Technical Report for the Kışladağ Gold Mine, Turkey

Effective Date: January 2010

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ABBREVIATIONS AND ACRONYMS

El Dorado Gold Corporation	Eldorado
Tüprag Metal Madencilik Sanayi Ve Ticaret Limited Sirketi	Tüprag
Nearest-neighbour	NN

UNITS OF MEASURE

Annum (year)	a
Centimetre	cm
Cubic metre	m^3
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Degree	°
Degrees Celsius	°C
Diameter	ø
Dry metric ton	dmt
Foot	ft
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
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
Kilovolt	kV
Kilovolt-ampere	kVA
Less than	<
Litre	L
Litres per minute	L/m
Litres per hour per square metre	L/h/m^2
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Micrometre (micron)	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million tonnes	Mt
Million tonnes per year (annum)	Mt/a
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Percent	%
Revolutions per minute	rpm

Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Short Ton (2,000 pounds)	st
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Weight/weight	w/w
Wet metric tonne	wmt
Year (annum)	a

SECTION 1 • SUMMARY

1.1 INTRODUCTION AND PROPERTY DESCRIPTION

Eldorado Gold Corporation (Eldorado), an international gold mining company based in Vancouver, British Columbia, owns and operates the Kişladağ Open Pit Gold Mine in Turkey through its wholly owned Turkish subsidiary, Tüprag Metal Madencilik Sanayi Ve Ticaret Limited Sirketi (Tüprag).

Eldorado has prepared this Technical Report of the Kişladağ Gold Mine to support a material change in mineral reserves and mineral resources relative to those quoted in previous Technical Reports (Technical Report Kişladağ Project Feasibility Study, March 2003 for the mineral reserves and 2003 Update of Resources, Kişladağ Project, Uşak, Turkey, September 2003 for the mineral resources).

Information and data for this report were obtained from Kişladağ Gold Mine. The work entailed review of pertinent geological, mining, process and metallurgical data in sufficient detail to support the preparation of this Technical Report. The Qualified Persons responsible for preparing this Technical Report are Stephen Juras, Ph.D., P.Geo. (geology and mineral resources), Richard Miller, P.Eng. (mineral reserves and mining), and Paul Skayman, MAusIMM (metallurgy and process). All are employees of Eldorado.

Construction of infrastructure, crushing and screening plant, the ADR plant, and the leach pad preparation started in 2004 and were completed in 2006. Commercial production was reported in July 2006. Current capacity is 10 Mt of ore per year. Work is underway to upgrade the facilities to 12 Mt. Current gold production is approximately 240,000 oz/annum.

The Kişladağ project land position consists of a single operating license, number IR 7302, with a total area of 15,717 ha. The fenced in project area covers approximately 6 km^2.

The Kişladağ gold mine is located in a rural area in west-central Turkey between the major centres of Izmir, lying 180 km to the west on the Aegean coast, and the capital city of Ankara, 350 km to the northeast. The Project site lies 35 km southwest of the city of Uşak (population 170,000) near the village of Gümüşkol . Access to the mine is via the 5.3 km long mine access road which connects to the highway between the towns of Ulubey and Esme. Electricity is provided via a 25 km long power line from a substation near Uşak. Water for the operation is pumped from wells near Ulubey to the site via a 13 km long pipeline.

The project site is located at an elevation of approximately 1,000 masl in gently rolling topography. The climate is temperate with an average annual rainfall of 425 mm, most of which occurs during the winter months.

1.2 GEOLOGY AND MINERALIZATION

The Kışladağ deposit consists of porphyry-style gold mineralization centred on a series of overlapping sub-volcanic intrusives of quartz-syenite to quartz-monzonite composition. It is located in one of the several mid- to late-tertiary volcanic complexes in western Turkey. In the Kişladağ region, these volcanics erupted onto a basement of schist and gneiss at the northeast margin of an uplifted terrain known as the Menderes Massif.

Within the deposit area, the main lithologies are quartz-trachyte to quartz-latite flows and volcaniclastic rocks intruded by a series of nested sub-volcanic porphyritic intrusives of alkalic affinities. Based on the intensity of alteration, mineralization and cross cutting relationships three mineralized (Intrusions #1, #2 and #2A) and one post mineral (Intrusion #3) intrusive bodies have been identified. West of the open pit the schistose basement has been intersected in a number of drill holes and outcrops of basement rocks have been mapped north and west of the leach pad area.

Intrusion #1 is the oldest, and generally, best mineralized porphyritic intrusive phase. It forms the core of the system, and is cross cut by the younger porphyritic intrusions. It forms a slightly elliptical body approximately 800 m across, which was exposed at the pre-mining surface. Intrusion #2A occurs in the southeast corner of the pit, where it intrudes the margin of Intrusion #1. It forms a circular stock 250 m to 300 m across. It appears to taper at depth. This unit carries economic gold grades, but is not as well mineralized as Intrusion #1. Intrusion #2 occurs as two separate semi-circular stocks, both approximately 150 m to 200 m across. One occurs in the center of the pit, cutting the core of Intrusion #1, and the second occurs on the northwestern margin of Intrusion #1. These stocks carry economic gold grades. Intrusion #3 is the youngest intrusive body at Kişladağ. It forms a semi-circular stock near the center of Intrusion #1 and extends into an elongate, steeply dipping to vertical dyke-like body to the west. The contacts of this unit with other rocks are generally well preserved, and the drop in gold grade is abrupt at the contacts.

The Kişladağ deposit lies in the eroded core of a Miocene stratovolcano complex, which has experienced relatively little structural modification. Lithologic contacts are primarily intrusive or depositional, with no mappable fault offsets documented. Despite the absence of major faults, the Kişladağ deposit and adjacent rocks contain a high density of low-displacement brittle fractures. Most of the observable fractures are best classified as joints and low-displacement faults and have continuity limited to a few metres to a few tens of metres.

Gold mineralization with traces of molybdenum, zinc, lead and copper encircles the late barren stock (Intrusion -#3). Higher-grade gold mineralization (> 1 g/t Au) is associated with Intrusion #1 and forms a horseshoe shaped zone around of Intrusion #3. The higher gold grades are associated with multiphase quartz sulphide stockwork and pervasive silicification. The mineralized zones dip outward in a bell-shaped body; sub parallel to the contact of the stock. Pyrite is the dominant sulphide mineral present with visual estimates averaging around 4% in the primary mineralized zone.

Oxidation has affected the Kışladağ deposit, being deeper on the uphill (southern) side of the deposit (from 30 m to 80 m deep) as compared to the downhill (northern) side of the deposit, (from 20 m and 50 m below surface). Limonite is the most abundant oxide mineral.

1.3 DRILLING, SAMPLING, AND ANALYSES

Several drilling campaigns by both core drilling and RC drilling took place from 1998 until 2009 for a total of some 76,000 m of which 50% was drilled in 2007 to 2009. It is this later drilling, mostly core holes, that provided information to enable upgrading of the mineral resource.

All diamond drilling in Kışladağ was done with wire line core rigs and mostly of HQ size. Drillers placed the core into wooden core boxes with each box holding about 4 m of HQ core. Geology and geotechnical data are collected from the core and core is photographed (wet) before sampling. SG measurements were done approximately every 5 m. Core recovery in the mineralized units was excellent, usually between 95% and 100%. The entire lengths of the diamond drill holes were sampled (sawn in half by diamond saw). The core library for the Kışladağ deposit is kept in core storage facilities on site.

Samples were prepared at Eldorado's in-country preparation facility at Çanakkale in north-western Turkey. A Standard Reference Material (SRM), a duplicate and a blank sample were inserted into the sample stream at every 8th sample. From there the sample pulps were shipped to the ALS Chemex 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 digestion and ICP analysis.

Monitoring of the quality control samples showed all data were in control throughout the preparation and analytical processes. In Eldorado's opinion, the QA/QC results demonstrate that the Kışladağ deposit assay database, particularly for new data obtained from 2007 to 2009, is sufficiently accurate and precise for resource estimation.

Since the start of production in 2006, the entire drill hole database was reviewed in detail. Checks were made to original assay certificates and survey data. Any discrepancies found were corrected and incorporated into the current resource database. Eldorado therefore concludes that the data supporting the Kışladağ resource work are sufficiently free of error to be adequate for estimation.

1.4 METALLURGICAL TESTWORK

Testwork completed by Kappes Cassiday and Associates has shown that the Kişladağ ore is amenable to heap leaching technology. A gold recovery of 80% to 81% is projected for the oxide ore. The primary ore has a higher sulphide content and gold recovery is projected to be 60%.

Kappes Cassiday and Associates completed a total of 45 heap leach column tests on Kişladağ ore. The tests showed that a fine crush size and a leach period of ninety days were required to maximize gold recovery. A crush size of 80% passing 6.3 mm was selected for both the oxide and primary ore. No cement agglomeration is required and the heap has been designed for stacking to a height of 60 m. The mine continues to carry out confirmatory metallurgical test work on a regular basis.

1.5 MINERAL RESOURCE

Eldorado used significant new data from the last three years of deep drilling campaigns and mining to update the geologic model. The latest resource and reserve work incorporated new lithology models and an alteration model, all constructed in 3D. Mineralized or grade shapes were also made. To constrain gold grade interpolation for the Kişladağ deposit, Eldorado created 3D mineralized envelopes, or shells. These were based on initial outlines derived by a method of Probability Assisted Constrained Kriging (PACK). A threshold value of 0.20 g/t Au was used.

Data analyses demonstrated that the lithologic units within the gold mineralized shell should be treated as separate domains. Grades for blocks within the respective domains will be estimated with a hard boundary between them. Also, the analyses showed that the distributions do not indicate a problem with extreme grades for gold.

Block model cell size was 20 m east x 20 m north x 10 m high. Modelling consisted of grade interpolation by ordinary kriging (OK) for all domains inside the mineralized shell. A two-pass approach was instituted for interpolation where the first pass required values from a minimum of two holes in order to interpolate a model grade value. The model was validated by visual inspection, checks for bias and for appropriate grade smoothing.

The mineral resources of the Kişladağ deposit were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization of the project satisfies sufficient criteria to be classified into Measured, Indicated, and Inferred mineral resource categories. The Kişladağ mineral resources as of 31 December 2009 are shown in Table 1-1. The Kişladağ mineral resource is reported at a 0.3 g/t Au cutoff grade and calculated to end of 2009 mining limits.

Table 1-1: Kişladağ Mineral Resources, as of 31 December 2009

Mineral Resource Category	Tonnes (t x '000)	Grade (Au g/t)	In Situ Gold (oz x '000)
Measured	82,904	0.93	2,490
Indicated	329,345	0.74	7,783
Measured+Indicated	412,249	0.78	10,273
Inferred	182,083	0.50	2,950

1.6 MINERAL RESERVE AND PIT DESIGN

The open pit was designed using Gemcom GEMS software based on a 10 m bench height with double benching for most pit walls. The design pit was based on the results of an optimization done using Whittle software. Berm width, face angle and bench stack heights varied by sector and rock quality. Spill berm widths varied from 6.7 m to 9.0 m and geotechnical berms of 12.5 m to 28.0 m widths were used to separate bench stacks and satisfy the overall slope angle limitations. Inter-ramp angles varied from 39° to 56°. The upper half of the pit (above 750 m elevation) has a double ramp network and the lower half of the pit is limited to a single ramp. Ramps were designed with a minimum width of 26.3 m for the two-way traffic ramps and a minimum width of 16.0 m for the single lane ramp used only for the bottom four benches. The pit exposure on surface ranges in elevation from 960 m to 1,080 m and the pit extends down to a bottom elevation of 500 m. The entire pit has a surface footprint of 125 ha.

The Kisaldag mineral reserves as stated in Table 1-2 are contained within the designed final pit as of 31 December 2009.

Table 1-2: Kışladağ Mineral Reserves, as of 31 December 2009

Mineral Reserve Category	Tonnes (T x '000)	Grade (Au g/t)	In Situ Gold (oz x '000)
Proven	68,230	1.05	2,312
Probable	149,240	0.94	4,504
Proven + Probable	217,470	0.97	6,816

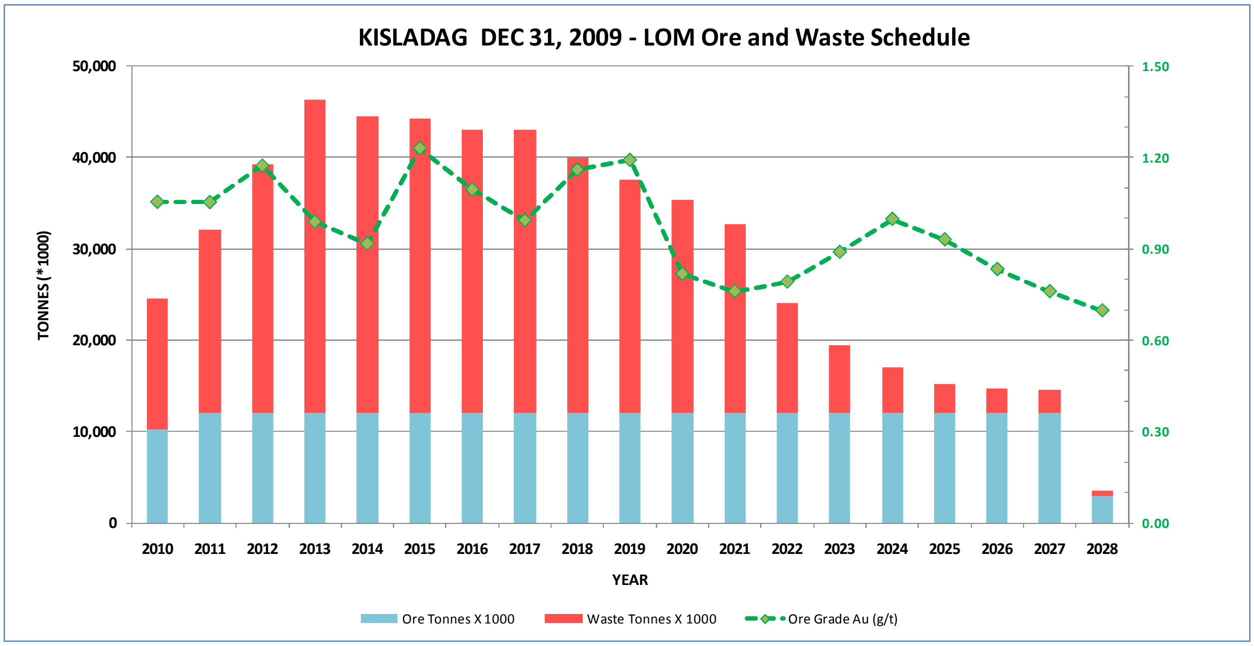
The mineral reserves were estimated using a cutoff grade of 0.35 g/t Au for oxide ore and 0.50 g/t Au for sulphide ore. Gold price was US$825/oz. Ore loss and waste dilution have been accounted for in the block model and therefore do not affect the reserve. This had been substantiated by production to model reconciliation over a period of almost four years of operation.

1.7 MINING OPERATION

The mine operation is a standard drill and blast truck and shovel open pit operation utilizing Caterpillar model 785C 150-ton trucks and Hitachi EX-3600 hydraulic shovels. The mine operates 24 hours a day seven days a week, which is also the schedule for the crusher.

Figure 1-1 shows the LOM plan, which foresees a production rate of 12 Mt of ore as of 2011.

Figure 1-1: Life-of-Mine Ore and Waste Schedule



1.8 PROCESSING

The Kişladag ore is processed in a standard heap leach facility containing a three stage crushing plant, an overland conveyor to the heap leach pad, mobile conveyors and a stacker for placing the ore and a carbon adsorption facility (ADR plant) for recovering the gold. The carbon is treated on site in a refinery and the final product is a gold doré bar.

The product from the crushing and screening circuit is transported to the heap leach pad by two stage overland conveyors. A series of portable conveyors and a radial stacker are used to place the ore onto the pad. Depending on the place and geometry of the pads, advance stacking is applied as well. The designed stack height is 60 m placed in 10 m high lifts.

The heap leach pad is a permanent facility employing a two part liner system of a compacted layer of low permeability clay soil, with a 2 mm thick HPDE/LLDPE synthetic liner.

The water management system has been designed to accommodate a 100 year, 24 hour storm event

Gold from the heap solutions is loaded onto activated carbon in the ADR plant. The gold is recovered from the carbon in a standard Zadra process consisting of pressure stripping, electro winning and smelting. The final product is a gold doré bar suitable for final processing to 99.999% purity in domestic or offshore refineries

1.9 ENVIRONMENTAL CONSIDERATIONS AND CLOSURE PLAN

The Kişladağ Project EIA study was completed in January 2003 and submitted to the Turkish Authorities at the Ministry of Forest and Environment. An Environmental Positive Certificate for the project was subsequently obtained in June 2003. The EIA document presents a number of potential socio-economic effects associated with the development of the Project, and defines a number of measures to avoid or minimize potential impacts.

An environmental monitoring plan has been developed to address the potential impacts of the mining operation. This plan was put in place prior to pre-production starting in 2005 and has been maintained throughout the production phase. The scope of the monitoring program within this plan includes elements of air quality, surface water and ground water monitoring. Data collected during the monitoring program is reported to the relevant government agencies on a monthly and annual basis. Additional issues addressed in the imbedded Environmental Management Plan include noise and blast vibration monitoring as well as waste and hazardous waste storage and disposal.

A Preliminary Closure Plan, based on the closure strategy presented in the EIA report, was prepared for the operation in August 2007 by a third party consultant. This plan is updated and revised regularly during operation of the mine, culminating in the establishment of a Final Closure Plan prior to decommissioning. A mine Reclamation Plan, based on EIA report and Preliminary Closure Plan has been prepared and submitted to the relevant Government Authority in August 2008 as part of regulatory requirements. Annual base improvements shall be reported to Government Authority every year.

1.10 OPERATING AND CAPITAL COSTS

1.10.1 OPERATING COSTS

The operating costs per unit of production have been relatively constant since the start of the mine life. The cost for processing and mine support are expected to remain constant for the remainder of the mine life, except when they are changed due to inputs that affect the entire gold mining industry, including, but not limited to, changes in, fuel costs, reagent costs, exchange rates, labour costs and inflation. The unit costs for mining are expected to increase as the pit deepens and they are also affected by the previously listed inputs.

1.10.2 CAPITAL COSTS

In 2009, the mine initiated a study to expand the current processing rate to 12 Mt of ore per year. The capital cost of this expansion is included in the 2010 budget, and sufficient capital has been budgeted to include an upgrade of the existing crushing and screening plant as well as the ADR plant. The project is expected to be completed by 2011.

The initial capital expenditure has already been paid back. The project is generating positive cash flow and the expansion capital proposed for 2010 will be paid back within the first year of the project completion.

1.11 CONCLUSIONS

The Kişladağ Gold Mine has been in operation for nearly four years. The mine has been successful in its implementation of construction and operations plans as described in the previous NI 43-101 technical report prepared by Hatch in 2003. Production tonnages and gold produced are matching forecasts. Eldorado believes that the mine will continue to perform as well in the future as it has during the first years of operations.

SECTION 2 • INTRODUCTION AND TERMS OF REFERENCE

Eldorado Gold Corporation (Eldorado), an international gold mining company based in Vancouver, British Columbia, owns and operates the Kişladağ Gold Mine in Turkey through its wholly owned Turkish subsidiary, Tuprag Metal Madencilik Sanayi Ve Ticaret Limited Sirketi (Tuprag). Eldorado has prepared this Technical Report of the Kişladağ Gold Mine to support a material change in mineral reserves and mineral resources relative to those quoted in previous Technical Reports (Technical Report Kişladağ Project Feasibility Study, March 2003 for the mineral reserves and 2003 Update of Resources, Kişladağ Project, Uşak, Turkey, September 2003 for the mineral resources).

Information and data for this report were obtained from Kişladağ Gold Mine. The work entailed review of pertinent geological, mining, process and metallurgical data in sufficient detail to support the preparation of this Technical Report.

The Qualified Persons responsible for preparing this Technical Report as defined in National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects and in compliance with 43-101F1 (the "Technical Report") are Stephen Juras, Ph.D., P.Geo., Richard Miller, P.Eng, and Paul Skayman, MAusIMM. All are employees of Eldorado.

Dr. Juras, Director, Technical Services for the Company, was responsible for the preparation of the sections in this report that concerned geological information, sample preparation and analyses and mineral resource estimation. He most recently visited the Kişladağ Gold Mine on November 14 to 16, 2009.

Mr. Miller, Manager, Mining for the Company, was responsible for the preparation of the sections in this report that dealt with mineral reserves estimation, mine operations and related costs. He most recently visited the Kişladağ Gold Mine on September 15 to 17, 2009.

Mr. Skayman, Vice President Operations for the Company, was responsible for the preparation of the sections in this report that dealt with metallurgy and process operations and related costs. He most recently visited the Kişladağ Gold Mine on September 15 to 17, 2009.

Drill programs executed in 2008 and 2009 has generated results that allow the company to report a substantial increase of the resource and reserve at the mine.

This document presents a summary of the current and forecast operation at the mine.

Turkish names frequently include Turkish characters. In some cases, the names may have been written using a standard US keyboard. The following table is provided as a cross reference list.

Kisladag ....Kışladağ

Kisla...Kışla

Usak ....Uşak

Tuprag ....Tüprag

Gokgoz Tepe...........................Gökgöz Tepe

Canakkale ....Çanakkale

Gumuskol ....Gümüşkol

Sogutlu .... Söğütlü

Katrancilar ....Katrancılar

Karapinar.................................Karapınar

Esme ....Eşme

Sayacik....................................Sayacık

Dag..Dağ

TEDAS ....Tedaş

SECTION 3 • RELIANCE ON OTHER EXPERTS

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

Encon Environmental Consultancy Company

Information from their report titled Kışladağ Gold Mine Environmental Implementation Plan, dated August 2004 was used in Section 18 and 19.4 of this document.

The Mines Group Inc.

Information from their report titled Kışladag Gold Mine Preliminary Closure Plan For the Uneconomic Rock Storage Area And Heap Leach and Other Facilities, August 2007, was used in Section 18 of this document.

Kappes Cassiday Associates

Information from numerous Metallurgical testwork reports was used in Section 16 of this document.

Norwest Corporation

Numerous Technical Reports and Memos on Waste Rock Management were used in Section 19.1 of this document.

HTA Geotechnical Consulting

Information from their report titled Updated Mining Geotechnical Study for Open Pit Slope Optimisation Kışladağ Gold Mine, Turkey, dated November 2009, was used in Sections 17.2 and 19.1 of this document.

Mines Group –

Numerous Technical Reports and Memos on Leach Pad Design were used in Section 19.2 of this document.

Professor Hasan Yazicigil (Middle East Technical University, Ankara)

Information from their report titled Determination of the Sites with potential to Supply Water to the Gümüşkol Mine Project, dated July 2000, was used in Section 19 of this document.

SECTION 4 • PROPERTY DESCRIPTION AND LOCATION

4.1 INTRODUCTION

Kişladağ gold mine is an operating open pit gold mine since 2005 with surface facilities consisting of heap leach pads and ADR plant, crushing plant and ancillary buildings.

4.2 PROPERTY LOCATION

Kişladağ is located in west-central Turkey between the major centres of Izmir, lying 180 km to the west on the Aegean coast, and the capital city of Ankara, 350 km to the northeast. The Project site lies 35 km southwest of the city of Uşak (population 170,000) near the village of Gümüşkol.

Approximate Project co-ordinates are:

UTM..06 87500E 42 61600N

UTM Zone 35S

Map Sheet................................Uşak-L22 (1:100.000 scale)

Longitude: ....29° 08' 58" E

Latitude....................................38° 28' 56" N

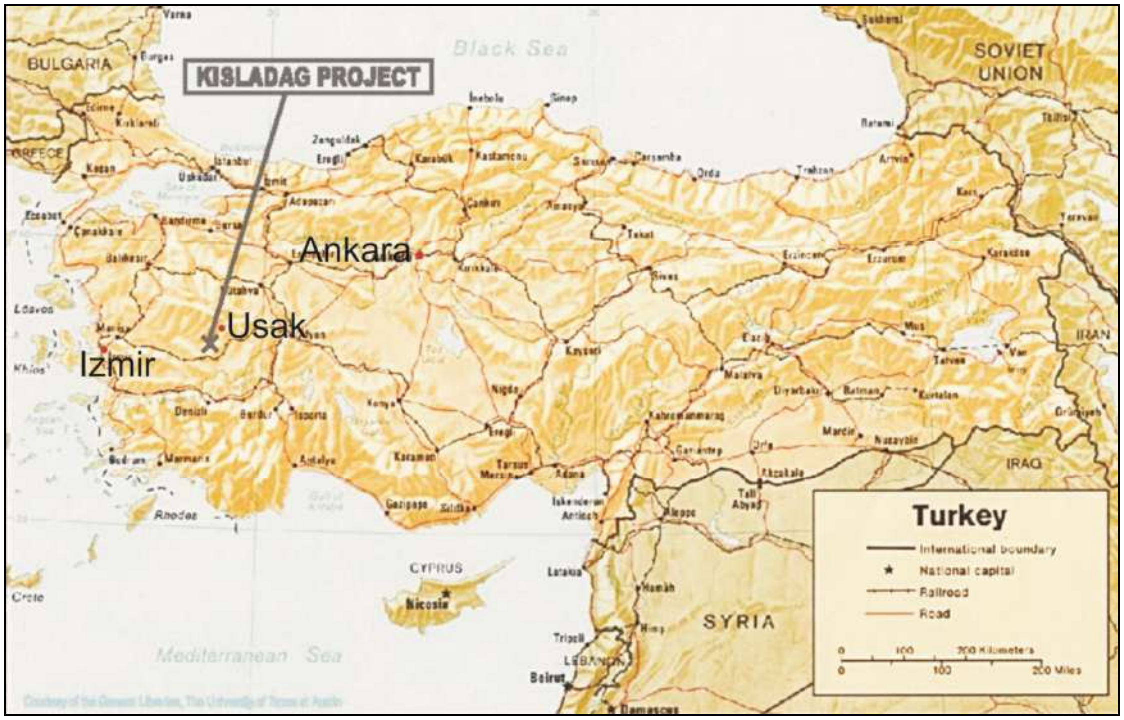
Land use within the concession area falls into three categories: inhabited (villages and dwellings) agricultural land (cropping and grazing) and barren lands (not suitable for agriculture). The non-irrigated farmlands of low productivity make up about 27% of the area. The remaining area is barren and is mainly used for communal grazing. An attempt to forest the area was initiated 20 years ago, however only approximately 24% of the concession area is currently planted with pine and cedar. Development of the plantation has been restricted by the poor soil conditions.

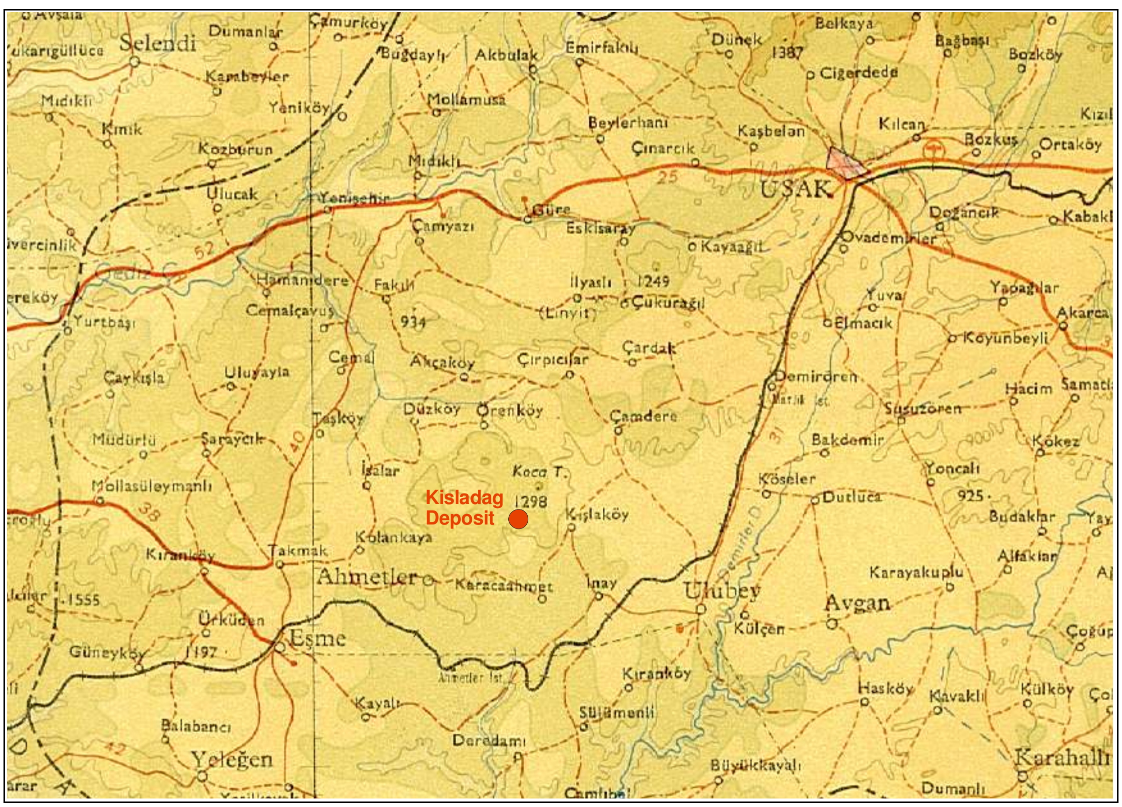
There are no permanent water bodies in the area and water supply is limited to ephemeral streams and shallow seasonal stock ponds. The geology of the area is dominated by volcanics with generally poor aquifer characteristics. The villages in the area are supplied with potable water piped from a source located approximately 5 km to the west of Kişla village.

The soil depth in the process plant area is less than 2 m deep and subsurface conditions appear to be weathered bedrock suitable for economical spread footing foundation design.

The Kişladağ site is located approximately 250 km south of the major North Anatolian Fault zone and is located between the first and second degree seismic zones as defined in the Turkish code. This is equivalent to an earthquake Zone 4 in the American Uniform Building Code. The effective ground acceleration coefficient is 0.4 g.

Figure 4-1: Location Map showing Western Turkey and Blow up of Project Area

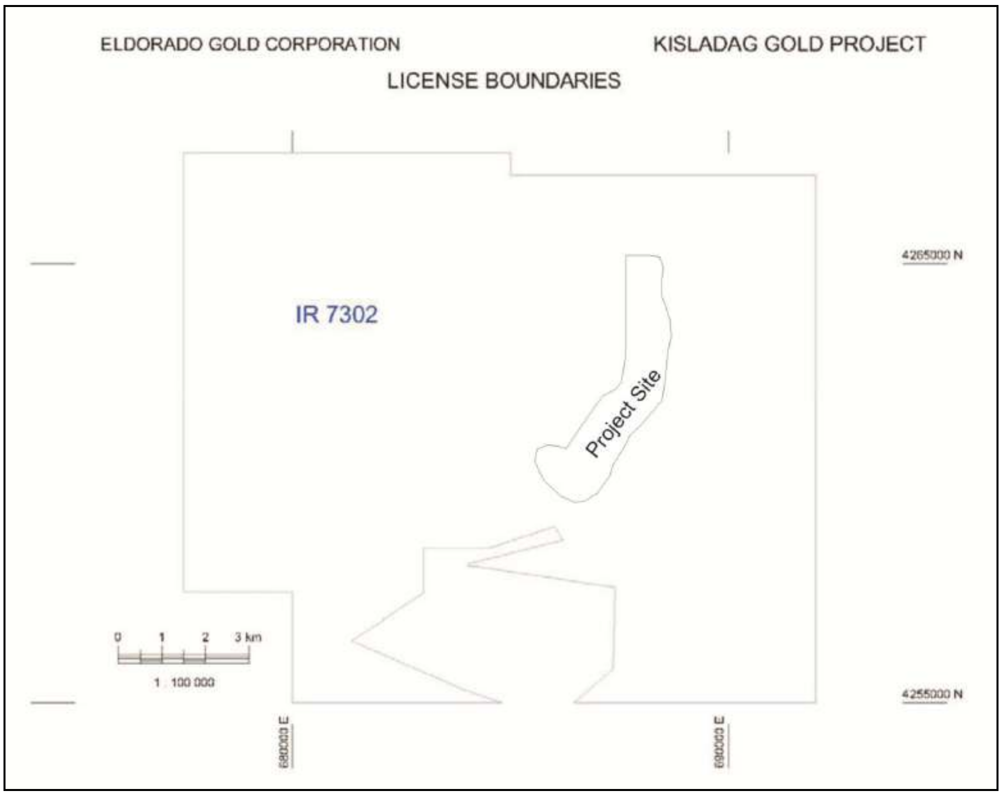




4.3 LAND TENURE

The Kişladağ Project land position consists of a single operating license, number IR 7302, with a total area of 15,717 ha. According to Turkish mining law, Tüprag retains the right to explore and develop any mineral resources contained within the license area provided fees and taxes are maintained. The license was issued on 9 April 2003 and is currently set to expire on 9 April 2013. Duration of mining license can be extended if the production is still going on at the end of license period. The license boundaries are defined in UTM coordinates and are shown in Figure 4-2.

Figure 4-2: Kişladağ Land Position



4.4 ROYALTIES

A royalty payment to the Government of Turkey is calculated on an annual basis at 1% of the mining cost times a factor of 130%, which is effectively 1.3% of the mining cost. No other royalties apply to the property.

4.5 ENVIRONMENTAL LIABILITIES

No environmental liabilities have been assumed with the project.

4.6 PERMITS AND AGREEMENTS

The process of obtaining the necessary permits for a mining operation in Turkey is similar to that in other developed countries. The following describes the process in place at the time of permitting Kışladağ and the timing involved in the process. This process has subsequently changed through government legislation.

The first permit required to initiate an industrial project in Turkey is the Site Selection Permit. This permit is intended to establish the legal right to the land for the concession owner to proceed with development of an industrial or commercial project. A review of project scope is carried out by a number of local provincial and federal government agencies to determine if conflicting land use issues exist in the Project area, or may develop in the future. Approval is obtained from each agency prior to issuing the Site Selection Permit. Environmental baseline and impact studies follow receipt of the Site Selection Permit. The Environmental Impact Assessment study marks the second major step in the permitting process, culminating in the issuance of an Environmental Positive Certificate, which precedes application for the remaining technical permits. While certain time constraints apply to different permits applications, there is no overall timeline that defines the total duration of the permitting process.

Listed below is a summary of some of the milestones for the Kişladağ Project and its permitting process until start of commercial production.

1997 .... Identification of ore body

1998-2002 .... Completion of feasibility stage drilling programs

1999 .... Approval of the site selection permit

2000-2003 .... Completion of feasibility study

2003 .... Approval of Environmental Positive Certificate and Mine Operation Permit

2003 .... Granting of Site Selection Permit

2004 .... Zoning Plan and Construction Permit approved

2005 .... Construction started

2006-April .... Commissioning and leaching started

2006-May .... First doré poured

2006-July .... Commercial production.

Table 4-1 lists key permits that have been obtained when they were issued and which government authority did the issuing.

Table 4-1: Permits

Name of Permit	Issue Date	Issuer
Site Selection Permit	1999	Governorship of Uşak
Mining License	2003-04-09	Ministry of Energy and Natural Resources
EIA Permit	2003-06-27	Ministry of Environment
Site Selection Permit	2003-12-12	Ministry of Health
Pre-Emission Permit	2004-03-03	Directorship of Environment of Uşak
Forestry Permit	2004-06-30	Directorship of Forestry
Zoning Plan and Construction Permit	2004-08-03	Governorship of Uşak
Establishment Permit	2005-12-19	Ministry of Labour
Operation Permit	2006-04-04	Ministry of Labour
Trial Permit	2006-04-06	Provincial Administration of Uşak
Discharge Permit	2007-03-28	Directorship of Environment of Uşak
Emission Permit	2007-03-28	Directorship of Environment of Uşak
Opening Permit	2007-04-06	Provincial Administration of Uşak
Opening Permit [1]	2008-03-06	Provincial Administration of Uşak

Notes: The Discharge Permit needs to be renewed every five years from the date of issue. The Emission Permit will be renewed 28 March 2010 and then renewed every two years thereafter.

[1] The second opening permit, 6 March 2008, was granted after the 2007 injunction against the project had been lifted, see section 6.0 History.

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

5.1 ACCESSIBILITY

Kişladağ is located in west-central Turkey between the major centres of Izmir, lying 180 km to the west on the Aegean coast, and the capital city of Ankara, 350 km to the northeast. The Project site lies 35 km southwest of the city of Uşak (population 170,000) near the village of Gümüşkol.

The major cities of Izmir and Ankara are served by international airlines, and there are regular internal flights by Turkish airlines to most major centres in the country. There is also an airport at Uşak for internal flights.

The highway from Ankara, through Uşak to Izmir is a major national trucking route and is in good condition. From the port city of Izmir, Kişladağ can be reached via an all-weather, paved road some 246 km distant. The preferred access route from Izmir is to travel east to Uşak on the Salihli/Uşak highway, and then south for approximately 50 km on a paved road to Ulubey/Esme. A new 5.3 km long, mine access road constructed in 2004, is connecting the site to the Ulubey/Esme road. The national rail system is accessible to the project.

Figure 5-1: Road Map

SECTION 6 • HISTORY

Eldorado acquired the Kişladağ property from Gencor Limited of South Africa in July 1996, as part of their portfolio of assets in Brazil and Turkey. The original prospect was identified in 1989 from satellite image interpretations, and confirmed through ground reconnaissance and geochemical sampling programs.

Since 1996, Eldorado's exploration activities at Kişladağ have focused primarily on the zone known locally as Gökgöz Tepe using stream sediment sampling, geochemical soil sampling and an Induced Polarization (IP) geophysical survey. On the basis of this work, a gold anomaly was identified along the north slope of Gökgöz Tepe extending approximately 1,200 m on strike by 600 m wide. This work was followed in 1997 by 2,745 m of trench sampling, and 1,638 m of percussion drilling.

In 1998, a six hole HQ diamond drilling program (1,059 m) probing the main anomaly target followed the gold mineralization to depths of greater than 250 m and effectively confirmed the potential for a low grade bulk tonnage gold deposit, and in 1999 an additional 5,000 m of HQ core drilling and 1,600 m of trenching extended the strike length and depth of the deposit. Based on the trenching, percussion drilling and core drilling data available to that date, Micon International and Eldorado identified a Measured and Indicated resource of 42.8 Mt of 1.49 g/t, plus an Inferred resource of 31.1 Mt at 1.35 g/t (all based on a 0.8 g/t cutoff grade).

In 2000, a reverse circulation (RC) drill program totalling 7,580 m (and 577 m of DDH) led to a revised resource estimate and a significant increase in the deposit's contained metal content. That year, Micon International reported a Measured and Indicated Resource of 125.97 Mt for the deposit at an average grade of 1.20 g/t Au. This is equivalent to 4.85 Moz of contained gold in oxides and primary ore (using a cutoff grade of 0.4 g/t Au).

In 2002, a combined total of 10,582 m (RC, DDH and Percussion) was completed.

In 2003 to 2004, the drilling campaigns continued and a total of 8,499 m (RC and DDH) were drilled including 1,384 m for open pit geotechnical purposes. These geotechnical holes were also assayed and results were used for resource-reserve calculations later on.

Metallurgical test work initiated during 1999 and 2000 by Eldorado indicated that the ore would be amenable to heap leaching, and in 1999 Eldorado was granted a Site Selection Permit by the Turkish authorities for a gold mining operation at the Kişladağ Project site. Early receipt of this permit was made possible by the high level of support the Project has received from within the Uşak province as well as at the central government level.

In 2001 Eldorado commissioned a Prefeasibility Study by Kilborn Engineering Pacific Limited (Kilborn), based on the concept of recovering gold by heap leaching. This study considered an operation to treat 3.4 Mt/a of material based on an owner operated mining fleet and a three stage crushing circuit generating a final crush size of 100% minus 8 mm. The objective of this approach was to minimize capital expenditure in the early years and allow for expansion to develop the total resource at a later date. Initial capital cost was estimated to be US$47.4 million with a cash operating cost estimated at US$154/oz and an average annual gold production of 103,600 troy ounces.

Subsequent to issuing the Prefeasibility Study, Kilborn was asked to review the Project conditions in light of devaluation of the Turkish currency and to incorporate the option of contracting the mining operation and utilising used crushing equipment. An Addendum to the Prefeasibility Study was issued in December 2001, presenting a revised initial capital cost estimate of US$29.6 million and a cash operating cost estimate of US$149/oz.

In April 2003, a bankable feasibility study was completed by Hatch. The study envisaged a staged increase in production over a five year period from an initially production target of 5 Mt/a increasing to 10 Mt/a in Year 5. An optimization study was subsequently completed in July 2003, which generated a total life of mine capital cost estimate for the project of approximately US$138.5 million.

The two pre-existing NI43-101 Technical Reports were composed at this time. Proven and Probable mineral reserves equal to 115 million tonnes at a grade of 1.23 g/t Au (oxide cut-off =0.35 g/t Au; sulphide cut-off 0.50 g/t Au; gold price – US 325/oz) were declared and supported in Hatch (2003). Measured and Indicated mineral resources of 215 million tonnes grading 1.04 g/t Au (0.40 g/t Au cut-off) were declared and supported in Micon (2003).

First construction work on the project started in 2004 with access road construction and continued through 2005 to 2006 with leach pad area preparation, construction of crushing, screening and ADR plants and ancillary buildings. Open pit production started in 2005. All construction work for the first phase was completed in early 2006 and commercial production was declared in July 2006.

Expansion of the crushing-screening plant to 10 Mt/a followed commercial production with completion of the additional capacity in April 2007.

The operation was shut down in August 2007 after the Environmental Positive Certificate for Kişladağ had been challenged by a third party. The injunction was lifted in February 2008 and the operation resumed in March 2008.

Owner operation in the pit replaced the mine contractor in September 2008.

SECTION 7 • GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Kişladağ deposit is located in one of the several mid- to late-tertiary volcanic complexes in western Turkey related to subduction along the Hellenic Trench that lies to the southwest of Turkey. In the Kişladağ region, these volcanics erupted onto a basement of schist and gneiss at the northeast margin of an uplifted terrain known as the Menderes Massif. The lithologies of the Project region have been described by the Middle East Technical University Geological Engineering Department based in Ankara (Yazicigil et al., 2000), see Figure 7-1.

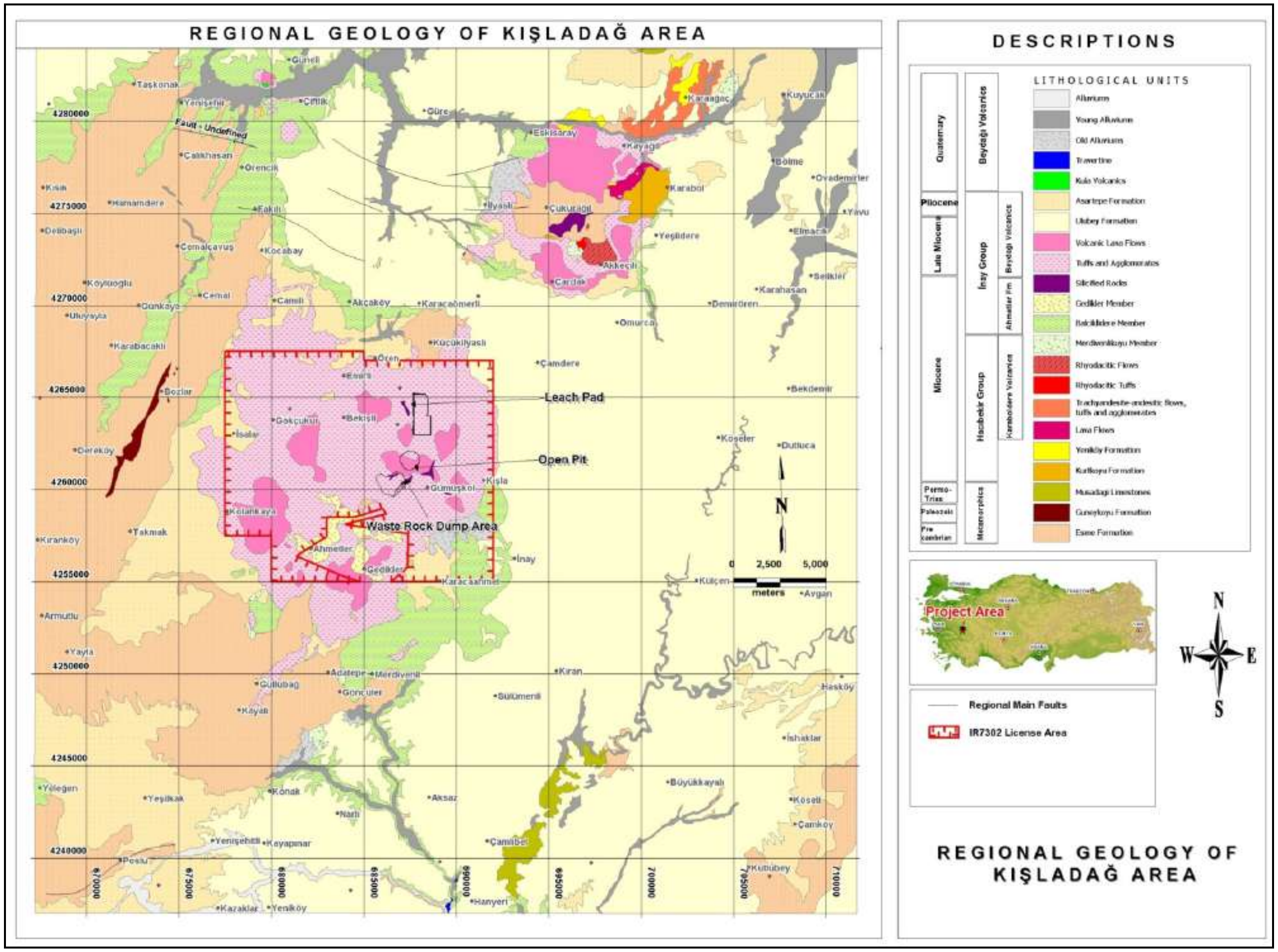
Within the Project area, the oldest rocks are the Paleozoic Menderes Metamorphic Complex, consisting of granitic gneisses crosscut by aplite and quartz dykes of the Güneyköyü Formation, overlain by calcareous schists, crystalline gneisses and augen gneisses of the Esme Formation. Southeast of the Project area the Menderes metamorphic rocks are overlain by Permo-Triassic Musadaği marbles.

The Menderes Complex is overlain by Early Miocene poorly sorted, angular to subangular conglomerates and sandstones (Kürtköyü Formation), and the Miocene Yeniköy Formation consisting of tuffs, sandy to clay rich limestones, and sandstones and polygenic conglomerates of variable composition.

The Early Pliocene Ahmetler Formation is made up of three distinct members; the Merdivenlikuyu member composed of a crudely layered sequence of old colluvium, consisting of angular gravels and cobble sized fragments derived from the Menderes metamorphic basement, the Balçiklidere member consisting of fluvial limestones, claystones, tuffs, sandstones and conglomerates that conformably overlies the Merdivenlikuyu, and the Gedikler member made up of light yellow to light greenish-grey siltstones, claystones and tuffs.

The Beydaği Volcanics, thought to have been deposited concurrently with the Gedikler member sediments, are composed of a series of purplish to pinkish volcanic flows with white-yellowish andesitic tuffs. The Beydaği formation is the dominant host rock in the Project area. Recent age dating of intrusive rocks at Kişladağ indicates a Middle Miocene age for the intrusive/volcanic complex and there is clearly a need for a revision of the stratigraphic relationships.

Figure 7-1: Kişladağ Regional Geology



The Middle Pliocene Ulubey Formation conformably overlies the Ahmetler formation and grades from altered siltstones and claystones through marl and upwards to pinkish and grayish lacustrine limestones.

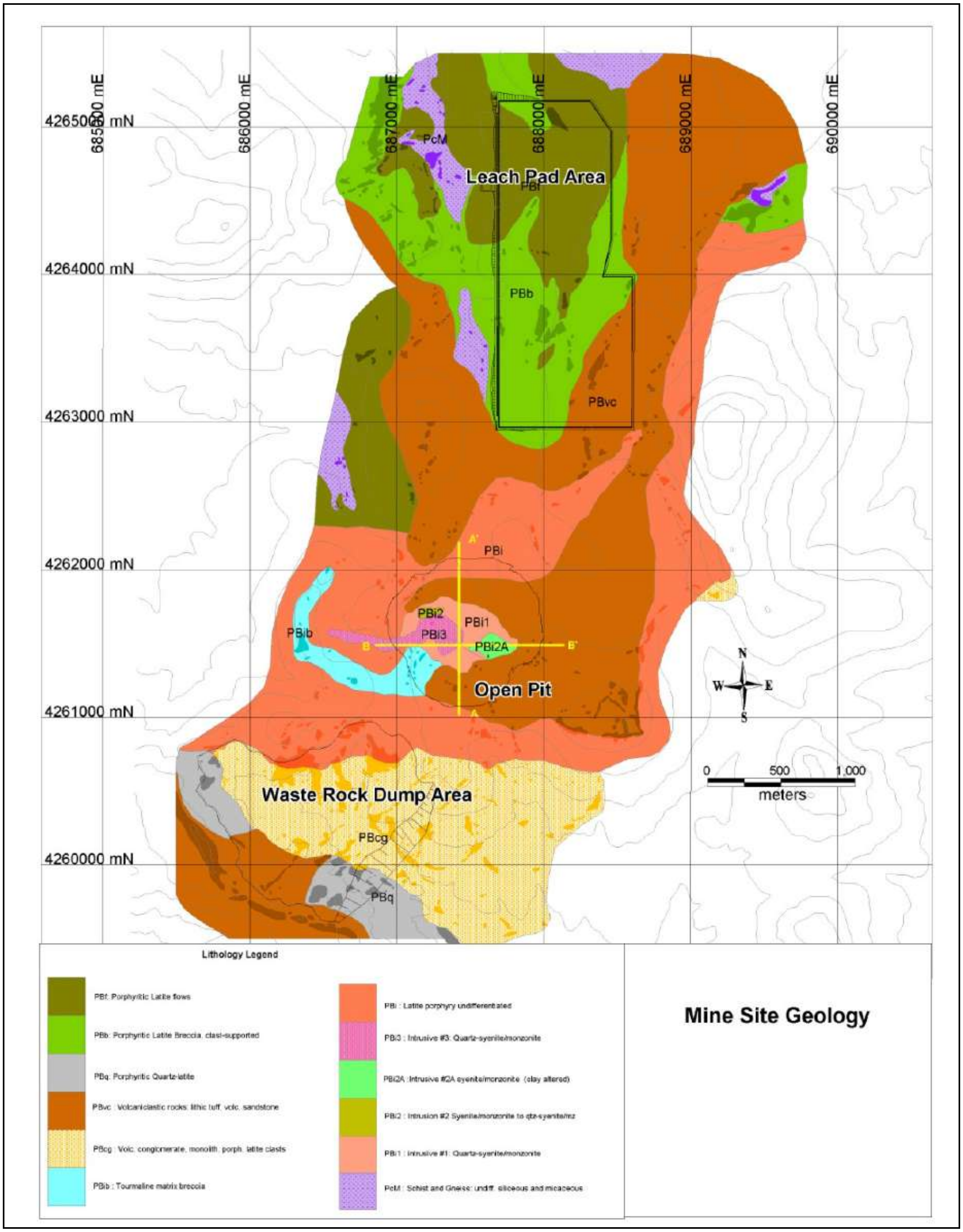
Overlying the Ulubey is the Quaternary Asartepe Formation of poorly cemented gravels, sandstones and siltstones with minor lenses of marl and claystone.

The last stage of volcanic activity in the Project region is represented by the Quaternary Kula Volcanics consisting of basaltic lavas and tuffs.

7.2 PROPERTY GEOLOGY

Within the deposit area, the main lithologies that have been logged and mapped by Eldorado are quartz-trachyte to quartz-latite flows and volcaniclastic rocks intruded by a series of nested sub-volcanic porphyritic intrusives of alkalic affinities. Based on the intensity of alteration, mineralization and cross cutting relationships three mineralized intrusives (Intrusions #1, #2 and #2A) and one post mineral body (Intrusion #3) have been identified. West of the open pit the schistose basement has been intersected in a number of drill holes and outcrops of basement rocks have been mapped north and west of the leach pad area (see Figure 7-2).

Figure 7-2: Mine Site Geology



7.3 LITHOLOGIES

7.3.1 BASEMENT SCHISTS

The basement schist varies from a fine grained mylonite to coarse augen gneiss with rounded porphyroclasts of orthoclase. The schist is composed dominantly of quartz, plagioclase/albite and white mica and contains quartz veins that have been affected by the intense deformation. In the vicinity of the open pit the foliation has a shallow dip to the south-east. Elsewhere on the property the dips are dominantly flat lying but may show a rapid transition to steeply dipping.

7.3.2 PYROCLASTICS

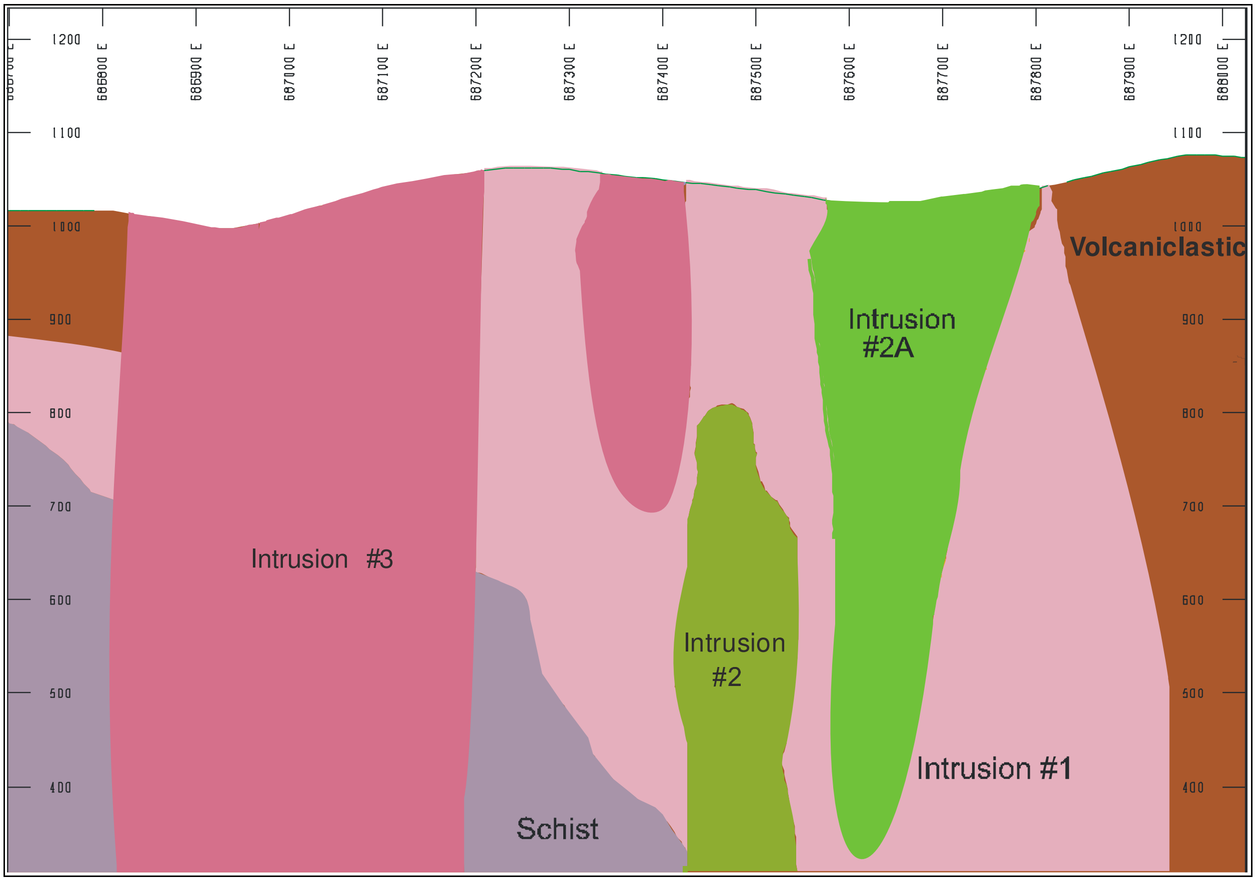
The volcanic rocks on the upper benches of the pit and in drill holes on the south side of the pit vary in texture from fine grained fragmental ash fall tuffs with pumice fragments to porphyritic flows with flow banding and auto brecciation. Eight hundred metres east of the pit two diamond drill holes intersected a number of fine grained plagioclase-hornblende porphyries with interlayered ash fall tuffs. The porphyries have been propylitically altered and it is not clear if they are intrusive bodies or flows.

The contact between volcaniclastics and Intrusive #1 is difficult to define due to the intensity of alteration but is steeply inclined and often marked by a tourmaline rich hydrothermal breccia.

7.3.3 INTRUSION #1

Intrusion #1 is the oldest, and generally, best mineralized porphyritic intrusive phase. It forms the core of the system, and is cross cut by the younger porphyritic intrusions. It forms a slightly elliptical body approximately 800 m across, which was exposed at the pre-mining surface. Recent drilling indicates the roots of Intrusion #1 lie on the southeastern side of the pit. In the western half of the deposit, the basement schist is encountered in deeper holes, at depths of 200 m to 350 m, and to the west of the proposed pit outline, Intrusion #1 may become sill-like in form, lying above the schist, and not coming to surface (see Figure 7-3). Contacts between Intrusion #1 and the surrounding volcanic rocks are generally obscured by alteration. Contacts with younger intrusions, particularly Intrusion #3 are better preserved.

Figure 7-3: E-W Cross-Section at 4261500N with Lithologies (looking north)



Intrusion #1 comprises abundant phenocrysts in a K-feldspar dominant groundmass. Plagioclase is the dominant phenocryst phase, comprising up to 30% of the rock by volume. It occurs as tabular crystals ranging in size from <1 mm to 5 mm. Biotite is the second most abundant phenocryst phase, comprising up to 10% of the rock. It is typically finer-grained than plagioclase. Blocky megacrystic K-feldspar phenocrysts, up to 1 cm, are a characteristic of this unit, but are low in abundance. Quartz phenocrysts are rare. Based on the primary mineral assemblages this rock would lie in the quartz-syenite to quartz-monzonite fields.

7.3.4 INTRUSION # 2A

Intrusion #2A occurs in the southeast corner of the pit, where it intrudes the margin of Intrusion #1. It forms a circular stock 250 m to 300 m across. It appears to taper at depth. This unit carries economic gold grades, but is not as well mineralized as Intrusion #1. It is a fine to medium-grained porphyritic rock. Intense pervasive clay-quartz alteration appears to have selectively overprinted this unit. The unit contains a significant amount of plagioclase phenocrysts (~20%), up to 2 mm in length, and possibly sparse quartz phenocrysts. This unit is very similar to texture to Intrusion #2, but is differentiated because of the intense clay alteration, which does not affect Intrusion #2.

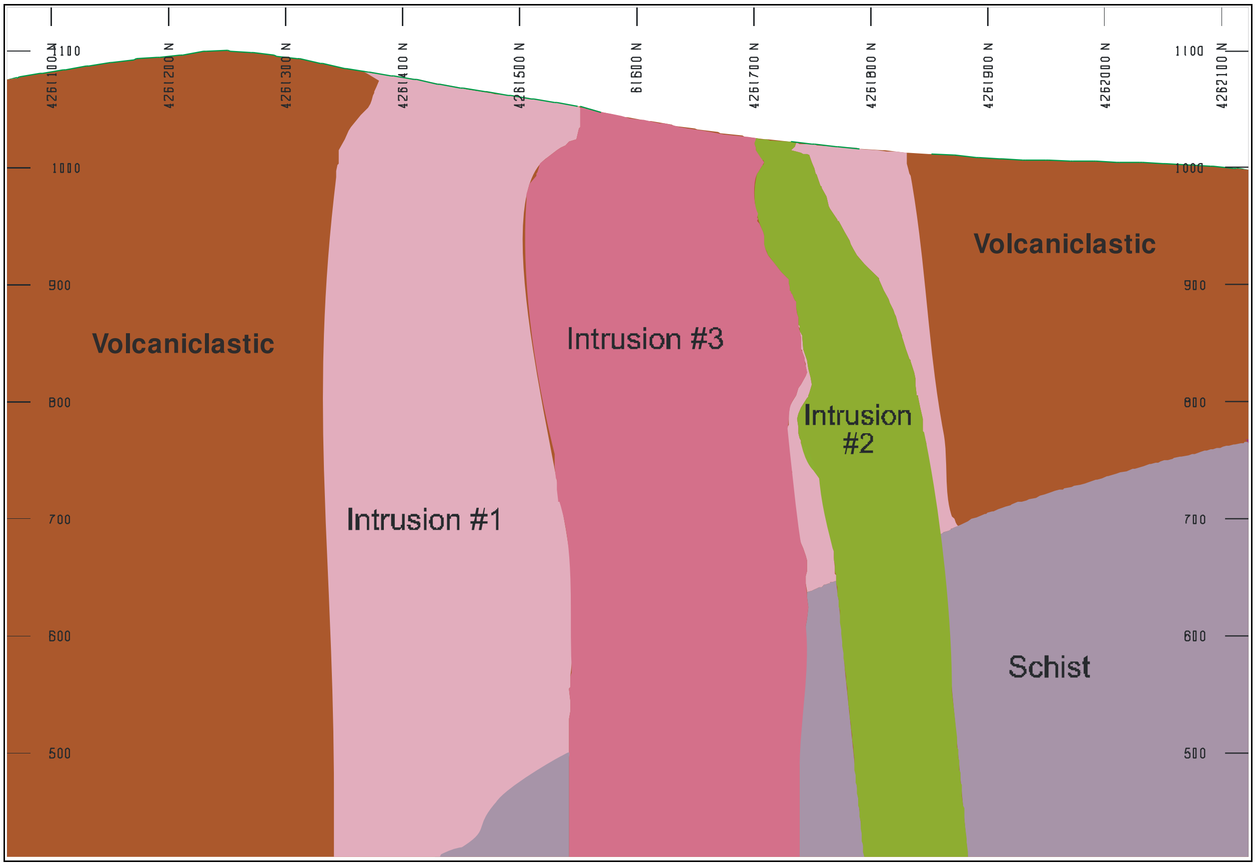
7.3.5 INTRUSION #2

Intrusion #2 occurs as two separate semi-circular stocks, both approximately 150 m to 200 m across. One occurs in the center of the pit, cutting the core of Intrusion #1, and the second occurs on the northwestern margin of Intrusion #1. Both intrusions are in contact with Intrusion #3, and there are large fragments of schist along the margin of the northwestern stock. The stocks carry economic gold grades. The rock is a fine-medium grained porphyry. It is comprised of abundant (20-30%) plagioclase phenocrysts up to 2 mm in length in a dominantly K-feldspar groundmass. No quartz phenocrysts were observed. This unit can be weakly-moderately magnetic. Compositionally this unit would lie in the syenite – monzonite fields.

7.3.6 INTRUSION #3

Intrusion #3 is the youngest intrusive body at Kişladağ. It forms a semi-circular stock near the center of Intrusion #1, west of the central Intrusion #2 stock (Figure 7-4), and extends into an elongate, steeply dipping to vertical dyke-like body to the west, extending beyond the limits of the proposed pit. It has been traced at least 450 m west of the pit (Figure 11-1). The contacts of this unit with other rocks are generally well preserved, and the drop in gold grade is abrupt at the contacts. It is a fine-grained porphyritic unit, comprised of 20% to 30% plagioclase phenocrysts, up to 4 mm in length, sparse quartz and biotite phenocrysts (both <5%), and amphibole phenocrysts (5-10%). The amphibole phenocrysts have been preferentially altered to secondary mafic minerals, but their prismatic shapes are preserved. This intrusion is typically magnetic, due to the presence of very fine-grained disseminated magnetite in the groundmass. The magnetite is probably a primary component of this unit. Its bulk composition places it in the quartz-syenite - quartz-monzonite fields.

Figure 7-4: N-S Cross-Section at 787300E, Looking West



7.4 STRUCTURAL GEOLOGY

The Kişladağ deposit lies in the eroded core of a Miocene stratovolcano complex, which has experienced relatively little structural modification. Lithologic contacts are primarily intrusive or depositional, with no mappable fault offsets documented. The most significant structure in the deposit area is a northeast-striking sub-vertical fault that passes just east of the deposit, and is inferred from the presence of highly linear silicified ridges on surfaces and the apparent truncation of gold grade and intrusive phases along on the eastern edge of the deposit. Alteration associated with this unnamed feature can be traced south of the deposit into the waste dump area.

Bedding dips within the stratified sequence are shallow to moderate. In most areas, strata dip away from the core of the deposit, mimicking the probable form of the original stratovolcano. The present-day dips mainly reflect primary depositional dips, rather than the results of structurally-induced tilting.

Despite the absence of major faults, the Kişladağ deposit and adjacent rocks contain a high density of low-displacement brittle fractures. Most of the observable fractures are best classified as joints (sub-parallel fractures with no movement) and low-displacement faults (displacement magnitudes measuring centimetres to decimetres) and have continuity limited to a few metres to a few tens of metres.

The most continuous faults visible in the pit correspond to sub-vertical, steeply-dipping, EW-striking faults that are localized within a corridor crossing the central portion of the pit. Intrusion 1 has an elongate form parallel to these faults, and the dyke-shaped portion of Intrusion 3 occurs along the western extension of this fault corridor. Collectively, these features suggest that the deposit and causative intrusions are localized along an east-west fracture/fault zone, evidence for which is largely obliterated by the intrusions and associated hydrothermal activity.

Multiple joint sets are ubiquitous in pit exposures, and are well characterized in orientation data collected from oriented drill core. Joints commonly occur in orientation-specific swarms or structural corridors, but exhibit neither radial nor concentric orientation distributions in the deposit area. Steeply west-dipping, NNE-striking joints form by far the most pervasive set, dominating the structural character of the eastern third of the present pit.

SECTION 8 • DEPOSIT TYPES

The Kışladağ deposit consists of porphyry-style gold mineralization centred on a series of overlapping sub-volcanic intrusives of quartz-syenite to quartz-monzonite composition. Geological consultant Richard Sillitoe visited the property in 2000, and concluded in his report:

"Kişladağ is confirmed to be a true porphyry gold deposit, albeit possessing several distinctive geological features. These include the paucity of quartz veinlets, the dominance of molybdenum over copper and the exceptionally high gold values. The deposit is centred on a steep, multi-phase latite porphyry intrusion of alkaline affiliation. Younger intrusive phases were emplaced progressively nearer the centre of the stock and are characterized by increasingly weaker alteration of lower gold contents. The centrally positioned late-mineral phase is essentially barren."

SECTION 9 • MINERALIZATION

The Kişladağ deposit consists of porphyry-style gold mineralization centered on a series of overlapping sub-volcanic and extending outward into the surrounding volcanic and volcaniclastic rocks.

Gold mineralization with traces of molybdenum, zinc, lead and copper encircles the late barren stock (Int-#3). Higher-grade gold mineralization (> 1 g/t Au) is associated with Intrusive #1 and forms a horseshoe shaped zone around the northern, southern and eastern sides of Intrusive #3. The higher gold grades are associated with multiphase quartz sulphide stockwork and pervasive silicification. The mineralized zones dip outward in a bell-shaped body; sub parallel to the contact of the stock.

Gold is associated with at least three phases of partially overlapping stockwork veining and brecciation. These include:

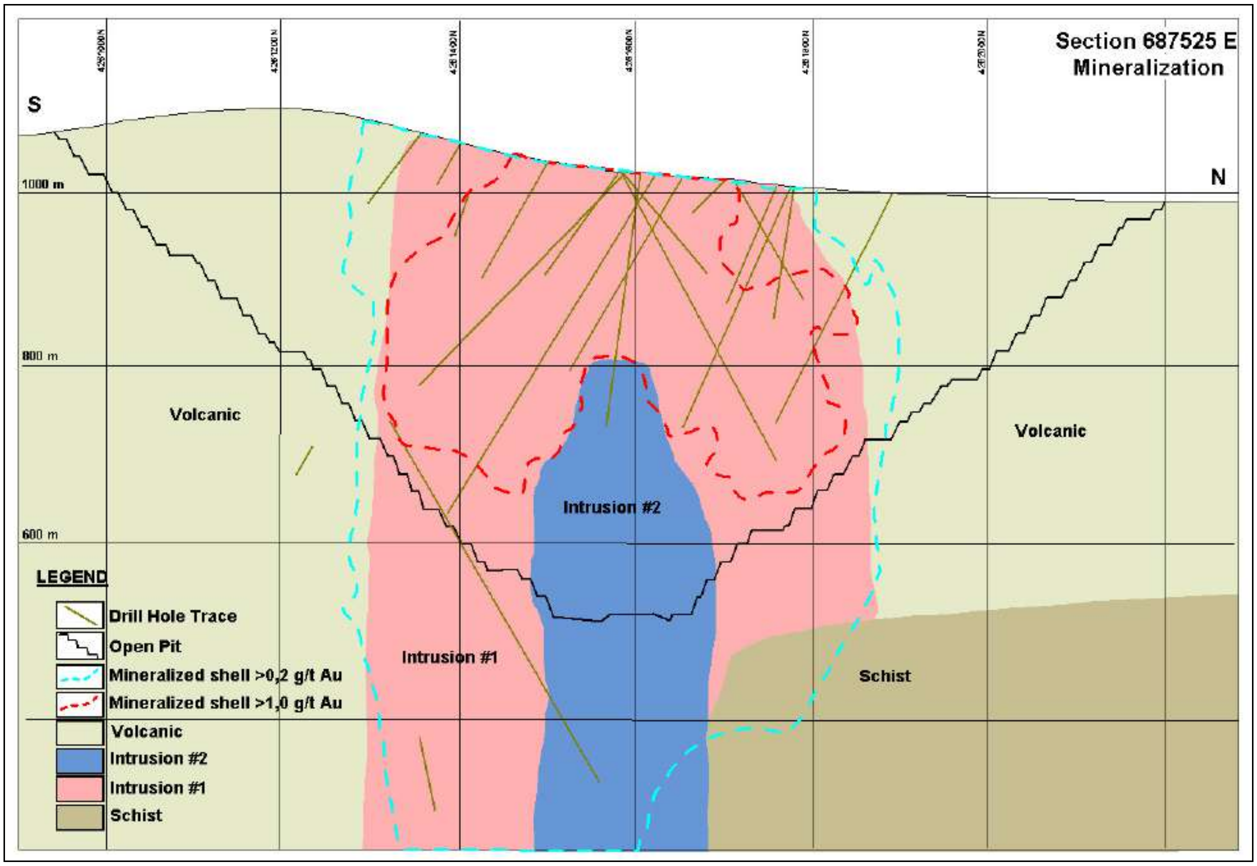
- Intense quartz-tourmaline stockwork veining
- Hydrothermal breccias (±gold)
- Multiple phases of quartz-pyrite veining with local silica flooding with gold
- Late sulphide rich quartz veining with traces of molybdenum, sphalerite, galena and tetrahedrite (± gold).

In general, the amount of stockwork veining decreases with depth, especially below 650 masl elevation. Higher-grade mineralization (above 2 ppm Au) has been traced from surface to depths greater than 250 m below surface. Lower-grade mineralization, grading between 0.5 and 1.0 ppm Au has been traced to the deepest levels drilled on the property approximately 600 m to 700 m below surface (Figure 9-1).

Tourmaline is locally abundant in every phase of mineralization except the late vuggy silica. It typically occurs as very fine, anhedral grains disseminated in the host rock and silica gangue. Pyrite is the dominant sulphide mineral present with visual estimates averaging around 4% in the primary ore zone. Locally as much as 15% pyrite can be present. Other sulphide minerals identified in microscopic studies include chalcopyrite, sphalerite, tetrahedrite, galena and molybdenite.

Oxidation tends to be deeper on the uphill (southern) side of the deposit (from 30 m to 80 m deep) as compared to the downhill (northern) side of the deposit, where oxidation is limited to between 20 m and 50 m below surface. There is also a broad east-west trend, with slightly deeper oxidation on the east side (50 m to 100 m) versus the west side of the deposit where oxidation ranges from 30 m to 60 m deep. Limonite is the most abundant oxide mineral, usually occurring along fractures in thin colloform layers and as disseminated patches around weathered pyrite and mafic minerals.

Figure 9-1: Section 687525 E Mineralized shells with Lithologies

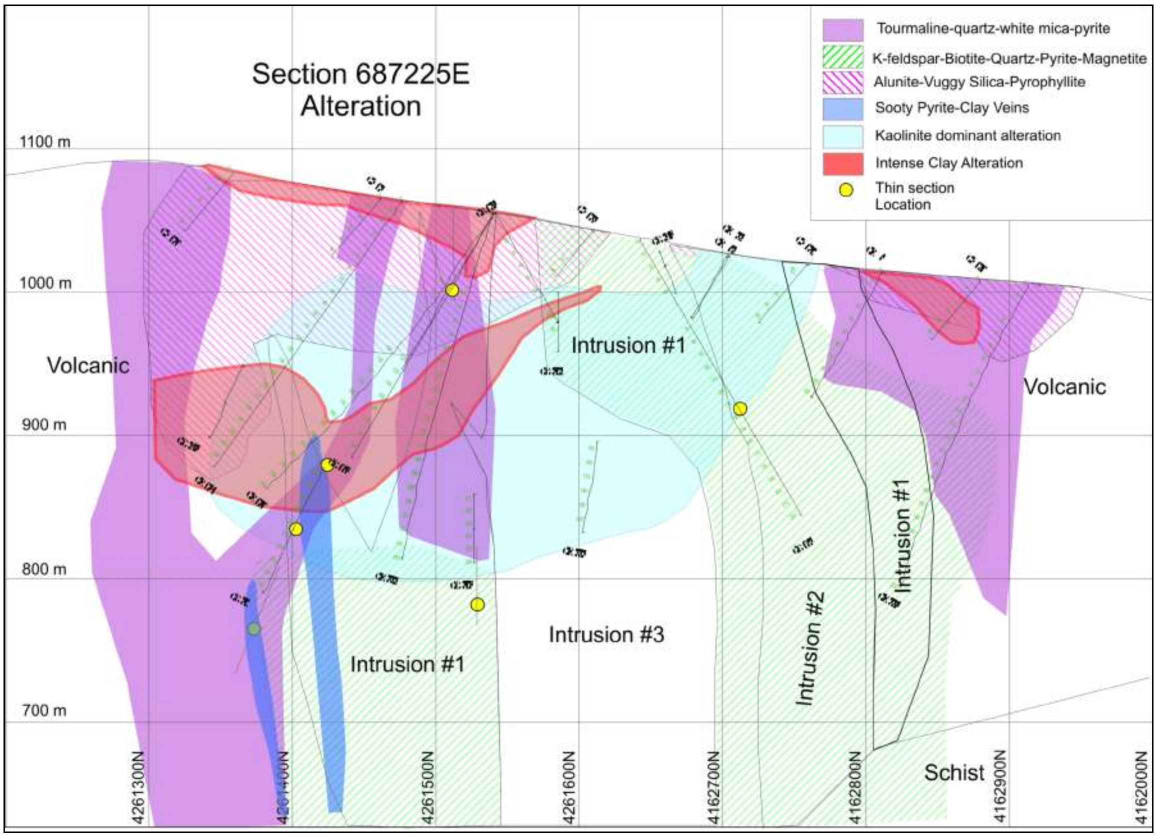


9.1 ALTERATION

A potassic assemblage characterized by secondary biotite, and focused on Intrusion #1 forms the core of the system. Tourmaline is present throughout the deposit, and in all phases of intrusions, but is most common in the volcanic rocks adjacent to Intrusion #1 and forms an outer alteration shell. An advanced argillic assemblage characterized mainly by alunite occurs on the outer margins, beyond and overprinting the tourmaline, partially capping the deposit and extending to depths of up to 350 m on the eastern side of the pit (see Figure 9-2). A pervasive retrograde argillic assemblage of clays +/-chlorite-montmorillite overprints all other alteration, and can extend to depths of 600 m. Surface clay weathering and oxidation is also present, and locally extends to depths of 250 m on the outer margin of the deposit. It may in part be preferentially developed on zones of advanced argillic alteration. Disseminated pyrite is ubiquitous, and may have been introduced which each stage of hydrothermal alteration, particularly the potassic and argillic stages. There is no propylitic alteration (chlorite-calcite-epidote) in or immediately adjacent to the deposit.

The alteration mineral assemblages associated with the deposit were formed by a complex interaction of hydrothermal and meteoric fluids. Each intrusive event contributed a high temperature phase dominated by magmatic fluids and vapours followed by a retrograde cooling stage associated with meteoric water telescoping in on the earlier alteration.

Figure 9-2: Section 687225E Interpreted Alteration Zones. Geologic contacts are shown for reference.



SECTION 10 • EXPLORATION

10.1 PRE-2007 WORK

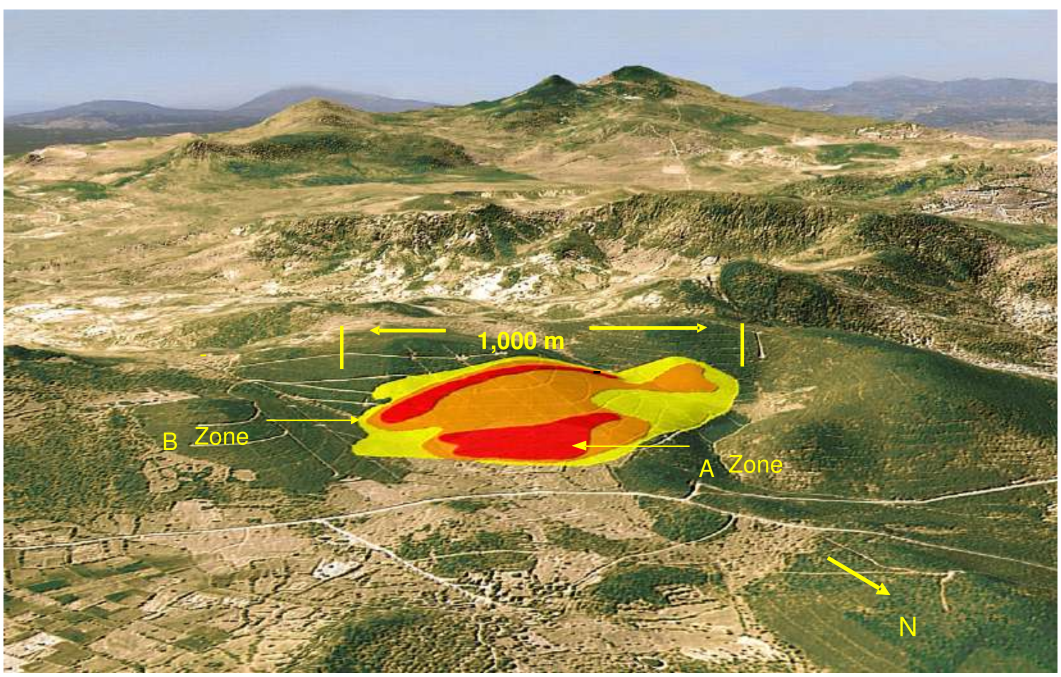
The deposit was discovered during a regional reconnaissance program in the late 1980s. LANDSAT imagery was utilized to identify potential alteration zones associated with volcanic centers. Stream sediment, soil and rock chip sampling and reconnaissance geological mapping were then applied to screen the anomalies for precious metal mineralization. Detailed exploration began at Kişladağ in 1997 when Tüprag acquired the key license covering the gold anomaly.

Detailed exploration commenced with geological mapping, a soil geochemical survey, trenching, and shallow percussion drilling. Induced Polarization and ground magnetic survey were also conducted on the property

10.1.1 SOIL GEOCHEMISTRY

The soil sampling program outlined a large oval shaped gold anomaly, defined by a greater than 25 ppb Au contour, approximately 1,100 m east-west along strike by 750 m north-south on the north slope of Gökgöz Tepe. Within this anomaly, two zones with greater than 200 ppb Au were identified as Anomaly A and B respectively, (see Figure 10-1)

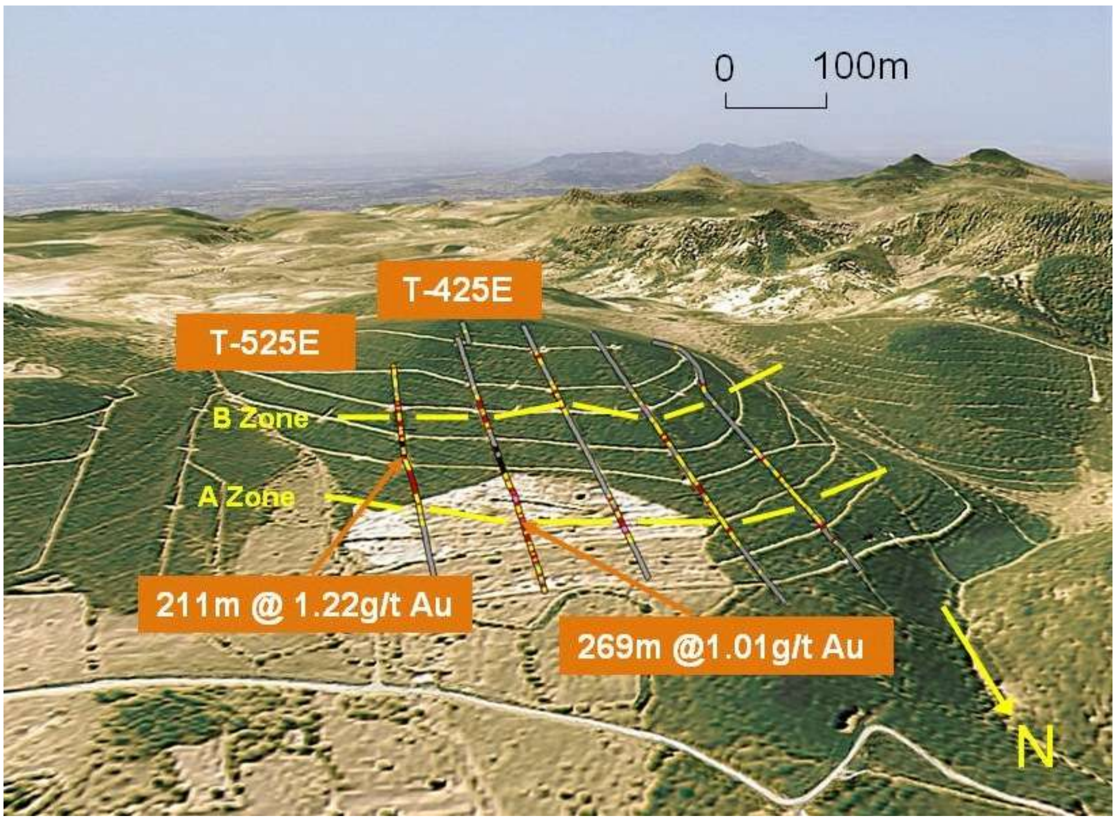
Figure 10-1: Soil Geochemical Anomaly Viewed from the North



10.1.2 TRENCHING

A follow-up program consisting of channel rock samples from 2,475 m of north-south backhoe trenches confirmed the anomalies (see Figure 10-2). In year 2000 another approximately 1,500 m of trenching is done towards the west side of the deposit.

Figure 10-2: View of Gökgöz Tepe Looking South Showing the Gold Trench Geochemistry



10.1.3 GEOPHYSICS

Geophysical techniques consisted of ground-base induced polarization (IP) and magnetic surveys. The most effective has been IP with chargeability highs coinciding with the higher grades of mineralization and resistivity highs associated with large zones of silica-tourmaline alteration.

10.2 2007 – 2009 WORK

Exploration work during this period consisted primarily of diamond drilling. Additional work included detailed mapping, at a scale of 1:5000, of the central parts of Kişladağ and adjacent Sayacik volcanic centres and a detailed study of the alteration associated with the Kişladağ deposit. The alteration study was supported by numerous PIMA™ analyses.

SECTION 11 • DRILLING

Diamond drill holes are the principal source of geologic and grade data for the Kişladağ Mine since the start of mining in 2006. Data from these holes are directly responsible for the significant increase in the year-end 2009 mineral resources and reserves (see Section 17). Drilling totals, organized by periods (pre-2004, covered in the previous Technical Report (Micon, 2003), 2004 to 2006, and 2007 to 2009) are shown in Table 11-1. The rest of this section focuses on the 2007 to 2009 drilling campaigns.

Table 11-1: Summary of Kişladağ Mine Drilling by Period

	Diamond Drilling		RC Drilling		Rotary Drilling	
Period	**# of holes**	**(m)**	**# of holes**	**(m)**	**# of holes**	**(m)**
Pre-2004	53	12,269	145	21,298	44	2,264
2004-2006	9	862	8	1,329		
2007-2009	81	34,603	14	3,558		
Total	**143**	**47,734**	**167**	**26,185**	**44**	**2,264**

The 2007-2009 campaign diamond drill holes range in length from 194 to 1,058 m, averaging 418 m. The RC holes were not as deep, averaging 273 m. The location of these holes are shown on a collar plan map in Figure 11-1.

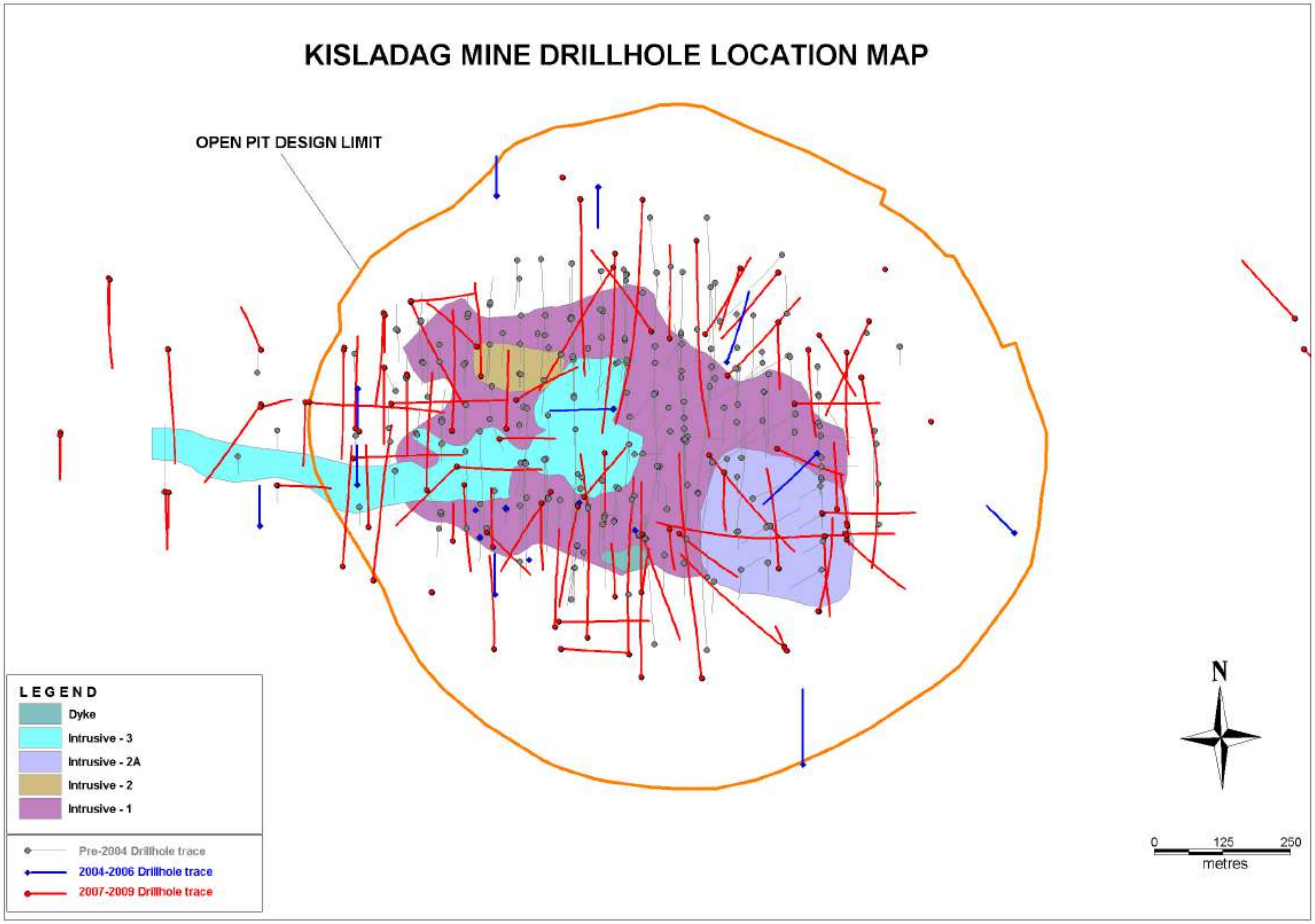
Drilling was done by wireline method with H-size (HQ, 63.5 mm nominal core diameter) and, less commonly, N-size (NQ, 47.6 mm nominal core diameter) equipment using up to four drill rigs. Upon completion, the collar and anchor rods were removed and a PVC pipe was inserted into the hole. Drill hole collars were located respective to a property grid. Proposed hole collars and completed collars were surveyed by the mine survey group.

The drill holes were drilled at an inclination of between 50° and 90°, with the majority between 60° and 70°. Holes were drilled along 0°, 090°, 135°, 180°, and 310° azimuths. Down-hole surveys were taken approximately every 50 m by the drill contractor using a single-shot measurement system (Reflex survey instrument).

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.

Eldorado reviewed the core logging procedures at site, and the drill core was found to be well handled and maintained. Material was stored as stacked pallets in an organized "core farm." Data collection was competently done. Core recovery in the mineralized units was excellent, usually between 95% and 100%. Overall, the Kişladağ drill program and data capture were performed in a competent manner.

Figure 11-1: Kişladağ Mine Drill hole Location Map



SECTION 12 • SAMPLING METHOD AND APPROACH

The new data collected in 2007 to 2009 comprised mostly diamond drill core. Minor Reverse Circulation (RC) holes were drilled and rock chips collected.

All diamond drilling in Kışladağ was done with wire line core rigs and mostly of HQ size. Some deeper drill holes required a reduction to NQ rods to complete the drill hole. Drillers placed the core into sturdy, locally made, wooden core boxes with each box holding about 4 m of HQ core. The driller keeps track of the drilling depth and places footage marker wooden blocks at the end of each run. These marker blocks are nailed into the boxes. Drill core was later delivered to logging site. Sample numbers were written on wooden core boxes allowing gaps in numbering sequence for control sample insertion. The entire lengths of the diamond drill holes were sampled. Geology and geotechnical data are collected from the core and core is photographed (wet) before sampling. SG measurements were done approximately every 5 m. The 2007 to 2009 core cutting and sampling was done on site at Kışladağ . The cut samples were then sent to the Canakkale prep lab. The entire core library for the Kışladağ deposit is kept in core storage facilities on site.

For the limited RC drilling during 2007 to 2009, Eldorado was responsible for the sampling. Sampling was done on 2.5 m intervals with the whole length of the hole sampled. A cloth or perforated sample bag was used for RC samples. When drilling dry, drill cuttings were collected from cyclone directly and splitted using Jones splitter after that. After each rod drill hole cyclone was blown clean. A small (~1 kg) sample from each interval was collected also for logging and chipboard making. Wet RC sampling was done using a rotating wet splitter mounted on the rig. If the ground water flow was insufficient, extra water was injected through the rods to maintain the necessary flow rate for rotating wet splitter. Wet samples were left to drain their water on a safe place and then they were shipped to companies prep lab facility in Canakkale province.

Significant composited assays (by intersected ore shell thickness) for the Kışladağ deposit are shown in Appendix A. Only values from the 2007 to 2009 drill campaigns were tabulated.

SECTION 13 • SAMPLE PREPARATION, ANALYSES AND SECURITY

13.1 SAMPLE PREPARATION AND ASSAYING

Split core samples are prepared for analysis at Eldorado's in-country preparation facility at Çanakkale in north-western Turkey. The samples are prepared according to the following protocol:

The entire sample is crushed to 90% minus 3 mm.

- A 1 kg subsample is riffle split from the crushed minus 3 mm sample and pulverized to 90% minus 75 µm (200 mesh)
- A 110 g subsample is split off by taking multiple scoops from the pulverized 75 µm sample
- The 110 g subsample is placed in a kraft envelope, sealed with a folded wire or glued top, and prepared for shipping. The rest of the pulverized sample is then stored in plastic bags.

All equipment is flushed with barren material and blasted with compressed air between each sampling procedure. Regular screen tests are done on the crushed and pulverized material to ensure that sample preparation specifications are being met.

The sample batches are arranged to contain regularly inserted control samples. A Standard Reference Material (SRM), a duplicate and a blank sample were inserted into the sample stream at every 8th sample. The duplicates are used to monitor precision, the blank sample can indicate sample contamination or sample mix-ups, and the SRM is used to monitor accuracy of the assay results.

The sample pulps are sent from the Çanakkale sample preparation facility to ALS Chemex Laboratories sample preparation facility in Izmir and were then shipped under the supervision of ALS Chemex to their Analytical Laboratory in North Vancouver, BC. All samples were assayed for gold by 30 g fire assay with an AA finish and for multi-element determination using fusion digestion and ICP analysis.

13.2 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 0.03 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.

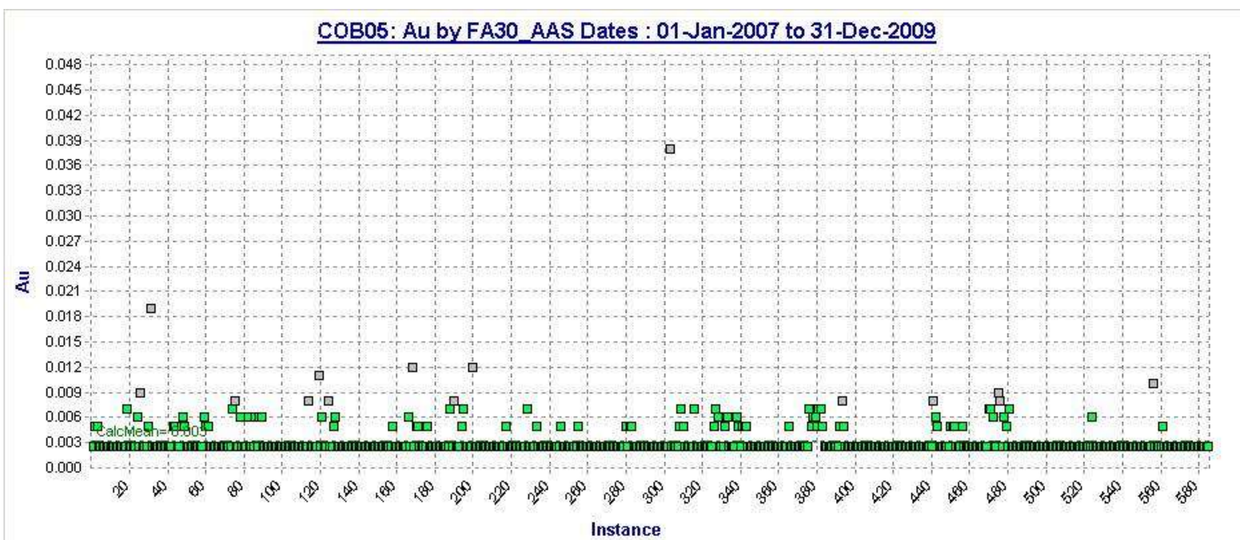
13.2.1 PRE-2007 PERFORMANCE

Assay performance of data collected prior to 2007 (essentially 2003 and earlier) has been described in detail in the previous Technical Report (Micon, 2003). It will only be summarized here. A system of SRMs, blanks and duplicates controlled the earlier exploration and delineation work. All results showed that the process was in control and that the data was sufficiently reliable to support resource estimation. The remaining sections will only deal with performance in 2007 to 2009.

13.2.2 BLANK SAMPLE PERFORMANCE

Assay performance of field blanks is presented in Figure 13.1 for gold. The analytical detection limit (ADL) for gold is 0.005 g/t. The rejection threshold was chosen to equal 0.03 g/t. The results show a no evidence of contamination. Rare higher values were investigated and found to be caused by sample mix-ups.

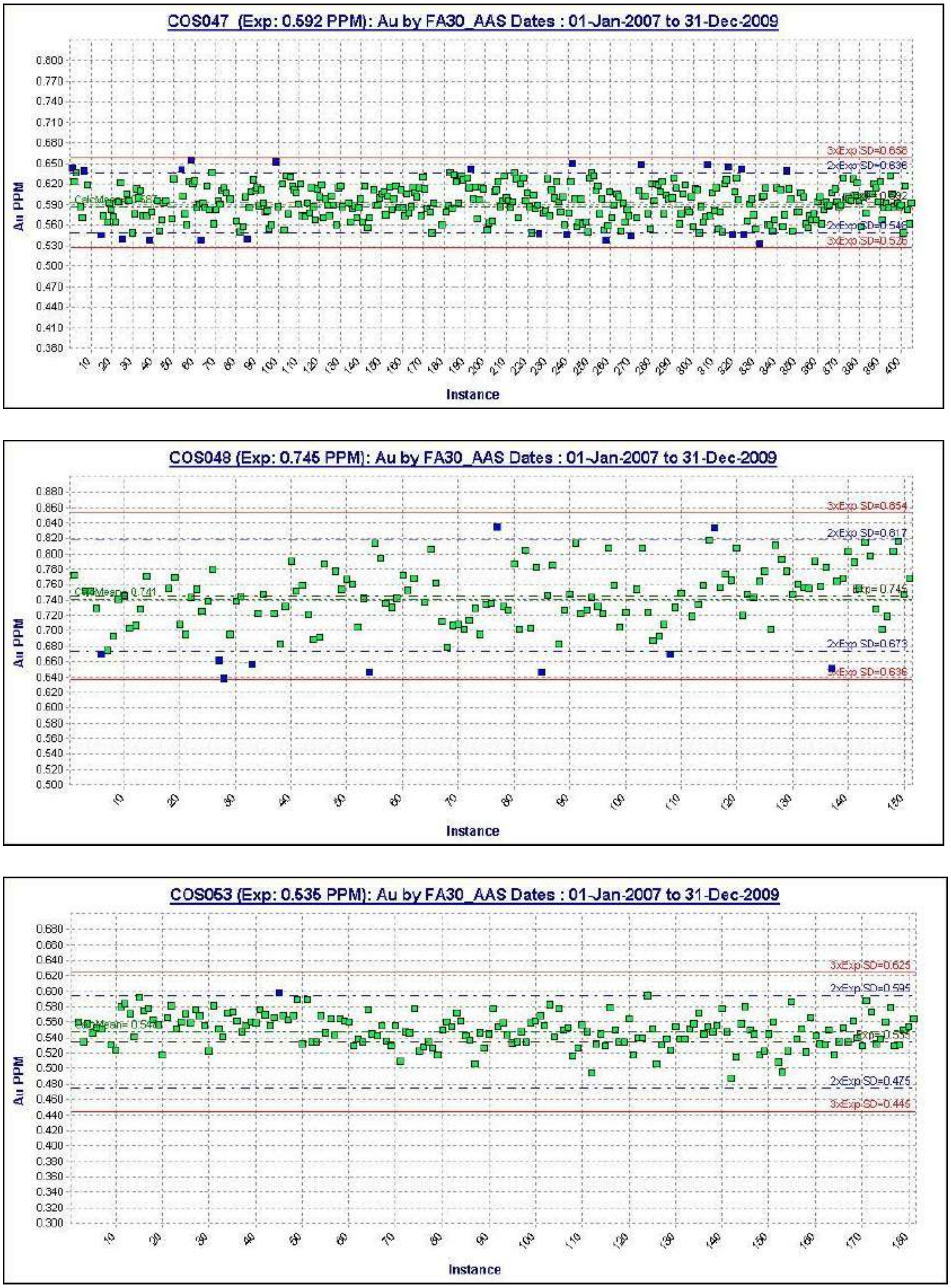
Figure 13-1 Kışladağ Blank Data – 2007 to 2009

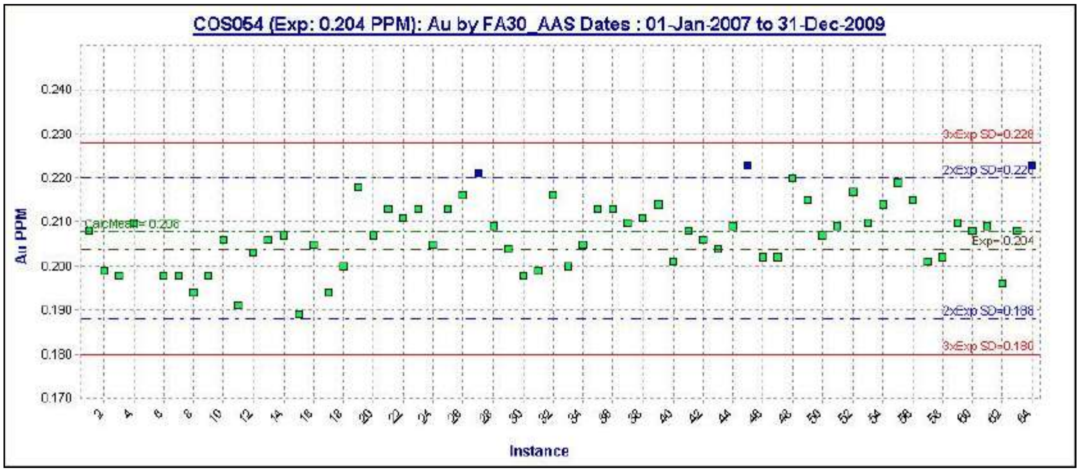


13.2.3 STANDARDS PERFORMANCE

Eldorado Gold strictly monitors the performance of the SRM samples as the assay results arrive at site. Four SRM samples are used, covering a grade range between 0.2 g/t to 0.75 g/t. Charts of the individual SRMs are shown in Figure 13-2. 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 cutoff date of 31 September 2009, all samples had passed acceptance criteria.

Figure 13-2: Standard Reference Material Charts, 2007 to 2009, Kışladağ





13.2.4 DUPLICATES PERFORMANCE

Eldorado implemented and monitored regularly submitted coarse reject duplicates. These data reproduced well. The duplicate data are shown in a relative difference chart in Figure 13-3 and percentile rank chart in Figure 13-4. Patterns observed in the relative difference plot are symmetric about zero suggesting no bias in the assay process. For the 90th percentile of the population as shown on the percentile rank plot, a maximum difference of 20% is recommended for the coarse reject duplicates because these duplicate types can be controlled by the subsampling protocol. The Kışladağ data shows 16% difference in the coarse reject data.

Figure 13-3: Relative Difference Plot of Kışladağ Duplicate Data, 2007 to 2009.

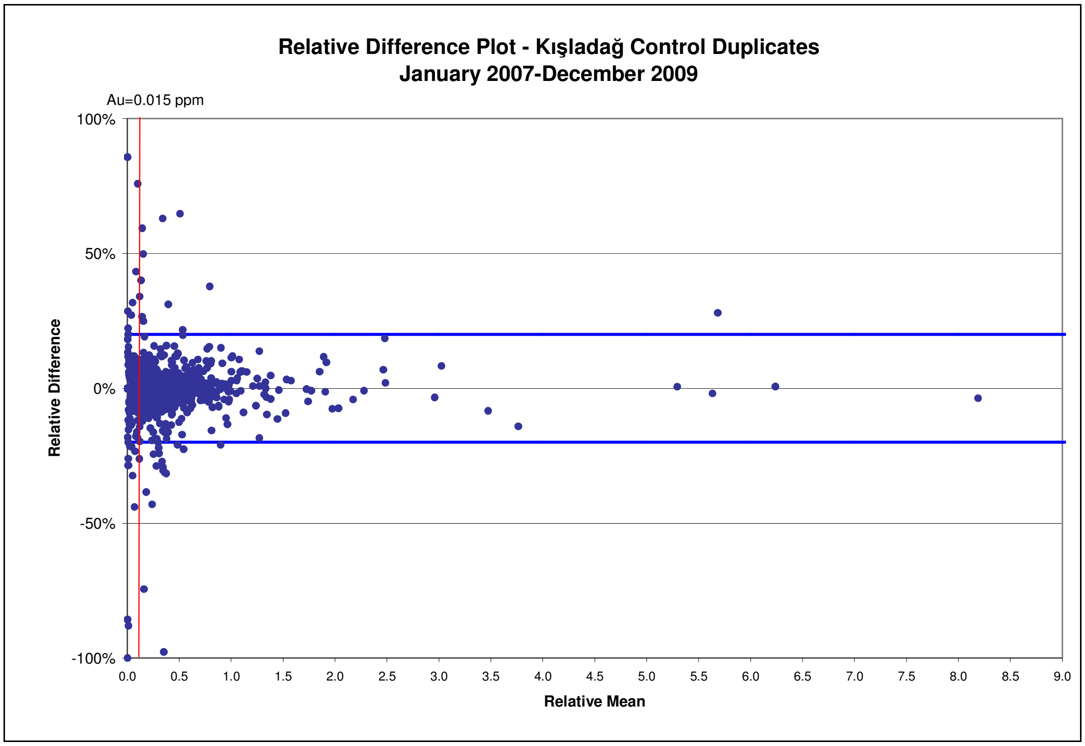
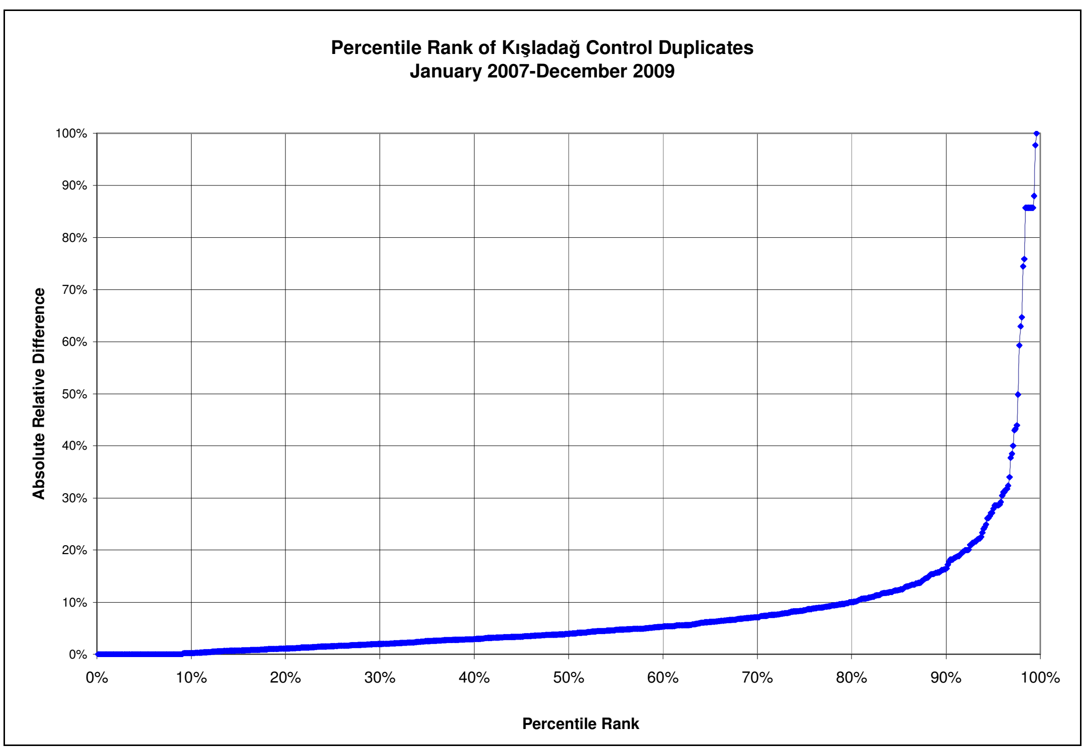


Figure 13-4: Percentile rank plot, Kışladağ duplicates, 2007 to 2009.



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 W1 / (W1-W2).

13.3 CONCLUDING STATEMENT

In Eldorado's opinion, the QA/QC results demonstrate that the Kışladağ deposit assay database, particularly for new data obtained from 2007 to 2009, is sufficiently accurate and precise for resource estimation.

SECTION 14 • DATA VERIFICATION

Since the start of production in 2006, the entire drill hole database was reviewed in detail. Checks were made to original assay certificates and survey data. Also, the descriptive information (lithology and alteration) was reviewed through relogging and pit mapping, and mineral identification by PIMA™ (a field portable, infrared spectrometer) analyses. Any discrepancies found were corrected and incorporated into the current resource database.

Another form of verification is the reconciliation to production of mined portions of the resource model. Results to date have shown excellent agreement between mined production and the long term resource model. This is discussed in Section 18.

Eldorado therefore concludes that the data supporting the Kışladağ resource work are sufficiently free of error to be adequate for estimation.

SECTION 15 • ADJACENT PROPERTIES

There are no mineral properties of importance adjacent to the Kişladağ Mine site.

SECTION 16 • MINERAL PROCESSING AND METALLURGICAL TESTING

Testwork completed by Kappes Cassiday and Associates has shown that the Kişladağ ore is amenable to heap leaching technology. A gold recovery of 80% to 81% is projected for the oxide ore. The primary ore has a higher sulphide content and gold recovery is projected to be 60%.

The Kişladağ deposit is characterized by an oxidation zone that extends from surface to approximately 50 m deep. The gold grades of the oxide ore is relatively less than primary ore and elemental analysis shows that the sulphur content of the oxidized ore is approximately 0.3% compared to 2.5% for the deeper primary ore. The ore contains minor amounts of silver and copper, which has not affected recovery of gold in the testwork. Concentrations of potentially deleterious elements such as Hg, As, and Sb are insignificant and should not present processing or environmental problems.

The mineralogy of the Kişladağ ore shows that the gold occurs in fine grains that are usually associated with pyrite, its oxidation products, or gangue. The rock types described are primarily andesite and dacite porphyry and hydrothermal breccias, showing various types of alteration, including silicification and clay alteration.

Grinding and crushing testwork indicated that the Kişladağ ore could be classified as medium to hard rock, abrasive, with a high comminution energy demand. Lakefield Research investigated the potential for fine grinding and cyanide leaching and concluded that the extraction of gold is not particularly sensitive to particle size. However, gold recoveries varied widely and a correlation between gold recovery and sulphide content was suggested. Gravity gold recovery was poor and froth flotation did not appear viable. A scoping study showed that the economics of a mill were less attractive than a heap leach.

Kappes Cassiday and Associates completed a total of 45 heap leach column tests on Kişladağ ore. The tests showed that a fine crush size and a leach period of ninety days were required to maximize gold recovery. A crush size of 80% passing 6.3 mm was selected for both the oxide and primary ore. Oxide ore is less sensitive to crush size and a coarser crush size may prove viable during operations. As per anticipated by KCA testing works, during the course of operations, oxide ore crushing size was coarsened to 80% passing -12.5 mm. On site laboratory column tests also supported coarse crushing of oxide ore. Reagent consumptions are moderate and cyanide consumption is projected to be 0.25 kg NaCN/t for oxide ore and 0.34 kg NaCN/t for the primary ore. The average cyanide consumption for Year 2009 is realized as 0.32 kg NaCN/t. Lime consumption is projected to be 4 kg $Ca(OH)_2$/t. Actual lime consumption is about 5.5 kg $Ca(OH)_2$/t. Percolation tests indicated that the Kişladağ ore does not require cement agglomeration and heap heights of up to 60 m are possible. Some of the cells at operation are already reached to 60 m lift heights. Percolation is moderate with applications rates averaging 5 to 8 L/m^2/h.

SECTION 17 • MINERAL RESOURCES AND MINERAL RESERVES ESTIMATES

17.1 MINERAL RESOURCES

The mineral resource estimates for the Kişladağ Mine were calculated under the direction of Dr. Stephen Juras, P.Geo. The estimates were made from a 3D block model utilizing commercial mine planning software. Projects limits, in UTM coordinates, are 686295 to 688175 East, 4260955 to 4262115 North, and 0 to +1110 m elevation. Block model cell size was 20 m east x 20 m north x 10 m high.

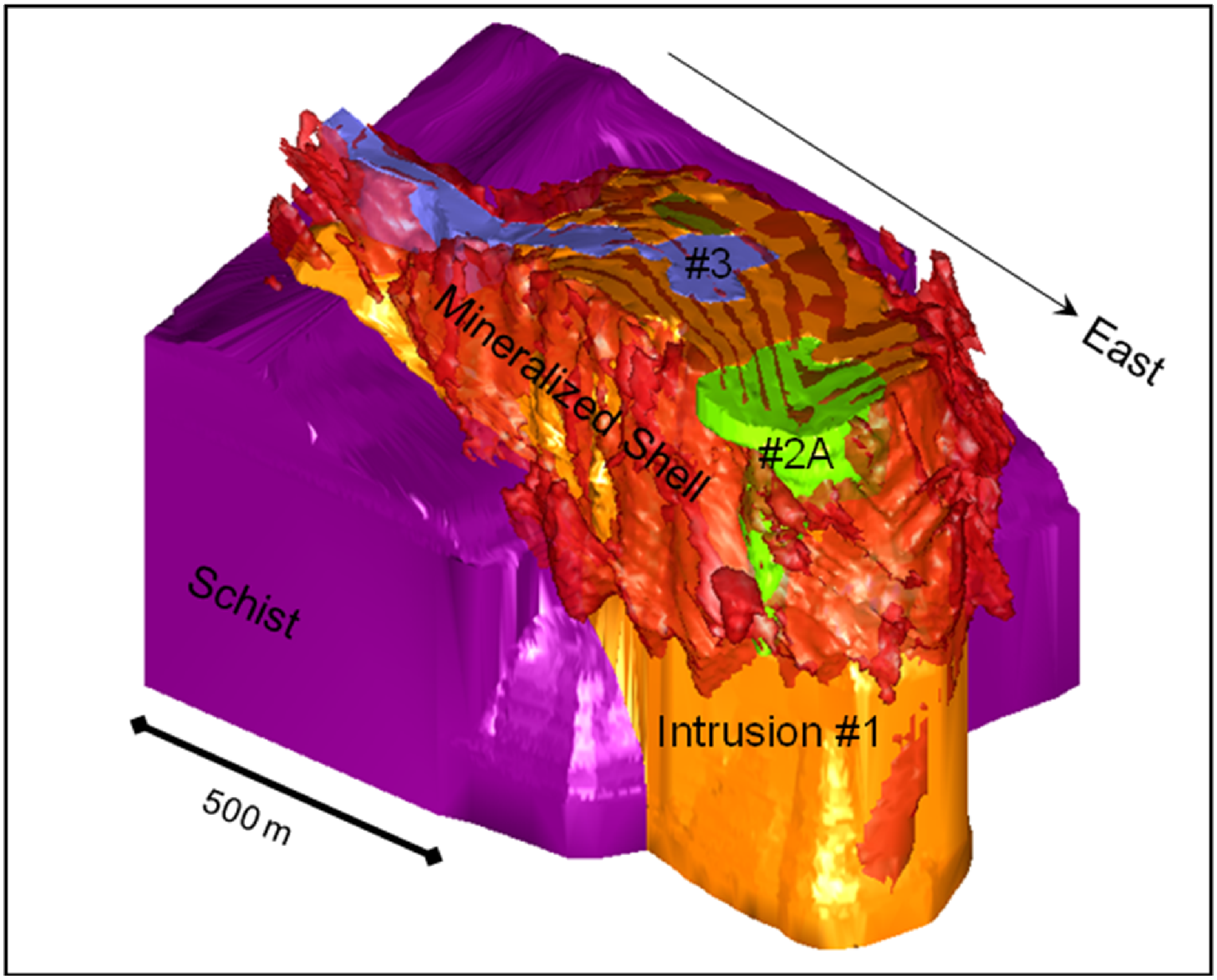
17.1.1 GEOLOGIC MODELS

Eldorado used significant new data from the last three years of deep drilling campaigns and mining to update the geologic model described in the previous Technical Report (Micon, 2003). The latest resource and reserve work incorporated new lithology models and an alteration model, all constructed in 3D. Most significant changes revolve around the principal gold-hosting unit, Intrusion #1, and the basement Schist unit. The neck of Intrusion #1 was found to occur at the south-eastern end of the deposit. Over the western half of the deposit, Intrusion #1 actually overlies the Schist unit in a sill-like configuration. Other changes include a much more irregular Intrusion #3 contact with Intrusion #1 resulting in a lower overall volume for Intrusion #3, the delineation of a second Intrusion #2 body in the south-eastern part of the deposit, and a conical shape that pinches with depth for Intrusion #2A.

Mineralized or grade shapes were also made. To constrain gold grade interpolation for the Kişladağ deposit, Eldorado created 3D mineralized envelopes, or shells. These were based on initial outlines derived by a method of Probability Assisted Constrained Kriging (PACK). The threshold value of 0.20 g/t Au was determined by inspection of histograms and probability curves as well as indicator variography. Shell outline selection was done by inspecting contoured probability values. These shapes were then edited on plan and section views to be consistent with the lithology model and the drill assay data so that the boundaries did not violate data and current geologic understanding of mineralization controls. Figure 17-1 shows the relationship between the PACK or mineralized shell and the lithology units.

All generated 3D shapes were checked for interpretational consistency on section and plan, and found to have been properly constructed. The shapes honoured the drill data and appear well constructed.

Figure 17-1: Relationship between the PACK or Mineralized Shell and Lithology Units



17.1.2 DATA ANALYSIS

The lithologic and mineralized domains were reviewed to determine appropriate estimation or grade interpolation parameters. Several different procedures were applied to the data to discover whether statistically distinct domains could be defined using the available geological objects. The lithology categories were investigated within and outside the mineralized shell.

Descriptive statistics, histograms and cumulative probability plots, box plots and contact plots have been completed for gold in the Kişladağ deposit. Results obtained were used to guide the construction of the block model and the development of estimation plans. The data analyses were conducted on 5 m down-hole composited assay data. The statistical properties from this analysis are summarized in Table 17-1.

Table 17-1: Kişladağ Deposit Statistics for 5 m Composites – Au g/t Data

Lithology	Mean	CV	Q25	q50	q75	Max	No. of Comps
Within PACK Shell							
Intrusion #1	1.03	0.90	0.45	0.75	1.31	17.17	7,447
Intrusion #2	0.65	0.76	0.37	0.54	0.75	4.50	513
Intrusion #2A	0.49	0.68	0.29	0.42	0.60	4.79	954
Intrusion #3	0.34	0.69	0.21	0.28	0.41	2.20	421
Pyroclastics	0.34	0.72	0.21	0.28	0.40	3.56	1,392
Schist	0.30	0.51	0.21	0.26	0.34	1.44	203
Outside PACK Shell							
Intrusion #1	0.15	0.93	0.08	0.12	0.18	1.23	155
Intrusion #2	0.13	0.67	0.07	0.12	0.16	0.26	26
Intrusion #2A	0.11	0.56	0.06	0.11	0.16	0.32	97
Intrusion #3	0.11	0.91	0.06	0.09	0.14	1.59	633
Pyroclastics	0.08	1.18	0.02	0.06	0.11	2.27	3,255
Schist	0.09	0.77	0.04	0.08	0.13	0.51	157

Gold grades are highest and most prevalent in Intrusion #1. Younger units Intrusions #2 and #2A are also mineralized but at more uniform lower values (means equalling 0.65 and 0.49, respectively). Intrusion #3 and the Pyroclastics unit generally contain weak to no gold mineralization thus have higher numbers of composites falling outside the mineralized shell. Gold mineralization occurring above the mineralized shell threshold in these units lie along the contact area with the other intrusive units, mainly Intrusion #1. Generally, the coefficient of variance (CV) values of all units are relatively low reflecting the porphyry style mineralization of the deposit.

Contact profiles or plots, generated to explore the relationship between grade and lithology units, allow for a graphical representation of the grade trends away from a "contact." All contact relationships within the PACK shell were investigated. Most show a small transitional zone of 10 to 15 m along the respective contacts. Beyond 20 m from a contact, though, the relationships become distinct and support treating each unit as separate domains for grade interpolation.

Estimation Domains

The data analyses demonstrated that the lithologic units within the gold mineralized shell should be treated as separate domains. Grades for blocks within the respective domains will be estimated with a hard boundary between them; only composites within the domain will be used to estimate blocks within the domain. For units outside the mineralized shell, only the schist and Intrusion #3 should be treated as separate estimation domains. The remaining units will be treated as having "soft" boundaries.

17.1.3 EVALUATION OF EXTREME GRADES

Extreme grades were examined for gold, mainly by histograms and cumulative probability plots. Generally, the distributions do not indicate a problem with extreme grades for gold. Less densely drilled areas of the deposit required the use of outlier restricted grades to prevent the possibility of grade smearing. This is described in the estimation section below.

17.1.4 VARIOGRAPHY

Variography, a continuation of data analysis, is the study of the spatial variability of an attribute. Eldorado prefers to use a correlogram, rather than the traditional variogram, because it is less sensitive to outliers and is normalized to the variance of data used for a given lag. Correlograms were calculated for gold in the main domains: Intrusion #1, Intrusion #3 and Pyroclastics. Variogram model parameters and orientation data of rotated variogram axes are shown in Tables 17-2 and 17-3.

Gold in Intrusion #1 displays two structures: a long ranged, E-W trending, moderately steep N dipping, and gently E plunging structure and a more cylindrical, N-S trending, gently SE plunging shorter ranged structure. The Pyroclastics unit displays similar trending structures, but with much shorter ranges. The nugget effects for both are low. Gold in Intrusion #3 contains a high nugget value. Its modelled structures contain small ranges and mirror the shape of this unit, i.e., steeply, southwest plunging.

17.1.5 MODEL SETUP

The block size for the Kişladağ model was selected based on mining selectivity considerations (open pit mining). It was assumed the smallest block size that could be selectively mined as ore or waste, referred to the selective mining unit (SMU), was approximately 20 m x 20 m x 10 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 5 m fixed-length down-hole composites. The composite data were back-tagged by the mineralized shell and lithology units (on a majority code basis). The compositing process and subsequent back-tagging was reviewed and found to have performed as expected.

Bulk density data were assigned to a unique assay database file. These data were composited into 10 m fixed-length down-hole values. This compositing honoured the lithology domains by breaking the composites on the domain code values.

Various coding was done on the block model in preparation for grade interpolation. The block model was coded according to lithologic domain and mineralized shell (on a majority code basis). Percent below topography was also calculated into the model blocks.

Table 17-2: Au Variogram Parameters for Kişladağ Deposit

	Model	Nugget Co	Sills		Rotation Angles						Ranges					
			C1	C2	Z1	X1'	Y1"	Z2	X2'	Y2"	Z1	X1'	Y1"	Z2	X2'	Y2"
Intrusive (#1, #2, #2A) Domain	SPH	0.367	0.329	0.304	-2	10	-4	-11	25	17	71	30	156	530	127	236
Pyroclastics Domain	SPH	0.176	0.515	0.309	-17	16	-34	-154	63	84	15	24	106	208	24	13
Intrusion #3 Domain	EXP	0.649	0.351	-	-4	72	-123	-	-	-	13	81	107	-	-	-

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

Table 17-3: Azimuth and Dip Angles of Rotated Variogram Axes, Kişladağ Deposit

	Axis Azimuth						Axis Dip					
	Z1	X1	Y1	Z2	X2	Y2	Z1	X1	Y1	Z2	X2	Y2
Intrusive (#1, #2, #2A) Domain	155	87	358	206	87	349	79	-4	10	60	16	25
Pyroclastics Domain	96	63	343	111	20	206	53	-32	16	3	27	63
Intrusion #3 Domain	54	321	356	-	-	-	-10	72	-15	-	-	-

Notes: Azimuths are in degrees. Dips are positive up and negative down.

A near surface oxidation of sulphide minerals has occurred at Kişladağ. Since leaching recoveries differ between the oxidized and primary mineralized rock, the boundary needs to be known for reserve conversion work. Past models used an interpreted oxide surface. Since the start of production, the oxide – primary boundary is defined through modelled total S % and ppm Co. The abundance of the latter element was found to be sensitive to the destruction of pyrite thus correlative to indentifying oxidized areas. Modelled Co values also helped to distinguish between S values due to sulphide (i.e., pyrite) and S concentrations due to sulphates (alunite and barite).

17.1.6 ESTIMATION

Modelling consisted of grade interpolation by ordinary kriging (OK) for all domains inside the mineralized shell and inverse distance weighting to the second power (ID2) for background model blocks. Nearest-neighbour (NN) grades were also interpolated for validation purposes. Blocks and composites were matched on estimation domain.

The search ellipsoids were oriented preferentially to the orientation of the respective domain as defined by the attitude of the gold grade shell and structures defined in the spatial analysis. Searches had the longest ranges for Intrusion #1 (130 to 350 m) and shortest for Intrusion #3 (20 to 80 m). Block discretization was 4 m x 4 m x 2 m.

A two-pass approach was instituted for interpolation. The first pass required a minimum of two holes from the same estimation domain whereas the second pass allowed a single hole to place a grade estimate in any uninterpolated block from the first pass. This approach was used to enable most blocks to receive a grade estimate within the domains, including the background domains. Blocks received a minimum of 2 to 3 and maximum of 3 to 4 composites from a single drill hole (for the two-hole minimum pass). Maximum composite limit ranged from 9 to 12.

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 in the validation section below.

In all domains, an outlier restriction was used to control the effects of high-grade composites in local areas of less dense drilling, particularly in background domains and poorly mineralized units (e.g., Intrusion #3). The threshold grades were generally set close to the threshold grade of the PACK shell in the case of the background domains, or through inspection of the cumulative probability plots for the mineralized units. Mineralized domains in Intrusion #1, #2 and #2A used an outlier of 7.0 g/t Au, whereas mineralized Intrusion #3 and Pyroclastics units used 1.2 and 1.5 g/t Au, respectively. All background domains used a 0.5 g/t Au outlier grade except for Intrusion #3, where the outlier grade equalled 0.3 g/t Au. The restricted distance was 50 m.

Bulk density values were estimated into the resource model by an averaging of composites. A maximum of six and minimum of two 10 m composites were used for the averaging. A rectangular search was used, measuring 200 m north x 125 m east x 50 m elevation. In the event a block was not estimated, default density values were assigned based on lithology and oxidation code.

Values for total S and ppm Co were interpolated by an averaging of composites using a rectangular search similar to that used for the bulk density. Estimation was constrained by matching model blocks to composites from the same lithology domain for Co only.

17.1.7 VALIDATION

Visual Inspection

Eldorado completed a detailed visual validation of the Kişladağ resource model. The model was 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 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 and, in general, all background domains. Examples of representative sections and plans containing block model grades, drill hole composite values, and domain outlines are included in Appendix B.

Model Check for Change-of-Support

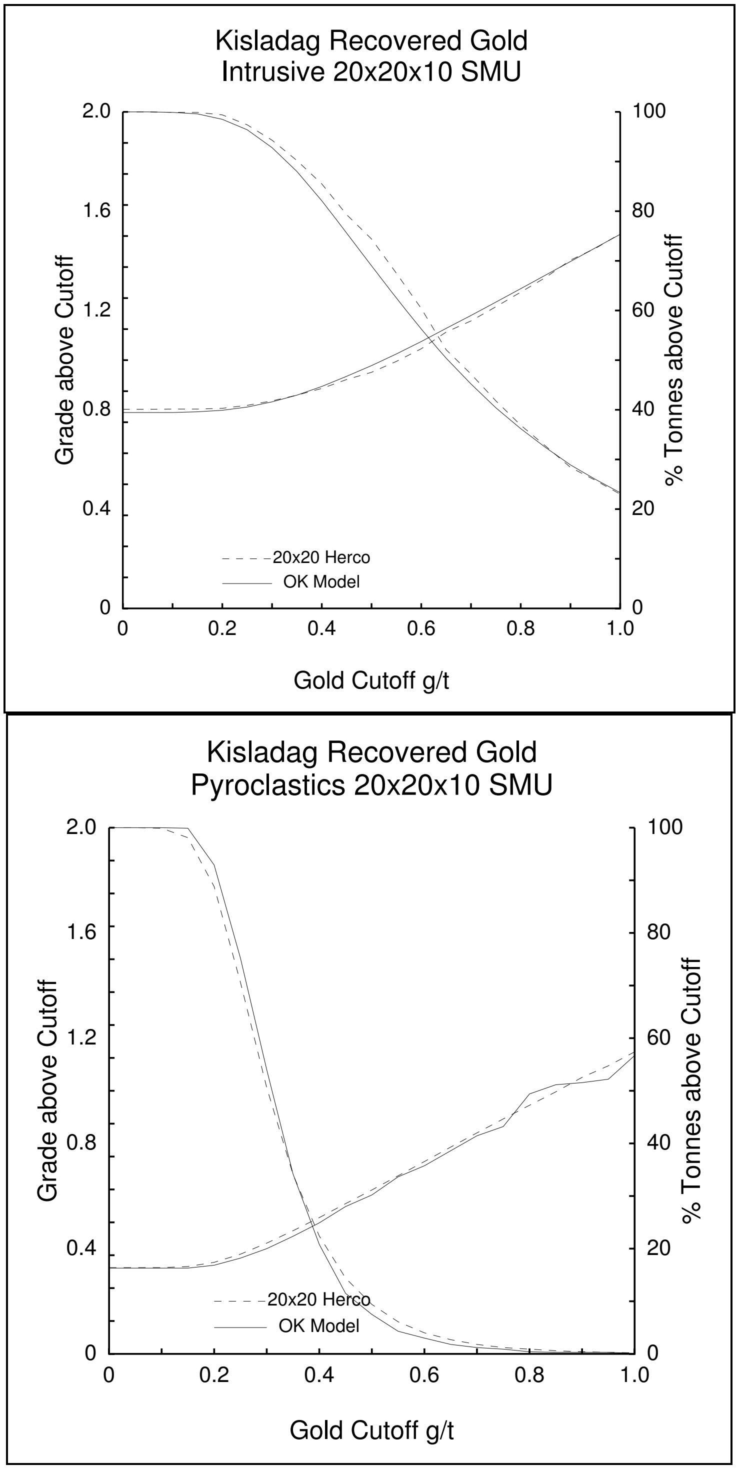
An independent check on the smoothing in the estimates was made using the Discrete Gaussian or Hermitian polynominal change-of-support method. This method uses the “declustered” distribution of composite grades from a nearest-neighbour or polygonal model to predict the distribution of grades in blocks. The histogram for the blocks is derived from two calculations:

- the block-to-block or between-block variance
- the frequency distribution for the composite grades transformed by means of hermite polynomials (Herco) 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 20 m x 20 m x 10 m block grades for gold in the Intrusion #1 and Pyroclastics mineralized domains is shown with dashed lines on the grade-tonnage curves in Figure 17-2 overleaf. This is the distribution of grades obtained from the change-of-support models. 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 range of mining grade cutoff values (0.3 g/t to 0.5 g/t Au).

Figure 17-2: Herco Plots for Intrusives Domain (top) and Pyroclastics Domain (bottom)



Model Checks for Bias

The block model estimates were checked for global bias by comparing the average metal grades (with no cutoff) from the model with means from nearest-neighbour estimates (NN). (The nearest-neighbour estimator declusters the data and produces a theoretically unbiased estimate of the average value when no cutoff grade is imposed and is a good basis for checking the performance of different estimation methods.) Results, summarized in Table 17-4, show no problems with global bias in the estimates.

Table 17-4: Global Model Mean Gold Values by Mineralized Shell Domain

Domain	NN Estimate	Kriged Estimate	% Difference
Intrusion #1	0.825	0.846	+2.5
Intrusion #2	0.584	0.598	+2.3
Intrusion #2A	0.512	0.509	-0.7
Pyroclastics	0.344	0.340	-1.1
Intrusion #3	0.289	0.295	+1.8
Schist	0.208	0.207	-0.7

The model was also checked for local trends in the grade estimates by grade slice or swath checks. This was done by plotting the mean values from the nearest-neighbour estimate versus the kriged results for benches (in 5 m swaths) and for northings and eastings (both in 20 m swaths). The kriged estimate should be smoother than the nearest-neighbour estimate, thus the nearest-neighbour estimate should fluctuate around the kriged estimate on the plots. The observed trends behave as predicted and show no significant trends of gold in the estimates in Kişladağ model.

17.1.8 MINERAL RESOURCE CLASSIFICATION

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

Inspection of the Kişladağ model and drill hole data on plans and sections, combined with spatial statistical work and investigation of confidence limits in predicting planned annual and quarterly production, contributed to the setup of various distance to nearest composite protocols to help guide the assignment of blocks into Measured or Indicated mineral resource categories. Reasonable grade and geologic continuity is demonstrated over most of the Kişladağ deposit, which is drilled generally on 40 m to 80 m spaced sections. A two-hole rule was used where blocks containing an estimate resulting from two or more samples, all within 80 m and from different holes, were classified as Indicated mineral resources. Where the sample spacing was about 50 m or less, the confidence in the grade estimates and lithology contacts were the highest thus permissive to be classified as Measured mineral resources. A three-hole rule was used where blocks containing an estimate resulting from three or more samples, all within 50 m and from different holes, were classified as Measured mineral resources.

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

A test of reasonableness for the expectation of economic extraction was made on the Kişladağ mineral resources by developing a series of open pit designs based on optimal operational parameters and gold price assumptions. Those pit designs enveloped most of the Measured and Indicated mineral resources thus demonstrating the economic reasonableness test for the new estimate and reporting cutoff grade of the Kişladağ mineral resources.

17.1.9 MINERAL RESOURCE SUMMARY

The Kişladağ mineral resources as of 31 December 2009 are shown in Table 17-5. The Kişladağ mineral resource is reported at a 0.3 g/t Au cutoff grade and calculated to end of 2009 mining limits. The cutoff grade differs from the 0.40 g/t value used in the previous Technical Report (Micon, 2003) and is supported by current mining practice and a markedly higher gold price regime.

Table 17-5: Kişladağ Mineral Resources, as of 31 December 2009

Mineral Resource Category	Tonnes (x '000)	Grade (Au g/t)	In Situ Gold (oz x '000)
Measured	82,904	0.93	2,490
Indicated	329,345	0.74	7,783
Measured+Indicated	412,249	0.78	10,273
Inferred	182,083	0.50	2,950

17.2 MINERAL RESERVES

Richard Miller, Manager Mine Engineering of Eldorado completed the reserve estimates in this report.

17.2.1 GEOTECHNICAL CHARACTERISTICS AND DESIGN

Kişladağ has been in operation since 2006 and detailed geo-technical analysis and monitoring has been ongoing since that time.

The ground conditions at Kişladağ are highly variable. Zones of geotechnical importance include the weathering profile that divides the oxide and sulphide horizons, the three intrusions, which have different alteration profiles and structural characteristics, and a series of late state brittle deformations called friable zones. These major zones are also affected by a local rock mass fabric, which includes multiple joint sets of varying persistence and orientation.

The open pit slopes have been monitored on a continuous basis since the start of operations. The monitoring program consists of measurements of slope displacement using prisms, changes in ground water levels using piezometers, regular inspections of the berms and highwalls and development of a hazard map for mine operations. A slope radar system is also planned, as the mine gets deeper.

Since 2007, the mine has conducted 18 oriented geo-technical core holes for a total of 5,500 m of drilling. From these holes a total of 384 UCS tests, 25 tri-axial tests, and 69 direct shear tests have been conducted. In addition to the geo-technical drilling, all resource drill holes have been logged for geo-tech parameters such as RQD, lithology, RMR and the geotechnical blockiness index. The Kişladağ mine staff has incorporated these aspects into a 3D model. This 3D model and the parameters gathered during the testing program are used in numerical analysis to predict the deformations and strain of a given slope design.

The open pit slopes have been designed by HTA Geotechnical Consulting based out of South Africa. The designs incorporate numerous aspects of geo-technical behaviour and predicted failure modes. These include double and single stacking of benches, geo-technical berms, limiting multiple bench stacks, de-pressurization of the slopes using horizontal boreholes, pre-split blasting along varying bench face angles and limiting of the overall slope angle for a given slope height.

The geo-technical designs incorporated in the Kişladağ reserves are divided into the sectors which are show in Figure 17-3. These sectors each have a different set of design criteria throughout the sector depending on weathering and the proximity to friable zones. An overall summary of the geotechnical designs parameters is shown in Table 17-6.

Figure 17-3: Kişladağ Geotechnical Design Sectors

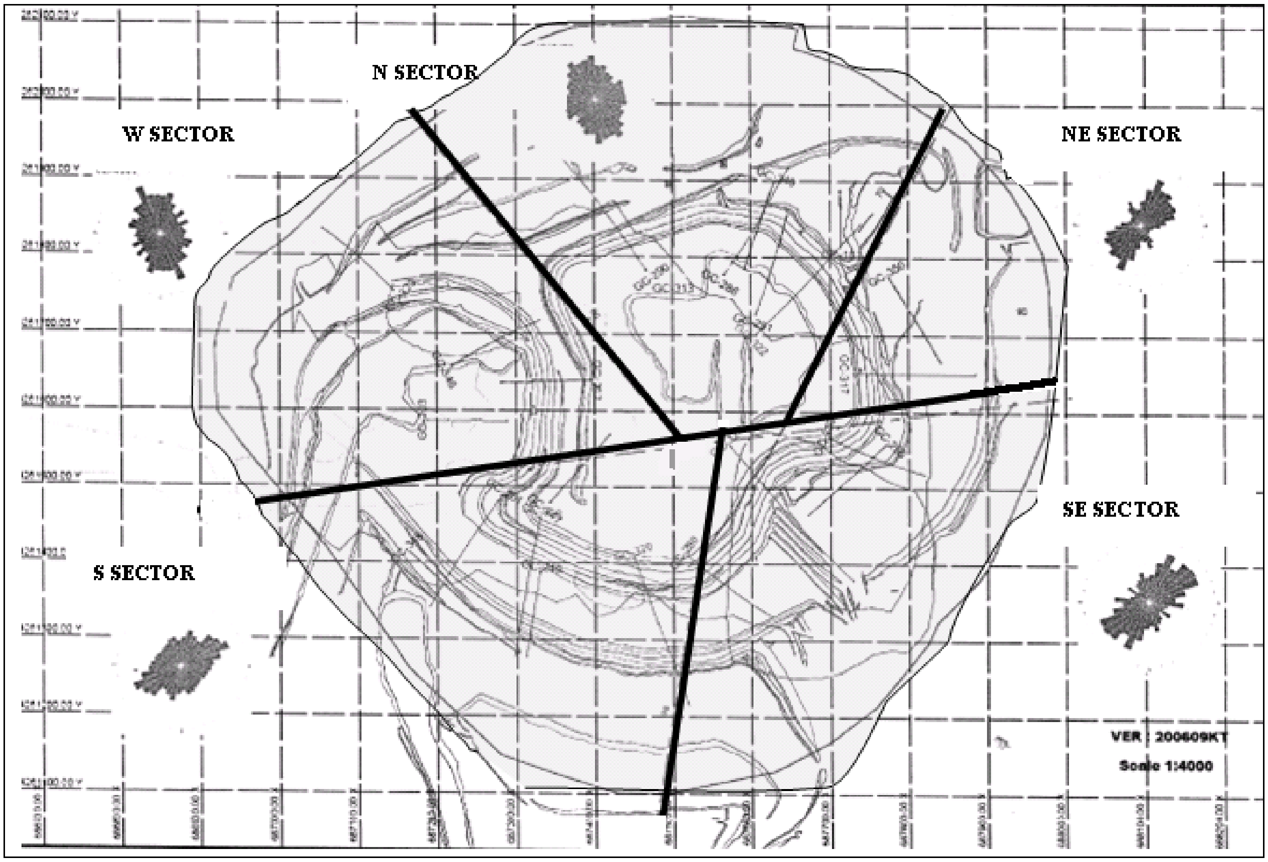


Table 17-6: Summary of Kişladağ Geotechnical Design Parameters

Sector	Oxide or Sulphide	Bench Face Angle	Inter-ramp Angles	Overall Slope Angle
N	Oxide	65	39-49	44.5
	Sulphide	70-75	50-55	
NE	Oxide	65	39-49	43.7
	Sulphide	70-75	49-56	
SE	Oxide	65-70	39-49	42.4
	Sulphide	70-75	51-56	
S	Oxide	65	39-51	44.7
	Sulphide	70-75	51-56	
W	Oxide	65	39-51	48.5
	Sulphide	70-75	51-56	

17.2.2 MINERAL RESERVE STATEMENT

The remaining mineral reserves as stated in Table 17-7 are contained within the designed final pit as of 31 December 2009. The design pit was based on the results of an optimization done using Whittle software and includes geotechnical parameters as outlined by the most recent report from HTA Geotechnical Consulting, dated November 2009.

Table 17-7: Kışladağ Mineral Reserves, as of 31 December 2009

Mineral Resource Category	Tonnes (t x '000)	Grade (Au g/t)	In Situ Gold (oz x '000)
Proven	68,230	1.05	2,312
Probable	149,240	0.94	4,504
Proven + Probable	217,470	0.97	6,816

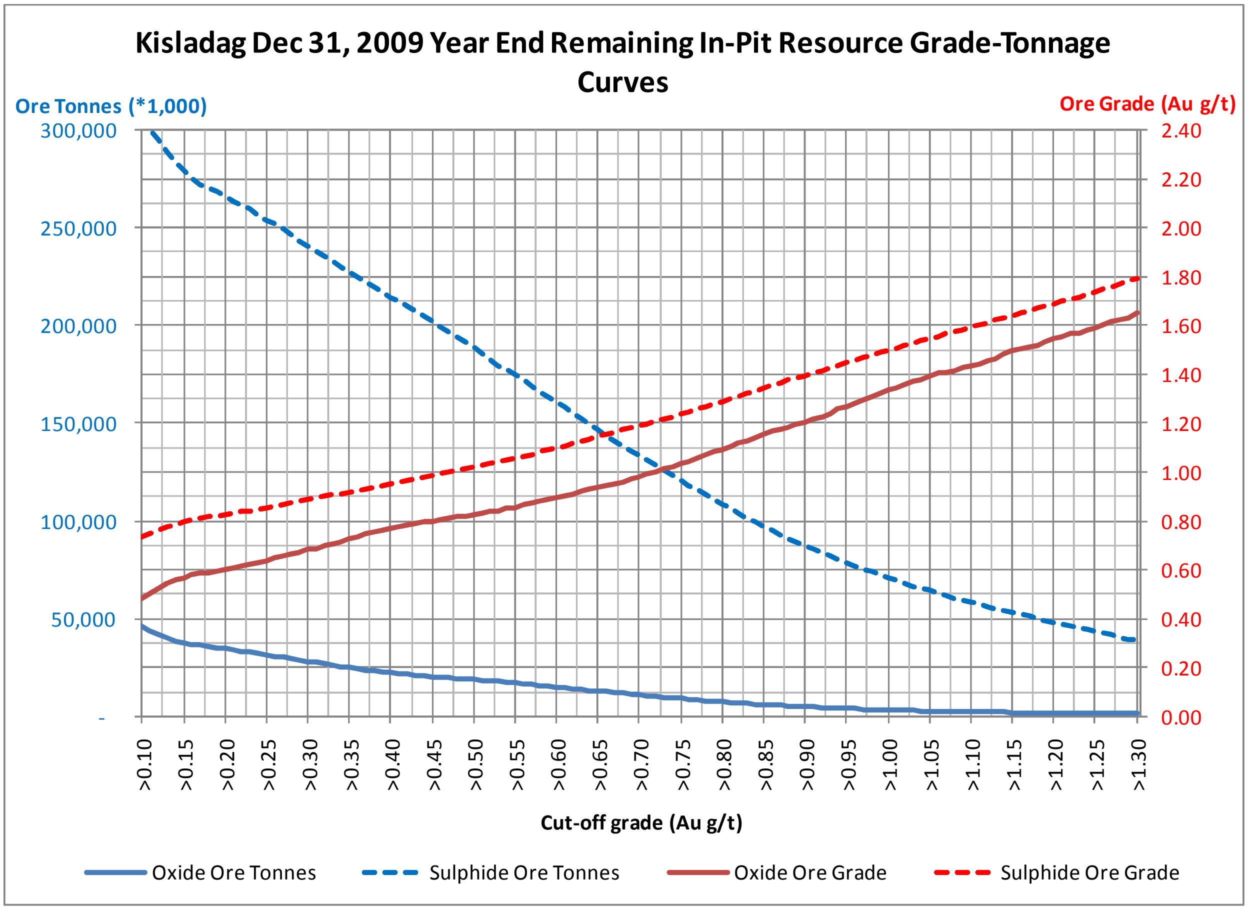
17.2.3 MINERAL RESERVE CHARACTERISTICS

Of the total proven and probable reserves, a total of 25,694,000 tonnes is oxide ore at a grade of 0.72 g/t and 191,776,000 tonnes is sulphide ore at a grade of 1.01 g/t based on a cutoff grade of 0.35 g/t Au for oxide ore and 0.50 g/t Au for sulphide ore.

Ore loss and waste dilution have been accounted for in the block model and therefore do not affect the reserve. Historical reconciliations as mentioned in Section 18 substantiate this principle.

The grade-tonnage curves for measured and indicated oxide and sulphide resource blocks within the final pit are shown in Figure 17-4.

Figure 17-4: Grade-Tonnage Curve



17.2.4 OPTIMIZATION PARAMETERS

Table 17-8 lists the parameters used in the pit optimization.

The cost estimates were current as of October 2009 and are reflective of anticipated costs for the life of the operation. Processing costs were reflective of the ore type being treated with sulphide ore having more reagent usage and therefore a higher unit cost. Likewise, metallurgical recovery was also calculated separately by rock type based on historic testwork.

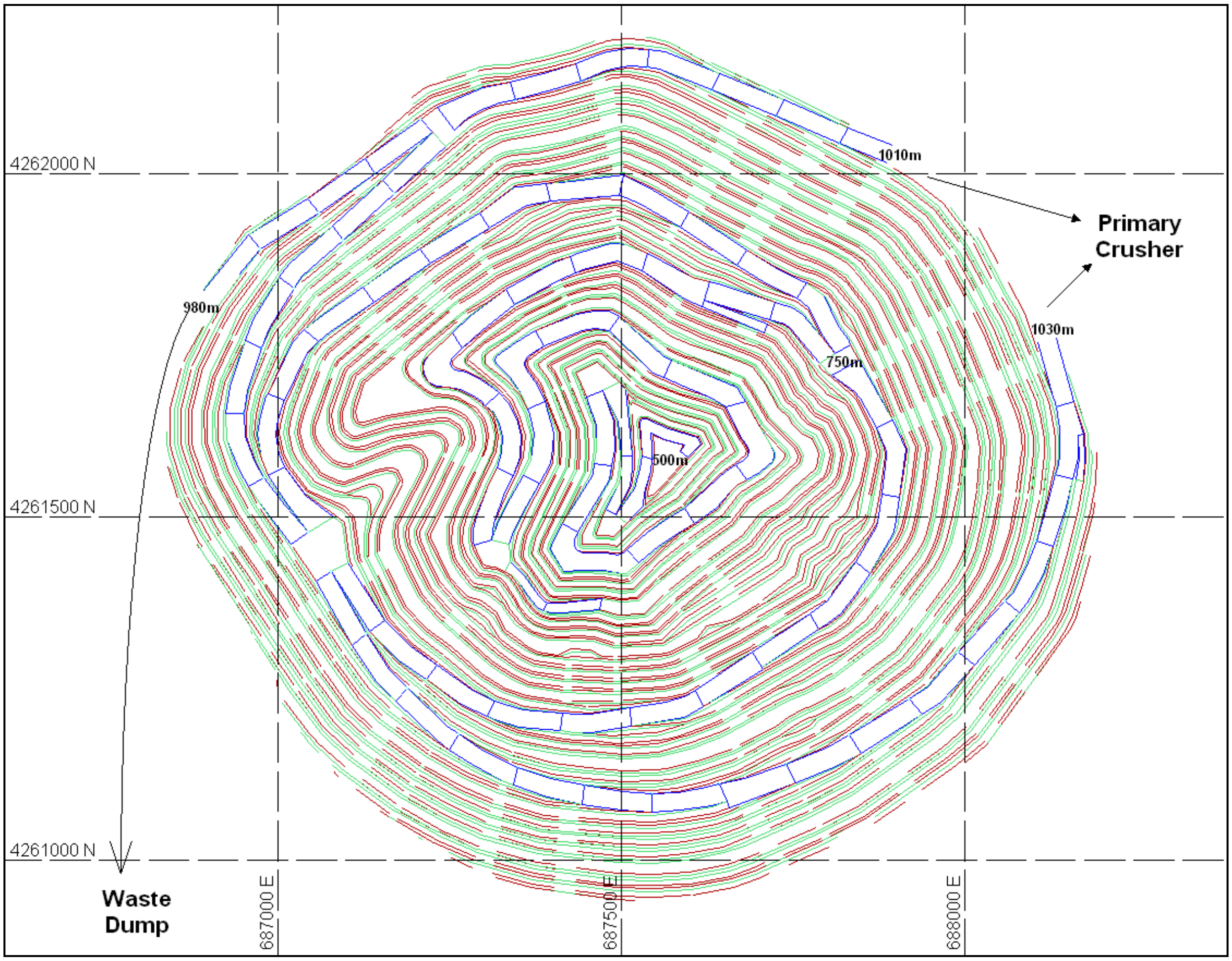
Table 17-8: Pit Optimization Parameters

Item	Unit	Value	Comments
Gold price	US$/oz	825	-
Production rate	Mt/a	12	12
Overall limiting pit slope angle	Degrees	42.4 to 48.5	Variable by sector as defined by HTA 2009
Base mining cost	US$/t mined	1.07	All-in Mine Department cost for a base unit of ore from 1000m bench to the primary crusher
Additional cost per bench	US$/t/10 m bench	0.03	Applies to all rock below 1,000 m elevation
Additional cost for waste	US$/t	0.15	Applies to all waste due to extra hauling costs
Mining dilution	%	0	Applicable mining dilution is built into the block model
Mining ore loss	%	0	Applicable mining ore loss is built into the block model
G&A cost	US$/t	0.94	Unitized cost per tonne of ore
Processing cost for oxide ore	US$/t	2.76	All-in Process Department unit cost for oxide ore per tonne treated
Processing cost for sulphide ore	US$/t	3.97	All-in Process Department unit cost for sulphide ore per tonne treated
Cut-off grade for oxide ore	g/t Au	0.35	-
Cut-off grade for sulphide ore	g/t Au	0.50	-
Oxide ore metallurgical recovery	%	80	-
Sulphide ore metallurgical recovery	%	60	-
Transport, refining & royalty (Net of Ag credit)	US$/oz	4.64	-

17.2.5 PIT DESIGN CRITERIA

The open pit was designed using Gemcom GEMS software based on a 10 m bench height with double benching for most pit walls. Berm width, face angle, and bench stack heights varied by sector and rock quality. Spill berm widths varied from 6.7 m to 9 m and geotechnical berms of 12.5 m to 28.0 m widths were used to separate bench stacks and satisfy the overall slope angle limitations. Inter-ramp angles varied from 39° to 56°. The upper half of the pit (above 750 m elevation) has a double ramp network and the lower half of the pit is limited to a single ramp. Ramps were designed with a minimum width of 26.3 m for the two-way traffic ramps and a minimum width of 16 m for the single lane ramp used only for the bottom four benches. The pit exposure on surface ranges in elevation from 960 m to 1,080 m and the pit extends down to a bottom elevation of 500 m. The entire pit has a surface footprint of 125 ha (Figure 17-5 and Appendix B).

Figure 17-5: Final Pit Design



SECTION 18 • OTHER RELEVANT DATA AND INFORMATION

18.1 RECONCILIATION

Over more than three years of history, the life to date reconciliation of calculated contained gold being placed on the pad for treatment versus the resource block model being used for the respective period has a full 100% reconciliation of resource gold ounces, inclusive of any mining losses such as ore recovery or dilution. The resource block modelling has shown a history of providing very reliable total contained gold predictions with a tendency for the ore tonnage being slightly under-predicted and ore grade being slightly over-predicted (Table 18-1).

Table 18-1: Historical Ore Reconciliation

Period	Resource Block Model Ore (t '000)	Resource Block Model Ore Grade (Au g/t)	Resource Block Model Ore (Au oz)	Grade Control Ore (t '000)	Grade Control Ore Grade (Au g/t)	Grade Control Ore (Au oz)	Ore Placed on Leach Pad (t '000)	Ore Placed on Leach Pad Grade (Au g/t)	Ore Placed on Leach Pad (Au oz)
Pre-2008	10,246	1.14	377,052	10,756	1.19	411,511	10,504	1.21	409,430
2008	7,357	1.33	314,973	8,048	1.25	323,443	7,556	1.27	308,029
2009	10,525	1.20	406,346	10,550	1.13	383,303	10,717	1.11	383,343
LTD	28,128	1.21	1,098,371	29,354	1.18	1,118,257	28,777	1.19	1,100,802
% of Resource Block Model				104%	98%	102%	102%	98%	100%

Although recoveries are difficult to confirm on a large-scale heap leach project, gold produced to date supports the recoveries indicated in the original study. Column tests on monthly composites support the high recoveries (80%) for the oxide ore and variable recoveries in the sulphide ore with an average sulphide recovery in excess of 60%. At this stage, there is no evidence that the forecasted recoveries will not be met or exceeded over the life of the operation.

18.2 CLOSURE PLAN

The Kişladağ Project EIA study was completed in January 2003 and has been submitted to the Turkish Authorities. An Environmental Positive Certificate was obtained in June 2003. A Preliminary Closure Plan, based on the closure strategy presented in the EIA report, has been prepared by The Mines Group in August 2007. The Mine Reclamation Plan was prepared based on the EIA report and the preliminary closure plan submitted to the Relevant Government Authority in August 2008 as part of relevant regulation requirement.

Post closure monitoring is consisting of quarterly sampling of all pad drain-down solutions and rates, pond solutions, monitoring wells and the leak detection systems for a period of five years following closure of the facilities. The post closure monitoring will mirror that of operations until it can be demonstrated that there is very little likelihood of changes occurring in the geochemistry of the surrounding ground water system in the open pit, overburden disposal dumps, the leach pad or leach pad drain-down solutions.

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

19.1 MINE PLAN AND PRODUCTION

19.1.1 INTRODUCTION

The Kışladağ Deposit is located in Western Turkey approximately 180 km east of the city of Izmir and 35 km south-west of the provincial capital of Uşak. It is situated in the western edge of the Anatolian plateau at an elevation of 1,000 masl.

19.1.2 PIT DEVELOPMENT

The final reserve pit has been designed to economically extract the oxide and sulphide resources that are convertible to ore reserves. It extends about 1,250 m from north to south and 1,350 m from east to west.

Pit designs have been completed for four mining phases, namely the initial pit, two intermediate pits and a final pit. The four phases are based on Whittle pit shells while considering practical mining widths for the intermediate pits.

The lowest bench in the initial pit is at elevation 870 masl. The pit wall depth measured from this bench to the highest point of the rim is 210 m. The final pit bottom is at elevation 500 masl, for a total depth of 580 m below the pit rim.

19.1.3 MINING COSTS

Kışladağ is an operating mine with well-known mining costs that however can vary over time subject to unpredictable fluctuations in items like fuel costs, currency exchange rates, local labour indices, etc. Actual costs from 2009 were used as the main basis for the future cost estimates.

19.1.4 MINING RISKS AND OPPORTUNITIES

The ore scheduling is based on the treatment of 12 Mt of ore per year from 2011. Current permitting and crushing capacities are at 10 Mt of ore per year. An application for increasing the permitted capacity to 12 Mt/a will be submitted early in 2010 and is expected to be accepted in time for the planned increase. Kışladağ has received budget approval for modifications to the crushing, screening and conveying infrastructure to complete the modification to produce at 12 Mt/a by the end of 2010 with no net loss in production during 2010. The timing of the permitting and/or the timing and success of the infrastructure upgrading can pose a risk to the ore schedule.

The rock dump has been designed by Norwest Consulting during the feasibility stage and has been updated by Norwest Consulting with all previous annual adjustments to the open pit reserves. The rock dump design update for the open pit design of this report is currently in progress. All previous redesigns were shaped into the originally permitted footprint of the rock dump. The new reserve will require a larger rock dump. There is sufficient scope in the project license area for disposal of additional rock.

The pit walls have been designed using the best information available at the time. It is known that there are zones within the rock that are friable and that these zones pose risk to the wall stability. When encountering these zones in the pit walls a degree of flattening the slope is required to mitigate that risk. Extensive work was done in 2007 to 2009 to characterize and outline these zones in three dimensions. Concurrently in 2009, the drill program extended the measured and indicated resources and resultantly the economical pit grew in size. The known friable zones were then extrapolated long distances out to the new reserve pit for the design. Further drilling will help outline the friable zones at the final pit walls. There is a risk (and in the same context an opportunity) that the friable outline used may be more (or less) extensive than modelled at the final pit walls.

A diamond-drilling program consisting of 20,000 m of exploration holes has been budgeted for 2010 and is currently in progress. This program is intended to test inferred mineralisation near the depths of the final pit and, if successful, may increase the measured and/or indicated resources. Any increase such as this in the measured or indicated resources may ultimately also increase the ore reserves.

19.1.5 GEOTECHNICAL CHARACTERISTICS

HTA Geotechnical Consulting from South Africa provided geotechnical input for the overall slope geometries. Their reports address pit material characteristics, highwall slope stability analysis and provide geotechnical design criteria for the open pit. A summary of this is presented below:

Pit Slope Design and Stability Analysis

Geotechnical pit slope design and stability analysis was conducted on interpreted likely geotechnical conditions based on the following information sources:

- current geological information (lithology, weathering, alteration)
- detailed geotechnical logging of boreholes and mapping of excavated pit faces
- structural data obtained from the drill holes and pit face mapping
- modelling of the open pit in terms of geotechnical blockiness as friable, blocky and semi-massive ground
- results of laboratory testing
- slope stability analysis for potential sliding or rotational failures with different material (rock) types and groundwater pressure conditions incorporated into the analysis model

- assigning generalised material properties to each rock type for the analysis based on geological interpretation of the logs, strength estimates, Hoek-Brown criterion and laboratory test results
- bench and berm inspections
- slope prism monitoring data for displacement
- rock failure reports
- rainfall and ground water level measurements
- accelerated weathering tests.

The information listed above was used for the geotechnical pit slope design and the following parameters were obtained:

Pit development will be varying for five geotechnical sectors, namely, North (N), North-East (NE), South-East (SE), South (S) and West (W). 20 m high benches will be developed in two 10 m steps. The 20 m face height will be reduced in some locations, where ground type is expected as friable. Slope face angles will change from 65 to 75 degrees depending on the sectors and oxide-sulphide type of ground. Spill berm widths will vary from 6.7 m to 9 m. Geotechnical berm widths will be in the range of 12.5 m to 28 m. The overall slope angle of the final pit design is 41.9 degrees for N, 42.4° for NE, 41.3 degrees for SE, 43 degrees for S and 46.5 degrees for W geotechnical sectors. The intermediate pit phases are shallower than the final pit design.

Ground Water

Ground water level has been monitored since July 2007 by taking level measurements from the five piezometer holes around the open pit. The ground water level is currently around 860 masl. Drainage holes will be implemented in order to locally lower the phreatic surface for slope depressurizing.

Blast Vibration Analysis

The company has two blast vibration measurement devices for blast monitoring. Each blast in the mine is monitored by both devices. One device is used in the pit and the other device is used in one of the villages close to the mine (at about 880 m to the nearest point on the final pit boundary).

19.1.6 MINE PRODUCTION SCHEDULE

The mine production schedule was produced using the Gemcom GEMS program which links the block model and final pit design with the required production criteria.

Mining Schedule

The mine is scheduled to operate twenty four hours per day for seven days per week. The crusher operates the same schedule with the mine and is fed with ore by trucks on an almost exclusively direct feed basis with a small stockpile available for use when required.

A mine production rate of 5 Mt of ore per year was maintained for the first two years of mine life while using a mining contractor. Subsequently with replacing the mining contractor with larger owner equipment and personnel, the production rate has been increased to 10 Mt/a. In 2011 the production rate will be increased again to 12 Mt/a and kept at that rate for the remainder of the mine's life. The highest annual mining production rate occurs in Year 2013 with a total movement of 46.34 Mt (ore and waste).

Scheduling Objectives and Constraints

The principal objectives and constraints for the mine production schedule are as follows:

- to meet plant feed ore tonnage requirements
- to avoid periods of low annual gold production
- to avoid equipment congestion by constraining the number of benches mined each period
- to smooth total material mined each period
- to smooth truck requirements as haul distances increase and equipment availabilities decrease.

The mine production schedule is based on providing the crusher with 10 Mt of ore per year in 2010 and 12 Mt/a thereafter.

Annual Mine Plans

Table 19-1 shows a summary of the life-of-mine production plan according to ore and waste type.

Figure 19-1 shows the life-of-mine ore and waste production and ore grade by year.

Figure 19-2, Figure 19-3 and Figure 19-4 show some selected annual year ending bench configurations of the pit, as it is developed to illustrate the phasing of the pit development. Figure 19-5 shows the final pit.

Table 19-1: Life-of-Mine Production Type by Ore and Waste Type

Year	Oxide Ore		Sulp. Ore		Total Ore		Oxide Wst	Sulp. Wst	Total Wst	SR
	(tX1000)	Au(g/t)	(tX1000)	Au(g/t)	(tX1000)	Au(g/t)	(tX1000)	(tX1000)	(tX1000)	
2010	4,066	0.76	6,178	1.25	10,244	1.05	10,727	3,640	14,367	1.40
2011	3,476	0.81	8,574	1.15	12,051	1.05	11,719	8,299	20,017	1.66
2012	1,542	0.72	10,479	1.24	12,021	1.17	16,567	10,662	27,230	2.27
2013	4,430	0.60	7,583	1.21	12,013	0.99	19,492	14,835	34,327	2.86
2014	2,542	0.78	9,534	0.95	12,076	0.92	15,608	16,742	32,350	2.68
2015	310	0.70	11,697	1.24	12,008	1.23	14,220	18,037	32,257	2.69
2016	117	0.45	11,898	1.10	12,015	1.09	11,410	19,628	31,039	2.58
2017	74	0.48	11,973	1.00	12,046	0.99	12,086	18,890	30,975	2.57
2018	130	0.48	11,875	1.17	12,005	1.16	8,438	19,502	27,940	2.33
2019	118	0.96	11,893	1.19	12,011	1.19	5,125	20,464	25,589	2.13
2020	33	0.87	11,970	0.82	12,003	0.82	269	23,140	23,408	1.95
2021	681	0.79	11,333	0.76	12,014	0.76	63	20,669	20,732	1.73
2022	72	0.63	11,936	0.79	12,008	0.79	63	12,034	12,097	1.01
2023	224	0.74	11,794	0.89	12,018	0.89	63	7,410	7,473	0.62
2024	803	0.70	11,203	1.02	12,006	1.00	63	4,898	4,961	0.41
2025	1,008	0.70	10,992	0.95	12,001	0.93	72	3,087	3,159	0.26
2026	1,946	0.73	10,062	0.85	12,008	0.83	63	2,629	2,691	0.22
2027	3,842	0.68	8,169	0.80	12,011	0.76	63	2,475	2,537	0.21
2028	279	0.62	2,633	0.71	2,912	0.70	63	521	583	0.20
Total	25,694	0.72	191,776	1.01	217,470	0.97	126,170	227,562	353,732	1.63

Figure 19-1: Life-of-Mine Ore and Waste Schedule

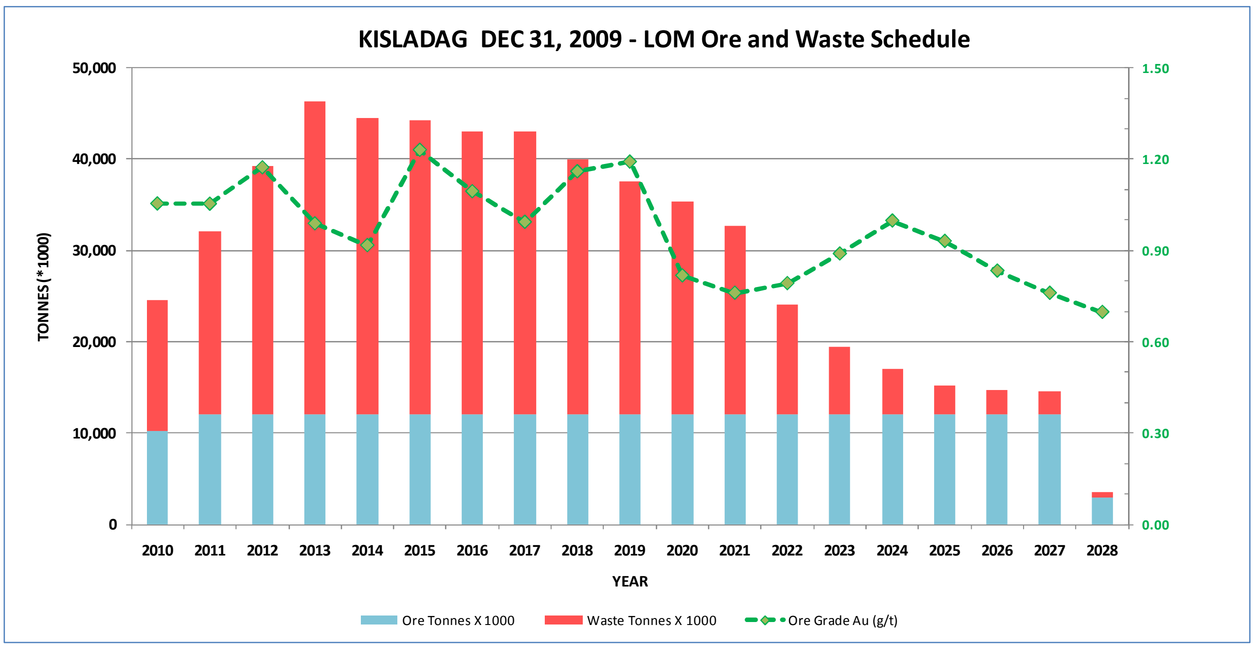


Figure 19-2: Open Pit Configuration at the End of 2010

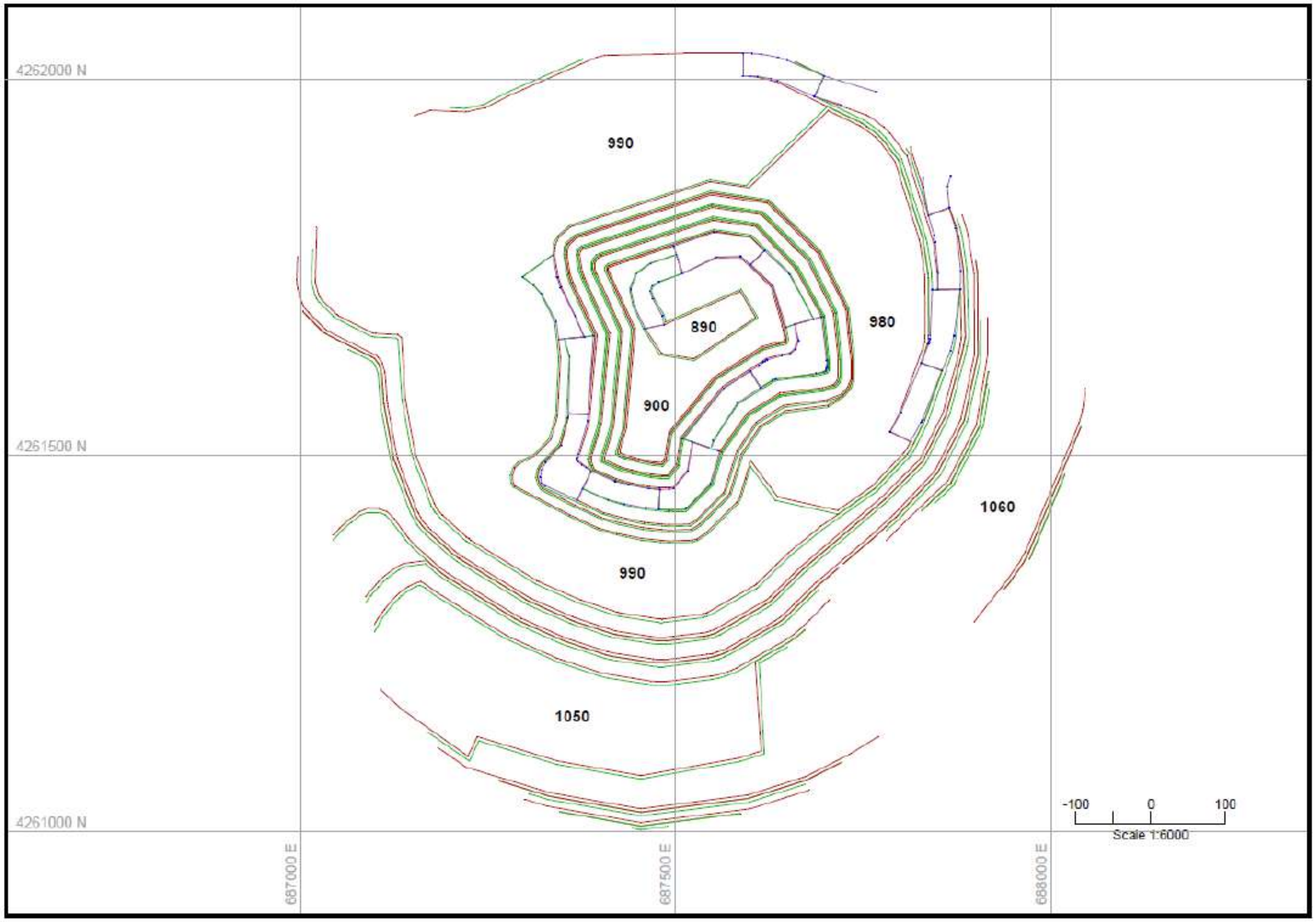


Figure 19-3: Open Pit Configuration at the End of 2015

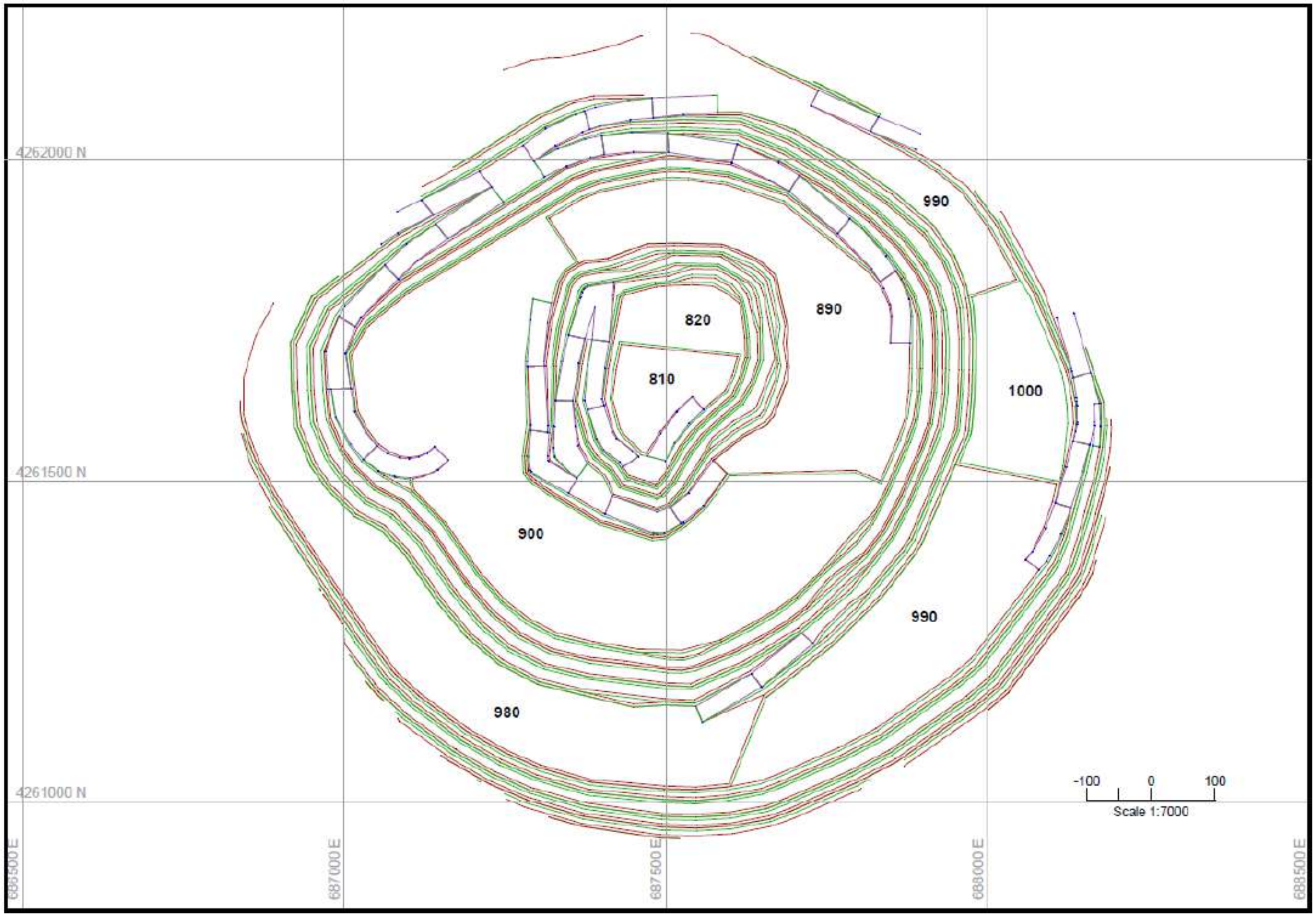


Figure 19-4: Open Pit Configuration at the End of 2020

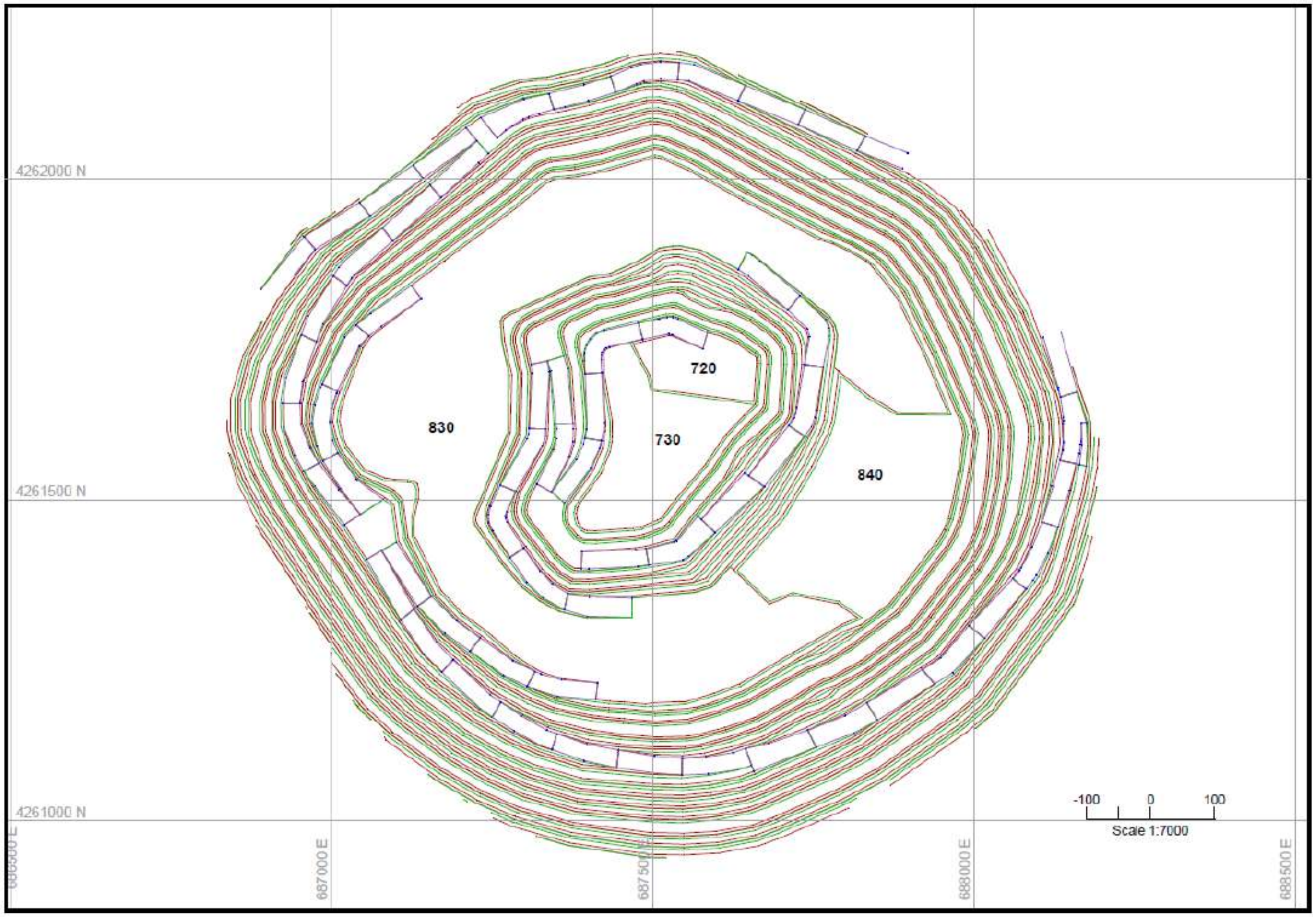
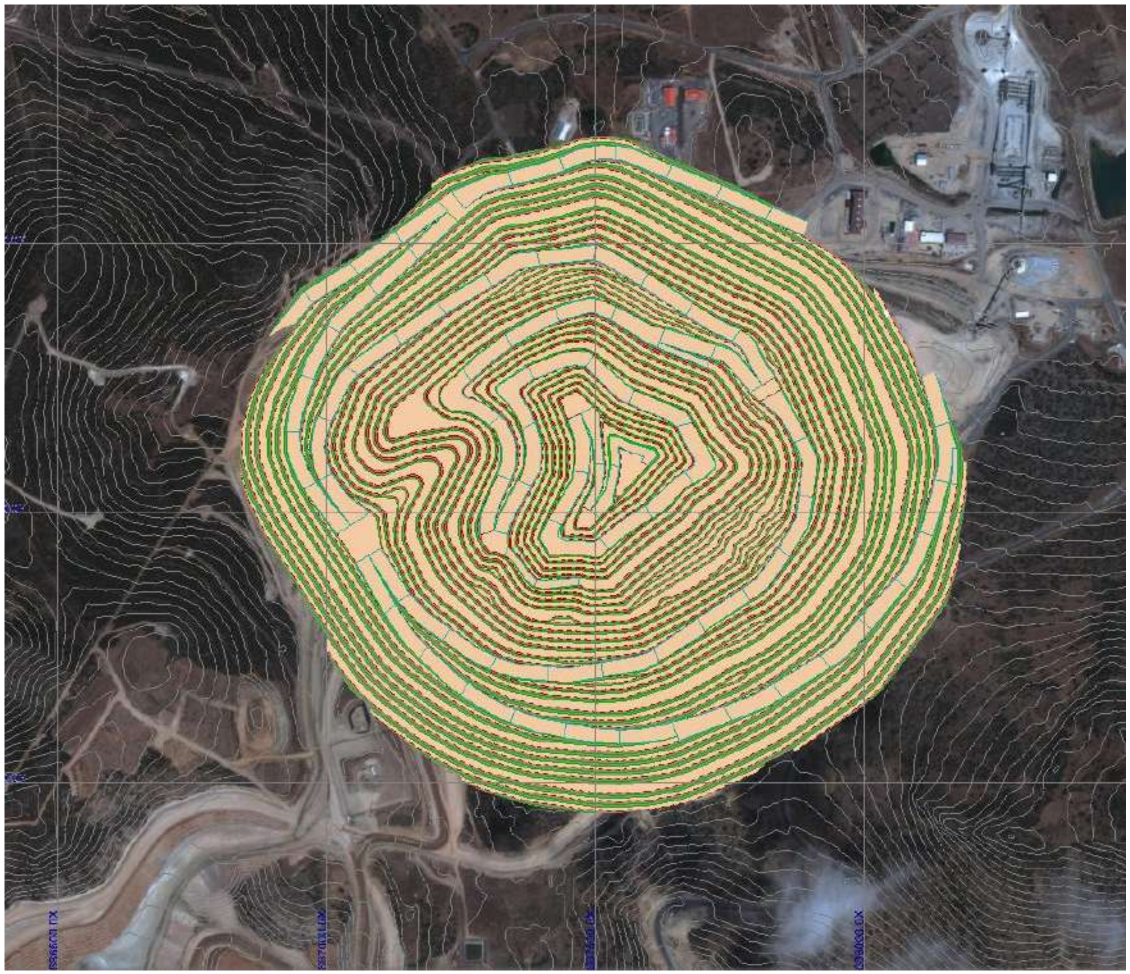


Figure 19-5: Final Pit



19.1.7 MINE OPERATION

One hundred percent drill and blast mining method has been assumed. A 10 m drilling bench height is used with about 1 m sub-drill totalling 11 m. Production drilling also provides samples for grade control purposes. Hydraulic excavators and a front-end loader complimented with off-highway trucks make up the production fleet.

Mining Method and Equipment

The currently selected excavating equipment consists of HITACHI EX 3600 hydraulic shovels with 21 m^3 bucket capacity loading into a fleet of Caterpillar 785C dump trucks each with a capacity of 136 tonnes. A Caterpillar 993K front-end loader has been selected as a secondary loading unit.

For blast hole drilling, Atlas Copco DM-45 rotary rock drills have proven to be capable of meeting production targets while also provide grade control data. One Atlas Copco Roc L6 drill rig has been selected for pre-split applications. All final walls and long-standing intermediate walls are pre-split to a 20 m face height (10 m in the friable zones).

Currently, the equipment required for constructing and maintaining haul roads, waste dumps and for in-pit duties include a Cat 834H wheel dozer, two Cat D9T dozers, two Cat M16 graders, three water trucks and a Cat 330 rock breaker. A number of smaller service and utility vehicles, together with pit pumps and tower lights are also used in the open pit.

Currently the open pit operates with a fleet of equipment as presented in Table 19-2.

Table 19-2: Equipment List

Model	Type	Units
Caterpillar 785 C	Haul truck	10
Caterpillar D9 T	Track dozer	2
Caterpillar 16 M	Grader	2
Caterpillar 993 K	Wheel loader	1
Caterpillar 834 H	Wheel dozer	1
Hitachi EX-3600 FS	Excavator	2
Atlas Copco ROC L6	Drill rig (PS)	1
Atlas Copco DM 45	Drill rig	2
Iveco Eurotracker	Trailer-low bed	1
Mercedes Benz 4140B 8X4	Tire handler	1
Mercedes Benz 4140B 8X4	Water truck	3
Mercedes Benz 3340K 6X4	Fuel truck	1
Mercedes Benz 3340K 6X4	Maintenance truck	1
Mercedes Benz 3340K 6X4	Service truck-16 t crane	1
Mercedes Benz 3028K 6X4	Anfo charger truck	1
Land Rover Defender 130 4X4	Welding truck	1
CAT 330D LME Excav. with Atlas Copco HB 2200 Rock Breaker	Backhoe / rock breaker	1

With an increase in production levels from 2011 and furthermore as the pit gets deeper, additional primary production equipment will be required. By 2012, an additional excavator will be required and the number of trucks will have increased to 18 in total. A fourth excavator will be required in 2013 by which time the total number of trucks will have climbed to 23. That loading capacity will be sufficient for the remainder of the mine life; however, some additional hauling capacity will eventually be required. The maximum number of trucks required to meet the life of mine production schedule will be a total of 28 trucks in the period of 2017 to 2021.

Drilling requirements will also necessitate the addition of a third production drill rig in 2011 and a fourth production drill rig in 2012.

Once the pit is developed below the water table in 2014, a permanent pumping system and a rig for drilling depressurization holes will also be required for the remainder of the mine life.

Loading and Hauling

The travelling surface of the mine haul roads inside the pit is 20 m wide. In addition, the high safety berm occupies an additional 5.3 m of width. Allowance has also been made for a 1 m wide drainage ditch. The minimum required width of the haul road for two-way traffic is therefore 26.3 m. Many of the intermediate pit haulage ramps have been designed at a conservative width of 30 m. Extensive use of multiple ramp access has been employed for easy sequencing between phases.

Future ore and waste production estimates in the range of 15,000 to 20,000 tonnes per operating shift for each 21 m^3 shovel with 136 tonnes haul trucks are based on actual operating statistics. Hauling productivities vary annually with the haul road profiles for both ore and waste. In estimating productivities, average haul road profiles were estimated from the designed pit benches to the crusher and the waste dump respectively for each year of the operation with a planned reduction of equipment availability over time. Haul roads have been designed with 10% gradients, except for the external road to the waste dump, which will have a 6% uphill grade. Haul distances will vary over the mine life from 700 m to 6,000 m for ore and 500 m to 6,500 m for waste.

19.1.8 WASTE ROCK DISPOSAL

Two types of waste material will be produced during the life of mine. These are oxide and sulphide. The sulphide waste rock is characterized as being potentially acid generating.

Waste Dumps and Haul Roads

Access roads to the dump are developed from the main and temporary haul roads during the life of mine as and when required. Waste rock from the open pit will be dumped to the southwest of the pit at a trucking distance that will vary from about 500 m to 6,500 m over the life of the operation. The rock dump has a currently designed capacity of 180 Mt with potential for future expansion. The waste dump is developed carefully to avoid any ground water or surface water contact with sulphide waste material.

The ultimate planned rock dump configuration is the filling in of an existing valley from the natural saddle in the west. Natural ground water seepage sources identified in the valley floor have been capped with coarse oxide rock from the sources right to the down slope edge of the rock dump design limits.

Sulphide rock is dumped in selected locations of the waste dump and is capped with oxide rock as the dumping progresses to limit any exposure of sulphide rock. Diversion ditches have been constructed along the periphery of the waste rock dump to divert surface water from making contact with the rock dump. All the water runoff from the rock dump area is collected in the two collection ponds, each constructed with membrane. Figure 19-6 shows the two collections ponds (middle), final contoured slope on the left and a non-contact water diversion ditch on the right. The maximum dumping height of waste rock is limited to 50 m in the waste dump area.

Figure 19-6: Rock Dump Collection Ponds

Waste Dump Closure

As part of closure at the end of the open pit life, the face of the rock dump will be resloped to angles of 22 degrees (2.5H:1V) or 27 degrees (2H:1V) depending on the area and capped with oxide waste and covered with a layer of soil for vegetation to form a store and release protective capping system.

Figure 19-7 shows a small part of the rock dump that has been resloped, covered with oxide rock and is in a stage with some top soil covering. This program was successful in testing the future reclamation work for mine closure.

Figure 19-7: Rock Dump Collection Ponds

19.1.9 MINE INFRASTRUCTURE

Explosive and Accessories Magazine

Explosives for mining operations will be supplied to site on a regular basis. Explosives are stored on site in the explosive magazine. The magazine is fenced and located within the property boundary according to the relevant Turkish regulations and safe mining practice.

Workshop and Laydown Areas

Facilities needed for the mine equipment maintenance include:

- change house space for operator, maintenance, and engineering staff and hourly employees
- a crane equipped mine workshop
- an outdoor concrete wash pad
- a concrete pad for changing large vehicle tires
- an outdoor lay down area
- a parking area for mobile equipment not in use or awaiting maintenance.

Fuel Depot

All mine equipment is diesel operated. Trucks are fuelled at the temporary fuelling depot within the open pit mining area. There is also a separate fuelling station near the warehouse for light vehicles.

A fuel truck and a lubrication truck are used for the pit equipment as required.

Pit Dewatering

Permanent pit dewatering system will be implemented in 2014 when the pit bottom reaches below the ground water level of 860 masl.

Surface Water Diversion Channel

For the most part the natural topography slopes away from the pit edge however a diversion channel will be needed on the west side of the pit outside of the waste dump haul road starting in Q4 of 2010.

19.1.10 CONCLUSIONS

Kışladağ Gold Mine is a currently operating mine with some 217 Mt of remaining ore reserves that will be depleted by 2028 at a rate of 10 Mt of ore in 2010 and 12 Mt of ore per year thereafter. It employs conventional drill and blast and truck and shovel mining methods where operating costs and productivity metrics are well known. Mining is done in 10 m high benches and the pit is phased with four phases. Extensive geotechnical work has defined parameters for the pit face geometry based on a number of controlling elements. The water table, which is roughly 150 m below the surface topography, will necessitate the installation of depressurization holes and a permanent pumping system from 2014. Waste rock will be dumped in a designated location with segregated management of the sulphide and oxide waste rock with careful control of natural ground water seepages, preventative measures related to the ingress of surface water, and management contact water.

19.2 PROCESS PLANT AND RECOVERABILITY

The Kişladağ ore is processed in a standard heap leach facility containing a three stage crushing plant, an overland conveyor to the heap leach pad, mobile conveyors and a stacker for placing the ore and a carbon adsorption facility (ADR plant) for recovering the gold. The carbon is treated on site in a refinery and the final product is a gold doré bar.

The initial design capacity was 5,000,000 dry tonnes of ore per year for the first two years of operation when predominantly oxide material was processed. The facilities were expanded to process 10 Mt/a after year two, when primary ore from the deeper, higher sulphide zone in the pit predominated although oxide ore will be recovered annually over the LOM in varying quantities (Table 19 -1) The crush size is 80% passing 6.3 mm for primary sulphide ore and 80% passing 12.5 mm for oxide ore. The overall availability of the crushing and screening plant is about 70%.

The primary crusher is a 1,270 mm by 1,651 mm gyratory crusher capable of processing the ultimate design rate of 1,751 t/h. Run of mine ore is hauled from the open pit and direct dumped into the primary crusher. Initially, contract miners delivered the ore in 37 tonne trucks and the layout provides for two trucks dumping simultaneously. The dump pocket has a capacity of 300 tonnes and is adequate for the larger owner operated mine trucks in later years. Owner operated mining training and transition with 150 tonne trucks started in May 2008. In October 2008, the mine started to use its own equipment. The crushed ore is conveyed to a 20,000 tonne coarse ore stockpile. This is used as surge capacity to feed the fine ore circuit. Material is then conveyed from the base of this to a 300 tonne coarse ore bin and then on to the secondary crusher.

The final crushed product was prepared in a circuit consisting of one scalping screen, one MP1000 standard secondary cone crusher, three MP800 short head tertiary cone crushers, and four fine ore screens. The capacity of the circuit ranges from 1,300 t/h to 1,400 t/h when delivering a product containing 80% passing 6.3 mm.

Final product from the crushing and screening circuit is transported to the heap leach pad by two stage overland conveyors and a series of portable conveyors. A radial stacker distributes the ore onto the pad. Depending on the location and geometry of the pads, advance stacking is applied as well. Stacking is carried out in 10 m lifts to a projected design height of 60 m.

When the overland conveying is not available, to keep the crushing unit in operation, the final product is conveyed to a 25,000 tonnes fine ore stockpile via a radial stacker. The fine ore is then manually reclaimed from the pile by front-end loaders and re-introduced onto the overland conveyor belt via a hopper.

The heap leach pad is a permanent facility employing a two-part liner system of a compacted layer of low permeability soil, with a 2 mm thick HPDE/LLDPE synthetic liner. The initial pad had a capacity of 15 Mt and sequential expansions to the pad are accommodating the total tonnage mined.

During the pre-production period, mostly oxide ore was mined, crushed, and screened for use as overliner material. Once the overliner material has been placed, the three stage crushing plant was used for a period of three months to deliver 400,000 tonnes of pre-production ore to the pad. Irrigation of the heap commenced in the second month of pre-production and by the end of the first month of production, sufficient leaching had taken place to allow the first gold to be poured.

The leach cycle for oxide ore, based on test work, is 90 days; this will change for the higher lifts, since the solution return takes longer as the pad height keeps increasing. The sulphide ore is much slower leaching. Testwork indicates that it could take as long as 120 days without taking increasing pad heights into account. Solution application rate is about 5 to 8 L/h/m^2 of crushed ore. There are three process ponds installed to contain the heap leach solutions. The pregnant solution pond has the live capacity of 8,600 m^3, barren pond has 17,400 m^3 and intermediate pond has 7,200 m^3 capacities. There is an 8,600 m^3 volume spare pond available as well. The process ponds have a double HDPE liner and fitted with leak detection pipes and pumps. The pond surfaces were covered with floating 100 mm diameter HDPE plastic balls in order to reduce evaporation.

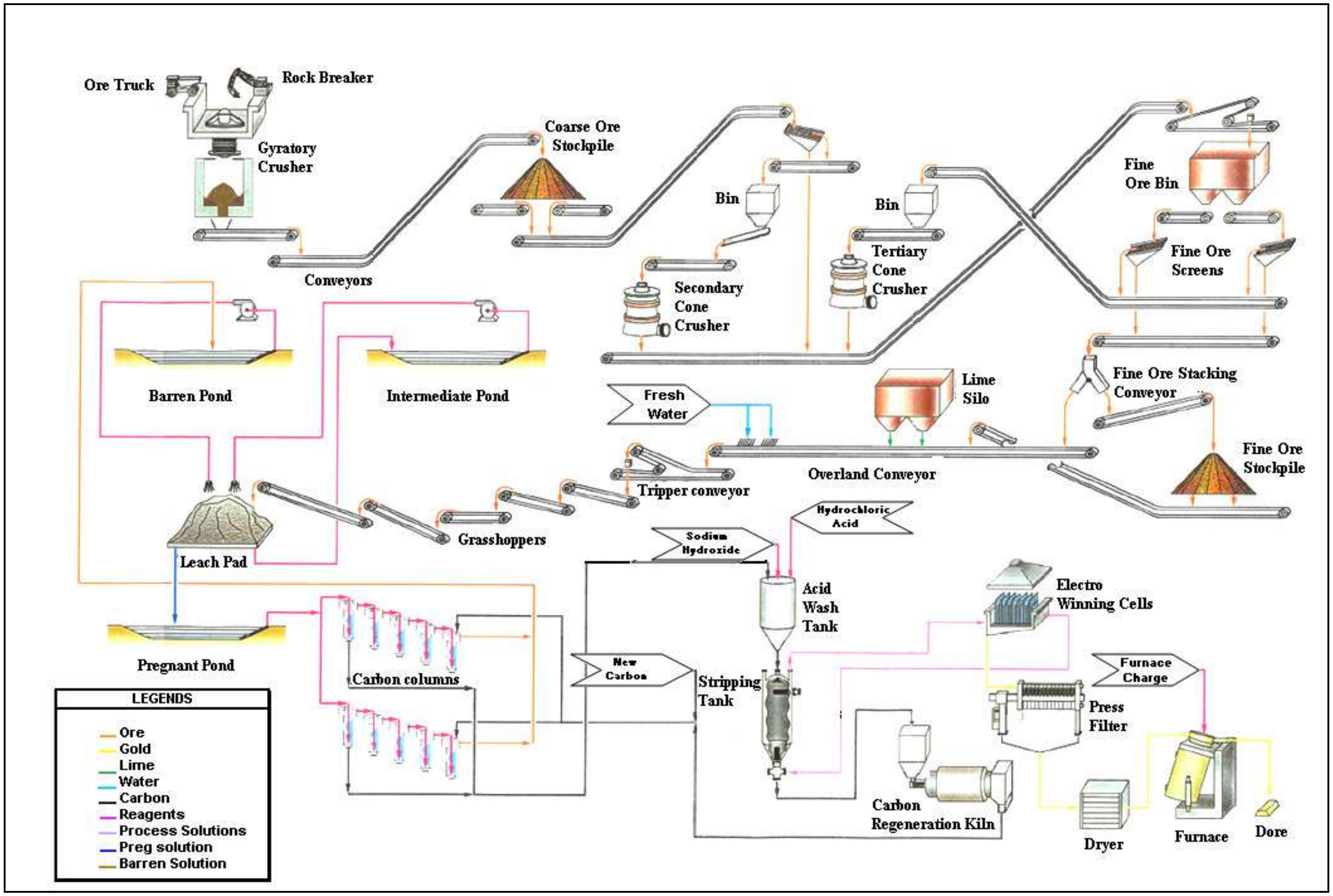
The water management system has been designed to accommodate a 100 year, 24 hour storm event. A 116,000 m^3 storm water event pond with an HDPE liner was provided to contain excess overflow solution from the pregnant solution pond. A second storm water event pond of 98,000 m^3 capacity, was installed in order to cater for the storm event that exceeds the one in a 100 year estimate, an emergency hydrogen peroxide detoxification plant will reduce the cyanide content of the solution to safe levels, in the event discharge becomes necessary.

The gold adsorption facility (ADR plant) consists of two trains of carbon columns with each train consisting of five columns. Gold from the heap solutions is loaded onto the activated carbon and the carbon is removed periodically for treatment. The gold is recovered from the carbon in a standard Zadra process consisting of pressure stripping, electro winning and smelting. The final product is a gold doré bar suitable for final processing to 99.999% purity in domestic or offshore refineries.

Process plant work force requirements have been based on a three shift, seven day per week operation. There are a total of 164 process positions.

A simplified flow sheet for the Kişladağ process plant is presented in Figure 19-8.

Figure 19-8: Simplified Flow Sheet



19.3 SITE INFRASTRUCTURE AND ANCILLARY FACILITIES

19.3.1 SITE LOCATION

The Project is located on the western edge of the Anatolian Plateau at an elevation of approximately 1,000 m. Local elevations range from a peak of 1,300 masl (Kişla Dag) to a valley of 700 masl. There are a number of small farming villages within the concession area and livestock breeding provides their main means of support.

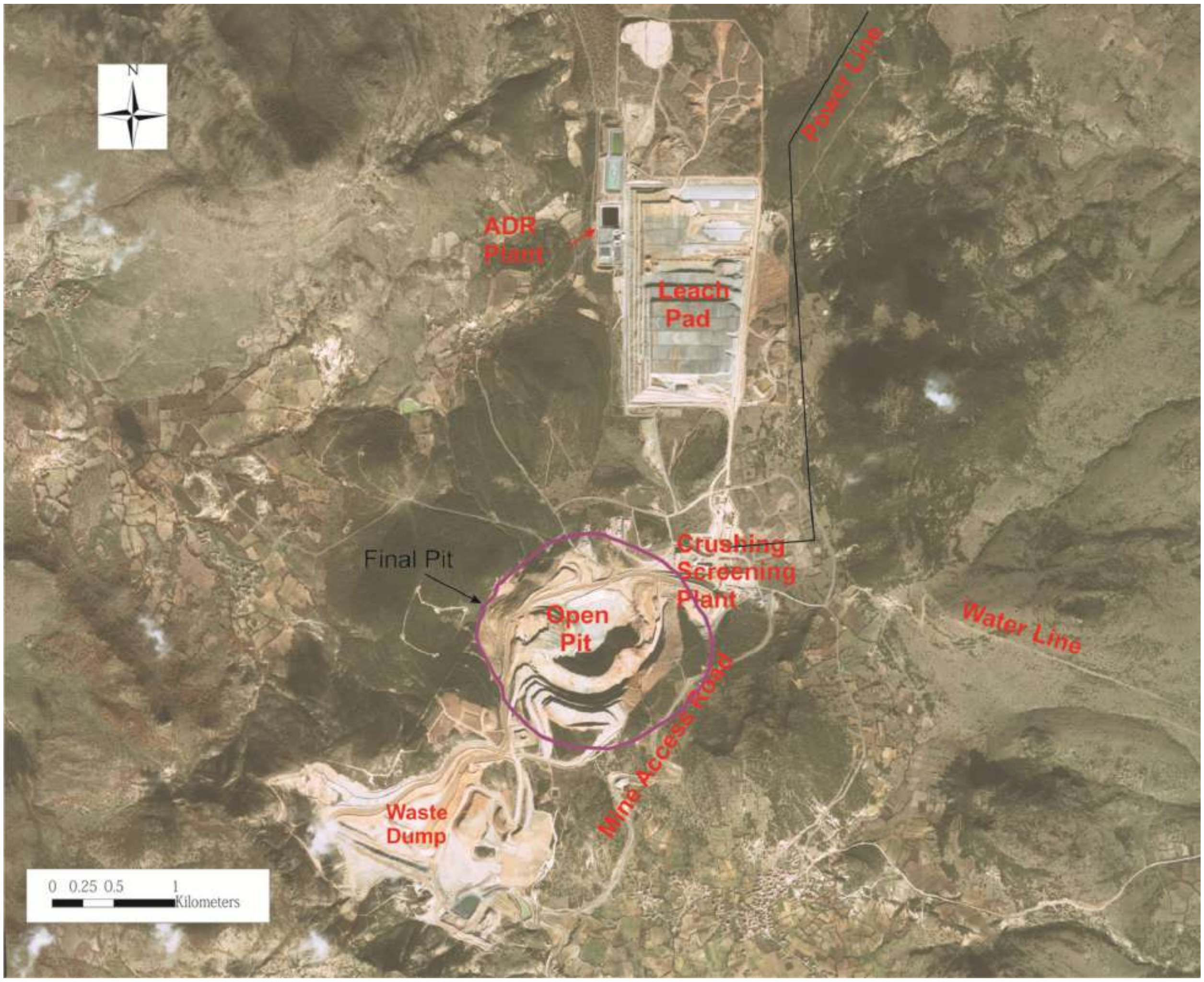
The crushing plant location (N 4262000 and E 688200) is adjacent to the open pit, approximately 2 km north of the village of Gümüşkol. The administration buildings are located on level ground between the pit and the crushing plant. The rock disposal site is located about 1 km southwest of the open pit, within the headwater area of a small valley drained by an intermittent stream.

The leach pad facility has been constructed to the north of the plant site. The leach-pad is located on the western flank of Kişladağ Mountain and bounded on the west side by the main basin drainage course. The leach pad extends northwards, approximately 1.2 km presently under operation, and a further expansion of 0.5 km under construction.

A 2 m high range fence has been installed along the property boundary and controls access to the mine site. There is one main access gate, which includes a gatehouse manned 24 h/d. Additional security fencing has been erected around the ADR plant and solution ponds, electrical substations, reagent and explosives storage areas.

Figure 19-9 presents a view of the project area as per end of year 2009.

Figure 19-9: Project Area



19.3.2 ACCESS ROAD

As the original access road was unsuitable for mine use, it is narrow and winds through the village of Gümüşkol; a new road has been constructed approximately 5.3 km long, 10 m wide connecting the mine site to the regional road from Ulubey to Esme. A portion of the old road connecting the villages of Gümüşkol and Katrancilar has been replaced by a new road, approximately 1.9 km long, constructed to bypass the crushing facilities.

19.3.3 WATER SUPPLY

There are no permanent water bodies in the area. Local water supply is limited to ephemeral streams and shallow seasonal stock ponds. The geology of the area is dominated by volcanics with generally poor aquifer characteristics. Fresh water for the Project is supplied from a well field located approximately 13 km to the east of the plant site, in Neogene sediments. Four wells have been constructed and are in operation; a fifth well is being constructed. Two water storage tanks and underground distribution system at site provides capacity for process, firewater, and potable water requirements.

An underground sewer reticulation is connected to the main buildings through to a treatment plant, with a capacity of 50 m^3/d.

19.3.4 POWER SUPPLY

The Turkish national power utility company, TEDAS, distributes electrical power to the Kışladağ site via a 25 km long 34.5 kV transmission line from the Uşak industrial zone. The main transformer at site is rated at 10 MVA, 35.5 kV to 6.6 kV and provides 6.6 kV power, which is distributed locally via overhead power lines and underground cables.

19.3.5 BUILDINGS

The permanent mine buildings have been designed and constructed by local Turkish contractors. The architecture of the facilities includes local building materials and methods compatible with the surrounding infrastructure.

The original workshop/warehouse was constructed and later a separate maintenance workshop was built and the **Warehouse** (760 m^2) modified to be used for warehousing, department offices, and IT control room. An adjacent outdoor fenced area together with covered hard standing has been constructed for storage of large equipment and miscellaneous reagents. A diesel depot for dispensing fuel to small vehicles is included.

A **Maintenance Workshop** (780 m^2) has been constructed and includes an electrical workshop, an instrument workshop, tool storage, a security store, offices, storage space for maintenance items, washroom, locker and change room. An overhead traveling crane has been installed inside the main workshop. Outside paved areas have been provided for work areas and storage.

Earlier, the mining contractor established their own temporary facilities to service their mining fleet.

The **Mine Truckshop** complex (1,850 m^2) has been constructed and designed to service the fleet of mining trucks. The complex includes three indoor heavy equipment repair bays equipped with an overhead traveling crane, a covered outdoor service bay and an outdoor wash bay equipped with an oil/water separator. A general repair area and a welding shop have also been included in the complex. A three-storey annex houses a mechanical room and office space. In addition, a two storey annex for change room and washroom has been built.

The **Administration Building** (400 m^2) is a single storey building and includes general areas for engineering, geology and administration personnel plus seven individual offices for management personnel.

The **Mine Dry and Canteen** (540 m^2) is a single storey concrete building. The canteen has been equipped with a kitchen area and a seating area for 72 people and includes a covered, enclosed patio that can seat a further 60 diners. Washrooms, shower facilities, and clean and dirty lockers are provided in the mine dry area. Additional space is provided for five offices and a meeting room.

The **Assay Laboratory Building** (440 m^2) houses the assay laboratory rooms, assayers and assistants offices, washrooms for personnel, and storage rooms. The assay laboratory is capable of handling 550 samples per day, which includes sample preparation, acid digestion, atomic absorption finish, fire assay, and a wet laboratory. The laboratory has received ISO 9001 certificate and Laboratory Accreditation.

The **Health and Security Building** (86 m^2) provides three consulting and treatment rooms for the mine's doctor, toilets, and separate attached offices for the security contractors manager.

The **Environment and Safety Building** (470 m^2) provides office accommodation, meeting room, small laboratory, toilet and tea room facilities for environment, safety and process engineering management personnel.

At the main entrance gate the **Public Relations Building** (132 m^2) provides office accommodation, reception room, toilet, and tearoom facilities for PR management personnel.

A small **Gate House** is provided for security guards on duty controlling the main entrance.

Miscellaneous prefabricated buildings provide additional office accommodation, washroom facilities, storage rooms, and work areas for the construction management team, geology core logging and storage, safety rescue and demonstration, laundry, archives and a prayer room.

Operations personnel reside in the surrounding towns and villages and there are no plans to erect a permanent camp for operations personnel or temporary construction camps. Personnel are bussed to site. During construction, contractors are responsible for providing their workforce with accommodation and transport.

19.4 ENVIRONMENTAL CONSIDERATIONS

The Kişladağ Project EIA study was completed in January 2003 and submitted to the Turkish Authorities at the Ministry of Forest and Environment. An Environmental Positive Certificate for the project was subsequently obtained in June 2003. The EIA document presents a number of potential socio-economic effects associated with the development of the Project, and defines a number of measures to avoid or minimize potential impacts.

An environmental monitoring plan has been developed to address the potential impacts of the mining operation. This plan was put in place prior to pre-production mining starting in 2005 and has been maintained throughout the production phase. The scope of the monitoring program within this plan includes elements of air quality, surface water and ground water monitoring. Data collected during the monitoring program is reported to the relevant government agencies on a monthly and annual basis. Additional issues addressed in the imbedded Environmental Management Plan include noise and blast vibration monitoring as well as waste and hazardous waste storage and disposal.

19.5 CAPITAL AND OPERATING COSTS ESTIMATES

19.5.1 OPERATING COSTS

Kişladağ has been in operation since 2005. The operating costs and production levels are well understood, and the mine has a history of achieving its budgeted production and forecast expenditures.

Table 19-3 shows the forecasted costs and production statistics for 2010.

Table 19-3: 2010 Forecast Operating Costs and Production Statistics

Item	2010 Forecast Operating Costs
Mining Costs ($/t mined)	1.32
Processing Costs ($/t placed on pad)	3.55
Mine Support ($/t placed on pad)	1.31
Tonnes Mined (000's t)	22,100
Tonnes Placed on Leach (000's t)	10,000
Gold Ounces Produced (oz)	230,000 – 240,000
Cash Operating Costs ($/oz)	310 – 330

The operating costs per unit of production have been relatively constant since the start of the mine life. The cost for processing and mine support are expected to remain constant for the remainder of the mine life, except when they are changed due to inputs that affect the entire gold mining industry, including, but not limited to, changes in, fuel costs, reagent costs, exchange rates, labour costs and inflation. The unit costs for mining are expected to increase as the pit deepens and they are also affected by the previously listed inputs.

19.5.2 CAPITAL COSTS

Kişladağ commenced gold production in 2006 and since that time, the mine has generated sufficient cash flow to pay back the initial capital expenditure.

In 2009, the mine initiated a study to expand the current processing rate to 12 Mt of ore per year. The capital cost of this expansion is included in the 2010 budget, and sufficient capital has been budgeted to include an upgrade of the existing crushing and screening plant as well as the ADR plant. The project is expected to be completed by 2011.

In addition to the expansion capital costs, Kişladağ requires sustaining capital expenditures to maintain the planned production rates over the life-of-mine. These capital expenditures include, but are not limited to, costs for leach pad expansion, costs for new mining equipment to increase the size of the fleet and costs for replacement of old equipment.

19.6 MARKETS

19.6.1 GOLD MARKET

Supply-Demand Balance

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

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

Price

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

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

19.7 CONTRACTS

Following contracts and purchase agreements are in place at Kişladağ:

CyPlus GmbH	Cyanide
Securitas Guvenlik Hizmetleri A.S.	Security
Efta Petrol Insaat Turizm Gıda Madencilik Ithalat Ihracaat San.ve Tic. A.S..	Diesel fuel and lube
Kapex Patlayıcı Maddeler San.ve Tic. A.S.	Explosives
Bilintur Bilkent Tur.Ins.Yat.ve Tic.A.S..	Meal catering
Forestry Department Denizli Region....	Lease of Forest Land

19.8 TAXES

Corporate taxation for Turkish businesses is currently 20%. Depreciation is based on a unit-of-production calculation. No book to tax differences (such as tax loss carryforward or investment incentive deductions) have been considered in the economic analysis.

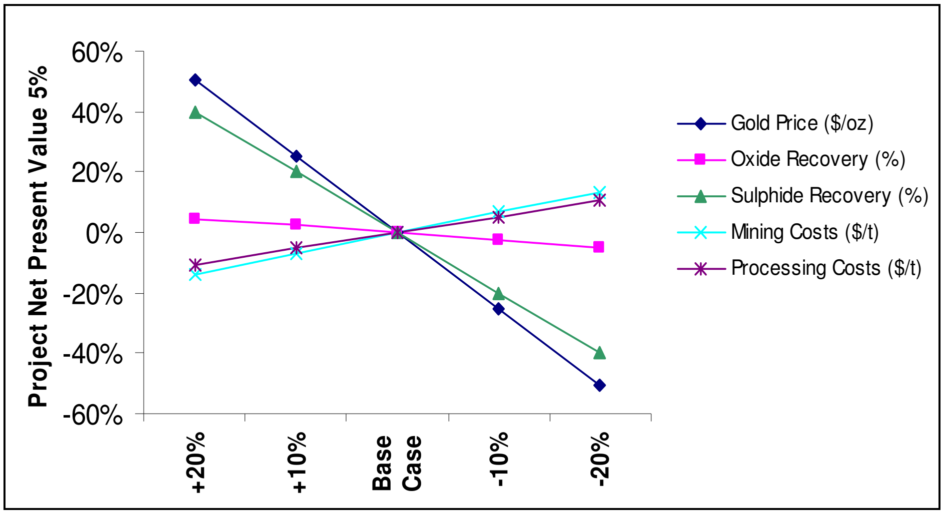
19.9 ECONOMIC ANALYSIS

Kişladağ is an operating mine and life-of-mine plans, are routinely generated to determine cash flows and capital requirements for the project.

The cash flow analysis indicates that the mine generates a positive cash flow (after taxes and capital expenditures) over the life of the proven and probable reserves.

The cash flow model was tested for sensitivity by varying the input parameters. The results of the sensitivity analysis are shown in Figure 19-10. The net present value of the Kişladağ project is most sensitive to changes in the gold price, followed by changes in the sulphide recovery. Changes in oxide recovery, mining costs and processing costs have a smaller effect on the net present value.

Figure 19-10: Kişladağ Economic Sensitivity Analysis



19.10 MINE LIFE AND PAYBACK

The current reserve at Kişladağ supports a mine life of 19 years.

The initial capital expenditure has already been paid back. The project is generating positive cash flow and the expansion capital proposed for 2010 will be paid back within the first year of the project completion.

SECTION 20 • CONCLUSIONS AND RECOMMENDATIONS

20.1 CONCLUSIONS

The Kişladağ Gold Mine has been in operation for nearly four years. The mine has been successful in its implementation of construction and operations plans as described in the previous NI 43-101 technical report prepared by Hatch in 2003. Production tonnages and gold produced are matching previous forecasts. There is no reason to believe that the mine will not continue to perform as well in the future as it has during the first years of operations.

The geology of the Kisaldag deposit is well understood. The deposit is considered to be an example of porphyry-style gold mineralization centred on a series of overlapping sub-volcanic intrusives of quartz-syenite to quartz-monzonite composition.

Diamond drill holes are the principal source of geologic and grade data for the Kişladağ Mine since the start of mining in 2006. Data from these holes are directly responsible for the significant increase in the year-end 2009 mineral resources and reserves.

Eldorado employs a comprehensive QAQC program for its gold analyses on drill core and RC samples. Monitoring of the quality control samples showed all data were in control throughout the preparation and analytical processes. In Eldorado's opinion, the QA/QC results demonstrate that the Kışladağ deposit assay database, particularly for new data obtained from 2007 to 2009, is sufficiently accurate and precise for resource estimation.

Since the start of production in 2006, the entire drill hole database was reviewed in detail. Checks were made to original assay certificates and survey data. Any discrepancies found were corrected and incorporated into the current resource database. Eldorado therefore concludes that the data supporting the Kışladağ resource work are sufficiently free of error to be adequate for estimation.

The Kışladağ deposit resource model and mineral reserve estimates were developed using industry-accepted methods. Both have been substantiated by production to model reconciliation over a period of almost four years of operation.

The metallurgy of the Kisladağ deposit is well understood. Recovery numbers gained throughout the first four years of operation agree very well with the results obtained during the Feasibility Study. Project to date there is excellent agreement between predicted gold recovered and that actual gold recovered. Operating costs and reagent requirements approximately agree and robust models have been developed and used both for production forecasting and cost control.

The initial capital expenditure has already been paid back. The project is generating positive cash flow and the expansion capital proposed for 2010 will be paid back within the first year of the project completion.

SECTION 21 • REFERENCES

Hatch, 2003. Technical Report Kışladağ Project Feasibility Study. National Instrument 43-101 Technical Report March 2003

Kappes, Cassiday and Associates, Kısladağ Project, Report of Metallurgical Testwork, August 10, 2000

Kappes, Cassiday and Associates, Kısladağ Project, Report of Metallurgical Testwork, October 30, 2000

Kappes, Cassiday and Associates, Kısladağ Project, Report of Metallurgical Testwork, December 2002

Kappes, Cassiday and Associates, Kısladağ Project, Report of Metallurgical Testwork, March 2003

Kappes, Cassiday and Associates, Kısladağ Project, Sulfide Material, Report of Metallurgical Tests, March 2008

Kappes, Cassiday and Associates, Kısladağ Project, Report of Metallurgical Testwork, January 2009

Micon, 2003. 2003 Update of Resources, Kışladağ Project, Usak, Turkey. National Instrument 43-101 Technical Report September 2003

SGS Lakefield Research, An Investigation of Gold Recovery from Kisladag Project Samples for Eldorado Gold Corporation, July 24, 2002

Yazicigil H, Karahanoglu N, Yilmaz K, Gundogdu A, Sakiyan J, Yesilnacar E, Tuzcu B, 2000. Investigation and management ofthe Kucuk Menderes Basin groundwater. Hydrogeology, vol 5, Middle East Technical University, Ankara

SECTION 22 • SIGNATURE PAGE AND DATE

The effective date of this report entitled "Technical Report for the Kişladağ Gold Mine, Turkey" is 28 January 2010. It has been prepared for Eldorado Gold Corporation by Richard Miller, P.Eng., S. Juras, P.Geo., and P. Skayman, MAusIMM, each of whom are qualified persons as defined by NI43-101.

Signed the 15th day of March, 2010.

SIGNED

"Richard Miller"	*"Stephen Juras"*	*"Paul Skayman"*
Richard Miller, P.Eng Manager, Mining Eldorado Gold Corp.	Stephen Juras, PhD, P.Geo Director, Technical Services Eldorado Gold Corp.	Paul Skayman, MAusIMM VP, Operations Eldorado Gold Corp.

SECTION 23 • CERTIFICATIONS AND CONSENTS

CERTIFICATE OF QUALIFIED PERSON

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

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 work for the Kışladağ Gold Mine in Turkey. The report entitled *Technical Report for the Kışladağ Gold Mine, Turkey,* with an effective date of January 2010, was prepared under my supervision except Sections 16, 17 (sub-section 2), 18 and 19. I visited the mine on numerous occasions in 2009 with my most recent visit occurring from November 15 to 17, 2009.

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

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

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

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

Dated at Vancouver, British Columbia, this 15th day of March, 2010.

"Stephen Juras"

____

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

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

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

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

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

I was responsible for reviewing matters related to the mining operations and directing the mineral reserve estimation work for the Kışladağ Gold Mine in Turkey. The report entitled *Technical Report for the Kışladağ Gold Mine, Turkey,* with an effective date of January 2010, was prepared under my supervision, specifically Sections 17 (sub-section 2), 18 and 19. I had worked as Manager Mining for the Kışladağ Project / Mine based in Turkey for three years and have visited the mine on numerous occasions since starting my current position in 2007. My most recent visit occurred from September 15 to 17, 2009.

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

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

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

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

Dated at Vancouver, British Columbia, this 15th day of March, 2010.

"Richard Miller"

____

Richard Miller, P.Eng

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

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

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

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

I was responsible for reviewing matters related to the metallurgical data for the Kisladag Gold Mine in Turkey. The report entitled Technical Report for the Kisladag Gold Mine, Turkey, with an effective date of January 2010, was prepared under my supervision, specifically Sections 1 to 3 and Section 16 and parts of Section 19. I visited the mine on numerous occasions in 2009 with my most recent visit occurring from September 14 to 18, 2009.

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

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

I have read National Instrument 43-101 and Form 43-101FI and the sections for which I am responsible in this report entitled, Technical Report for the Kisladag Gold Mine, Turkey, with an effective date of January 2010, has been prepared in compliance with same.

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

Dated at Vancouver, British Columbia, this 15th day of March, 2010.

"Paul Skayman"

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Paul J Skayman, B. Sc (Ext. Met).

APPENDIX A
TABLE OF MINERALIZED INTERSECTIONS

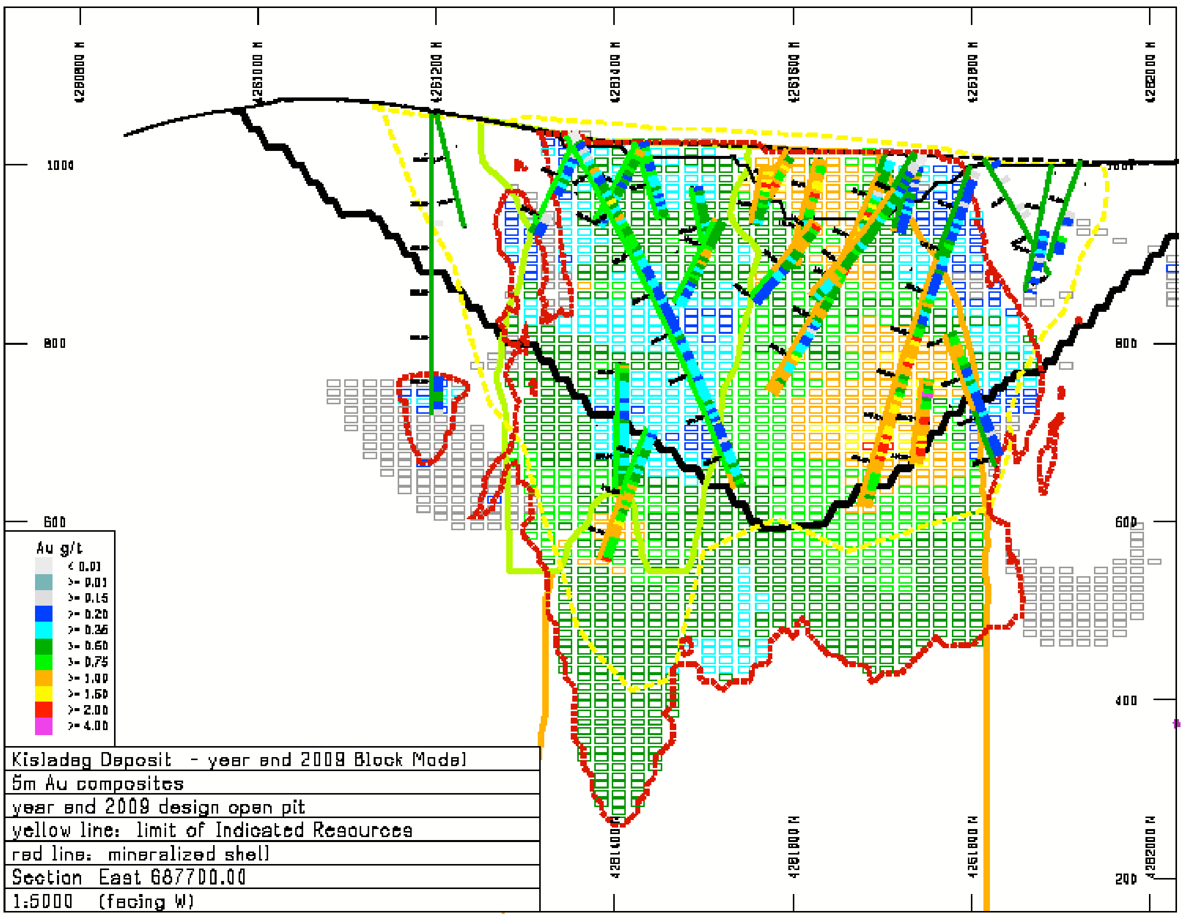
Listing of Kışladağ mineralized intersections (contained within the gold shell) from 2007 to 2009 drill holes.
(coordinates represent mid-point coordinates of intercepts).

DHID	East	North	Elevation	Au (g/t)	Length (m)
GC-271	687755	4261727	966	0.50	23
GC-271	687759	4261645	789	0.78	303
GC-272	687694	4261462	833	0.46	415
GC-273	687823	4261612	836	0.36	183
GC-274	687807	4261524	909	0.79	300
GC-275	687701	4261729	805	0.91	388
GC-276	687174	4261444	935	0.62	275
GC-277	686970	4261636	891	0.33	206
GC-278	687400	4261308	1013	0.43	58
GC-278	687397	4261359	821	1.08	300
GC-279	687154	4261760	919	0.56	240
GC-279	687147	4261846	749	0.44	70
GC-280	687571	4261364	868	1.11	301
GC-281	687502	4261377	836	1.70	340
GC-282	687108	4261714	866	0.80	399
GC-283	687762	4261536	853	0.84	325
GC-284	687596	4261464	850	0.40	324
GC-285	687056	4261595	838	0.70	391
GC-286	687569	4261783	989	0.22	28
GC-286	687608	4261842	873	0.57	218
GC-286	687640	4261901	738	0.23	20
GC-287	687263	4261405	936	1.01	194
GC-288	687123	4261481	1010	0.45	25
GC-288	687125	4261435	931	0.45	138
GC-289	687381	4261462	802	1.07	198
GC-290	687443	4261818	929	1.28	165
GC-291	687239	4261431	893	1.01	275
GC-292	687200	4261637	958	0.66	185
GC-292	687203	4261715	766	0.53	174
GC-293	687221	4261586	986	0.53	133
GC-293	687272	4261586	902	0.37	30
GC-294	687031	4261553	831	0.43	311
GC-295	687101	4261654	861	0.75	316
GC-296	687239	4261527	809	0.69	125
GC-297	687051	4261780	886	0.24	20
GC-297	687093	4261697	751	0.48	213
GC-298	686948	4261430	996	0.28	20
GC-298	686948	4261449	964	0.22	30
GC-298	686947	4261492	890	0.30	15
GC-298	686945	4261560	770	0.98	67
GC-299	687092	4261517	985	0.29	25

DHID	East	North	Elevation	Au (g/t)	Length (m)
GC-299	687073	4261498	931	0.33	25
GC-299	687027	4261454	805	0.44	178
GC-303	686904	4261425	894	0.34	148
GC-303	686911	4261515	733	0.20	30
GC-303	686913	4261547	676	0.26	38
GC-303	686915	4261570	634	0.21	19
GC-304	686831	4261589	740	0.74	265
GC-313	687500	4261782	969	0.60	48
GC-313	687501	4261845	857	0.80	175
GC-317	687793	4261650	912	0.38	203
GC-317	687873	4261650	811	0.26	15
GC-320	687467	4261359	913	0.66	188
GC-320	687492	4261291	807	0.24	15
GC-321	687821	4261478	793	0.69	519
GC-322	687679	4261776	802	0.68	293
GC-323	687124	4261777	952	0.63	123
GC-324	687575	4261553	974	0.24	15
GC-324	687663	4261446	747	0.60	456
GC-325	686958	4261375	938	0.34	33
GC-325	686967	4261447	824	0.41	168
GC-325	686978	4261553	662	0.35	120
GC-327	687174	4261332	671	0.27	33
GC-327	687173	4261350	611	0.34	68
GC-327	687171	4261367	556	0.22	38
GC-328	687319	4261397	879	0.74	258
GC-328	687299	4261284	634	0.77	43
GC-329	686716	4261601	870	0.48	108
GC-329	686666	4261532	693	0.36	15
GC-329	686650	4261510	633	0.38	27
GC-330	686971	4261722	724	0.39	43
GC-330	686968	4261691	638	0.32	90
GC-331	687421	4261250	863	0.74	18
GC-333	687852	4261411	991	0.33	50
GC-334	687784	4261349	918	0.43	155
GC-334	687791	4261284	779	0.23	23
GC-335	687827	4261448	973	0.49	155
GC-335	687948	4261449	799	0.20	17
GC-336	687297	4261392	716	0.52	156
GC-337	687102	4261436	829	0.42	451
GC-338	687358	4261458	771	0.63	345
GC-339	686901	4261649	707	0.40	118
GC-340	686787	4261654	765	0.37	231
GC-341	686900	4261647	881	0.27	25
GC-341	687017	4261644	665	0.36	276
GC-342	686829	4261499	752	0.44	363
GC-344	686576	4261473	975	0.34	25

DHID	East	North	Elevation	Au (g/t)	Length (m)
GC-344	686576	4261438	855	0.34	45
GC-344	686578	4261400	714	0.23	58
GC-345	686592	4261581	739	0.46	157
GC-346	687242	4261672	992	0.68	93
GC-346	687310	4261709	888	0.47	38
GC-348	687189	4261390	992	0.43	135
GC-349	687628	4261894	989	0.19	20
GC-349	687601	4261819	879	0.93	117
GC-350	687839	4261670	846	0.21	18
GC-351	687865	4261631	844	0.40	63
GC-351	687896	4261485	432	0.44	783
GC-352	687398	4261752	697	0.58	532
GC-353	687544	4261725	669	0.84	560
GC-354	687682	4261871	902	0.28	48
GC-354	687636	4261810	616	0.39	500
GC-355	687453	4261337	556	0.84	206
GC-357	687569	4261324	529	0.44	473
GC-358	687786	4261317	867	0.23	60
GC-358	687799	4261395	544	0.53	422
GC-359	687840	4261385	993	0.26	80
GC-360	687426	4261390	664	0.83	383
GC-361	687543	4261281	816	0.20	15
GC-361	687522	4261404	546	0.76	530
GC-362	687340	4261789	676	0.52	302
GC-363	687436	4261742	618	0.67	355
GC-364	687806	4261408	996	0.45	63
GC-364	687630	4261390	502	0.49	955
GR-307	686924	4261623	955	0.19	90
GR-307	686926	4261747	721	0.37	40
GR-309	687019	4261626	798	0.59	421
GR-310	687829	4261715	850	0.32	20
GR-310	687805	4261664	744	0.57	170
GR-311	687827	4261413	980	0.44	140
GR-311	687829	4261394	870	0.26	38
GR-312	687348	4261822	854	0.52	238
GR-315	687449	4261346	981	0.31	95
GR-315	687449	4261440	804	1.01	285
PZ-01	687773	4261269	898	0.33	60
PZ-01	687773	4261269	804	0.22	68
PZ-06	686930	4261600	852	0.63	288
PZ-07	687715	4261196	740	0.35	39

APPENDIX B
RESOURCE BLOCK MODEL FIGURES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>= 0.20
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>= 0.60
>= 0.75
>= 1.00
>= 1.50
>= 2.00
>= 4.00
Kisladag Deposit - year end 2009 Block Model
5m Au composites
year end 2009 design open pit
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red line: mineralized shell
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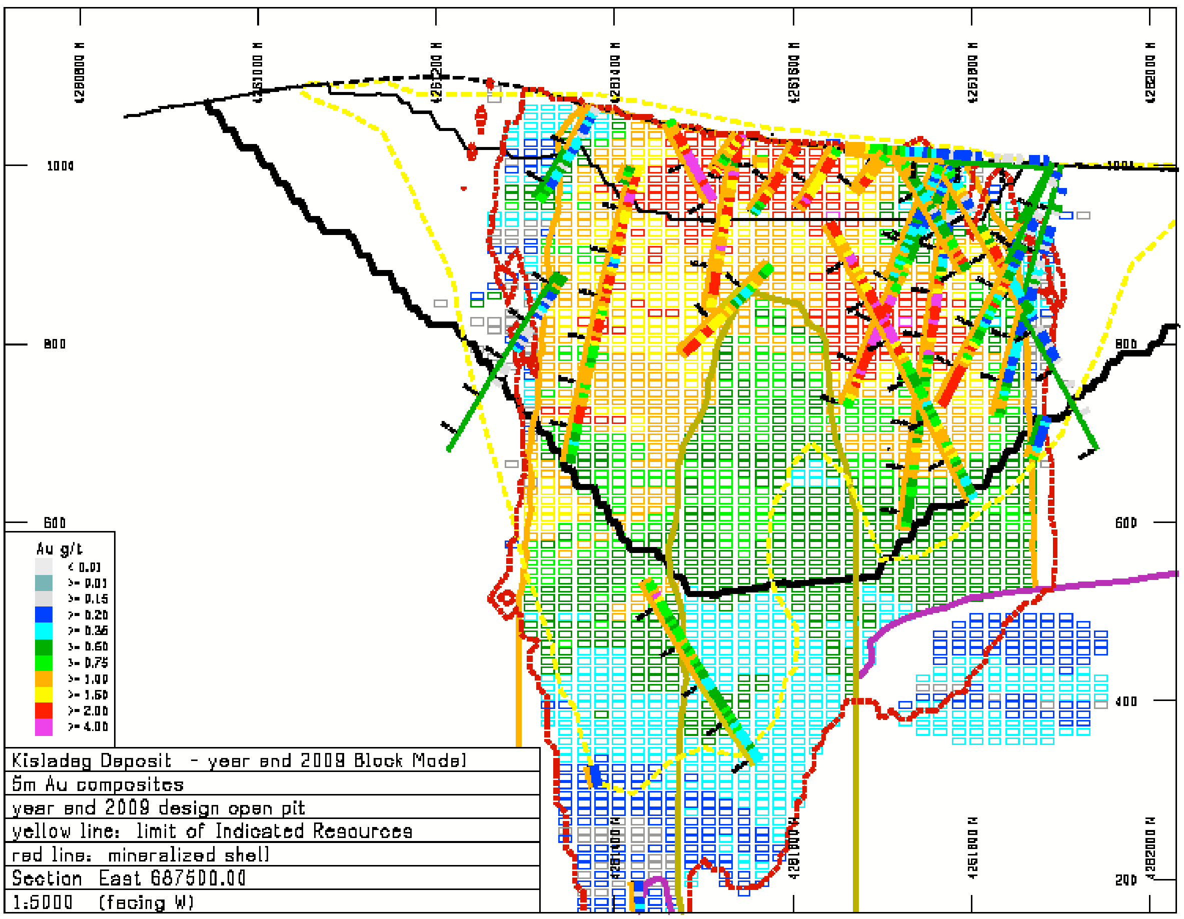
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>= 0.60
>= 0.75
>= 1.00
>= 1.50
>= 2.00
>= 4.00
Kisladag Deposit - year end 2009 Block Model
5m Au composites
year end 2009 design open pit
yellow line: limit of Indicated Resources
red line: mineralized shell
Section East 687500.00
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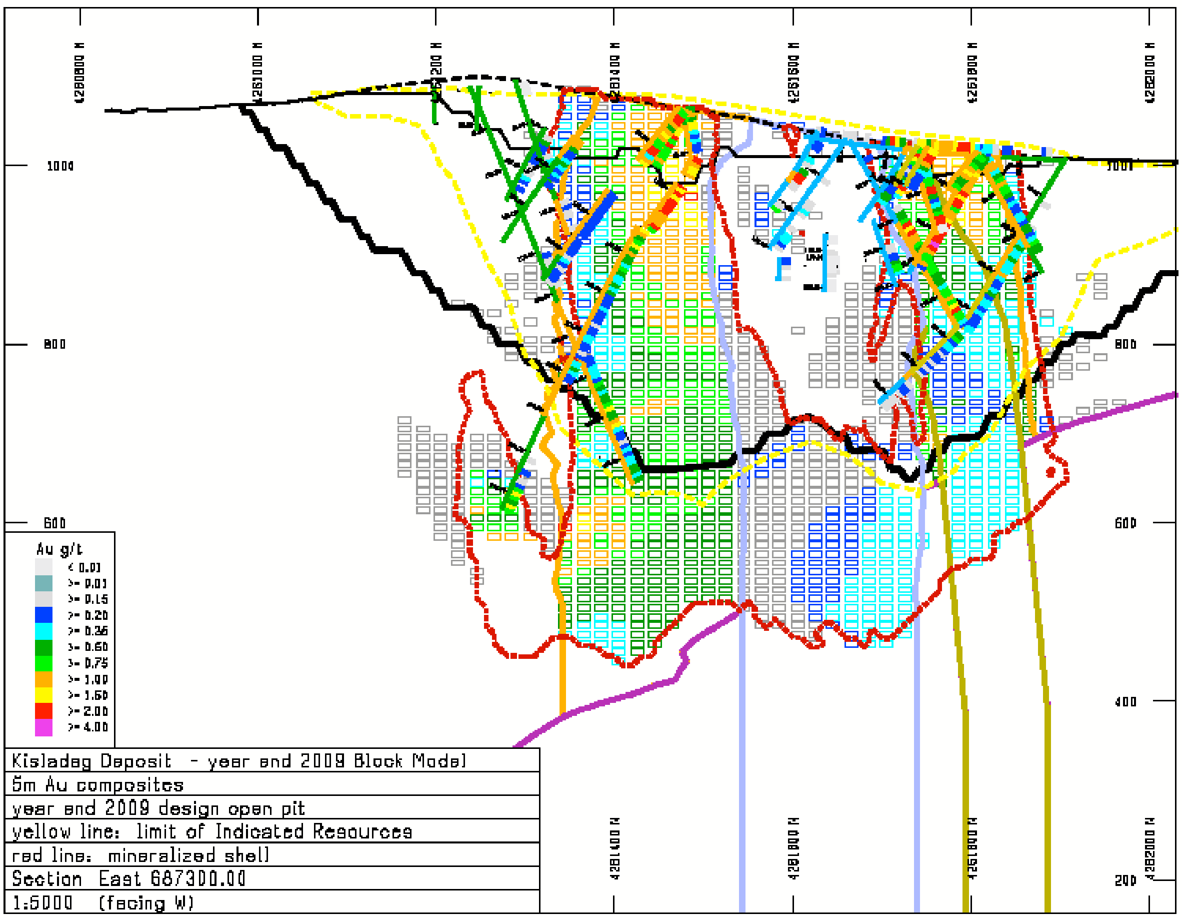
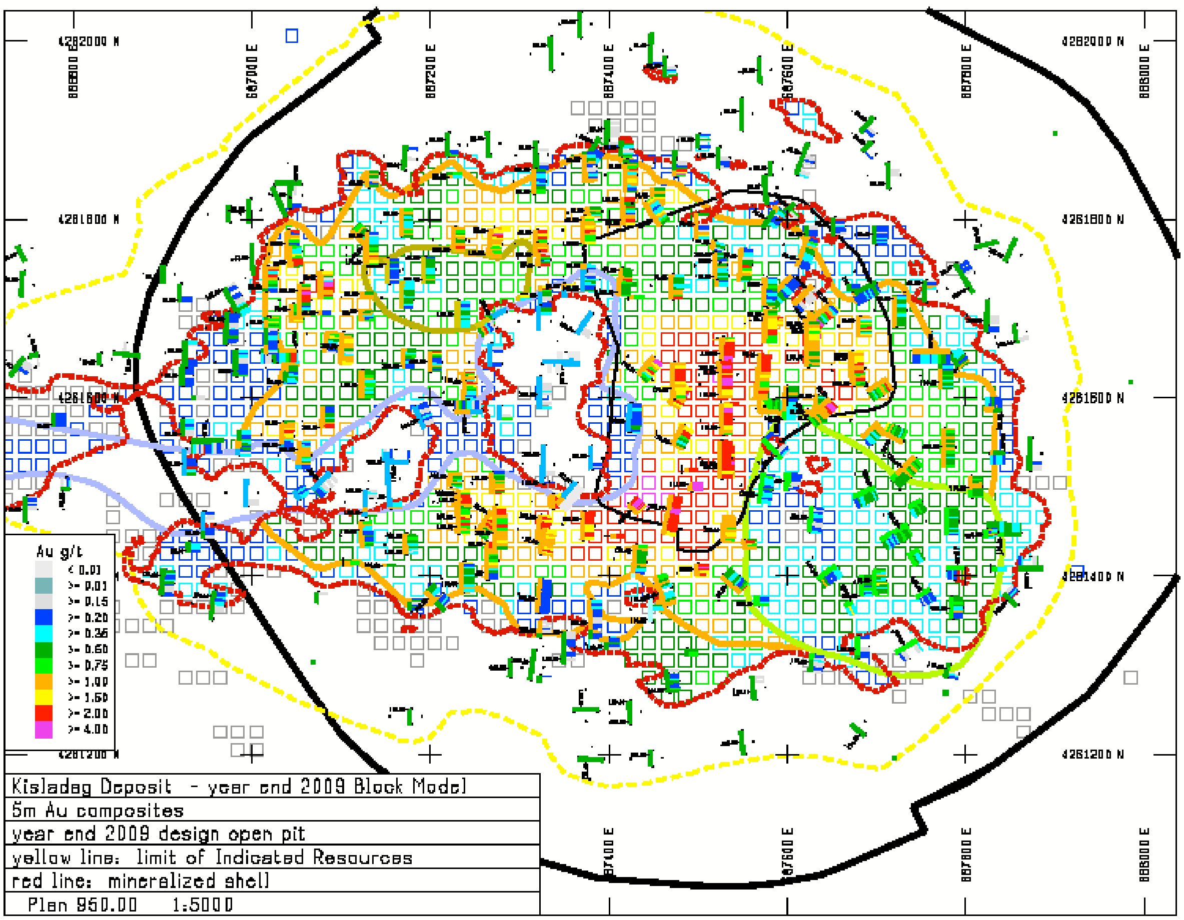
eldoradogold

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>= 0.01
>= 0.15
>= 0.20
>= 0.35
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>= 0.75
>= 1.00
>= 1.50
>= 2.00
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year end 2009 design open pit
yellow line: limit of Indicated Resources
red line: mineralized shell
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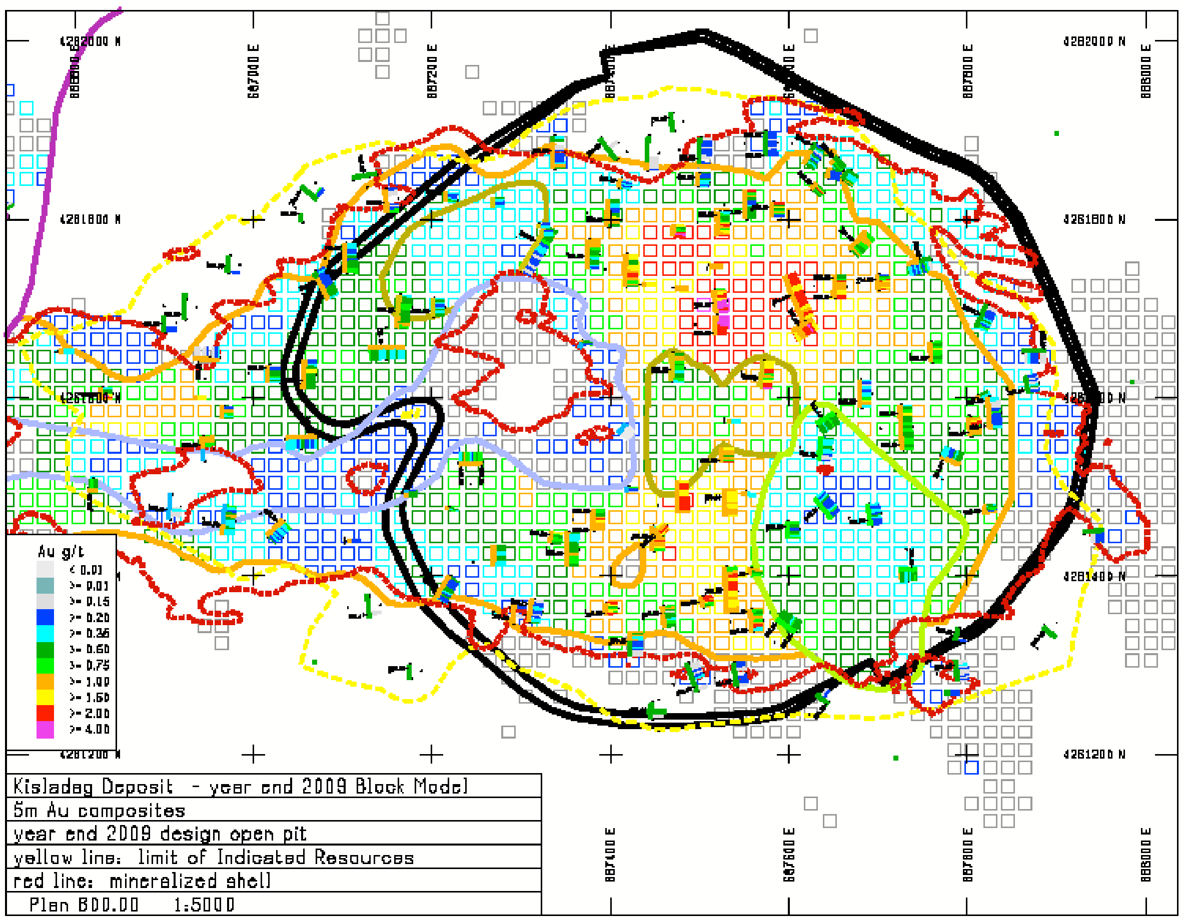
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>= 0.15
>= 0.20
>= 0.35
>= 0.60
>= 0.75
>= 1.00
>= 1.50
>= 2.00
>= 4.00
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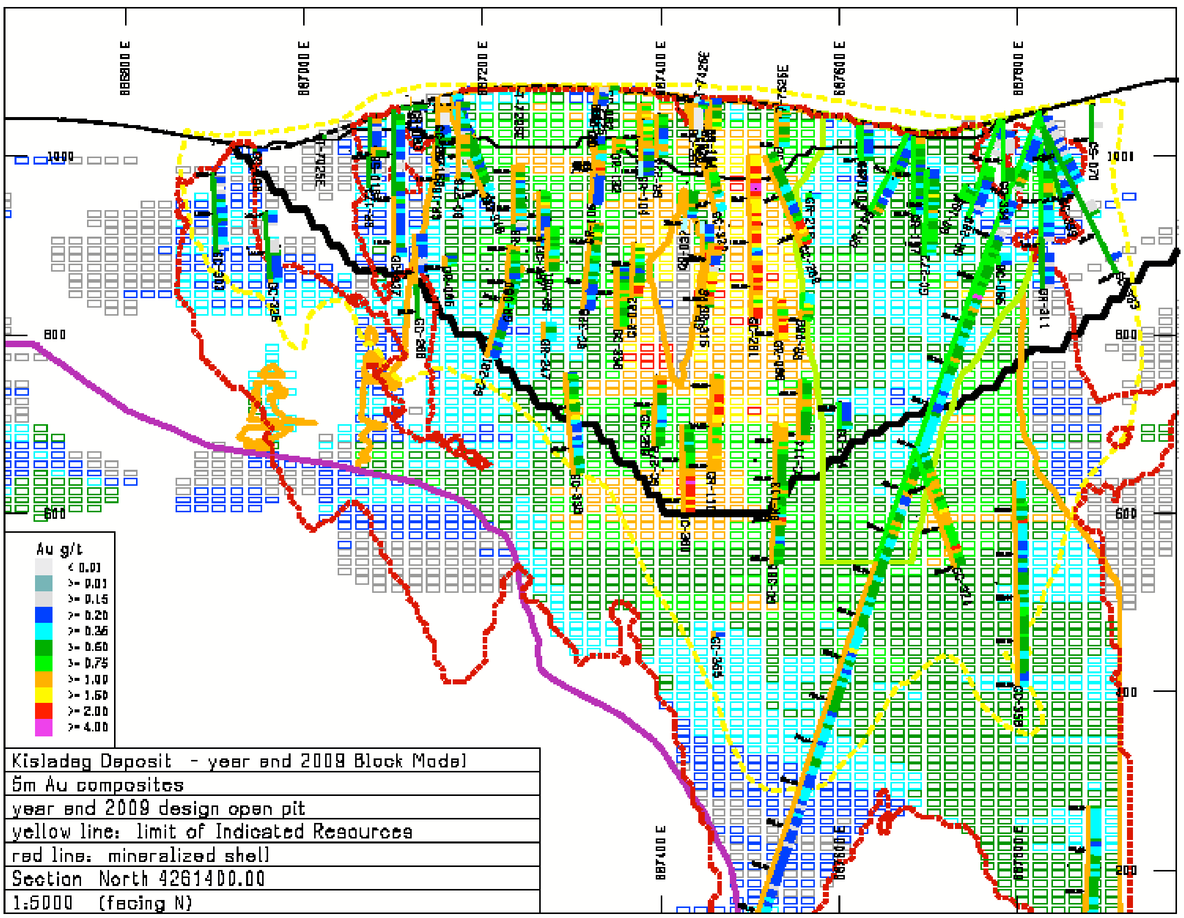
5m Au composites
year end 2009 design open pit
yellow line: limit of Indicated Resources
red line: mineralized shell
Plan 800.00 1:5000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800
600
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200
Au g/t
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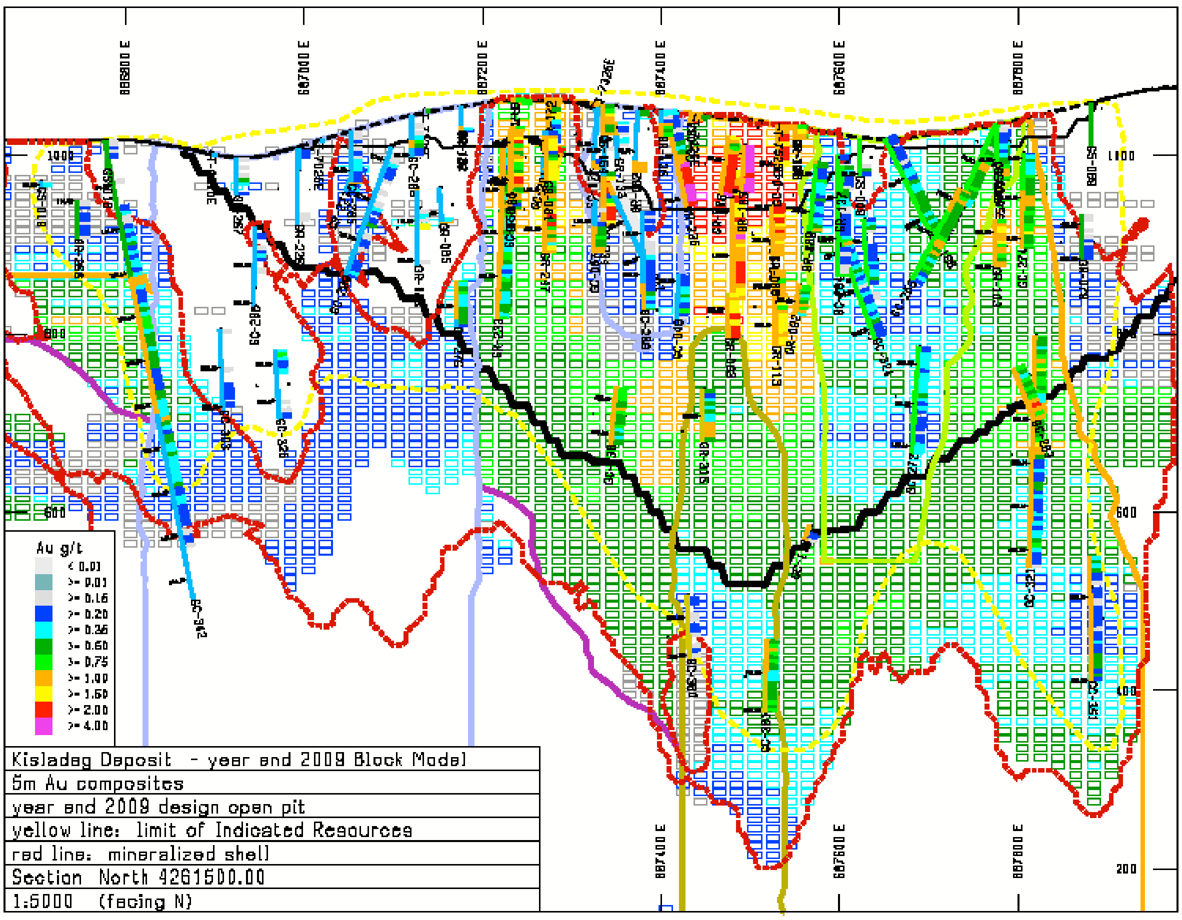
>= 0.01
>= 0.15
>= 0.20
>= 0.35
>= 0.60
>= 0.75
>= 1.00
>= 1.50
>= 2.00
>= 4.00
Kisladag Deposit - year end 2009 Block Model
5m Au composites
year end 2009 design open pit
yellow line: limit of Indicated Resources
red line: mineralized shell
Section North 4261400.00
1:5000 (facing N)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400
200
Au g/t
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>= 0.01
>= 0.16
>= 0.20
>= 0.36
>= 0.60
>= 0.75
>= 1.00
>= 1.60
>= 2.00
>= 4.00
Kisladag Deposit - year end 2009 Block Model
5m Au composites
year end 2009 design open pit
yellow line: limit of Indicated Resources
red line: mineralized shell
Section North 4261500.00
1:5000 (facing N)