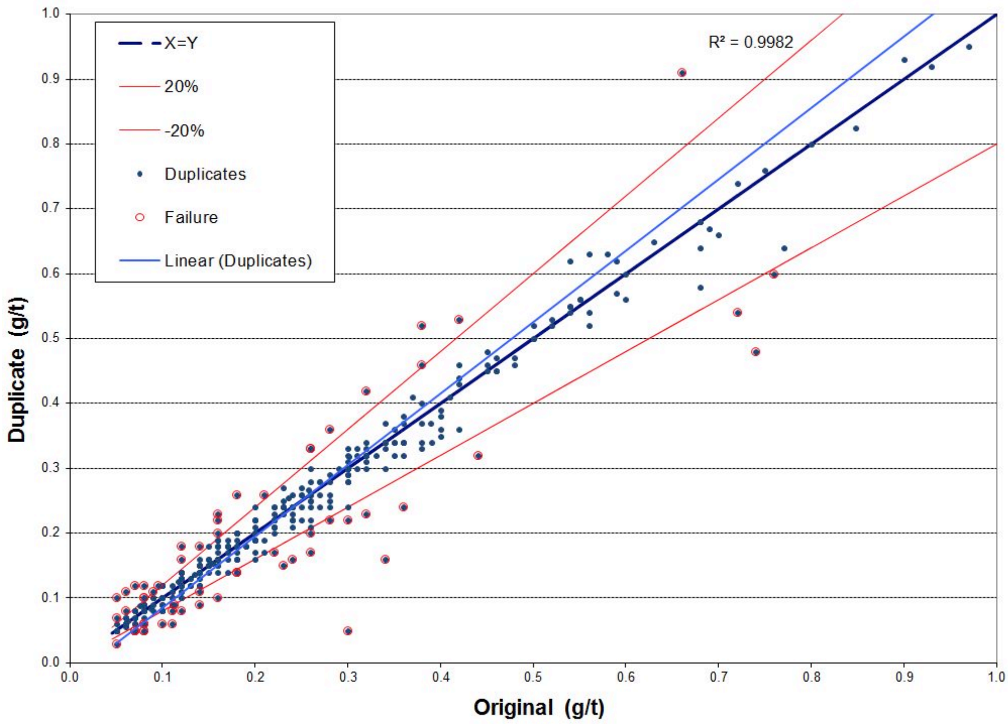
The failure percentage associated with standard 0 is higher than desired, however it is noted that this material has only been submitted 21 times. There are no failures associated with the gold assays and the copper failures are all associated with the results being lower than the accepted value.

11.6.2 Coarse Duplicates

Coarse duplicates were submitted to the laboratory approximately every 20th sample. The duplicates were created when the samples were crushed and split in the facilities in Strumica. To prevent the duplicate from being recognised by the laboratory, these samples were not identified as such when submitted.

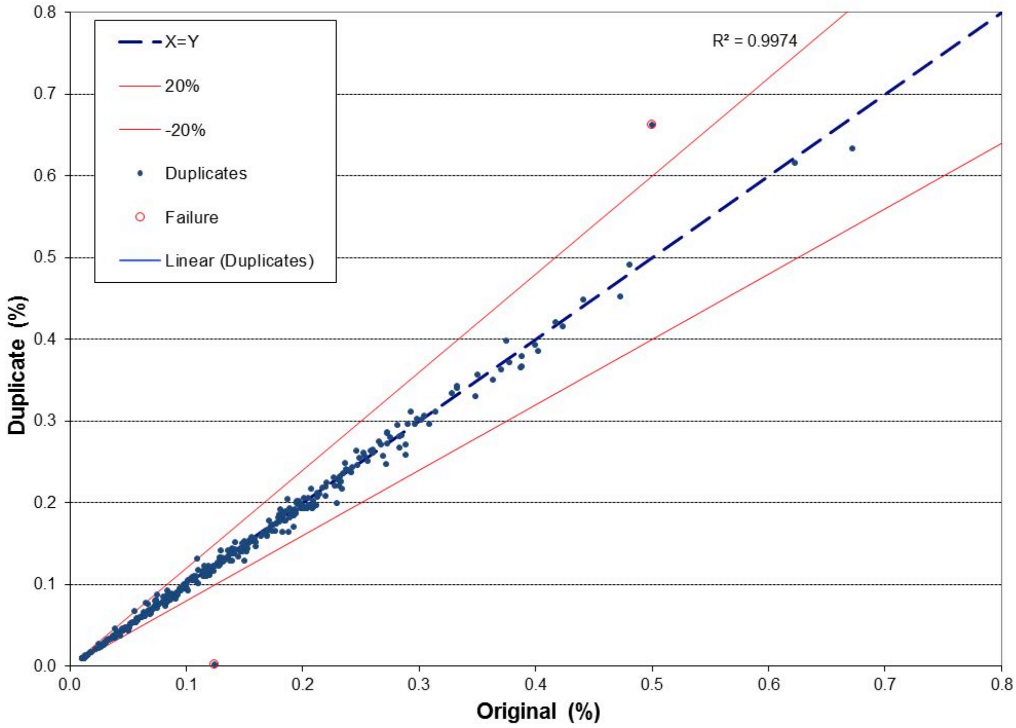
In total 441 duplicate samples were submitted, of which 60 failures occurred for gold and 4 failures occurred for copper. Failures were defined by the difference between the original and the duplicate assay being >20%. Therefore, the total failure rate was 9% for gold and 2% for copper, which is well within acceptable limits for coarse duplicate analysis. Correspondingly the slope of regression value of R_2 = 0.99 for gold and 0.98 for copper indicates a very good correlation between the original and the duplicates, see Figure 11.7 and Figure 11.8.

Figure 11.7 Coarse Reject Duplicates (Au g/t)



Source: Tetra Tech

Figure 11.8 Coarse Reject Duplicates (Cu %)



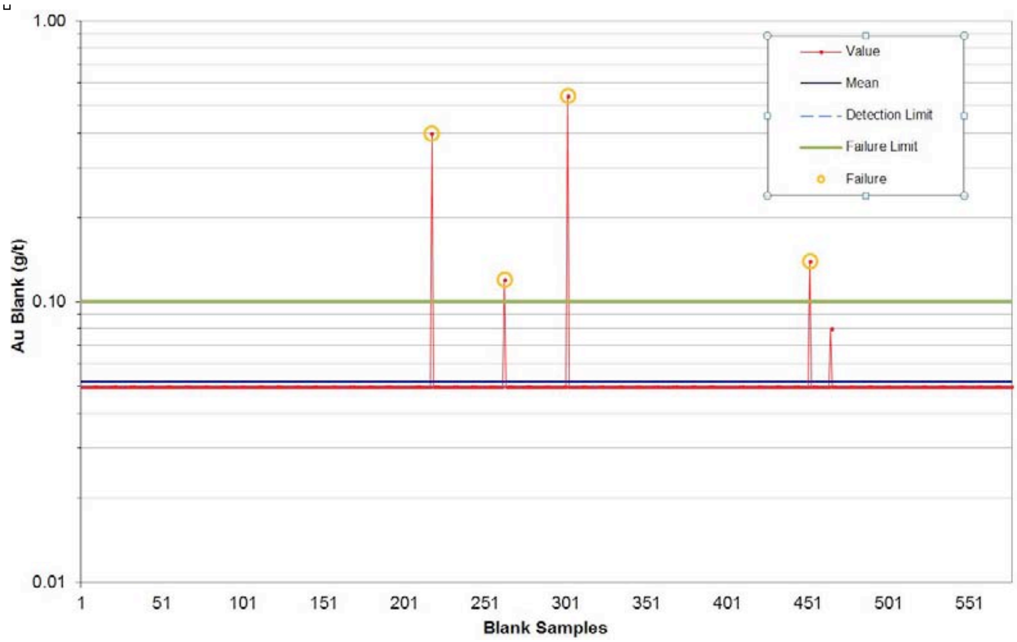
Source: Tetra Tech

11.7 BLANKS

Euromax has created blank materials out of construction limestone. The limestone is processed through the crushing and splitting procedure using the same techniques as those adopted for the genuine samples. By putting the material through the same process as the genuine samples, the blanks act as a verification of the sample preparation process, indicating the presence of any contamination.

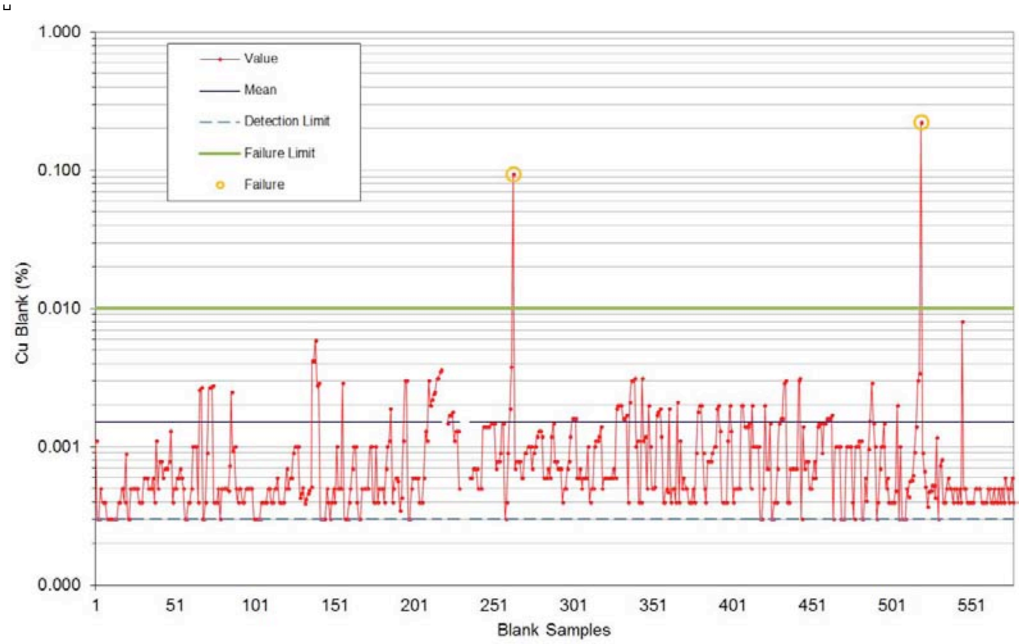
A total of 576 blank samples were inserted. With respect to gold assays, four samples (0.8%) failed, where failure is defined by the values being two times the detection limit, see Figure 11.9. This is well within acceptable limits.

Figure 11.9 Blank Compliance Chart (Au g/t)



With respect to the copper assays, two samples (0.4%) failed. For copper, the failure level was set at 0.01% (100 ppm), see Figure 11.10. This is well within acceptable limits.

Figure 11.10 Blank Compliance Chart (Cu %)



11.8 Tetra Tech Opinion

Whilst there are failures associated with certain aspects of the QA/QC programme, they are well within acceptable limits and do not show any particular trends or patterns which would suggest any systemic or procedural issues. The QA/QC programme conducted by Euromax was appropriate and meets industry standards.

In Tetra Tech's opinion, the sample preparation and security procedures are acceptable and the data can be relied upon for Resource estimation.

12.0 DATA VERIFICATION

12.1 SUMMARY

The following steps have been taken to verify the data used to estimate the Mineral Resource and prepare the technical report:

- A site visit was completed to review collar locations, geological outcrops, suitability and accuracy of the core logging and other relevant activities, as well as the suitability of the facilities and equipment.
- Completed office based verification of assay records against original certificates and validation of drillhole database.

12.2 SITE VISIT

A site visit to the project was carried out by Mr. Robert Davies, B.Sc., CGeol, EurGeol, of Tetra Tech from the 17th to the 20th of June 2013. The property was visited on the 18th June 2013. Mr. Davies was accompanied on the site visit by:

- Patrick Forward, Chief Operating Officer, Euromax
- Dimitar Dimitrov, Senior Vice President, Exploration, Euromax
- Mitko Ligovski, Project Geologist, Euromax
- Dragi Peltechki, Mining Engineer, Euromax
- Steve Hill, Senior Project Manager, Tetra Tech.

The site visit included an inspection of the following aspects of the project:

- Sampled drill core from a selection of holes associated with recent and historic campaigns.
- Sample cutting, preparation and storage facilities.
- Coarse rejects and sample pulps.

12.2.1 PROJECT SITE AND DRILLHOLE LOCATIONS

Geological outcrops (Figure 12.1) and drilling locations were inspected during the site visit. Drillhole collars are not formally marked at the project, however access roads and drilling pads are easily identifiable, allowing drill locations to be verified within acceptable limits.

Figure 12.1 Outcrop of Weathered Granite along Access Track

Source: Tetra Tech

Notes: Outcrop of weathered granite, showing phyllic alteration and a steeply dipping fault that includes quartz-sericite-clay alteration. Photo taken at grid ref: (UTM zone 34) E: 0653122 N: 4594224.

The lithologies and alteration observed in outcrop correspond to those recorded on Euromax's geological maps and sections.

Where collars were not visible, earthworks associated with the drilling platforms were recognisable. Several drilling locations were checked in the field and compared to the database records. Drilling platform locations and collars (where visible) were verified using a hand-held Garmin Etrex™ GPS. The locations were found to be within an acceptable tolerance of GPS unit accuracy (average of ±10 m) and consistent with the database. The location of the drillholes listed in Table 12.1 were verified.

Table 12.1 Verified Drillhole Locations

Drillhole No.
EOIC0813
EOIC1127
EOIC0811
EOIC1357
EOIC1361
EOIC1367

Whilst undertaking the site visit, EOIC1367 was being drilled. Tetra Tech inspected the drill rig setup and observed the drilling crew retrieving, handling and recording the drill core samples. Tetra Tech is satisfied that the drilling and core handling is appropriate for the project.

12.2.2 Core Logging and Sampling Facilities

The core logging and sampling facility is located at Strumica, 20 km to the west of the property. The facilities are located in a secure, clean, dry and well equipped permanent building, Figure 12.2. Logging is completed on well-lit sturdy benches.

Figure 12.2 Core Logging and Storage Facilities in Strumica

Source: Tetra Tech

12.2.3 CORE STORAGE

The core is permanently stored in stacked aluminium trays within Euromax's facility at Strumica. The core storage facilities were inspected and found to be clean, dry and secure (Figure 12.3).

Figure 12.3 Core Storage Facilities in Strumica

Source: Tetra Tech

The aluminium core trays are clearly labelled with indelible pen. The labels defined the drillhole number, the core box number and the 'from' and 'to' depth intervals (Figure 12.4).

Figure 12.4 Labelled Aluminium Core Boxes

Source: Tetra Tech

Wooden markers, labelled with depth intervals are inserted within the core boxes to identify the end of individual core runs. Core boxes are sealed on the bottom, but do not have lids and are, therefore, exposed to potential contamination from above.

Sample preparation is carried out by Euromax onsite before the samples are sent to the assay laboratories. Both coarse rejects and sample duplicates are stored in the facilities at Strumica, see Figure 12.5 and Figure 12.6.

Figure 12.5 Duplicate Samples Stored at Strumica

Source: Tetra Tech

Figure 12.6 Coarse Reject Material Stored in Strumica

Source: Tetra Tech

12.3 DRILL CORE CHECK

The six drillholes detailed in Table 12.2 were selected by Tetra Tech to be brought out of storage for inspection.

Table 12.2 Drillhole Cores Inspected

Drillhole No.
EOIC1357
EOIC1255
EOIC1246
EOIC1123
EOIC1016
EOIC0812

The geological interpretation and logging of lithology and alteration has been reviewed against the core samples and discussed with Euromax's senior project geologist. In addition, several core runs were reviewed to ascertain the accuracy of the recovery records.

Whilst onsite, Tetra Tech also reviewed the Euromax procedures in relation to logging, data transcription, database input and data security.

12.4 OFFICE-BASED DATA VERIFICATION

12.4.1 Assay Certificates

Euromax provided Tetra Tech with signed Portable Document Format (PDF) assay certificates for all of the assays contained within the database.

Tetra Tech checked 10% of the assay certificates against the values contained within the database. The certificates checked were chosen to provide examples from each drilling campaign. In total, 987 assay results were checked against the certificates.

Database Validation

The drillhole database was provided to Tetra Tech in Excel and Microsoft Access™ (Access) database format. On import of the records into Surpac, Tetra Tech validated the database to identify incorrectly entered or inconsistent data. Any errors were communicated to Euromax for review and correction.

12.5 LIMITATIONS OF DATA VERIFICATION

Check samples were not taken during the site visit because of the difficulty of exporting representative samples.

12.6 TETRA TECH OPINION

In the opinion of Tetra Tech's QP, the exploration, drilling and sampling activities completed by Euromax on the property meet or exceed industry norms and form an appropriate basis for Mineral Resource estimation.

It is recommended that permanent borehole collar markers are installed. This should be done at the earliest opportunity whilst the temporary location markers / earthworks are still discernible. Permanent markers will be necessary for drillhole location verification exercises to be undertaken for future Mineral Resource updates and studies. These markers should be made from resilient material and display the drillhole number.

13.0 METALLURGICAL TESTWORK REVIEW

This report details the mineralogical and metallurgical test work completed to date on the Ilovitza Copper Gold project ore.

The historical investigations were conducted by two organisations: ITMNS in Belgrade, Serbia and SGS, UK (SGS). A high level summary of these investigations is presented in this report.

As part of the prefeasibility study, additional mineralogical and metallurgical testwork was carried out by SGS. They have tested a composite sample that was composed from samples collected from dedicated metallurgical drill holes within the mineralised zones.

The mineralogical investigations completed at SGS indicate that significant pyrite liberation occurs at a grind size of approximately P80 = 150 microns (µm). Further it was suggested that a significant proportion of the gold is locked in pyrite, and a pyrite concentration step would be beneficial to increase the overall gold recovery.

Tetra Tech analysed the metallurgical testwork results with the objective of identifying the optimal process design flowsheet. The metallurgical studies indicated that the ore is amenable to flotation and cyanidation and is efficiently processed with a flotation and Carbon in Leach flowsheet.

Based on the metallurgical testwork results, the life of mine recovery for Copper (Cu) and Gold (Au) were estimated at 84 percent (%) and 88% respectively.

13.1 HISTORICAL TESTWORK REVIEW

Tetra Tech has reviewed the available information pertaining to the historical metallurgical testwork.

The historical metallurgical testwork includes:

- Flotation testwork completed by ITMNS in Belgrade, Serbia as cited in the MMTS NI 43-101 resource estimate (MTS 2012 has not been presented to Tetra Tech for review).
- Scoping level flotation testwork work SGS.

A short summary of each testwork programme is presented below.

13.1.1 FLOTATION TESTWORK BY ITMNS

The testwork described in Section 13.0 of MMTS (2012) provides the following details:

- Flotation testwork was conducted on a single composite sample representing stockwork mineralisation.

- The sample was described as being diamond drill core, from the central core of the Ilovitza copper-gold-porphyry system.
- Composite sample head grades were 0.24% copper, 0.25 grams per tonne (g/t) gold and 1.6 g/t silver (Ag).
- Simple flotation tests were conducted on the composite sample at a grind size of 80% passing 75 µm (P80 -75 µm).
- Flotation recoveries were indicated as 84% copper, 58% gold, and 68% silver.
- Molybdenum (Mo) recovery was not indicated.
- A flotation concentrate grading 22% copper, 16.6 g/t gold, and 145.7 g/t silver was produced.

It was stated that the Bond work index (Bwi) for the same composite sample at a grind size of 80% passing -75 µm was 11.6 kilowatt hours per tonne (kWh/t).

The composite sample identification, depth, and the material type were not described in the MMTS (2012) technical report. In addition to this, the testwork parameters have not been provided and therefore no further comment can be provided as to the reliability of the testwork data.

13.1.2 FLOTATION TESTING BY SGS

Euromax has employed the services of SGS to complete initial scoping test work on a single composite sample from the sulphide mineralised zones.

The sulphide composite sample head grade analysis, as recorded by SGS, is shown in Table 13.1.

Table 13.1 Sulphide Composite Head Grade Analysis

Material	Unit	Concentration
Copper	%	0.220
Iron	%	3.220
Molybdenum	%	0.008
Sulphur	%	1.100
Gold	g/t	0.350

SGS has completed initial flotation scoping testwork on the sulphide composite sample.

Reagent sighting testwork was performed on approximately 1 kilogram (kg) samples ground to 80% passing 45 µm. The aim of these tests was to establish maximum copper and gold recovery while suppressing the iron and sulphur (i.e. pyrite). The sighting tests were conducted at a pH of 10.5 which suppresses pyrite flotation.

The sighting reagents tested were typical for copper gold flotation (Table 13.2).

Table 13.2 Copper Flotation Reagents Tested

Frother	Gold Promoter	Collectors
MIBC*	Copper Sulphate	Cytec MX3601 Cytec 3418A SEX* 5100

Note: *MIBC – methyl isobutyl carbinol, SEX – sodium ethyl xanthate

It was reported by SGS that, the Cytec MX3601 and 3418A collectors performed consistently well for copper and gold recovery while limiting iron and sulphur concentration to flotation concentrates.

MX3601 was selected for further rougher flotation trials to evaluate grind size versus recovery of copper, gold, iron, and sulphur. From the quantitative evaluation of minerals by SGS's scanning electron microscopy (QEMSCAN) work and gold grain size analysis, it has been established that copper recovery is likely to be less sensitive to grind size than gold. The grind sizes selected were based on the QEMSCAN work, with 75, 63, 53, 50 and 45 µm being chosen. Rougher flotation was performed over a period of 10 to 15 minutes.

It was reported that the grind size made little difference to the iron, sulphur, and copper rougher float recoveries, whereas, a finer grind benefited gold recovery.

Copper was recovered well below a 75 µm grind. It was found that grinding finer had slowed the initial copper recovery rate but not changed the overall recovery.

Copper recoveries after 15 minutes of flotation time were recorded as 90% with a mass pull of 8%. The low concentration of iron in the final flotation concentrate indicated that the rougher stage is far more selective for copper than pyrite.

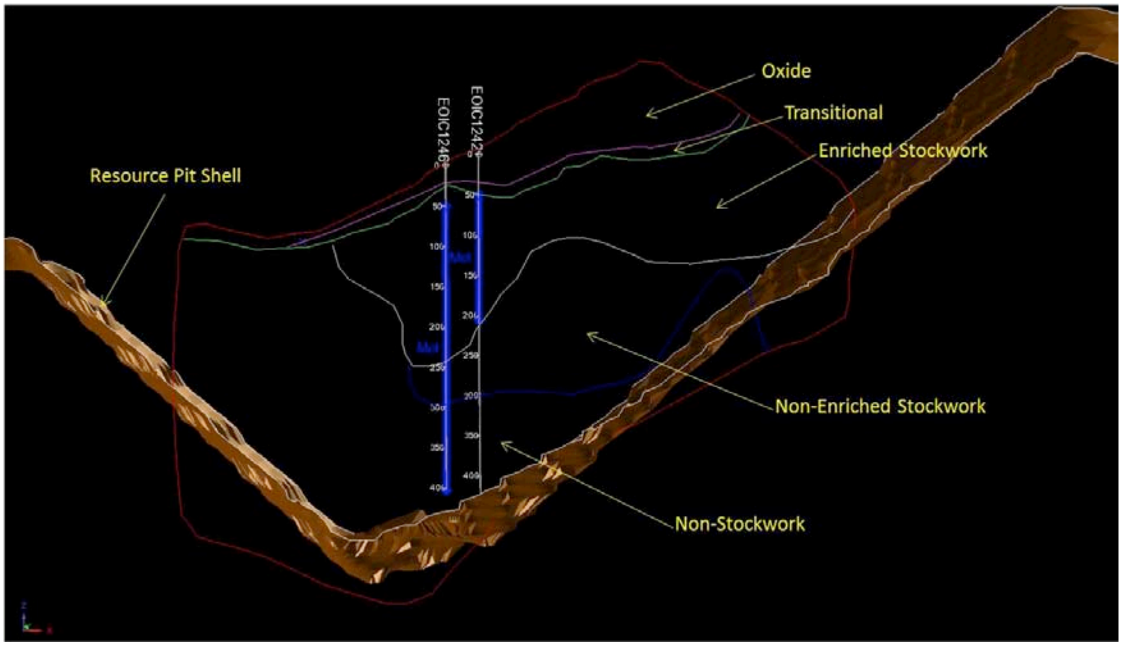
It was reported that the gold recovery was improved with a grind finer than 75 µm. The recovery at this grind was 63% gold with 83% gold being recovered at 80% passing 50 µm. It was concluded that the primary driver for grind size is gold recovery rather than copper recovery, since copper is less sensitive.

13.2 METALLURGICAL SAMPLING

13.2.1 SAMPLE BACKGROUND

Tetra Tech was not involved with the sample selection for metallurgical testwork but reviewed the sample lists provided by Euromax and SGS. It is understood that the testwork samples were collected from two dedicated metallurgical drill holes (Figure 13.1) within the mineralised zones. It appears that the drill holes covered the central part of the proposed pit shell model and include the non Stockwork, Stockwork and the enriched Stockwork mineralisation.

Figure 13.1 Location of Metallurgical drill holes



Metallurgical samples were shipped to SGS (UK) in 10 boxes as shown in Table 13.3.

Table 13.3 Metallurgical Samples sent to SGS (UK)

Hole ID	Box ID	Depth (m)		Head Grade		Weight (kg)
		From	To	Copper (%)	Gold (g/t)	
IC-1242	Box 1	52.0	104.0	0.37	0.68	41.1
	Box 2	104.0	156.0	0.17	0.31	41.3
	Box 3	156.0	205.5	0.17	0.35	39.3
IC-1246	Box 4	51.0	101.0	0.37	0.77	39.1
	Box 5	101.0	150.0	0.40	0.86	39.3
	Box 6	150.0	199.0	0.25	0.39	38.8
	Box 7	199.0	248.0	0.27	0.53	39.5
	Box 8	248.0	299.0	0.19	0.25	40.6
	Box 9	299.0	352.0	0.19	0.26	42.2
	Box 10	352.0	403.3	0.19	0.19	41.0

Note: ID = Identification, m = metres

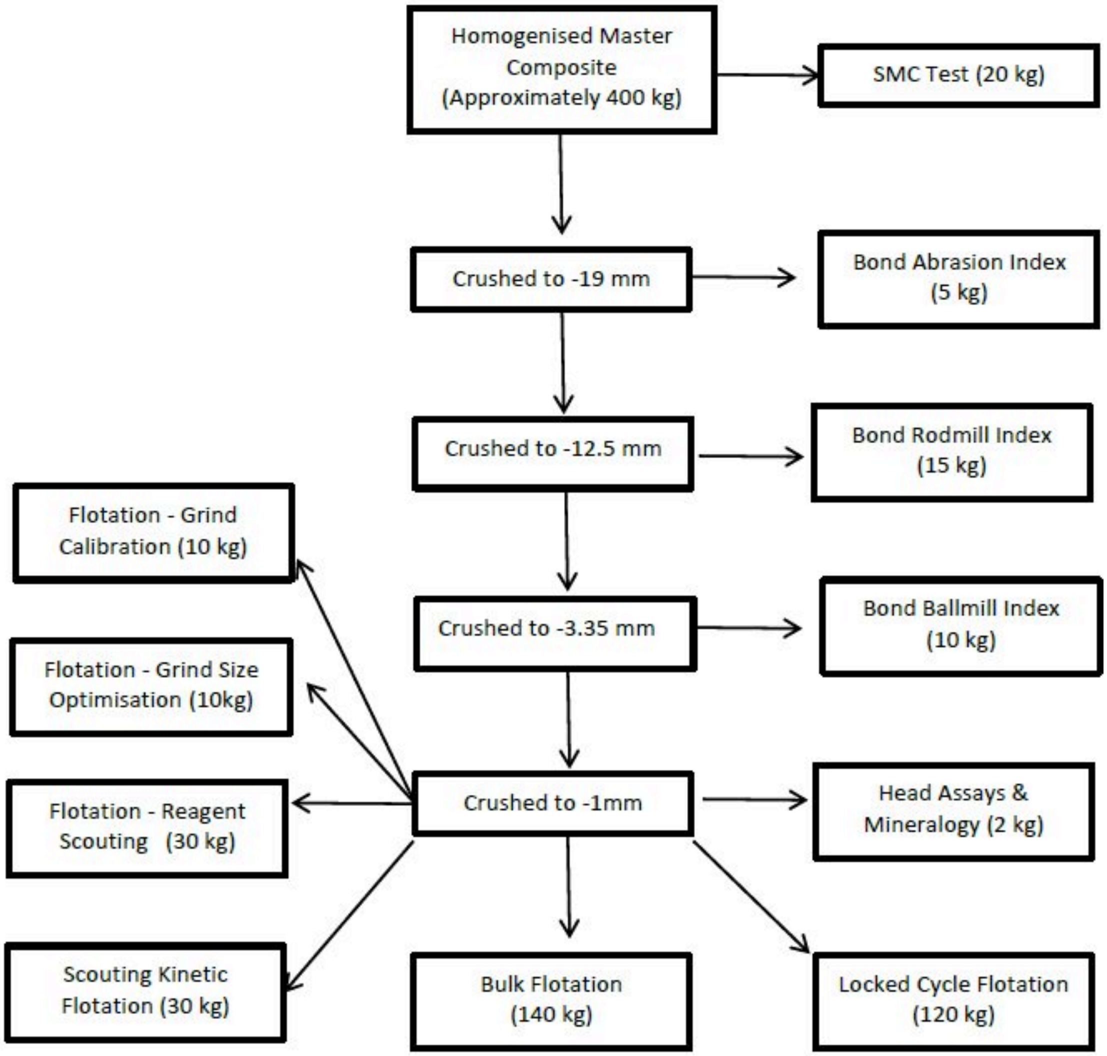
13.2.2 SAMPLING PROGRAMME

Tetra Tech was not directly involved with the selection or quality control / quality assurance aspects of the sampling programme. However, it is understood that trained personnel were involved in all stages of the sampling and industry best practices in sampling were adhered to. A list of the samples used in the metallurgical testwork programmes are shown in Table 13.3.

13.2.3 Sample Preparation

The samples were delivered to SGS in two boxes as quarter HQ (63.5mm diameter) core. SGS blended all the quarter core samples together in order to make a homogenised composite and then took sub-samples for various tests as shown in Figure 13.2.

Figure 13.2 Sample Preparation Flowsheet



13.3 Ore Characteristics

13.3.1 Characterisation

The Ilovitza deposit is characterised as a porphyry-copper-gold deposit, consisting of a supergene oxide material at the surface and hypogene sulphide material at depth.

The supergene oxide materialisation forms a near surface layer, but has been weathered and leached to the extent that nearly all evidence of copper mineralisation has been removed.

The majority of the Ilovitza deposit consists of hypogene sulphide mineralisation lying below the oxide layer. The most abundant copper mineral in the Ilovitza sulphide deposit is chalcopyrite, which is typically associated with complex particles and pyrite. Secondary copper sulphides minerals consist of chalcocite and bornite, which exist in minor quantities.

Gold and silver are present in small amounts with minor amounts of molybdenite, galena, and sphalerite.

13.3.2 PHYSICAL PROPERTIES

The physical properties of the ore are shown in Table 13.4.

Table 13.4 Physical Properties of the Ore

Ore Characteristics	Unit	Values
Specific Gravity	g/cm^3	2.5
Bulk Density	t/m^3	1.6
Angle of Repose	degrees	35
Moisture Content	%	5

Note: g/cm^3 = grams per cubic centimetres, t/m^3 = tonnes per cubic metre

13.3.3 CHEMICAL ANALYSIS

SGS analysed the head sample for gold, silver, and copper assays. SGS also conducted a whole rock analysis for all other major elements. A summary of the assay and whole rock analysis are shown in Table 13.5 and Table 13.6.

Table 13.5 Summary of Head Assays

Au (g/t)	Ag (g/t)	Cu (%)	Fe (%)	Mo (%)	S (%)
0.34	1.76	0.24	2.62	0.003	0.95

Note: Fe = Iron, S = Sulphur

Table 13.6 Summary of Whole Rock Analysis

Element	Assay (ppm)	Element	Assay (ppm)	Element	Assay (ppm)
Aluminium	22,940	Mercury	<1	Antimony	<1
Arsenic	9	Holmium	<1	Scandium	3
Boron	<1	Indium	<1	Selenium	<1
Barium	98	Iridium	<1	Samarium	<1
Beryllium	<1	Potassium	5,512	Tin	3
Bismuth	<1	Lanthanum	16	Strontium	10
Calcium	3,570	Lithium	5	Talc	<1
Cadmium	1	Lutetium	<1	Terbium	<1
Cerium	24	Magnesium	11,050	Thorium	<1
Cobalt	9	Manganese	717	Titanium	260
Chromium	48	Sodium	195	Thallium	<1
Dysprosium	<1	Niobium	3	Thulium	<1
Erbium	1	Neodymium	14	Vanadium	44
Europium	<1	Nickel	13	Tungsten	3
Gallium	7	Phosphorus	248	Yttrium	12
Gadolinium	2	Lead	84	Ytterbium	1
Hafnium	<1	Rhenium	<1	Zinc	170
				Zirconium	15

Note: ppm = parts per million, < = less than

13.4 MINERALOGICAL TESTWORK

13.4.1 SAMPLE SELECTION

SGS Lakefield, Canada (SGS Lakefield) completed bulk mineralogy and gold deportment studies on the Ilovitza metallurgical sample which was prepared and sent by SGS UK.

The sample was stage-crushed by SGS Lakefield to a P_{80} of 150 µm size, and then screened into four fractions (+150 µm, -150 to +106 µm, -106 to +38 µm, and -38 µm). These fractions were submitted to heavy liquid separation (HLS) at a specific gravity of 2.8 g/cm^3, and subjected to pre-concentration using a super-panner (SP) separately. Representative HLS and SP sub-samples were collected for assays and polished section preparation for the mineralogy study.

13.4.2 MINERALOGICAL RESULTS

13.4.2.1. BULK MINERALOGY

Bulk model mineralogy was determined by QEMSCAN using the particle mineral analysis (PMA) mode of operation, and was also supported by X-ray Diffraction (XRD) analysis.

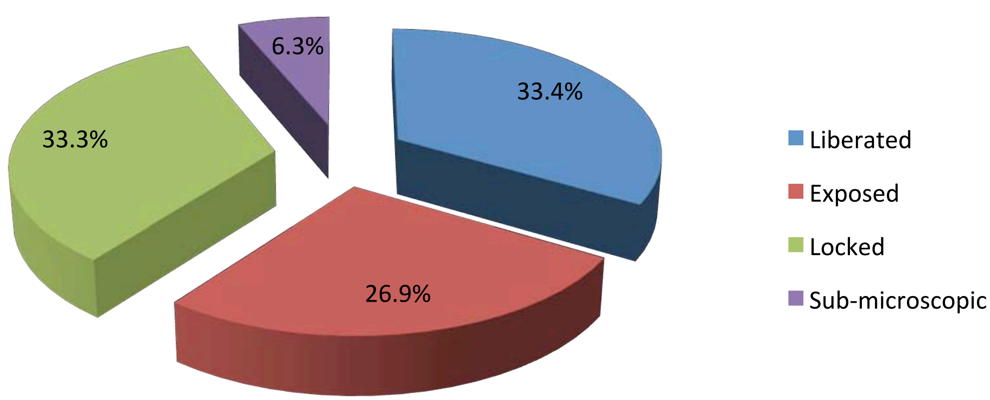
It was reported by SGS that the sample is comprised mainly of silicate minerals, and small amounts of iron-oxides (2.9%), pyrite (0.7%), chalcopyrite and other sulphides. It was also reported that pyrite is well liberated (83%) and chalcopyrite is moderately liberated (66%) at a particle size of P_{80} of 150 µm.

13.4.2.2. GOLD DEPORTMENT STUDY

In summary, the following observations were made by SGS:

- The observed gold minerals include mainly native gold, minor calaverite ($AuTe_2$), and trace amounts of petzite (Ag_3AuTe_2) and other Au-Ag-Te minerals (**Note**: Te = Tellurium).
- The sub-microscopic gold accounts for 6.3% and the microscopic gold accounts for 93.7% of the total gold grade in the sample.
- The major sub-microscopic gold carriers are iron oxides and pyrite.
- The liberated, exposed and locked gold minerals account for 33.4%, 26.9% and 33.3% of the total gold grade, respectively (Figure 13.3).
- Most of the exposed and locked gold is associated with chalcopyrite (~48%) and pyrite (~37%).

Figure 13.3 Overall Gold Distribution



The following conclusions can be drawn from the mineralogical investigations:

- The liberated and exposed gold accounts for approximately 60% of Au; this gold is amenable to Cyanidation. Run of mine (ROM) leach dissolutions of approximately 60% can be expected from Ilovitza ore.
- Locked gold (33.3% of the total Au) may not be amenable to Cyanide leaching at the specified grind. However the leach dissolutions can be potentially increased by exposing the locked gold particles by finer grinding.

- Sub-microscopic gold, carried by sulphides and iron oxides (6.3% of the Au) is considered refractory and is not amenable to direct cyanide leaching.
- Liberation of pyrite (83%), chalcopyrite (66%) and iron-oxides (70%) (calculated from the QEMSCAN analysis) indicate the potential to concentrate these minerals in order to recover a large proportion of the sub-microscopic gold. This, coupled with the fact that 88% of the locked and exposed gold is associated with chalcopyrite and pyrite, tentatively suggests that concentration of the sulphides is critical for the gold recovery.

13.5 METALLURGICAL TESTWORK

13.5.1 COMMINUTION TESTWORK

SGS completed comminution testwork on a composite sample in order to determine the parameters required for the design of the comminution circuit.

The following tests were conducted:

- Bond Rod Mill Work Index.
- Bond Ball Mill work index.
- Bond Abrasion index.
- SAG Mill Comminution test (SMC).

A summary of the comminution test results are shown in Table 13.7.

Table 13.7 Summary of Comminution Results

Bond Rod Mill Index		
Limiting Screen Size	µm	1,180
F_{80}	µm	7,969
P_{80}	µm	897
Work Index	kWh/t	15.1
Bond Ball Mill Index		
Limiting Screen Size	µm	75
F_{80}	µm	2,149
P_{80}	µm	61
Work Index	kWh/t	15.9
Bond Abrasion Index		
Abrasion Index	-	0.1150
SMC Test Results		
DW_i	kWh/m³	3.64
Mi_a	kWh/t	12.70
Mi_h	kWh/t	8.30
Mi_c	kWh/t	4.30
A	-	58.50
b	-	1.19
Axb	-	69.62
SG	g/cm³	2.53
ta	-	0.71

Note:

A = Maximum breakage
AG = Autogenous
Axb = Overall AG-SAG Hardness
b = relationship between energy and impact breakage
HPGR = High Pressure Grinding Roll
kWh/m³ = kilowatt hours per cubic metre
Mi_a = Coarse particle component
Mi_c = Crusher component
Mi_h = HPGR Component
SAG = Semi Autogenous
SG = Specific Gravity
Ta = Low energy abrasion component of breakage

The Bond Ball Mill index (BWi = 15.9 kWh/t) indicates that ore is categorised among the moderately hard range of the BW_i scale. This means the fine grinding using a ball mill would be an energy intensive operation.

However, the SMC test results indicated that the ore is soft. It was reported that approximately 75% of the ores represented in the JKTech database, an industry standard database of ore hardness, are harder than the Ilovitza ore. This indication is in line with the general expectation that it is harder to grind fine because this may involve breaking the grain boundaries of the minerals. In this context, a moderately hard ball mill index and a soft SAG mill index are supported.

The abrasion index of the Ilovitza ore is moderately high and indicates a high wear rate of mill liners and media compared to similar copper projects.

13.5.2 FLOTATION TESTWORK

Two different flowsheet options (bulk sulphide flotation and selective copper flotation) were evaluated by SGS in order to identify the optimal flotation flowsheet design. Effect of grind and reagent scouting tests were conducted for both options in order to establish the optimal flotation conditions. This was followed by locked cycle flotation testwork. A summary of the effect of grind tests is shown in Figure 13.4 and Figure 13.5.

Figure 13.4 Copper Recovery versus Effect of Grind

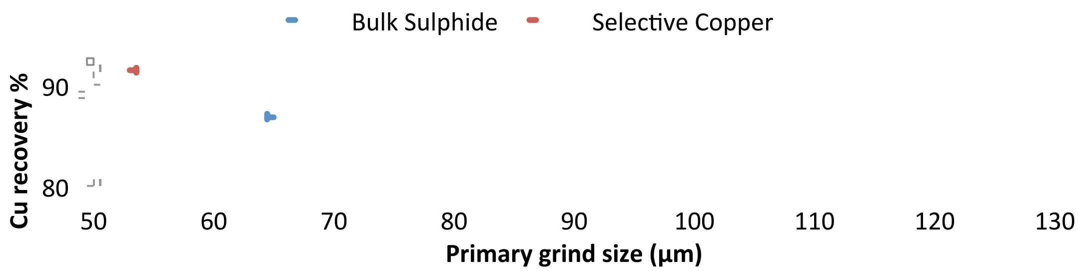
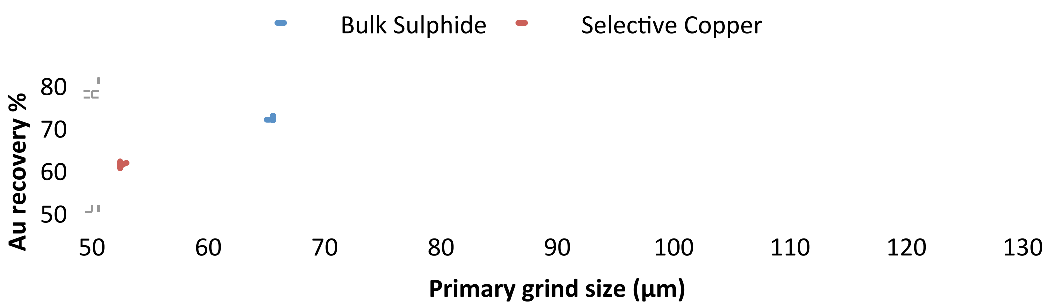


Figure 13.5 Gold Recovery versus Effect of Grind



The results shown in Figure 13.4 indicate that in general, the bulk flotation copper recoveries are lower than the selective flotation copper recoveries. The optimal copper recoveries are reported at approximate 85 µm grind size for the bulk flotation and 75 µm for the selective flotation.

Figure 13.5 suggests that the gold recoveries of the bulk flotation tests are comparatively higher compared with the selective flotation. However, no clear trends were observed between the gold recovery and the feed particle size distribution.

The cleaner tests (on the concentrates collected from the effect of grind tests) indicated that an acceptable concentrate grade was not achievable via the bulk flotation route. As a result, the bulk flotation was not taken in to locked cycle tests.

As shown in Figure 13.4, optimal copper recovery of selective flotation has been obtained at approximately 75 µm. Figure 13.5 suggests that the selective flotation gold recoveries are not influenced by the particle size distributions within the tested range (P_{80}=53 µm to 96 µm). Based on these results, a primary grind size of 75 µm was selected for the selective flotation locked cycle testwork.

13.5.2.1. LOCKED CYCLE TESTWORK

The summary of the locked cycle test on the selective flotation option is shown in Table 13.8.

Table 13.8 Summary of Locked Cycle Test Results (Selective Flotation)

Product Description	Weight%	% Cu	ppm Au	Cu Rec%	Au Rec%
Copper Cleaner Concentrate	0.9	24.03	26.82	87.0	64.9
Copper Rougher Concentrate	1.3	17.07	19.17	87.2	65.5
Copper Cleaner Tailings	0.4	0.17	0.36	0.3	0.6
Copper Rougher Tailings	98.7	0.03	0.13	12.8	34.5
Total Copper Tailings	99.1	0.03	0.13	13.0	35.1
Pyrite Cleaner Concentrate	1.3	0.31	2.96	1.6	10.6
Pyrite Cleaner Tailings	0.3	0.24	0.63	0.3	0.5
Final Tailings	97.2	0.03	0.09	10.8	23.4
Feed	100.0	0.24	0.37	100.0	100.0

The locked cycle testwork results presented in Table 13.8 indicate that a flotation concentrate at 0.9% mass pull could potentially recover 87% of the copper and 64.9% gold at saleable grades (Cu 24% and Au 26.8 g/t). The results also indicate that a separate pyrite concentrate can recover an additional 10.6% of gold at a 1.3% mass pull. However, the gold grade is much lower (2.96 g/t) than the expected saleable concentrate grade.

13.5.2.2. PH MODIFIER TESTS

All of the above flotation tests were conducted with Sodium Carbonate as the pH modifier but it was noted that the consumption of this reagent for pH modification was excessively high at approximately 12 kilograms per tonne (kg/t) of ore. The higher consumption rates result in significantly increased flotation operating costs.

As a result of higher operating costs, a set of additional rougher tests were conducted with different pH levels in order to identify the optimal pH where a compromise in rougher recovery can be justified against the cost savings. These tests were conducted with sodium carbonate and lime as the pH modifiers and a range of pH conditions were tested as shown in Table 13.9. The results of pH modifier tests are presented in Table 13.9.

Table 13.9 Summary of pH Modifier Tests

pH modifier	Lime		Sodium Carbonate	
pH	Cu Recovery (%)	Au Recovery (%)	Cu Recovery (%)	Au Recovery (%)
8.0	N/A	N/A	81.3	52.4
9.0	84.0	54.9	85.6	57.1
9.5	83.7	54.4	86.1	60.2
10.0	84.4	61.1	86.6	59.9

The results have indicated two different trends (for lime and Sodium Carbonate) between the recoveries and pH levels. For the Sodium Carbonate scenario, the copper and gold recoveries were steadily increased with pH increments. However for the lime scenario, copper recovery was consistent at approximately 84% for all of the tested pH values; the gold recovery was consistent at approximately 55% for pH 9 and 9.5 but increased to 61% for pH 10.

An economic analysis was conducted based on these results in order to identify the optimal conditions for plant design.

13.5.2.3. ECONOMIC ANALYSIS OF PH MODIFIERS

The dependence of copper recovery on pH is clearly demonstrated in Table 13.9. It is clear that, for the sodium carbonate scenario, the higher the pH, the greater the copper recovery. Although copper recovery considerations are generally overriding, higher sodium carbonate consumption incurs a cost.

The net copper revenue and the incremental reagent cost for raising the pH level for improved percent recovery were considered in this analysis for a set of standard parameters (Table 13.10) in order to determine the optimum pH for plant operation. The gold revenues were excluded in this analysis, since a captive carton in leach (CIL) plant is considered for the treatment of flotation tails.

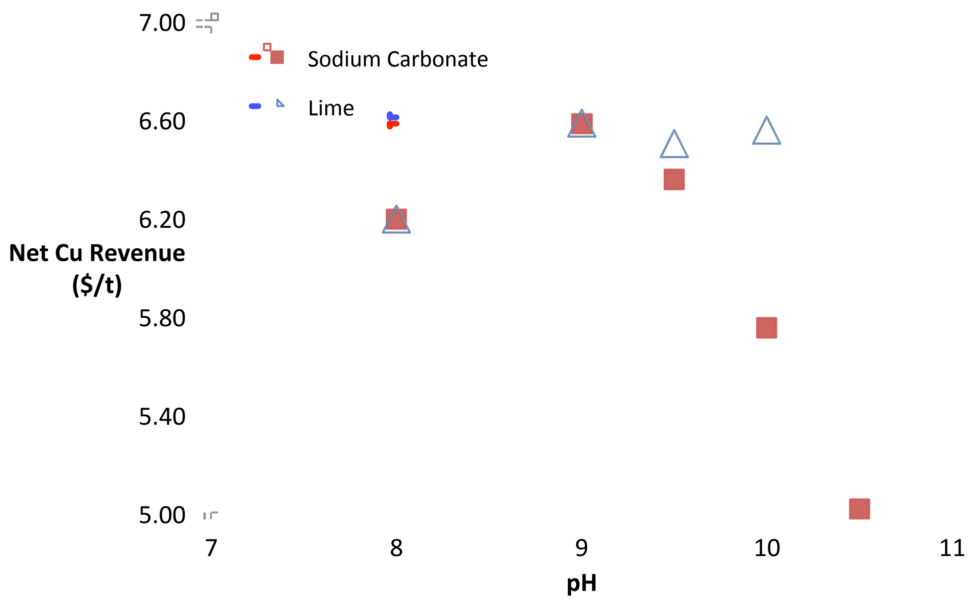
Table 13.10 Economic Analysis Parameters

Throughput (t/a)	6,000,000
Head grade - % Cu	0.24
Cu Price ($/lb)	3.30
Sodium Carbonate Price ($/kg)	0.17
Lime Price ($/kg)	0.14
Cleaner Cu recovery (%)	95%

Note: $/kg = dollars per kilogram, $/lb = dollars per pound, t/a = tonnes per annum

The summary of economic analysis is shown in Figure 13.6 .

Figure 13.6 Variation of Net Copper Revenue with pH



Note: $/t = dollars per tonne

Figure 13.6 clearly indicates that the net copper revenue is maximised at approximately pH 9 for both sodium carbonate and lime as pH modifiers. It can be seen that the net revenue is almost the same for both Sodium Carbonate and Lime at pH 9 despite the differences in the percent recovery (Table 13.9). Based on the additional testwork and economic analysis Tetra Tech recommends lime as the pH modifier and a pH value of 9; copper recovery of 84% and gold recovery of 55% can be attained at this pH level.

13.5.3 KINETIC LEACH TESTS

A bulk flotation test was conducted at pH 9 with lime in order to generate samples for the CIL test. The grind for the flotation feed was fixed at approximately P_{80}=75 µm for consistency.

Kinetic leach tests were conducted on the flotation tails for 48 hours but samples collected and analysed in 2, 4, 6 and 24 hour (h) intervals. The cyanide strength and pH were varied among the tests. A summary of the leach test results is shown in Table 13.11.

Table 13.11 Summary of Kinetic Leach Tests

Test	pH	CN (g/l)	Gold Dissolution (%)	
Leach Retention Time			24h	48h
CN1	11.0	0.3	52.5	65.9
CN2	11.0	0.1	69.1	70.5
CN3	11.0	0.2	55.8	77.6
CN7	11.0	0.5	73.7	76.4
CN8	10.5	0.3	76.2	79.0
CN9	11.5	0.3	71.1	73.6

Note: g/l = grams per litre

As seen from the results the 48 h gold dissolution was comparable for all of the tests with the lowest dissolution reported for CN1 and highest dissolution reported for CN8.

It appears that the residual cyanide in solution for leaching may not be sufficient for the tests carried out for the CN2 and CN3 tests.

The CN1 test showed significantly lower gold dissolution compared to the remaining tests. The reason for the lower dissolution is not known but this may be due to sampling and assaying errors.

As a result of these anomalies, tests CN1, CN2 and CN3 were not considered in any further analysis in order to avoid bias.

It was found that the gold dissolutions observed after 24 h leach were only marginally less than the 48 h leach gold dissolutions for tests CN7, CN8 and CN9.

Based on the reported testwork, Tetra Tech recommends a 24 h leach retention time for the CIL plant design. It is understood that the mean gold dissolution of 73.7% (tests CN7, CN8 and CN9) could be attained at the proposed design retention time.

13.5.4 SETTLING TESTS

Settling tests were conducted on the flotation tailings samples in order to determine the optimum flocculent addition, settling rate and required thickener area. Based on the results, the required specific thickener areas were calculated by SGS according to the Talmage & Fitch method. A summary of the settling test results is presented in Table 13.12.

Table 13.12 Summary of Settling Test Results

Feed Pulp Density (% Solids)	Product Pulp Density (% Solids)	Unit Thickener Area (m^2/t/day)
5	20.6	0.22
	24.4	0.55
	28.2	0.79
6	24.5	0.21
	28.3	0.48
	32.1	0.68
8	20.6	0.27
	24.8	0.75
	29.1	1.09
10	18.7	0.64
	24.1	1.08
	29.4	1.35

Note: m^2/t/day = square metres per tonne per day

The results shown in Table 13.12 indicate that the required thickener areas remain identical for the different feed pulp densities tested. However it can be seen that the required thickener area has gradually increased with the product pulp density requirements.

Tetra Tech recommends a required thickener area of approximately 1.35 m^2/t/day for the prefeasibility level equipment selections but more tests are required in the future to precisely define the thickener area requirements.

13.6 PROCESS SELECTION AND FLOWSHEET DEVELOPMENT

The completed metallurgical testwork indicated that Ilovitza ore is low grade and moderately hard but amenable to flotation and cyanidation at approximately P_{80}=75 µm size. The process flowsheet has been developed based on the testwork findings with the objective of producing a saleable copper concentrate and maximising the gold recovery.

The comminution testwork indicated that the ore is generally rated as moderately hard. Based on the testwork results a comminution circuit that contains primary crushing followed by a SAG and ball mill grinding is recommended.

The flotation testwork has indicated that saleable copper concentrates can be produced (24% copper grade) with approximately 1% mass pull and good recoveries (84% copper and 55% gold). Based on this testwork a copper flotation circuit that contains rougher, scavenger flotation followed with concentrate regrinding and cleaner flotation is recommended as the primary processing method. It is also recommended that copper concentrates should be dewatered as required for shipping to a smelter.

The cyanidation of flotation tails has shown additional potential for gold recovery. It was reported that at 24 h leach retention time the overall gold recovery can be increased up to 88%. Based on this data, a CIL circuit with 24 h retention time is recommended.

A cyanided detoxification process is recommended on the CIL tails in order to meet the industry best practice for Cyanide management and use.

The use of tailings thickener is recommended in order to facilitate the pumping of final tails (detoxified CIL tails) to the tailings management facility (TMF).

In summary, the conceptual process flowsheet included in the preliminary economic assessment (Tetra Tech, 2013) has been confirmed with the addition of a CIL circuit to treat the flotation tails in order to maximise the gold recovery.

14.0 MINERAL RESOURCE ESTIMATES

14.1 SUMMARY

Tetra Tech has adopted the definition of Mineral Resource as outlined within the CIM Definition Standards on Mineral Resources and Mineral Reserves (CIM, 2010).

Tetra Tech has re-estimated the Mineral Resources for the project, with an effective date of 27th November 2013. The most recent data included in the estimate was received on 2nd October 2013. The Mineral Resources have been estimated by Mr. Robert Davies, B.Sc., EurGeol, CGeol, supervised by Mr. Simon McCracken, BAppSc, MAIG, FGS. Euromax provided geological and analytical data in Excel and Access database format. A topographic survey was provided in drawing exchange format file (.dxf) format and consisted of a satellite radar DEM. Modelling and estimation has been completed using Geovia Surpac version 6.3.1.

Exploratory data analysis highlighted a number of statistically differentiated grade populations, which were interpreted to be controlled by the following:

- Level of hydrothermal alteration
- Oxidation state
- Supergene leaching and enrichment.

Wireframe models were used to isolate grade populations into domains for the purpose of sample selection and to constrain the grade interpolation.

Statistical and grade continuity analyses were completed to characterise the mineralisation and subsequently used to develop grade interpolation parameters. Grade estimation was completed using ordinary kriging. The search ellipsoid dimensions and orientations were chosen to reflect the continuity revealed by geostatistical studies and optimised using quantitative kriging neighbourhood analysis.

Estimates for silver and molybdenum were not made as it is Tetra Tech's opinion that the potential for incremental value to be added by these commodities is limited.

A Mineral Resource classification scheme consistent with the CIM guidelines (2010) was applied. The estimates are categorised in the Inferred, Indicated and Measured Mineral Resource categories, reported above a dollar equivalent cut-off grade that defines the Resource as potentially mineable by open pit mining methods.

Dollar equivalent cut-offs were calculated based upon spot metal prices as of 19th August 2013. The metal prices used are US $1,366 /oz Au and US $3.30 /lb Cu.

The dollar equivalent is calculated using the following formula:

Dollar eq = [Au * Recovery * Au Price] + [Cu * Recovery * Cu Price]

A pit optimisation was performed using the Lerchs & Grossman algorithm as implemented in Vulcan. The pit shell was generated to define blocks within the model that have reasonable prospects for economic extraction.

Resource grade / tonnage sensitivity tables were created based upon a range of dollar equivalent cut-offs for blocks within the overall Resource pit shell. A base case cut-off of US $16 /t was chosen for sulphide materials and US $8 /t for oxide materials.

14.2 Geological Interpretation

The Ilovitza porphyry system is approximately 1.5 km in diameter and is associated with a poorly exposed dacite-granodiorite plug, emplaced along the north eastern border of the northwest-southeast elongate Strumitza graben. The exact location of the deposit is controlled by major north-south cross cutting faults and minor northwest faults, parallel to the tectonised border of the graben (see Figure 7.3 and Figure 7.4).

At surface, the Ilovitza intrusive complex consists of a central dacitic breccia diatreme, approximately 1.3 km in diameter. The diatreme is intruded by at least one dacite and two granodiorite porphyry stocks that have generated several hydrothermal pulses, resulting in widespread multi-phase veining within a mineralised stockwork.

The Ilovitza porphyry is centred on a hill of more than 400 m of absolute relief, surrounded at lower elevations by numerous small dikes and irregular bodies of dacitic tuff / breccias and intermediate volcanic rocks.

Alteration related to Tertiary magmatic activity at Ilovitza is variably present over an area of about 8 km^2. Pervasive alteration is largely confined to a roughly 1.5 km^2 area in and adjacent to the main intrusive complex. Smaller areas of pervasive and structurally-controlled alteration extend asymmetrically to the south and east of the intrusive complex. Interpreted alteration zones are illustrated in Figure 7.3 and Figure 7.4.

Subsequent supergene activity is understood to have re-mobilised metal from the higher elevations within the deposit, resulting in a leached and depleted cap. The metal appears to have been deposited beneath the depleted cap, resulting in an enriched layer of between 150 m and 180 m in thickness, located at an elevation of between 325 m and 754 m.

14.3 Wireframe Models

A 3D wireframe model of mineralisation has been prepared for the entire deposit, using an in-situ dollar equivalent cut-off of $10, calculated using the following formula:

In-situ dollar Equivalent = [Au grade * Au price] + [Cu grade * Cu price]

Where the mineralisation is still open laterally, the wireframe has been extended 50 m beyond the drillholes. The dimensions of the mineralisation wireframe are summarised in Table 14.1.

The overall mineralisation wireframe has been sub-divided into seven domains based upon the exploratory data analysis described in Section 14.4. The seven wireframes divide the deposit based upon:

- Level of hydrothermal alteration
- Oxidation state
- Supergene leaching and enrichment.

The dimensions of the seven domain wireframes are summarised in Table 14.1.

Table 14.1 Summary of Wireframe Dimensions

Target	Min. (m) X	Max. (m) X	Min. (m) Y	Max. (m) Y	Min. (m) Z	Max. (m) Z	Volume (m^3)
Total Mineralisation	7,653,544	7,654,589	4,594,652	4,595,657	-13	771	225,288,079
Oxide Stockwork (OS)	7,653,646	7,654,370	4,594,746	4,595,581	476	754	76,113
Oxide Non-Stockwork (ONS)	7,653,574	7,654,381	4,594,763	4,595,580	476	771	4,480,295
Transitional Stockwork (TS)	7,653,649	7,654,376	4,594,746	4,595,579	445	719	4,034,613
Transitional Non-Stockwork (TNS)	7,653,567	7,654,442	4,594,657	4,595,553	447	768	2,035,152
Fresh Enriched Stockwork (FES)	7,653,649	7,654,518	4,594,746	4,595,451	325	754	47,962,809
Fresh Non-Stockwork (FNS)	7,653,544	7,654,589	4,594,652	4,595,657	-13	736	92,721,644
Fresh Non-Enriched Stockwork (FNES)	7,653,682	7,654,502	4,594,745	4,595,653	16	669	73,977,452

Note: m^3 = cubic metre Max. = maximum Min. = minimum

14.4 Exploratory Data Analysis

Euromax provided Tetra Tech with drillhole data in Excel and Access database format. The data consisted of collar, assay, lithology, downhole survey, alteration structural and sample recovery information. Euromax is responsible for the database management of the drill programmes.

Tetra Tech linked the supplied data to Surpac via the database utility. Validation checks were made following connection and any errors noted and communicated to Euromax for correction.

Only diamond core drilling data was used for the Resource estimation.

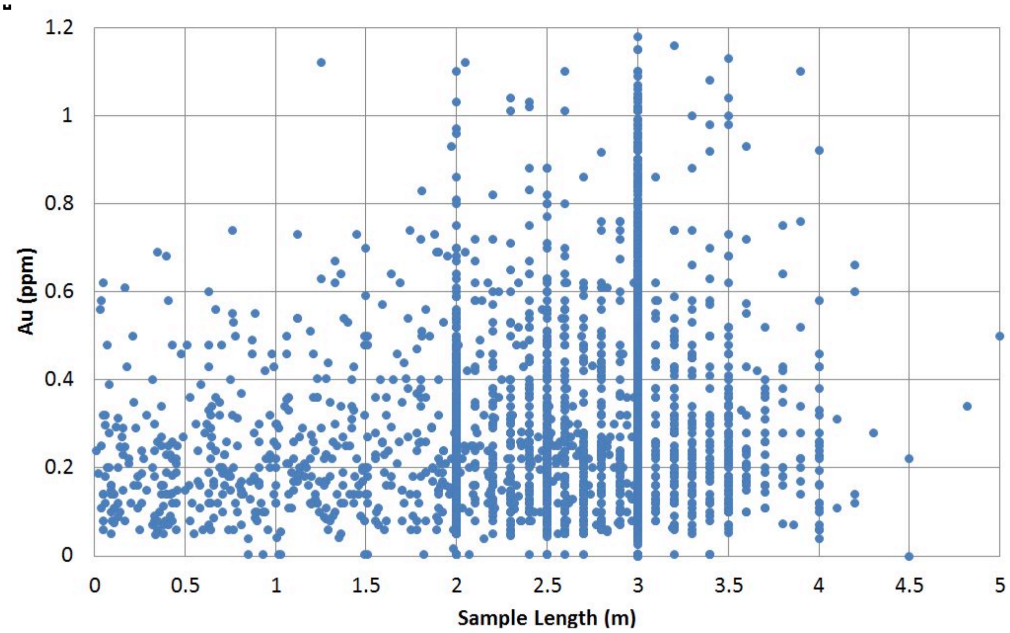
Descriptive statistics for all of the raw samples from within the mineralised wireframes are presented in Table 14.2.

Table 14.2 Descriptive Statistics for All Raw Assays

Measure	Au (ppm)	Cu (%)
Mean	0.28	0.17
Standard Error	0.00	0.00
Median	0.23	0.16
Mode	0.14	0.01
Standard Deviation	0.38	0.12
Coefficient of Variation	1.33	0.72
Sample Variance	0.14	0.02
Kurtosis	1678.91	43.09
Skewness	33.06	3.74
Range	22.20	2.86
Minimum	0.00	0.00
Maximum	22.20	2.86
Sum	2393.88	1442.04
Count	8470.00	8470.00
Confidence Level (95.0%)	0.01	0.00

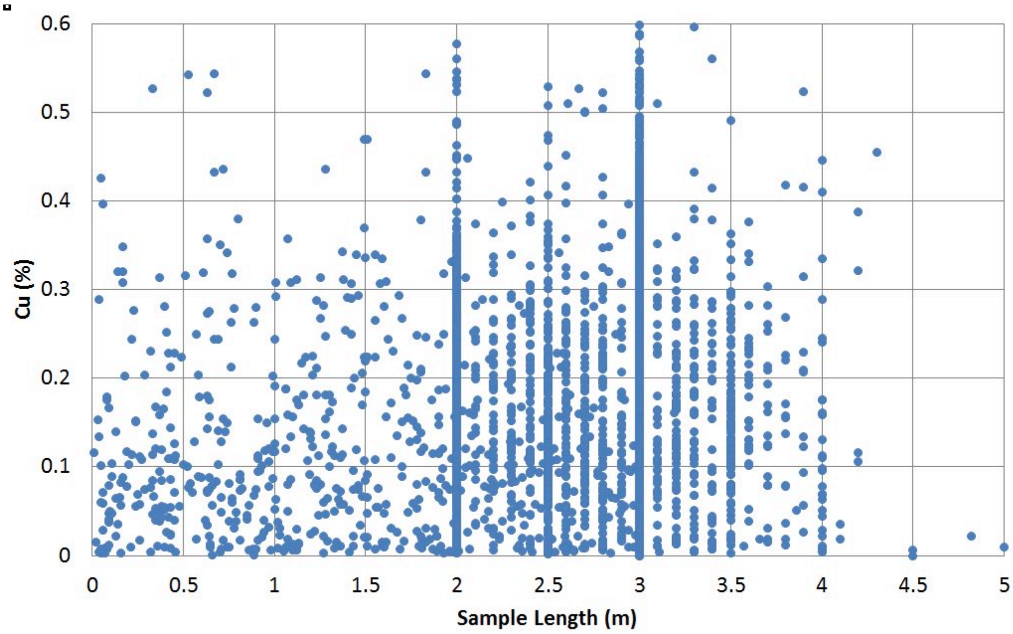
There is no relationship noted between sampled length and grade, as illustrated in Table 14.1 and Table 14.2.

Figure 14.1 Raw Sample Length Plotted Against Gold Grade



Source: Tetra Tech

Figure 14.2 Raw Sample Length Plotted Against Copper Grade



Source: Tetra Tech

14.5 Compositing

Lithology and alteration observations dictated sample interval selection for all drilling campaigns at Ilovitza.

The mean sample length for all raw samples within the mineralised wireframe is 2.74 m. A 3 m best fit routine was utilised to produce composites within hard domain boundaries. The compositing was completed in Surpac. Descriptive statistics associated with all of the composited samples within the mineralised wireframe are given in Table 14.3.

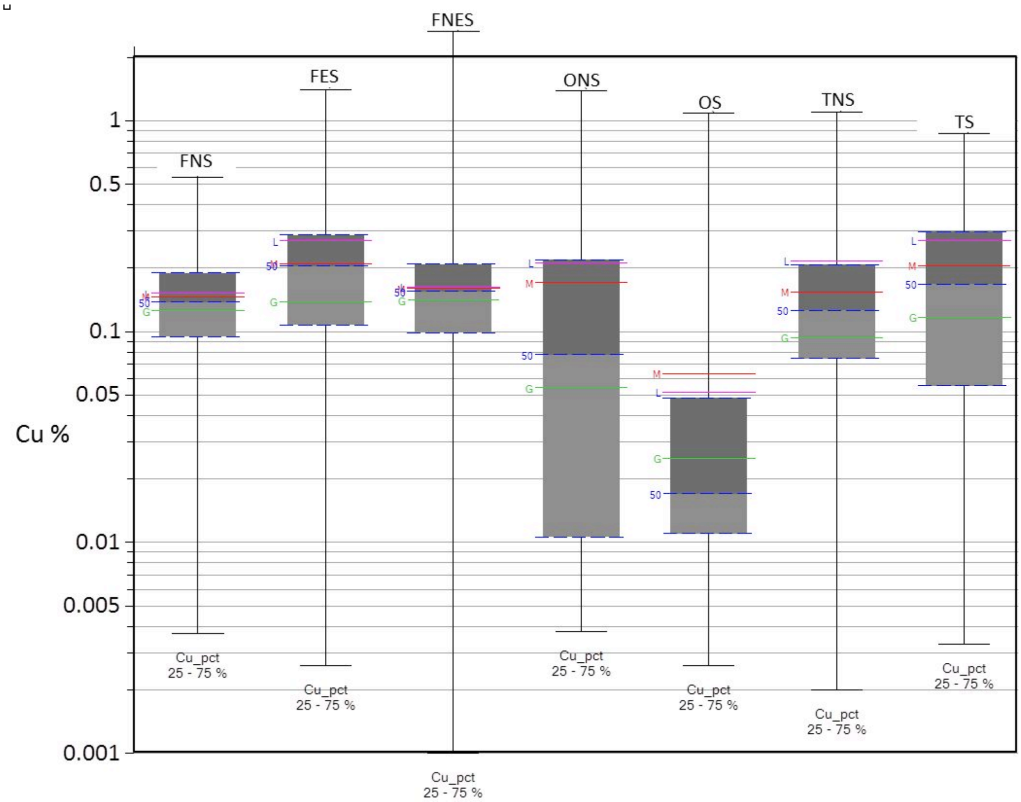
Table 14.3 Descriptive Statistics for All 3 m Composites

Measure	Composites	
	Au	Cu
Mean	0.28	0.17
Standard Error	0.00	0.00
Median	0.23	0.16
Mode	0.12	0.01
Standard Deviation	0.32	0.12
Coefficient of Variation	1.13	0.69
Sample Variance	0.10	0.01
Kurtosis	826.11	33.20
Skewness	22.12	3.29
Range	15.20	2.45
Minimum	0.00	0.00
Maximum	15.20	2.45
Sum	2130.45	1273.12
Count	7476.00	7476.00
Confidence Level (95.0%)	0.01	0.00

14.6 Population Analysis and Domaining

Multiple statistical grade populations were noted in the samples contained within the overall mineralisation wireframe. The box and whisker plots presented in Figure 14.3 and Figure 14.4 suggest multiple differentiated grade populations, thus supporting the interpreted domaining.

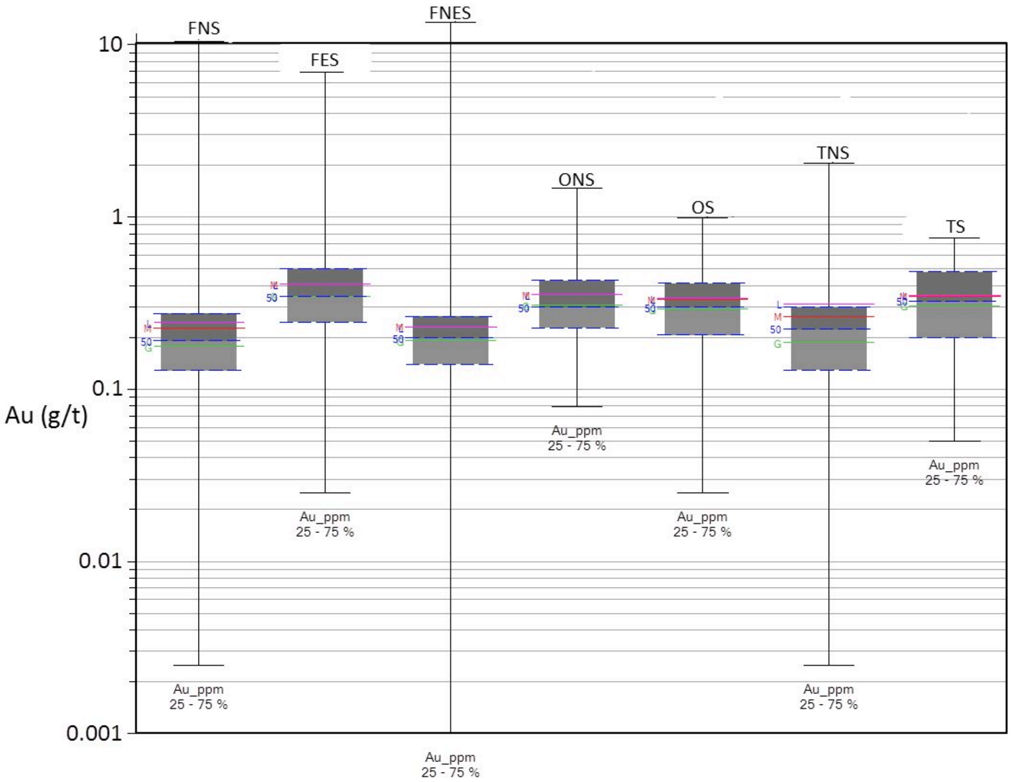
Figure 14.3 Box and Whisker Plot of log Copper against Domain



Source: Tetra Tech

Notes: L = Log Estimated Mean, G = Geometric Mean, M = Arithmetic Mean, pct = Percentage

Figure 14.4 Box and Whisker Plot of log Gold against Domain



Source: Tetra Tech

Separating the samples into the seven domains resulted in a series of single, well distributed, log normal grade populations for gold. The copper grade distribution in the fresh domains also presented as single log normal populations. The copper grade distribution within the oxide and transitional zones were greatly improved through domaining, but signs of mixing of populations still remain.

Alteration and lithology were shown to be ineffective in isolating grade populations, even when combined.

Table 14.4 presents a pivot table, showing grade against alteration style for the oxide and transitional materials. The analysis shows that the gold distribution is not controlled by alteration style, however there are elevated average copper grades associated with the potassic alteration styles.

Euromax RESOURCES

Table 14.4 Pivot Table of Alteration Style against Grade

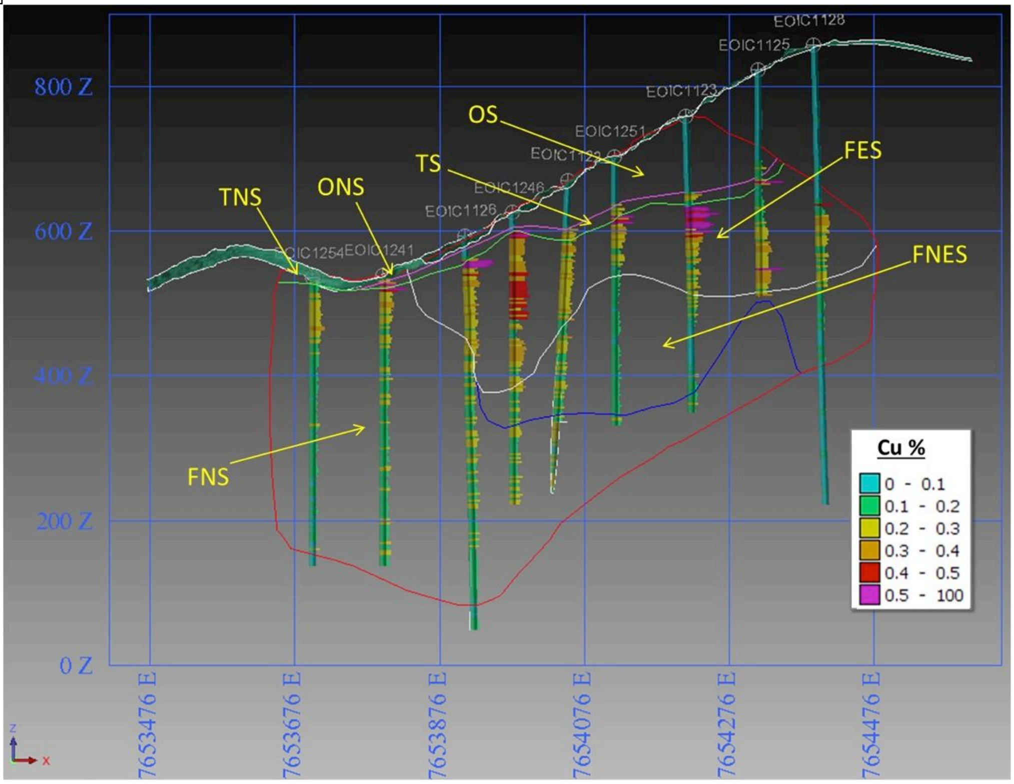
Alteration	Au (ppm)		Cu (%)	
	Average	Max.	Average	Max.
Argillic	0.27	1.31	0.04	0.83
Biotite Chlorite Quartz	0.26	0.43	0.19	0.45
Chlorite Quartz	0.30	0.48	0.44	1.10
Chlorite Sericite Clay Quartz	0.30	0.90	0.25	1.52
Quartz Sericite	0.23	1.48	0.01	0.36
Sericite Clay	0.20	1.52	0.03	1.05
Not Altered	0.22	0.38	0.05	0.09

A combination of oxidation state, style of hydrothermal disturbance and alteration style could provide a suitable means for separating the oxide and mixed copper grade populations into separate domains. However, with the current drill hole spacing, further subdivision of the oxide and transitional material would result in too few composites being available for effective grade interpolation.

The existence of multiple copper grade populations within the oxide zone has no effect on the Mineral Resource, due to the overall grades being very low and it is assumed that no attempt will be made to recover copper from this material. It is also noted that the mixed zone is thin and therefore the effect of mixed copper populations is likely to have negligible impact on the overall Mineral Resource.

Figure 14.5 illustrates an east-west section through the centre of the deposit, presenting the final domains used for the estimation.

Figure 14.5 East-West Section through Wireframe Models



Source: Tetra Tech

14.7 CAPPING

Capping analysis was completed for copper and gold in each domain using the 3 m composites. Decile analysis (Parrish, 1997) was completed and the grade distributions were plotted as histograms and probability plots on normal and log scales.

Capping requirements were based upon the need to exclude outlier results and to avoid having a significant concentration of metal within the final deciles / centiles in the Parrish analysis. The cappings applied to the composites prior to estimation are presented in Table 14.5.

Table 14.5 Summary of Grade Capping

Domain	Metal	Cap	Number of Composites Changed
Oxide	Au	No cap required	
	Cu	0.80%	6
Mixed	Au	1 ppm	2
	Cu	No cap required	
Fresh	Au	3 ppm	8
	Cu	No cap required	

14.7.1 Bulk Density

Euromax provided density results for 62 samples, taken from the 2008, 2010, 2011, 2012 and 2013 drilling campaigns. The samples included granite, granodiorite and dacite, and were taken from the oxide transitional and fresh zones.

The density values presented in Table 14.6 were adopted for the reporting of tonnages.

Table 14.6 Density Values

Lithology	Oxidation State	Density (t/m^3)
Mean of all Lithologies	Oxide	2.30
Dacite	Fresh/ Transitional	2.48
Granite	Fresh/ Transitional	2.58
Granodiorite	Fresh/ Transitional	2.54

Note: t/m^3 = Tonnes per cubic metre

The density testing was completed by the faculty of civil engineering, Sts. Cyril and Methodius University, Skopje, Macedonia.

14.7.2 Variography

In order to maintain sufficient numbers of composites, variography was completed separately for the oxide, transitional and fresh material, without sub-dividing the dataset into all seven separate domains.

Variography was completed in Visor software Version 8.1. A normal scores experimental variogram was produced in plan view initially, with the strike, dip and plunge established independently thereafter. The nugget value has been established from a downhole variogram.

The modelled variograms were back transformed to establish the variogram structures to be utilised within the kriging estimation.

Weak to moderate directional control has been established in the majority of the domains for both copper and gold. Table 14.7 presents the variogram parameters used for the estimation.

Table 14.7 Directional Variogram Parameters

Oxidation	Metal	Nugget	Sill		Range (m)		Azimuth (°)	Plunge (°)	Dip (°)	Ratio	
			1	2	1	2				Semi-Major	Minor
Oxide	Au	0.13	0.87	-	144.00	-	10.00	0.00	-80.00	1.19	2.48
Oxide	Cu	0.11	0.70	0.19	144.00	247.00	124.00	14.00	5.00	3.98	7.48
Transitional	Au	0.06	0.94	-	353.00	-	238.20	-15.00	-13.16	1.14	4.30
Transitional	Cu	0.06	0.54	0.40	196.00	274.00	20.00	0.00	-20.00	1.96	3.75
Fresh	Au	0.43	0.40	0.17	186.00	256.00	319.00	-58.00	16.70	1.16	1.80
Fresh	Cu	0.12	0.40	0.48	132.00	595.00	34.00	-29.50	78.50	1.50	1.81

14.8 Resource Block Models

A single block model has been constructed in Surpac for the project. The block model parameters are given in Table 14.8.

Table 14.8 Block Model Parameters and Extents

Measure	Y	X	Z
Minimum Coordinates	4593900	7652533	0
Maximum Coordinates	4596400	7655433	1050
User Block Size	25	25	10
Minimum Block Size	25	25	10
Rotation	0	0	0

Block partial percentages were used to record the percentage of each block contained within the mineralisation wireframe, and also within each of the seven domains. A comparison of the overall mineralisation wireframe and block model volumes (as block percentages) is given in Table 14.9.

Table 14.9 Comparison of Block Model and Wireframe Volumes

Description	Unit	Volume
Wireframes	m^3	225,288,079
Block model	m^3	219,562,118
Difference	%	2.6

14.8.1 Interpolation Strategy

Grades were estimated using ordinary kriging, adopting a multi-pass methodology. The kriging employed variogram parameters as presented in Section 14.7.2. Each of the seven domains was estimated independently for both gold and copper.

Quantitative kriging neighbourhood analysis was undertaken to optimise the block size, number of informing samples, discretisation and search distances used in the estimation.

Quantitative measures of the kriging performance (e.g. slope of regression, kriging efficiency, kriging variance, block variance, proportion of negative weights) were used to test the appropriateness and optimise the kriging parameters. The analysis was undertaken on large representative portions of the block model as well as on isolated test blocks. Where test blocks were used, these were chosen as examples of one well informed block and one poorly informed block.

A summary of the estimation strategy is show in Table 14.10.

Table 14.10 Estimation Strategy

Domain	Metal	Pass 1			Pass 2			Pass 3		
		Samples		Search	Samples		Search	Samples		Search
		Min.	Max.	Distance	Min.	Max.	Distance	Min.	Max.	Distance
FES	Au	12	37	100	12	37	150	12	37	300
FES	Cu	10	30	90	10	30	150	10	30	300
FNES	Au	12	37	100	12	37	150	12	37	300
FNES	Cu	10	30	90	10	30	150	10	30	300
FNS	Au	12	37	100	12	37	150	12	37	300
FNS	Cu	10	30	90	10	30	150	10	30	300
ONS	Au	20	40	60	20	40	100	20	40	250
ONS	Cu	15	45	60	15	45	120	15	45	250
OS	Au	20	40	60	20	40	100	20	40	250
OS	Cu	15	45	60	15	45	120	15	45	250
TNS	Au	15	45	60	15	45	150	15	45	350
TNS	Cu	45	45	75	15	45	150	15	45	250
TS	Au	15	45	60	15	45	150	15	45	350
TS	Cu	45	45	75	15	45	150	15	45	250

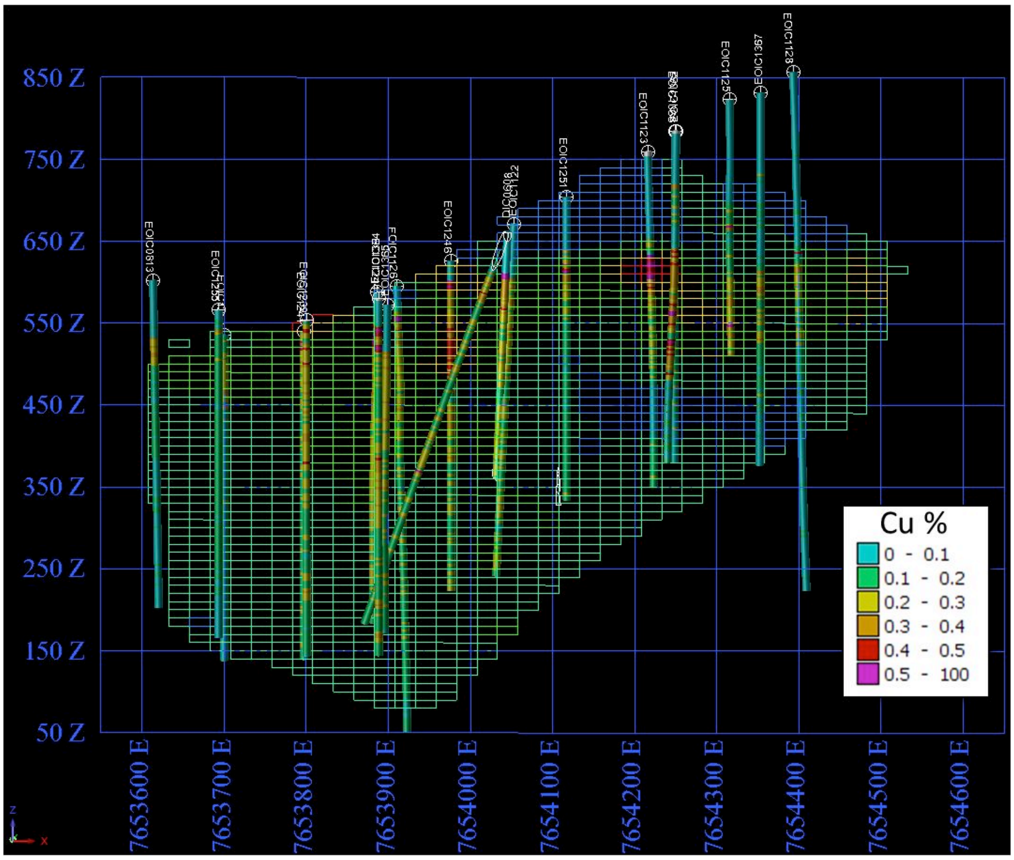
The discretisation of X = 3, Y = 3 and Z = 3 was found to be optimal for all zones. A maximum of six samples per drillhole were allowed to inform the estimation of any one block.

14.9 Block Model Validation

Block model validation was completed using graphical and statistical methods, to confirm that the estimated block model grades appropriately reflect the local composite grades.

Graphical analysis of the informing samples versus estimated block grades was undertaken using horizontal and vertical sections, (selected vertical sections are presented in Figure 14.6 and Figure 14.7).

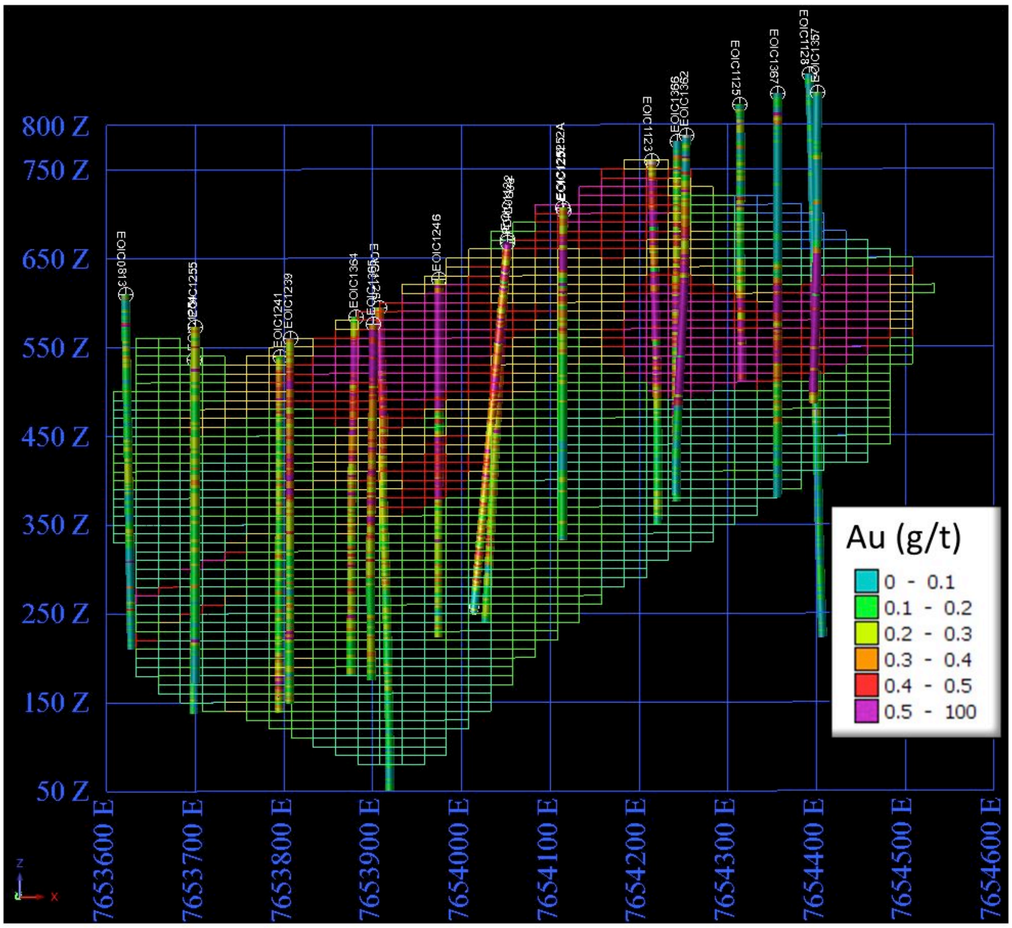
Figure 14.6 **East-West Section through Block Model and Local Drillholes with Copper Grades Presented**



Source: Tetra Tech
Note: Section taken along East- West along a Y coordinate of 4,595,200.

The visual inspection demonstrated excellent correlation between composite and block grades, with the anisotropic directionality noted in the variography reflected in the grade distribution.

Figure 14.7 East-West Section through Block Model and Local Drillholes with Gold Grades Presented

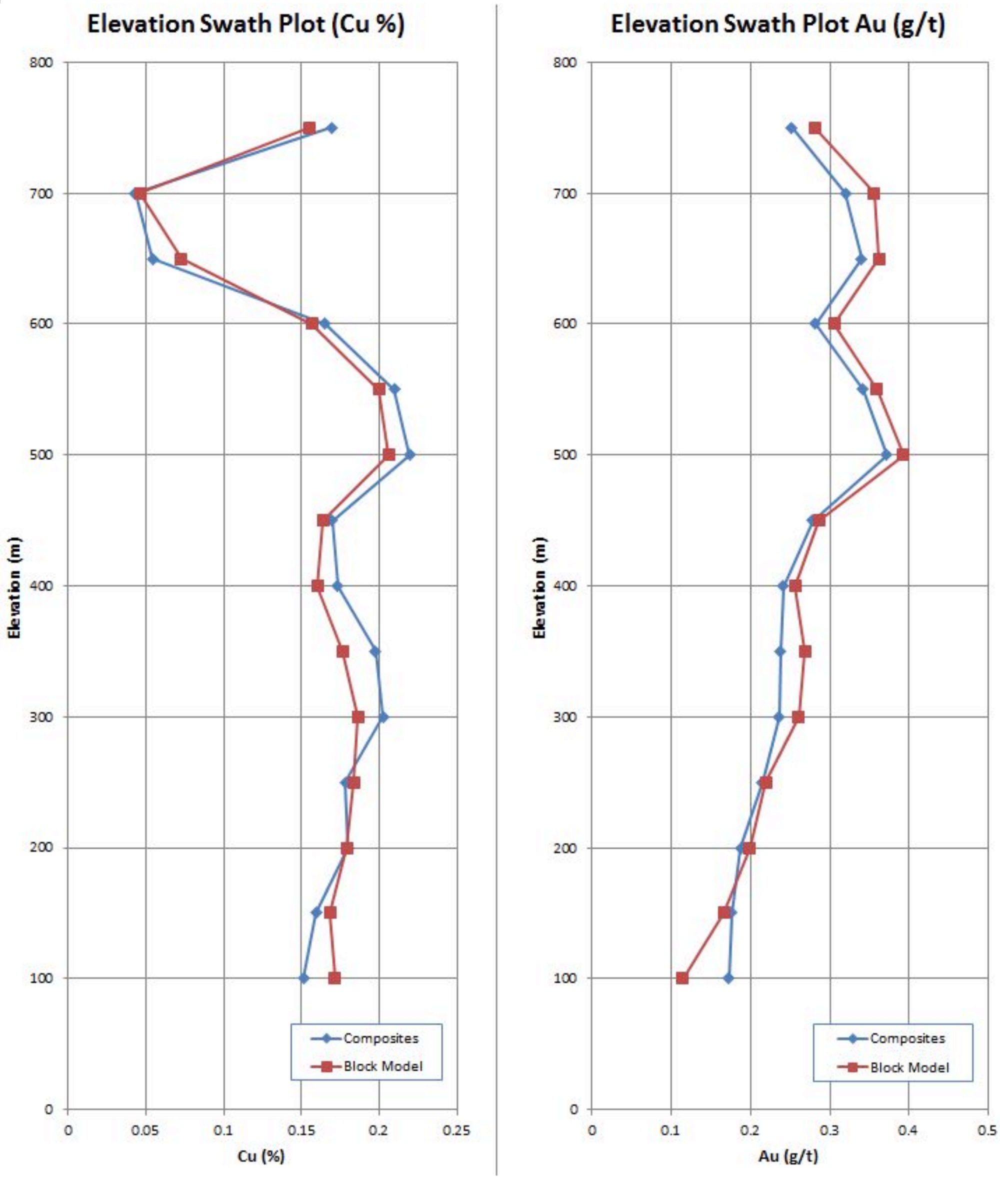


Source: Tetra Tech

Note: Section taken along East- West along a Y coordinate of 4,595,200.

Figure 14.8 illustrates good correlation between the block grades estimated by ordinary kriging, and the informing composites. The supergene leaching and enrichment is exhibited as a depleted cap (particularly with regard to copper) and an enriched zone around the 500 m elevation.

Figure 14.8 Elevation Swath Plots for Copper and Gold



Source: Tetra Tech

Note: Numbers of blocks and composites are limited above 700 m.

A comparison was made between the estimated block grades and the entire informing composite populations for copper and gold. This was undertaken through the use of a range of statistical measures (see Table 14.11).

A number of the measures indicate a reduction in variance. This is as a result of the change of support associated with the estimation process and the kriging interpolation. Overall, the statistics present excellent conformance.

Table 14.11 Statistics Comparing Block Estimate and Composite Grades

Measure	Composites		Block Model	
	Au (g/t)	Cu (%)	Au (g/t)	Cu (%)
Mean	0.28	0.17	0.26	0.16
Standard Error	0.00	0.00	0.00	0.00
Median	0.23	0.16	0.23	0.15
Mode	0.12	0.01	0.22	0.00
Standard Deviation	0.32	0.12	0.14	0.08
Coefficient of Variation	1.13	0.69	0.52	0.47
Sample Variance	0.10	0.01	0.02	0.01
Kurtosis	826.11	33.20	30.16	2.57
Skewness	22.12	3.29	3.36	0.88
Range	15.20	2.45	2.74	0.84
Minimum	0.00	0.00	0.00	0.00
Maximum	15.20	2.45	2.74	0.84
Sum	2130.45	1273.12	10665.34	6561.27
Count	7476.00	7476.00	40480.00	40480.00
Confidence Level (95.0%)	0.01	0.00	0.00	0.00

14.9.1 Conclusion

The various comparators described in the foregoing sub sections serve to illustrate that the block model estimate is robust and satisfactorily models the distribution and variability of the informing sample grades without undue bias or smoothing. It is suitable for the current level of study.

14.10 Mineral Resource Classification and Tabulation

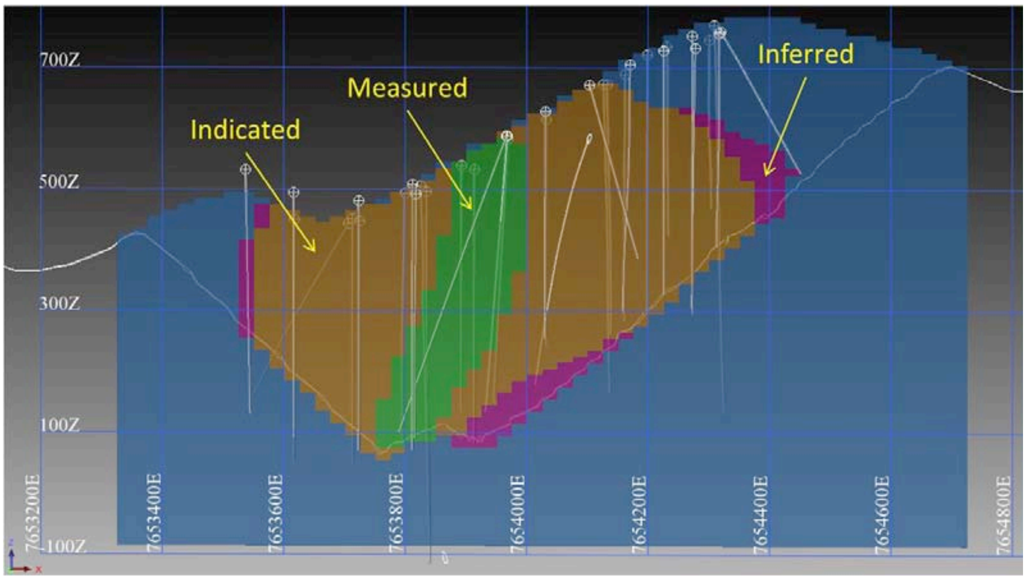
The model was classified according to CIM Definition Standards on Mineral Resources and Mineral Reserves (CIM, 2010).

Wireframe models have been generated to represent the limit of the Measured, Indicated and Inferred Resources. The models consider the following criteria:

- Confidence in the sampling data and geological interpretation
- Analysis of variogram parameters
- The data distribution (based upon graphical analysis and average distance to informing composites)
- Kriging efficiency
- Slope of Regression.

The models reflected the trends in the classification parameters as presented on a block by block basis, whilst ensuring that the classification resulted in appropriately coherent units. Figure 14.9 presents an east-west section illustrating the adopted Resource classification.

Figure 14.9 East-West Section Illustrating the Resource Classification



Source: Tetra Tech

Note: Section taken along East- West along a Y coordinate of 4,595,200.

In order to statistically validate the adopted classification, the average block values within each classification domain were reported, see Table 14.12.

Table 14.12 Statistical Validation of Classification

Classification	Average Distance to Informing Composites (m)	Kriging Efficiency	Slope of Regression
Measured	66	64	90
Indicated	89	48	80
Inferred	144	20	58

14.10.1 Mineral Resource

For the purpose of Mineral Resource reporting, the transitional material has been grouped with either the oxidised or fresh material based upon the copper content. Where the transitional material has less than 0.2% copper, it is regarded as oxide and where greater than 0.2% it is considered as fresh. This approach reflects the fact that there would not be a separate process route for transitional material.

The Mineral Resource for fresh material is summarised in Table 14.13 and Table 14.14.

Table 14.13 Measured and Indicated Fresh Mineral Resource Based upon a Dollar Equivalent cut-off of $16 /t

Classification	Tonnage (Kt)	Grade		Contained Metal	
		Au (g/t)	Cu (%)	Au (Koz)	Cu (Klb)
Measured	18,440	0.34	0.22	200	88,677
Indicated	218,640	0.33	0.22	2,341	1,036,427
Total M + I	237,080	0.33	0.22	2,541	1,125,104

Table 14.14 Inferred Fresh Mineral Resource Based upon a Dollar Equivalent cut-off of $16 /t

Classification	Tonnage (Kt)	Grade		Contained Metal Tonnage (Kt)	
		Au (g/t)	Cu (%)	Au (Koz)	Cu (Klb)
Inferred	19,850	0.36	0.22	226	96,942

The oxide Mineral Resources within the constraining pit shell are summarised within Table 14.15.

Table 14.15 Measured and Indicated Oxide Mineral Resource based upon a Dollar Equivalent cut-off of $8 /t

Classification	Tonnage (Kt)	Grade Au (g/t)	Contained Metal Au (Koz)
Measured	1,340	0.38	16
Indicated	34,540	0.33	365
Total	35,880	0.33	381

Table 14.16 Inferred Oxide Mineral Resource Based upon a Dollar Equivalent cut-off of $8 /t

Classification	Tonnage (Kt)	Grade Au (g/t)	Contained Metal Au (Koz)
Inferred	6,750	0.25	55

Notes:

1. Dollar equivalent cut-offs are based upon the following calculation:
   *Dollar Eq = (Au * recovery *price) + (Cu * recovery *price)*
2. The following assumptions were adopted for the calculation of the dollar equivalent:
   - Au recovery in oxide of 86%
   - Cu recovery in oxide of 0%
   - Au recovery in mixed and fresh 65%
   - Cu recovery in mixed and fresh 85%
   - Recoveries based on previous test work are not viewed by Euromax as materially different from the final recoveries in this study and do not warrant re-reporting of the resource
   - Spot metal prices effective 19th August 2013 of US $1,366 /oz Au and US $3.30 /lb Cu.
3. Numbers may not add exactly due to rounding.
4. Tonnages calculated using the densities outlined in table14.6.
5. Mineral Resources that are not mineral reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
6. Contained gold within this report is quoted in Troy ounces. It is noted that in a press release dated 03/12/2013 ("Euromax Announces Increased Mineral Resource Estimate Prior to Pre-Feasibility Completion") reported the contained gold in standard ounces.

A constraining pit shell has been applied to the 3D block model to ensure reasonable prospects of economic extraction for the above reported Resources. This does not represent a formal pit optimisation. The pit was generated using the Lerchs & Grossman algorithm as implemented in Vulcan. The economic value of the blocks was calculated by the multi-element pit optimiser module of Vulcan based on the following financial and technical parameters:

- Mining cost $2 (US $/t)
- Mining dilution 1%
- Mining recovery 99%
- Pit slope variable according to geotechnical conditions
- Processing cost $6.58 (US $/t)
- Gold selling cost $59.9 (US $/oz) (10% of metal price)
- Copper selling cost $0.47 (US $/lb) (10% of metal price)
- Spot metal prices effective August 19th 2013:
- US $1,366 /oz Au
- US $3.30 / lb Cu.
- Dollar equivalents based upon the following calculation:

*Dollar Eq = (Au * recovery * price) + (Cu * recovery * price)*

Note: The following assumed recoveries were applied:

- Au Recovery in oxide 86%
- Cu Recovery in oxide 0%
- Cu Recovery in fresh 85%

- Au Recovery in fresh 65%
- Recoveries based on previous test work are not viewed by Euromax as materially different from the final recoveries in this study and do not warrant re-reporting of the resource.

A dollar equivalent cut-off was applied to blocks within the overall Resource pit shell to define the Mineral Resource presented within Table 14.13 to Table 14.16. Base case dollar equivalent cut-offs have been chosen based upon assumed processing and General and Administration (G&A) costs of US $14 /t for sulphide/mixed materials and US $6 /t for oxide materials. The assumed mining cost for both mineralised material and waste is US $2 /t.

14.11 Grade Tonnage Sensitivity Analysis

The block model has been reported at a range of dollar equivalent cut-offs, as presented in Table 14.17. It should be noted that the figures presented in Table 14.17 do not constitute a Mineral Resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grades.

Table 14.17 Grade Tonnage Sensitivity for Fresh Materials

Classification	Material	Dollar Equivalent Cut-off (US $)	Tonnage (Koz)	Grade		Contained Metal	
				Au (ppm)	Cu (%)	Au (Koz)	Cu (Klb)
Measured	Fresh	12	18,780	0.34	0.22	204	89,855
		16	18,440	0.34	0.22	200	88,677
		24	7,470	0.45	0.26	108	42,652
		36	2,020	0.67	0.34	43	8,091
Indicated	Fresh	12	290,220	0.3	0.19	2786	1,231,269
		16	218,640	0.33	0.22	2,341	1,036,427
		24	80,040	0.45	0.27	1153	480,740
		36	14,530	0.66	0.35	307	111,630
Inferred	Fresh	12	39,670	0.27	0.18	343	153,087
		16	19,850	0.36	0.22	226	96,942
		24	7,930	0.52	0.29	132	50,875
		36	2,660	0.59	0.35	50	20,195

It is noted that even with higher dollar equivalent cut-offs, the Mineral Resource remains spatially coherent and relatively close to surface. This presents options with regard to the scale of the potential mining operation and offers opportunities with regard to mine scheduling.

14.12 Previous Mineral Resource Estimates

This report presents an updated Mineral Resource for the project with an effective date of 27th November 2013. The previous Mineral Resource was estimated by Tetra Tech, with an effective date of 26th July 2013. The previous Resource estimate was presented in a Technical report released on 16th September 2013.

The previous Mineral Resource statement is summarised in Table 14.18.

Table 14.18 Summary of the July 2013 Mineral Resource Statement

Material	Classification	Resource (Kt)	In Situ Grades		Contained Metal	
			Cu (%)	Au (g/t)	Cu (Mlb)	Au (koz)
Fresh & Mixed	Measured	15,770	0.22	0.35	78,039	194
	Indicated	168,250	0.21	0.33	797,183	1,949
	Inferred	8,200	0.20	0.29	35,952	83
Oxide	Measured	850	n/a	0.37	n/a	11
	Indicated	15,200	n/a	0.36	n/a	192
	Inferred	3,410	n/a	0.32	n/a	38

Note: n/a = not applicable

The July 2013 Resource was calculated using a dollar equivalent cut-off of $16 for the fresh and mixed materials and $8 for oxide materials. The dollar equivalent was calculated as follows:

$$\textit{Dollar Eq} = (Au * recovery * price) + (Cu * recovery * price)$$

Note: The following assumptions were adopted for the calculation of the dollar equivalent:

- Au recovery in oxide of 70%
- Cu recovery in oxide of 0%
- Au recovery in mixed and fresh 83%
- Cu recovery in mixed and fresh 90%
- Recoveries based on previous test work current at the time of reporting
- Spot metal prices effective 17th June 2013 of US $1,385 /oz Au and US $3.18 /lb Cu.

The July 2013 Resource tonnages were calculated using an in-situ-density of 2.15 t/m^3 for oxide materials and 2.45 t/m^3 fresh and mixed. The July 2013 Resource was pit constrained.

Table 14.19 presents a comparison between the July 2013 Resource and the November 2013 Resource statement.

Table 14.19 Comparison between the July 2013 and November 2013 Resource for Sulphide and Mixed Materials

Classification	July 2013 Resource			November 2013 Resource			Tonnage Difference (Kt)
	Tonnage (Kt)	Au (g/t)	Cu (%)	Tonnage (Kt)	Au (g/t)	Cu (%)	
Measured	15,770	0.35	0.22	19,780	0.34	0.22	4,010
Indicated	168,250	0.33	0.21	253,180	0.33	0.22	84,930
Inferred	8,200	0.29	0.20	26,600	0.33	0.22	18,400

The grades have remained relatively constant, however the tonnage has increased significantly, particularly with regard to the indicated materials. The November 2013 Resource included the following updates, which have resulted in the changes observed:

- New lithological and alteration logging instigated as part of the acid rock drainage reassessment allowed separate domaining of enriched supergene stockwork zones.
- Revised interpolation of density data based upon lithological and oxidation modelling.
- The incorporation of three additional drill holes.
- Revised constraining pit shell based upon the new block model and updated assumptions.

15.0 MINERAL RESERVES

A mining plan and schedule were developed for mining the mineral resources that are identified in Section 14. An economic analysis of this proposed mining project was carried out (refer to Section 22). The results were positive. The preliminary mine plan is based on Measured and Indicated mineral resources. This report's preliminary feasibility level of detail requires that both the Measured and Indicated mineral resources be classified as a Probable mineral reserve. No Proven mineral reserves have been designated.

The main assumptions that were used in identifying the mineral reserves are discussed in Section 16, particularly Tables 16.2 and 16.4.

Table 15.1: Mineral reserves (diluted and recovered).

Probable Reserve, Oxide (Diluted and Recovered)	16 Million tonnes
Gold Grade	0.33 g/tonne
Gold Ounces	172,000
Primary/Transitional Probable Reserve (Diluted and Recovered)	209 Million tonnes
Gold Grade	0.34 g/tonne
Gold Ounces	2.28 Million
Copper Grade	0.20%
Copper Pounds	905 Million
Total Probable Reserve (Diluted and Recovered, Rounded)	225 Million Tonnes
Gold Grade	0.34 g/tonne
Gold Ounces (Rounded)	2.45 Million
Copper Grade	0.20%
Copper Pounds	905 Million

Notes:
1. Unplanned dilution equals 5% at diluting grades of 0.17 g/tonne gold and 0.05 % copper.
2. Mining losses = 5%.
3. Mineral reserves are a subset of mineral resources.

15.1 In-Pit Inferred Mineral Resources

Though the mine plan was based on Measured and Indicated mineral resources, Table 15.2 shows the Inferred mineral resources that occur within the planned pit.

Figure 15.1 and Figure 15.2 illustrate where these blocks are located within the pit. They are located mainly at the periphery of the pit. The Inferred blocks are planned to be mined but are not considered to be part of the Mineral Reserve. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves.

With additional drilling, it is possible that these in-pit inferred mineral resources could be upgraded to higher mineral resource categories. However, there is no guarantee that this would occur.

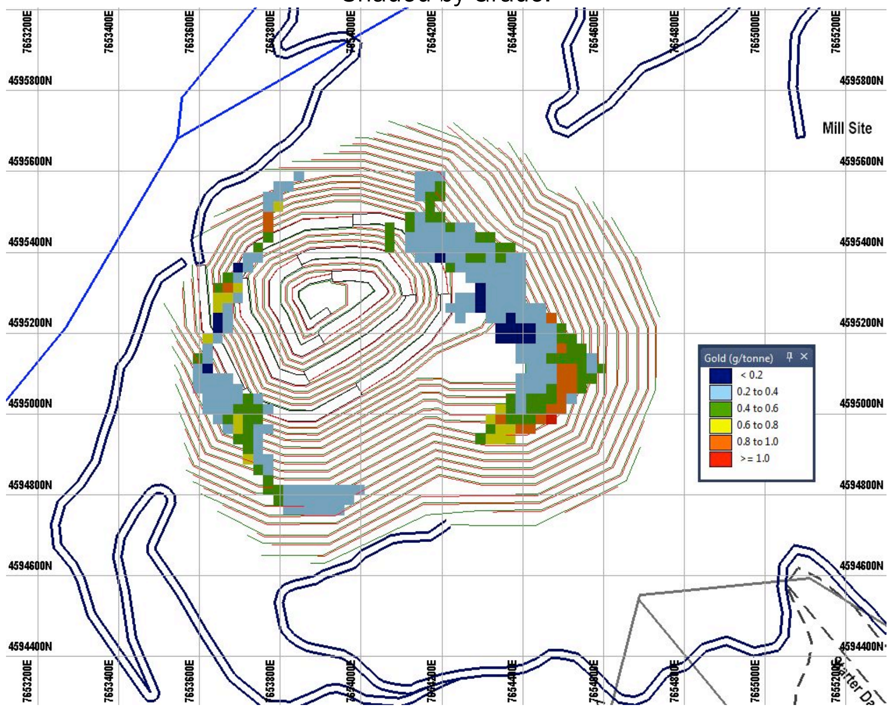
For the purpose of mine scheduling, the in-pit Inferred mineral resource blocks were considered to be waste rock.

Table 15.2: In-pit Inferred mineral resources.

Oxide	
Tonnes (Millions)	2.14
In-Situ Ounces (000s)	19.7
In-Situ Gold Grade (g/tonne)	0.29
Primary/Transitional	
Tonnes (Millions)	15.34
In-Situ Ounces (000s)	166
In-Situ Gold Grade (g/tonne)	0.34
In-Situ Copper Pounds (Millions)	73.70
In-Situ Copper Grade	0.22%
Total Inferred (Rounded)	
Tonnes (Millions)	17.5
In-Situ Ounces (000s)	186
In-Situ Gold Grade (g/tonne)	0.33
In-Situ Copper Pounds (Millions)	73.7
In-Situ Copper Grade	0.22%

Figure 15.1: Inferred blocks within the final pit.

Shaded by Grade:



Shaded by Elevation:

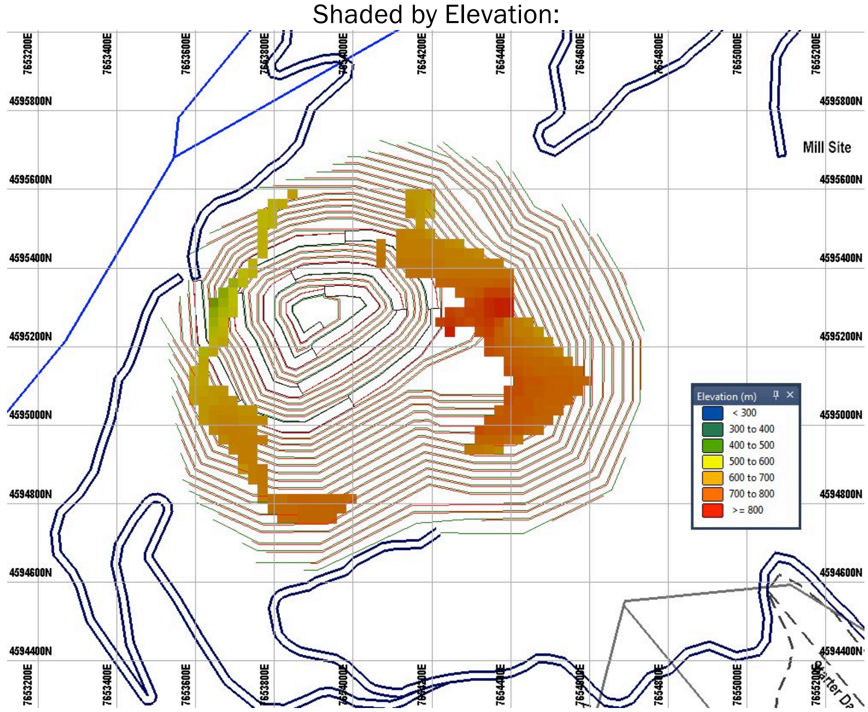
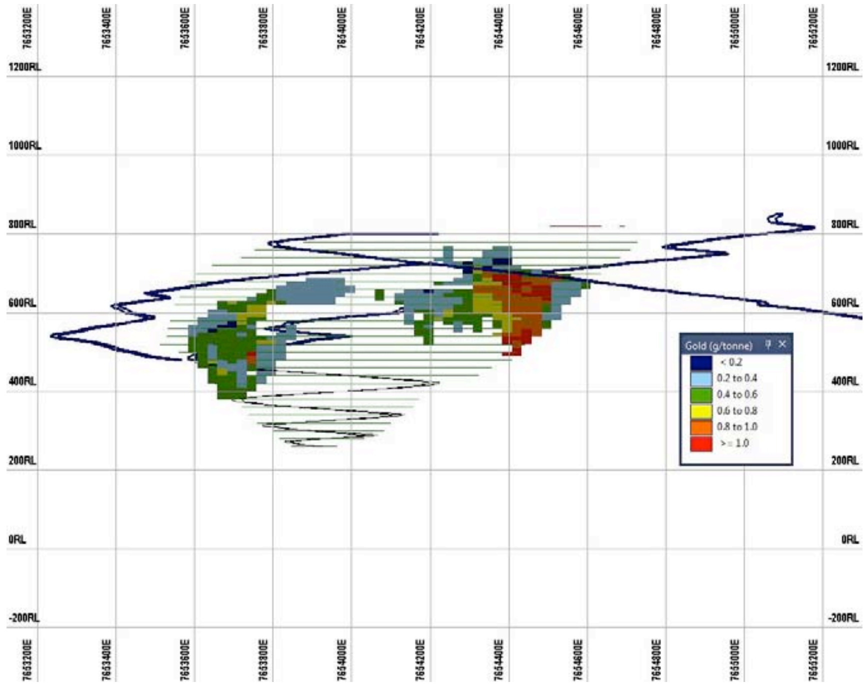


Figure 15.2: Inferred blocks within the final pit, facing north, shaded by grade.



15.2 Mineral Reserve Risk

The quantity and grade of mineral reserves can be affected by economic and political factors such as mining costs, geotechnical parameters, mineral processing costs and recoveries, and permitting. Increasing costs or decreasing prices could decrease the quantity of material that can profitably be mined. Conversely, decreasing costs or increasing prices could increase the quantity of material that can profitably be mined.

As discussed in Section 22, the greatest risks to the project are (i) a severe decrease in metal price, and (2) and marked increase in operating cost. Those represent the riskiest parameters of most mining projects. In that regard, this project is no more or no less risky than any other mining project of this type.

16.0 MINING METHODS

16.1 INTRODUCTION

In January, 2014, Pat Forward, Chief Operating Officer of Euromax Resources engaged ACA Howe International Limited ("Howe") to carry out pit optimisation, detailed pit design, production scheduling, capital cost estimation, and operating cost estimation, to a detail that would be suitable for a preliminary feasibility study for the Ilovitza gold-copper deposit in Macedonia. Euromax supplied all of the data, including optimisation parameters, the current block model, and other digital files.

16.2 PIT SLOPES

Tetra Tech carried out a detailed geotechnical study. They concluded that an overall pit slope angle of 39° would provide long-term slope stability.

16.3 PIT OPTIMISATION

The following pit optimisation parameters were supplied (Table 16.1). Nested pits were optimised at a range of gold prices (refer to Table 16.2).

Table 16.1 Pit optimisation parameters.

Parameter	Value
Gold Price	$US 1250 per Ounce
Copper Price	$US 3.00 per Pound
Exchange Rate	$US 1 = $MKD 43.50
SG, Non-Mineralised Rock	2.45
Mining Cost, Ore & Waste	$1.80 per tonne
Mining Cost Escalation With Depth	$0.01 per metre below 400 m MSL
Mining Dilution	5%
Mining Losses	5%
Processing Cost	Oxide $6.24 per tonne Fresh $9.25 per tonne
Milling Rate	10 Million Tonnes per Year
Milling Recovery	
Oxide	90% Gold 0% Copper
Fresh	90% Gold 85% Copper
General & Administration	$1 per tonne
Pit Slope	39°
Reclamation Cost, Ore & Waste	$0.25 per tonne
Selling Costs	Gold $62.50 per Ounce Copper $0.15 per Pound

Note: Mill recoveries based on earlier test results but difference from final recoveries not viewed as material by Euromax.

Table 16.2 Nested pit gold prices.

Pit	Percent of Base Case Gold Price	Gold Price ($US per Oz)	
1	45%	$	563
2	50%	$	625
3	52%	$	650
4	54%	$	675
5	56%	$	700
6	57%	$	713
7	58%	$	725
8	60%	$	750
9	62%	$	775
10	64%	$	800
11	66%	$	825
12	68%	$	850
13	70%	$	875
14	72%	$	900
15	74%	$	925

16.4 Results of Pit Optimisation

Fifteen nested pits were outlined at various gold prices (refer to Table 16.2 for gold prices). Figure 16.1 and Figure 16.2 show a plan view and cross-sections of the optimum pits, respectively.

Four nested pits were selected for more detailed pit design and production scheduling.

Figure 16.1: Plan view of nested pits.

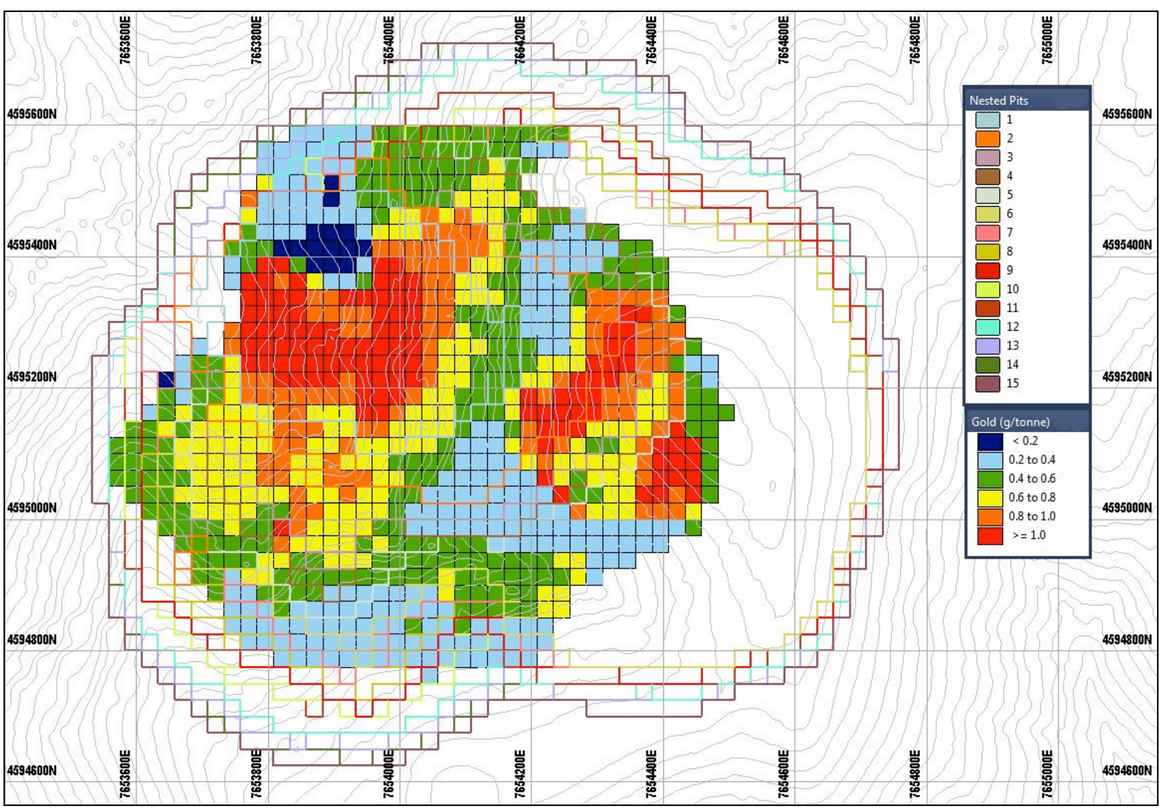
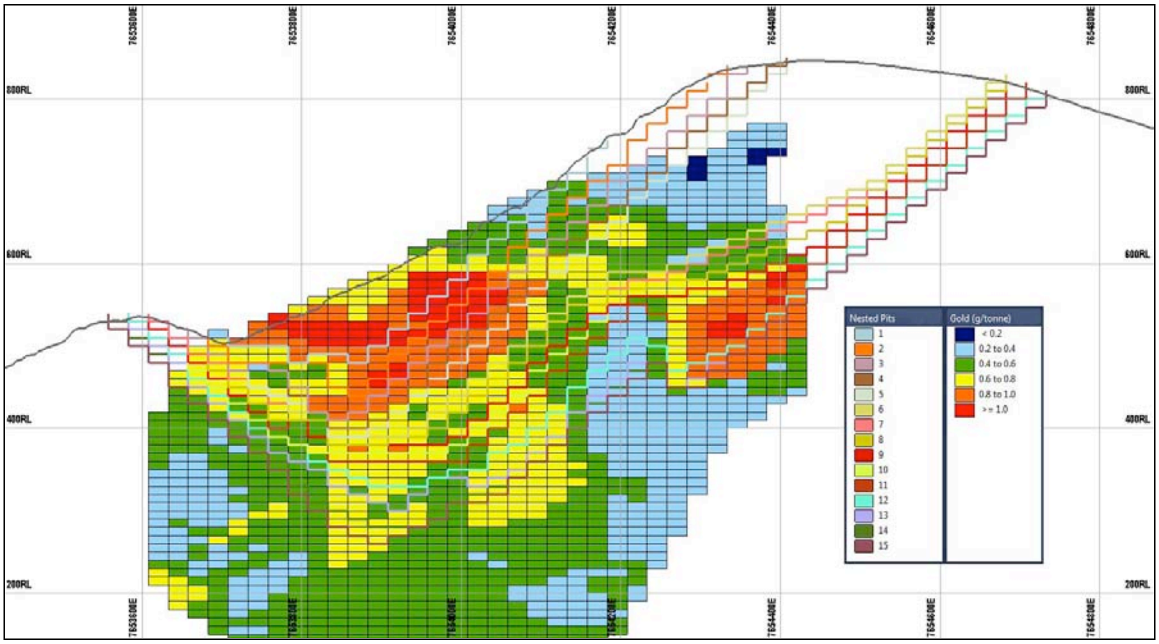
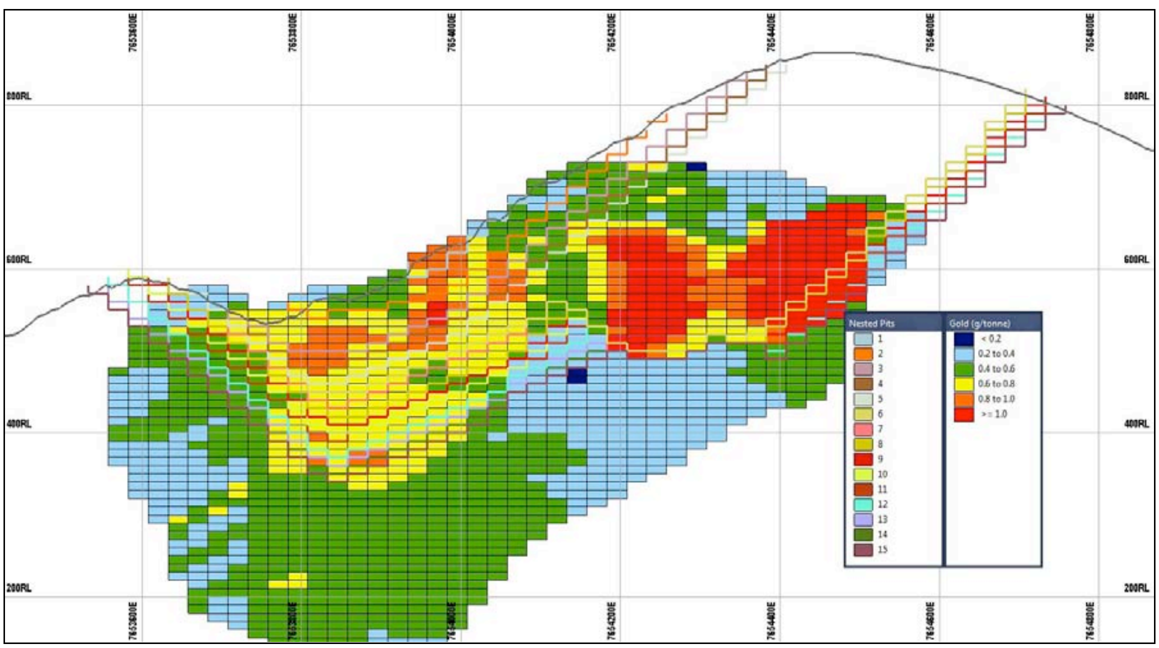


Figure 16.2: Sections through optimum pit shells with block grades (Au equivalent), preliminary work

Section 4,595,285 North, Facing North



Section 4,595,145 North, Facing North



16.5 GOLD EQUIVALENCY

Using the supplied parameters, one percent copper is equivalent to 1.55 g/tonne gold.

1% Copper Grade	1 g/tonne (Gold Grade)
2200 lb/tonne	31.1 g/oz
22 lb Copper in Feed	0.0322 oz of Gold in Feed
85% Copper Recovery	90% Gold Recovery
18.7 lb Copper Recovered	0.0289 oz of Gold Recovered
$3 Revenue per lb	$1,250 Revenue per oz
$56.10 Revenue for 1% in Feed	$36.17 Revenue for 1 g/tonne in Feed

1% Copper = 1.55 g/tonne gold.

16.6 DILUTING GRADE

Preliminary analysis indicates that the break-even operating cut-off grade was approximately 0.3 g/tonne Au-Equivalent. Mineralised blocks below cut-off had an average grade of 0.17 g/tonne gold and 0.05% copper. The majority of the dilution contained within the deposit would be internal dilution at approximately those grades.

For conservatism, no diluting grades were applied during pit optimisation. However, they were employed later during production scheduling and economic analysis.

16.7 CUT-OFF GRADE

The *in situ* block cut-off grades were calculated for oxide and primary/transitional mill feed as 0.21 and 0.25 g/tonne of gold or gold-equivalent, respectively (Table 16.3). Considering the need for some profit to be realised, those grades were increased to 0.23 and 0.27 g/tonne, respectively.

The oxide requires stockpiling and re-handling, so the *in situ* oxide cut-off was increased to 0.25 g/tonne. To account for a longer than initially-expected mill feed haul distance, the primary/transitional cut-off was increased to 0.30 g/tonne.

Table 16.3 Cut-off grade determination.

	Oxide	Primary / Transitional	Comments
Milling Cost per Diluted Tonne Milled	$ 6.00	$ 7.00	
Milling Cost per Tonne In Situ	$ 6.90	$ 8.05	
Revenue per Gram per Tonne In Situ			
Item	Value	Value	
Mining Recovery	95%	95%	5% mining losses
Mill Recovery	90%	90%	90% milling recovery
Smelter Return	95%	95%	5% selling and smelter costs
Grams of Gold Returned	0.81	0.81	revenue from gold net of smelter
Revenue	$ 32.60	$ 32.60	$1250 gold price.
Milling Cut-off Grade (g/tonne)	0.21	0.25	Milling Cost Per In Situ Tonne / Revenue per In Situ Gram Per Tonne
Cut-off Including Profit (g/tonne)	0.23	0.27	Considers 10% profit
Selected Cut-off (g/tonne)	0.25	0.30	Considers oxide re-handling and longer haul distances.

16.8 Detailed Pit Design (Pit "De-Optimisation") and Production Scheduling

Based on the results of pit optimisation, the following refinements to the governing parameters were made (refer to Table 16.4).

Table 16.4 Production scheduling parameters.

Parameter	Value
Milling Rate	10 million Tonnes per Year Fresh Rock (Oxide to be Stockpiled for Milling at End of Mine Life)
Diluting Grade*	Gold 0.17 g/tonne Copper 0.05%

* Refer to Section 16.6.

16.9 Pre-Production Stripping

It was desired to keep pre-production stripping to a minimum. With the small starter pit, pre-production stripping could be as little as 300,000 tonnes. Pre-production stripping and initial mining would concentrate on a near-surface pod of higher-grade material (refer to Figure 16.4).

16.10 Production Stripping

The waste stripping schedule was brought forward in order to provide sufficient material for constructing the tailings dam (refer to Table 16.7).

16.11 Pit Phases

The 21-year mine life of the pit was subdivided into four phases. Table 16.5 shows the timing of the phases and the nested pits to which they correspond. The phases were designed to balance:

- early capital payback;
- operational constraints;
- overall profitability; and,
- a reasonable mine life.

Table 16.6 shows the totals for mill feed and in-pit Inferred mineral resources. Table 16.7 illustrates the proposed mining and milling schedule. The starter and final pits are shown in Figure 16.3 and Figure 16.4, respectively. Detailed illustrations of each phase are located in an Appendix to this report.

Table 16.5 Pit phases by year.

Phase	Nested Pit*	Year(s)	Bottom Elevation	Description
1	1	1-2	480 m	Starter Pit
2	2	2-3	440 m	Expansion of Starter Pit
3	6	3-9	400 m	Pushback and Deepening
4	15	9-21	260 m	Pushback and Deepening
Oxide Stockpile Milling		21-23		

Note: The detailed, de-optimised design *closely* follows the nested pit.

Figure 16.3 Starter pit (Phase 1).

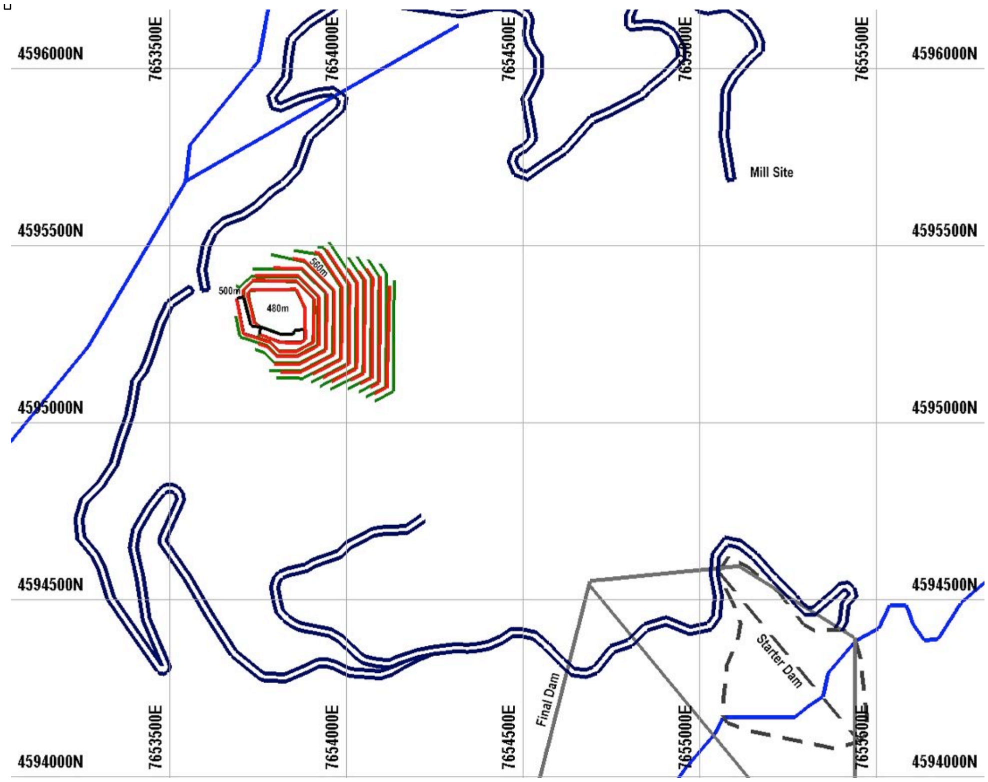


Figure 16.4 Final pit (Phase 4).

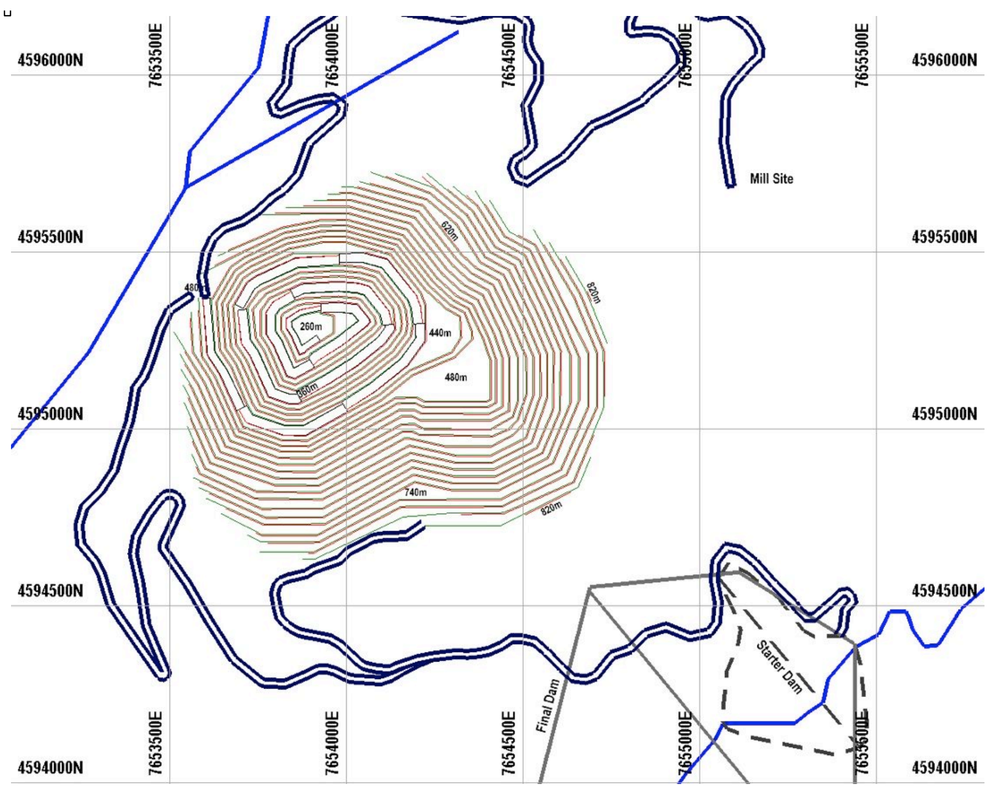


Table 16.6 Potential mill feed and in-pit Inferred mineral resources.

Potential Mill Feed (Consists of In-Pit Measured and Indicated Mineral Resources)

Phase	Feed Type	Cut-off Grade (Au-Eq, g/tonne)	Non-Diluted Volume (m^3, Millions)	Non-Diluted Tonnes (Millions)	SG	Diluted Tonnes (Millions) [2.]	In Situ Grade, Gold (g/tonne)	Diluted Grade, Gold (g/tonne) [1.]	In Situ Grade, Copper	Diluted Grade, Copper [1.]	Destination
Phase 1	Oxide	0.25	0.95	2.19	2.30	2.30	0.41	0.40	0.17%	0.16%	Stockpile
Phase 1	Primary	0.30	5.37	12.71	2.37	13.35	0.43	0.42	0.26%	0.25%	Mill
Phase 2	Oxide	0.25	1.28	2.95	2.30	3.09	0.37	0.36	0.06%	0.06%	Stockpile
Phase 2	Primary	0.30	6.24	15.30	2.45	16.07	0.43	0.42	0.24%	0.23%	Mill
Phase 3	Oxide	0.25	3.07	7.05	2.30	7.40	0.34	0.33	0.06%	0.06%	Stockpile
Phase 3	Primary	0.30	19.84	48.97	2.47	51.42	0.40	0.39	0.22%	0.21%	Mill
Phase 4	Oxide	0.25	1.41	3.24	2.30	3.41	0.26	0.26	0.06%	0.06%	Stockpile
Phase 4	Primary	0.30	48.81	121.73	2.49	127.82	0.31	0.30	0.19%	0.18%	Mill
Total			86.97	214.14	2.46	224.85	0.35	0.34	0.20%	0.19%	

Notes:

1. Includes dilution (5%) and mining recovery (95%). Diluting grades comprise 0.17 g/tonne gold and 0.05% copper.
2. Includes dilution (5%).

Undiluted In-Pit Inferred Mineral Resources (Considered as Waste)

Phase	Feed Type	Cut-off Grade (Au-Eq, g/tonne)	Non-Diluted Volume (m^3, Millions)	Non-Diluted Tonnes (Millions)	SG	In Situ Grade, Gold (g/tonne)	In Situ Grade, Copper	Destination
Phase 1	Oxide	0.25	0.00	0.01	2.30	0.32	0.09%	Waste
Phase 1	Primary	0.30	0.03	0.07	2.30	0.24	0.30%	Waste
Phase 2	Oxide	0.25	0.01	0.02	2.30	0.33	0.10%	Waste
Phase 2	Primary	0.30	0.03	0.08	2.30	0.34	0.14%	Waste
Phase 3	Oxide	0.25	0.13	0.31	2.30	0.31	0.06%	Waste
Phase 3	Primary	0.30	1.67	4.10	2.46	0.43	0.26%	Waste
Phase 4	Oxide	0.25	0.78	1.81	2.30	0.28	0.05%	Waste
Phase 4	Primary	0.30	4.47	11.08	2.48	0.30	0.20%	Waste
Total			7.13	17.48	2.45	0.33	0.20%	

Table 16.7 Mining and milling schedule.

Phase	Type	Diluted Tonnes (Millions)[1]	Diluted Grade, Gold (g/tonne)[2]	Diluted Grade, Copper[2]	Pre-Production	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	*Total*
Primary/Transitional Mining & Milling																													
Phase 1	Primary	13.35	0.42	0.25%		10.00	3.35	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	***13.35***
Phase 2	Primary	16.07	0.42	0.23%		-	6.65	9.41	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	***16.07***
Phase 3	Primary	51.42	0.39	0.21%		-	-	0.59	10.00	10.00	10.00	10.00	10.00	0.83	-	-	-	-	-	-	-	-	-	-	-	-	-	-	***51.42***
Phase 4	Primary	127.82	0.30	0.18%		-	-	-	-	-	-	-	-	9.17	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	8.65	-	-	***127.82***
Oxide Mining & Stockpiling																													
Phase 1	Oxide	2.30	0.40			1.72	0.58	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	***2.30***
Phase 2	Oxide	3.09	0.36			-	1.28	1.81	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	***3.09***
Phase 3	Oxide	7.40	0.33			-	-	0.08	1.44	1.44	1.44	1.44	1.44	0.12	-	-	-	-	-	-	-	-	-	-	-	-	-	-	***7.40***
Phase 4	Oxide	3.41	0.26			-	-	-	-	-	-	-	-	0.24	0.27	0.27	0.27	0.27	0.27	0.27	0.27	0.27	0.27	0.27	0.27	0.23	-	-	***3.41***
Oxide Stockpile Milling		16.20	0.33			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.35	10.00	4.85	***16.20***
Total Mining: Primary, Transitional, & Oxide (Millions of Tonnes)						11.7	11.9	11.9	11.4	11.4	11.4	11.4	11.4	10.4	10.3	10.3	10.3	10.3	10.3	10.3	10.3	10.3	10.3	10.3	10.3	8.9	-	-	***225***
Tonnes Primary Mill Feed						10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	8.65	-	-	***209***
	Mill Feed Gold Ounces (000s)					133.7	133.6	132.9	124.1	124.1	124.1	124.1	124.1	99.6	97.4	97.4	97.4	97.4	97.4	97.4	97.4	97.4	97.4	97.4	97.4	84.2	-	-	***2,276***
	Mill Feed Gold Grade (g/tonne)					0.42	0.42	0.41	0.39	0.39	0.39	0.39	0.39	0.31	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	-	-	***0.34***
	Mill Feed Copper Pounds (Millions)					54.4	51.3	49.5	47.2	47.2	47.2	47.2	47.2	40.5	39.9	39.9	39.9	39.9	39.9	39.9	39.9	39.9	39.9	39.9	39.9	34.5	-	-	***905***
	Mill Feed Copper Grade					0.25%	0.23%	0.23%	0.21%	0.21%	0.21%	0.21%	0.21%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.00%	0.00%	***0.20%***
Tonnes Oxide Mill Feed						-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.35	10.00	4.85	***16***
	Mill Feed Gold Ounces (000s)					-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	14.4	106.1	51.4	***172***
	Mill Feed Gold Grade (g/tonne)					-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.33	0.33	0.33	***0.33***
Total Tonnes Mill Feed						10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	4.85	***225***
	Mill Feed Gold Ounces (000s)					133.7	133.6	132.9	124.1	124.1	124.1	124.1	124.1	99.6	97.4	97.4	97.4	97.4	97.4	97.4	97.4	97.4	97.4	97.4	97.4	98.6	106.1	51.4	***2,448***
	Mill Feed Gold Grade (g/tonne)					0.42	0.42	0.41	0.39	0.39	0.39	0.39	0.39	0.31	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.31	0.33	0.33	***0.34***
	Mill Feed Copper Pounds (Millions)					54.4	51.3	49.5	47.2	47.2	47.2	47.2	47.2	40.5	39.9	39.9	39.9	39.9	39.9	39.9	39.9	39.9	39.9	39.9	39.9	34.5	-	-	***905***
	Mill Feed Copper Grade					0.25%	0.23%	0.23%	0.21%	0.21%	0.21%	0.21%	0.21%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.16%	0.00%	0.00%	***0.20%***
Waste Stripping (Millions of Tonnes)[3]					10	11	11	10	10	10	10	10	10	10	10	8	8	6	6	6	6	6	5	1					***164***
	Cumulative				10	21	32	42	52	62	72	82	92	102	112	120	128	134	140	146	152	158	163	164	164	164	164	164	***164***
Waste Taken From Phase					1, 4	1, 2, 4	2, 4	3, 4	3, 4	3, 4	3, 4	3, 4	3, 4	4	4	4	4	4	4	4	4	4	4	4	4	4	5	6	
Stripping Ratio						1.1:1	1.1:1	1:1	1:1	1:1	1:1	1:1	1:1	1:1	1:1	0.8:1	0.8:1	0.6:1	0.6:1	0.6:1	0.6:1	0.6:1	0.5:1	0.1:1	0:1	0:1	0:1	0:1	***0.7:1***

Notes:
1. Includes dilution (5%).
2. Includes dilution (5%) and mining recovery (95%). Diluting grades comprise 0.17 g/tonne gold and 0.05% copper.
3. Includes in-pit Inferred mineral resources.

16.12 Equipment Fleet Selection

Euromax expressed a desire to use 90-tonne haul trucks. To determine the number of trucks that would be needed, haul roads were designed (refer to Figure 16.4) and cycle times for each phase were estimated.

16.12.1 Ore Haulage Distance to Mine Mouth

The straight line, along gradient, average distance from each ore block to the mine mouth was calculated for each block. The mine mouth coordinates are 7,653,660 m East, 4,595,360 m North, 480 m RL. The average gradient for the straight line distance was also calculated. For each Phase, an average distance and slope gradient were calculated.

For Phase 1, the average haul distance was 1.2 kilometres (refer to Table 16.8). The average slope gradient was downhill at the maximum in-pit haul road gradient of +/- 8% (refer to Figure 16.6).

Table 16.8 Average ore haulage distance by Phase.

Phase	Average Ore Haulage Distance (m)	Average Distance Plus 25%*	Average Slope Gradient
1	1,200	1,500	-8%
2	1,100	1,400	-5%
3	1,600	2,000	-7%
4	1,000	1,300	0

** To account for curves - rounded up to the nearest 100.*

Figure 16.5: Histogram of haul distances (in metres), Phase 1.

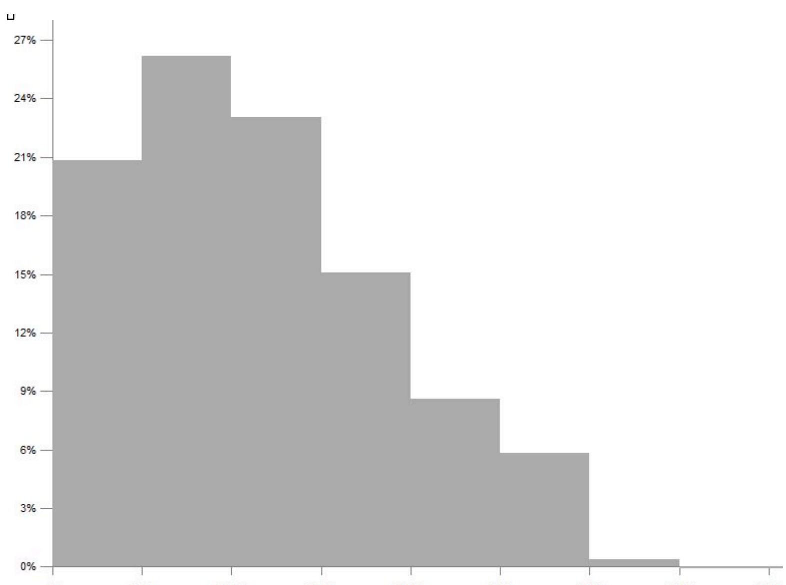


Figure 16.6: Average slope gradient for Phase 1, in percent.

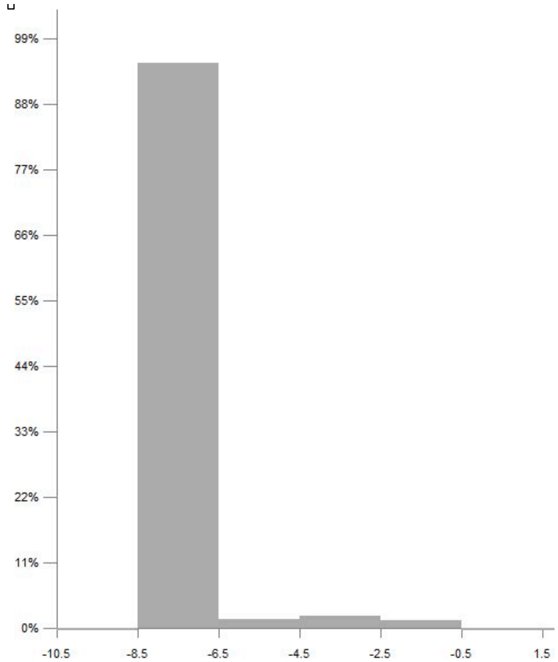


Table 16.9: Average waste haulage distance by Phase.

Phase	Average Waste Block Northing (m)	Average Waste Block Elevation (m)	Average Horizontal Waste Haulage Distance (m)[1.]	Distance at +8% to Top of Hill (m)	Distance at -10% to Dam Area (m)
1	4,595,360	580	1,200	2,400	1,300
2	4,595,260	640	1,000	1,100	1,300
3	4,595,130	745	1,400	0	1,500
4	4,595,120	650	900	1,100	1,300
Average of 3&4	4,595,125	700	1,150	550	1,400

1. To main waste haulage road located on south side of pit.

16.12.2 Waste Haulage Distance to the South Side of Pit

Figure 16.7: Histogram of waste block elevations for Phase 4.

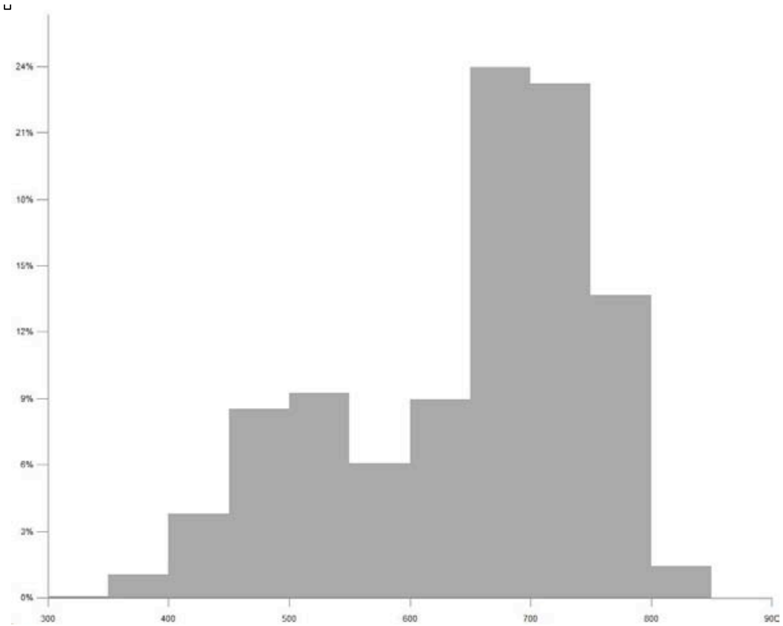
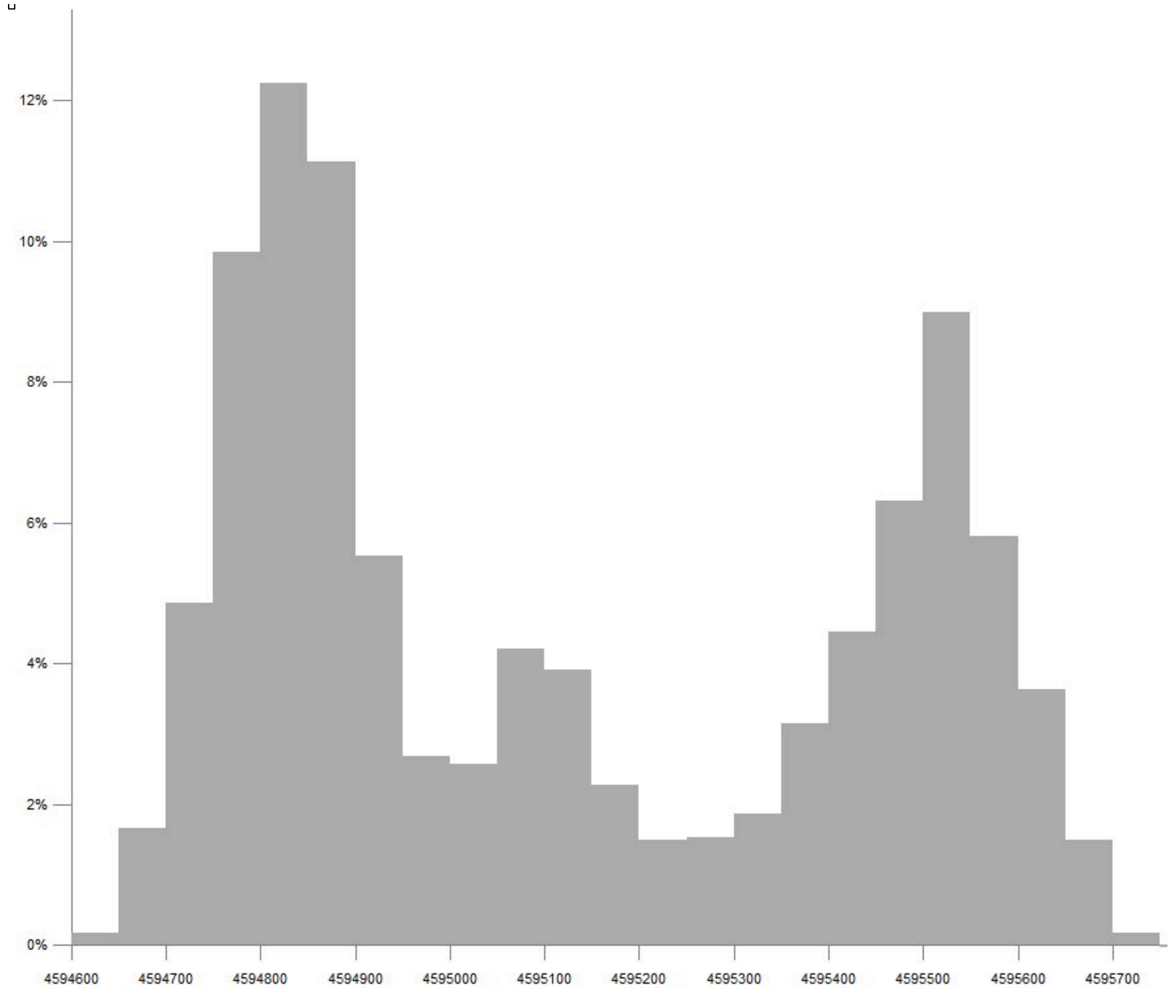


Figure 16.8: Histogram of waste block northings for Phase 4.



16.13 Mobile Equipment Fleet

Based on production requirements and the average haul distances from Sections 16.12.1 and 16.12.2, and the assumption from Table 16.10 , a mobile equipment fleet was selected (Table 16.11).

Table 16.10 Equipment selection parameters.

Truck		CAT 777
Truck Capacity (tonnes)		90
Loader		CAT 990
Loader Capacity (tonnes)		15
Truck Speed Downhill:		
Empty (km/hr)[3.]		40
Loaded (km/hr)[3.]		25
Truck Speed Uphill (8%):		
Empty (km/hr)[3.]		30
Loaded (km/hr)[3.]		13
Speed, Loaded, Uphill, 2%		30
Overall Availability		75%
Working Hours per Day[7.]		21
Working Days per Year		350
Fuel Price	$	1.21
Exchange Rate (USD/CAD)	$	0.90
Explosives & Accessories		
Ore (per tonne)	$	0.32
Waste (per tonne)	$	0.32

Table 16.11 Mobile equipment fleet.

Item	Description	Number
CAT 777 Truck	90 tonne Capacity	8 Preproduction 13 Years 1-6 11-12 Years 7-11 10 Years 12-21 5 Years 22-23
CAT 990 Loader	15 tonne Bucket Capacity	1 Pre-production 3 Years 1+
CAT 375 Excavator	75 tonne	1
CAT 345 Excavator	45 tonne	1
CAT D10 Bulldozer	430 kWatt (580 HP), Waste Pile	1
CAT D8 Bulldozer	300 kWatt (405 HP), Pit Work	2
CAT 770 Water Truck	35 tonne Capacity	1
CAT 24 Motor Grader	400 kWatt (530 HP), 7.3 m Blade	1
CAT 16 Motor Grader	220 kWatt (300 HP), 4.9 m Blade	1
Sandvik D75 Drill	Up to 280 mm Hole, Production	2
Sandvik DX800 Drill	75-125 mm Hole, Road Work / Secondary Blasting	1
CAT 825 Compactor	260 kWatt (350 HP), Dam Compaction	1
ANFO Prill Truck	20 tonne	1
Boom Truck	5 tonne	2
Lube/Fuel Truck		1
Man Bus		2

Table 16.12 Equipment replacement schedule.

Item	Life (Years)
CAT 777 Truck	6
CAT 990 Loader	3
CAT 375 Excavator	3
CAT 345 Excavator	3
CAT D10 Bulldozer	3
CAT D8 Bulldozer	3
CAT 770 Water Truck	6
CAT 24 Motor Grader	6
CAT 16 Motor Grader	6
Sandvik D75 Drill	3
Sandvik DX800 Drill	3
CAT 825 Compactor	3
ANFO Prill Truck	6
Boom Truck	6
Lube/Fuel Truck	6
Man Bus	6
Pick Ups (Toyota double cab)	6
Lights	6
Pumps	6

Based on the mobile fleet requirements by year from Table 16.11 and the replacement schedule from Table 16.12, an equipment purchasing schedule was constructed (Table 16.13).

Table 16.13 Equipment purchasing schedule.

		Preproduction -2	Preproduction -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19
CAT 777 truck	90 tonne capacity	4	4	5	0	0	0	4	4	4	0	0	0	3	3	4	0	0	0	4	4	2
CAT 990 Loader	15 tonne bucket capacity	1	2	0	1	2	0	1	2	0	1	2	0	1	2	0	1	2	0	1	2	0
CAT 375Excavator	75 tonne	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0
CAT 345 Excavator	45 tonne	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0
CAT D10 Bulldozer	430 kWatt Waste Pile	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0
CAT D8 Bulldozer	300kWatt Pit Work	1	1	0	1	1	0	1	1	0	1	1	0	1	1	0	1	1	0	1	1	0
CAT 770 Water Truck	35 tonne capacity	1	0	0	0	0	0	1	0	0	0	0	0	1	0	0	0	0	0	1	0	0
CAT 24 Motor Grader	400kWatt 7.3m Blade	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0
CAT 16 Motor Grader	220kWatt 4.9m Blade	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0
Sandvik D75 Drill	Up to 280mm Hole, Production	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0
Sandkvic DX800 Drill	75-125mm Hole, Road work, secondary bla	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0
CAT 825 Compactor	260 kWatt, dam compaction	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0	1	0	0
ANFO Prill Truck	20 tonne	1	0	0	0	0	0	1	0	0	0	0	0	1	0	0	0	0	0	1	0	0
Boom Truck	5 tonne	1	0	0	0	0	0	1	0	0	0	0	0	1	0	0	0	0	0	1	0	0
Lube/Fuel Truck		1	0	0	0	0	0	1	0	0	0	0	0	1	0	0	0	0	0	1	0	0
Man Bus		1	0	0	0	0	0	1	0	0	0	0	0	1	0	0	0	0	0	1	0	0
Pick Ups	Toyota HiLux double cabs	2	5	0	0	0	0	2	5	0	0	0	0	2	5	0	0	0	0	2	5	0
Lights	Self contained units c/w genset	2	3	0	0	0	0	2	3	0	0	0	0	2	3	0	0	0	0	2	3	0
Pumps	Godwin or Pleuger submersibles	2	3	0	0	0	0	2	3	0	0	0	0	2	3	0	0	0	0	2	3	0

Equipment capital and operating costs were developed based on manufacturers' quotations and published data (Table 16.14 and Table 16.15). The most expensive piece of equipment to operate is the 90 tonne haul truck, mainly because of the fuel it consumes and the wear-and-tear from near-continuous operation.

Table 16.14 Equipment capital costs.

Unit	Capital Cost US$	Source
CAT 777 truck or Equivalent	$1.22 Million	Komatsu
CAT 990 Loader or Equivalent	$1.40 Million	Caterpillar
CAT 375 Excavator or Equivalent	$650,000	Caterpillar
CAT 345 Excavator or Equivalent	$425,000	Caterpillar
CAT D10 Bulldozer or Equivalent	$1.12 Million	Komatsu
CAT D8 Bulldozer or Equivalent	$550,000	Komatsu
CAT 770 Water Truck or Equivalent	$870,000	Published Data
CAT 24 Motor Grader or Equivalent	$2.45 Million	Caterpillar
CAT 16 Motor Grader or Equivalent	$590,000	Komatsu
Sandvik D75 Drill or Equivalent	$1.50 Million	Sandvik
Sandvik DX800 Drill or Equivalent	$600,000	Sandvik
CAT 825 Compactor or Equivalent	$150,000	Caterpillar
ANFO Prill Truck	$430,000	Published Data
Boom Truck	$185,000	Published Data
Lube/Fuel Truck	$94,000	Published Data
Man Bus	$75,000	Published Data
Pick Ups	$25,000	Published Data
Lights	$26,242	Dealer Quotation
Pumps	$14,228	Dealer Quotation

Table 16.15 Equipment operating costs. US$

Unit	Fuel Consumption (L/hr)	Fuel Cost (per hour)	Wear Items (per hour)	Planned Repair & Maintenance (per hour)	Labour (per hour)	Total Hourly Cost
CAT 777 Truck	75	$ 91	$ -	$ 101	$ 7	$ 199
CAT 990 Loader	71	$ 86	$ 1	$ 95	$ 7	$ 189
CAT 375 Excavator	55	$ 67	$ 4	$ 27	$ 7	$ 105
CAT 345 Excavator	44	$ 53	$ 3	$ 20	$ 7	$ 83
CAT D10 Bulldozer	80	$ 97	$ 16	$ 23	$ 7	$ 143
CAT D8 Bulldozer	41	$ 50	$ 16	$ 20	$ 7	$ 93
CAT 770 Water Truck	41	$ 50	$ -	$ 56	$ 7	$ 113
CAT 24 Motor Grader	50	$ 61	$ 2	$ 52	$ 7	$ 122
CAT 16 Motor Grader	34	$ 41	$ 1	$ 23	$ 7	$ 73
Sandvik D75 Drill	90	$ 108	$ 17	$ 48	$ 7	$ 180
Sandvik DX800 Drill	43	$ 52	$ 10	$ 44	$ 7	$ 112
CAT 825 Compactor	67	$ 81	$ 16	$ 20	$ 7	$ 124
ANFO Prill Truck	24	$ 29	$ -	$ 4	$ 7	$ 40
Boom Truck	13	$ 15	$ -	$ 13	$ 7	$ 35
Lube/Fuel Truck	10	$ 12	$ -	$ 3	$ 7	$ 22
Man Bus	10	$ 12	$ -	$ 3	$ 7	$ 22
Pick-Up	5	$ 6		$ 2		$ 8
Lighting Plant	2	$ 2		$ 1		$ 3

16.14 Comparison Between Waste Disposal Options

After the tailings dam is complete, there would be 40 million tonnes of waste left over. There are two main disposal options: (1) use the waste as tailings dam ballast, or (2) construct a waste pile west of the pit.

Based on the road designs shown in Figure 16.9, the return travel times for each option were estimated (refer to Table 16.16). The waste pile option was 50% longer than the tailings ballast option. Based on this, and the following factors, it was determined that trucking the remaining waste to the tailings pond would easily be the more economical option than constructing a separate waste pile.

1. The waste pile would need runoff water collection ditches and settling ponds.
2. A river crossing is required for the road to the waste pile.
3. Additional, significant reclamation costs would be incurred for the waste pile.
4. Adding the additional waste rock to the dam's ballast increases its mass and decreases its slope, which increases the dam's safety factor.

Figure 16.9: Comparison of waste disposal options.

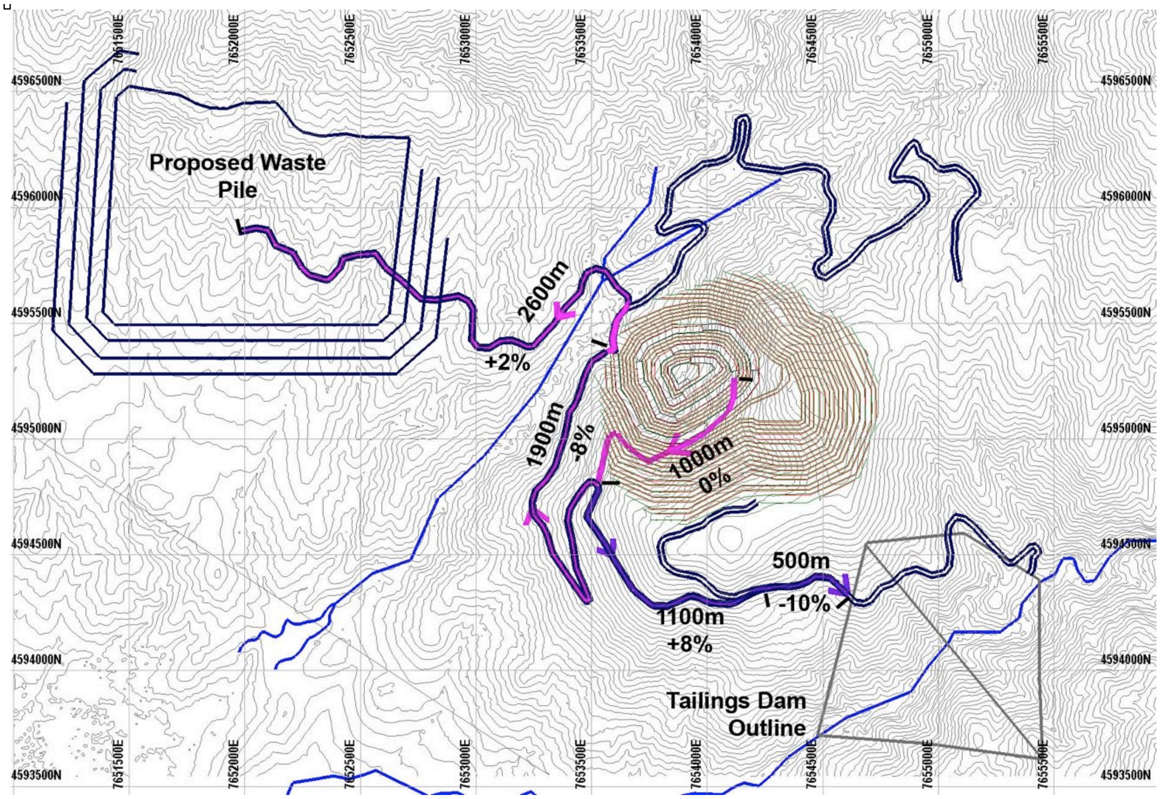


Table 16.16 Return travel times for waste disposal options.

Distances

Destination	Metres (One Way)
To Waste Pile	
To Pit Mouth (-8%)	1,900
To Waste Pile (+2%)	2,600
To Tailings Dam Ballast	
Up to Top of Hill (+8%)	1,100
Down to Dam (-10%)	500

Average Return Travel Times (min)

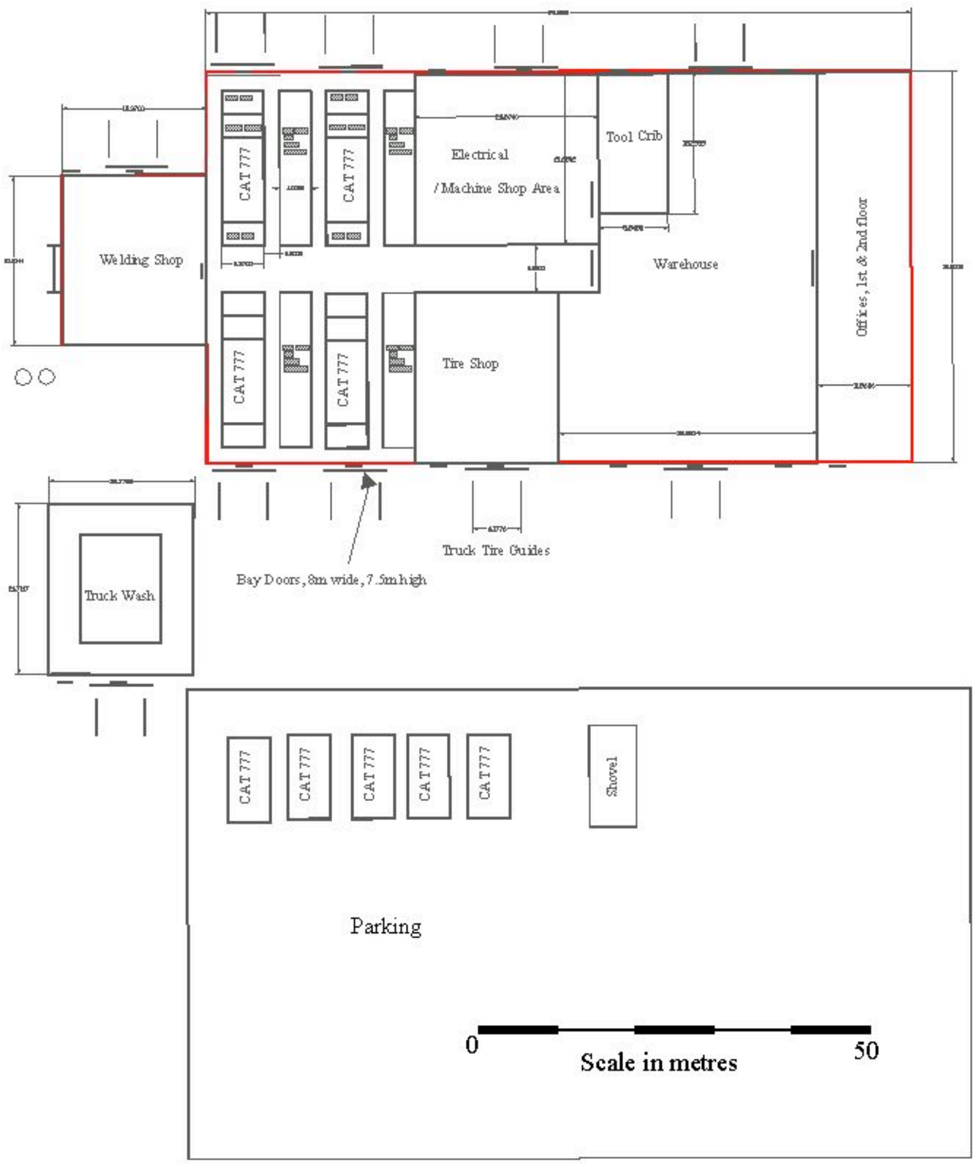
Destination	Minutes (Return)
To Waste Pile	
To Pit Mouth (-8%)	13.3
To Waste Pile (+2%)	14.0
Total for Waste Pile	27.3
To Tailings Dam Ballast	
Up to Top of Hill (+8%)	11.6
Down to Dam (-10%)	7.1
Total for Tailings Dam	18.7

16.15 Truck Workshop

A truck workshop was designed that would incorporate four bays, each sized for the largest piece of mobile equipment: the 90 tonne truck (refer to Figure 16.10). The shop would also have space for an electrical and machine shop, a tyre shop, a tool crib, a warehouse, a dry (changing and showering facility), a welding shop, and office space.

The building would have an area of 3,600 square metres. A Quantity Surveyor estimated the total construction cost at $US 9.7 million, exclusive of contingency and sales tax.

Figure 16.10 Preliminary truck workshop design.



17.0 RECOVERY METHODS

17.1 SUMMARY

The process plant will be constructed for a 10 million tonnes per annum (Mt/a) capacity based on a flowsheet that produces a saleable copper concentrate and maximises the overall copper and gold recovery. The Ilovitza ore derived from a porphyry copper gold deposit is moderately hard, and is amenable to both flotation and cyanidation. The process flowsheet has been developed based on the test work reported in Section 13.0 with the objective of producing a saleable copper concentrate and maximising the gold recovery.

The run of mine (ROM) ore will be crushed by a gyratory crusher and then ground in a two-stage (semi autogenous grinding [SAG] mill and ball mill) conventional milling circuit in order to produce slurry with an optimum size distribution for flotation and leaching. The ground slurry, with a particle size of P80 = 75 µm, is fed into flotation to produce a saleable copper concentrate. The copper concentrate at an expected copper grade of 24% is dewatered in the concentrate thickener and filter and shipped for smelting.

The flotation tails are fed into a pre-leach thickener and the thickener underflow will then be pumped through Carbon in Leach (CIL) tanks. Flotation tailings slurry will be leached in 16 CIL tanks which utilise cyanide leaching and recovery of the dissolved precious metals onto activated carbon. The carbon is then pressure stripped with a hot caustic solution to elute the precious metals into a pregnant solution which, in turn is treated by conventional electrowinning to produce a gold sludge that is suitable for direct smelting on site.

Tailings from the process plant will be pumped to the Tailings Management Facility (TMF).

The installed plant capital cost of the proposed process plant is estimated at US $249.6 million ±25%.

The total process plant operating cost has been estimated based on the process design work and the reagent consumptions estimated based on the prefeasibility study test work results. The estimated process plant operating cost is US $6.50 per tonne (/t) within ±25%.

The overall copper and gold recoveries are estimated at 84% and 88% respectively.

17.2 PROCESS DESIGN CRITERIA

The process design criteria have been derived from the following sources of. information:

- Test work results as reported in Section 13.0
- Mill throughput of 10,000,000 t/a as defined by Euromax.
- Information published in the public domain, industry standard assumptions, and knowledge gained from similar projects and/or unit operations.
- Equipment manufacturer's recommendations.

- Previous studies completed on the project.

Table 17.1 lists a summary of the principle process design criteria established for the project.

Table 17.1 Process Design Criteria

PROCESS DESIGN CRITERIA			
DESCRIPTION	UNIT	VALUE	SOURCE
GENERAL			
Type of Deposit		Porphyry Copper Gold - Sulphides	N/A
Economic Metals		Gold, Copper	8
Annual Processing Rate, Overall	t/a (dry)	10,000,000	1
PLANT OPERATING SCHEDULE			
Shifts /d		2	6
Hours /Shift	h	12	6
Hours /d	h	24	6
Days /a	Days	365	6
AVAILABILITY / UTILISATION			
Crushing Plant Utilisation	%	85	6
Daily Processing Rate	t/d	32,232	2
Processing Rate, Operating	t/h	1,343	2
Milling Plant Utilisation	%	96	6
Milling Plant Availability	%	95	
Daily Processing Rate	t/d	30,041	2
Processing Rate, Operating	t/h	1,252	2
ORE CHARACTERISTICS			
Ore Specific Gravity	t/m^3	2.95	7
Ore Bulk Density	t/m^3	1.6	6
Ore Moisture Content	%	5	6
Abrasion Index A_i - Bond	g	0.115	4
Bond Rod Mill Work Index	kWh/t	15.1	4
Bond Ball Mill Work Index	kWh/t	15.9	4
SMC Test Coarse Size Work Index (Mia)	kWh/t	12.7	4
Head Grade - Copper	%	0.23%	5
Head Grade - Gold	g/t	0.35	5
METALLURGICAL RECOVERY & METAL PRODUCTION			
Copper Recovery	%	84%	4
Gold Recovery	%	88%	4
Concentrate Copper Grade	%	24%	4
Concentrate Gold Grade	g/t	26.8	4

Table continues...

Copper Production	lb/a	42,593,316	2
Gold Production	oz/a	98,837	2
CRUSHING			
ROM discharge bins		400m³/640 tonnes live capacity	7
No. of ROM Bins		1	2
Crusher Type	Primary Gyratory	Metso Superior MK-II 42-65 or Equivalent	9
Number of Crushers		1	2
Operating Shifts /d	Shift /d	2	6
Operating Hours /Shift	h /shift	12	6
Operating days /week	d/Week	7	6
Crusher Operating Time	%, overall	85%	7
Crusher Operating Rate (Design)	t/h	1,343	2
Crusher Operating Rate (Design)	m³/h	839	2
Crusher Feed Particle Size F80	mm	416	6
Crusher Product Particle Size, P80	mm	150	6
Reduction Ratio		2.78	2
Open Side Setting	mm	150	9
Crusher Discharge onto Apron Feeder		Primary Crusher Discharge Apron feeder rated at 2,000 t/h	7
No. of Discharge Conveyors		1	6
Product size on discharge conveyor (design) P80	mm	150	6
COARSE ORE STOCKPILE			
Crushed Ore Stockpile (Live Capacity)	t	60,082	6,7
Crushed Ore Bulk Density	t/m³	1.6	7
Angle of Repose	degrees	35	7
Angle of Reclaim	degrees	60	7
Reclaim method		Sub terrainian Apron Feeders	7
No. of Feeders		4	6
Average Tonnage Rate, Operating	t/h	1,252	3
Average Tonnage Rate (Design Each)	t/h	313	3
GRINDING			
Primary Mill Type		SAG	
Mill Size D x EGL	m	32 ft x 14 ft	2
Number of Trains		1	6
Number of Mills per Train		1	6
Total Number of Mills		1	6
Average Fresh Feed Tonnage Rate	t/h	1,252	3
Solids /Pulp Density	%	70	2,3,7
Top feed Size	mm	250	6

Table continues…

Feed Size, F80	mm	150	6,7
Product Size, P80	mm	1.5	6,7
Mill Recycle Load	%	25	6,7
Mill Speed	% CS	75	7
Mill Ball Charge	% VB	12	7
Load Volume	% V	24	7
Discharge Trommal Aperture	mm	12	6
Pebble Port Aperture	mm	50	6
Pebble Crusher		Cone HP800	6
Operating Shifts /d	Shift /d	2	6
Operating Hours /Shift	h /shift	12	6
Operating days /week	d /Week	7	6
Crusher Operating Time	%, overall	95%	6
Mill discharge pebble generation	%	25	6
Required Processing Rate, Operating	t/h	313	2,3
Total Volumetric Process Rate (Design)	m^3/h	196	2
Feed F80	mm	50	6
Product Particle Size, P80	mm	13	2,3,7
Reduction Ratio		3.85	2
Closed Side Setting	mm	13	2,3,7
Secondary Mill Type		Ball Mill - Overflow Discharge	7
Mill Size D x L	m	22 ft x 32 ft	2,6
No. of Trains		2	6
No. of Mills per Train		1	2,3
Total No. of Mills		2	6
Average Fresh Feed Tonnage Rate (Each)	t/h	626	3
Solids /Pulp Density	%	70	2,3
Top feed Size	μm	2,000	6,7
Feed Size, F80	μm	1,500	6,7
Product Size, P80	μm	75	6
Ball Mill Recycle Load	%	250	2,3
Mill Speed	% CS	75	7
Mill Ball Charge	% VB	35	7
Classification		Cyclones	7
No. of Operational Cyclones per Mill		8	2
No. of Cyclones per Mill		10	6
Total No. of Cyclones		20	2
Cyclone Size Diameter	mm	914	2
Product size P80	μm	75	2

Table continues…

DEWATERING CYCLONES			
No. of Trains		2	6
No. of Operational Cyclones per Mill		42	2
No. of Cyclones per Mill		48	6
Total No. of Cyclones		96	2
Cyclone Size Diameter	mm	254	2
Product size P80	µm	10	2
COPPER ROUGHER FLOTATION			
Configuration		Rougher / Scavenger	7
No. of Trains		1	
Flotation Cell	Type	Metso RCS / OK Equivalent	6
Feed Rate/train	t/h	1,252	2
Required Bank Volume	m^3/h	2,725	2
No. of Cells		18	2
Cell Volume	m^3	158.6	2
Masspull	%	1.35	4
CONCENTRATE REGRINDING			
No. of trains		1	6
Regrind Mill Type		Verti /SMD Mill	6
No. of mills/train		1	2
Total No. of Mills		1	2
Circuit Type		Open	7
Feed Solids	%	70	2
Motor Power	kW	355	2
Mill Size		SMD 355	2
Mill Product Size (P_{80})	µm	25	4
Classification		Cyclones	7
No. of Operational Cyclones per Mill		2	2
No. of Cyclones per Mill		3	6
Total No. of Cyclones		3	2
Cyclone Size Diameter	mm	22.8	2
Product size P80	µm	25	2
COPPER CLEANER FLOTATION			
Configuration		Cleaner	
No. of Trains		1	
Flotation Cell	Type	Metso RCS / OK Equivalent	6
Feed Rate/train	t/h	16	2
Required Bank Volume	m^3/h	45	2
No. of Cells		4	2

Table continues…

Cell Volume	m^3	14.2	2
Overall Mass pull	%	0.90	4
Concentrate Grade	Cu - %	24.0	4
	Au - g/t	26.82	4
Cu Recovery	%	84.0	4
Au Recovery	%	55.0	4
COPPER CONCENTRATE DEWATERING			
Concentrate Thickener Type		5 m High rate Thickener	2,7
No. of Trains		1	6
No. of Thickeners / Trains		1	6
No. of Thickeners		1	2
Thickener Diameter	m	5.00	2
Thickener Underflow Solid Density	%	57.5	3
Slurry Transportation Method		Pumping	7
Flocculent Consumption	g/t	25	4
Solids Specific Gravity		1.6	6
Flocculent Solution Strength	g/l	0.5	7
Filter Type		Filter Press or Similar	7
Filter Feed rate	t/h	11.3	3
Filtration rate	$kg/m^2/h$	480.00	6
Filtration Area Required	m^2	23.47	2
Filter Size Selected	m^2	20.00	2
No. of Filters		2	2
Final Product Moisture	%	8-10	7
PRELEACH THICKENERS			
Cyclone Overflow Thickener Type		80m High rate Thickener	2,9
No. of Trains		1	6
No. of Thickeners / Trains		1	6
No. of Thickeners		1	2
Thickener Diameter	m	80	2,9
Thickener Underflow Solid Density	%	45	3
Slurry Transportation Method		Pumping	7
Flocculent Consumption	g/t	25	4
Solids Specific Gravity		1.6	6
Flocculent Solution Strength	g/l	0.5	7
Flocculent Plant Capacity	kg/h	50	2
Flocculent Holding Tank Capacity	h	2	2
CARBON IN LEACH			
Solids Feed rate	t/h	1,240	3

Table continues…

No. of Parallel trains	-	2	6
CIL Feed rate per train	t/h	620	2
Pulp Density	% Solids	45%	2,3
Nominal Flow rate	m^3/h	1,928	2,3
Processing method	-	Carbon In Leach	5
Leach feed grade	g/t	0.16	
Absorption & Leach Tanks			
Type		Mechanical Agitation	6
No. of Trains		2	6
No. of tanks	per train	8	2
Operating Volume , each	m^3	3,144	2
Nominal Resident Time, Total	h	3	4
Nominal Resident Time, each	h	24	2
Tank Dimensions	m x m	16 x 16.5	2
Bottom Clearance (approx.)	mm	4,000	8
Impeller clearance (approx.)	mm	5,500	8
Leach Air Requirement (per tank)	m^3/h	2,163	2,7
Carbon Inventory & Movement			
Carbon Concentration	g/l	10	2
Carbon Loading	g/t	2,500	2,7
Carbon Density	t/m^3	0.48	7
Carbon per tank	t	31	2,8
Carbon per stream	t	252	2,8
Carbon total circuit	t	503	2
Carbon Flow rate			
Per Stream	t/d	2.1	2
Total circuit	t/d	4.3	2
Carbon Advance Method		Recessed Pumps	8
Carbon Advance Sequence		Both streams every day	2,8
Inter-stage Screen Aperture	mm	0.8	7,8
Inter-stage Screen Area	m^2	3.9	2
Loaded Carbon Dewatering Screen Aperture	mm	0.6	7
Loaded Carbon Dewatering Screen Area	m^2	48.2	2
ACIDWASH, STRIPPING, REFINING AND CARBON REGENERATION			
No. of Parallel Trains		1	6
Loaded Carbon Density	t/m^3	0.48	
Acid Wash			
Acid Type		Dilute Hydrochloric Acid	
No. of units per Train		1	2,8

Table continues...

Total No. of units		1	6
Column Type		Conical Bottom	2
Wash Schedule		Every Batch	
Operating temperature	^{0}C	135	
Acid Column Capacity	t/unit	4.3	
Bed Expansion	%	50	
Bed Volume (bV)	m^3	13.3	
<u>**Acid Wash Pump box**</u>			
Type		Flat Bottom	7
Volume per bed volume	m^3	13.3	2
Solution flow Rate	bV/h	2	7
Solution flow Rate	m^3/h	26.6	2
<u>**Acid Wash Cycle**</u>			
Acid wash	min	180	7
Water Rinse	min	60	7
Total Acid wash Cycle	min	240	7
<u>**Elution**</u>			
Type		Pressure Zadra	1
Operating Schedule	Batch/d	1	7
Operating Schedule	d/week	7	7
Maximum Carbon Loading	g/t	10,000	7
Optimum Carbon Loading Gold	g/t	2,500	6
Elution Efficiency	%	99	6
<u>**Strip Vessel**</u>			
Number of units per train	-	1	6
Total number of units	-	1	6
Capacity	t/unit	4.3	2,8
Bed Expansion (design)	%	50	6
Bed Volume	m^3	13.3	2
<u>**Strip Operating Conditions**</u>			
Copper Strip Temperature	^{0}C	Ambient	0
Gold Strip Temperature	^{0}C	135	7
Gold Strip Solution		1% NaOH + 0.2% NaCN	7
Gold Strip Pressure	kPa	450	7
<u>**Strip Cycle**</u>			
Caustic Strip Solution Flow Rate - gold	bV/h	2	7
Caustic Strip Solution Flow Rate - gold	m^3/h	26.6	2
Preheat Solution	min	60	7
Carbon Transfer	min	30	7

Table continues...

Preheat Carbon	min	120	7
Caustic Heated Strip	min	180	7
Cool down/ rinse	min	60	7
Carbon Transfer to Kiln	min	30	7
Total Strip Cycle	min	480	7
Heat Recovery Exchanger			
Type	-	Plate and Frame	7,8
Temperatures:			
Pregnant Solution In	^0C	135	7
Pregnant Solution Out	^0C	95	7
Barren Solution In	^0C	85	7
Barren Solution Out	^0C	111	7
Strip Solution Heater			
Type	-	Steam heater - Diesel Fired	7,8
Temperatures:			
Inlet	^0C	111	7
Desired outlet	^0C	135	7
Cooler			
Pregnant Solution In	^0C	95	7
Pregnant Solution Out	^0C	85	7
Electrowinning			
No. of parallel trains	-	1.00	6
Electrowinning Cell	Type	Standard Rectangular	6,8
Cells in parallel per train	-	1.00	2
Cell Area	m^2	0.19	2
Cell Temperature	^0C	85	7
Operating Schedule	h/d	16	6
Actual Solution Flowrate /cell	m^3/h	26.64	2
Solution linear velocity	m/s	0.40	2
Type of cathode	-	Stainless Steel mesh(basketless)	7,8
No. of Cathodes	per cell	39.00	7,8
Cathode sludge removal method	-	Pressure Wash	7
Carbon Regeneration			
No. of Parallel Trains	-	1.00	6
No. of Kilns /train	-	1.00	2,8
Furnace	Type	Diesel Fired Horizontal Rotary Kiln	8
Capacity	t/d	4	2,8
Schedule	-	Batch	7
Batches /day	-	1	2

Table continues…

Resident Time	min	120	2,8
Cycle Time	h	10	8
Feed Rate	t/h	0.43	8
Capacity	t	4	2
Reactivation Temperature	^0C	750	7
Carbon Dewatering Screen			
Type		Inclined Vibrating	7
Aperture	mm	0.60	7
Carbon Moisture Content	%	50	6
Carbon Feed Bin			
Capacity	t	4	2
Capacity	m^3	13	2
Quench Tank			
Capacity	t	4	2
Capacity	m^3	13	2
Smelting			
Type		Diesel-Fired Tilting Furnace	8
Smelting Temperature	^0C	1,050	7
Operating Schedule	-	Daily	6
Fluxes	-	Borax, silica, Soda Ash, Fluorspar	7
CYANIDE DESTRUCTION			
Method	-	SO_2/Air	4
Operating pH	-	8.5	4
No. of Parallel trains	-	1	6
Feed rate	t/h	1,240	3
Pulp Density	% Solids	45%	3
Volume Flow rate	m^3/h	1,928	3
Reagents	-	$CuSO_4$, Sodium Metabisulphite, Lime	4,7
Detox Tanks			
Type	-	Mechanical Agitation	5
No. of tanks	per train	2	2
Operating Volume , each	m^3	646	2
Tank Dimensions	m x m	9.3 x 9.8m	2
Method of Aeration	-	Single stage blower	7
FINAL TAILINGS THICKENER			
Cyclone Overflow Thickener Type		100 m High rate Thickener	2,7
No. of Trains		1	6
No. of Thickeners/Train		1	6
No. of Thickeners		1	2

Table continues…

Thickener Diameter	m	100	2
Thickener Underflow Solid Density	%	60	3
Slurry Transportation Method		Pumping	7
Flocculent Consumption	g/t	25	4
Solids Specific Gravity		1.6	6
Flocculent Solution Strength	g/l	0.5	7

Key:

1 = Client	g/l = grams per litre	N/A = Not Applicable
2 = Calculation	g/t = grams per tonne	NaCN = Sodium Cyanide
3 = Mass Balance	h = hour	NaOH = Sodium Hydroxide
4 = Met Test work	h/d = hours per day	No. = number
5 = Resource Estimate	h /shift = hours per shift	OK = Outokumpu
6 = Assumption	kg/h = kilograms per hour	oz /a = ounces per annum
7 = Industry	kg/m²/h = kilograms per square metre per hour	RCS = Reactor Cell System
8 = Preliminary Economic Assessment	kPa = kilo Pascals	Shift /d = shifts per day
9 = Vender's recommendation	kW = kilowatt	SMC = SAG Mill Comminution
% = percent	kWh/t = kilowatt hours per tonne	SO2 = Sulphur dioxide
a = annum (year)	L – Length	t = tonnes
bV/h = Bed volume per hour	lb/a = pounds per annum	t/a = tonnes per annum
CS = Critical Speed	m = metres	t/d = tonnes per day
$CuSO_4$ = Copper Sulphate	m/s = metres per second	t/h = tonnes per hour
d = day	m^2 = square metre	t/m^3 = tonnes per cubic metre
D = Diameter	m^3 = cubic metres	µm = micron
^{0}C = degrees Celsius	m^3/h = cubic metres per hour	V = Volume
ft = foot	Min = minutes	VB = Volume of Balls
g = gram	mm = millimetres	

17.3 Availability and Utilisation

The crushing plant has been designed based on 353 operational days per year to allow 1 day per month of planned shut down for scheduled maintenance and housekeeping. The crushing plant is expected to have an availability of 88% to give an effective utilisation of 85%.

The milling and leaching plants have been designed based on 354 operational days per year with approximately 97% availability to give 95% effective utilisation.

The ore stockpile has been designed with sufficient surge capacity to provide an uninterrupted supply to the mill in order to maintain a high utilisation during the crusher shut down periods.

17.4 Location and Layout

The proposed process plant will be located approximately 850 metres above sea level and northeast of the open pit mining area. The ROM pad, crushers, and conveyers will be located west of the process plant (between the open pit and the process building) to facilitate the material handling and crushing from the mining area.

The TMF will be located south of the process plant and the reclaim water ponds will be located northwest of the plant building (in-between the process plant and the TMF area).

The gold room, office buildings, and laboratory will be located adjacent to the process plant.

The plant access road will connect the process plant to the local area roads.

17.5 PRIMARY EQUIPMENT LIST

The primary equipment list (Table 17.2) has been prepared based on the process design criteria presented in Table 17.1

Table 17.2 Primary Equipment List

EQUIPMENT	NUMBER	kW	TOTAL kW
Crushing			
Crusher Feedbin/Hopper (360 t)	1	-	-
Primary Gyratory Crusher (50" x 65", 375 kW)	1	375	375
Crusher Discharge Bin (400 t)	1	-	-
Crushed ore apron feeder (1.5 m x 6 m)	1	95	95
Conveyor (1.2 m x 800 m)	1	400	400
Conveyor Belt Scale	1		-
Tramp Magnet	1		-
Belt Sampler	1		-
Dust Collector Bin	1	6	6
Coarse ore stockpile (72,000 t)	1	-	-
Sump Pump - Crusher Station	1	19	19
Sump Pump - Stockpile Area	4	19	75
Grinding			
Stockpile reclaim apron Feeder (1.2 m x 6 m)	4	23	90
SAG Mill Feed Conveyor	1	200	200
SAG Mill (32 ft x 16 ft, 8.4 MW motor)	1	8,400	8,400
SAG Mill Pebble discharge conveyor	1	40	40
Pebble Crusher Vibrating Feeder	1	1	1
Pebble Crusher -HP800	1	450	450
Pebble Crusher Discharge Conveyor	1	40	40
Metal Detector	1	-	-
Tramp Magnet	1	-	-
Mill Discharge Pump	4	750	1,500

Table continues...

Ball Mill (24 ft x 34 ft, 10.6 MW motor)	2	10,575	21,150
Ball Mill Cyclone Pack (8 x 91.4 cm Cyclones)	2		-
Desliming Cyclone Pack (12 x 25.4 cm)	8	-	-
SAG and Ball Mill bridge Crane	3	4	11
Mill Liner Crane	1	56	56
SAG Mill Media Hopper	1	8	8
Ball Mill Ball Hopper	2	8	15
Media Charge Electro Magnet	3	5	15
Monorail Hoist	1	11	11
Cyclone overflow slurry sampler	2	-	-
Sump Pump - SAG Mill	1	188	188
Sump Pump - Ball Mills	2	75	150
Linear Safety Screen (40 m^2)	1	75	75
Copper Flotation			
Floating Conditioning Tank + Agitator	1	45	45
Rougher Cells 156 m^3	9	150	1,350
Scavenger Feed Pump	2	375	375
Scavenger Cells 156 m^3	9	150	1,350
Scavenger Tailings Pumps	2	375	375
Concentrate Discharge Box (Rougher- Scavenger)	1	-	-
Concentrate Feed Pump	2	4	4
Concentrate Regrinding Cyclone Pack (3 x 22.8 cm Cyclones)	3	-	-
Regrind Mill - SMD Mill (SMD 355)	1	355	355
Cleaner Conditioning Tank + Agitator	1	2	2
Cleaner Feed Pump	2	11	11
Cleaner Cells 14.2 m^3	4	19	75
Cleaner Tailings pumps	2	4	4
Cleaner Concentrate Pump	1	4	4
Slurry Sampler	2	-	-
Sump Pumps	1	19	19
Pre Leach Thickener			
In Line Floc Mixer	1	-	-
Thickener underflow pump	2	375	375
Thickener over flow surge tank	1	-	-
Thickener over flow pump	2	375	375
Pre-Leach thickener (80 m high rate)	1	11	11
Sump Pump	2	19	38
Carbon in Leach			
Leach Circuit Feed Sump	1		-

Table continues...

Leach feed pumps	4	83	330
Pump Motors	4		-
Feed Trash Screen	2	15	30
Trash screen motor	2		-
L each Tanks (16.0 m x 16.5 m Stainless Steel Cyanide tank)	16		-
Leach Impeller (Mixertech)	16	75	1,200
Impellor motors	16		-
Recessed Impellor Pumps (carbon)	32		-
Pump Motors	32	1	24
Interstage screen MPS(P)	16		-
Safety Screen	2	15	30
Screen motor	2		-
Screen feed pump	4	83	330
Pump Motors	4		-
Leach Circuit Feed Sump Pump	4	83	330
Pump Motors	4		-
Leach Feed Samplers	2		-
Leach Tail Samplers	2		-
Cyanide Detoxification Reactor & Agitator	2	75	150
Cyanide Detoxification Reactor Pumps	6	83	496
Pump Motors	6		-
Cyanide Detoxification Surge Tank & Agitator	1	75	75
Oxitrol measuring unit	1		-
Tailings Discharge			
Tailings Thickener (100 m high rate)	1	56	56
Sump Pumps	1	19	19
Concentrate Handling			
Concentrate Thickener	1	1	1
Concentrate Thickener Underflow Pump	2	2	2
Filter Press (29.7 m^2)	1	8	8
Filtrate Pumps	2	2	2
Cloth wash Pumps	2	2	4
Filter Cake Product Conveyor	1	3	3
Load Out Filter Cake Product Conveyor	1	3	3
Conveyor Scales	1	-	-
Belt Sampler (Capacity 262 cm^3)	1	-	-
Truck Scales	1	-	-
Sump Pumps	2	19	38
Reagent Preparation & Distribution			

Table continues…

Reagent Mixing Tanks	8	-	-
Reagent Mixing Tanks - Agitators	8	2	12
Reagent Holding Tanks	8	-	-
Reagent Feeders with Metering system	8	-	-
Reagent Distributors with pumps	2	0	0
Lime Storage Bin (181)	1	4	4
Lime Slaker System	1	75	75
Flocculant Plant	1	6	6
Sump Pumps	2	19	38
Plant Supply & Utilities			
Plant Air Compressor	1	56	56
Flotation Air blowers	20	38	750
Process Water Tank	1	-	-
Potable Water Tank	1	-	-
Water Pumps	4	19	75
Elution, Electrowinning & Gold Room Package			
Elution, Electrowinning & Gold Room Package	1	257	257

Key:

cm = centimetre | MPS = Mineral Process Separating | P = Pumping

cm^3 = cubic centimetre | MW = megawatt | SMD = Stirred MediaDetritor

" = Inches

17.6 Process Description

17.6.1 Process Description Overview

Mine haul trucks will haul ROM ore from the open pit mine to the crushing plant 7 d/week. ROM ore will be fed into the primary crusher feed hopper system directly by mine trucks. The gyratory crusher will discharge the product into an apron feeder that feeds the stockpile feed conveyor.

The gyratory crusher will operate 7 d/week, 24 h/d, with an operational utilisation of 85%. A stockpile feed conveyor will feed the open coarse ore stockpile with two days of operational capacity.

Apron feeders situated below the coarse ore stockpile will feed the conveyors, sending material directly to the SAG mill. The slurry will discharge from the SAG mill through discharge grates fitted with 75 mm pebble port. The SAG mill discharge will pass through a classifying trommel screen with 12.5 mm apertures. The trommel undersize will pass into the plant mill sump. The +12.5 mm -75 mm pebbles will move via conveyor to a pebble bin feeding an open circuit type pebble crusher, which will crush the oversize pebbles before discharging back to the SAG mill feed.

The discharge from the plant mill sump will be pumped to parallel clusters of classifying hydrocyclones and the hydrocyclone underflows will feed the ball mills. The ball mills will discharge back into the plant mill sump, creating a recirculating load of 250%, to ensure efficiency of grind.

The mill cyclone overflows will be treated by clusters of dewatering cyclones in order to prepare the mill discharge for flotation. Dewatering cyclone overflows will be pumped to the process water tank. The underflows at approximately 35% pulp density will be discharged to the flotation conditioning tank. Dewatering cyclones are selected instead of mill thickener in order to minimise capital cost and plant foot print requirements. However, it is essential to revisit this equipment selection with a detailed trade off study in the future in order to identify the optimal solution.

The conditioned slurry will be treated by rougher scavenger copper flotation and the concentrates will be cleaned by cleaner flotation after regrinding. The scavenger tails will be discharged to the preleach thickener for thickening.

The cleaned copper concentrates will be dewatered in a concentrate thickener followed by filtration. The dewatered concentrates will be shipped to the smelter packed in super sacks.

The flotation tailings thickener underflow will be pumped into leach feed sumps and subsequently into CIL tanks. The CIL tanks will be constructed on progressively lower elevations to allow the slurry to flow by gravity from tank to tank. Each tank can be by-passed by manually operated valves which will direct the flow through a by-pass line traversing the tank. Each CIL tank will be equipped with a gas sparger for injection of air into the slurry, in order to maintain the oxygen level in the solution required for the process kinetics. The carbon from the CIL tanks will be moved in a counter current flow to the slurry and the loaded carbon will be emptied sequentially for stripping. The CIL tails will be discharged to the tailings thickener after cyanide destruction which employs the SO_2 Air method. The tailings thickener discharge will be pumped to a conventional TMF for permanent disposal.

The loaded carbon from the CIL tanks will be pumped to the loaded carbon recovery screen, ahead of the acid wash/stripping circuits. A linear recovery screen will recover the loaded carbon. The slurry from the loaded carbon recovery screen (screen undersize) will return to the CIL slurry feed launders and the loaded carbon will be charged directly into the acid wash vessels from the loaded carbon screen.

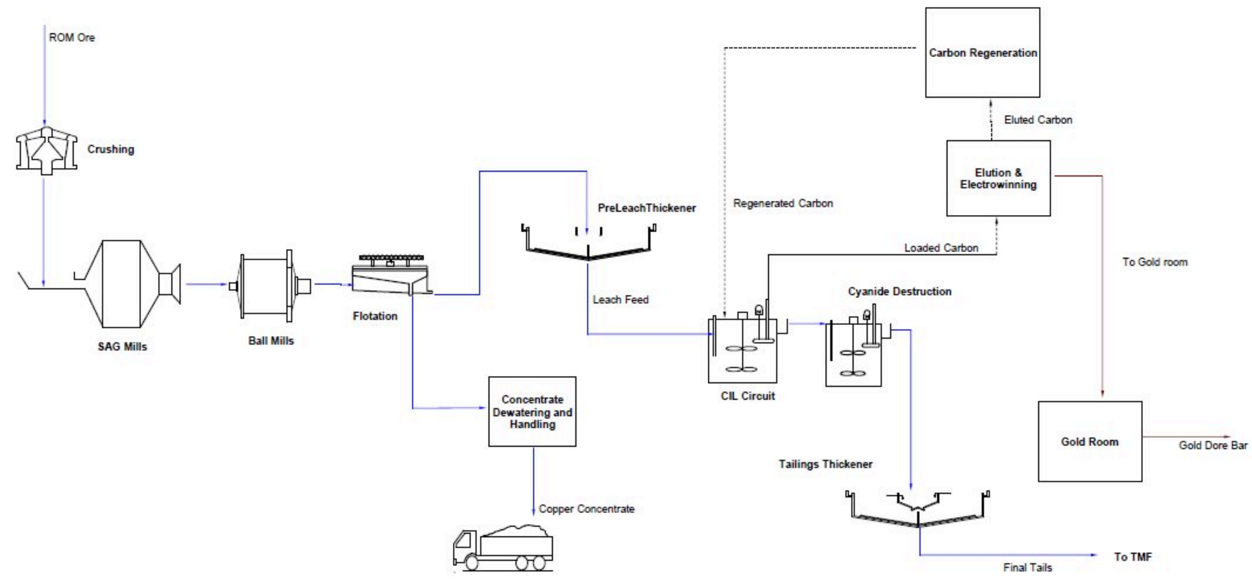
Recessed impeller pumps will be used for the carbon transfer throughout the circuit.

Acid washed carbon will then be pressure stripped with a hot caustic solution to elute the gold into a high-grade pregnant solution, which will then be treated by conventional electrowinning to produce cathode sludge or cathodes suitable for direct smelting. The cathode sludge will be filtered, dried, and smelted on site to produce gold doré bars.

Sampling of the various process streams will be carried out in order to quantify the plant performance on a shift and daily basis and to be able to control areas of the process on a continuous/semi-continuous basis. The location and type of sampling is described in each subsection of the process plant description.

A block flow diagram of the process plant is presented in Figure 17.1.

Figure 17.1 Block Flow Diagram of the Process Plant



17.6.2 Primary Crushing, Conveying and Stockpiling

The primary crusher will be a 42” x 65” gyratory crusher with a 375 kW drive motor. The crusher has been designed to crush at an average rate of 1,343 t/h, at a product size of P_{80} = 150 mm.

A mobile crane with an 80 t capacity will serve to lift and place materials during crusher mantle and main shaft replacement.

The crusher building will include a dust collection system in order to maintain a safe and clean working environment inside the crusher building. The building has been designed with a surge pocket under the crusher with a live capacity of 400 t. An apron feeder will extract crushed ore from the surge pocket under the crusher to feed the crushed ore conveyor at an average rate of 1,343 t/h. The apron feeder will be equipped with a 95 kW variable speed drive which will control the loading of the crushed ore conveyor.

The crushed ore conveyor, 1.2 m wide, approximately 800 m long and driven by a 400 kW motor, will feed the crushed ore stockpile.

The crushed ore stockpile has a live capacity of 60,000 t, which represents approximately 48 hours of process plant operation. The total live plus dead storage capacity in the crushed ore stockpile is approximately 75,000 t, which represents approximately 60 hours of process plant operation. This will allow the process plant to continue operating for the duration of a complete crusher mantle relining.

17.6.3 Ore Reclaim and SAG Mill Feed Conveyor

The crushed ore will be reclaimed from the crushed ore stockpile by four apron feeders located in a reclaim tunnel beneath the stockpile. The apron feeders will each be 1.2 m wide, 6 m long, and extract ore from a lined opening below the stockpile. Two apron feeders will feed the SAG mill conveyor, with each apron feeder having a full operational capacity of 80% of the SAG mill feed. This will allow continued SAG mill operation during apron feeder maintenance or unforeseen down time.

The reclaim tunnel will incorporate emergency exits, drainage, and ventilation to allow maintenance to be carried out in a safe working environment.

The SAG mill feed conveyor will be approximately 180 m long and carry the ore from the stockpile reclaim apron feeders to the two SAG mill feed chutes. The conveyors will be 900 mm wide and driven by a 200 kW drive motor to provide a full load capacity of 1,252 t/h. The feed system will incorporate conveyor scales and trash magnets (installed before the scales) to record accurate mill feed tonnage while removing tramp material.

Sampling of the SAG mill feed will not be practical due to the material size, therefore on-line samplers have not been included. For sampling of the mill feed a stop belt sampling procedure will be employed.

17.6.4 Grinding Circuit

The milling plant will receive fresh ore feed at an average rate of 1,252 t/h operating 7 d/week, 365 days per annum (d/a), to process on average 10,000,000 t/a at an overall plant effective utilisation of 95%.

Primary grinding will be achieved via a SAG mill with secondary grinding completed by a ball mill.

The 9.75 m diameter by 4.35 m SAG mill will be driven by a fixed speed, 8,600 kW motor via a single pinion drive, receiving, on average, 1,260 t/h of fresh ore feed. The slurry will discharge from the SAG mills through discharge grates fitted with 75 mm pebble ports. The SAG mill discharge will pass through a classifying trommel screen with 12.5 mm apertures. The trommel undersize will pass into the plant mill sump and the +12.5 mm -75 mm pebbles will be conveyed to a pebble bin feeding an open circuit type pebble crusher, which will crush the oversize pebbles before discharging back to the SAG mill feed.

The recirculating load conveyor will receive the pebble port oversize and transfer this material (pebbles) to feed the pebble crusher. The single cone type pebble crusher will be driven by a 400 kW drive motor. The product from the pebble crusher will be sized at P_{80} = 12 mm and will be conveyed back to the SAG mill feed conveyor.

The recirculating load circuit will include a self-cleaning belt magnet to remove tramp material in order to protect the pebble crusher from tramp, metallic materials. It will also include a metal detector ahead of the pebble crusher which will actuate the crusher feed shuttle to divert feed from the pebble crusher to the discharge conveyor of the pebble crusher when metal is detected on the crusher feed belt.

The pebble crusher can be by-passed during any unplanned shut down of the pebble crusher station.

The final product from the SAG circuit will then be fed to the ball mill and will have a product size of P_{80} = 1.5 mm, with top size of approximately P_{100} = -2 mm.

Two ball mills, rated at 10,600 kW, 7.3 m in diameter, with an effective grinding length (EGL) of 10.6 m long, will be operated in closed-circuit with classifying hydrocyclones to produce a ball mill product size P_{80} = 75 µm.

The mill sump will be equipped with two heavy duty slurry pumps, one operational and one 100% duty standby, feeding closed-circuit classifying hydrocyclone cluster. Hydrocyclone cluster will consist of ten 914 mm in diameter cyclones (8 operating and 2 on standby). The hydrocyclone underflow will be piped back to the mill sump where the returning load will create a recirculating feed to the ball mills in the region of 250%. The overflow stream from the hydrocyclone cluster will pass through a dewatering cyclone cluster consisting of 48 25.4 mm diameter cyclones (42 operational and 8 on standby). The dewatering cyclone underflow, at approximately 35% solids will feed the flotation circuit. The overflow from the dewatering cyclones will be pumped to the process water tank.

Each of the two mill bays will be equipped with overhead bridge cranes used for maintenance and grinding media charging.

Two ball pits will allow a different size grinding media to be delivered to the mills. In order to replenish the mill charge, the grinding balls will be loaded into a 15 t capacity ball bucket with a ball magnet suspended from an overhead crane. The ball bucket will be transported by overhead crane to the mills and the balls transferred into the ball charging hoppers located at each mill feed end. Each ball pit will have its own access door on the exterior building wall, which will be open to provide access for delivery trucks to discharge balls into the ball pits. Additional ball storage will be situated outside the main plant building.

One overhead mill liner portal crane will be available for the refurbishment of the grinding mills. Driven by a 75 kW electric motor, the hydraulic unit will have a lifting capacity of 1,800 kg enabling the removal and repositioning of liners and liner plates inside the grinding mills during routine maintenance.

The product from the grinding circuit (dewatering hydrocyclone underflow) will be sampled with automated in-line samplers. The particle size distribution and solids content recorded from these samplers will be used to monitor and control the grind size of the product from the milling circuit (leach circuit feed).

17.6.5 Flotation

The ground slurry from the ball mill grinding circuit will be pumped to a flotation feed tank, where flotation reagents will be added and conditioning can take place, before being pumped to rougher flotation.

Rougher flotation consists of 9 by 156 m³ flotation cells in series. The rougher tailings will be pumped as feed for 9 by 156 m³ scavenger cells. The residence time over the rougher and scavenger flotation is approximately 25 min each.

The rougher and scavenger concentrates will be combined with and pumped to a classifying cyclone pack in closed circuit with an SMD mill for regrinding. The underflow from the cyclone pack will feed the regrind mill, while the cyclone overflow, classified to P_{80} -25 µm will be presented to the cleaner flotation directly.

The regrind mill will be a Metso Stirred Media Grinding mill (SMD 355) rated at 355 kW. The cyclone pack consists of three 22.8 mm hydrocyclones (2 operational and 1 on standby).

The cleaner stage consists of of 4 by 14.2 m^3 cells in series offering approximately 16 min of residence time. Tailings from this cleaner stage are pumped back to the scavenger flotation feed tank. The concentrates from the cleaner stage will be the final copper concentrate.

The overall flotation mass pull is approximately 1.3% of the plant feed, producing 9.9 t/h concentrate at a grade of 24% copper and 26.8 g/t gold.

The tailings from the scavenger flotation cells will be pumped directly to an 80 m diameter preleach thickener. The thickener underflow, thickened to approximately 45% solids will be pumped to the CIL circuit. Clarified water from the thickener overflow underflow will be pumped back to the process water system.

17.6.6 Concentrate Thickening and Filtration

The final copper-gold flotation concentrate from the cleaner cells will be directed to a 5 m diameter high rate thickener. The thickened concentrate will be drawn from the thickener underflow at approximately 60% solids and sent to one of two parallel 29 m^2 filter presses, where the concentrate will be dewatered to approximately 8 to 10% moisture. The dewatered concentrate filter cake will discharge through the bomb bay doors on to a conveyor that will deliver the concentrate to a stockpile at one end of the mill building.

Front-end loaders will load the concentrate into bulk concentrate haul trucks for transportation. A truck scale in the load out area will ensure proper legal loading of the concentrate trucks.

Filtrate from the filter press operation will be collected and pumped back to the concentrate thickener. The clarified water overflow from the concentrate thickener will be pumped to the process water system.

17.6.7 Leach Feed Thickeners

The flotation tails will be pumped from the scavenger flotation to the thickener de-aeration drop boxes. A high rate thickener will control the feed density to the leach circuit at approximately 45% solids (w/w).

The thickener tank will be 80 m in diameter, with a constant sloped base (7.5° from the horizontal), and constructed with mild steel. The slurry feed will be gravity fed from the de-aeration drop box on the perimeter of the tank to an auto dilution feed well at the centre of the unit. Flocculant will be introduced into the thickener feed wells by an automated flocculation plant.

The thickener underflow pumps will be located beneath the central discharge cone at the centre of the thickener in a sub-terrain tunnel. There will be two thickener underflow pumps, fitted with variable speed drive motors. One pump will be operational, while the other will be a full-duty standby. The pump speed will vary, in order to control the solids density of the feed to the leach circuit.

The thickener overflow will flow by gravity to a water tank dedicated to the thickener, with the water pumped back to the process water system.

17.6.8 Carbon In Leach Circuit

The leach feed thickener underflow of 45% (w/w) solids will flow into the thickener underflow sump from where the slurry will be fed to two parallel rows of leach tanks. There will be a total of 16 agitated leach tanks arranged in 2 rows of 8 tanks. Each row of eight tanks will be fed with a thickener underflow sump pump.

The leach circuit has been designed to provide a total leach residence time of approximately 24 hours. Each tank will be equipped with a 75 kW motor and a double impeller agitator mounted on the superstructure on the top of each tank.

The leach tanks will have a working volume of approximately 3,140 m^3, dimensions of approximately 16.0 m diameter and 16.5 m high, and sit at progressively lower elevations to allow the slurry to flow by gravity from tank to tank in each of the two, eight-tank series.

Each tank will hold 31 tonnes of carbon achieving a gold loading of 2,500 g/t. Inter-stage screens will be installed within the tank to prevent loaded carbon leaving with the overflowing slurry. Carbon will be withdrawn from each leach train using recessed pumps at a rate of 2.15 t/d. This equates to a circuit total of 4.3 t/d of carbon being fed to the elution circuit.

Each tank can be by-passed by manually operated valves which will direct the flow through a by-pass line traversing the tank. Each leach tank will be equipped with a gas sparger for injection of air into the slurry in order to maintain the oxygen level in the solution required for acceptable process kinetics.

The loaded carbon will be recovered on a linear screen. The slurry from the loaded carbon recovery screen (screen undersize) will return to the carousel slurry feed launders by gravity and the loaded carbon charged directly into the acid wash vessels from the loaded carbon screen. Loaded carbon screening will be located directly above the acid wash vessels.

The barren slurry from the CIL circuit will flow continuously by gravity to a linear safety screen. From the linear safety screen the tailings slurry will flow into the cyanide detoxification (Detox) circuit.

Access for the mobile crane will be provided along each row of eight tanks.

17.6.9 Cyanide Detoxification

The final tailings from the CIL circuit will be treated by an INCO SO_2-AIR cyanide detoxification (detox) process. It is expected that this process will reduce the cyanide content below acceptable levels for storage in the TMF.

The INCO process destroys the cyanide through oxidation to cyanate a combination of oxygen and sulphur dioxide. Sulphur dioxide can be supplied in a number of forms. In this case sodium metabisulphite has been used as the source as it allows for storage in powder form and simple addition to the circuit. Oxygen is supplied through the addition of compressed air. Copper acts as a catalyst to this reaction and added to the circuit in the form of hydrated copper sulphate. As sulphuric acid is produced during the oxidation of cyanide to cyanate, lime slurry will be added to maintain pH between eight and nine.

The CIL tails will be pumped to two mechanically agitated cyanide destruction tanks in series. Although test work has not been completed to date the circuit has been designed with a residence time of 30 min of slurry at 45% solids (w/w) to achieve an acceptable final weak acid dissociable cyanide (WAD) concentration.

The agitated detox tanks were sized according to the quoted residence time of 30 min in total, two tanks with a diameter of 9.3 m and a working volume of 645 m^3 per tank. The agitators will be designed to provide fine dispersion of compressed air introduced directly below the impellers.

The final detox discharge stream will be pumped to the tailings thickener.

17.6.10 Carbon Acid Washing

The slurry containing the loaded carbon from the CIL tanks is pumped to the loaded carbon recovery screens located above the acid wash tank. The screened (washed) carbon from the linear screen falls through a chute into an acid wash tank. The moisture removed from the carbon on the recovery screens will combine with the barren slurry from the CIL circuit and be fed to the cyanide detoxification circuit.

The loaded carbon will then flow by gravity into a circular acid wash tank. The acid wash tank is constructed with a conical bottom and a fibre reinforced plastic interior.

The loaded carbon will first be drained and then washed using fresh water acidified with commercial hydrochloric acid solution pumped directly from the acid wash pumpbox. This solution will be pumped upwards through the carbon bed, allowed to overflow and recycled. The pH of this operation will be monitored and controlled at 1.0 during acid recirculation which will be carried out for up to three hours.

This solution will then be neutralised and pumped to the cyanide destruction tank. The final acid washed carbon will then be pumped directly to the striping circuit using the same recessed impeller carbon pump.

A fan-induced ventilation system will be incorporated above the acid wash tank. The acid wash area will have its own sump pump with any solution collected pumped directly to the first cyanide destruction tank.

17.6.11 Carbon Stripping

Carbon stripping is accomplished with Zadra pressure stripping technology. The circuit consists of a barren solution tank, a strip vessel, a natural gas fired strip solution heating system and heat exchangers.

Hot stripping solution composed of fresh water with 1% NaOH and 0.2% NaCN at 135°C will be pumped through the column normally at the rate of 2 bV/h for a total strip cycle of approximately 8 hours. This includes a cycle of two bed volumes preheat period lasting one hour where the carbon is heated up to 135°C, 30 min carbon transfer, a 10 bV strip, a 2 bV cool down/rinse and finally a 30 min carbon transfer period.

During stripping this solution will pass through the first plate and frame heat exchanger to recover heat from the pregnant solution before being contacted under pressure with a diesel fired steam heater. The heated solution will then pass through the carbon strip column where the carbon releases the precious metals. Exiting the column, the solution is cooled in a heat exchanger to drop the temperature below boiling point before entering the electrowinning cells. After the electrowinning cells, the solution is cooled by a trim heat exchanger and returned to the barren strip solution tank to be pumped back and recycled back through the heat exchangers and the strip column. When the carbon stripping is complete, the strip solution will be cooled to approximately 70°C after which the stripped carbon will be pumped to the dewatering screen mounted above the carbon reactivation kiln feed bin. After stripping, the barren carbon is cooled down by a water wash step in the strip vessel. Stripped carbon is then pumped from the strip vessel to the carbon reactivation area.

Sampling of the solution both before and after the electrowinning will be used to control the carbon stripping circuit. The barren solution from the cells will enter the strip solution tank where sodium hydroxide, sodium cyanide and make-up water will be added on a batch basis before each strip is initiated.

17.6.12 Electrowinning

The hot (less than 90°C) pregnant strip solution will flow through the electrowinning cells in parallel where the gold will be deposited on the stainless steel cathodes.

The barren solution, returned from electrowinning cells, is continually recycled via a barren pump box and returned to the barren strip solution tank. An anticipated 10% will be bled off back to the grinding thickener at the end of each strip cycle to avoid slow poisoning of the strip solution.

The loaded stainless steel cathodes will be cleaned periodically by applying a high-pressure water spray in situ or, if necessary, by hoisting each cathode above a purpose built wash tank to remove the precious metals loosely attached to the stainless steel cathode mesh. The sludge resulting from cleaning will be recovered from a sludge holding tank using a compressed air sludge pump, feeding a plate and frame filter press. Before emptying the press an air blow will ensure a drier, more manageable precious metal concentrate at an approximate 20% moisture content.

A fan-induced ventilation system will be incorporated above the cells. The electrowinning area will have its own sump pump with solution pumped back to the barren pump box.

17.6.13 Smelting

The precious metal sludge recovered as filter cake will be treated in an electric drying oven with a cycle time of 10 hours and temperatures of up to 450°C.

The dried, partially-calcined sludge will be mixed with fluxes (sodium nitrate, sodium carbonate, sand, and borax) and smelted in a diesel fired tilting furnace at 1,050°C and cast into moulds. The subsequent gold bars will be cleaned using a bar cleaner before being sampled and transferred to a vault for later transport to the smelter.

The induction furnace will be fitted with its own ventilation system whereby a refinery blower will evacuate all adjacent fumes/dust through a refinery dust collector. This dust will be collected in a bag house and returned to the flux mixing area for mixing with the fresh calcined sludge. Slag arising from this operation can be re-melted if significant inclusion of precious metals occurs. Finally relatively clean slag and other rejects (such as used crucibles) will be recycled to the ball mill.

17.6.14 Carbon Reactivation

The carbon to be re-activated will be pumped onto a dewatering screen located above the carbon storage tank. The carbon will be screw fed into a diesel fired horizontal rotary reactivation kiln. The carbon will be reactivated as it travels through the tubular rotary kiln being subjected to temperatures of 750°C. The design capacity of each kiln is 210 kg/h, the equivalent of approximately 4.3 t/d per kiln with 85% availability.

The hot re-activated carbon will be screened on a carbon sizing screen and the coarse carbon will be quenched in water and pumped to the activated carbon storage tank.

Fresh carbon will be added to a carbon attrition tank before being pumped over the carbon sizing screen. Fines from this process (and the size screening of reactivated carbon) will be fed to the fine carbon bin.

Carbon from the reactivated carbon storage tank will be pumped to the CIL circuit as required. The carbon stripping and reactivation area will also have a sump pump that will pump any spillage/washings into the carbon fines tank.

17.6.15 Tailings Thickeners

The final tails will be pumped from the cyanide destruction tanks to the thickener de-aeration drop boxes. A high rate thickener will control the feed density of the final tails at approximately 60% solids (w/w).

The thickener tank will be 100 m in diameter, with a constant sloped base (7.5° from the horizontal), and constructed with mild steel. The slurry feed will be gravity fed from the de-aeration drop box on the perimeter of the tank to an auto dilution feed well at the centre of the unit. Flocculant will be introduced into the thickener feed wells by an automated flocculation plant.

The thickener underflow pumps will be located beneath the central discharge cone at the centre of the thickener in a sub-terrain tunnel. There will be two thickener underflow pumps, fitted with variable speed drive motors. One pump will be operational, while the other will be a full-duty standby. The pump speed will vary, in order to control the solids density of the feed to the leach circuit.

The thickener overflow will flow by gravity to a water tank dedicated to the thickener, with the water pumped back to the process water system.

17.7 UTILITIES AND REAGENTS

Plant services will include blowers to supply air, potable and plant water supply, and electric power supply.

The principal reagents will be lime and cyanide, copper sulphate and liquid sulphur dioxide.

17.7.1 AIR

The air supply will include the flotation cells, CIL tanks, and the instrument air requirements. Twenty air blowers each rated at 37.5 kW and supply air at approximately 45 m^3/min (69 kPa); will supply the required plant air.

17.7.2 WATER

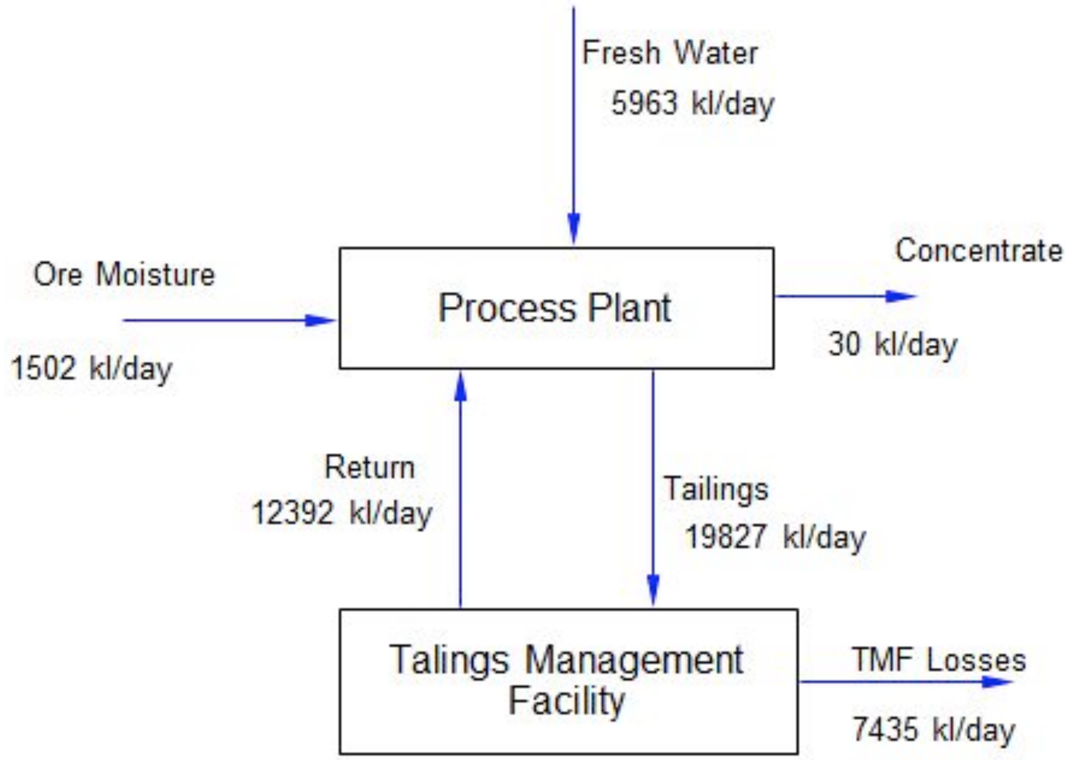
Fresh water will be supplied via direct pumping from the pit, boreholes, and local reservoirs to a dedicated fresh water tank. The overflow from the fresh water tank and the water take up from the TMF will be pumped into the process water tank and subsequently delivered to the process plant as required, principally as SAG and ball mill dilution water and to the carbon regeneration circuits.

The elution circuit and gold room will be serviced directly from the fresh water tank and recycled back to the process water tank.

A preliminary water balance (Figure 17.2) indicates that the process plant will operate with a negative water balance and require 5,693 kilo litres (kl) of fresh water every day. It is understood that the Tailing Management Facility has been designed on the assumption that the tailings will settle from 60% solid by mass to a final settled volume of 80% solid by mass. Based upon this assumption it is anticipated that the process plant will discharge 19,827 kl of water with final tailings every day and approximately 62% of the discharged water (i.e. 12,392 kilo litres per day [kl/d]) can be recovered (subject to evaporation and precipitation) from the tailings dam for process water requirements.

A comprehensive water balance will be established during the feasibility study to identify the water requirements more precisely.

Figure 17.2 Preliminary Process Water Balance



17.7.3 Power and Mobile Equipment

Power will be supplied to the site via a 5 to 8 kilometre, 110 kiloVolt spur from the main Macedonian national power grid. The power requirements for the process plant are approximately 42.5 MW.

One dedicated wheeled crane will be required for general plant use, maintenance and reagent preparation together with the additional mobile plant equipment listed in the primary equipment list.

17.7.4 Laboratory

A fully equipped laboratory will be made available on site to have separate sectors as described below:

- Sample preparation sector with space/bench area for sample receiving, drying ovens, size reduction equipment and adequate bench space for the preparation of mine and geology samples.
- Assay laboratory with separate sample preparation and storage areas. The assay laboratory will contain fire assay equipment, scale room, chemical analysis laboratory and chemical storage.
- Metallurgical laboratory including pressure filters, grinding simulation equipment, bench flotation cells, bottle roll leach test equipment and other miscellaneous metallurgical laboratory equipment as required.

17.7.5 LIME

Quick lime will be delivered in bulk carriers, equipped with pneumatic unloading systems. The lime will be unloaded from the trucks into two silos. The silos will be equipped with a dust collection system, a pneumatic unloading system, and unloading hoppers at the bottom. Screw feeders at the bottom of the silos will convey the quick lime to the lime slaker where water is added and the quick lime hydrated and suspended in the water. The lime slurry will flow from each slaker to an agitated storage tank, where the lime is then distributed to two different distribution loops. Each distribution loop will have two horizontal slurry pumps, one operating and one standby, which will feed the loop. From each distribution loop lime slurry will be metered into its usage point by an automated valve. The unused flow in each loop will be piped back to the agitated storage tank forming a close circuit, minimising line plugging occurrences. The lime slurry distribution loops will be piped to the leach tank area and the detox plant.

17.7.6 CYANIDE

Cyanide will be delivered in bulk bags containing sodium cyanide briquettes. The briquettes will be added to an agitated cyanide mixing tank with the required volume of water. The bag containing the briquettes will be hoisted onto a bag-breaker situated above the tank and enclosed in a steel chute for dust control. The cyanide solution will be made up to approximately 25% concentration. A split tank system can be used whereby two-thirds of the tank is dedicated for cyanide mixing and the other third as storage from which the cyanide solution will be pumped via the ring main to the leaching circuit as required to maintain an initial cyanide concentration in the first leach tank.

A bunded concrete spillage area with a dedicated sump pump will be installed.

Cyanide bags are normally contained within wooden crates and stored on a concrete platform, under cover, in an open-sided structure with suitable access for a forklift.

17.7.7 COPPER SULPHATE

Copper sulphate is delivered in 1,000 kg super sacks transported in batches of 24 t per truck load. The super sack is lifted onto the feed bin of the agitated tank and the copper sulphate powder discharges into the water that has been metered into the tank. The mixture is then transferred by a chemical pump to a storage tank from where it is pumped to the detox plant via a metering chemical pump.

17.7.8 SULPHUR DIOXIDE

Sulphur dioxide is delivered in liquid form by tanker trucks of approximately 26 t and stored in an 80 t horizontal storage tank. The storage tank comes with a pressure regulator to regulate the pressure of the sulphur dioxide gas.

An evaporator and a rectifier are connected to the sulphur dioxide storage tank to deliver/inject the sulphur dioxide gas to the INCO reactor.

17.7.9 Caustic Soda

Caustic soda is delivered in liquid form (50% concentration) in bulk transport trucks of approximately 35 t. At the process plant the caustic soda is unloaded into a storage tank which is equivalent to about 15 to 20 days storage. It is pumped to the barren strip solution tank where it is diluted with fresh water.

17.8 Control Philosophy

Appropriate control philosophy of the process plant will be developed during the feasibility study when the process parameters are fixed and piping and instrumentation diagrams (P&ID) are completed.

A control philosophy will be developed to achieve the following objectives as a minimal requirement:

- Maximise throughput and stabilise the control of the product particle size.
- Provide better control of the grinding circuits by control of the cyclone feed density.
- Monitor and control of flotation and regrinding allowing optimum recovery and concentrate grade control.
- Control of preleach, concentrate and tailing thickener underflow density to maintain the efficiency.
- Monitor flotation reagents additions and flotation concentrate mass.
- Monitor and control the filter press operation to ensure effective dewatering.
- Monitor the carbon loading and control the movement in the CIL circuit.
- Monitor and control the carbon elution and electrowinning circuits to increase the elution efficiency and reduce the soluble gold losses.
- Ensure the safety and security of the gold room.

17.9 Capital Cost Estimation

The direct capital costs have been estimated by generating an equipment list and using the latest edition of Mine and Mill Equipment Costs developed by Infomine USA, Incorporated (InfoMine 2013). This is appropriate for prefeasibility study purposes, where the estimate is typically within the range of ±25% using top down cost estimation techniques.

The installed motor sizes were also estimated using the same method as above. Where relevant data was not available from the manual, other estimates such as vendor packages or budget prices provided by equipment suppliers were made.

Once the basic equipment cost was calculated, factors were applied to estimate the direct construction capital costs, these factors being derived from similar projects.

By this method, the estimated capital cost for treating the Ilovitza sulphides ore by milling flotation and CIL is US $249.6 million ±25%.

Sustaining funds will be required for the on-going maintenance and upkeep of the processing plant throughout the life of the project. These funds have been incorporated into the operating cost section, based on a cost per tonne throughput. Additional sustaining capital, for major capital equipment replacement, has not been included. Table 17.3 summarises the estimation of the capital cost.

Table 17.3 Capital Cost Summary

Description	Million US $
Crushing	4.99
Grinding	30.87
Preleach Thickener	3.43
Copper Flotation	8.33
Carbon in Leach	16.34
Tailings Discharge	3.70
Concentrate Handling	0.51
Reagent Preparation & Distribution	0.77
Plant Supply & Utilities	0.96
Sub Total - Primary Equipment Cost	*69.91*
Civils	20.97
Structural Steel	16.78
Piping & Valves	24.47
Electrical & Instrumentation	27.96
Transport	20.97
Erection of Items	10.49
Vendor Services	2.10
First Fills	2.10
Sub Total - Indirect Capital Cost	*125.83*
Total - Installed Plant Capital Cost	195.74
Site Preparation & Construction Management	29.36
Coarse Ore Stockpile Construction	14.38
Elution, Electrowinning & Gold Room Package	3.38
Plant Mobile Equipment Cost	6.66
TOTAL PLANT CAPITAL COSTS	$249.52

17.10 Operating Cost Estimate

The operating cost includes the following estimated components:

- Labour (operations and maintenance).
- Reagents and consumables (plant).
- Maintenance spares.
- Power.

Labour has been estimated assuming a typical structure and using approximate labour rates based on information from Euromax.

A 12 h shift roster has been assumed with three shifts required to provide cover for rostered days off. A 40% allowance has been assumed for the salary burden.

Reagent consumptions have been estimated based on the laboratory test work results. Maintenance spares have been calculated assuming 2.5% of the cost of the mechanical equipment capital.

Power costs have been build-up from the capital equipment list absorbed power requirements, with the absorbed power assumed to be approximately 80% of the installed motor power. The power cost of (Euro) €56.6 per megawatt hour (/MWh) or US $0.08 /kWh (€ Euro:US Dollar exchange rate of 1.35), has been based on Euromax's figure for Macedonian grid power.

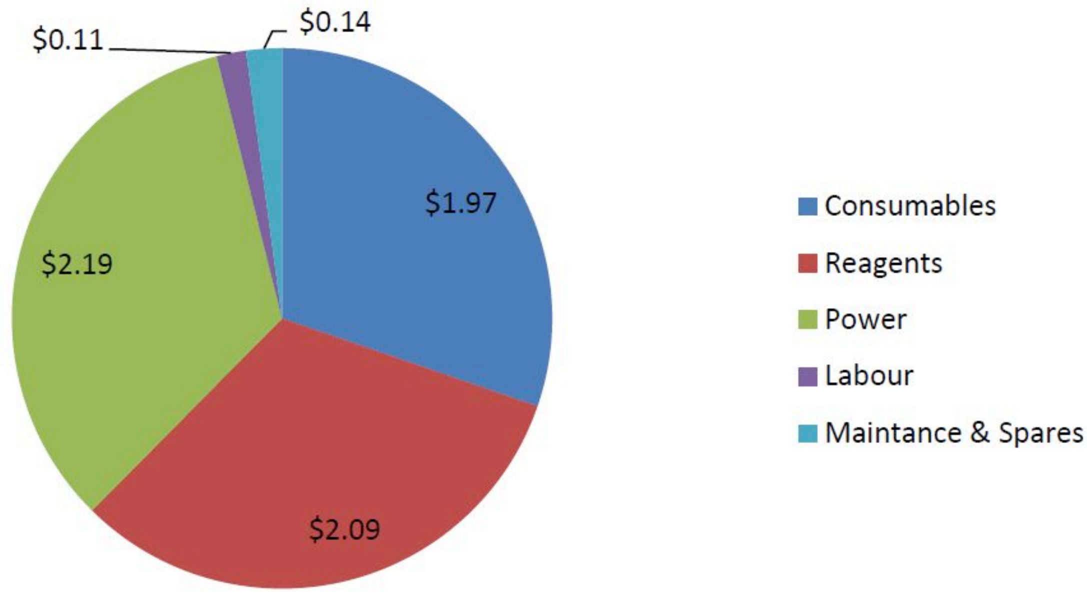
The total plant operating cost for treating the sulphide material by milling, flotation and CIL is estimated at US $6.50 /t ±25%.

A summary of operating costs is shown in Table 17.4 and Figure 17.3.

Table 17.4 Process Operating Cost

Description	Cost ($/t)
Consumables	1.97
Reagents	2.09
Power	2.19
Labour	0.11
Maintenance & Spares	0.14
TOTAL	$6.50

Figure 17.3 Process Operating Cost



17.11 ENVIRONMENTAL ISSUES

Tetra Tech is not involved with the environmental aspects of this project. However a high level review is provided as a summary view from a plant perspective.

17.11.1 PERMITS, STANDARDS AND REQUIREMENTS

Tetra Tech recommends that all the necessary permits and standards to operate the processing plant should be identified and obtained, or at least a reasonable prospect of obtaining such permits must be established by the time of the feasibility study.

Permits and standards requirements will be those required by Macedonian legislation governing mining and mineral processing operations. A general list of permits standards and requirements that are expected for processing plant construction and operation is provided below. It is important to note that this list is only indicative and not comprehensive.

- Plant construction permit.
- Plant operating permit.
- Industrial water usage and discharge permit.
- Air quality and emission standards requirements.
- Environmental compliance requirements.
- Effluent treatment and discharge requirements.

17.11.2 AIR QUALITY AND EMISSIONS

Air quality on the site and surrounding area may be affected by project activity during construction and operation of the processing plant.

Emissions that would result from the processing plant and their effects on air quality should be evaluated. Considerations should be given to ground level concentrations of hydrogen cyanide gas (HCN) (by potential evolution) and any other potential harmful emissions that may arise during the operation of the plant.

Tetra Tech recommends comparing these concentration levels with appropriate Macedonian air pollution control regulations to identify the necessary actions (if any) for compliance.

17.11.3 WATER

Fresh water sources for makeup should be identified during the feasibility study.

Non-acid generating (NAG) tests will be conducted during the feasibility study to identify any Potentially Acid Generating (PAG) characteristics of tailings and to identify an appropriate neutralisation programme.

Tetra Tech recommends conducting Toxicity Characteristic Leaching Procedure (TCLP) tests in the final tails to identify and quantify any leachable toxic metals left un-leached. Depending on the results, consideration should be given during the feasibility study to establish any additional treatment requirements to eliminate potential ground water contamination by toxic metals.

Total Dissolved Solids (TDS) of the process water will increase over time due to the use of lime for pH modification in the leach circuit. It is recommended potential treatment methods for reducing TDS levels below expected standards are investigated during the feasibility study.

17.11.4 EFFLUENTS (CYANIDE MONITORING, CONTROL AND DESTRUCTION)

Online cyanide monitors will be installed in the detox tailings stream to monitor the cyanide levels on the discharge streams. This will ensure that the discharge will be well below the acceptable levels. It is recommended that the mine should adopt the international cyanide management code and follow the best practices for cyanide management listed below.

1. Implement an overall planning procedure, from conception to closure and rehabilitation, based on an assessment of risks that maximises the benefits and minimises liabilities and environmental impacts.
2. Establish, implement and regularly review a cyanide management strategy as part of the mine's environmental management plan for implementing best practice.
3. Implement initial and ongoing cyanide safety and management training for all personnel involved in cyanide including contractors, who have management, operational or maintenance responsibilities or who handle or are exposed to cyanide (this training should cover both the everyday roles of personnel and how they respond to cyanide-related emergencies).

4. Establish well-defined responsibilities for individuals with clear chains of command and effective lines of communication within the workforce.
5. Institute safe procedures for cyanide handling governing transport, storage, containment, use and disposal.
6. Integrate the mine's cyanide and water management plans.
7. Identify and implement appropriate options for minimising demand for cyanide and reusing, recycling and disposing of residual cyanide from plant operations.
8. Conduct regular cyanide audits and revise cyanide management procedures where appropriate.
9. Develop comprehensive cyanide occupational and natural environment monitoring and management programme, supported through a sampling, sample preservation, analysis and reporting protocol.
10. Establish a carefully considered and regularly practiced emergency response procedure.

17.12 Health and Safety

A Health and Safety Policy will be drafted in accordance with the Macedonian health and safety legislation.

A general list of health and safety requirements that are considered for processing plant operation is provided below. It is important to note that this list is only indicative and not comprehensive.

- Risk assessments for process equipment.
- Hazard and Operability (HAZOP) studies for relevant equipment and processes.
- Developing a procedure for the management and control of substances that are hazardous to health (COSHH).
- Safety training including emergency response plan for all the staff.
- Use of personal protective equipment (PPE) when required.
- Displaying appropriate warning signs at relevant places

17.13 Risks and Opportunities

Potential risks and opportunities associated with this project from a processing/metallurgical perspective are identified and listed below.

17.13.1 Potential Risks

- Laboratory copper / gold recoveries are not always attainable under operating plant conditions.
- The mill may not effectively handle the design throughput throughout the mine life at the design conditions.

- Leach gold recoveries have a direct correlation to the grind size and therefore grinding efficiency is critical for the target gold recovery. Any grinding circuit overloading will result in lower than anticipated gold recovery.
- Ineffective operation and poor control of desliming cyclones may result in lower copper recoveries and increased reagent consumptions.
- Flotation gold recoveries may be reduced as a result of lime as a pH modifier
- Gold theft during processing is a common problem. A tight schedule and gold inventory control particularly in the adsorption and elution circuit is highly recommended.
- The plant is designed as a single line throughout with the exception of ball mills only. Unplanned breakdown of key equipment such as a SAG Mill or the thickeners can completely suspend the production temporarily.

17.13.2 Potential Opportunities

- Recovery of copper may be sustained at relatively coarser grinds; this could possibly offer operating cost savings.

18.0 INFRASTRUCTURE

18.1 INTRODUCTION

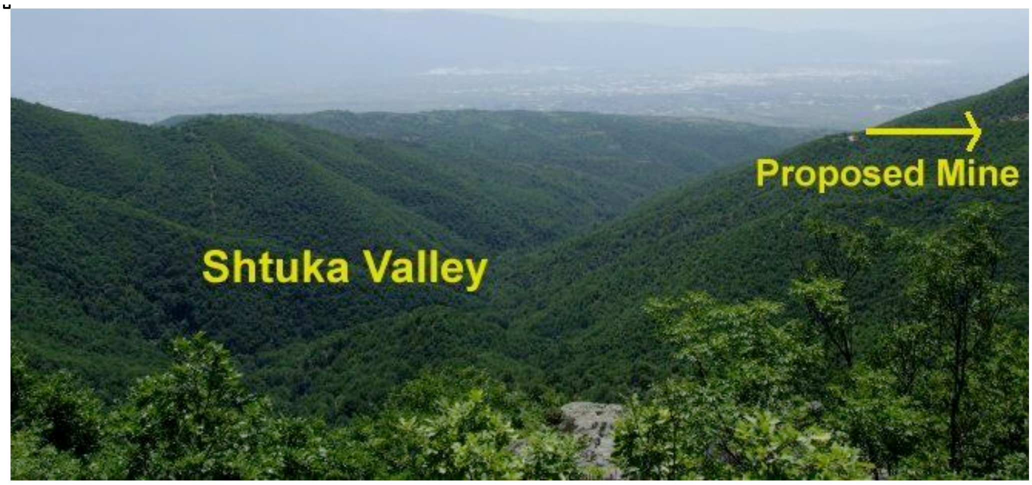
The site of the Ilovitza Gold/Copper project is located at the south-eastern hills of the Malesevski Mountains, approximately 3 km northeast of the village of Ilovitza, in south-eastern Macedonia. Ilovitza is located approximately 20 km east of the town of Strumica, as shown on Figure 18.1. The mine property is about a three-hour drive from either Skopje, the capital city of Macedonia or Sofia, the capital city of Bulgaria and two hours from the Greek port of Thessaloniki.

Figure 18.1 Ilovitza Project Location Map

Source: Tetra Tech

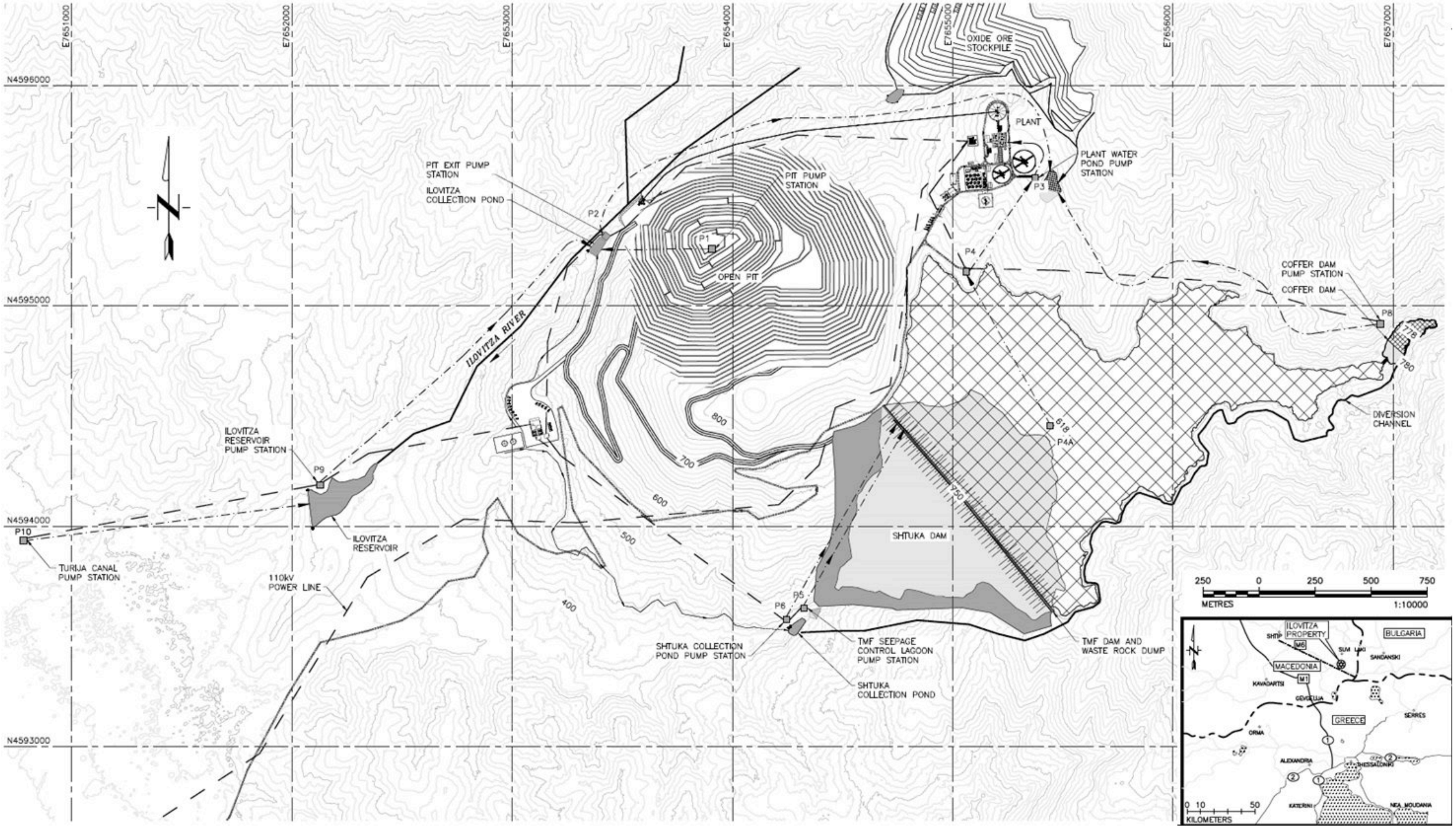
The infrastructure components of the proposed mining project will generally be constructed on rugged terrain in a mountainous region, featuring high hills and steep valleys. The elevation of the proposed mine site area varies between 350 m and 950 m above mean sea level (mamsl). The lower areas are located at the south southwest end of two relatively narrow valleys, within the Municipality of Bosilovo, while the high ground is predominantly located in the north northeast part of the site, close to the Bulgarian border, on the territories of Novo Selo and Berovo Municipalities. The two main valleys, next to the proposed open pit area, are the Ilovitza and Shtuka valleys, as shown on the images of Figure 18.2. The junction of the two valleys is located approximately 3 km southwest from the centre of the proposed open pit. The area surrounding the mine site is, in the most part, forested with medium to large trees, as shown on the images below.

Figure 18.2 Images of the Ilovitza and Shtuka Valleys, Surrounding the Proposed Mine



Source: Tetra Tech

Figure 18.3 **Proposed Site Layout**



Source: Tetra Tech

18.2 SITE LAYOUT

The topography of the site presents a number of challenges for the development of the project, with the location of the tailings management facility (TMF) as the primary driver. The PEA identified a number of alternative locations, which were assessed as part of the Prefeasibility Study and Euromax's preferred option is the Shtuka valley adjacent to the open pit.

A trade-off study for the tailings disposal method was completed which recommended the co-disposal of filtered tailings with waste rock. However at the time of publication no filtering testwork had been completed and as co-disposal is a relatively new technology Euromax favoured a more conventional thickened liquid tailings and tailings dam solution for the Prefeasibility study. It is recommended that further tailings test work is conducted to establish the characteristic and filterability of the tailings during the next stage of the study.

To reduce the tailings pumping requirements, Euromax requested that the processing plant be located on the Upper site to the northeast of the open pit, on a saddle between the Ilovitza and Shtuka valleys. The site layout shown in Figure 18.3 has been developed to meet these requirements.

The mine site and facilities are based in two main areas, an Upper site and Lower site. The Lower site has the Run of Mine (ROM) pad and primary crusher adjacent to the mine and pit exit around the 480 m elevation. The haul truck workshop and main fuel storage area are also adjacent around the 450 m elevation. The remaining facilities are located on an upper site around the 850 m elevation. This upper site includes the crushed ore stockpile, process plant, with gold room and product dispatch, as well as the ancillary facilities including the administrative and social building, stores and workshops.

Ore delivered to the ROM pad / primary crusher is crushed and conveyed via a cable conveyor to the crushed ore stockpile at the upper site for processing. Following processing the resulting tailings are discharged to a tailings management facility (TMF), located in the Shtuka valley.

Waste rock from the mining process will be transported by haul tracks to the Shtuka valley for use in constructing the tailings dam with surplus waste rock dumped on the downstream face of the tailings dam. Oxide ore will be stored in a temporary stockpile close to the plant for processing at the end of the life of mine.

Water around the site will be managed, primarily to ensure that the environment is protected, while providing a secure water supply to the processing plant and maintaining clean water flows in the existing rivers/creeks.

The proposed open pit and site facilities are located as shown on Figure 18.3. The upper and lower sites will be connected to the existing highway M6 by a newly constructed paved road. The new intersection at M6 is presently proposed for construction between Turnovo and Sekirnik. A network of internal gravel covered roads will connect the site facilities.

A new power supply will also be constructed to support operations. This will include a 7.5 km high voltage transmission line from the existing 110 kV transmission line some 2.5 km southeast of the village of Ilovitza to the upper site substation. A medium and lower voltage distribution network will supply power from the main upper site substation to the other site facilities.

18.3 SITE FACILITIES

The property has no existing infrastructure, services or facilities. The existing access road, through the town of Ilovitza to a water reservoir and unpaved tracks beyond on the property, are currently utilised for exploration. It is anticipated that some limited service can be obtained from the local area, but the site will require the construction of additional facilities to support the mining and processing operations including the following:

- Mineral processing plant and facilities.
- Associated buildings including the administration building, workshops, stores, truck shop and maintenance building.
- Access road to site and site roads.
- Water management including fresh make-up, fire, and potable water supplies.
- Sewage collection and treatment.
- Power supply and distribution.
- Fuel storage.

It is understood that the mine area is in an active seismic zone and all facilities should be designed accordingly and form part of the engineering design requirements for future studies.

18.3.1 Lower Site

The lower site, as shown in Figure 18.4, will be primarily focused upon the mining activities and will be semi self-contained.

Access to the lower site will be via a new road constructed from the existing highway M6. Power supplies will be via a medium voltage supply from the upper site substation.

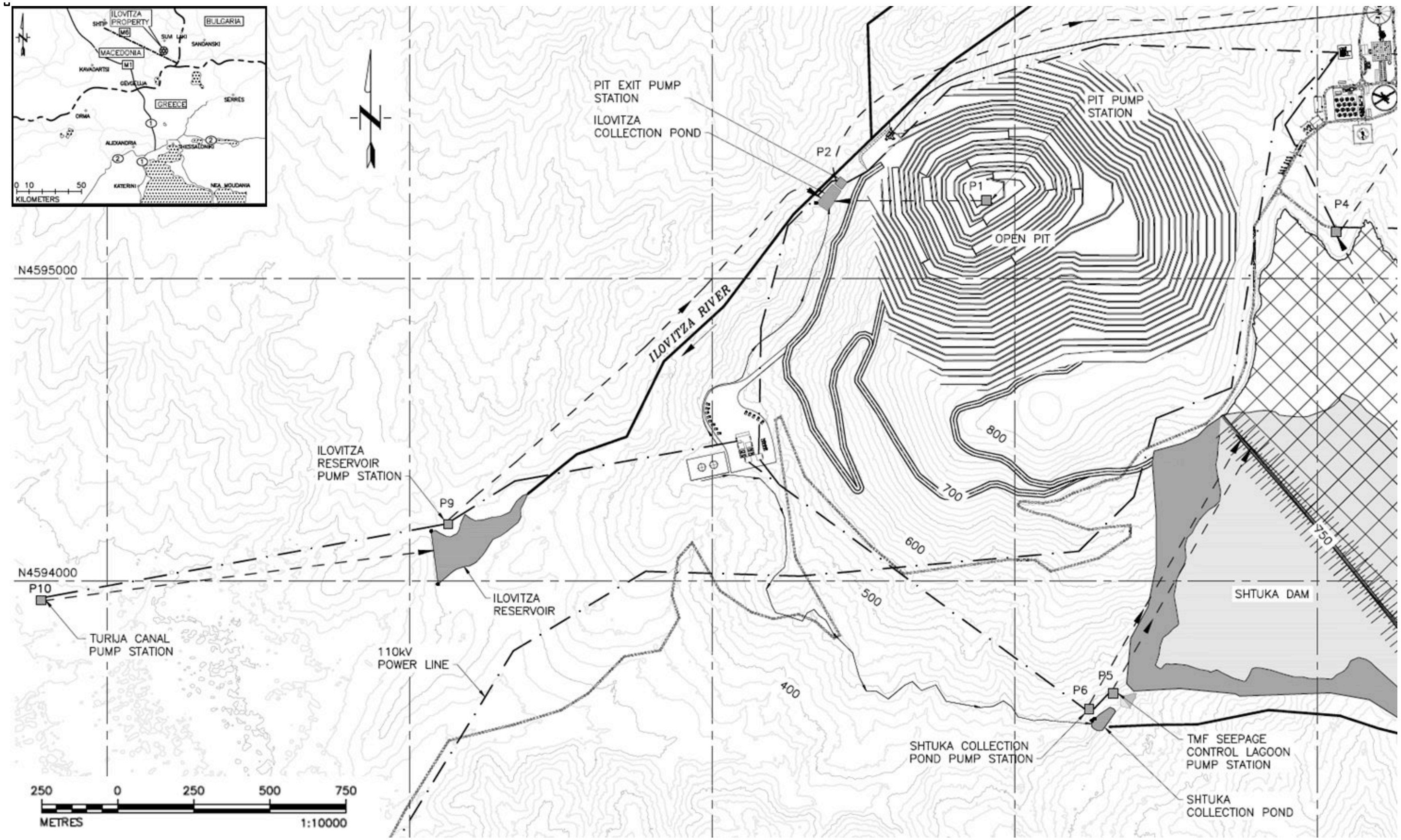
18.3.1.1 ROM Pad and Primary Crusher

The ROM Pad and Primary Crusher will be located adjacent to the pit exit in the Ilovitza valley. Haul trucks will deliver ore either directly to the primary crusher or to temporary stockpiles on the ROM pad, which will be reclaimed and delivered to the primary crusher by front end loaders. Euromax have designed the ROM pad with a 5 degree ramp with the crusher to be set in a small excavation below. The pad construction will require excavation, movement, crushing and partial compaction of material and Euromax plan for this to be carried out during the pre-production period. During this period an explosives magazine and access road will also be constructed further to the northeast (See Section 18.3.1.2). Euromax have estimated a total budget of US $6.6 million for the construction of the ROM pad explosives store and access road.

The primary crusher will be a single gyratory crusher with associate control facilities and maintenance hoists.

The crushed ore will be conveyed from the primary crusher (480 m elevation) to the crushed ore stockpile (850 m elevation) by a cable conveyor. Euromax have designed the conveyor with a 12 degree inclination and an approximate 800 m radial curve. Euromax have received a budget quotation of US $9 million from the manufacturer who also verified the design as viable. Euromax have estimated as installation cost of some US $1.2 million which has been added to the overall capital cost. Using parameters provided from the supplier Euromax have estimated a conveyor total operating cost of $0.10 per tonne of feed.

Figure 18.4 Pit and Lower Site Layout



Source: Tetra Tech

18.3.1.2 Explosives Store

The explosive store will be located in the Ilovitza valley, approximately 500 m northeast of the ROM Pad. Euromax currently plan for the store to be constructed and operated under a drill and blast contract with an explosives contractor.

Euromax have made an allowance for the cost of the store construction and access road within the US $6.6 million allowance for the ROM pad area, as described above for the crusher, see Section 18.3.1.1.

18.3.1.3 Truck Maintenance Workshop and Stores

The truck shop facility is anticipated to be equipped with an overhead crane and all the facilities necessary to maintain the mining fleet and other mobile plant. Adjacent to the truck shop, the stores facility will include both internal space and secure external storage space for larger components, tyres and capital spares.

The maintenance floor space provides areas for maintenance shop activities including welding and repair, as well as a warehouse, local offices, and associated facilities to support warehouse and truck maintenance personnel. Additional truck driver facilities will be provided for rest breaks and ablutions.

A truck parking area adjacent to the truck shop will be provided suitable for the whole mining fleet.

Further details of the maintenance workshop and facilities are given in the Mining Section 14.

18.3.1.4 Fuel Storage

Diesel fuel requirements for the mining equipment, and the process and ancillary facilities will be supplied from above-ground diesel fuel storage tanks located at the lower end of the plant site. Fuel will be pumped to vehicle refuelling points near the truck shop and at the ROM pad. The diesel fuel storage tank will have a capacity sufficient for approximately 10 days of operation. Diesel storage will consist of above-ground tanks in a bunded area for containment, complete with loading and dispensing equipment conforming to National and European regulations. The facility will be enclosed within suitable security fencing.

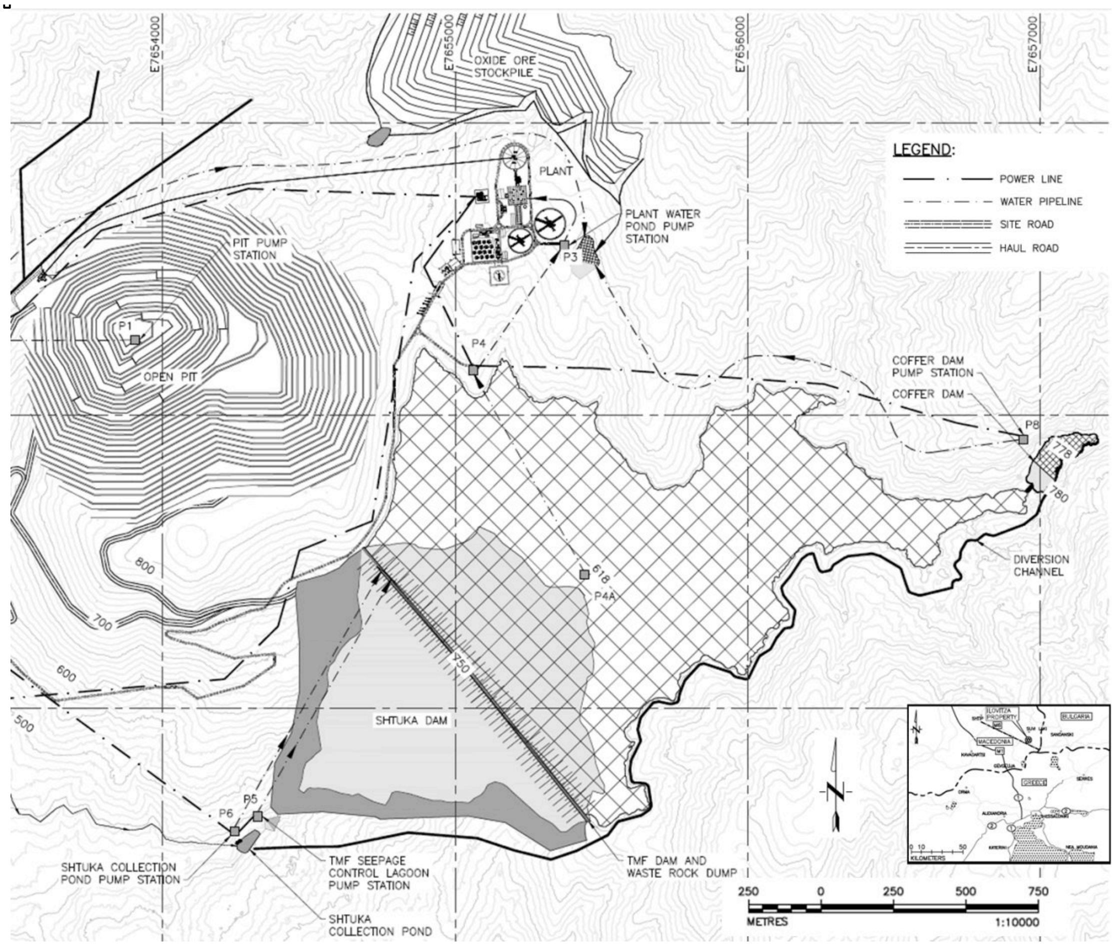
Euromax plans for the fuel storage facility to be constructed and operated under a fuel supply contract.

18.3.2 Upper Site

The upper site, as shown in Figure 18.5 will primarily be focused upon the mineral processing activities and will be self-contained.

Access to the upper site will be via a new road constructed from the existing highway M6 to the lower site and extended on up the Shtuka valley. Power supplies will be via a new high voltage supply line.

Figure 18.5 Upper Site Layout Showing the Processing Plant and TMF



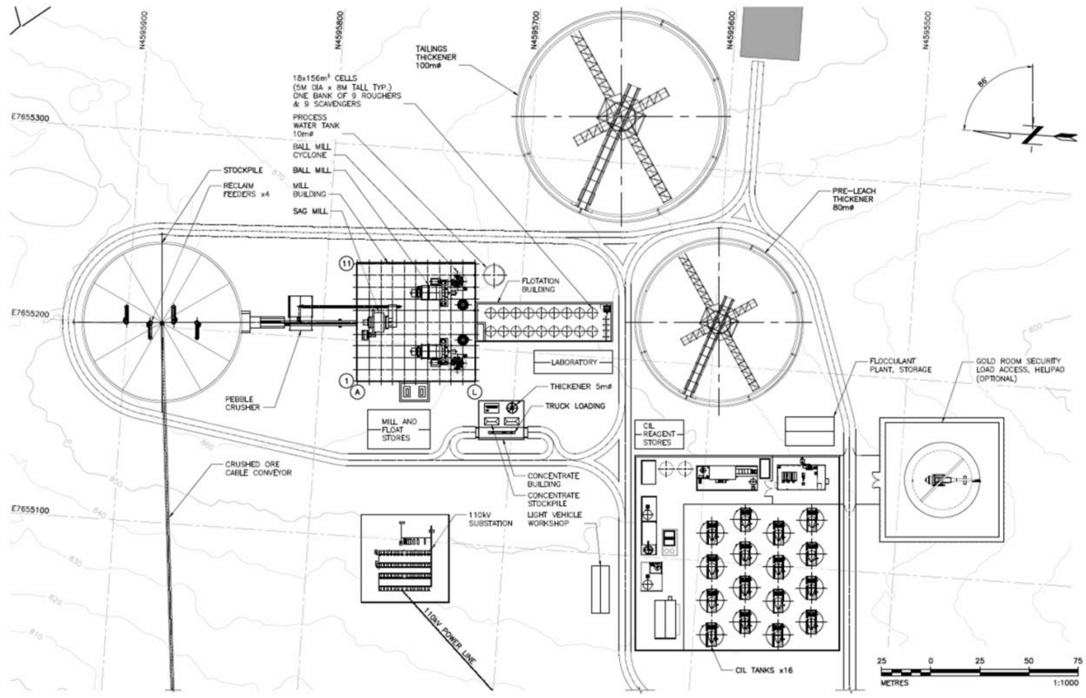
Source: Tetra Tech / Euromax

18.3.2.1 Crushed Ore Stockpile

It is anticipated that the main crushed stockpile will be situated on the higher level adjacent to the concentrator building. The crushed ore will be conveyed from the primary crusher on the edge of the ROM pad, via a cable conveyor. The crushed ore material will be reclaimed via a tunnel which will be of reinforced concrete construction, with one level of elevated steel platforms supporting four apron feeders. It is connected at one end to a conveyor tunnel of corrugated steel construction and concrete floor slab which leads to the concentrator building.

The crushed ore feed conveyor will also include a pebble crusher to allow oversized material to be reintroduced to the mill.

Figure 18.6 Process Plant Layout



Source: Tetra Tech

18.3.2.2 Concentrator Facilities

The Concentrator facilities are illustrated in Figure 18.6.

It is anticipated that the mill building will be a steel structure with a steel roof deck and wall cladding. The design of the building and process plant should make use of terracing and the natural slope of the hill side where appropriate. Heavy mat reinforced concrete foundations and piers are provided to support mills.

The flotation and CIL areas will be open and situated in bunded areas.

The Gold Room will be located in a secure area adjacent to the CIL area.

The mill building and flotation area will include overhead cranes.

The concentrator buildings will house electrical switch gear, motor control centre, control room, local maintenance workshops and a laboratory.

Separate stores buildings will be provided for the mill and float consumables and secure stores for the CIL reagents, acid and cyanide.

18.3.2.3 Product Storage and Load out

It is proposed that the concentrate will be stored inside a building for environmental and practical reasons. The concentrate will be reclaimed and loaded to trucks for transport to smelters. The stock and load-out building will be situated at the lower end of the flotation area.

Doré bars produced will be stored in secure area in the Gold Room and will periodically be transported by armoured truck by road to clients or alternatively by helicopter. (The location for a helipad is indicated on the site layout, but this is not included in the base case, which assumes road transport.)

18.3.2.4 Administration and Canteen Buildings

The administration and canteen buildings will be situated on the upper site and will also house changing rooms, a canteen, and first-aid and firefighting facilities. They will also house the administrative, engineering and geology staff.

18.3.2.5 Maintenance shop and Stores

The maintenance and light vehicle workshop facility is anticipated to be equipped with an overhead crane and all the facilities necessary to maintain the process plant facilities and other mobile plant.

Adjacent to the maintenance shop, the stores facility will include both internal space and secure external storage space for larger components and capital spares.

The maintenance floor space provides areas for maintenance shop activities including welding and repair, as well as a warehouse, offices, and associated facilities to support warehouse and maintenance personnel.

18.3.2.6 Substation

The main high voltage substation will be located near the plant and will step down the incoming supply to a medium voltage for distribution around the site.

18.3.2.7 Gate House and Security

A security gate house and weighbridge will be located at the entrance to the upper site to control access.

A truck and employee vehicle parking area will be provided outside the upper site to manage vehicle movements.

18.3.3 Water Supply and Distribution

Process water will be recirculated and pumped from the tailings facility and from the water ponds.

It is assumed that the potable water supply will be taken from boreholes and suitably filtered, treated and distributed around the site.

18.3.4 Sewage and Water Treatment

The sewage treatment plant will be designed to meet with local standards. Once treated, the sewage treatment plant effluent will be discharged into the environment in accordance with the requirements of the Environmental Impact Assessment (EIA).

18.3.5 Remote Facilities

The remote facilities will primarily consist of water pumping stations. These will be automated and un-manned, controlled from a central water management control room. Each pump station will be housed in a brick building in a fenced off area and it is anticipated that they will require regular inspections.

18.3.6 Building List

The Lower site facilities are detailed in the Mining Section 14. The main Upper site facilities are listed in Table 18.1.

Table 18.1 Upper Site Facilities

DESIGNATION	Quantity	Unit
PLANT FACILITIES		
Plant offices contingency	216	m^2
Laboratory building	546	m^2
Laboratory equipment and facilities		
Reagent & Cyanide Stores	756	m^2
Acid Stores	432	m^2
SITE FACILITIES		
Administration building	1,008	m^2
Canteen, changing rooms	288	m^2
Guard House	128	m^2
Weighbridge	1	
Security fencing	6,000	m
Fire & Ambulance garage	144	m^2
First aid facilities (inc. Admin Building)		
Light Vehicle / Maintenance workshop	216	m^2
Warehouse building for maintenance equipment, tools and materials	756	m^2
Laydown and parking areas	5,000	m^2

18.4 ROADS

The Ilovitza mine site is located approximately 187 km southeast of the Macedonian capital city of Skopje and has relatively good road access.

The road link between Skopje and Strumica, the closest town to the mine site, comprises paved highways in good condition. The initial 15 km road section from Skopje to Petrovec is the M3 national highway. The M3 merges into the M1 just south of the Alexander the Great International Airport of Skopje. The M1 is part of the main E-75 route that connects nine European countries from Norway to Greece. After 40 km on the M1 (E-75), at the town of Veles, the road to Strumica continues on highway M5 for 45 km until the town of Shtip. Shtip is connected with the Bulgarian border by national highway M6 that crosses the towns of Strumica and Turnovo. The distance between Shtip and Turnovo is approximately 80 km. The last, approximately 7 km of the journey from Turnovo to the mine site is, in the most part, a relatively narrow paved road, as pictured in Figure 18.7.

Road access is required to both the Upper and Lower sites for construction and operation purposes. The access road has to be constructed to facilitate the transport of construction equipment and facilities, including rigid frame heavy haul vehicles and tyres.

For operations the access road will be used for the delivery of plant supplies and consumables including fuel and reagents as well as operations personnel. The road will also be used to dispatch the concentrate product in 30 tonne articulated trucks and armoured vehicles for the gold doré bars.

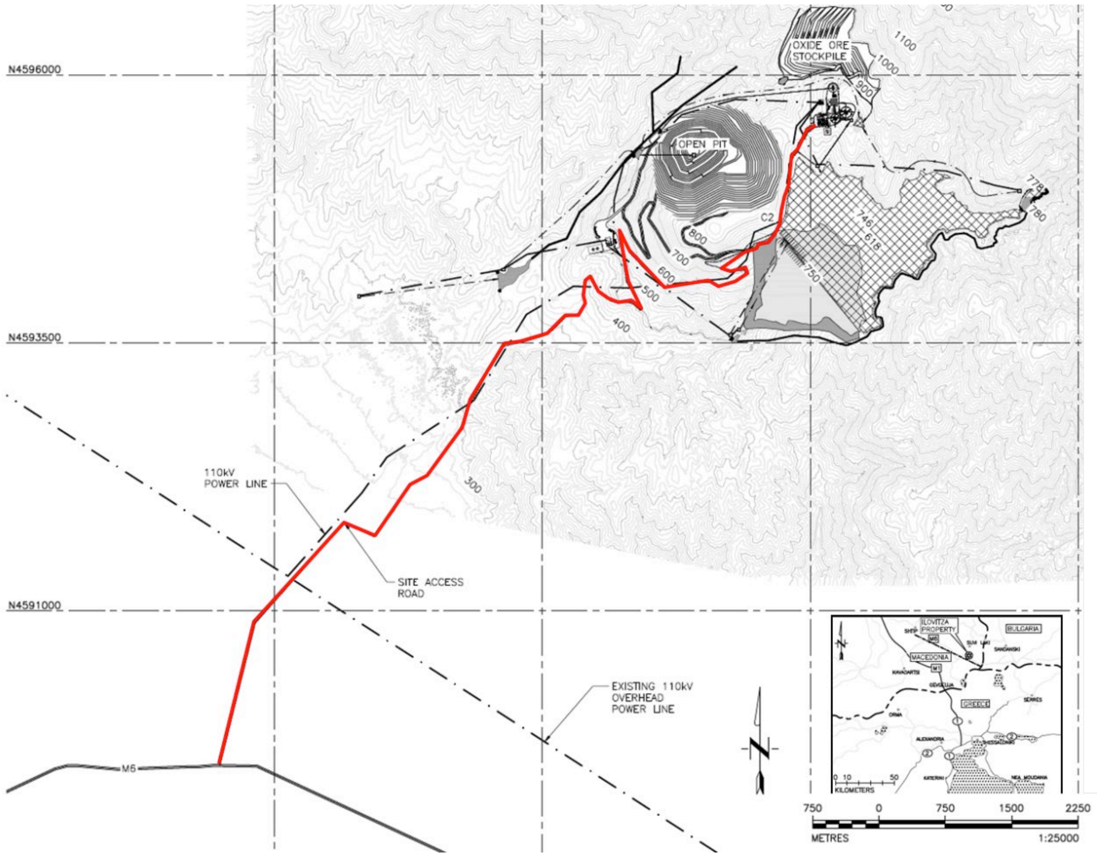
Figure 18.7 Existing Narrow Paved Road from Highway M6 to the Village of Shtuka

Source: Tetra Tech

This road, in its current state, may cause accessibility issues for large trucks, because of the width and potential load bearing issues. Further restrictions in access along this route also exist around the villages of Turnovo and Shtuka, which have narrow, unpaved streets and many pedestrians. The road between the villages is also frequently restricted by local slow moving or stationary farm vehicles. Consequently this route is not considered suitable for mine construction or operation.

In order to facilitate easy access to the mine's main plant site, it is proposed to build a new connecting road from highway M6 to the process plant. The new access road from the plant site will be constructed winding down through the Shtuka valley between the open pit and the proposed TMF site, to limit the maximum gradient to 8%. This road will pass by the fuel storage and haul truck workshop and cross the Shtuka valley (creek) over a new bridge or a culvert. The road will then bypass the village of Shtuka to the east and then follow the Shtuka creek until highway M6, west of the village of Sekirnik. The proposed alignment of the access road is shown (red line) on Figure 18.8 however the final route will need to be confirmed in subsequent studies.

Figure 18.8 Suggested Access Road Alignment from Highway M6 to the Mine's Plant Sites



Source: Tetra Tech

The existing condition at the junction of the proposed access road and highway M6 is shown on Figure 18.9. The view of the Shtuka creek next to Sekirnik is shown in Figure 18.10. Construction of the new access road may also require the realignment and clearing of the Shtuka creek between the villages of Shtuka and Sekirnik. The new road will also bypass a small wetland area, formed behind an existing concrete/rock gravity dam.

Figure 18.9 Suggested Junction Location for the Proposed Access Road at Highway M6

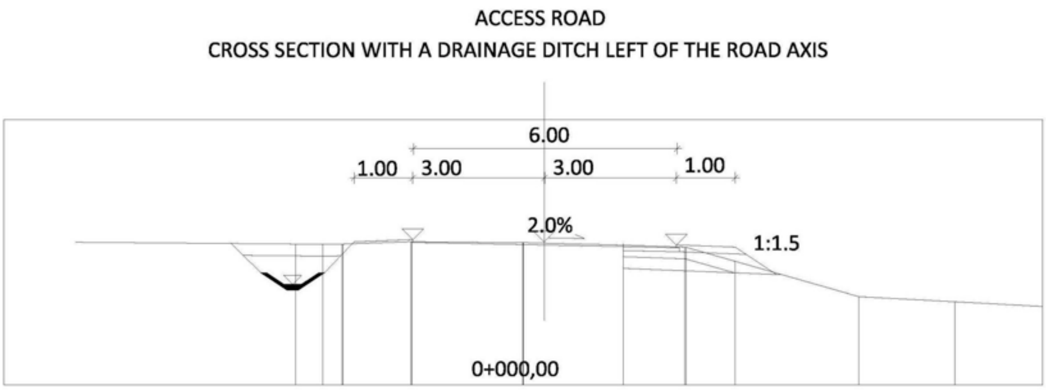
Source: Tetra Tech

Figure 18.10 View of the Shtuka Creek along the Proposed Access Road

Source: Tetra Tech

The access road is proposed with 120 mm thick asphalt concrete flexible pavement, placed over selected granular base and sub-base layers. The proposed cross section is shown in Figure 18.11.

Figure 18.11 Proposed Cross Section of the Paved Access Road

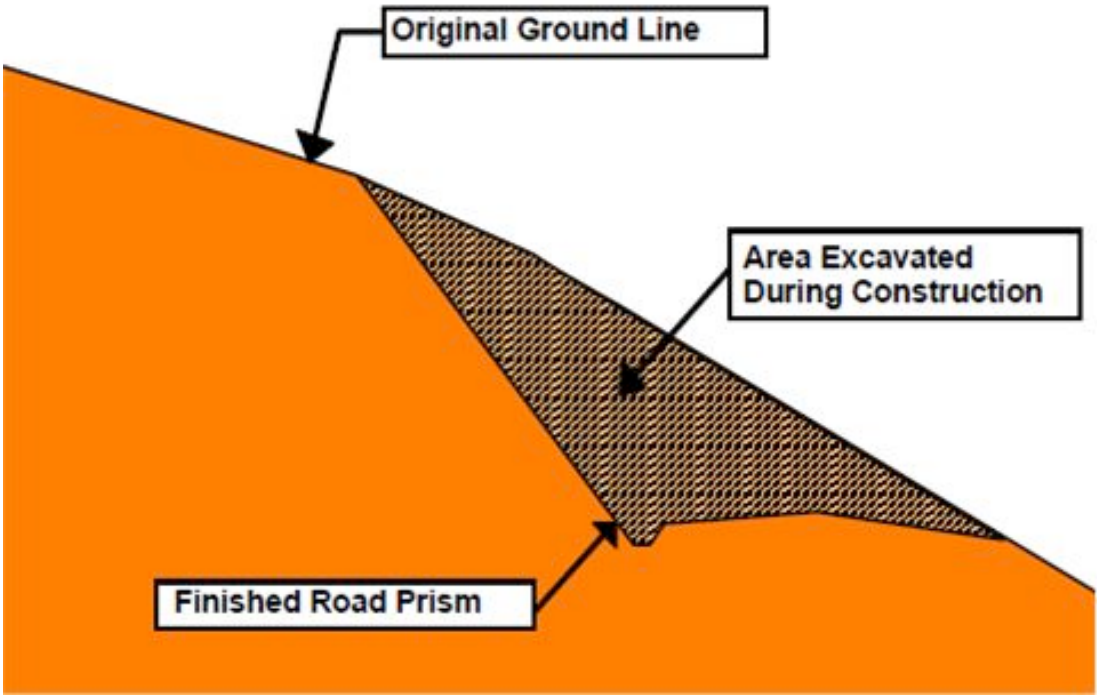


Source: Geodet, 2014

The road profile shown in Figure 18.11 can be constructed up to a maximum hillside slope angle of around 34°. This corresponds to a ratio of 1 : 1.5 (run over rise). The maximum fill slope angle, shown on the right side of this Figure, is a function of the shear strength of the soil used as embankment fill, specifically the internal angle of friction and in some cases, the cohesion.

Compacted side cast fills which must support part of the road become more difficult to construct with increasing native side slopes. For side slopes in excess of 25° to 27° (50 to 55%), the full road width should be moved into the hillside (benched construction). Excavated material can be side cast or used somewhere else, but should not form part of the roadbed.

Figure 18.12 Full Bench Type Road Construction on Steep Native Slopes



Source: Wikipedia

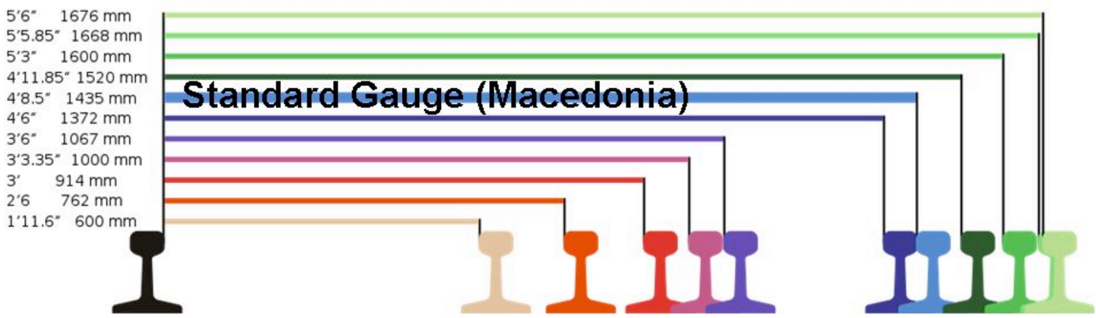
Hence, in Ilovitza, where side slopes will possibly exceed 50 to 55% around the open area of the proposed access road and unstable slope conditions may also be present, it will be mandatory to consider full bench construction, shown in Figure 18.12.

18.5 RAIL CONNECTIONS

Euromax provisionally plans to sell the concentrate product to a local smelter via a combination of road and rail transport, however the option of exporting the concentrate product by sea is retained.

The railway infrastructure in the Republic of Macedonia has been in place since 1873 when the first railway track from Skopje to Thessaloniki in Greece was constructed. Today the railways network is about 925 km in single track standard gauge lines, of which 315 km are electrified with a 25 kV 50 Hz AC system. Common railway gauges are shown in Figure 18.13.

Figure 18.13 Railway Gauges of the World



Source: Locomotive Wiki

The Macedonian railway network system is connected north-south by “Corridor X”, with the railway network systems of Serbia and Greece, as shown in Figure 18.14. Its northern point at the border with Serbia is Tabanovce. The line from that town passes through Kumanovo, Skopje, Veles, Gradsko, Negotino, Demir Kapija and finishes at Gevgelija, at the border with Greece. This railway line is electrified, modernised and runs by the E-75 motorway. A branch of this line is the Veles - Prilep - Bitola - Medzitlija line that connects these Macedonian cities with northwestern Greece and Thessaloniki. A further line is proposed in the north to Bulgaria; however, it is understood that this is still under construction.

Figure 18.14 Railway Network of Macedonia

Source: Locomotive Wiki

The railway transport system is managed by the publicly owned Macedonian Railways (Makedonski Železnici, MŽ) and at present, is the only provider of railway services in the country.

The main station in Skopje is located approximately 180 km from Ilovitza. The line’s alignment generally follows the E-75 corridor with loading sidings in Gevgelija at the Greek border. The distance between the line at Miravci and the mine site is approximately 60 km, as shown on Figure 18.14.

The Bulgarian rail network extends as far as Petrich in the south-western corner of Bulgaria, approximately 45 km from the Ilovitza site, as shown on Figure 18.15. This line also links to the Pirdorp smelter at Chelopech, the Black Sea Ports of Varna and Burgas, and also to the Mediterranean Port of Thessaloniki in Greece and to the Serbian rail network and links to Bor.

Figure 18.15 Railway Network of Bulgaria

Source: Locomotive Wiki

A direct rail link from the mine site to the existing rail network in Macedonia would require approximately 60 km of new rail line to be built from the project site to Gevgelija. However, this would have to rise up 250 m to cross the Belassica Mountains and would be a very expensive option to consider. Alternatively, to link the site to the Bulgarian rail network at Petrich would require a new 40 km rail line, although this too would be costly and would incur cross border issues, which would increase costs and construction timeframes. Consequently at present a direct rail link is not considered for the project.

The location of the Ilovitza mining operation offers the potential to export the concentrate product to smelters in Bulgaria, Serbia or overseas, via road or a combination of road and rail transportation.

Euromax's preferred option is to export the concentrate to the Pirdop refinery in Bulgaria. Euromax have commissioned a Bulgarian consultancy to complete a preliminary transport study, which has indicated that a combined truck and rail option, trucking the ore to Petrich first and then using rail transportation through Bulgaria to the Pirdop refinery (300 km), is the preferred option.

Figure 18.16 Petrich Station

Source: Tetra Tech August 2013

The station at Petrich is at the end of a branch line on the Bulgarian National rail network. The station has a number of sidings and appears to be relatively underutilised with capacity to accommodate the Euromax concentrate shipments.

It is understood that Petrich station has facilities to load rail wagons which could be adapted to load concentrate in containers. Alternatively concentrate could potentially be stocked in a secure site alongside a siding and reclaimed by frontend loader to be loaded in wagons. However, further study is required to determine the most appropriate transport method and discussions are also required with the management of Petrich Station and the Bulgarian National Rail operator, to determine the respective costs of each option.

18.6 PORT FACILITIES

Macedonia is a landlocked country with the nearest port that of Thessaloniki, located in neighbouring Greece, which is a major deep sea port on the Northern Aegean Sea. The distance between the proposed mine site and the port is approximately 140 km by road, as shown on Figure 18.17.

Figure 18.17 Location of the Deep Sea Port of Thessaloniki, in Relation to the Ilovitza Site

Source: Tetra Tech

18.6.1 PORT OF THESSALONIKI

The Port of Thessaloniki, shown in Figure 18.18, is one of the largest seaports in Greece and the Aegean Sea basin and has good road and rail links. As a free port, it functions as a major gateway for the Balkan hinterland and southeastern Europe, including Macedonia, as well as a major transhipment hub in the Aegean-Black Sea area.

The port has the capability of handling general and bulk dry cargos, liquid cargos, containers and passenger ships, with some 3,000 ships each year. The port has a total annual traffic capacity of some 16 million tonnes (Mt), of which 7 Mt is general and bulk dry cargo, 9 Mt is liquid fuel cargo, 370,000 TEUs and 220,000 passengers.

Figure 18.18 Thessaloniki Port

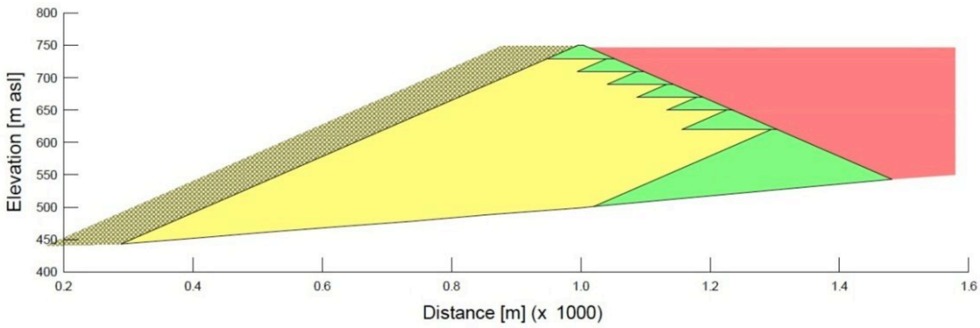
Source: Thessaloniki Port Authority (THPA)

It should be possible to export concentrate product, either as a bulk commodity or containerised, through the Port of Thessaloniki to international smelters. Concentrate could be transported by road the 140 km to the port or via a combination of road and rail transport. However, for this study it is understood that Euromax's preferred option is the sale of concentrate to smelters in the Balkans region and that export by sea is a secondary option at this stage.

18.7 WASTE ROCK DUMPS

At this stage all waste rock is proposed to be used for the construction of the tailings and water storage dams. Excess waste rock totalling some 50 Mt will be placed as a buttress on the downstream face of the TMF embankment. This has been modelled by the Faculty of Civil Engineering in Skopje and the final profile is given in Figure 18.19.

Figure 18.19 Tailings Embankment and Waste Rock Buttress



Source: Faculty of Civil Engineering, Skopje

Preliminary investigations into the acid generating potential of the waste rock have been undertaken by Euromax, although the exact ratio between Non-Acid Generating ("NAG") and Potentially Acid Generating ("PAG") waste rock has not yet been determined, however it is recommended that all water reservoir dams should be constructed from NAG rock. The central body of the large TMF dam may include PAG rock material. The anticipated contact water seepage with this dam will be collected in the TMF seepage control dam and the collected water will be pumped back into the TMF.

18.8 TAILINGS MANAGEMENT FACILITY (TMF)

The Faculty of Civil Engineering in Skopje were commissioned to provide a preliminary design for a Tailings Management Facility using the following assumptions:

- That the embankment would be where possible constructed from waste rock as provided in the mining schedule.
- Use a downstream construction approach.
- Position the embankment in the Shtuka valley so as to maximise capacity for tailings storage. Assume thickened tailings of about 60 to 65% solids.
- Drainage would ideally be with perimeter drainage channels.
- Utilise a spillway design and lagoon downstream to cope with any large precipitation events.
- Take into account the seismic conditions of the area.
- Use the known geotechnical features of the valley based on an earlier test pit programme supervised by the faculty.
- Use a single embankment.

18.8.1 General Approach to Design of TMF

Seven profiles were considered within the Shtuka valley: Three profiles at downstream locations (A1, A2, A3), at a river bed elevation of 430m amsl; profile (B) at river bed elevation of 490m amsl and three upstream profiles (C1, C2, C3) at river bed elevations of 490-550m amsl. Profiles were also considered for an upstream coffer-dam for each location (P). The volume curves for the potential dam profiles were calculated in 50 m slices relative to the vertical surface of the dam profile. Final profiles for further consideration were selected on the basis of suitable storage volume and geotechnical characteristics.

18.8.2 EMBANKMENT LINER

In selecting the type of liner for the upstream embankment face it was noted that there is no borrow location available in the vicinity for supply of clay as material for waterproof element of the dam and therefore two possible alternatives were considered, namely using asphalt core (diaphragm) or with geo-membrane facing. The choice of most favourable alternative was made by comparison of five criteria:

(1) the available quantities of mine rock, (2) the practicality for construction of the waterproof element, (3) the cost of the diversion pipe for protection of the construction pit for construction of initial dam at crest elevation of 620 m amsl (4) possibility for dam heightening, and (5) cost of the waterproof element. A geo-membrane was selected as providing the best solution on this basis.

18.8.3 GEOTECHNICAL INVESTIGATIONS AND EMBANKMENT SITE SELECTION

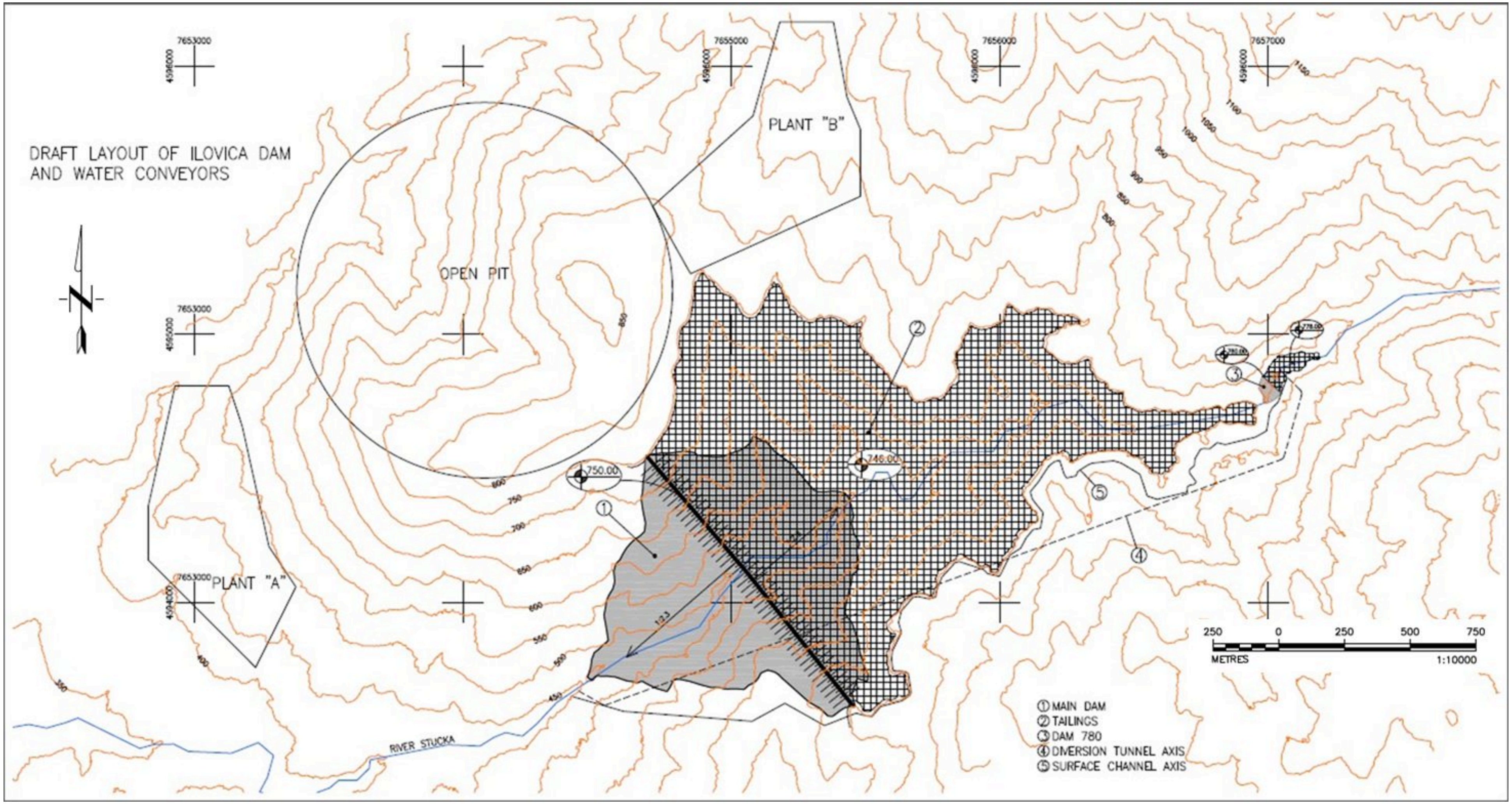
Preliminary geotechnical investigations including test pits and mapping were carried out in the Shtuka valley during 2013. The results from these investigations are mainly presenting the geotechnical characteristics of the soil overburden in the riverbed and of the lowest part of the abutments, while for the quality of the soils and rocks in the higher parts of the abutments relatively limited data had been collected, mainly due to terrain inaccessibility. For the needs of this Preliminary Feasibility level of Design these data have been appropriate, however for more detailed stages of Feasibility and Basic Engineering, a higher level of geotechnical survey will be required. The conclusions from the analysis of the geotechnical data are the following:

Dam profiles (C2) and (P) are appropriate for construction of conventional dam types, and the soil and rock medium has significantly higher strength compared to other local areas and so in the structural analysis can be treated as stiff and non-deformable.

In the preliminary structural analysis of the dams for profiles (C2) and (P) the following material parameters can be adopted for the filter layer and dam shell: Bulk unit weight of 20.0 kN/m^3 (natural water content), and 21.0 kN/m^3 (saturated unit weight). The angle of internal friction of the soil overburden may be estimated at 33º.

The final layout for the embankment profiles is given in Figure 18.20 below

Figure 18.20 Tailings Facility Layout



Source: Faculty of Civil Engineering, Skopje

18.8.4 Water Diversion

The maximum flood inundations were determined for 1,000 year rainfall events by the application of the Synthetic Hydrograph Method - SHM, in accordance with guidance from the Soil Conservation Service, USA. The time duration of the raise of the inflow hydrograph is obtained through regional analysis for the catchment of the river Vardar. Local base line readings are being taken at the moment by the Company and will be applied to designs during the Feasibility Study stage of design. Given the dams location, at Feasibility Study stage it will likely be designed for storm two thirds 1,000 year storm and Probable Maximum Flood.

Two alternatives were considered for capturing of the flood from the catchment area upstream of the coffer dam both during the mine operation and also after mine closure, namely via diversion tunnel or via diversion channel. A diversion channel was the approach that was found to be the most cost effective. For the diversion channel a coupling is required between the coffer-dam (P) and the diversion channel itself. The hydrodynamic parameters of the flow for this channel, for a discharge of 36.8 m^3/s, were calculated using non-uniform gradually variable flow estimated from the free water table. A chute is required from the top of the dam to the terminal structure downstream of the toe of the structure.

For conveying of the construction water during construction of the diversion dam at profile (P), for discharge of 4.2 m^3/s and for the starter dam at profile C2, for a discharge of 4.5 m^3/s a coupling between the diversion pipe and the upstream coffer-dam is required. The dimensions of the diversion pipes for flood protection were calculated by hydraulic analysis of pressure flow. The thickness of the internal steel lining is calculated using a cylinder formula. The thickness of the external reinforced-concrete lining of the diversion pipes, as well and thickness of the reinforced-concrete linings of the all remaining hydraulic structures were estimated so as to be structurally sound.

A seepage water collection system is included and seepage water will be stored in a temporary storage lagoon before being pumped back to the main TWF storage area and will contribute to water circulated back to the process plant to contribute to make up water.

18.8.5 Embankment Stability

The stability of the main dam embankment has been verified using the Limit Equilibrium Method (LEM) for the downstream slope, for first stage of construction and before lake filling, as well and for final stage of the dam construction and filling with tailings. Once the facility is filled the pore pressure coefficient of the flotation tailings is adopted at ru = 0.4. The seismic safety has been checked by application of pseudo-static method for the downstream slope at final stage of dam construction and tailings lake filling with a seismicity coefficient Kc = 0.15.

18.8.6 Tailings Properties

The adopted tailings properties are summarised in Table 18.2.

Table 18.2 Tailings Properties For the Slurry (pulp) Density of 1.64 t/m^3 or 62% Solid in Weight

Characteristics of Flotation Tailings	Adopted Amount of Tailing Qjal = 31,609.6 t/day
Mass of solid phase, t/day	31,609.6
Mass ratio S: L (Solid : Liquid)	1 : 0.613
Mass of liquid phase, t/day	19,373.63
Mass of pulp, t/day	50,983.23
Volume of solid phase in the pulp, m^3/day	11,707.26
Volume of liquid phase in the pulp, m^3/day	19,373.63
Volume of the pulp, m^3/day	31,080.89
Pulp density, t/m^3	1.64
Volume ratio S : L	1 : 1.655
Pulp flow: Hour, m^3/hour Minute, m^3/min. Second, l/s	 1,295.04 21,584.00 359.73

18.8.7 Embankment Construction Schedule

The embankment will be constructed with mine waste rock. Table 18.3 below shows the schedule of filling using a combination of crushed and compacted (engineered) fill and engineered ballast and the storage volume achieved with each lift. It should be noted that an additional volume of some 30 Mt of tailings was achieved by increasing the height of the dam by 10m.

Table 18.3 TMF Embankment Staged Development

	Initial	Sustaining						
Year required by end	-1	0.5	2	3	5	8	11	15
Engineered fill (Mm^3)	4.2	1.3	0.8	1	1.1	1.1	1.3	0.423
Ballast (Mm^3)		4.2	4.9	6.5	8.2	10	12.9	8.03
Waste required (t)	8.4	10.16	10.42	13.7	16.96	20.2	25.82	15.3
Capacity gained (Mm^3)	5.2	9	9.7	13.2	17.3	21.9	27.4	15.8
Cumulative capacity gained (Mm^3)	5.2	14.2	23.9	37.1	54.4	76.3	103.7	119.5
ROM Tonnage accommodated (t)	7.8	13.5	14.55	19.8	25.95	32.85	41.1	23.7
Cumulative ROM tonnage accommodated (t)	7.8	21.3	35.85	55.65	81.6	114.45	155.55	179.25
Percentage of Sustaining Engineered fill		19.70%	12.12%	15.15%	16.67%	16.67%	19.70%	

18.8.8 Tailings Distribution System

Euromax has designed a tailings distribution system from the thickener underflow comprising a header tank, pumping system using centrifugal pumps and a piping distribution network allowing two distribution pipes to the upper and lower parts of the TMF. The distribution system is gravity assisted and pressure relief valves are included to regulate flow.

18.8.9 Estimated Costs for the TMF

The costs in Table 18.4 have been developed by the faculty of Civil Engineering in Skopje and verified by Euromax and Construction group Geing of Macedonia. An additional US $3.1 million has been allowed for the distribution system.

Table 18.4 TMF Estimated Costs

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15
Coffer dam	€ 470,274															
Grouting coffer dam	€ 90,275															
Starter dam embankment	€ 13,679,119															
Phased embankment	-	€6,165,646	€3,794,244	€4,742,805	€2,608,543	€2,608,543	€1,739,028	€1,739,028	€1,739,028	€2,055,215	€2,055,215	€2,055,215	€251,245	€251,245	€251,245	€251,245
Crushing of Engineered fill	€ 480,000	€148,571	€91,429	€114,286	€62,857	€62,857	€41,905	€41,905	€41,905	€49,524	€49,524	€49,524	€12,086	€12,086	€12,086	€12,086
Grouting embankment **	€ 300,000	€150,315	€92,502	€115,627	€63,595	€63,595	€42,397	€42,397	€42,397	€50,105	€50,105	€50,105	€6,125	€6,125	€6,125	€6,125
Temporary coffer dam,	€ 2,984															
Diversion for (C2) profile*	€ 902,705															
Temporary coffer dam*	€ 3,342															
Diversion for (P) profile*	€ 211,706															
Diversion channel*	€ 4,546,308															
Chute from (C2), *	€ 1,721,580															
Crusher	€ 3,928,571															
TOTALS Euro	€ 26,336,865	€6,464,533	€3,978,174	€4,972,718	€2,734,995	€2,734,995	€1,823,330	€1,823,330	€1,823,330	€2,154,844	€2,154,844	€2,154,844	€269,456	€269,456	€269,456	€269,456
TOTALS DOLLAR	$36,871,611	$9,050,346	$5,569,444	$6,961,805	$3,828,993	$3,828,993	$2,552,662	$2,552,662	$2,552,662	$3,016,782	$3,016,782	$3,016,782	$377,238	$377,238	$377,238	$377,238

*Increased from Faculty of Civil Engineering estimate by 10% to accommodate raised dam.

**Calculated pro rata for final lift.

18.8.10 TMF Review

The co-author and responsible QP of this section has reviewed all the input data and they were found to be reasonable and to comply with normal industry practice. Some minor technical variations were tested, which had no effect or led to very minor improvements and none had a material negative effect. The various cost build-ups by Skopje University contain minor arithmetical errors but these have not significantly affected the overall estimates.

18.9 OXIDE ORE TEMPORARY STOCKPILE

A design for the oxide stockpile has been developed by Euromax together with the Faculty of Natural and Technical Sciences at the University of Stip, Macedonia. The oxide dump was designed to accommodate the oxide material above cut-off grade mined throughout the mine life and to be processed once all the sulphide and transition material is exhausted. The total amount of oxide material that the stockpile is required to accommodate is 16.2 Mt.

The main points of the design of the current study are as follows:

- Stockpile location.
- Stockpile formation.
- Erosion and drainage water control measures.
- Bill of quantity/cost estimation.

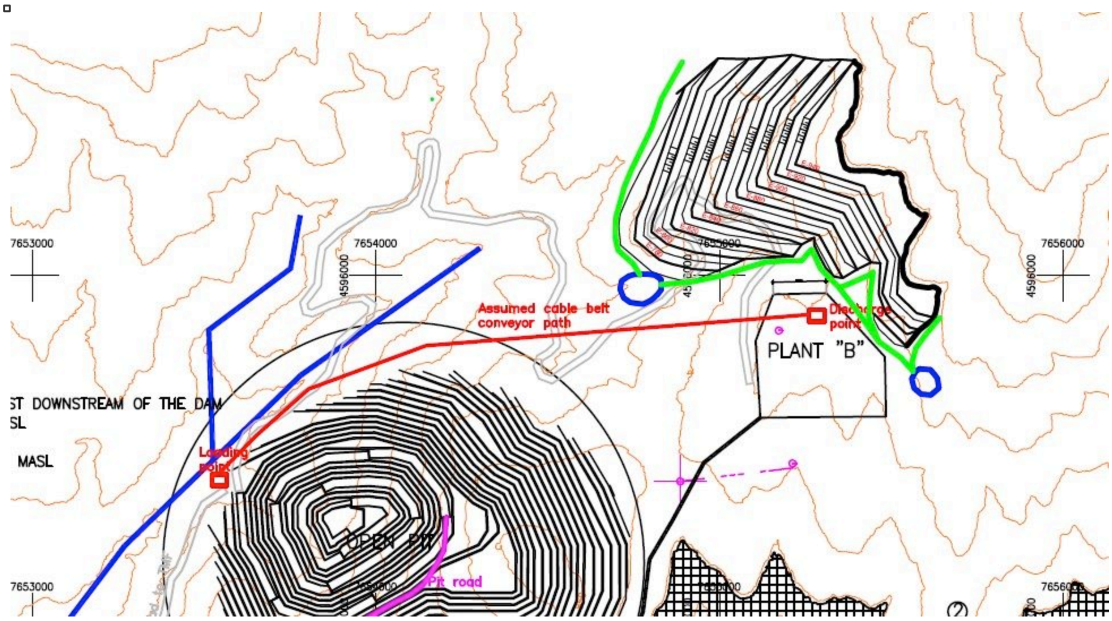
The following issues are also addressed in the design of the oxide ore stockpile:

- Surface water management facilities.
- Groundwater protection features including basal drains.
- Stability.
- Closure geometry.
- Possible closure covers to control air entry, limit water infiltration, and hence limit seepage.

18.9.1 OXIDE STOCKPILE LOCATION

The location of the temporary storage pile for oxide ore is selected on a basis of the planned ore transportation system and future processing plant location. The site selected by Euromax is above the processing plant up against the hill.

Figure 18.21 Proposed Oxide Stockpile Location



Source: Faculty of Civil Engineering, Skopje

18.9.2 Oxide Stockpile Design and Material Placement

Oxide material placement is assumed to be by contractor for this study. Material will be placed along the side of the hill in six benches with an average height of 20 m each. The minimum bench width is set to 25 m in order to reduce the general slope and keep the stockpile slopes as safe as possible. This geometry will also allow for recovery of the material at the end of mine life.

The operation of the stockpile involves the following activities:

- Ore transport from the mine to the transfer station by conveyor.
- Off-loading of the ore at the transfer station and loading on trucks by wheel loader.
- Transport the ore to the stockpile site (an average distance of approx. 2000 m).
- Dump in accordance with the planned stockpile development and operating plans including lift height and location.

The continuing disposal site preparation, which will include:

- Access road construction and maintenance.
- Clearing of new areas for dumping, foundation preparation and drain construction as required in new areas.
- Maintenance, upgrade, and expansion of surface water management facilities.
- Environmental monitoring of conditions around the stockpile including seepage water, surface water, groundwater quantities and quality.
- Performance monitoring and documentation including stability, erosion, consolidation, and creep.

18.9.3 Oxide Stockpile Environmental Control Measures

In order to provide proper runoff management and prevent water penetration inside the stockpile, diversion ditches surrounding the stockpile from the upper side are envisaged. The trapezoidal diversion ditches are designed with a bottom width of 50 cm and 60 cm height. The water from these ditches is clean and can be directed toward natural water flows or to the plant water collection system in order to be used as processing water.

In order to collect the possible drainage waters coming from the stockpile itself, concrete lined collection channels are planned both below the stockpile and also surrounding the structure on all three sides are envisaged. The shape of these concrete channels is also trapezoidal and size is provisionally set to bottom width of 50 cm and 60 cm height.

Drainage water collected should be directed to the flotation tailing pond where high alkalinity will be used to settle the pollutants and reuse water in the system. It is recommended that some retention space for this water stream is constructed in order to avoid possible spillages during the extreme precipitations.

In order to reduce the intensity of any further oxidation processes, it is recommended that the stockpile surface is sprayed with binders. The binders will create surface crust and thus reduce water and air movement through the pile.

18.9.4 Oxide Stockpile Cost Estimation

Table 18.5 details the estimated cost per year of constructing and maintaining the oxide stockpile, including the placement of material based on using a Macedonian contractor. The Faculty of Natural and Technical Sciences at the University of Stip estimated costs based on local experience from current operations in the country.

Table 18.5 Oxide Dump Cost Estimation

Description	Unit	Quantity	Unit Price [€]	Total Price [€]	Total Price [$]
Preliminary Works					
Land survey, mark, peg and safeguard the microsites	ha	48.00	40.00	1,920	2,688
Cut trees, bushes, etc. to clear and prepare the dump site; effective area: 33% of the total dump area.	m²	16,000.00	0.50	8,000	11,200
Preparation of roads including dozing, grading and excavation where necessary including loading, transport and disposal of the excavated topsoil (for reuse in future reclamation).	m³	1,000.00	3.00	3,000	4,200
Total Preliminary Works carried to Summary				*12,920*	*18,088*
Operation					
Loading, local transport and disposal of the excavated material onto the designated fill areas for levelling. All works should be done as per drawings and technical specifications.	m³	10,820,000	0.80	8,656,000	12,118,400
Mechanical levelling (dozing) dump body to achieve the designed levels. All works should be done as per drawings and technical specifications.	m²	1,176,744	0.20	235,349	329,488
Environmental monitoring.	per year	23	8,000.00	184,000	257,600
Performance monitoring.	per year	23	10,000.00	230,000	322,000
Total Dump Reshaping Works				*9,305,349*	*13,027,488*
Environmental Control Measures					
Water control measures					
Excavation of trapezoid earth channel - diversion ditch (bottom width – 50 cm, height – 60 cm) for rainwater interception (clean).	m	2400	15	36,000	50,400
Construction of trapezoid concrete channel - (bottom width – 50 cm, height – 60 cm) for drainage water collection.	m	1500	200	300,000	420,000
Erosion control					
Spraying stockpile surface with binders lump sum/per year.		23	50000	1,150,000	1,610,000
Total Dump Works Carried to Summary				*1,486,000*	*2,080,400*
Total Health and Safety Measures Works				*148,600*	*208,040*
Total				10,952,869	15,334,016

18.10 WATER

18.10.1 WATER MANAGEMENT SCHEME

The water management scheme has been developed to provide a continuous supply of process water for the process plant, manage the flow of dirty water and control the release of clean water to the rivers and creeks.

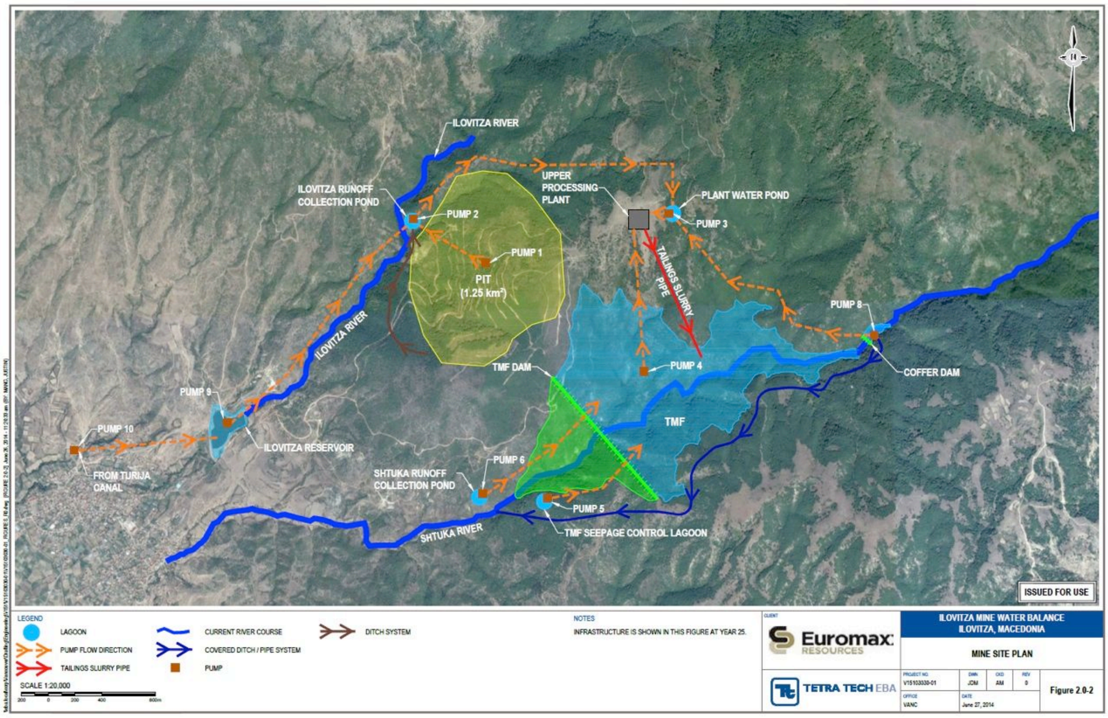
The process plant discharges tailings with some 20,000 m^3 of water per day, which allowing for plant availability and utilisation equates to some 6.6 Million m^3/year. It is understood from the TMF designers that approximately half of this water (or 3.5 Million m^3/year) will be retained within the settled tailings. However, it is understood that further tailings testing is required at the next stage of study to determine the tailings settling characteristics. For the purposes of this study it has been assumed that some 3.1 Million m^3/year of tailings water will be available for reclaim, with a net additional water requirement of some 3 Million m^3/year.

The sources of additional water considered include:

- Pit Inflow water. As the open pit progresses more water will seep into the pit from the surrounding ground water. For pit slope stability, it is proposed to include horizontal drains in the pit walls as the pit develops to depressurise the pit walls. Pit inflows are expected to rise to some 2 Million m^3/year by the end of the life of mine. In addition to ground water the pit will also receive rainfall into the pit, which in the winter months will exceed evaporation. All water entering the pit is considered as dirty water and will be pumped to the process plant as a priority for use in the process. Excess pit water will be stored either in the plant reservoir or the TMF.
- Rainfall on the TMF dam, waste rock dump and haul roads will also contain particulates and can be considered “dirty”. This surface water will be collected in collection ponds in the Ilovitza and Shtuka valleys and then pumped to the plant or TMF for use in the processing plant.
- The Shtuka valley will also receive rainfall upstream of the TMF which will be captured behind a lined dam. The dam will be used to control the release of water to a covered canal that will divert water past the TMF and will maintain a minimum flow of water in the Shtuka River of some 50 m^3/hr. Excess water will be stored in the reservoir behind the dam and can be pumped to the plant for process water requirements.
- The availability of water will vary throughout the year due to rainfall and evaporation. To manage these fluctuations it is intended to use the TMF as a temporary storage facility. The TMF will be constructed to maintain a minimum 3 m of freeboard and with a water storage capacity of some 1,000,000 m^3.
- Further sources of water are available from the existing Ilovitza reservoir and the Turija Canal. When additional water is required it can be pumped from the Ilovitza reservoir and in turn the water level in the Ilovitza can be maintained by pumping water from the Turija Canal. It is understood that water extraction from these water sources will be charged at some 5 Denar/m^3 (US $0.11/m^3). Preliminary discussions with the local water authority have suggested that the Turija Canal could supply some 2 to 3 Million m^3/year, which should be sufficient to support the mining project.

The water management scheme is illustrated in Figure 18.22.

Figure 18.22 Indicative Water Management Scheme



Source: Tetra Tech EBA

18.10.2 WATER MODELLING

To assess the water requirements from the different water sources a Goldsim site surface water model was prepared. This also took into account the seasonal rainfall and modelled the water pumping requirements on a monthly basis for the project for the first and final years of operation.

The model has used the available ground water data and local and regional historical climate information, however there are some discrepancies and further information will be required from site hydrology and hydrogeology studies to improve the accuracy of the model.

The resulting predicted monthly pumping rates in the first years of operation assuming an average year, are given in Table 18.6

Table 18.6 Average Yearly Monthly Pump Rates (First Year of Operation)

Month	First Year of Operation: Pump Rates (m^3/month)									
	Pump 1	Pump 2	Pump 3	Pump 4	Pump 5	Pump 6	Pump 7	Pump 8	Pump 9	Pump 10
January	-	136,191	160,832	348,325	-	8,883	-	24,504	127,599	27,249
February	-	200,216	219,407	289,758	-	11,443	-	19,163	192,782	84,732
March	-	178,399	206,982	302,183	-	7,498	-	28,630	169,199	51,082
April	-	258,391	295,024	214,150	-	14,120	-	36,879	248,407	103,982
May	-	237,441	281,241	227,925	-	9,122	-	44,252	227,774	84,916
June	-	329,141	373,641	135,525	-	10,718	-	45,163	323,482	198,282
July	-	385,582	414,674	94,492	-	5,462	-	29,578	379,207	293,491
August	-	309,266	335,991	173,183	-	9,632	-	26,908	302,453	225,391
September	-	165,614	181,103	328,067	-	9,067	-	15,422	155,665	108,149
October	-	139,266	163,007	346,158	-	15,854	-	23,543	128,132	38,666
November	-	106,141	120,102	389,058	-	12,588	-	13,673	92,831	42,307
December	-	142,841	173,882	335,283	-	10,234	-	30,918	135,716	22,582
Annual (m^3/year)	**-**	**2,588,486**	**2,925,886**	**3,184,108**	**-**	**124,621**	**-**	**338,633**	**2,483,246**	**1,280,828**

Source: Tetra Tech EBA

In the final years of operation it is anticipated that in an average rainfall year the project would need to pump some 0.3 Million m^3 from the Turija canal, largely in July and August. However even in a dry year this would only rise to 1.3 Million m^3. The model results assuming an average year, are given in Table 18.7

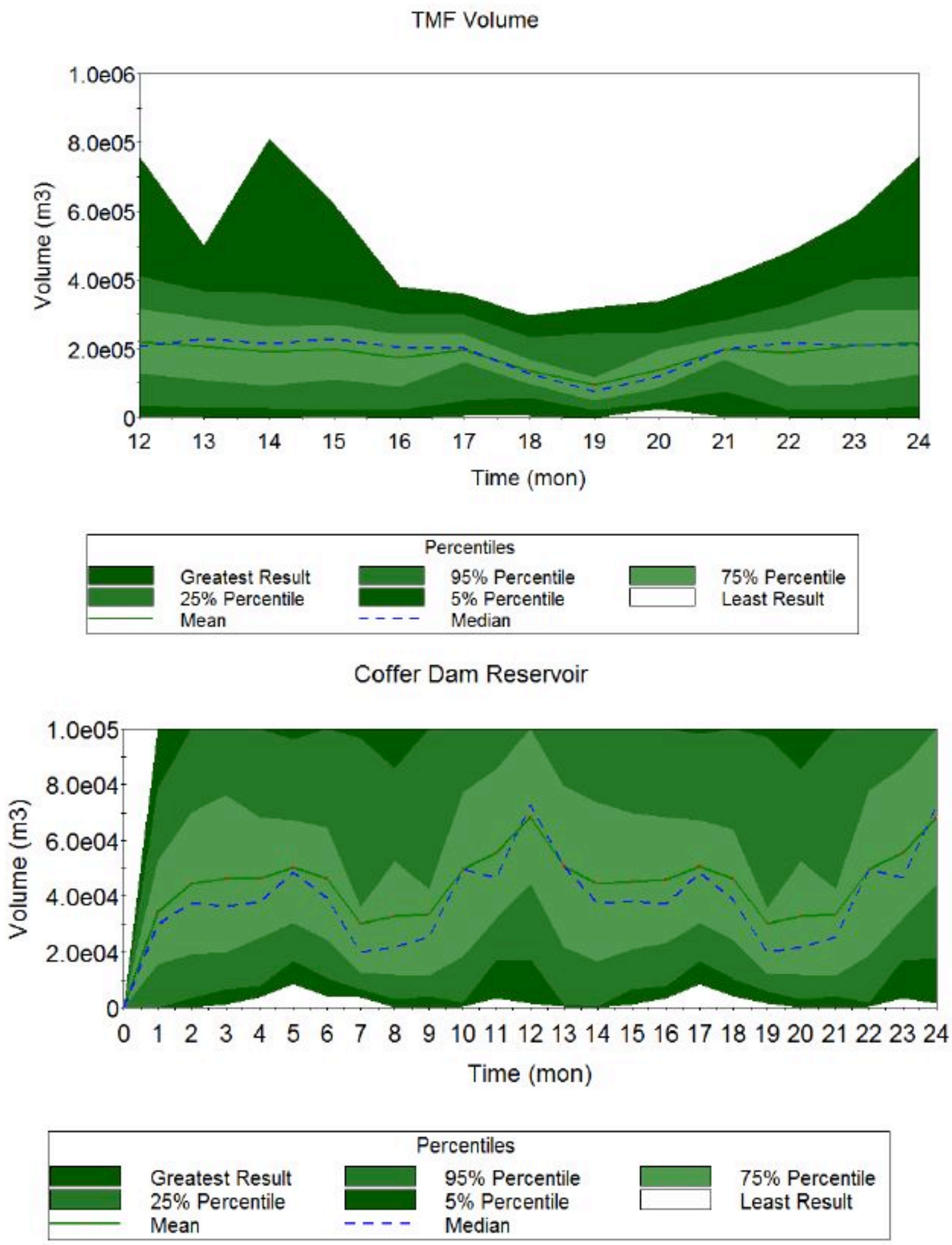
Table 18.7 Average Yearly Monthly Pump Rates (Final Year of Operation)

Month	Final Year of Operation: Pump Rates (m^3/month)									
	Pump 1	Pump 2	Pump 3	Pump 4	Pump 5	Pump 6	Pump 7	Pump 8	Pump 9	Pump 10
January	223,517	227,874	228,116	281,058	-	8,810	-	96	1,448	0
February	195,133	201,999	202,141	307,025	-	10,558	-	120	4,406	374
March	177,325	191,307	192,366	316,800	-	6,707	-	1,097	11,001	349
April	127,158	162,341	166,216	342,958	-	14,124	-	4,122	32,075	3,074
May	75,464	175,549	200,474	308,692	-	8,819	-	25,373	97,224	8,441
June	30,494	255,499	303,416	205,742	-	10,288	-	48,577	223,599	53,782
July	67,999	360,274	390,991	118,175	-	5,806	-	31,202	290,507	132,182
August	142,550	302,507	318,182	190,975	-	8,816	-	15,860	157,850	93,082
September	205,475	235,374	241,341	267,825	-	9,674	-	5,899	26,591	2,932
October	237,875	244,932	245,291	263,875	-	14,718	-	163	3,274	2,574
November	260,900	266,849	267,266	241,892	-	12,658	-	128	1,390	591
December	219,833	222,491	222,649	286,517	-	9,534	-	34	237	57
Annual (m^3/year)	**1,963,724**	**2,846,995**	**2,978,445**	**3,131,533**	**-**	**120,513**	**-**	**132,670**	**849,601**	**297,286**

Source: Tetra Tech EBA

Figure 18.23 illustrates the monthly variation in the volumes of water stored in the TMF and in the water coffer dam respectively. This indicates that in an average dry year there would be less than 200,000 m^3 stored in the TMF at any one time, although in a significantly wet year this could peak at 800,000 m^3. The water reservoir upstream from the TMF on the other hand indicates that the release of a minimum of 50 m^3/hr should be able to be maintained for all but the driest years.

Figure 18.23 Year 1 – Monthly Variations in Water Storage Volumes in the TMF and Water Reservoirs



Source: Tetra Tech EBA

18.10.3 Water Management Operating Costs

Based upon the required volumes of water to be pumped from each water source for an average rainfall year, it has been estimated that the average water cost in the initial year will be US $0.15/m^3$, although this could rise to US $0.23/m^3$ in a dry year.

18.11 DAMS

18.11.1 TAILINGS AND WATER STORAGE DAMS

Six new earth-fill dams are proposed for construction at the Ilovitza mining project:

- One main TMF dam.
- A main coffer dam, constructed upstream from the TMF area.
- One seepage water collection dam, immediately downstream from the TMF dam.
- Two surface water collection dams. One planned further downstream from the seepage water collection dam in the Shtuka valley, and one in the Ilovitza valley.
- One water storage dam, located immediately adjacent to the process plant.

In addition to the six new earth-fill dams, there will be other small run-off collection ponds and diversion channels as required to manage the rain water run-off from such areas as the Oxide Ore Stockpile.

The existing Ilovitza water reservoir dam, located next to the water treatment plant in the Ilovitza valley will also be utilised in the mine water supply and management system. It is believed that the capacity of this water reservoir is currently reduced due to a build-up of sediment, which may require dredging. It is recommended that a survey of the reservoir and the stability of this dam should be reviewed during the detailed design phase, prior to any dredging of this existing reservoir.

The TMF and associated Dams and Water management facilities are described in detail in Section 10.

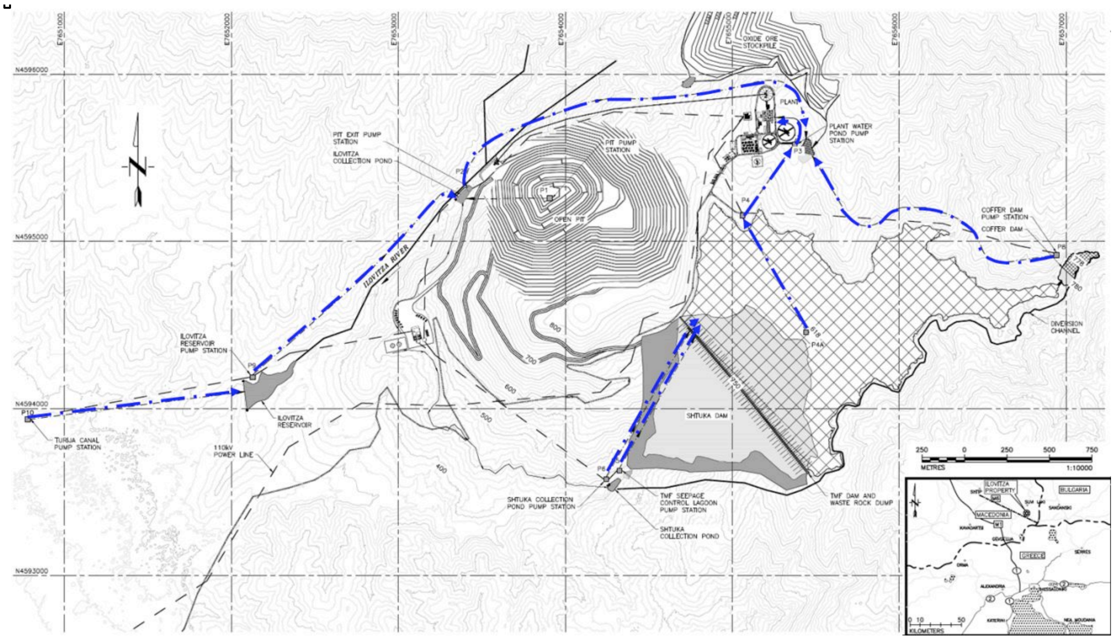
The new water dams will be constructed from local overburden material, stripped during the development of the mine site and from Non-Acid Generating (NAG) waste rock, excavated from the open pit. The soil and rock fill materials will be placed in thin lifts and compacted with a heavy vibratory roller, in order to increase the fill's dry density and internal shearing resistance. The height and crest elevations of the dams will vary significantly across the site. Given the geographic location in a seismically active region, the side slopes of the dams are presently suggested with a safe inclination of 1 m vertical lift to 3 m horizontal splay on both sides, to meet good practice requirements in the European Union, in accordance with sustainability requirements and with International Commission on Large Dams (ICOLD) guidelines. The upstream face of the dams is proposed to be covered with High Density Poly Ethylene (HDPE) or other appropriate liner, in order to prevent water seepage through the dams. Curtain grouting of the underlying weathered rock zone beneath some of the dams may also be required, in order to prevent water seepage from the water reservoirs. Alternatively, a lined base of the water reservoirs may be considered instead of curtain grouting. The final decision about the seepage control alternative should be made during the feasibility design stage.

At this stage all waste rock is proposed to be used for the construction of the tailings and water storage dams. Excess waste rock will be placed on the downstream face of the TMF. This is described in more detail in 18.7.

18.12 PIPELINES

The water supply scheme has been developed to provide a continuous supply of process water for the process plant. A corresponding water pumping and pipeline scheme has been developed which is illustrated in Figure 18.24. This will require some 16 km of pipelines.

Figure 18.24 Water Pumping and Pipeline Scheme



Source: Tetra Tech

The pipelines will be constructed using high pressure HDPE and either buried in trenches or run at surface level, secured on mounting blocks. The pipeline diameter will vary depending upon the required flow rate.

The pipeline from the Turija Canal - Ilovitza Reservoir - Ilovitza Collection pond to the plant will be designed to supply the full plant demand of 826 m³/day to enable the plant to start up and operate in an emergency. It is anticipated that these pipelines will be 315 mm diameter.

Return water from the TMF will be collected by a floating platform and pump station in the centre of the TMF area which will be connected to a fixed pump station at the edge of the TMF. Periodically as the TMF is raised, this pump station will have to be relocated and moved up the valley.

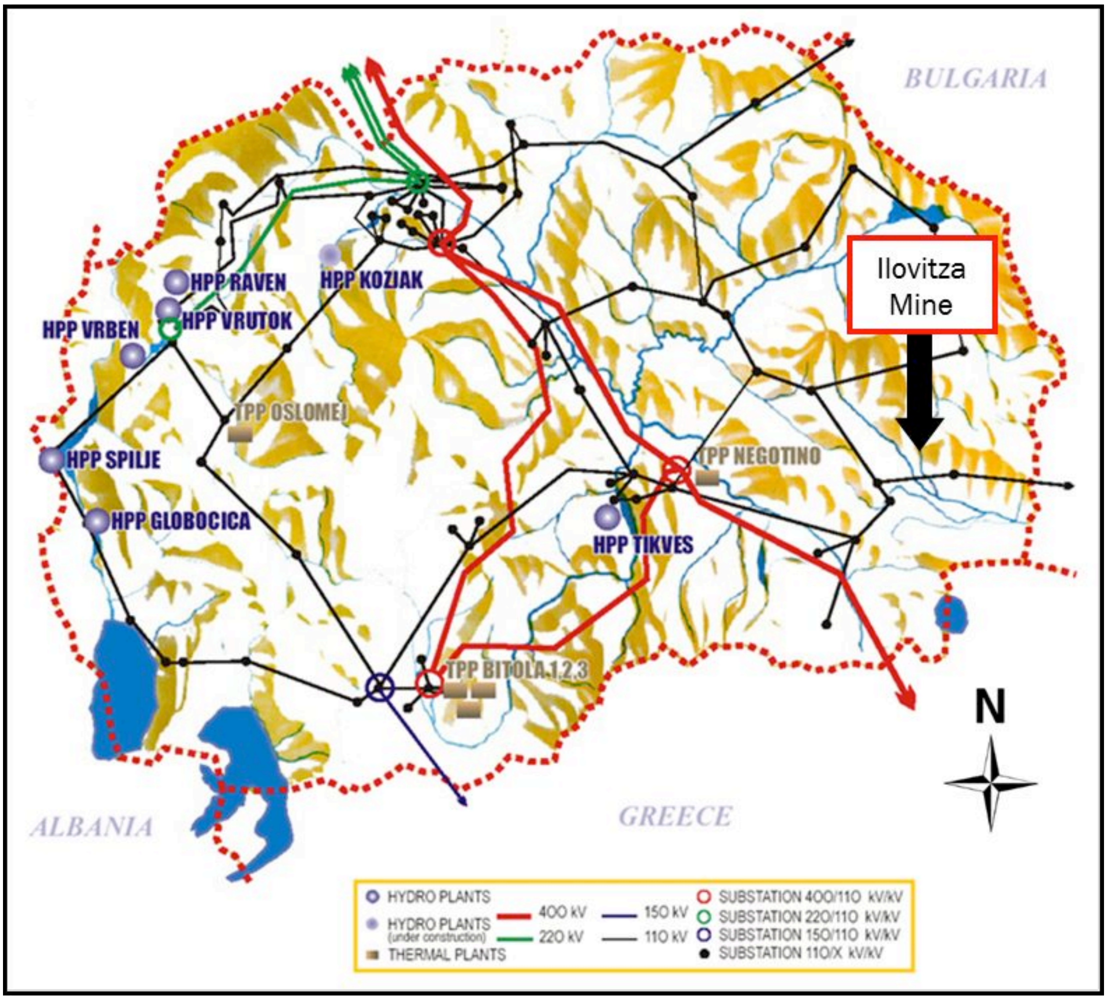
18.13 POWER

18.13.1 HIGH VOLTAGE POWER SUPPLY

The power requirement for the mining and processing operations is estimated to be approximately 70 Megawatts (MW) for the base case 10 Mt/year production scale.

Macedonia is understood to be connected to the European grid via the National Grids of Bulgaria, Greece and Serbia as illustrated by Figure 18.25. However the stability and capacity of the grid network in the eastern Macedonia has yet to be confirmed.

Figure 18.25 Macedonian Power Distribution Network



Source: Google Maps / Tetra Tech (not to scale)

It is understood that there is a 110 kV line passing within 7.3 km of the plant site, with an existing substation near the town of Sushica, approximately 10.5 km from the site. However, it is understood that this substation may not have the capacity to support the project although this should be confirmed with discussions with the local power distribution company during the next stage of study.

Figure 18.26 Existing Power Transmission Line

Source: Tetra Tech August 2013

For the purposes of this study it is proposed to construct a new 7.3 km high voltage 110 kV power line from a T-off of the existing 110 kV power line, through to the site's primary substation on the edge of the plant site, as illustrated in Figure 18.27.

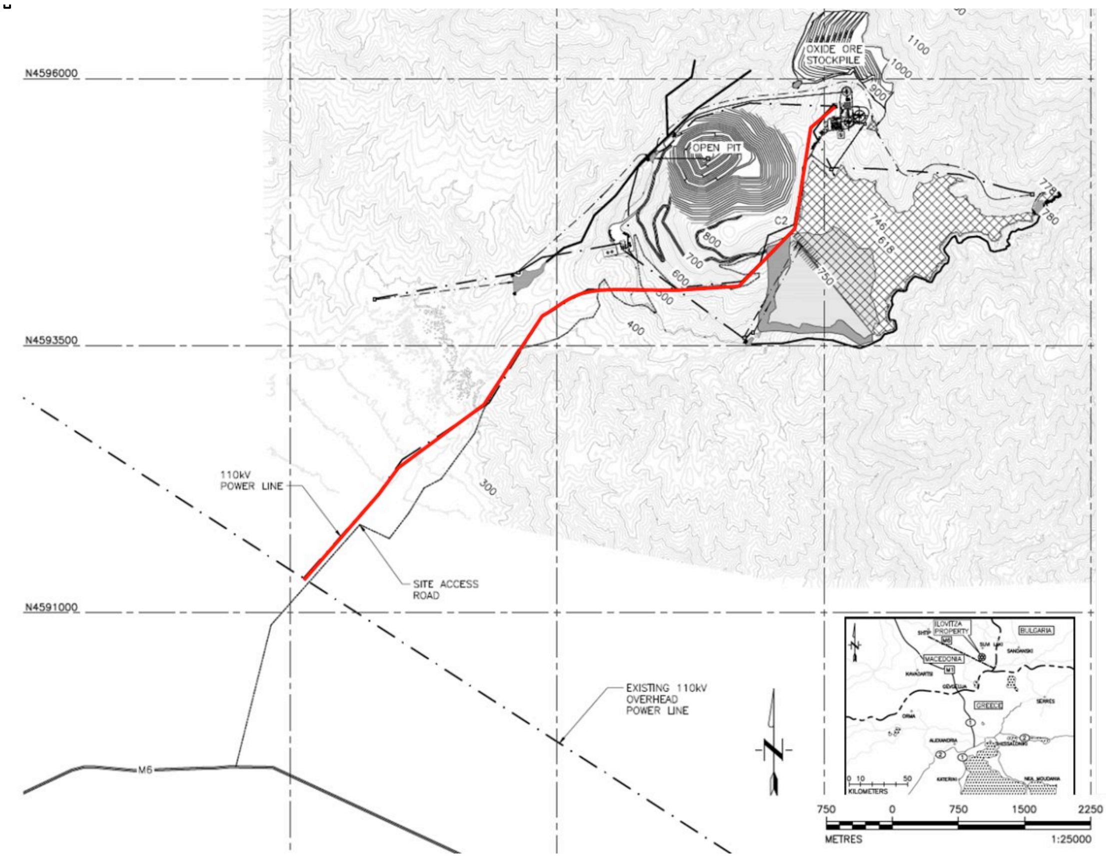
As this will be a dedicated line for the project the whole capital cost of the new line is likely to have to be paid for by Euromax. However, it may be advisable for the ownership of this line to be handed over to the distribution company for maintenance purposes and the point of common coupling located at the plant substation.

A preliminary grid connection scheme and transmission scheme has been developed consisting of a simple T-off connection on the passing Overhead Transmission Line (OHTL) and 7.3 km 110 kV dual circuit to the site substation. This connection will be completed with;

ground frame gantry, surge arrestor bushings, termination connections to primary, transformer bushings, optical fibre grounding cables and optical fibre management rack.

It is recommended that discussions are held with the local electricity company during the next stages of the study to confirm the connection details and to complete any system modelling for the new load.

Figure 18.27 Suggested 110 kV Power Transmission Alignment from Existing 110 kV Line to the Plant Sites

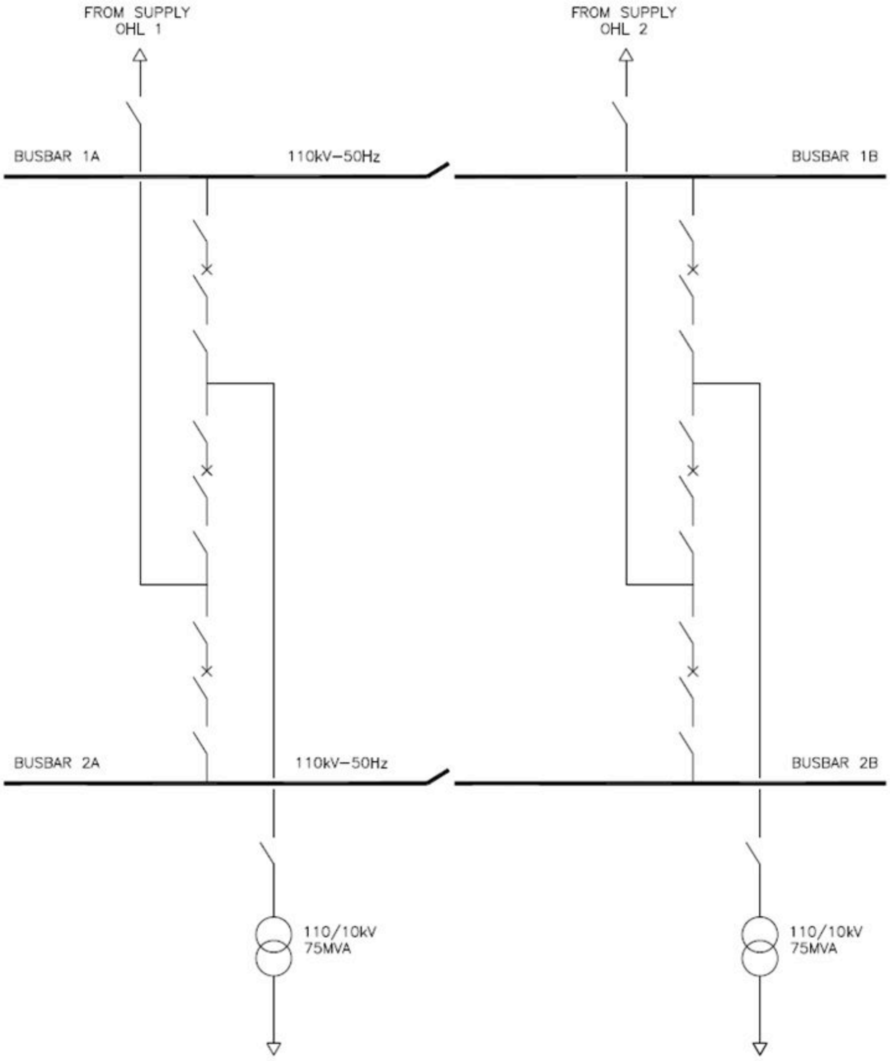


Source: Tetra Tech

18.13.2 Site Primary Substation

A primary substation will be established on site to transform the incoming power to a suitable medium voltage for site distribution to consumers. A single line diagram for the primary substation connection is illustrated in Figure 18.28.

Figure 18.28 High Voltage Supply Connection Single Line Diagram



Source: McLellan and Partners Electrical Infrastructure Report June 2014

It is proposed that the plant's main substation will consist of 110 kV to 10 kV step-down power transformers. The 10 kV line will be the plant's main distribution voltage. This substation will consist of 10 kV switchgear line-ups that will be used to distribute power to the various plant areas as required by either overhead line or land based cable tray/conduit. The mills will also be powered at 10 kV.

Each major plant area will require an electrical room where the 10 kV distribution will be stepped down to the process level distribution voltages. There will be a selection of switchgear (breakers and starters) and motor control centres.

A critical process motor control centre in each electrical room (where critical loads are identified) connected to a stand-alone generator system will transfer power from one source to another via an automatic transfer switch.

18.13.3 Site Distribution

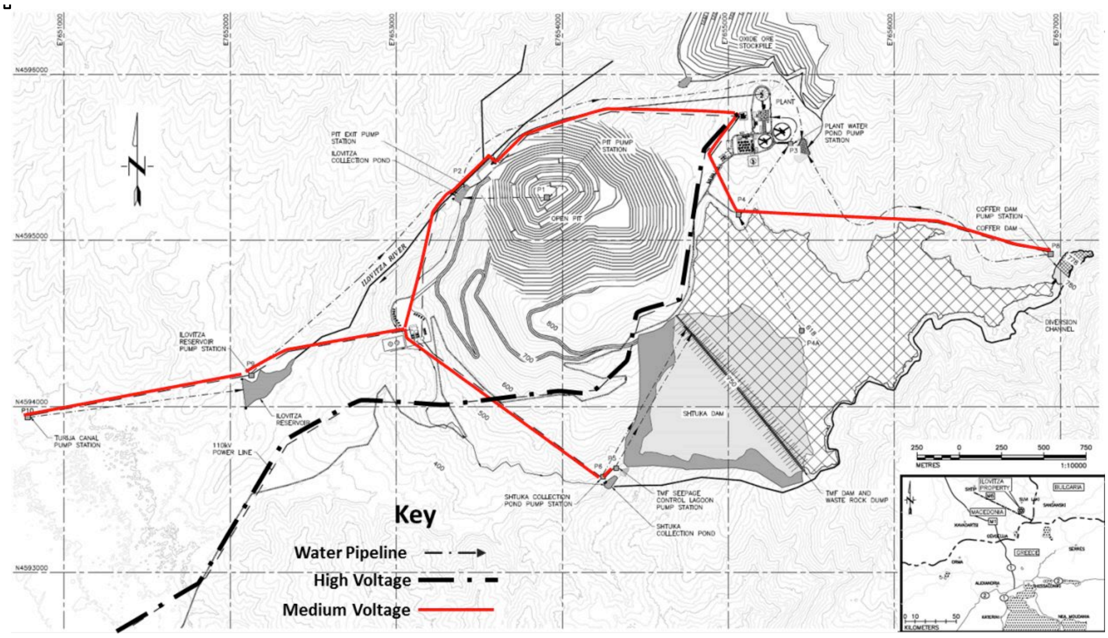
From the primary substation power will be distributed to remote consumers via a medium voltage distribution network as shown in Figure 18.29. This will link the primary substation with secondary local substations at each remote location which will support the power requirements in those locations.

The secondary substations include:

- Plant Site
  - Mill Substation.
  - Flotation Substation.
  - Mill Thickener Substation (inc. P3).
  - CIL Substation.
  - Workshops and Stores Substation.
  - Admin Block.
- ROM Pad and Primary Crusher Substation (including Explosive store, P1 & P2).
- Truck-shop Substation.
- Ilovitza pump station (P9) Substation.
- Turva Canal pump station (P10) Substation.
- Shtuka Collection Pond pump stations (P5 & P6) Substation.
- Tailings pump station (P4) Substation.
- Coffer Dam pump station (P8) Substation.

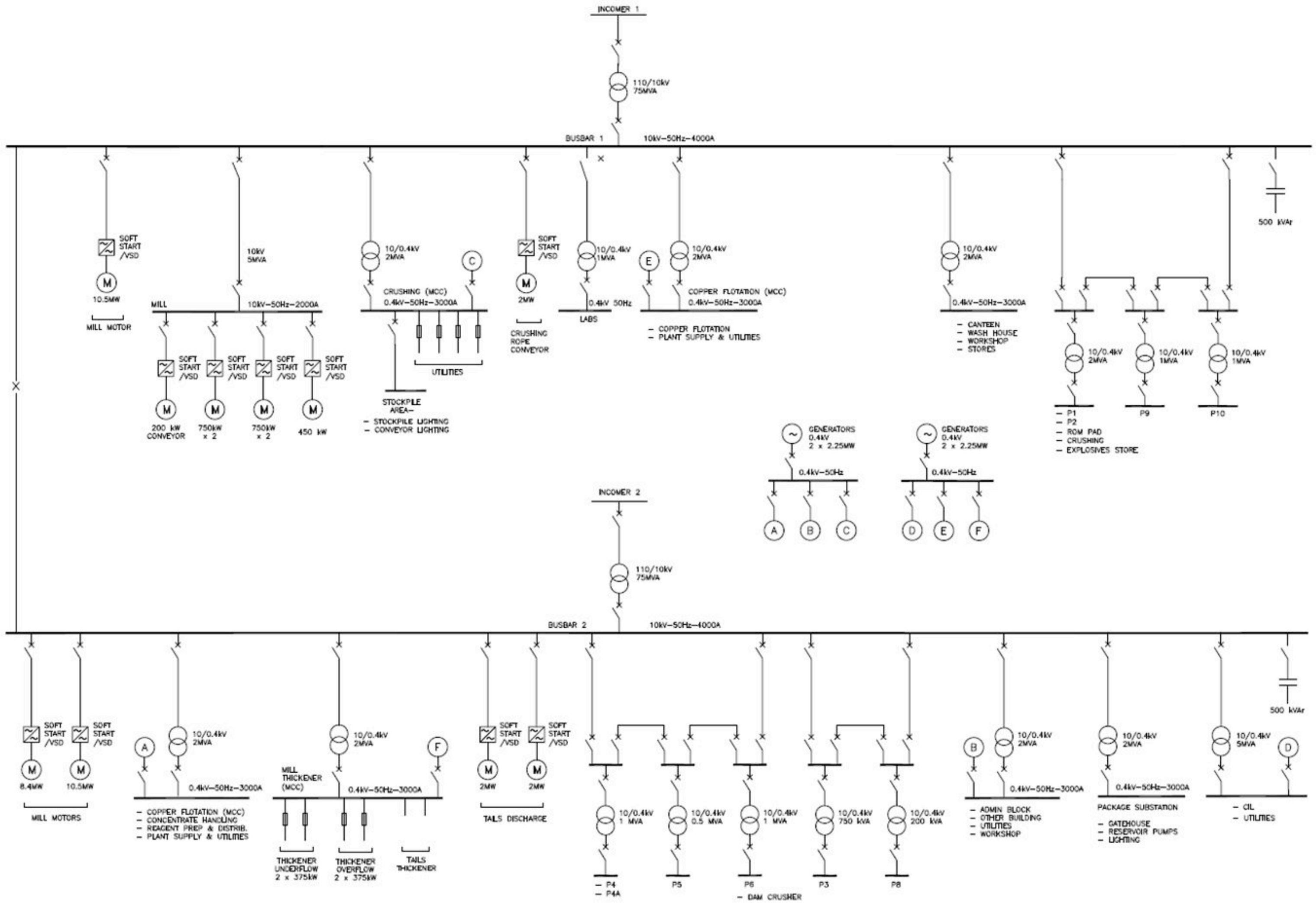
The preliminary single line diagram is shown in Figure 18.30.

Figure 18.29 Site Power Distribution Network



Source: Tetra Tech

Figure 18.30 Preliminary Single Line Diagram



Source: McLellan and Partners Electrical Infrastructure Report June 2014

18.14 CAPITAL AND OPERATING COSTS

18.14.1 CAPITAL COSTS

The site infrastructure capital cost estimate is given in Table 18.8.

Table 18.8 Summary of Site Infrastructure Capital Cost Estimate

DESIGNATION	COSTS US $
PLANT FACILITIES	Inc. Plant Costs
Miscellaneous – Crusher concrete perimeter retaining wall (with foundation)	705,280
ROM Pad construction, inc. explosive store and access roads (Euromax Estimate)	*6,600,000*
Primary Crusher	Inc. Plant Costs
Crushed Ore Cable Conveyor (Euromax Estimate of $10,200,000, based on Metso quote)	Inc. Mining Capital Costs
INFRASTRUCTURE AND BUILDINGS	
Telecommunications	70,000
Plant Facilities	3,482,325
Site Facilities, Buildings & Workshops	4,791,600
Other Facilities	800,200
Tailings Return Water Collection	1,376,209
Raw Water Collection and Distribution	9,611,567
Services	300,000
Mine Services	Inc. Mining Costs
Power Supply and Distribution	50,634,520
Fuel Farm	Supplier supply
Mining Fleet	Inc. Mining Costs
Mine Truck Shop (ACA Howe Estimate)	*9,685,000*
Explosives Store Facilities	Supplier supply
First Fill and Consumables	Inc. Mining Costs
Access & Internal Roads, Supplementary Earthworks and Fencing	8,640,798
Miscellaneous Facilities	4,020,625
TAILINGS MANAGEMENT FACILITY	
Tailings Dam Construction	Inc. Sustaining Capital
Tailings Distribution System (Euromax Estimate)	*3,094,755*
PROJECT INFRASTRUCTURE CAPITAL COST	**$103,812,879**

Note: Excluding contingency and EPCM costs.
Infrastructure Capital Costs Estimate sources: Tetra Tech / *Euromax*

18.14.2 Operating Costs

The Infrastructure operating costs are largely associated with the water supply costs, which have been calculated based upon the predicted availability of the different water sources for an average rainfall year. In a dry year water costs could increase by up to $0.10 /t concentrate.

Table 18.9 Summary of Site Infrastructure Operating Cost Estimate

INFRASTRUCTURE OPEX	Year 1	Final Year
Water Pumping Cost	$ 1,549,126	$ 1,395,331
Power Maintenance Cost	$ 854,432	$ 854,432
Site maintenance - excl. power (1% CapEx)	$ 464,636	$ 464,636
Total Annual Infrastructure OpEx	**$ 2,868,193**	**$ 2,714,399**
Infrastructure OpEx per tonne ROM	$ 0.29 /t concentrate	$ 0.27 /t concentrate

Note: Excluding contingency

In addition the operating cost of the crushed ore cable conveyor will add $0.10/t ore feed to the plant processing costs.

19.0 MARKETING AND CONTRACTS

There are no market studies or contracts material to the Project.

20.0 ENVIRONMENTAL AND SOCIAL STUDIES

20.1 CURRENT STATUS OF ENVIRONMENTAL STUDIES

20.1.1 INITIAL EIS AND ENVIRONMENTAL INVESTIGATIONS

An Environmental Impact Statement (EIS), prepared by a Macedonian company, Rudplan DOOEL, based on the project Conceptual Study prepared by Phelps Dodge Vardar DOOEL (now known as Euromax Resources DOO,) was presented to the Macedonian Government in October 2011. Approval of the EIS was received from the Ministry of Environment and Physical Planning (MEPP) in November 2011. At that time, the project definition in the Conceptual Study was not comprehensive. The impact assessment was therefore of a generic nature and tailored to Macedonian regulatory requirements. Further definition of the project development, building on the work completed in the PEA, has been undertaken during the PFS. This has addressed aspects such as:

- Finalising production rates and the consequent definitions to infrastructure and project footprint;
- Revised waste rock and tailings management practices to reflect the results from the geochemistry test work;
- Refining TMF locations to accommodate increased Mineral Reserves;
- Assessing options for process water supply; and
- Transportation of concentrate to a suitable smelter, which is likely to be in Bulgaria.

Euromax recognises that further baseline investigations and impact assessments are needed to assess these refined definitions of the project scope and ensure that the Ilovitza project is Equator Principle (and by definition IFC Performance Standard) compliant. These further investigations must be completed by Euromax as part of an Environmental and Social Impact Assessment (ESIA).

Although the principle for developing the Ilovitza project has already been accepted by MEPP; an amended EIS report will be required. It is expected that this will take the form of an ESIA report and associated Management Plans that will meet the requirements for project financing as well as Macedonian regulatory requirements and be based on the findings from extended baseline studies and impact assessment based on the project description as defined in the FS. The proposed ESIA will form an integral part of the Feasibility Study.

20.1.2 ONGOING BASELINE PROGRAMMES

Additional environmental and social baseline studies have been commissioned by Euromax from Golder Associates (UK) Ltd, together with further ground and surface water studies to be completed by Schlumberger Water Services (SWS), UK. The aspects to be investigated are shown in Table 20-1. The scope of the baseline studies will be designed to characterise the environment in which the project will be located, to a sufficient level to allow an impact assessment to be completed.

Table 20.1 Aspects to be investigated during the baseline studies

Environmental Baseline	
Geology	Biodiversity and ecosystem services
Geomorphology and landscape	Climate
Soils and Land Capability	Air quality
Land Use	Noise
Agriculture and forestry	Traffic
Water Studies	
Geochemistry	Groundwater
Surface water	Water supply
Social Baseline	
Social and economic data including livelihoods	Archaeology and cultural heritage
Stakeholder Engagement	Landscape and Visual assessment

Investigations of ore and overburden geochemistry commenced in the summer of 2013. The results from the initial test work (Ilovitza Field Tests, J Crummy, Jun 2014) have allowed a preliminary classification of the rock types sampled into Oxidised, Intermediate and Acid Generative. The oxidised material is not expected to be acid generating and some is reported as potentially acid consuming. Some acid generation from un-oxidised material, for example the dacites and the hydrothermally altered and disturbed granites, may be expected. However it is too soon to confirm the extent of any acid rock drainage (ARD) and metals leaching (ML) or the likely effects these may have on groundwater or surface water quality. Further test work will be planned based on results to date, but at this stage it appears that sufficient oxide material may be available to encapsulate or buffer Intermediate and Acid Generative material in deposited waste rock, should this be necessary.

Field work from seasonally dependant studies, such as ecology, air quality, climate, surface and ground water monitoring commenced during Quarter 3, 2013. Much of this initial work was related to establishing appropriate monitoring networks and sampling stations for the ecological surveys and the water and air quality programmes, together with training local Macedonian consultants and Euromax technicians in the sampling procedures and data management required. Initial results from the first rounds of monitoring and sampling have not yet been interpreted; this will form part of the next phase of work in 2015.
A key element of the social investigations will be to develop the household surveys so they can commence early in 2015, before the agricultural season gets underway. In parallel stakeholder consultation will also commence in early 2015. The remainder of the non-seasonal studies will be completed in 2015.

20.1.3 CURRENT PROGRAMME

The anticipated environmental and social programme in relation to the PFS and FS schedules, will continue with baseline studies into 2015. Revised identified impacts for the project will then be assessed in relation to the base line data and the more advanced engineering definitions achieved as the project approaches completion of FS. Impacts will be assessed through quantitative and qualitative methods and the effects of mitigation measures incorporated, in order to develop an understanding of any identified residual impacts. Mitigation measures will be developed in conjunction with the FS engineers to ensure that they are integrated into project design.

Management Plans will form an essential part of the ESIA, which itself will be an integral part of the FS report. The Management Plans (MPs) will in turn be incorporated by Euromax into the ESMS that will be developed for the project and will be focussed on Construction, Operations and Closure. Management Plans will likely include the following:

Environmental	Social
Environmental, Social and Health and Safety Management System	Stakeholder Engagement
Pollution Prevention (inc noise, dust, air emissions, vibrations)	Community Health, Safety and Security
Water resources	Cultural Heritage
Soil	Human Resources
Biodiversity	Workers Occupational Health, Safety and security
Sediment and erosion control	Community Development
Waste	Transport
Mine waste	Emergency Preparedness and Response
Hazardous Materials	Land Acquisition and Economic Displacement
Spill response	Influx
Cyanide	
Footprint	
Conceptual Closure	

While each plan will contain a section on monitoring requirements, these may also be consolidated into a separate Monitoring Plan.

At the start of construction, Euromax propose that contractors will be pre-qualified on various aspects including their HS&E capabilities. They will sign off on the relevant MPs to confirm that they understand the standards which Euromax require. Suitable contract clauses will be included by Euromax in the main EPC/EPCM contract, with a requirement for sub contractors to adhere to the same standards.

Close supervision and monitoring of activities will be carried out by Euromax during the construction phase.

20.1.4 Standardised Place Names

The following English translations of Macedonian place names have been determined by the Euromax communities team on site. It is likely that additional place names will be added in the future but in the intervening period the following transliterations will be used in all technical reports and plans:

- Ilovitza village;
- Shtuka village;
- Ilovitza River;
- Shtuka River;
- Skopje;
- Strumica;
- Bosilovo;
- Novo Selo;
- Shtip;
- Turnovo;
- Sekirnik;
- Radovo;
- Drvosh;
- Barbarevo;
- Staro Baldovci;
- Sushica;

- Borievo;
- Ednokukevo;
- Robovo;
- Monospitovo;
- Petralinci; and
- Novo Konjarevo.

20.2 Environmental And Social Implications Of Project Options

Sections 16, 17 and 18 of the PFS study identify a variety of different project options in order to select the preferred project design recommended for more detailed investigation during the FS. Significant elements of this evaluation relate to the engineering effectiveness and the capital and operational costs of the different options, but international ESIA practice requires that their environmental and social implications also contribute to this assessment. The different options should be further examined during the FS programme, by the ESIA consultants, when the existing baseline characteristics of the Ilovitza area are better understood, and constraints mapping is available, in order to substantiate this initial review from an environmental and social perspective. At this stage, options around the following six aspects of the project have been considered:

- Mining, including methods and mine production rates;
- Ore processing and location of processing plant and ancillary infrastructure;
- Disposal of tailings and waste rock;
- Water supply;
- Power supply; and
- Transportation of concentrate.

20.2.1 Mining

Section 16 of the PFS states that the preferred mining method is for open pit mining, using blasting and conventional trucks and shovels to remove the ore and waste rock. Underground mining is not feasible because of the nature of the orebody, its shape and proximity to the surface.

A final throughput of 10Mtpa was selected as optimal from a mining perspective and this is considered likely to have no particular adverse environmental or social impacts over other options considered.

Land take will be limited as a far as possible by minimising the project footprint, to reduce impacts on agriculture and biodiversity that may be identified during the ESIA. The visual impact of the mine will probably be limited to long views from elsewhere in the Strumica valley. Landscape and visual impact will be assessed during the ESIA process and mitigated through project design elements such as building orientation and colour and the use of screening vegetation.

The development of ARD and metals leaching (ML) has been identified as a potential impact, and its management will depend upon realistic interpretation of current and additional test work, careful planning and implementation of management and control measures and their day-to-day execution, both during operations and following closure. These aspects will be addressed in the baseline studies, the ESIA and the site environmental Management Plans.

20.2.2 Plant Site

Section 17 of the PFS describes that flotation to produce a copper/gold concentrate with Carbon in Leach processing of the tailings and oxide ore has been selected as the preferred processing route. Reagent transport, storage and handling will be carried out in accordance with good international industry practice (GIIP), in compliance with Macedonian and EU requirements and those of the International Cyanide Management Code (ICMC or The Code).

Two relatively flat locations for the process plant were considered in the PFS, both with similar sized footprints. Site A lies to the south-west of the open pit at an elevation of some 500 m whereas Site B is on the north-east side of the open pit at an elevation of some 780 m.

Section 18 of the PFS states that Site B, the Upper Site, will be used for the process plant, primarily because it provides easier distribution of tailings into the planned Tailings Management Facility (TMF) described below. It is also further from the nearest houses in Shtuka village. The different options should be further examined during the FS programme, by the ESIA consultants, when the existing baseline characteristics of the Ilovitza area are better understood, and constraints mapping is available, in order to substantiate this initial review from an environmental and social perspective.

20.2.3 Disposal Of Tailings And Waste Rock

Sections 16 of the PFS states that over the life of the mine it is estimated that approximately 164Mt of waste rock and just over 200Mt of primary ore tailings will be produced. Two key aspects of the storage of the tailings and waste rock have been examined:

- Potential locations of the storage facilities; and
- Possible methods for disposal.

The two aspects are closely inter-linked as there are limited areas for their disposal in the vicinity of the mine site. Section 18 of the PFS presents how the Shtuka valley has been selected as the most favourable site based on technical and environmental criteria.

Three possible methods for disposal and storage were considered:

- Conventional thickened slurry tailings, in which the tailings slurry is pumped into the impoundment. Separate disposal facilities would be required for the waste rock, although quite large amounts could be used to construct the impoundment embankments;

- Dry stacked (filtered) tailings, where relatively dry tailings (~20% moisture content) are hauled to the impoundment and stacked and spread by front-end loaders and bulldozers. Dry-stacked tailings are more stable than slurry impoundments. Again, separate disposal facilities are required for the waste rock although some could be used to construct embankments to contain the dry-stacked tailings; and

- Co-disposal of filtered tailings and waste rock. Here filtered tailings and waste rock are disposed of in successive layers so that the tailings fill the voids within the mass of coarse run-of-mine waste rock. This leads to a geotechnically stable facility. A further advantage is that the facility footprint is smaller compared with separate disposal sites.

The trade off study identified Co-disposal as a sound technical option. However the capital costs of the filtration plant and the elevated operating costs of filtering and trucking or conveying the tailings for dry stacking and / or co-disposal were considered by Euromax to be prohibitive for such a scale of operation. A further evaluation reported in a study by the University of Skopje (Ilovitca Preliminary Solution Book 2 Final Univ of Skopje) determined that by producing conventional thickened tailings and using waste rock to build the TMF embankment using the downstream construction method, with excess waste rock used as a downstream buttress to the retaining structure, the advantages of stability and a compact overall mine waste storage site were achieved. The different options should be further examined during the FS programme, by the ESIA consultants, when the existing baseline characteristics of the Ilovitza area are better understood, and constraints mapping is available, in order to substantiate this initial review from an environmental and social perspective.

20.2.4 Water Supply

A number of options for water supply have been considered in the PFS. See section 18. Current options include supernatant water from the tailings, supplemented by inflows into the pit, run-off from the tailings/waste rock facility and rain/snow falling within the catchment of the overall mine site; make-up water from the Turija reservoir via the Turija Canal to the Ilovitza reservoir which could supply some 2 to 3 Million m^3/year; and a borehole field around the uphill side of the open pit. Water supply options will be further investigated during the FS and the impacts assessed as part of the ESIA in a way that evaluates impacts to the water supply of third parties and impacts on other environmental or social receptors.

20.2.5 Power Supply

The project base case and Euromax's preference is to obtain power directly from the Macedonian National Grid via a spur that would be constructed from the main power line near Turnovo. This line would initially follow the route of an old drainage line and then the alignment of the new access road to the mine that will by-pass Shtuka village to the south. This will reduce land take and visual impact in the immediate vicinity of the village. The different options should be further examined during the FS programme, by the ESIA consultants, when the existing baseline characteristics of the Ilovitza area are better understood, and constraints mapping is available, in order to substantiate this initial review from an environmental and social perspective.

Other power options, such as geo-thermal or hydro-electrical, will be examined during the FS, as part of the overall trade-off studies to limit greenhouse gas emissions for the mining operation.

20.2.6 Transportation Of Concentrate

Section 18 of the PFS states that concentrate will most probably be sent to the Pirdop smelter in Bulgaria. Three transport options have been considered from an engineering perspective:

- In covered truck by road from Ilovitza to Pirdop, a distance of 310km;

- In covered truck by road to a railhead at Petrich and then in covered wagons by train to Pirdop; and

- In closed containers by road to a railhead at Petrich and then loaded onto flat-bed wagons by train to Pirdop.

Using closed containers ,the opportunity for the loss of concentrate into the environment from the moving trucks and rail wagons or at any of the load-out and discharge points is minimized. There is also a reduced risk of spillage of concentrate in the event of a road or rail accident.

It is unlikely that a detailed impact assessment of the handling and transfer of the full or empty containers at the Petrich railhead will be required either for the project ESIA (as an associated facility) or by the Bulgarian authorities. The viability of this option will be examined during FS.

20.3 Mine Closure

20.3.1 Closure Planning

At this stage of project development, it is assumed that closure will entail the removal of all structures associated with the mining operation and rehabilitation of the operating area so it integrates with the natural landscape surrounding the site. In order to meet Euromax's corporate needs, a comprehensive conceptual closure management plan will be developed as part of the FS programme that complements the project definition developed. This will take the form of a Conceptual Closure Plan (CCP) annexed to the ESIA, as one of the suite of Management Plans that will be developed. The CCP will be developed by the ESIA and FS team and will include a provisional estimate of closure costs.

The following broad aims will be incorporated in the closure management plan:

- to plan for a geochemically and physically stable site that needs minimal ongoing aftercare;
- to plan for productive and sustainable after-uses of the site that are acceptable to Euromax, local communities and the Macedonian Government;
- to protect public health and safety;
- to alleviate or eliminate environmental damage;
- to minimise adverse socio-economic impacts; and
- to consider re-use of valuable attributes from the project.

20.3.2 Closure At Ilovitza

The final vision for closure of the project must be to create a productive and sustainable after-use for the site that is acceptable to Euromax, the Macedonian authorities, the local communities and any future users of the site. The pit lake, if one is predicted to develop and persist, and tailings and waste rock impoundments must remain after closure, but at this stage it is necessary to plan to remove all other plant, buildings, structures and associated infrastructure, and restore the site to grassland and/or woodland similar to that which is currently present on the site.

In this regard, a number of environmental and social aspects are important. These include:

- *Deposit geochemistry*. The geochemistry of the tailings and mineralised waste rock have the potential for ARD/ML from the tailings, waste rock and the pit lake slopes. It is reported that initial studies have indicated that any acid generative material may be managed by encapsulating with oxidised rocks. In addition, it is important to consider the potential effects from process reagents post closure.
- *Revegetation*. The regular spring and autumn rainfall will encourage re-vegetation of the mine site but it is proposed that this will be enhanced with the use of appropriate seed mixes and planting shrubs and trees of local provenance, using seed collected during mine life.
- *Water*. Many of the local people around the project site require water supply for personnel use, irrigation and farming . The ESIA will assess ways to maintain acceptable water supply in quantity and quality at closure and post closure.
- *Biodiversity*. The biodiversity of the existing habitats present on the site is currently being assessed through seasonal baseline surveys, including vegetation, small and large mammals, birds and aquatic biodiversity. The project will adopt a No Net Loss approach to Biodiversity, in line with IFC Performance Standard 6 (PS6), and will use the mitigation hierarchy to develop the best approach to meet PS6.
- *Socio-economic* impact on local communities and labour at closure. The impact of the cessation of the socio economic benefits the project offers will be evaluated in the ESIA and measures put in place to maintain skills and manage influx/efflux.

PIT LAKE

The likelihood of the development of a pit lake and the potential impacts associated to such a pit lake will be evaluated in the ESIA and mitigation and management measures will be put in place to ensure no unacceptable impacts at closure and post closure Detailed studies on closure will be taken prior to closure, building on the regularly reviewed and updated Conceptual Closure Plan.

20.4 MATERIAL ENVIRONMENTAL AND SOCIAL ISSUES FOR PROJECT DEVELOPMENT

On the basis of the current understanding of the Project Definition and the limited information available on environmental and social aspects, five potentially material issues have been identified. These are:

- Water supply;
- Geochemistry of the ore and waste rock;
- Stakeholder engagement and relationships;
- Socio-economic impact on livelihoods and agriculture;
- Biodiversity;

- Traffic and associated effects (air quality, noise and vibrations);
- Closure.

Identifying a sustainable water supply for the project is currently considered to be the most critical issue. None of the other issues are considered to be fatal flaws at this stage of the programme although there is a level of uncertainty associated with each of them. These uncertainties will be addressed during the FS and the ongoing ESIA investigations.

The material issues, their potential environmental and social impacts for the project and proposed next steps aimed at understanding these potential issues are shown in Table 20.4. Mitigation measures for Predicted impacts will be developed during the FS.

Table 20.2 Material social and environmental issues

Issue	Potential impacts	Proposed next steps
Water supply	At the present time the process water supply for the project is proposed to come from the tailings lake and the Turija canal. This may result in lower flows in the Turija canal available for local water supply and irrigation Lower flows in the Shtuka River available for local water supply and irrigation may result from the presence of the TMF. This may also have an effect on the groundwater levels and therefore spring water supply in local villages	The water supply studies in the FS will identify a reliable and long-term water supply for the project that does not adversely affect local communities, agriculture or environmental receptors. This will involve establishment of a comprehensive water balance for the project during construction, commissioning, operations and post closure. Mitigation measures to address identified impacts to communities, agriculture or environmental receptors will be developed. Project design will incorporate water reduction and recycling measures to reduce the amount of make up water required. Potential water supply options could include surface waters, aquifers and existing reservoirs, or a combination of these, depending on project requirements, existing use of the water sources and the timing of project water needs.

Issue	Potential impacts	Proposed next steps
Geochemistry of the ore and waste rock.	Oxidation of sulphides in the tailings and waste rock could lead to acidic conditions, metal leaching and acid rock drainage into ground and surface water.	ARD investigation programme underway as part of baseline studies. Buffering capacity has been identified in some rock types. Buffering will be maximised in the design to minimise ARD issues. Complete ABA testing on ore, waste rock and tailings and if necessary carry out kinetic testwork to assess rates of release of metals and sulphate. These will be scaled up to predict likely release within tailings/waste rock storage facilities. Refine block model so that oxidised, intermediate and acid generative rocks are identified and their placement can be scheduled to achieve encapsulation of acid generative rock. On the basis of a refined block model, develop a plan to mitigate ARDML, probably by encapsulating acid generative waste rock types within oxidised material in tailings/waste dumps. Assess drainage from these impoundments and likely contamination into groundwater and develop treatment measures to be included in project design.

Issue	Potential impacts	Proposed next steps
Stakeholder engagement and relationships	Failure to implemement effective stakeholder engagement could lead to impacts on Euromax reputation direct opposition to development of the mine from within the region and possibly the country; · difficulties in accessing land; · increased costs and delays; risks to obtaining the legal and social licence to operate; and adverse publicity from local and international NGOs	Euromax began its community relations programme in 2012 with the opening of a local Information Centre in Ilovitza village and its staffing with a local Administrator. A team of Community Liaison Officers are being recruited to ensure effective implementation of the consultation plan, developed as part of the ESIA This team will also be based in Ilovitza village and the intention is for local people to have free access to the Community office and Communities team. This team will form the basis for the Community Relations (CR) group required during construction, operations and closure. Regular effective engagement with stakeholders, and a good grievance mechanism will allow them to better understand the current and planned activities of the project, so that their expectations are based on an informed understanding. At the same time Euromax will better understand stakeholder concerns so that these can be incorporated into the ESIA and other aspects of the FS as appropriate. Project-wide training on stakeholder communication will also be put in place to ensure project activities continue to contribute to positive community relations, such as prompt response to grievances via the grievance mechanism or agreed codes of staff conduct when dealing with local people.

Issue	Potential impacts	Proposed next steps
Socio-economic impact on livelihoods and agriculture; ▪	The influx of labour, changes to local and regional industry and , the use of local workers in the mine could have an impact on livelihoods and agricultural skills in the local area	Baseline data gathering will establish the baseline for agriculture and livelihoods. Baseline data and stakeholder engagement information and the grievance mechanism will feed into the key issues to be assessed in the ESIA.
Biodiversity and ecosystem services	The project footprint, associated infrastructure and mining activities may have an effect on ecological and biodiverse receptors and ecosystem services	Baseline data gathering will characterise the existing ecology, biodiversity and ecosystem services. The ESIA will assess the potential impacts and use the mitigation hierarchy to minimise impacts in line with IFC PS6
Traffic and associated effects (air quality, noise and vibrations	Increased traffic on existing and potential new roads may have an impact on receptors living, working or using social and environmental receptors	Baseline data gathering will characterise the existing environment. The ESIA will assess the potential impacts and use the mitigation hierarchy to minimise impacts

Issue	Potential impacts	Proposed next steps
Mine closure	Ineffective mine closure planning can result in unacceptable environmental and social impacts, delays at the permitting stage, significant unnecessary closure costs when mining ceases and a detrimental impact on Euromax's reputation.	The conceptual closure plan (Section 20.3) provides an overview of the need for managed closure and the measures required. It covers the physical closure of the mine but not social or community aspects because these will be addressed during the FS. As the project becomes better defined during the FS the conceptual plan will be revised accordingly. The ESIA will assess the impacts associated with closure and post closure and the results will feed into the closure plan and the project design, which will include mitigation measures to address identified impacts. . The PFS conceptual plan addresses the following elements: · Legal and other obligations; · Closure planning vision; · Open pit; · Process plant; · Tailings and waste rock; · Site infrastructure; · Site rehabilitation/re-vegetation; and · Estimate of closure costs · Environmental monitoring and management · Land issues The Closure Plan will be revised at regular intervals during mine life and in the case of any significant changes to the mine as described in the FS.

21.0 CAPITAL AND OPERATING COSTS

The Pre-Feasibility Study on the Ilovitza gold-copper project has defined operating and capital costs as detailed in this section.

All costs have been estimated in US dollars. Euro values have been converted to dollars at a long-term exchange rate of 1.4 dollars to the euro.

21.1 Capital Costs

A summary of the total estimated capital costs is given in Table 21.1 below.

Table 21.1 Capital Cost Summary

Description (US$ million)	Initial Capex	Sustaining Capex
Mining Fleet (incl. conveyor)	34.8	128.0
Processing Plant	249.5	(in opex)
Owners costs	10.0	-
Infrastructure	103.8	30.6
Tailings (incl. pre-strip)	58.1	47.5
Reclamation (end of mine life)	-	30.0
Sub-total	**456.2**	**236.1**
Contingency (10%)	45.6	-
Total	**501.8**	**236.1**

21.1.1 Mining Capital Costs

Mining capital costs comprise the mining fleet, as detailed in section 16.0, along with the conveyor that runs between the primary crusher at the pit exit and the process plant. Cost details of the mining fleet are given in Table 21.2.

Table 21.2 Mining Fleet Capital Costs

Equipment	Unit Cost	Initial Cost (pre-production)	Sustaining costs
CAT 777 truck or equivalent	$1,221,640	$9,773,120	$45,200,680
CAT 990 Loader or equivalent	$1,400,000	$4,200,000	$25,200,000
CAT 375Excavator or equivalent	$650,000	$650,000	$3,900,000
CAT 345 Excavator or equivalent	$425,000	$425,000	$2,550,000
CAT D10 Bulldozer or equivalent	$1,118,460	$1,118,460	$6,710,760
CAT D8 Bulldozer or equivalent	$547,820	$1,095,640	$6,573,840
CAT 770 Water Truck or equivalent	$873,000	$873,000	$2,619,000
CAT 24 Motor Grader or equivalent	$2,450,000	$2,450,000	$14,700,000
CAT 16 Motor Grader or equivalent	$593,880	$593,880	$3,563,280
Sandvik D75 Drill or equivalent	$1,500,859	$1,500,859	$9,005,153
Sandvic DX800 Drill or equivalent	$596,407	$596,407	$3,578,442

Equipment	Unit Cost	Initial Cost (pre-production)	Sustaining costs
CAT 825 Compactor or equivalent	$150,000	$150,000	$900,000
ANFO Prill Truck	$433,000	$433,000	$1,299,000
Boom Truck	$185,000	$185,000	$555,000
Lube/Fuel Truck	$94,000	$94,000	$282,000
Man Bus	$75,000	$75,000	$225,000
Pick Ups (Toyota double cab or equivalent)	$25,000	$175,000	$525,000
Lights	$26,242	$131,208	$393,624
Pumps	$14,228	$71,141	$213,423
Total		**$24,590,715**	$127,994,202

The cable conveyor capital cost is estimated at $9 million cost based on a quote from equipment supplier Metso and a further $1.16 million to install. The capital cost of the gyratory crusher has been allocated to the process plant.

21.1.2 PROCESS CAPITAL COSTS

The process plant equipment capital costs have been built up from budget quotes provided to Tetra Tech by suppliers. Indirect costs have been factored in line with experience from other projects.

Table 21.3 Process Plant Capital Costs

Equipment	Cost/Unit	Total Cost
Crushing		
Crusher Feedbin / Hopper (360 t)	$164,056	$164,056
Primary Gyratory Crusher (50" X 65" -375kW)	$2,464,000	$2,464,000
Crusher Discharge Bin (400 t)	$180,925	$180,925
Crushed ore apron feeder (1.5m x 6m)	$322,530	$322,530
Conveyor (1.2m x 800m)	$1,692,138	$1,692,138
Conveyor Belt Scale	$15,100	$15,100
Tramp Magnet	$12,600	$12,600
Belt Sampler	$35,000	$35,000
Dust Collector Bin	$6,575	$6,575
Sump Pump - Crusher Station	$18,895	$18,895
Sump Pump - Stockpile Area	$18,895	$75,580
		$4,987,399
Grinding		
Stockpile reclaim apron Feeder (1.2m x 6m)	$226,510	$906,040
SAG Mill Feed Conveyor	$315,600	$315,600
SAG Mill (32ft x 16ft - 8.4MW motor)	$10,160,000	$10,160,000
SAG Mill Pebble discharge conveyor	$147,100	$147,100
Pebble Crusher Vibrating Feeder	$15,500	$15,500
Pebble Crusher- HP800	$1,638,000	$1,638,000
Pebble Crusher Discharge Conveyor	$147,100	$147,100
Metal Detector	$1,410	$1,410
Tramp Magnet	$12,600	$12,600

Equipment	Cost/Unit	Total Cost
Mill Discharge Pump	$272,650	$1,090,600
Ball Mill (24ft x 34ft - 10.6MW motor)	$6,988,000	$13,976,000
Ball Mill Cyclone Pack (8 x 91.4 cm Cyclones)	$141,600	$283,200
Desliming Cyclone Pack (12 x 25.4cm)	$64,200	$513,600
SAG and Ball Mill bridge Crane	$166,400	$499,200
Mill Liner Crane	$411,750	$411,750
SAG Mill Media Hopper	$80,600	$80,600
Ball Mill Ball Hopper	$80,600	$161,200
Media Charge Electro Magnet	$22,040	$66,119
Monorail Hoist	$67,100	$67,100
Cyclone overflow slurry sampler	$18,350	$36,700
Sump Pump - SAG Mill	$67,600	$67,600
Sump Pump - Ball Mills	$42,141	$84,282
Linear Safety Screen (40m^2)	$188,940	$188,940
		$30,870,241
Copper Flotation		
Flotation Conditioning Tank + Agitator	$166,040	$166,040
Rougher Cells 156m^3	$336,213	$3,025,917
Scavenger Feed Pump	$129,175	$258,350
Scavenger Cells 156m^3	$336,213	$3,025,917
Scavenger Tailings Pumps	$129,175	$258,350
Concentrate Discharge Box (Rougher- Scavenger)	$21,740	$21,740
Concentrate Feed Pump	$12,858	$25,716
Concentrate Regrinding Cyclone Pack (3 x 22.8cm Cyclones)	$5,150	$15,450
Regrind Mill -SMD Mill (SMD 355)	$1,095,000	$1,095,000
Cleaner Conditioning Tank + Agitator	$35,740	$35,740
Cleaner Feed Pump	$15,835	$31,670
Cleaner Cells 14.2m^3	$68,700	$274,800
Cleaner Tailings pumps	$12,858	$25,716
Cleaner Concentrate Pump	$12,858	$12,858
Slurry Sampler	$18,350	$36,700
Sump Pumps	$18,895	$18,895
		$8,328,859
Pre Leach Thickener		
Thickener underflow pump	$129,175	$258,350
Thickener over flow surge tank	$78,290	$78,290
Thickener over flow pump	$129,175	$258,350
Pre-Leach thickener (80m high rate)	$2,750,000	$2,750,000
Sump Pump	$42,141	$84,282
		$3,429,272
Carbon in Leach		
Leach feed pumps	$34,132	$136,528
Pump Motors	$8,423	$33,692

Equipment	Cost/Unit	Total Cost
Feed Trash Screen	$87,150	$174,300
Trash screen motor	$2,402	$4,804
Leach Tanks (16.0 m x 16.5 m Stainless Steel Cyanide tank)	$610,969	$9,775,505
Leach Impeller (Mixertech)	$97,697	$1,563,157
Impellor motors	$8,423	$134,768
Recessed Impellor Pumps (carbon)	$8,074	$258,367
Pump Motors	$548	$17,536
Interstage screen MPS(P)	$159,300	$2,548,800
Safety Screen	$87,150	$174,300
Screen motor	$2,402	$4,804
Screen feed pump	$34,132	$136,528
Pump Motors	$6,043	$24,172
Leach Circuit Feed Sump Pump	$34,132	$136,528
Pump Motors	$6,043	$24,172
Leach Feed Samplers	$18,350	$36,700
Leach Tail Samplers	$18,350	$36,700
Cyanide Detoxification Reactor & Agitator	$268,655	$537,309
Cyanide Detoxification Reactor Pumps	$34,132	$204,792
Pump Motors	$8,423	$50,538
Cyanide Detoxification Surge Tank & Agitator	$268,655	$268,655
Oxitrol measuring unit	$61,900	$61,900
		$16,344,555
Tailings Discharge		
Tailings Thickener (100m high rate)	$3,686,000	$3,686,000
Sump Pumps	$18,895	$18,895
		$3,704,895
Concentrate Handling		
Concentrate Thickener	$75,600	$75,600
Concentrate Thickener Underflow Pump	$9,757	$19,514
Filter Press (29.7m^2)	$31,425	$31,425
Filtrate Pumps	$9,757	$19,514
Cloth wash Pumps	$9,757	$19,514
Filter Cake Product Conveyor	$114,673	$114,673
Load Out Filter Cake Product Conveyor	$114,673	$114,673
Conveyor Scales	$15,100	$15,100
Belt Sampler (Capacity 262cm^3)	$35,000	$35,000
Truck Scales	$26,990	$26,990
Sump Pumps	$18,895	$37,790
		$509,793
Reagent Preparation & Distribution		
Reagent Mixing Tanks	$8,975	$71,800
Reagent Mixing Tanks - Agitators	$14,000	$112,000
Reagent Holding Tanks	$8,975	$71,800
Reagent Feeders with Metering system	$2,060	$16,480

Equipment	Cost/Unit	Total Cost
Reagent Distributors with pumps	$2,420	$4,840
Lime Storage Bin (181)	$115,300	$115,300
Lime Slaker System	$236,500	$236,500
Flocculant Plant	$100,500	$100,500
Sump Pumps	$18,895	$37,790
		$767,010
Plant Supply & Utilities		
Plant Air Compressor	$32,400	$32,400
Flotation Air Blowers	$6,000	$120,000
Process Water Tank	$387,030	$387,030
Potable Water Tank	$387,030	$387,030
Water Pumps	$9,570	$38,280
		$964,740
Sub Total - Primary Equipment Cost		**$69,906,764**
Indirect Capital Costs		
Civils	30%	$20,972,029
Structural Steel	24%	$16,777,623
Piping & Valves	35%	$24,467,367
Electrical & Instrumentation	40%	$27,962,706
Transport	30%	$20,972,029
Erection of Items	15%	$10,486,015
Vendor Services	3%	$2,097,203
First Fills	3%	$2,097,203
Sub Total - Indirect Capital Cost		**$125,832,175**
Total - Installed Plant Capital Cost		**$195,738,939**
Site preparation & Construction Management		$29,360,841
Plant Mobile Equipment Cost		$6,664,697
Coarse Ore Stockpile Cost		$14,380,801
Elution, Electrowinning & Gold Room Package		$3,377,838
Total Plant Capital Cost		**$249,523,116**

Note: numbers may not sum due to rounding

The mill plant will require sustaining capital over the life of the operation, in order to repair and replace the original equipment. This has been assumed to be 2.5% of the initial direct capital cost of the equipment per year of operation. This cost has been incorporated into the operating cost

21.1.3 Infrastructure Capital Costs

Project initial infrastructure capital costs are detailed in Table 21.4.

Table 21.4 Initial Infrastructure Capital Costs

Designation	Costs US $
Plant Facilities	
Miscellaneous– Crusher concrete perimeter retaining wall (with foundation)	$705,280
ROM Pad construction, inc. explosive store and access roads (Euromax Estimate)	$6,600,000
Infrastructure And Buildings	
Telecommunications	$70,000
Plant Facilities	$3,482,325
Site Facilities, Buildings & Workshops	$4,791,600
Other Facilities	$800,200
Tailings Return Water Collection	$1,376,209
Raw Water Collection and Distribution	$9,611,567
Services	$300,000
Power Supply and Distribution	$50,634,520
Mine Truck Shop (ACA Howe estimate)	$9,685,000
Access & Internal Roads, Supplementary Earthworks and Fencing	$8,640,798
Miscellaneous Facilities	$4,020,625
Tailings Distribution System (Euromax Estimate)	$3,094,755
Project Infrastructure Capital Cost	**$103,812,878**

Note: numbers may not sum due to rounding

Sustaining infrastructure capital costs are associated with maintaining the oxide ore stockpile and its subsequent closure and total $30.6 million.

21.1.4 Tailings Capital Costs

The cost of the tailings management facility was estimated in euros and the final totals were converted to dollars, as summarised in Table 21.5.

Table 21.5 Tailings Management Facility Capital Costs

	Initial US$	Sustaining US$
Coffer dam	€ 470,274	
Grouting coffer dam	€ 90,275	
Starter dam embankment	€ 13,679,119	
Phased embankment	-	€ 32,307,489
Crushing of Engineered fill	€ 480,000	€ 802,629
Grouting embankment **	€ 300,000	€ 787,641
Temporary coffer dam,	€ 2,984	
Diversion for (C2) profile*	€ 902,705	
Temporary coffer dam*	€ 3,342	
Diversion for (P) profile*	€ 211,706	
Diversion channel*	€ 4,546,308	
Chute from (C2), *	€ 1,721,580	
Crusher	€ 3,928,571	
TOTALS Euro	€ 26,336,865	€ 33,897,759
TOTALS DOLLAR	$36,871,611	$47,456,862

Some 10 million tonnes of waste rock need to be pre-stripped in order to construct the initial phase of the tailings embankment. This has been capitalised at a total cost of $21.2 million.

21.1.5 Reclamation Capital Costs

A global allowance for the closure of the mine has been estimated at $30 million and includes credits for salvage. This amount will be refined at the FS stage as detailed plans are drawn up for mine closure in compliance with local and international regulations and guide lines, including the Equator Principles and IFC Performance Standards. It is assumed that the costs are incurred in the final year of operation. A closure fee for the temporary oxide stockpile has also been included in the sustaining capital for infrastructure equal to the total upkeep costs of the facility throughout the mine life.

21.1.6 Working Capital Costs

Working capital has been calculated based on road and rail transport and the proximity of the project to the Pirdop smelter in Bulgaria and gold refinery in the region allowing for small lot shipments and avoiding utilising sea freight, thus avoiding associated delays. Receivables for the concentrate are calculated based on two months of concentrate revenue after offsite costs whilst for doré gold they are based on one month of doré revenue after offsite costs. Note that it is likely that 90% of the revenue from concentrate would be payable at mine gate, as is usual in concentrate contract terms, which would reduce this receivable amount. With respect to payables, one month's total operating costs is used as it is expected that payment terms on average would be 30 days. Owing to the ability to ship in small lot sizes, no inventory allowance is made in the working capital calculation.

21.1.7 Contingency

An additional 10% of contingency capital has been allowed for the initial pre-production period. This amounts to US$45.6 million. The amount is thought to be appropriate overall for the project, though there are some areas where the project is sufficiently well defined for this estimate of 10% to be over stated, there are other where it may be under estimated and overall these will balance out.

21.2 Operating Costs

Operating costs were calculated on a first principles basis.

A summary of the key operating costs is given in Table 21.6 below.

Table 21.6 Summary of Operating Costs

Mining - Average LOM cost (US$/t ore)	
Mining - Oxide (incl. rehandle cost)	1.96
Mining - Sulphide	1.72
Mining - Waste (excl. pre-strip)	1.59
Conveyor	0.10
Processing	
Oxide Processing	5.23
Sulphide Processing	6.50
Infrastructure opex	0.29
G&A	1.00

21.2.1 Mining Operating Costs

Quotations were received for the key unit costs from reputable suppliers in country as shown in Table 21.7.

Table 21.7 Mining Operating Cost Local Inputs

Fuel	l	US1.19
Explosives (bulk ANFO)	kg	US$1.11

Labour rates were obtained from the Macedonian office of national statistics and are from January 2014. Year by year operating costs for the period of open pit operation are given in Table 21.8. It should be noted that labour for the explosives is assumed to be under contract from the supplier and covered by general and administration operating costs. Likewise, management and technical services are assumed to be covered under general and administration operating costs. A conveying cost of US$0.10 has been estimated to transport the ore from the gyratory crusher to the coarse ore stockpile. Crushing costs have been allocated to the process costs.

Table 21.8 Mine Operating Costs by Year in US Dollars

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21
CAT 777 Truck	$8,790	$14,280	$14,280	$15,380	$14,280	$14,280	$14,280	$14,280	$14,280	$14,280	$16,480	$14,280	$14,280	$10,990	$10,990	$10,990	$10,990	$10,990	$9,890	$5,490	$4,390	$4,390
CAT 990 Loader	$1,040	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130	$3,130
CAT 375 Excavator	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580	$580
CAT 345 Excavator	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460	$460
CAT D10 Bulldozer	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790	$790
CAT D8 Bulldozer	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020	$1,020
CAT 770 Water Truck	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620
CAT 24 Motor Grader	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670	$670
CAT 16 Motor Grader	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400	$400
Sandvik D75 Drill	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980	$1,980
Sandvik DX800 Drill	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620	$620
CAT 825 Compactor	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680	$680
ANFO Prill Truck	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220	$220
Boom Truck	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190	$190
Lube/Fuel Truck	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120
Man Bus	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120
Total	$18,300	$25,880	$25,880	$26,980	$25,880	$25,880	$25,880	$25,880	$25,880	$25,880	$28,080	$25,880	$25,880	$22,590	$22,590	$22,590	$22,590	$22,590	$21,490	$17,090	$15,990	$15,990
Per tonne Mill Feed	n/a	$2.20	$2.20	$2.30	$2.30	$2.30	$2.30	$2.30	$2.30	$2.50	$2.70	$2.50	$2.50	$2.20	$2.20	$2.20	$2.20	$2.20	$2.10	$1.70	$1.60	$1.80
Explosives Ore	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32
Explosives Waste	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32
Grade Control labour and assay	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02
Light vehicle costs	$-	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05	$0.05
Total Ore	**n/a**	**$1.48**	**$1.48**	**$1.58**	**$1.58**	**$1.58**	**$1.58**	**$1.58**	**$1.58**	**$1.68**	**$1.78**	**$1.78**	**$1.78**	**$1.78**	**$1.78**	**$1.78**	**$1.78**	**$1.78**	**$1.78**	**$1.88**	**$1.98**	**$2.18**
Total Waste	**$2.12**	**$1.42**	**$1.42**	**$1.52**	**$1.52**	**$1.52**	**$1.52**	**$1.52**	**$1.52**	**$1.62**	**$1.72**	**$1.72**	**$1.72**	**$1.72**	**$1.72**	**$1.72**	**$1.72**	**$1.72**	**$1.72**	**$1.82**	**$1.92**	**$2.12**

21.2.2 PROCESS OPERATING COSTS

Power supply was quoted for from two different suppliers distribution costs were applied for an estimated overall cost of US$0.076 per kWh. A breakdown of the operating costs for the primary sulphide feed is given in Table 21.9.

Table 21.9 Plant Operating Costs for Sulphide Ore

OPEX	10Million tonne throughput				
	Consumption		Cost		
Consumables	kg/t Ore	tons/yr.	US$/Unit	US$/yr	US$/t
Gyratory Crusher	N/A	N/A	$122.22	$910,050	$0.09
Cone Crusher	N/A	N/A	$91.09	$678,256	$0.07
SAG Mill Liners	0.033	329	$5.50	$1,810,130	$0.18
SAG Mill Media	0.404	4,044	$1.10	$4,448,924	$0.44
Ball Mill Liners	0.061	611	$5.50	$3,361,671	$0.34
Ball Mill Media	0.751	7,511	$1.10	$8,262,287	$0.83
Regrind Mill Liners	0.002	0.3	$5.50	$ 1,699	$0.00
Regrind Mill Media	0.029	3.8	$3.00	$11,388	$0.00
Laboratory Supplies -Fixed		1	$100,000	$100,000	$0.01
General Consumables -Fixed		1	$100,000	$100,000	$0.01
Sub Total					**$1.97**
Reagents					
Flocculant	0.03	250	$3.71	$928,125	$0.09
Sodium Silicate	0.50	5000	$0.68	$3,375,000	$0.34
MX3601	0.06	550	$9.28	$5,104,000	$0.51
MIBC	0.03	340	$3.24	$1,101,600	$0.11
Lime - Flotation	0.50	5000	$0.20	$1,000,000	$0.10
Cyanide	0.21	2100	$2.20	$4,620,000	$0.46
Lime - Leaching	0.60	6000	$0.20	$1,200,000	$0.12
Sodium Hydroxide (kg/t of Carbon)	15	23.5	$0.88	$20,717	$0.00
Hydrochloric Acid (Litre / t of Carbon)	150	235.4	$0.37	$87,107	$0.01
Activated Carbon (kg / t of Carbon)	25	39.2	$2.20	$86,323	$0.01
Sodium Metabisulphite	0.22	2170	$0.42	$900,637	$0.09
Copper Sulphate	0.09	856	$2.60	$2,225,546	$0.22
Lime - Detox	0.12	1211	$0.20	$242,178	$0.02
Fluxes (Smelting - kg/Oz of Gold)	0.15	5.5	$2.00	$11,084	$0.00
Sub Total					**$2.09**
Power	Absorbed kW	kWh/yr.	$/kWh	$/yr.	$/t
Crushing & Material Handling	824	6,135,271	$0.076	468,796	0.05
Grinding	27539	220,014,423	$0.076	16,811,302	1.68
Mill / Pre leach Thickeners	679	5,424,113	$0.076	414,456	0.04
Copper Flotation	3373	26,944,006	$0.076	2,058,792	0.21
Carbon in Leach	2546	20,343,342	$0.076	1,554,435	0.16
Tailings Pumping	64	509,306	$0.076	38,916	0.00
Concentrate handling	51	404,899	$0.076	30,938	0.00
Reagent Preparation	114	910,979	$0.076	69,608	0.01
Plant Utilities	749	5,984,350	$0.076	457,264	0.05
Sub Total					**2.19**
Labour					
Plant Labour				771,120	0.08
Maintenance Labour				370,440	0.04
Sub Total					**0.11**
Maintenance & Spares					
Maintenance & Spares (2.5% of capital Cost)				1,387,498	0.14
Sub Total					**0.14**
Total Plant Operating Cost					**6.50**

Oxide material mined throughout the mine life and stockpiled adjacent to the process plant will be processed at the end of mine life. A rehandling charge of US$0.35 has been estimated for getting the ore to the mill. Estimated process operating costs for the oxide feed are given in Table 21.10.

Table 21.10 Process Plant Operating Costs for Oxide Ore

OPEX	10MTPA				
	Consumption		Cost		
Consumables	kg/t Ore	tons/yr.	$/Unit	$/yr	$/t
Gyratory Crusher	N/A	N/A	$122.22	$910,050	$0.09
Cone Crusher	N/A	N/A	$91.09	$678,256	$0.07
SAG Mill Liners	0.033	329	$5.50	$1,810,130	$0.18
SAG Mill Media	0.404	4,044	$1.10	$4,448,924	$0.44
Ball Mill Liners	0.061	611	$5.50	$3,361,671	$0.34
Ball Mill Media	0.751	7,511	$1.10	$8,262,287	$0.83
Regrind Mill Liners	0.002	0.3	$5.50	$1,699	$0.00
Regrind Mill Media	0.029	3.8	$3.00	$11,388	$0.00
Laboratory Supplies -Fixed		1	$100,000	$100,000	$0.01
General Consumables -Fixed		1	$100,000	$100,000	$0.01
Sub Total					**$1.97**
Reagents					
Flocculant	0.03	250	$3.71	$928,125	$0.09
Cyanide	0.21	2100	$2.20	$4,620,000	$0.46
Lime - Leaching	0.60	6000	$0.20	$1,200,000	$0.12
Sodium Hydroxide (kg/t of Carbon)	15	23.5	$0.88	$20,717	$0.00
Hydrochloric Acid (Litre / t of Carbon)	150	235.4	$0.37	$87,107	$0.01
Activated Carbon (kg / t of Carbon)	25	39.2	$2.20	$86,323	$0.01
Sodium Metabisulphite	0.22	2170	$0.42	$900,637	$0.09
Copper Sulphate	0.09	856	$2.60	$2,225,546	$0.22
Lime - Detox	0.12	1211	$0.20	$242,178	$0.02
Fluxes (Smelting - kg/Oz of Gold)	0.15	5.5	$2.00	$11,084	$0.00
Sub Total					**$1.03**
Power	Absorbed kW	kWh/yr.	$/kWh	$/yr.	$/t
Crushing & Material Handling	824	6,135,271	$0.076	$ 468,796	$0.05
Grinding	27539	220,014,423	$0.076	$16,811,302	$1.68
Mill / Pre leach Thickeners	679	5,424,113	$0.076	$ 414,456	$0.04
Carbon in Leach	2546	20,343,342	$0.076	$1,554,435	$0.16
Tailings Pumping	64	509,306	$0.076	$ 38,916	$0.00
Reagent Preparation	114	910,979	$0.076	$ 69,608	$0.01
Plant Utilities	749	5,984,350	$0.076	$457,264	$0.05
Sub Total					**$1.98**
Labour					
Plant Labour				$771,120	$0.08
Maintenance Labour				$370,440	$0.04
Sub Total					**$0.11**
Maintenance & Spares					
Maintenance & Spares (2.5% of capital Cost)				$1,387,498	$0.14
Sub Total					**$0.14**
Total Plant Operating Cost					**$5.23**

21.2.3. INFRASTRUCTURE OPERATING COSTS

The Infrastructure operating costs are largely associated with the water supply costs, which have been calculated based upon the predicted availability of the different water sources for an average rainfall year. Infrastructure operating costs are summarised in Table 21.11.

Table 21.11 Summary of Site Infrastructure Operating Cost Estimate

INFRASTRUCTURE OPEX	Year 1	Final Year
Water Pumping Cost	$ 1,549,126	$ 1,395,331
Power Maintenance Cost	$ 854,432	$ 854,432
Site maintenance - excl. power (1% Capex)	$ 464,636	$ 464,636
Total Annual Infrastructure OpEx	**$ 2,868,193**	**$ 2,714,399**
Infrastructure Opex per tonne ROM	$ 0.29 /t concentrate	$ 0.27 /t concentrate

Note: Excluding contingency

21.2.4 GENERAL AND ADMINISTRATION OPERATING COSTS

The estimated general and administration costs are based on a cost of US$1.00 per tonne of mill feed or some US$10 million per annum, which is comparable to similar size operations.

21.3 CAPITAL & OPERATING COST REVIEW

The co-author and responsible QP of this section has reviewed the source capital and operating costs detailed above. This review relies on the inputs from the QP's responsible for the preceding Sections of this report. The review demonstrated that the raw cost data was typical and comparable with similar projects and that the method in which the final project cost was estimated to be in line with industry practise.

22.0 ECONOMIC ANALYSIS

22.1 Summary

An economic evaluation of the Project using discounted cashflow was prepared on a pre-tax and a post-tax basis. For the 23-year mine life, 225Mt total throughput, operating at 10 Mt/a, the PFS returns the following financial results:

- 18.6% Internal Rate of Return (IRR) pre tax, 16.5% IRR post-tax
- 6.3 years pre-tax payback, 6.8 years post-tax payback on $501.8 million initial capital
- US$675 million pre-tax Net Present Value (NPV) at a 5% discount value.
- US$558 million post-tax NPV at 5% discount value.

22.2 Principal Assumptions

22.2.1 Metal Prices

The base case copper and gold prices used in this analysis are $3/lb copper and $1,250/oz gold. These are estimated by Euromax to be realistic long-term prices for the current market.

22.2.2 Off-Site Charges

In the absence of letters of intent, the following off-site charges were assumed for the copper concentrate produced:

- 95.83% payable copper (based on a copper grade of 24% for concentrate and a deduction of 1% Cu)
- 97.00% payable gold
- $75/dmt concentrate – copper treatment charge
- $0.075/payable lb copper – copper refining charge
- $5.00/toz gold – gold refining charge
- 0.1% net invoice value (NIV) – insurance losses and marketing
- $45.00/wmt concentrate – transport charge.

It was assumed that concentrate would have a copper grade of 24% and an average moisture content of 10%.

For doré production the following terms were applied:

- 99% payability gold
- $1.00/toz gold – gold refining charge
- $5.00/toz gold – insurance and transport charge

22.2.3 By-Product Revenue

The concentrate produced from metallurgical test work was found to have payable quantities of silver in all assays completed. Concentrates from previous test work also all had payable silver in similar quantities. The assay from the most recent work was the lowest of the three available assays at 110 g/t silver (a smelter analysed a concentrate from a different batch of test work and obtained 113 g/t silver and a sample of concentrate produced at the Institute of Mining and Metallurgy at Bor, Serbia was assayed at the same laboratory and returned 145 g/t silver). The lowest of these values i.e. 110 g/t silver was adopted to generate a by-product revenue in the model. Silver adds some US$53 million to the NPV at a 5% discount rate.

22.2.4 Taxes, Royalties

Starting from 1 January 2009 a new tax regime became effective in Macedonia whereby the base for income tax computation had been shifted from “profit before tax” concept to “profit distribution” concept.

As per the Macedonian Corporate Income Tax (“CIT”) Law, tax is calculated and payable at a rate of 10% on two components, and both components are taxed separately from each other:

- Component 1: Expenses not recognized for tax purposes and understated revenues;
- Component 2: Profit distribution.

The tax rate on both components is currently set at 10% and this has been applied to the financial model with no tax holiday. Total tax payable for the life of the project is US$193 million.

A state royalty has been applied at 2% of the net smelter return (NSR). This is estimated as an average annual cost of $4.3 million during full production from Sulphide ore. The total royalty payable is $92.7 million for the LOM.

22.3 Discounted Cashflow

The production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered metal production from the relationships of tonnage processed, head grades and recoveries. Assumed market prices for copper and gold have been adjusted to realised price levels by applying refining and transportation charges from mine site to refinery to determine the net revenue contributions for each metal. It was noted that the mine schedule has resulted in higher than average grades being delivered to the mill in the first eight years of the mine life.

Unit operating costs for mining, processing and general and administrative areas were applied to annual milled tonnages to determine the overall mine site operating cost which has been deducted from the net revenue to derive the operating cash flow.

The initial capital costs and allowances for sustaining capital have been incorporated on a year-by-year basis over the mine life and deducted from the operating cash flow to determine the net cash flow before taxes. Initial capital costs include costs accumulated prior to first production of copper gold concentrate; sustaining capital includes expenditures for mining and processing additions, replacement of equipment and tailings embankment construction.

The working capital is recovered at the end of the mine life and aggregated with the salvage value contribution and applied towards reclamation during closure.

The discounted cash flow is given in Table 22.1.

Table 22.1 Project Discounted Cashflow

	Unit	Total	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year
			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Production																											
Oxide																											
Oxide Ore	t	16,230,000	0	0	1,720,000	1,860,000	1,890,000	1,440,000	1,440,000	1,440,000	1,440,000	1,440,000	360,000	270,000	270,000	270,000	270,000	270,000	270,000	270,000	270,000	270,000	270,000	270,000	230,000	0	0
Sulphide																											
Sulphide Ore Tonnes	t	208,650,000	0	0	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	8,650,000	0	0
Au grade	g/t	0.34	0.00	0.00	0.42	0.42	0.41	0.39	0.39	0.39	0.39	0.39	0.31	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.00	0.00
Cu grade	%	0.20%	0.00	0.00	0.25%	0.23%	0.22%	0.21%	0.21%	0.21%	0.21%	0.21%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.00%	0.00%
Waste																											
Pre-strip	t	10,000,000		10,000,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Waste	t	154,000,000	0	0	11,000,000	11,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	8,000,000	8,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	5,000,000	1,000,000	0	0	0	0
Strip ratio (incl Oxide)	Waste:ore	0.73	0	0	0.94	0.93	0.84	0.87	0.87	0.87	0.87	0.87	0.97	0.97	0.78	0.78	0.58	0.58	0.58	0.58	0.58	0.49	0.10	0	0	0	0
Total Waste (incl Prestrip)	t	164,000,000	-	10,000,000	11,000,000	11,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	8,000,000	8,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	5,000,000	1,000,000	0	0	0	0
In-Situ Metal Au - Oxide	oz	172,218	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	14,400	106,100	51,718
In-Situ Metal Au - Sulphide	oz	2,275,900	0	0	133,700	133,600	132,900	124,100	124,100	124,100	124,100	124,100	99,600	97,400	97,400	97,400	97,400	97,400	97,400	97,400	97,400	97,400	97,400	97,400	84,200	0	0
In-Situ Metal Au - Total	oz	2,448,118	0	0	133,700	133,600	132,900	124,100	124,100	124,100	124,100	124,100	99,600	97,400	97,400	97,400	97,400	97,400	97,400	97,400	97,400	97,400	97,400	97,400	98,600	106,100	51,718
In-Situ Metal Cu	t	410,546	0	0	24,675	23,269	22,453	21,410	21,410	21,410	21,410	21,410	18,370	18,098	18,098	18,098	18,098	18,098	18,098	18,098	18,098	18,098	18,098	18,098	15,649	0	0
Flotation Plant Feed																											
Sulphide Ore	t	208,650,000	0	0	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	10,000,000	8,650,000	0	0
Au grade	g/t	0.34	0.00	0.00	0.42	0.42	0.41	0.39	0.39	0.39	0.39	0.39	0.31	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.00	0.00
Cu grade	%	0.20%	0.00%	0.00%	0.25%	0.23%	0.22%	0.21%	0.21%	0.21%	0.21%	0.21%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.00%	0.00%
Au recovery to Conc	%		0	0	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%
Cu recovery to Conc	%		0	0	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%
Recovered Au to Conc	oz	1,251,745	0	0	73,535	73,480	73,095	68,255	68,255	68,255	68,255	68,255	54,780	53,570	53,570	53,570	53,570	53,570	53,570	53,570	53,570	53,570	53,570	53,570	46,310	0	0
Recovered Cu to Conc	t	344,859	0	0	20,727	19,546	18,860	17,984	17,984	17,984	17,984	17,984	15,431	15,203	15,203	15,203	15,203	15,203	15,203	15,203	15,203	15,203	15,203	15,203	13,145	0	0
Concentrate Grade (Cu)	%		0	0	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%	24%
Moisture Content	%		0	0	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%
Concentrate Produced	wmt	1,485,604	0	0	95,000	89,587	86,443	82,427	82,427	82,427	82,427	82,427	70,726	69,679	69,679	69,679	69,679	69,679	69,679	69,679	69,679	69,679	69,679	69,679	60,248	0	0
Concentrate Produced	dmt	1,436,913	0	0	86,364	81,443	78,585	74,933	74,933	74,933	74,933	74,933	64,297	63,344	63,344	63,344	63,344	63,344	63,344	63,344	63,344	63,344	63,344	63,344	54,771	0	0
CIL FEED																											
Oxide	t	16,230,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1,350,000	10,000,000	4,880,000
Au grade	g/t	0.33	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.33	0.33	0.33
Sulphide	t	206,563,500	0	0	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	8,563,500	0	0
Au grade	g/t	0.15	0.00	0.00	0.19	0.19	0.19	0.17	0.17	0.17	0.17	0.17	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.00	0.00
Au recovery to Dore	%		0	0	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%	73.7%
Recovered Au Dore	oz	874,179	0	0	43,898	43,865	43,636	40,746	40,746	40,746	40,746	40,746	32,702	31,980	31,980	31,980	31,980	31,980	31,980	31,980	31,980	31,980	31,980	31,980	38,258	78,196	38,116
Gross Au Recovery		87%	0	0	87.83%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	86%	74%	74%
Capital																											
Initial Capital																											
Infrastructure	USD	*103,812,878*	51,906,439	51,906,439																							
Tailings	USD	*58,071,611*	18,435,806	39,635,806																							
Mining Fleet	USD	*24,590,715*	16,109,925	8,480,789																							
Conveyor	USD	*10,162,591*	5,081,296	5,081,296																							
Processing Plant	USD	*249,523,116*	124,761,558	124,761,558																							
Owners costs	USD	*10,000,000*	5,000,000	5,000,000																							
Sub-Total		*456,160,911*																									
Contingency	USD	*45,616,091*	22,129,502	23,486,589																							
Total		*501,777,002*																									
Sustaining Capital																											
TMF	USD	*47,456,862*	0	0	9,050,346	5,569,444	6,961,805	3,828,993	3,828,993	2,552,662	2,552,662	2,552,662	3,016,782	3,016,782	3,016,782	377,238	377,238	377,238	377,238	0	0	0	0	0	0	0	0
Mining Fleet	USD	*127,994,202*	0	0	6,108,200	9,432,426	3,347,820	0	16,109,925	8,480,789	4,886,560	9,432,426	3,347,820	0	14,888,285	7,259,149	4,886,560	9,432,426	3,347,820	0	16,109,925	8,480,789	2,443,280	0	0	0	0
Infrastructure	USD	*30,565,833*	0	0	1,956,913	1,528,760	1,551,769	1,206,634	1,206,634	1,206,634	1,206,634	1,206,634	378,308	309,281	309,281	309,281	309,281	309,281	309,281	309,281	309,281	309,281	309,281	309,281	278,603	102,200	15,334,016
Reclamation Costs	USD	*30,000,000*	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	10,000,000	10,000,000	10,000,000
Working Capital	USD	*0*	0	0	29,511,205	-1,144,633	-814,756	-2,034,050	0	0	0	0	-5,963,269	-688,803	286,667	0	286,667	0	0	0	0	143,333	479,417	66,083	-2,518,583	-14,998,626	-2,610,652
Total Capex	USD	*737,793,898*	243,424,526	258,352,476	46,626,663	15,385,997	11,046,638	3,001,576	21,145,552	12,240,085	8,645,855	13,191,721	779,641	2,637,260	18,501,015	7,945,669	5,859,746	10,118,945	4,034,339	309,281	16,419,207	8,933,404	3,231,978	375,365	7,760,020	-4,896,426	22,723,364
Working Capital Required			0	0	29,511,205	28,366,572	27,551,816	25,517,766	25,517,766	25,517,766	25,517,766	25,517,766	19,554,497	18,865,694	19,152,361	19,152,361	19,439,027	19,439,027	19,439,027	19,439,027	19,439,027	19,582,361	20,061,777	20,127,861	17,609,278	2,610,652	1,269,506
Operating Cost																											
Mining Ore (Oxide)	USD	26,208,117	0	0	2,553,412	2,761,248	2,994,784	2,281,740	2,281,740	2,281,740	2,281,740	2,281,740	606,435	481,826	481,826	481,826	481,826	481,826	481,826	481,826	481,826	481,826	508,826	535,826	502,445	0	0
Mining Ore (Sulphide)	USD	359,804,688	0	0	14,845,420	14,845,420	15,845,420	15,845,420	15,845,420	15,845,420	15,845,420	15,845,420	16,845,420	17,845,420	17,845,420	17,845,420	17,845,420	17,845,420	17,845,420	17,845,420	17,845,420	17,845,420	18,845,420	19,845,420	18,896,288	0	0
Mining Waste	USD	245,380,000	0	0	15,620,000	15,620,000	15,200,000	15,200,000	15,200,000	15,200,000	15,200,000	15,200,000	16,200,000	17,200,000	13,760,000	13,760,000	10,320,000	10,320,000	10,320,000	10,320,000	10,320,000	8,600,000	1,820,000	0	0	0	0

	Unit	*Total*	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year	Year
			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Conveyor (Ore)	USD	22,488,000	0	0	1,172,000	1,186,000	1,189,000	1,144,000	1,144,000	1,144,000	1,144,000	1,144,000	1,036,000	1,027,000	1,027,000	1,027,000	1,027,000	1,027,000	1,027,000	1,027,000	1,027,000	1,027,000	1,027,000	1,027,000	888,000	0	0
Rehandle Oxide	USD	5,680,500	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	472,500	3,500,000	1,708,000
Processing Oxides	USD	84,963,982	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	7,067,244	52,349,958	25,546,780
Processing Sulphides - Conc + CIL	USD	1,356,648,307	0	0	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	65,020,288	56,242,549	0	0
Water Pumping	USD	59,246,408	0	0	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,839,512	2,456,177		
G&A	USD	222,793,500	0	0	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,900,000	9,913,500	10,000,000	4,880,000
Total Opex	USD	2,383,213,502	0	0	111,950,632	112,172,468	112,989,004	112,230,960	112,230,960	112,230,960	112,230,960	112,230,960	112,447,655	114,314,046	110,874,046	110,874,046	107,434,046	107,434,046	107,434,046	107,434,046	107,434,046	105,714,046	99,961,046	99,168,046	96,438,704	65,849,958	32,134,780
Revenue																											
Au payability	%				97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%	97.00%
Cu payability	%				95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%	95.83%
Ag per ton Conc	g/t																										
Contained Ag in Conc	oz	5,589,936			335,977	316,831	305,714	291,509	291,509	291,509	291,509	291,509	250,130	246,424	246,424	246,424	246,424	246,424	246,424	246,424	246,424	246,424	246,424	246,424	213,073	0	0
Ag payability	%				90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%
Au Payable	oz	*1,214,193*			71,329	71,276	70,902	66,207	66,207	66,207	66,207	66,207	53,137	51,963	51,963	51,963	51,963	51,963	51,963	51,963	51,963	51,963	51,963	51,963	44,921	0	0
Ag Payable	oz	*5,030,942*			302,379	285,148	275,143	262,358	262,358	262,358	262,358	262,358	225,117	221,782	221,782	221,782	221,782	221,782	221,782	221,782	221,782	221,782	221,782	221,782	191,766	0	0
Cu Payable	t	*330,490*			19,864	18,732	18,075	17,235	17,235	17,235	17,235	17,235	14,788	14,569	14,569	14,569	14,569	14,569	14,569	14,569	14,569	14,569	14,569	14,569	12,597	0	0
Au Revenue	USD	*1,517,740,813*			89,161,188	89,094,500	88,627,688	82,759,188	82,759,188	82,759,188	82,759,188	82,759,188	66,420,750	64,953,625	64,953,625	64,953,625	64,953,625	64,953,625	64,953,625	64,953,625	64,953,625	64,953,625	64,953,625	64,953,625	56,150,875	0	0
Ag Revenue	USD	*90,556,961*			5,442,823	5,132,662	4,952,568	4,722,449	4,722,449	4,722,449	4,722,449	4,722,449	4,052,101	3,992,070	3,992,070	3,992,070	3,992,070	3,992,070	3,992,070	3,992,070	3,992,070	3,992,070	3,992,070	3,992,070	3,451,790	0	0
Cu Revenue	USD	*2,185,199,409*			131,338,910	123,854,524	119,508,751	113,955,819	113,955,819	113,955,819	113,955,819	113,955,819	97,779,887	96,331,296	96,331,296	96,331,296	96,331,296	96,331,296	96,331,296	96,331,296	96,331,296	96,331,296	96,331,296	96,331,296	83,293,978	0	0
Total Revenue	USD	*3,702,940,222*			220,500,098	212,949,024	208,136,439	196,715,007	196,715,007	196,715,007	196,715,007	196,715,007	164,200,637	161,284,921	161,284,921	161,284,921	161,284,921	161,284,921	161,284,921	161,284,921	161,284,921	161,284,921	161,284,921	161,284,921	139,444,853	0	0
Offsite Charges																											
Illovitsa to Pirdop	USD/t	*71,127,180*			4,275,018	4,031,405	3,889,952	3,709,207	3,709,207	3,709,207	3,709,207	3,709,207	3,182,688	3,135,537	3,135,537	3,135,537	3,135,537	3,135,537	3,135,537	3,135,537	3,135,537	3,135,537	3,135,537	3,135,537	2,711,179	0	0
Insurance	%	*3,702,940*			220,500	212,949	208,136	196,715	196,715	196,715	196,715	196,715	164,201	161,285	161,285	161,285	161,285	161,285	161,285	161,285	161,285	161,285	161,285	161,285	139,445	0	0
TC (Cu)	USD/dmt	*107,768,455*			6,477,300	6,108,189	5,893,866	5,620,010	5,620,010	5,620,010	5,620,010	5,620,010	4,822,254	4,750,814	4,750,814	4,750,814	4,750,814	4,750,814	4,750,814	4,750,814	4,750,814	4,750,814	4,750,814	4,750,814	4,107,846	0	0
RC (Cu)	USD/lb	*54,645,413*			3,284,400	3,097,238	2,988,563	2,849,700	2,849,700	2,849,700	2,849,700	2,849,700	2,445,188	2,408,963	2,408,963	2,408,963	2,408,963	2,408,963	2,408,963	2,408,963	2,408,963	2,408,963	2,408,963	2,408,963	2,082,938	0	0
RC (Au)	USD/oz	*6,070,963*			356,645	356,378	354,511	331,037	331,037	331,037	331,037	331,037	265,683	259,815	259,815	259,815	259,815	259,815	259,815	259,815	259,815	259,815	259,815	259,815	224,604	0	0
RC (Ag)	USD/oz	*2,012,377*			120,952	114,059	110,057	104,943	104,943	104,943	104,943	104,943	90,047	88,713	88,713	88,713	88,713	88,713	88,713	88,713	88,713	88,713	88,713	88,713	76,706	0	0
Total Offsite Charges		243,405,508			14,740,257	13,925,350	13,450,038	12,816,334	12,816,334	12,816,334	12,816,334	12,816,334	10,974,112	10,809,117	10,809,117	10,809,117	10,809,117	10,809,117	10,809,117	10,809,117	10,809,117	10,809,117	10,809,117	10,809,117	9,346,169	0	0
Dore																											
Payability	%				99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%	99.90%
Payable Dore	oz	*873,305*			43,854	43,821	43,592	40,705	40,705	40,705	40,705	40,705	32,669	31,948	31,948	31,948	31,948	31,948	31,948	31,948	31,948	31,948	31,948	31,948	38,220	78,118	38,078
Revenue	USD	*1,091,630,818*			54,817,863	54,776,863	54,489,858	50,881,801	50,881,801	50,881,801	50,881,801	50,881,801	40,836,643	39,934,629	39,934,629	39,934,629	39,934,629	39,934,629	39,934,629	39,934,629	39,934,629	39,934,629	39,934,629	39,934,629	47,775,278	97,646,880	47,597,505
Refining	USD/oz	*873,305*			43,854	43,821	43,592	40,705	40,705	40,705	40,705	40,705	32,669	31,948	31,948	31,948	31,948	31,948	31,948	31,948	31,948	31,948	31,948	31,948	38,220	78,118	38,078
Transport/Insurance	USD/oz	*4,370,894*			219,491	219,327	218,178	203,731	203,731	203,731	203,731	203,731	163,510	159,898	159,898	159,898	159,898	159,898	159,898	159,898	159,898	159,898	159,898	159,898	191,292	390,979	190,581
Dore Net Revenue	USD	*1,086,386,619*			54,554,518	54,513,715	54,228,089	50,637,365	50,637,365	50,637,365	50,637,365	50,637,365	40,640,464	39,742,783	39,742,783	39,742,783	39,742,783	39,742,783	39,742,783	39,742,783	39,742,783	39,742,783	39,742,783	39,742,783	47,545,765	97,177,784	47,368,847
Royalty	USD	*92,689,318*			5,315,144	5,173,401	5,077,341	4,785,170	4,785,170	4,785,170	4,785,170	4,785,170	3,958,382	3,884,213	3,884,213	3,884,213	3,884,213	3,884,213	3,884,213	3,884,213	3,884,213	3,884,213	3,884,213	3,884,213	3,621,925	1,943,556	947,377
Net Revenue	USD	*4,541,776,598*	0	0	260,442,038	253,496,649	248,789,717	234,473,317	234,473,317	234,473,317	234,473,317	234,473,317	193,960,709	190,326,444	190,326,444	190,326,444	190,326,444	190,326,444	190,326,444	190,326,444	190,326,444	190,326,444	190,326,444	190,326,444	177,474,315	95,234,229	46,421,470
Pre-Tax Net Cashflow	USD	*1,420,769,198*	-243,424,526	-258,352,476	100,958,514	125,989,827	124,784,061	119,279,096	101,096,806	110,002,272	113,596,502	109,050,636	80,845,026	73,385,133	60,951,383	71,506,730	77,032,652	72,773,453	78,858,059	82,583,117	66,473,192	75,678,994	87,133,420	90,783,034	73,365,548	34,855,419	-8,436,674
NPV	USD	675,125,139																									
IRR	%	18.6%																									
Tax Paid	USD	*193,098,287*	0	0	10,095,851	12,598,983	12,478,406	11,927,910	10,109,681	11,000,227	11,359,650	10,905,064	8,084,503	7,338,513	6,095,138	7,150,673	7,703,265	7,277,345	7,885,806	8,258,312	6,647,319	7,567,899	8,713,342	9,078,303	7,336,555	3,485,542	0
Post-Tax Net Cashflow	USD	1,227,670,911	-243,424,526	-258,352,476	90,862,662	113,390,844	112,305,655	107,351,186	90,987,125	99,002,045	102,236,852	98,145,572	72,760,523	66,046,620	54,856,245	64,356,057	69,329,387	65,496,108	70,972,253	74,324,805	59,825,873	68,111,095	78,420,078	81,704,730	66,028,993	31,369,877	-8,436,674
NPV	USD	558,403,580																									
IRR	%	16%																									

22.4 SENSITIVITY

Sensitivity analyses were carried out on the following parameters:

- discount rate
- copper price
- gold price
- capital cost
- on-site operating costs

The results for changes in discount rate and metal price are presented in Table 22.2.

Table 22.2 Sensitivity of the Project NPV to Metal Price and Discount Rate

Gold (US$/oz)	Copper (US$/lb)	NPV @ 0% discount (US$M)	NPV @ 5% discount (US$M)	NPV @ 7.5% discount (US$M)	Pre-tax IRR (%)
1,100	2.50	757.5	284.3	146.4	11.4%
1,250	3.00	1,420.8	675.1	459.0	18.6%
1,400	3.50	2,084.0	1,066.0	771.6	24.9%

Clearly the project is sensitive to both changes in discount rate and metal prices but the project still does offer positive returns on investment at lower prices and rates.

The sensitivities to changes in copper prices are shown in Table 22.3.

Table 22.3 Project NPV Sensitivity to Copper Price

Copper Price US$/lb	$2.5	$2.75	$3.0	$3.25	$3.5
NPV US$M @ 5% discount	$463M	$569M	$675M	$781M	$887M

Sensitivities to a change only in gold prices are shown in Table 22.4

Table 22.4 Project NPV Sensitivity to Gold Price

Gold Price US$/oz	$1,100	$1,175	$1,250	$1,325	$1,400
NPV US$M @ 5% discount	$496M	$586M	$675M	$765M	$854M

The results indicate that the project is more sensitive to gold prices which cause the same magnitude of change in NPV for a much smaller swing in price as a percentage of the base case.

An analysis of sensitivity to operating and capital costs has also been completed. Changes in project NPV in response to variations in capital and operating costs are given in Table 22.5.

Table 22.5 Project NPV Sensitivity to Operating and Capital Costs

	-5%	-2%	0%	+2%	+5%
NPV US$M with changes in capex	$707M	$688M	$675M	$662M	$643M
NPV US$M with change in opex	$744M	$703M	$675M	$648M	$606M

The results indicate the project is more sensitive to changes in operating costs than to changes in capital but that the project remains viable within the ranges tested.

22.5 Model Review

The co-author and responsible QP of this section has reviewed all the input data and the construction of the Financial Model and they were found to be reasonable and to comply with normal industry practise. Some minor variations were tested, which had no effect or led to very minor improvements and none had a material negative effect.

22.5.1 Metal Recoveries

Recovery to a copper flotation concentrate of 84% (Cu) and 55% (Au) was used as detailed in Section 13.5.2.3. Higher recoveries of 87% and 64.9% had been achieved in the Locked Cycle test (Section 13.5.2.1) but this was with a different reagent regime.

A gold leach recovery of the sulphide flotation tailings of 73.7% was used (Section 13.5.3). Although the cyanidation tests reported a silver leach recovery of 37.4%, from an average cyanidation feed of 0.51 gpt Ag, this silver revenue was not used in the model.

The same gold leach recovery of 73.7% was used for the treatment of the oxide ore in the latter years although a much higher average recovery of 93% had been achieved in the testwork.

22.5.2 Capital and Operating Costs

All costs were taken from Section 21, which in turn relies on the inputs from the QP's responsible for the preceding Sections of this report.

22.5.3 Concentrate Revenue

No indicative terms had been received from local copper smelters but terms for typical smelter contracts were used and are reasonable.

The refinery terms for the doré are for a high gold content doré with no silver content or payment.

23.0 ADJACENT PROPERTIES

There are no material properties adjacent to the Property.

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no other additional information or explanation necessary to make the Technical Report understandable and not misleading.

25.0 INTERPRETATION AND CONCLUSIONS

The preparation of this Section of the Report was supervised by Dr David Patrick PhD, CEng, FIMMM, FAusIMM of A C A Howe International Limited, and Daniel Leroux, M.Sc., P.Geo of ACA Howe International Limited, both independent QPs as defined by NI 43-101. The contents of this Section rely on the information supplied by the other QPs responsible for the individual Sections this report.

The current Pre-Feasibility Study has defined a mining project at the Euromax owned Ilovitza Gold-Copper project that justifies continuing development to Feasibility Study and Front End Engineering level. Overall the study complies with industry standard practices for PFS level and is considered to have an accuracy of plus or minus 15% or better. The scope of the project including preliminary mine design, mine schedule, process flow sheet and process plant design, waste management and tailings management facility have been assessed and viable solutions defined in each case.

Based on the supplied data and application of appropriate methods, an open pit mining project was outlined that could be profitably mined with the parameters applied that are set out in Table 25.1.

Table 25.1 Mining Parameters

Milling Rate	10 Million Tonnes Per Year
Mine Life	22-23 Years
Average Diluted Mill Feed Grades: Gold Copper	 0.34 g/tonne 0.20 %
Average Yearly Metal Production Delivered to Mill: Gold Copper	 100,000-110,000 ounces 38-39 Million Pounds
Total Tonnes Milled	225 Million
Total Waste Tonnes (Incl. Sub-Grade and In-Pit Inferred Mineral Resources)	164 Million
Stripping Ratio: Year 1 Life of Mine	 1.1:1* 0.7:1
Pre-production Stripping	10 Million tonnes*
Minimum Pit Elevation	260 metres

* Early stripping requirements are high because of tailings dam construction.

Based on the results of a positive economic analysis of the proposed mine, mineral reserves were identified. The preliminary mine plan is based on Measured and Indicated mineral resources. This report's PFS level of detail requires that both the Measured and Indicated mineral resources be classified as a Probable mineral reserve. No Proven mineral reserves have been designated.

Table 25.2 Mineral Reserves

Probable Reserve, Oxide (Diluted and Recovered)	16 Million tonnes
Gold Grade	0.33 g/tonne
Gold Ounces	172,000
Primary/Transitional Probable Reserve (Diluted and Recovered)	209 Million tonnes
Gold Grade	0.34 g/tonne
Gold Ounces	2.28 Million
Copper Grade	0.20%
Copper Pounds	905 Million
Total Probable Reserve (Diluted and Recovered, Rounded)	**225 Million Tonnes**
Gold Grade	**0.34 g/tonne**
Gold Ounces (Rounded)	**2.45 Million**
Copper Grade	**0.20%**
Copper Pounds	**905 Million**

Notes:
1. Unplanned dilution equals 5% at diluting grades of 0.17 g/tonne gold and 0.05 % copper.
2. Mining losses = 5%.

Some of the blocks that are part of the mine plan belong to the Inferred mineral resource category. These are planned to be mined but are not considered to be part of the Mineral Reserve, because Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves.

With additional drilling, it is possible that these in-pit Inferred mineral resources could be upgraded to higher mineral resource categories. However, there is no guarantee that this would occur.

For the purpose of mine scheduling, the in-pit Inferred mineral resource blocks were considered to be waste rock.

Table 25.3 In-Pit Inferred Mineral Resources

Oxide	
Tonnes (Millions)	2.14
In-Situ Ounces (000s)	19.7
In-Situ Gold Grade (g/tonne)	0.29
Primary/Transitional	
Tonnes (Millions)	15.34
In-Situ Ounces (000s)	166
In-Situ Gold Grade (g/tonne)	0.34
Total (Rounded)	
Tonnes (Millions)	17.5
In-Situ Ounces (000s)	186
In-Situ Gold Grade (g/tonne)	0.33

Based on the positive results of this report, the project warrants additional, more detailed mine design and production scheduling work to at least Feasibility study level.

Processing of the ore using a flow sheet comprising crushing, grinding by SAG and then ball mill, flotation of a copper concentrate and treatment of the flotation tailings for additional recovery of gold has been defined as a viable process route. The process route has had sufficient testwork carried out in order to establish process operating and capital costs to the required level of detail for a PFS level study. Costs have been established for a 10 million tonne per year operation. Oxide ore will be processed at the end of mine life once the Sulphide ore is exhausted. The process will produce a copper concentrate with payable gold credits and gold doré.

Existing infrastructure has been examined and the required additional infrastructure designed to a level appropriate for the study. This comprises a system of roads and power lines to connect to the local networks, a water pumping network to ensure sufficient make up water for the plant, a series of buildings and workshops to house the various parts of the project and accommodate the required support for this and a tailings and waste management facility in the Shtuka valley, adjacent to the mine. The footprint is as compact as possible and has the advantage of impacting only the drainage systems, which pass directly by the deposit.

No fatal flaws have been found with respect to environmental and social issues and the project remains within the parameters of the EIS approved in 2012.

Financial analysis of the parameters defined by the PFS demonstrate a viable project. The mine schedule taken forward to the financial model delivers higher than average grade in the first eight years. As with all bulk mining projects, the project is sensitive to changes in metal price but still returned a positive return within the range of sensitivities tested for metal price, discount rate, operating costs and capital costs.

26.0 RECOMMENDATIONS

The preparation of this Section of the Report was supervised by Dr David Patrick PhD, CEng, FIMMM, FAusIMM of A C A Howe International Limited, and Daniel Leroux, M.Sc., P.Geo of ACA Howe International Limited, both independent QPs as defined by NI 43-101. The contents of this Section rely on the information supplied by the other QPs responsible for the individual Sections the Report.

Specific recommendations have been made in the various sections of this report. In general terms the definition of a viable project from the positive results of this pre-feasibility study leads to the recommendation that the project should be advanced to the next stages of feasibility study and front end engineering. Key aspects of this are as follows.

26.1 Infill Drilling

It is recommended that an infill-drilling programme is undertaken on the property. Drill holes are currently spaced on a 100 m grid, with small distinct areas being infill drilled on a 50 m grid. Extending the infill drilling on a 50 m grid over central region of the porphyry is recommended. The proposed drilling programme is concentrated within an area defined by a 200m radius from the deepest part of the Resource pit shell. This targets the infill drilling on the centre of the porphyry, where the mineralisation outcrops and the initial phases of mining are likely to be concentrated. Inclusion of inclined holes would enable the optimisation of variography and would also allow the investigation of structures.

The total additional drill length would be approximately 10,000 m, calculated from hole trajectories from the topography to beyond the Resource currently within the constraining pit shell. The additional information would provide the following benefits:

- An increased understanding of the short-range variability of the grade continuity.
- Allow further Measured mineral resources to be defined in areas likely to be in the mine plan of the early years from which Proven mineral reserves might be derived
- Conversion of Inferred mineral resources that fall within the current reserve pit
- Sufficient data to allow further segregation of mineralisation populations on the basis of lithology and alteration.
- Increased understanding of the structural geology and controls of the deposit.
- Additional material for metallurgical testwork.
- Additional geotechnical information.

Two specific holes are recommended for investigating a northwest trending structure on the eastern side of the deposit that possibly has controlled emplacement of the porphyry or subsequently acted as a conduit for a gold mineralizing overprint. The structure coincides with a feature identified from the IP survey along a contact between zones of high and low resistivity. The two holes are inclined at 55 degrees to the southwest.

The approximate costs associated with completing the proposed infill-drilling programme are included in the proposed feasibility study budget in Table 26.1.

It is recommended that geotechnical data collection is integrated into the infill drilling campaign to increase the knowledge in relation to rock mass characterisation.

With respect to other geological and field work the following is recommended:

- Prepare a full set of procedural documentations in relation to all aspects geological field work, sample preparation and database management.
- Install permanent drill hole collar markers on all locations drilled on the property. Markers should include drill hole reference number.
- Complete a programme of in-situ density testwork using existing core samples and material collected during the proposed infill drilling campaign.

26.2 Mining

More-detailed mine design, production scheduling, and equipment selection work should be carried out to support a definitive feasibility study. This will require further geotechnical drilling and modelling of ground water. It is recommended that sufficient Measured mineral resources be defined to support Proven mineral reserves for the first three to five years of the mine schedule, assuming the required engineering detail is also achieved.

26.3 Metallurgical Testwork and Mineral Processing.

Further metallurgical testwork is required to support feasibility and front end engineering studies. This should include geo-metallurgy to investigate variability within the deposit. In particular the higher-grade areas which fall within the early years of the current mine plan. Further optimisation of the process route may also be achieved by investigating the following:

- Testing a gravity circuit for the extraction of gold
- Batch testing using a much higher mass pull for the rougher stage of concentration in order to recover more gold and then using the cleaner stage to improve the concentrate quality with respect to copper. This would potentially make it possible to process only the cleaner tailings for further gold recovery rather than the rougher and cleaner tailings. The reduced quantity of throughput capacity required for the carbon in leach plant could lead to considerable savings in operating cost and capital.

Feasibility level design for the process plant should be advanced. Commissioning of a feasibility study and front end engineering and design from an engineering group and / or equipment supplier could streamline the process for this scale of plant. Integration of mine infrastructure studies is advisable to ensure the integrity of the study.

26.4 Infrastructure

More detailed geotechnical investigations are required over the proposed sites for the mine infrastructure, in particular the plant site, tailings management facility, truck workshop and proposed road corridor. More detailed design of the tailings facility is required and this should be supported by an appropriate level of tailings testwork to investigate tailings rheology and any potential for acid drainage.

26.5 Water

The current studies into water should be continued with the drilling of boreholes to investigate ground water levels, flows and quality, and the continued monitoring of existing drill holes, wells and surface water. This will enable modelling of surface and underground water which will be vital for all the engineering aspects of the project as well as for environmental and social considerations,

26.6 Environmental and Social

The current baseline monitoring should continue in order to allow the impacts of the feasibility study and engineering to be correctly assessed in an updated environmental and social impact assessment. This assessment should include stakeholder engagement and aim to be equator principle and IFC performance standard compliant to ensure there are no barriers to financing construction.

26.7 Feasibility Study and Front End Engineering Budget

Table 26.1 gives the budget cost for the feasibility study and front end engineering of the Ilovitza project.

Table 26.1 Ilovitza Feasibility Study and Front End Engineering Provisional Budget

Category	Item	Description	VAT	Cost
Drilling	Drilling	14,000 metres extra drilling to cover infill, sterilisation, pit geotechnical	Y	$1,600,000
	Mobilization/Demobilization		Y	$60,000
	Road, drill pads repairing		N	$90,000
	Core boxes, sample bags etc.		N	$60,000
Metallurgy	Metallurgical testing with mineralogy	Four additional samples	N	$380,000
Hydrology & Geotechnical	Hydrogeology, including drilling and 3D modelling	Five 400 metre holes fully equipped	Y	$350,000
	Geotechnical Study for Infrastructure	Includes test pits and drilling	Y	$400,000
Land & Roads	Land survey and urbanisation of production area	Based on local quotes	Y	$300,000
	Access road construction study and project	Based on local quotes	Y	$50,000
Environmental & Social	Environmental monitoring / weather station	Quoted from Golder Associates	N	$740,000
	Water Studies	Quoted from Schlumberger Water Services	N	$300,000
	Local E&S consultants		Y	$100,000
	Waste Rock Characterisation, ARD studies		N	$100,000
	Environmental impact reporting		N	$1,000,000
	Local consultation		N	$100,000
Mining	Mining Study		N	$350,000
Plant design	Plant Engineering		N	$2,640,000
Tailings	TMF Engineering including site investigation		N	$200,000
	Paste Tailings Rheology and characterisation		N	$30,000
Infrastructure & marketing	Marketing Studies		N	$75,000
Reporting	PFS		N	$30,000
	Macedonian Technical Report for Construction Permit		N	$500,000
	Translation of technical documentation		Y	$15,000
Permitting	Elaborate on Ilovitza 11, including expert revision		N	$100,000
	Conceptual study for Ilovitza 11 for Mining concession		N	$100,000
	Cadastral report for Ilovitza 11 for Mining concession		N	$50,000
	Concession fees	180,000 MKD * 1.68km = 302,400 MKD	N	$24,000
Tendering	Preparation of bid requests		N	$50,000
Community	Community Projects		N	$100,000
Other	Vehicle purchases		N	$186,000
	Equipment purchases		N	$77,500
Subtotal				$10,157,500
Contingency at 10%				$1,015,750
VAT at 18%				$517,500
Total				$11,690,750

27.0 REEFERENCES

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Civil Engineering Faculty, Skopje. Preliminary Solution on Tailings Dam of Mine for Copper and Gold Ilovica, Municipality Bosilovo May 2014. Civil Engineering Faculty, Skopje. Macedonia

SGS Mineral Services UK Ltd. Phase II Metallurgical Testing on the Illovitza Deposit, Macedonia 23rd May 2014. SGS Mineral Services UK Ltd. Truro, Cornwall, UK

CERTIFICATE OF QUALIFIED PERSON

Robert Davies
Swindon, UK
Telephone: +44 1793 512305
Email: robert.davies@tatratech.com

I, Robert Davies do hereby certify that:

1. I am a Resource Geologist with Tetra Tech WEI Inc. located at Unit 2, Apple Walk, Kembrey Park, Swindon, SN2 8BL, UK. This certificate applies to the technical report titled "Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in southeast Macedonia" for Euromax Resources dated 19th December 2014, effective date 5th June 2014 (the "Technical Report").

2. I graduated with a Bachelor of Science (Honours) degree in Geology from the University of Liverpool, UK in 2004 and have practiced the profession of Geology since my graduation. I have been employed with Tetra Tech since 2012; as a Resource Geologist. My relevant experience is ten years professional practise as a consulting geologist working in mineral exploration and resource estimation, most recently completing a JORC compliant resource estimations for the Zegen Gol Gold Project in Russia, NI 43-101 compliant resource estimations for the Mangazeisky Silver Project in Russia, the Trun Gold/Silver Project in Bulgaria and technical due diligence reviews of the Red Chris copper gold project in British Columbia, and the Jaquipe Iron Ore project in Brazil. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

3. I am a Chartered Geologist registered and in good standing of the Geology Society of London (#1013629), and a European Geologist registered and in good standing of the European Federation of Geologists (#965).

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

5. I completed a site visit to the Ilovitza Property between 17 and 20 June 2013.

6. I am responsible for the preparation of Sections 7, 8, 9, 10, 11, 12 and 14 of the technical report titled "Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia", dated 19th December 2014, effective date 5th June 2014 (the "Technical Report") relating to the Ilovitza Property held in Macedonia by Euromax Resources Limited.

7. I am independent of issuer applying all of the tests in section 1.5 of National Instrument 43-101.

8. I have had no prior involvement with Euromax Resources Ilovitza Property that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.

9. As of the effective date of this report, 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.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: June 5th 2014
Signed Date: December 19th 2014

QPCertificate-19December2014
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QPCertificate-19December2014

____
Robert Davies, BSc (Hons), CGeol, EurGeol,
FGS, MIQ

CERTIFICATE OF QUALIFIED PERSON

Arunasalam Vathavooran
Swindon, UK
Telephone: +44 1793 512305
Email: arun.vathavooran@tatratech.com

I, Arunasalam Vathavooran do hereby certify that:

1. I am a Senior Metallurgical Engineer with Tetra Tech WEI Inc. located at Unit 2, Apple Walk, Kembrey Park, Swindon, SN2 8BL, UK. This certificate applies to the technical report titled “Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in southeast Macedonia” for Euromax Resources dated 19th December 2014, effective date 5th June 2014 (the “Technical Report”).

2. I graduated with a Doctorate in Mineral Processing from the University of Nottingham, UK in 2006 and have practiced the profession of Metallurgical Engineering since my graduation. I have been employed with Tetra Tech since 2011; as a Metallurgical Engineer and from 2012 as a Senior Metallurgical Engineer. My relevant experience for 12 years in Metallurgical engineering covers a wide range of commodities including precious metals and base metals. I have delivered detailed studies including preliminary economic assessments, prefeasibility studies and feasibility studies for mining projects located in Europe, Asia and Africa.

3. I am a Chartered Engineer registered and in good standing of the Engineering Council (579205), the Institute of Materials, Minerals and Mining (444570), and the Society for Mining, Metallurgy and Exploration (4149245).

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

5. I have not completed a site visit to the Ilovitza Property.

6. I am responsible for the preparation of Sections 13 and 17 of the technical report titled “Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia”, dated 19th December 2014, effective date 5th June 2014 (the “Technical Report”) relating to the Ilovitza Property held in Macedonia by Euromax Resources Limited.

7. I am independent of issuer applying all of the tests in section 1.5 of National Instrument 43-101.

8. I have had no prior involvement with Euromax Resources Ilovitza Property that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.

9. As of the effective date of this report, 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.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: June 5th 2014
Signed Date: December 19th, 2014

____
Arun Vathavooran, PhD, CEng, MIMMM, SME

CERTIFICATE OF QUALIFIED PERSON

Laszlo Bodi
Mississauga
Ontario, Canada
Telephone: +1 - 289-981-9521
Email: Laszlo.Bodi@tetratech.com

I, Laszlo Bodi do hereby certify that:

1. I am a Principal Civil/Geotechnical Engineer with Tetra Tech Inc. located at 6835A Century Avenue, Mississauga, Ontario, Canada, L5N 2L2. This certificate applies to the technical report titled "Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia" for Euromax Resources dated 19th December 2014, effective date 5th June 2014 (the "Technical Report").

2. I graduated with a Master of Science Civil Engineering degree from Budapest University of Technology and Economics, Budapest, Hungary in 1978 and have practiced the profession of Civil/Geotechnical Engineering since my Masters graduation. I have been employed with Tetra Tech since 2011; as Principal Civil/Geotechnical Engineer. My relevant experience for 36 years in geotechnical and civil engineering covers a wide range of areas including detailed studies for a wide variety of mining projects such as mine extensions, mine infrastructure design (including airfields and access roads), tailings dams, waste rock dumps, seismic studies and mine backfill design in Canada, USA, Spain, Burkina Faso, Gabon, Turkey, Russia, China and Armenia.

3. I am a Professional Engineer registered and in good standing of the Professional Engineers of Ontario (#90235631), the Ontario Society of Professional Engineers (11386128) and the Canadian Geotechnical Society (#060398).

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

5. I completed a site visit to the Ilovitza Property between May 19 and 24, 2013.

6. I am responsible for the preparation of Sections 18.1, 18.2, 18.3 (*except 18.3.1.1, 18.3.1.2, 18.3.1.3, 18.3.1.4*) 18.4, 18.5, 18.6, 18.10, 18.11, 18.12, 18.13 and 18.14 of the technical report titled "Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia", dated 19th December 2014, effective date 5th June 2014 (the "Technical Report") relating to the Ilovitza Property held in Macedonia by Euromax Resources Limited.

7. I am independent of issuer applying all of the tests in section 1.5 of National Instrument 43-101.

8. I have had no prior involvement with Euromax Resources Ilovitza Property that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.

9. As of the effective date of this report, 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.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: June 5, 2014
Signed Date: December 19, 2014

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LICENSED PROFESSIONAL ENGINEER
C. Bodi
L. BODI
90235631
Dec. 19/2014
PROVINCE OF ONTARIO

C. Bodi

Laszlo Bodi, P.Eng.

A.C.A. HOWE INTERNATIONAL LIMITED
Mining and Geological Consultants

CERTIFICATE OF QUALIFIED PERSON

William Douglas Roy, M.A.Sc., P.Eng.

I, William Douglas Roy, M.A.Sc., P.Eng., do hereby certify that:

1. I am an Associate Mining Engineer with ACA Howe International Limited, whose office is located at 365 Bay Street, Suite 501, Toronto, Ontario, Canada.
2. I graduated with a Bachelor of Engineering ("B.Eng.") degree in Mining Engineering from the Technical University of Nova Scotia (now Dalhousie University) in 1997 and with a Master of Applied Science ("M.A.Sc.") degree in Mining Engineering from Dalhousie University in 2000.
3. I am a Professional Mining Engineer registered with the Association of Professional Engineers of Nova Scotia (Registered Professional Engineer, No. 7472). I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), the Prospectors and Developers Association of Canada ("PDAC"), and the Society of Mining, Metallurgy, and Exploration ("SME" - USA).
4. I have worked as a mining engineer for more than fifteen years since graduating from university. This work has included mine design, the estimation of mineral resources and mineral reserves for precious metals, base metals and industrial minerals, surface and underground mine design, and mine feasibility studies.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43- 101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am co-author of the technical report titled "Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia ", dated 19th December 2014 (the "Technical Report") relating to the Ilovitza Property held in Macedonia by Euromax Resources Limited. I am responsible for Sections 15, 16 18.3.1.1.-18.3.1.4, 18.7, 18.9 and Section 18.14.I have read NI 43-101 and Form 43-101 F1. This Technical Report has been prepared in compliance with that Instrument and form.
7. I visited the Mineral Property that is the subject of this technical report on April 7-10, 2014.
8. I have had no prior involvement with Euromax Resources or the Ilovitza Property that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.
9. I am not aware of any material fact or material change with respect to the subject matter of this Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.
10. I am independent of the issuer, Euromax Resources, applying all of the tests in Section 1.5 of NI 43-101.
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 report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes.

Effective Date: December 19, 2014

Signed Date: December 19, 2014

REGISTERED PROFESSIONAL ENGINEER
DATE Dec 19, 2014
W. D. ROY
7472
PROVINCE OF NOVA SCOTIA

William Douglas Roy, M.A.Sc., P. Eng.

DATE AND SIGNATURE PAGE

A C A HOWE INTERNATIONAL LIMITED
Geological Consultants

CERTIFICATE of AUTHOR

I, David J Patrick, do hereby certify that:

1. I am Director and Senior Geologist of ACA Howe International Limited of Wingbury Courtyard Business Village, Upper Wingbury Farm, Wingrave, Aylesbury, Buckinghamshire HP22 4LW, UK
2. I graduated with a Bachelor of Science degree in geology from the University of Manchester in 1967. In addition, I have obtained a PhD in geochemistry from the University of Manchester in 1970. I am a Fellow of the Institution of Mining and Metallurgy, a Fellow of the Australasian Institute of Mining and Metallurgy and a Chartered Engineer. I have worked as a geologist for a total of 40 years since my graduation from university. I joined ACA Howe International Limited as a senior consultant geologist in 1980 and since 2007 have been Principal Geologist and Director. My work experience includes a background in mineral exploration, resource evaluation and valuation studies for precious metals, base metals, diamonds and industrial minerals projects in many parts of the world. In addition, I have completed numerous National Policy 2A, and NI 43-101 technical reports for gold and / or base metals projects located worldwide and have also prepared technical reports for companies listing on exchanges in Canada, UK, Australia and South Africa. I have worked on porphyry deposits in Argentina, Chile, Indonesia, Kazakhstan, Papua New Guinea and Slovakia. I have managed and/or co-authored several multidisciplinary due diligence studies and exploration projects and contributed to various scoping studies.
3. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purposes of NI 43-101.
4. I am co-author of the technical report titled “Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia ”, dated 19th December 2014 (the “Technical Report”) relating to the Ilovitza Property held in Macedonia by Euromax Resources Limited. I am responsible for co-authoring Sections 1, 2, 3, 4, 5, 6, 23, 24, 25, 26 and 27.
5. I have not visited the property.
6. I have not had prior involvement with the property that is the subject of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
8. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
9. 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 and I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

Dated this 19th December 2014.

David J. Patrick

CERTIFICATE OF QUALIFIED PERSON

DANIEL C. LEROUX, P. GEO
2001 rue du Clairon
St-Lazare, QC
J7T 0C2
Telephone: +1-416-388-6446
Email: dcleroux@acahowe.ca

I, Daniel C. Leroux, M.Sc., P. Geo. (ON, SASK), do hereby certify that:

1. I am a Vice President with the firm of A.C.A. Howe International Limited, Mining and Geological Consultants (“Howe”) located at 365 Bay St., Suite 501, Toronto, Ontario, Canada, M5H 2V1. This certificate applies to the technical report titled “Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia” for Euromax Resources dated December 19, 2014.” (the “Technical Report”) with an effective date of December 19, 2014 and a signing date of December 19, 2014.

2. I graduated with a Bachelor of Science, Geology degree from Laurentian University in 1993 and a Master of Science degree in Mineral Exploration in 2013 from Laurentian University and have practiced the profession of geoscience since my Bachelor of Science graduation. I have been employed with Howe since 1993; since 2007 as Vice President, from 2005 to 2007 as a Senior Consulting Geologist, from 1999 to 2004 as an associate consulting geologist and from 1993 to 1999 as Project Geologist. I have a total of 23 years experience in the mining industry including a background in international mineral exploration, evaluation and valuation studies for precious metals, base metals, diamonds and industrial minerals projects. Additional experience includes the completion of various National Policy 2A and NI 43-101 technical reports for gold and / or copper projects located worldwide.

3. I am a Professional Geoscientist (P. Geo.) registered with the Association of Professional Geoscientists of Saskatchewan (APEGS, No. 10475) and with the Association of Professional Geoscientists of Ontario (APGO, No. 742), a member of the CIMM and of the Society of Economic Geologists.

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

5. I have not completed a site visit to the Ilovitza Property.

6. I have jointly prepared reviewed and supervised the preparation of Sections 1.0, 2.0, 3.0, 4.0., 5.0, 6.0, 23.0, 24.0, 25.0, 26.0 and 27.0 of the technical report titled “Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia”, dated December 19th, 2014 (the “Technical Report”) relating to the Ilovitza Property held in Macedonia by Euromax Resources Limited.

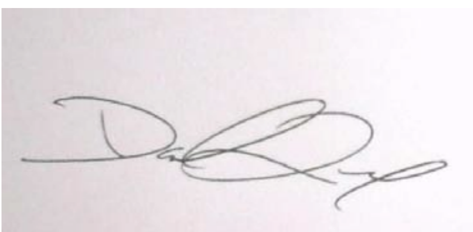
7. I am independent of issuer applying all of the tests in section 1.5 of National Instrument 43-101.

8. I have had no prior involvement with Euromax Resources Ilovitza Property that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.

9. As of the effective date of this report, 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.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: December 19, 2014
Signed Date: December 19, 2014



Daniel C. Leroux, M.Sc., P. Geo.

CERTIFICATE OF AUTHOR

David Carter
Phoenix Mining Consultants Limited
MBL House, 16 Edward Court,
Altrincham Business Park,
George Richards Way,
Altrincham, WA14 5GL,
Cheshire, United Kingdom
Email: carter-david@btconnect.com

I, David Carter, do hereby certify that:

1. I am a Geotechnical Engineer of 2 Mayor’s Walk Close, Pontefract, West Yorksire WH8 2SP, UK.

   I graduated with a BSc in Civil Engineering from the University of Leeds in 1965 and with a PhD in Soil Mechanics from the same university in 1969. I became a Member of the Institution of Civil Engineers (MICE) and a Chartered Engineer (C Eng) in 1972. I became a Fellow, Geological Society (FGS) 1973. I was a Fellow of the Institution of Mining Engineers until its merger and am now a Fellow of the Institute of Materials, Minerals and Mining.

   I have over 40 years postgraduate geotechnical experience specialising in waste and tailings disposal at mines and in the waste water industry. I have worked in over 20 countries and have had wide experience of tailings dam appraisal and design.

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

3. I have not completed a site visit to the Ilovitza property.

4. I am responsible for Section 18.8 of the technical report titled “Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia” dated 19th of December 2014 (the “Technical Report”) relating to the Ilovitza property held in Macedonia by Euromax Resources Ltd.

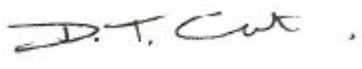
5. I have had no prior involvement with Euromax Resources Ilovitza Property that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been in compliance therewith.

6. As of the effective date of this report, 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.

7. I am independent of Euromax Resources Ltd. pursuant to section 1.5 of NI 43-101.

8. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the the public company files on their websites accessible by the public, of the Technical Report.

Dated this 19th day of December, 2014.



____

David Carter BSc, PhD, MICE, FIMMM, C Eng, FGS

CERTIFICATE OF AUTHOR

Gordon Antony Jackson
TJ Metallurgical Services Ltd
6 Coxburn Brae, Bridge of Allan, Stirling, United Kingdom, FK9 4PS
Email: tony@TJ-Met.co.uk

I, **Gordon Antony Jackson** do hereby certify that:

1. I am a Consultant Metallurgist at TJ Metallurgical Services Ltd of 6 Coxburn Brae, Bridge of Allan, Stirling, United Kingdom. My residential address is 6 Coxburn Brae, Bridge of Allan, Stirling, United Kingdom.

2. I graduated with a B.Sc. Hons. (Eng) from Royal School of Mines, Imperial College of Science, Technology & Medicine, London University in 1980 and I have practiced my profession for a total of 34 years since my graduation from university.

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

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

5. I have not completed a site visit to the Ilovitza property.

6. I am responsible for of Sections 19, 21 and 22 of the technical report titled "Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia" dated 19th of December 2014 (the "Technical Report") relating to the Ilovitza property held in Macedonia by Euromax Resources Ltd.

7. I have had no prior involvement with Euromax Resources Ilovitza Property that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been in compliance therewith

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

9. I am independent of Euromax Resources Ltd. pursuant to section 1.5 of NI 43-101.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the the public company files on their websites accessible by the public, of the Technical Report.

Dated this 19th day of December, 2014.

Tony Jackson

Gordon Antony Jackson

CERTIFICATE OF AUTHOR

Gareth Digges La Touche
Golder Associates
Cavandish House
Bourne End Business Park
Bourne End, Buckinghamshire SL8 5AS
T: 01628 851851
Email: gdiggeslatouche@golder.com

I, Gareth Digges La Touche, do hereby certify that:

1. I am a Chartered Geologist.

2. I graduated with a B.Sc. in Geology and Geography from the University of Keele in 1990, an M.Sc. in Computing in Earth Science from the Univeristy of Keele in 1991 and an M.Sc. in Hydrogeology, from the University of Birmingham in 1995. I have approximatly 20 years geoscience consulting experience of which the last 15 have been spent working in the extractive minerals sector.

3. I am a Fellow of the Geological Society of London and a Chartered Geologist (C.Geol.). I am a member of the European Federation of Geologists and accredited as a European Geologist (Eur.Geol.).

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

5. I have not completed a site visit to the Ilovitza property.

6. I am responsible for Section 20 of the technical report titled “Pre-Feasibility Study Technical Report for the Ilovitza Gold-Copper Project in Southeast Macedonia” dated 19th of December 2014 (the “Technical Report”) relating to the Ilovitza property held in Macedonia by Euromax Resources Ltd.

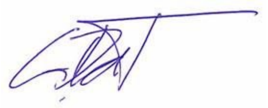
7. I have had no prior involvement with Euromax Resources Ilovitza Property that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been in compliance therewith.

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

9. I am independent of Euromax Resources Ltd. pursuant to section 1.5 of NI 43-101.

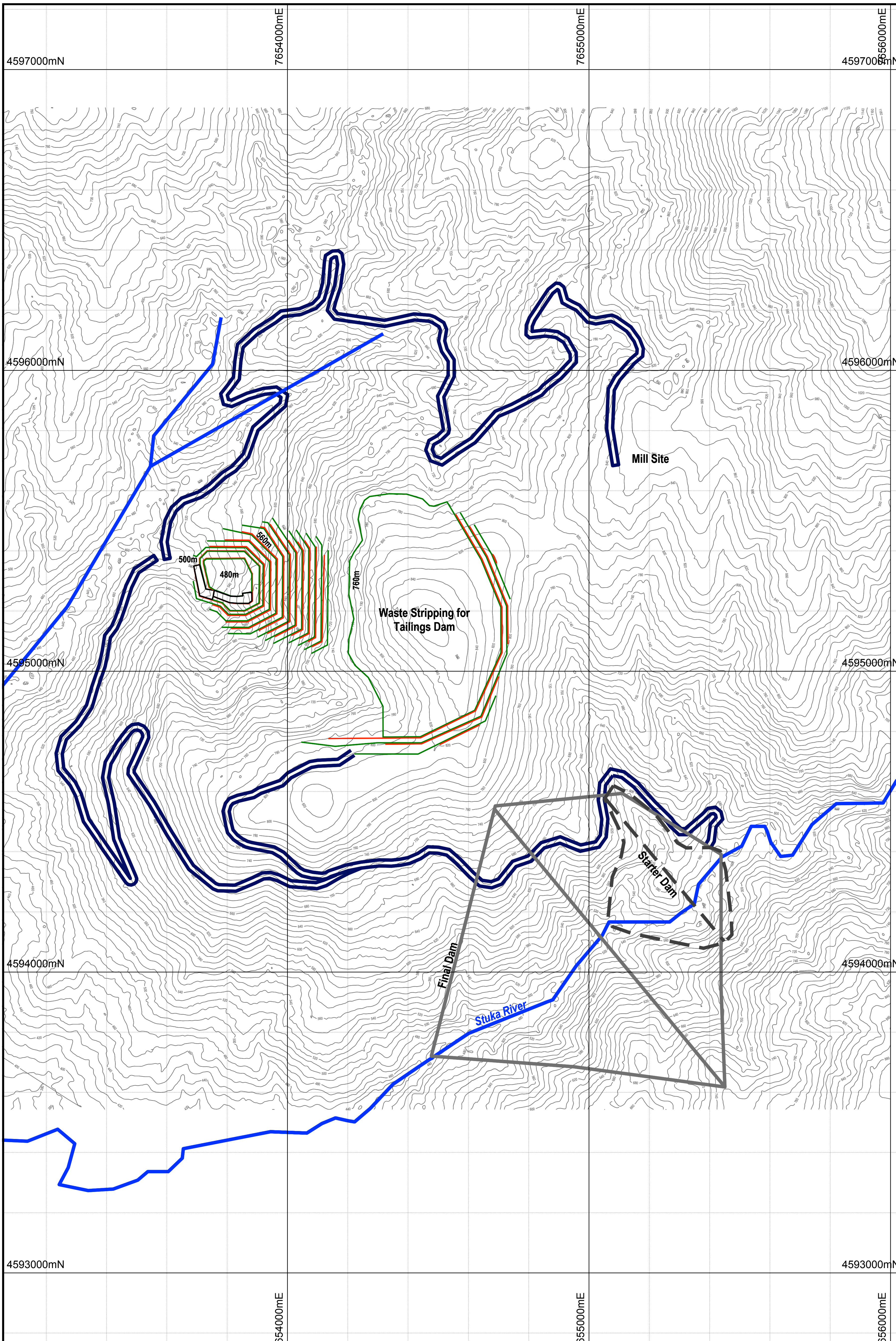
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the the public company files on their websites accessible by the public, of the Technical Report.

Dated this 19th day of December, 2014.



____
Gareth Digges La Touche, BSc, MSc, FGS, CGeol, EurGeol.

APPENDICIES



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Mining and Geological Consultants
UK - Toronto - Halifax
www.acahowe.co.uk

Drawn by:
Doug Roy, M.A.Sc., P.Eng.
Associate Mining Engineer

Coordinate System:
Hermannskogel Projection
HK_3_Degree_GK_Zone_7_9_BASE

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Scale
1 : 12500

Plot Date
24-May-2014

Sheet
1 of 4

Plot File: Phase 1

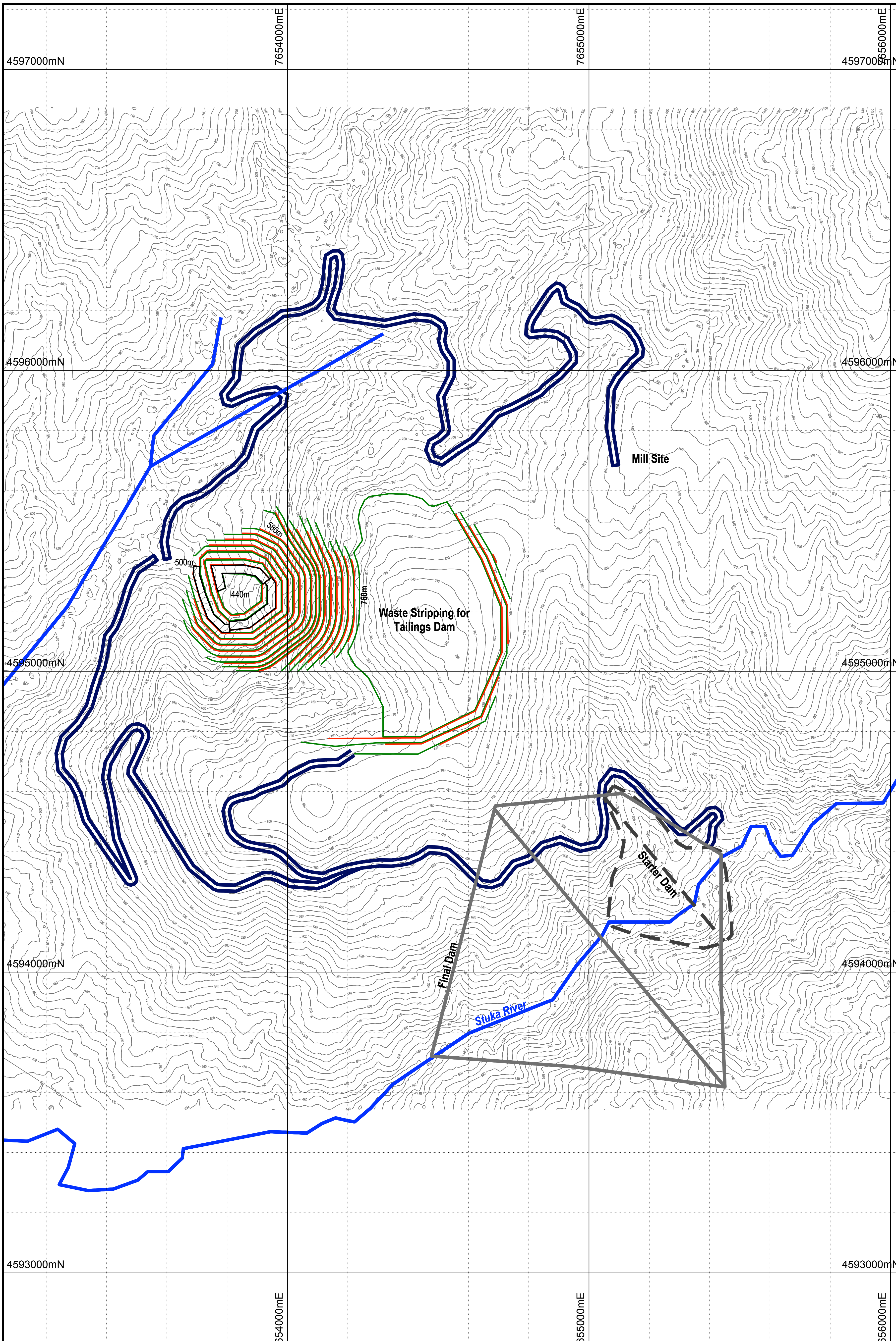
100 0 100m

Preliminary Pit Design

Phase 1

Euromax RESOURCES

Ilovitza Deposit
Macedonia



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Plot Date
24-May-2014

Sheet
2 of 4

Plot File: Phase 2

100 0 100m

Preliminary Pit Design

Phase 2

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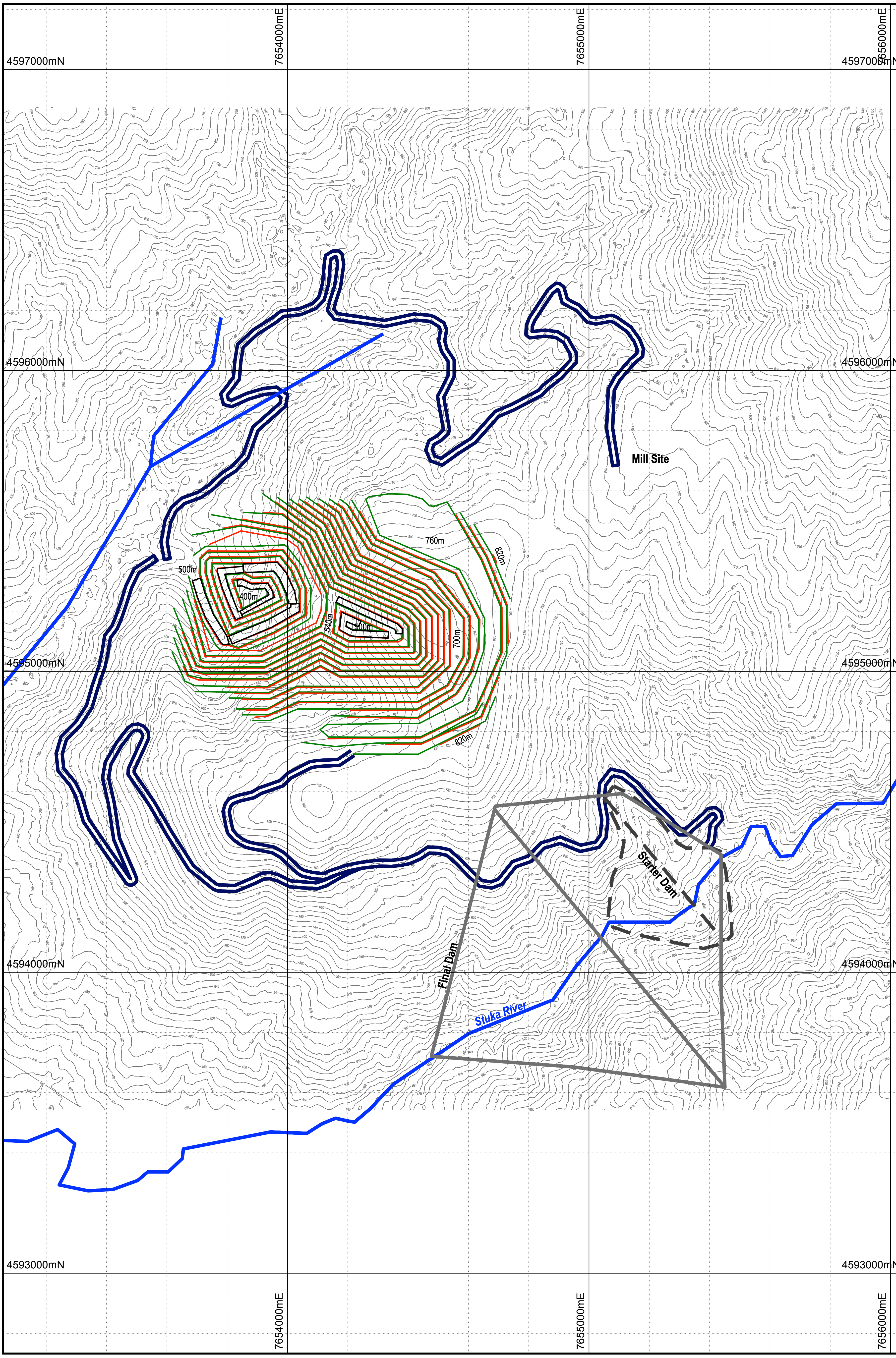
Macedonia



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Mining and Geological Consultants
UK - Toronto - Halifax
www.acahowe.co.uk

Drawn by:
Doug Roy, M.A.Sc., P.Eng.
Associate Mining Engineer

Coordinate System:
Hermannskogel Projection
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1 : 12500

Plot Date
24-May-2014

Sheet
3 of 4

Plot File: Phase 3

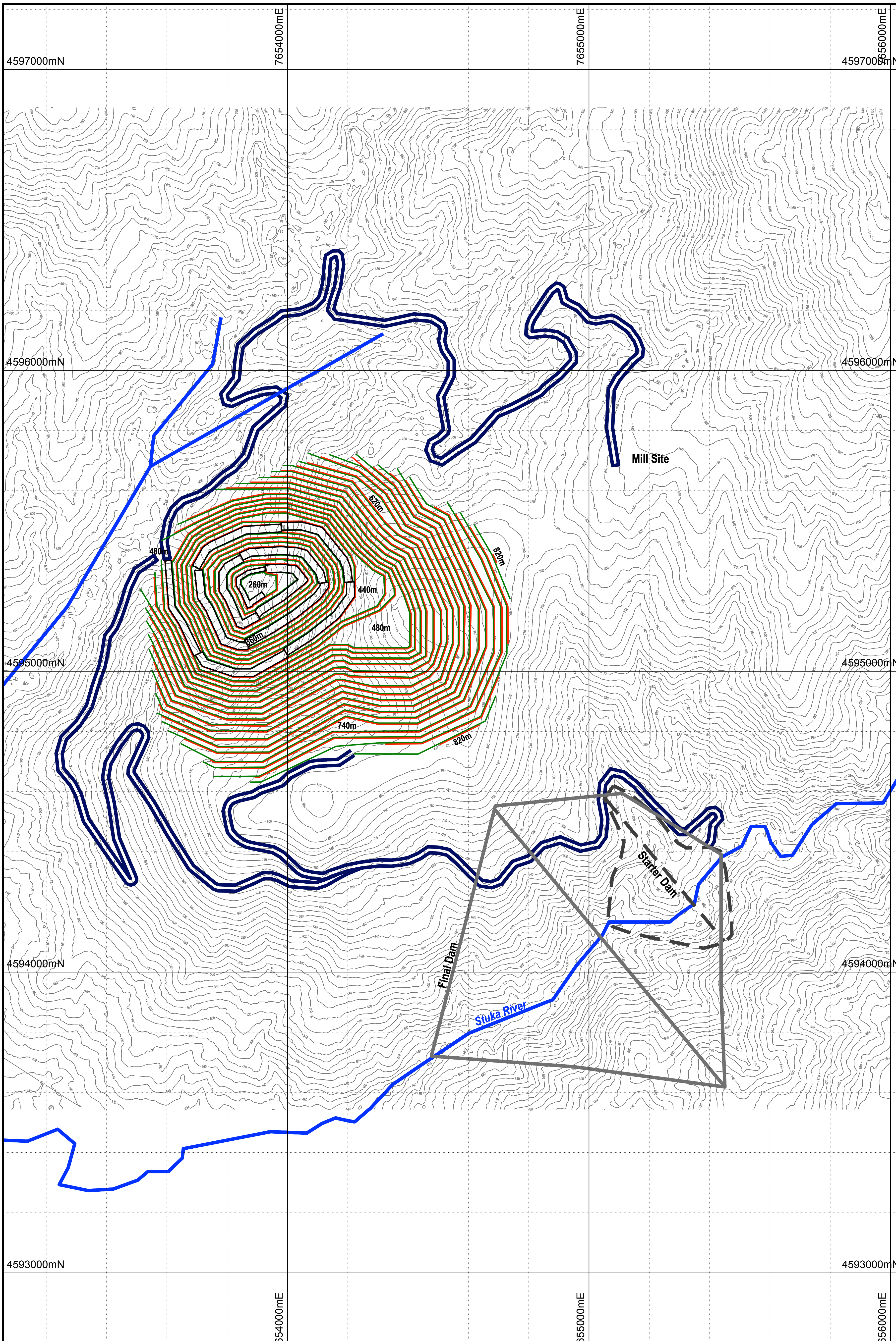
100 0 100m

Preliminary Pit Design

Phase 3

Euromax RESOURCES

Ilovitza Deposit
Macedonia



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Mining and Geological Consultants
UK - Toronto - Halifax
www.acahowe.co.uk

Drawn by:
Doug Roy, M.A.Sc., P.Eng.
Associate Mining Engineer

Coordinate System:
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HK_3_Degree_GK_Zone_7_9_BASE

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Plot Date
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Sheet
4 of 4

Plot File: Phase 4

100 0 100m

Preliminary Pit Design

Phase 4

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