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43-101

EFEMÇUKURU PROJECT Efemçukuru, Turkey

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WARDROP

Report to:

ELDORADO GOLD CORP.

Technical Report on the Efemçukuru Project

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TECHNICAL REPORT ON THE EFEMÇUKURU PROJECT

EFFECTIVE DATE AUGUST 2007

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GLOSSARY

UNITS OF MEASURE

Above mean sea level	amsl
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	Btu
Canadian Dollars	Cdn$
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km

Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Micron	µ
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million bank cubic metres per annum	Mbm^3/a
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM
Tonne (1,000 kg)	t

Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per cubic metre hour	ts/m^3h
United States Dollar	US$
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

ABBREVIATIONS AND ACRONYMS

Acid Rock Drainage	ARD
Adsorption - Desorption - Recovery	ADR
ALS Chemex Laboratories	Chemex
Ammonium Nitrate/Fuel Oil	ANFO
Atomic Absorption	AA
Bulk Densities	BD
Canadian Environmental and Metallurgical Incorporated	CEMI
Canadian International Minerals	CIM
Carbon-in-leach	CIL
Confromite Europeenne	CE
Cumulative Distribution Function	CDF
Eldorado Gold Corporation	Eldorado
Encon Environmental Consultancy Company	Encon
Engineering, Procurement, and Construction Management	EPCM
Environmental Impact Assessment	EIA
Environmental Management Plan	EMP
Free Carrier	FCA
Free On Board	FOB
Gemcom Geology, Mine Planning and Production Scheduling software	GEMS
Gemcom Integrated Geology, Resource Modelling, Mine Planning and Production software	SURPAC
Gemcom MineSched Surface and Underground Scheduling software	MineSched
General and Administration	G&A
Golder Associates Limited	Golder
Gravity Recoverable Gold	GRG
Gross Vehicle Weight	GVW
H.A. Simons Ltd.	Simons
High Pressure Sodium	HPS
Inductively Coupled Plasma Spectroscopy	ICP
In situ Densities	D
In-the-Hole	ITH
Internal Rate of Return	IRR
Inverse Distance	ID
Load-Haul-Dump unit	LHD
London Metal Exchange	LME
Longitudinal Longhole stoping	LLH
Mechanized Cut-and-Fill	MCF
Micon International Limited	Micon
Middle Ore Shoot	MOS
Ministry of Environment and Forestry	MoEF
Motor Control Centre	MCC
National Instrument 43-101	NI 43-101
Norwegian Geotechnical Institute	NGI
Norwest Corporation	Norwest

Nearest Neighbour	NN
Net Present Value	NPV
North Ore Shoot	NOS
Ordinary Kriging	KG
Polyvinyl Chloride	PVC
Potentially Acid Generating	PAG
Quality Assurance/Quality Control	QA/QC
Quantile-Quantile	QQ
Reverse Circulation	RC
Rock Mass Rating	RMR
Rotating Biological Contactor	RBC
Runoff Coefficient	Rc
Selective Mining Unit	SMU
South Ore Shoot	SOS
Specific Gravity	SG
Standard Reference Materials	SRMs
Three Dimensional	3D
Transverse Longhole stoping	TLH
Tüprag Metal Madencilik Sanayi Ve Ticaret Limited	Tüprag
Türkiye Elektrik Dağıtım A.Ş.	Tedaş
Turkish Air Pollution Control Regulations	APCR
Turkish Water Pollution Control	WPC
Uniaxial Compressive Strength	UCS
Ventsim Mine Ventilation Simulation Software	Ventsim
Wardrop Engineering Incorporated	Wardrop
Work Breakdown Structure	WBS
X-ray Fluorescence	XRF

SECTION 1 • SUMMARY

1.1 INTRODUCTION

Eldorado Gold Corporation (Eldorado) commissioned a study of their 100% owned Efemçukuru Project located in western Turkey. This Technical Report has been prepared by Wardrop Engineering Inc. (Wardrop), in accordance with standards set out in National Instrument 43-101 (NI 43-101) and based on information contained in the draft Efemçukuru Feasibility Report.

The report defines an operation based on underground mining and milling of the ore on site at Efemçukuru with post treatment of a gold concentrate at Eldorado's Kişladağ gold mine in Turkey. The mine will operate at a production rate of 1,100 tonnes per day, producing an average of 112,400 ounces of gold annually at a cash cost of $226/ounce.

Table 1.1 Operating Highlights of Feasibility Study Mineral Resources

Classification	Tonnes	Grade g/t Au	Ounces
Measured	1,150,000	14.07	520,000
Indicated	2,732,000	9.99	877,000
Measured and Indicated	3,882,000	11.20	1,397,000
Inferred	753,000	8.79	213,000

1. Mineral resources at the Efemçukuru Project are reported at a 3.0 g/t Au cut-off grade.
2. The contained gold represents estimated contained metal in the ground and has not been adjusted for the metallurgical recoveries of gold.
3. Resource classification conforms to CIM Standards on Mineral Resources and Mineral Reserves referred to in National Instrument 43-101. Mineral Resources that are not Reserves do not have demonstrated economic viability. Measured and Indicated Mineral Resources are that part of a Mineral Resource for which quantity, grade can be estimated with a level of confidence sufficient to allow the application of technical and economic parameters to support mine planning and evaluation of the economic viability of the deposit. An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified.

Table 1.2 Mineral Reserves

Classification	Tonnes	Grade g/t Au	Ounces
Proven	1,320,000	11.89	505,000
Probable	2,465,000	9.04	716,000
Proven and Probable	3,785,000	10.04	1,221,000

1. Cut-off grade 4.5 g/t Au
2. Reserve classification conforms to CIM Standards on Mineral Resources and Mineral Reserves referred to in National Instrument 43-101.
3. Mineral reserves estimated using a gold price of $530 per ounce are included in the mineral resource estimate.

Table 1.3 Project Performance

Project Data	Feasibility Results
Production Data	
Life of Mine	9.4 Years
Mine Throughput	3.785 M Tonnes
Metallurgical Recovery Au	86.5%
Average Annual Gold Production	112,400 Ounces
Total Gold Produced	1,056,566 Ounces
Operating Costs	
Mining	$27.20/Tonne
Processing	$30.95/Tonne
G&A	$4.98/Tonne
Total Operating Cost/Tonne Ore	$63.14/Tonne
Cash Operating Costs	$226.23/Ounce
Total Cash Costs	$227.20/Ounce
Capital Cost	
Initial Investment Capital	$104,204,000
Working Capital	$6,001,000
Sustaining Capital	$21,308,000
Economics @ $530 Au	
Net Present Value (NPV) After Tax @ 0%	$155.5 M
Net Present Value After Tax @ 5%	$86.7 M
Internal Rate of Return (IRR) After Tax	19.0%

1.2 PROJECT OVERVIEW

1.2.1 DEPOSIT

The Efemçukuru deposit is a high grade epithermal hosted vein structure located in the Menderes Massive of western Turkey. Eldorado has completed approximately 38,600 metres of diamond core and reverse circulation drilling at the Efemçukuru Project to May 30, 2007, including the 2006/2007 drill campaign. The known Kestani Beleni vein structure extends over 1,200 metres on surface. The deposit as defined in the report comprises two ore shoots, Middle Ore Shoot (MOS) and South Ore Shoot (SOS), with an average dip angle of approximately 60°. The vertical extent of the currently defined resource from surface is approximately 350 metres.

Exploration drilling will continue through 2007 to define the downdip extension of the deposit and explore other structures to the north of the MOS. Assay data from recent and ongoing drilling will be used to update the current resource model by the end of 2007.

1.2.2 RESOURCE ESTIMATION

The report incorporated the updated mineral resources of the SOS and MOS deposits along the Kestani Beleni Vein system. The previous mineral resource estimates for these deposits were completed on work done prior to the year 2000, and described in an updated NI 43-101 technical report by Micon International Ltd. (Micon) dated January 2006. The SOS and MOS mineral resources in that work were based on data from 84 diamond drill core holes totalling 10,438 m. An additional deposit described in the Micon report, the North Ore Shoot (NOS), remains at an early exploration stage and was outside the scope of the report. The mineral resources supporting the report utilized drill data as of a cut-off date of 30 May, 2007. As of that date, data from 89 additional holes supplemented the existing Efemçukuru database: 52 new core holes totalling 12,082 m and 37 reverse circulation (RC) holes totalling 3072 m.

1.2.3 RESERVE ESTIMATION

A proven and probable reserve estimate has been prepared by Wardrop using the resource model provided by Eldorado as a basis for the mine design. Stope designs and production schedule were prepared by Wardrop in Gemcom Integrated Geology, Resource Modelling, Mine Planning and Production software (SURPAC) and reconciled for tonnes and grade in commercial software (Gemcom). An overall cut off grade of 4.5 grams per tonne based on a gold price of $530/ounce has been used for all mining methods. Overall dilution from all mining methods is estimated at approximately 11%. Mining recovery of ore is estimated at 92% including mining losses due to pillars and ore in narrow vein structures.

1.2.4 MINING

Conventional trackless equipment will be employed to extract ore from mechanized cut-and-fill (MCF) as well as longitudinal longhole (LLH) and transverse longhole (TLH) stopes. Mining stope widths in the deposit vary from 2 metres up to 29 metres. Consolidated paste backfill will be placed in the stopes to act as a working floor for mining.

Access to the underground operation will be via two opposing adits intersecting the workings at mid elevation. A twin internal ramp system located in the footwall will deliver ore to an underground crusher station. Crushed ore is then conveyed to surface storage bins by an inclined conveyor system.

A mine contractor will carry out preproduction development of the haulage levels and ramps in preparation for mining. Underground mine production of 1,283 tonnes per day is based on a 6 day week.

1.2.5 METALLURGY

A review of the available metallurgical test data by Wardrop has confirmed that gravity concentration followed by flotation will achieve acceptable recovery of gold in the first stage. This will be followed by direct cyanidation of the flotation concentrate after regrinding to provide an overall gold recovery from ore of approximately 86.5%.

The concentrator will operate at 1,100 tonnes per day using a semi autogenous primary grinding mill followed by a ball mill for secondary grinding. Concentration of the gold after gravity treatment will be achieved through a flash cell in conjunction with rougher and cleaner cells. The flotation circuit design will be further optimized with a pilot scale test program prior to completion of the detailed engineering.

Flotation concentrate will be transported by road for treatment at the Kişladağ mine facility using a regrind mill and carbon-in-leach (CIL) circuit for final recovery of gold. Residue will be transferred to the Kişladağ leach pad. Tailings from the Efemçukuru concentrator will be processed through a filtration plant to generate dry stack tailings for surface disposal.

1.2.6 INFRASTRUCTURE

The Efemçukuru Project is located approximately 45 kilometres by road south of the city of Izmir at an elevation of approximately 700 metres. Access to the site is via all weather tarred roads. Power will be provided to the site via a dedicated transmission line from the Urla substation approximately 20 kilometres distance. Mine infrastructure will include administration buildings, the concentrator, filtration plant, tailings and waste rock impound areas. Concentrate treatment will be done at a dedicated facility located at the Kişladağ mine site.

1.2.7 OPERATING COSTS

Life of mine operating costs, based on annual production of 402,000 tonnes of ore, are estimated at $63.14 per tonne of ore mined, including production royalties. This cost equates to $226.23 per ounce of gold produced, which includes mining, general and administrative, process costs at Efemçukuru, transportation of concentrate to Kişladağ, and process costs at Kişladağ. The total processing costs are broken down as follows:

Table 1.4 Process Cost Breakdown

Category	Cost per Tonne Ore	Cost per Ounce Produced
Processing Efemçukuru	$21.21	$75.99
Bagging and Transportation	$3.24	$11.61
Processing Kişladağ	$6.50	$23.30

1.2.8 CAPITAL COSTS

Capital costs for the mine infrastructure, on site concentrator, and off site concentrate treatment have been developed using construction data from the recently completed Kişladağ gold mine in neighbouring Uşak province. Preproduction development costs are based on the use of a Turkish mining contractor.

Table 1.5 Investment Capital Cost Estimate

Area	Feasibility Results
Efemçukuru	
Overall Site	$10,383,000
Mining	$16,334,000
Process	$15,305,000
Tailings Disposal	$5,061,000
Ancillary Buildings and Services	$8,387,000
Total Direct Costs Efemçukuru	**$55,470,000**
Kişladağ	
Process	$8,478,000
Total Direct Costs Kişladağ	$8,478,000
Total Project Indirects	$24,282,000
Owners Costs	$4,020,000
Contingency	$11,954,000
Total Project Capital Costs	**$104,204,000**

1. Mining costs include preproduction development of 3,400 metres.

1.2.9 FINANCIAL ANALYSIS

Wardrop has completed a financial analysis of the Efemçukuru Project using a discounted cashflow model incorporating tax and royalty schedules as employed at the Kişladağ mine. Gold price has been fixed at a 3 year average of $530/ounce. No allowance has been made for inflation or escalation. Currency exchange rates for the Turkish Lira, US Dollar, and Canadian Dollar are fixed at a 180 day average, July 6, 2007.

Table 1.6 Efemçukuru Project Financial Analysis Summary

Project Data	Estimated Value
Life of Mine	9.4 Years
Total Gold Produced	1.056 Moz
Total Ore Mined	3.785 M Tonnes
Initial Project Capital Cost	$104.2 M
Cash Operating Cost	$226.23/oz
Total Cash Cost	$227.20/oz
Base Case Gold Price	$530/oz
After Tax Net Present Value @ 0%	$155.5 M
After Tax Net Present Value @ 5%	$86.7 M
After Tax Internal Rate of Return	19%

Table 1.7 Financial Sensitivity Analysis – NPV Value at 5%

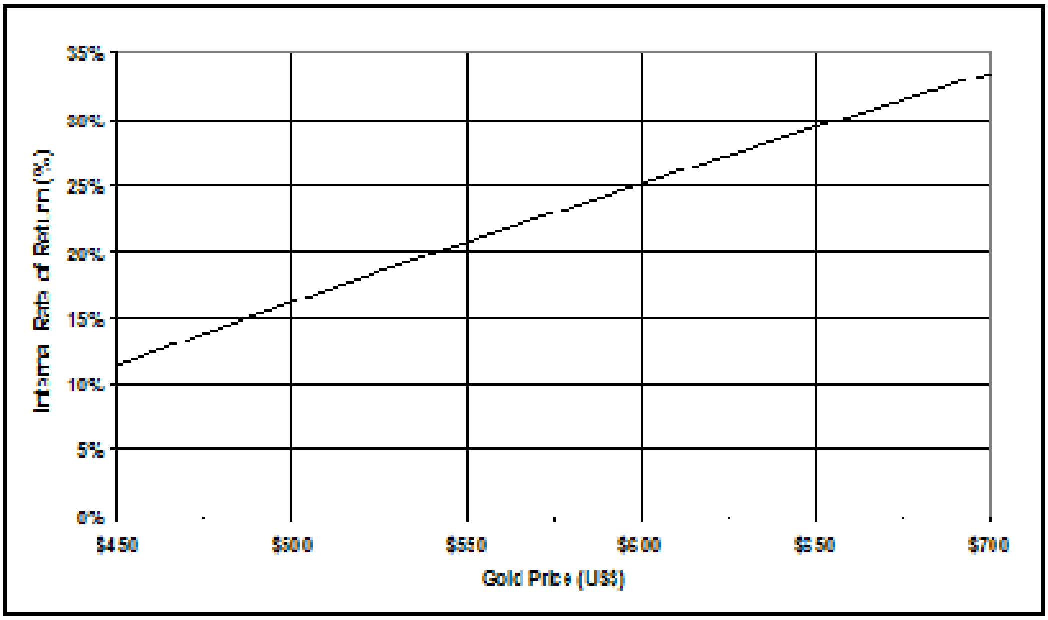
	-20%	-10%	0%	+10%	+20%
Au Price	21.5	54.1	86.7	119.2	151.8
Op Cost	113.2	99.9	86.7	73.4	60.1
Initial Capex	103.1	94.9	86.7	78.5	70.2

(million US dollars)

Table 1.8 IRR Value

	-20%	-10%	0%	+10%	+20%
Au Price	8.8%	14.0%	19.0%	23.7%	28.2%
Op Cost	22.9%	21.0%	19.0%	17.0%	14.9%
Initial Capex	24.4%	21.5%	19.0%	16.8%	15.0%

Figure 1.1 IRR Sensitivity – Gold Price (Post-tax)



1.2.10 SCHEDULE

Eldorado continues to complete the remaining land acquisition and permitting requirements with the objective to commence construction activities by December 31, 2007. Eldorado is proceeding to finalize orders for long lead time orders in the 3rd Quarter 2007. An eighteen month construction schedule is envisaged for the project with initial production anticipated in the 3rd Quarter 2009.

1.3 CONCLUSIONS

The Efemçukuru Project is feasible from an economical, technical, and practical aspect as described by the parameters set in this report.

SECTION 2 • INTRODUCTION

Eldorado is proposing to develop their Efemçukuru Project, a greenfield site located in western Turkey 20 km southwest of Izmir.

Wardrop was retained to prepare a feasibility study to evaluate the project economics of a 1,100 t/d underground mine and processing facility; and a concentrate processing plant at Eldorado's existing Kişladağ mine site located 180 km west in west central Turkey.

Information and data for this report were obtained from Efemçukuru Project Draft Feasibility Study Report (September 2007) and data provided by Eldorado.

The work represents all aspects of the project design, including mining, waste handling and storage, tailings handling and storage, power and electrical distribution, milling and processing, concentrate transportation and processing, in compliance with the permitted plans and design for the Efemçukuru Project development. The feasibility study work was completed to a +15%-5% level of accuracy and to a design level suitable for submission to financing institutions.

Andy Nichols, P.Eng., an employee of Wardrop, served as the Qualified Person responsible for preparing this technical report as defined in NI 43-101 Standards of Disclosure for Mineral Projects and in compliance with 43-101F1. Mr. Nichols also conducted and supervised the review of matters pertaining to the mineral reserves. This includes equipment requirements and operating cost developments. He has not visited the project site.

Stephen Juras, Ph.D, P.Geo., an employee of Eldorado, provided Qualified Person assistance by directing the review of the geological data and mineral resource estimation work. Dr. Juras was responsible for the preparation of the sections in this report that concern geological information and matters pertaining to the mineral resource. He most recently visited the project site on April 20 to 22, 2007.

Mr. Rick Alexander, P.Eng., a Senior Mechanical Consulting Engineer, served as the Qualified Person for the infrastructure design of the project and related sections in this technical report. He visited the property on September 15 to 17, 2006.

Mr. Andre de Ruijter, P.Eng., a Wardrop employee, served as the Qualified Person who supervised and reviewed matters pertaining to process design and metallurgical testwork, and was responsible for the preparation of sections concerning process design and metallurgy in this report. He has not visited the project site.

The term “ore” is used for convenience throughout this report to denote that portion of the Measured and Indicated mineral resources that have been converted to Proven and Probable mineral reserves.

SECTION 3 • RELIANCE ON OTHER EXPERTS

Information contained in this Technical Report has been provided by additional consultants, as listed below. It is assumed that information for this Technical Report including estimates, concepts, designs, and conclusions supplied by other consultants have been prepared by Qualified Persons.

NORWEST CORPORATION

- filtered tailings storage facility design
- mine development rock dump facility design
- site water balance.

GOLDER ASSOCIATES LTD.

- site hydrology
- site hydrogeology
- water management design

CANADIAN ENVIRONMENTAL AND METALLURGICAL INC. (CEMI)

- water treatment plant design

THE MINES GROUP INC.

- rehabilitation and closure design.

ENCON ENVIRONMENTAL CONSULTANCY CO. (ENCON)

- Environmental Impact Assessment (EIA) report.

ELDORADO GOLD CORP.

- matters relating to taxation in the economic modelling of the report
- information on permitting and status of permitting
- information regarding location and property title.

SECTION 4 • PROPERTY DESCRIPTION AND LOCATION

4.1 INTRODUCTION

The Efemçukuru Project is a greenfield site consisting of a proposed underground mine, and surface facilities consisting of the process plant and ancillary buildings with additional offsite facilities for treatment of gold concentrate.

4.2 LOCATION AND DESCRIPTION

The Efemçukuru Project area is licensed as a 2261.49 ha greenfield site near the west coast of Turkey, located approximately 20 km from the provincial capital city of Izmir on the Aegean coast, in a mountainous area known as Tepe Daği (Figure 4.1). The fenced area enclosing the mine and process site is 70 ha, of which approximately 22 ha will be disturbed land.

Tüprag Metal Madencilik Sanayi Ve Ticaret Limited (Tüprag), a wholly owned subsidiary of Eldorado, holds the mineral rights to the Efemçukuru Project. The mine and process facilities will be located within the licence area. Both private landholders and the State as Forest Lease Land hold surface rights in the area.

The project co-ordinates are:

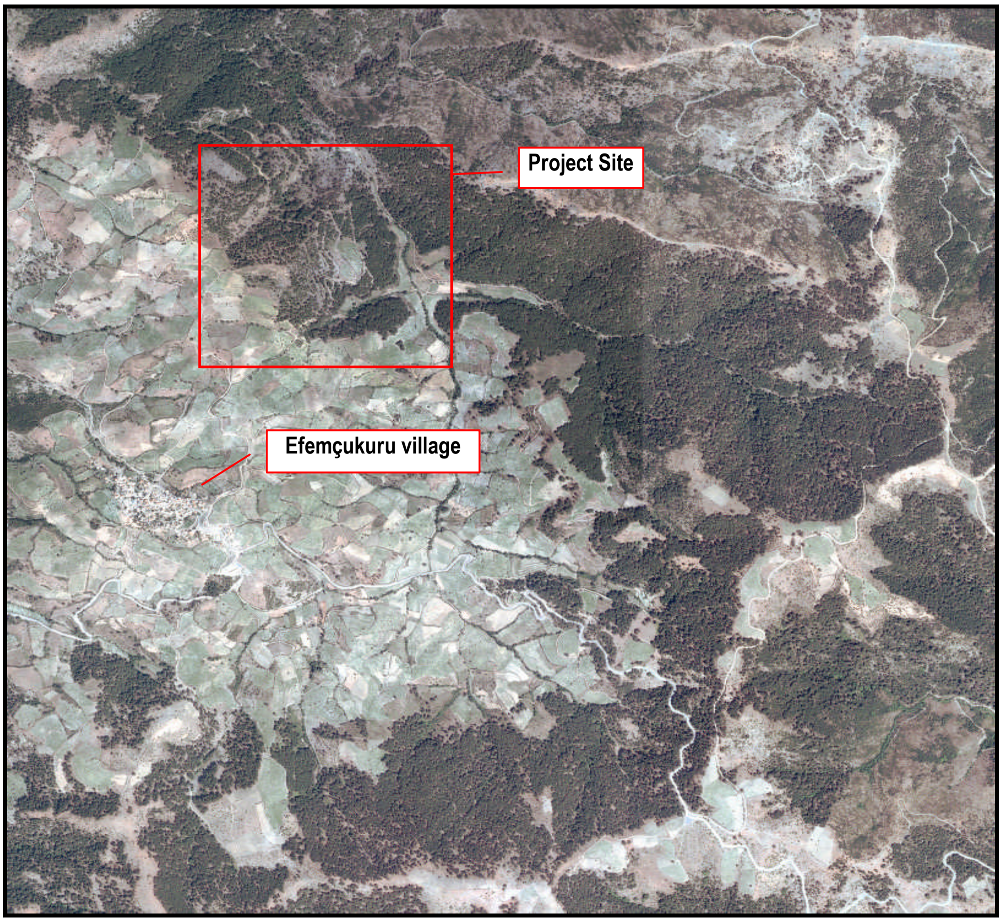
- UTM: 04 97524E 42 38507N
- UTM Zone: 37
- Longitude: 38° 17' 30"
- Latitude: 26° 58' 15"
- Map Sheet: Urla-L

Small rural villages populate the area south of the mine site and the inhabitants rely primarily on viticulture for livelihood. The village of Efemçukuru, with a population of approximately 500 people, lies 2 km southwest of the project site.

Figure 4.2 shows the Efemçukuru Project area in relation to Efemçukuru village. The exploration roads on the project site can clearly be seen in the upper left hand corner of the photograph.

Figure 4.1 Location Map

Figure 4.2 Efemçukuru Project Area and Village



4.3 SURFACE AND SUB-SURFACE CONDITIONS

Steep hills and narrow valleys characterize the project site with the elevation on site ranging from 580 masl in the valley to 770 masl in the surrounding hills. The deposit outcrops the Kestane Beleni and Lena Hills, which slope steeply to the Kokarpinar Creek Valley.

Vegetation consists of mature pine trees with sparse undergrowth covering the hillsides. The flatter land in the valleys and upper slopes of the hills has been cultivated with grape vines.

Photo 4.1 and

Photo 4.2 show the general topography and vegetation in the plant site area.

Photo 4.1 View Looking North at Plant Site

Photo 4.2 View Looking South at Plant Site

A detailed geotechnical investigation was completed to determine the site subsurface conditions in order to support detailed engineering. Site investigations indicated a thin layer of scree overlying the site, typically less than one meter underlain by 2 to 3 m of weathered bedrock, then unweathered hornfels or phyllite bedrock. The subsurface conditions indicate the site will be suitable for shallow economical spread footings.

4.4 Royalties

A royalty set at 1% of mine production costs is payable to the Turkish government on an annualized basis. No other royalties apply to the property.

4.5 Environmental Liabilities

Wardrop is not aware, nor has been made aware, of any significant environmental liabilities associated with the Efemçukuru property.

4.6 Permitting

Development of a mining project in Turkey must follow the permitting regulations set out for all industrial development. The list of key permits and status of the Efemçukuru Project are shown below in Table 4.1.

Table 4.1 Permit Status – Efemçukuru Project

Permit	Status
Environmental Positive Certificate	Received
Blasting and Explosives Permit	To be applied for prior to startup
Trial Operating Permit	To be applied for prior to startup
Opening Permit	To be applied for after startup and inspection
Work Place Labour Permit	To be applied for after startup and inspection
Air Emission and Discharge Permit	To be applied for after startup and inspection

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

5.1 ACCESS AND INFRASTRUCTURE

Site access is via approximately 40 km of regional paved and gravel roads from the city of Izmir to the north. Access is gained from the Seferihisar coast road to the west and north or the Izmir-Menderes highway to the east. A two km unimproved public 'forestry' road currently provides access to the mine property from the regional roads. The travel time by road from Izmir is between 45 and 60 minutes.

The only infrastructure on the site is a power line originating from the village of Efemçukuru that supplied power to a well used to supply water to the town and for irrigation. The pumps and the power line are currently decommissioned. The power line has limited capacity and is not useable for the permanent operations, it can be commissioned for limited supply during construction.

The proximity of the site to Izmir, one of the largest industrial centers in Turkey and the second largest port is advantageous to the project for logistics and material supplies as well as a source of qualified personnel for construction and operations.

5.2 CLIMATE

The planned mine site is situated within the Aegean climatic zone, which is characterized by hot and dry summers and warm and rainy winters. Temperatures in the region range between 30ºC in summer and 0ºC in winter with an annual average of approximately 17ºC. The study area is susceptible to orographic effects caused by the lifting of moisture-laden air from the Aegean Sea. Accordingly, the study area experiences a significant amount of rainfall variation on a monthly basis.

Long-term climatic records are not available for the Koçadere River Catchment. Although a meteorological station does exist at the Efemçukuru mine site, it has only been in operation since 1998. The next closest meteorological station located at Beyler, approximately seven km to the southwest, has a longer period of record, however the lack of data overlap prevents correlation with the meteorological data recorded at the mine site. The next closest stations with overlapping periods of record are Balçova, roughly 14 km to the northeast; Seferihisar, approximately 15 km to the southwest; and Izmir, approximately 20 km to the northeast.

Owing to the relatively long period of record, and strong correlation with data collected at the mine site, the precipitation record from the Izmir meteorological station (1969-2004) has been used in the evaluation of long-term precipitation conditions at the Efemçukuru mine site (Golder, 2005). Table 5.1 below presents the expected seasonal variation of monthly climate data for the study area. The expected annual average precipitation is 740 mm, while precipitation extremes for wet and dry years (1:100 year return period) are respectively 1,255 mm and 383 mm (Golder, 2005).

Table 5.1 Distribution of Annual Climate Data

Month	Temp	Precipitation		Run-Off (mm)			Potential Evapotranspiration		Lake Evapotranspiration	
	°C	mm	% of Annual	Rc* = 0.40	Rc = 0.55	Rc = 0.80	mm	% of Annual	mm	% of Annual
January	6.5	128.0	17.3%	51.2	70.4	102.4	0.0	0.0%	39.2	3.3%
February	6.2	103.6	14.0%	41.4	57.0	82.9	2.4	0.3%	47.6	4.0%
March	9.9	83.6	11.3%	33.4	46.0	66.9	41.0	4.3%	72.8	6.1%
April	12.6	47.4	6.4%	19.0	26.1	37.9	82.1	8.7%	82.6	
May	18.1	27.4	3.7%	11.0	15.1	21.9	131.0	13.9%	124.6	10.4%
June	22.1	9.6	1.3%	3.8	5.3	7.7	174.0	18.5%	165.9	13.8%
July	25.5	6.7	0.9%	2.7	3.7	5.4	204.9	21.7%	195.3	16.2%
August	25.0	6.7	0.9%	2.7	3.7	5.4	176.3	18.7%	176.4	14.7%
September	20.4	15.5	2.1%	6.2	8.5	12.4	98.3	10.4%	129.5	10.8%
October	17.0	48.8	6.6%	19.5	26.8	39.0	33.2	3.5%	81.9	6.8%
November	12.8	111.0	15.0%	44.4	61.1	88.8	0.0	0.0%	42.0	3.5%
December	6.3	151.7	20.5%	60.7	83.4	121.4	0.0	0.0%	42.0	3.5%
Totals	**N/A**	**740**	**100%**	**296**	**407**	**592**	**943**	**100%**	**1203**	**100%**

*** Rc =** runoff coefficient

Notes: a Golder 2005. **b** Annual runoff coefficient applied to undeveloped areas. **c** Annual runoff coefficient applied to tailings and development rock storage piles. **d** Annual runoff coefficient applied to mill site areas, roads, and adit laydown areas.

Average annual precipitation is 750 mm due to the moderating influence of the Aegean Sea. There is limited snowfall. Average high wind velocity is 30 km/h with a maximum of 50 km/h.

5.3 PHYSIOGRAPHY

The Efemçukuru Project is located at the western end of the Izmir-Ankara Suture Zone, a major regional structure that extends northeast and then east from Izmir for almost 800 kilometres.

SECTION 6 • HISTORY

6.1 HISTORY

While carrying out reconnaissance work in western Turkey, Tüprag discovered the Efemçukuru Project in 1992. The area was noted on geological plans as the site of old mine workings. Surface evidence of these workings has been found in the form of shallow excavations in the main Kestane Beleni structure.

Between 1992 and 1996, Tüprag conducted extensive exploration work including a magnetic survey and mapping, soil, rock chip, and channel sampling and surface trenching, and in excess of 6,000 m of HQ drilling. The exploration work identified a high-grade vein-hosted gold system consisting of three separate ore zones along the Kestane Beleni structure known as the SOS, MOS, and NOS. A metallurgical testwork program was completed to support a conceptual study in 1994 by Tüprag, described a 1,000 t/d underground operation using CIL ore processing.

In 1997 and 1998, a 4,092 m HQ infill drilling program was undertaken along the SOS, MOS, and North Ore Shoot (NOS) to further delineate the initial identified resource. Additional diamond drilling was carried out for hydrogeological testing in the vein structure as well as the hanging wall and foot wall rocks. In 1998, Micon evaluated the geological model and confirmed a measured and indicated resource of 1.87 Mt at 14.26 g/t, with an inferred resource of 660,000 tonnes at 11.99 g/t Au.

Permitting for the project was initiated in 1998 and an Environmental Impact Assessment (EIA) study was completed in May 2004.

The Efemçukuru Project reached an advanced stage of development with the completion of a full prefeasibility study in 1999 that describes an 800 t/d underground mine supported by a gold flotation recovery plant producing both gravity concentrate and flotation concentrate. The gravity concentrate was to be smelted on site and the flotation concentrate was to be shipped out of the country from the port of Izmir for smelting.

Limited work completed after the 1999 prefeasibility study as Eldorado was focusing on the development of the Kisladağ Project, also located in western Turkey the Kisladağ Project was commissioned in the 4th quarter of 2005.

Eldorado has recently resumed exploration work on the property and are currently advancing the engineering and permitting requirements for construction of the project. Photo 6.1 shows the current drilling program in progress.

Photo 6.1 Current Drilling Program

Drilling on the Efemçukuru property has been carried out in several phases. The first drill program was started in September 1992 (KV-001 through KV-015) and continued in November 1992 (KV-016 through KV-026) after a one month break to await and evaluate assay results. The second phase of drilling was carried out in May/June 1993 (KV-027 through KV-43). The third phase of drilling was carried out from August to October 1996 (KV-44 through KV-056) after a 3-year hiatus during which Eldorado acquired control of Tüprag from Gencor. The fourth and fifth phases of drilling occurred between March and December 1997 (KV-57 through KV-108).

Infill and exploratory drilling commenced again in August 2006 and has continued throughout 2007. The following table summarizes the drilling that has been completed on the property.

Table 6.1 Summary of Drilling on the Efemçukuru Deposit

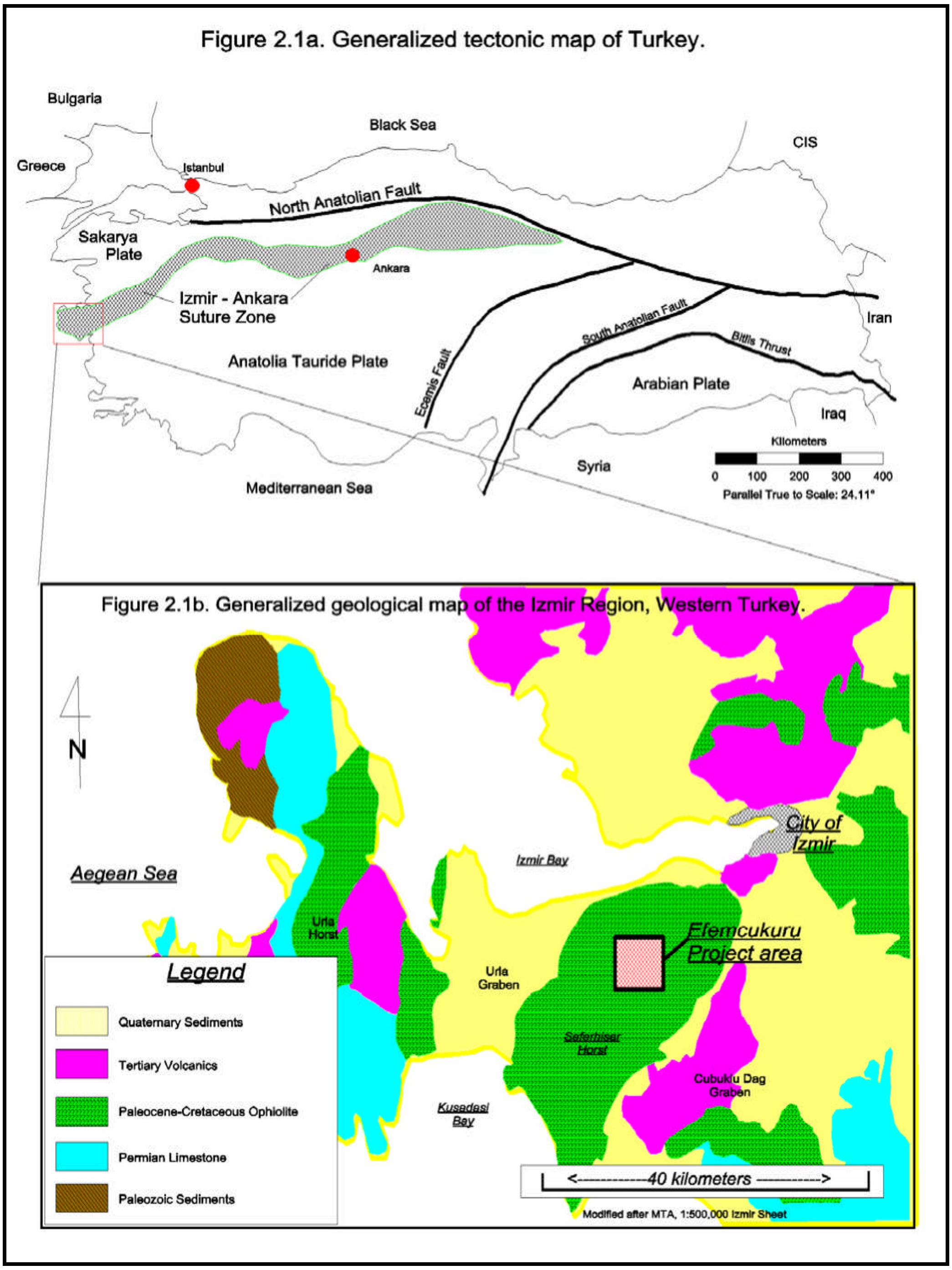
Vein	Type of Drilling	Year	# of Holes	Metres
Kestane Beleni	Core	1993, 96, 97, 2006,07	186	30108
Mezarlik Tepe	Core	1993 & 96	2	103
Kokarpinar	Core	1993	4	465
Subtotal			**192**	**30,676**
Kokarpinar	Percussion	1997	8	393
Kestane Beleni	RC	2006, 07	51	4,631
Total			**251**	**35,700**

SECTION 7 • GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Efemçukuru project is located at the western end of the Izmir-Ankara Suture Zone, a major regional structure that extends northeast and then east from Izmir for almost 800 km (Figure 7.1). The Izmir-Ankara Suture Zone marks the closure point of a subduction zone that separated the Sakarya and Anatolide-Tauride microplates plates during the late Cretaceous and early Paleocene Age. As the subduction zone closed, Neo-Tethyian sea floor between the two microplates was obducted onto the Anatolide-Tauride plate. Lenses of serpentine often associated with thrust faults and large olistoliths of recrystallized limestone were caught up in the melange-like complex that formed during the suturing process. Regionally extensive volcanism and intrusive activity were also associated with the subduction process. Subsequent mid-Tertiary dilation in western Turkey resulted in block faulting and the formation of the north-south orientated Seferihisar horst. The Efemçukuru project is situated in the central part of the Seferihisar horst (Figure 7.1). Younger Neogene sediments and volcanics fill the flanking graben structures.

Figure 7.1 Regional Geology



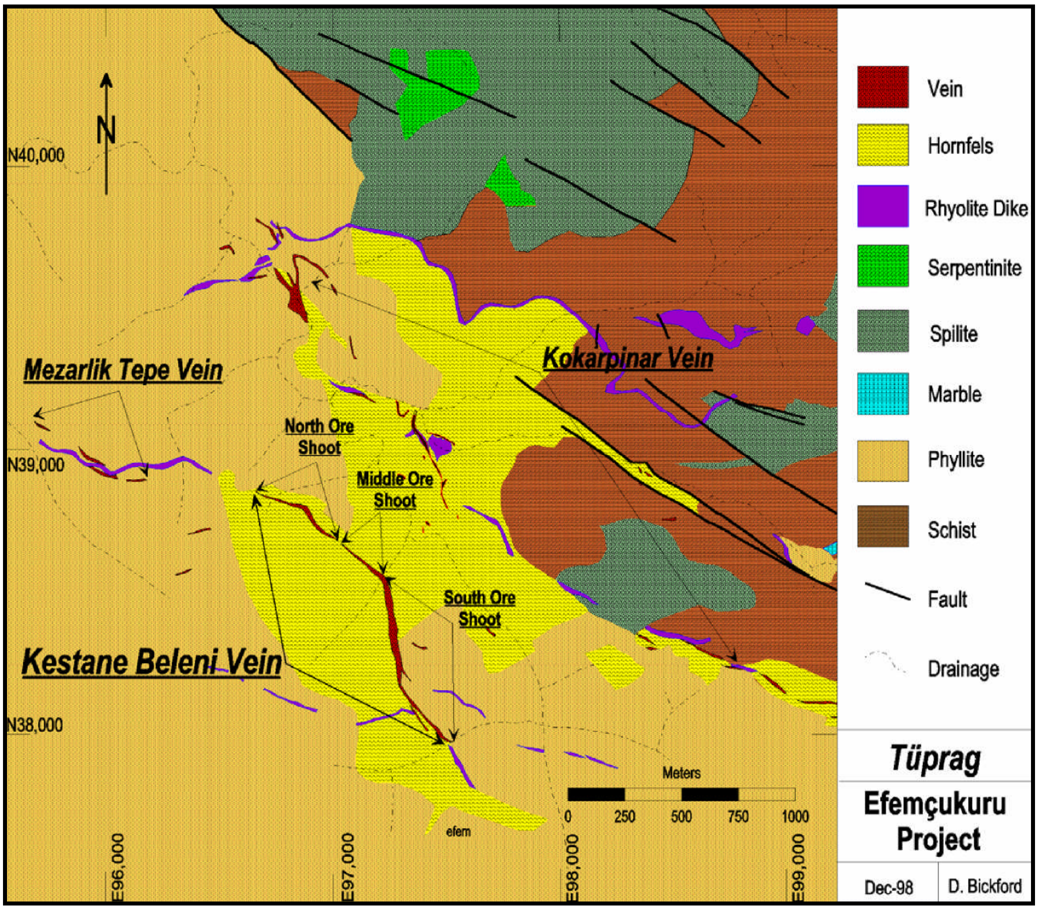
7.2 LOCAL GEOLOGY

The immediate project area is comprised of a late Cretaceous to Paleocene-age volcano-sedimentary sequence, which has been regionally metamorphosed to greenschist facies (Figure 7.2). Intermediate to mafic submarine volcanics and interbedded mafic sediments (schist) in the northeast corner of the project area grade southward and westward into phyllites. Granitic intrusive reportedly outcrops in a restricted military radar station located approximately 3.5 km north of the deposit area (outside the area of the map). The granite is probably subduction related.

Narrow rhyolitic dykes cut the immediate host rock. These are unmetamorphosed and largely undeformed, and therefore post-date the regional metamorphic collision-related event. Age is reported as a Late Miocene age (11.9 Ma, K-Ar) for rhyolitic rocks (dikes) in the region, and they are thought to be related to the post-collisional extensional magmatism. They clearly pre-date the gold mineralization event because they are cross-cut by the auriferous veins, and also appear to pre-date hornfelsing of the metasediments because they are also cross-cut by early quartz-calcite-sphalerite-galena-pyrite veinlets. The rhyolite dikes are thought to be the surface expression of a deeper intrusive body, which is not exposed in the vicinity of the deposit.

Phyllites and hornfels are the primary host rock for mineralization on the property. Where unaffected by hydrothermal alteration, the phyllites are typically soft, fissile, and have a well-developed S1 foliation. Fractures in the phyllite are locally filled with thin metamorphic quartz-microcline veinlets. The phyllites were strongly deformed during regional tectonic events. Foliation strike and dip directions change quickly over short distances. Near the center of the deposit area, the phyllites have been thermally metamorphosed to hornfels over a 2 km x 2 km area. De-carbonation and silicification of the calcareous phyllites has generated an assemblage of epidote, tremolite and actinolite rich rocks with varying amounts of pyrite, pyrrhotite and base metals. The hornfelsing has embrittled the host rocks, rendering them more susceptible to fracturing and brecciation than the more ductile pelitic phyllites. Within the deposit, the highest gold grades and thickest vein intersections are commonly found within these hornfelsed rocks.

Figure 7.2 Local Geology of the Efemçukuru Project Area



7.3 Vein Descriptions

Gold and base metal mineralization in the Efemçukuru deposit is hosted in three north to northwest-trending epithermal veins, which crosscut hornfels, phyllites, and rhyolite dikes. The main vein, which is the focal point of this study, is the Kestane Beleni Vein. A second sub-parallel structure known as the Kokarpinar Vein outcrops approximately 450 m northeast of the Kestane Beleni Vein and the Mezarlik Tepe Vein is located approximately 500 m west of the Kestane Beleni Vein. The Mezarlik Tepe Vein has been treated as an extension of the Kestane Beleni Vein in previous studies, but is separated here for discussion purposes.

The Kestane Beleni vein is characterized by multi-stage breccias containing abundant wall rock and vein fragments, and to a lesser extent, layered vein textures. The vein was emplaced along an active fault system, and the abundant breccias are a result of fault induced hydro-fracturing within a dilational segment of the controlling fault. The vein has a sigmoidal form in plan view, the geometry of which supports oblique dextral+normal movement on the controlling fault system. Most vein intersections contain fault zones along one or both vein contacts, or internal to the vein itself. The position of these faults is difficult to predict, and they do not correlate easily between adjacent drill holes. They likely form an anastamosing network formed during post-mineral reactivation of the controlling fault.

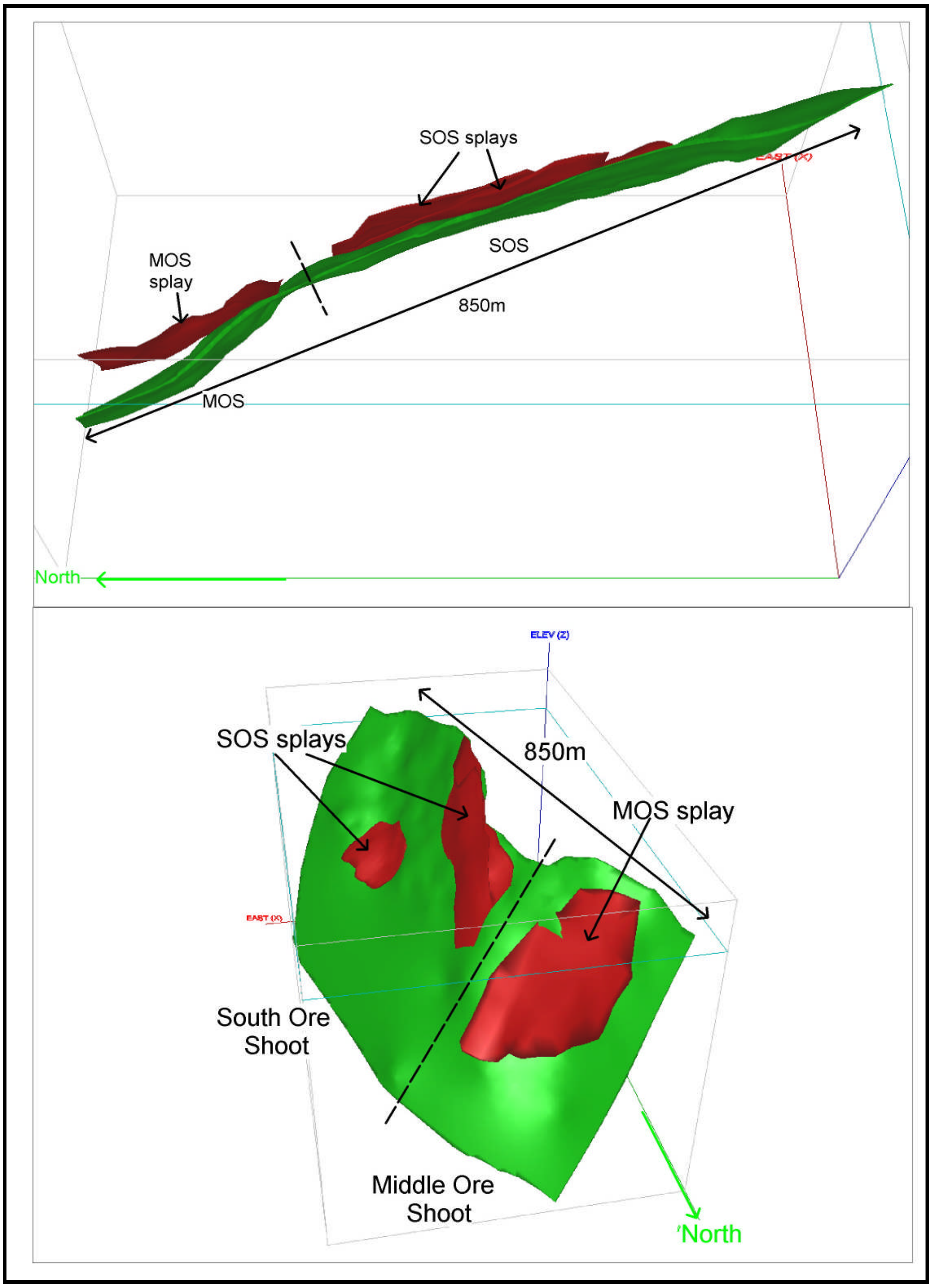
The Kestane Beleni Vein has an approximate strike length of 1,100 m. Three ore shoots have been identified along the vein; they are: (1) SOS, (2) MOS, and (3) NOS.

The SOS can be traced on surface along a bearing of 338° for 500 m and dips between 60° to 68° northeast. The southern half of the shoot consists mostly of a single vein. A small split develops at depth in the SOS in the central part of the ore shoot and a second larger split develops nearer the surface, close to mid-point of the SOS and continues approximately 230 m to the north end of the shoot, (Figure 7.3). Drilling indicates that the base of the second split rakes to the northeast at approximately -60°. The location of this split and its trend is important because it coincides with a thickening of high-grade gold mineralization that developed at the base of the split. Above the split, the vein breaks into two and sometimes three branches of varying thickness and grade. Generally the middle branch contains lower gold grades while the footwall and particularly the hanging wall branches contain higher gold grades. Additionally, significant stockwork type mineralization is locally present between the vein splays where they cut hornfels. Limited amounts of stockwork mineralization are also present where the vein hanging wall consists of phyllite, however phyllite hosted zones are more restricted in size and continuity.

Where the SOS consists of a single vein, its thickness generally ranges from 3 m to 5 m, however, locally it can reach more than 10 m in thickness. Gold mineralization is generally not distributed across the whole vein, but more typically occurs as discrete zones within the vein. Where the vein breaks into small splays, the splays are generally narrower, with thicknesses of 1 m to 2 m.

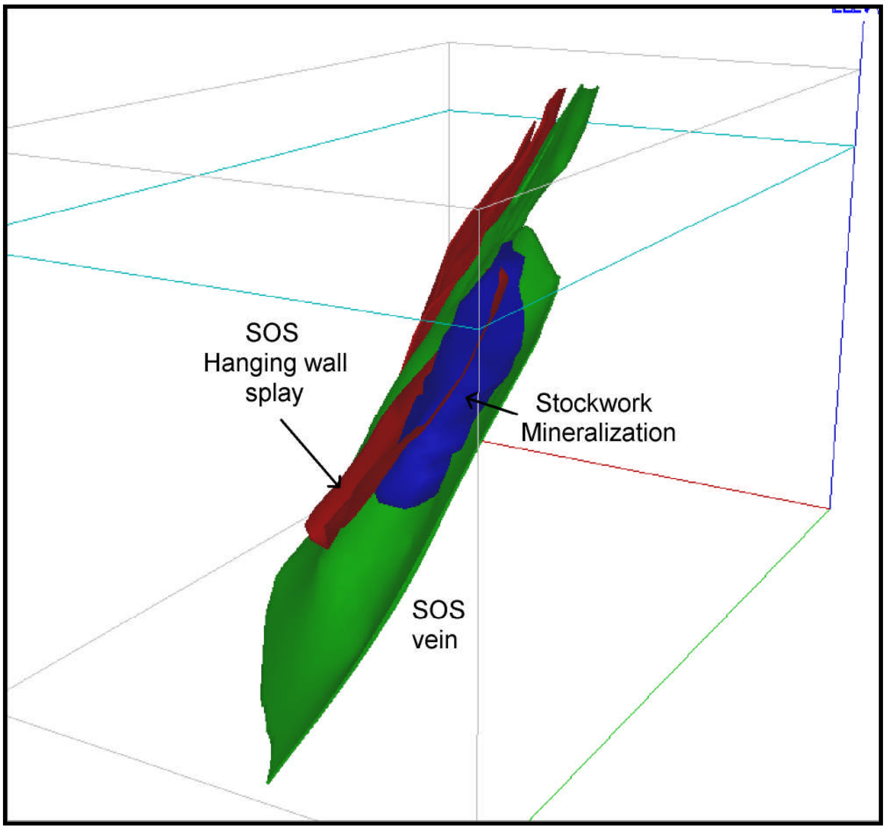
Infill drilling between the South and Middle Ore shoots has confirmed the continuity of the vein in the hinge zone and the shoots have been combined into one geological olid for modelling. Figure 7.3 provides a three dimensional view of the combined SOS and MOS shoots and also shows the splays that have been modelled for both shoots.

Figure 7.3 Views of the Combined SOS and MOS Shoots and Hanging Wall Splays



In the MOS area the Kestane Beleni Vein strikes 320° for approximately 230 m and has an average dip of 60° - 65° to the northeast. The vein is hosted completely in hornfels on both the footwall and hanging wall sides. High-grade gold mineralization in the MOS forms a steeply plunging shoot in the central part of the MOS. The shoot has a relatively narrow surface expression (±3 m at elevation 675 m), widens rapidly with depth to more than 20 m at elevation 525 masl. A single splay diverges from the main vein in the central part of the shoot and an extensive zone of stockwork mineralization is found between the two veins (Figure 7.4) and extends into the hanging wall above the splay. The stockwork zone is best developed between 550 m and 600 m elevation with some extensions above and below these elevations. The zone is traceable along strike for approximately 75 m.

Figure 7.4 View Along the Vein from the North



The NOS is separated from the MOS by a “gap” zone, which coincides with an additional 25° westward bend in the strike of the Kestane Beleni Vein to approximately 300° and a flattening of the vein dip to 45°. The NOS outcrops intermittently for approximately 200 m along this bearing before it pinches out. Based on limited drilling results, two narrow veins make up the NOS down to a depth of approximately 100 m. The veins merge at depth, and along strike to the north into a single structure. Where two veins are present, high-grade gold mineralization is localized in the upper vein.

SECTION 8 • DEPOSIT TYPES

The Efemçukuru deposit is a typical low sulphidation epithermal vein deposit developed in an active fault environment resulting in numerous fragments of the surrounding country rock being included in the quartz rhodinite vein. Vein textures are typical of an epithermal system where the gold was precipitated by boiling of the hydrothermal fluids and also by chemical reaction with the surrounding wall rocks.

SECTION 9 • MINERALIZATION

The Kestane Beleni Vein is a low-sulfidation epithermal vein. Early veining consisted of three or more episodes of sulphide poor quartz-rhodonite gangue, followed by later multiple phases of quartz and quartz-sulphide veining. Open-space textures are present in the vein along with complex breccia textures. Sulphides consist of pyrite, sphalerite, galena, and trace amounts of chalcopyrite. The base metal and overall sulphide content of the MOS is considerably higher.

Silver content of the deposit is highly variable. The average silver grade for both the MOS and SOS is low (11.9 and 8.3 g/t), however, parts of the MOS shoot contain silver values in excess of 100 g/t. The higher grade silver zones tend to be peripheral to the high grade gold zones.

The majority of the gold mineralization is very fine-grained (0.5 to 30 microns) occurring as free grains in quartz and rhodonite gangue, and as partially locked grains in pyrite, chalcopyrite and sphalerite. Gold is also, to a lesser extent, present in galena. Higher gold grades, however, are not directly related to sulphide percentages.

Oxidation is most common in near surface intercepts; however, isolated zones of oxidized vein material can be found at all depths explored by drilling and appears to be related to faulting. Locally, vein footwall and hanging wall rocks can be oxidized to iron and manganese gossan, while only a few metres away the host rocks are unoxidized.

Old surface workings, usually consisting of shallow pits and deeper selective cuts are found intermittently along the surface trace of each shoot. The largest surface workings are located above holes KV-1 in the SOS and KV-13 in the NOS. During drilling, voids that are interpreted as old workings were occasionally encountered at shallow depths. Most of these voids occur in soft oxidized vein material within the upper 30 to 40 metres of the vein structure. Occasionally, they are back filled with soft, black manganese rich mud containing wall-rock fragments. The voids probably terminate around the oxide-sulphide boundary.

9.1 VEIN PARAGENESIS AND ALTERATION

The vein paragenesis are summarized as follows:

- early quartz veining with minor sulphides (sphalerite, pyrite, galena) at margins

- anastomosing quartz-rhodonite stockwork veining, with marginal sphalerite, pyrite, and galena
- brecciation of hornfels host rocks
- quartz-rhodonite-rhodochrosite-axinite cavity fill and breccia cement, with coarse grained sphalerite, pyrite, and galena in symmetrically banded layers. Chalcopyrite also occurs in particularly high grade intersections. These veins and breccias repeat and cross-cut one another to form thick, high-grade intervals
- late unmineralized quartz (locally amethystine or chalcedonic) with minor calcite.

TerraSpec analyses had been conducted on drill core and chip samples returned from crushed rejects of drill core, reverse circulation holes and surficial trenches. Hornfels alteration in the metasediments is well developed but there is a lack of hydrothermal alteration in the host rocks to the vein system. This probably indicates a combination of two effects.

1. The fluids were not particularly reactive towards the already clay-chlorite-rich mineral assemblage in the argillites and hornfels, suggesting near-neutral pH and temperatures similar to that of the background greenschist facies and hornfels metamorphism. These observations are consistent with the 200°–300°C temperatures, moderate salinities, and sparsity of dissolved gases (e.g., CO_2) in fluid inclusions.

2. The highly channelled nature of fluid flow in the veins and breccias limited the physical degree of wall-rock interaction. This behaviour is in accordance with the hydraulically fractured and brecciated nature of the veins, which suggests that fluid flowed in pulses following build up of fluid pressure to the point of rupture of the fault system (not necessarily as high as lithostatic pressure, but higher than hydrostatic pressure).

SECTION 10 • EXPLORATION

10.1 SURFACE AND SUBSURFACE EXPLORATION WORK

Tüprag began work on the Efemçukuru project in 1992. Since then, the following work has been completed: district geological mapping at 1:5000 scale (10 km^2) and detailed prospect mapping at 1:500 scale (2 km^2), reconnaissance stream sediment sampling (147 samples), rock chip sampling (650 samples), soil sampling (891 samples), trenching (1820 m with 867 samples in 20 trenches), and drilling (approximately 12,000 m in 112 drill holes).

10.1.1 MAPPING

District scale geological mapping on 1:5000 scale topographic maps enlarged from the 1:25,000 scale maps has been carried out over approximately 10 km^2 surrounding the prospect. Detailed geological mapping on 1:500 scale topographic maps, prepared by a contract surveyor, has been carried out over the entire strike length of the Kestane Beleni Vein. Vein contacts with wall rock were established through a combination of outcrops, trenches and road cuts.

10.1.2 SURFACE SAMPLING AND TRENCHING

The Kestane Beleni Vein has been sampled extensively along its strike with rock chip sampling and trenches. Systematic sampling across the vein was not possible because of locally thick overburden and occasional old workings back-filled with rubble. Because of the patchy nature of the sampling, assays obtained from the surface sampling have been used to help project the strike and dip of the vein but have not been used in resource calculations.

SECTION 11 • DRILLING

Drilling on the Efemçukuru property has been carried out in several phases. The first drill program was started in September 1992 (KV-001 through KV-015) and continued in November 1992 (KV-016 through KV-026) after a one month break to await and evaluate assay results. The second phase of drilling was carried out in May/June 1993 (KV-027 through KV-43). The third phase of drilling was carried out from August to October 1996 (KV-44 through KV-056) after a 3-year hiatus during which Eldorado acquired control of Tüprag from Gencor. The fourth and fifth phase of drilling occurred between March and December 1997 (KV-57 through KV-108).

Infill and exploratory drilling commenced again in August 2006 and has continued throughout 2007. The following table summarizes the drilling that has been completed on the property. A list of the project drill holes used in the Efemçukuru mineral resource estimate, together with the coordinates and lengths, is provided in Appendix A, along with a drill hole location plan map.

Table 11.1 Summary of Drilling on the Efemçukuru Deposit

Vein	Type of Drilling	Year	# of Holes	Metres
Kestane Beleni	Core	1993, 96, 97, 2006,07	186	30,108
Mezarlik Tepe	Core	1993 & 96	2	103
Kokarpinar	Core	1993	4	465
Subtotal			**192**	**30,676**
Kokarpinar	Percussion	1997	8	393
Kestane Beleni	RC	2006, 07	51	4,631
Total			**251**	**35,700**

Core drilling for the 1993 through 1997 programs on the Kestane Beleni and Mezarlik Tepe veins was completed using a skid mounted Longyear 38 drill operated by Kennebec Drilling of Canada. The same core drill was used on the Kokarpinar Vein together with a Stenuick percussion drill owned and operated by Tüprag.

The core drilling program that started in August 2006 was carried out using the Longyear 38, and an IDC D-120 and CS-14 rig. RC drilling was completed with Tüprag's Explorer rig and an IDC Mustang rig.

Drilling has been carried out along the Kestane Beleni Vein on profiles spaced from 20 m to 40 m apart. The closest profile spacing is on the Middle Ore Shoot, followed by the South and North Ore Shoots. The down dip spacing along profiles ranges from 20 m to over 50 m. The deeper exploratory holes were drilled to intersect the vein over 100 m below previous vein intercepts.

Most core holes, in the early drilling programs, with the exception of KV-94, were drilled approximately perpendicular to the vein at dips ranging from 45° to 85°. Hole KV-94, in the MOS, was drilled almost down the dip of the vein because of access problems. The inclination and direction of drilling for the 2006 and 2007 program was variable due to the limited number of collar location available from which to drill.

Down hole deviation for holes drilled in 2007, 2008 was measured using a Reflex EZ Shot instrument with readings taken at 25 m down the hole.

Standard logging and sampling conventions were used to capture information from the drill core. The core was logged in detail onto paper logging sheets, and the data were then entered into the project database. The core was photographed before being sampled.

Core recovery in the mineralized units was very good, averaging 97% for over 92% of core intervals in the mineralized zones. The quite small number of poorer recovery intervals should have negligible impact on the Efemçukuru mineral resource estimate.

SECTION 12 • SAMPLING METHOD AND APPROACH

Samples collected during core drilling used either a 5-ft or a 10-ft single tube HQ core barrel. Some deeper drill holes required a reduction to NQ rods to complete the drill hole. Core material was removed from the core barrel and placed into wooden boxes with a capacity of 4 m per box. The end of each core run was marked with a wooden block showing the depth of the hole at the bottom of the run. Geological logs were prepared for the complete hole and geotechnical logging was done over selected intervals (± 20 m from mineralized zones). Sample intervals from 0.1 m to 1.6 m were selected by the geologist and marked in the core boxes. Individual samples were then cut using a diamond rock saw. One half of the split core was reduced through a two stage crushing and pulverizing circuit. After initial crushing, the sample was split to approximately 1 kg in a Jones type splitter and then pulverized. After pulverizing, the sample was split again into two 200 g pulps. One 200 g pulp was shipped to the analytical laboratory and the second 200 g pulp, together with the approximately 1 kg of pulp reject, was put into storage.

Holes KV-1 through KV-83 and KV-96 through KV-108 were prepared at Tüprag's sample preparation laboratory in Çanakkale, Turkey. Holes KV-84 to KV-95 were prepared at the SGS laboratory in Izmir. Pulp rejects from samples prepared at SGS were returned to Tüprag for storage.

Core cutting for the 2006 to 2007 drilling program was carried out initially at the project site and then at the company's core logging and storage facility in Gaziemeer, an industrial area close to the Izmir airport. The half cores were shipped to Tüprag's sample preparation facility in Çanakkale and pulps prepared as per the previous drilling campaigns.

The RC drill holes were sampled at 1 m intervals outside the ore zone and at 0.5 m intervals for vein and stockwork intervals. The samples were split at the drill site and a 1.0 to 1.5 kg sample was sent to Tüprag's sample preparation facility in Çanakkale. There the pulps were prepared in the same manner as for the core samples.

Significant composited assays (by intersected ore shell thickness) for the Efemçukuru Project are shown in Appendix B. Only values equal to or above 3.0 g/t gold grade were tabulated.

SECTION 13 • SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 ASSAY METHOD

Primary assaying up to 1997 was completed at SGS laboratories in Canada and France and check assays were done at Chemex and Bondar-Clegg laboratories in Vancouver, Canada. Holes KV-1 through KV-26 were fire-assayed at SGS-Xral in Toronto, Canada and holes KV-27 through KV-108 were fire assayed at the SGS laboratory in Carcassonne, France. The initial fire-assay was done on a 1 assay-ton charge with an atomic absorption (AA) finish. Over-range samples (>10 ppm Au) were re-assayed with a gravimetric finish.

Besides gold, multi-element analyses, including silver were completed on approximately 75% of the samples from drill holes KV-01 to KV-43, and on 35% of the samples from drill holes KV-44 to KV-95.

Sample pulps from the 2006-2007 drilling program were sent from the Çanakkale sample preparation facility to ALS Chemex Laboratories (Chemex) sample preparation facility in Izmir and were then shipped under the supervision of Chemex to their analytical laboratory in North Vancouver. All samples were assayed for gold by 30 g fire assay with an AA finish and for multi-element determination using fusion digest and inductively coupled plasma spectroscopy (ICP) analysis.

Samples that returned assays greater than 5 g/t were re-assayed by fire assay with a gravimetric finish. During the latest program, all samples greater than 5 g/t and less than 10 g/t Au from the pre-109 holes were re-assayed also. All geological and assay data for the project is stored in a database program developed by Maxwell Geoservices.

13.2 QUALITY ASSURANCE AND QUALITY CONTROL (QA/QC) PROGRAM

Assay results are provided to Eldorado in electronic format and as paper certificates. Upon receipt of assay results, values for Standard Reference Materials (SRMs) and field blanks are tabulated and compared to the established SRM pass-fail criteria:

- automatic batch failure if the SRM result is greater than the round-robin limit of three standard deviations
- automatic batch failure if two consecutive SRM results are greater than two standard deviations on the same side of the mean.
- automatic batch failure if the field blank result is over 0.5 g/t Au.

If a batch fails, it is re-assayed until it passes. Override allowances are made for barren batches. Batch pass/failure data are tabulated on an ongoing basis, and charts of individual reference material values with respect to round-robin tolerance limits are maintained.

Laboratory check assays are conducted at the rate of one per batch of 20 samples, using the same QA/QC criteria as routine assays.

13.2.1 Pre-2006/2007 QA/QC

The QA/QC for the initial 108 drill holes has been described in an earlier Technical Report (Estimation of Resources, Kestani Beleni Structure, WT/ Efemçukuru Project, Turkey, February 1999 updated October 2004). Summarizing, the QA only consisted of duplicate samples, internal laboratory pulp duplicates, and pulp duplicate samples sent to a second laboratory. No discussion dealt with SRM or blank QA samples. Results were deemed acceptable based on the performance of the duplicate samples.

Eldorado submitted over 50% of these older samples, taken from intervals that fell within the interpreted ore shell, for re-analysis with the current QA/QC protocol. Results of gold values generally agreed with the earlier values, except in the 5 to 10 g/t range which experienced a few percent upgrading in grades. This, however, was primarily due to a change in analytical procedures (imposing a gravimetric finish on re-assays of samples greater than 5 g/t with the initial AA finish method) and not deficient QA.

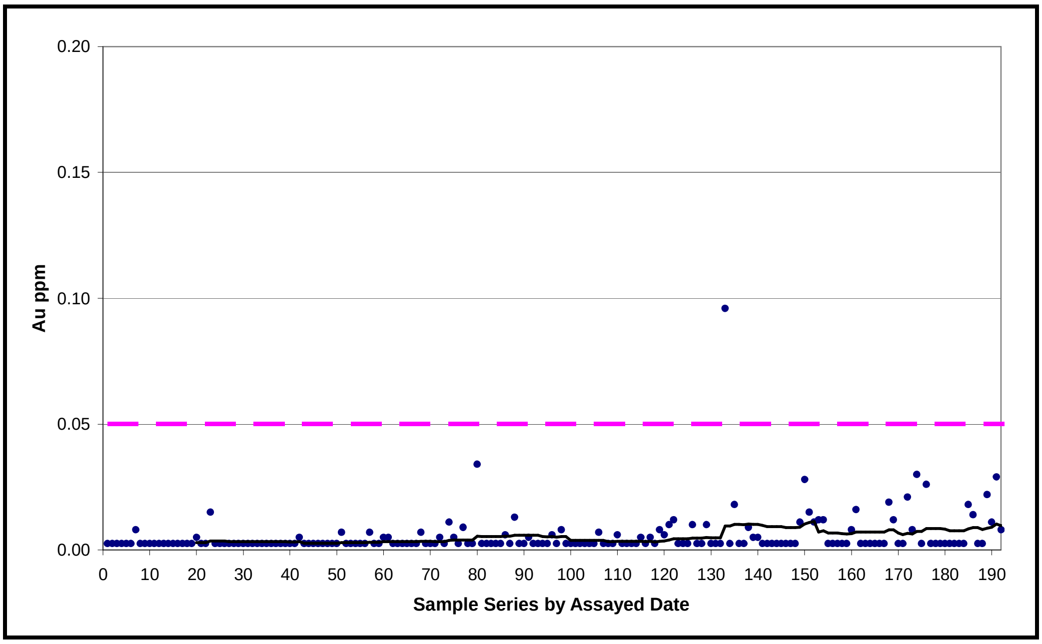
13.2.2 Standards Performance

Eldorado strictly monitors the performance of the SRM samples as the assay results arrive at site. Six SRM samples are used, covering a grade range between 0.5 g/t to 35 g/t. Charts of the individual SRMs are included in Appendix B. All samples are given a “fail” flag as a default entry in the project database. Each sample is re-assigned a date-based “pass” flag when assays have passed acceptance criteria. At the data cut-off date of 30 May 2007, only a very small number of assayed samples still had the “fail” flag. The relative uncertainty introduced to the mineral resource estimate by using this very small number of temporarily failed samples is considered negligible.

13.2.3 Blank Sample Performance

Assay performance of field blanks is presented in Figure 13.1 for gold. The analytical detection limit for gold is 0.005 g/t. The rejection threshold was chosen to equal 0.05 g/t (dashed horizontal line). The results show a very low incidence of contamination, essentially none close to grades considered for mining cut-off purposes. The few cases of sample mix-ups were investigated and corrected.

Figure 13.1 Efemçukuru Blank Data – 2006/2007 Drill Program



13.2.4 DUPLICATES PERFORMANCE

Eldorado implemented and monitored two types of duplicate data: regularly submitted coarse reject duplicates and an approximate 20% re-submission of samples from mineralized intervals to a second laboratory (Assayers Canada Laboratory, Vancouver, Canada). The latter was particularly important to help verify the very high gold grades occasionally analyzed.

The duplicate data are shown as relative difference charts in Figure 13.2 and Figure 13.3. Patterns are symmetric about zero, suggesting no bias in the assay process. The coarse reject chart shows almost all data greater than 1 g/t falling well within the 20% limits. Of note on the two-lab comparative chart is the excellent replication of samples with values greater than 30 g/t.

Additionally, a Quantile-Quantile (QQ) plot comparing data between the two laboratories was generated (Figure 13.4) to check for any bias in the analysis, particularly at grades greater than 30 g/t. The generated trend shows no indication of any bias.

Figure 13.2 Relative Difference Chart – Coarse Reject Data

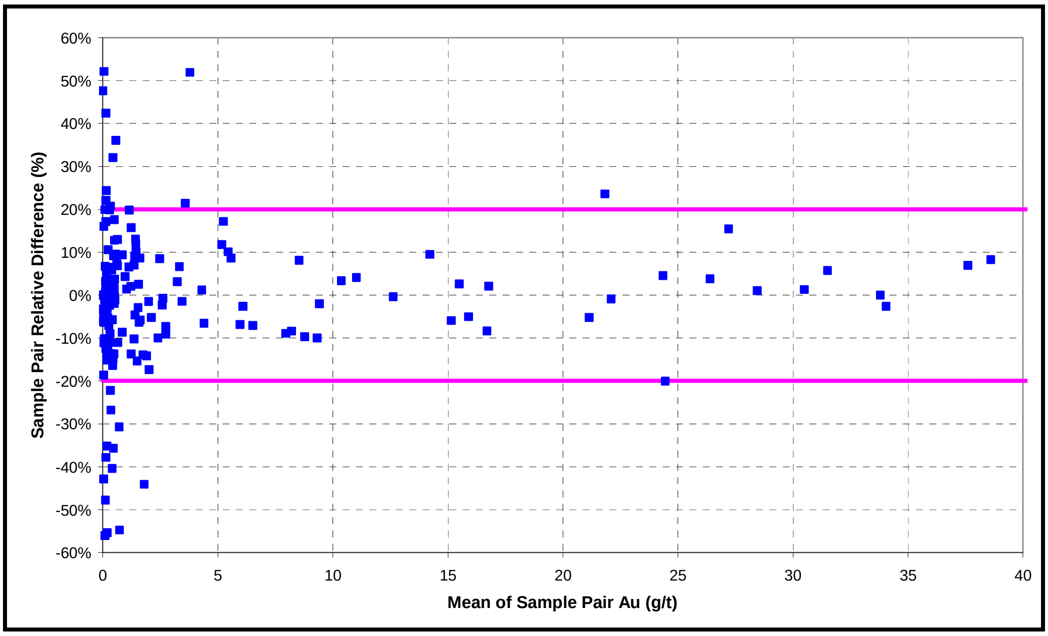


Figure 13.3 Relative Difference Chart – Second versus Original Laboratory Duplicated Data

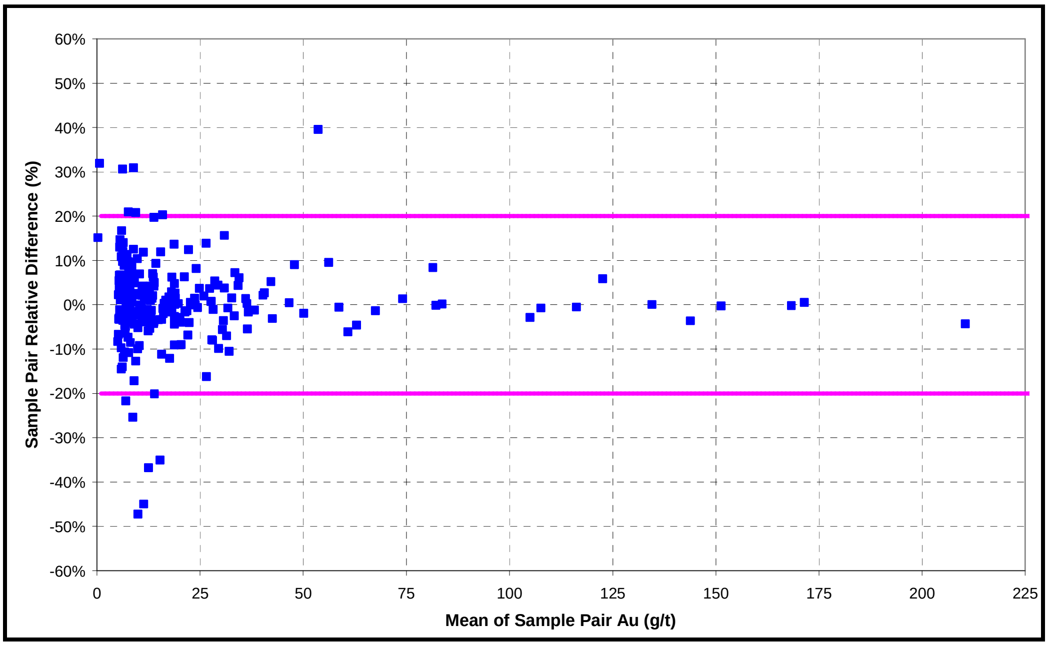
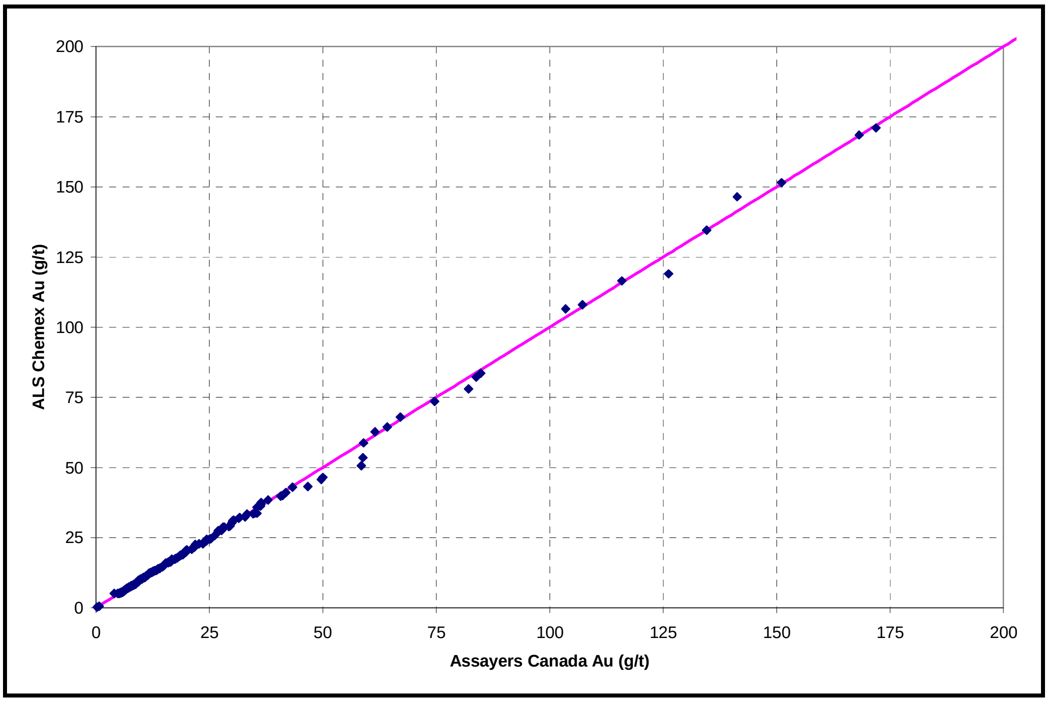


Figure 13.4 QQ Plot of Duplicate Samples Analyzed at Both the Second and Original Laboratory



13.2.5 Specific Gravity Program

Samples taken for assay from core holes are being measured for specific gravity and tabulated by rock type. The specific gravity for non-porous samples (the most common type) is calculated using the weights of representative samples in water (W2) and in air (W1). The bulk density is calculated by:

$$\frac{\mathbf{W1}}{\mathbf{(W1 - W2)}}$$

Less-common porous samples are dried and then coated with paraffin before weighing. Allowance is made for the weight and volume of the paraffin when calculating the specific gravity.

13.3 Concluding Statement

In Eldorado's opinion, the QA/QC results demonstrate that the Efemçukuru project assay database is sufficiently accurate and precise for resource estimation.

SECTION 14 • DATA VERIFICATION

As a test of assay data integrity, the data used to estimate the 2007 Efemçukuru mineral resource were verified against original source data. This process was implemented as part of database upgrading program with the installation of a Datashed system for the Efemçukuru project. Survey (collar and down hole) data and assay data were checked. Any discrepancies found were corrected prior to entry into the new database. Newer data entered directly into the database are periodically compared to original electronic certificates (assays) and down hole measurements and collar survey data. As a result, the data transferred for use in resource modelling are considered sufficiently free of error to be adequate for resource estimation of the Efemçukuru Project.

SECTION 15 • ADJACENT PROPERTIES

There are no properties adjacent to the Efemçukuru project site, nor properties in the local region. The closest active operating gold mine is located at Ovacki, Izmir province some 100 km north of Efemçukuru.

SECTION 16 • MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 INTRODUCTION

The testwork results summarized in Sections 16.3 and on, detail the relevant metallurgical test programs undertaken since 1993 up to 2007 and forms the basis for the process design parameters and criteria listed in Table 16.1 and Table 16.2 of Section 16.2. The flowsheets are attached in Appendix D.

16.2 PROCESS DESIGN

16.2.1 PROCESS DESIGN PARAMETERS

Whole-ore cyanidation, gravity concentration, flotation concentrate cyanidation, and combinations of the above, have been tested over the years. The recommended process flowsheet for the recovery of gold was a combination of gravity concentration, flotation, and concentrate cyanidation. The primary gravity concentrate, as well as the cleaner flotation concentrate, would be subjected to further upgrading by gravity concentration techniques to produce a smeltable grade product. This product would be dried and smelted on-site at Efemçukuru. The remaining concentrate would be de-watered, bagged and trucked to the processing facility at Kişladağ. This concentrate would then be reground and leached with cyanide, with the gold recovered by electrowinning.

16.2.2 PROCESS DESIGN CRITERIA

The process design criteria summary for the Efemçukuru process and Kişladağ process are shown in Table 16.1 and Table 16.2 respectively. The design parameters selected have been based on metallurgical testwork results which are detailed in subsequent pages of this section.

Table 16.1 Process Design Criteria - Efemçukuru

	Unit	Amount
Availability / Utilization		
Annual Processing Rate	t/y	401,500
Daily Processing Rate (calendar day)	t/d	1,100
Crusher Plant Operating Time	%	83.3
Crushing Processing Rate	t/h	66.9
Grinding and Flotation Plant Operating Time	%	90.0
Grinding and Flotation Processing Rate	t/h	50.9
Ore Properties		
Head Grade (Life-on-Mine Average):		
Gold	g/t Au	10.04
Silver	g/t Ag	17.8
Sulphide Sulphur	%	2.73
Bond Abrasion Index:	-	0.67
Bond Crushing Work Index:	kWh/t	17.0
SAG/Autogenous Mill Work Index:	kWh/t	18.1
Ball Mill Work Index	kWh/t	17.2
SG		2.91
Gold Recoveries		
Gold Recovery to Flotation Concentrate	%	62.0
Gold Recovery to Gravity Concentrate	%	30.0
Overall Gold Recovery	%	92.0
Doré Production		
Gold in Doré	kg	1,209
	oz	38,862
Gold in Concentrate	kg	2,498
	oz	80,315
Mass Deportment		
Primary Gravity Concentrate	%	0.039
Upgraded Cleaner Flotation Concentrate	%	0.001
Table Gravity Concentrate	%	0.002
Flash Flotation Concentrate (average)	%	11.7
Scavenger Flotation Concentrate	%	21.4
Cleaner-(Final) Flotation Concentrate (average)	%	8.1
Flotation Tailings	%	91.9

Table 16.2 Process Design Criteria - Kişladağ

	Units	Amount
Flotation Concentrate Properties		
Gold	g/t	76.5
Silver	g/t	134
Sulphide Sulphur	%	37
S.G.		3.3
Feed Size		
Particle Size P80	µm	20
Moisture	%	8
Bulk Density	t/m³	2.31
Production Criteria		
Concentrate Treatment Rate	t/h	4.14
Gold Recoveries		
Cyanide Leach Extraction	%	91.2
Doré Production		
Gold Content in Doré	kg	2277
	oz	73210
Cyanide Leaching		
Number of Stages:		
Pre-Aeration		1
Leaching		4
Pulp Density	%	30.0
Leach Residue	ppm CN	300
Leach Residence Time	h	48

16.2.3 PROCESS DESCRIPTION

The Efemçukuru process plant will be designed to treat a nominal 401,500 tonnes of gold and silver-bearing ore per year for a treatment rate of 1,100 tonnes per day at an overall plant availability of 90% (83% for the crushers). The processing facilities include crushing, followed by a SAG mill - ball mill grinding circuit with a classification step to produce an 80% minus 67 microns grind product size. A centrifugal gravity concentrator will treat a portion of the cyclone underflow feeding the ball mill. The cyclone overflow will be floated in a scavenger flotation circuit with the concentrate returned to join the feed to the flash flotation cell, which serves as the rougher flotation stage. The scavenger flotation tailings will be the final tailings at the Efemçukuru plant.

The ball mill discharge will be treated in a flash flotation cell, together with the scavenger concentrate. The flash flotation concentrate will be upgraded in a cleaner flotation circuit, and the cleaner tailings will be returned to the SAG mill discharge pumpbox which feeds the classification cyclones. The gravity concentrate will be upgraded to produce a smeltable gold product. The cleaner flotation concentrate will similarly be upgraded by gravity means to be combined with the upgraded gravity concentrate, prior to drying and smelting. The balance of the flotation concentrate will be combined with the upgraded gravity concentration tailings and will be thickened in the concentrate thickener, and then filtered to a low-moisture content concentrate. This concentrate will be bagged and transported to the Kişladağ plant where the gold will be recovered by further processing. The plant tailings will be thickened prior to filtration for use in a paste backfill plant, or disposal as dry stack tailings.

The Kişladağ plant will receive the bagged flotation concentrate. This concentrate will be re-slurried and milled in a regrind mill to obtain a particle size of 80% passing 20 microns. The reground concentrate product will be pumped to a pre-aeration tank, and this will be followed by leaching with cyanide. The leached slurry will be filtered to recover the pregnant solution. The pregnant solution will be heated and will then be fed to the electrowinning circuit to recover the gold and the silver metals. The resulting electrowinning sludge product will be washed, filtered, dried, and then smelted to produce Doré metal. The barren solution will be re-used in the regrinding and the leaching circuits. The washed leach residue will constitute the tailings which will be disposed of by discharging onto the conveyor belt feeding the Kişladağ leach pads.

16.3 METALLURGICAL TEST

Table 16.3 is a chronological summary of the testwork reports reviewed in order to derive the process design criteria required for the Efemçukuru and Kişladağ treatment facilities. The table also includes pre-feasibility reports, as well as review reports, issued since 1993. Some of the testwork programs conducted utilized processes which were not considered in the design of the two processing plants and will therefore not be referred to in this report.

Table 16.3 Reports Reviewed

Author	Date	Report Title
Genmin Process Research	02 Feb 1993	Kavacik Project.
Genmin Process Research	16 Mar 1993	Kavacik.
Vancouver Petrographics	02 Jul 1993	Petrographic Description of 13 samples from Tuprag Metals.
Anonymous	19 Jul 1996	Efemçukuru Project Metallurgical Testwork
Gencor (Genmin) Process Research	08 Jan 1997	Milling and Thickener Optimisation Tests for Efemçukuru.
Pocock Industrial Inc	00 Jun 1997	Flocculant Selection, Gravity Sedimentation, Pressure Filtration and Pulp Rheology Studies Conducted for Tuprag Metal Madencilik.
Anamet Services	00 Jul 1997	Results of Cyanide Leaching and Flotation Scoping Tests for Three Samples of Gold-Bearing Ore from the Efemcukura (sic) Deposit, Turkey.
Anamet Services	22 Sep 1997	Test Report Summary Efemçukuru Project.
Billiton (Gencor) Process Research	27 Oct 1997	Cyanidations of Tuprag Efemçukuru Ore and Concentrate.
Billiton Process Research	31 Oct 1997	Flotation Testwork on Efemçukuru Ore.
CSMA Minerals Ltd	30 Apr 1998	Metallurgical Testwork on Samples from the Efemçukuru Deposit.
A.R. MacPherson Consultants Ltd	05 May 1998	Proposed Grinding Circuit for Tuprag Efemçukuru Project.
U.I. Minerals	00 May 1998	Efemçukuru Project Metallurgical Testwork Review - Executive Summary.
U.I. Minerals	00 May 1998	Efemçukuru Project Metallurgical Testwork Review - Volume 1.
U.I. Minerals	00 May 1998	Efemçukuru Project Metallurgical Testwork Review - Volume 2.
CSMA Minerals Ltd	23 Jun 1998	Further Flotation and Cyanide Leach Testwork on Efemçukuru Ore.
U.I. Minerals	24 Jun 1998	Efemçukuru Project Metallurgical Testwork Review - Update.
A.R. MacPherson Consultants Ltd	24 Nov 1998	Proposed Ball Mill Circuit for Milling Tuprag Efemçukuru Ore.
Tuprag Metal Madencilik	10 Dec 1998	Efemçukuru Metallurgical Review.
Sao Bento	22 Dec 1998	Efemçukuru Flotation Concentrate Transport to and Treatment at Sao Bento, Brazil.
Kilborn Engineering Pacific Ltd	00 Jan 1999	Efemçukuru Project Prefeasibility Study.
U.I. Minerals	00 Feb 1999	Summary of Testwork Conducted on Efemçukuru Ores.
U.I. Minerals	26 Feb 1999	Mineralogical Study on Efemçukuru Ores.
Eldorado Gold Corporation	00 Mar 1999	Efemçukuru Gold Project - Prefeasibility Study, Volume 1.
Kilborn Engineering Pacific Ltd	00 Jan 2002	Efemçukuru Gold Project Pre-Feasibility Study, Process and Ancillary Facilities - Addendum.
J.R. Goode and Associates	19 Jun 2002	Efemçukuru Project, Review of Metallurgical Data.
U.I. Minerals	31 Jan 2006	Efemçukuru Project Testwork Developments.
Wardell Armstrong International (CSMAMinerals)	00 Feb 2006	Further Flotation and Environmental Testing of two Samples from the Efemçukuru Deposit.
Knelson Research and Technology	03 Jun 2007	Eldorado Gold; Gravity-Recoverable-Gold Test Results.

16.4 Testwork Program Components

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

- head analysis and specific gravity (SG) determinations
- mineralogical examination
- comminution testwork
- gravity amenability testwork
- gold and sulphide mineral flotation investigations
- cyanide leaching of flotation concentrates
- thickening - static tests
- pressure and vacuum filtration test; pulp viscosity.

16.5 Head Analysis and Specific Gravity Determinations

Table 16.4 to 16.8 summarizes the head assay analyses obtained for the various ore samples selected to reflect the variability of the ore deposit. Table 16.4 gives the assays obtained for the samples tested by Anamet Services in July 1997, while Table 16.5 provides analyses for samples used in a testwork program by Billiton Process Research in October 1997. Table 16.6 gives the head assay values for gold, as well as listing the SG determinations obtained from the CSMA Minerals testwork program conducted during April 1998. Anamet Services reported elemental analyses conducted on metallurgical test samples in July 1997 and by CSMA Minerals in April 1998. The results are presented in Table 16.7 and Table 16.8 respectively.

Table 16.4 Head Analyses for Tüprag Samples - Anamet Services

Sample	Tüprag 1		Tüprag 2		Tüprag 3	
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)
1	10.41	20.00	11.22	13.00	10.82	30.70
2	11.86	21.00	10.69	12.20	10.96	31.40
3	10.87	23.80	11.10	13.10	12.43	26.90
4	11.65	22.90	11.68	14.30	11.45	33.70
5	13.34	21.00	11.54	15.60	11.78	31.80
6	10.82	20.80	11.30	17.60	11.03	31.70
7	11.00	19.70	12.30	13.30	12.53	33.70
8	11.78	20.00	12.15	14.10	12.39	34.10
Arithmetic Mean	11.47	21.15	11.50	14.15	11.67	31.75

The multiple gold assays for each sample results reported in Table 16.4 above show good consistency with a range of between 10.41 and 13.34 g/t Au, and with an overall gold grade of about 11.5 g/t Au. The silver grades also show good consistency for each sample assayed, but overall show a wide range of values from a minimum of 12.2 g/t Ag to a maximum of 34.1 g/t Ag. The gold assay results reported in Table 16.5 show a similar high degree of variation from sample to sample in accordance with the selection process used.

Table 16.5 Head Assays - Billiton Process Research

Sample	Au (g/t)
1	17.70
2	16.50
3	37.80
4	20.10
5	16.90
Arithmetic Mean	21.80

Table 16.6 Head Assays and SG Determinations - CSMA Minerals

Sample	SG	Au (g/t)	Ag (g/t)
EFG1	2.897	7.45	7.90
EFG2	2.991	61.15	53.40
EFG3	2.985	8.46	14.40
EFG4	3.070	34.48	35.60
EFG5	2.902	13.74	12.80
EFG6	2.822	13.53	39.30
EFG7	2.721	20.12	22.20
Arithmetic Mean	2.913	22.70	26.51
GC2 - Composite Sample	2.993	11.55	20.90

The CSMA Minerals assay results reported in Table 16.6 also shows significant variability in the gold grade of the ore with values ranging from 7.45 g/t Au to 61.15 g/t Au reflecting their origin from different parts of the orebody. Similarly, the silver head grade varies markedly from a low value of 7.9 g/t Ag to a high of 53.4 g/t Ag. Sample GC2, a global composite sample, was constituted to represent the average ore plant feed material.

In order to define a basis for design, it was determined that the overall mine plan value grade would be 10.035 g/t Au and 17.8 g/t Ag, detailed in Section 17.2. These mine grade values are reasonably close to the values shown in the tables above, and for the GC2 sample in particular.

The SG values of the test samples reported in Table 16.6 above are consistent with an average value of 2.91 g/cm^3 which was the value used for the process design criteria.

The elemental analyses conducted by Anamet Services and CSMA Minerals has been presented in Table 16.7 and Table 16.8 respectively. The Total Base Metal analysis reflects the combined content of copper, lead, and zinc.

Table 16.7 Elemental Analysis - Anamet Services

Element	Sample		
	Tüprag 1	Tüprag 2	Tüprag 3
As (ppm)	455	500	1,448
Bi (ppm)	5	5	5
Cd (ppm)	8	8	34
Cu (ppm)	600	400	1,100
Hg (ppm)	0.003	0.064	0.081
Pb (ppm)	3,000	2,600	13,800
Total S (%)	1.96	2.26	6.87
Sulphide S (%)	1.88	1.25	3.84
Sb (ppm)	75	74	81
Se (ppm)	5	5	5
Te (ppm)	10	11	5
Zn (ppm)	2,000	2,800	10,000
Total Base Metals (%)	0.56	0.58	2.49

A review of the two sets of elemental analyses indicates that the data is relatively consistent and that the metal contents of the different samples vary greatly.

The data in Table 16.7 and Table 16.8 also highlights the variability of the ore samples selected for testing. Notable is the sulphur content which ranged in value from 0.26 to 5.5% as sulphide sulphur.

This variation in ore feed grade will have varying effects on the processing of these ores, and control will have to be kept on the origin and the type of ore being treated by the Efemçukuru plant and the subsequent leaching at the Kişladağ plant. The sulphide sulphur content will influence the flotation recovery with regard to mass recovery and the subsequent treatment of the flotation concentrate. The arsenic content could influence the degree of refractoriness of the ore, as may the pyrite content, while the base metal content will effect the consumption of sodium cyanide.

Table 16.8 Statistical Analysis of Assays - CSMA Minerals

Element	Mean	Minimum Value	Maximum Value
Au (g/t)	21.31	7.45	61.15
Ag (g/t)	25.81	7.90	53.40
Cd (ppm)	17.4	0.8	32.2
Sb (ppm)	7.6	5.0	16.1
As (ppm)	776	400	1,507
Hg (ppm)	0.2	0.1	0.5
Cu (ppm)	769	199	1,610
Pb (ppm)	6,300	1,500	13,500
Zn (ppm)	6000	2400	11,400
Fe (%)	5.78	2.86	8.57
Total S (%)	3.02	0.27	6.27
Sulphide S (%)	2.73	0.26	5.50
Total Base Metals (%)	1.31	0.46	2.49

16.6 MINERALOGICAL EXAMINATION

Vancouver Petrographics conducted an examination of 13 samples submitted by Tüprag in 1993. The major gangue minerals were identified to be quartz with varying amounts of rhodochrosite and rhodonite, as well as chlorite, hematite, jarosite, fluorites, calcite and manganese-silicates. The dominant sulphide mineral was found to be pyrite with subordinate and varying amounts of chalcopyrite, sphalerite, and galena, and generally trace amounts of arsenopyrite, tetrahedrite, covellite, pyrrhotite and marcasite. Gold was identified in five of the 13 samples. Generally, the gold was associated with pyrite as grains between 3 and 53 microns in size, and at times, the gold-containing pyrite was associated with sphalerite and/or chalcopyrite. Gold grains were also observed to be present as blebs in sphalerite, and chalcopyrite, and galena, and carbonate gangue. However, the most common association of the gold grains was found to be with the pyrite.

Anamet Services conducted a mineralogical examination in 1997 and identified the main gangue and ore-bearing minerals to be essentially the same as were reported by Vancouver Petrographics. Anamet also confirmed the presence of an unidentified bismuth-lead-silver sulphide mineral, and metallic bismuth, while the gold-bearing mineral was identified as being electrum, which is native gold, but containing varying amounts of silver in solid solution. The electrum was found to occur mainly as inclusions and along fractures with mainly pyrite, while sulphide-electrum intergrowths were also observed. The maximum liberated size particle of electrum was about 30 microns. Intergrowths and partial intergrowths of electrum particles of

15 microns, or larger, with mainly pyrite, were observed. Electrum particle sizes ranged from 0.5 to 30 microns, but were found to be mostly associated or locked in pyrite. The pyrite was generally liberated at a grind size of about 75 microns, although a significant proportion of the electrum would be locked in the pyrite at this grind size. Galena, sphalerite and chalcopyrite would generally be liberated at this grind size, although individual grains would contain intergrowths and/or inclusions of the other minerals.

16.7 Comminution Testwork

A.R. MacPherson Consultants conducted grinding testwork on Efemçukuru samples in 1998. Table 16.9 shows the results obtained.

Table 16.9 Efemçukuru Grindability Data – MacPherson Consultants

Tüprag Composite Sample	Unit	Value
Feed Size, F_{80}	µm	21,370
Product Size, P_{80}	µm	231
Gross Autogenous Work Index	kWh/t	22.45
Correlated Autogenous Work Index	kWh/t	18.10
Rod Mill Bond Work Index	kWh/t	18.40
Ball Mill Bond Work Index	kWh/t	17.20
Bond Abrasion Index	g	0.6701
SAG Mill Products SG	g/cm^3	3.06

The rod mill value was determined at the closing size of 1,410 microns (14 mesh), and the ball mill value at the closing size of 149 microns (100 mesh). The testwork reflects that the ore is relatively hard. In addition, the ore is abrasive and resistant to impact breakage, highlighting the need for a pebble crusher in the SAG mill circuit. This data was incorporated into the design of the grinding circuit.

16.8 Gravity Concentration and Flotation Testwork

Several testwork programs were conducted using gravity concentration and flotation testing of the Efemçukuru samples. The most comprehensive program was that conducted by CSMA Minerals in April 1998. Supplementary flotation testwork was conducted by WAI and reported in February 2006. Gravity-gold-recoverable (GRG) testwork was completed by Knelson Research on three samples submitted. The results obtained were reported in June 2007. The results obtained are discussed below.

16.8.1 CSMA MINERALS, APRIL 1998

The material presented to gravity concentration and flotation was milled to 85% passing 75 microns without any additional grinding step between the processes. The gravity concentration tests were performed with a Knelson centrifugal concentrator, and the concentrate obtained was further upgraded using a Mozeley separator. Table 16.10 presents the results of the eight tests conducted. The eight samples tested were EFG1, EFG2, EFG3, EFG4, EFG5, EFG6 and EFG7 and GC2. The first seven samples represented different types of mineralization of the Efemçukuru deposit. Sample GC2 was a composite sample representing the anticipated mill feed average mineralization.

The gold and silver in all the samples was shown to be clearly amenable to gravity recovery. The gold recovery values obtained varied between 26.1 to 57.0% while the silver recoveries varied between 5.4 and 30.8%. The subsequent flotation recovery was also good, with between 30 and 65% of the residual gold being recovered into a flotation concentrate resulting in an overall recovery ranging between 81 and 95%. The flotation recovery for the residual silver after gravity concentration varied between 44 and 77% resulting in an overall silver recovery of between 49 and 90%. The differences in the recoveries give an indication of the variability of the ore in the deposit. There was insufficient sample available from the gravity concentrate for a sulphur analysis to be determined and this presents a slightly distorted view of the sulphur balance as given in Table 16.10. The design grind adopted was 80% passing 67 microns.

The testwork also indicated that selectivity was highly variable, with flotation concentrate mass recoveries generally ranging from 5.4% to 17.1%, although the test from sample EFG7 resulted in a very high mass recovery of 28.9%. This sample was taken from the transition zone where oxidation levels are higher.

The GC2 sample, a composite of all the ore types representing the expected average mineralization, gave the lowest gravity concentration recovery of all the samples, highlighting the variability of the Efemçukuru ore samples. However, the combined recovery from the gravity and flotation processes was 91% which is a reasonable overall gold recovery value.

Table 16.10 Gravity Concentration and Flotation Test Results - CSMA Minerals

Sample Tested and Products	Recovery (%)			
	Weight	Au	Ag	Sulphur
Sample GC2				
Gravity Concentrate	0.20	26.1	6.8	-
Flotation Concentrate	15.37	64.8	63.8	95.9
Total Recovery (gravity + flotation)	15.57	90.9	70.6	95.9
Sample EFG1				
Gravity Concentrate	0.14	45.2	19.0	-
Flotation Concentrate	8.30	49.3	66.3	99.1
Total Recovery (gravity + flotation)	8.44	94.5	85.3	99.1
Sample EFG2				
Gravity Concentrate	0.35	40.3	30.8	-
Flotation Concentrate	11.57	49.1	56.4	99.8
Total Recovery (gravity + flotation)	11.92	89.4	87.2	99.8
Sample EFG3				
Gravity Concentrate	0.14	26.2	10.6	-
Flotation Concentrate	13.51	60.4	77.3	93.4
Total Recovery (gravity + flotation)	13.65	86.6	87.9	93.4
Sample EFG4				
Gravity Concentrate	0.29	44.9	28.1	-
Flotation Concentrate	15.19	47.2	61.7	94.1
Total Recovery (gravity + flotation)	15.48	92.1	89.8	94.1
Sample EFG5				
Gravity Concentrate	0.28	45.5	16.9	-
Flotation Concentrate	5.43	47.0	69.7	99.4
Total Recovery (gravity + flotation)	5.71	92.5	86.6	99.4
Sample EFG6				
Gravity Concentrate	0.33	38.1	5.4	-
Flotation Concentrate	17.05	42.9	44.0	97.2
Total Recovery (gravity + flotation)	17.38	81.0	49.4	97.2
Sample EFG7				
Gravity Concentrate	0.15	57.0	16.6	-
Flotation Concentrate	28.93	29.7	43.6	96.9
Total Recovery (gravity + flotation)	29.08	86.7	86.7	96.9

16.8.2 KNELSON RESEARCH & TECHNOLOGY CENTRE, JUNE 2007

Three ore composites, made up from a number of individual core intervals, were tested to determine the amount of gravity recoverable gold under standard test conditions utilizing four stages of size reductions. The samples were selected to represent the two sources of ore, namely SOS (South Ore Shoot) and MOS (Middle Ore Shoot), and also to test a high-grade MOS sample. The results are summarized in Table 16.11. These results indicate that ores from both SOS and MOS contain a significant amount of free gold, resulting in 41 to 57% gold recovery in these gravity concentration tests. Gold grains in the recovered gravity concentrates had a particle size of 80% passing 93, 88, and 115 microns, respectively for samples EFG10 (SOS sample), EFG11 (MOS sample) and EFG12 (MOS high-grade sample).

Since the recovery of gold under plant conditions varies with regard to mass recovery, and gold will be lost during the subsequent upgrading with a shaking table, the actual gravity gold recovery will be less than the values given in Table 16.5. However, the Knelson results confirm the gravity recovery values previously obtained by CSMA Minerals, and also validates the selection of the gravity concentration process in the flowsheet. The assumed recovery of 30% gold in the process design criteria recovery by gravity will be a conservative estimate for the planned ore feed to the plant consisting of 50% EFG10-type and 50% EFG11-type material.

Table 16.11 Summary of Gravity-Recoverable-Gold Tests - Knelson Research

Sample ID			EFG10	EFG11	EFG12
Ore Shoot			SOS	MOS	MOS
Head Assay	Gold	g/t	11.9	16.7	66.3
	Silver	g/t	9.9	30.0	59.9
	Copper	%	0.01	0.07	0.07
	Iron	%	1.57	8.06	9.07
	Lead	%	0.15	1.14	0.95
	Total Sulphur	%	1.11	6.11	6.06
	Zinc	%	0.34	1.44	0.92
Cumulative Pass for Gravity Recoverable Gold	Passing 20 µm	%	29	27	31
	Passing 50 µm	%	50	48	63
	Passing 80 µm	%	93	88	115
Gravity- Recoverable-Gold Value	Concentrate Mass	%	1.80	2.09	2.15
	Gold Recovery	%	57.0	41.1	49.8
	Stage 1 Size, P80	microns	796	717	1,304
	Stage 2 Size, P80	microns	166	198	196
	Stage 3 Size, P80	microns	85	91	91
	Stage 4 Size, P80	microns	66	63	75

16.8.3 DIAGNOSTIC LEACH TEST - GENCOR

In February 1999, U.I. Minerals reported the results of diagnostic leach test results carried out by Gencor Process Research in 1993. In the absence of the actual Gencor results, the data will be quoted from the U.I. Minerals report. The results of the mineralogical distribution and association of gold present in a composite sample of borehole core material is outlined in Table 16.12 below.

Table 16.12 Diagnostic Leach Results – Gencor Process Research (quoted by U.I. Minerals)

Item	% Distribution	Remarks
Gold recovered by direct cyanidation	89.94	Test Details: grind 80% passing 45 microns; 1.5 kg/t CN; 5.0 kg/t $Ca(OH)_2$; pH = 11.5; 1 to 2 hours pre-oxidation; 12 to 24 hours leaching. The sample head grade was calculated to be 11.60 g/t Au.
Gold adsorbed on carbonaceous material	0.14	
Gold associated with pyrrhotite and carbonate minerals	1.38	
Gold associated with sulphide minerals	0.59	
Gold occluded within quartz	7.95	
Total	100.00	

These results indicate that less than 2% of the gold is associated, or occluded, by sulphide and carbonate minerals. The high degree of variability between the different samples tested is evident.

16.9 FLOTATION TESTWORK

16.9.1 WARDELL ARMSTRONG INTERNATIONAL

The most definitive flotation testwork of Efemçukuru samples was reported by WAI (CSMA Minerals) during 2005 and 2006 when batch and locked-cycle flotation tests were undertaken on two samples. The two samples tested represented the MOS (Middle Ore Shoot) and the SOS (South Ore Shoot). The results from the locked-cycle testwork conducted on these two samples, MOS (sample GC4) and SOS (sample GC3), were reported together with the flotation testwork results from the composite sample GC2 which had been tested in 1998. All three sets of data obtained are reproduced in Table 16.13. The MOS/GC4 and SOS/GC3 information constitutes the average data from the last two cycles of the locked-cycle tests. The results quoted in Table 16.13 are reported directly from the respective reports. The test conditions for all three tests were similar, namely a grind size of 80% passing 67 microns, natural pulp pH, and a standard reagent suite which is described in detail in

Table 16.14.

Table 16.13 Summary of Flotation Results of Composite Samples - WAI

Sample	Product	Mass Recovery (%)	Assays			Distribution (%)		
			Au (g/t)	Ag (g/t)	S (%)	Au	Ag	S
GC2	Concentrate	8.10	183.00	212.00	42.8	88.6	83.7	91.7
	Tailings	91.90	2.07	3.64	0.34	11.4	16.3	8.3
	Feed	100.00	16.70	20.50	3.77	100.0	100.0	100.0
MOS/GC4	Concentrate	16.35	116.20	197.00	41.60	94.2	94.1	98.4
	Tailings	83.65	1.39	2.42	0.14	5.8	5.9	1.6
	Feed	100.00	20.16	34.23	6.92	100.0	100.0	100.0
SOS/GC3	Concentrate	3.91	285.80	315.40	35.50	92.2	85.9	95.6
	Tailings	96.09	0.98	2.12	0.07	7.8	14.1	4.4
	Feed	100.00	12.12	14.37	1.45	100.0	100.0	100.0

The variation in the head grades of the three representative samples is apparent, although the GC2 head grade values are a reasonable average of the MOS/GC4 and SOS/GC3 samples with respect to gold, silver and sulphur. Similarly, the mass recovery for GC2 at 8.1% is reasonably positioned between the MOS/GC4 sample with the higher mass recovery of 16.4% and the relatively high 6.92% sulphur grade, and the SOS/GC3 sample with the lower mass recovery of 3.9% and the lower sulphur grade of 1.45%. However, the GC2 sample recoveries for gold, silver and sulphur would be expected to range between those obtained for the MOS/GC4 and the SOS/GC3 samples, but this was not found to be the case. The gold, silver and sulphur recoveries for GC2 were in fact lower than the SOS/GC3 sample results. The probable presence of oxidized material in this sample could have contributed to these results. However, it is apparent that the two different areas of the deposit, namely the MOS/GC4 and SOS/GC3, are mineralogically distinct as typified by the different grades, and particularly the sulphur grade. A sulphide flotation recovery process will therefore be expected to give varying results depending on the origin, and the sulphur grade, of the ore reporting to the processing plant.

The overall design of the flotation circuit will be based on GC2 conditions, namely a final flotation concentrate mass recovery of 8.1% and an overall gold recovery of 92% of which 30% would be recovered by gravity concentration and 62% by flotation. This is in accord with the results given in Table 16.10 which indicated an overall gold recovery of 90.9% of which 26.1% was recovered by gravity concentration, and 64.8% recovered by flotation.

Table 16.14 Standard Flotation Conditions

Reagents/Conditions	Addition Rate/Remarks
NaSH	100 g/t; sulphidizing agent
Copper sulphate	100 g/t; sulphide mineral activator/surface modifier
SIBX	40 g/t; collector reagent
S8649	40 g/t; collector reagent
AF70	20 g/t; frother reagent
OPT45 - guar gum	100 g/t; depressant reagent
Slurry pH	Natural; was found to vary between 6.2 and 6.8 for MOS; 7.5 to 8.0 for SOS
Pulp density	30%
Grind/particle size	80% passing 67 microns

16.9.2 CLEANER FLOTATION TESTWORK

It was recognized that, from an economic perspective, it would be beneficial to conduct testwork to establish the practical minimum amount of concentrate mass that could be generated given the long trucking distance and the expense involved in hauling the concentrate from the Efemçukuru plant to the Kişladağ site. The results obtained indicate that a low-mass recovery option be implemented during plant operations to avoid excess gangue being recovered into the concentrate.

16.10 CYANIDE LEACHING OF FLOTATION CONCENTRATES

CSMA Minerals conducted a detailed series of tests using flotation concentrates produced from each of the samples (except EFG6 which had insufficient sample) collected and tested for the April 1998 testwork program. The flotation concentrate generated from the basic flotation tests was reground to 100% passing 38 microns, and then subjected to cyanide leaching. The cyanide concentration at the start of the test was 6.7 kg/L CN and was allowed to degrade as the leach progressed. The lime was added in sufficient amounts to maintain the pH at about 11.0.

Table 16.15 shows the initial starting conditions and the results of the tests.

Table 16.15 Leaching of 100% Passing 38 µm Flotation Concentrate – CSMA Minerals

Leaching Conditions		
Parameter	**Unit**	**Value**
Grind	microns	100% passing 38
Sample Mass	kg	2
Pre-Aeration Time	hours	2
Pulp Density	% solids	40.0
Leaching Time	hours	24.0
NaCN Added to Leach	kg/t	10.0
pH of Pulp	-	11.0

Sample	Head Grade Au (g/t)	Head Grade Ag (g/t)	Dissolution Au (%)	Dissolution Ag (%)	NaCN Used (kg/t)	Initial pH	Final pH	Lime Used (kg/t)
EFG1	35.25	42.30	95.48	71.29	5.21	8.88	11.71	0.60
EFG2	238.73	181.80	97.59	81.98	8.69	8.85	11.70	0.65
EFG3	23.45	44.70	82.55	53.97	5.67	8.97	11.75	0.75
EFG4	126.81	175.10	22.71	0.37	9.86	9.66	11.80	0.60
EFG5	89.56	89.80	93.94	73.59	6.28	8.73	11.75	0.80
EFG7	38.06	42.20	97.43	82.04	5.05	8.70	11.36	1.55
GC2	60.74	93.90	92.82	61.64	7.80	9.32	11.53	0.60

Sample GC2 indicated that gold extractions of 92.8% could be achieved under the leach conditions as specified in the above table. The dissolution was observed to be very rapid since the leach test was only conducted over a 24-hour period.

In order for CSMA to assess the potential for improved leach performance of the flotation concentrate generated from samples EFG3 and EFG4 by finer grinding, a sample of each concentrate previously tested was reground to 100% passing 10 microns, and subsequently subjected to cyanide leaching.

Table 16.16 shows the initial starting conditions and results of the two tests.

Table 16.16 Leaching of 100% Passing 10 µm Flotation Concentrate - CSMA Minerals

Leaching Conditions					
Parameter		**Unit**		**Value**	
Grind		microns		100% passing 10	
Pre-aeration Time		hours		2	
Pulp Density		% solids		10.0	
Leaching Time		hours		24.0	
NaCN Added to Leach		kg/t		10.0	
pH of Pulp		-		11.0	
Sample	**Head Grade Au (g/t)**	**Head Grade Ag (g/t)**	**Dissolution Au (%)**	**Dissolution Ag (%)**	**NaCN Used (kg/t)**
EFG3	23.5	44.7	78.7	57.1	14.3
EFG4	126.8	175.1	91.9	69.5	6.7

Sample EFG3 did not respond to fine grinding yielding a slightly lower extraction of 78.7% compared with an extraction value of 82.6% attained originally (compare the results of

Table 16.15 and

Table 16.16). However, a significant improvement in the extraction of sample EFG4 was obtained with the finer regrind test which yielded a gold dissolution of 91.9%.

Further differential regrind work conducted by CSMA indicated that fine grinding of the flotation concentrates would be beneficial to the leach recovery of both gold and silver, although this would be accompanied by a corresponding increase in cyanide consumption, particularly in regrinding from 100% passing 55 microns to a particle size of 100% passing 20 microns. In addition, the extraction results recorded were found to be consistent with the previous results obtained with sample GC2 in that a leach extraction of 93.2% was obtained in this test with the regrind to 100% passing 20 microns. Table 16.17 presents the results of these tests.

Table 16.17 Effect of Regrind Size on Gold Extraction of GC2 - CSMA Minerals

Leaching Conditions				
Parameter	**Unit**		**Value**	
Grind	microns		80% passing	
Sample Mass	kg		2	
Pre-Aeration Time	hours		2	
Pulp Density	% solids		10.0	
Leaching Time	hours		24.0	
NaCN Added to Leach	kg/t		10.0	
pH	-		11.0	
Sample	**Regrind P80 (microns)**	**Dissolution Au (%)**	**Dissolution Ag (%)**	**NaCN Used (kg/t)**
GC2	55	87.2	56.6	6.82
GC2	40	87.9	61.1	8.45
GC2	20	93.2	62.9	8.82

While the statistical basis for selecting fine grinding as a process route has clearly not been conclusively established by the above tests, generally high gold dissolution values were observed at the regrind particle size of 100% passing 38 microns, or approximately 80% passing 20 microns. This regrind size of 80% passing 20 microns has therefore been used for design purposes.

16.11 THICKENING TESTS

Static thickening testwork has limited usefulness for definitive thickener design, but in the absence of continuous rake test data, this will be used in the design of the thickeners and filters required. The available testwork from Gencor Process Research and CSMA Minerals were not conducted on the same basis, but the testwork does provide useful data for establishing preliminary settling and filtration rates for the process products.

16.11.1 GENCOR PROCESS RESEARCH - JANUARY 1997

Table 16.18 shows results obtained by Gencor Process Research in 1997 for thickening tests on flotation concentrate at various grind sizes.

Table 16.18 Thickening Test Results - Gencor Process Research

Grind Size 80% Passing	Flocculant Addition (g/t)	Thickening Rate (t/d per m^2)
45 microns	12.0	24
63 microns	6.5	40
75 microns	10.0	34
106 microns	8.3	30

Magnafloc 155 was found to be the optimal flocculant.

16.11.2 U.I. Minerals – February 1999

This report covers work done by CSMA Minerals during 1998, but the source data was not identified in either of the two CSMA documents made available for the review, and the discussion will be based on the information supplied by U.I. Minerals.

The settling tests were conducted on flotation concentrate and flotation tailings produced from the flotation test campaign. The pulp density and pH were not altered and no flocculant was added. The grind size for these tests was about 80% passing 67 microns. The settling area for the flotation concentrate was determined to be 0.711 t/d per m^2. The equivalent tailings settling area was found to be 0.404 m^2 per t/d.

The filtration tests were also conducted on flotation concentrate and flotation tailings samples produced during the flotation test campaign. The pulp density and slurry pH values were not altered and no flocculant was added. The filtration rate for the flotation concentrate sample was found to be 137.1 kg/day per m^2 and the filtration rate for tailings was determined to be 350.8 kg/day per m^2. No additional information was made available. These filtration rates are poor and probably reflect the lack of flocculant addition.

16.12 Pressure and Vacuum Filtration Tests - Pocock 1997

All the testwork work undertaken by Pocock Industrial during 1997 adopted a grind of 80% passing 30 microns with application to tailings disposal after whole-ore cyanide leaching tests. The work of interest was the sedimentation and pressure filtration of cyanide leach residue thickener underflow, and the results obtained have been presented in Table 16.19 for the sedimentation tests to serve as a guide for the design of the thickening and filtration of leach residue. Table 16.20 presents the pressure filtration tests.

Table 16.19 Leach Residue Sedimentation Test Results – Pocock Industrial

Test Parameters and Results	Cyanide Leach Residue Sample
Flocculant - anionic (g/t)	25 to 35
pH	8.6
Feed Density, % Solids	<35
Solids Loading (t/d per m^2)	40 to 45
Feed Hydraulics (m^3/h per m^2)	3.5 to 4.6
Underflow Density (% Solids)	60 to 65

An anionic flocculant, Magnafloc 155, was identified as the optimal flocculant. The actual solids loading value, without scale-up, is given in the above table, while the feed hydraulics values are also given without the scale-up value. The predicted underflow density values were determined by static extension tests with a minimum 60-minute compression retention time.

Recommendations for a high rate thickener design include:

- limiting the feed density to less than 35% solids (to about 20% solids) by internal or external dilution
- flocculant dilution at 0.1 g/L solution strength to be added prior to pulp contact at a dosage rate 25 to 35 g/t
- the thickener sizing should be based on the maximum feed loading of 4.6 m^3/h per m^2 and the solids loading of 45 t/d per m^2
- the target thickener underflow density should be 60% solids.

The results of pressure filtration testwork conducted by Pocock Industrial on a sample of cyanide leach residue and thickened cyanide residue thickener underflow slurry is presented below in Table 16.20.

Table 16.20 Leach Residue Pressure Filtration Test Results - Pocock Industrial

Test Parameters and Results	Cyanide Leach Residue	
	Leach Residue Sample	Thickener Underflow Residue Sample
Feed Solids (%)	42.00	58.00
Bulk Cake Density (t/m^3)	1.58	1.57
Cycle Time (min)	9.10	6.00
Filter Cake Moisture (%)	15.00	15.00
Sizing Basis (m^3/t)	0.79	0.80

The testwork was performed using an automatic filter press with a cycle time, which included a 3.5-minute dead time for cake discharge and cleaning. The sizing basis also included a 1.25 scale-up factor. Of particular interest are the pressure filtration tests on high rate thickener underflow to examine the effect of cake thickness and air blow duration on production rate and filter cake moisture. The cycle time listed in the table above included cake formation time, pumping time, air blow time, and dead time during the cake removal and cleaning phases. Of interest is the fact that the cake moisture value for both tests, namely a leach residue sample at 42% solids feed to the filter and a thickener underflow sample at 58% solids feed to the filter, was the same at 15% solids. However, the cycle time was significantly reduced in the case of the thickener underflow sample. No wash cycle was included with these tests and therefore the wash efficiency cannot be stated. In addition, it was not specified whether a filter cake sample containing less than 15% moisture could be produced.

16.13 PULP VISCOSITY TESTS - POCOCK 1997

During 1997, Pocock Industrial also undertook preliminary viscosity testwork on samples of cyanide leach residue slurry identical to that used for the filtration testwork described in Section 16.12 above. The pulp viscosity data was collected using a Brookfield Model LVT rotating viscometer. The data presents a comparison of apparent viscosity versus shear rate for the various solid concentrations. The results obtained are presented below in Table 16.21.

Table 16.21 Cyanide Leach Residue Viscosity Test Results - Pocock Industrial

Solids (%)	Temperature (°C)	pH	Viscosity cP at $5s^{-1}$	Viscosity cP at $25s^{-1}$
62.8	18.2	8.6	900	300
54.0	18.2	8.6	280	90
44.7	18.2	8.6	80	20

The decreasing apparent viscosity with increasing shear rate (shear thinning) is characteristic of pseudo-plastic non-Newtonian fluids. It demonstrates the necessity of achieving and maintaining a specific velocity gradient to initiate and maintain slurry flow conditions.

16.14 CONCLUSIONS

The test results reported above have formed the basis for the design of the process plants at Efemçukuru and Kişladağ. The process unit operations are described in Section 19.2.

SECTION 17 • MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 MINERAL RESOURCE ESTIMATE

The mineral resource estimates for the Efemçukuru Project were calculated under the direction of Dr. Stephen Juras, P.Geo. The estimates were made from 3D block models utilizing commercial software (Gemcom). The Efemçukuru Project for 2007 comprised two deposits along the Kestane Beleni Vein system: SOS and MOS. Both were represented in the same block model. Project limits, in truncated UTM coordinates, are 97770 to 98280 East, 37970 to 38870 North, and +770 to +1290 m elevation. Cell size for the project was 4 m east x 4 m north x 4 m high. The model was rotated 10° to the west (that is, model north is 10° west of project north).

17.1.1 GEOLOGIC MODELS

Nearly all known gold mineralization is contained within the Kestane Beleni Vein, closely associated hanging wall splays and local marginal stockwork veins to the principal veins. However, in areas of greater vein thickness, the gold distribution is only locally consistent. It may occur either along the hanging wall or footwall portions, in the central part, or throughout the entire thickness of the vein. Though the occurrence of sulphide minerals correlates with the presence of gold, grade information can only be obtained by assaying. Therefore domains to constrain grade interpolation are by necessity grade based.

Eldorado used new data from the infill drill program and revised structural interpretations of the Kestane Beleni Vein system to create mineralized or grade shapes for the Efemçukuru gold mineralization. These 3D shapes were based on approximately a 2.0 g/t Au grade threshold and general vein geometry. The threshold value was chosen by inspection of histograms and probability curves, and further supported by indicator variography. Areas of narrow or absent above threshold mineralization were included by implementing a minimum 2 m interval rule. The shapes were checked in plan and section and edited to be consistent with the structural and vein models and the drill assay data.

Seven sets of mineralized shapes or envelopes were made: one each for the Main Vein in the SOS and MOS, one for the folded south end of the SOS Main Vein, one each for the Upper Splay Veins in the SOS and MOS, one for a deeper or Lower Splay Vein at the south end of the SOS, and a mineralized stockwork envelope in the MOS. These solids were used to code the drill hole data and the block model.

VOIDS

There is significant surface evidence to indicate some of the veins have been mined in ancient times with small surface openings and waste dumps. During the drill program numerous voids were logged in drill core, primarily in the SOS. Each of these voids was examined in cross-section, a void shape digitally drawn on the relevant sections. These were subsequently meshed to form 3D objects and used to calculate a VOID percentage in the model blocks. This number was subtracted from the ORE percent calculated for each block prior to calculating resource and reserve tonnage and grade. About 60,000 tonnes were subtracted due to the interpreted voids.

17.1.2 DATA ANALYSIS

The seven mineralized domains were reviewed to determine appropriate grade interpolation parameters. Descriptive statistics, histograms and cumulative probability plots have been completed for gold in each domain. Results obtained were used to guide the construction of the block model and development of estimation plans. The data analyses were conducted on length weighted assay values and 1 m down-hole composited assay data.

HISTOGRAMS AND CUMULATIVE FREQUENCY PLOTS

Histograms and cumulative probability plots display the frequency distribution of a given variable and demonstrate graphically how that frequency changes with increasing grade. With histograms, the grades are grouped into bins, and a vertical bar on the graph shows the relative frequency of each bin. Cumulative frequency or cumulative distribution function (CDF) diagrams demonstrate the relationship between the cumulative frequency (expressed as a percentile or probability) and grade on a logarithmic scale. They are useful for characterizing grade distributions and identifying multiple populations within a data set.

Initial analyses were done on gold assays irrespective of hosting lithology within the SOS and MOS deposits, respectively. The positively skewed trends for both deposits show multi-modal populations. Lower grade thresholds of around 0.2 g/t Au (35^{th} percentile) and ~ 2.0 g/t Au (70^{th} percentile) occur in both distributions. The ~ 2.0 g/t Au threshold lends support for the use of the gold grade shells at SOS and MOS.

Subsequent analyses were done on composited data inside the generated mineralized envelopes. Within shell gold grades in the SOS domains (Main Vein, Upper Splay and Lower Splay) show positively skewed trends due to a strong 2 to 8 g/t population (55% of the distribution). In addition, the Main Vein and Upper Splay domains show a high grade "break" in the CDF trends, occurring at the 95^{th} percentile at a grade of 35 g/t for the Main Vein and a grade of 30 g/t for the Upper

Splay. The multiple thresholds likely correspond to multiple pulses of mineralization responsible for the SOS deposit, with the main pulse corresponding to the 2 - 8 g/t grade range.

Gold grades inside the MOS mineralized shells show strongly positively skewed trends for the Splay and Stockwork domains, but only slightly skewed pattern in the MOS Main Vein. Multiple thresholds are present but no distinct high grade "break" is observed in the CDF distribution. An exception occurs in the Stockwork domain with a "break" in its CDF trend at ~50 g/t (90th percentile). The multiple populations in the Main Vein domain are at higher grades with the 2 - 8 g/t range population only comprising 30% of the MOS distribution. The distribution here is less varied as shown by the relative low CV value (Table 17.1). Clearly the MOS deposit experienced a somewhat different mineralization history than that in the SOS.

Appendix C contains histograms and CDFs plots for gold for the project area. The statistical properties of the gold data are summarized in Table 17.1.

Table 17.1 Efemçukuru Statistics for 1 m Capped Composite Au Data (g/t)

Lithology / Zone	Mean	CV	q25	q50	q75	Max	# of Comps
South Ore Shoot							
Main Vein	11.89	1.64	3.70	6.95	13.32	200	544
Upper Splays	11.83	1.81	3.26	5.88	11.83	200	144
Lower Splay	7.69	0.58	4.41	6.35	9.36	21.3	19
Middle Ore Shoot							
Main Vein	18.65	1.23	4.84	11.55	23.30	200	562
Upper Splay	18.62	1.57	3.32	6.50	17.15	145	101
Stockwork	2.90	2.94	0.26	0.77	2.14	108	946

Evaluation of Extreme Grades

Extreme grades were examined for gold, mainly by histogram and CDF plots. Very high grade outlier samples were given a high level cap equal to 200 g/t Au (approximately 99.7% level of the grade distribution). This was applied to the assay data prior to compositing in all domains in the SOS and MOS. Six assay intervals were capped in the SOS and five capped in the MOS.

In the SOS deposit, a distinct higher grade population was noted beginning at 35 g/t Au. Examination of the distribution of these grades in section and plan shows that rather than behaving randomly these high grade samples tend to occur in clusters. This behaviour combined with the drill density and search ellipse strategy (see below) would serve to limit over-extrapolation of these higher grades. In areas of less dense drill coverage, an outlier restriction was used (at 35 g/t) to prevent over-extrapolation of high grade into areas of lower grade.

A similar though much less distinct higher grade population is seen in the MOS distribution. Occurring at about 70 g/t Au, these values also were observed to be present in small groups. Thus a similar approach was implemented on the treatment of higher grade values at the MOS. For areas using the outlier restriction, a threshold grade of 70 g/t Au was used in the MOS.

ESTIMATION DOMAINS

The data analysis and geologic interpretation of the gold mineralization at Efemçukuru supports the use of a grade based shell to define the mineralized portions of the vein systems. As described above, grade shells were constructed in the veins using a threshold grade of about 2 g/t Au. The spatial relationships between the main vein and splay veins in both SOS and MOS deposits also necessitate the use of separate domains for grade interpolation. The MOS stockwork zone with its marked grade contrast to the vein mineralization was also treated as a separate estimation domain.

Grades in these domains would be estimated with a hard boundary logic, that is, only composites within a domain would be used to interpolate grade into blocks defined by that domain.

17.1.3 VARIOGRAPHY

Variography, a part of data analysis, is the study of the spatial variability of an attribute. Correlograms, rather than the traditional variograms, were used on the Efemçukuru data because of its lower sensitivity to outliers, and its normalization to the variance of the data for a given lag.

Correlograms were calculated for gold in the SOS and MOS Main Vein domains. Multiple directional sample correlograms were calculated, and then modelled via a best fit model (SAGE software). The model consist of a nugget effect (measure of the random variation component), single or two-nested structure variance contribution, range for the variance contribution, and the model type (spherical in this analysis). After fitting the variance parameters, the modelling algorithm fits an ellipsoid to the ranges from the directional models for each structure. The anisotropy in grade variation is given by these ellipsoids. Variogram model parameters and orientation data of rotated variogram axes are shown in Table 17.2 for both deposits.

Table 17.2 Variogram Parameters for SOS and MOS Main Vein Domains

	Model	Nugget	Sills		Rotation Angles						Ranges					
		Co	C1	C2	Z1	Y1'	Z1"	Z2	Y2'	Z2"	Z1	Y1	X1	Z2	Y2	X2
SOS Main Vein	SPH	0.450	0.464	0.086	11	70	-60	2	-1	27	4	70	35	16	150	25
MOS Main Vein	SPH	0.591	0.409	--	52	-34	89	--	--	--	110	40	20	--	--	--

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

Gold in the SOS Main Vein domain displays dominantly NW-SE trending, moderate NE dipping and SE plunging structures. Ranges along strike and down the dip have short to moderate lengths. The across the dip range is quite short. The fitted model is supported by the observed geology and gold distribution. The SOS gold nugget effect is moderately high comprising 45% of the total variation.

MOS Main Vein gold distribution displays a NW-SE trending, moderately steep NE plunging structure. Ranges are somewhat longer than in the SOS model, particularly the across the dip range. The latter reflects the thicker nature of the mineralization in the MOS. The nugget effect for the MOS distribution is somewhat higher than in the SOS, comprising close to 60% of the total variation.

17.1.4 MODEL SET-UP

The block size for the Efemçukuru model was selected based on mining selectivity considerations (underground mining). It was assumed the smallest block size that could be selectively mined as ore or waste, referred to as the selective mining unit (SMU), was approximately 4 m x 4 m x 4 m. In this case the SMU grade-tonnage curves predicted by the restricted estimation process adequately represented the likely actual grade-tonnage distribution.

The assays were composited into 1 m down-hole composites. The compositing honoured the estimation domain by breaking the composites on the domain code values. The capping limits were applied to the assay data prior to compositing. The compositing process was reviewed and found to have performed as expected.

Various coding was done on the block model in preparation for grade interpolation. The block model was coded according to deposit (SOS and MOS) and estimation domain and percent inside the domain (ore percent). Up to two estimation codes and percents were permitted per block. Percent below topography was also calculated into the model blocks as was percent void or previously mined (see above). The sum of the two ore percent values represented the final ore percent for the block. That value was adjusted downward by the void percent number, where applicable.

17.1.5 Estimation

The Efemçukuru estimation plans, or sets of parameters used for estimating blocks, were designed using a philosophy of restricting the number of samples for local estimation. Eldorado has found this to be an effective method of reducing smoothing and producing estimates that match the Discrete Gaussian change-of-support model and ultimately the actual recovered grade-tonnage distributions. While local predictions based on the small number of samples are uncertain, this method can produce reliable estimates of the recovered tonnage and grade over the entire deposit, i.e., the global grade-tonnage curves from the estimations are accurate predictors of the actual grade-tonnage curves.

Modelling consisted of grade interpolation by ordinary kriging (KG) and inverse distance weighting to the third power (ID). Kriged grades were used for the Main Vein domains in both SOS and MOS, as well as the MOS Upper Splay domain. ID grades were used for the SOS Upper and Lower Splay domains. The MOS stockwork domain was interpolated by distance weighting to the second power because of the smaller data population. Only capped grades were interpolated. Nearest-neighbour (NN) grades were also interpolated for validation purposes. Blocks and composites were matched on estimation domain. All blocks straddling contacts were estimated twice with each of the composite sets on either side of the contact. The final block grade was calculated with a volume-weighted average of the two domain grades in that block. No grades were interpolated in the background areas. Where next to weakly mineralized material, default dilution grades were obtained by inspection of composite data and used as part of the mine planning work.

The search ellipsoids were oriented preferentially to the orientation of the vein in the respective domain. In the SOS, the Main Vein - South search ellipsoid orientation was 322° with a 52° NE dip. The Main Vein ellipsoid had an azimuth of 342° with a 63° NE dip whereas the Upper Splay domain used a 345° azimuth ellipsoid with a 65° NE dip. The Lower Splay domain ellipsoid orientation was 336° with a 51° NE dip. All SOS search ellipsoids were also given a -50° NW plunge. The MOS Main Vein and stockwork domains used a 320° trending, 65° NE dipping ellipsoid whereas the MOS Upper Splay ellipsoid had an orientation of 330° with a 56° NE dip.

A three-pass approach was instituted for interpolation. The first pass required a minimum of two holes from the same estimation domain, and the second and third passes allowed a single hole to place a grade estimate in a block. This approach was used to enable most blocks to receive a grade estimate within the domains. Blocks received a minimum of 4 and maximum of 3 composites from a single drill hole (for the two-hole minimum pass) in the Efemçukuru model. Maximum composite limit was 10. For the MOS stockwork domain, a minimum of 3 and maximum of 10 composites were used for the multiple hole first pass, with a limit of 2 composites from a single drill hole. All second pass runs used a minimum of 2 and a maximum of 8 composites (maximum of 2 composites from a single drill hole) and all third pass runs used a minimum of 1 and a maximum of 6 composites (maximum of 2 composites from a single drill hole).

Search ranges for the first two passes in the SOS domains were 70 m along the long axis (down the plunge direction), 35 m down the dip direction, and 5 m across the dip. The ranges for the third pass were increased to 100 m, 40 m and 8 m. Ranges for passes 1 and 2 in the MOS domains comprised 75 m along the long axis (down the dip), 30 m along the strike direction, and 15 m across the dip. The ranges for the third pass were increased to 100 m, 40 m, and 20 m. Block discretization was 3 x 3 x 3.

These parameters were based on the geological interpretation, data analyses, and variogram analyses. The number of composites used in estimating grade into a model block followed a strategy that matched composite values and model blocks sharing the same ore code or domain. The minimum and maximum number of composites were adjusted to incorporate an appropriate amount of grade smoothing. This was done by change-of-support analysis (Discrete Gaussian or Hermitian polynomial change-of-support method), as described below.

For both deposits, an outlier restriction was used to control the effects of high-grade composites in the second and third passes. The threshold grades were 35 g/t Au for SOS domains and 70 g/t Au for MOS domains. The restricted distances were set to half the original search ranges.

The bulk density was assigned to the model using averaged values from measured data. All Main Vein and Splay domains were assigned a value of 2.80. The MOS Stockwork domain was assigned a value of 2.69. Eldorado feels that this is a conservative approach and that future model updates will be able to interpolate model bulk density values by utilizing a sufficiently large set of measured data.

VALIDATION

Visual Inspection

Eldorado completed a detailed visual validation of the SOS and MOS resource models. Models were checked for proper coding of drill hole intervals and block model cells, in both section and plan. Coding was found to be properly done. Grade interpolation was examined relative to drill hole composite values by inspecting sections and plans. The checks showed good agreement between drill hole composite values and model cell values. The hard boundaries between grade shells appear to have constrained grades to their respective estimation domains. The addition of the outlier restriction values succeeded in minimizing grade smearing in regions of sparse data. Examples of representative sections and plans containing block model grades, drill hole composite values, and domain outlines are included in Appendix C for the SOS and MOS deposits.

Model Checks for Bias

Eldorado checked the block model estimates for global bias by comparing the average metal grades (with no cut-off) from the model (KG and ID) with means from nearest-neighbour estimates (the nearest-neighbour estimator declusters the data and produces a theoretically unbiased estimate of the average value when no cut-off grade is imposed and is a good basis for checking the performance of different estimation methods). Results, summarized in Table 17.3, show no problems with global bias in the estimates.

Eldorado also checked for local trends in the grade estimates (grade slice or swath checks). This was done by plotting the mean values from the nearest-neighbour estimate versus the model (kriged or ID) results for benches (in 12 m swaths) and for northings (in 20 m swaths). The model estimate should be smoother than the nearest-neighbour estimate, thus the nearest-neighbour estimate should fluctuate around the model estimate on the plots. Results for gold for SOS and MOS domains are shown in Appendix C. The two trends behave as predicted and show no significant trends in the estimates in both SOS and MOS models.

Table 17.3 Global Model Mean Grade Gold Values (g/t) by Domain

Domain	NN Estimate	ID Estimate	KG Estimate	NN vs ID %	NN vs KG %
South Ore Shoot					
Main Vein	11.78	11.53	11.69	-2.2	-0.9
Main Vein - South	9.07	8.24	8.35	-9.3	-7.8
Upper Splays	10.95	10.99	11.44	+0.4	+4.4
Lower Splay	8.61	7.82	9.11	-10.0	+5.6
Middle Ore Shoot					
Main Vein	12.30	12.50	12.57	+1.6	+2.2
Upper Splay	14.79	15.73	14.73	+6.0	-0.4
Stockwork	2.07	2.01	--	-2.7	--

Model Check for Change-of-Support

An independent check on the smoothing in the estimates was made using the Discrete Gaussian or Hermitian polynominal change-of-support method described by Journel and Huijbregts (Mining Geostatistics, Academic Press, 1978). The distribution of hypothetical block grades derived by this method is compared to the estimated model grade distribution by means of grade-tonnage curves. The grade-tonnage curves allow comparison of the histograms of the two grade distributions in a format familiar to mining. If the estimation procedure has adequately predicted grades for the selected block size, then the grade-tonnage curves should match fairly closely. If the curves diverge significantly, then there is a problem with the estimated resource.

This method uses the "declustered" distribution of composite grades from a nearest-neighbour or polygonal model to predict the distribution of grades in blocks. In this case the blocks used in the model are 4 m x 4 m x 4 m. The unadjusted polygonal model assumes much more selectivity for ore and waste than is actually possible in mining practice, since many sample-sized volumes are averaged together within a block. This means that part of the sample-sized volumes in the block may be ore (above the mining cut-off) and part may be waste. Hence, the distribution of the grade of the blocks is not likely to resemble the distribution of grades from composite samples derived from the polygonal estimate. The method assumes that the distribution of the blocks will become more symmetric as the variance of the block distribution is reduced (i.e., as the mining blocks become bigger).

The histogram for the blocks is derived from two calculations:

- the block-to-block variance (sometimes referred to in statistics as the between-block variance), which is calculated by subtracting the average value of the variogram within a block from the variance for composite samples (the sill of the variogram)
- the frequency distribution for the composite grades transformed by means of hermite polynomials (Herco: hermite correction) into a less skewed distribution with the same mean as the declustered grade distribution and with the block-to-block variance of the grades.

The distribution of hypothetical block grades derived by the Herco method is then compared to the estimated grade distribution to be validated by means of grade-tonnage curves.

The distribution of calculated 4 m x 4 m x 4 m block grades for gold in the Main Vein domains of the SOS and MOS are shown with dashed lines on the grade-tonnage curves in Figure 17.1 and Figure 17.2. The continuous lines in the figures show the grade-tonnage distribution obtained from the block estimates. The grade-tonnage predictions produced for the model show that grade and tonnage estimates are validated by the change-of-support calculations over the likely range of mining grade cut-off values (about 4 g/t Au).

Figure 17.1 Recovered Grade - Tonnage Chart, SOS, Model Gold Grades (Kriged and HERCO transformed NN)

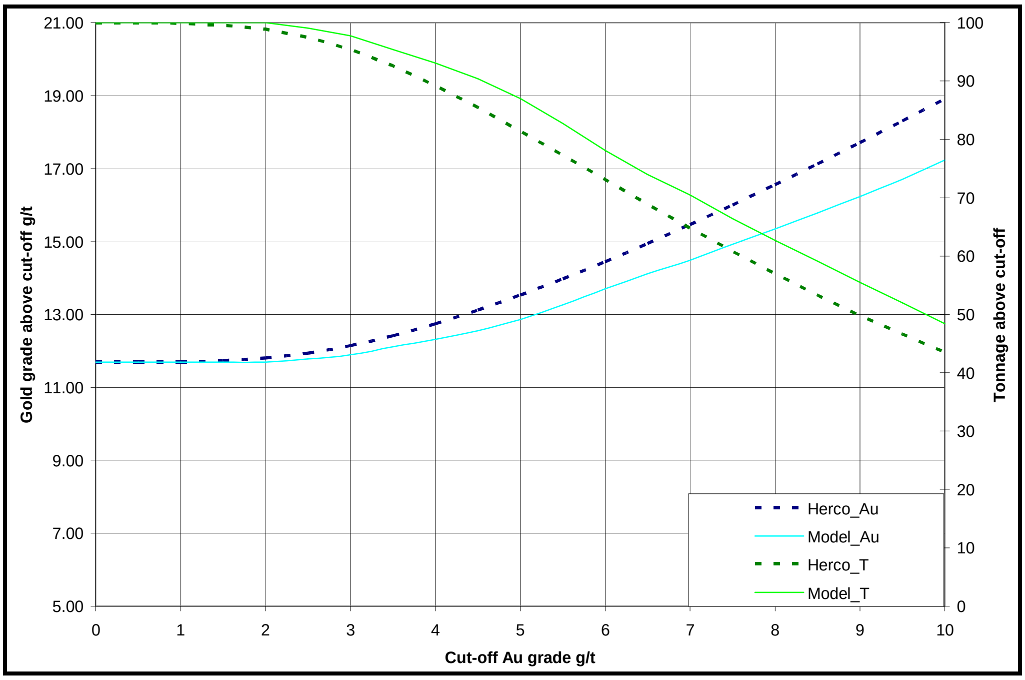
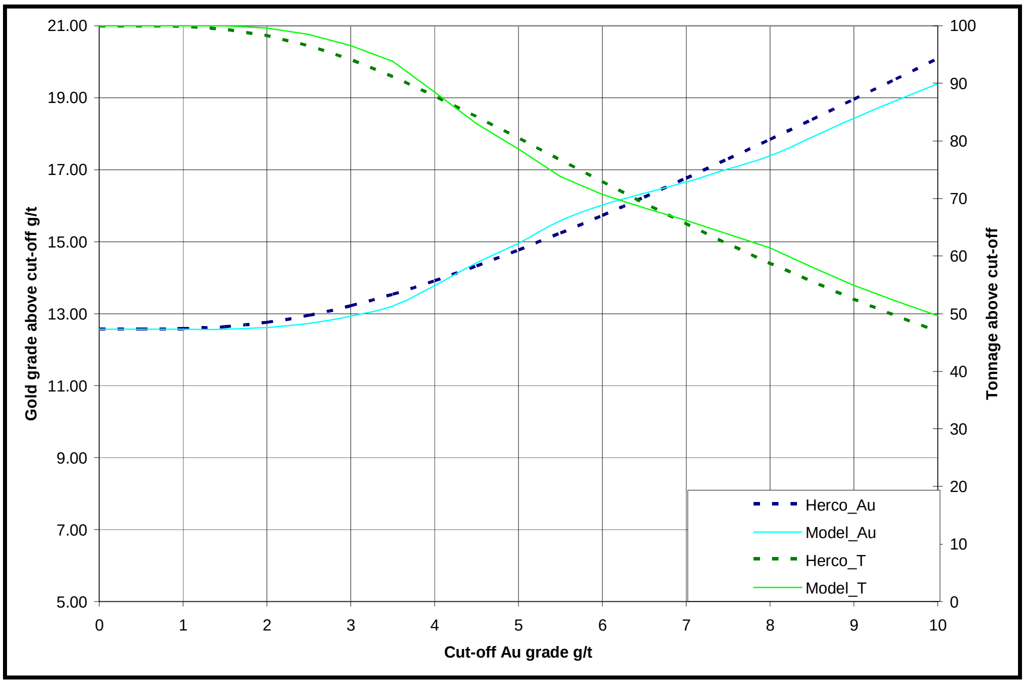


Figure 17.2 Recovered Grade - Tonnage Chart, MOS, Model Gold Grades (Kriged and HERCO transformed NN)



Histograms and Probability Plots

Histograms were constructed to show the frequency of sample grades within the mineralized domains. Both model (kriged and ID) and nearest-neighbour plots were made. The nearest-neighbour plots mimic the respective composite value distribution. The model results show the formation of a more symmetric distribution because of the smoothing effect caused by using multiple values from multiple drill holes to interpolate a model block value.

17.1.6 MINERAL RESOURCE CLASSIFICATION AND SUMMARY

The mineral resources of the Efemçukuru Project were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization of the project satisfies sufficient criteria to be classified into Measured, Indicated, and Inferred mineral resource categories.

Inspection of the model and drill hole data on plans and sections combined with spatial statistical work showed good geologic and grade continuity in areas where sample spacing was about 25 to 30 m. When taken together with all observed factors, blocks covered by this data spacing at both SOS and MOS deposits may be classified as Measured mineral resource. A three-hole rule was used, where blocks containing an estimate resulting from three or more samples from different holes (used a search ellipse equal to one half the ranges of the first pass: 35 to 40 m along the long axis, 15 to 17 m along the intermediate direction) were tagged. Then, in longitudinal section view, polygons were digitally drawn around contiguous zones of three-hole assigned blocks in both deposits. These shapes were used to classify SOS and MOS blocks as Measured mineral resources.

The Indicated Mineral Resource category is supported by the present drilling grid over most of the remaining part of the SOS and MOS deposits. The drill spacing is at a nominal 30 m on and between sections. Geologic and grade continuity is demonstrated by inspection of the model and drill hole data in plans and sections over the various zones, combined with spatial statistical work. Considering these factors, blocks covered by this data spacing may be classified as Indicated Mineral Resource at both SOS and MOS deposits. A two-hole rule was used by limiting potential blocks to those interpolated by the first pass. As in the measured resources, the pass one blocks were viewed in longitudinal section and polygons digitally drawn around contiguous zones of the two-hole assigned blocks. These polygons were used to classify SOS and MOS blocks not already assigned as Measured resources as Indicated mineral resources.

All interpolated blocks that did not meet the criteria for either Measured or Indicated mineral resource at SOS and MOS were assigned as Inferred mineral resources.

The mineralization of the Efemçukuru Project as of June 2007 is classified as Measured, Indicated, and Inferred mineral resources. The total project mineral resources are shown in Table 17.4 and are reported at a gold cut-off grade of 3.0 g/t.

Table 17.4 Efemçukuru Project Mineral Resources – June 2007

Mineral Resource Category	Tonnes	Au (g/t)	Contained Au (oz)
South Ore Shoot			
Measured	396,000	11.46	146,000
Indicated	1,654,000	10.88	579,000
Measured + Indicated	2,050,000	11.00	725,000
Inferred	586,000	8.80	166,000
Middle Ore Shoot			
Measured	754,000	15.44	374,000
Indicated	1,078,000	8.61	298,000
Measured + Indicated	1,832,000	11.42	672,000
Inferred	167,000	8.74	47,000
Total Efemçukuru Project			
Measured	1,150,000	14.07	520,000
Indicated	2,732,000	9.99	877,000
Measured + Indicated	3,882,000	11.20	1,397,000
Inferred	753,000	8.79	213,000

17.2 MINERAL RESERVE ESTIMATE

The Efemçukuru Project mineral reserve is 3.785 million diluted tonnes at an average grade of 10.04 g/t Au. The reserve estimates are included in the resource estimate. The mine cut-off grade used for the mine reserve calculation was 4.5 g/t Au. Silver was not considered as part of this study. The projected mine life is 9.4 years at the proposed production rate of 1,100 tonnes per day, with 10 months of pre-production underground mine development.

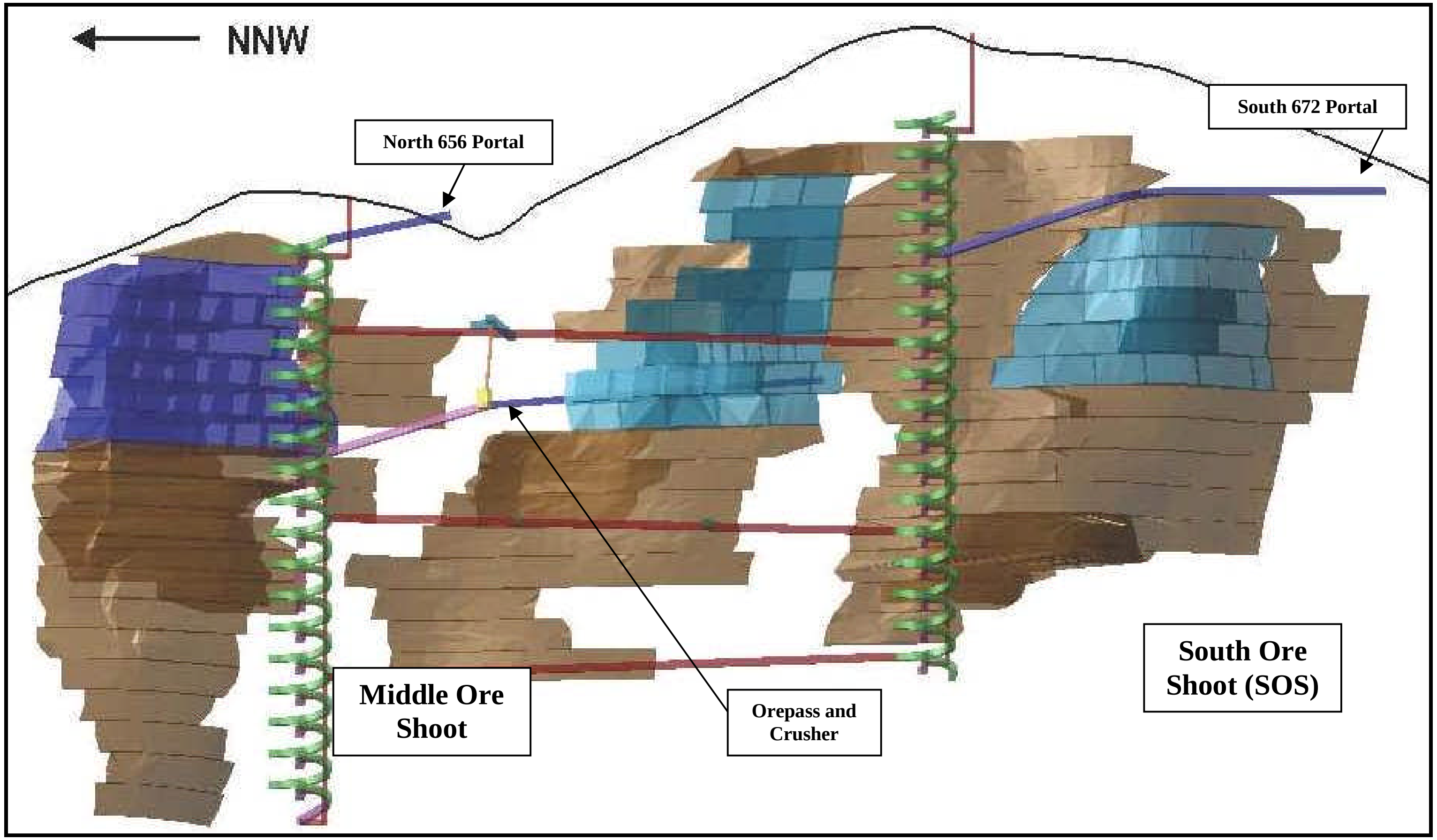
The mineral reserve is defined as the economically mineable part of a measured or indicated mineral resource. Table 17.5 outlines the diluted mineral reserve by mining method.

Table 17.5 Mineral Reserve

	Tonnes	Grade (g/t)	Gold (oz)
Mineral Reserve			
Proven Reserve	1,320,000	11.89	505,000
Probable Reserve	2,465,000	9.04	716,000
Proven and Probable Reserve	3,785,000	10.04	1,221,000
Mineral Reserve by Orebody			
Middle Ore Shoot	1,797,000	10.25	592,000
South Ore Shoot	1,988,000	9.84	629,000
Mineral Reserve by Mining Method			
Mechanized Cut-and-Fill	1,716,000	9.72	536,000
Longitudinal Longhole	825,000	8.40	223,000
Transverse Longhole	1,244,000	11.56	462,000

Figure 17.3 shows the mining blocks by type with the mine development along the strike of the orebody looking east from the footwall.

Figure 17.3 Mineral Reserve – Mining Blocks at 4.5 g/t



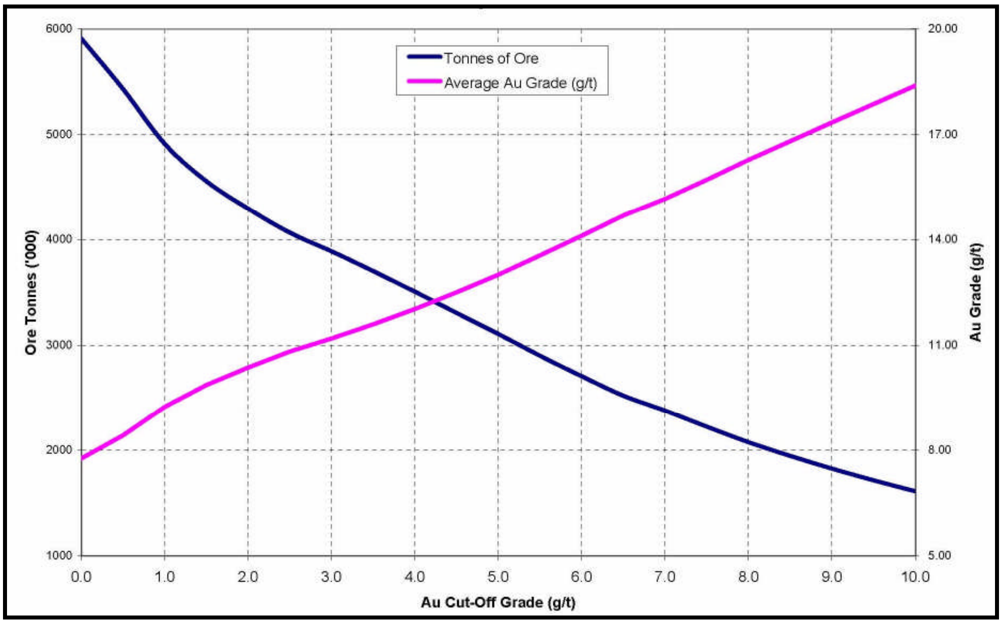
Dark Blue = transverse longhole *Light Blue = longitudinal longhole* *tan = mechanized cut-and-fill*

17.2.1 CUT-OFF GRADE

The proven and probable mineral reserve was determined using a mine cut-off grade of 4.5 g/t. The mine cut-off grade was developed from a preliminary economic evaluation based on feasibility work completed in 2006.

The grade-tonnage curve in Figure 17.4 shows the consistent relationship between cut-off grade and tonnes and grade at Efemçukuru.

Figure 17.4 Reserve Grade Tonnage Curve



The mine cut-off grades by mining method were updated at the end of the project and are shown in Table 17.6.

Table 17.6 Mine Cut-Off Grades by Mining Method

Mining Method	Gold Price (US$/oz)	Total Cash Cost (US$/t)	Cut-off Grade (g/t)
Mechanized Cut-and-fill	530	67.54	4.6
Longitudinal Longhole	530	60.85	4.2
Transverse Longhole	530	59.43	4.0

An updated mine cut-off grade of 4.3 g/t was calculated at the completion of the study with the following inputs:

- gold price of US$530/oz (3 year average gold price)
- metallurgical recovery of 86.5%
- total cash operating cost of US$63.41/t.

The updated mine cut-off grade calculation was not used to redefine the mining blocks for the following reasons:

- grade decreases quickly outside the orebody
- the practical mining width will be the full width of the geological vein in narrow areas
- the grade distribution is not regular or predictable within the orebody
- the grade control will be an assay cut-off and will impact the selectivity of the mining operation.

In particular, in the MOS where the transverse longhole method will be used, the lower cut-off grade of 4.0 g/t would have little impact due to the factors listed above. The difference between the assumed 4.5 g/t and the updated cut-off grades at the end of the study was not significant.

17.2.2 CUT-OFF GRADE CALCULATION

Cut-off grades for the deposit were calculated using the financial model. The following definitions were used:

- **Mineral Reserve** = mineral resource *x* mining recovery
- **Gross Revenue** = mineral reserve *x* metallurgical recovery *x* metal price
- **Off-site Costs** = concentrate transport, insurance, bagging, and metal processing at Kişladağ
- **On-site Direct Operating Costs** = mining, milling, and general and administrative costs
- **Total Cash Operating Cost** = off-site and on-site direct operating costs
- **Sustaining Capital Cost** = capital costs incurred after initial project capital
- **Initial Capital Cost** = capital costs required for construction and project start-up.

Cut-off grades are defined as follows:

- **Break Even Grade** = (gross revenue) *less* (total cash operation operating costs) *less* (sustaining capital) *less* (initial capital cost)
- **Mine Cut-off Grade** = (gross revenue) *less* (total cash operation operating costs)
- **Mill Cut-off Grade** = (gross revenue) *less* (off-site costs) *less* (milling and general and administrative operating costs)

A cut-off grade sensitivity analysis is shown in Table 17.7 with 3-year average, 2-year average, and current gold prices.

Table 17.7 Cut-Off Grade Sensitivity

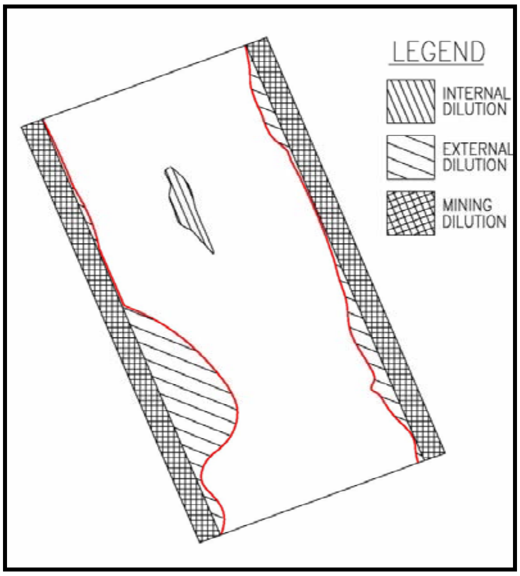
	Gold Price (US$/oz)	Total Cash Cost (US$/t)	Cut-Off Grade (g/t)
Mine Cut-Off Grade			
Design - March 2007	534	69.24	4.5
2 Year Average	585	63.41	3.9
Current	670	63.41	3.4
Mill Cut-Off Grade			
Base Case (3 Year Average)	530	35.93	2.4
2 Year Average	585	35.93	2.2
Current	670	35.93	1.9

17.2.3 DILUTION

Dilution is the ratio of waste to ore. The following three types of dilution were considered, and are shown in Figure 17.5:

- dilution within the orebody (internal dilution)
- dilution within the mining block but outside the orebody (external dilution)
- dilution outside the mining block due to mining operation (mining dilution).

Figure 17.5 Dilution by Type



Generally, geological block models can either be used with orebody solids or independently as a percent block model. The mineral resource and reserves were calculated using a percent block model. The mine design and mining blocks were created in SURPAC, then converted and imported into Gemcom Geology, Mine Planning and Production Scheduling software (GEMS).

Mining blocks were developed by slicing horizontal sections through the orebody on 4 m intervals, based on the MCF breasting of 4 m high cuts. The mining blocks were converted from SURPAC into GEMS and incorporated into the block model.

The block model blocks contained the following information:

- volume
- Au grade
- %Vein (volume of the block inside the gold mineralized shell)
- %Stope (volume of the block inside the mining block).

The dilution quantities are shown in Table 17.8.

Table 17.8 Dilution by Type

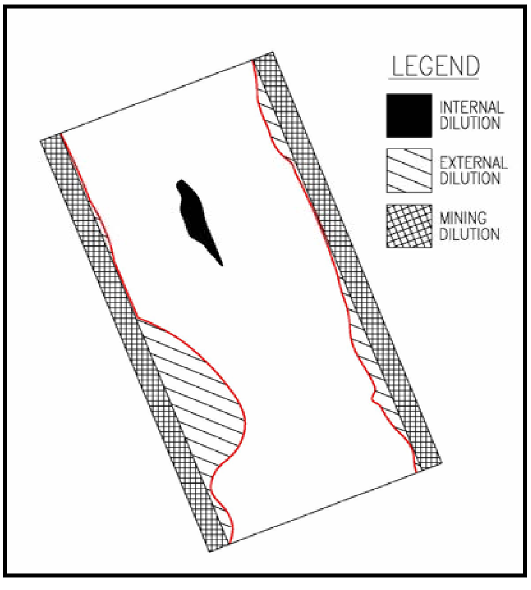
Dilution	Tonnes	Grade	Gold (oz)	% of Total Dilution
Internal	597,400	2.64	50,700	55
External	207,800	3.04	20,300	19
Mining	282,000	0.99	9,000	26

INTERNAL DILUTION

Mining blocks were developed using a mine cut-off grade of 4.5 g/t, based on the initial project economic parameters described above. The mining blocks represent the volume of material to be mined, including material both inside and outside the orebody.

Internal dilution is the volume of material inside the orebody and inside the mining block below the mine cut-off grade of 4.5 g/t. This may be referred to as incremental ore; low grade ore that will be recovered in the mining operation. The block model developed in GEMS includes internal dilution of 0.0 to 4.5 g/t. Internal dilution is shown in Figure 17.6.

Figure 17.6 Internal Dilution

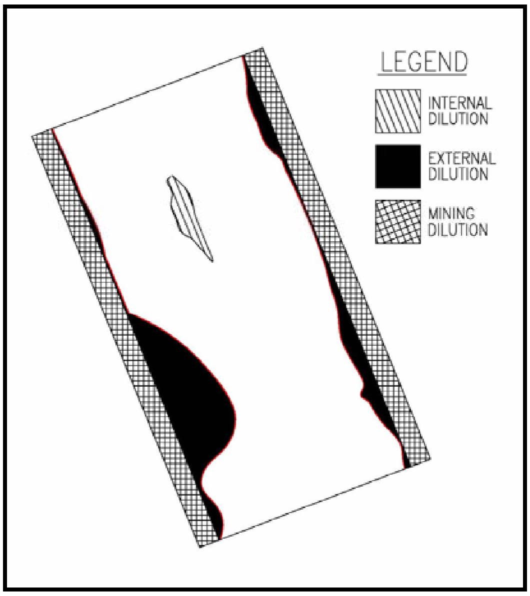


For the purpose of this study, all material within the mining block, fully diluted, will report to the mill. For example, a mining block may include blocks that range between 1.0 and 4.4 g/t. These blocks will be mined and milled. In any given mining block, a block with an average grade less than the cut-off grade, is included in the mineral reserve and is considered internal dilution.

EXTERNAL DILUTION

External dilution is the volume of material recovered inside the mining block but outside the orebody, as shown in Figure 17.7. Eldorado developed a dilution model from the drill hole data estimating gold grades for the ground adjacent to the orebody. The gold grade varies depending on the area of the mine and was added as attributes into the block model for the final mineral reserve calculation.

Figure 17.7 External Dilution



MINING DILUTION

Mining dilution is the overbreak sustained during blasting as shown in Figure 17.8. The following parameters were used based on typical dilution averages for the various mining methods:

- MCF mining method = 0.25 m each side of the mining block
- LLH and TLH mining methods = 0.5 m each side of the mining block.

The average width of each mining block was determined and the mining dilution calculated manually in Microsoft Excel. Mining dilution was calculated and added to each mining block for the final mineral reserve.

Figure 17.8 External Dilution

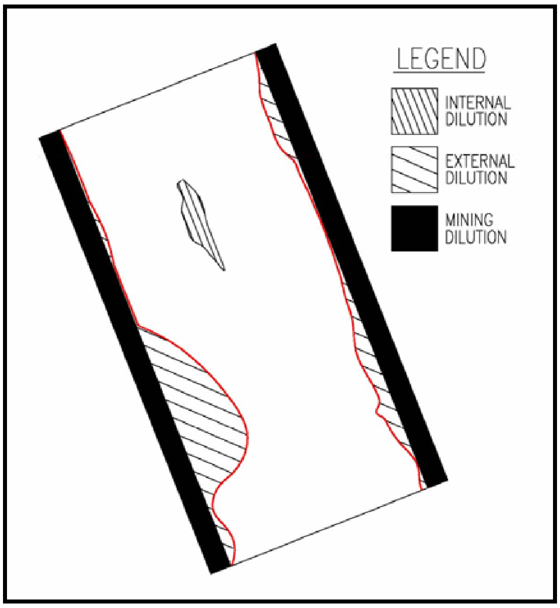
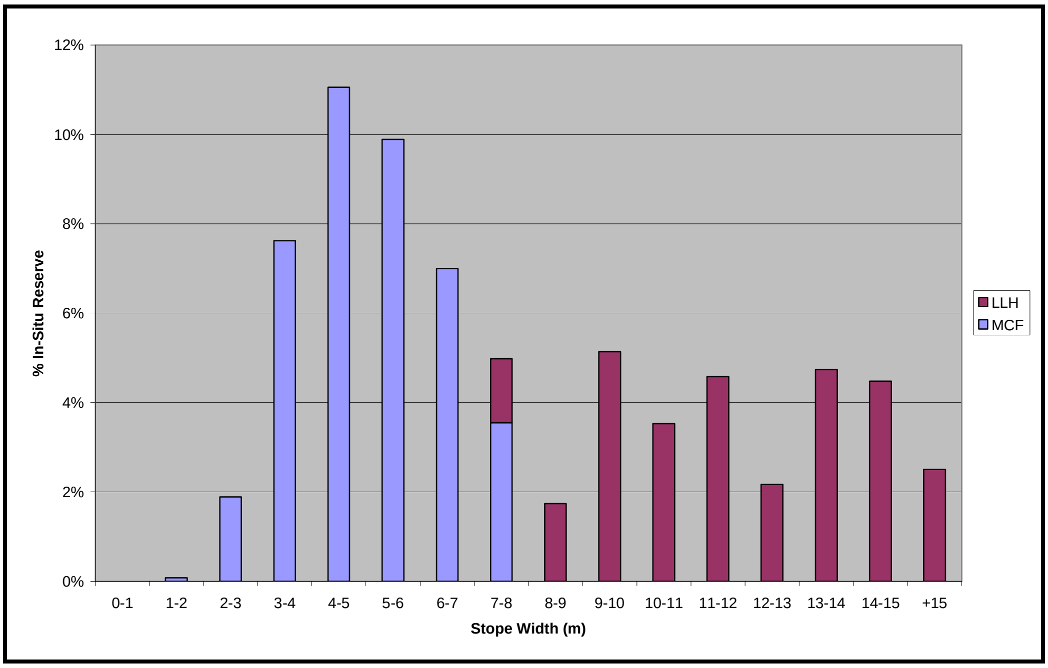


Figure 17.9 illustrates the range of mining block widths at Efemçukuru for mechanized cut-and-fill and longitudinal longhole mining. Mechanized cut-and-fill widths range from 2 m to 8 m and represent 45% of production. Transverse longhole mining widths will be greater than 15 m and represent 33% of production. Longitudinal longhole mining widths range from 8 m to 15 m and represent 21% of production.

The minimum mechanized mining width was determined to be 2 m based on the size of a Microscoop with a width of 1.4 m leaving clearance of 0.3 m each side of the machine. High orebody grade may allow mining in areas of the vein less than 2 m wide with additional dilution to bring the heading up to 2 m width.

Figure 17.9 Orebody Profile – Mining Block Width by Mining Method



The average mining widths and estimated mining dilution is shown in Table 17.9.

Table 17.9 Mining Dilution by Mining Method

Mining Method	Average Width	Average Dilution (%)	Minimum Dilution (%)	Maximum Dilution (%)
Mechanized Cut-and-fill	4.7	12	7	29
Transverse Longhole	23.5	5	3	8
Longitudinal Longhole	11.7	9	6	14

The overall mining dilution for the project is estimated at 11%. Paste backfill dilution and end wall dilution from the interaction of mining methods are included in these dilution estimates.

17.2.4 MINING RECOVERY

The measured and indicated mineral resource at 4.5 g/t cut-off contains a total of 1.3 million ounces of gold. Overall mining recovery of gold ounces is estimated at 92%.

Table 17.10 Mining Recovery

	Tonnes (Mt)	Grade (g/t)	Gold (Moz)
Measured and Indicated Resource (@ 4.5 g/t)	3.299	12.51	1.327
Proven and Probable Reserve (Diluted)	3.785	10.04	1.221

The mining recovery assumes minimal pillars will remain in the orebody. Using 100% paste back fill will enable ore to be recovered adjacent to the paste fill masses with minimum dilution.

Ore will be sterilized in the crown pillar and the conveyor pillar. The conveyor pillar is unlikely to be recovered. The potential for recovering the crown pillar at the end of mine life was not included in this study. Ore was not recovered in some ore zone widths less than 2 m.

The production schedule was based on three phases, mining bottom to top in each phase. The number of phases is related directly to the number of working places available and the number of sill matts required. For more detailed information on the mine sequencing refer to Section 19.1.4

The measured and indicated resource continues to be developed through ongoing drill programs on site. The resource in the feasibility was updated during the mine design. Mining blocks were based on the current block model. The mineral reserve reporting within the stope outlines was performed with the latest updated block model. Some newly defined measured and indicated resources are therefore not included in the mineral reserve and fall within the 92% recovery. The mine resource and reserve will be updated at the end of the current drilling program.

OLD WORKINGS

Surface evidence in the SOS indicates that ancient mining occurred at Efemçukuru with small surface openings and waste dumps. During the 1995 surface drilling program, some 22 voids were logged in the drill core from this area. It is not certain how much ore was mined in ancient times; however, Micon estimated it to be 9,000 tonnes in the SOS.

This tonnage has been updated on the basis of void modelling in the block model resulting in a revised estimate of 51,100 tonnes at 16.2 g/t. A total of 26,700 ounces of gold have been removed from the mineral resource to account for these voids. This includes a number of geological voids that have been identified within the orebody.

The voids vary between 1 m and 5 m. During the mining operation the voids would be located using delineation drilling. Voids would normally be filled with paste backfill to allow mining to continue through the orebody. Safety will be critical in the extraction of ore around the voids.

17.2.5 GRADE CONTROL

Grade control at Efemçukuru will be through assay cut-off and not visual or structural cut-offs. The mining operation will rely on an assay cut-off to classify the ore. Mine development in ore will be sampled across the face. Sampling may include channelling of the face using an electric twin blade diamond saw with chip sampling to depth as required. Delineation drilling will be required in both mine development and production work areas.

An assay laboratory will be constructed on site at the start of the project to ensure quick turn around of samples.

Longhole blasthole chips will be assayed to increase the sample database. The grade control program will be developed to supplement the exploration drill hole information, and the delineation drilling, and to provide reconciliation of modelled gold grades. The geological model will be updated regularly on the basis of new data.

Broken rock in the orebody will typically report directly to the mill. There may be exceptions in marginal areas where the direction of the face advance will be dependent on the results of the sampling; the broken rock may remain in the remuck bay until the results of the sampling are received.

A hand held X-Ray Fluorescence (XRF) Spectral Analysis may be trialled for instantaneous direction decisions at the face. It is not expected to give accurate or absolute values but may provide a relative benchmark.

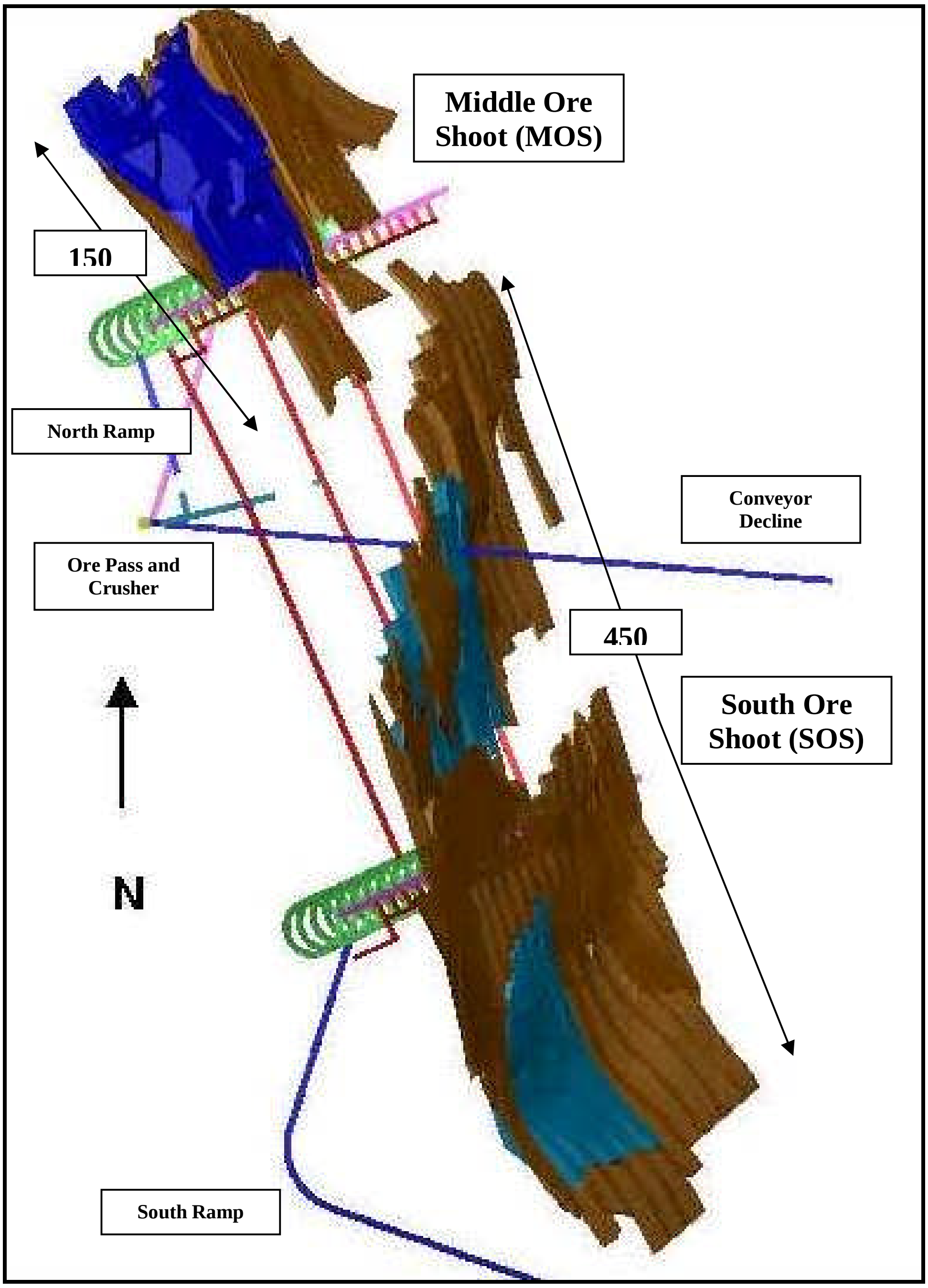
17.2.6 OREBODY PROFILE

Table 17.11 lists the dimensions and mineral reserve of each orebody including the stockwork.

Table 17.11 Orebody Profile

	MOS	SOS
Strike length (m)	150	450
Average Orebody Width (m)	27	8
Vertical Orebody Height (m)	288	288
Mineral Reserve (kt)	1,797	1,988
Gold Grade (g/t)	10.252	9.839
Tonnes per vertical metre (t/m)	6,240	6,900

Figure 17.10 Plan View of Underground Mining Blocks and Development



SECTION 18 • OTHER RELEVANT DATA AND INFORMATION

18.1 SURFACE LAYOUT

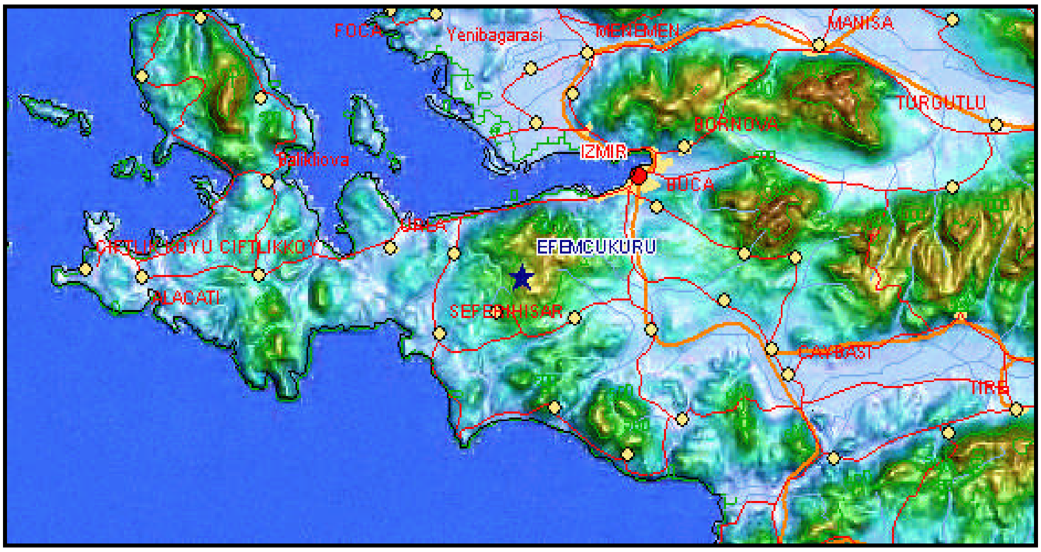
The Efemçukuru Project consists of an underground mine with a process plant and ancillary facilities on surface situated southwest of Izmir in an area easily accessible by road. The site has local access to a large sea port in Izmir, approximately a 60 km drive from site, and an international airport in Menderes, approximately a 30 km drive from site. The surface infrastructure at the Efemçukuru project site to support the mining and processing operations include the following:

- site access roads
- plant site roads
- water supply and distribution
- sewage collection and disposal
- diesel fuel storage
- power supply and distribution
- ancillary facilities.

The infrastructure has been designed to conform to locally available materials and methods of construction. Due to the project's close proximity to Izmir and surrounding towns (as shown in Figure 18.1) the infrastructure to support the operations, including power supply and site access, is readily available.

The infrastructure required to support the concentrate treatment plant to be located at the Kisladag mine site will be available from the present installation.

Figure 18.1 Efemçukuru Area Map



18.2 Site Access and Local Roads

The access to site is from Izmir through regional paved and gravel roads. The roads are narrow and winding with some isolated steep grades; however, the roads are paved and in good condition and easily passable by commercial trucks.

The size of the trucks utilized for construction and operations will be limited to loads of 4 meters high by 4.5 meters wide by 7 meters long with a maximum weight of 75 tonnes. For major equipment a logistics company and a local construction company performed separate investigations on access to the site and confirmed the maximum load size permissible over several routes. Generally, transportation will need to be limited to medium gross vehicle weight (GVW) trucks with 18 tonne maximum loads to safely access the site and precautions such as utilization of pilot vehicles will need to be utilized. Photo 18.1 and

Photo 18.2 show the regional access road to site.

Photo 18.1 Regional Access Road

Photo 18.2 Regional Access Road

18.2.1 SITE ACCESS ROAD

The site is currently accessed from the regional roads by a 2 km unimproved forestry road which will require upgrades including alignment improvements and resurfacing to meet the minimum standards of the regional roads. The road will be paved and designed to limit deviation from the existing alignment in order to limit impact on forestry lands and removal of trees. Retaining walls rather than cut slopes will be used in order to limit disturbance. The access road design will be completed to a maximum 12% grade and 7 m width with turnouts to suit 18 tonne GVW trucks, speed will be limited on the access road to 30 km/h.

A local construction company has complete an additional survey along the access road; preliminary designs were completed for the feasibility study.

Photo 18.3 shows the current site forestry access road.

Photo 18.3 Current Site Forestry Access Road

During operations, concentration will be bagged on site and transported to Kişladağ in 18 tonne GVW trucks, suitable for the narrow access roads. The concentrate will be unloaded at Kişladağ using a forklift and stored in a new warehouse prior to processing. A local freight company will provide the vehicles and operators for the haulage. A convoy of three 18 tonne trucks will haul concentrate to Kişladağ twice a day. A pilot vehicle will lead the convoy and provide warning for oncoming vehicles; the convoys will leave the plant at the same time each day so locals are aware. The trucks will be utilized to back haul reagents and spares for the operation. Reagents will be bulk stored at the Menderes warehouse reducing the warehousing requirements at site.

18.3 SITE LAYOUT

Reference: Appendix D, Dwg A0-10-001 and A0-10-002

The Efemçukuru project site consists of the concentrator process plant, ancillary buildings, tailings filtration/backfill plant at the North 656 Portal; the fitered tailings storage and development rock dumps in the valley below the South 672 Portal. The site layout was designed to limit the disturbed footprint and the amount of trees removed and to blend in with the surroundings. A range fence primarily for control of domestic and wild animals surrounds the entire site. The site excavation is intended to balance the cut and fill quantities to limit aggregates that would need to be hauled to site or rock needing to be hauled off the site. The cut-and-fill quantities are approximately 60,000m^3 and 42,000m^3 respectively.

The plant site will be located on the west side of the Kokarpinar valley at an elevation of 605 masl. It consists of the ore storage bins, concentrator building, water treatment plant, and ancillary facilities. The ore bins are fed from the underground crusher by a 800mm wide belt conveyor which daylights at elevation 619 masl at a small pad with and access road to allow service vehicles. The site has been designed to limit the disturbed footprint by terracing the facilities into the topography to avoid a large excavation. The location of the plant site lends itself to the utilization of the existing forestry road for access. The plant site drains to the catchment pond located at the north of the site. Water from the catchment pond will be pumped to the water treatment plant.

Site roads have been designed to follow the alignment of the existing exploration roads where possible in order to minimize site disturbance and removal of trees. The roads allow access from the plant to the filtration plant and the North 656 Portal; the development rock dump, the South 676 Portal, and filtered tailings storage. A haul road allows access from the filtration plant to the filtered tailings storage area. Internal roads will be sealed for dust control.

Access to the firewater tanks and ventilation raises will be by four-wheel drive service vehicles along upgraded exploration roads.

Photo 18.4 shows the area for the development rock dump. The South 676 Portal will be located on the north side hill above the valley (right side of photo), the sedimentation pond at the toe of the valley (forefront of photo) with the development rock filling the valley to the narrow section. Filtered tailings storage will be located further up the valley (not shown in photo).

Photo 18.5 is a view of the valley where the water treatment plant will be located looking north towards the toe of the rock dump and towards the process plant site.

Photo 18.4 View Looking West at Future Rock Dump Area

Photo 18.5 View Looking North Towards the Plant Site

Photo 18.6 is located near the top of the hillside above the South Ore Shoot viewing southeast towards the Valley showing the terrain and vegetation typical of the mine site.

Photo 18.7 looks west towards the tailing filter/backfill plant and Portal 656 located in the higher up the valley with access by a road built which will be built on the southern side of the valley.

Photo 18.6 View Looking East Towards Rock Dump from South 676 Portal

Photo 18.7 View Looking West Towards Tailings Dump

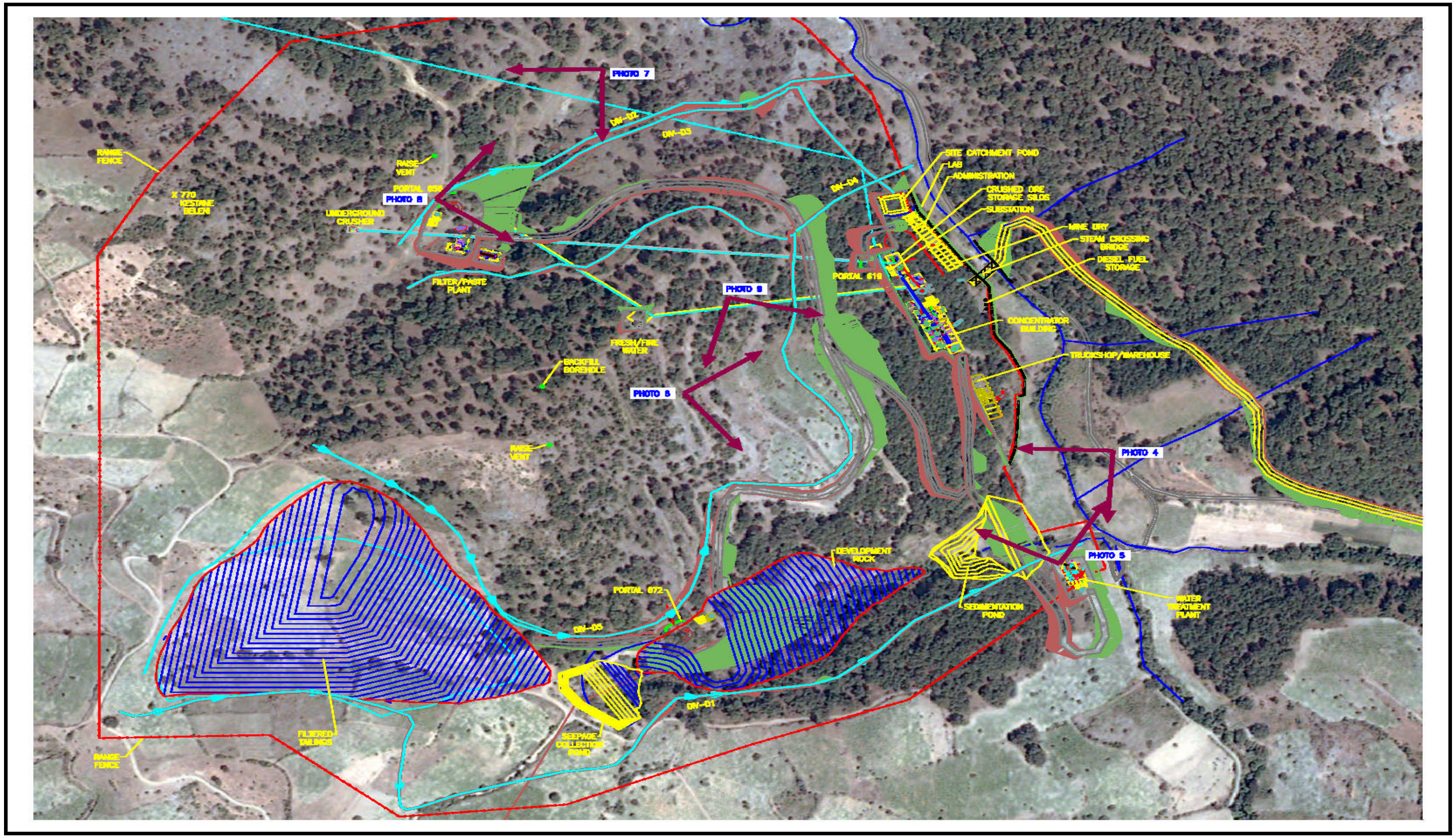
The tailing filter/backfill plant will be located in the valley beside the access to the North 656 Portal in an area near the location of Photo 18.8 (looking east). The exploration road on the right side of the photo will be upgraded for access to the filtration plant and adit. A large quantity of cut-and-fill will be required in this area to build a pad for the plant foundation, mine access laydown area, and allowances for turning the radius of haulage trucks and mining equipment.

Photo 18.9 is taken on the hillside looking south into the Kokarpinar Valley and future site location.

Photo 18.8 View Looking East Towards Filtration Plant and North 656 Portal

Photo 18.9 View Looking Northwest towards Plant along the Main Access Road

Figure 18.2 Viewpoint and Range of View Photos 18.4 to 18.9



18.3.1 FIRE/FRESH WATER SUPPLY STORAGE AND DISTRIBUTION

The plant site will require nine cubic metres of water per hour, which will be provided from mine dewatering, site collection and recycled water from the process. Mine water will be pumped from underground to the sedimentation pond located below the development rock dump. The sedimentation pond will store mine water and contact water collected by the diversion ditches around the perimeter of the project site footprint. Mine and contact water stored in the sedimentation pond will be treated and used in the process with the excess water treated, tested, and released back to the environment.

The water treatment plant located on the southeast of the plant site will supply fresh-treated water to a fire and fresh water storage tank located on the hill above the process plant which will then be distributed by gravity to the process plant. The fresh water tank will serve double duty as storage of both fire and fresh water. Fire and fresh water reservoirs inside the tank will be separated with the use of a standpipe inside the tank to draw off the fresh water off the top of the fresh water standpipe assuring storage for firewater is maintained.

The firewater distribution system will consist of a dedicated buried firewater main and hydrant system for the plant site and ancillary buildings. Hose cabinets will be placed within the process plant and ancillary facilities, supplemented by portable fire extinguishers in all facilities. Hose stations located at 50 m intervals and automatic sprinklers over the drive will protect underground conveyor. Ancillary buildings will be provided with automatic wet sprinkler systems throughout.

The hypochlorinator and potable water storage tank will be located at the mill site. The potable water tank has a capacity of 15 m^3. Buried piping will distribute potable water to the ancillary facilities. The potable water is suitable for general use in the facilities but not for consumption. A tanker truck will supply drinking water to a storage tank located at the process plant.

Emergency showers and eyewash stations have been situated throughout the process building.

18.3.2 DIESEL FUEL STORAGE AND DISTRIBUTION

Diesel fuel requirements for the mining equipment and process and ancillary facilities will be supplied from a buried diesel fuel storage tank located near the truck shop. The diesel fuel storage tank will have a capacity of 10,000 L sufficient for approximately two days of operation. Diesel storage will consist of an underground tank and will be complete with loading and dispensing equipment conforming to Turkish regulations. A fuel dedicated service truck will transport diesel to the underground equipment.

18.3.3 Sewage Collection and Treatment

The sewage disposal system will comprise of a buried gravity collection system from the process and ancillary facilities to the sewage treatment plant located at the Southeast of the property. The plant site layout allows for gravity sewage collection throughout.

The sewage treatment plant will be a pre-packaged Rotating Biological Contactor (RBC). The plant will be manufactured off site and containerized for simple connection to the collection system on site. Once treated, the sewage treatment plant effluent will be discharged into the environment in accordance with the requirements of the Environmental Impact Assessment.

18.3.4 Waste Disposal

Solid waste from the kitchen and non hazardous waste from operations will be hauled off site for disposal.

Hazardous waste will be collected and shipped off site for their disposal into approved facilities.

18.4 Power Supply and Electrical Distribution

18.4.1 General

The electrical system has been sized to take into account the process loads, water treatment plant loads, mining loads, and the ancillary loads, such as the workshop/warehouse mine dry/canteen and administration building. The estimate load list is included in Table 18.1. Spare capacity is available within the electrical distribution system to allow for limited future expansion of the process plant.

Table 18.1 Estimated Load List – Efemçukuru

Area	Total Rating (kW)	Connected Load (kW)	Standby Load (kW)	Operating Load (kW)	Annual Operating (MWh/a)
Process Plant Area					
C1 – Primary Crushing	170	170	0	143	957
D0 – Crushed Ore Storage & Reclaim	45	45	0	38	332
E0 – Air Supply & Distribution	134	104	30	88	767
E1 – Grinding & Classification	1,603	1,569	34	1,458	12,775
E2 – Gravity Concentration	41	39	2	33	288
E3 – Pebble Crushing	147	147	0	124	1,082
E4 – Flotation	265	164	100	134	1,173
E5 – Concentrate Dewatering & Loadout	157	114	43	85	743
E6 – Reagents	51	35	16	21	188
E7 – Gold Room	172	168	4	84	735
F2 – Tailings Filtrations & Paste	139	72	67	59	516
K4 – Water Supply & Distribution	514	469	45	245	2,146
Total Process Plant	**3,437**	**3,096**	**341**	**2,512**	**21,702**
Mine Area					
B1 – Mining Equipment	3,956	3,956	0	2,004	17,552
J3 – Truck Shop	44	44	0	23	203
Total Mine	**4,000**	**4,000**	**0**	**2,027**	**17,754**
Site Area					
G1 – Water Treatment	77	34	44	22	194
F1 – Tailings Thickening	22	22	0	10	81
K3 – Fresh Water	63	34	30	28	247
Total Site	**163**	**89**	**74**	**60**	**522**
Total Efemçukuru	**7,600**	**7,185**	**415**	**4,598**	**39,978**

18.4.2 POWER SUPPLY

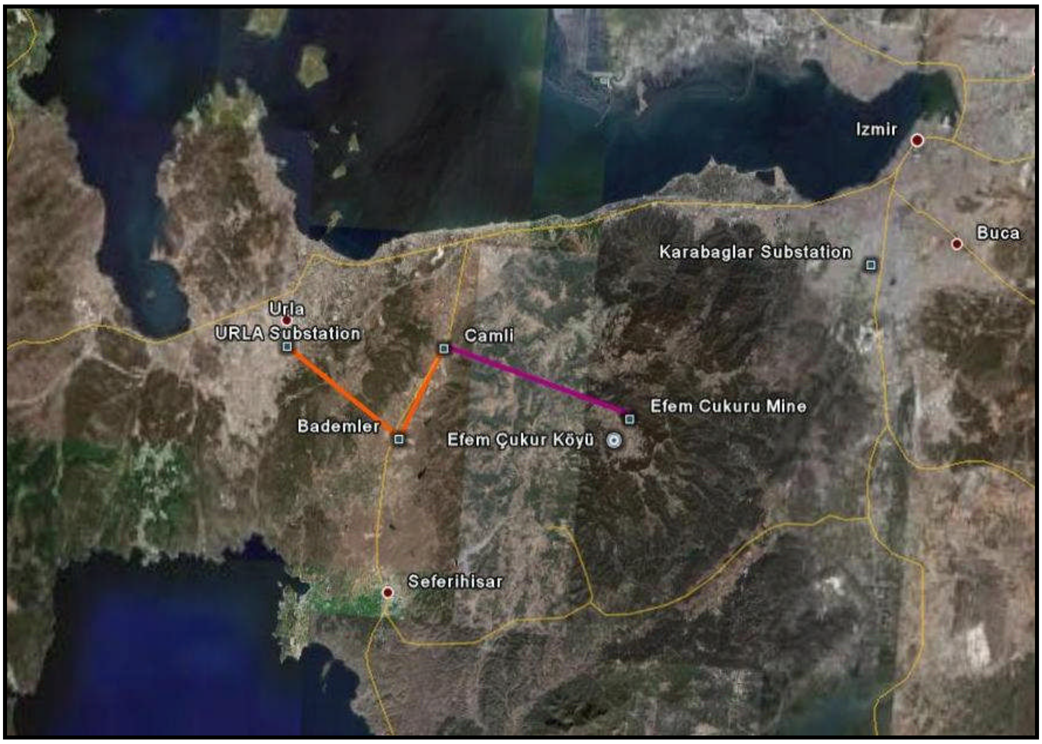
The existing power supply to the site is via an overhead line from Efemçukuru village to the well and pumphouse located at the plant site. The existing line is not adequate to serve operations; however, with some repairs and modifications to the alignment the existing line can be commissioned for use during construction.

The incoming power supply to serve operations will be a new 34.5 kV, 50 Hz overhead pole line from the National Grid to the owner-supplied substation at site. The new power line will be commissioned and turned over to Türkiye Elektrik Dağıtım

A.Ş. (Tedaş), the regional power authority; Tedaş will assume ownership of the power line and maintain the system up to the substation at the mine site.

Options for the routing of the new power line have been evaluated by a local electrical consultant who has identified three supply options to support the feasibility study. For the purpose of the study, the selected option is a new power line originating from the substation located in the district of Urla, which has the required capacity and is located approximately 20 km to the east of the plant site. As shown in Figure 18.3, the proposed route will require approximately 200 m of underground cable and 20 km of overhead transmission line.

Figure 18.3 Power Line



18.4.3 Site Power Distribution

Reference: Appendix D, Dwg A0-10-001

The site layout (Dwg A0-10-001) shows the location of the incoming power line, main substation, and site power distribution. The following description of the plant site electrical distribution system is in accordance with the site layout, electrical single line diagrams.

The new incoming 34.5 kV overhead power line from the national grid will terminate at the main substation located at the north of the plant site. The location of the substation will remove the need to route overhead utility lines above the vehicular traffic areas of the main plant. The main substation will consist of a main disconnect, metering facilities, and 6 x 34.5 kV feeder positions in a walk-in type outdoor rated enclosure. Distribution from the main substation location will include the following feeders:

- a 34.5 kV overhead power line extending west to provide power to the fire/fresh water tank area, the filter plant area (electrical room – ER4) and the north adit area (ER5) which will supply underground mining equipment and services
- a 34.5 kV feeder cable installed along the primary crusher conveyor structure to underground distribution which serves the primary crusher (ER1)
- a 34.5 kV feeder cable to the crushed ore storage electrical room (ER2)
- a 34.5 kV feeder cable to serve the 4 kV grinding area motor loads in the concentrator building (ER3)
- a 34.5 kV feeder cable to serve the low voltage loads in the concentrator building (ER3) and buildings adjacent to the concentrator building such as the administration, mine dry, lab and gatehouse buildings
- a 34.5 kV overhead power line extending south and west to provide power to the ancillary facility loads including:
  - the shop/warehouse, fuel storage and existing south pump house facilities (ER6)
  - the water treatment plant (ER7)
  - the South 676 Portal and underground equipment (ER8).

All surface electrical rooms on site will be pre-manufactured and shipped pre-assembled and tested. This will minimize the amount of effort needed for on-site wiring and installation, and will facilitate pre-check out of much of the low voltage electrical room equipment. These rooms will contain area low voltage motor control, control system cabinets, HVAC, lighting, and provision for power correction equipment where required. The electrical rooms will be installed on concrete supports where appropriate and adjacent to structures where there are concentrations of electrical equipment needing power and control.

Underground 400 V cables will exit from ER3 location to provide power to the power distribution centers and motor control centers of the ancillary buildings. Motor control centers will be complete with motor starters, contactors, disconnect switches, transformers, panels, circuit breakers, and fuses.

A standby diesel generator rated at 250 kVA, 400 V complete with radiator cooling, exhaust system and muffler, fuel tank, transfer pump, and auto transfer switch for auto-starting on main power failure will be located adjacent to the electrical room. The diesel generator will be supplied for operation of emergency lighting and essential drives in the event of a power outage in the immediate concentrator building areas. Allowance is made for two 1000 kVA, 400 V emergency standby generators located at each adit site to drive vital underground equipment.

18.5 Electrical Equipment and Materials

18.5.1 Equipment and Materials

All electrical equipment will be rated for a minimum elevation of 750 masl and an ambient temperature range of 0°C to 40°C and will be certified by Conformité Européenne (CE).

General Power and Lighting

Power outlets will consist of 15 A, 220 V, 1 Phase, 50 Hz plug in receptacles for small tools, and 60 A, 400 V, 3 Phase, 50 Hz disconnect and receptacle for welders, and others. The lighting will comprise of the following types of fixtures:

- high pressure sodium (HPS) fixtures, sized as required for lighting of the mill areas
- fluorescent fixtures for office and electrical rooms
- HPS flood light fixtures mounted on the buildings will be supplied for yard lighting.

18.5.2 Power and Control Cables

Distribution cables will be aluminium-armoured polyvinyl chloride (PVC) jacketed, cross-linked polyethylene insulated conductors.

Cables will run from the electrical room to the electrical equipment and devices, mounted on cable tray/racking throughout the mill building and direct buried between buildings, unless otherwise noted.

18.5.3 Communications

A communications network will be established using satellite technology for voice, fax, and Internet service.

18.6 ANCILLARY FACILITIES

The ancillary facilities have been designed utilizing concrete panels and blockwork as far as practical to maximize the use of locally available materials and methods and blend in with the local architecture. The following is a general description of the ancillary facilities included on the Efemçukuru site.

18.6.1 PROCESS BUILDINGS

The process buildings including the concentrator building and filtration plant building will be structural steel, stick built buildings with concrete panel siding. The reagent area, bag storage area, electrical, and mechanical rooms and offices will be located in a two-storey structure annexed to the main process building.

The concentrator building will have a 20 tonne overhead crane servicing the grinding and flotation areas, and a 7.5 tonne overhead crane servicing the reagent and concentrate storage areas.

The filtration plant will be located beside the North 656 Portal and will be built onto the side of the hill to utilize the topography.

18.6.2 LABORATORY

A laboratory will be located in a separate building at the south end of the mill building. It will be equipped to perform daily analysis of mine and process samples including ICP and fire assaying. The laboratory will be a single storey concrete frame/block wall structure of 150 m^2.

18.6.3 WORKSHOP AND WAREHOUSE

The workshop and warehouse will be a pre-fabricated concrete/blockwork building. The building has been designed to provide facilities for maintenance and repair.

The workshop and warehouse include two indoor truck bays and an outdoor wash bay. Waste oil storage will be provided for removal and disposal to an approved facility. Also included are a machine shop, welding shop, and electrical/instrumentation work area. Maintenance and planning personnel will have offices located on the second floor.

Indoor storage of 144 m^2 has been allowed in the warehouse area and an outdoor fenced secure storage is included.

Warehousing will be provided in Menderes for temporary concentrate storage and reagent and spares storage.

18.6.4 ADMINISTRATION BUILDING

The administration building will be a two-storey building of concrete and blockwork construction; and will house 32 staff.

The administration building is approximately 500 m^2 on two levels, including space for engineering, geology, and administration personnel. The general manager, mine manager, mill superintendent, and security chief will also have offices in this building.

18.6.5 MINE DRY AND CANTEEN

The mine dry and canteen will be of two-storey concrete and blockwork construction.

The mine dry and canteen are approximately 540 m^2 on two levels. The mine dry is equipped with lockers and baskets for 80 miners. This includes offices for the mine captain, shift supervisor, and safety officer as well as a lamp room and first aid room on the ground floor. Also included on the ground floor are the clean and dirty dry areas and a security area. The upper floor contains the lunchroom and kitchen.

18.6.6 GATEHOUSE

The main gatehouse will be a simple one storey blockwork building located at the access to the plant site. The gatehouse will include a reception area and space for safety and security personnel. A covered area adjacent to the gatehouse will serve for emergency vehicle parking.

18.6.7 PERSONNEL ACCOMMODATION AND TRANSPORTATION

Personnel for construction and operations will be from Izmir and the surrounding communities. No temporary camp for construction or permanent camp will be installed at site. It is assumed personnel will be hired and trained from the labour pool living in Izmir and local area; no subsistence allowance will be required.

The contractors will transport personnel for construction to site and Eldorado will provide transportation for operations. An allowance has been made for the contracting of three 20-seat buses during operations to transport people from Izmir and Menderes.

18.7 KIŞLADAĞ CONCENTRATE PROCESS PLANT

18.7.1 KIŞLADAĞ FACILITES KIŞLADAĞ

Reference: Appendix D, Dwg P0-10-001

The infrastructure for the concentrate process plant at Kişladağ includes the following:

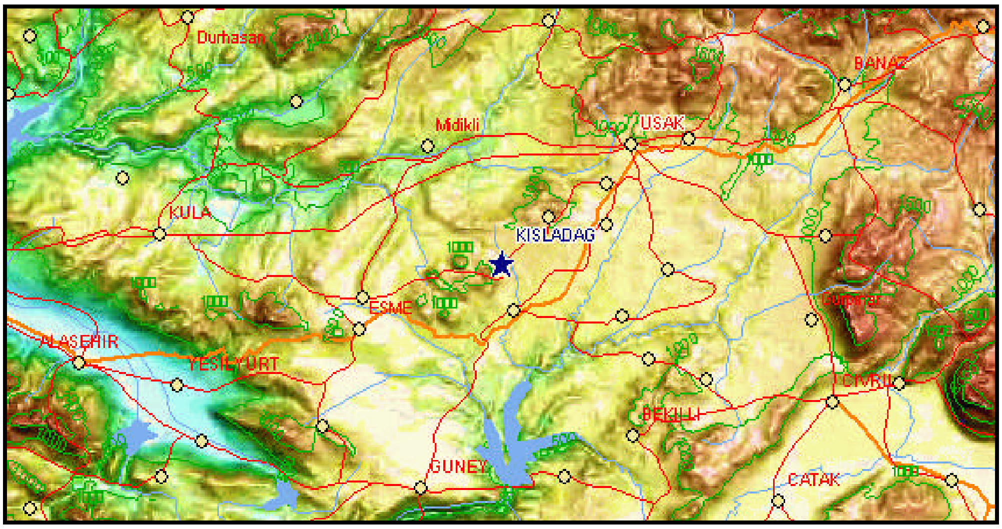
- plant site road and site preparations
- water supply and distribution
- sewage collection and disposal
- power supply and distribution.

The existing infrastructure for Eldorado's Kişladağ mine and process facility will support all the needed requirements of the new concentrate processing facility – power supply, water, sewage, and site access. Dwg. P0-10-001 illustrates the site layout.

18.7.2 SITE ROADS AND SITE PREPARATIONS

The proposed concentrate process plant will be located west of the Kişladağ's tertiary and secondary crushing and screening complex. The site will be leveled and cleared of organic debris with a stable base for construction. No road improvements will be required for access roads to site. Figure 18.4 shows the Kişladağ site located between Esme and Uşak. The Kişladağ mine is located approximately 180 km from the Efemçukuru site by road.

Figure 18.4 Kişladağ Area Map



18.7.3 WATER SUPPLY

A water main is located 125 m south of the proposed site. Water requirements are minimal, estimated at 0.6 cubic meter per hour which will be used in the process and recycled into Kişladağ's leaching process.

18.7.4 SEWAGE COLLECTION AND DISPOSAL

Sewage will be collected and added to Kişladağ's existing system for treatment in the RBC. The added volume from the new facility can be handled in the existing RBC unit. The unit's total capacity was verified and compared with Kişladağ's current and future operational requirements.

18.7.5 POWER SUPPLY AND DISTRIBUTION

The existing power transmission lines run beside the road 50 m east of the proposed site. A pad mounted load break switch will be located next to the existing power lines with underground cables feeding a transformer adjacent to the electrical room in the new process plant. Table 18.2 outlines the power requirements of the new facility.

Table 18.2 Estimated Load List Kişladağ

Area	Total Rating (kW)	Connected Load (kW)	Standby Load (kW)	Operating Load (kW)	Annual Operating (MWh/a)
Kişladağ Process Plant Area					
P1 – Concentrate Rehandling & Milling	604	552	30	510	4,467
P2 – Cyanide Leaching & Leach Residue Dewatering	651	630	6	521	4,565
P3 – Gold Room	369	355	0	253	2,217
P4 – Reagents	53	45	11	27	235
P5 – Services	141	130	21	81	711
Total Kişladağ	**1,819**	**1,712**	**67**	**1,392**	**12,195**

Currently Kişladağ's power requirement is 11.4 MW, the concentrate process plant and associated equipment will require 1.39 MW adding approximately 10% to Kişladağ's load.

18.7.6 ANCILLARY FACILITIES

Labour requirements for the concentrate process plant are minimal. Existing facilities will be used for maintenance, and metallurgical testing. The site cafeteria will be utilized and security functions will be run from the existing office. An existing modular office on site will be refurbished to house the process foreman and clerk. Costs have been factored into the study for the additional utilization of Kişladağ's existing site services.

Concentrate will be stored in a new 20 x 30 m sprung structure capable of handling 1000 two-tonne bags and reagents.

18.8 SOCIOECONOMIC CONSIDERATIONS

Socioeconomic impacts of the Efemçukuru Project on the surrounding community were addressed in Eldorado's comprehensive EIA report, prepared by Encon. The report deals with the socioeconomic concerns including culture; archaeological structures; impact on income sources; physical impact including noise, vibration, and visual nuisance; and immigration and emigration. The report also proposed a management plan to prevent or mitigate any impacts of the project including, socioeconomy, public health, and public safety. On the basis of the mitigation measures proposed to alleviate impacts, the project received an Environmental Positive Certificate from the Ministry of Environment and Forestry (MoEF) in September 2005.

Eldorado plans to continue the high level of attention given to the influence the planned operation will have on the local population. The successful programs implemented at the Kişladağ mine will be used as templates to insure the community will see long term benefit from the presence of the mine and that a sustainable economy can be developed. The Company has embarked on agricultural projects associated with land within the project boundary as an initial step in this direction and will continue to build on these efforts.

18.9 RECLAMATION AND CLOSURE

The potential impacts of the mine operation at Efemçukuru on the physical, biological and sociological environment around the project site have been addressed in the Environmental Impact Assessment Report (by Encon). Mitigation measures have also been proposed in the EIA to deal with these impacts during both the operation of the mine and at closure. A closure strategy has been developed in the EIA which will be compiled into a preliminary closure plan for Efemçukuru to be issued prior to start up of operations and subsequently revised on a regular basis prior to decommissioning and closure of the mine.

An evaluation of the Efemçukuru site for reclamation costing has been prepared by The Mines Group Inc.

18.9.1 LAND DISTURBANCE

The project site encompasses approximately 40 hectares of disturbed land including all access and site roads, process plant area, tailings and development rock dumps with associated water collection ponds, water treatment plant area, ditches and collection ponds, and all other areas within the footprint of the facilities and infrastructure noted above.

18.9.2 RECLAMATION AND CLOSURE ACTIVITIES

GOALS

The primary goal of the reclamation plan for the mine is to remove or mitigate any short or long term hazards to the environment posed by the operation and return the site to a state as close to original condition as possible. Maximum consideration will be given to closure issues during the design stage of the project to reduce impacts of the plant and underground operations on long term closure of the site.

The use of new proven technology will be promoted to advance reclamation efforts to a successful early conclusion.

Concurrent reclamation will provide the opportunity to refine the closure plan while reducing short term impacts in sensitive areas such as tailings and rock disposal sites.

A successful transfer of stewardship is sought between Eldorado and local communities to insure the reclaimed land will meet the needs of future generations.

PLANNING & STRATEGIES

Planning for rehabilitation of the surface site will commence during the construction phase of the project. All productive topsoil will be stripped as part of the construction activities and will be stored in a dedicated site north of the tailings dump. Volumes of cut and fill for the project site will be balanced as well as possible to reduce the amount of materials handling required at closure of the site and access roads. Where possible modular construction will be used to again reduce the amount of disturbance to the site and promote ease of salvage and reclamation.

Concurrent reclamation is a strategic approach to reduce impacts on the environment during operation and accelerate the return of the site to a usable condition as quickly as possible at the end of the mine life.

Reclamation and Closure Activities

Mine Rock Storage Area

The permanent storage facility designated for the mine development rock covers an area of approximately 2.3 hectares. The design of the rock dump has focused on mitigation of potential Acid Rock Drainage (ARD) during operations and after closure of the mine. As part of this design a multi-layered soil cover will be placed over exposed rock in the dump. The "store and release" cover will be comprised of a 1.0 m thick layer of well-graded soil material underlain by a 0.3 m thick drain layer. The purpose of the cover is to limit the amount of seepage of meteoric water through the dump and into the containment system below. The cover will be composed of materials with high moisture loading characteristics to retain rain water. Suitable vegetation will be propagated to promote transpiration of the trapped moisture. Prior to capping the dump, slopes will be contoured to reduce erosion and promote rapid run off of rain water. Details of the cover design for the rock dump and tailings storage area are presented in Section 19.1.13.

Tailings Storage Area

The tailings impound area at Efemçukuru will provide permanent storage of filtered mill tailings. The material will be placed in the storage area after mechanical filtration to remove excess water. The "Dry Stack" concept of tailings storage eliminates the need for downstream storage dams and resulting water and slimes handling installations. Direct placement of the tailings on the dump and mechanical compaction presents the opportunity to create a stable storage dump similar to the rock dump. Due to the low reactivity of the tailings material and low moisture content in the dump, the potential for acid drainage from the dump is extremely low.

Closure treatment of the tailings dump will be similar to that of the rock dump. Based on the multilayer cover system described above, the site, covering an area of 6.2 hectares, will be contoured and capped to control migration of water and provide a soil base to establish plant growth.

Plant Site and Mine Portals

All buildings, equipment and structures will be removed from the plant site and the ground resloped to conform to the natural contours of the area before being reclaimed with topsoil and vegetation. Prior to demolition of buildings the local community and local government will be approached to discuss alternative uses of the structures. The access road to the site and site roads will be removed and planted. The local Forestry Department will be consulted on potential future use of these roads prior to removal.

The three mine portals and two ventilation raises will be plugged and capped with concrete to restrict access. These structures will be designed to withstand seismic events and pressures from underground water buildup.

Underground Workings

As an ongoing part of mining paste backfill will be placed in the underground workings including stopes and abandoned access drifts. Following mining all underground equipment, air and water services as well as all consumable supplies will be removed. No materials with the potential to contaminate the mine water will be left. The mine dewatering pumps will be removed and the workings allowed to flood up to a stable ground water level. All flowing drill holes will be plugged to restrict outflow from the workings.

Water Treatment

During the operating life of the mine, surface runoff and seepage water will be collected and treated on site before discharge. After closure, ground water in the mine workings will be allowed to build up to normal levels. All boreholes and openings into the workings will be sealed. During operations seepage water from the tailings and rock dump will be collected and treated for recirculation and process use. After closure a passive bioremediation system will be installed to deal with the seepage. A series of groundwater monitoring wells will be maintained and monitored to insure compliance with water quality standards.

18.9.3 Reclamation and Closure Costs

A $10 million reclamation cost has been included in the project’s economic evaluation which includes dismantling and removal of all equipment and buildings on the project site; rehabilitation of the project site; and long term maintenance, monitoring, and testing of the tailings dump, development rock dump, and monitoring wells.

Currently there are no specific regulations for bonding of mine closure costs. In lieu of a regulation, the Ministry of Environment and Forestry deals with the issue on a case by case basis. General practice has been to have the owner secure a line of credit or hold funds relating to closure costs, the amount to be established between both parties.

18.9.4 Monitoring and Reporting

Closure and reclamation are iterative processes requiring ongoing planning and modification. To maintain this process steps will be taken to:

- obtain additional site specific information on mine components as they are constructed and operated including the physical, chemical, and biological characteristics
- prepare and retain accurate as build records and data throughout the life of the mine
- perform site specific testing and evaluation of techniques and methodologies included in the closure plan
- carry out ongoing review to the state of knowledge on reclamation and closure to assure a high level of performance and implementation
- review annually and revise the closure plan in accordance with changes in conditions and practises at the operation
- maintain and monitor groundwater monitoring wells to insure compliance with water quality standards.

18.9.5 Post-Closure Public Access and Safety

All disturbed lands on site will be rehabilitated to a natural state similar to the original site conditions including the revegetation of the disturbed land. The removal of all equipment, buildings, and infrastructure from the project site; plugging and capping of all mine portals and ventilation raises will remove all access to the underground workings and the removal of any other potential physical hazards from the project site will be carried out at closure to insure public safety.

The security fence around the project site will be removed to allow the free movement of fauna and public access to private and public lands.

18.10 Project Schedule

The critical path of the project is driven by land expropriation and receipt of permits. Currently it is estimated expropriation and permitting will be completed by the end of June 2008. Work on site is not scheduled to begin prior to this date. The progress of the permitting and its effect on commitments required to maintain the schedule will need to be closely monitored by the Engineering, Procurement, and Construction Management (EPCM) contractor.

Currently, it is envisioned that some limited offsite construction work, such as the access road upgrade and preparation of the construction infrastructure, will be able to commence prior to permitting being completed.

The project milestones are summarized below.

Feasibility Study Complete	August 2007
Prepare and Issue Bid Packages for Long Delivery Equipment	September 2007
Eldorado's Board Project Approval	November 2007
Begin Planning and Preliminary Engineering	November 2007
Award Long Lead Equipment	November 2007
Award Detailed Engineering	December 2007
Award the access road construction	December 2007
Begin Detailed Engineering	January 2008
Site Earthworks	April 2008
Award Pre-Production Mining Contract	May 2008
Mobilize Pre-Production Mining Contractor	June 2008
Mobilize Aggregate Plant and Batch Plant	July 2008
Begin Concrete Placement on Site	August 2008
Begin Process Building Erection	October 2008
Begin Process Equipment Installation	November 2008
Mechanical Completion	July 2009

18.10.1 Methodology

The implementation of the Efemçukuru Project is based on a standard EPCM project delivery method. Eldorado will engage a North American EPCM consultant for the overall execution of the project.

The overall construction period from commencement of detailed engineering to mechanical completion for the Efemçukuru Project is 18 months.

Engineering will be completed in North America and Turkey according to North American and European Standards, maximizing the use of Turkish standard materials and methods where appropriate. Quality of work and productivity observed on visits to fabricators and contractors during the feasibility study indicated that a high quality level and efficiency could be achieved locally.

Modular construction methodology is being utilized for the Efemçukuru Project to ensure high quality standards, accelerate the implementation of the construction schedule, and minimize the construction site workforce. The proximity to the site of

Izmir is a definite advantage for the utilization of this methodology. Modularization will be completed in Izmir in a controlled environment where quality can be closely monitored. Modules will be erected to the greatest extent possible in the shop and then broken down and shipped to site for erection. This will ensure accurate and expedited installation. Electrical motor control centres (MCC) will be modularized similarly.

Procurement of equipment will be completed based on competitive bidding to qualified international vendors, suppliers with service and parts readily available in Turkey, and specifically Izmir will be a critical selection criteria. The installation contractor will generally complete construction bulk materials.

Where practical, construction contracts will be competitively bid to qualified Turkish contractors. Tüprag has a comprehensive database of qualified contractors in the Izmir area to draw on. Ancillary buildings including the gatehouse, truckshop, administration building, and laboratory will be tendered as design build packages.

The EPCM contractor will be required to carefully plan and coordinate the construction work due to the limited footprint of the site. Access will be maintained on the construction site at all times. A construction laydown site in Menderez is being investigated to allow staging of materials and equipment so they are available on an as required basis. Work will be completed on site minimizing site disturbance and utilizing modular construction methodology to accelerate the schedule, ensuring high quality of workmanship, and limiting the construction equipment, workforce, and congestion on site.

18.10.2 DISCUSSION

The project will be divided into three periods: the project approval period, the permitting period, and the implementation period.

PROJECT APPROVAL PERIOD

The project approval phase extends from the completion of the feasibility study in September 2007 to Eldorado's Board approval in November 2007.

During the period between completion of the feasibility study and Eldorado's Board approval, limited activities will be completed. Work completed during this period will be restricted to critical path activities. The site survey, geotechnical, and hydrogeological investigation will be completed during this period in order to support the detailed design. Engineering work will be limited to preliminary engineering, which includes the design of the access road and the design and specification of the temporary construction infrastructure. The long lead mining and process equipment, including the SAG mill and Ball mill specifications, will be prepared for bid during this period but no commitments for equipment need to be made to suit the schedule requirements.

At the end of the current drill program, the mine reserve will be updated and attached as an addendum to this feasibility study.

During this period, a contract will be negotiated and awarded for the EPCM of the project. A contract for the design of the access road will be negotiated and awarded, and a contract from a qualified Turkish mining company is to be negotiated and awarded for preproduction mining.

PERMITTING PERIOD

Immediately following Eldorado's Board approval in November 2007 but prior to permits being secured in June 2008, engineering will commence in order to support the schedule and allow mobilization on site quickly upon receipt of permits.

The detailed mine design will commence in cooperation with the selected mining contractor, EPCM contractor, and Eldorado. With the site geotechnical completed detailed design of the tailings and waste areas and ponds can commence as well as the detailed design of the site roads and grading. Additionally, detailed design of the flowsheets, layouts, ancillary buildings, process modules, plant infrastructure, and power supply will begin.

Purchase orders for long delivery equipment to support the construction schedule or detailed design of the modules will be issued in 2007. Purchase orders with cancellation clauses or initial procurement of vendor engineering will need to be negotiated in the event permitting can not be secured for the project. The contracts for the access road construction will be tendered and awarded. Contracts for concrete and aggregate production, site civil and roads, tailings and development rock storage and ponds will be tendered to qualified contractors. Tenders will be evaluated; however, the contracts will not be awarded until permits are in place. Construction of the access road improvements will take place during this period. No on-site construction work will take place during this period, as permits are not in place.

The plant site rough grading will require a substantial quantity of blasting of bedrock. A mobile aggregate plant will be mobilized to site for crushing and screening to produce the required construction aggregates. The mobile aggregate plant will be diesel, as sufficient temporary construction power will not be available from the grid. Due to a limited availability of mobile crushers, Eldorado will refurbish their Kisladağ aggregate plant and relocate it to the Efemçukuru site.

IMPLEMENTATION PERIOD

Detailed design of the mechanical, piping, electrical, and instrumentation disciplines will begin in early 2008. The balance of the process equipment will be bid and awarded during this time. Once the final permits are received in June 2008, construction on site can begin. The civil contracts adjudicated previously will allow Eldorado to issue contracts as soon as the permits are secured and the contractors to mobilize immediately. The priority work on site will be the mobilization of the site grading and road contract in order to support mobilization of the mining contractor and placement of concrete. The balance of the lump sum contract packages will be assembled and issued as the engineering is completed. Mechanical completion is currently scheduled as July 2009, 12 months from mobilization at site.

Construction at the Kişladağ site is not restrained by permitting; however, for efficiency and cost, it is considered advantageous to complete the construction of the Kişladağ facilities in parallel with the Efemçukuru facilities.

The construction workforce for the Efemçukuru project site will primarily come from Izmir. The city of Izmir, at approximately four million people, has a large labour pool to draw from and a large number of vendors, fabricators, and contractors available.

18.10.3 LONG DELIVERY/CRITICAL PATH EQUIPMENT

The following equipment will need to be procured in advance of receipt of the permit in order to meet the requirements of engineering and construction:

- mining equipment
- aggregate plant
- batch plant
- SAG and ball mills
- Knelson concentrators
- flotation cells
- thickeners
- paste mixers
- Isamill.

Equipment will fit into process modules fabricated in Izmir and transported to site. Prefabrication of modules in a controlled environment assures quality and ease of installation at site. Highly skilled industrial fabricators are available in Izmir for this work.

18.10.4 CONTRACT BREAKDOWN STRUCTURE

While lump sum contracts are preferred, unit rate construction contracts for concrete supply and aggregate supply prior to having an aggregate plant will allow these critical path contracts to be awarded early. These contracts suit unit rate methodology well and have been successful at the Kişladağ project. EPCM and pre-production mining contracts are traditionally awarded as unit rate contracts. The balance of the contracts will be tendered and awarded as lump sum contracts as the engineering will be sufficiently advanced to support this preferred contract methodology. A large number of local contractors were interviewed during the production of the feasibility study and local construction contractors are familiar with a variety of contract packaging philosophies. At this time the only design build, turn-key packages envisioned are for the ancillary facilities including the maintenance shop/warehouse, administration building, mine dry and cafeteria, laboratory, and gatehouse.

The following is a breakdown of the contracts envisioned for the Efemçukuru Project:

C-001	Access Road Engineering	Unit Rate
C-002	EPCM Contract	Unit Rate
C-003	Pre-production Mining	Unit Rate
C-004	Temporary Construction Power	Reimbursable
C-005	Access Road	Lump Sum
C-006	Concrete Supply	Unit Rate
C-007	Aggregate Production	Unit Rate
C-008	Site Civil – Efemçukuru	Lump Sum
C-008A	Site Civil – Kişladağ	Lump Sum
C-009	Tailings and Waste Rock Storage	Lump Sum
C-0010	Concrete – Efemçukuru	Unit Rate
C-0010A	Concrete – Kişladağ	Unit Rate
C-0011	Power Line	Lump Sum
C-0012	Mechanical & Electrical – Efemçukuru	Lump Sum
C-0012A	Mechanical & Electrical – Kişladağ	Lump Sum
C-0013	Process Building – Efemçukuru	Lump Sum
C-0013A	Process Building – Kişladağ	Lump Sum
C-0014	Ancillary Buildings	Design Build
C-0015	Site Services – Efemçukuru	Lump Sum
C-0015A	Site Service – Kişladağ	Lump Sum

SECTION 19 • REQUIREMENTS FOR TECHNICAL REPORTS ON PRODUCTION

19.1 MINE PLAN AND PRODUCTION

19.1.1 INTRODUCTION

The Efemçukuru Project mineral reserve is 3.785 million diluted tonnes at an average grade of 10.04 g/t Au. The mine cut-off grade used for the mine reserve calculation was 4.5 g/t Au. Silver was not considered as part of this study. The projected mine life is 9.4 years at the proposed production rate of 1,100 t/d, with 10 months of pre-production underground mine development.

This study describes mining two of the three known orebodies, or ore shoots at Efemçukuru, namely the SOS and MOS, both of which are open down-dip. The NOS is poorly defined at this time and has not been included in this study.

The mine design has been developed to allow flexible access to both the MOS and SOS. Two spiral footwall ramps at each orebody provide access for moving men, equipment, and supplies underground. Advantages of the two-ramp system include increased stope availability, more robust ventilation with increased equipment and labour productivity. One disadvantage of this approach is the additional cost of waste development for the ramps.

Ore will be truck hauled to a central ore pass system above the underground crusher before being conveyed to surface via an 800 mm belt conveyor. The orepass will provide 1,500 tonne surge capacity for underground production with a further 2,700 tonne capacity in bins on surface. Waste rock will be hauled to surface via the South 672 Portal.

MCF will be the primary stoping method used for widths between 2 m and 8 m. This method allows selective recovery of ore within the orebody. TLH will be used in the MOS where the orebody is wider than 5 m. LLH will be used in the SOS where the orebody is wider than 8 m.

Mining from longhole stopes will easily achieve the full target production rate. The key will be maintaining balance between the longhole and mechanized cut-and-fill production to minimize operating costs and labour requirements. Ore from the MOS and SOS orebodies will be blended to balance high and low sulphide ore and provide a consistent head grade to the mill. The transverse longhole stope access is planned in ore, limiting the number of working stopes available but reducing waste development.

Paste backfill will be used as a "free standing" structure to control stability of walls, dilution, and safety for the longhole stopes. In the mechanized cut-and-fill stopes, paste backfill will be used to stabilize the working floor. The paste plant will be located near the North 656 Portal.

In November 1997, H.A. Simons Ltd. (Simons) completed a prefeasibility study, which included a conceptual underground mine design and schedule. A production rate of 800 t/d was selected in the study. The report recommended the primary mining method to be cut-and-fill with multiple-entry footwall access, and suggested that ore widths in the MOS might allow for sub-level longhole stoping.

Wardrop has revised the work carried out by Simons. The findings of this study are in general agreement with the previous studies; they differ in the following details:

- primary crusher will be located underground
- ore will be conveyed to the surface via a 800 mm conveyor belt
- two portals will be installed (one for each ramp)
- mechanized cut-and-fill will be adopted in selected narrow areas of the MOS and SOS
- transverse longhole stoping will be adopted in selected wider areas of the MOS
- longitudinal longhole stoping will be adopted in selected wider areas of the SOS
- stope access will be developed in ore.

Grade control at Efemçukuru will be through assay cut-off and not visual or structural cut-offs. Narrow mechanized cut-and-fill stopes will not be selective within the vein. Narrow stopes will be mined from the footwall to hanging wall.

19.1.2 Mine Production Rate and Mine Life

The mine production rate is based on supplying the mill with 7,700 tonnes per week of ore. The mill will operate seven days per week with an availability of 90%. The mine will operate 6 days per week, 312 days per year. The average mine production rate will be 1,283 t/d.

Taylor's Rule of Thumb suggests 1,375 t/d for this mineral resource and reserve. In light of best practices at mines with multiple, narrow orebodies, 1,283 t/d is considered appropriate. A combination of longhole and cut-and-fill mining will be required to meet this target.

Taylor's formula is:

$$\text{Optimum Production Rate} = \frac{5 \text{ x } (\text{Economic Resource})^{3/4}}{(\text{Production Days per year})}$$

Mine operating life is estimated at 9.4 years, with 10 months of pre-production underground development.

Longhole stopes can easily achieve the target production rate; the key will be maintaining the required balance between the longhole and mechanized cut-and-fill to meet the daily production targets listed in Table 19.1. This balance is required for:

- blending of ore between MOS and SOS – grade control and sulphide blending
- minimize operating costs
- optimize labour requirements
- minimize mine development
- achieve consistent mill throughput for life of mine.

If the longhole methods overproduce there will be more pressure on the mechanized cut-and-fill, potentially increasing equipment requirements.

The transverse longhole stope access has been planned in ore to reduce the potential for over production. This also reduces the amount of waste development and enforces the sequencing of stopes.

The mechanized cut-and-fill will require an average of 4 rounds per day to meet the target of 582 t/d. Each round is approximately 179 tonnes based on an average stope width of 4 m. Transverse longhole will target 427 t/d and longitudinal longhole will target 274 t/d. The summary of target and capacity production by mining method is shown in Table 19.1.

Table 19.1 Summary of Target Daily Production

Mining Method	Production by Method (%)	Maximum Production Capacity (t/d)	Target Production (t/d)
MCF	45	751	582
TLH	33	1,974	427
LLH	21	1,228	274
Effective Mine Productivity	-	-	**1,283**

The mine productivity by mining method is shown in Table 19.2. The productivity per manshift is calculated as the target production for each mining method divided by the number of direct and indirect personnel (including supervision) required. The productivity per total manshift includes all personnel related to mining including technical services, maintenance, and the mine manager.

Table 19.2 Summary of Mine Productivity

Mining Method	Production (t/d)	Direct & Indirect Labour	Productivity (t/manshift)	All Labour	Productivity (t/total manshift)
MCF	582	13	45	28	21
LLH	274	5	56	10	26
TLH	427	7	62	15	29

19.1.3 Mining Methods

Factors taken into account when selecting the mining method at Efemçukuru included:

- continuity, size, and shape of the orebody
- local orebody ground conditions (ground support requirements)
- dip angle of the orebody
- achievable production rate based on mucking requirements
- value of in situ ore, mining dilution and recovery.

The proposed mining methods are sub-level transverse longhole stoping, sub-level longitudinal longhole stoping, and mechanized cut-and-fill. All methods will require paste backfill.

To minimize development and allow flexibility between mining methods all mining methods will utilize mining block heights of 16 m, floor to floor. Figure 19.1 shows the shared drilling and extraction levels between the methods.

Figure 19.1 Interaction between Mining Methods – Cross-Section

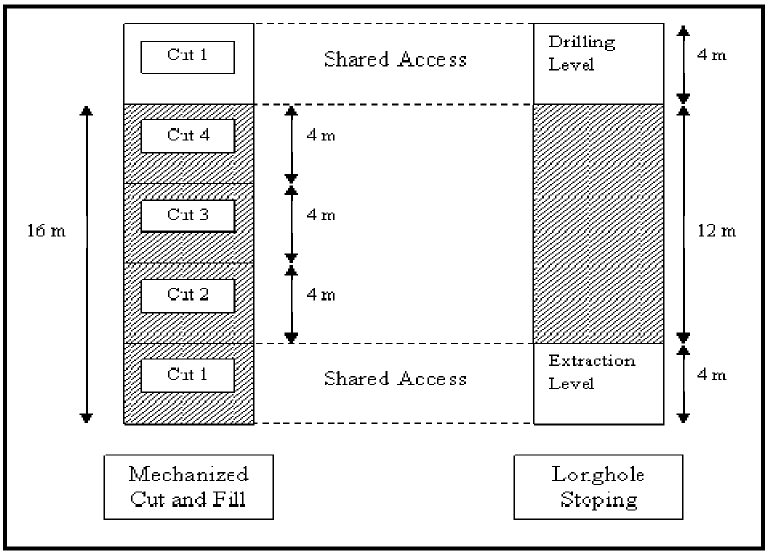
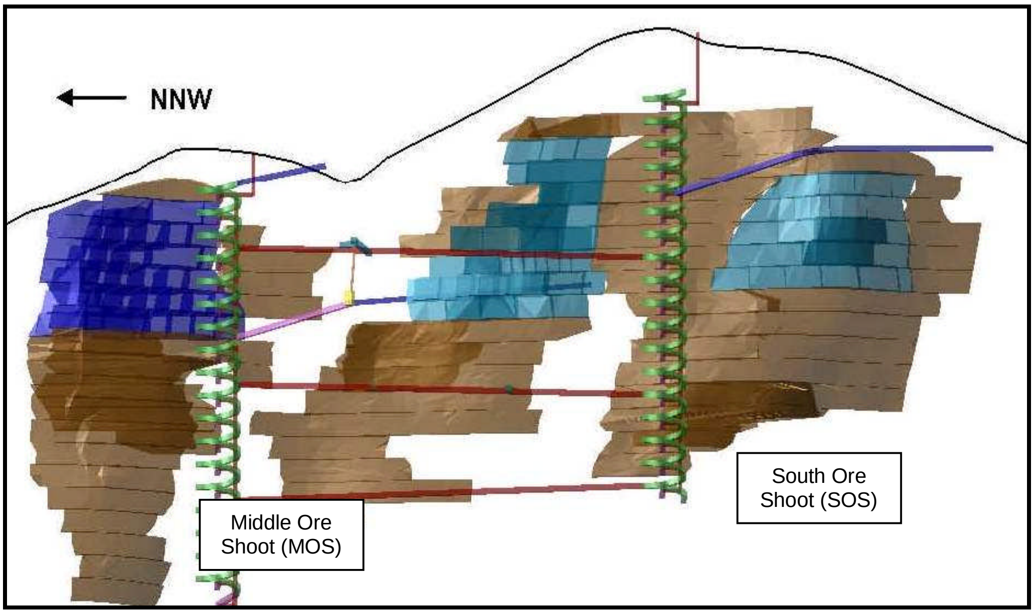


Figure 19.2 shows the mining blocks by type with the mine development looking east across the strike of the orebody.

Figure 19.2 Mining Method



Dark Blue = transverse longhole Light Blue = longitudinal longhole tan = mechanized cut-and-fill

MECHANIZED CUT-AND-FILL

MCF is the primary mining method and accounts for 45% of the total production; the target MCF production is 582 t/d. This selective mining method is more expensive, has lower productivity, and requires more working faces to meet production targets. MCF stoping will generally be used for orebody widths less than 8 m. At widths greater than 4 m, double breasting with tight backfill will be utilized.

MCF lifts will typically be the width of the orebody and 4 m high x 4 m deep. The block height will be 16 m floor to floor. Sub-level development will provide access for orebody drilling and blasting, ore and waste haulage, materials and services supply, and ventilation.

The mechanized cut-and-fill production will require an average of four rounds per day blasted to meet the target of 582 tonnes per day. Each round is approximately 179 tonnes based on an average 4 m wide stope. A minimum of eight working faces should be accessible at any one time. This will include six stopes in the production cycle (including drilling, charging, mucking, or ground support) and two stopes being backfilled or curing. This is based on the average 4 m wide stope. Where production stopes are wider than this average, the number of working places may be reduced.

The productivity of the MCF should be maximized where possible due to the narrow and complex nature of the orebody. The longhole methods should be used to supplement the MCF mining method.

Table 19.3 Average Mechanized Cut-and-Fill Mining Blocks

Mining Method	Average Mining Block Properties				Productivity		
	Width (m)	Length (m)	Height (m)	Tonnage (t)	Target Production (t/d)	Drilling Required (m/d)	Explosives Loaded (kg/d)
MCF	4.0	115.0	16.0	20,610	582	464	637

The overall average length of 115 m for MCF represents a number of stopes in one mining block. Average lengths of individual stopes will vary according to the number of working places required.

A one boom jumbo will drill the face, advancing an estimated four metres per round. Two boom jumbos will be used as required. Blast holes will be 45 mm diameter, drilled on a standard overhand heading pattern. ANFO explosives will be initiated by dynamite primers with non-electric detonators. Emulsion will be required for loading wet holes.

TRANSVERSE LONGHOLE STOPING

The target TLH production is 427 tonnes per day. TLH production accounts for 33% of the total production. Transverse longhole stoping provides high productivity from a small number of work areas. Sub-level development will be 4 m wide x 4 m high to accommodate 42" diameter ventilation tubing and 20 tonne haulage trucks. TLH stopes will be developed across the strike of the orebody using a drilling sublevel on top of the stope, with an extraction level at the bottom. TLH will be used for stoping widths greater than 15 m. The block height will be 16 m floor to floor.

Table 19.4 Average Transverse Longhole Mining Blocks

Mining Method	Average Mining Block Properties				Productivity		
	Width (m)	Length (m)	Height (t)	Tonnage (t)	Target Production (t/d)	Drilling Required (m/d)	Explosives Loaded (kg/d)
TLH	23.5	25.0	16.0	26,320	427	125	282

Stope development will be in ore on the footwall side of the orebody. The access will remain until the end of the stope cycle to provide access for the paste backfill. This will reduce operating cost and restrict the availability of stopes to prevent over production of this mining method early in the mine life.

Sub-levels will be at 16 m vertical intervals. Sub-level development will provide access for orebody drilling and blasting, ore and waste haulage, materials and services supply, and ventilation.

An in-the-hole (ITH) drill will perform blasthole drilling in longhole stopes. Average drilling depth will be 12 m from the upper sill to the lower extraction level. Blast holes will be 64 to 89 mm diameter, drilled on a 1.5 m square pattern. Ammonium Nitrate/Fuel Oil (ANFO) will be the bulk explosive initiated by high explosive with non-electric detonators.

The stope development sequence will commence with a slot raise in the corner of the stope. The slot raise will be developed by longhole drilling, and stage blasted from the bottom up (i.e. drop raised). The raise will then be enlarged to form a slot across the full width of the stope. Vertical rings of drill holes will be blasted into the slot as required.

Transverse longhole stopes will be mucked from a single draw point on the extraction level on the footwall side of the stope. Ore will be mucked directly into 20 tonne haulage trucks before being hauled to the central ore pass system.

LONGITUDINAL LONGHOLE STOPING

The target LLH production is 274 tonnes per day. LLH production accounts for 21% of the total production. LLH stoping also provides high productivity from a small number of work areas. LLH stopes will be along the strike of the orebody using a drilling sublevel on top of the stope, followed by an extraction level at the bottom. LLH will be used for stoping widths between 8 m and 15 m. The block height will be 16 m floor to floor.

Table 19.5 Average Longitudinal Longhole Mining Blocks

Mining Method	Average Mining Block Properties				Productivity		
	Width (m)	Length (m)	Height (m)	Tonnage (t)	Target Production (t/d)	Drilling Required (m/d)	Explosives Loaded (kg/d)
LLH	11.7	120.0	16.0	62,900	274	99	150

Stope access, and drill and blast will be similar to the TLH method. The overall mining block length of 120 m represents a number of stopes. Average lengths of individual stopes will be determined by geotechnical analysis during the detailed engineering stage.

The stope development sequence will commence with a slot between the drilling level and extraction level at the end of the stope. Stope development will be in ore. The slot raise will be developed by longhole drilling, and stage blasted from the bottom up. Vertical rings of drill holes will be blasted as required into the slot during production.

Longitudinal longhole stopes will be mucked from a single draw point on the extraction level from the stope access. Ore will be mucked directly into 20 tonne haulage trucks before being hauled to the central ore pass system.

MINING BLOCK DIMENSIONS

"Average" mining block dimensions are used for cost estimation and productivity analysis. Mining block dimensions will vary according to the factors listed at the start of this section. Average mining block dimensions are compared in Table 19.6.

Table 19.6 Average Dimensions

Mining Method	Average Width (m)	Average Length (m)	Height (m)	Tonnage (t)
Transverse Longhole	23.5	25.0	16.0	26,320
Longitudinal Longhole	11.7	120.0	16.0	62,900
Mechanized Cut-and-fill	4.0	115.0	16.0	20,610

The production by orebody and mining method is shown in Table 19.7.

Table 19.7 Orebody Delineation by Mining Method

Orebody	% TLH	% LLH	% MCF
MOS	69	0	31
SOS	0	42	55

19.1.4 MINING SCHEDULE

DEVELOPMENT SCHEDULE

Reference: Appendix D, Dwg B4-40-042

All lateral and ramp waste development will be performed by two boom mining jumbos. Ground support will be completed using dedicated ground support jumbos. Load-haul-dump units (LHDs) will muck broken rock to a remuck bay before loading into articulated haul trucks.

The development productivity is based on the activities listed in Table 19.8. The development schedule by year is shown in Table 19.9.

Table 19.8 Mine Development Cycle

Activity	Time (h)
Drill	2.8
Charge	2.2
Fire	0.3
Muck	2.0
Ground Support	2.0
Total	9.9

Table 19.9 Mine Development Schedule

	Unit	Year											Total
		2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	
Development Waste Rock													
Waste Rock Tonnes	t	159,286	101,527	29,498	28,337	107,083	5,940	-	-	-	-	-	431,671
Waste Rock Productivity	t/d	613	325	95	91	343	19	-	-	-	-	-	-
Mine Development by Material													
Waste Development	m	2,781	1,871	585	523	2,147	115	-	-	-	-	-	8,022
Ore Development	m	0	2,774	3,977	2,062	-	2,245	974	-	-	-	-	12,031
Total Development	m	2,781	4,644	4,561	2,585	2,147	2,360	974	-	-	-	-	20,052
Mine Development by Type													
Ramp Waste Development	m	1,875	977	331	221	993	-	-	-	-	-	-	4,398
Lateral Waste Development	m	906	893	254	302	1,153	115	-	-	-	-	-	3,624
Conveyor Drift	m	414	-	-	-	-	-	-	-	-	-	-	414
Advance per Day	m/d	18	15	15	8	7	8	3	-	-	-	-	-
Raise Development	m	176	276	73	55	164	36	-	-	-	-	-	781

The development cycle is illustrated in Appendix D, Dwg B4-40-42.

There will be two mine development crews available during years 1 and 2. One crew will remain after this time to continue mine access development and assist with production development and training as required.

Single heading development rates are estimated at 7.7 m/d, andmultiple heading at 12.0 m/d.

The conveyor decline development rate was reduced to 5 m/d to account for productivity losses due to the -18% gradient.

The mine development schedule was completed in Gemcom MineSched Surface and Underground Scheduling software (MineSched). Productivities were entered by development type and by crew.

The development schedule was based on the following targets:

- minimize pre-production development
- maintain access to six months of blasted stocks
- maintaining ramp development six to nine months ahead of production.

PRE-PRODUCTION DEVELOPMENT

Over a period of 10 months, the underground mine pre-production development will include the items listed in Table 19.10.

Table 19.10 Pre-production Development Requirements

Development	Priority	Development Length (m)	Time Required (days)
Mine Entry Portals	1	40	30
South Ramp	2	1,000	125
North Ramp	3	794	99
590 Level Connector Drift	4	337	42
Conveyor Decline	5	415	83
Underground Crusher & Orepass Installation	6	140	150

A mining contractor will complete the pre-production work. Initially three development crews will be required to develop the North 656, South 676, and conveyor adits concurrently. Only two development crews will be required once the conveyor decline is completed. The crusher and orepass installation will require development of a bypass to allow the orepass can be developed independently of the crusher installation.

The contractor will also be required to provide training to Eldorado employees prior to handover from pre-production to production.

PRODUCTION SCHEDULE AND SEQUENCING

Reference: Appendix D, Dwg B4-40-036

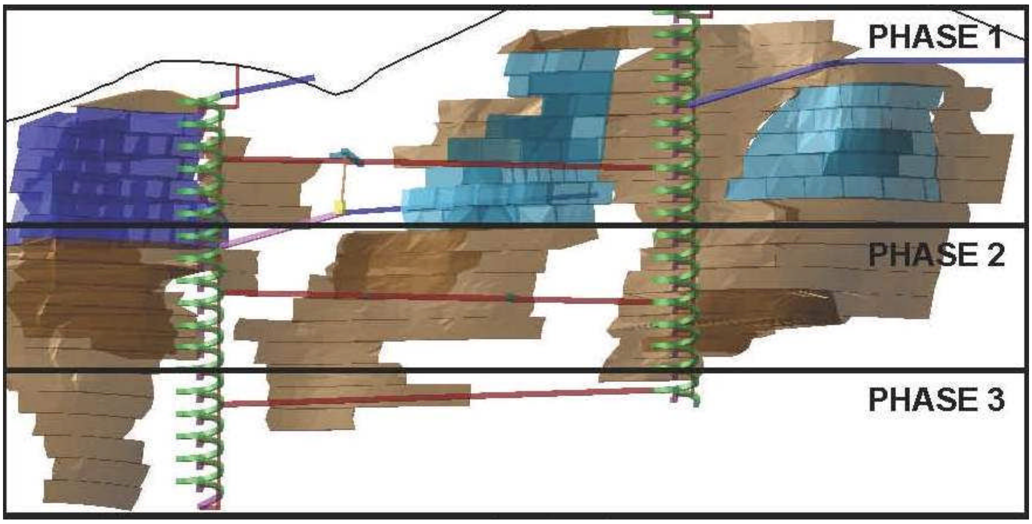
The production schedule was developed using the target production rates discussed in Section 19.1.4 above. The production schedule targets high-grade ore early in the mine life where possible. The focus of the production schedule was to ensure sustainability of mill throughput and grade for the life of mine. The high productivity longhole methods were restricted to the target production. During mining operation, the balance between daily production targets and long-term production will require adhering to the life of mine plan.

The production schedule is based on three phases of mining, each phase being mined from bottom to top as shown in Figure 19.3. The number of phases is related directly to the number of working places available and the number of sill mats required.

The number of working faces available in the schedule is based on the average mechanized cut-and-fill stope width of 4 m. A more detailed approach will be required for the final mine design in detailed engineering. There is potential for the first phase to be divided into two phases to increase the number of work places available in the early mine life, creating a total of four phases for life of mine.

This study assumes three phases and four areas where sill mats will be required. Sill mats will be required at the lower level of each phase and in addition at the lowest transverse longhole stope in the MOS where there is a transition between mining methods. Working floors will be equally important for all stoping operations. In particular, high strength working floors will be required to minimize dilution and maximize the productivity of the production LHDs.

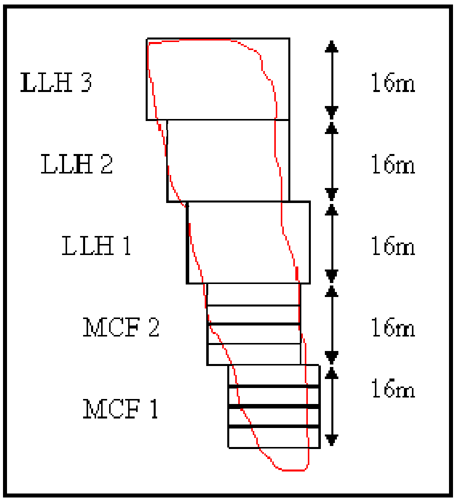
Figure 19.3 Production Areas



All mining methods will use mining block heights of 16 m floor to floor to allow sharing of stope access as discussed in Section 19.1.3.

An example of the interaction between the MCF and LLH mining methods is shown in Figure 19.4.

Figure 19.4 Mining Method Sequence



In the example (Figure 19.4), the lower mechanized cut-and-fill stope (MCF1) is typically mined first. The sequence would progress from MCF1 to MCF2 to LLH1 to LLH2 to LLH3. No sill mat would be required.

If in the same example the longitudinal longhole stopes are required first, a sill mat would be required at the bottom of the longitudinal longhole stope (LLH1). This would be installed on the extraction level prior to stoping.

The mining sequence would therefore be LLH1 to LLH2 to LLH3 to MCF1 to MCF 2. One sill mat would be required.

For the purpose of this study sill mats were assumed to include:

- rockbolts in the side wall up to approximately 1 m
- support cables with anchor pins in the side wall
- reinforcing with screening
- high strength cement (8% cement content for working floor).

Ideally, the sill mat will be installed in the narrowest width of the stope to minimize the support required. The extraction of ore beneath the sill pillar will be on a retreat basis using upholes. The ore will be mucked using a remote controlled LHD. Underground personnel will not work directly under the sill mat.

Table 19.11 shows the ore production schedule by year.

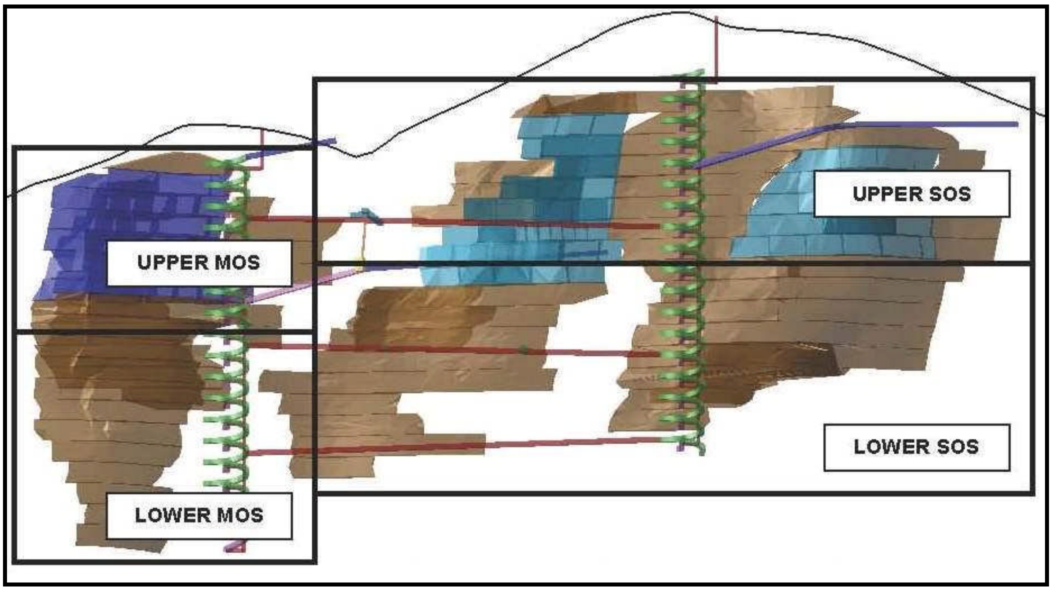
Table 19.11 Mine Production Schedule

	Unit	2008 Year -1	2009 Year 1	2010 Year 2	2011 Year 3	2012 Year 4	2013 Year 5	2014 Year 6	2015 Year 7	2016 Year 8	2017 Year 9	2018 Year 10	Total
Mine Operating Time													
Days	d	0	156	312	312	312	312	312	312	312	312	289	2,941
Total MCF Production													
MCF Lower MOS Production*	t	0	0	0	0	90,509	73,927	24,447	134,249	127,524	102,536	0	553,192
	t/d	0	0	0	0	290	237	78	430	409	329	0	188
MCF Upper SOS Production*	t	0	1,354	94,519	49,893	9,952	0	37,679	50,328	57,283	51,524	23,494	376,026
	t/d	0		303	160	32	0	121	161	184	165	81	128
MCF Lower SOS Production*	t	0	89,955	0	152,610	80,828	116,221	128,056	11,243	0	34,587	172,688	786,188
	t/d	0		0	489	259	373	410	36	0	111	553	267
Total Tonnes	**t**		**91,309**	**94,519**	**202,503**	**181,289**	**190,148**	**190,182**	**195,820**	**184,807**	**188,647**	**196,182**	**1,715,406**
Total Productivity	t/d		585	303	649	581	609	610	628	592	605	678	583
Total LLH Production													
Total Tonnes	**t**	**0**	**109,441**	**306,981**	**135,987**	**0**	**0**	**110,114**	**82,884**	**54,887**	**25,248**	**0**	**825,542**
Total Productivity	t/d	0	702	984	436	0	0	353	266	176	81	0	281
Total TLH Production													
Total Tonnes	**t**	**0**	**0**	**0**	**63,010**	**220,211**	**211,352**	**101,204**	**122,796**	**161,806**	**187,605**	**176,172**	**1,244,156**
Total Productivity	t/d	0	0	0	202	706	677	324	394	519	601	609	423
Total Mine Production													
Total Tonnes	**t**	**0**	**200,750**	**401,500**	**401,500**	**401,500**	**401,500**	**401,500**	**401,500**	**401,500**	**401,500**	**372,354**	**3,785,104**
Average Gold Grade	**g/t**	**0**	**10.37**	**9.65**	**10.03**	**9.69**	**10.52**	**10.20**	**10.44**	**9.18**	**9.52**	**10.99**	**10.04**
Gold Mined	oz	0	66,920	124,623	129,426	125,051	135,805	131,644	134,822	118,466	122,877	131,598	1,221,233
Gold Produced	oz	0	57,886	107,799	111,953	108,169	117,472	113,872	116,621	102,473	106,289	113,832	1,056,367

Notes: *Lower MOS = below Level 525 Upper MOS = Level 577and above Lower SOS = below Level 577

Figure 19.5 illustrates the areas shown in the production schedule.

Figure 19.5 Production Areas



19.1.5 MINE ACCESS

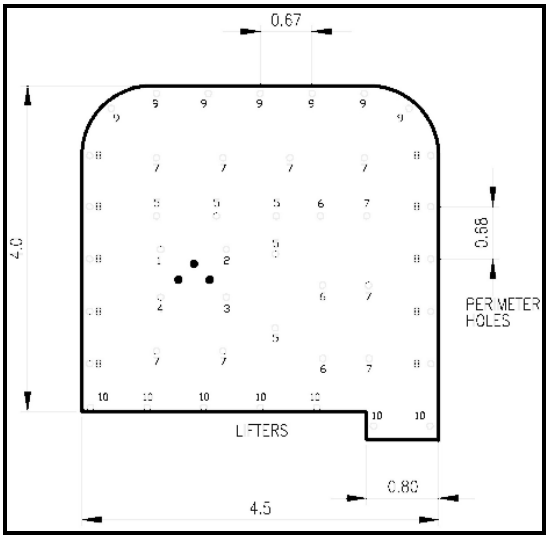
Reference: Appendix D, Dwg A0-10-001

There are three access points to the Efemçukuru orebody. The north access adit will be developed from the 656 m elevation. This adit will be developed 90 m to the west at 4.5 m wide and 4.0 m high at a gradient of -15% to connect with the North Ramp. The hornfels hanging wall rock is competent and will be supported by conventional rock bolts, with the addition of mesh and shotcrete, as local conditions require. This adit is the main mine access for equipment, personnel, material, and supplies. The development plan is shown in Figure 19.6.

The north adit will carry all mine services including the compressed air line, fresh water line, return water line, the backfill line, and the electrical and communication cables.

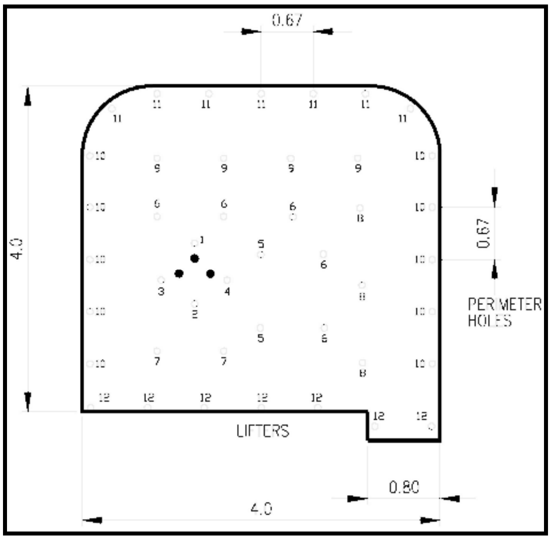
The south access adit and South Ramp will be developed to connect access from underground to the development rock dump. This decline will be driven for approximately 300 m at 4.5 m wide x 4.0 m high and at -15% grade to connect with the South Ramp. All underground mine waste rock from initial and subsequent development will be transported by truck via this adit to surface. The south adit will provide secondary access to the mine for personnel and materials.

Figure 19.6 Adit – Haulage Drives, Ramps, Access to Ore – 4.0 m x 4.5 m



The conveyor portal and decline will be developed from 619 m elevation. The decline will be developed 415 m south at 4 m wide x 4 m high and at -18% grade. The conveyor decline will connect to the crusher room at 545 m elevation. The decline intersects a mineralized zone through a pillar (15 m x 15 m x 42 m) sterilizing 19,533 tonnes at 6.0 g/t, a total of 3,772 oz of gold. The cross section of the conveyor decline is illustrated in Figure 19.7.

Figure 19.7 Conveyor Drift – 4 m x 4 m



19.1.6 Mine Development

Development Ground Support

The geotechnical evaluation detailed in Section 19.1.8 provides the technical basis for the ground support at Efemçukuru.

Standard Development Back Support

The Norwegian Geotechnical Institute (NGI) support method was used to determine the safe spans for mining. For both MOS and SOS, the maximum unsupported safe span is 9.0 m.

The minimum recommended ground support for the access development is summarized as follows:

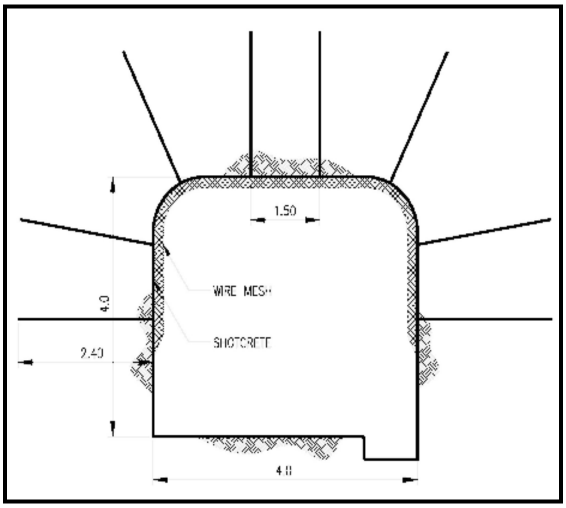
- pattern bolting using 2.4 m long mechanical rock bolts on a 1.2 m x 1.2 m pattern is recommended for the back and shoulder of all ramps/drifts with widths of 4.5 m and life expectancies of more than 3 years
- only spot bolting, as and where required, for all access sills with widths of 3.5 m, and life expectancies of less than three years.

Additional mine development back support, as required, may include:

- discretionary additional bolting density at intersections
- at portals, 50 mm thick shotcrete applied for a minimum length of 40 m
- welded mesh screen, 1.5 m x 3.0 m sheets of #9 Gauge 75 mm x 75 mm, 8 bolts per screen on a 3-2-3 pattern continuously overlapped with shotcrete
- all development will require regular (monthly) maintenance scaling and inspection.

Figure 19.8 illustrates the maximum required ground support with mesh and shotcrete.

Figure 19.8 Maximum Ground Support Requirements

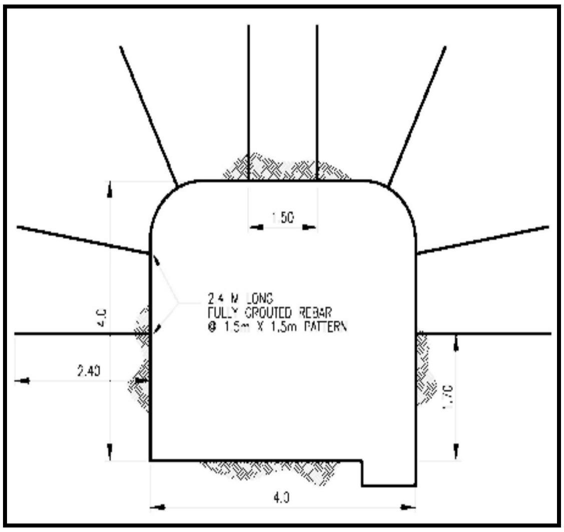


Standard Development Wall Support

Figure 19.9 illustrates standard wall support including:

- bolted and screened to within 2 m of the floor
- welded mesh screen, 1.5 m x 3.0 m sheets of #9 Gauge 75 mm x 75 mm, six bolts secure the screen as required
- discretionary 1.2 m or 1.5 m resin-grouted bolts where ground conditions are weak.

Figure 19.9 Typical Wall Bolting Pattern



The first 40 m of mine development from surface is expected to be in broken and weathered rock, necessitating increased ground support. The first blast at the portals will use perimeter drilling and pre-shear blasting to minimize overbreak and fracturing of the surrounding rock. A 1.2 m square pattern of 2.4 m resin-grouted bolts with 100 mm square welded mesh screen will be installed at the brow of the portal and on the sidewalls. Shotcrete 50 mm thick will also be required. This support system will be applied until competent rock is mapped.

19.1.7 MINE EXPLORATION

Mine exploration from underground will be required early in the mine life. During pre-production access to the orebody from the North Ramp will be used for exploration drilling of the GAP or transition zone. The SOS and the MOS are both open down-dip and require further exploration drilling.

Delineation drilling will be required for grade control.

19.1.8 GEOTECHNICAL EVALUATION

CROWN PILLAR

The stability of the Efemçukuru mine crown pillar has been evaluated using analytical and empirical methods. The results of the NGI Unsupported Safe Span and Carter's Crown Pillar Stability Analysis indicate that a 10 m thick crown pillar is stable for both MOS and SOS.

The results of Carter's empirical procedure show that the design crown pillar will be stable. The ground conditions would have to deteriorate to a critical value of Q<0.3 for spans between 5 and 15 m in both MOS and SOS before a potentially unstable condition is reached. This level of deterioration is not likely to occur on a large enough scale in a 10 m thick crown pillar. For larger spans, 20 m or larger, the ground conditions would have to deteriorate to a critical value of Q <0.7 for MOS, and Q<0.8 for SOS before a potentially unstable condition is reached. This level of deterioration is possible and additional ground control may be required. Stopes below pillars will be tight filled to achieve long-term stability.

The results of the Hoek and Brown pillar strength procedure indicate a crown pillar strength between 32 and 48 MPa. The analysis indicates a maximum pre-mining horizontal stress of 8.60 MPa and a mining-induced maximum stress in the order of 30 MPa. This level of stress is less than the calculated strength of the crown pillar; therefore, a stress-driven failure mechanism is not likely.

The results of CPILLAR, the crown pillar analysis software, show the crown pillar is stable.

OREBODY

Two major ore shoots were analyzed for the feasibility study, the MOS and the SOS. The host rocks for MOS consist of hornfels in the hanging and footwalls. The host rocks for SOS consist of hornfels in the hanging wall, with phyllites more prevalent in the footwall.

Minor structure in the site included two joint sets and one bedding plane, which are summarized in Table 19.12.

Table 19.12 Minor Structures

Type	Dip	Azimuth
Bedding	46	298
Joint	66	79
Joint	80	355

A stereographic analysis showed that wedge failures in the roof of mine heading are possible kinematic failures. The rock mass classification data for MOS and SOS is summarized in Table 19.13.

These results represent ground conditions that can be described as Fair to Good. There is little difference in the average and the range of values between the vein and the immediate abutments (both hanging wall and footwall). This is most likely due to the presence of stockwork in association with the vein.

Table 19.13 Rock Mass Classification – MOS and SOS

Zone	RMR (1976)		Q	
	Relative Minimum	Average	Minimum	Average
MOS				
HW	40	57	1.1	7.4
Ore	41	58	1.2	8.3
FW	46	65	2.2	18.0
SOS				
HW	44	64	1.7	16.1
Ore	42	61	1.4	11.5
FW	47	65	2.4	18.0

In the context of the mining method, it is generally recognized that a Rock Mass Rating (RMR) >30 represents the threshold for the successful implementation of long hole mining. Modified long hole methods, (i.e., short lift long hole), are commonly applied in ground conditions with an RMR between 30 and 40.

A total of 976 point-load tests were performed on drill core samples to derive the equivalent Uniaxial Compressive Strength (UCS) for the rock types. Table 19.14 shows a summary of the results.

Table 19.14 Uniaxial Compressive Strength Analysis

MOS	UCS (MPa)		# Tested
	Average	Minimum	
MOS			
Both Directions	162	8	350
Longitudinal Direction	161	8	44
Conjugate Direction	167	12	306
Hornfels	163	8	218
Vein Breccias	163	25	132
SOS			
Both Directions	154	6	617
Longitudinal Direction	174	7	98
Conjugate Direction	160	6	519
Hornfels	158	14	330
Phyllites	138	16	25
Vein Breccias	179	6	204

EARTHQUAKE SEISMIC ZONE

The United States Geological Survey describes Western Turkey as an active earthquake area. The Efemçukuru Project is located in a Turkish Seismic Zone 1 for seismic activity. This equates to a level 8 on the Mercali Scale (MSK-64). The earthquake hazard map of Western Turkey indicates that the region has a 10% chance of experiencing an earthquake that exceeds an acceleration coefficient of 0.5 in 50 years. An acceleration coefficient of 0.5 is equivalent to a peak horizontal ground velocity of 600 mm/s. Possible damage resulting from this level of earthquake would be moderate to heavy.

The results of the Mathew's/Potvin Stability Analysis indicates the proposed cut-and-fill and longhole stope geometries will be stable.

The Phase^2, numerical modelling, finite element analysis of the proposed longhole extraction sequence indicates that the stoping geometry and crown pillar are stable, and no mining induced stress related problems are anticipated.

19.1.9 PASTE BACKFILL

Paste backfill will be required to provide working floor and sill mat construction, and longhole stope stability.

Some of the advantages of using paste backfill compared with hydraulic backfill include:

- higher strengths will be achieved with an equivalent cement content
- higher paste backfill strength potentially reduces backfill dilution
- shorter stope cycle times can be achieved because an equivalent strength can be achieved in a shorter time with paste backfill
- paste backfill will be deposited as a non-segregated mass providing more predictable and consistent strength properties
- drainage of water and slimes from the fill are minimized, reducing the need for bulkhead construction and extensive drainage works.

Some of the disadvantages of paste backfill include:

- the paste backfill distribution network requires a greater level of engineering design to manage pipeline pressures
- paste backfill systems typically have higher capital costs
- paste backfill systems typically have higher operating costs

- the pumpability of the paste backfill is potentially sensitive to small changes in water content and grain size distribution.

The components for the consolidated backfill for the Efemçukuru Project have been designed using Mohr-Coulomb failure criterion, average cut-and-fill stope dimensions, and previous case studies for backfill mix designs.

Based on the results of the analysis, the following conclusions have been made:

- a uniaxial compressive strength of 0.5853 MPa of the backfill must be attained in order to satisfy the following parameters for the back fill requirements:
  - freestanding height
  - regional stiffness
  - resistance to blast damage
  - flexibility
- the percent cement by weight should be 4 to 5% in order to obtain a backfill compressive strength of 0.5853 MPa.

A number of quality control issues that may arise include the following:

- quality and consistency of the backfill materials
- degree of mixing
- configuration of the fill distribution and delivery system
- effect of segregation during the transportation of the fill
- effect of dropping the fill during placement.

The following further recommendations are made as a result of this study:

- the following tests are required in detailed engineering:
  - rheologic index testing
  - cement and binder screening
  - uniaxial compressive strength testing.
- the working floor of mechanized cut-and-fill stopes will require 8% cement by weight to obtain a backfill compressive strength of 1.2 MPa to support the load of the production LHDs
- quality control issues should be satisfied prior to the emplacement of backfill
- ongoing regular strength testing of paste backfill during operations.

Working Floors

The required backfill compressive strength for the working floors is 1.2 MPa based on the distributed weight of a 6700 kg LHD. A cement content of 8% will be required to achieve the target compressive strength.

The working floor will be critical to minimize paste backfill dilution and maximize productivity of the production LHDs for all mining methods. Working floors and sill mats were considered concurrently during mine scheduling to manage the interaction between the mining methods.

19.1.10 Material Handling

Ore

Ore from stopes will be mucked using 6,700 kg capacity LHDs with 3.7 m^3 buckets. Ore will be directly loaded into 20 tonne articulated dump trucks before being hauled to the central orepass and crusher system. Crushed ore will be conveyed on a 800 mm conveyor into two 1,200 tonne surface bins before entering the process plant. Passing bays are designed to minimize haulage delays at the orepass.

Initially, three haulage trucks will be required to meet the production targets with an additional truck required in Year 4.

The haulage cycle distance and times for different locations in the orebody and by mining method are shown in Table 19.15.

Remote mucking will be required for the longitudinal longhole mining and for mechanized cut-and-fill when extracting the cut directly below a sill mat. Productivity will be reduced when remote mucking. The microscoop will be required in cut-and-fill stopes less than three metres wide.

Table 19.15 Production Capacity by Mining Method

Orebody Location	Haul Distance (m)	Cycle Time (min)	Maximum Production Capacity (t/h)
Mechanized Cut-and-Fill (Lower MOS)	2,810	19.5	45
Mechanized Cut-and-Fill (Upper SOS)	1,620	10.6	83
Mechanized Cut-and-Fill (Lower SOS)	2,260	14.2	62
Transverse Longhole (Upper MOS)	1,060	8.4	104
Longitudinal Longhole (Upper SOS)	1,620	13.6	65

WASTE

Underground waste rock will be loaded into articulated haul trucks by LHDs and hauled to surface via the South Ramp. The centre of the development rock dump will be located just below South 672 Portal; during early mine life the rock dump will accessed along the haul road for initial filling, an approximately 500 m drive from the portal. The final development rock dump, including 25% contingency, will be approximately 530,000 tonnes.

One truck will be required over the life of mine for waste haulage.

UNDERGROUND BACKFILL DISTRIBUTION SYSTEM

Tailings not reporting underground will be stored in a dry surface tailings facility. The tailings facility is discussed in more detail in Section 19.1.13. Table 19.16 lists the initial estimates for tailings production used in the development of the backfill design at Efemçukuru.

Table 19.16 Estimated Tailings Production and Disposal

Surface Tailings Facility Storage	Unit	Value
Mine Tonnage (dry)	t	3,785,750
Concentrate Tonnage (dry)	t	307,438
Tailings Tonnage (dry)	t	3,478,301
Backfill Tonnage (dry)	t	1,659,237
Surface Tonnage (dry)	t	1,819,064
Surface Volume	m^3	972,762
Surface Volume with 15% contingency	m^3	1,118,676

The density factors used in this calculation are shown in Table 19.17.

Table 19.17 Density Factors

Bulk Densities (BD)	t/m^3
Tailings to Underground (dry)	1.42
Tailings to Surface (solid density – packed)	2.15
Tailings to Surface (wet density – packed)	2.36
Waste rock, all types	1.67
Concentrate (dry)	3.20
In Situ Densities (D)	**t/m^3**
Waste Rock	2.80
Ore	2.80
Cement	3.15

Paste will be transported into underground openings through a series of 6” pipeline arteries from the surface paste plant. Paste from the plant will be pumped through the main trunk line for each orebody.

The profile of paste distribution by gravity underground involves vertical and horizontal runs of pipe. Horizontal distance is two to three times the vertical distance. Horizontal pumping of paste is highly dependent on material characteristics, frictional losses, and the moisture content of the paste. A frictional loss of 6 kPa/m is assumed based on similar operations, but testing will be required at the detailed engineering stage.

The paste backfill will be distributed throughout the orebody via lined boreholes between levels at the orebody cross-cuts. The paste backfill will be delivered by pipe to the MOS via the north portal to the first cross-cut on Level 641, then distributed vertically through the lined boreholes. The paste backfill will be delivered to the SOS by pipe on surface to a borehole south of the backfill plant. The paste backfill will be piped to Level 705, then distributed vertically through the lined boreholes. Wardrop recommends that two lined boreholes be installed between each level in each orebody to provide contingency in the event of blockage of the primary borehole.

Since mechanized cut-and-fill is the primary mining method, the availability of the paste backfill plant and infrastructure will be critical to the operation to ensure that the mining cycle is not delayed.

The paste plant will produce dewatered tailings for underground backfill and surface disposal on a continuous basis. Approximately 48% of the dewatered tailings will be placed underground in old stopes and the remaining 52% will be transported to the surface disposal facilities for dry stacking.

The backfill piping will be capable of sustaining static full column pressure assuming that a pipeline blockage might occur including allowances for additional pressure transients. While a factor of safety of two is commonly assigned for paste distribution lines, the abrasiveness of the material will reduce wall thickness over time. For this reason, the main and secondary delivery line is recommended to be Schedule 120 and the tertiary lines will be HDPE lines within 100 m of the target stope. A target transport velocity of approximately one metre per second will be achieved with 100 mm diameter pipe.

The pipeline will be lubricated or “slicked” with either water or a low cement content/water slurry before pumping. This procedure prevents the water in the cement from being removed from the paste and adhering to the potentially dry inner pipe wall.

Cleanout of the backfill lines will be a difficult task. High pressure flushing with mechanical pipe cleaners (pigs) will be required to clean the lines effectively. The flushing water will need to be diverted appropriately at low points to avoid flooding and potential mud rushes.

19.1.11 MINE EQUIPMENT

Table 19.18 lists of underground development, production, and service equipment.

Table 19.18 Underground Mine Equipment

Equipment	Type	Quantity
Drilling Equipment		
Development Jumbo	Two boom	2
Development Jumbo	One Boom	2
Rock Bolting Drill Rig	Diesel-electro-hydraulic	2
Longhole DTH Drill	Diesel-electro-hydraulic	2
Jackleg	Pneumatic	12
Stoper	Pneumatic	12
Exploration Drill	Electric	1
Loading & Hauling Equipment		
Development LHD	4.0 m^3 bucket	1
Production LHD	3.7 m^3 bucket	2
LHD	1.5 m^3 bucket	1
Microscoop	-	1
Underground Haulage Truck	20 tonne	3
Service Vehicles		
Grader	Underground	1
ANFO Loader	Light Vehicle	2
Scissor Lift	Heavy Vehicle	2
Boom Truck	Heavy Vehicle	1
Personnel Carrier	Light Vehicle	4
Heavy Duty Pick-up	Light Vehicle	4
Maintenance Vehicle	Light Vehicle	2
Fuel – Lube Truck	Heavy Vehicle	2
Forklift	Telescopic Boom	1
Light Vehicles	Supervisor/Maintenance	2

An additional truck will be required in Year 4 to meet increasing production needs and a further additional truck in Year 6 to replace one of the heavy utilized trucks.

19.1.12 SERVICES

VENTILATION

The ventilation system designed for the Efemçukuru mine is an exhaust system delivering approximately 165 m^3/s or 625,000 m^3/h. Four main exhaust fans and a number of auxiliary fans will control the primary ventilation circuit. To minimize surface ventilation noise the main ventilation fans will be installed underground. Four fans offer increased flexibility in ventilation flow control throughout the life of the mine. During production, fresh air will downcast through the two spiral ramps and upcast through the two exhaust raises and the conveyor drift. Double ventilation doors and auxiliary fans will be required to assist with ventilation circuit control.

The two access ramps and the conveyor drift will be developed during pre-production. Temporary fans installed at the portals will provide initial primary ventilation through ventilation tubing to the working area at approximately 30 m^3/s. This arrangement will continue until the two exhaust raises are completed.

Permanent ventilation will be by four twin 55 kW vane axial fans, supplying 165 m^3/s of air at full production. The fans will be located in the ventilation drift connecting the ramp to the exhaust raise in each orebody. During full production, vane axial fans installed at the ramp connection will provide secondary ventilation. Exhaust air from each working face will return through the stope access crosscuts to the exhaust raises. The ventilation system designed for the Efemçukuru underground mine is consistent with regulations applied by the Province of Ontario and follows general practices employed throughout Canadian underground mines.

Design Parameters

The mine ventilation requirements were derived from the diesel equipment list and based on the requirement of 0.06 $m^3/s/kW$. Table 19.19 lists the total air volume required. The full production ventilation requirements are 165 m^3/s. A utilization factor was applied to the diesel-electric-hydraulic equipment and low utilization equipment. Ventilation losses are included at 5% of the total ventilation requirements.

Table 19.19 Ventilation Requirements at Full Production

Equipment	Qty.	Equipment Utilization (%)	Utilized Power (kW)	Air Volume (cfm)	Air Volume (m^3/s)
Heavy Equipment					
Two boom Jumbo	2	50%	74	9,900	4
One boom Jumbo	2	50%	55	7,400	3
Rock Bolting Drill Rig	2	50%	55	7,400	3
Longhole Drill	2	50%	116	15,600	7
LHD 1.5 m^3	1	100%	63	8,400	4
LHD 3.7 m^3	2	100%	300	40,200	18
LHD 4.0 m^3	1	100%	186	25,000	11
20 t Haulage Truck	3	100%	705	94,500	42
Microscoop Loader	1	50%	19	2,500	1
Utility Vehicles					
Scissor Lift	2	50%	111	14,900	7
Cassette Carrier	1	50%	54	7,250	3
Maintenance Vehicle	2	50%	100	13,400	6
Supervisor's Vehicle	3	50%	150	20,100	9
Heavy Duty Pick-Up	2	50%	100	13,400	6
Personnel Carrier	4	50%	200	26,800	12
Grader	1	100%	108	14,500	6
Boom Truck	1	50%	56	7,450	3
Fuel-Lube Truck	1	50%	50	6,700	3
ANFO loader	2	50%	111	14,900	7
Ventilation Losses	5%			17,515	8
TOTAL			**2,612**	**367,815 cfm (625,000 m^3/h)**	**165**

The ventilation system design was modelled using Ventsim Mine Ventilation Simulation Software (Ventsim). This software allows input parameters including resistance, k-factor (friction factor), length, area, perimeter, and fixed quantities (volume) of air. The variable parameters used in this model were k-factor, fixed quantity, area, perimeter, and length. The k-factors used are average standards for various types of drifts, raises, and openings. Underground ventilation control requires several sets of ventilation control doors, regulators, and auxiliary fans (of various kW) to direct air quantities to the workings.

System Description

Pre-Production Ventilation

Pre-production development ventilation will be the responsibility of the mining contractor. The contractor will supply air underground via a series of primary and secondary ventilation fans at the portals and in the conveyor drift. These fans will be from 45 kW to 75 kW each, with booster fans required in development headings. Exhaust air will exit the mine through these development heading openings until the development reaches a completed exhaust raise to surface. The pre-production is shown in Figure 19.10.

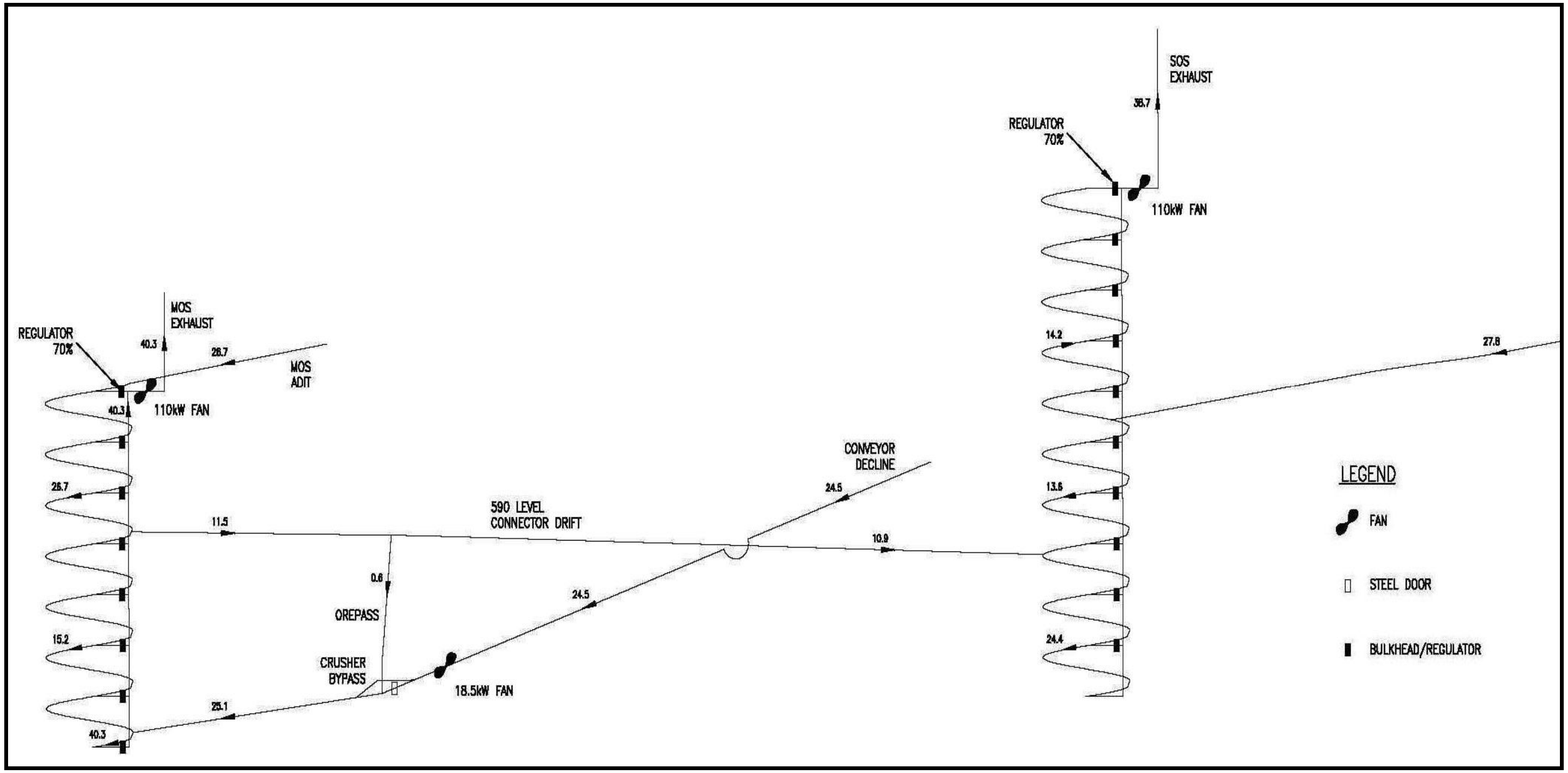
The exhaust raises will be developed incrementally as the mine deepens. A regulator or bulkhead will be installed in the access to the ventilation raise on each level once the next level is connected to the ramp. Regulators will control the air flow on each level throughout the mine life.

Once the exhaust raises are completed, exhaust fans will be installed in the upper crosscut of each raise. In the MOS this will be Level 641 and in the SOS Level 705. Each exhaust raise will have one twin 55kW vane axial fans and will exhaust approximately 40.3 and 38.7 m^3/s respectively. The bulkhead installation should include sufficient fan inserts for additional fans required at full production. During pre-production the conveyor decline may be used to provide fresh air. A total of 24.5 m^3/s will enter the conveyor decline. This will reversed once production begins.

These identical twin 55kW vane axial fans were chosen to facilitate simplicity in spare parts inventory and ensure that tuning of the fans is minimized. These multi-stage fans will need to have the same blade pitch. Dampeners will also be required to prevent recirculation in the event of a fan failure.

Double airlock doors will be installed at the crusher chamber to control the airflow during production. A bypass will be required to enable air to be exhausted out the conveyor decline using an auxiliary fan.

Figure 19.10 Pre-production Ventilation



Full Production Ventilation

The full production ventilation requirements are 168 m^3/s. During production, fresh air will travel through the spiral ramps. The North Ramp will intake 89.6 m^3/s at a velocity of 5.0 m/s. The South Ramp will intake 78.9 m^3/s at a velocity of 4.4 m/s. Each exhaust raise will have two twin 55 kW vane axial fans and will exhaust approximately 77.2 and 78.3 m^3/s for the MOS and SOS respectively. Regulators on each level, located in the access to the exhaust raise, will control the airflow during development and production cycles.

The conveyor decline will exhaust approximately 13 m^3/s of return air during production. The orepass and lower crusher access will also exhaust through the conveyor decline. The conveyor adit will be a mine exhaust to minimize the high risk of a conveyor fire polluting the underground mine workings.

The regulators on the exhaust shafts will control the ventilation on the connector drifts. For example, the regulator on Level 481 controls the 490 level connector drift. The regulator will be typically set at 70% to ventilate the connector drift through the life of mine. The full production ventilation is shown in Figure 19.11.

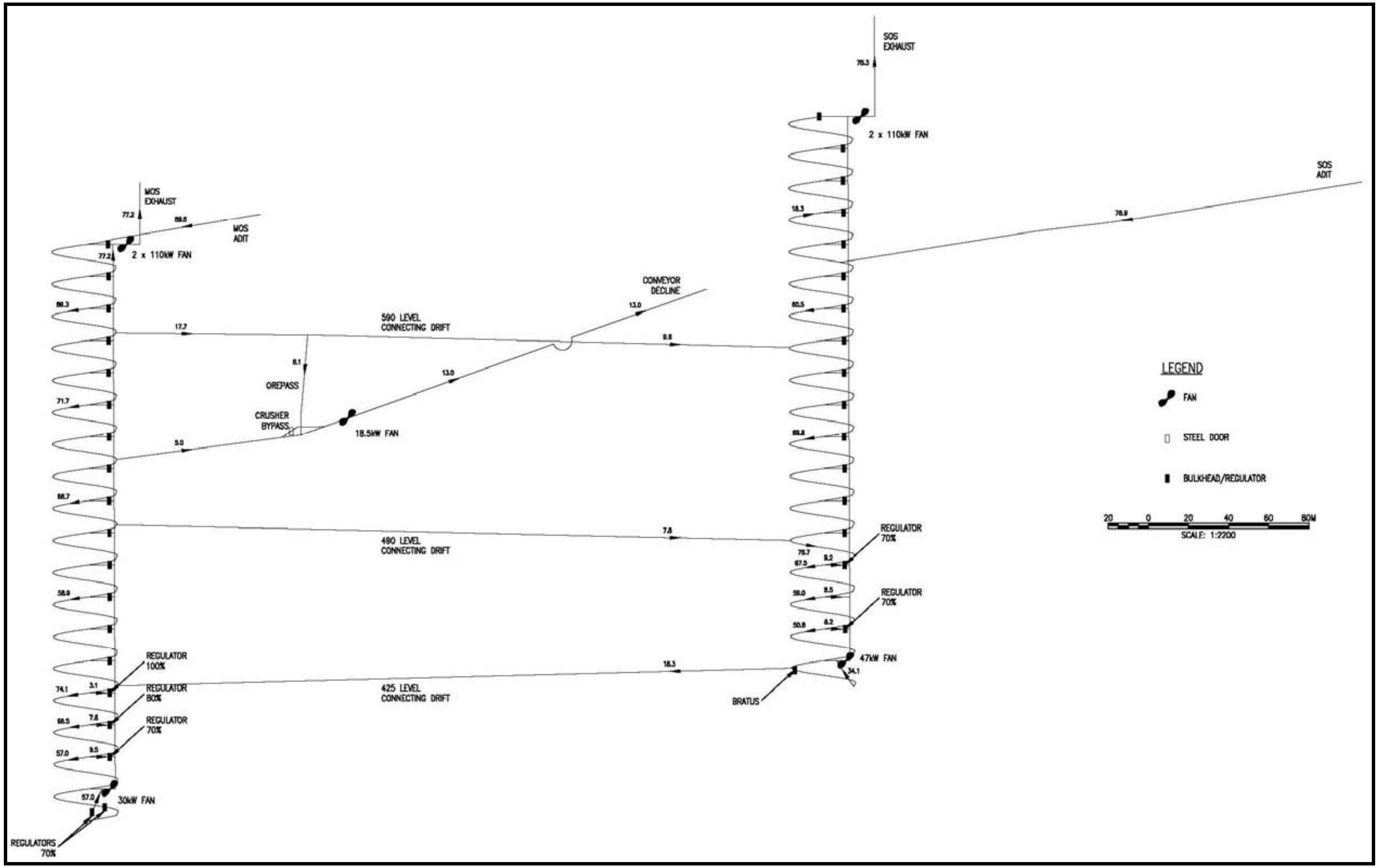
Secondary fans will be used on production levels to provide approximately 39 m^3/s for two trucks and one LHD or 53 m^3/s for three trucks and one LHD.

UNDERGROUND POWER AND ELECTRICAL DISTRIBUTION

The total Efemçukuru operating load is approximately 4,598 kW or 39,978 MWh/a. The underground mine operating load contributes approximately 2,027 kW or 17,754 MWh/a. Power will be required for drilling equipment including jumbos and drill rigs, ventilation fans, dewatering pumps and air compressors. The LHDs and articulated haul trucks will be diesel powered.

The regional 34.5 kV power grid will supply power to site. A sub-station near the concentrator plant will distribute power on site. Power will be stepped down to a lower voltage and fed underground through insulated cables. Underground power supply cables will terminate at disconnect switches providing total isolation of underground power in an emergency. Power will be distributed from the disconnect switches through cables to power centres located underground. All equipment and cables will be fully protected to prevent electrical hazards to personnel.

Figure 19.11 Full Production Ventilation



WATER AND MINE DRAINAGE

Initial studies on mine water inflow were estimated by Encon (April 2005) using their numerical hydrogeologic model of the site, and then by Golder (March 2006) using a numerical model that was based on the Encon model. Both models incorporated site data collected until 2005. Since that time, additional field investigations have been conducted near the proposed workings. From the recent work, a pumping test for predicting mine inflows was completed in well PW1 in March 2007.

The 2006 model was updated using the hydrogeologic data collected during this pumping test and then used to provide updated predictions of potential mine inflow. The Golder 2006 model was updated in May 2007.

The updated hydrogeologic model was used to predict groundwater inflow to the proposed mine, and to evaluate the potential impact of mine dewatering on groundwater regime. The progress of mining was simulated based on a mining schedule that was provided by Eldorado and used in the 2006 model.

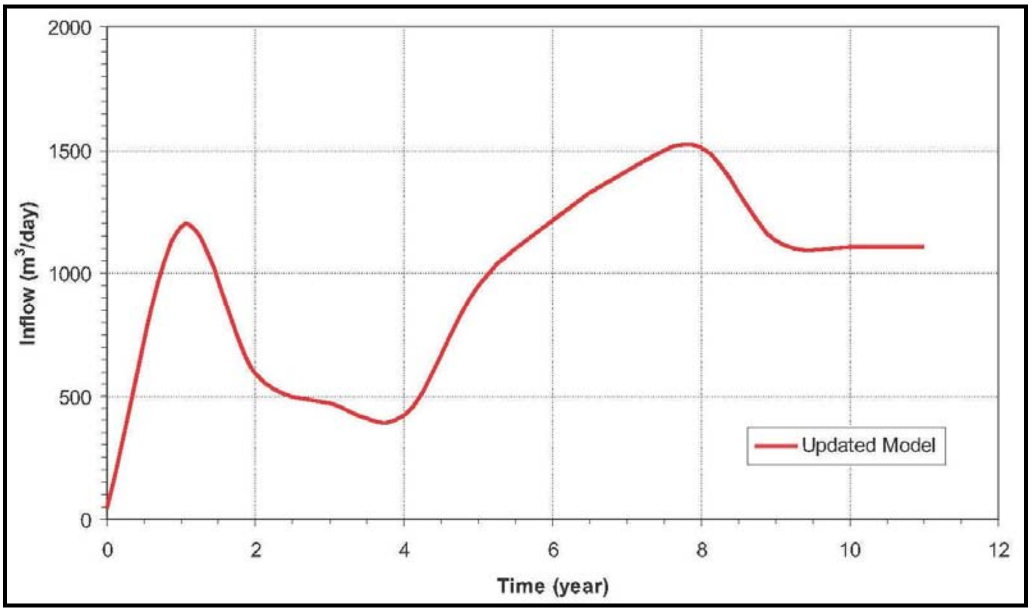
In summary, this mining schedule assumes:

- **Year 0:** access to the orebody via a decline at elevation of 580 m geodetic
- **Year 1 to 4:** mining at elevations above 580 m geodetic
- **Year 5 to 8:** mining between 490 m and 580 m geodetic
- **Year 9 to 11:** mining between 370 m and 490 m geodetic in the central portion of the orebody.

The updated model suggests that mine dewatering could result in a 2 m drawdown contour extending up to 1000 m west of the proposed mine. Drawdown is also predicted to occur east of Kokarpinar Creek. The mine inflow predicted by the calibrated model is presented in Figure 19.12.

Inflow is predicted to increase to approximately 1,200 m^3/d during decline and access construction in the first year of mining, and then gradually decrease to about 400 m^3/d in year 4 when mining occurs above the elevation of 580 m geodetic. Model results suggest that later in the mine life inflow would gradually increase to approximately 1,500 m^3/d when mining extends below 580 m elevation. This updated inflow is approximately four times greater than the inflow predicted by the 2006 model for the base case scenario (380 m^3/d), and approximately two times greater than the one predicted in the 2006 sensitivity simulation (670 m^3/d).

Figure 19.12 Mine Water Inflow



Groundwater inflows and water from development and production drilling will be collected in underground sumps. A permanent sump will be required at the lowest point on each ramp. Holding tanks will be used every 100 m to pump the mine water to surface. Submersible 150 kW electric pumps will be installed at each holding tank and at the permanent sumps at the bottom of each ramp. Mine water will be discharged via the dewatering pipelines to the surface treatment facility.

Auxiliary low-head pumps will deliver water from underground workings to the main dewatering lines. A 75 kW high-head pump will pump mine water to the surface treatment facility.

COMPRESSED AIR

Compressed air will be required for the following:

- development and production jumbo drilling
- production ITH drilling
- jackleg and stoper drilling
- explosive loading
- cleaning or dewatering blast holes with blowpipes
- shotcreting
- emergency ventilation at the refuge stations.

Two mobile air compressors each delivering 2,550 m^3/h at 0.70 MPa will be located on surface at the North 656 Portal and South 676 Portal. Compressed air will be distributed via steel piping with other mine services suspended in the upper corners of development and stope headings. A 200 mm diameter pipe will be required in the main ramps, with 100 mm to 50 mm diameter pipes in secondary headings and stopes.

EXPLOSIVES STORAGE AND HANDLING

ANFO will be the bulk explosive for underground production and development. Emulsion will be used for wet holes. During pre-production there will be blasting at anytime for the development headings. After the pre-production period all blasting will be at the end of each shift. All personnel underground will be required to be in a designated Safe Work Area during blasting.

Initially cap and powder magazines will be located near the surface on a drift off the North and South Ramps. Permanent installations will be installed on the 490 Level Connector Drift. The cap and explosive magazines will be installed approximately 30 m apart and have sufficient storage for one week of explosives. Transport of explosives underground will be by an underground flatbed logistics truck.

The explosive supplier will provide training for explosive handling and blasting.

DELIVERY OF SUPPLIES AND PERSONNEL TRANSPORTATION

Flatbed diesel-powered utility vehicles will move supplies including drill parts, explosives, and other consumables from surface to underground work areas. Two diesel-powered enclosed personnel carriers will transport the crews. Supervisors, engineers, geologists, surveyors, mechanics, and electricians share smaller diesel-powered vehicles.

COMMUNICATIONS

A leaky-feeder radio system will provide the primary communication underground. Supervisors and mobile maintenance crews will utilize hand held radios. The leaky-feeder radio system will be linked to the surface PABX system.

MAINTENANCE

Preventive maintenance encompasses all activities that prolong the life of equipment and reduce premature failures. Management of the preventive maintenance program will be implemented early in the mine life. Maintenance personnel underground will perform preventative and corrective maintenance work including adjustments, lubrication, and refuelling.

All major repair and maintenance on mining equipment including drills, loaders, and trucks will be performed on surface in the heavy vehicle workshop located between the mine dry and the concentrator plant. The maintenance planner on-site will develop maintenance schedules.

A number of specialized mechanics will be required to maintain and train Eldorado's employees for maintenance on primary underground development and production equipment.

FUEL STORAGE AND DISTRIBUTION

Diesel fuel will be delivered to the mine-site by road tanker and stored in fuel tanks buried on surface, compliant with local Turkish regulations. The storage fuel tanks will be installed on a concrete pad with concrete berms to prevent contamination in the event of a spillage. All bulk lubricants for operations will be stored in the warehouse.

Mine trucks hauling waste rock will be refuelled on surface. A lube-fuel truck with a 1,000 US gallon tank (3,785 L) will fuel LHD units, drills, and other underground diesel equipment not reporting to the surface each shift.

FIRE PROTECTION, SAFETY, AND MINE RESCUE

The North and South ramps will be designated fresh air escape routes. A total of three portable refuge stations will be required at full mine production; one on the North Ramp, one on the South Ramp and one on the 425 connector drift. These portable enclosures provide a self-contained atmosphere. The refuge station will provide oxygen at controlled rates, and will remove carbon dioxide from the air.

Mine rescue equipment and facilities will be maintained on the mine site. Two mine rescue teams will be trained with the necessary fire fighting and rescue skills. Detailed ventilation plans will need to be updated regularly for the mine rescue teams.

Fire extinguishers will be located at key infrastructure locations and at strategic points along each underground sub level. All underground miners will be trained in basic safety, first aid, and underground mine survival techniques. A stench gas system will be used to warn all employees of an emergency underground and will be installed at both the MOS and SOS portals.

Fire suppression systems will be fitted to all mobile equipment.

19.1.13 SURFACE TAILINGS AND DEVELOPMENT ROCK MANAGEMENT

The tailings area is sized for the disposal of 1,920,000 tonnes (approximately 1,200,000 m^3) of dry tailings, and occupies a footprint area of 62,300 m^2. The development rock storage area is sized for the disposal of approximately 544,000 tonnes of development rock (approximately 253,000 m^3), and occupies a footprint area of 22,700 m^2.

The feasibility-level tailings storage design includes the following elements:

- storage of dry stacked filtered tailings in a facility incorporating compacted tailings structural shells with 3H:1V outer slopes on the downstream and upstream sides of the facility to provide structural stability for the pile
- an underdrain and base liner system comprised of a fully-lined base, a central rock drain, and a toe drainage blanket to collect seepage from the tailings pile
- an engineered closure cover system comprised of a synthetic cover over the tailings, overlain by a 1 m-thick store and release soil cover system.

The feasibility-level development rock storage facility includes the following design elements:

- an overall 3H:1V (18°) slope contoured to promote drainage, reduce erosion and to provide long-term stability
- an underdrain and grouted/sealed foundation system to promote drainage of any collected mine rock pile seepage and limit infiltration into the underlying bedrock
- a closure cover system comprised of a synthetic cover over the mine rock, overlain by a 1 m-thick store and release soil cover system.

The detailed design for both storage facilities will be completed during a subsequent stage of the project. This design will incorporate information from the planned geotechnical site investigations in the area of the storage facilities, along with other relevant project information as it becomes available. The detailed design will also include detailed staging plans for waste disposal for performing concurrent reclamation during the mining operations, as well as coverage of other design factors such as:

- detailed tailings placement scheme
- NAG/PAG mine rock blending and placement scheme (if required)
- site foundation investigation findings
- site groundwater investigation including mapping of groundwater seeps/springs
- measurement of near-surface bedrock permeability
- assessment of tailings pile geotechnical stability
- assessment of development rock pile geotechnical stability

- detailed water balance for operations and closure
- detailed design and specification of the underdrain and liner systems
- design of run-on controls
- design of final covers and surface water drainage schemes
- design of sedimentation ponds for operations and closure
- design of monitoring systems and contingency actions.

19.2 Unit Operations & Process Metal Recoveries

The Efemçukuru gold ore contains a significant amount of free gold and is considered to be equivalent to a free-milling ore, despite the presence of sulphide minerals (mainly as pyrite). The process adopted to recover gold consists of crushing, grinding, flotation, concentrate regrinding and cyanide leaching, and direct electrowinning. Free gold will be recovered by gravity concentrators from the classification cyclone underflow and from flotation concentrate.

The following sections describe the unit operations followed by the projected process metal recoveries for both the Efemçukuru and Kişladağ operations.

19.2.1 Process Unit Operations

Efemçukuru Plant

The Efemçukuru plant consists of the following unit operations:

- underground ROM ore bin and primary crushing system with conveyor belts to transport the crushed ore to the crushed ore bins on surface
- belt feeders reclaiming ore from the crushed ore bins
- primary SAG milling with recycle pebble crushing, and secondary ball milling
- classification of the SAG mill discharge and flotation tailings streams to a final product of 80% passing 67 microns
- continuous primary gravity centrifugal concentration operating on a portion of the cyclone underflow in order to recover free gold
- flash flotation operating on the ball mill discharge to recover readily floatable free gold and coarse liberated sulphide mineral particles
- scavenger flotation of cyclone overflow to recover finer liberated gold and sulphide mineral particles
- upgrading of the flash flotation concentrate and scavenger flotation concentrate in the cleaner flotation circuit , together or separately, as required

- recovery of free gold by gravity concentration from the cleaner flotation concentrate
- recovery of free gold by gravity concentration from the primary gravity concentrates
- recleaner table concentration to upgrade the primary gravity concentrates
- recleaner table concentration to upgrade the cleaner flotation concentrates (optional process utilizing the same shaking table as used for the gravity concentrate)
- drying and smelting of the combined upgraded recleaner gravity and flotation concentrates
- thickening and pressure filtration of the concentrate consisting of the gravity tailings of the cleaner flotation and the primary gravity concentrates
- thickening and filtration of the flotation (plant) tailings for disposal as paste backfill or as dry stack tailings
- reagents and services.

KIŞLADAĞ CONCENTRATE TREATMENT PLANT

The leaching of the flotation concentrate received from Efemçukuru, and the subsequent recovery of the gold and silver, will be conducted at a process plant, which will be located at Kişladağ. This recovery plant will consist of the following unit operations:

- repulping of the flotation concentrate
- regrinding of the concentrate to 80% passing 20 microns
- cyclone classification
- pre-aeration of slurry prior to cyanidation
- cyanide leaching
- leach residue filtration
- leach residue solids disposal to the existing Kişladağ facility
- direct electrowinning of gold and silver from the pregnant solution
- smelting of metals to produce gold doré
- reagents and services with some commonality with the existing Kişladağ facility.

19.2.2 METAL RECOVERY

The projected metallurgical recovery values are given in Table 19.20.

Table 19.20 Projected Metallurgical Recovery Values

	Grade (g/t)		Recovery (%)
	Au	Ag	Au
Efemçukuru			
Feed	10.0	17.5	100.0
Doré	-	-	30.0
Flotation Concentrate	76.5	133.3	62.0
Tailings	0.87	1.52	8.0
Kişladağ			
Feed	76.5	133.3	100.0
Doré	-	-	91.2
Tailings	6.8	42.1	8.8
Overall Recovery			**86.5**

The Efemçukuru process will produce 38,700 oz of gold doré and 33,000 tonnes of concentrate at a head grade of 76.5 g/t. The concentrate is processed at Kişladağ to produce a further 73,000 oz of gold shown in Table 19.21.

Table 19.21 Projected Feed and Gold Production

		Tonnage	Gold Production
		t/a	oz
Efemçukuru Plant	Ore	401,500	-
	Doré bar	-	38,700
	Flotation Concentrate	33,000	-
Kişladağ Plant	Flotation Concentrate	33,000	-
	Doré bar	-	73,000
Total	**Doré bar**	**-**	**111,700**

19.3 MARKETS

19.3.1 GOLD MARKET

SUPPLY – DEMAND BALANCE

The market for the sale of gold is subjected to supply and demand trends as are most metals markets. Current forecasts of this trend suggest demand will continue to grow while supply will fluctuate as older mines are depleted and fewer world class deposits are brought into production. Although price demands will change over the life of the Efemçukuru mine, the nature of the project indicates that a swing to significantly lower prices will not jeapardize the operation.

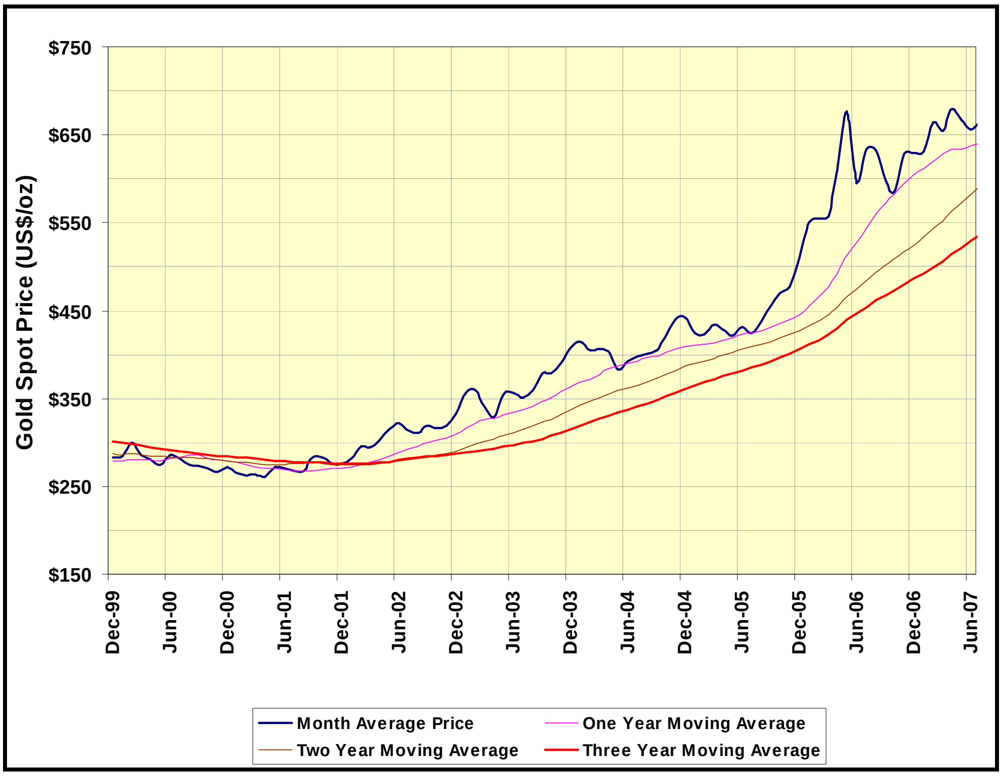
PRICE

Over the last five years, gold has been in a sustained bull market; the dollar price more than doubled during this period. Figure 19.13 shows the monthly gold spot price averages and moving averages from December 1999 to July 2007.

In 2005, prices followed an upward trend closing the year at US$513/oz. During 2006, gold broke through a series of multi-year highs, initially the US$500-level and later the US$600 and US$700-marks. The first 2006 milestone saw gold reach a 25-year high of US$572/oz in early February. This mark was drastically surpassed when gold achieved the 2006 price high of US$725/oz on May 12, 2006. At the end of the year, gold for immediate delivery traded at US$635.20 an ounce. Price support of US$600 continued into 2007 with a low of US$604.90 on January 4, 2007 to a high of US$692 on April 20, 2007. The price has levelled around US$650 into Q3 2007.

The three-year average has shown consistent growth since Q2 2002. The base case for the financial evaluation of the project used a gold price of US$530, slightly lower than the three-year average on August 1, 2007.

Figure 19.13 Gold Price



19.4 CONTRACTS

Other than a contract with IDC drilling for diamond core and RC drilling, no contracts have been entered into at this time.

Budgetary quotations have been assembled for key areas of construction and development to support the feasibility estimate as follows:

- budgetary contract terms for initial mine development have been provided by Yertas, a reputable Turkish mine contractor
- budgetary contract terms for shipping concentrate from Efemçukuru to Kişladağ have been provided by Ayhan Nakliyat Sanayi ve Ticaret A.Ş.
- budgetary contract terms for construction have been provided by Izmir Engineering.

Commercial contracts for goods and services will be prepared and issued by the EPCM contractor in accordance with Turkish regulations and laws and according to the procedures developed during the Kişladağ Phase 1 and 2 construction.

19.5 ENVIRONMENTAL CONSIDERATIONS

19.5.1 PROJECT DESCRIPTION

Eldorado Bold Corp. through its 100% owned Turkish subsidiary Tüprag metal Madencilik intends to develop and operate the Efemçukuru gold mine and ancillary processing facilities on its mineral licences near Efemçukuru village of Menderes District, Izmir Province, in the Aegean Region of Turkey. The reserve base of the operation is 3.785 Mt of ore, at a grade of 10.04 g/t Au. The total area of the project site will be approximately 73 ha and will be enclosed by a fence.

Over an expected life of 10 years, the project will mine and process approximately 401,500 tonnes of ore per year. The proposed underground mining methods include mechanized cut-and-fill and longhole stoping. Mine development rock will be excavated and stored on surface in engineered facilities designed to accommodate approximately 560,000 tonnes. The proposed beneficiation process aims to produce a gravity concentrate and a flotation concentrate. The flotation concentrate will be transported off-site by truck for further processing at Eldorado's Kişladağ site. Over the operating period of the project, tailings from ore processing will total about 3.5 Mt, of which 48% will be returned underground as backfill. The remaining tailings will be dewatered to about 10% moisture and stored on surface in engineered facilities.

Eldorado's comprehensive EIA)report, prepared by Encon in 2005, received an Environmental Positive Certificate from the MoEF in September 2005. This report determined that the proposed mine development will affect physical components of the existing environment at the site. The report also proposed an Environmental Management Plan (EMP) to prevent or mitigate any impacts of the project.

At the request of Eldorado, Wardrop conducted a review of the EIA report to identify any issues that could incur unforeseen environmental liability for the company. Each component and issue is summarized briefly below, focussing on the EMP and the conclusions about possible residual effects.

19.5.2 AIR QUALITY

Air quality on the site and in the surrounding rural area may be affected by project activity during construction and operation of the mine.

Emissions that would result from project activity and their effects on local air quality were evaluated by modelling studies. Ground level concentrations of NO_2, CO, HC, SO_2, and Pb in Particulate Matter and total Particulate Matter were estimated and these values were compared with limits given in the Turkish Air Pollution Control Regulation, 2004 (APCR). All predicted levels of these contaminants were below the APCR limits. Additional studies showed that dust concentrations measured at 3 m from the crushing, office, and filtered tailings disposal sources would be below limits set in the Turkish Industrial Originated Air Pollution Control Regulations, 2004. Particulate matter will also be measured regularly at construction areas, along truck routes, and near settlements. Mitigation measures, such as the use of dust filters, road paving, and water spraying on haul roads, at the crushers, and conveyors will be undertaken to ensure compliance with APCR limits. After mine closure and restoration of the site, no residual sources of particulate or gaseous emissions will exist.

19.5.3 WATER QUALITY

The water component of the local environment at the project site includes groundwater and surface runoff. Baseline surface water quality was well documented in the EIA. Surface water contained insignificant traces of fertilizer and pesticides but fecal coliform and biological oxygen demand levels were above regulatory norms and indicated pollution by agricultural runoff. No industrial sources of water pollution exist in the area.

With mine development, several changes in water quantity and quality will occur. Runoff from precipitation and emergent springs on the site will be collected, channelled around the engineered disposal sites for tailings and uneconomic rock and held in a settling pond for use as process water. Excess water will be discharged to the Kokarpinar Stream.

Domestic waste water emanating from the project will be treated by an RBC and tile field and will comply with Turkish Water Pollution Control, 1987 (WPC) regulations. This source will cease to exist post-closure.

Underground mine workings will receive groundwater inflows and seepage. Some of this inflow may contact acid generating rock and consequently contain dissolved metals.

This water will be pumped to the surface and monitored for contaminants. If it does not meet regulatory criteria or process water requirements, it will be directed to a settling pond and treated before being used as process water. The pond will operate with 0.6 m freeboard and will be sized to contain an additional 1-in-100 year 24-hour storm event.

The filtered tailings were tested and are predicted to be non-acid generating and therefore contact and underdrain water will not generate ARD post-closure.

Contact and underdrain water from the development rock dump area may contain levels of metals non-compliant with WPC Regulations. It will be monitored throughout the life of the project and treated similarly to mine drainage water before being used as process water or discharged as surface drainage. Of all the issues considered herein, the quality of uneconomic rock dump runoff has the most potential to be of concern post-closure. However, as discussed below, the expected volume is small and the neutralizing potential of 80% of the blended development rock will be sufficient to neutralize any ARD generated within the dump over the long term.

The volume of potentially non-compliant drainage is expected to be small. This expectation is based on the following information and assumptions, derived from the EIA and humidity cell test results subsequently provided by Eldorado:

- average annual precipitation at Efemçukuru: 877.0 mm
- average annual evaporation: 435.1 mm
- net annual average precipitation: 441.9 mm
- surface area of development rock dump: 22,700 m^2
- volume of precipitation on dump: 0.4419 m x 22,700 m^2 = 10,031 m^3/a
- amount of development rock in dump at closure: approximately 540,000 tonnes
- potentially acid generating (PAG) development rock comprises 18% or: 0.18 x 560,000 = 100,800 tonnes.

Given the diversion of surface runoff and on-site spring discharges away from the dump, cell-by-cell macro-encapsulation of the PAG rock and establishment of a substantial vegetated soil cap, as proposed, conservatively assumes that at most only 25% of precipitation infiltrates the development rock dump soil cover. This would equal 0.25 x 10,031 = 2,507 m^3/a. Since only 20% of the development rock is PAG, the volume of potentially non-compliant contact water would be about (0.2 x 2,507) = 502 m^3/a, or 1,375 L/d, on average

Acid-base-accounting, short-term leachability tests, and long-term humidity cell tests of PAG development rock indicated that the leachate is weakly acidic and that the neutralization potential of 80% of the development rock greatly exceeds the acid generating potential of the remaining 20%. It is suggested that selectively stacking the 20% of PAG development rock over and in contact with the non-PAG rock will utilize the available excess NP and reduce the dissolution of a hazardous material (As) in the residual underflow or seepage from the dump.

19.5.4 LAND USE

Construction and operation of the mine will change the present land use of the site for at least its 10-year life. Present agricultural land use will be reduced and owners will be compensated by agreement with the owners or lessees for lost production. Government owned forest will be cleared under permit from the work area. Vegetation will be removed from approximately a third of the project area for building sites, processing facilities, haul roads, and waste disposal dumps. Where vegetation is removed, the topsoil will be stockpiled in reserves for use in future rehabilitation.

Progressive reclamation of the waste storage areas will be undertaken. As the waste storage areas are filled on a cell-by-cell basis, soil cover will be applied from the stored reserve and a vegetation cover will be planted using native species. At closure, only the last waste cells, the demolished building sites and roadways will remain to be re-contoured, covered with the stored topsoil and seeded, or replanted with native species.

The success of re-vegetation will be monitored throughout the life of the project and a limited post-closure period, allowing for supplementary seeding or plantings if required. Adherence to the proposed land use management plan will ensure that the mix of plant species on the site will be restored and soil erosion prevented.

Reversion of the project site to its current uses will depend on socio-economic conditions prevailing after closure and restoration.

19.5.5 Flora and Fauna

With regard to plant and animal species of concern, the Efemçukuru Project site is not designated as a protected habitat area under Turkish or international law. However, several species are listed as being under one or more conservation guidelines and 63 animal species of international concern and under some degree of protection occur on the site. Of the 218 plant species identified, most are globally distributed, 15 are endemic to Turkey but comprise an insignificant proportion of all plants listed. Only one species was identified as under threat, but it is not endemic to Turkey.

The baseline vegetation studies for the EIA were carried out in September, March, April, and May, representing all seasons, and were well documented. This plant community information will be used to design the pattern for re-vegetating the project site during operations and after closure. As well, adjacent areas, undisturbed by the project, will provide a reservoir of local species for the natural re-population of the plant communities.

Wildlife species (mammals, birds, reptiles, and amphibians) were identified and enumerated during four visual surveys of the three habitat types on the site: forest, degraded forest, agricultural land, and riparian (streamside) habitat. Altogether, there were eight mammal, 62 bird, seven reptile, and one amphibian species recorded. None are endemic (restricted) to the local area or severely threatened by habitat loss.

While the project will displace some species through human presence and activity, including the generation of noise and traffic and habitat removal, theses species will be able to return to the restored site after closure. The small total area of disturbance; about a third of the licensed 73 ha site and the relatively short duration

(10-year) of the entire project, will likely lead to natural recovery of the plant and animal populations at the site.

19.5.6 APPROVALS AND PERMITS

The MoEF is now the sole body responsible for the EIA, the Environmental Positive Certificate, and Site Selection Permit. The company completed the EIA study at Efemçukuru in May 2005 and received an Environmental Positive Certificate in September 2005. At that point, the government had not passed a regulation regarding the mine permitting process. As of October 2006, various individual permits were still being negotiated but there have been no roadblocks identified to the proponent. Preliminary engineering is underway.

19.5.7 CONCLUSIONS

The following conclusions were drawn from this review:

- air quality is not an issue
- water and mine waste water quality is adequately addressed and mitigated in the project design
- given the adequate restoration of wildlife habitat including re-vegetation with the species now present and removal of the 750 m culvert on Kokarpinar Creek, natural repopulation by displaced species will take place and post-closure liability for terrain and wildlife and habitat restoration is unlikely
- at this time, no roadblocks to project permitting have been identified to Eldorado by the Turkish Government.

19.6 CAPITAL AND OPERATING COST ESTIMATES

19.6.1 CAPITAL COST ESTIMATE

The capital cost for mining, processing, and infrastructure is estimated at US$104.2 million. The capital cost estimate is considered accurate to ±15%/-5%. This is suitable for project appropriation, project financing, and establishing the cost basis for the EPCM phase of the project. A summary of the capital cost estimate is shown in Table 19.22. The cost estimate has been prepared in 2nd Q 2007 US dollars.

Table 19.22 Capital Cost Estimate Summary

Area Code	Area Description	Manhours	Labour ($)	Materials ($)	Construction Equipment ($)	Process Equipment ($)	Cost ($)
A0	**OVERALL SITE**						
A1	Access road	6,218	112,820	160,800	81,350	-	354,970
A2	Creek diversion	1,100	16,500	15,000	18,750	-	50,250
A3	Power supply – Tecmar	35,105	702,100	1,659,075	362,555	405,740	3,129,470
A4	Power distribution	6,975	138,800	1,219,370	114,890	3,617,950	5,091,010
A5	Plant site control system	3,884	88,480	186,301	3,955	646,912	925,648
A6	Communication system	4,175	83,500	73,000	9,650	390,000	556,150
A7	Fire alarm system	900	18,000	95,000	5,250	-	118,250
A8	Fencing/perimeter & miscellaneous	3,944	78,880	62,300	16,060	-	157,240
Area A	**Site Subtotals**	**62,301**	**1,239,080**	**3,470,846**	**612,460**	**5,060,602**	**10,382,988**
B0	**MINING**						
B1	Mining equipment	13,700	274,000	1,252,530	67,440	6,171,406	7,765,376
B3	Underground development	4,171	1,414,481	3,622,133	3,020,335	-	8,056,949
B4	Mine plan	56	1,120	171,140	40	161,190	333,490
B5	Explosive storage area	434	8,070	157,520	595	12,000	178,185
Area B	**Mining Subtotals**	**18,361**	**1,697,671**	**5,203,323**	**3,088,410**	**6,344,596**	**16,334,000**
C/D/E	**PROCESS**						
C1	Primary crushing area	16,609	332,187	482,032	66,829	713,611	1,594,659
D0	Crushed ore storage/reclaim	17,637	352,738	745,898	46,647	454,260	1,599,543
E0	Concentrator building	37,857	757,130	1,274,217	72,609	176,830	2,280,786

table continued…

Area Code	Area Description	Manhours	Labour ($)	Materials ($)	Construction Equipment ($)	Process Equipment ($)	Cost ($)
E1	Grinding & classification	25,403	508,117	479,061	28,826	3,254,229	4,270,233
E2	Gravity concentration	2,276	45,511	77,910	4,941	160,354	288,716
E3	Pebble crushing	6,250	124,990	98,297	9,648	389,919	622,854
E4	Flotation	11,264	225,280	269,064	16,047	1,503,963	2,014,353
E5	Concentrate dewatering/load-out	8,829	176,607	189,862	11,949	627,006	1,005,423
E6	Reagents & plant services	8,953	179,066	248,379	13,601	262,800	703,845
E7	Gold room	6,384	131,288	161,242	13,527	618,570	924,626
Area C-E	**Process Subtotals**	**141,462**	**2,832,914**	**4,025,962**	**284,624**	**8,161,542**	**15,305,038**
F0	**TAILINGS & WASTE DISPOSAL**						
F1	Tailings thickening	5,741	114,820	253,980	26,072	268,569	663,441
F2	Tailings filtration/paste b/fill	26,895	537,935	854,742	48,836	1,728,748	3,170,261
F3	Dry tailings containment	9,025	180,500	669,530	63,950	0	913,980
F4	Mine waste storage	3,991	79,820	171,900	61,607	0	313,327
Area F	**Tailings Subtotals**	**45,652**	**913,075**	**1,950,152**	**200,465**	**1,997,317**	**5,061,009**
G0	**WATER TREATMENT FACILITY**						
G0	Site drainage & catchment	2,552	51,035	33,300	58,665	0	143,000
G1	Water treatment facility	16,128	342,550	259,941	24,000	1,199,943	1,826,434
Area G	**Water Treatment Subtotals**	**18,680**	**393,585**	**293,241**	**82,665**	**1,199,943**	**1,969,434**
J0	**ANCILLARY BUILDINGS**						
J1	Administration building	5,700	114,000	255,000	10,500	25,000	404,500
J2	Mine dry & canteen	7,090	141,800	198,700	13,400	20,000	373,900

table continued…

Area Code	Area Description	Manhours	Labour ($)	Materials ($)	Construction Equipment ($)	Process Equipment ($)	Cost ($)
J3	Truckshop & warehouse	15,193	303,860	488,948	33,825	419,835	1,246,468
J4	Lube storage	-	-	-	-	-	-
J5	Security/gatehouse/first aid	924	18,480	42,715	1,206	10,000	72,401
J6	Assay/environmental/met laboratory	5,946	120,120	101,061	9,652	350,000	580,833
Area J	**Ancillary Subtotals**	**34,853**	**698,260**	**1,086,424**	**68,583**	**824,835**	**2,678,102**
K0	**SITE SERVICES**						
K1	Site preparation & roads	20,396	407,930	153,020	574,812	-	1,135,762
K2	Sewage collection/treatment	1,653	33,060	43,030	14,646	140,000	230,736
K3	Fresh water supply	2,375	47,500	37,320	20,300	24,000	129,120
K4	Fire/fresh water storage/distribution	20,211	404,215	470,233	63,100	140,367	1,077,915
K5	Fuel storage area	1,193	23,862	26,059	3,110	53,500	106,531
K6	Yard lighting	630	12,600	183,000	984	-	196,584
K7	Plant site mobile fleet	454	9,080	1,700	330	851,772	862,882
Area K	**Site Services Subtotal**	**46,912**	**938,247**	**914,362**	**677,282**	**1,209,639**	**3,739,530**
TOTAL	**EFEMÇUKURU DIRECT COSTS**	**368,221**	**8,712,832**	**16,944,310**	**5,014,489**	**24,798,474**	**55,470,101**
P0	**KIŞLADAĞ EXPANSION**						
P1	Concentrate rehandling & milling	13,062	547,840	1,413,004	42,981	1,301,959	3,305,784
P2	Cyanide leaching & leach residue	24,324	486,489	571,857	47,708	1,233,069	2,339,123
P3	Gold room	9,227	188,146	462,897	17,980	1,592,083	2,261,106
P4	Reagents	2,715	54,306	63,513	4,957	103,011	225,787
P5	Services	4,482	89,630	110,557	9,343	136,254	345,784

table continued…

Area Code	Area Description	Manhours	Labour ($)	Materials ($)	Construction Equipment ($)	Process Equipment ($)	Cost ($)
TOTAL	**KIŞLADAĞ DIRECT COSTS**	**53,810**	**1,366,411**	**2,621,828**	**122,969**	**4,366,376**	**8,477,583**
TOTAL	**PROJECT DIRECT COSTS**	**422,031**	**10,079,243**	**19,566,138**	**5,137,458**	**29,164,850**	**63,947,684**
N	**INDIRECTS**						
Efemçukuru							
N1-1	Construction indirects	46750	1,257,250	2,363,500	163,000	-	3,783,750
N2-1	Spare parts	-	-	-	-	1,586,250	1,586,250
N3-1	Initial fills & w/house inventory	-	-	435,000	0	-	435,000
N4-1	EPCM	65340	7,353,900	1,680,000	230,000	-	9,263,900
N5-1	Freight	0	0	2,101,547	70,000	3,519,344	5,690,891
Kişladağ							
N1-2	Construction indirects	11290	377,000	534,000	50,500	-	961,500
N2-2	Spare parts	-	-	-	-	218,319	218,319
N3-2	Initial fills & warehouse inventory	-	-	55,000	-	-	55,000
N4-2	EPCM	10060	1,096,100	266,000	30,000	-	1,392,100
N5-2	Freight	0	0	369,746	17,500	508,007	895,253
TOTAL	**PROJECT INDIRECT COSTS**	**133,440**	**10,084,250**	**7,804,793**	**561,000**	**5,831,920**	**24,281,963**
Y1	**OWNER COSTS**						
Y1-1	Owners costs (Efemçukuru)	109,600	2,364,000	884,298	96,000	-	3,344,298
Y1-2	Owners costs (Kişladağ)	16,000	420,000	224,000	32,000	-	676,000
TOTAL	**Owner Costs**	**125,600**	**2,784,000**	**1,108,298**	**128,000**	**-**	**4,020,298**
TOTAL	**PROJECT INDIRECT & OWNER COSTS**	**259,040**	**12,868,250**	**8,913,091**	**689,000**	**5,831,920**	**28,302,261**

table continued…

Area Code	Area Description	Manhours	Labour ($)	Materials ($)	Construction Equipment ($)	Process Equipment ($)	Cost ($)
TOTAL	**DIRECT/INDIRECT COSTS**	**681,071**	**22,947,493**	**28,479,229**	**5,826,458**	**34,996,770**	**92,249,945**
Z	**CONTINGENCY**						
Z1	Efemçukuru direct costs	56,546	1,433,957	2,818,140	933,431	2,828,324	8,013,851
Z2	Kişladağ direct costs	8,072	202,246	390,098	18,197	436,638	1,047,180
Z3	Efemçukuru indirect/owner costs	22,169	903,115	785,482	53,000	686,527	2,428,124
Z4	Kişladağ indirect & owner costs	3,735	189,310	163,362	13,875	98,033	464,580
Z5	Escalation (not included)	-	-	-	-	-	-
Total	**Contingency (Average 11%)**	**90,522**	**2,728,628**	**4,157,082**	**1,018,503**	**4,049,522**	**11,953,734**
Total	**Project Costs**	**771,593**	**25,676,121**	**32,636,311**	**6,844,961**	**39,046,292**	**104,203,680**

BASIS OF ESTIMATE

The capital cost estimate is developed from engineering first principals and current construction costs in Western Turkey. The following costing information is included:

- project work breakdown structure (WBS)
- design criteria, flowsheets, and equipment list
- piping and instrumentation diagrams
- detailed site and facility general arrangement drawings
- single line diagrams
- multiple quotations for all "tagged" equipment
- Turkish vendor and contractor information
- Kişladağ historic and current construction costs
- in-house costing database
- preliminary engineering
- material take-offs by discipline.

During the feasibility study, Wardrop visited the proposed Efemçukuru mine site. Potential suppliers, contractors, and fabricators in Izmir were also visited. The equipment, material, and construction costs supplied by local contractors and fabricators are included in the estimate. Labour rates were also supplied to Wardrop.

Wardrop personnel inspected Eldorado's Kişladağ site in western Turkey. Historic and current construction costs were collaborated for the estimate.

The estimate assumes that all material and equipment will be purchased new on a competitive bid basis.

ESTIMATE STRUCTURE

The estimate is assembled and coded based on the approved project WBS. The WBS is a hierarchical roll up structure of project areas and discipline codes described below.

Project Areas

The following project areas are utilized in the development of the WBS:

A0 – Overall Site
A1 – Access Roads
A2 – Creek Diversion
A3 – Power Supply
A4 – Power Distribution
A5 – Control Systems
A6 – Communications
B0 – Mining
B1 – Mining Equipment
B2 – Mining Structures
B3 – Underground Development
B4 – Mine Plan
B5 – Explosives Storage Area
C1 – Primary Crushing
G1 – Water Treatment Facility
J0 – Ancillary Facilities
J1 – Administration Building
J2 – Mine Dry & Canteen
J3 – Truck shops && Warehouse
J5 – Security & Gatehouses
J6 – Laboratory
K0 – Site Services
K1 – Site Preparation && Road
K2 – Sewage Collection && Treatment
K3 – Fresh Water Supply
K4 – Fresh/Fire Water Storage & Distribution
K5 – Fuel Storage Area
K6 – Yard Lighting
K7 – Plant Mobile Fleet
D0 – Crushed Ore Storage && Reclaim

E0 – Concentrator Building
E1 – Grinding & Classification
E2 – Gravity Concentration
E3 – Pebble Crushing
E4 – Flotation
E5 – Concentrate Dewatering & Loadout
E6 – Reagents & Services
E7 – Gold Room
F0 – Tailings & Waste Disposal
F1 – Tailings Thickening
F2 – Tailings Filtration & Paste Backfill
F3 – Dry Tailings Containment
F4 – Mine Waste Storage
G0 – Site Drainage & Catchment
M0 – Temporary Services
M1 – Construction Services
N1 – Construction Indirects
N2 – Spare Parts
N3 – Initial Fills & Warehouse Inventory
N4 – EPCM
P0 – Kişladağ
P1 – Concentrate Reh&ling && Milling
P2 – Cyanide Leaching && Leach Residue Dewatering
P3 – Gold Room
P4 – Reagents
P5 – Services

Y1 – Owner Costs
Y2 – Work by Others
Y3 – Project Exclusions
Z1 – Contingency

Discipline Codes

Each area of the estimate is shown under the following discipline codes:

09 – Process Equipment	19 – Civil Site Services including minor Earthworks
11 – Architectural	20 – Major Earthworks (Dams, General Grading)
12 – Mechanical & Plate work	21 – Mobile Equipment
13 – Piping	40 – Mining
14 – Buildings including Services	43 – Mining Underground
15 – Concrete	44 – Mining Equipment
16 – Structural Steel	45 – Mining Adit-Drift Development
17 – Instrumentation	47 – Mining Ventilation
18 – Electrical	49 – Mining Paste Distribution

PRICING AND QUANTITY

General

All quantities are developed from engineering first principals. Design allowances for bulk materials are based on discussions between the respective discipline and the estimator.

Equipment and Materials

The estimate assumes that all equipment will be purchased new. In almost all cases, the process equipment will be imported. Detailed and comprehensive equipment specifications were prepared and issued for bid to qualified vendors for budgetary quotations. The vendors supplied an equipment price, delivery lead times, freight costs to marshalling yard and a spare parts allowance. In some instances vendors provided estimates for installation hours for specified equipment.

All equipment and material costs are Free Carrier (FCA) or Free On Board (FOB) to the manufacturer plant. The cost of spare parts, taxes, duties, freight, and packaging are included in the indirect costs in the estimate.

Items valued under US$50,000 are priced from information gathered from site visits or in-house data from similar projects.

Bulk materials will be supplied within Turkey. Bulk material budget prices are from local vendors, information from site visits and in-house data from similar projects.

Mining

Underground mining quantities are based on detailed mine plans. Underground pre-production development costs are based a quotation received from a qualified Turkish mining contractor.

Bulk Earthworks

Bulk earthworks quantities are based on rough grading designs. Excavation of topsoil and allowance for rock excavation is based on the geotechnical information available at the time of the study. Structural fill is cost based on aggregates being produced at site utilizing a portable crushing and screening plant; the price for relocating and refurbishing the aggregate plant from Kişladağ is included in the capital cost estimate. Earthwork quantities do not include an allowance for bulking or compaction of materials, these allowances are included in the unit prices.

Concrete

Concrete quantities are based on “neat” line quantities from engineering designs and sketches with a 10% allowance made for over-pour and wastage. Designers have provided quantities to the estimator, including:

- lean mix leveling concrete
- footings and foundations
- retaining walls
- grade beams and pedestals
- perimeter walls
- slab on grade
- elevated slabs
- pads curbs and sumps
- imbedded metals and anchor bolts.

WBS defined quantities were calculated by area. Unit rates for each type of work include formwork, reinforcing steel, placement, and finishing of concrete. Cement costs are based on redi-mix concrete delivered to site by local contractors.

Structural Steel

Steel quantities are developed from engineering designs with 5% contingency for growth and wastage. Contingency is included for cut-offs, bolts, and connections.

Designers utilized the following steel classification and units:

- light weight steel (0 to 30 kg/m sections)
- medium weight steel (31 to 60 kg/m sections)
- heavy steel (61 to 90 kg/m sections)
- extra heavy steel (> 90 kg/m sections)
- platform framing (tonnes)
- stairways (tonnes)
- grating (m^2)
- handrail (m).

Fabricated steel in Turkey is currently available at substantially lower prices than world markets, due to excess production capacity of local steel mills. This has presented a significant capital cost opportunity for the project. Recent fabricated steel costs were confirmed by local fabricators and acquired from Kişladağ construction costs.

Platework and Liners

All platework and metal liners for tanks, launders, pump boxes, and chutes have been calculated from detailed quantity takeoffs, developed from designs, and provided in kilograms of steel. Rubber lining for pump boxes has been provided on a square meter basis. A 10% allowance for growth and wastage has been included. Sample tank, pump box, and platework drawings were issued to an Izmir based fabricator for pricing of both shop fabrications and field-erected tanks.

Heating, Ventilation, and Air Conditioning (HVAC)

Where appropriate, HVAC is based on a cubic metre cost calculated from in-house data based on building function and site-specific climatic conditions.

Dust Collection/Suppression

Wardrop designed and sourced dust suppression and collection equipment for required items according to industry norms for dusty environments.

Piping

Piping quantities are based on detailed quantity take-offs for pipe over 3" diameter, including pipe lengths and fittings. The quantity take-offs are developed from pipe routing ‘red line’ drawings based on the detailed general arrangement drawings and the P&IDs. Piping is provided as separate line items, sorted by WBS area and pipe specification. Special piping including stainless steel is listed separately; flanges and bolt-ups are included. Allowances are included for supports, painting, and tagging.

Valves

All valves are listed as separate line items in the estimate.

Electrical

The electrical engineers data program includes total electrical costs for each piece of equipment including cables. These costs are categorized under the WBS structure and based on single line diagrams, project drawings, and sketches. Electric motors are included in the vendor packages and all electrical rooms are modularized. The incoming power line development cost is based on an estimate from Tekmar, an Izmir based electrical contractor. Separate line items are included for communications and fire alarm systems.

Instrumentation

Instrumentation is included as part of the design data program. The total instrument cost for each piece of equipment, including cables, is based on project drawings and sketches. The most competitive technically compliant bid was included in the estimate.

Buildings

Specifications and architectural designs were developed for each building. Process buildings are stick built structural steel buildings; detailed designs and quantities were prepared in accordance with the structural steel methodology outlined above.

The ancillary buildings are constructed of concrete frame and blockwork fill in accordance with Turkish standard construction methods. The buildings are estimated on a square metre basis, derived from current design/build construction costs obtained from Kişladağ, which includes civil, foundations, structural, and services.

ESTIMATE CURRENCY

The currency used for the estimate is United States Dollars (US$). The capital estimates were categorized by regional supply, shown in Table 19.24. Approximately 50% of all costs are based on Turkish supply.

Table 19.23 shows the relevant foreign exchange rates. The foreign exchange rates are current at 01 August 2007.

The capital estimates were categorized by regional supply, shown in Table 19.24. Approximately 50% of all costs are based on Turkish supply.

Table 19.23 Foreign Exchange Rates

Base Currency	Foreign Currency
US$1.00	Cdn$1.13
US$1.00	EUR 0.75
US$1.00	YTL 1.36

Table 19.24 Capital Estimate Regional Supply Summary – US$

Area	Turkish				European		Canadian		American		Totals
	Labour	Materials	Construction	Equipment	Labour	Equipment	Labour	Equipment	Labour	Equipment	
Direct Costs											
Efemçukuru	8,713,000	15,787,000	5,014,000	1,169,000	-	22,308,000	-	2,064,000	-	414,000	55,470,000
Kişladağ	1,366,000	2,622,000	123,000	943,000	-	2,139,000	-	-	-	1,285,000	8,478,000
Indirect Costs											
Efemçukuru	1,295,000	4,975,000	463,000	-	538,000	5,131,000	6,683,000	1,565,000	96,100	15,000	20,760,000
Kişladağ	385,000	967,000	98,000	-	71,900	746,000	848,000	219,000	169,000	19,000	3,522,000
Owner Costs											
Efemçukuru	1,106,000	572,000	96,000	-	-	-	1,258,000	312,000	-	-	3,344,000
Kişladağ	60,000	56,000	32,000	-	-	-	360,000	168,000	-	-	676,000
Contingency											
Efemçukuru	1,560,000	3,413,000	986,000	116,000	53,700	3,357,000	714,000	190,000	9,630	41,400	10,442,000
Kişladağ	240,000	504,000	32,100	94,300	7,120	310,000	127,000	49,000	17,000	131,000	1,512,000
Total	**14,725,000**	**28,897,000**	**6,845,000**	**2,322,000**	**670,000**	**33,991,000**	**9,990,000**	**4,568,000**	**292,000**	**1,905,000**	**104,204,000**

LABOUR RATE

An average labour rate of US$20.00 per hour is used in the estimate. Labour rates are based on information from Kişladağ construction records and discussions with Turkish construction companies.

Labour rates do not include allowance for delays caused by industrial relation issues or unforeseen disruptions.

Labour Rate Calculation

The labour rates include:

- base rate
- payroll burdens including pension, health premiums, employment insurance and vacation pay
- overtime shift premium rates with allowance for incidental overtime
- minimal allowance for expatriate labour
- small tools and consumables
- applicable local labour taxes
- contractor overhead and profit.

Contractor overheads include:

- field supervision including managers and general foremen (supervision)
- general and administrative including office supplies, vehicle costs, and office personnel
- transportation, accommodation, and meals
- sub-contractors mark-up.

Table 19.25 shows the development of the labour cost.

Table 19.25 Labour Rate Calculation

Item	Rate (%)	Value (US$)
Base Rate	-	7.00
Overtime Allowance	25	1.75
Subtotal	**-**	**8.75**
Taxation	18	1.58
Social, Pension, Health & Unemployment Benefits	34	2.98
Overheads and Profit	55	4.81
Supervision	20	1.75
Total (Rounded to US$20.00 for Estimate)		**19.87**

Scheduled site hours are 6 x 10 hour days; a total of 60 hours per week. Roster rotation is a 6 weeks in, 2 weeks out schedule for expatriates. A productivity factor of 1.3 is applied to North American labour hours.

PROJECT SCHEDULE

The estimate is based on key milestone project dates below.

Award Long Lead Equipment..	November 2007
Award Detailed Engineering	December 2007
Award Pre-Production Mining Contract	December 2007
Award the access road construction	December 2007
Begin Detailed Engineering	January 2008
Receive Permits and begin Site Earthworks	June 2008
Mobilize Aggregate Plant and Batch Plant	July 2008
Mobilize Pre-Production Mining Contractor	July 2008
Begin Concrete Placement on Site	August 2008
Begin Process Building Erection.....................................	October 2008
Begin Process Equipment Installation..............................	November 2008
Mechanical Completion	July 2009

PROJECT INDIRECTS

The total project indirects are US$24.3 million or 23% of the initial capital cost. This includes temporary facilities, construction Indirects, incidental overtime, freight, spare parts, EPCM, contracted work, and contingency.

Temporary Facilities

Temporary facility costs are allocated by percentage of direct costs. These include facilities supplied by Eldorado not included in contractor prices, which are not permanent and will be demobilized after construction.

Temporary facilities for construction include, but are not limited to the following:

- access roads
- power (including contractor requirements)
- barricades and security fencing
- lighting
- water supply

- toilets and sewage
- communication infrastructure
- general waste disposal (contractors will be responsible for work area and to deliver to common collection area)
- solid waste and hazardous waste disposal
- miscellaneous cranes
- miscellaneous equipment rentals
- mobilization and demobilization including labour, materials, and equipment
- warehousing and associated labour
- laydown areas
- warehouse fencing and gates
- site surveying.

Construction Indirects

Construction indirects include:

- doctor on call
- other medical and first aid
- bus transportation and personnel carriers
- personnel turn-around costs (senior Turkish personnel, expatriates, and 3rd country nationals)
- personnel travel time to site each shift
- personnel accommodation and meal costs
- safety including safety officer, equipment, and vehicle
- quality assurance and control
- security.

INCIDENTAL OVERTIME

Incidental overtime covers person hours required in addition to the normal 60-hour week. Incidental overtime will be required for complex or large concrete pours.

Freight

Freight allowances to Efemçukuru are included for equipment and materials being delivered to site. The allowances by region are:

- North America: 12%
- Europe or South America: 10%
- Turkey: 5%

Freight allowances to Kişladağ are included for equipment and materials being delivered to site. The allowances by region are:

- North America: 13%
- Europe or South America: 11%
- Turkey: 7%

The bids received suggest that approximately 80% of the mining and process equipment will be purchased in Europe. The breakdown of the costs by region were previously shown in Table 19.24. The freight allowance includes:

- land transportation from point of origin to port
- marshalling yard at port
- vessel loading and ocean transportation
- vessel unloading and marshalling in Turkey (Port of Izmir)
- land transportation to site and offloading
- minimal allowance for air-freight
- customs duties and agents fees.

Spare Parts, Initial Fills, and Warehouse Inventory

Allowances are included as a percentage of either total process equipment cost or quantities determined from the process flowsheets. Spare parts are approximately 5% of total process direct cost and initial fills and warehouse inventory is approximately 2% of the total material direct cost.

EPCM

EPCM costs total US$10.7 million and are derived from the following:

- engineering design, procurement, expediting and inspection, and contracts administration
- construction management and control
- specialist consultants
- an allowance for commissioning of the project facilities.

Sub-Contracting

Potentially sub-contracted work will include:

- geotechnical consultants
- laboratory testing equipment and personnel
- vendor support to train Eldorado's employees on plant or equipment maintenance and procedures.

Contingency

An average contingency of 11% or a total of US$12.0 million was estimated for this study. Contingency varies by project area, depending on the predicted level of risk.

OWNERS COSTS

Owners' costs total US$4.02 million and were developed by Wardrop based on previous project experience. Exclusions

The following items are excluded from the capital cost estimate:

- interest during construction
- financing costs
- exchange rate fluctuations
- lost time due to severe weather and seismic conditions
- lost time due to force majeure
- additional costs for accelerated or delayed deliveries of equipment, materials, and services resulting from a change in project schedule
- unplanned warehouse inventories
- reclamation and rehabilitation at mine closure
- escalation (past 2nd Q 2007).

Table 19.26 presents the items included in the owners costs.

Training

The Owners' costs include a training program estimated at US$1.98 million training of all staff and includes costs of staff, trainers, and all training and office overheads.

The cost for vendors to commission equipment and provide the equipment specific training required is also included in the indirects.

EXCLUSIONS

The following items are excluded from the capital cost estimate:

- interest during construction
- financing costs
- exchange rate fluctuations
- lost time due to severe weather and seismic conditions
- lost time due to force majeure
- additional costs for accelerated or delayed deliveries of equipment, materials, and services resulting from a change in project schedule
- unplanned warehouse inventories
- reclamation and rehabilitation at mine closure
- escalation (past 2nd Q 2007).

Table 19.26 Owners Cost Inclusions

1	Permits
2	Fees – Government & International
3	Regulatory Body Costs
4	Owner's Engineering
5	Owner's Consultants
6	Owner's Admin & Management
7	Environmental Studies & Costs
8	Environmental Impacts Assessments
9	Owner's Insurance – Marine & All Risk
10	Owner's Legal Costs
11	Owner's Risk & Opportunity Assessment
12	Reclamation Costs (End of Mine-Life)
13	Possible Salvage Values (Economic Model)
14	Owner's Exploration/Drilling Program Costs
16	Owner's Development Costs
17	Owner's Overseeing Team during Construction
18	Project Financing Costs
19	Land Acquisition
20	Taxes & Duties
21	Working or Deferred Capital
22	Hiring and Relocation Costs – Operations
23	Community relations/Local Programs
24	Safety Awards
25	Mine Closure Costs
26	Currency Fluctuations (Economic Model)

27	Laboratory Work
28	Testing Work Except Concrete/Aggregates
29	Soils Investigation and Reports
30	Sunk Costs including Feasibility Study

19.6.2 MINE SUSTAINING CAPITAL COST

Table 19.27 is a summary of the sustaining capital costs. The mining sustaining capital costs include exploration, mine development, paste backfill borehole development, purchase of additional equipment, and equipment leasing costs.

Table 19.27 Mine Sustaining Capital Cost Summary by Year – US$

	2009 Year 1	2010 Year 2	2011 Year 3	2012 Year 4	2013 Year 5	2014 Year 6	2015 Year 7	2016 Year 8	2017 Year 9	Total
Mine Development	1,850,059	1,019,067	950,767	2,253,794	515,683	-	-	-	-	**6,589,370**
Exploration	178,668	357,335	357,335	357,335	357,335	357,335	357,335	357,335	-	**2,680,013**
Mining Equipment	-	-	-	382,826	-	382,826	-	-	-	**765,652**
Backfill Boreholes	100,000	100,000	100,000	-	-	-	-	-	-	**300,000**
Equipment Leasing Payments	875,388	1,750,775	1,750,775	1,750,775	1,750,775	875,388	-	-	-	**8,753,876**
Total	**3,004,114**	**3,227,177**	**3,158,877**	**4,744,731**	**2,623,793**	**1,615,549**	**357,335**	**357,335**	**-**	**19,088,911**

19.6.3 Operating Cost Estimate

The operating costs at the Efemçukuru Project are estimated at US$63.14/t of ore processed or US$226.23/oz of gold produced. The total cash costs are estimated at US$63.42/t of ore processed or US$227.20/oz of gold produced. Annual tonnes mined and milled will be 401,500 tonnes at a gold grade of 10.04 g/t. Table 19.28 is a summary of the average annual operating costs over life of mine.

Operating costs are defined as the direct operating costs including mining, ore treatment at Efemçukuru, concentrate treatment at Kişladağ and general and administrative. Total Cash Costs include the project royalty.

Concentrate treatment costs at Kişladağ are based on the Efemçukuru production rate for inclusion in the projects overall operating cost.

Cost Estimate Basis

Production Rate

The mine operating costs are based on mine ore production of 1,283 tonnes per day. Mining operations will be six days per week. Ore processing will be seven days per week at an average production feed rate of 1,100 t/d. The process will produce a gravity concentrate and a flotation concentrate. The gravity concentrate will be smelted on site into doré. The flotation concentrate will be trucked to Kişladağ, approximately 200 km to the east, for processing.

Mining operations have been limited to six days per week to follow Turkish Labour codes and costs for overtime standby have been included in the cost estimate.

Labour Rate

Turkish labour rates obtained from site visits to Izmir are included in the operating cost. Labour rates from the Kişladağ operations were the basis of the calculated annual labour cost at Efemçukuru. Labour rates were also compared with wages obtained from other underground operating mines in Turkey.

Consumable and Reagents

Kişladağ costs were the basis of costing for consumables and reagents. Other reagent costs not available from the Kişladağ process were obtained from European suppliers. The potential economies of scale from increasing production through Kişladağ with Efemçukuru production are not quantified in this report. An allowance for laboratory and administration costs is included.

Table 19.28 Efemçukuru and Kişladağ Operating Cost by Year

	units	2009 Year 1	2010 Year 2	2011 Year 3	2012 Year 4	2013 Year 5	2014 Year 6	2015 Year 7	2016 Year 8	2017 Year 9	2018 Year 10	Total	Life of Mine US$/t milled	US$/oz
Underground Material Movements by Year														
Total Production (diluted) - 1170 tonnes/day	t	200,750	401,500	401,500	401,500	401,500	401,500	401,500	401,500	401,500	372,349	3,785,099	-	-
Au Grade (diluted)	g/t	10.37	9.65	10.03	9.69	10.52	10.20	10.44	9.18	9.52	10.99	10.04	-	-
Mined Ounces per Year	oz	66,920	124,623	129,426	125,051	135,806	131,645	134,822	118,466	122,878	131,597	1,221,234	-	-
Processed Tonnes per day	t/day	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	-	-	-
Processed Ounces per Year	**oz**	**57,806**	**107,799**	**111,954**	**108,169**	**117,472**	**113,873**	**116,621**	**102,473**	**106,289**	**113,832**	**1,056,368**	-	-
Underground Operating by Year														
Mechanized Cut-and-Fill Stoping	US$	1,517,667	1,562,327	3,361,470	3,099,019	3,231,369	3,181,154	3,378,571	3,188,433	3,228,934	3,258,863	29,007,807	7.66	27.46
Transverse Longhole Stoping	US$	0	0	571,193	1,996,240	1,915,932	917,427	1,113,161	1,466,791	1,700,662	1,596,976	11,278,382	2.98	10.68
Longitudinal Longhole Stoping	US$	1,129,447	3,168,088	1,403,405	0	0	1,136,392	855,375	566,442	260,563	0	8,519,712	2.25	8.07
Labour	US$	1,836,192	3,497,364	3,497,364	3,169,617	3,169,617	3,169,617	3,169,617	3,169,617	3,011,342	2,792,714	30,483,062	8.05	28.86
Underground Power	US$	380,157	1,224,765	1,350,619	1,031,939	1,430,712	1,899,729	1,899,729	1,899,729	1,899,729	1,761,309	14,778,418	3.90	13.99
Mine Training	US$	250,000	250,000	0	0	0	0	0	0	0	0	500,000	0.13	0.47
Lateral Ore Development	US$	1,941,724	2,783,554	1,443,224	0	1,571,254	681,766	0	0	0	0	8,421,522	2.22	7.97
TOTAL Underground Operating	US$	7,055,187	12,486,098	11,627,276	9,296,815	11,318,885	10,986,085	10,416,453	10,291,013	10,101,230	9,409,861	102,988,903	27.21	97.49
TOTAL Operating Unit Cost	**US$/milled**	**35.14**	**31.10**	**28.96**	**23.16**	**28.19**	**27.36**	**25.94**	**25.63**	**25.16**	**25.27**			
Efemcukuru Process Operating by Year														
Consumables	US$	777,395	1,943,488	1,943,488	1,943,488	1,943,488	1,943,488	1,943,488	1,943,488	1,943,488	1,802,381	18,127,678	4.79	17.16
Power - Process	US$	1,277,164	2,554,329	2,554,329	2,554,329	2,554,329	2,554,329	2,554,329	2,554,329	2,554,329	2,368,872	24,080,668	6.36	22.80
Liners	US$	109,753	548,764	548,764	548,764	548,764	548,764	548,764	548,764	548,764	109,753	4,609,614	1.22	4.36
Labour - Plant Supervision	US$	335,691	671,382	671,382	671,382	671,382	671,382	671,382	671,382	671,382	622,636	6,329,380	1.67	5.99
Labour - Plant Operations	US$	276,927	553,854	553,854	553,854	553,854	553,854	553,854	553,854	553,854	513,642	5,221,404	1.38	4.94
Labour - Maintenance	US$	286,351	572,703	572,703	572,703	572,703	572,703	572,703	572,703	572,703	531,122	5,399,097	1.43	5.11
Service Vehicle Fuel	US$	69,482	138,964	138,964	138,964	138,964	138,964	138,964	138,964	138,964	128,874	1,310,068	0.35	1.24
Service Vehicle Maintenance	US$	32,989	82,472	82,472	82,472	82,472	82,472	82,472	82,472	82,472	41,236	734,003	0.19	0.69
Maintenance Consumables	US$	165,073	825,367	825,367	825,367	825,367	825,367	825,367	825,367	825,367	412,684	7,180,694	1.90	6.80
Laboratory Consumables	US$	20,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	10,000	430,000	0.11	0.41
Dry Tailings Transport	US$	365,000	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,000	657,000	6,862,000	1.81	6.50
TOTAL Process Operating	US$	3,715,826	8,671,322	8,671,322	8,671,322	8,671,322	8,671,322	8,671,322	8,671,322	8,671,322	7,198,199	80,284,605	21.21	76.00
TOTAL Operating Unit Cost	**US$/milled**	**18.51**	**21.60**	**21.60**	**21.60**	**21.60**	**21.60**	**21.60**	**21.60**	**21.60**	**19.33**			
Kisladag Process Operating by Year														
Consumables	US$	136,986	342,464	342,464	342,464	342,464	342,464	342,464	342,464	342,464	171,232	3,047,930	0.81	2.89
Power	US$	717,669	1,435,338	1,435,338	1,435,338	1,435,338	1,435,338	1,435,338	1,435,338	1,435,338	1,331,125	13,531,496	3.57	12.81
Liners	US$	17,229	57,431	57,431	57,431	57,431	57,431	57,431	57,431	57,431	0	476,677	0.13	0.45
Labour - Plant Supervision	US$	117,279	234,559	234,559	234,559	234,559	234,559	234,559	234,559	234,559	217,529	2,211,279	0.58	2.09
Labour - Plant Operations	US$	98,712	197,424	197,424	197,424	197,424	197,424	197,424	197,424	197,424	183,090	1,861,190	0.49	1.76
Labour - Maintenance	US$	36,635	73,271	73,271	73,271	73,271	73,271	73,271	73,271	73,271	58,616	681,416	0.18	0.65
Vehicle Fuel Costs	US$	19,285	48,213	48,213	48,213	48,213	48,213	48,213	48,213	48,213	43,391	448,378	0.12	0.42
Maintenance Consumables	US$	69,862	174,655	174,655	174,655	174,655	174,655	174,655	174,655	174,655	104,793	1,571,895	0.42	1.49
Laboratory Consumables	US$	10,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	15,000	225,000	0.06	0.21
Sub Total Process	US$	1,223,657	2,588,353	2,588,353	2,588,353	2,588,353	2,588,353	2,588,353	2,588,353	2,588,353	2,124,776	24,055,262	6.36	22.77
General and Administrative	US$	10,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	18,548	188,548	0.05	0.18
Hired Services	US$	21,000	42,000	42,000	42,000	42,000	42,000	42,000	42,000	42,000	21,000	378,000	0.10	0.36
Sub Total General and Administration	US$	31,000	62,000	62,000	62,000	62,000	62,000	62,000	62,000	62,000	39,548	566,548	0.15	0.54
TOTAL Kisladag Operating	US$	1,254,657	2,650,353	2,650,353	2,650,353	2,650,353	2,650,353	2,650,353	2,650,353	2,650,353	2,164,324	24,621,809	6.50	23.31
TOTAL Operating Unit Cost	**US$/milled**	**6.25**	**6.60**	**6.60**	**6.60**	**6.60**	**6.60**	**6.60**	**6.60**	**6.60**	**5.81**	**6.50**		
Efemcukuru Conentrate Transportation														
Bagging	US$	26,400	52,800	52,800	52,800	52,800	52,800	52,800	52,800	52,800	38,960	485,760	0.13	0.46
Transportation (4.06tonne/hour $59/tonne)	US$	615,600	1,231,200	1,231,200	1,231,200	1,231,200	1,231,200	1,231,200	1,231,200	1,231,200	1,141,809	11,607,009	3.07	10.99
Transportation Insurance	US$	9,075	18,150	18,150	18,150	18,150	18,150	18,150	18,150	18,150	16,832	171,107	0.05	0.16
TOTAL Concentrate Transportation	US$	651,075	1,302,150	1,302,150	1,302,150	1,302,150	1,302,150	1,302,150	1,302,150	1,302,150	1,195,601	12,263,876	3.24	11.61
TOTAL Operating Unit Cost	**US$/milled**	**3.24**	**3.24**	**3.24**	**3.24**	**3.24**	**3.24**	**3.24**	**3.24**	**3.24**	**3.21**	**3.24**		
Efemcukuru General Administration by Year														
Power - Site	US$	30,705	61,409	61,409	61,409	61,409	61,409	61,409	61,409	61,409	56,951	$ 578,931	0.15	0.55
Labour - Senior Management	US$	180,059	360,118	360,118	360,118	360,118	360,118	360,118	360,118	360,118	333,971	$ 3,394,971	0.90	3.21
Labour - General Operations	US$	151,949	303,897	303,897	303,897	303,897	303,897	303,897	303,897	303,897	281,833	$ 2,864,958	0.76	2.71
Labour - Support Staff	US$	96,168	192,335	192,335	192,335	192,335	192,335	192,335	192,335	192,335	178,371	$ 1,813,221	0.48	1.72
General and Administraive	US$	36,250	72,500	72,500	72,500	72,500	72,500	72,500	72,500	72,500	67,236	$ 683,486	0.18	0.65
External Laboratory Testing	US$	75,000	150,000	150,000	150,000	150,000	150,000	150,000	150,000	150,000	105,000	$ 1,380,000	0.36	1.31
Office Overheads	US$	191,000	382,000	382,000	178,000	178,000	178,000	178,000	178,000	178,000	124,600	$ 2,147,600	0.57	2.03
Insurance, Licences & H/O	US$	225,000	450,000	450,000	450,000	450,000	450,000	450,000	450,000	450,000	360,000	$ 4,185,000	1.11	3.96
Support Vehicle Fuel	US$	80,281	160,561	160,561	160,561	160,561	160,561	160,561	160,561	160,561	148,904	$ 1,513,675	0.40	1.43
Support Vehicle Maintenance	US$	16,693	33,385	33,385	33,385	33,385	33,385	33,385	33,385	33,385	20,031	$ 303,804	0.08	0.29
TOTAL General Administration	US$	1,083,103	2,166,206	2,166,206	1,962,206	1,962,206	1,962,206	1,962,206	1,962,206	1,962,206	1,676,896	$ 18,865,646	4.98	17.86
TOTAL Operating Unit Cost	**US$/milled**	**5.40**	**5.40**	**5.40**	**4.89**	**4.89**	**4.89**	**4.89**	**4.89**	**4.89**	**4.50**	**4.98**		
Efemcukuru Royalties														
1% of Mine Operating Costs		67,266	119,046	110,858	88,638	107,917	104,744	99,313	98,117	96,308	89,716	981,925		
TOTAL Project Royalties		**0.34**	**0.30**	**0.28**	**0.22**	**0.27**	**0.26**	**0.25**	**0.24**	**0.24**	**0.24**	**0.26**		
TOTAL Cash Costs	**US$/annum**	**13,827,115**	**27,395,176**	**26,528,166**	**23,971,485**	**26,012,834**	**25,676,862**	**25,101,790**	**24,975,162**	**24,783,570**	**21,734,598**	**240,006,765**		
	US$/t milled	**68.88**	**68.23**	**66.07**	**59.70**	**64.79**	**63.95**	**62.52**	**62.20**	**61.73**	**58.37**	**63.41**		
	US$/oz produced	**238.87**	**254.13**	**236.96**	**221.61**	**221.44**	**225.49**	**215.24**	**243.72**	**233.17**	**190.94**	**227.20**		

Liners and Maintenance

Major equipment maintenance including crushers, grinding mills, and mining equipment is included in the operating costs. Jaw crusher, SAG mill, ball mill, pebble crusher and vertimill consumable parts were obtained directly from the appropriate vendors. Other process equipment maintenance costs are based on 5% of the process equipment capital costs. Maintenance costs for heavy equipment, service and light vehicles are based on 10% of the direct purchase price.

Power and Fuel

The power cost used in this study is US$0.107/kWh excluding VAT, this was based costs received from Kişladağ. An electrical equipment list was compiled with connected loads, motor efficiencies and utilization factors. The power requirements are estimated at 4,597 kW and 1,392 kW for Efemçukuru and Kişladağ, respectively.

The diesel cost used in this study is US$1.30/L based on the Tüpraş (Turkish Petroleum Refineries Co.) distributor cost listed May 25, 2007. Tüpraş refines Diesel Oil 50, a low sulphur diesel fuel (50 mg/L) in Izmir. Fuel costs were calculated based on individual equipment horsepower, engine efficiencies, and utilization factors.

HUMAN RESOURCES

STAFFING

Izmir with an estimated population 4 million people (2007) will provide an adequate labour pool to staff all operations including mining, process, and general & administrative (G&A) staff.

Eldorado's corporate policy of 80% local labour will be met by the region's abundance of skilled and semi-skilled labour used to support both oil and gas refineries, and steel manufactures along the Aegean coast. Turkey also has experienced an abundance of mining activities in past decade, which will provide the work force for mining operations. The climate, life style, and services available on the west coast of Turkey especially in the Izmir area are positive attributes for attracting potential staff.

Contractors

The mining operations will rely heavily on contractors for pre-production development. The mine development will be completed to a level to provide a minimum of 12 months of future production. The maintenance department will contract two mechanics to service the specialized equipment during development and early production phases. The process department will not employ any contractors.

Training

A training program has been budgeted in the capital cost for a 12 week training schedule of all staff. The budget includes the cost of staff, trainers, and all training and office overheads.

The development mining contractor will be utilized provide some of the training for the mine production staff and is provided for in operating cost estimate during the first year of operation.

The cost for vendors to commission equipment and the equipment specific training required is also included in the capital cost estimate.

A full-time safety & mine trainer will be on staff for the life of the mine. A comprehensive mine training program has been accounted for during pre-production for mining personnel training on equipment and developing mine rescue crews.

Transportation and Meals

Staff will be transported to site in 20-person busses from Menderes and Izmir. This will minimize local traffic and requirements on site.

A cafeteria will provide meals to all staff working on site.

Expatriates

Early operations include allowances for eight expatriates to manage the operation, oversee commissioning and start-up, set up procedures, and train staff. During year 3, this will be scaled back to two expatriates, the general manager, and controller.

Figure 19.14 is the organization chart for the Efemçukuru Project.

Figure 19.14 Efemçukuru Organizational Chart

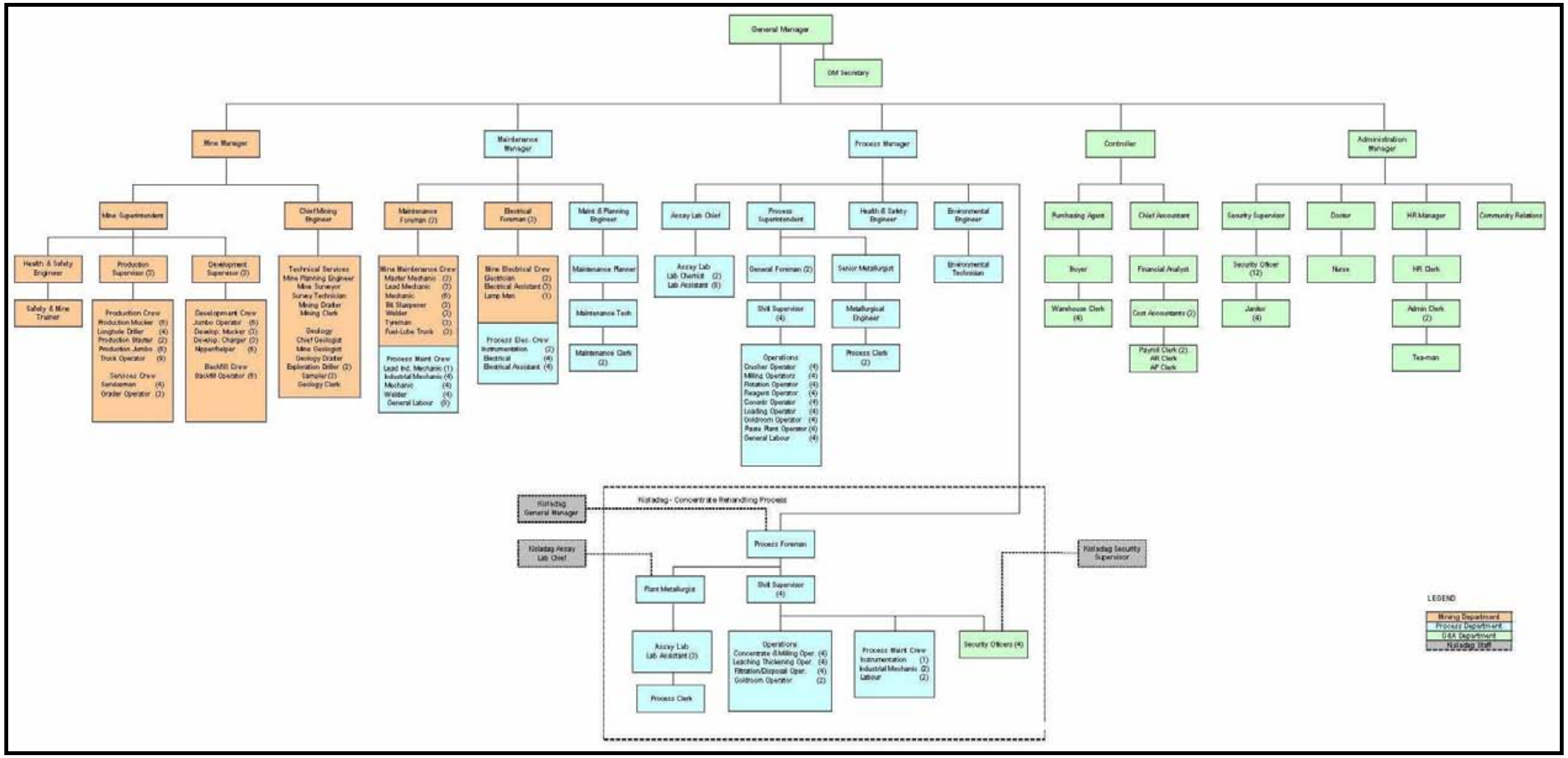


Table 19.29 to Table 19.32 show the labour requirements for the Efemçukuru Project and the Kişladağ processing. A total 295 full-time staff will be required for the all operations; 262 full-time staff will be required for the Efemçukuru site, with a further 33 full-time staff required to operate the concentrate process at Kişladağ.

Table 19.29 G&A Labour Requirements

Position	Labour
Senior Management	
General Manager	1
Controller	1
Administration Manager	1
Subtotal	**3**
General Operations	
Community Relations	1
Chief Accountant	1
Human Resources Manager	1
Financial Analyst	1
Purchasing Agent	1
Buyer	1
Security Supervisor	1
Security Officer	12
Cost Accountant	2
Doctor	1
Nurse	1
Subtotal	**23**
Support Staff	
Tea Man	1
General Manager Secretary	1
Human Resources Clerk	1
Administration Clerk	2
Payroll Clerk	4
Warehouse Clerk	4
Janitor	4
Subtotal	**17**
TOTAL G&A LABOUR	**43**

Table 19.30 Mining Labour Requirements

Position	Labour
Mine Supervision	
Mine Manager	1
Mine Superintendent	1
Electrical Foreman	3
Electrical Foreman	3
Shift Supervisor (production/development)	6
Subtotal	**14**
Mine Technical Services	
Chief Mining Engineer	1
Mine Planning Engineer	1
Mine Surveyor	1
Survey Technician	1
Chief Geologist	1
Mine Geologist	1
Draftsperson	2
Administration Clerk	2
Samplers	3
Subtotal	**13**
Production Crew	
Production Mucker	6
Truck Operator	9
Longhole Driller	4
Production Blaster	2
Production Jumbo	6
Subtotal	**27**
Development Crew	
Jumbo Operator	6
Development Mucker	3
Development Charger	3
Nipper/offsider	6
Subtotal	**18**
Services Crew	
Servicemen	4
Grader	3
Backfill Operator	6
Subtotal	**13**
Maintenance	
Master Mechanic	3
Lead Mechanic	3
Mechanic	6
Bit Sharpener	3

table continued…

Position	Labour
Welder	3
Tyreman	3
Fuel-lube truck operator	3
Electrician	3
Electrician Assistant	3
Lamp person	1
Subtotal	**31**
Non Production-Maintenance	
Exploration Driller	2
Health & Safety Engineer	1
Safety & Mine Trainer	1
Subtotal	**4**
Total Mine Labour	**120**

Table 19.31 Process Labour Requirements

Position	Labour
Plant Supervision	
Process Manager	1
Process Superintendent	1
Process General Foreman	2
Health and Safety Engineer	1
Shift Supervisors	4
Senior Metallurgist	1
Assay Lab Chief	1
Metallurgy Engineer	1
Environmental Engineer	1
Environmental Technician	1
Subtotal	**14**
Plant Operations	
Crusher Operator	4
Milling Operator	4
Flotation Operator	4
Reagent Operator	4
Gravity Concentrator Operator	4
Concentrate Filter & Loading Operator	4
Goldroom Operator	4
Paste Plant Operator	4

table continued…

Position	Labour
General Labour	4
Laboratory Chemist	2
Lab Assistant	8
Process Clerk	2
Subtotal	**48**
Maintenance	
Maintenance Manager	1
Maintenance and Planning Engineer	1
Maintenance Planner	1
Maintenance Technician	1
Maintenance Clerk	2
Electricians	4
Electricians Assistant	4
Instrumentation Technician	2
Lead Industrial Mechanic	1
Industrial Mechanic	4
Mechanic	4
Welder	4
General Labour	8
Subtotal	**37**
Total Process Labour	**99**

Table 19.32 Kişladağ Labour Requirements

Position	Labour
Plant Supervision	
Process Foreman	1
Shift Supervisor	4
Security Officer	4
Subtotal	**9**
Plant Operations	
Concentrate and Milling Operator	4
Leaching and Thickening Operator	4
Filtration and Disposal Operator	4
Gold Room Operator	2
Plant Metallurgist	1
Labour Assistant	3
Process Clerk	1
Subtotal	**19**
Maintenance	
Industrial Mechanic	2
Instrument Technician	1
General Labour	2
Subtotal	**5**
Total Kişladağ Labour	**33**

19.6.4 EFEMÇUKURU AND KIŞLADAĞ LABOUR COST

The total life of mine average annual labour cost for Efemçukuru and Kişladağ is US$6.4 million. Table 19.33 and Table 19.34 summarize the labour requirements and costs for the Efemçukuru and Kişladağ process facilities. During peak operating conditions, the operation of the mine and both process facilities will require 295 personnel.

Table 19.33 Efemçukuru Labour Cost

Position	Labour	Annual Cost (US$)	Cost/Tonne Ore (US$)
Administration			
Senior Management	5	360,118	0.90
General Operations	21	303,897	0.76
Support Staff	17	192,335	0.48
Total G&A	**43**	**856,350**	**2.13**
Mining			
Mine Supervision	14	673,528	1.68
Mine Technical Services	13	414,015	1.03
Production Crew	27	944,811	2.35
Development Crew*	18	620,032	1.54
Services Crew	13	392,031	0.98
Maintenance	31	586,816	1.46
Equipment Supplier	2	240,000	0.60
Non Production Maintenance	4	151,499	0.38
Total Mine	**120**	**4,022,732**	**10.02**
Process Department			
Plant Supervision	14	671,382	1.67
Plant Operations	48	553,654	1.38
Maintenance	37	572,703	1.43
Total Process	**99**	**1,797,939**	**4.48**
Total Efemçukuru	**262**	**6,677,021**	**16.63**

* The development crew labour cost is included in the total personnel and labour cost. A portion of this cost, based on the waste development requirements by year, is carried as a sustaining capital cost.

The mining labour cost in Table 19.33 is based on a full production year (years 2 and 3) including the capitalized waste development labour cost. Labour costs were determined on a year-by-year basis according to production and development requirements.

The total life of mine labour cost is US$33.2 million with US$30.8 million in operating labour cost. This equates to US$8.76/t (including sustaining capital) and US$8.05/t (excluding sustaining capital).

Table 19.34 Kişladağ Labour Cost

Position	Labour Force	Annual Cost (US$)	Cost/Tonne Ore (US$)
Kişladağ			
Plant Supervision	11	234,559	0.58
Plant Operations	17	197,424	0.49
Maintenance	5	73,271	0.18
Total Kişladağ	**33**	**505,253**	**1.25**

19.6.5 Mine Operating Cost

Annual mine operating costs will average US$10.8 million in full production years. The mine operating cost is based on annual production of 401,500 tonnes of mill feed. Mine operating costs were developed from first principals and based on the following:

- mechanized cut-and-fill stoping (approximately 45% of ore production)
- transverse longhole (TLH) stoping (approximately 33% of ore production)
- longitudinal longhole (LLH) stoping (approximately 22% of ore production)
- ore development and access
- ore haulage to central ore-pass system
- average haul distances from the upper and lower Middle Ore Shoot (MOS) and South Ore Shoot (SOS)
- underground backfill
- employee training
- supervising and operating labour
- underground mine power.

All pre-production development and operating costs are capitalized in the capital cost estimate. Equipment leasing costs have been included in the sustaining capital cost. The mine operating cost by mining method is shown in Table 19.35.

Mine productivities including equipment operating hours, labour requirements, and consumables were estimated based on eight-hour shifts, with an effective work time of 6.3 hours.

Table 19.35 Mine Operating Cost by Mining Method – US$/t

	Mechanized Cut-and-Fill			LLH	TLH
	Lower MOS	Upper SOS	Lower SOS	Upper SOS	Upper MOS
Load and Haul	3.25	2.34	2.42	4.11	2.64
Drilling	3.10	3.10	3.10	1.34	1.41
Blasting	2.61	2.61	2.61	1.26	1.52
Maintenance	2.29	2.16	2.17	1.35	1.18
Ground Support	3.06	3.06	3.06	0.00	0.00
Piping & Services	0.19	0.19	0.19	0.03	0.09
Electrical	0.02	0.02	0.02	0.00	0.00
Ventilation	0.26	0.26	0.26	0.00	0.00
Backfill	2.79	2.79	2.79	2.23	2.23
Power	4.22	4.06	3.75	3.50	4.08
Labour	7.82	8.13	8.09	8.49	7.82
Operating Cost	**29.62**	**28.72**	**28.46**	**22.31**	**20.96**

The underground equipment cost estimation is based on hourly operating costs from mine equipment suppliers, Western Mine Engineering and the personal experience of Wardrop engineers. Quotes for drilling consumables, ground support, ventilation tubing, and electrical cables were obtained from suppliers. Quotes were received from Turkish suppliers for explosives, piping, diesel and cement. Allowances have been made for material wastage where applicable.

The equipment selection and labour requirements are based on the utilization of equipment required to meet the target daily production for each mining method. The productivity per manshift is summarized in the Table 19.36 and discussed in Section 19.1.2.

Table 19.36 Summary of Target Daily Productivity

Mining Method	Production (t/d)	Direct and Indirect Labour	Productivity (t/manshift)	All Labour	Productivity (t/total manshift)
MCF	582	13	45	28	21
LLH	274	5	56	10	26
TLH	427	7	62	15	29

19.6.6 UNDERGROUND BACKFILL COST

The operating cost for backfill is US$3.64/t of ore including labour. Table 19.37 shows the cement, filtering, maintenance and underground labour costs.

Table 19.37 Average Backfill Operating Cost

Description	Unit	Value
Cement Content in Backfill	%	4.3
Tailings Tonnage	t/d	645
Cement Tonnage	t/d	28
Cement Price (per tonne cement)	US$	90.00
Backfill Cost per Tonne of Ore		
Cement	US$/t	2.50
Filtering	US$/t	0.29
Filter Belt	US$/t	0.29
Maintenance	US$/t	0.06
Labour	US$/t	0.49
Total Backfill Cost per Tonne of Ore	**US$/t**	**3.64**

19.6.7 PROCESS OPERATING COST

EFEMÇUKURU PROCESS OPERATING COST

Average annual process operating costs will be US$9.845 million, including power. The process operating costs are based on an annual processing of 401,500 tonnes of ore. The process operating costs are based on the following:

- power
- consumables including reagents and bagging
- liner replacement
- labour including supervision and maintenance
- service vehicle maintenance
- maintenance consumables
- laboratory consumables
- filtered tailings transport contract
- concentrate bagging and transportation to Kişladağ.

Table 19.38 and show the Efemçukuru process operating cost by year.

Table 19.38 Efemçukuru Process Operating Cost

Section	US$/a	US$/t
Consumables	1,928,476	4.79
Power - Process	2,561,773	6.36
Liners	490,384	1.22
Labour - Plant Supervision	673,338	1.67
Labour - Plant Operations	555,469	1.38
Labour - Maintenance	574,372	1.43
Service Vehicle Fuel Costs	139,369	0.35
Service Vehicle Maintenance	78,085	0.19
Maintenance Consumables	763,904	1.90
Laboratory Consumables	45,745	0.11
Tailings Trucking and Handling	730,000	1.81
Subtotal	**8,540,915**	**21.21**
Bags	51,677	0.13
Shipping	1,234,788	3.07
Insurance	18,203	0.05
Subtotal	**1,304,668**	**3.24**
Total	**9,845,583**	
Cost per Tonne	**24.45**	
Cost per oz Au Produced	**87.61**	

KIŞLADAĞ PROCESS OPERATING COST

Average annual process operating costs will be US$2.619 million, including power. The process operating costs are based on an annual processing of 36,000 tonnes of flotation concentrate. The process operating costs are based on the following:

- power
- consumables including reagents and bagging
- liner replacement
- labour including supervision and maintenance
- service vehicle maintenance
- maintenance consumables
- laboratory consumables.

Table 19.39 show the process operating cost by year for both tonnes of concentrate processed at Kişladağ and tonnes of ore processed at Efemçukuru.

Table 19.39 Kişladağ Process Operating Cost

Section	US$/annum	US$/t Concentrate	Cost/Tonne Ore (US$)
Process and Maintenance			
Consumables	324,248	9.43	0.805
Power	1,439,521	39.54	3.575
Liners	50,710	1.58	0.126
Labour - Plant Supervision	235,242	6.46	0.584
Labour - Plant Operations	197,999	5.44	0.492
Labour - Maintenance	72,491	2.02	0.180
General Admin Costs	30,542	0.55	0.050
Hired Services	60,319	1.16	0.100
Vehicle Fuel Costs	47,700	1.33	0.118
Maintenance Consumables	167,223	4.81	0.415
Laboratory Consumables	23,936	0.69	0.059
Total	**2,619,341**	**73.01**	**6.50**
Cost per Tonne Milled	**6.50**		
Cost per Ounces Produced	**23.31**		

19.6.8 Power Requirement

The total power cost will be approximately $13.99/t of ore including Mining and processing on the Efemçukuru and Kişladağ sites. Power will be supplied at 34.5 kVA through overhead lines from the national grid at US$0.107/kWh. Table 19.40 and Table 19.41 show the total power requirements for Efemçukuru and Kişladağ respectively. The Efemçukuru operating load is approximately 4,598 kW or 39,978 MWh/a. The Kişladağ operating load is approximately 1,392 kW or 12,195 MWh/a.

Table 19.40 Power Requirements for Efemçukuru

Area	Total Rating (kW)	Connected Load (kW)	Standby Load (kW)	Operating Load (kW)	Annual Operating (MWh/a)
Process Plant Area					
C1 – Primary Crushing	170	170	0	143	957
D0 – Crushed Ore Storage & Reclaim	45	45	0	38	332
E0 – Air Supply & Distribution	134	104	30	88	767
E1 – Grinding & Classification	1,603	1,569	34	1,458	12,775
E2 – Gravity Concentration	41	39	2	33	288
E3 – Pebble Crushing	147	147	0	124	1,082
E4 – Flotation	265	164	100	134	1,173
E5 – Concentrate Dewatering & Loadout	157	114	43	85	743
E6 – Reagents	51	35	16	21	188
E7 – Gold Room	172	168	4	84	735
F2 – Tailings Filtrations & Paste	139	72	67	59	516
K4 – Water Supply & Distribution	514	469	45	245	2,146
Total Process Plant	**3,437**	**3,096**	**341**	**2,512**	**21,702**
Mine Area					
B1- Mining Equipment	3,956	3,956	0	2,004	17,552
J3 – Truck Shop	44	44	0	23	203
Total Mine	**4,000**	**4,000**	**0**	**2,027**	**17,754**
Site Area					
G1 – Water Treatment	77	34	44	22	194
F1 – Tailings Thickening	22	22	0	9	81
K3 – Fresh Water	63	34	30	28	247
Total Site	**163**	**89**	**74**	**60**	**522**
Total Efemçukuru	**7,600**	**7,185**	**415**	**4,598**	**39,978**

Table 19.41 Power Requirements for Kişladağ

Kişladağ Process Plant Area	Total Rating (kW)	Connected Load (kW)	Standby Load (kW)	Operating Load (kW)	Annual Operating (MWh/a)
P1 – Concentrate & Mill	604	552	30	510	4,467
P2 – Leaching	651	630	6	521	4,565
P3 – Gold Room	369	355	0	253	2,217
P4 – Reagents	53	45	11	27	235
P5 – Services	141	130	21	81	711
Total Kişladağ	**1,819**	**1,712**	**67**	**1,392**	**12,195**

19.6.9 GENERAL AND ADMINISTRATION

The annual general and administration (G&A) cost is US$3,020,361. The G&A are costs that do not relate directly to the mining or processing operating costs. Table 19.42 shows the estimated G&A costs.

Table 19.42 G&A Costs

Area	Annual Cost ($/a)	Unit Cost ($/t Ore)
Power - Site	93,779	0.15
Labour - Senior Management	549,938	0.90
Labour - General Operations	464,083	0.76
Labour - Support Staff	293,717	0.48
General Admin Costs	110,715	0.18
External Laboratory Testing	221,809	0.36
Office Overheads	317,468	0.57
Insurance, Licenses & H/O	675,001	1.11
Support Vehicle Fuel Costs	245,194	0.40
Support Vehicle Maintenance	48,657	0.08
Total	**$3,020,361**	**4.98**
Cost per Tonne Milled	**4.98**	
Cost per oz Produced	**17.86**	

19.6.10 SUMMARY OF CASH COSTS

The total cash costs for the project are US$63.41/t, US$227.20/oz. and $25.5 million per year and are summarized in Table 19.43.

Table 19.43 Cash Costs

	US$/t Milled	US$/oz	US$/annum
Underground Operating	27.21	97.49	10,924,429
Efemçukuru Process Operating	21.21	76.00	8,516,096
Kişladağ Process Operating	6.50	23.31	2,611,730
Efemçukuru Conentrate Transportation	3.24	11.61	1,300,876
Efemçukuru General Administration	4.98	17.86	2,001,152
Efemçukuru Royalties	0.26	0.93	104,390
Total Cash Costs	**$63.41**	**$227.20**	**$25,458,673**

19.7 ECONOMIC ANALYSIS

19.7.1 INTRODUCTION

Economic evaluation indicates a pre-tax internal rate of return (IRR) of 23.2% and a pre-tax net present value (NPV) of US$115 million at a discount rate of 5.0%. The pre-tax base case financial model is calculated with the following parameters:

- 3 year average metal gold price of US$530/oz (London Metal Exchange)
- concentrate transport costs
- treatment costs at Eldorado's Kişladağ operation
- project Royalty of 1% of direct mine operating costs.

Wardrop prepared the model on a pre-tax basis.

Variations of the pre-tax base case are presented to show the effect of using five-year and two-year historical gold prices and the current gold price.

19.7.2 NPV AND IRR SUMMARY

This study presents the predicted NPV and IRR for the project, and a sensitivity analysis of the project to key variables including metal prices, exchange rates, and operating and capital costs.

Initial and sustaining capital has been considered on a year-by-year basis for the life of project. Initial capital comprises all capital expenditure prior to first production of mineral concentrate from the process plant; sustaining capital comprises all subsequent capital expenditure, including equipment replacement based on predicted equipment life. Contingency varies by project area, depending on the predicted level of risk.

Salvage costs are based on 10% of the initial direct process capital cost.

Reclamation costs are not provided in this study. An allowance of Cdn$10.0 million is included in the financial evaluation to balance salvage value of equipment.

Working capital of Cdn$6.0 million is included in year 1. This is based on three months of operating costs. Working capital is recovered at the end of mine life.

The discounted cash flow rate of 5.0% is considered an industry standard for gold projects.

Marketing costs were estimated at US$0.50/oz of the value of the metal in concentrate.

The net revenue is defined as the gross revenue less costs incurred subsequent to concentrating, which includes transportation, insurance, royalties and refining. No provision is made for deducting mine operating costs for this calculation. Operating cash flow is defined as the net revenue less mine operating costs.

19.7.3 ANALYSIS OF SENSITIVITY TO METAL PRICE

A number of gold price scenarios were run on the pre-tax model to evaluate the sensitivity on NPV and IRR. The current, five year, three year and two year rolling average gold prices from the London Metal Exchange are included as shown in Table 19.44.

Table 19.44 Summary of Gold Price Scenarios

Scenario	Gold Price (US$/oz)	Pre-Tax NPV (million US$)	IRR (%)
5 Year Average	465	65.1	15.8
3 Year Average (Base Case)	530	115.1	23.2
2 Year Average	580	153.5	28.5
Current Price	670	222.6	37.7

The pre-tax model used in these calculations is shown in Table 19.45.

Table 19.45 Pre-Tax Economic Evaluation – Base Case

	Source	Units	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	TOTAL
Period Ending			2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	
GROSS REVENUE														
Process Feed	W	000'S t	0	201	402	402	402	402	402	402	402	402	373	3,786
Average Grade Au	W	g/t	0.0	10.368	9.654	10.026	9.687	10.521	10.198	10.444	9.177	9.519	10.993	10.035
Contained Au	W	oz	0	66,920	124,623	129,426	125,051	135,806	131,645	134,822	118,466	122,878	131,827	1,221,464
Total Annual Recovery	W	%	0.00	86.50	86.50	86.50	86.50	86.50	86.50	86.50	86.50	86.50	86.50	86.50
Total Recovered Au	W	oz	0	57,886	107,799	111,954	108,169	117,472	113,873	116,621	102,473	106,289	114,031	1,056,566
Average Au Price	E	US$/oz	0	530.00	530.00	530.00	530.00	530.00	530.00	530.00	530.00	530.00	530.00	530.00
Gross Revenue (US $)	W	000's US$	0	30,679	57,134	59,335	57,330	62,260	60,352	61,809	54,311	56,333	60,436	559,980
OPERATING COSTS														
Mining Operating Costs	W	000's US$	0	7,055	12,486	11,627	9,297	11,319	10,986	10,416	10,291	10,101	9,410	102,989
		US$/t	0.00	35.14	31.10	28.96	23.16	28.19	27.36	25.94	25.63	25.16	25.23	27.20
		US$/oz	0.00	121.88	115.83	103.86	85.95	96.35	96.48	89.32	100.43	95.04	82.52	97.48
Process Operating Costs Efemcukuru	W	000's US$	0	3,716	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	7,198	80,285
		US$/t	0.00	18.51	21.60	21.60	21.60	21.60	21.60	21.60	21.60	21.60	19.30	21.21
		US$/oz	0.00	64.19	80.44	77.45	80.16	73.82	76.15	74.35	84.62	81.58	63.13	75.99
Process Operating Costs Kisladag	W	000's US$	0	1,255	2,650	2,650	2,650	2,650	2,650	2,650	2,650	2,650	2,164	24,622
		US$/t	0.00	6.25	6.60	6.60	6.60	6.60	6.60	6.60	6.60	6.60	5.80	6.50
		US$/oz	0.00	21.67	24.59	23.67	24.50	22.56	23.27	22.73	25.86	24.94	18.98	23.30
Transport & Insurance Costs	W E	000's US$	0	651	1302	1302	1302	1302	1302	1302	1302	1302	1196	12264
		US$/t	0.00	3.24	3.24	3.24	3.24	3.24	3.24	3.24	3.24	3.24	3.21	3.24
		US$/oz	0.00	11.25	12.08	11.63	12.04	11.08	11.44	11.17	12.71	12.25	10.49	11.61
Direct Operating General and Administration Costs	W	000's US$	0	1083	2166	2166	1962	1962	1962	1962	1962	1962	1677	18866
		US$/t	0.00	5.40	5.40	5.40	4.89	4.89	4.89	4.89	4.89	4.89	4.50	4.98
		US$/oz	0.00	18.71	20.09	19.35	18.14	16.70	17.23	16.83	19.15	18.46	14.71	17.86
Cash Operating Costs = Direct Operating + Trans & Insur		000's US$	0	13,760	27,276	26,417	23,883	25,905	25,572	25,002	24,877	24,687	21,645	239025
		US$/t	0.00	68.54	67.94	65.80	59.48	64.52	63.69	62.27	61.96	61.49	58.03	63.14
		US$/oz	0.00	237.71	253.03	235.97	220.79	220.52	224.57	214.39	242.77	232.27	189.82	226.23
Royalties	E	000's US$	0	71	125	116	93	113	110	104	103	101	94	1030
		US$/t	0.00	0.35	0.31	0.29	0.23	0.28	0.27	0.26	0.26	0.25	0.25	0.27
		US$/oz	0.00	1.22	1.16	1.04	0.86	0.96	0.96	0.89	1.00	0.95	0.83	0.97
Total Cash Costs = Cash Operating Costs + Royalties		000's US$	0	13,830	27,401	26,534	23,976	26,018	25,682	25,107	24,980	24,788	21,739	240,055
		US$/t	0.00	68.89	68.25	66.09	59.72	64.80	63.97	62.53	62.22	61.74	58.28	63.41
		US$/oz	0.00	238.93	254.19	237.01	221.65	221.48	225.53	215.28	243.77	233.22	190.64	227.20
CAPITAL COSTS														
Preproduction Capital	W	000's US$	60,724	43,478										104,202
Sustaining Capital	W	000's US$		3,031	3,331	3,313	4,999	2,928	1,942	707	707	350	0	21,308
Working Capital	E	000's US$	6,001										-6,001	0
Reclamation Cost	W	000's US$											10,000	10,000
Salvage at End of Mine Life	W	000's US$											-10,000	(10,000)
Total Capital Costs	W	000's US$	66,725	46,509	3,331	3,313	4,999	2,928	1,942	707	707	350	-6,001	125,510
FINANCIAL SUMMARY														
Gross Revenue		000's US$	0	30,679	57,134	59,335	57,330	62,260	60,352	61,809	54,311	56,333	60,436	559,980
Total Cash Costs		000's US$	0	13,830	27,401	26,534	23,976	26,018	25,682	25,107	24,980	24,788	21,739	240,055
Capital Costs		000's US$	66,725	46,509	3,331	3,313	4,999	2,928	1,942	707	707	350	-6,001	125,510
Post-Tax Cash Flow														
Cash Flow		000's US$	-66,725	-29,660	26,402	29,489	28,355	33,314	32,728	35,995	28,624	31,195	44,699	194,415
Accumulated Cash Flow		000's US$	-66,725	-96,386	-69,984	-40,495	-12,140	21,175	53,903	89,898	118,522	149,717	194,415	

Discount Rate		%	5.0%
Post-Income Tax Net Present Value (NPV)		million US$	115.1
Post-Income Tax Internal Rate of Return (IRR)		%	23.2%
Initial Capital		million US$	104.2
Cash Operating Costs		US$/t	63.14
		US$/ounce	226.23
Total Cash Costs		US$/t	63.41
		US$/ounce	227.20
Mine Life		Yrs	9.4
Payback Period		Yrs	3.9

Note: W = Wardrop; LME = London Metal Exchange; E = Eldorado

19.7.4 SENSITIVITY ANALYSIS

Sensitivities to metal price, capital cost, and operating cost on the IRR and NPV were considered on the post-tax base case model. Figure 19.15 and Figure 19.16 show the sensitivity trends. Figure 19.17 shows the cash flow projection, Figure 19.18 shows the sensitivity of the IRR to the gold price and Figure 19.18 shows the sensitivity of the NPV to the gold price.

The project is most sensitive to metal price. The project NPV is more sensitive to operating cost than initial capital cost.

Figure 19.15 NPV Sensitivity Analysis

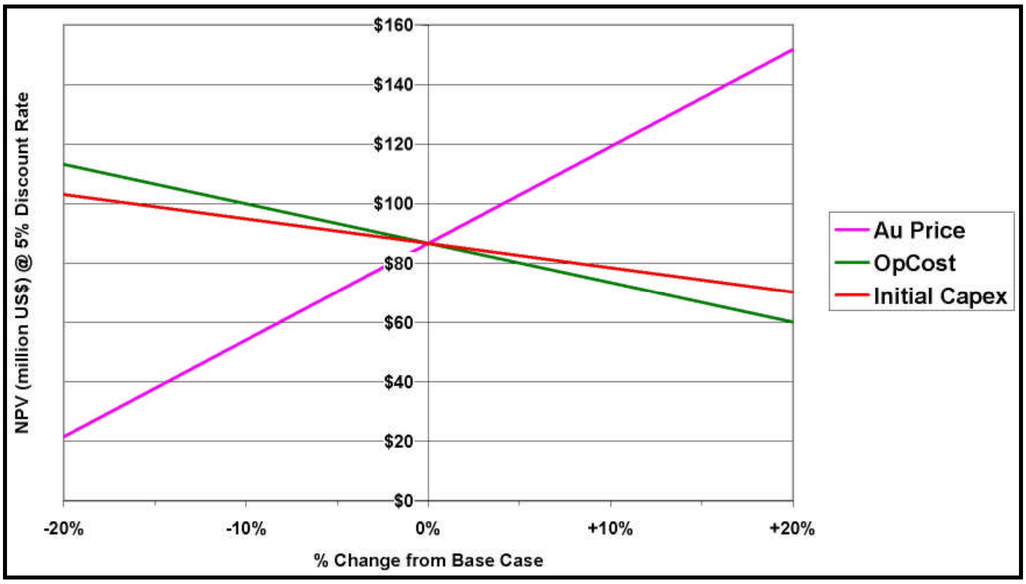


Figure 19.16 IRR Sensitivity Analysis

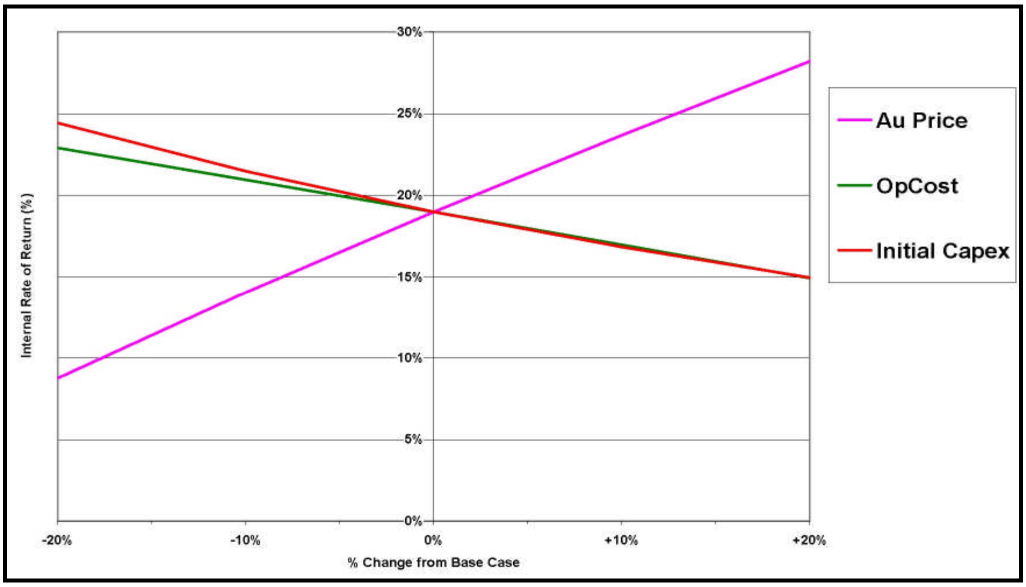


Figure 19.17 Cash Flow

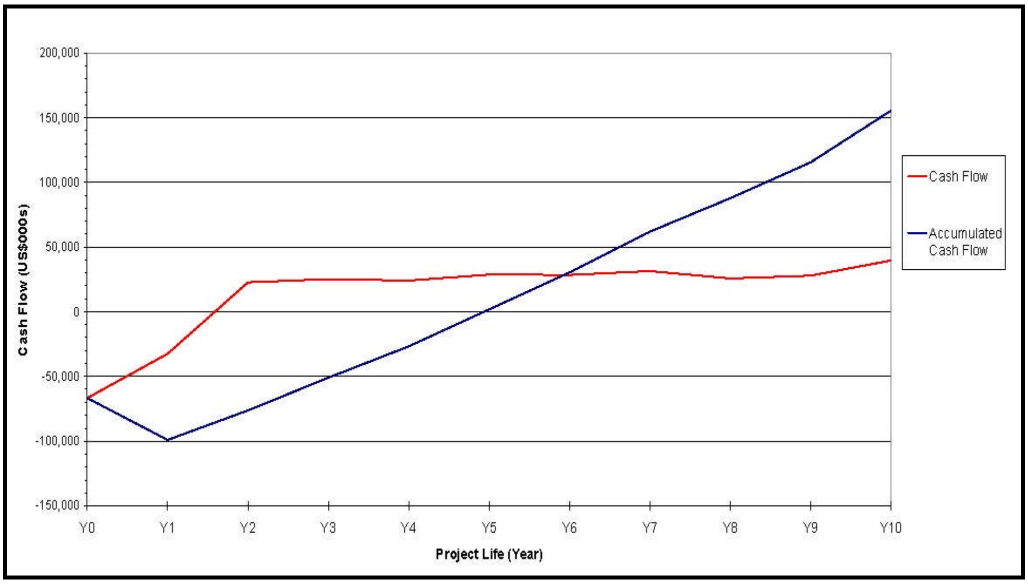


Figure 19.18 IRR Sensitivity to Gold Price

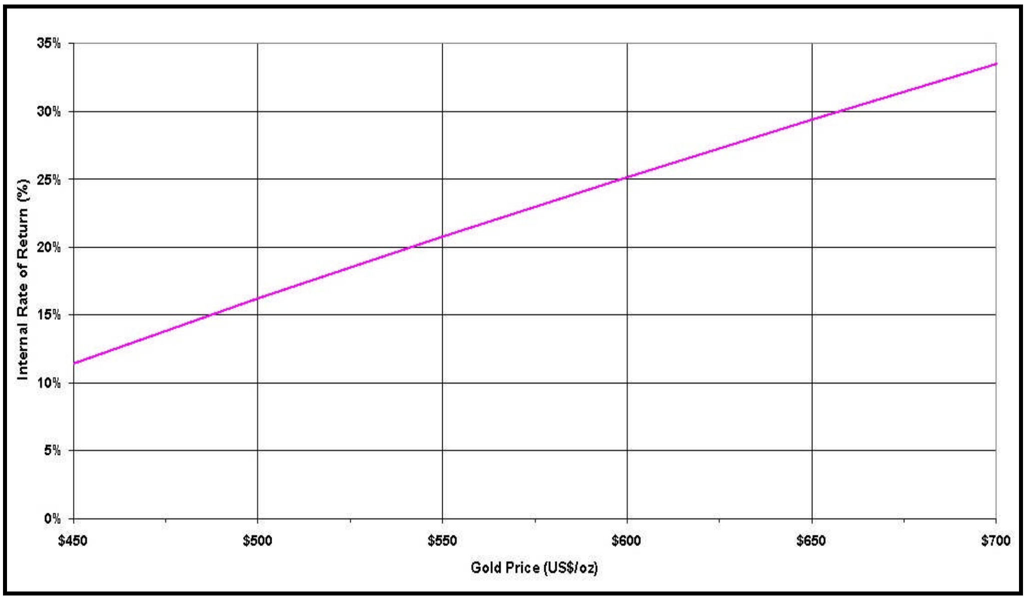
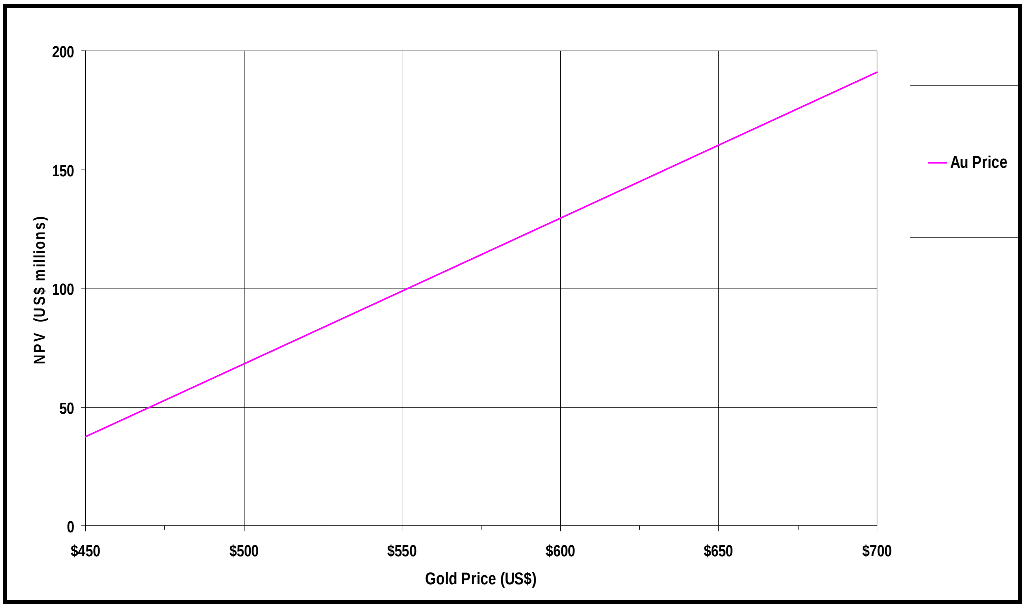


Figure 19.19 NPV Sensitivity to Gold Price (Post-tax)



19.7.5 PAYBACK

The payback period for the base case is 3.9 years. This is the time required after revenue is first received in year 1 to achieve break-even cumulative cash flow. The payback period is based on the annual un-discounted cash flows. There is no consideration for inflation, interest, or depreciation in this calculation.

19.7.6 ROYALTIES

Eldorado is required to pay 1% of the direct mine operating cost as a royalty to a third party. The average annual payment will be US$108,150 in a full production year. The total royalty included in this evaluation is US$1.03 million. This equates to US$0.27/t milled or US$0.97/oz produced.

19.7.7 TAXES

Economic evaluation indicates a post-tax IRR of 19.0% and a post-tax NPV of US$86.7 million at a discount rate of 5.0%. The post-tax base case financial model used the same inputs as the pre-tax economic evaluation:

- 3 year average metal gold price of US$530/oz (London Metal Exchange)
- concentrate transport costs
- treatment costs at Eldorado's Kişladağ operation
- project Royalty of 1% of direct mine operating costs.

The sensitivity of the project to discount rate is shown in Table 19.46.

Table 19.46 NPV Sensitivity to Discount Rate

Discount Rate	Post-tax NPV (million US$)
8.0%	$58.3
5.0%	$86.7
3.0%	$110.4
0.0%	$155.5

The post tax model is shown in Table 19.47.

Table 19.47 Post-tax Economic Evaluation – Base Case

	Source	Units	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	TOTAL
Period Ending			2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	
GROSS REVENUE														
Process Feed	W	000'S t	0	201	402	402	402	402	402	402	402	402	373	3,786
Average Grade Au	W	g/t	0.0	10.368	9.654	10.026	9.687	10.521	10.198	10.444	9.177	9.519	10.993	10.035
Contained Au	W	oz	0	66,920	124,623	129,426	125,051	135,806	131,645	134,822	118,466	122,878	131,827	1,221,464
Total Annual Recovery	W	%	0.00	86.50	86.50	86.50	86.50	86.50	86.50	86.50	86.50	86.50	86.50	86.50
Total Recovered Au	W	oz	0	57,886	107,799	111,954	108,169	117,472	113,873	116,621	102,473	106,289	114,031	1,056,566
Average Au Price	E	US$/oz	0	530.00	530.00	530.00	530.00	530.00	530.00	530.00	530.00	530.00	530.00	530.00
Gross Revenue (US $)	W	000's US$	0	30,679	57,134	59,335	57,330	62,260	60,352	61,809	54,311	56,333	60,436	559,980
OPERATING COSTS														
Mining Operating Costs	W	000's US$	0	7,055	12,486	11,627	9,297	11,319	10,986	10,416	10,291	10,101	9,410	102,989
		US$/t	0.00	35.14	31.10	28.96	23.16	28.19	27.36	25.94	25.63	25.16	25.23	27.20
		US$/oz	0.00	121.88	115.83	103.86	85.95	96.35	96.48	89.32	100.43	95.04	82.52	97.48
Process Operating Costs Efemcukuru	W	000's US$	0	3,716	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	7,198	80,285
		US$/t	0.00	18.51	21.60	21.60	21.60	21.60	21.60	21.60	21.60	21.60	19.30	21.21
		US$/oz	0.00	64.19	80.44	77.45	80.16	73.82	76.15	74.35	84.62	81.58	63.13	75.99
Process Operating Costs Kisladag	W	000's US$	0	1,255	2,650	2,650	2,650	2,650	2,650	2,650	2,650	2,650	2,164	24,622
		US$/t	0.00	6.25	6.60	6.60	6.60	6.60	6.60	6.60	6.60	6.60	5.80	6.50
		US$/oz	0.00	21.67	24.59	23.67	24.50	22.56	23.27	22.73	25.86	24.94	18.98	23.30
Transport & Insurance Costs	E	000's US$	0	651	1302	1302	1302	1302	1302	1302	1302	1302	1196	12264
		US$/t	0.00	3.24	3.24	3.24	3.24	3.24	3.24	3.24	3.24	3.24	3.21	3.24
		US$/oz	0.00	11.25	12.08	11.63	12.04	11.08	11.44	11.17	12.71	12.25	10.49	11.61
General and Administration Operating Costs	W	000's US$	0	1083	2166	2166	1962	1962	1962	1962	1962	1962	1677	18866
		US$/t	0.00	5.40	5.40	5.40	4.89	4.89	4.89	4.89	4.89	4.89	4.50	4.98
		US$/oz	0.00	18.71	20.09	19.35	18.14	16.70	17.23	16.83	19.15	18.46	14.71	17.86
Cash Operating Costs = Direct Operating + Trans & Insur		000's US$	0	13,760	27,276	26,417	23,883	25,905	25,572	25,002	24,877	24,687	21,645	239025
		US$/t	0.00	68.54	67.94	65.80	59.48	64.52	63.69	62.27	61.96	61.49	58.03	63.14
		US$/oz	0.00	237.71	253.03	235.97	220.79	220.52	224.57	214.39	242.77	232.27	189.82	226.23
Royalties	E	000's US$	0	71	125	116	93	113	110	104	103	101	94	1030
		US$/t	0.00	0.35	0.31	0.29	0.23	0.28	0.27	0.26	0.26	0.25	0.25	0.27
		US$/oz	0.00	1.22	1.16	1.04	0.86	0.96	0.96	0.89	1.00	0.95	0.83	0.97
Total Cash Costs = Cash Operating Costs + Royalties		000's US$	0	13,830	27,401	26,534	23,976	26,018	25,682	25,107	24,980	24,788	21,739	240,055
		US$/t	0.00	68.89	68.25	66.09	59.72	64.80	63.97	62.53	62.22	61.74	58.28	63.41
		US$/oz	0.00	238.93	254.19	237.01	221.65	221.48	225.53	215.28	243.77	233.22	190.64	227.20
CAPITAL COSTS														
Preproduction Capital	W	000's US$	60,724	43,478										104,202
Sustaining Capital	W	000's US$		3,031	3,331	3,313	4,999	2,928	1,942	707	707	350	0	21,308
Working Capital	E	000's US$	6,001										-6,001	0
Reclamation Cost	W	000's US$											10,000	10,000
Salvage at End of Mine Life	W	000's US$											-10,000	(10,000)
Total Capital Costs	W	000's US$	66,725	46,509	3,331	3,313	4,999	2,928	1,942	707	707	350	-6,001	125,510
FINANCIAL SUMMARY														
Gross Revenue	W	000's US$	0	30,679	57,134	59,335	57,330	62,260	60,352	61,809	54,311	56,333	60,436	559,980
Total Cash Costs	W	000's US$	0	13,830	27,401	26,534	23,976	26,018	25,682	25,107	24,980	24,788	21,739	240,055
Taxes	E	000's US$	0	2,726	3,617	4,147	4,161	4,570	4,137	4,445	2,922	3,292	4,866	38,883
Capital Costs	W	000's US$	66,725	46,509	3,331	3,313	4,999	2,928	1,942	707	707	350	-6,001	125,510
Post-Tax Cash Flow														
Cash Flow		000's US$	-66,725	-32,386	22,785	25,343	24,194	28,744	28,591	31,551	25,701	27,903	39,832	155,532
Accumulated Cash Flow		000's US$	-66,725	-99,112	-76,327	-50,984	-26,790	1,954	30,545	62,095	87,796	115,700	155,532	

		Units	
Discount Rate		%	5.0%
Post-Income Tax Net Present Value (NPV)		million US$	86.675
Post-Income Tax Internal Rate of Return (IRR)		%	19.0%
Initial Capital		million US$	104.2
Cash Operating Costs		US$/t	63.14
		US$/ounce	226.23
Total Cash Costs		US$/t	63.41
		US$/ounce	227.20
Mine Life		Yrs	9.4
Payback Period		Yrs	4.4

19.8 TAXES

Eldorado performed the tax evaluation using the pre-tax model developed by Wardrop. Corporate taxation for Turkish business is currently 20% as reported for Eldorado's Kişladağ operation. Depreciation is based on a unit of production calculation.

19.9 RISKS

19.9.1 INTRODUCTION

Risks are inherent to any major mining project. Early risk identification allows mitigating strategies to be devised and resources to be allowed for their implementation, making the project more robust.

While this section cannot include surprise risks, the risks identified here are comprehensive.

19.9.2 GEOLOGY AND MINERAL RESERVES

Geological risk exists in the offsets between the predicted orebody shape and the actual shape, potentially increasing dilution. This will be mitigated by one or more of the following:

- delineation drilling will define the local orebody geometry and reduce mining development rock
- the mechanized cut-and-fill method will allow the convoluted, discontinuous, and narrow areas of the orebody to be selectively mined
- smaller diameter blastholes at closer spacing will reduce overbreak, thus minimizing dilution and increasing fragmentation
- exploration drilling from underground will improve the certainty of some areas of the orebody.

ANCIENT MINING AND GEOLOGICAL VOIDS

Significant risk surrounds the dimensions and locations of the identified ancient workings and geological voids. Eldorado has modelled the voids and they are incorporated into the block model.

There is a risk that mining personnel safety will be compromised in these areas. This will be mitigated by using delineation drilling to identify the void and then using paste to fill the void. Water or mud in-rush must be considered and assessed. As the development approaches the ancient workings cover drilling will be required to probe the void. Risk assessment will be critical before mining through these areas.

There is a risk that recovery of ore will be affected. This will be mitigated by using paste as an engineered product and increasing ground support systems as required.

19.9.3 MINING RISK

SUSTAINABLE MINE PRODUCTION

There is a risk that overproduction from the longhole stoping methods will impact on the ability to achieve long term production targets. The mining methods must be balanced throughout the life of mine. Mechanized cut-and-fill is the primary mining method. The narrow orebody requires selective mining. This should be managed as outlined in the mining section of this report.

SELECTIVE MINING

There is a risk that mining could become too selective, especially in the longhole mining areas and production targets may not be met. The grade is irregular throughout the orebody and a broad approach will be required in certain areas. The mining operation will need to trust the delineation and assaying results and not restrict the productivity by being too selective. For both mining and processing grade control is critical, mining must diligently follow the production plan for proper ore blending and maximum recoveries in ore processing.

GROUNDWATER INFLOW

Groundwater inflow might be higher than anticipated. In March 2007, a pumping test was conduced in well PW1. The hydrogeologic model was updated after this test and indicates significant increases from previous studies in the predicted mine inflows peaking at 1500 m^3/d. The inflows should be reviewed further in detailed engineering. This risk will be mitigated by installing excess pumping capacity, increasing the number of holding tanks and by grouting drain holes to reduce localized inflows.

MINING CONTRACTOR NON-PERFORMANCE

There is a risk that the mining contractor will not meet the schedule, incurring cost over-runs and delaying the start of production. Monitoring and managing the mining contractor's progress closely will minimize this risk. Unavoidable over-runs are covered by the contingency in the capital cost estimate and conservative development productivity targets have been assumed.

NON-AVAILABILITY OF MINING PERSONNEL

There is a risk that personnel required for underground mining, and other key personnel, will not be available when production begins. This will be mitigated by providing overlap between the pre-production contractor work and the start of owner personnel. Expatriate personnel will be used in key roles at the project start-up.

TRAINING OF MINING PERSONNEL

Although there is an established mining industry and a pool of experienced mine personnel, some untrained local people will be employed by Eldorado. There is a risk that productivity may be adversely affected and production targets not met. This will be mitigated by the mining contractor providing specialized training personnel. Training will be undertaken during the pre-production period and the first six months of production. A training department will be on site during life of mine to train Eldorado employees. Major underground equipment suppliers should provide specialized mechanics to train local personnel on the maintenance and operation of the underground mining equipment.

NON-AVAILABILITY OF MINING EQUIPMENT

There is a risk that mining equipment is not available when required due to long lead times. This will be mitigated by retaining the services of the mining contractor, or the mining contractor's equipment, to cover the shortfall.

The use of São Bento equipment or other second-hand equipment may be possible to cover delays in equipment delivery.

BACKFILL

All mining methods will require backfill. The mining cycle is dependant on backfill, especially the mechanized cut-and-fill method. The availability of the filtration/backfill plant will be critical. Pump spares and sufficient operational consumables including lining and piping must also be available to repair line failures quickly and efficiently.

There is a risk that the lined backfill boreholes could become unserviceable due to a blockage. This will be mitigated by installing two backfill boreholes between each level.

OVERSIZE BROKEN ROCK

The longhole methods may produce oversize rock as a result of ground water inflows or geological structures in the orebody. This risk may be mitigated by using emulsion in wet holes and increasing the powder factor in areas of the orebody that are considered as harder rock. A mobile rock breaker will be used in the case that large rocks report to the drawpoint. Explosives may also be used at the end of shift.

19.9.4 PROCESS AND OPERATIONAL RISK

Further metallurgical testing will ensure that many of the risks identified in this section are mitigated. In particular flotation plant design, thickening and filtering design, concentrate processing, and water balance may be improved through further testing.

MINING PRODUCTION RATES

There is a risk that, at the end of the mining week, ore supply will be insufficient to feed the continuous mill operation on Sundays. There is limited surge capacity in the underground mining process. Broken ore can be stored in the orepass or in the storage bins on surface. This risk will be mitigated by ensuring that the designed surge capacity is filled at the end of the mining week.

ORE ZONE PRODUCTION RATES

The plant has been designed to accommodate a composited or mixed ore feed. In the case that SOS-only or MOS-only unblended ore is delivered to the plant, the operating conditions may become imbalanced, potentially reducing the gold recovery in the process. This risk will be mitigated by adhering to the life of mine production schedule, which includes a proportional balance between the orebodies and mining methods.

SULPHIDIC AND OXIDIZED ORE

The plant design was based on the ore feed being sulphidic in nature. Oxidized ore recoveries are generally lower and the presence of an excessive proportion of oxidized material in the plant feed may lead to reduced recoveries and operational imbalance in the treatment processes. This risk may be mitigated by delineation drilling ahead of production to identify the ore type in each stope to achieve a balance between sulphidic and oxidized material.

FLOTATION PLANT DESIGN

The flotation procedure selected for the Efemçukuru ore employs a unit cell configured with the rougher and cleaner cells which has been operated in other operations but will require larger scale testing. This system has been designed with flexibility and allowances for reconfiguration if necessary. Testwork is slated for the immediate future but is considered to be imperative in order to confirm the viability of this flotation procedure. This risk will be mitigated by further testing and larger scale pilot plant testing.

THICKENING AND FILTRATION DESIGN

Thickening and filtration design parameters have not been tested to the appropriate level of confidence for the design of the plant. Unless the design data is verified, this may affect the plant operations by creating bottlenecks, or resulting in higher moisture products, or possibly leading to higher dissolved gold losses.

TAILINGS FILTRATION AND BACKFILL

Tailings filtration is a potential risk to overall plant operations if the availability does not meet the process plants capacity. To mitigate this situation the design has included a storage tank with approximately 3 hours of tailings surge from the plant. The filters have also been designed to accommodate a 20% increase in filter plates for added capacity. Filtration design parameters will be further tested prior to detailed engineering.

KIŞLADAĞ CONCENTRATE PROCESSING

The regrind mill selected at Kişladağ has not been tested using Efemçukuru flotation concentrate. There is the possibility of under-sizing the regrind mill, or selecting less suitable equipment required for this purpose. The regrind product particle size also requires testing and confirmation.

Testwork is required to confirm the leaching conditions that will be utilized in treating the flotation concentrate, with respect to duration of leach, slurry density, pH, and cyanide concentration.

WATER BALANCE

The water policy has assumed that water can be re-used by recycling continuously in the plant. However, the build-up of dissolved salts and/or impurities could result in plant operations being affected detrimentally. Specifically, there is no data available to indicate whether the build-up of impurity metals such as copper, lead, and zinc in the leach and electrowinning circuit at Kişladağ will reduce the recovery of gold and silver. The system has been designed to introduce fresh water into the system but has emphasis on water conservation and environmental conservation. The availability of fresh water will be further quantified at Efemçukuru but currently indicates that there is excess water available. This risk will be mitigated by further hydrological testing at the Efemçukuru site and further metallurgical testing.

19.9.5 TRANSPORTATION AND LOGISTICS RISK

There is a risk that the Turkish contractor engaged to transport the concentrate will be expensive and not meet availability targets. This may be mitigated by appropriate contractor management including monthly reviews and penalty clauses in the contract. The company has performed preliminary assessments and can assume transportation of the concentrate at anytime. The transportation costs-benefits will be further demonstrated in advance of operations. Alternatively, a separate cooperative could also be setup to handle transportation of concentrate, goods and services, and personnel to both the Efemçukuru and Kişladağ operations.

19.9.6 ENVIRONMENTAL AND PERMITTING RISK

There is a risk of mining being delayed or interrupted by environmental or mine permitting. This will be mitigated by ensuring the appropriate government departments are involved and all permitting requirements satisfied as soon as possible. Various individual permits are still being negotiated but there have been no roadblocks identified to the proponent.

There is a risk of mining being delayed or interrupted by objections from the local residential community due to concerns about dust and noise. The project design mitigates the risk with the placement of mine fans and crusher underground, and minimizing surface truck haulage; storage of run-of-mine ore will be in bins on surface for dust suppression; and internal site roads and access road will be sealed to reduce dust.

19.9.7 PROJECT EXECUTION AND COMPLETION RISK

A number of risks may affect the project execution plan including:

- timely completion of permitting and land acquisition
- shortage of key personnel (management, engineering, supervisory, and artisans) will be mitigated by ensuring early placement of contracts, prompt and effective recruiting at start of project, and the expanded use of contractors and consultants as required
- shortage of construction equipment (cranes, modular site buildings, etc.) will be mitigated by ensuring early placement of orders for purchase and contracts for lease of construction equipment and followed by effective expediting
- shortage of contractors (mining, construction, earthworks, and catering) will be mitigated by obtaining early commitment from contractors
- long lead times on capital equipment delivery will be mitigated by ensuring orders are placed early with different vendors and followed by appropriate expediting
- increased excavation time and cost from adverse geotechnical conditions will be mitigated by assessing site conditions and re-evaluating the ground support systems.

19.9.8 MANAGEMENT RISK

In the early project life, senior management will include Western expatriate employees. These employees will be responsible for setting up an appropriate organizational structure to implement and manage the operational systems on site to achieve the long-term life of mine plan. The expatriate personnel will be replaced by local personnel once sufficient training is completed. Management and control systems implemented at Kişladağ will be used as a guideline for the setup of the operation at Efemçukuru.

19.9.9 POLITICAL RISK

There is a risk that the mine operation may be affected by litigation or other political risks. This may be mitigated by following the permitting process diligently and ensuring that all permitting is completed as soon as possible, in addition to working closely with all levels of government to insure confidence in a responsible execution and operation of the project.

19.9.10 FORCE MAJEURE RISK

Eldorado reserves the right to cancel, vary, or suspend the operation of contract of sale if events occur which are in the nature of force majeure including (without prejudice to the generality of the foregoing): fire, floods, storm, plant breakdown, strikes, lock-outs, riots, hostilities, non-availability of materials or supplies, or any other event outside the control of Eldorado. Eldorado shall not be held liable for any breach of contract resulting from such events.

19.9.11 ECONOMIC RISK

19.9.12 ECONOMIC RISKS WILL BE MITIGATED BY IMPLEMENTING STRATEGIES TO MONITORING EXCHANGE RATES, METAL PRICES, AND CONTRACT TERMS OVER LIFE OF MINE (OVERALL RISK ASSESSMENT

The risk factors listed for this project are typical for mining projects of this size. The greatest risk will be the definition of the orebody and controlling the mining direction to minimize dilution and maximize the recovery of gold ounces.

Table 19.48 overleaf).

19.9.13 OVERALL RISK ASSESSMENT

The risk factors listed for this project are typical for mining projects of this size. The greatest risk will be the definition of the orebody and controlling the mining direction to minimize dilution and maximize the recovery of gold ounces.

Table 19.48 Economic Risks

Identified Economic Risk	Mitigating Action
Fluctuating exchange rates	Monitor exchange rate trends and implement strategies (e.g. hedging)
Fluctuating metal prices	Monitor metal price trends and implement forward selling strategies (e.g. hedging) if deemed necessary
Changing smelter terms	Monitor smelter term trends and implement strategies including the use of alternate smelters and long-term contracts
Capital over-run	Financial resilience of the project will make moderate capital over-runs non-fatal
Operating cost over-run	Operating cost estimations have been conservatively calculated, with a high degree of detail backed by operational data from Kişladağ and other mining operations. Project strategies will include long term labour and consumable contracts to minimize and moderate operating cost over-runs.

19.10 OPPORTUNITIES

A number of project opportunities have been identified during the feasibility study. The following discussion has not been included in any planning, costing or economic evaluation for this project.

19.10.1 EXPLORATION POTENTIAL

The measured and indicated resource continues to be developed through ongoing drill programs on site. The resource in the feasibility was updated during the mine design. Stope outlines were based on the current block model. The mineral reserve reporting within the stope outlines was estimated with the latest updated block model. Some newly defined measured and indicated resources are therefore not included in the reserve and fall within the 92% recovery. The mine reserve will be updated at the end of the current drilling program.

Inferred mineral resources of 753,000 tonnes at a gold grade of 8.79 g/t totalling 213,000 oz may be converted to Measured or Indicated mineral resources by in-fill and exploration diamond drilling.

Figure 19.20 shows the current drilling progress at Efemçukuru.

Mine exploration from underground will be required early in the mine life. During pre-production access to the orebody from the North Ramp will be used for exploration drilling of the GAP or transition zone between the SOS and MOS. The SOS and the MOS are both open down-dip and require further exploration drilling.

19.10.2 SILVER

Exploration drilling at the Efemçukuru property has revealed the presence of silver in the orebody. The presence of silver in the deposit was not considered as part of this study and was not included in the economic evaluation.

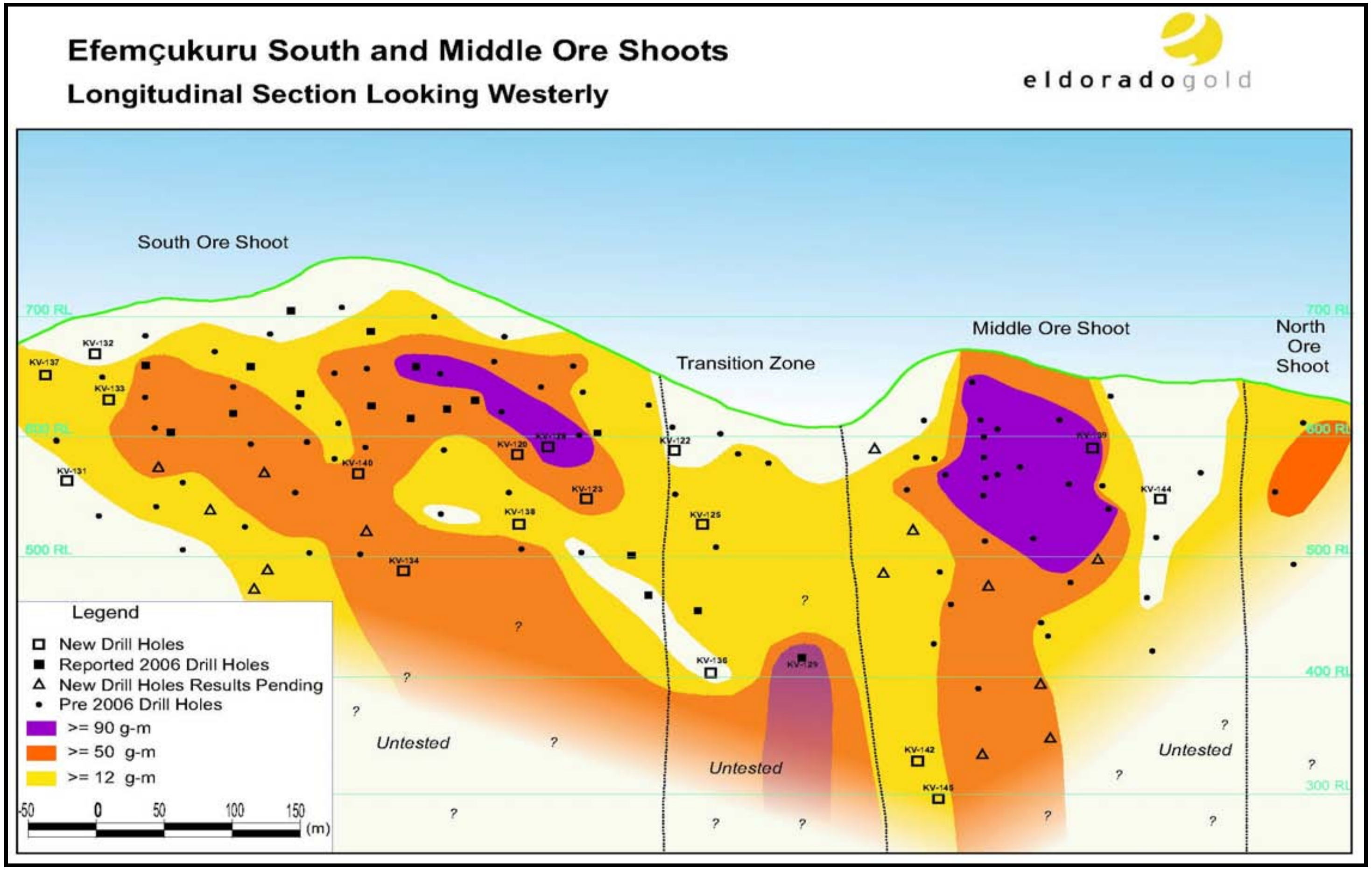
The results from Eldorado's geological model estimated with the ID3 (inverse distance to the third power) method are shown in Table 19.49. The silver resources listed are considered inferred mineral resources. The silver content is based on the results from the measured and indicated gold blocks in the current geological model and are inside the same mineralized shells.

Table 19.49 Inferred Silver Resources

Gold Cut-off Grade (g/t)	Ore (Mt)	Silver Grade (g/t)	Silver (M oz)
6.0	2.7	18.42	1.6
5.0	3.2	17.96	1.8
4.5	3.4	17.79	1.9
3.0	4.0	17.11	2.2

These inferred resources have not been adjusted for voids in the orebody.

Figure 19.20 Exploration Drilling Longitudinal Section



19.10.3 Mining

The minimum mining width used in this study is 2 m. Areas in the orebody that are less than 2 m are currently excluded from the reserve tonnage. Some of these areas include measured and indicated reserves above the 4.5 g/t design mine cut-off grade. These narrow areas may account for approximately 2-3% of resource ounces not recovered. If the narrow area is of sufficient grade, the full 2 m may be mined with the additional dilution. Alternatively, the narrow areas may also be mined using slushing or resue stoping to minimize the dilution. These options were not considered in this study.

Overhand mechanized cut-and-fill mining method has been used in this study. Attack ramps have been designed for each lift with ground support for each cut. On a retreat basis, the top lift could be mined using a double lift to reduce stope bolting requirements and costs. Increased remote controlled mucking may reduce the mucking productivity. Taking double lifts will depend on ground conditions and dilution would need to be considered.

The overall cement content in the paste may be reduced.after the required material strength and reactivity tests are completed during detailed engineering.

The proposed ground support has been conservatively designed based on the geotechnical data received from Eldorado. Further review of the geotechnical data once mine development commences may allow for reduction of the standard ground support requirements. Swellex ground support systems may be used in short term openings.

The mine design may be reviewed at detailed engineering to optimize the access to the orebody on each level reducing the waste development costs.

Eldorado's São Bento mine in Brazil has a range of underground mining equipment available for use at Efemçukuru from the mine closure. The equipment would have to meet EU standards for importation into Turkey. Underground equipment may include loaders, light vehicles, pneumatic longhole drills, surface forklifts, and ventilation fans.

The orebody at Efemçukuru is complex and narrow in some areas. The mining productivity may be enhances at detailed engineering. An increase in productivity would increase the cash flow and potentially improve the internal rate of return of the project. This would be achieved by increasing the number of working areas available underground and by modifying the process plant to accommodate the increased throughput. However, this will increase the number of sill mats required and potentially create the need for pillars in the orebody.

19.10.4 Processing

Recovery of gold may be improved by optimizing the gravity and flotation processes, as well as the concentrate leaching process.

Metallurgical testing will indicate the required changes to the processes, and such changes may be implemented at little cost. Improving concentrate quality will significantly benefit the project economics.

- The potential exists to increase plant efficiencies in the following areas:
- improve the recovery, and possibly the grade, of the flotation concentrate produced
- optimize the primary grind size and the regrind size
- optimize the cyanidation leach conditions.

Recoveries for the feasibility study were calculated using existing metallurgical test work results. Early indications from current test work indicate that the average recovery of 86.5% may be improved.

The process designed is very robust and capable of handling large variations in ore grade, variations between the ore types found in the MOS and SOS ore zones, and both oxidized and sulphur ore. With further metallurgical testing it is believed that the system can be further refined and downsized to reduce the capital cost during detailed engineering after the current metallurgical test program.

Eldorado's São Bento mine in Brazil has a range of process equipment available for use at Efemçukuru from the mine closure including flotation cells, gravity concentrators and pumping equipment. Other instrumentation and lab equipment may also be available. This equipment could be used to reduce the initial capital cost.

19.10.5 Logistics

Concentrate transport cost was quoted by Turkish logistic companies. A trade-off study may be conducted during detailed engineering, comparing third party shipping versus owner shipping.

A trade-off study will also be performed to compare the cost of delivering the concentrate by rail verse the road transport.

19.10.6 Initial Capital Reduction

The site layout was designed around the property boundary adjacent to the process plant. Eldorado is currently negotiating the acquisition of neighbouring properties to allow further expansion of the surface facilities and potentially improve the layout design.

Site buildings were designed using typical Turkish design and construction methods. Pre-fabricated structures may provide some potential cost savings.

All equipment was quoted as new from both Western and European suppliers. Eldorado is currently investigating Asian suppliers and the used equipment market for purchase of the SAG and ball mills. As previously mentioned São Bento mining equipment may also improve the initial capital cost of the project.

SECTION 20 • INTERPRETATION AND CONCLUSIONS

Wardrop reviewed pertinent data from the Efemçukuru Project to obtain a sufficient level of understanding to assess the mineral resource and mineral reserve estimate for the Efemçukuru deposit, and to assess the economic and practical feasibility of the Efemçukuru mine. Wardrop's conclusions from this review are:

1. The geological data of the Efemçukuru project is of adequate density and reliability to provide a good understanding of the Efemçukuru orebody.

2. The mineral resource estimate was developed using industry-accepted methods, and ore was classified using logic consistent with the CIM definitions referred to in NI 43-101 into Measured, Indicated, and Inferred Mineral Resources.

3. The mineral reserve estimate contained within the resource estimate was based on the conservative application of standard mining practices, and ore was classified using logic consistent with the CIM definition referred to in NI 43-101 into Probable and Proven Mineral Reserves.

4. The proposed metallurgical process and predicted metal recovery values are the result of a systematic evaluation of samples adequately representing the ore body, based on the results of testwork programs. The proposed mineral extraction process is consistent with industry-accepted mineral concentration methods. Plant efficiency could be improved with additional test work.

5. Infrastructure and logistics are based on proven technologies and industry-accepted standards which are consistent with similar mining operations in North America and Turkey

6. Revenue projections based on mineral marketing costs and future metal pricing are considered adequate to predict revenue from this project. Projected metal prices are conservative when compared to the method of a two or three-year historical average metal price, and Wardrop confirms that this is a fair assessment of probable future revenue from the project.

7. A before-tax economic analysis of this project predicts a pre-tax IRR of 23.2% and a pre-tax NPV of US$115.1 million at a discount rate of 5.0. Payback is 3 years 9 months. An after-tax economic analysis which considers all current tax rules indicates that an IRR of 19% and a NPV of $86.7 M based on a discount rate of 5% can be expected from this project.

SECTION 21 • RECOMMENDATIONS

21.1 EXPLORATION RECOMMENDATIONS

21.1.1 EXPLORATION POTENTIAL

In addition to the current in fill and deep drilling program, Eldorado has begun the drilling for metallurgical test samples. When the current program is completed the resources and reserve estimates will be reassessed. The vein remains open both at depth and strike length may be increased in both the south and north directions. Additional drilling is also planned in the North Ore Shoot which to date has not been evaluated.

21.1.2 SILVER EXPLORATION AND MODELING

This report does not include silver in either the economic evaluation or ore body resource and reserve modelling. Eldorado is currently assaying for silver and has completed preliminary block modeling to define the silver. The high silver grades shown in areas will allow for some additional value to be added to the ore, increasing the value of the current model.

In Dr. Steve Juras's opinion, the Qualified Person for geological matters in this Technical Report, the character of the property is of sufficient merit to justify the programs recommended above.

21.2 METALLURGICAL TESTWORK RECOMMENDATIONS

Although the previous testwork showed that the mineral samples responded well to metallurgical processes, further testing is recommended to improve confidence in the results over a larger sampling of the ore body. The proposed metallurgical drilling has targeted the ore zones associated with the first 4 years of mining.

The following process review summary and remarks highlight the need to undertake additional metallurgical testwork to ensure that the concentrator and concentrate treatment plant designs will be sufficiently robust to operate within the design parameters selected in treating a variable ore feed grade. A general description of the proposed testwork follows below. These testwork programs are planned for completion during 2007 and early 2008.

21.2.1 EFEMÇUKURU PLANT

MATERIALS HANDLING

A bulk sample of ore (main ore type or blended sample) will be submitted for materials flow testwork to establish design parameters for bins, conveyors, feeders and chutes, as well as to establish the likely bounds of retained moisture in the various process products.

COMMINUTION

Confirmatory testing of the main ore type, or a blended composite sample in the projected feed blend, will be required to determine the following parameters:

- unconfined compressive strength tests
- impact crushing tests on tumbled whole rock or drill core (minimum of HQ size, but preferably PQ size) to establish impact crushing profile for at least four size classes, and encompassing the size range 19 mm to 85 mm, if possible. Core or rock is tumbled in an Allis-Chalmers 6 ft x 1 ft tumble drum for 500 revolutions to break up imperfections. Pebbles produced are removed, screened into size classes and individually tested (preferably 20 per size class) in a twin pendulum breakage device. Where whole rock is used, the product of the tumble drum is sized, so that the data can be used for comparison with autogenous tumble test key indicators
- bond ball mill work index tests, the latter at various grind sizes
- bond crushing work index test
- bond abrasion index test
- JK Simet drop weight test.

Data from these tests can then be used to construct models for simulation of the grinding circuit and generating size distributions that can be used to help vendors correctly size screens, cyclones and pumps.

In addition, the following tests are recommended to characterize the variability of the ore samples:

- unconfined compressive strength tests
- bond abrasion index test
- bond ball millwork index tests, the latter at optimized grind sizes as determined in the flotation tests.

Gravity Concentration

Tests have been undertaken on the main ore types, or a blended composite sample, to gain a full appreciation of the concept of gravity recoverable gold (GRG) and to implement the results of these tests into the design, wherever applicable. This work should follow the standard GRG test procedures.

Flotation

Additional testwork will be carried out to further characterize flotation performance to confirm that the flowsheet developed is sufficiently robust to cater for a wide range of metallurgical ore type behaviour. Other aspects will be aimed at reducing reagent consumption or simplifying the operability of the flotation flowsheet, thus offering operating and/or capital cost savings.

The following testwork will be undertaken on the main composite sample with variability testing of the different ore sample types being subjected to optimize the flotation conditions:

- flash flotation behavior concerning mass recovery, gold/silver grade and recovery, sulphide sulphur grade and recovery, required residence time, and the effect of using a surface activator reagent, on the appropriate sample material (i.e. after gravity concentration)
- optimization of primary grind size. A coarser grind than 80% passing 67 microns may be achievable, especially with flash flotation ahead of scavenger flotation
- optimization of the present reagent suite
- optimization of scavenger flotation residence time
- cleaner circuit optimization and locked-cycle testing with scavenger flotation
- the proposed cleaning of the scavenger concentrate in the flash flotation cell requires testing. The alternative of upgrading the scavenger concentrate in the cleaner flotation circuit requires validation
- investigation into an additional stage of cleaner flotation to reduce concentrate mass and boost concentrate grade should be tested
- investigation into the use of centrifugal gravity concentration to produce a smeltable grade of product, or alternatively an upgrade of the cleaner gravity and flotation concentrates to reduce the mass of each respective concentrate, should also be tested
- the effect of recycling tailings water to flotation feed thereby emulating the closed water balance of the Efemçukuru plant should be examined

- flash flotation concentrate upgrading using gravity concentration techniques should be quantified.

Bulk Flotation Concentrate and Tailings Preparation

Bulk flotation concentrate and flotation tailings will be required for dewatering testwork and concentrate treatment testwork.

Flotation Concentrate Dewatering

Concentrate thickening test data is required to confirm sustainable thickener underflow density, overflow clarity, settling rates, flocculant selection, and flocculant addition rates.

Flotation concentrate filtration tests are required to confirm filtration form rates, the number of wash ratios required, the number of wash stages required, the optimum filter cake thickness, the filter cake moisture content, and the filtrate recovery rates. This information will be used for the detailed design of the required filtration circuit.

21.2.2 Kişladağ Plant

Materials Handling

Samples of flotation concentrate and leach residue will be submitted for materials flow testwork to establish the design parameters for the bins, conveyors, feeders, and the chutes, as well as to determine the bounds of retained moisture in the various process products.

Concentrate Regrinding

Flotation concentrate representing average conditions will be subjected to IsaMill Xstrata Technology testwork in order to produce a relationship of grind size versus specific power draw for screened sand grinding media and ceramic grinding media.

A preliminary analysis using cost input and consumables data supplied by Xstrata Technology indicates that ceramic media is more cost effective, but confirmatory tests are recommended, especially if acceptable local sand media in Turkey can be found. Optimum grind size will be determined during the cyanide leaching testwork.

Cyanide Leaching

The following cyanidation parameters require confirmation:

- confirmation will be obtained of the optimum regrind particle size in the leaching process using a bench scale glass bead stirred mill to prepare flotation concentrate samples ground to different grind sizes. Sizes of 80%

passing 45, 30, 25, 20, 15, and 10 microns should be examined. Grind size will be selected based on incremental net revenue taking into account power, reagents, grinding media consumption, and gold/silver extraction

- the leach residence time to be characterized
- the dissolved oxygen concentration requirements to be determined
- the optimized cyanide and caustic additions and consumption values to be established
- quantification is required regarding the cyanide concentration in the leach feed slurry and the leach residue slurry
- the use of spent electrolyte in the leach circuit should be investigated concerning the potentially detrimental effects to dissolution and reagent consumption because of the building up of deleterious dissolved impurity elements
- the potential benefit of lead nitrate addition during the leach process to enhance leach kinetics should be investigated
- determine the milling of flotation concentrate using cyanide-bearing solution, and its effect on the leach kinetics and reagent consumptions
- a full ICP analysis on leach feed, leach residue solids, and leach pregnant solution from the optimum condition leach tests carried out
- a bulk leach slurry to be tested to determine the residue solids thickening and dewatering characteristics, and to establish the gold/silver recovery parameters from pregnant solution
- the particle size distribution analysis of solids is required for agitator and pump selection
- viscosity testwork for agitator and pump selection will also be conducted.

LEACH RESIDUE DEWATERING

Leach residue filtration data is required to confirm filtration form rates, the wash ratio required, the number of wash stages required, filter cake thickness formation, typical filter cake moisture contents, filtrate recovery, and ultimately the required filter size and/or type of filter best suited to the application.

METALS RECOVERY

Electrowinning of gold and silver from leach liquor will be tested to determine whether there could be any potential problems with interference from impurity metals such as copper, zinc, iron and other deleterious impurity elements. Confirmatory testwork will also be conducted to determine a recommended solution temperature for the electrowinning process. The initial test should be performed at 85°C in open circuit and closed circuit to determine the pass efficiency of each cell and quantify the resultant spent electrolyte metal tenors. These results can then be used to confirm the selection of the equipment.

21.3 Geotechnical Testwork Recommendations

Wardrop recommends further geotechnical assessment to determine ground support for development intersections and optimizing group support for the detailed design including the following:

- the crown pillar should not fall below a thickness of 10 m in both the MOS and SOS
- structurally controlled failures and unravelling failures could be potential problems at the site. Ground control crew should make note of brecciated zones and jointing, and review the need for additional ground support in those areas
- the minimum recommended ground support for the access development is based on the NGI support method, and can be summarized as follows:
    - pattern bolting using 2.4 m long rock bolts on a 1.2 m x 1.2 m pattern is recommended for the back and shoulder of all Ramps/Drifts with widths of 4.5 m and life expectancies of more than 3 years
    - only Spot bolting, as and where required, for all Access Sills with widths of 3.5 m, and life expectancies of less than 3 years.
- for the purpose of Cost Estimating in the Efemçukuru Feasibility Study, the minimum recommended Ground Support Requirements for stopes in the MOS and SOS zones are:
    - spot bolting only, as and where required, for spans up to 6 m
    - pattern bolting using 2.4 m long rock bolts on a 1.2 m x 1.2 m pattern for spans up to 10 m.

21.4 Paste Backfill Testwork Recommendations

A detailed rheology, cement screening, and strength testing is recommended on the tailings from Efemçukuru ore to determine its engineering parameters for detailed engineering design and economic analysis.

The tailings will be tested to determine different binder contents to optimize binder consumption, and to determine the final strength requirements, the following tests are required in detailed engineering:

- rheologic index testing
- cement and binder screening
- uniaxial compressive strength testing.

21.5 Mine Ventilation Airway Survey Recommendations

A detailed design for the planned ventilation circuit is recommended.

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SECTION 23 • CERTIFICATES OF QUALIFIED PERSONS

SECTION 23 • CERTIFICATES OF QUALIFIED PERSONS

23.1 SIGNATURE PAGE, DATE, AND CERTIFICATES

The effective date of this report is August, 2007. Signed the 17th of September, 2007.

SIGNED

"Original Document, Revision 00 signed and sealed by Andy Nichols, P.Eng."

Andy Nichols, P.Eng.
Chief Mining Engineer
Wardrop Engineering Inc.

"Original Document, Revision 00 signed and sealed by Steve Juras, P.Geo."

Steve Juras, P.Geo.
Manager of Geology
Eldorado Gold Corp.

"Original Document, Revision 00 signed and sealed by Richard Alexander, P.Eng."

Richard Alexander, P.Eng.
Senior Mechanical Consulting Engineer
RPM Technical Services Ltd

"Original Document, Revision 00 signed and sealed by Andre de Ruijter, P.Eng."

Andre de Ruijter, P.Eng.
Senior Metallurgist
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Michael Andrew Nichols, of Vancouver, British Columbia, do hereby certify that as the author of this **TECHNICAL REPORT ON THE EFEMÇUKURU PROJECT**, dated August, 2007, I hereby make the following statements:

- I am Chief Mining Engineer with Wardrop Engineering Inc. with a business address at #800 – 555 W. Hastings St., Vancouver, BC.
- I am a graduate of Camborne School of Mines, England (ACSM, 1973).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia *(*Registration #125865*).*
- I have practiced my profession continuously since graduation.
- 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 purpose of NI 43-101.
- My relevant experience with respect to underground mining includes 31 years.
- I am responsible for the preparation of Sections 1.0 to 5.0, 17,19.0, and 21.0 to 22.0 of this technical report titled "Technical Report on the Efemçukuru Project ", dated August 31, 2007.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 17th day of September, 2007 at Vancouver, British Columbia.

"Original Document, Revision 01 signed and sealed by M.A. Nichols, P.Eng.,"

M.A. Nichols, P.Eng.
Chief Mining Engineer
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

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

- I am a member of the Association of Professional Engineers and Geoscientists of British Columbia.
- I graduated from the University of Manitoba with a Bachelor of Science (Honours) degree in geology in 1978 and subsequently obtained a Master of Science degree in geology from the University of New Brunswick in 1981 and a Doctor of Philosophy degree in geology from the University of British Columbia in 1987.
- I have practiced my profession continuously since 1987 and have been involved in: mineral exploration and mine geology on copper, zinc, gold and silver properties in Canada, United States, Brazil, China and Turkey; and ore control and resource modelling work on copper, zinc, gold, silver, tungsten, platinum/palladium and industrial mineral properties in Canada, United States, Mongolia, China, Brazil, Turkey, Peru, Chile, Portugal, Australia, Vietnam and Russia.
- As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.
- I was responsible for reviewing matters related to the geological data and directing the mineral resource estimation and classification for the Efemçukuru Project in Turkey. I am responsible for the preparation of Sections 6 to 15 and Section 17 (sub-section 1) of the report entitled Efemçukuru Project, Turkey, Technical Report with an effective date of August 2007.
- I visited the project site on numerous occasions in 2006 and 2007. My most recent visit to the project site was from 20 April 2007 to 22 April 2007.

  I have not had prior involvement with the property that is the subject of this technical report.
- I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.4 of National Instrument 43-1 01.
- I have read National Instrument 43-101 and Form 43-1 01 Fl and the sections for which I am responsible in this report entitled, Efemçukuru Project, Turkey, Technical Report with an effective date of August 2007, has been prepared in compliance with same.

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

I consent to the filing of the technical report entitled, Efemçkuru Project, Turkey, Technical Report with an effective date of August 2007, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of this report.

- Signed and dated this 17th day of September, 2007 at Vancouver, British Columbia.

"Original Document, Revision 01 signed and sealed by Stephan J. Juras, P.Eng., P.Geo."

Stephan J. Juras, Ph.D., P.Geo.
Manager of Geology
Eldorado Gold Corp.

CERTIFICATE OF QUALIFIED PERSON

I, Richard C. Alexander, P.Eng, of Vancouver, British Columbia, do hereby certify that as the author of this **TECHNICAL REPORT ON THE EFEMÇUKURU PROJECT**, dated August, 2007, I hereby make the following statements:

- I am a Senior Mechanical Consulting Engineer with RPM Technical Services Ltd. with a business address at 5922 Boundary Place, Surrey, B.C.
- I graduated with a degree in B.Sc. (Mechanical Engineering) from the University of Alberta in 1985.
- I am a member in good standing of the Association of Professional Engineers in the Province of British Columbia.
- I have worked in engineering, construction management and project management in the minerals industry for 20 years.
- 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.
- I am responsible for the preparation of Section 18.0 of this technical report titled "Technical Report on the Efemçukuru Project ", dated August 2007. In addition, I visited the Efemçukuru property on September 18, 2006.
- I have had prior involvement with the property while working on the Efemçukuru Project Prefeasibility dated January 1999 with Kilborn Engineering Pacific Ltd.
- As of the date of this Certificate, to my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 17th day of September, 2007 at Vancouver, British Columbia

"Original Document, Revision 01 signed and sealed
by Richard C. Alexander, P.Eng.,"

Richard C. Alexander, P.Eng.
Senior Mechanical Consulting Engineer
RPM Technical Services Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Marinus Andre de Ruijter, of Delta, British Columbia, do hereby certify that as the author of this **TECHNICAL REPORT ON THE EFEMÇUKURU Feasibility Study**, dated August 2007, I hereby make the following statements:

- I am a Senior Metallurgical Engineer with Wardrop Engineering Inc. with a business address at #800 – 555 W. Hastings St., Vancouver, B.C.
- I am a graduate of the University of Witwatersrand, Johannesburg, South Africa (M.Eng., 1979).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia *(*Registration #31031*).*
- I have practiced my profession continuously since graduation, except during the years 2000 to 2004.
- 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 purpose of NI 43-101.
- My relevant experience includes gravity concentration and flotation research and development work on cassiterite, wolframite, and chromite ores.
- I am responsible for the preparation of Section 16.0 and 20 of this technical report titled “Efemçukuru Feasibility Study “, dated August 2007.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 17th day of September, 2007 at Vancouver, British Columbia

“Original Document, Revision 01 signed and sealed by M.A. de Ruijter, P.Eng.”

M.A. de Ruijter, P.Eng.
Senior Metallurgist
Wardrop Engineering Inc.

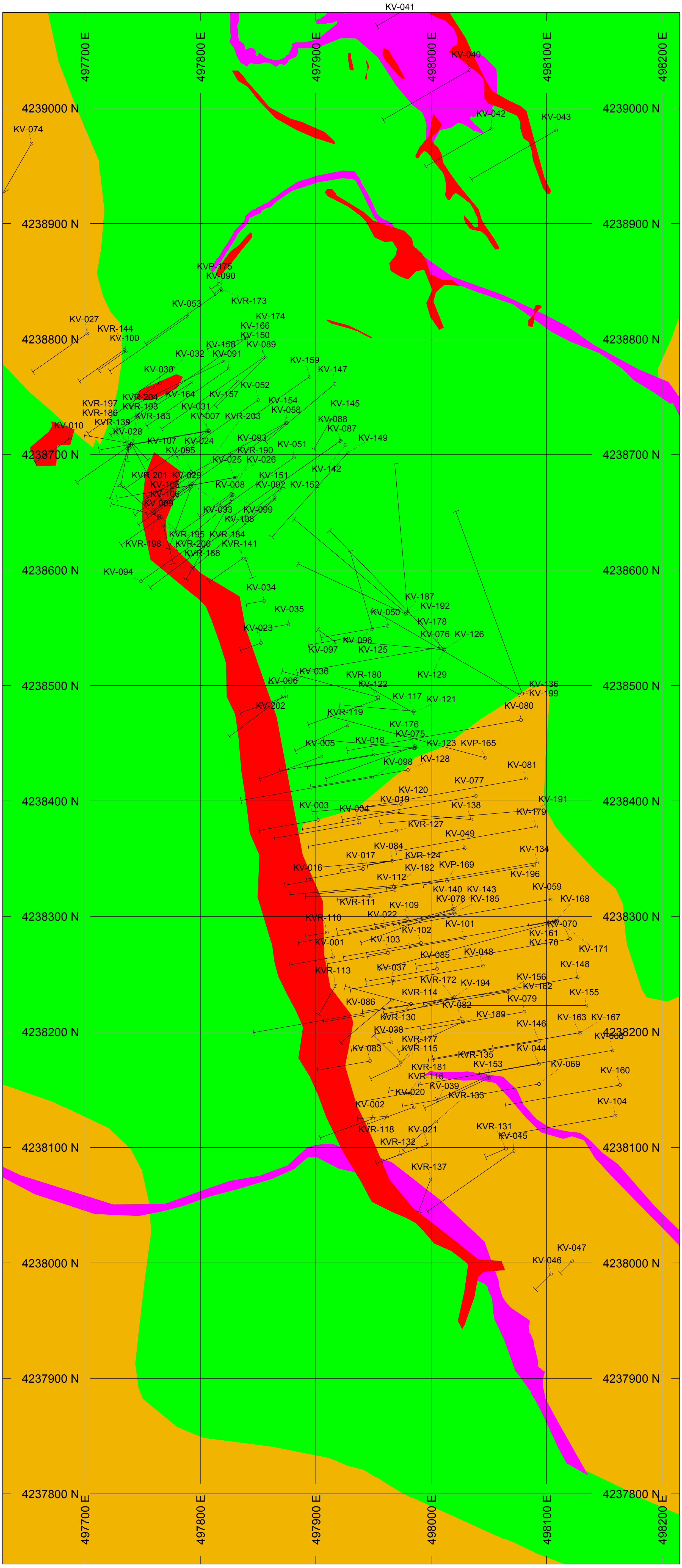
APPENDIX A

DRILL HOLE LIST
DRILL HOLE LOCATION MAP
LIST OF COMPOSITE DATA

Hole ID	Easting	Northing	Elevation	Azimuth	Inclination	Depth(m)	Shoot	Hole Type
KV-001	497,915	4,238,265	729.5	260	-45	54.00	SOS	CORE
KV-002	497,950	4,238,125	697.2	250	-45	69.00	SOS	CORE
KV-003	497,902	4,238,384	708.4	260	-45	73.50	SOS	CORE
KV-004	497,938	4,238,381	707.4	260	-45	105.20	SOS	CORE
KV-005	497,905	4,238,438	683.3	250	-45	80.20	SOS	CORE
KV-006	497,874	4,238,491	652.1	250	-45	60.00	SOS	CORE
KV-007	497,806	4,238,721	636.2	260	-45	134.70	MOS	CORE
KV-008	497,828	4,238,661	629.0	210	-45	112.80	MOS	CORE
KV-009	497,766	4,238,646	658.1	235	-45	59.40	MOS	CORE
KV-010	497,687	4,238,714	641.7	235	-45	51.80	MOS	CORE
KV-016	497,896	4,238,331	725.7	260	-65	55.50	SOS	CORE
KV-017	497,941	4,238,341	706.4	260	-55	86.90	SOS	CORE
KV-018	497,950	4,238,440	697.5	260	-50	128.10	SOS	CORE
KV-019	497,973	4,238,390	694.2	260	-50	132.60	SOS	CORE
KV-020	497,985	4,238,135	679.3	250	-60	85.10	SOS	CORE
KV-021	497,997	4,238,103	672.8	250	-45	59.40	SOS	CORE
KV-022	497,959	4,238,290	705.7	260	-45	106.70	SOS	CORE
KV-023	497,853	4,238,537	627.9	250	-70	54.90	SOS	CORE
KV-024	497,808	4,238,720	636.0	260	-70	201.00	MOS	CORE
KV-025	497,830	4,238,680	630.0	260	-45	146.30	MOS	CORE
KV-026	497,831	4,238,680	629.9	260	-70	167.60	MOS	CORE
KV-027	497,702	4,238,805	597.0	235	-45	81.68	MOS	CORE
KV-028	497,738	4,238,708	644.0	235	-45	78.06	MOS	CORE
KV-029	497,791	4,238,670	651.0	235	-45	76.10	MOS	CORE
KV-030	497,765	4,238,762	620.8	235	-45	107.70	MOS	CORE
KV-031	497,807	4,238,721	636.1	235	-45	123.83	MOS	CORE
KV-032	497,776	4,238,775	619.7	235	-70	121.00	MOS	CORE
KV-033	497,826	4,238,659	629.0	230	-50	96.31	MOS	CORE
KV-034	497,855	4,238,573	612.0	260	-70	46.14	SOS	CORE
KV-035	497,876	4,238,553	631.1	260	-70	80.65	SOS	CORE
KV-036	497,884	4,238,506	652.9	260	-70	73.76	SOS	CORE
KV-037	497,968	4,238,244	710.7	260	-70	111.60	SOS	CORE
KV-038	497,965	4,238,191	692.8	260	-70	102.71	SOS	CORE
KV-039	498,006	4,238,141	668.8	260	-70	92.00	SOS	CORE
KV-044	498,093	4,238,172	666.9	260	-45	149.96	SOS	CORE
KV-045	498,072	4,238,097	640.7	235	-45	128.62	SOS	CORE
KV-046	498,104	4,237,990	631.6	225	-65	44.81	SOS	CORE
KV-047	498,122	4,238,001	630.3	225	-75	55.42	SOS	CORE
KV-048	498,045	4,238,257	675.8	260	-60	159.71	SOS	CORE
KV-049	498,029	4,238,359	665.1	260	-60	171.28	SOS	CORE
KV-050	497,962	4,238,552	654.4	260	-70	171.29	SOS	CORE
KV-051	497,881	4,238,697	606.3	235	-60	149.96	MOS	CORE
KV-052	497,850	4,238,747	604.2	235	-65	203.27	MOS	CORE
KV-053	497,789	4,238,819	595.3	235	-65	194.13	MOS	CORE
KV-058	497,875	4,238,727	606.8	235	-80	256.02	MOS	CORE
KV-059	498,103	4,238,315	640.0	260	-50	206.20	SOS	CORE
KV-068	498,157	4,238,184	647.2	260	-55	203.70	SOS	CORE
KV-069	498,093	4,238,155	663.9	260	-65	154.80	SOS	CORE
KV-070	498,120	4,238,280	646.3	260	-50	210.80	SOS	CORE
KV-075	497,985	4,238,446	686.2	260	-50	146.30	SOS	CORE
KV-076	498,008	4,238,533	666.3	260	-45	177.80	SOS	CORE
KV-077	498,039	4,238,404	662.5	260	-45	166.30	SOS	CORE
KV-078	498,020	4,238,302	673.8	260	-45	131.30	SOS	CORE
KV-079	498,081	4,238,218	672.6	260	-60	182.30	SOS	CORE
KV-080	498,078	4,238,470	642.3	260	-45	216.30	SOS	CORE
KV-081	498,082	4,238,419	641.3	260	-45	203.30	SOS	CORE
KV-082	498,026	4,238,211	689.3	260	-55	131.30	SOS	CORE
KV-083	497,947	4,238,175	693.2	260	-45	65.30	SOS	CORE
KV-084	497,967	4,238,348	695.0	260	-70	135.00	SOS	CORE
KV-085	498,005	4,238,255	693.0	260	-60	125.30	SOS	CORE
KV-086	497,941	4,238,215	711.4	260	-45	49.70	SOS	CORE
KV-087	497,921	4,238,712	585.7	235	-55	203.30	MOS	CORE
KV-088	497,922	4,238,712	585.7	235	-75	242.30	MOS	CORE
KV-089	497,855	4,238,784	580.3	235	-60	215.30	MOS	CORE
KV-090	497,819	4,238,842	574.9	235	-65	190.50	MOS	CORE
KV-091	497,825	4,238,775	602.2	235	-60	173.30	MOS	CORE
KV-092	497,864	4,238,661	607.6	230	-45	131.30	MOS	CORE

Hole ID	Easting	Northing	Elevation	Azimuth	Inclination	Depth(m)	Shoot	Hole Type
KV-093	497,827	4,238,700	632.4	235	-45	116.30	MOS	CORE
KV-094	497,749	4,238,590	663.3	55	-45	89.30	MOS	CORE
KV-095	497,794	4,238,683	651.7	235	-45	87.00	MOS	CORE
KV-096	497,949	4,238,549	654.3	344	-55	122.20	SOS	CORE
KV-097	497,917	4,238,538	652.4	305	-77	83.30	SOS	CORE
KV-098	497,949	4,238,420	699.8	260	-50	179.30	SOS	CORE
KV-099	497,828	4,238,665	629.1	235	-55	101.30	MOS	CORE
KV-100	497,735	4,238,789	594.9	235	-75	110.30	MOS	CORE
KV-101	498,029	4,238,281	674.6	260	-50	133.30	SOS	CORE
KV-102	497,991	4,238,277	694.0	260	-55	119.30	SOS	CORE
KV-103	497,963	4,238,268	709.1	260	-50	83.30	SOS	CORE
KV-104	498,160	4,238,127	637.7	260	-50	156.50	SOS	CORE
KV-105	497,793	4,238,674	650.9	235	-55	83.30	MOS	CORE
KV-106	497,793	4,238,674	650.9	235	-65	96.80	MOS	CORE
KV-107	497,793	4,238,674	650.9	235	-75	107.30	MOS	CORE
KV-108	497,793	4,238,674	650.9	235	-85	110.30	MOS	CORE
KV-109	497,979	4,238,298	695.5	260	-50	96.10	SOS	CORE
KVR-110	497,910	4,238,286	729.7	260	-72	60.00	SOS	RC
KVR-111	497,948	4,238,317	706.4	270	-45	80.00	SOS	RC
KV-112	497,968	4,238,323	695.9	260	-60	100.45	SOS	CORE
KVR-113	497,917	4,238,240	730.4	210	-50	45.00	SOS	RC
KVR-114	497,966	4,238,228	710.9	230	-50	90.00	SOS	RC
KVR-115	497,974	4,238,174	683.8	315	-67	85.20	SOS	RC
KVR-116	497,982	4,238,147	680.4	280	-78	90.50	SOS	RC
KV-117	497,985	4,238,478	684.2	275	-72	208.50	SOS	CORE
KVR-118	497,962	4,238,127	693.4	265	-60	52.50	SOS	RC
KVR-119	497,927	4,238,466	682.6	244	-65	119.00	SOS	RC
KV-120	497,974	4,238,397	694.4	265	-60	154.55	SOS	CORE
KV-121	497,985	4,238,476	684.2	300	-77	236.50	SOS	CORE
KV-122	497,954	4,238,490	680.6	285	-52	140.00	SOS	CORE
KV-123	497,986	4,238,446	687.6	275	-63	175.00	SOS	CORE
KVR-124	497,966	4,238,348	695.9	265	-52	104.50	SOS	RC
KV-125	497,926	4,238,541	653.6	260	-77	146.50	SOS	CORE
KV-126	498,011	4,238,531	667.8	275	-70	245.00	SOS	CORE
KVR-127	497,970	4,238,374	695.4	260	-45	109.50	SOS	RC
KV-128	497,980	4,238,427	690.8	260	-52	139.10	SOS	CORE
KV-129	498,011	4,238,531	667.6	316	-65	338.00	SOS	CORE
KVR-130	497,983	4,238,224	703.8	285	-53	98.00	SOS	RC
KVR-131	498,065	4,238,099	642.2	247	-78	92.20	SOS	RC
KVR-132	497,973	4,238,094	685.1	250	-60	44.00	SOS	RC
KVR-133	498,004	4,238,122	672.3	230	-55	66.00	SOS	RC
KV-134	498,092	4,238,347	640.5	260	-52	215.00	SOS	CORE
KVR-135	498,049	4,238,162	669.9	245	-60.5	98.50	SOS	RC
KV-136	498,077	4,238,492	643.4	300	-60	295.50	SOS	CORE
KVR-137	497,999	4,238,072	672.3	200	-55	51.50	SOS	RC
KV-138	498,035	4,238,384	666.4	275	-55	184.40	SOS	CORE
KVR-139	497,736	4,238,710	641.6	280	-60	73.50	MOS	RC
KV-140	498,020	4,238,306	674.3	250	-55	149.20	SOS	CORE
KVR-141	497,837	4,238,610	613.4	235	-45	50.00	MOS	RC
KV-142	497,926	4,238,708	586.0	0	-90	311.00	MOS	CORE
KV-143	498,019	4,238,306	674.3	255	-73	181.80	SOS	CORE
KVR-144	497,734	4,238,791	595.4	235	-50	73.80	MOS	RC
KV-145	497,913	4,238,730	587.6	210	-85	338.50	MOS	CORE
KV-146	498,094	4,238,192	669.2	260	-55	170.00	SOS	CORE
KV-147	497,916	4,238,761	586.7	230	-75	300.00	MOS	CORE
KV-148	498,127	4,238,247	651.8	260	-55	236.30	SOS	CORE
KV-149	497,928	4,238,702	585.8	222	-49	150.00	MOS	CORE
KV-150	497,857	4,238,784	581.4	225	-80	350.30	MOS	CORE
KV-151	497,865	4,238,663	608.5	233	-62	74.50	MOS	CORE
KV-152	497,869	4,238,669	607.9	233	-62	300.00	MOS	CORE
KV-153	498,050	4,238,160	669.7	245	-60.5	125.40	SOS	CORE
KV-154	497,874	4,238,727	607.2	242	-57.0	250.00	MOS	CORE
KV-155	498,134	4,238,223	654.4	270	-60	220.50	SOS	CORE
KV-156	498,066	4,238,235	675.9	260	-47	160.00	SOS	CORE
KV-157	497,832	4,238,763	603.0	225	-80	240.00	MOS	CORE
KV-158	497,820	4,238,781	602.1	244	-52	145.00	MOS	CORE
KV-159	497,895	4,238,767	584.2	235	-82	420.00	MOS	CORE
KV-160	498,164	4,238,154	642.1	260	-58	190.00	SOS	CORE
KV-161	498,107	4,238,295	646.6	255	-45	222.00	SOS	CORE
KV-162	498,067	4,238,236	674.9	262	-63	281.50	SOS	CORE
KV-163	498,129	4,238,199	656.7	260	-55	225.00	SOS	CORE
KV-164	497,792	4,238,762	620.9	237	-50	130.00	MOS	CORE
KVP-165	498,047	4,238,437	659.0	285	-62.5	306.00	SOS	RC/CORE
KV-166	497,840	4,238,801	578.7	255	-85	400.00	MOS	CORE
KV-167	498,129	4,238,199	656.6	260	-70	225.00	SOS	CORE
KV-168	498,108	4,238,296	644.3	260	-62	50.00	SOS	CORE
KV-170	498,109	4,238,296	644.6	260	-62	17.00	SOS	CORE

Hole ID	Easting	Northing	Elevation	Azimuth	Inclination	Depth(m)	Shoot	Hole Type
KV-171	498,109	4,238,297	644.5	260	-62	277.50	SOS	CORE
KVR-172	498,019	4,238,229	691.9	255	-47	90.00	SOS	RC
KVR-173	497,818	4,238,843	576.0	235	-86	100.00	MOS	RC
KV-174	497,839	4,238,800	577.8	235	-60	250.00	MOS	CORE
KVP-175	497,816	4,238,848	575.6	235	-86	116.00	MOS	RC
KV-176	497,986	4,238,447	688.1	250	-62	175.50	SOS	CORE
KVR-177	497,972	4,238,171	683.9	245	-70	78.50	SOS	RC
KV-178	497,978	4,238,562	654.6	295	-70	300.00	MOS	CORE
KV-179	498,091	4,238,378	642.5	260	-57	226.40	SOS	CORE
KVR-180	497,954	4,238,488	681.3	250	-65	152.00	SOS	RC
KVR-181	497,980	4,238,147	682.2	260	-45	64.00	SOS	RC
KV-182	497,968	4,238,325	696.9	265	-47	97.00	SOS	CORE
KVR-183	497,738	4,238,706	642.5	190	-65	28.00	MOS	RC
KVR-184	497,839	4,238,609	613.6	160	-80	98.00	MOS	RC
KV-185	498,020	4,238,303	674.7	260	-66	171.00	SOS	CORE
KVR-186	497,737	4,238,705	642.4	190	-65	80.50	MOS	RC
KV-187	497,979	4,238,562	654.7	310	-65	300.00	MOS	CORE
KVR-188	497,818	4,238,631	627.4	235	-75	130.00	MOS	RC
KV-189	498,028	4,238,209	689.2	245	-63	136.00	SOS	CORE
KVR-190	497,827	4,238,666	629.7	235	-75	130.00	MOS	RC
KV-191	498,091	4,238,392	641.1	265	-50	211.10	SOS	CORE
KVR-193	497,741	4,238,710	642.2	138	-79	102.00	MOS	RC
KV-194	498,020	4,238,230	691.6	260	-50	274.50	SOS	CORE
KVR-195	497,768	4,238,639	658.4	165	-55	60.00	MOS	RC
KV-196	498,089	4,238,344	640.4	260	-67	250.30	SOS	CORE
KVR-197	497,739	4,238,708	642.2	200	-45	70.00	MOS	RC
KVR-198	497,764	4,238,645	657.9	311	-61	92.00	MOS	RC
KVR-200	497,764	4,238,647	658.3	284	-45	60.00	MOS	RC
KVR-201	497,792	4,238,685	652.3	239	-62	98.00	MOS	RC
KVR-203	497,792	4,238,672	651.9	200	-55	100.00	MOS	RC
KVR-208	497,956	4,238,490	680.8	270	-65	144.40	SOS	RC



Phyrolite
Vein
Hornfels
Phyllite

PLAN SPECS:

REF. PT. E, N	497900 m	4238000 m
EXTENTS	586.4 m	1344 m

SCALE 1 : 3600

(m)

-20 0 20 40 60 80 100 120

*unknown

N W E S

Efemçukuru Project (June 2007)

Drill Hole Location Plan Map

DHID	East	North	Elev	Au	Ag	Length	Domain	Zone
KV-001	497892	4238260	706	6.90	8.7	1.0	Main Vein	SOS
KV-002	497935	4238120	681	5.14	6.4	3.0	Main Vein	SOS
KV-003	497876	4238378	682	7.87	28.3	3.0	Upper Splay	SOS
KV-003	497881	4238379	686	7.70	19.5	1.6	Upper Splay	SOS
KV-004	497882	4238372	651	4.61	5.7	4.2	Main Vein	SOS
KV-004	497890	4238374	659	9.69	37.4	8.9	Upper Splay	SOS
KV-005	497880	4238431	658	34.30	32.2	1.5	Upper Splay	SOS
KV-006	497851	4238481	624	8.06	-1.0	2.1	Main Vein	SOS
KV-016	497880	4238326	688	3.94	2.7	4.9	Main Vein	SOS
KV-016	497884	4238327	698	6.17	9.4	3.8	Upper Splay	SOS
KV-017	497901	4238336	648	9.86	10.4	13.4	Main Vein	SOS
KV-018	497895	4238440	635	13.60	-1.0	3.6	Upper Splay	SOS
KV-019	497903	4238382	610	5.63	-1.0	9.1	Main Vein	SOS
KV-019	497908	4238383	617	11.64	-1.0	8.8	Upper Splay	SOS
KV-020	497957	4238125	629	9.05	7.4	12.2	Main Vein	SOS
KV-021	497974	4238094	648	6.01	-1.0	6.6	Main Vein	SOS
KV-022	497909	4238282	655	27.36	-1.0	2.7	Main Vein	SOS
KV-023	497843	4238533	601	4.03	-1.0	1.5	Main Vein	SOS
KV-034	497843	4238571	578	4.56	26.1	5.4	Main Vein	SOS
KV-035	497860	4238550	585	8.90	26.3	2.2	Upper Splay	SOS
KV-036	497866	4238502	602	3.33	13.1	1.5	Main Vein	SOS
KV-036	497869	4238503	611	7.13	106.0	3.4	Upper Splay	SOS
KV-036	497878	4238504	634	3.49	1.7	1.9	Upper Splay	SOS
KV-037	497940	4238236	621	16.41	11.7	6.0	Main Vein	SOS
KV-038	497945	4238188	634	7.70	5.5	12.8	Main Vein	SOS
KV-039	497981	4238137	599	10.59	17.3	16.4	Main Vein	SOS
KV-044	497995	4238160	561	5.70	14.7	5.8	Main Vein	SOS
KV-045	498036	4238073	596	21.62	-1.0	3.2	Main Vein	SOS
KV-048	497978	4238240	545	9.10	8.9	13.9	Main Vein	SOS
KV-050	497909	4238542	506	6.46	11.0	4.4	Main Vein	SOS
KV-050	497913	4238543	518	4.84	23.0	9.8	Upper Splay	SOS
KV-059	497989	4238290	496	9.28	7.1	4.8	Main Vein	SOS
KV-059	497992	4238291	500	5.72	2.3	4.9	Lower Splay	SOS
KV-068	498059	4238167	505	3.08	4.7	3.0	Main Vein	SOS
KV-069	498039	4238147	538	9.61	10.0	5.5	Main Vein	SOS
KV-070	497997	4238260	501	8.52	13.1	4.5	Main Vein	SOS
KV-070	498002	4238260	507	6.90	4.3	2.4	Lower Splay	SOS
KV-070	498014	4238260	522	6.87	-1.0	2.3	Lower Splay	SOS
KV-075	497898	4238439	582	5.69	7.2	9.0	Main Vein	SOS
KV-075	497907	4238439	593	163.34	133.5	19.0	Upper Splay	SOS
KV-076	497896	4238509	552	17.42	46.2	3.4	Main Vein	SOS
KV-076	497903	4238511	560	7.91	31.7	3.0	Upper Splay	SOS
KV-077	497929	4238390	551	6.80	8.4	6.5	Main Vein	SOS
KV-078	497940	4238287	589	25.22	17.5	4.5	Main Vein	SOS
KV-079	498004	4238207	522	4.19	4.7	5.4	Main Vein	SOS
KV-080	497939	4238444	501	5.04	11.6	3.3	Main Vein	SOS
KV-082	497965	4238197	590	8.60	5.4	9.0	Main Vein	SOS
KV-083	497925	4238170	670	7.32	12.6	6.3	Main Vein	SOS
KV-084	497928	4238343	586	14.50	10.1	4.2	Main Vein	SOS
KV-085	497953	4238245	593	12.83	30.3	5.9	Main Vein	SOS
KV-098	497885	4238409	623	5.93	10.9	9.6	Main Vein	SOS

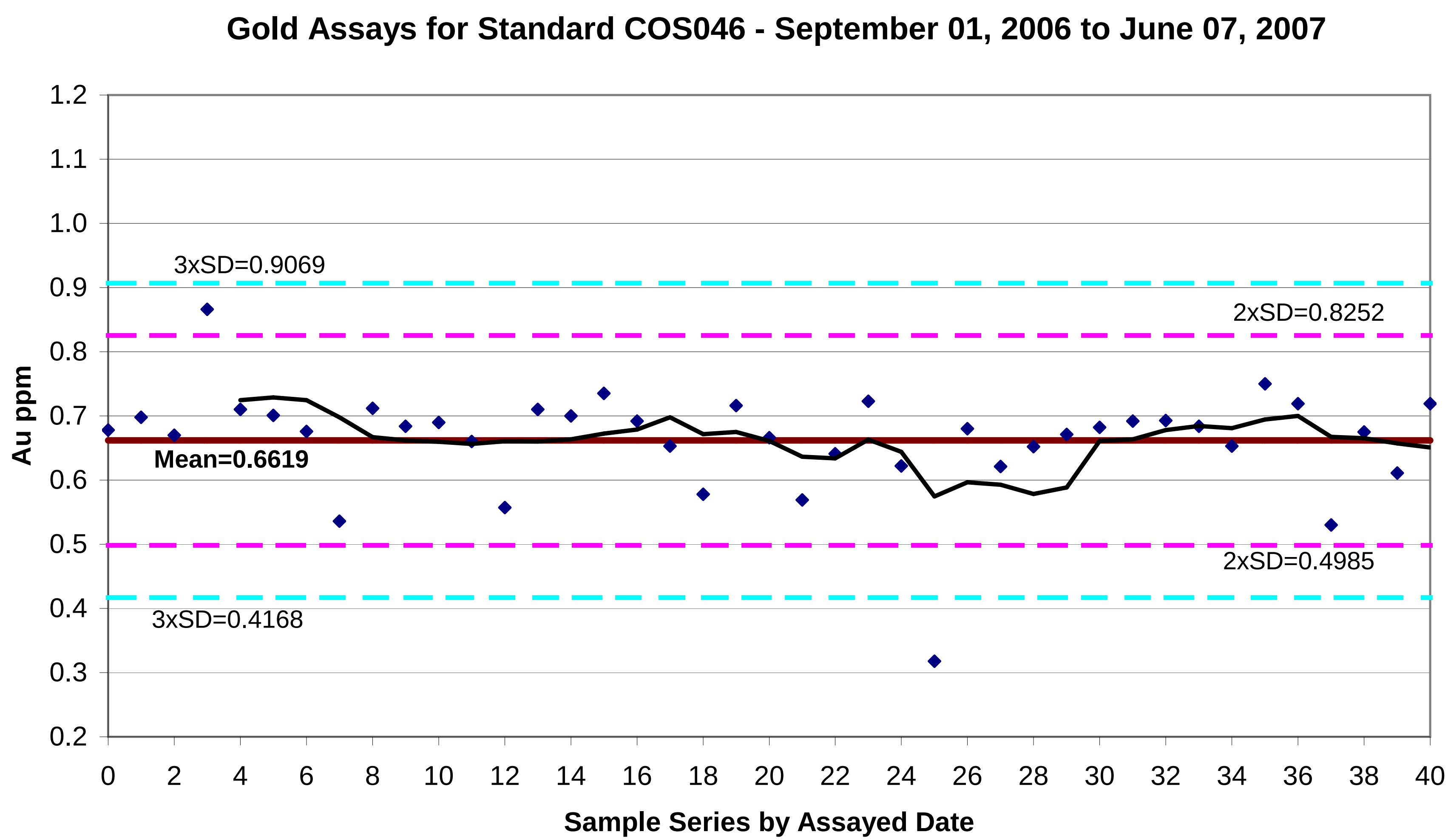
DHID	East	North	Elev	Au	Ag	Length	Domain	Zone
KV-098	497900	4238412	641	23.58	28.9	3.3	Upper Splay	SOS
KV-101	497951	4238268	581	5.93	5.7	7.0	Main Vein	SOS
KV-102	497934	4238267	611	6.79	7.4	6.0	Main Vein	SOS
KV-103	497914	4238260	650	20.49	12.0	5.3	Main Vein	SOS
KV-109	497924	4238289	625	14.80	12.4	4.7	Main Vein	SOS
KV-112	497922	4238319	614	12.73	13.5	6.2	Main Vein	SOS
KV-117	497930	4238485	501	5.70	6.9	5.4	Main Vein	SOS
KV-120	497911	4238397	579	11.10	14.2	7.5	Main Vein	SOS
KV-120	497917	4238397	591	6.39	14.3	6.7	Upper Splay	SOS
KV-122	497890	4238509	590	3.41	15.2	2.4	Upper Splay	SOS
KV-123	497917	4238449	546	10.99	21.0	8.1	Main Vein	SOS
KV-125	497900	4238533	531	13.17	30.4	1.8	Main Vein	SOS
KV-125	497905	4238535	556	5.71	22.9	4.6	Upper Splay	SOS
KV-126	497938	4238536	455	8.04	9.1	5.0	Main Vein	SOS
KV-126	497955	4238534	506	6.05	104.5	3.8	Upper Splay	SOS
KV-128	497903	4238420	587	7.27	9.5	8.8	Main Vein	SOS
KV-128	497909	4238420	595	11.78	16.3	11.6	Upper Splay	SOS
KV-129	497934	4238607	421	23.70	23.6	20.7	Main Vein	SOS
KV-134	497989	4238325	492	8.71	9.5	9.5	Main Vein	SOS
KV-138	497941	4238402	531	8.72	9.0	2.8	Main Vein	SOS
KV-140	497950	4238284	569	16.22	11.8	5.2	Main Vein	SOS
KV-143	497976	4238296	521	66.16	20.9	15.5	Main Vein	SOS
KV-148	498017	4238228	480	3.35	6.2	3.4	Main Vein	SOS
KV-148	498029	4238230	500	3.49	3.1	3.0	Lower Splay	SOS
KV-153	497999	4238143	573	7.39	14.7	9.9	Main Vein	SOS
KV-155	498043	4238223	480	3.67	4.9	1.2	Main Vein	SOS
KV-156	497969	4238221	565	13.01	9.3	7.9	Main Vein	SOS
KV-161	497986	4238264	516	11.09	9.3	6.2	Main Vein	SOS
KV-161	497991	4238265	521	11.11	6.5	5.2	Lower Splay	SOS
KV-162	497994	4238223	524	7.61	6.3	5.0	Main Vein	SOS
KV-163	498038	4238179	521	14.91	23.5	2.1	Main Vein	SOS
KV-167	498073	4238187	485	16.53	10.0	2.0	Main Vein	SOS
KV-171	498017	4238280	463	6.98	4.8	4.8	Main Vein	SOS
KV-171	498023	4238281	475	5.53	2.9	2.8	Lower Splay	SOS
KV-176	497922	4238425	554	40.58	33.9	8.0	Main Vein	SOS
KV-182	497910	4238322	633	8.78	11.3	4.8	Main Vein	SOS
KV-185	497965	4238295	546	5.79	6.2	5.0	Main Vein	SOS
KV-189	497979	4238181	572	4.31	3.7	11.6	Main Vein	SOS
KV-194	497953	4238218	602	8.17	5.7	8.5	Main Vein	SOS
KV-196	498005	4238340	448	5.21	4.8	21.8	Main Vein	SOS
KVP-165	497967	4238467	448	8.61	17.6	7.0	Main Vein	SOS
KVR-110	497895	4238284	686	9.85	15.1	5.5	Main Vein	SOS
KVR-111	497900	4238315	657	19.11	16.9	8.5	Main Vein	SOS
KVR-113	497909	4238223	707	3.04	8.5	2.0	Main Vein	SOS
KVR-114	497933	4238201	655	6.19	10.5	8.0	Main Vein	SOS
KVR-115	497954	4238193	618	12.31	8.4	18.0	Main Vein	SOS
KVR-116	497969	4238148	603	9.99	8.0	17.5	Main Vein	SOS
KVR-118	497943	4238124	658	8.62	13.5	9.0	Main Vein	SOS
KVR-118	497948	4238125	667	7.77	6.0	4.0	Upper Splay	SOS
KVR-119	497890	4238451	593	7.96	16.8	3.0	Main Vein	SOS
KVR-119	497893	4238452	600	3.58	10.1	6.5	Upper Splay	SOS

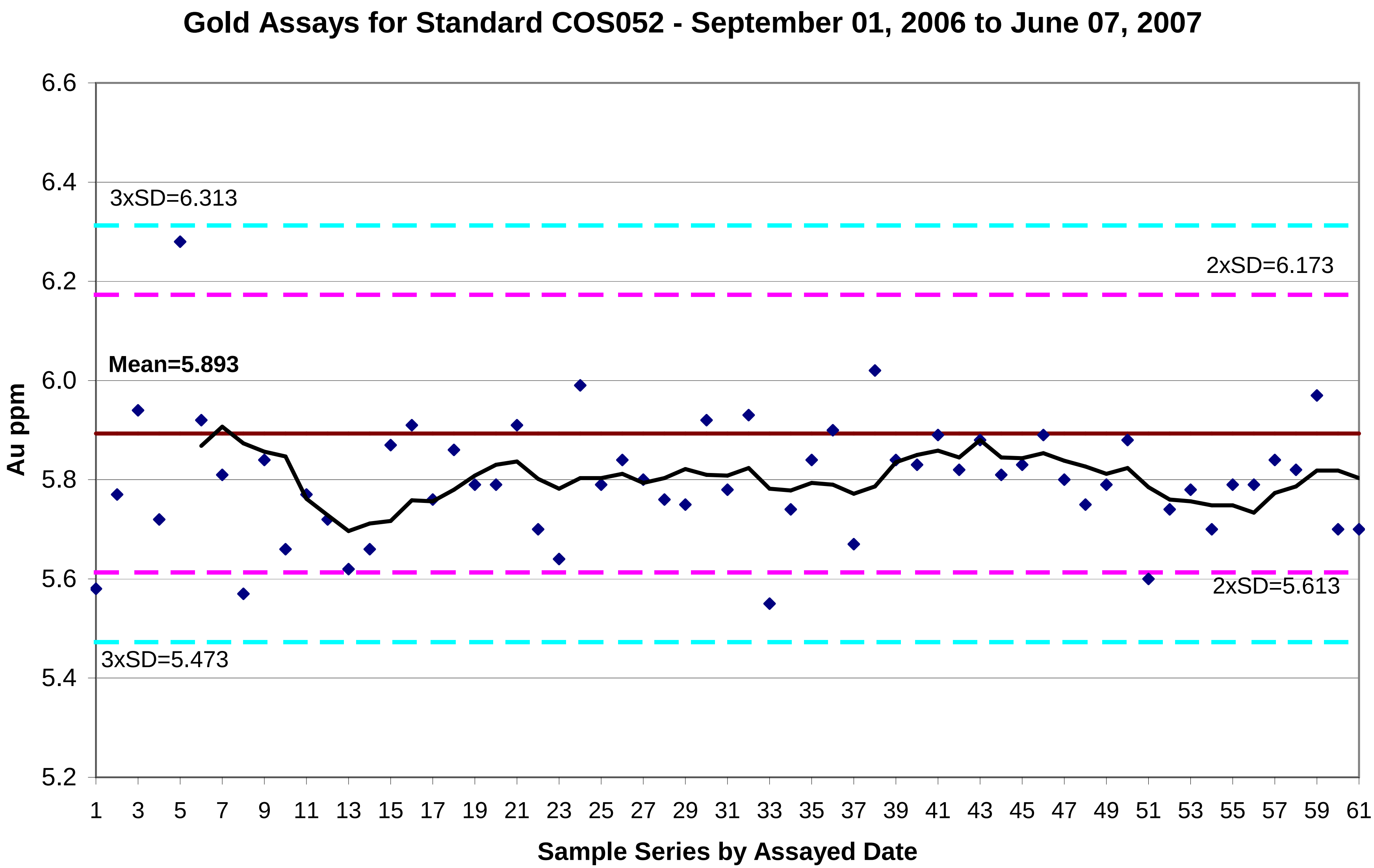
DHID	East	North	Elev	Au	Ag	Length	Domain	Zone
KVR-119	497898	4238454	612	6.67	13.8	6.0	Upper Splay	SOS
KVR-124	497911	4238342	622	12.14	11.0	8.5	Main Vein	SOS
KVR-127	497901	4238363	628	3.40	5.6	6.0	Main Vein	SOS
KVR-127	497905	4238364	632	10.18	17.9	7.0	Upper Splay	SOS
KVR-130	497933	4238238	634	11.22	9.2	3.5	Main Vein	SOS
KVR-133	497982	4238104	629	5.08	5.4	6.0	Main Vein	SOS
KVR-137	497994	4238059	651	16.93	10.6	1.0	Main Vein	SOS
KVR-177	497951	4238161	620	7.52	6.6	17.0	Main Vein	SOS
KVR-180	497899	4238468	567	6.70	17.1	7.0	Main Vein	SOS
KVR-180	497906	4238470	580	4.58	11.3	10.5	Upper Splay	SOS
KVR-181	497945	4238138	644	7.80	6.1	11.0	Main Vein	SOS
KVR-181	497952	4238140	651	3.04	4.3	2.5	Upper Splay	SOS
KVR-208	497898	4238489	556	15.31	16.4	2.5	Main Vein	SOS
KVR-208	497908	4238489	576	16.05	18.6	10.5	Upper Splay	SOS
KVR-215	497897	4238594	501	8.88	12.0	10.0	Upper Splay	SOS
KVR-227	497917	4238243	661	8.46	5.5	4.5	Main Vein	SOS
KV-007	497731	4238707	560	11.09	14.3	6.3	Main Vein	MOS
KV-007	497752	4238711	581	26.12	43.8	6.4	Upper Splay	MOS
KV-007	497762	4238713	591	5.21	3.1	23.5	Stockwork	MOS
KV-008	497804	4238621	583	10.88	21.2	5.0	Main Vein	MOS
KV-009	497754	4238638	644	20.27	45.1	6.2	Main Vein	MOS
KV-024	497767	4238701	512	5.44	21.5	5.3	Main Vein	MOS
KV-024	497777	4238706	543	43.33	18.3	5.4	Upper Splay	MOS
KV-025	497768	4238669	567	23.56	13.2	18.4	Main Vein	MOS
KV-026	497793	4238674	525	6.85	10.5	36.1	Main Vein	MOS
KV-028	497717	4238689	614	12.09	36.4	15.2	Main Vein	MOS
KV-029	497760	4238649	613	48.95	41.0	13.5	Main Vein	MOS
KV-030	497714	4238726	558	7.39	9.7	2.3	Main Vein	MOS
KV-030	497733	4238739	581	4.03	6.5	1.6	Upper Splay	MOS
KV-031	497756	4238682	574	12.15	21.9	20.6	Main Vein	MOS
KV-031	497763	4238688	583	6.58	18.5	6.5	Stockwork	MOS
KV-031	497766	4238690	587	8.78	47.8	2.9	Upper Splay	MOS
KV-031	497773	4238695	595	3.13	6.8	19.1	Stockwork	MOS
KV-032	497752	4238758	538	33.75	24.4	4.1	Upper Splay	MOS
KV-032	497767	4238769	589	4.74	25.5	2.4	Upper Splay	MOS
KV-033	497797	4238632	581	8.35	18.0	5.0	Main Vein	MOS
KV-051	497824	4238657	486	5.07	2.6	6.1	Main Vein	MOS
KV-051	497834	4238664	506	4.95	21.5	4.6	Upper Splay	MOS
KV-052	497796	4238712	445	6.67	25.5	9.1	Main Vein	MOS
KV-052	497812	4238722	490	4.34	17.9	4.6	Upper Splay	MOS
KV-058	497844	4238706	390	20.16	12.9	7.6	Main Vein	MOS
KV-087	497845	4238673	473	8.79	49.0	10.0	Upper Splay	MOS
KV-088	497876	4238690	426	3.98	17.4	4.1	Upper Splay	MOS
KV-089	497785	4238727	433	6.85	-1.0	7.2	Main Vein	MOS
KV-089	497807	4238744	477	3.67	6.4	6.8	Upper Splay	MOS
KV-090	497754	4238799	417	3.70	-1.0	9.0	Main Vein	MOS
KV-090	497789	4238822	500	4.78	-1.0	4.9	Upper Splay	MOS
KV-091	497767	4238735	477	13.74	41.3	7.4	Main Vein	MOS
KV-092	497822	4238626	554	9.86	8.4	6.0	Main Vein	MOS
KV-093	497772	4238661	565	23.98	33.1	17.6	Main Vein	MOS
KV-093	497779	4238667	574	3.40	-1.0	7.7	Stockwork	MOS

DHID	East	North	Elev	Au	Ag	Length	Domain	Zone
KV-093	497783	4238669	578	10.41	-1.0	4.3	Upper Splay	MOS
KV-093	497789	4238673	585	4.30	-1.0	15.0	Stockwork	MOS
KV-094	497787	4238617	613	4.55	14.5	7.2	Main Vein	MOS
KV-095	497755	4238661	606	25.06	27.9	19.2	Main Vein	MOS
KV-099	497792	4238640	566	10.83	12.0	21.7	Main Vein	MOS
KV-100	497718	4238777	514	4.22	17.0	4.2	Main Vein	MOS
KV-105	497763	4238653	598	23.55	22.2	19.7	Main Vein	MOS
KV-106	497767	4238656	581	23.06	39.7	21.9	Main Vein	MOS
KV-106	497772	4238660	596	4.71	-1.0	9.7	Stockwork	MOS
KV-107	497775	4238661	566	27.83	27.2	20.9	Main Vein	MOS
KV-107	497778	4238664	581	3.53	-1.0	8.7	Stockwork	MOS
KV-108	497786	4238669	549	29.95	26.7	16.2	Main Vein	MOS
KV-142	497927	4238709	329	6.23	34.6	5.3	Main Vein	MOS
KV-145	497903	4238714	361	6.10	25.9	3.8	Upper Splay	MOS
KV-147	497867	4238725	353	3.76	27.6	3.2	Main Vein	MOS
KV-149	497869	4238637	489	6.88	34.0	2.0	Main Vein	MOS
KV-150	497826	4238757	341	3.78	42.5	2.5	Main Vein	MOS
KV-154	497803	4238686	475	18.86	20.9	5.6	Main Vein	MOS
KV-154	497817	4238694	501	14.52	20.2	1.4	Upper Splay	MOS
KV-157	497807	4238736	381	6.17	34.7	8.0	Main Vein	MOS
KV-157	497816	4238745	463	4.84	10.9	5.8	Upper Splay	MOS
KV-158	497745	4238744	495	14.56	68.0	4.1	Main Vein	MOS
KV-158	497766	4238754	525	18.80	12.7	5.2	Upper Splay	MOS
KV-159	497862	4238747	320	3.56	43.2	5.0	Main Vein	MOS
KV-164	497732	4238725	535	10.75	118.4	5.7	Main Vein	MOS
KV-164	497748	4238734	557	15.84	13.1	4.2	Upper Splay	MOS
KV-174	497773	4238749	430	5.82	34.4	6.9	Main Vein	MOS
KVR-139	497706	4238716	589	8.38	17.9	17.0	Main Vein	MOS
KVR-141	497818	4238597	590	6.89	47.0	6.0	Main Vein	MOS
KVR-184	497844	4238599	551	6.75	12.9	5.5	Main Vein	MOS
KVR-186	497734	4238685	601	14.94	24.3	22.0	Main Vein	MOS
KVR-186	497736	4238698	627	9.20	12.9	17.0	Stockwork	MOS
KVR-188	497808	4238623	581	10.75	15.8	11.0	Main Vein	MOS
KVR-190	497807	4238650	541	7.27	13.8	24.0	Main Vein	MOS
KVR-193	497755	4238695	549	12.27	8.4	12.5	Main Vein	MOS
KVR-193	497750	4238700	581	3.59	7.1	41.5	Stockwork	MOS
KVR-193	497745	4238705	613	41.32	36.6	14.5	Upper Splay	MOS
KVR-193	497743	4238708	630	7.64	9.4	15.5	Stockwork	MOS
KVR-195	497770	4238630	645	5.70	9.2	4.0	Upper Splay	MOS
KVR-197	497730	4238683	614	17.86	24.2	16.5	Main Vein	MOS
KVR-197	497735	4238699	632	11.25	12.0	16.5	Stockwork	MOS
KVR-198	497747	4238658	621	34.33	41.8	33.0	Main Vein	MOS
KVR-200	497742	4238653	635	3.33	4.1	3.0	Stockwork	MOS
KVR-200	497747	4238652	641	58.77	48.1	13.0	Main Vein	MOS
KVR-201	497761	4238667	586	35.17	31.6	21.5	Main Vein	MOS
KVR-201	497768	4238671	600	13.75	15.1	5.0	Stockwork	MOS
KVR-201	497771	4238673	607	10.05	10.9	1.5	Upper Splay	MOS
KVR-203	497781	4238637	599	30.70	34.7	9.0	Main Vein	MOS

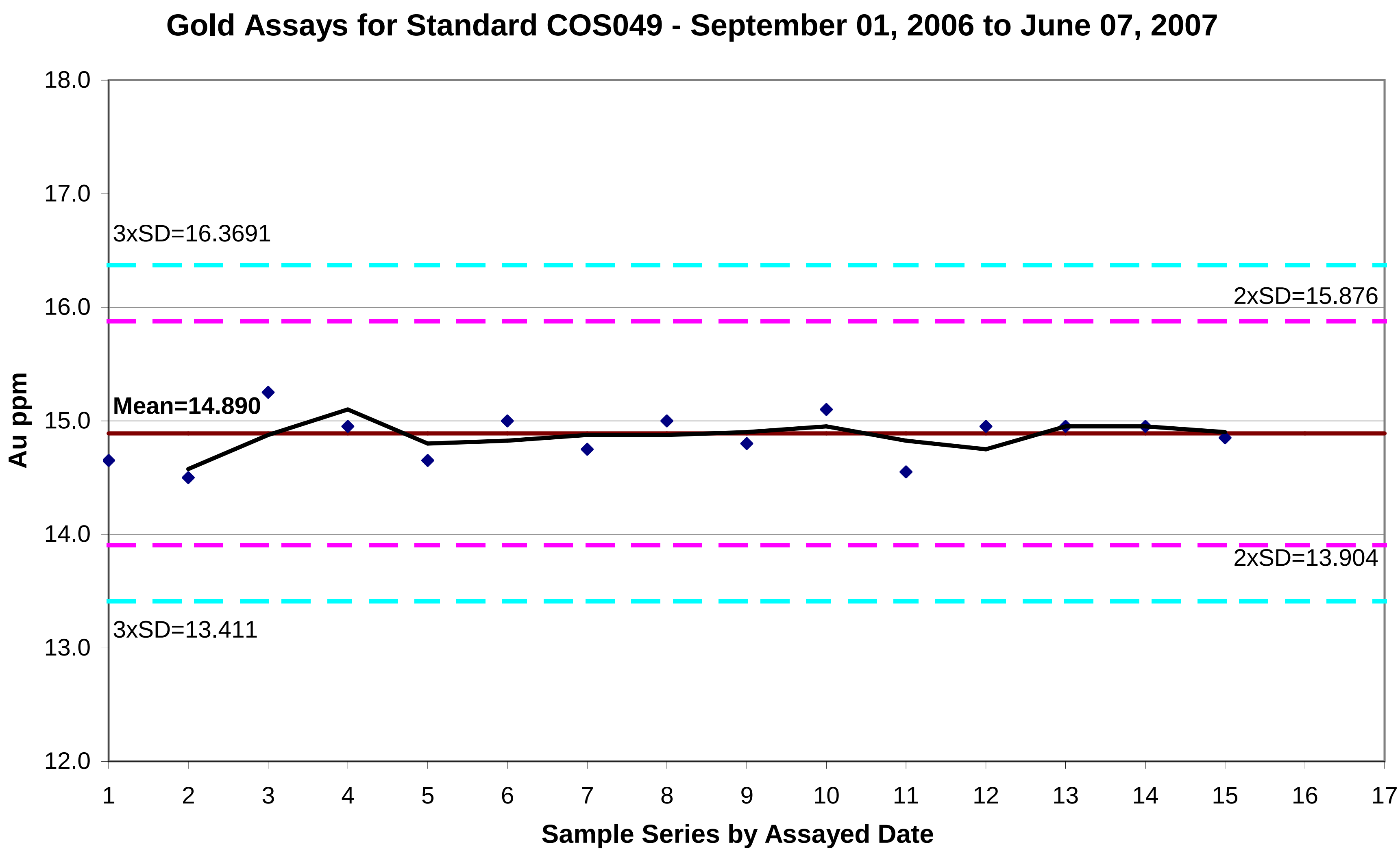
APPENDIX B

Standard Reference Charts
Histogram Plots
Grade Swath Plots

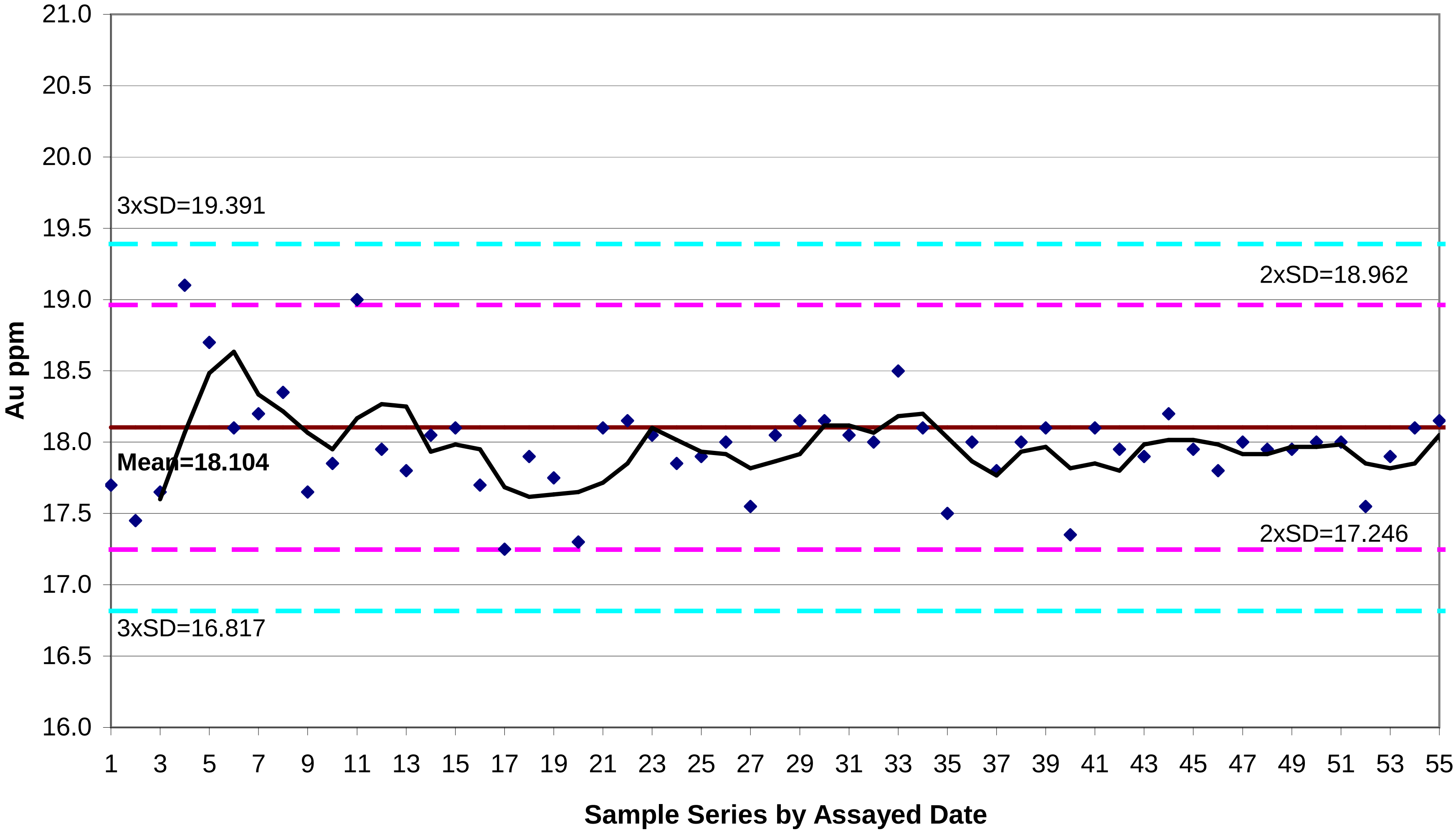

Gold Assays for Standard COS046 - September 01, 2006 to June 07, 2007
3xSD=0.9069
2xSD=0.8252
Mean=0.6619
2xSD=0.4985
3xSD=0.4168
Au ppm
1.2
1.1
1.0
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0
2
4
6
8
10
12
14
16
18
20
22
24
26
28
30
32
34
36
38
40
Sample Series by Assayed Date

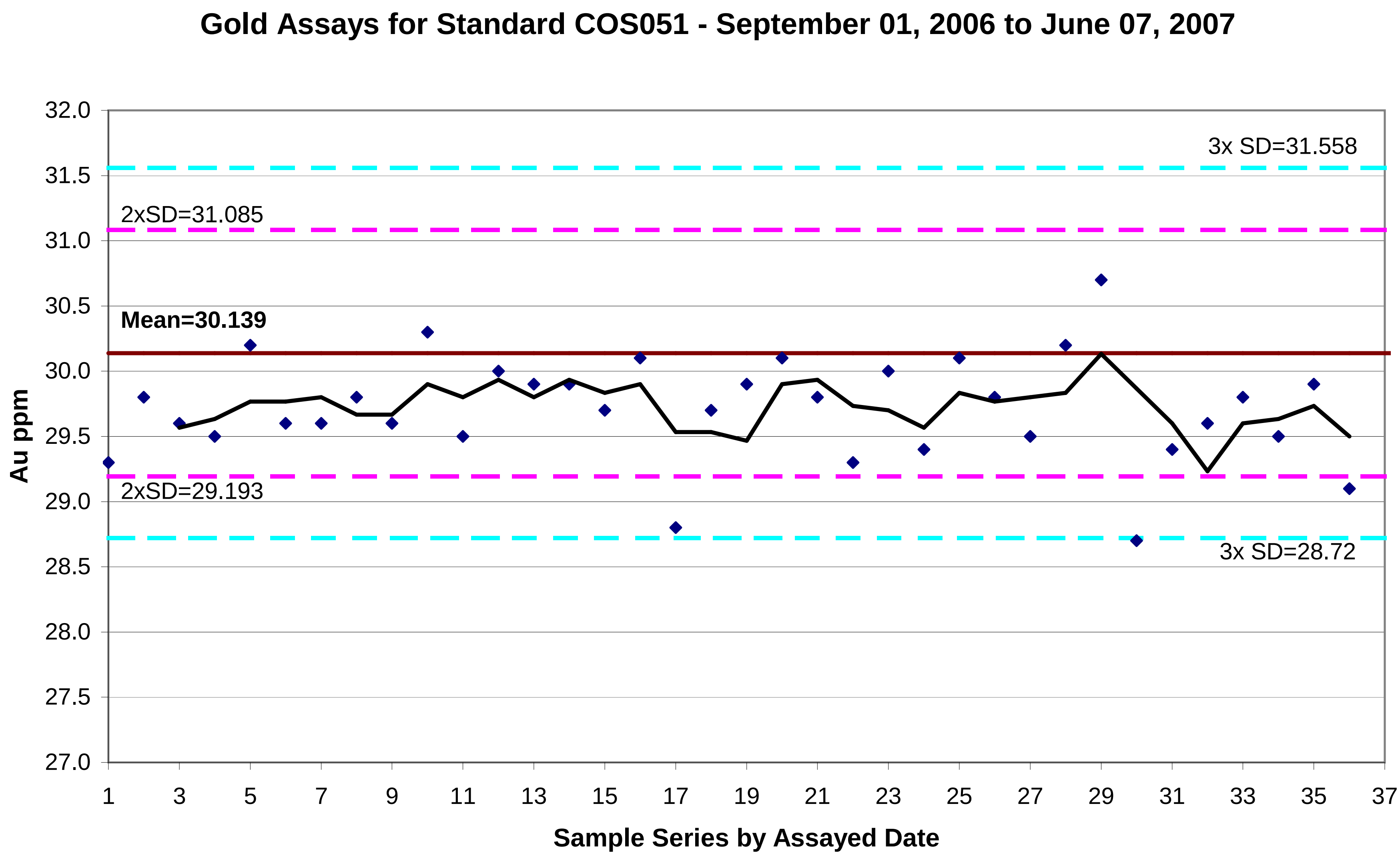


Gold Assays for Standard COS052 - September 01, 2006 to June 07, 2007
Au ppm
6.6
6.4
6.2
6.0
5.8
5.6
5.4
5.2
3xSD=6.313
2xSD=6.173
Mean=5.893
2xSD=5.613
3xSD=5.473
1 3 5 7 9 11 13 15 17 19 21 23 25 27 29 31 33 35 37 39 41 43 45 47 49 51 53 55 57 59 61
Sample Series by Assayed Date


Gold Assays for Standard COS049 - September 01, 2006 to June 07, 2007
18.0
17.0
16.0
15.0
14.0
13.0
12.0
Au ppm
3xSD=16.3691
2xSD=15.876
Mean=14.890
2xSD=13.904
3xSD=13.411
1
2
3
4
5
6
7
8
9
10
11
12
13
14
15
16
17
Sample Series by Assayed Date

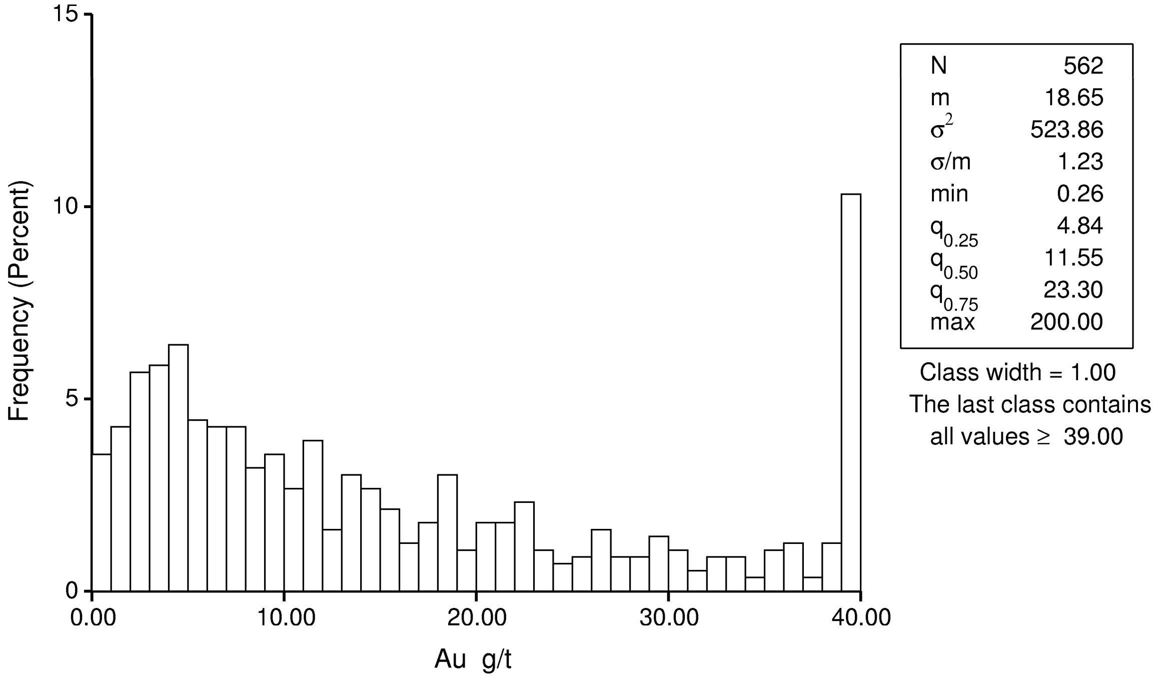
Gold Assays for Standard COS050 - September 01, 2006 to June 07, 2007



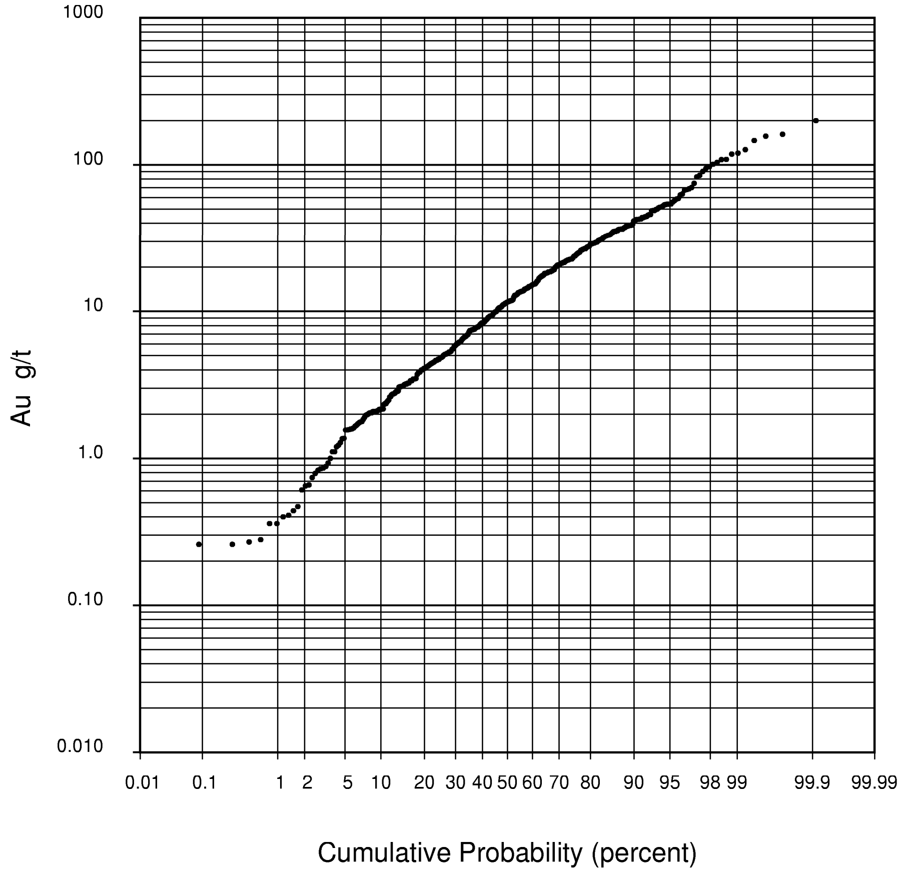


Gold Assays for Standard COS051 - September 01, 2006 to June 07, 2007
3x SD=31.558
2xSD=31.085
Mean=30.139
2xSD=29.193
3x SD=28.72
32.0
31.5
31.0
30.5
30.0
29.5
29.0
28.5
28.0
27.5
27.0
1 3 5 7 9 11 13 15 17 19 21 23 25 27 29 31 33 35 37
Au ppm
Sample Series by Assayed Date

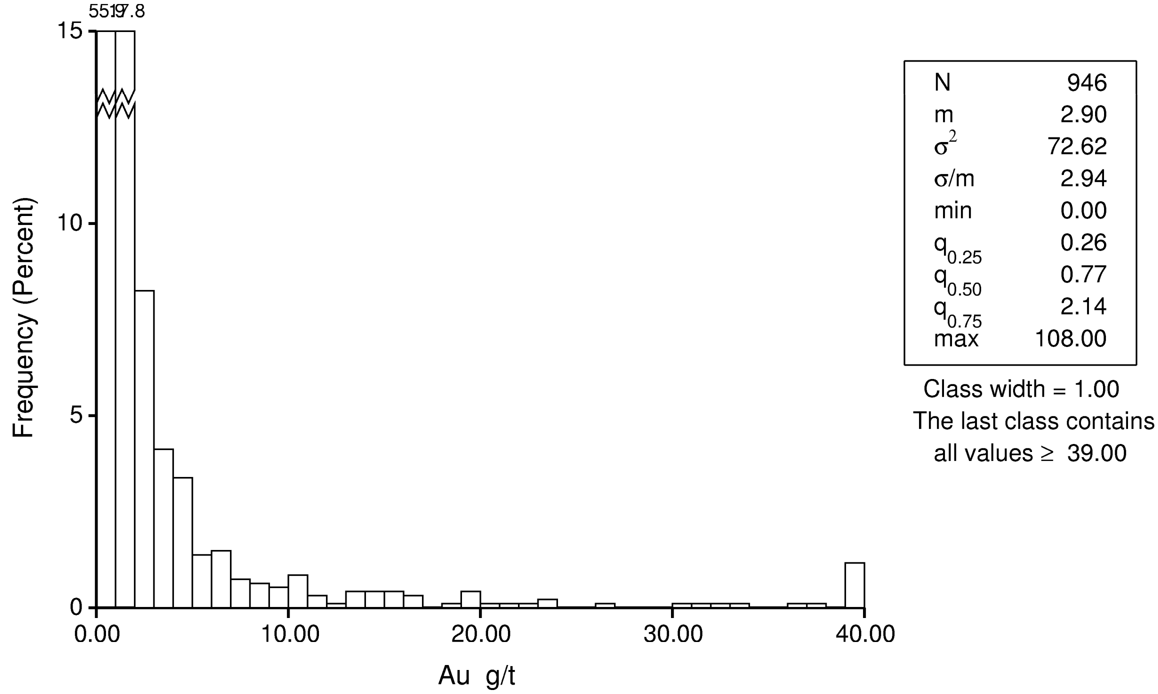
Efem MOS Main Vein Au 1m composites (Assays cap to 200g/t)



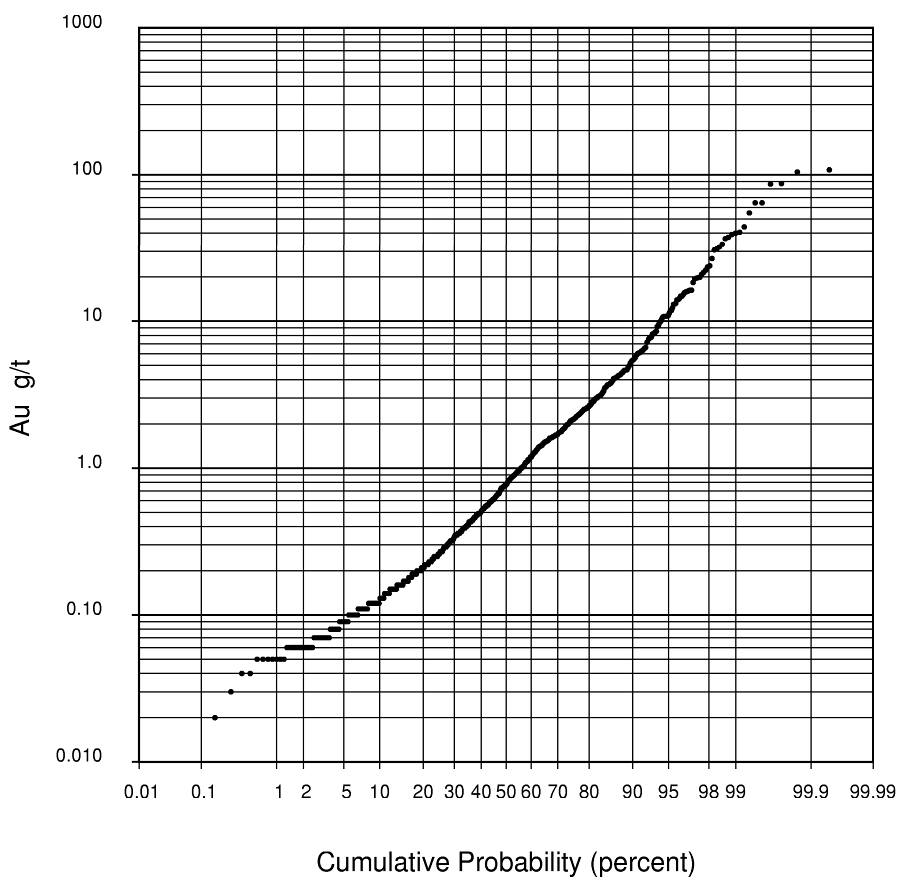
Efem MOS Main Vein Au 1m composites (Assays cap to 200g/t)



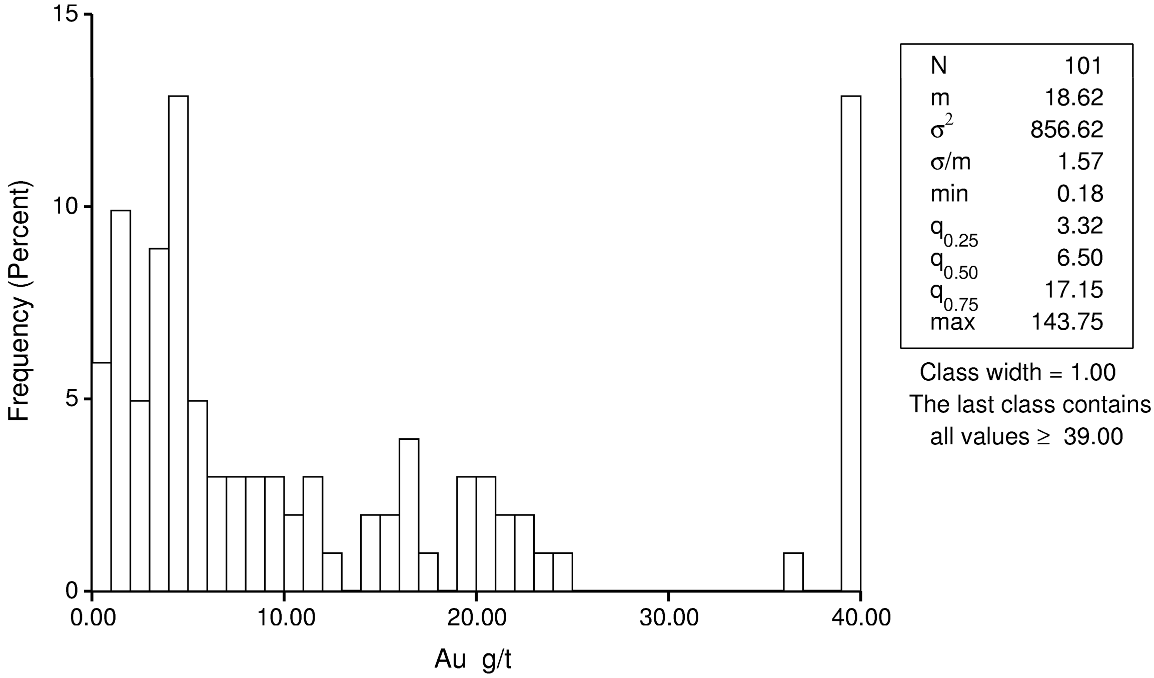
Efem MOS Stockwork Zone Au 1m composites (No Cap)



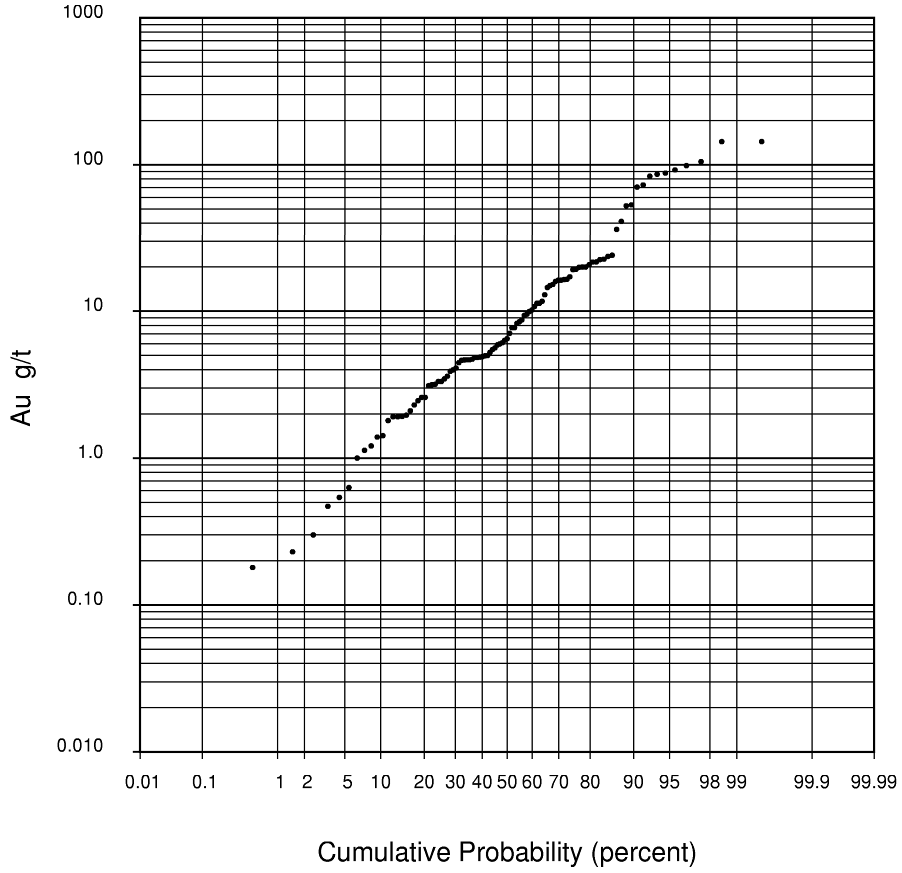
Efem MOS Stockwork Zone Au 1m composites (No Cap)

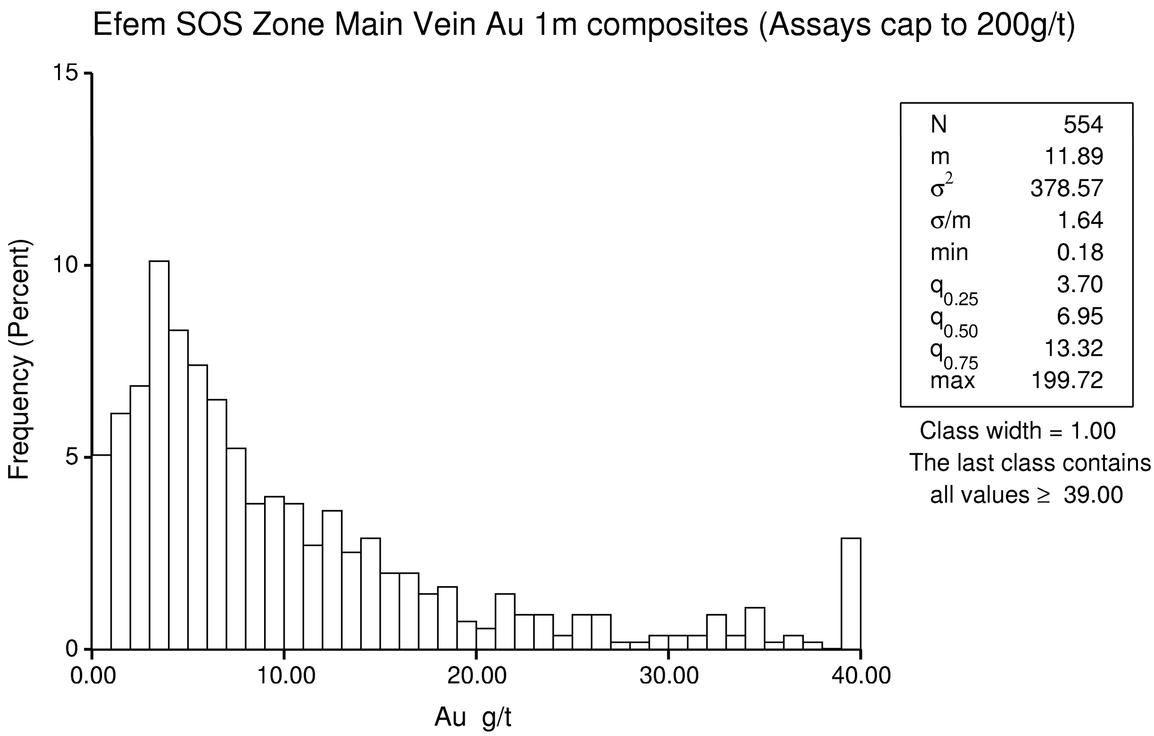


Efem MOS Upper Splays Au 1m composites (Assays cap to 200g/t)

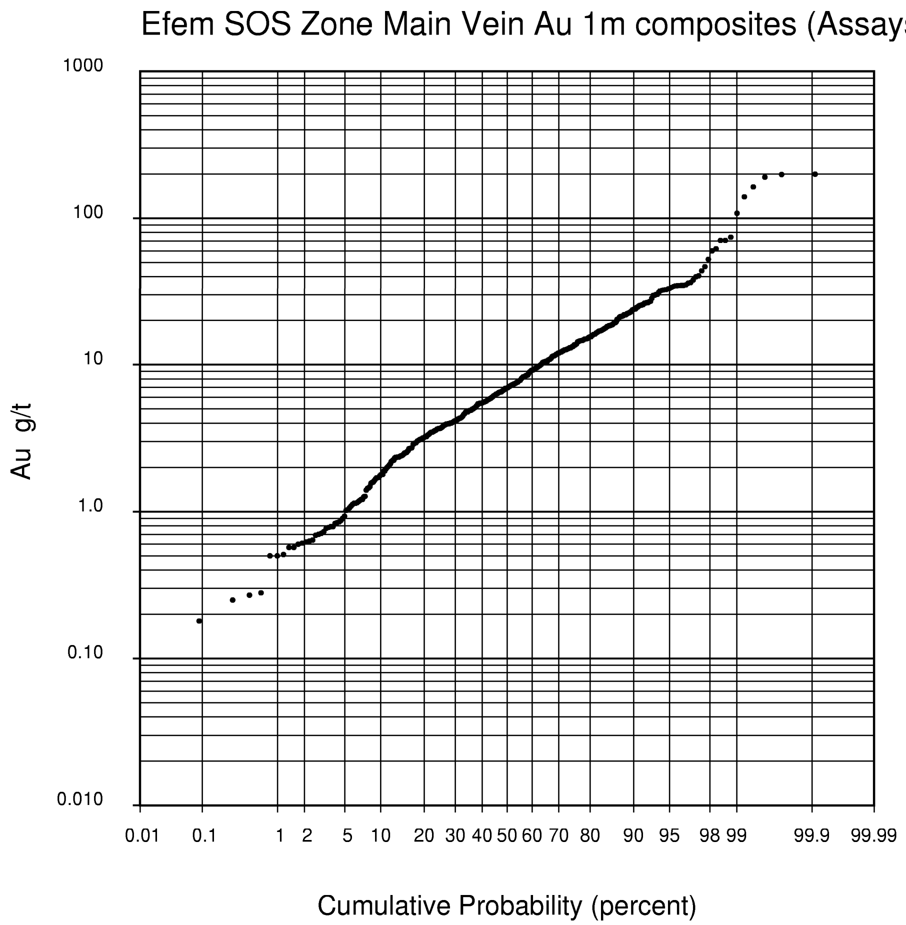


Efem MOS Upper Splays Au 1m composites (Assays cap to 200g/t)

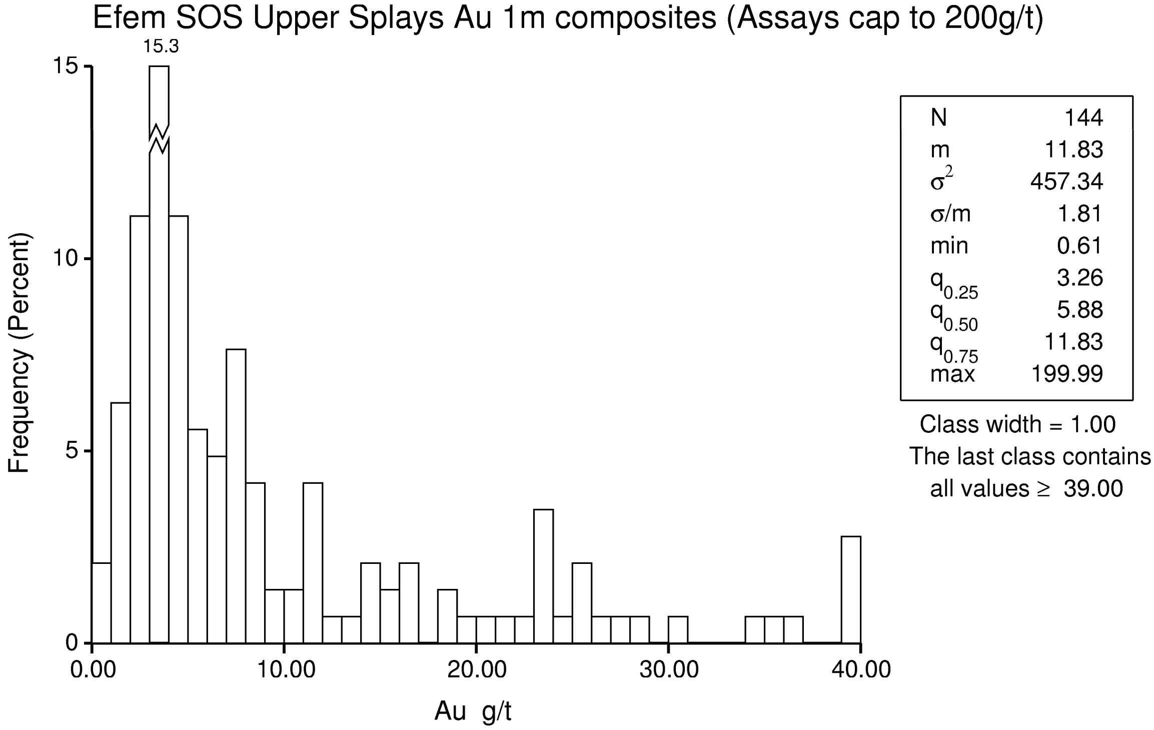


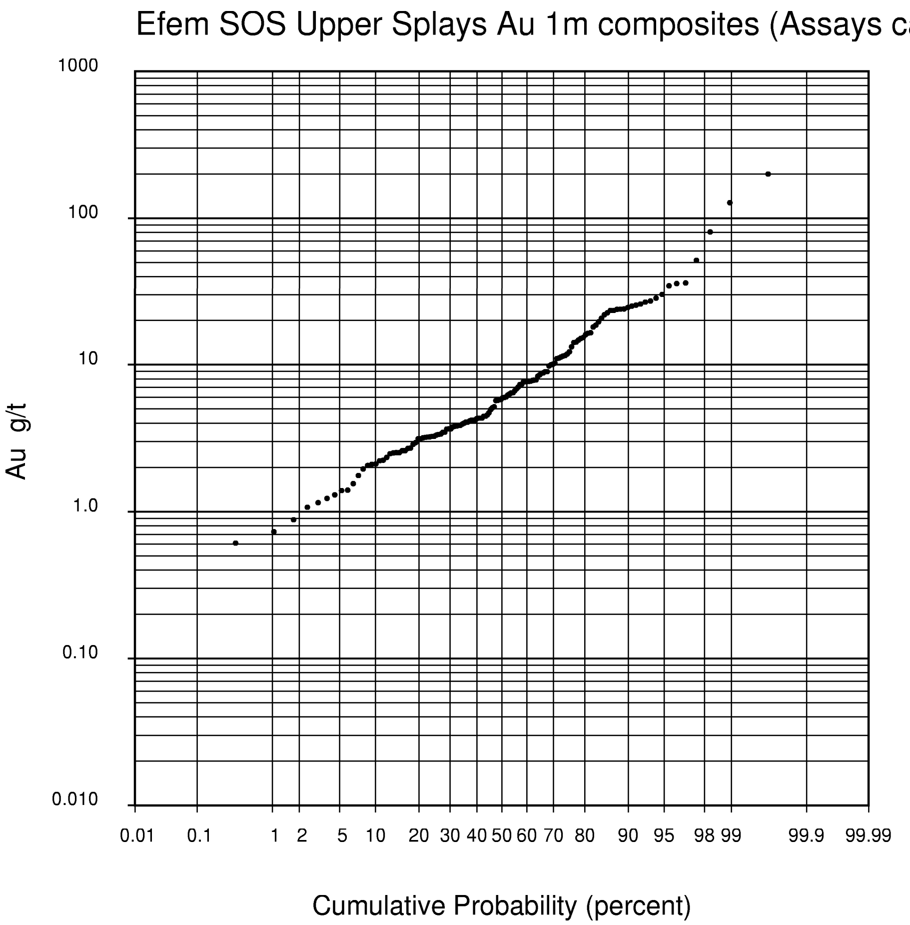


Efem SOS Zone Main Vein Au 1m composites (Assays cap to 200g/t)
15
10
5
0
Frequency (Percent)
0.00
10.00
20.00
30.00
40.00
Au g/t
N 554
m 11.89
σ2 378.57
σ/m 1.64
min 0.18
q0.25 3.70
q0.50 6.95
q0.75 13.32
max 199.72
Class width = 1.00
The last class contains
all values ≥ 39.00

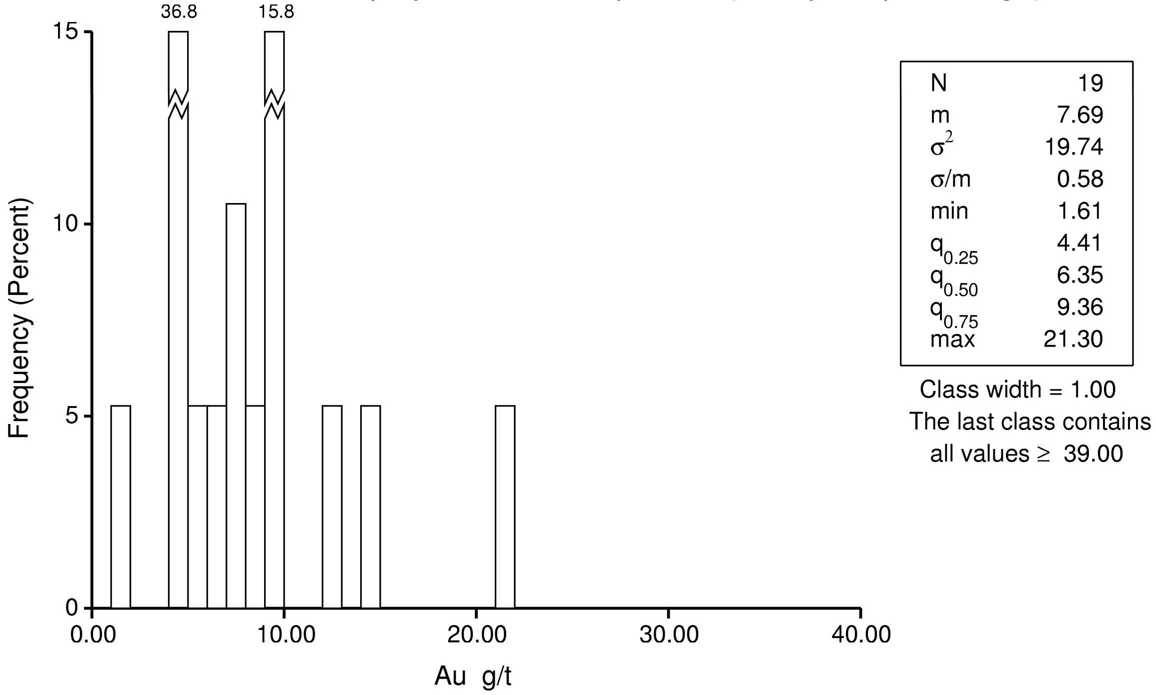


Efem SOS Zone Main Vein Au 1m composites (Assays cap to 200g/t)
1000
100
10
1.0
0.10
0.010
Au g/t
0.01 0.1 1 2 5 10 20 30 40 50 60 70 80 90 95 98 99 99.9 99.99
Cumulative Probability (percent)

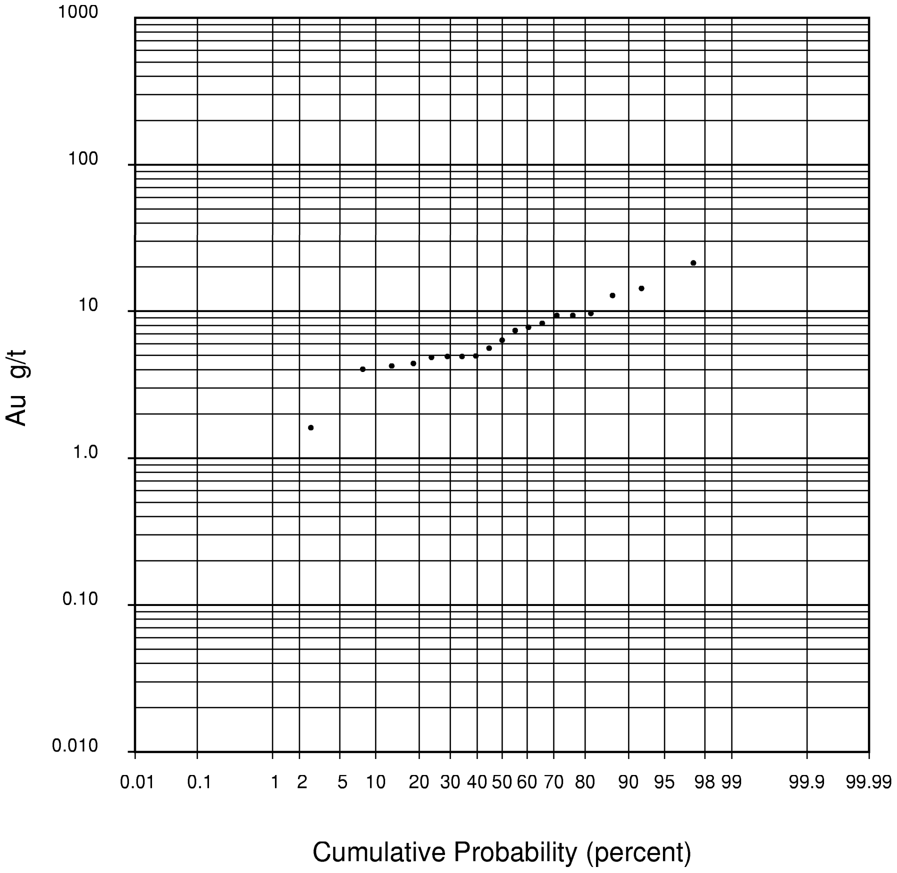

Efem SOS Upper Splays Au 1m composites (Assays cap to 200g/t)
15.3
Frequency (Percent)
Au g/t
N 144
m 11.83
σ2 457.34
σ/m 1.81
min 0.61
q0.25 3.26
q0.50 5.88
q0.75 11.83
max 199.99
Class width = 1.00
The last class contains
all values ≥ 39.00

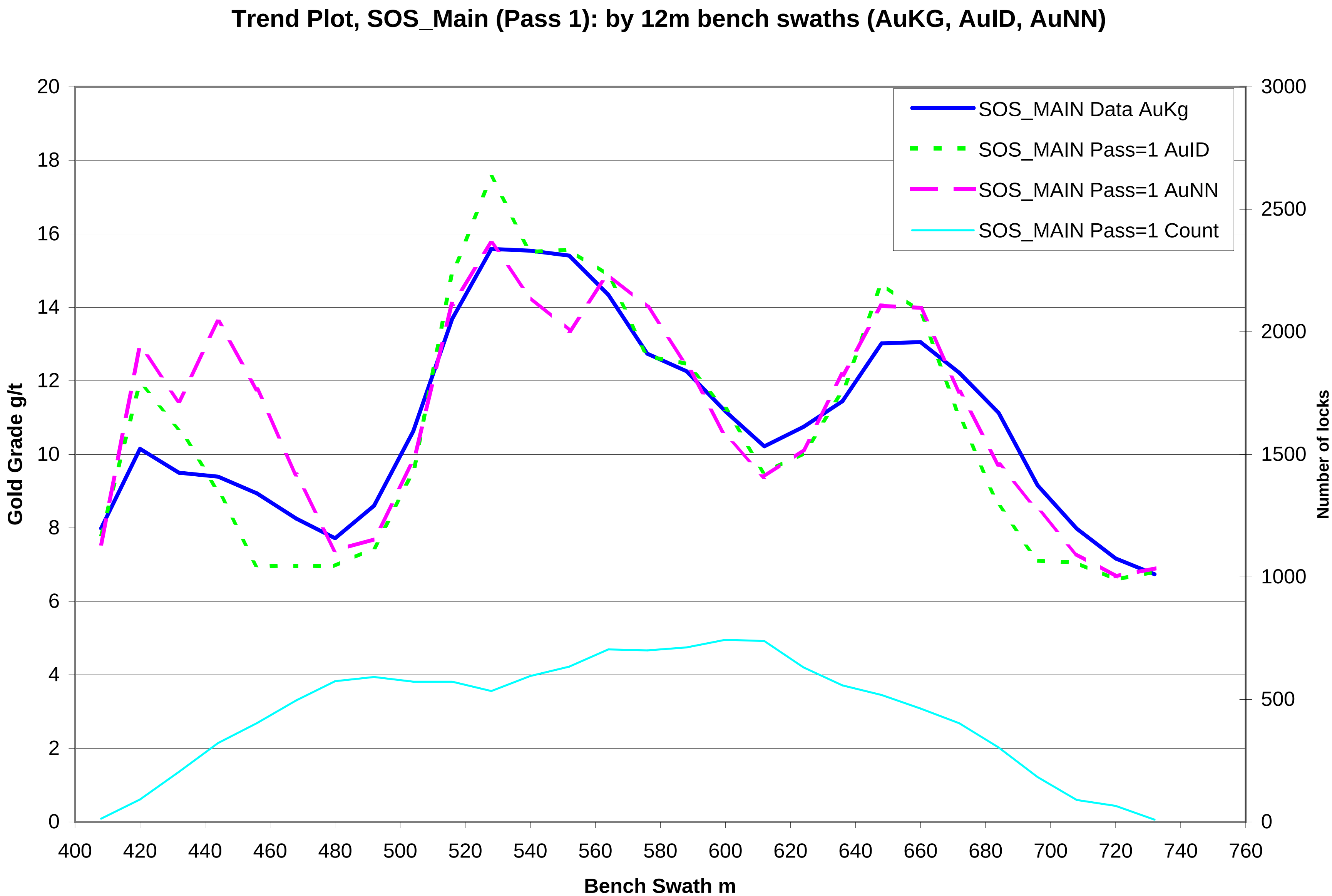

Efem SOS Upper Splays Au 1m composites (Assays cap to 200g/t)
Au g/t
Cumulative Probability (percent)

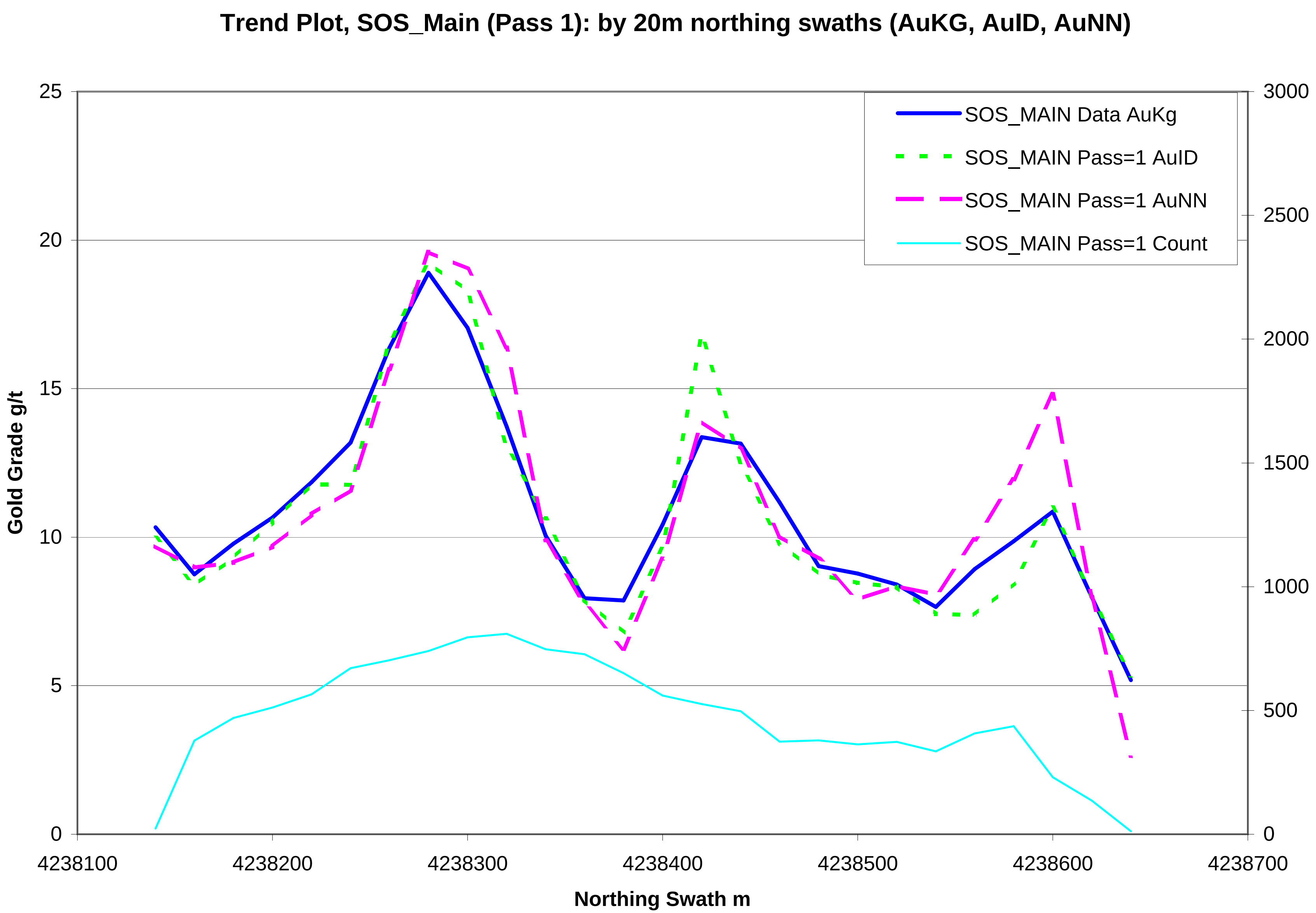
Efem SOS Lower Splays Au 1m composites (Assays cap to 200g/t)



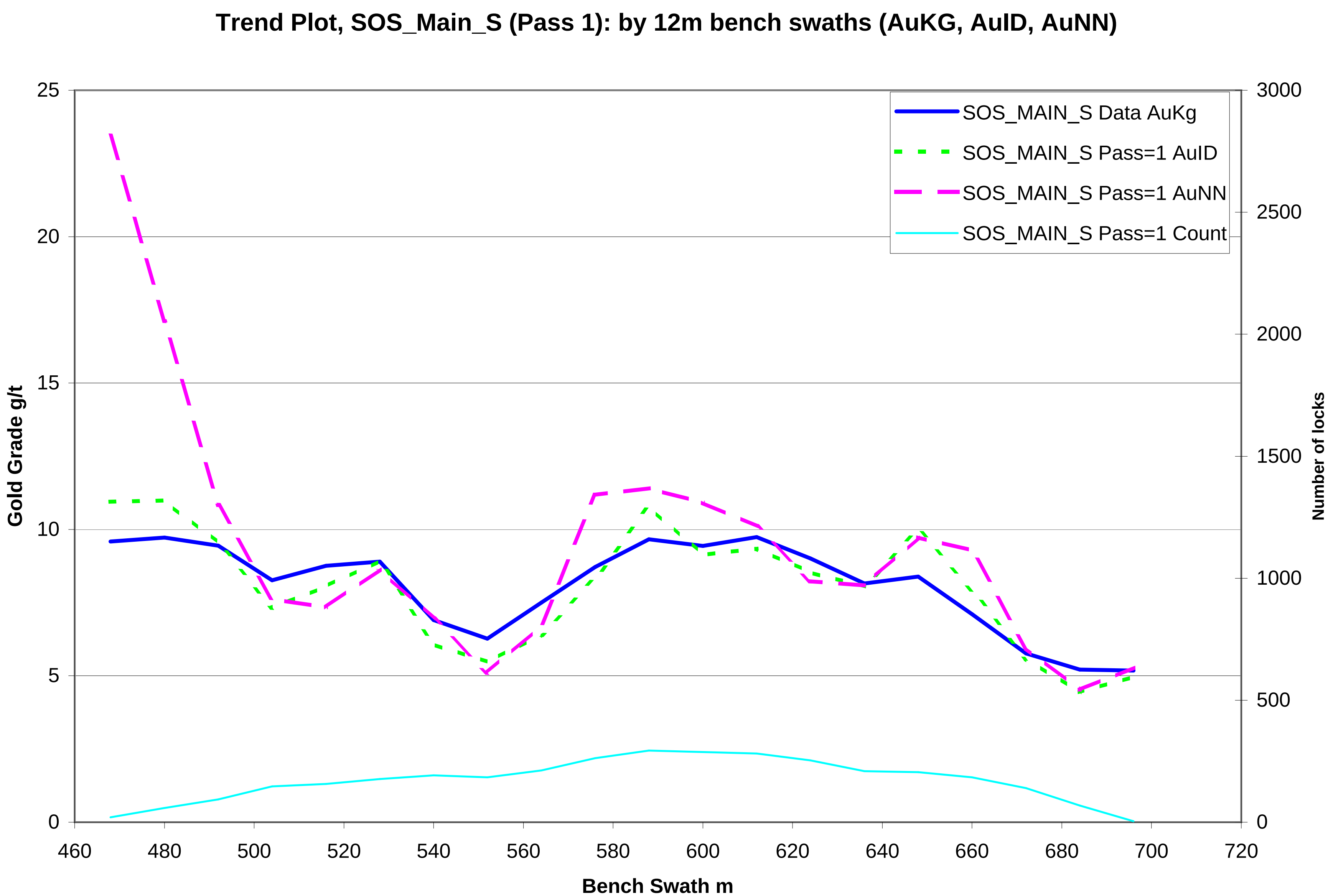
Efem SOS Lower Splays Au 1m composites (Assays cap to 200g/t)




Trend Plot, SOS_Main (Pass 1): by 12m bench swaths (AuKG, AuID, AuNN)
Gold Grade g/t
Number of locks
Bench Swath m
SOS_MAIN Data AuKg
SOS_MAIN Pass=1 AuID
SOS_MAIN Pass=1 AuNN
SOS_MAIN Pass=1 Count
0
2
4
6
8
10
12
14
16
18
20
0
500
1000
1500
2000
2500
3000
400
420
440
460
480
500
520
540
560
580
600
620
640
660
680
700
720
740
760

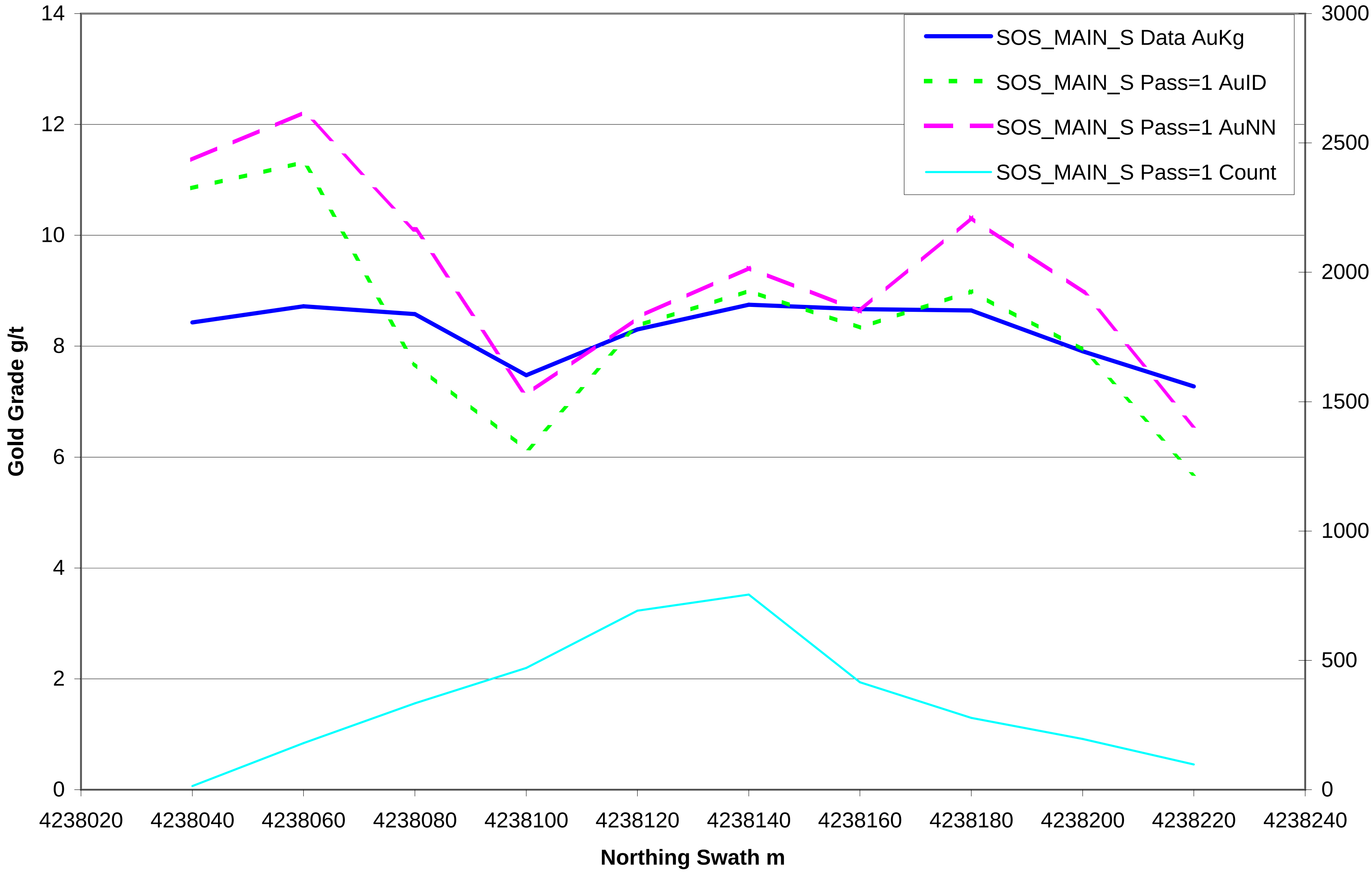


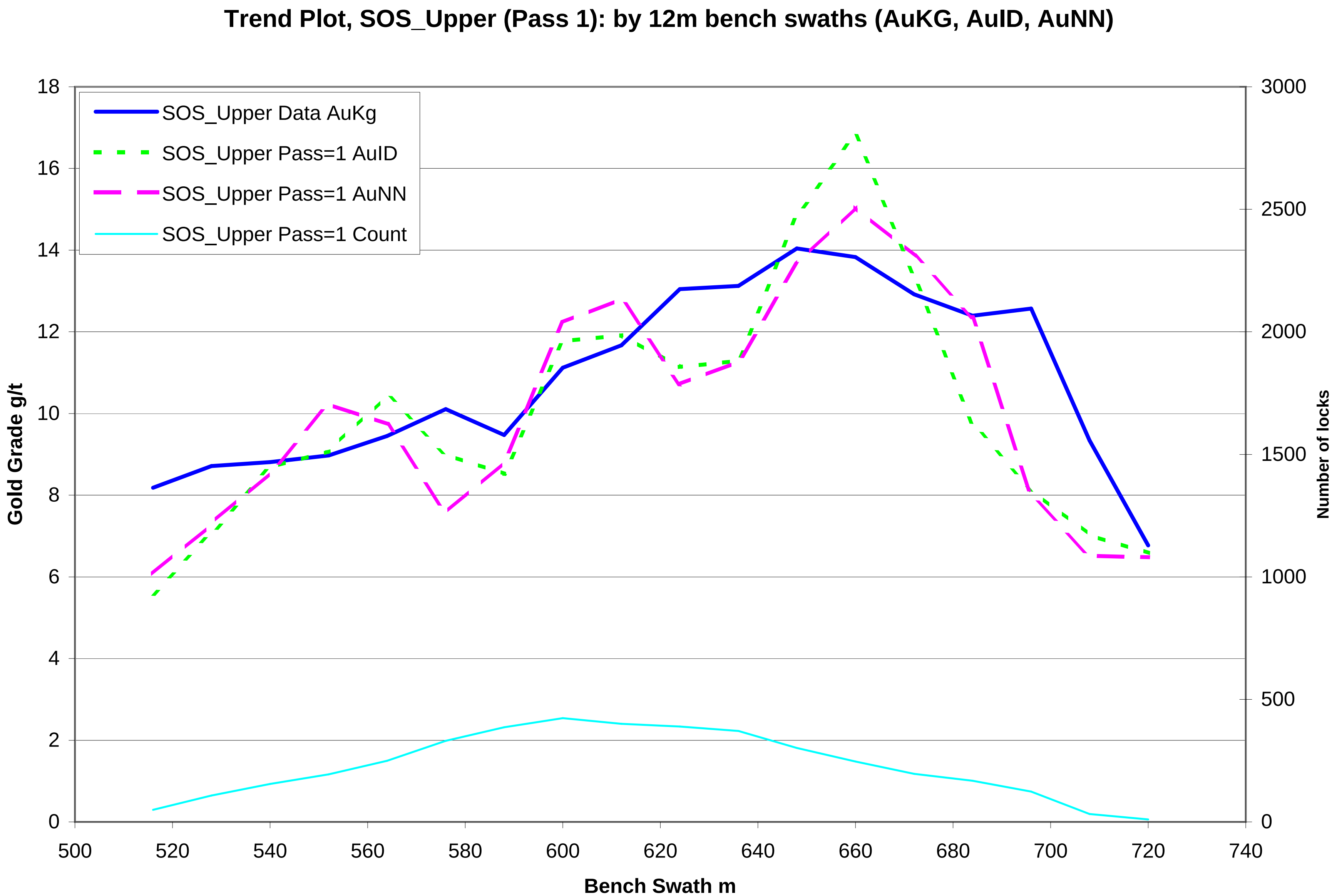
Trend Plot, SOS_Main (Pass 1): by 20m northing swaths (AuKG, AuID, AuNN)
SOS_MAIN Data AuKg
SOS_MAIN Pass=1 AuID
SOS_MAIN Pass=1 AuNN
SOS_MAIN Pass=1 Count
Gold Grade g/t
Number of locks
Northing Swath m
0
5
10
15
20
25
0
500
1000
1500
2000
2500
3000
4238100
4238200
4238300
4238400
4238500
4238600
4238700



Trend Plot, SOS_Main_S (Pass 1): by 12m bench swaths (AuKG, AuID, AuNN)
Gold Grade g/t
Number of locks
Bench Swath m
SOS_MAIN_S Data AuKg
SOS_MAIN_S Pass=1 AuID
SOS_MAIN_S Pass=1 AuNN
SOS_MAIN_S Pass=1 Count
0
5
10
15
20
25
0
500
1000
1500
2000
2500
3000
460
480
500
520
540
560
580
600
620
640
660
680
700
720

Trend Plot, SOS_Main_S (Pass 1): by 20m northing swaths (AuKG, AuID, AuNN)





Trend Plot, SOS_Upper (Pass 1): by 12m bench swaths (AuKG, AuID, AuNN)
SOS_Upper Data AuKg
SOS_Upper Pass=1 AuID
SOS_Upper Pass=1 AuNN
SOS_Upper Pass=1 Count
Gold Grade g/t
Number of locks
Bench Swath m
0
2
4
6
8
10
12
14
16
18
500
1000
1500
2000
2500
3000
500
520
540
560
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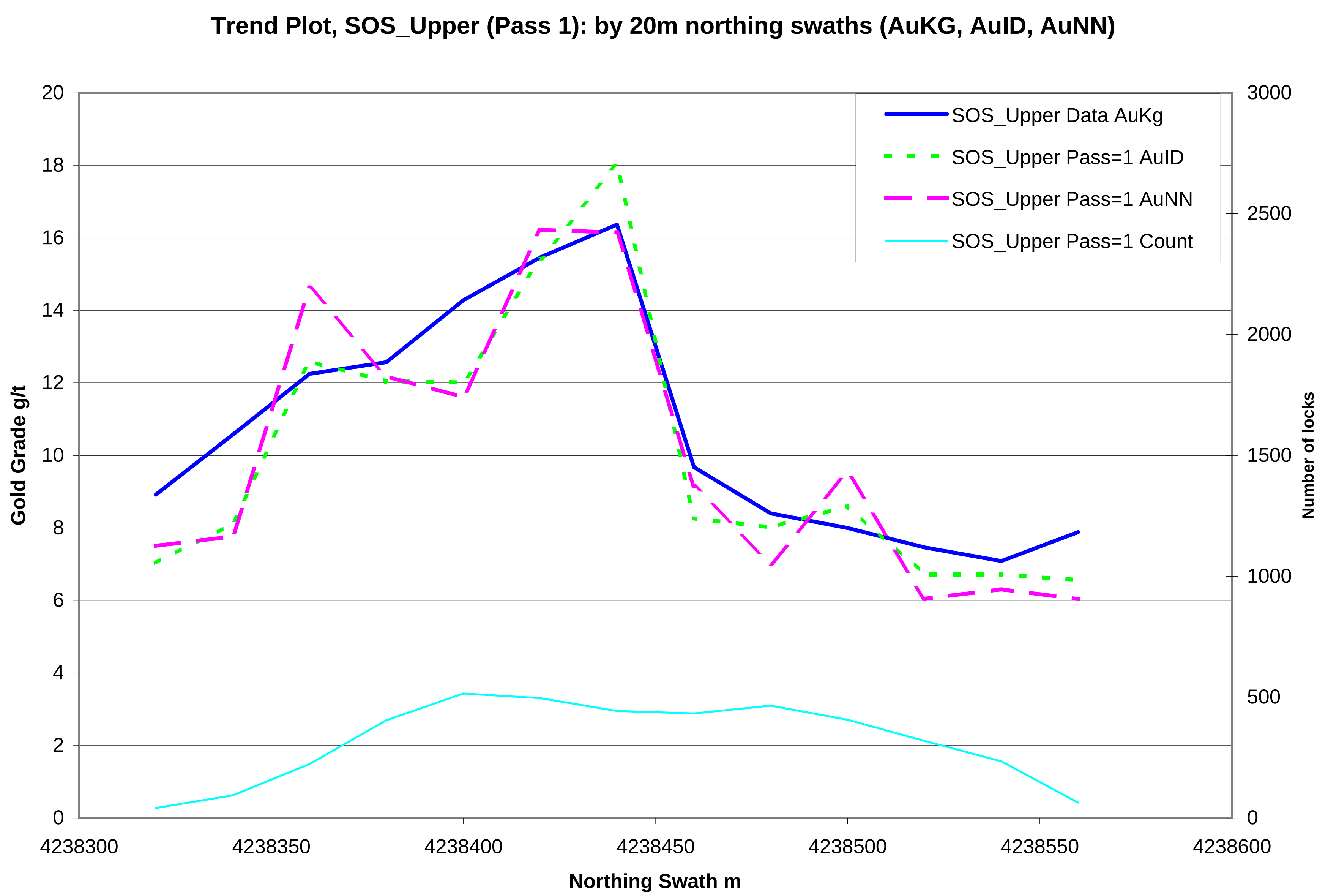
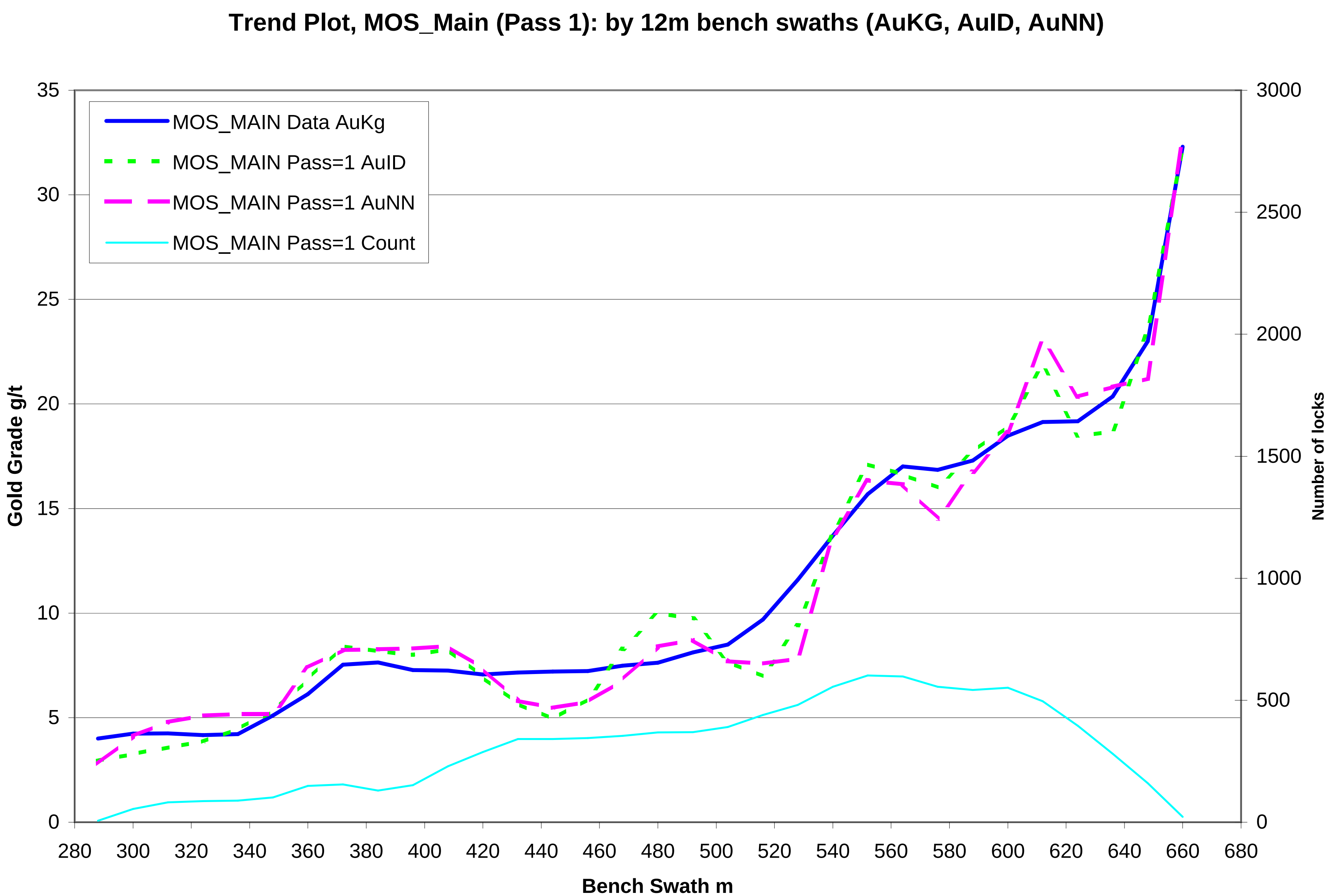
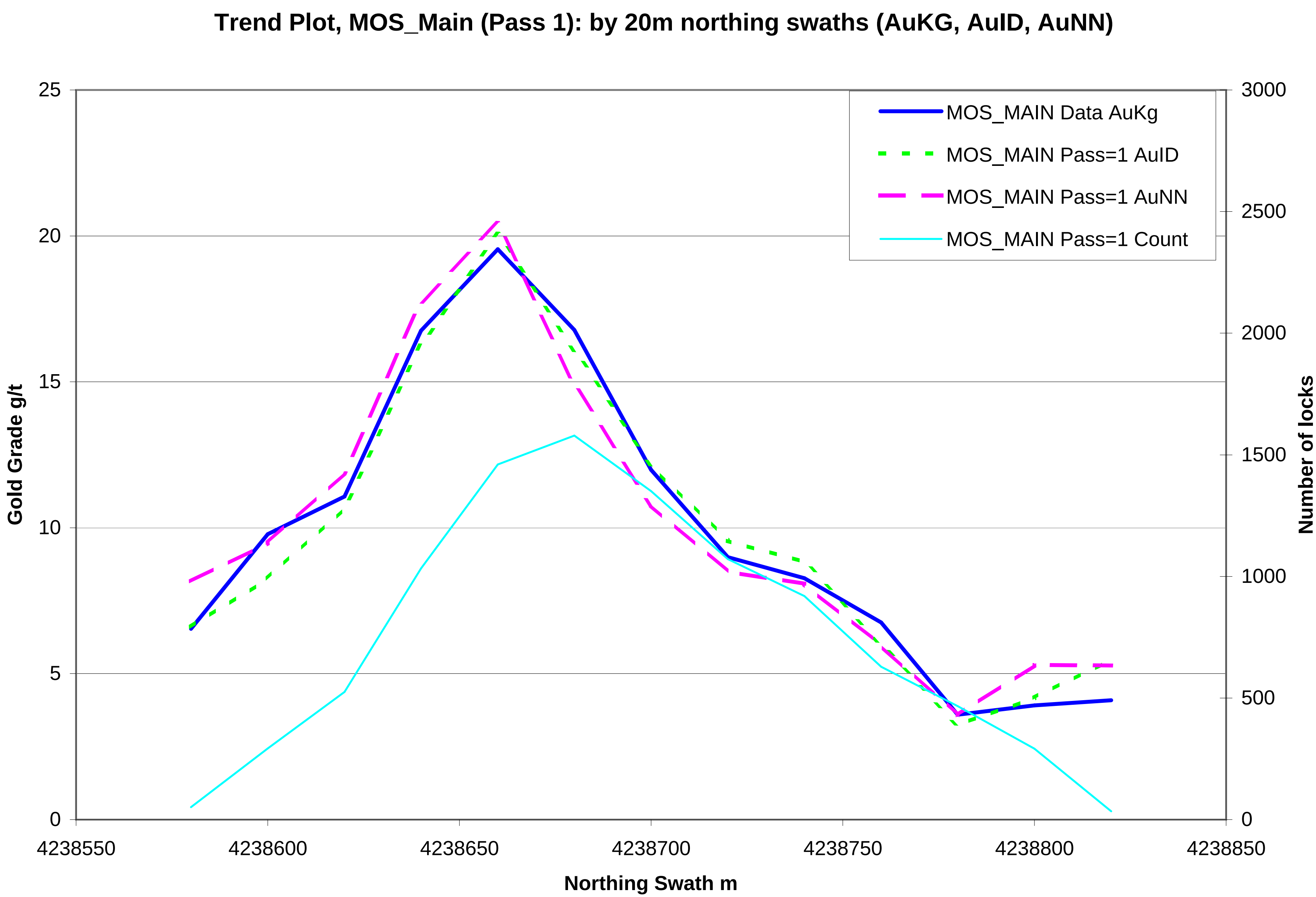
660
680
700
720
740

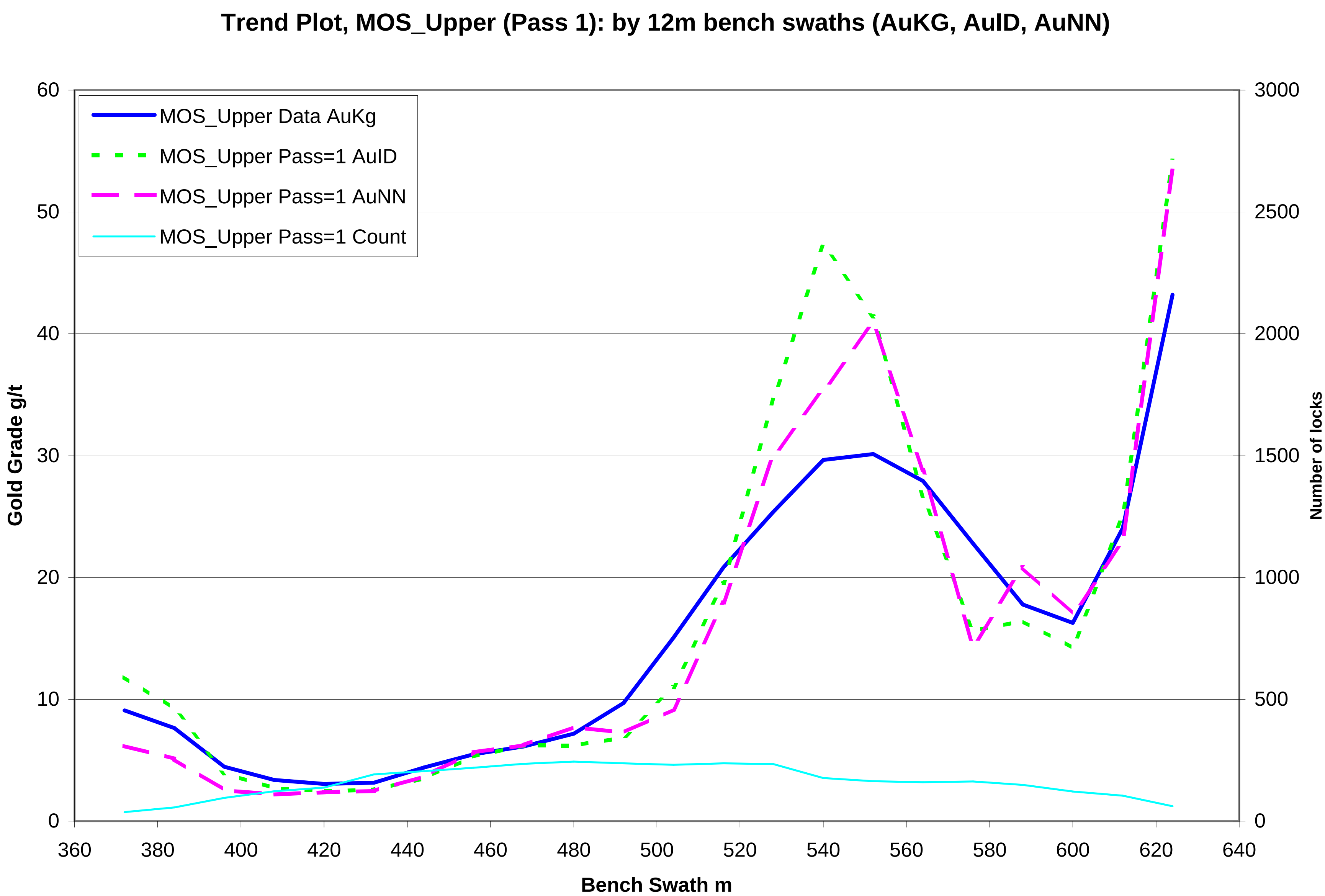


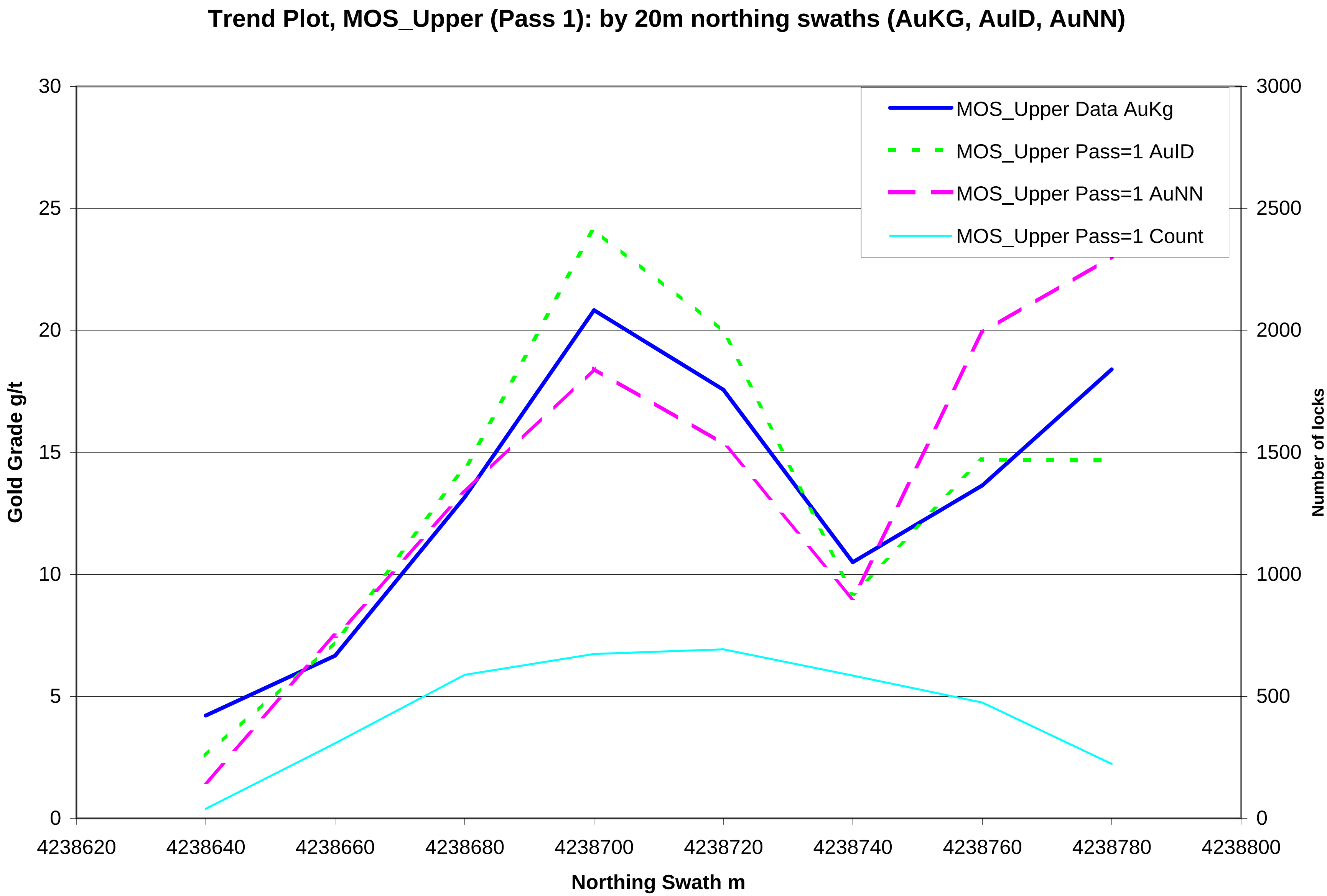
Trend Plot, SOS_Upper (Pass 1): by 20m northing swaths (AuKG, AuID, AuNN)
SOS_Upper Data AuKg
SOS_Upper Pass=1 AuID
SOS_Upper Pass=1 AuNN
SOS_Upper Pass=1 Count
Gold Grade g/t
Number of locks
Northing Swath m
0
2
4
6
8
10
12
14
16
18
20
0
500
1000
1500
2000
2500
3000
4238300
4238350
4238400
4238450
4238500
4238550
4238600

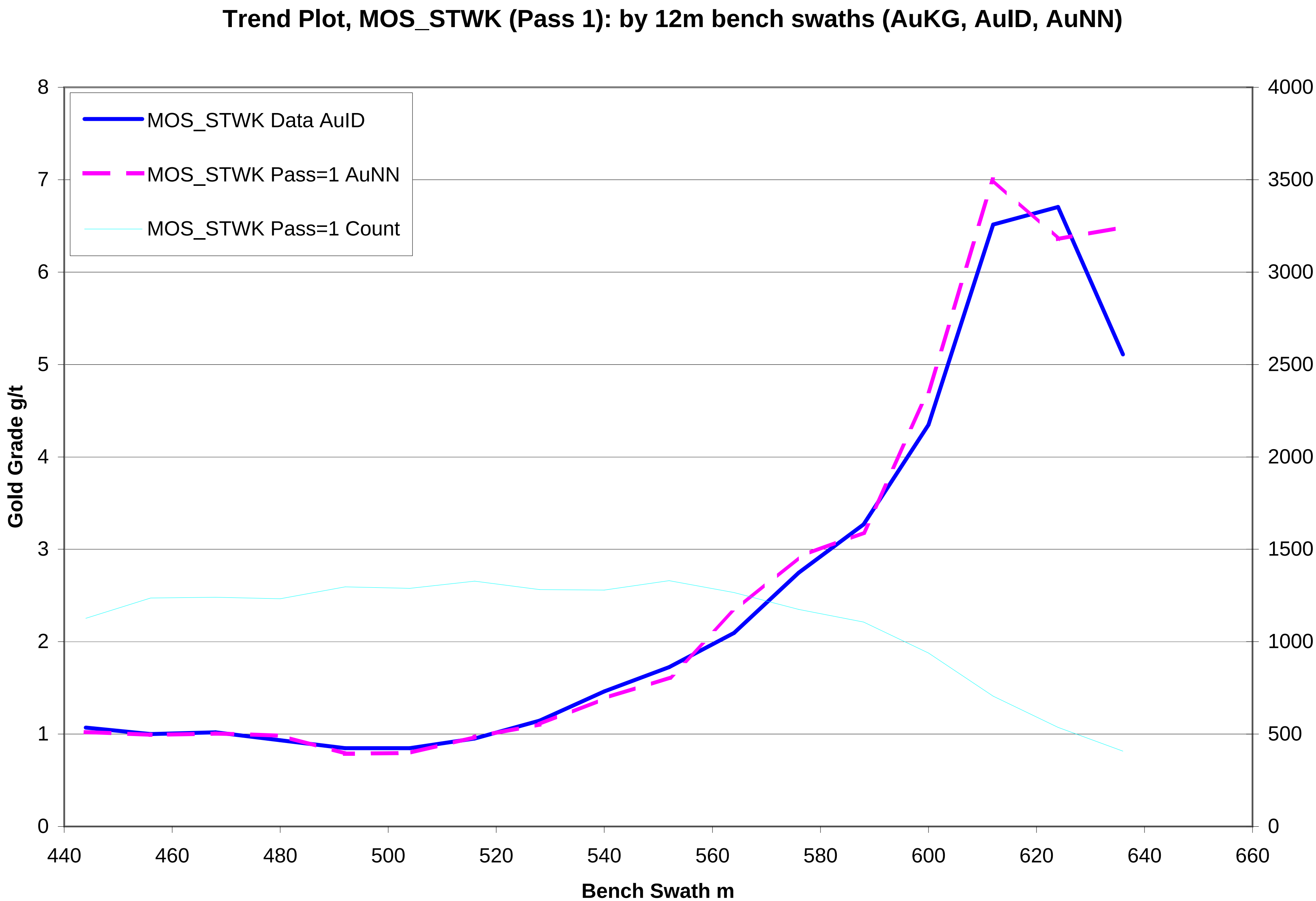


Trend Plot, MOS_Main (Pass 1): by 12m bench swaths (AuKG, AuID, AuNN)
MOS_MAIN Data AuKg
MOS_MAIN Pass=1 AuID
MOS_MAIN Pass=1 AuNN
MOS_MAIN Pass=1 Count
Gold Grade g/t
Number of locks
Bench Swath m
0
5
10
15
20
25
30
35
500
1000
1500
2000
2500
3000
280
300
320
340
360
380
400
420
440
460
480
500
520
540
560
580
600
620
640
660
680

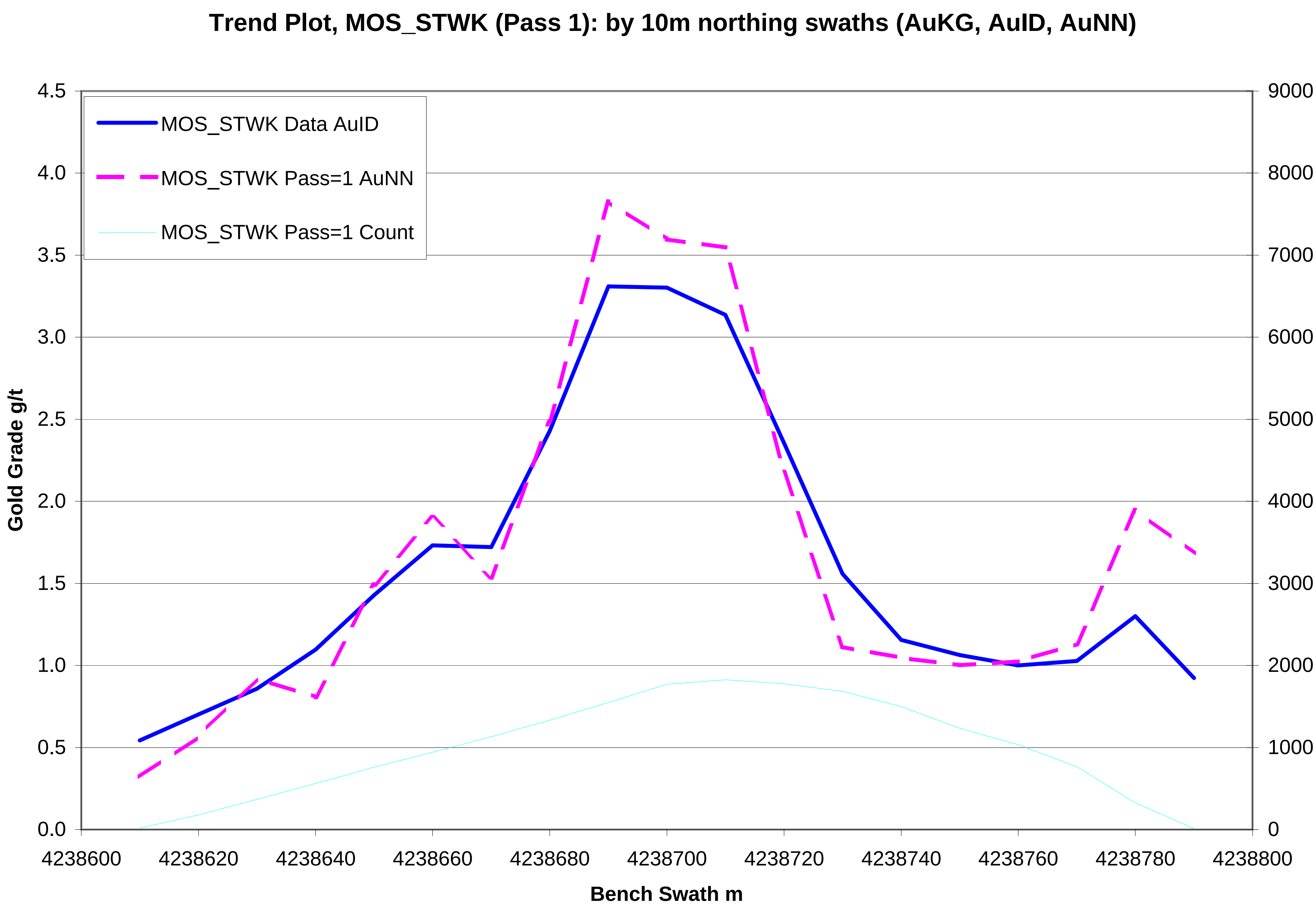

Trend Plot, MOS_Main (Pass 1): by 20m northing swaths (AuKG, AuID, AuNN)
Gold Grade g/t
Number of locks
Northing Swath m
MOS_MAIN Data AuKg
MOS_MAIN Pass=1 AuID
MOS_MAIN Pass=1 AuNN
MOS_MAIN Pass=1 Count
0
5
10
15
20
25
0
500
1000
1500
2000
2500
3000
4238550
4238600
4238650
4238700
4238750
4238800
4238850


Trend Plot, MOS_Upper (Pass 1): by 12m bench swaths (AuKG, AuID, AuNN)
MOS_Upper Data AuKg
MOS_Upper Pass=1 AuID
MOS_Upper Pass=1 AuNN
MOS_Upper Pass=1 Count
Gold Grade g/t
Number of locks
Bench Swath m
0
10
20
30
40
50
60
0
500
1000
1500
2000
2500
3000
360
380
400
420
440
460
480
500
520
540
560
580
600
620
640


Trend Plot, MOS_Upper (Pass 1): by 20m northing swaths (AuKG, AuID, AuNN)
MOS_Upper Data AuKg
MOS_Upper Pass=1 AuID
MOS_Upper Pass=1 AuNN
MOS_Upper Pass=1 Count
Gold Grade g/t
Number of locks
Northing Swath m
0
5
10
15
20
25
30
0
500
1000
1500
2000
2500
3000
4238620
4238640
4238660
4238680
4238700
4238720
4238740
4238760
4238780
4238800



Trend Plot, MOS_STWK (Pass 1): by 12m bench swaths (AuKG, AuID, AuNN)
MOS_STWK Data AuID
MOS_STWK Pass=1 AuNN
MOS_STWK Pass=1 Count
Gold Grade g/t
Bench Swath m
0
1
2
3
4
5
6
7
8
0
500
1000
1500
2000
2500
3000
3500
4000
440
460
480
500
520
540
560
580
600
620
640
660



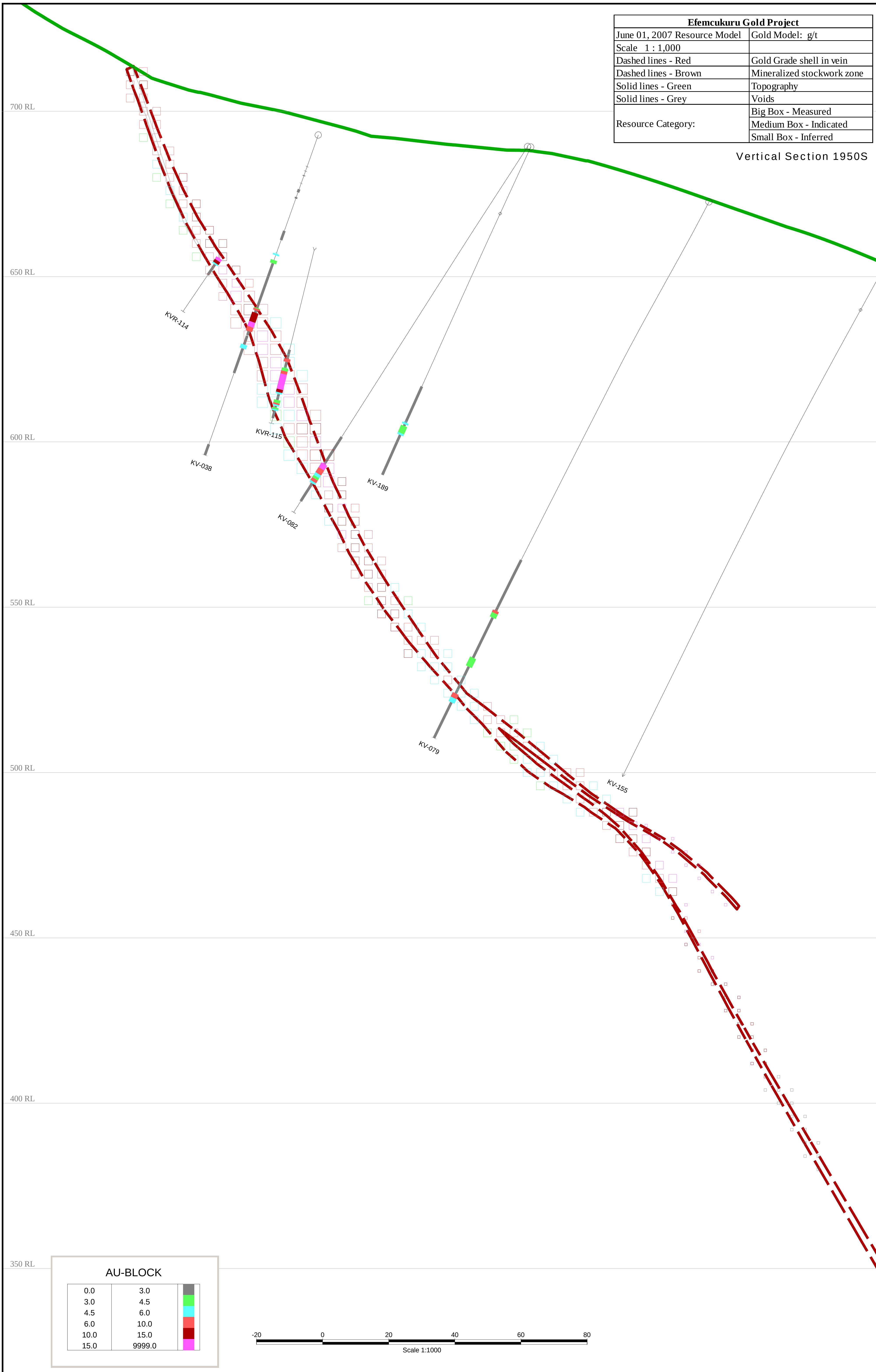
Trend Plot, MOS_STWK (Pass 1): by 10m northing swaths (AuKG, AuID, AuNN)
MOS_STWK Data AuID
MOS_STWK Pass=1 AuNN
MOS_STWK Pass=1 Count
Gold Grade g/t
Bench Swath m
4.5
4.0
3.5
3.0
2.5
2.0
1.5
1.0
0.5
0.0
9000
8000
7000
6000
5000
4000
3000
2000
1000
0
4238600
4238620
4238640
4238660
4238680
4238700
4238720
4238740
4238760
4238780
4238800

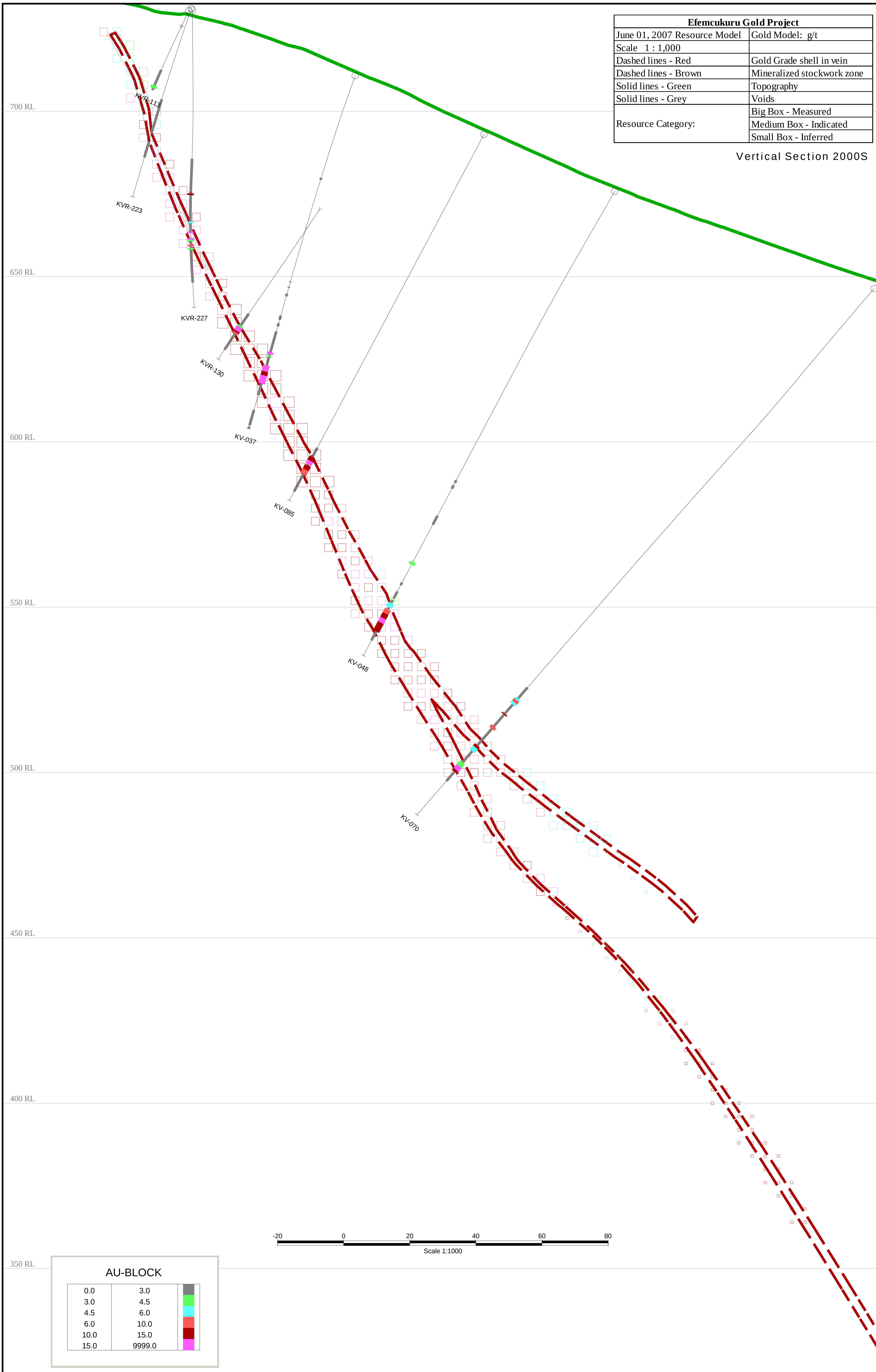
WARDROP

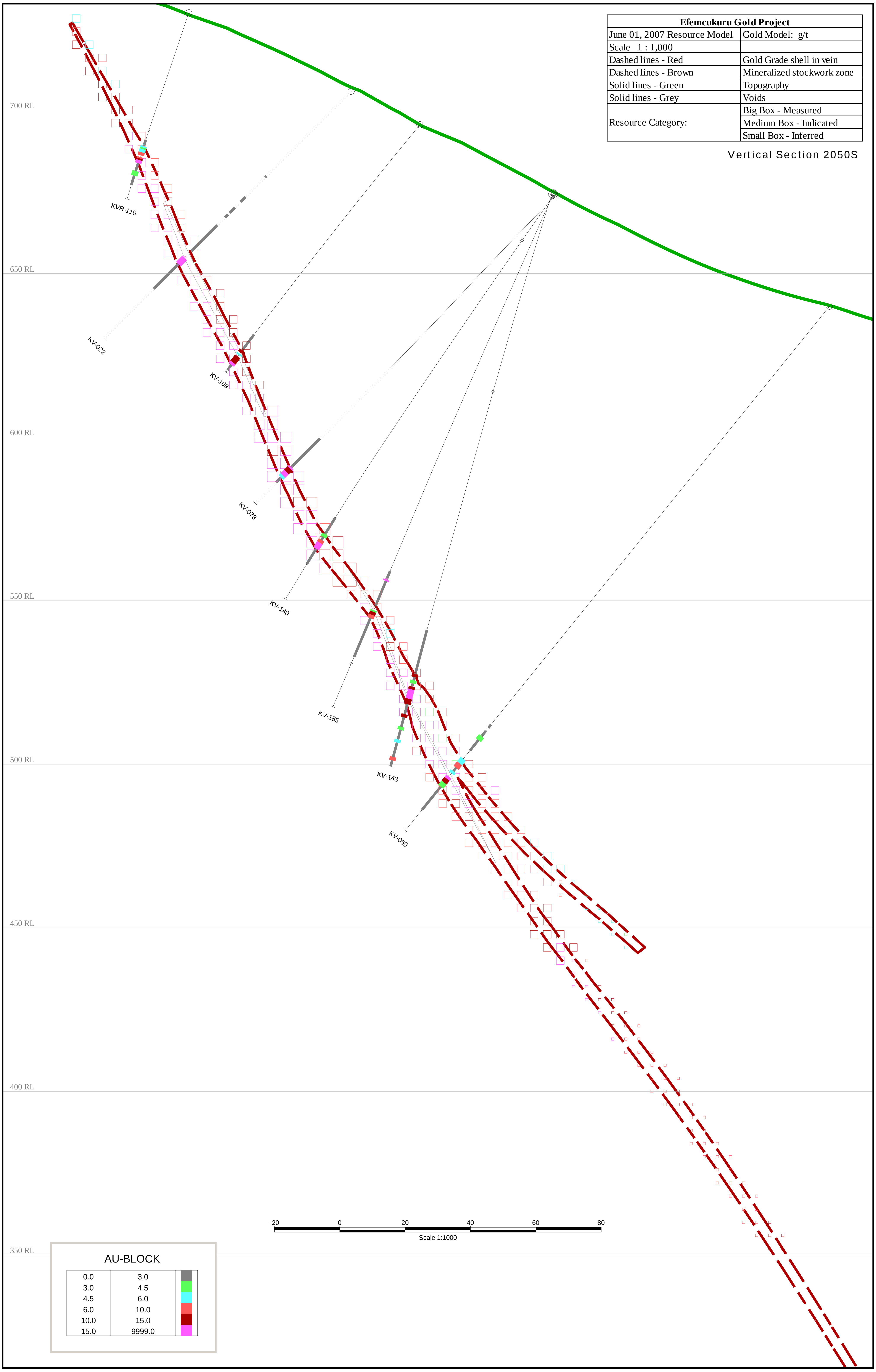
eldoradogold

APPENDIX C

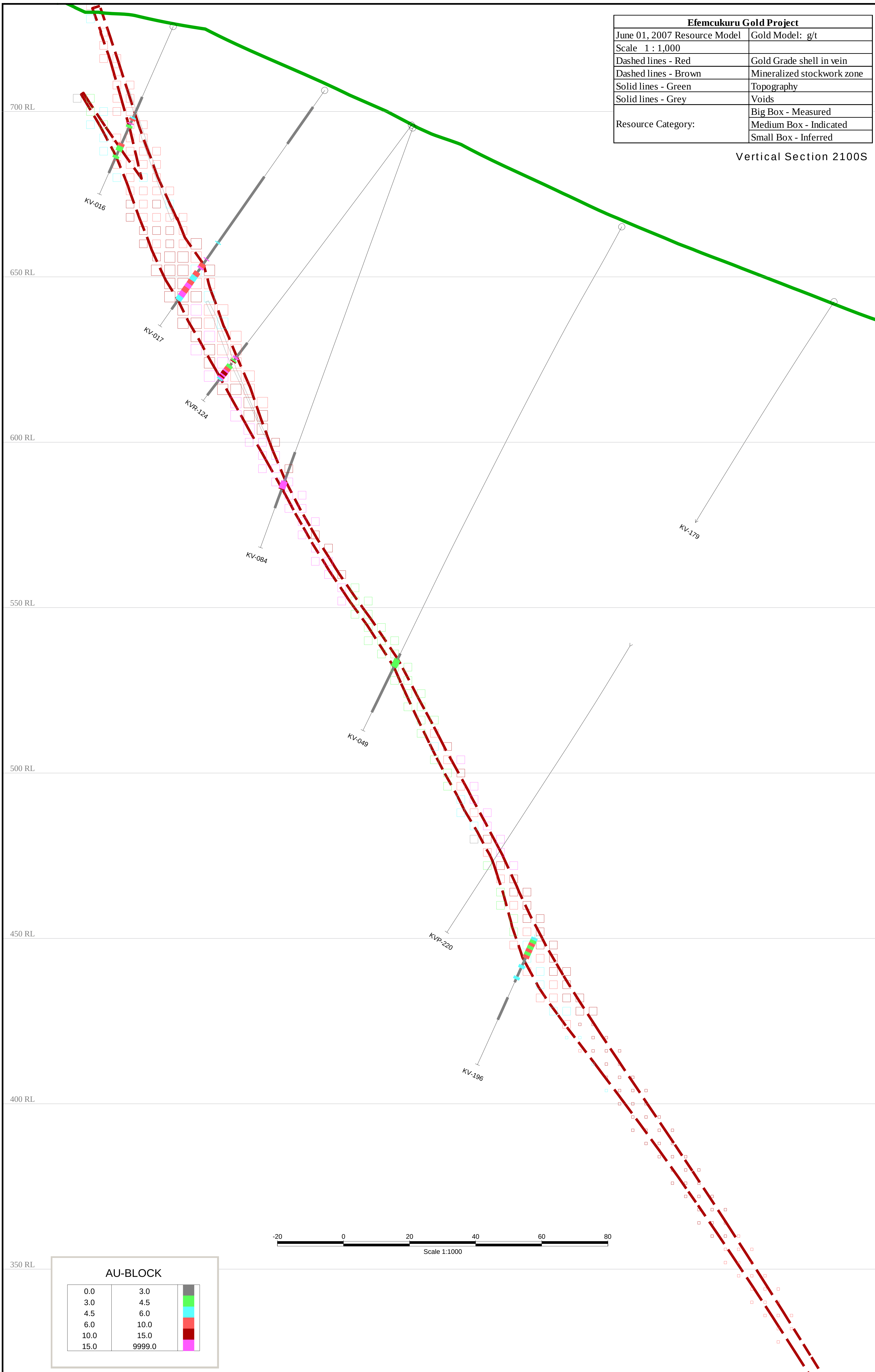
SECTIONS SOS
SECTIONS MOS

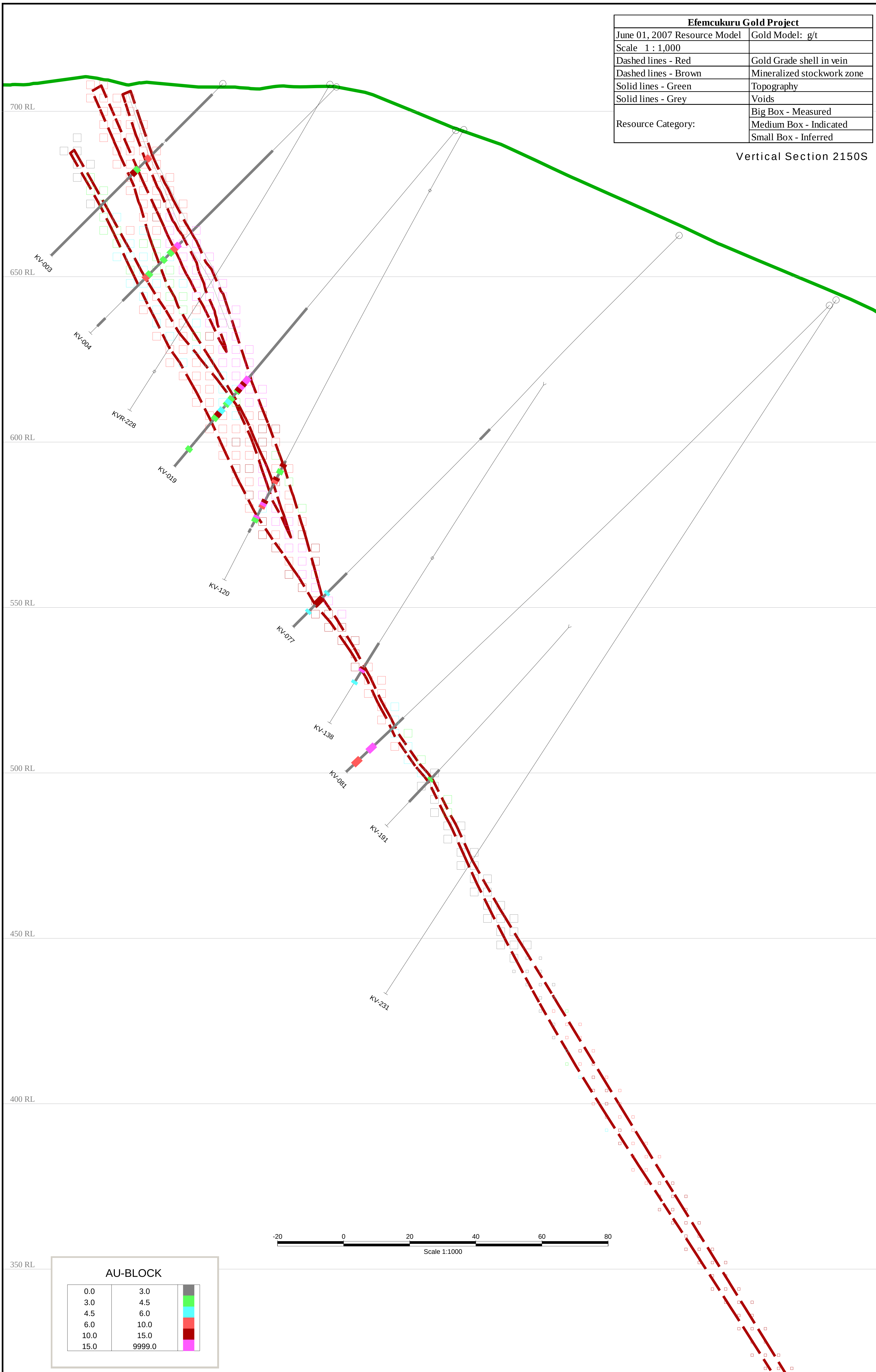

Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 1950S
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KVR-114
KVR-115
KV-038
KV-082
KV-189
KV-079
KV-155
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0
-20
0
20
40
60
80
Scale 1:1000

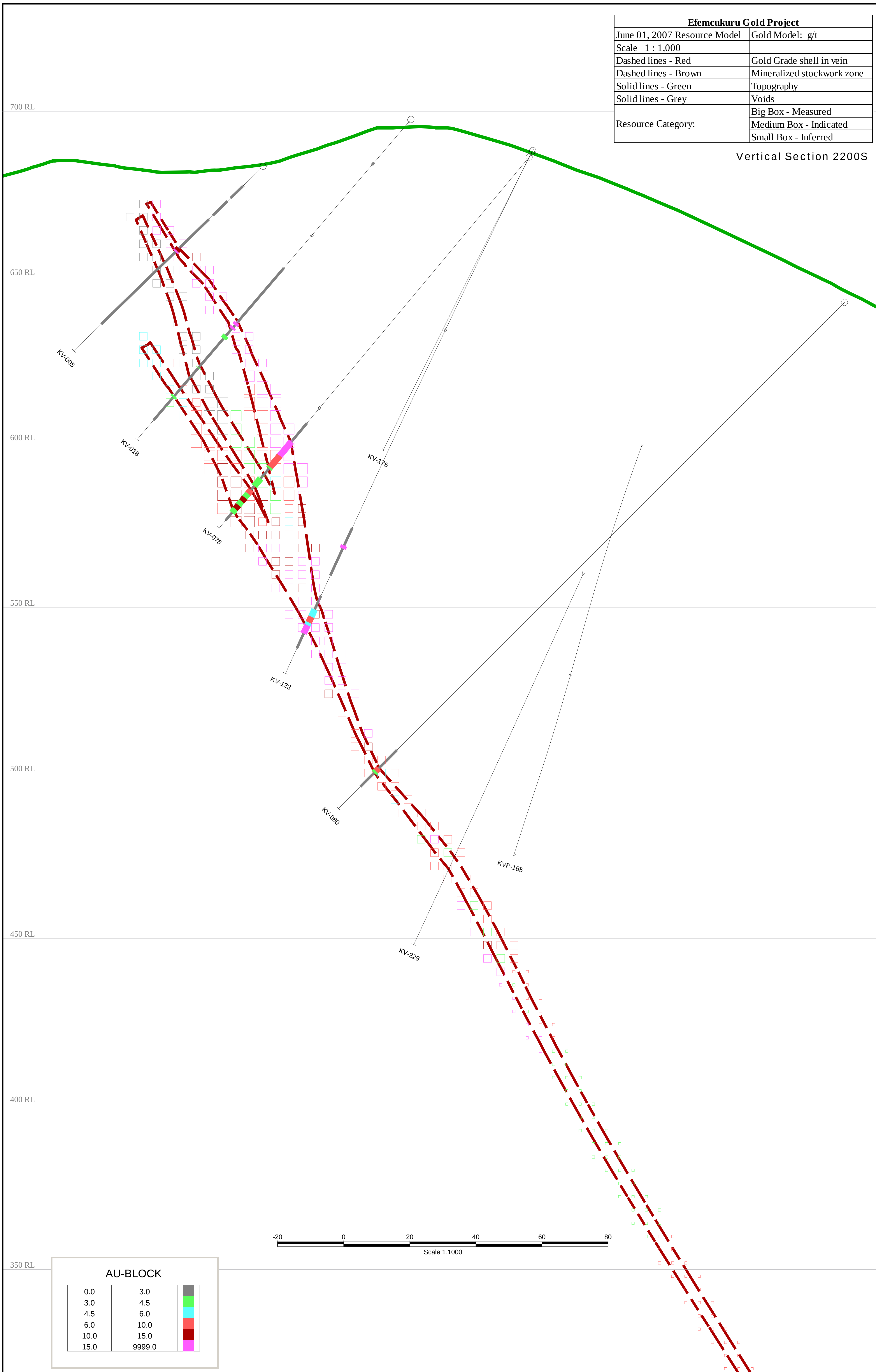

Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 2000S
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KVR-113
KVR-223
KVR-227
KVR-130
KV-037
KV-085
KV-048
KV-070
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0

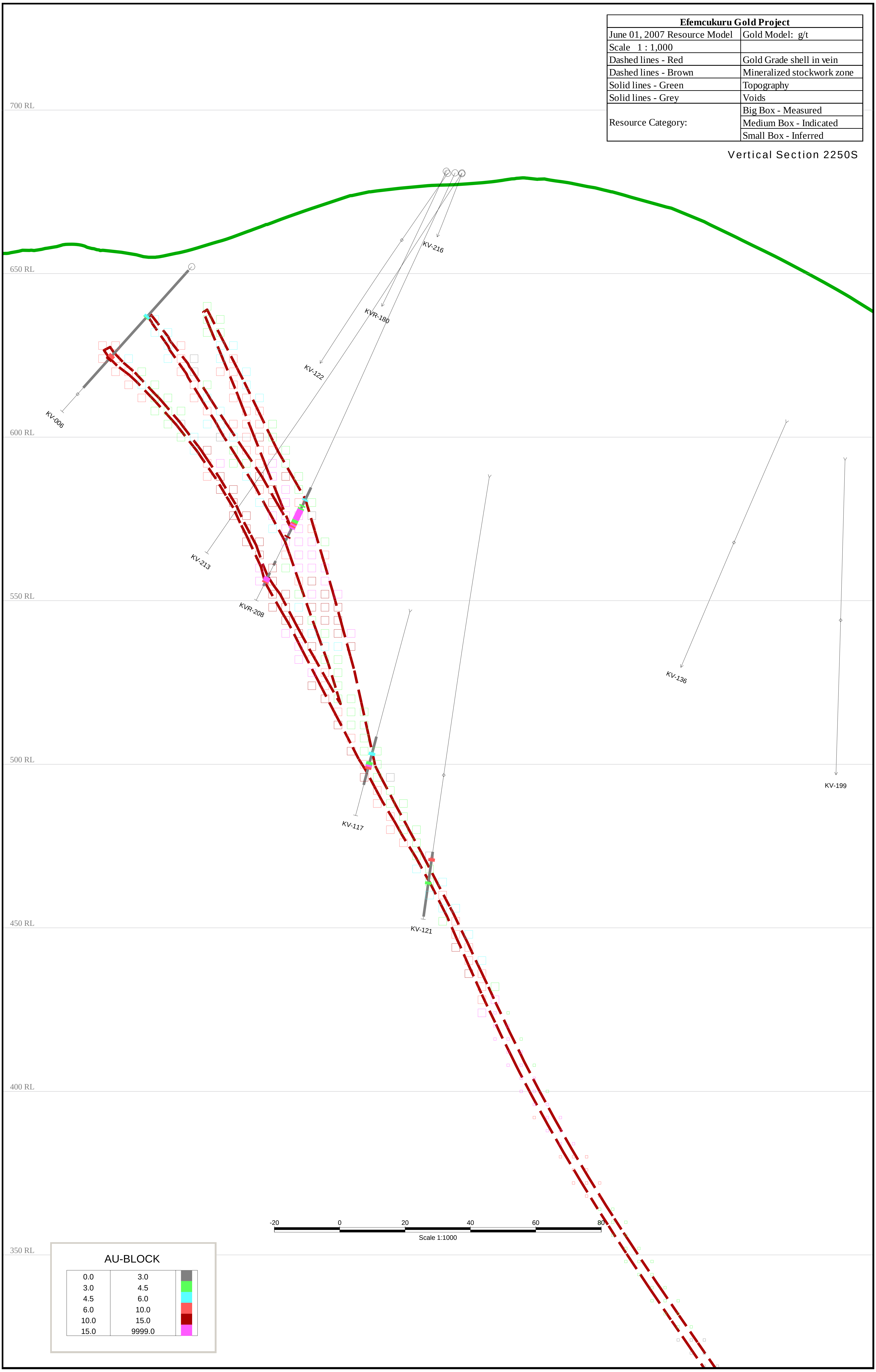


Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 2050S
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KVR-110
KV-022
KV-109
KV-078
KV-140
KV-185
KV-143
KV-059
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0

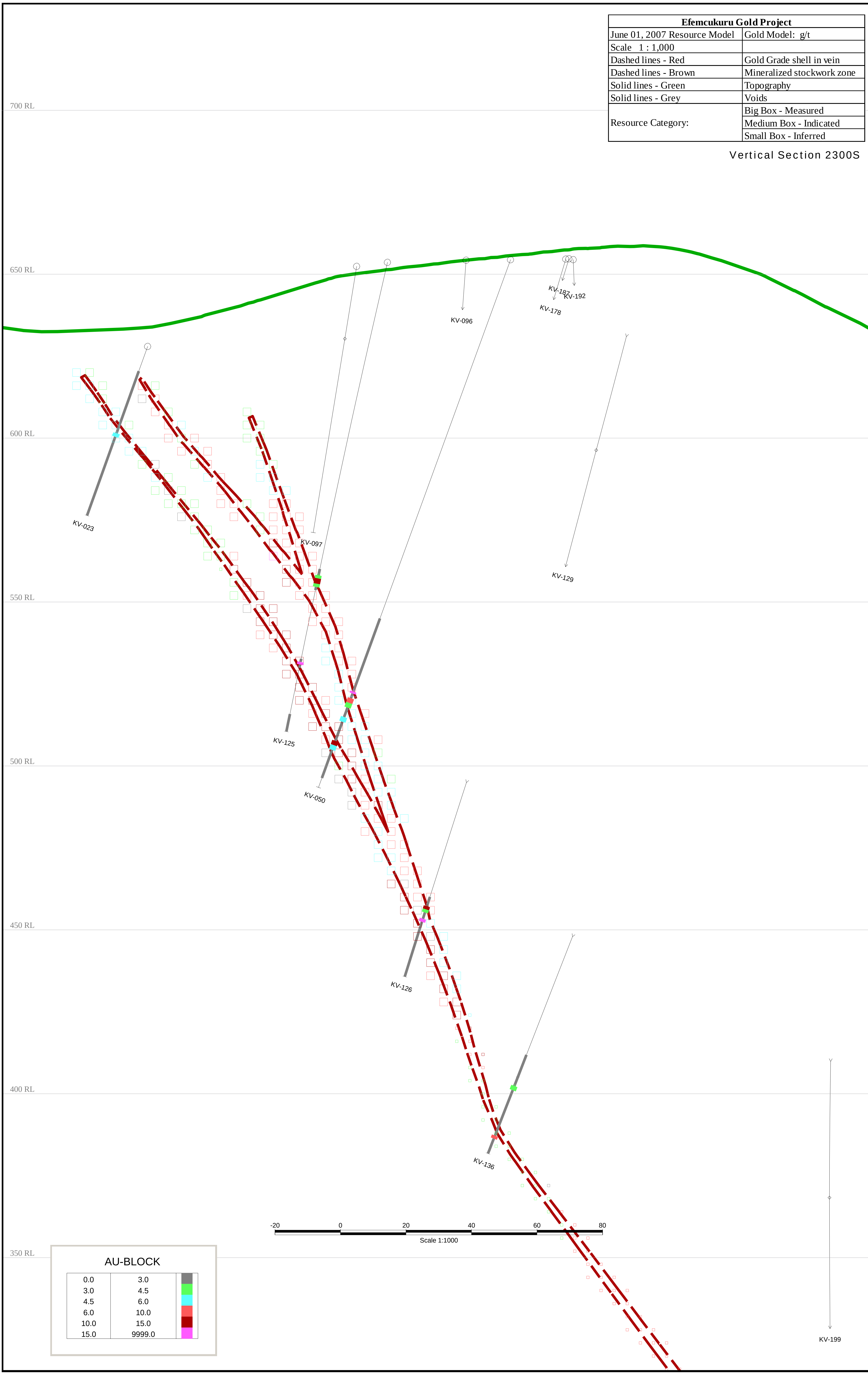

Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 2100S
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KV-016
KV-017
KVR-124
KV-084
KV-049
KVP-220
KV-196
KV-179
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0


Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 2150S
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KV-003
KV-004
KVR-228
KV-019
KV-120
KV-077
KV-138
KV-081
KV-191
KV-231
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0


Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 2200S
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KV-005
KV-018
KV-075
KV-176
KV-123
KV-080
KV-229
KVP-165
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0



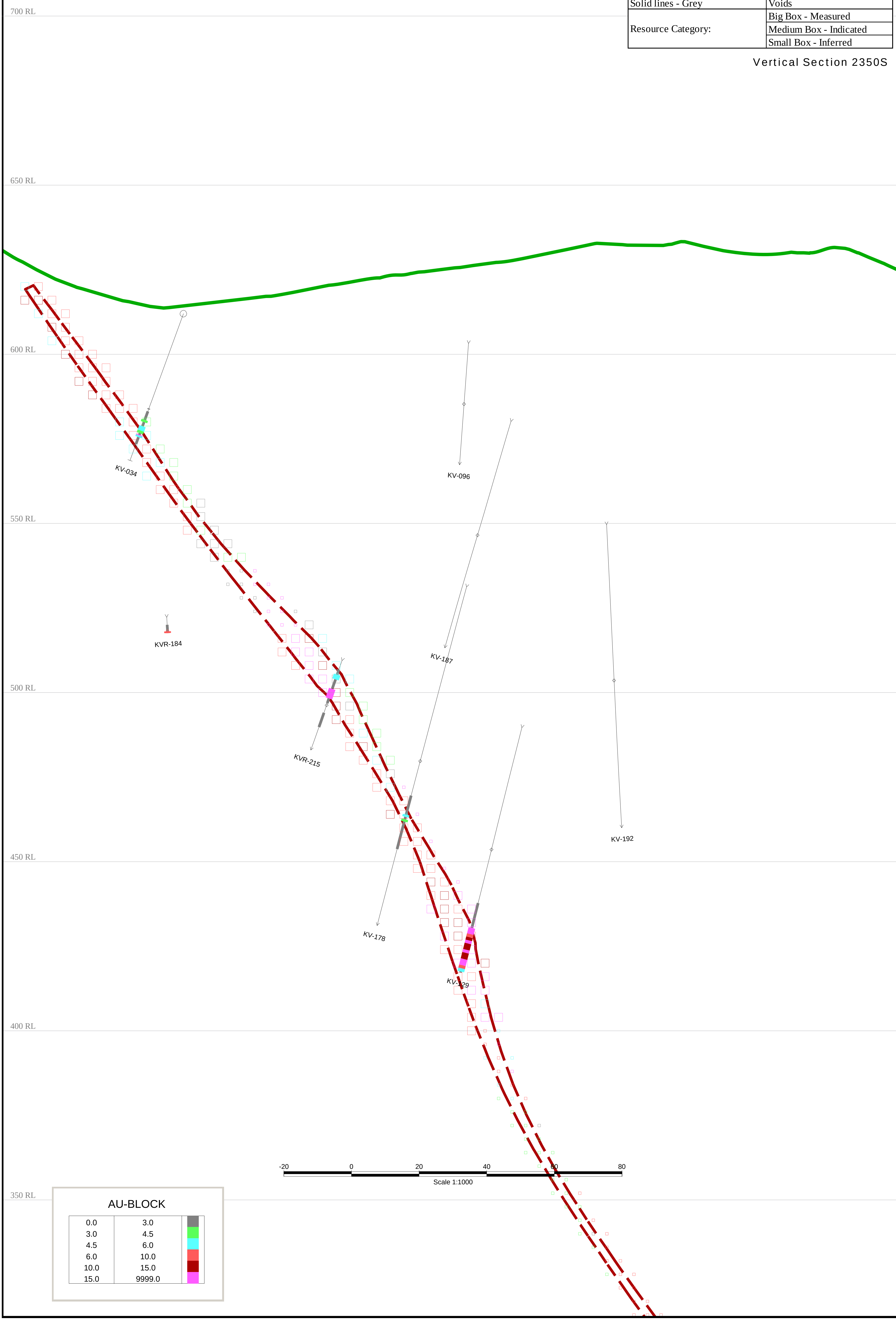
Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 2250S
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KV-216
KVR-180
KV-122
KV-006
KV-213
KVR-208
KV-117
KV-121
KV-136
KV-199
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0

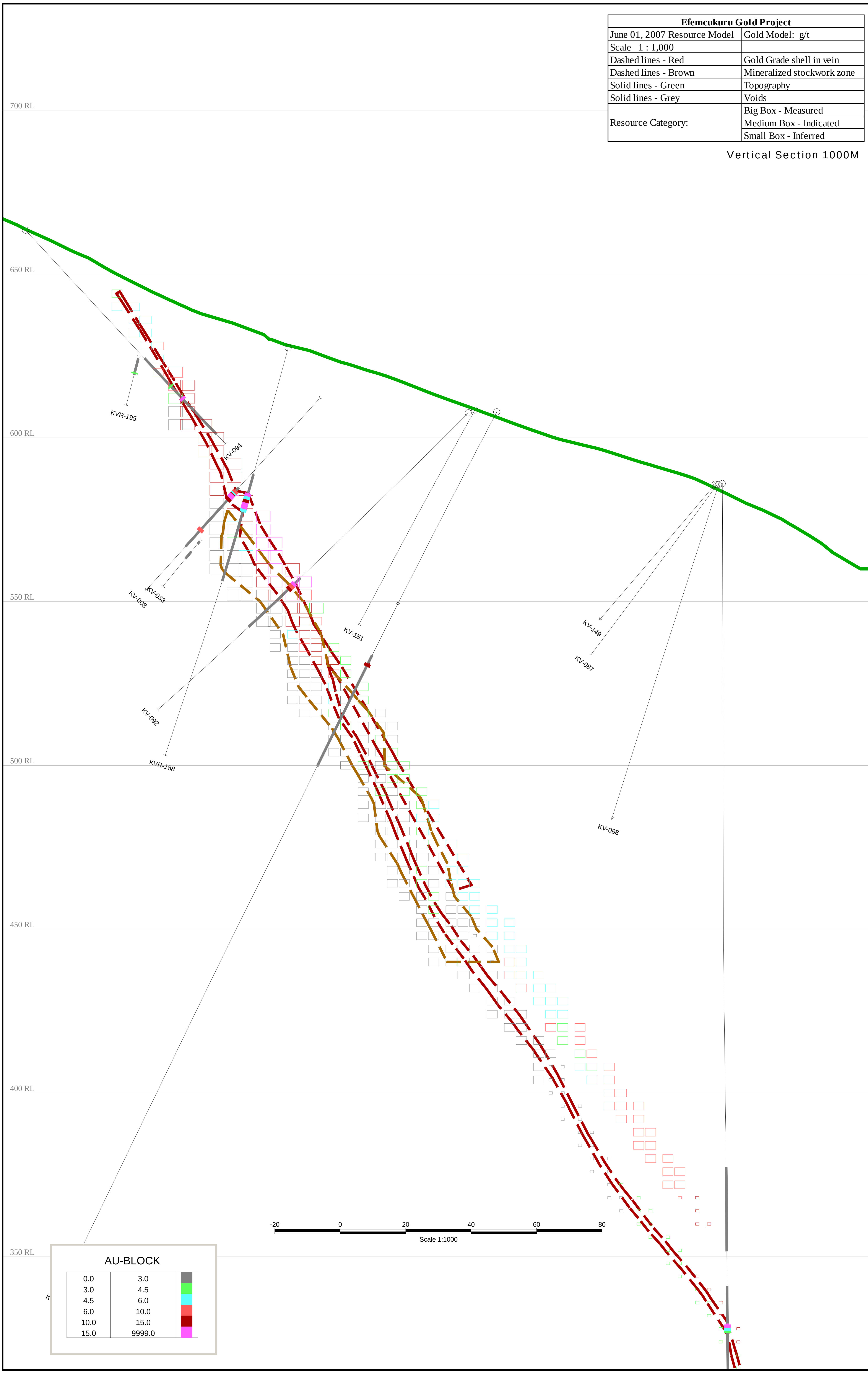


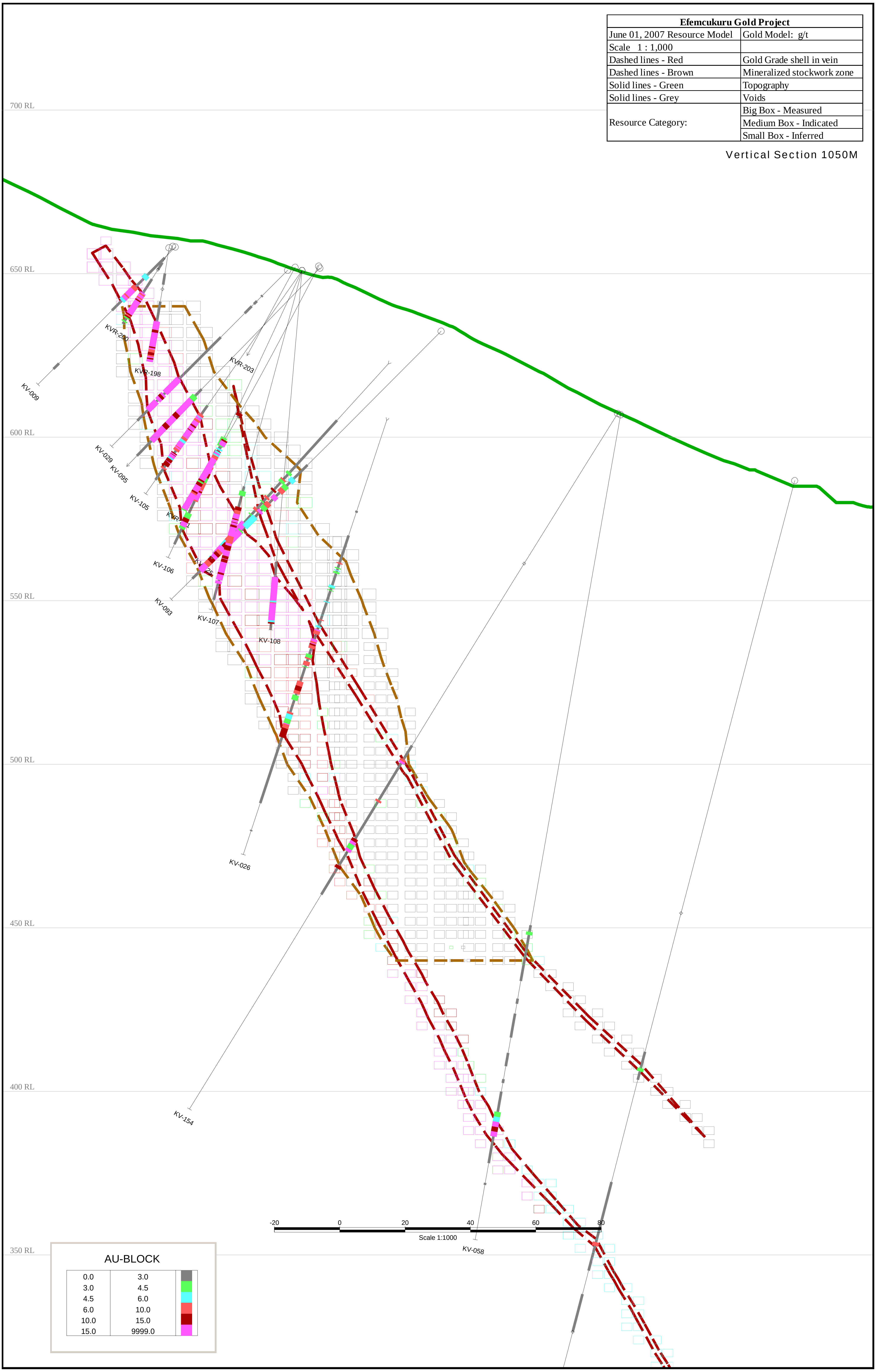
Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 2300S
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KV-187
KV-192
KV-178
KV-096
KV-023
KV-097
KV-129
KV-125
KV-050
KV-126
KV-136
KV-199
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0

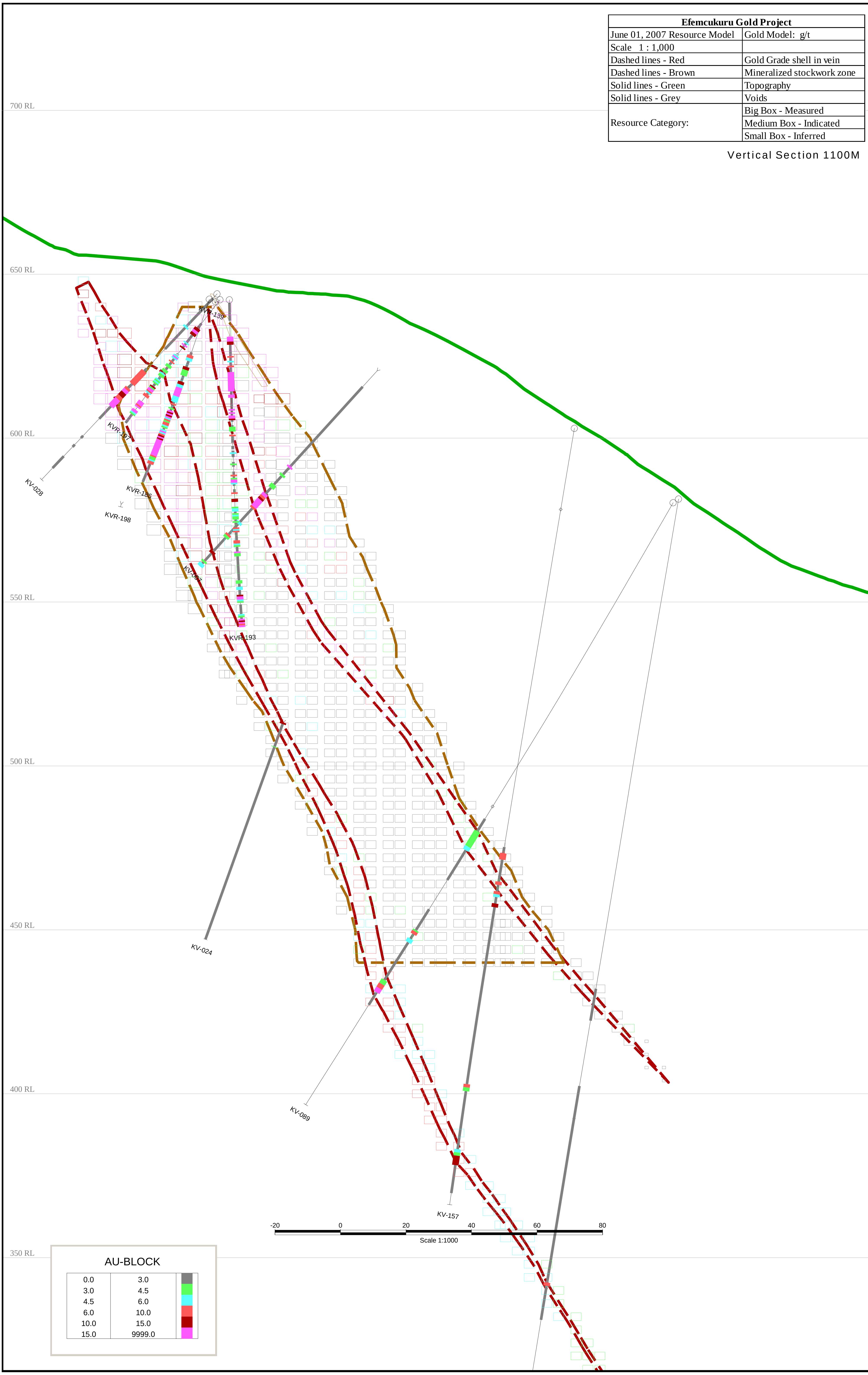
Efemcukuru Gold Project	
June 01, 2007 Resource Model	Gold Model: g/t
Scale 1 : 1,000	
Dashed lines - Red	Gold Grade shell in vein
Dashed lines - Brown	Mineralized stockwork zone
Solid lines - Green	Topography
Solid lines - Grey	Voids
Resource Category:	Big Box - Measured
	Medium Box - Indicated
	Small Box - Inferred

Vertical Section 2350S




Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 1000M
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KVR-195
KV-094
KV-033
KV-008
KV-151
KV-092
KVR-188
KV-149
KV-087
KV-088
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0

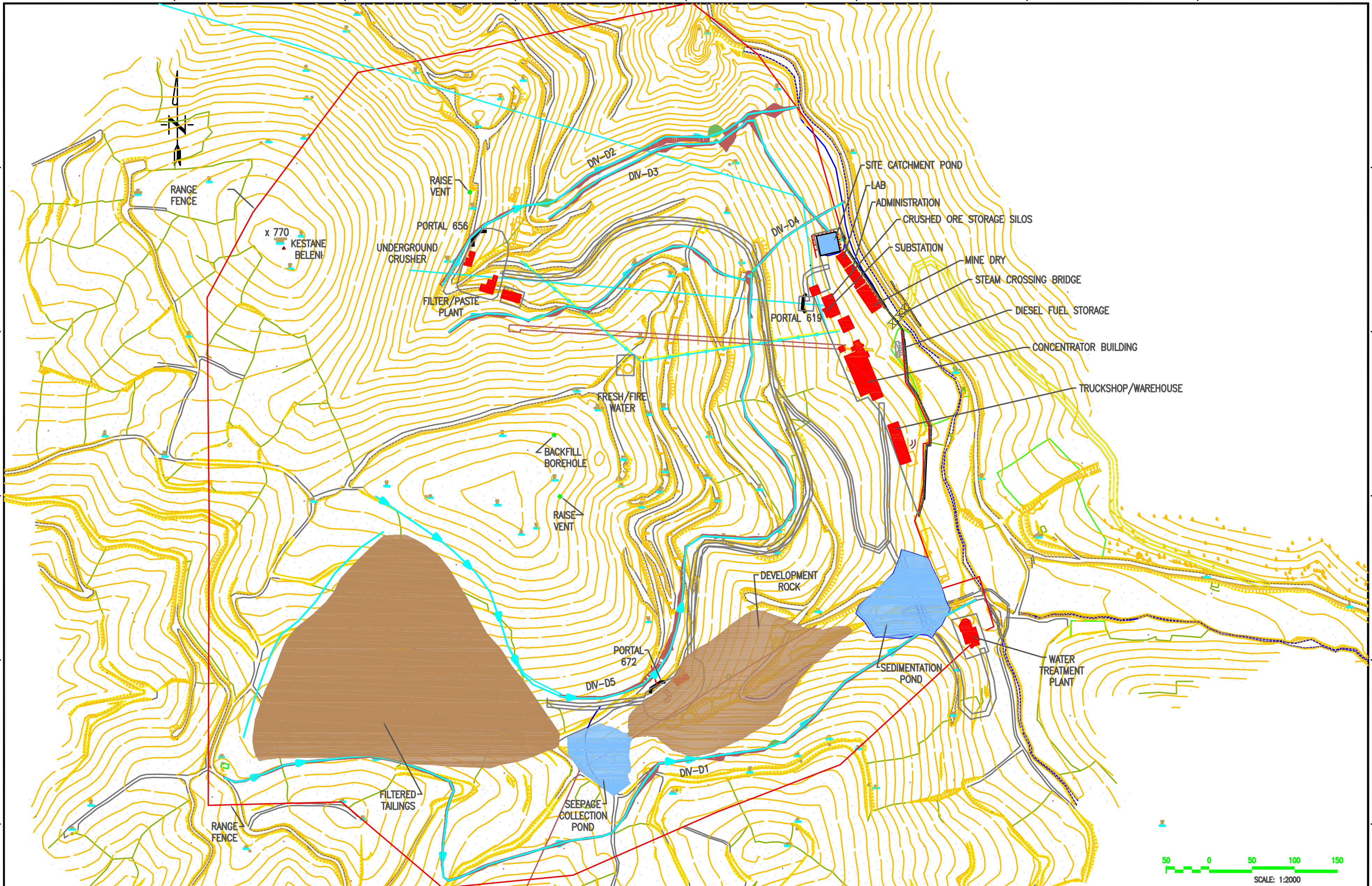

Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 1050M
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KVR-200
KVR-198
KV-009
KVR-203
KV-029
KV-095
KV-105
KV-106
KV-093
KV-107
KV-108
KV-026
KV-154
KV-058
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0



Efemcukuru Gold Project
June 01, 2007 Resource Model
Gold Model: g/t
Scale 1 : 1,000
Dashed lines - Red
Gold Grade shell in vein
Dashed lines - Brown
Mineralized stockwork zone
Solid lines - Green
Topography
Solid lines - Grey
Voids
Resource Category:
Big Box - Measured
Medium Box - Indicated
Small Box - Inferred
Vertical Section 1100M
700 RL
650 RL
600 RL
550 RL
500 RL
450 RL
400 RL
350 RL
KV-028
KVR-197
KVR-186
KVR-198
KVR-139
KV-067
KVR-193
KV-024
KV-089
KV-157
-20
0
20
40
60
80
Scale 1:1000
AU-BLOCK
0.0 3.0
3.0 4.5
4.5 6.0
6.0 10.0
10.0 15.0
15.0 9999.0

APPENDIX D

SITE PLANS
MINE PLANS
PROCESS DIAGRAMS- EFEMÇUKURU
PROCESS DIAGRAMS – KIŞLADAĞ



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REV. No.	ISSUE No.	DESCRIPTION	DATE	BY

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DRAWN BY:	J.DIAZ	MAY07
CHECK. BY:		
APP. BY:		

WESTERN TURKEY

eldoradogold

WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT

OVERALL SITE LAYOUT

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
A010001.DWG	05517201.00	A0-10-001	C

07/08/10 08:05

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REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
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B	1	ISSUED FOR FINAL CLIENT COMMENTS	03AUG07	DAS
A	1	ISSUED FOR COMMENT	31JUL07	JD

CLIENT | PROJ.MAN. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

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SCALE:	1:750	
DESIGN. BY:	J.DIAZ	MAYO7
DRAWN BY:	J.DIAZ	JUN/07
CHECK. BY:		
APP. BY:		

WESTERN TURKEY

eldoradogold

WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT

PLANT SITE PLAN

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
A010002.DWG	05517201.00	A0-10-002	C

U:\A-G\ELDORADO GOLD CORPORATION - 5172\05517201.00 - EFEMCUKURU PROJECT (FEASIBILITY STUDY)\10-LAYOUT\DWG\A010002.DWG

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07/09/16 12:14

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DWG. NO.	REFERENCE DRAWINGS

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
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E	1	ISSUED FOR FINAL CLIENT COMMENTS	03AUG07	DAS
D	1	ISSUED FOR FEASIBILITY STUDY	20JUL07	JD
C	1	ISSUED FOR EIS UPDATE	14NOV06	PAZ
B	1	ISSUED FOR FEASIBILITY STUDY	31OCT06	RAH
A	1	ISSUED FOR REVIEW	31AUG06	AK

CLIENT | PRO.MAN. | PRO.ENG. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

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CHECK. BY:		
APP. BY:		

WESTERN TURKEY

eldoradogold

WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT

KISLADAG CONCENTRATE REHANDLING SITE LAYOUT

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
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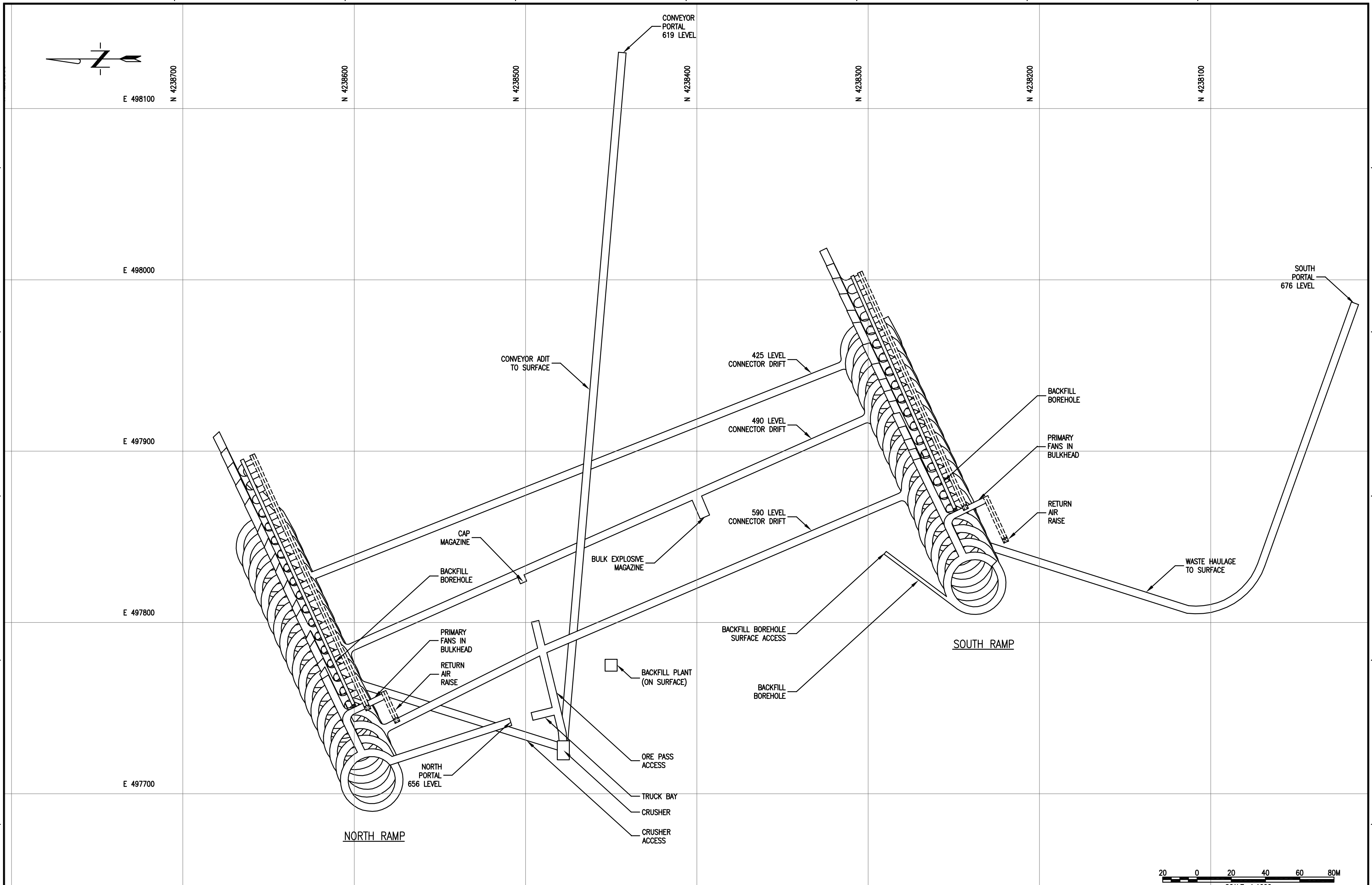

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CLIENT | PRELIMIN. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

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C	1	ISSUED FOR FEASABILITY STUDY REPORT DRAFT	03AUG07	T.S.

SECTION:	MINING	
SCALE:	1:1000	DATE
DESIGN. BY:	S.C.	03AUG07
DRAWN BY:	T.S.	03AUG07
CHECK. BY:		
APP. BY:		

WESTERN TURKEY

eldoradogold

WARDROP Engineering Inc.

REFERENCE No.: 0551720100-DWG-B440001

TITLE

EFEMÇUKURU PROJECT

MINING DEVELOPMENT PLAN
PLAN VIEW

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
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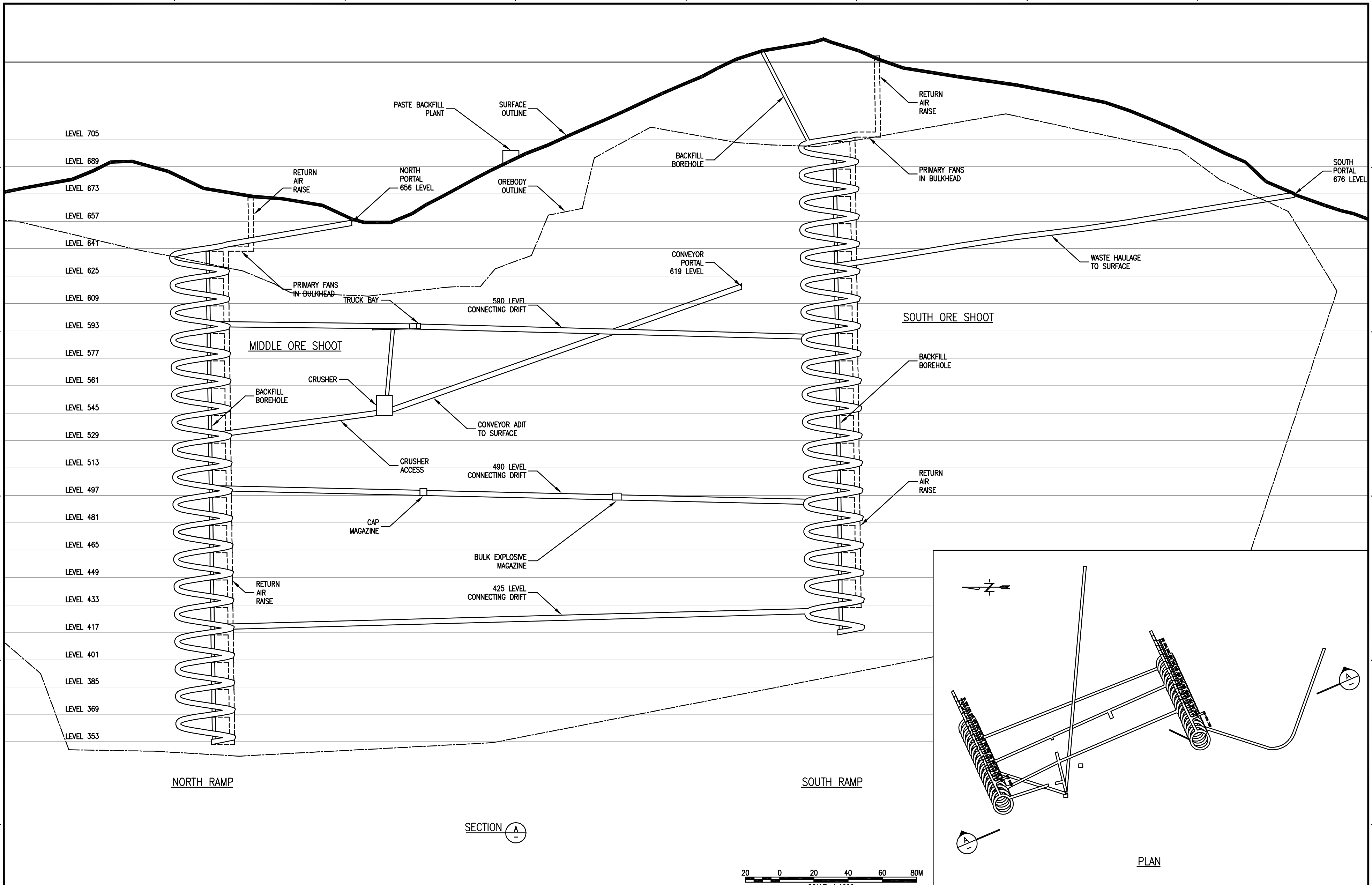

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DWG. NO.	REFERENCE DRAWINGS

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
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C	1	ISSUED FOR FEASABILITY STUDY REPORT DRAFT	03AUG07	T.S.

CLIENT / PROJ.MAN. / PROCESS / ELECTR. / INSTR. / PIPING / MECH. / STRUCT. / SERVICES / ARCH. / LAYOUT

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APP. BY:		

WESTERN TURKEY

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WARDROP Engineering Inc.

REFERENCE No.: 0551720100-DWG-B440002

TITLE

EFEMÇUKURU PROJECT

MINING DEVELOPMENT PLAN LONG SECTION VIEW

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
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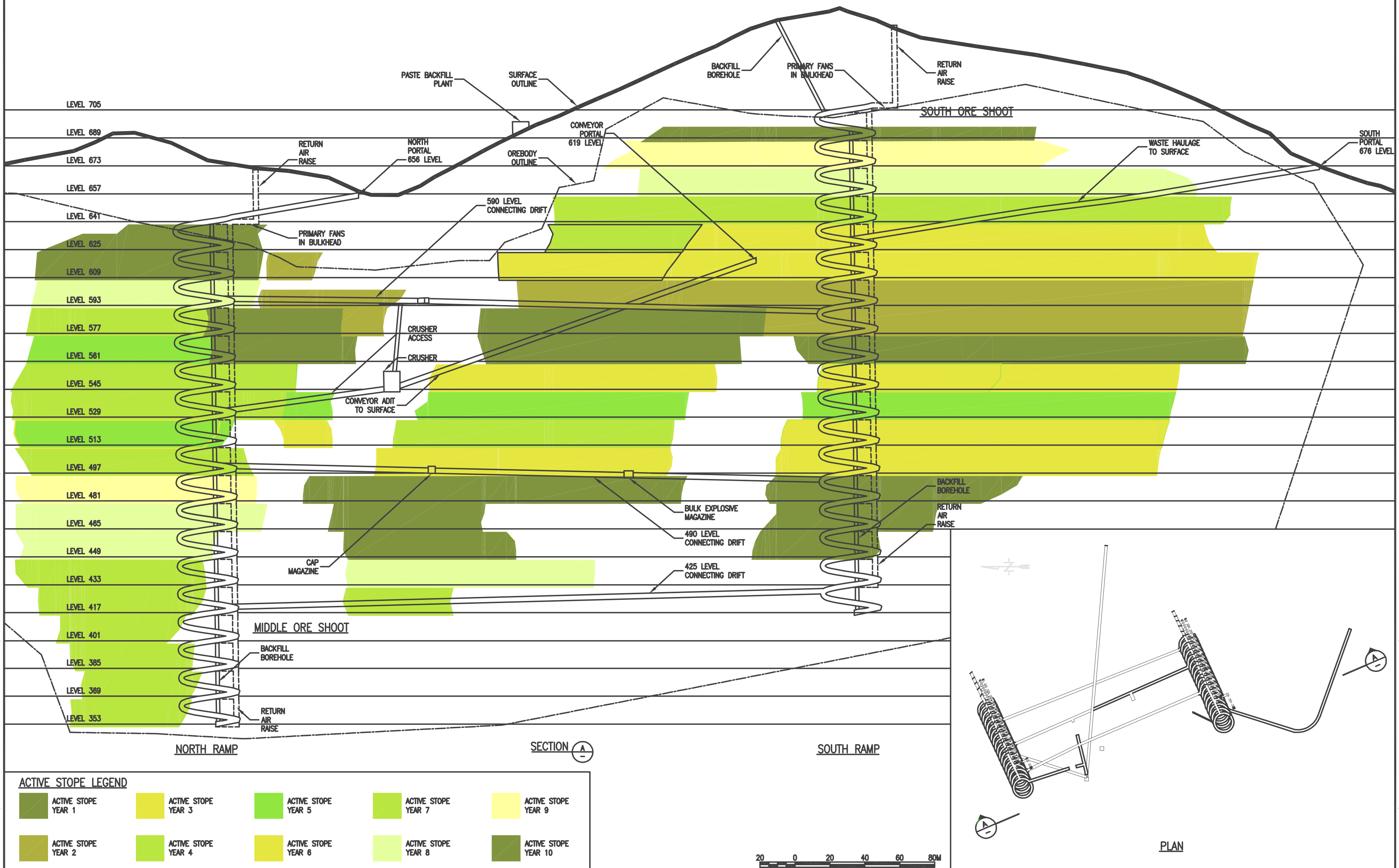

ACTIVE STOPE LEGEND

ACTIVE STOPE YEAR 1	ACTIVE STOPE YEAR 3	ACTIVE STOPE YEAR 5	ACTIVE STOPE YEAR 7	ACTIVE STOPE YEAR 9
ACTIVE STOPE YEAR 2	ACTIVE STOPE YEAR 4	ACTIVE STOPE YEAR 6	ACTIVE STOPE YEAR 8	ACTIVE STOPE YEAR 10

DWG. NO.	REFERENCE DRAWINGS

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C	1	ISSUED FOR FEASABILITY STUDY REPORT DRAFT	03AUG07	T.S.

CLIENT PROJMNG PROCESS ELECTRL INSTR. PIPING MECH. STRUCT. SERVICES ARCH. LAYOUT

		DATE
SECTION:	SECTION	
SCALE:	SCALE	
DESIGN. BY:	S.C.	03AUG07
DRAWN BY:	T.S.	03AUG07
CHECK. BY:	S.C.	03AUG07
APP. BY:		

LOCATION

eldoradogold

WARDROP Engineering Inc.

REFERENCE No.: 0551720100-DWG-B440036

TITLE

EFEMÇUKURU PROJECT

MINE PRODUCTION CUMULATIVE ANNUAL YEAR 10

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
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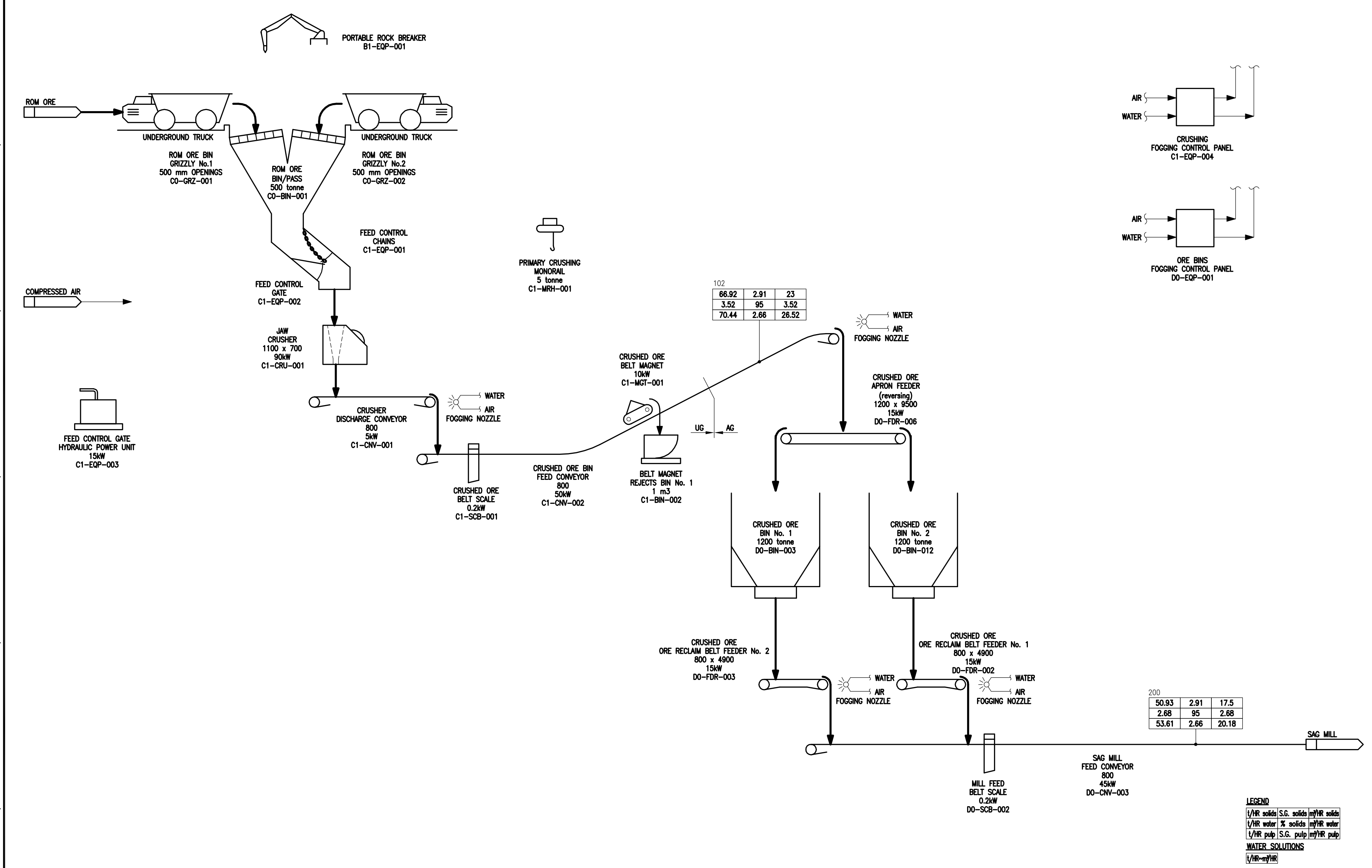

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G	1	ISSUED FOR FINAL CLIENT COMMENTS	03AUG07	DAS
F	1	ISSUED FOR FEASIBILITY STUDY	31JUL07	JD
E	1	ISSUED FOR FEASIBILITY STUDY	31OCT06	RAH
D	1	ISSUED FOR CLIENT COMMENT	11AUG06	PAZ
C	1	ISSUED FOR APPROVAL	04AUG06	JJM
B	1	ISSUED FOR APPROVAL		

CLIENT | PROJMGR. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

SECTION: PROCESS
SCALE: NONE
DATE
DESIGN. BY: GH JUN06
DRAWN BY: DH JUN06
CHECK. BY:
APP. BY:

WESTERN TURKEY

eldoradogold

WARDROP Engineering Inc.

TITLE
EFEMÇUKURU PROJECT
PRIMARY CRUSHING
STORAGE AND RECLAIM
PROCESS FLOWSHEET No. 1
1100 tpd

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
A009001.DWG	05517201.00	A0-09-001	H

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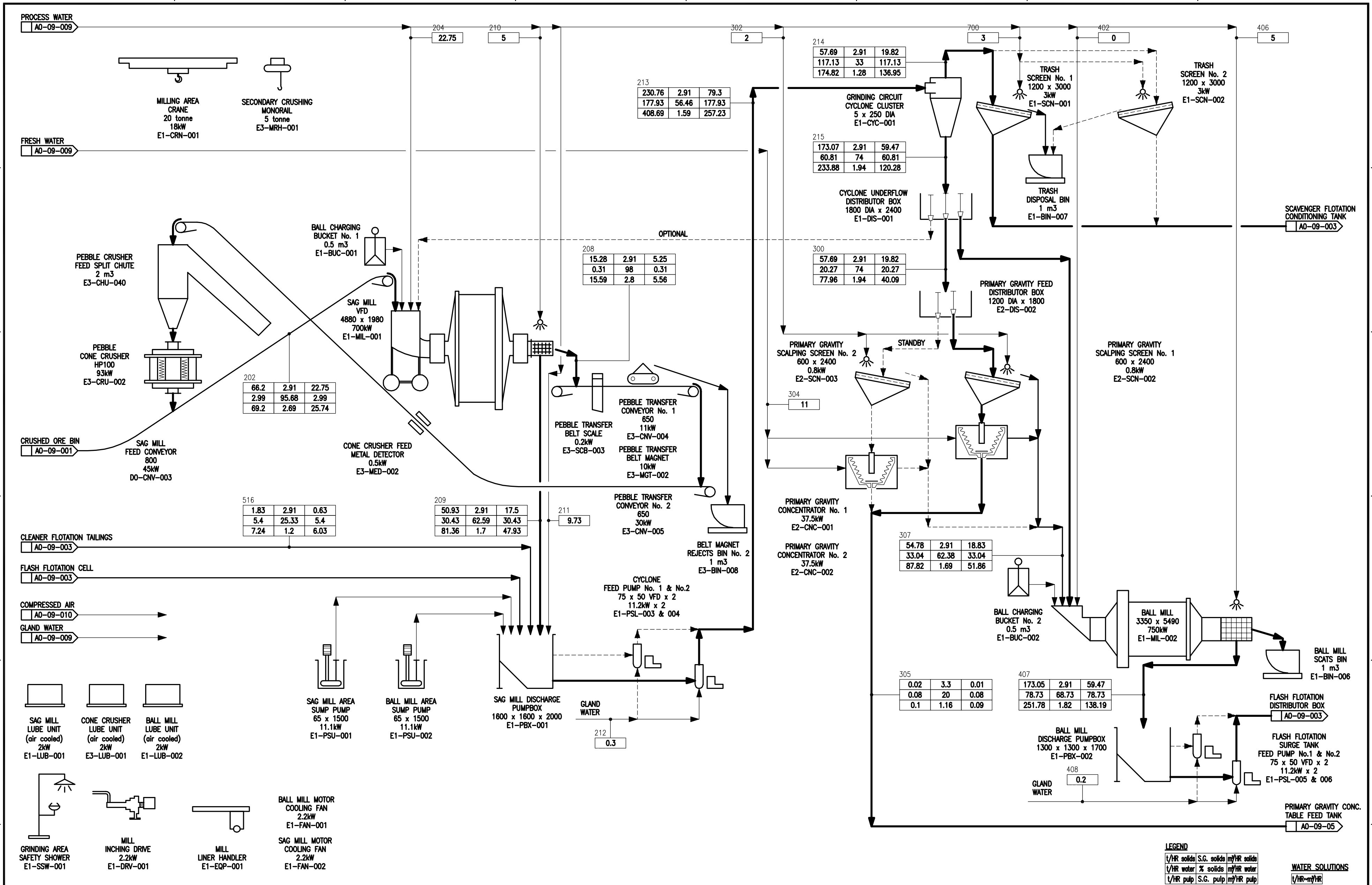

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B	1	ISSUED FOR APPROVAL		

CLIENT | PROJ.MAN. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

SECTION: PROCESS
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DRAWN BY: DH | JUN06
CHECK. BY:
APP. BY:

WESTERN TURKEY

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WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT

GRINDING AND CLASSIFICATION PROCESS FLOWSHEET No. 2 1100 tpd

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
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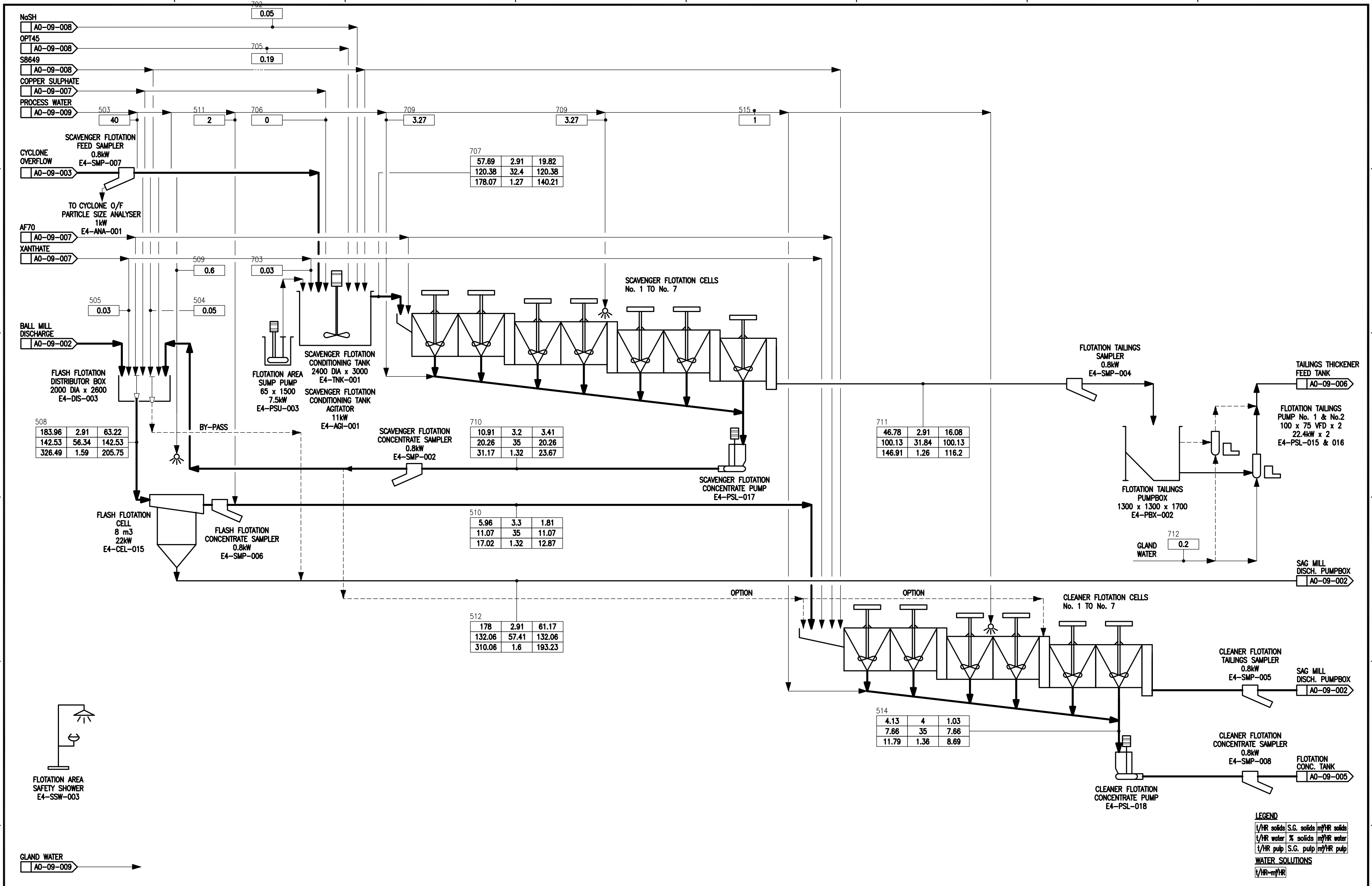

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C	1	ISSUED FOR APPROVAL	04AUG06	JJM
B	1	ISSUED FOR APPROVAL		

CLIENT PROJ.MAN. PROCESS ELECTR. INSTR. PIPING MECH. STRUCT. SERVICES ARCH. LAYOUT

SECTION: PROCESS
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DRAWN BY: DH | JUN06
CHECK. BY:
APP. BY:

WESTERN TURKEY

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WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT

FLOTATION CIRCUIT PROCESS FLOWSHEET No. 3 1100 tpd

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
A009004.DWG	05517201.00	A0-09-003	H

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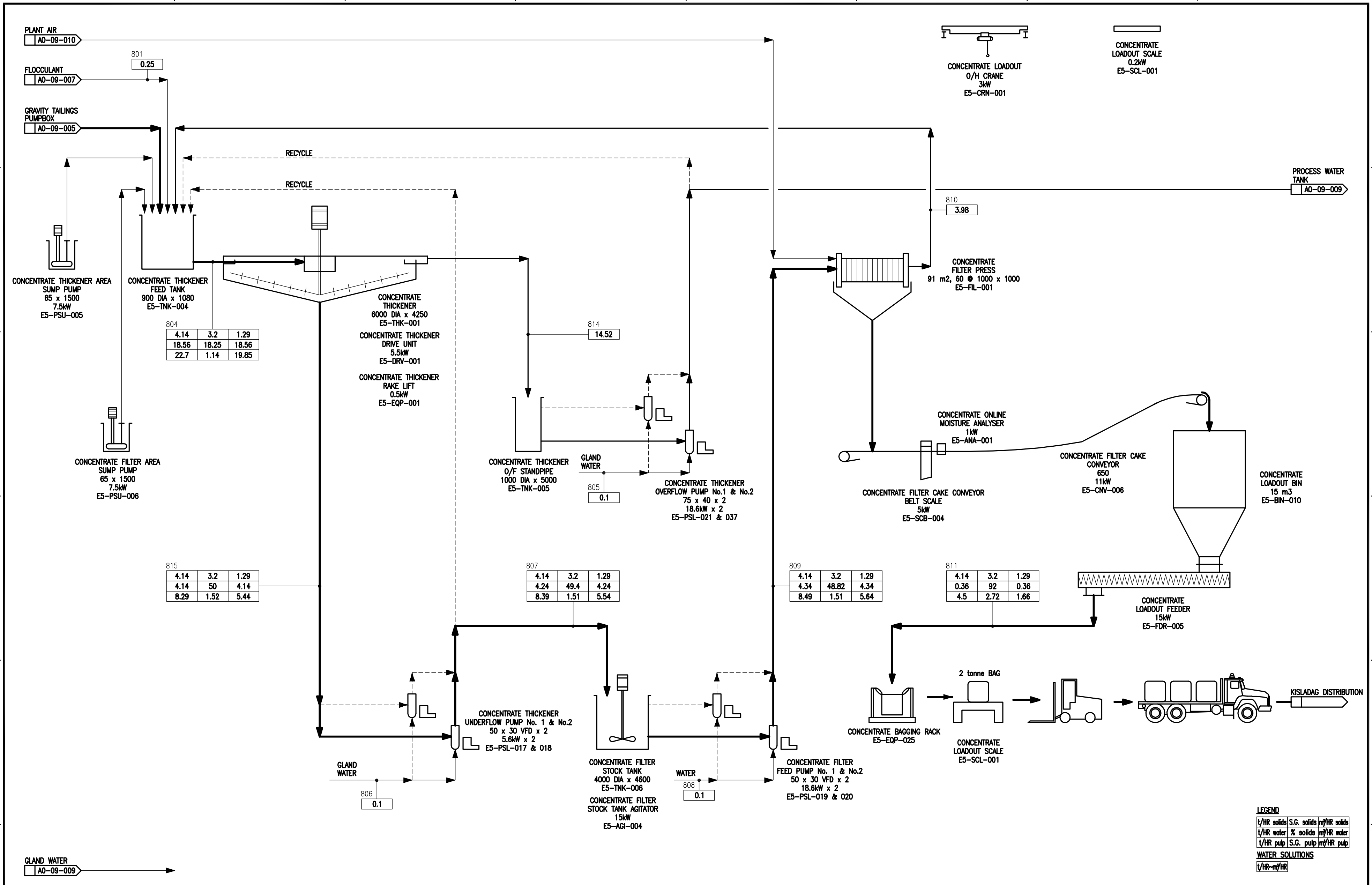

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C	1	ISSUED FOR APPROVAL	04AUG06	JJM
B	1	ISSUED FOR APPROVAL		

CLIENT | PROJ.MAN. | PROJ.ENG. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

SECTION:	PROCESS	
SCALE:	NONE	DATE
DESIGN. BY:	GH	JUN06
DRAWN BY:	DH	JUN06
CHECK. BY:		
APP. BY:		

WESTERN TURKEY

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WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT

CONCENTRATE DEWATERING PROCESS FLOWSHEET No. 4 1100 tpd

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
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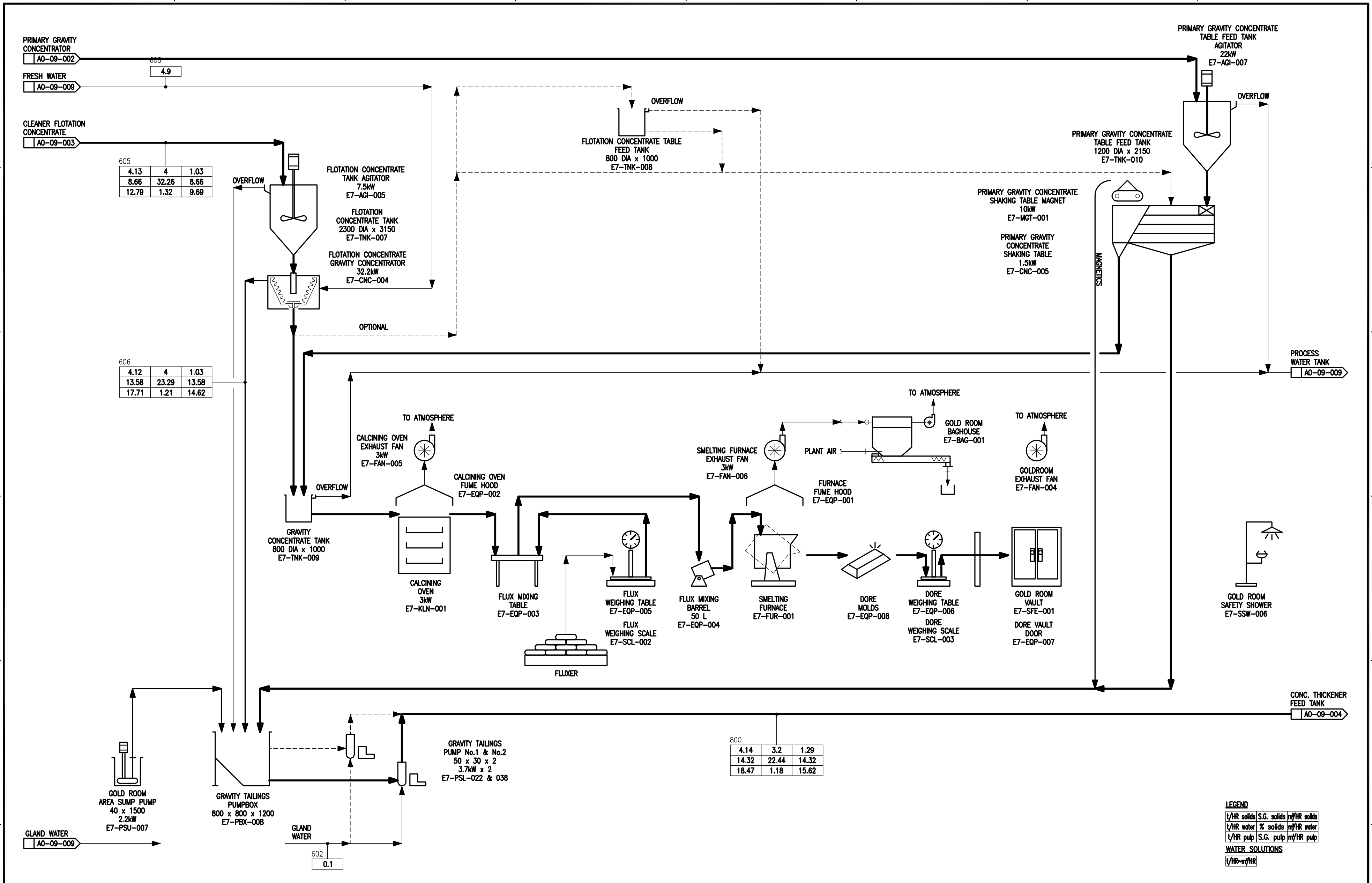

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B	1	ISSUED FOR APPROVAL		

CLIENT | PROJMAN. | PROJENG. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

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DESIGN. BY: GH — JUN06
DRAWN BY: DH — JUN06
CHECK. BY:
APP. BY:

WESTERN TURKEY

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WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT

GOLD ROOM
PROCESS FLOWSHEET No. 5
1100 tpd

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
A009005.DWG	05517201.00	A0-09-005	H

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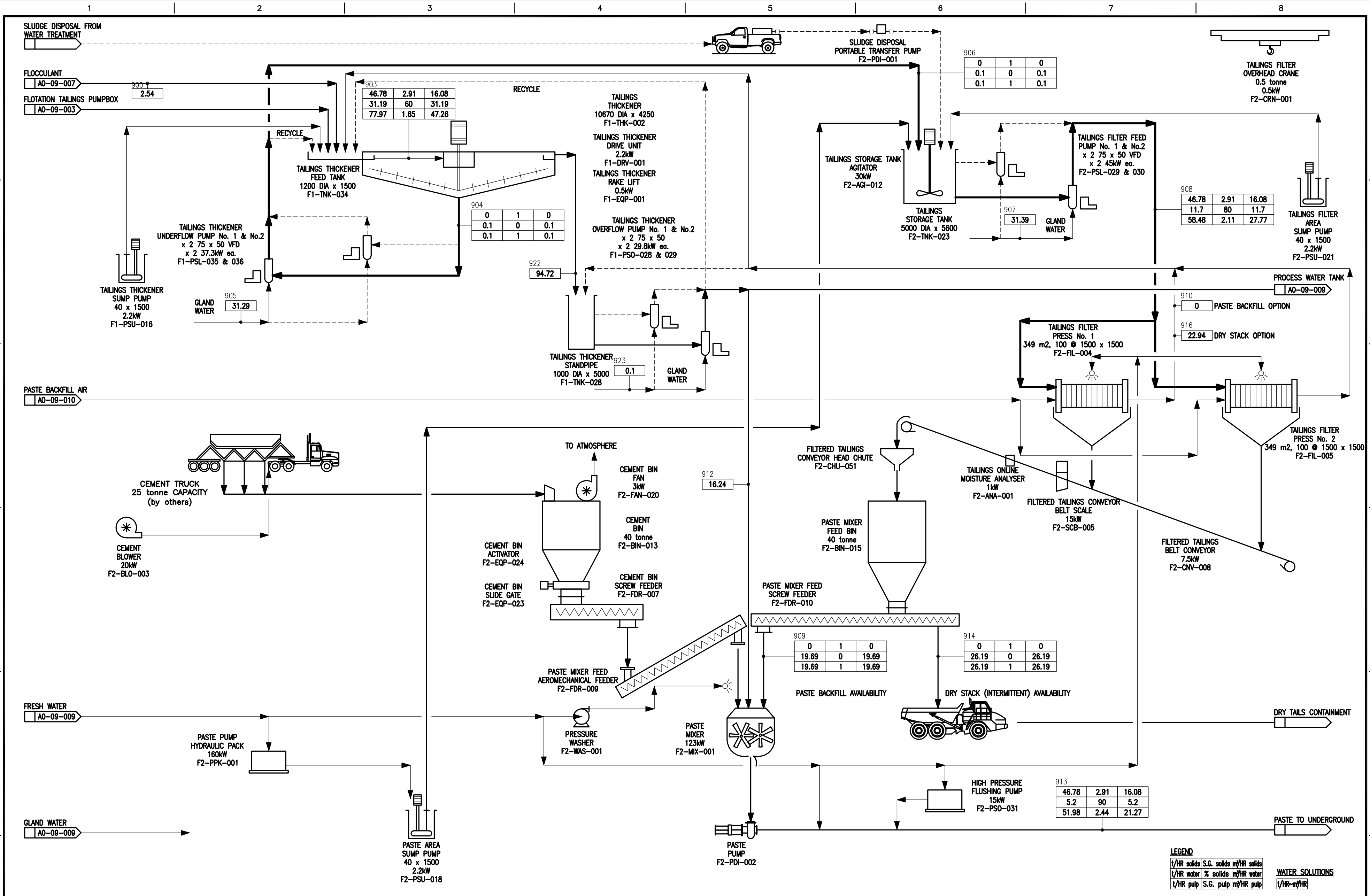

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DWG. NO.	REFERENCE DRAWINGS

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F	1	ISSUED FOR FEASIBILITY STUDY	31JUL07	JD
E	1	ISSUED FOR FEASIBILITY STUDY	31OCT06	RAH
D	1	ISSUED FOR CLIENT COMMENT	11AUG06	PAZ
C	1	ISSUED FOR APPROVAL	04AUG06	JJM
B	1	ISSUED FOR APPROVAL		

CLIENT PROJMG. PROCESS ELECTR. INSTR. PIPING MECH. STRUCT. SERVICES ARCH. LAYOUT

SECTION:	PROCESS	
SCALE:	NONE	DATE
DESIGN. BY:	GH	JUN06
DRAWN BY:	DH	JUN06
CHECK. BY:		
APP. BY:		

WESTERN TURKEY

eldoradogold

WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT
TAILINGS DISPOSAL
AND PASTE BACKFILL
PROCESS FLOWSHEET No.6
1100 tpd

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
A009006.DWG	05517201.00	A0-09-006	H

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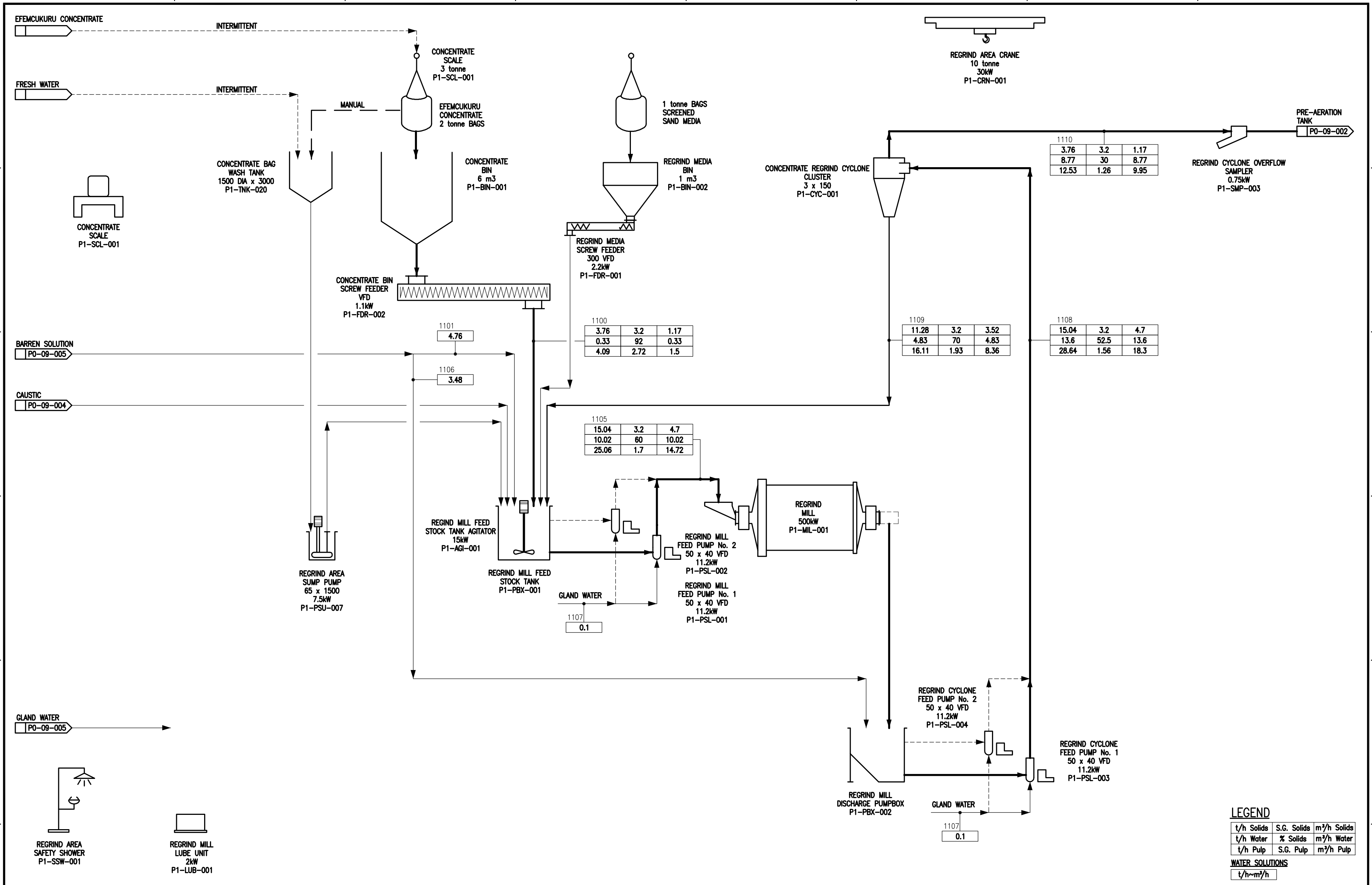

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B	1	ISSUED FOR CLIENT COMMENT	11AUG06	PAZ
A	1	ISSUED FOR INFORMATION	09AUG06	JJM

CLIENT, PROJ.MAN., PROCESS, ELECTR., INSTR., PIPING, MECH., STRUCT., SERVICES, ARCH., LAYOUT

SECTION: PROCESS	
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DESIGN. BY: GH	08AUG06
DRAWN BY: JJM	08AUG06
CHECK. BY:	
APP. BY:	

WESTERN TURKEY

eldoradogold

WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT
KISLADAG PLANT
CONCENTRATE REHANDLING AND REGRIND MILLING PROCESS FLOWSHEET No. 1

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
P009001.DWG	05517201.00	P0-09-001	G

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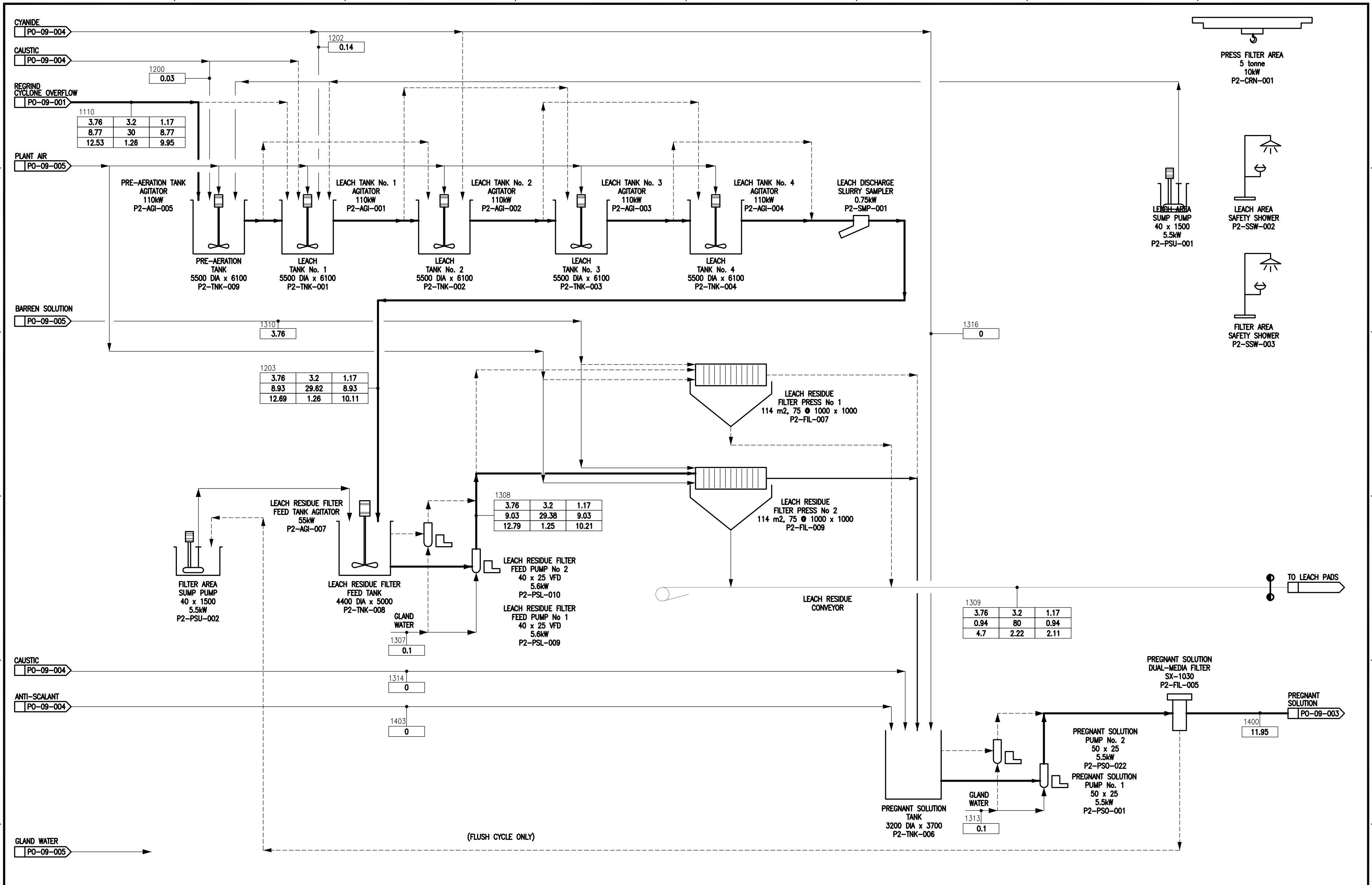

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DWG. NO.	REFERENCE DRAWINGS

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
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B	1	ISSUED FOR CLIENT COMMENT	11AUG06	PAZ
A	1	ISSUED FOR INFORMATION	09AUG06	JJM

CLIENT, PRELIMIN., PROCESS, ELECTR., INSTR., PIPING, MECH., STRUCT., SERVICES, ARCH., LAYOUT

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DRAWN BY: JJM — 08AUG06
CHECK. BY:
APP. BY:

WESTERN TURKEY

eldoradogold

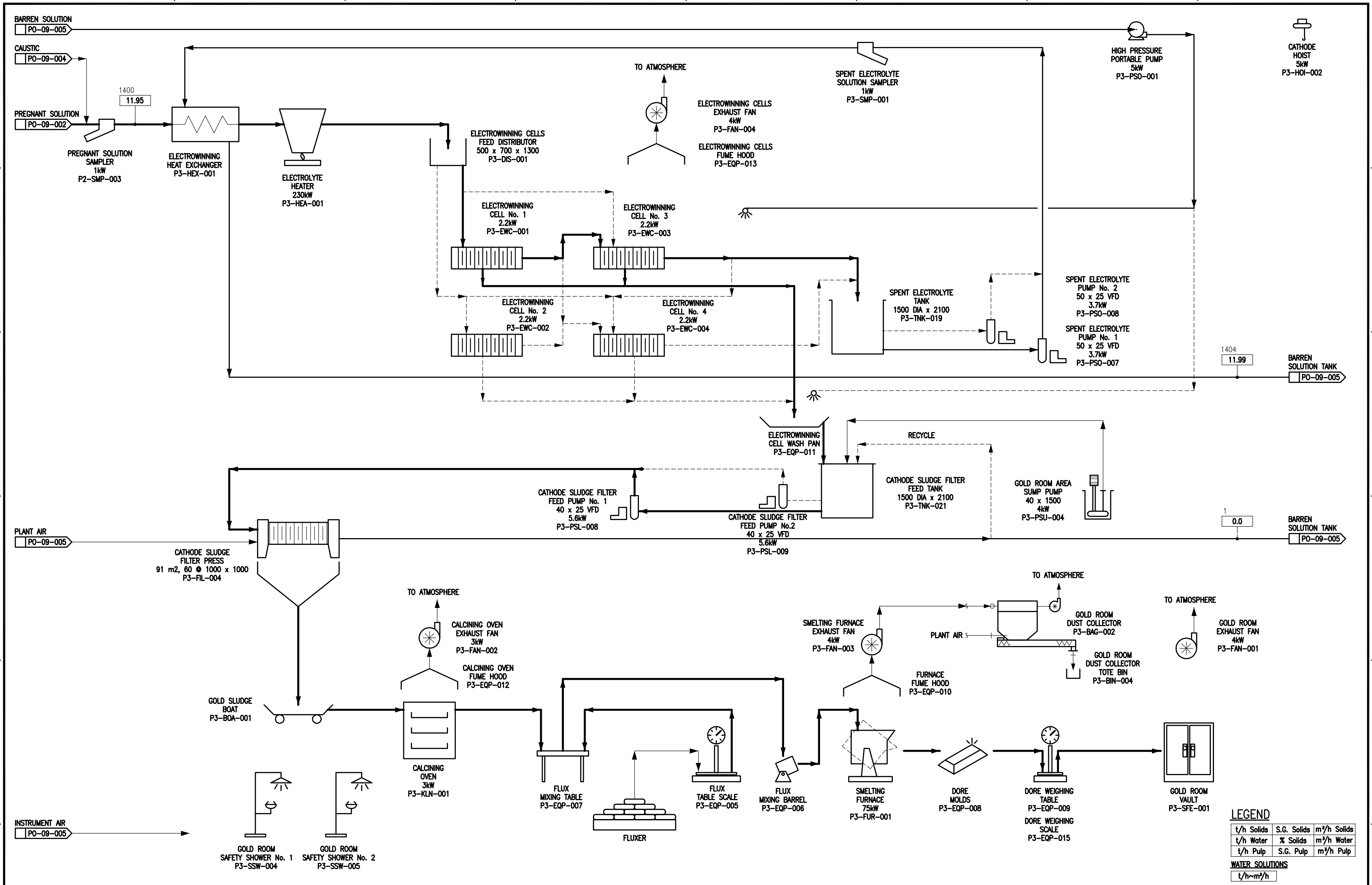
WARDROP Engineering Inc.

TITLE
EFEMÇUKURU PROJECT
KISLADAG PLANT
CYANIDE LEACHING AND
LEACH RESIDUE FILTRATION
PROCESS FLOWSHEET No. 2

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
P009002.DWG	05517201.00	P0-09-002	G

1201
07/08/15
A1

U:\A-G\ELDORADO GOLD CORPORATION – 5172\05517201.00 – EFEMCUKURU PROJECT (FEASIBILITY STUDY)\09-PROCESS\PFD\P009002.DWG



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DWG. NO.	REFERENCE DRAWINGS

REV. No.	ISSUE No.	DESCRIPTION	DATE	BY
G	1	ISSUED FOR FEASIBILITY STUDY	10AUG07	DAS
F	1	ISSUED FOR FINAL CLIENT COMMENTS	03AUG07	DAS
E	1	ISSUED FOR FEASIBILITY STUDY	06JULY07	JD
D	1	ISSUED FOR EIS UPDATE	14NOV06	PAZ
C	1	ISSUED FOR FEASIBILITY STUDY	31OCT06	RAH
B	1	ISSUED FOR CLIENT COMMENT	11AUG06	PAZ
A	1	ISSUED FOR INFORMATION	09AUG06	JJM

CLIENT | PRELIMIN. | PROCESS | ELECTR. | INSTR. | PIPING | MECH. | STRUCT. | SERVICES | ARCH. | LAYOUT

SECTION:	PROCESS	
SCALE:	NONE	DATE
DESIGN. BY:	GH	08AUG06
DRAWN BY:	JJM	08AUG06
CHECK. BY:		
APP. BY:		

WESTERN TURKEY

eldoradogold

WARDROP Engineering Inc.

TITLE

EFEMÇUKURU PROJECT

KISLADAG
GOLDROOM
PROCESS FLOWSHEET No.3

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
P009003.DWG	05517201.00	P0-09-003	G

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