ELDORADO GOLD CORP /FI
Form 6-K
April 17, 2003

FORM 6K

UNITED STATES
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

Report of Foreign Issuer

Pursuant to Rule 13a-16 or 15d-16 of
the Securities Exchange Act of 1934

For the month of **April, 2003**

Commission File Number **001-31522**

Eldorado Gold Corporation

(Translation of registrant s name into English)

Suite 920 - 1055 West Hasting Street
Vancouver, British Columbia V6E 2E9
(Address of principal executive offices)

Indicate by check mark whether the registrant files or will file annual reports under cover Form 20-F or Form 40-F

Form 20-F ___ Form 40-F X

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(1): ______

Note: Regulation S-T Rule 101(b)(1) only permits the submission in paper of a Form 6-K if submitted solely to provide an attached annual report to security holders.

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(7): ______

Note: Regulation S-T Rule 101(b)(7) only permits the submission in paper of a Form 6-K if submitted to furnish a report or other document that the registrant foreign private issuer must furnish and make public under the laws of the jurisdiction in which the registrant is incorporated, domiciled or legally organized (the registrant s home country), or under the rules of the home country exchange on which the registrant s securities are traded, as long as the report or other document is not a press release, is not required to be and has not been distributed to the registrant s security holders, and, if discussing a material event, has already been the subject of a Form 6-K submission or other Commission filing on EDGAR.

Indicate by check mark whether by furnishing the information contained in this Form, the registrant is also thereby furnishing the information to the Commission pursuant to rule 12g3-2(b) under the Securities Exchange Act of 1934.

Yes ___ No _X_

If Yes is marked, indicate below the file number assigned to the registrant in connection with Rule 12g3-2(b) 82___

SIGNATURE

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

ELDORADO GOLD CORPORATION

Date: April 17, 2003

/s/ Dawn Moss
Dawn Moss, Corporate Secretary

Suite 200, 1550 Alberni Street
Vancouver, British Columbia, Canada V63 1A5
Tel. (604) 689-5767 Fax: (604) 689-3918
www.hatch.ca

CONSENT OF AUTHOR

To: Toronto Stock Exchange
American Stock Exchange

I, Robert Duncan Henderson, P. Eng, do hereby consent to the filing, with the regulatory authorities referred to above, of the technical report titled Kisladag Project Feasibility Study and dated March 2003 (the Technical Report) and to the written disclosures of the Technical Report, and of extracts from or a summary of the Technical Report in the written disclosure being filed by Eldorado Gold Corporation.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the Technical Report.

Dated this 14th day of April 2003

(Robert Henderson)

Robert Henderson, P.Eng.
Project Manager
Hatch Vancouver

CERTIFICATE OF AUTHOR

I, Callum Leith Brown Grant, P.Eng., do hereby certify that:

I. I am currently employed as Manager of Geology & Mining by:

HATCH Associates Ltd.,
Suite 200, 1550 Alberni Street,
Vancouver, British Columbia,
CANADA V6G 1A5

II. I graduated with the degree of B.Sc. Geology (Honours) from the University of Aberdeen, Scotland in 1971. In addition I obtained the degree of M.Eng. (Mining) from McGill University in 1977.

III. I am a member of the Association of Professional Engineers and Geoscientists of the Province of British Columbia, and of the Association of Professional Engineers of the Province of Ontario.

IV. I have worked as a geologist and mining engineer for 27 years since my graduation from my first university.

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

VI. I am responsible for reviewing the mining section of this report (the Technical Report). I visited the property and sample preparation facilities of Eldorado in Turkey in September 2002.

VII. I have not had any prior involvement with the property that is the subject of this Technical Report.

VIII. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

IX. I am independent of the issuer applying all the tests in section 1.5 of National Instrument 43-101.

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

XI. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 14th day of April, 2003

(Callum Grant)

CLB Grant, P.Eng.
Manager Geology & Mining
Hatch Vancouver

CERTIFICATE OF AUTHOR

I, Robert Duncan Henderson, P.Eng., do hereby certify that:

I. I am currently employed as Project Manager by:

HATCH Associates Ltd.,
Suite 200, 1550 Alberni Street,
Vancouver, British Columbia,
CANADA V6G 1A5

II. I graduated with the degree of B.Sc. Chemical Engineering (Honours) from the University of Cape Town, South Africa in 1984. In addition I obtained the degree of MBA from the University of Cape Town, South Africa in 1991.

III. I am a member of the Association of Professional Engineers and Geoscientists of the Province of British Columbia (License number 23330).

IV. I have worked as a mineral processing engineer for 19 years since graduation.

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

VI. I supervised the preparation of the Kisladag Feasibility Study and am responsible for the overall preparation of this Technical Report. I visited the Kisladag property in October 2002.

VII. I have had prior involvement with the Kisladag property. The nature of my prior involvement was preparation of conceptual metallurgical process options in 1999.

VIII. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

IX. I am independent of the issuer applying all the tests in section 1.5 of National Instrument 43-101.

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

XI. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 14th day of April, 2003

(Robert Henderson)

Robert Henderson, P.Eng.
Project Manager
Hatch Vancouver

STATEMENT OF QUALIFICATIONS

CERTIFICATE

I, G.H. Giroux, of 982 Broadview Drive, North Vancouver, British Columbia, do hereby certify that:

1) I am a consulting geological engineer with an office at #513 675 West Hastings Street, Vancouver, British Columbia, working as an Associate with Micon International Limited.

2) I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M.A. Sc., both in Geological Engineering.

3) I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.

4) I have practised my profession continuously since 1970.

5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.

5) This report is based on a study of the data and literature available on the KiÕlada Project. I am responsible for the resource estimations completed in Vancouver during 2002-3. A site visit and examination of the property, drill core and sample preparation facility was made between the dates September 8 and September 16, 2002.

5) I have had prior involvement with the property completing earlier resource estimations in 1999, 2000 and 2002 as described in the Bibliography.

5) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report.

9) I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

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

11) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public files on their websites accessible by the public.

Dated this 14th day of March, 2003

(Gary H. Giroux)

G. H. Giroux, P.Eng., MASc.

TECHNICAL REPORT

KISLADAG PROJECT

FEASIBILITY STUDY

PR 311235.001
FL311235.201
Rev. 0, March 2003

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NOTICE

This report was prepared for the sole and exclusive benefit of Eldorado Gold Corporation (Eldorado) by Hatch Associates Limited (Hatch). This document is meant to be read as a whole, and sections should not be read or relied upon out of context. This document contains the expression of the professional opinion of Hatch based on information available at the time of preparation. The quality of the information, conclusions and estimates contained herein is consistent with the intended level of accuracy as well as the circumstances and constraints under which the mandate was performed. The report includes information generated or provided by other outside sources identified herein. Hatch does not warrant the accuracy or completeness of data supplied by outside sources. This report is to be used by Eldorado only, subject to the terms and conditions of its contract with Hatch. Any other use or, or reliance on this report by any third party shall be at that party s sole risk.

ELDORADO GOLD CORPORATION
KISLADAG PROJECT

FEASIBILITY STUDY

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1. Project Overview

The Kisladag Project is planned to be a 10 million tonne per annum (mtpa) open pit, heap leach gold mine located in west-central Turkey. Since 1997, the Project has advanced through various stages of exploration to final feasibility stage. Preparation of this Feasibility Study follows an extensive drilling program in 2002, which culminated in a further increase in the mineral resource at Kisladag as reported in November 2002. Current activity is focused on obtaining the necessary permits and approvals to advance the project to a construction decision in 2003.

This Feasibility Study is prepared in accordance with the Standards of Disclosure for Mineral Projects as defined by National Instrument 43-101. The Measured and Indicated Mineral Resource estimated at a cut off grade of 0.4 g/t Au is 166.4 million tonnes at a grade of 1.13 g/t Au containing 6.05 million ounces of gold. In order to meet regulatory requirements, a mine production schedule has been developed to include only Measured and Indicated Resources. Inferred mineral resources within the design pit have not been considered reserve and have been assigned as waste material. The total Proven and Probable Mineral Reserves are estimated to be 115 million tonnes at a grade of 1.23 g/t Au. Of the total reserve, approximately 24% is oxide ore and 76% is primary ore. This quantity of ore will sustain the feed to a heap leach facility for a period of 14 years and gold will continue to be recovered in Year 15.

Production is scheduled to start at the end of 2004 following an 18-month design and construction period. The initial capital cost for construction of the Project is estimated to be US$54.4 million. In Year 5 of the operation, a further US$39.4 million will be required for mining equipment and expansion to the crushing plant. The cash operating cost in the first four years of production is estimated to be US$3.97 per tonne which is equivalent to US$138 /oz. Life of mine cash operating cost is estimated to be US$152 /oz based on US$3.82 per tonne of ore processed.

A mine production rate of 5 million tonnes per year of ore has been set for the first four years of the mine s life. Average daily production rates will be 16,100 tonnes per day (tpd) in ore and 13,000 tpd in waste during these initial four years. Annual ore production will increase to 7.5 million tonnes in Year 5, and to 10 mtpa the following year, remaining at that level until the end of mine life. The highest daily production rate occurs in Year 7 with a total movement of 79,000 tpd (ore plus waste). Total quantities of ore and waste will be 115 million tonnes and 106 million tonnes respectively over the mine life. The overall strip ratio will be 0.92. A mining contractor will initially be employed for waste movement and ore mining. In Year 4 of operations, Eldorado will begin to phase in its own mining fleet and mine workforce for completion of the Project.

Extensive metallurgical bench scale studies and column leach tests have identified that Kisladag ore is amenable to heap leaching technology. A gold recovery of 81% is projected for the oxide ore. The primary ore has a higher sulphide content and gold recovery is projected to be 60%. The ore will require a crush size of 80% passing 6.3 mm and a leach period of ninety days.

The Kisladag ore will be processed in a standard heap leach facility containing a three stage crushing circuit, 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 will be treated on site in a refinery and the final product will be gold doré bar. The average gold production in the first four years of operation is expected to be 143,000 ounces per annum increasing to 230,000 ounces per annum after Year 5.

Situated on the western edge of the Anatolian plateau, 200 km inland from the port city of Izmir, the Kisladag Project is well serviced by national roads and rail services. The project site is located at an elevation of approximately 1,000m in gently rolling topography. The climate is temperate with an average annual rainfall of 450 mm, most of which occurs from November to March.

The Project will employ 356 people at maximum production, the majority of workers being drawn from the local region. Infrastructure to support the mine will include an access road, a water well field with a 13 km water pipeline and a 30 km power transmission line. Supplies and services are available in the city of Usak, 35 km to the north.

An Environmental Impact Assessment (EIA) report has been submitted to the Turkish Ministry of Environment. The EIA has identified potential impacts the Project will have on the local environment and social structure and presents the mitigation measures required to minimise impacts while maximizing the benefits of the Project locally and nationally.

2. Introduction

Eldorado Gold Corporation (Eldorado) is an international gold mining company based in Vancouver, Canada, whose shares trade on the Toronto Stock Exchange (TSX: ELD) and American Stock Exchange (AMEX: EGO). Eldorado has gold assets in Turkey and in Brazil where gold production in 2002 from its Sao Bento mine was 103,000 ounces at a total cash cost of US$184/oz.

Eldorado owns a 100% interest in the Kisladag Project (the Project) through its wholly owned Turkish subsidiary, Tüprag Metal Madencilik Sanayi Ve Ticaret Limited Sirketi (Tüprag).

In September 2002, Eldorado commissioned Hatch Associates Limited (Hatch) to prepare a Kisladag Project Feasibility Study with the purpose of assisting Eldorado to obtain Project financing from institutional lenders, or through a public offering. This document presents a summary of the detailed information contained in the following Feasibility Study volumes:

VOLUME I	Section 1.0	Introduction

3. Disclaimer

The Kisladag Feasibility Report summarises the findings of a feasibility study of the Kisladag gold project undertaken by Hatch Associates Limited (Hatch) in late 2002 and early 2003.

This study evaluates all aspects of the Project including geology, ore reserves, mining, metallurgy, processing, infrastructure, environmental requirements, and financial evaluation. The study sets out the costs to construct and operate the mine and includes a financial analysis of the Project.

Hatch was responsible for the processing, costing, and site infrastructure aspects of the study (and preparation of the final report), while other consultants prepared the following components:

Micon International Limited (Micon) was contracted by Eldorado to complete the geology description and mineral resource estimate;

Eldorado completed the mine plan and pit design with input from Hatch;

Recommendations on geotechnical aspects and pit slope design were provided by Rockland Limited (Rockland);

Norwest Corporation (Norwest) completed the waste rock management report;

The Mines Group Inc. (Mines Group) was contracted by Eldorado to undertake the design of the heap leach pad.

Hatch summarized the EIA report prepared by Encon Environmental Consultancy Company (Encon) and Hatch included metallurgical recommendations from Kappes Cassiday Associates (KCA).

No due diligence of a legal or environmental nature was included in the Terms of Reference for the work, and where comments to this effect appear in the report, they have been extracted from other documents.

4. Property Description and Location

Kisladag is located in west-central Turkey between the major centres of Izmir, lying 180 kilometres to the west on the Aegean coast, and the capital city of Ankara, 350 kilometres to the northeast. The Project site lies 35 km southwest of the city of Usak (population 165,000) near the village of Gumuskol. Approximate Project co-ordinates are latitude $38^{0}30$ N and longitude $29^{0}12$ E.

Figure 1 provides information on the location of the Project.

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. The highway from Ankara to Usak/Izmir is a major national trucking route and is in good condition. From the port city of Izmir, Kisladag can be reached via an all-weather, paved road some 246 kms distant. The preferred access route from Izmir is to travel east to Usak on the Salihli/Usak highway, and then south for 37 km on a paved road to Ulubey/Esme. A new 5.3 km long access road will connect the site to the Ulubey/Esme road once the Project is established.

Although Usak has an airport, the facility is currently closed due to cutbacks. The national rail system is active and there is a rail siding at Inay, approximately 10 km from the site.

The Kisladag 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, Tuprag retains the right to explore and develop any mineral resources contained within the license area for an indefinite period of time, providing fees and taxes are maintained. The license boundaries are shown on Figure 2.

Figure 1: General Location Map

Figure 2: Kisladag Land Position

5. Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The Project area sits on the western edge of the Anatolian Plateau at an elevation of approximately 1,000 metres, in gentle rolling topography. Local elevations range from 1,300 m above sea level (asl) to 700 m asl. The climate in this region is transitional between Mediterranean and Continental regimes and is characterized by warm dry summers and mild wet winters. Temperatures average 14°C for the year, varying from an average minimum of 3°C in January to an average maximum of 33°C in August. Annual rainfall is approximately 425 millimetres occurring mainly in the winter months (of the total annual precipitation, 31% falls in spring (March to May) and 48% in winter (November to February). The maximum daily precipitation recorded is 47 mm. According to the records of the Esme and Ulubey stations, mild northerly winds are dominant in the area with maximum wind speeds of 51 61 km/hour.

The flora in the area is transitional between a coastal Aegean type and the more continental Anatolian regions with sparse vegetation, small oak and pine trees, shrubs, and meadow grasses.

There are a number of small villages within the concession area where local residents are engaged in the grazing of domestic livestock and marginal farming of wheat from non-irrigated lands. The primary means of support for the locals is agricultural production. The soils of the area are dominantly thin brown forest soils up to 0.4 m deep. Approximately 95% of the concession area falls within Turkish land category VI and VII, which is generally regarded as unsuitable for agriculture.

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 10 years ago, however only approximately 24% of the concession area is currently planted with pine and cedar.

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 Kisla village.

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

The Kisladag site lies within the first and second degree seismic zones as defined by the relevant Turkish design code. This is equivalent to earthquake Zone 4 in the American Uniform Building Code. The effective ground acceleration coefficient used in the study is 0.4 g.

Significant population centres near the Project site are the municipalities of Kisla located 4 km to the east with a population of 2,150; Ulubey 14 km to the south east with a population of 5,100 and Esme which is 18 km to the south west with a population of 11,600. The villages of Gumuskol, Ovacik, Katrancilar and Sogutlu are located within 3 kilometres of the Project site. The city of Usak (population 166,000, 42 km by road from site) is the provincial capital of Usak province and is a significant industrial centre. Although Usak is not recognized as a major mining district, there are a number of industrial minerals operations in the region including lime production and marble quarries.

Turkey has a substantial mining industry supported by a well-developed infrastructure. Mineral production is dominated by industrial minerals, energy (coal) and base metal sectors by both domestic and international mining companies. The majority of the Turkish mining operations are public sector companies and coal, iron, boron and chromite are the major products. Although the history of gold mining in Turkey predates Roman times, production of gold in modern times only began in 2001 with the start-up of the Ovacik Mine (Newmont Mining) located approximately 250 km to the west of Kisladag.

6. History

Eldorado acquired the Kisladag 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 Kisladag 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 metres on strike by 600 metres wide. This work was followed in 1997 by 2,745 metres of trench sampling, and 1,541 metres 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 metres and effectively confirmed the potential for a low grade bulk tonnage gold deposit, and in 1999 an additional 5,212 metres of HQ core drilling and 1,600 metres 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 million tonnes of 1.49 g/t, plus an Inferred resource of 31.1 million tonnes at 1.35 g/t (all based on a 0.8 g/t cut-off grade).

In 2000, a reverse circulation (RC) drill program totalling 7,605 metres (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 million tonnes for the deposit at an average grade of 1.20 g/t gold, that is 4.85 million ounces of contained gold in oxides and primary ore (using a cut-off grade of 0.4g/t Au).

Early in 2002, a combined total of 9,134 m (RC, DDH and Percussion) was completed.

Metallurgical testwork 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 Kisladag Project site. Early receipt of this permit was made possible by the high level of support the Project has received from within the Usak province as well as at the central government level.

Based on the concept of recovering gold by heap leaching, in 2001 Eldorado commissioned a Prefeasibility Study by Kilborn Engineering Pacific Limited (Kilborn). This study considered an operation to treat 3.4 million tonnes per annum 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.

7. Geology

7.1 Geological Setting

The Kisladag Project is located within one of several mid-to late-Tertiary volcanic complexes of western Turkey that have been related to a regional structural subduction zone along the Hellenic Trench lying to the southwest. In the Kisladag region, these volcanics erupted onto a basement of schist at the northeast margin of an uplifted terrain known as the Menderes Massif.

The Kisladag gold deposit itself is located on the north-facing side of the hill Gökgöz Tepe , within a prominent volcanic structure that forms part of a 90 km^2 northeast-southwest trending volcanic complex. The volcanic centre is estimated to be Pliocene in age (based on lacustrine sediments lying within the volcanics).

Within the deposit area, the main lithologies have been logged and mapped by Eldorado geologists, and are generally classed as pyroclastics with intruded latite porphyry. Intra-mineral breccias indicate that at least two, and possibly three, separate mineralized intrusives are present. These intrusives are compositionally and texturally similar, consisting of a few percent of feldspar and hornblende phenocrysts in a fine grained latite matrix. A late, weakly mineralized to barren stock, marks the end of mineralizing activity.

7.2 Deposit Types

The Kisladag deposit consists of porphyry-style gold mineralization centred on a series of overlapping sub-volcanic intrusives. A lesser amount of mineralization is hosted by subaerial volcanics, which surround and partially overlap the mineralized intrusives along their southern and eastern margins. Geological consultant Richard Sillitoe visited the property from August 29 to September 2, 2000, and concluded in his report:

> *Kisladag 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.*

Figure 3: Property Geology Map

1.1

ELDORADO GOLD CORPORATION - KISLADAG PROJECT
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7.3 Mineralization

Gold mineralization with traces of molybdenum, zinc, lead and copper encircles a late barren stock at the Kisladag deposit. Higher-grade gold mineralization forms a horseshoe shaped zone around the northern, southern and eastern sides of the late intrusive stock, which is 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 and quartz flooding of hydrothermal breccias, multiple phases of quartz-pyrite veining containing gold and late sulphide-rich quartz veining with traces of molybdenum, sphalerite, galena and tetrahedrite. A final phase of vuggy barren silica veining is associated with intense acid leaching but is effectively barren of gold mineralization. Outcrops of this late silicification form prominent vein and sill-like bodies away from the main deposit area and have been interpreted as eroded remnants of the original leached cap (lithocap).

In general, the amount of stockwork veining decreases with depth, especially below the 650 metres (asl) elevation. Higher grade mineralization (above 2 ppm Au) has been traced from surface to depths greater than 250 metres 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 400 metres below surface).

Pyrite is the dominant sulphide mineral with other sulphide minerals identified in microscopic studies as chalcopyrite, sphalerite, tetrahedrite, galena and molybdenite. Traces of cinnabar and orpiment have also been described in trench samples and traces of scheelite, magnetite and rutile are also present. Trace amounts of secondary chalcocite are also present at or below the oxide-primary boundary.

Oxidation tends to be deeper on the uphill (southern) side of the deposit (from 30 m to 80 m deep) compared to the downhill (northern) side of the deposit where oxidation is limited to between 20 and 50 metres 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 to 60 m in depth. 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.

The centre of the volcanic complex is marked by a broad northwest trending alteration zone measuring approximately 5 kms by 3 kms with the Kisladag deposit located near the centre of this zone. A complex pattern of partially overlapping alteration types is present in the deposit area. High-grade mineralization

is typically associated with early potassic altered (feldspar-biotite) replacement overprinted by pervasive silica-illite, tourmaline and sericite. Locally, there are relict zones of an early potassic assemblage of chlorite, biotite and magnetite. A later advanced argillic to intermediate argillic phase, consisting of kaolinite in the main deposit and grading outward from the deposit to kaolinite cut by secondary alunite, overprints the early alteration phases.

7.4 Exploration

The Kisladag deposit has been explored using sequential campaigns of regional satellite evaluation, regional reconnaissance, stream sediment sampling, and geophysics leading to more detailed phases of local mapping and drilling in the period 1998 to 2002.

7.5 Drilling and Trenching

The first phase of drilling was completed in 1997 and consisted of 32 short percussion holes totalling 1,640 metres. These holes, labelled GS, tested the top 50 metres of the main gold soil anomalies and confirmed the mineralized zones.

In 1998 a second phase of six HQ core drill holes (GC-33 to GC-38) totalling about 1,059 metres tested the main gold anomaly. Drilling extended the gold values found in trenching to approximately 250 metres in depth and confirmed the potential for a low grade, bulk tonnage gold deposit. In 1999 another 23 core holes totalling approximately 5,212 metres were completed to depths of 150 to 450 metres.

During the 2000 field season an additional 11 percussion drill holes (576 m), 6 diamond drill holes (917 m), 30 reverse circulation holes (7,582 m) and 3 surface trenches (1,611 m) were completed. In 2001, 5 RC holes totalling 1,020 metres were drilled as pre-collars. An additional two diamond drill holes were drilled and five previously drilled holes were deepened to test the deposit at depth. Another 5,995 metres were also drilled in 2002 with reverse circulation, targeting the mineralized boundary of the porphyry gold deposit to the south, north and east of the main zone in addition to 8 large diameter core holes (PQ) for metallurgical sampling.

7.6 Sampling Method and Approach

Percussion drill holes completed in 1997 and 2000 were drilled using a pneumatic drill operated by an Eldorado technician and supervised by an Eldorado geologist. The drill used a downhole hammer with a 76 mm diameter bit. Holes were drilled dry, and the rock chips and cuttings were collected on a plastic

sheet spread around the drill stem. Samples were taken on 1 m intervals by collecting the entire 7.5 kg sample and transferring it into a numbered plastic bag. A small amount of sample was taken from the top of the bag and saved for logging and chipboard construction. The remaining sample was shipped to Eldorado s sample preparation facility at Canakkale.

The majority of diamond drill holes at Kisladag were completed using a conventional Longyear 38 diesel-hydraulic drill rig recovering HQ and NQ core, and all holes were surveyed with a downhole instrument. In some cases where difficult ground conditions near surface were encountered, holes were collared with larger HQ rods and finished with NQ. Collection and placement of the core in the core boxes followed accepted industry standards under the supervision of qualified geological staff.

A track mounted reverse circulation drill was used in 2002 drilling. For holes GR-118 to GR-149 samples were collected through a cross over sub about 1 m up the drill stem from the drill bit. For the later holes starting at GR-150 to the end of the program, a face sampling bit was used. Studies have shown that in gold deposits a face sampling bit will recover more fine gold that might be lost with a regular bit and in general are less susceptible to down hole contamination. The entire sample from a 2.5 metre run was collected in cloth bags from the cyclone. The sample was then quartered at the drill site with a Jones splitter and ¼ (or about 6 to 8 kg) was sealed in a marked plastic bag. A small sub sample of chips was collected from the rejects for logging and chipboard creation. The samples were shipped via bonded courier to the sample preparation facility at Canakkale.

7.7 Sample Preparation, Analyses and Security

Samples from trenching, percussion, diamond drilling and reverse circulation drilling were received at Eldorado s sample preparation and storage facility at Canakkale. This facility houses a modern, clean and well run laboratory equipped with two large dryers, three crushers, LM-2 pulverizer and Jones splitter.

In the 1998 programs, each trench sample was transferred to drying trays and dried. The entire dried sample was crushed to -$^1/_8$ inch using jaw and cone crushers. The sample was then homogenized in a mixing barrel and split using a Jones splitter to produce a 1.5 kg fraction for pulverization. A LM-2 ring and puck pulverizer was used to grind to 100% -30 mesh; a 250 gm sub-sample was then taken for final assay. The rejects from both crushing and pulverization are labelled and stored at the Canakkale facility. Trench samples were shipped to SGS Laboratories in France where they were pulverized to 95% -150 mesh and assayed using a one assay tone fire assay with gravimetric finish.

Samples collected in 1998 from percussion drilling were transferred to drying trays and dried overnight. The entire sample was crushed to -1/8 inch using jaw and cone crushers. A Jones splitter was used to produce a 1.5 kg split that was pulverized in a plate grinder to 100% -30 mesh. A 250 g sub sample was split and sent for assay at the SGS Laboratories in France where they were pulverized to 95% -150 mesh and assayed using a one assay tone fire assay with gravimetric finish. Rejects from crushing and pulverizing are stored. Percussion samples taken in 1999 were dried at Canakkale and crushed to 70% -2 mm. A 1 2 kg split was taken with a Jones splitter with rejects stored. The sample was pulverized to 95% -150 mesh in an LM2 mill with a 150 g sample selected and shipped to ALS Chemex Laboratories in Vancouver (Chemex) for a one assay ton fire assay with gravimetric finish. The reject pulp was re-bagged and stored.

Detailed geological, mineralogical, alteration and geotechnical data were logged and recorded during the diamond drilling programs at Kisladag in 1998 and 1999. Half core cut with a diamond saw, collected from the drillcore in 1998, was dried, crushed to 80% passing -6.5 mm, and a 1 2 kg split taken with a Jones splitter. The sample was then pulverized to 100% -30 mesh and a 250 gm sub-sample taken for final pulverizing and assaying. Core samples taken after 1998 (Holes from GC-39 to present) were halved with a diamond saw, dried and crushed to 70% -2 mm and a 1-2 kg split taken for grinding. This sub-sample was then pulverized to 95% -150 mesh in a LM-2 mill then a 50 g split taken for assay.

Samples from all drill holes are shipped to Tuprag s sample preparation laboratory in Canakkale where they are dried at 35 40°C overnight. The samples are then split with a Jones Splitter to produce a 1 2 kg sample for pulverization. The remaining reject sample is stored. The sample is pulverized to 95% -150 mesh in a LM-2 ring and puck pulverizer. A 100 g sample is taken directly from the pulverizer, at various locations around the bowl, using a spoon. This sample is shipped to Chemex in Vancouver for a one assay ton fire assay with gravimetric finish. The reject pulp was re-bagged and stored.

Specific Gravity Determination

A total of 573 measurements for specific gravity were collected from drill core (Holes GC-33 to GC-61) using a wax technique. Samples of drill core from oxide, mixed oxide/primary and primary horizons were weighed dry. The samples were then coated with wax and weighed again. Finally, the wax-coated samples were weighed in water. The volumes of the sample and the sample in water were determined and the appropriate specific gravity calculated. An additional 1,319 specific gravity determinations were collected from drill core in 2002. A method of weighing the core dry and weighting the core in water was used for these determinations.

The combined database of density measurements was used for the block modelling and estimation of the Kisladag resources. The specific gravity for ore and waste in the model ranged from 2.37 to 2.42.

Independent Care and Custody Sampling

Two independent checks of sampling at Canakkale were completed by Micon. In 1999, 5 core samples were selected and followed through Eldorado s sample preparation facility maintaining strict care and custody procedures at all times. The samples were carried back to Canada by Micon where they were submitted (along with two blanks) to ALS Chemex Laboratories in Mississauga, Ontario. These samples were analyzed in duplicate and all returned gold values within the range expected, based on visual inspection of the core.

During another site visit in September 2002, Micon collected two splits at the reverse circulation drill (hole GR-176). The samples were then transported to Canakkale where a pulp was prepared for shipment to Chemex in Vancouver, where final assaying was completed. While both samples returned slightly higher gold values compared to the originals, both sets of data were similar and within a range consistent with accuracies possible in sampling and assaying for gold.

7.8 Data Verification

A Quality Control and Quality Assurance (QA/QC) program at Kisladag was instituted from the beginning of the exploration program with blanks, standards and duplicate samples submitted on a routine basis. For every 20th sample a second split is taken and submitted with a different sample number. In addition, a blank sample is added about every 20th sample, as is a standard, resulting in 1 out of every 7 samples being a duplicate, blank or standard.

After pulverizing, sample pulps are sealed in plastic bags, placed in a locked metal box and shipped to Chemex laboratory in Vancouver, for a standard gold fire assay.

During the 1999 drill program a total of 84 blank samples were submitted along with samples from drill holes GC-39 to GC-61. Of these samples 80 returned values of below detection for gold. Four samples showed detectable gold values, with the most significant error occurring in Hole GC-48 Sample 98494 with a blank assayed at 0.54 g/t Au. This resulted from a numbering error and was resolved. The remaining three samples above detection were less than 0.1 g/t Au.

A total of 166 blank samples were submitted during the 2000 drill campaign along with samples from drill holes GR-79 to GR-108. Of these samples, 163 returned values of below detection for gold and two reported at the detection limit of 0.03 g/t Au. Only one sample reported above the detection limit, sample # 152896 from Hole GR-87. This sample had a grade of 0.75 g/t Au. Upon receipt of these results Eldorado asked for and received a second set of assays for the samples run with this blank.

A total of 60 blank samples were submitted during the 2002 drill campaign along with samples from drill holes GC-114 to GC-117 and GR-109 to GR-113. Of these samples, 50 returned values of below detection for gold, 9 samples reported at or below 10 ppb Au. One blank was reported at over 300 ppb and this sample and those around it were re-assayed. The false value was a result of a numbering error and was corrected. One other blank reported at 15 ppb was re-assayed.

During 1999, a total of 71 samples of a Standard Y were submitted to the assay stream, at the sample preparation facility. Standard Y was prepared by Tüprag in large batches and history has shown it is repeatable. The mean value for Standard-Y is 0.51 g/t Au. An allowable range for values would be within 2 standard deviations above or below the mean. Only one sample from Hole GC-59, sample 99532, was outside these limits with a value of 1.11 g/t Au.

A total of 94 samples of a Standard Y were submitted at the sample preparation facility for the 2000 drill campaign. Five samples were outside the limits of the mean ± 20% for Standard-Y (0.51 g/t Au). Three with sample numbers 153216 (0.36 g/t), 154752 (0.39 g/t) and 155176 (0.39 g/t) were below, and two with numbers 153936 (0.69 g/t) and 153024 (0.66 g/t) were above. A second standard called KIS-2 was also used in the 2000 drill campaign with an average grade of 0.75 g/t Au. A total of 50 samples of this standard were placed in the sample stream at similar intervals, mentioned above. All samples were within a range of the mean ±12%.

A total of 113 standard samples were submitted to the Canakkale sample preparation facility for the 2002 drill campaign. Several different standards were used during the 2002 drill program.

A total of 41 standard samples designated KIS-1 were used. The expected value for this standard, based on 67 analyses, is 0.24 g/t Au. All but 1 of the assays for standard KIS-1 were within ±2 standard deviations of the mean value (see Figure 4). Sample 159630 returned a value of 0.505 g/t Au, a value more likely Standard KIS-10.

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A standard KIS-10 was used 64 times with the expected mean of 0.44 g/t Au. All but one sample (158790 which assayed 0.405 g/t Au), were within the limits ±2 standard deviations of the mean (see Figure 5).

Standard KIS-9 with an expected value of 0.5 g/t Au was also used 8 times with all results within the range of ± 2 standard deviations of the expected mean (see Figure 6).

Figure 4: Results for Standard KIS-1 Tested in 2002

Figure 5: Results for Standard KIS-10 Tested in 2002

Figure 6: Results for Standard KIS-9 Tested in 2002

Reject Duplicates

During the 1999 drill campaign about 1 in 20 samples were duplicated in the sample laboratory, with a second split taken after crushing and submitted with a different sample number. In this way, a blind duplicate was subjected to the same pulverization and assay procedure during the same time frame as the original sample. A total of 87 duplicated samples were re-assayed in this manner. The results showed an excellent agreement (coefficient of correlation of 0.996) and no indication of bias. In addition to a blind confirmation of the assay procedure, these checks indicate a low level of sampling variability.

As a further test, the sample pulps for 79 of these reject duplicates were sent to Intertek Testing Services Bondar Clegg (Bondar) in Vancouver, for a gold assay. A total of 79 of the 87 samples were assayed for gold at Bondar. The correlation coefficient was lower, but still excellent at 0.976 and there was no indication of any bias. There was, however, more scatter about the equal value line indicating more variability. This could be introduced by a second laboratory and different assay procedures or by simple sample variability from the smaller sample volume contained within the pulp.

A total of 208 samples were re-assayed in 2000 by submitting a second split of the crushed sample. There is excellent agreement (coefficient of correlation of 0.992) and no indication of bias. In addition to a blind confirmation of the assay procedure, these checks indicate a low level of sampling variability. A Thompson-Howarth test calculates the precision at concentration 1.0 g/t Au of ± 9.25 %.

A total of 116 samples were routinely duplicated during the 2002 drill campaign and sent to the primary laboratory Chemex Vancouver, as blind checks. There is no indication of any sampling or analytical bias with the best fit regression line almost mirroring the equal value line. The correlation coefficient is 0.9.

Micon concluded that quality control tests at Kisladag have shown that the analytical results used in the resource calculations are reliable. The standards and blanks submitted routinely in the assay stream are reporting back within acceptable limits. The case where blanks or standards assayed outside these limits resulted in samples being re-analysed. The routine duplication of reject samples, submitted blind to the primary lab, is another check on the assay procedure. This test again showed good reproducibility (Correlation Coefficient of 0.992) and is a good indication of overall sampling variability of ± 9.25%. The second laboratory checks with Bondar indicate that Chemex, results on average, are conservative with Bondar overestimating gold relative to Chemex by 0.023 g/t.

7.9 Adjacent Properties

There are no known mineral properties of interest lying adjacent to the Project site, the nearest prospect being the Sayacik silver showing to the southwest of the Beydag volcano (within the Kisladag volcanic complex).

8. Mineral Processing and Metallurgical Testing

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

The Kisladag deposit is characterized by an oxidation zone that extends from surface to approximately 50 m deep. The gold grades of the oxide and primary ore are similar 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 impacted 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 Kisladag 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 Kisladag ore can 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 Kisladag ore. The tests showed that a fine crush size and a leach period of ninety days was 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. 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. Lime consumption is projected to be 4 kg $Ca(OH)_2$/t. Percolation tests indicated that the Kisladag ore does not require cement agglomeration and heap heights of up to 60m are possible.

9. Mineral Resource & Mineral Reserve Estimates

9.1 Mineral Resources

The geologic block model for the Kisladag deposit was developed by Eldorado and its consultants using GEMCOM software. A three-dimensional interpretation of the mineralized envelope and a grid of 20 m x 20 m x 10 m blocks form the basis of the block model with each block assigned a geological domain code based on the block s centroid falling inside or outside the three dimensional mineralized envelope.

Search strategies for ordinary kriging interpolation were assigned to each geologic domain. Within Domain 100, blocks in the North Limb of the deposit (i.e. north coordinate greater than 4261560 N) were estimated using a different semi-variogram model than those within the South Limb. For data selection, however, this boundary was considered soft and composites from one side were allowed to influence the grade of blocks on the other side.

Specific Gravity values were interpolated into the block model from a database of specific gravities collected from drillcore measurements.

Mr. Gary Giroux, Associate with Micon is the independent Qualified Person responsible for preparation of the resource estimate in accordance with National Instrument 43-101. Mr. Giroux visited the Kisladag site and the Company s sample preparation laboratory in 2002.

The following tables summarise the Kisladag resource estimates as of November 2002 (tabulated in Measured, Indicated, and Inferred categories):

Table 1: Kisladag Resource Estimate Summary

MEASURED RESOURCE				INDICATED RESOURCE			
Cut-off Au (g/t_	Tonnes Above Cut-off	Grade Above Cut-off Au (g/t)	Total Contained Ounces of Gold	Cut-off Au (g/t_	Tonnes Above Cut-off	Grade Above Cut-off Au (g/t)	Total Contained Ounces of Gold

0.00	55,500,000	1.129	2,010,000	0.00	199,700,000	0.717	4,600,000
0.40	47,500,000	1.283	1,960,000	0.40	118,900,000	1.070	4,090,000
0.50	45,100,000	1.328	1,930,000	0.50	108,500,000	1.130	3,940,000
1.00	27,900,000	1.684	1,510,000	1.00	52,600,000	1.555	2,630,000
1.50	13,600,000	2.177	950,000	1.50	21,300,000	2.065	1,410,000

	INFERRED RESOURCE				MEASURED + INDICATED		
Cut-off Au (g/t_	**Tonnes Above Cut-off**	**Grade Above Cut-off Au (g/t)**	**Total Contained Ounces of Gold**	**Cut-off Au (g/t_**	**Tonnes Above Cut-off**	**Grade Above Cut-off Au (g/t)**	**Total Contained Ounces of Gold**
0.00	195,500,000	0.394	2,480,000	0.00	255,200,000	0.806	6,610,000
0.40	69,100,000	0.814	1,810,000	0.40	166,400,000	1.131	6,050,000
0.50	54,600,000	0.912	1,600,000	0.50	153,600,000	1.188	5,870,000
1.00	14,500,000	1.494	700,000	1.00	80,600,000	1.600	4,150,000
1.50	5,300,000	2.011	340,000	1.50	34,800,000	2.109	2,360,000

9.2 Mineral Reserves

To establish mineable quantities and grade, Eldorado employed the Whittle 4X pit optimization program using the latest resource model completed by Micon in November 2002. This optimization program uses a technique whereby the gold price is gradually increased, generating a series of nested pits with each successive outline corresponding to a slightly higher gold price. These pits were then analyzed with a selected set of current cost and price assumptions (for example $325 per ounce of gold) to establish their respective values. An optimal pit shell was then selected on the basis of the highest Net Present Value (NPV).

Rockland, Ltd. of North Vancouver, Canada (Rockland) provided detailed pit wall slope angle recommendations. Since inter-ramp slope angles did not accommodate haul road segments, the original overall slope angles were flattened to reflect at least three loops of in-pit haul roads, as follows:

Table 2: Pit Wall Slope Angles

Rockland Recommended Pit Slope Geometry				
Design Sectors	**Pit Wall Location**	**Azimuth Range (degrees)**	**Inter-Ramp Angle (degrees)**	**Final Overall Slope Angles Used in Optimization (degrees)**
1	Northeast	000-040	54	45
2	East	040-105	54	45
3	Southeast	105-170	56	48
4	Southwest	170-225	56	49
5	West	225-282	50	43
6	Northwest	282-360	54	45

Smoothed pit designs incorporating appropriate pit access ramps, wall slope angles, catchment berm designs and minimum mining widths for the selected mining equipment were produced within Whittle Pit Shells 12, 19 and Pit Shell 33.

The mineral reserve estimates for the Project include only Measured and Indicated blocks within the final pit limits. Inferred blocks within these limits cannot be considered reserve according to international standards of reporting, and have been assigned as waste material.

Based on breakeven analysis, mill or internal cut-off grades were set at 0.35 g/t Au for oxide and 0.50 g/t Au for primary material respectively. These cut-off grades are based on representative metallurgical gold recovery parameters and appropriate fixed and variable operating costs.

A detailed pit design incorporating practical mining considerations, recommended slope angles, ramp design, and haul road layouts was completed and a reserve estimate of 115 million tonnes at 1.23 g/t Au was generated for the deposit, including appropriate allowances for dilution and mining losses and a gold price of US$325/oz. A total of 221 million tonnes of ore and waste will be mined and moved over the 14 year life of mine. The overall average waste to ore ratio is therefore projected to be 0.92:1.

Mr. Callum Grant, Manager, Geology and Mining for Hatch is the independent Qualified Person responsible for preparation of the reserve estimate in accordance with National Instrument 43-101. Mr. Grant visited the Project site in 2002. The following table summarises the Kisladag reserve estimate (tabulated in Proven and Probable categories).

Table 3: Mineral Reserve Estimate

Reserve	Tonnage	Average Grade	Metal Content

Ore Type	Category	(tonnes)	(g/t Au)	(kg Au)	(oz Au)
Oxide	Proven	13,332,000	0.778	10,372	333,000
	Probable	13,907,000	1.068	14,853	478,000
	Total	**27,239,000**	**0.925**	**25,225**	**811,000**
Prima	Proven	29,388,000	1.392	40,908	1,315,000
	Probable	58,512,000	1.279	74,837	2,406,000
	Total	**87,900,000**	**1.317**	**115,745**	**3,721,000**
Total	Proven	42,720,000	1.200	51,280	1,648,000
	Probable	72,419,000	1.239	89,690	2,884,000
	Total	**115,139,000**	**1.225**	**140,970**	**4,532,000**

10. Mine Operations and Scheduling

10.1 Mine Design

Pit designs have been completed for two major mining phases, namely the initial pit and a final, ultimate pit. In addition, a provisional pit shell was used as an intermediate pit between the initial pit and the final pit. The three pits are based on Whittle pit shells number 12, 19 and 33.

The lowest bench in the initial pit is at elevation 870 m asl. The pit wall measured from this bench to the highest point of the rim is 240 m. The final pit bottom is at elevation 670 asl, for a total depth of 440 m below the pit rim. A layout and a section drawing of the pit are presented in Figures 7 and 8.

The travelling surface of the mine haul roads inside the pit is 20 m wide. In addition, there is a 3 m high safety berm requirement that will occupy an additional 5 m of width. Allowance has also been made for a 1 m wide drainage ditch. The total required width of the haul road is therefore 26 m. The gradient of the haul road in the pit is generally 10%. The main pit haul ramp is constructed as a downwards anti-clockwise spiral. This allows for right lane driving at the pit wall travelling downhill into the pit while the loaded trucks heading out of the pit will travel on the pit side. Exceptions from this will occur during mining of the upper benches that will require the loaded trucks to travel downhill. The last six segments of the haul road close to the pit floor are designed for single lane traffic only with a road width of 15 m.

10.2 Production Schedule

The mine production schedule includes a nine month pre-production period during which topsoil will be removed, surface drainage and haul roads will be constructed, and pre-production mining of waste and ore will begin. Surface runoff collection and drainage ditches will be constructed around the pit to divert water and minimize pit inflows.

The first phase of pre-production mining will provide some 600,000 tonnes of waste rock for construction of the crusher pad and for other fill requirements. During the second phase, 390,000 tonnes of ore to be used as leach pad overliner will be mined, crushed and screened by a mobile crushing plant temporarily erected at the north end of the pit. After the overliner material has been laid, a further 400,000 tonnes of ore will be crushed and placed on the leach pad to coincide with the commissioning of the process plant.

In Phase I, a mine production rate of 5 million tonnes per annum (tpa) of ore has been set for the first four years of the mine s life (except for Year 1 when the planned production rate will be 4.8 million tonnes). Average daily production rates will be 16,100 tpd in ore and 13,000 tpd in waste during these initial four

years. In Phase II, annual ore production will increase to 7.5 million tonnes in Year 5, and to 10 million tpa the following year, remaining at that level until the end of mine life. The highest daily production rate occurs in Year 7 with a total movement of 79,000 tpd (ore and waste).

10.3 Waste Disposal

Waste rock from the open pit will be dumped to the southwest of the pit at a trucking distance that will vary from about 900m to 4,300m over the life of the operation. The rock dump has a designed capacity of 110 million tonnes with potential for future expansion, and will be developed to avoid any ground water contact with sulphide waste material. On completion of mining, the face of the rock dump will be graded and covered with a layer of soil.

10.4 Mining Equipment (Phase II)

During the pre-production and early ore mining phases of the Project, a mining contractor will be employed for both waste movement and ore mining. In Year 4, Eldorado will phase in its own equipment fleet and mine workforce for completion of the operation.

In Phase II, the nominal production rate of 16,000 to 32,000 tpd of ore and some 13,000 to 31,000 tpd of waste will require a fleet of medium to large size mining equipment. Principal units will consist of 18m^3 loaders, 150 t trucks, and rotary drills capable of 152 203 mm holes. Equipment requirements have been based on a three shift basis over a six-day week, or a total of 310 days per year.

10.5 Drill and Blast Design

The drill pattern has been based on a powder factor of 0.20 kg/t and 10 m high benches. Drilling will be on a 5 m x 5 m pattern in ore with 1.5 m sub-drilling and a 152 mm hole, and 6.5 m by 6.5 m in waste with a 203 mm hole. Drilling productivities in ore and waste are estimated at 50 t/metre and 85 t/metre respectively.

AN/FO blasting agent will be used in the pit except if wet conditions occur during heavy rains, in which case plastic liners will be used down-the-hole to keep the blasting agent dry. On average, blasting will only be carried out three to four times a week, and only on day shift. For the purpose of this study, it is assumed that smooth blasting or pre-splitting will be applied at the final pit wall.

10.6 Loading and Hauling

Since the oxide, primary and waste rock have similar densities and hardness, loading productivities in ore and waste are expected to be similar and have been estimated at 9,940 tonnes per operating shift using 18 m^3 wheel loaders and 136 tonne haul trucks. Hauling productivities will vary with the haul road profile, and ore versus waste destinations. 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. Haul roads have been designed with 10% gradients, except for the road to the waste dump, which will have a 6% uphill grade. Haul distances will vary over the mine life from 450 m to 1,800 m for oxides, 100 m to 3,000 m for primary, and up to 4,300 m for waste. The mine fleet will include three track dozers, a wheel dozer and two graders for construction and maintenance of the bench roads, haul roads and the waste dump road.

10.7 Mine Services

The mine services complex will include a repair and maintenance facility and fuel facilities for mobile mining equipment, a mine dry, storage lockers and washroom/shower facilities.

Explosives for mining operations will be supplied to site on a regular basis. Explosives will be stored in two magazines, a detonator magazine and a powder magazine. The magazines will be fenced and located within the property boundary at least 0.5 km from the nearest mine facilities and working or populated areas according to the relevant Turkish regulations and safe mining practice.

Power supply to the pit for mine dewatering will be distributed from the main site substation over the 6.6 kV overhead power line. A transformer will be installed to step down power to the pumps.

10.8 Manpower

The required manpower has been estimated based on a three shift, six day per week operation (with the exception of drilling which will operate two shifts per day). This will result in good utilization of mine equipment and also provide sufficient time for maintenance of the primary crusher system. In general, one operator has been assigned to each major mine equipment unit on each shift.

The maximum complement of mining department personnel has been estimated at 177 people in Year 7 of the operation (mine operators, maintenance, supervision, and technical support).

Figure 7: Ultimate Pit Layout

Figure 8: Pit Sections

11. Process Plant

The Kisladag ore will be 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 will be treated on site in a refinery and the final product will be a gold doré bar.

The initial design capacity will be 5,000,000 dry tonnes of ore per annum for the first four years of operation when predominantly oxide material will be processed. The facilities will be expanded to process 10 mtpa after year five when primary ore from the deeper, higher sulphide zone in the pit will predominate. Oxide ore will be recovered through to year 9. The crush size will be 80% passing 6.3 mm and the overall availability of the crushing and screening plant is estimated to be 70%.

The primary crusher will be a 1300 mm by 1750 mm gyratory crusher capable of processing the ultimate design rate of 1,653 tonnes per hour. Run of mine ore will be hauled from the open pit and direct dumped into the primary crusher. Initially, contract miners will deliver the ore in 40 tonne trucks and the layout provides for two trucks dumping simultaneously. The dump pocket has a capacity of 300 tonnes and will be adequate for the larger owner operated mine trucks in later years. The crushed ore will be conveyed to a 300 tonne coarse ore bin and then on to the secondary crushers. The bin will also feed a 20,000 tonne stockpile when the crushers are not available.

The final crushed product will be prepared in a circuit consisting of one scalping screen, one MP800 standard secondary cone crusher, one MP800 shorthead tertiary cone crusher and two fine ore screens. The capacity of the circuit will be 827 tonnes per hour when delivering a product containing 80% passing 6.3 mm. This capacity will be doubled in year five of operation by installing a second parallel train between the gyratory crusher and overland conveyor. The coarse ore bin will be equipped with an additional reclaim feeder to facilitate the future installation.

Final product from the crushing and screening circuit will be transferred to an overland conveyor via a radial stacker. This stacker will have the capability to form a 7,000 tonne fine ore stockpile with a swing of 45 degrees. The fine ore will be manually reclaimed from the pile by front-end loaders and re-introduced onto the overland conveyor belt via a hopper. Crushed ore will be transported to the heap leach pad by an overland conveyor and a series of portable conveyors and a radial stacker will place the ore onto the pad. The design includes a total of five 10 m high lifts with a total heap height of 50 m.

The heap leach pad will be 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 will have a capacity of 15 million tonnes and sequential expansions to the pad will accommodate the total tonnage mined.

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

The leach cycle, based on testwork, is 90 days and the solution application rate will be 12 litres per hour per square metre of crushed ore. There will be three 10,000 m^3 process ponds installed to contain the heap leach solutions. The process ponds will have a double HDPE liner and will be fitted with leak detection pumps. The pond surfaces will be covered with floating 100 mm diameter HDPE plastic balls in order to prevent bird access.

The water management system has been designed to accommodate a 100 year, 24 hour storm event. A 77,000 ^{m3} storm water event pond with a single HDPE liner will be provided to contain excess overflow solution from the pregnant solution pond. A second storm water event pond will be installed in year seven of operation. 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) will consist of two trains of carbon columns with each train consisting of five columns. Gold from the heap solutions will be loaded onto the activated carbon and the carbon will be removed periodically for treatment. The gold will be recovered from the carbon in a standard process consisting of pressure stripping, electrowinning and smelting. The final product will be a gold doré bar suitable for final processing to 99.9% purity in an offshore refinery.

Process plant manpower requirements have been based on a three shift, seven day per week operation. There will be a total of 88 process positions initially, increasing to 107 positions by year six.

A simplified flowsheet for the Kisladag Process plant is presented overleaf.

Figure 9: Simplified Flowsheet

12. Infrastructure and Ancillary Facilities

12.1 Site Location

The Project is located on the western edge of the Anatolian Plateau at an elevation of approximately 1000 m. Local elevations range from a peak of 1,300 m asl (Kisla Dag) to a valley of 700 m asl. There are a number of small farming villages within the concession area and livestock breeding provides their main means of support.

The planned crushing plant location (N 4262000 and E 688200) is adjacent to the open pit, approximately 2 km north of the village of Gumuskol. The administration buildings are located on level ground between the pit and the crushing plant. The mining contractor will be allocated space to the east of the primary crusher site. The rock disposal site is located about 1 km northwest of the open pit, within the headwater area of a small valley drained by an intermittent stream.

The leach pad facility will be constructed to the north of the Ovacik settlement. The pad will be located on the western flank of Kisladag Mountain and bounded on the westside by the main basin drainage course flowing from Ovacik northwards. Future leach pad extensions will be to the north.

A 2m high range fence installed along the property boundary will control access to the mine site. There will be one main access gate, which will include a gatehouse manned 24 hours a day. Additional security fencing will be erected around the ADR plant and solution ponds, electrical substations, reagent storage and explosives storage areas.

Figure 10 overleaf presents a summary view of the overall site layout and Figure 11 is an artistic rendition of the ultimate open pit and crushing facilities prior to reclamation.

12.2 Access Road

The existing paved road is unsuitable for mine use as it is narrow and winds through the village of Gumuskol. A new site access road will be 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 existing road connecting the villages of Gumuskol and Ovacik will be abandoned and a new road, approximately 1.9 km long will be constructed to bypass the crushing facilities.

Figure 10: Initial Site Plan

Figure 11: Ultimate Open Pit and Heap Leach Facilities Prior to Reclamation

12.3 Water Supply

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. Fresh water supply for the Project will be supplied from a well field located approximately 13 km to the east of the plant site, in Neogene sediments. A water tank and distribution system at site will provide capacity for process, firewater and potable water requirements.

12.4 Power Supply

The Turkish national power utility company, TEDAS, will supply the electric power for the Project. A new 30 km long, 34.5 kV transmission line will be constructed to provide power from the city of Usak to the site. A main transformer at site rated at 10 MVA, 34.5 kV   6.6 kV will provide 6.6 kV power to be distributed locally via overhead power line.

12.5 Buildings

The permanent mine buildings will be designed and constructed by local Turkish contractors and the schedule will allow sufficient time for construction. Where feasible, the architecture of the facilities will include local building materials and methods to be compatible with the surrounding infrastructure.

The workshop/warehouse (760 m^2) will house an electrical workshop, an instrument workshop, tool storage, a security store, offices and storage space for maintenance items. A floor pit and an overhead traveling crane have also been included. An adjacent outdoor fenced area will be used to store large equipment and miscellaneous reagents. The mining contractors will establish their own temporary facilities to service the contract mining fleet.

The truckshop complex (782 m^2) installed in year five of operation has been designed to service the fleet of larger owner mining trucks. The complex will include three indoor heavy truck 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 will house a mechanical room and office space.

The administration building (400 m^2) will be a single storey concrete building and will include general areas for engineering, geology and administration personnel plus seven individual offices for management personnel.

The mine dry and canteen (540 m^2) will be a single storey concrete building. The canteen has been equipped with a kitchen area and a seating area for 72 people and there is provision for washrooms, shower facilities and clean and dirty lockers. Additional space has been provided for first aid, safety, an assembly area plus three offices.

Figure 10: Initial Site Plan

The assay laboratory building (270 m^2) will house the assay laboratory, assayers offices, metallurgists offices, separate washrooms for male and female personnel, a core logging room and a storage room for laboratory supplies. The assay laboratory has been sized to process approximately 200 samples per day and will include sample preparation, acid digestion, atomic absorption (AA) finish, fire assay and a wet laboratory.

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

13. Environmental

The draft Kisladag Project EIA study was completed in January 2003 and has been submitted to the Turkish authorities. An Environmental Positive Certificate which is issued upon acceptance of the Project by the Ministry of Environment is anticipated by the third quarter of 2003.

ENCON Environmental Consultancy Company in association with Knight Piesold Limited UK and the Planning Alliance were responsible for the environmental baseline studies and the environmental impact assessment. 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.

The Kisladag Project area is located in a transition zone between continental and Mediterranean weather regimes. The Project site is relatively dry, with no significant lakes, perennial rivers or major aquifers in close vicinity. The geology of the area is dominated by a volcanic sequence with generally poor aquifer characteristics. Surface waters in the area are comprised of intermittently flowing ephemeral streams, shallow ponds dug for livestock and temporary surface water accumulation following heavy precipitation events. The deposit is located on the watershed divide separating the Gediz River and the Buyuk Menderes River basins, and the ephemeral streams exiting the area eventually end up in either the Gediz River or the Buyuk Menderes River, which flow into the Aegean Sea. None of the ephemeral streams or artificial shallow ponds in the area support any notable aquatic life.

The majority of soil cover in the area falls within Turkish land use category VI and VII, which are restricted in terms of cultivation potential (stony, rocky and erosion-prone) and are generally deemed unsuitable for economic agriculture. Approximately 5% of the area is covered with Class III Colluvial soils suitable for agriculture use but require special control measures to take account of topography and erosion potential.

The majority of the area (Gokgoz, Koru, Aktepe and Arap Tepe) is underlain by volcanic rock and the dominant flora species in this area are scrub oak and pine trees and grasses. The widespread flora species encountered in the Kisla Mountain and Kabaagac Mountain areas, which is calcareous in nature, is oak. Most of the land is inhabited or in active use for such activities as cultivation (non-irrigated), grazing, and fuel gathering. Wildlife interest is therefore not high and the Project area does not include habitat of designated conservation value.

There are no known archaeological artefacts, inscriptions or other remains within and the near vicinity of the proposed operation. Further more, there are no protected areas within the Project area.

The region is characterized by low employment opportunity, and the proposed mining project will significantly improve this situation. However, the Project is also likely to increase pressure on community resources, such as health, and education services. Other important factors for communities include potential impacts on current and customary uses of the land (e.g. for agriculture, grazing, and fuel gathering), increase in traffic volumes, and also potential for nuisance, noise, vibration, dust and visual impact.

The most significant emission that may affect air quality at Kisladag will be the generation of dust and mitigation methods will be required. As confirmed by experience from operating mines worldwide, the dust problem can be contained by adopting dust avoidance and control measures, including installation of dust collecting devices, effective road maintenance and application of water mists and sprays during dry weather.

Acid rock drainage (ARD), also referred to as acid mine drainage, is a potential impact generally associated with mining projects. Extensive testwork has been undertaken to assess the potential for the generation of ARD at the Kisladag Project, with the main focus on ARD potential of the waste rock. Acid base accounting and humidity cell testwork has shown that neutralizing potential is low and that reactivity of sulphide minerals in the waste is very low. The conclusion is that there is little probability of ARD being a major impact particularly given the design characteristics of the rock dump.

The rock dump is designed to minimize exposure of such sulphide bearing rock to air and water. During operation, runoff water from precipitation will be diverted by interception and diversion ditches and segregated to reduce the volume of water coming in contact with sulphide rock. Internal run-off or seepage from the rock dump will be intercepted in a lined pond at the base of the rock dump, and treated for particulate matter and pH as required prior to release or subsequently used for operations purposes. In addition, site specific conditions exist that will inhibit the onset of ARD, such as; a favourable ratio of precipitation/evaporation, the apparent lack of aquifers of importance in the underlying strata and low permeability of these formations, the presence of considerable amounts of oxidized waste rock with no ARD potential which can be used to isolate ground water sources, and the proposed strategy of progressively capping and vegetating the dump for final reclamation.

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Figure 10: Initial Site Plan

Permanent surface water bodies or productive aquifers do not exist in the Project area. Therefore, no significant disruption of surface flow is expected due to proposed development. Mine water, which will be generated in limited quantities, will be recycled to be used for dust control and process water. Process water in the heap leach facility will be contained in a closed, zero discharge circuit. Solutions from the heap will be contained in lined ponds sized to accommodate 100 year 24 hour storm events. Project design incorporates measures for the prevention and prompt mitigation of uncontrolled releases of liquid effluent to the environment resulting from accidents, equipment failures or natural catastrophes. The Environmental Management Plan for the Project will be designed to reduce potential for such releases through the adoption of best working practices.

The heap leach facility will use cyanide solutions to recover gold from crushed ore. This widely used mining chemical is hazardous, and during operations, a comprehensive Cyanide Management Plan will be in place to ensure proper procedures are followed. The heap leach facility will be equipped with a lining system consisting of a 2 mm thick synthetic liner and a 30 centimetre layer of compacted soil to prevent cyanide from entering the environment. On closure the heap will be washed with fresh water to bring the cyanide levels in the heap to below regulated levels. For final reclamation, the leach pad will be contoured, capped with a soil layer to trap meteoric water and re-vegetated to promote trans-evaporation.

Under normal operating conditions all process liquids will be recycled within the process, there will be no discharge of liquids to the environment other than the discharge of clean water that meets the discharge limits under the relevant water discharge regulations. The design of the Project incorporates measures for the prevention and prompt mitigation of uncontrolled releases of liquid effluent to the environment resulting from accidents, equipment failures or natural catastrophes.

A Preliminary Closure Plan, based on the closure strategy presented in the EIA report, will be prepared as part of the detailed engineering of the Project following on from the feasibility study. This plan will be updated and revised through development and operation of the mine, culminating in the establishment of a Final Closure Plan prior to decommissioning. This document will detail the closure works in line with land uses, goals and after-care provisions agreed with the authorities and following appropriate consultation with the local community and other stakeholders.

14. Project Implementation

14.1 Permitting

The process of obtaining the necessary permits for a mining operation in Turkey is similar to that in other developed countries. The first permit required to initiate an industrial project in Turkey is the Site Selection Permit. This permit was issued to Eldorado by the Provincial Governors Office in 1999 and confirms that there are no development conflicts in the proposed Project area.

The key Environmental Positive Certificate is issued by the Ministry of Environment following a successful review of the Environmental Impact Assessment (EIA) Report. The permit contains agreed protocols between the proponent and Ministry for mitigations methods, monitoring standards, closure procedures and financial guarantees. The EIA has been submitted and receipt of the Environmental Positive Certificate is anticipated by the third quarter of 2003.

In order to obtain the Construction Permit from the Municipality, Eldorado first has to apply for an Establishment Permit from the Ministry of Public Health. This permit covers a number of activities and licenses including a waste deposition license, an electric use permit, a water use permit, an air emission permit, project zoning approval, tailing facility design approval and Forestry, Treasury or Private land acquisition.

Once the facilities have been constructed, a Trial Permit from the Ministry of Health will allow Eldorado one year to demonstrate that the plant operates at standards defined in the Establishment Permit. The Ministry of Health will issue the final Project Operating Permit and the Air Emission Permit.

14.2 Construction

Construction of the Kisladag facilities will be completed in two major phases. The first phase will comprise the bulk of the infrastructure, equipment and earthworks required to process predominantly oxide ore during the first four years of operation. The second phase, in year five of operations will entail an expansion to the crushing circuit and purchase of larger mining equipment required to increase production throughput to final design capacity of 10 million tonnes per annum. There will also be minor subsequent construction phases associated with expansion of the heap leach pad and closure of the Project.

Construction of the initial facilities will be the most critical phase. Long delivery equipment will need to be purchased early and shipped to site. Engineering activities must be completed ahead of construction and competent contractors will be selected to install and build the facilities. The table below presents a summary of the construction logistics required for the initial phase.

Table 4: Summary Construction Logistics

Description	**Quantity**
Steel (tonnes)	1,400
Concrete (m3)	4,700
Pipe (m)	120,000
Conveyors (tonnes)	900
Process Equipment (tonnes)	1,100
Construction Manhours (hrs)	654,000

Peak Construction Workforce (men)	475

The duration of the engineering, design and construction activities will be approximately 18 months. The receipt of an Environmental Positive Certificate by the third quarter of 2003 should enable construction to commence in late 2003. The critical path activity will be procurement of long delivery process equipment. Pre-production mining activities are scheduled to commence in the fourth quarter 2003 and the crushing plant will be commissioned in the third quarter 2004. Crushed pre-production ore will be delivered to the heap leach pad for three months until full production commences. The first gold will be poured at the end of 2004.

A summary construction schedule is presented overleaf.

Figure 12: Summary Schedule

15. Capital Cost

The capital cost estimate has been prepared with an intended level of accuracy of plus or minus 15% and is intended to support the Feasibility Study financial analyses.

The estimated capital costs for constructing and sustaining the operation of the Kisladag Project are presented in Table 5 below. The costs are expressed as third quarter 2002 US dollars without escalation. Total capital costs over the life of the operation are estimated to be US$130,990,000.

Table 5: Capital Cost Summary

Description	Initial Capital Cost (k$US)	Expansion Capital Cost (k$US)	Sustaining Capital Cost (k$US)
Infrastructure	9,352	812	
Crushing	10,875	8,134	
Leach Pad Conveying	3,572	1,147	
Heap Leach Pad and Ponds	4,746	429	15,031
ADR Facilities	3,260	26	
Waste Dumps	327		
Mining	381	25,829	14,778
Closure Capital			7,400
Sub-Total Direct Costs	**32,514**	**36,377**	**37,209**
EPCM	3,908	949	
Construction Indirects	2,749	663	
Freight	946	533	
Spares	1,825		
First Fill	118		
Sub-Total Indirect Costs	**9,546**	**2,145**	
Owners Project Management	4,773		
Pre-Production Mining	2,764		
Sub-Total Owners Costs	**7,537**		
Contingency	4,777	885	
Total Project	**54,374**	**39,407**	**37,209**

16. Operating Cost

Cash operating costs in the first four years of production are estimated to be US$3.97 per tonne of ore, equivalent to US$138 per ounce of gold produced. Life of mine cash operating costs are estimated to be US$3.82 per tonne of ore and US$152 per ounce. The estimated operating costs for the Project expressed in 2002 US dollars are summarised below.

Table 6: Life of Mine Operating Cost Summary

	Life-of mine US$ millions	US$ per tonne	US$ per ounce
Mining	196	1.70	68
Process	173	1.51	60
General and Administrative	54	0.47	19
Heap Rinse and Detox	9	0.08	3
Transport & Refining	7	0.06	2
Cash Operating Cost	**440**	**3.82**	**152**
Royalties	13	0.11	5
Total Cash Cost	**453**	**3.93**	**157**
Depreciation	114	0.99	39
Amortisation	13	0.12	4
Closure Cost	7	0.06	3
Total Production Cost	**583**	**5.10**	**203**

Table 7: Cash Operating Cost Summary

Cost Area	First Four Years Of Operations		Life of Mine Operation	
	US$/t	US$/oz	US$/t	US$/oz
Mining	1.61	56	1.70	68
Processing	1.63	57	1.51	60
General and Administrative	0.67	23	0.47	19
Heap Rinse and Detox	0.00	0	0.08	3
Transport & Refining	0.07	2	0.06	2
Total	**3.97**	**138**	**3.82**	**152**

The distribution of annual operating costs in the first and sixth year of operations at Kisladag is presented below.

Figure 13: Year One Operating Cost Distribution

Figure 14: Year Six Operating Cost Distribution

Operating expense is defined as any recurring expenditure that can be expensed in the tax year in which it occurs. The operating cost estimate includes all recurring costs for payroll, contractors, maintenance parts and consumables, reagents, freight, etc to operate all facilities as described in this study.

Operating expenses commence at the end of the three month pre-production period. Ongoing capital expenditures required for heap leach pad expansions and purchase of mining equipment are included in the capital cost estimate. The final Project closure cost is also included in the capital cost estimate. The costs for rinsing and detoxification of the heap leach facilities which will commence in year ten of operations, have been included as an operating cost.

In general, the detailed operating costs have been developed by applying vendor budget pricing to estimates of quantities. The mine operating costs in the first five years of operation were obtained from Turkish mining contractors.

The Kisladag Project will employ 170 people in the first year of operation and a mining contractor will provide additional mining labour. By year six of operation, the mining contract will be complete and the total number of employees will have risen to 356. In general, the operations and maintenance personnel will reside in nearby communities and will work three eight-hour shifts or two twelve hour shifts to provide 24 hour coverage.

Table 8: Summary Staffing Requirement

	Number of Employees	
Operations Area	**First Year of Operation**	**Year Six of Operation**
Mining	14	165
Processing	88	107
General and Administrative	68	84
Total	**170**	**356**

17. Financial Analysis

Hatch completed a financial analysis of the Kisladag Project using a discounted cash flow model incorporating the most recent Turkish tax and royalty schedules. Project construction capital cost estimates including pre-production costs, ongoing capital costs and mine closure costs have been included in the Project cashflow projection. Operating costs presented as fourth quarter 2002 US dollars remain constant over the mine life and no allowance for inflation has been included. The economic analysis excludes considerations of alternative financing options and is based on zero debt in order to present a base case cash flow analysis. In order to meet regulatory requirements, the mine production schedule includes only Measured and Indicated Resources. A summary of the financial analysis is presented below.

Table 9: Kisladag Project Financial Analysis Summary

Project Data	Estimated Value
Life of Mine	15 years
Total Gold Produced	2.9 million oz
Total Ore Mined	115 million tonnes
Total Material Mined	221 million tonnes
Open Pit Strip Ratio	0.92
Initial Project Capital Cost	US$54.4 million
Cash Operating Cost	US$152 /oz
Total Production Cost	US$203 /oz
Base Case Gold Price	US$325 /oz
Before Tax Net Present Value @0%	US$356 million
After Tax Net Present Value @0%	US$255 million
After Tax Net Present Value @5%	US$146 million

After Tax Net Present Value @10%	US$85 million
After Tax Internal Rate of Return	33%
Payback Period (from start-up)	2.6 years

A summary of the annual production schedule and cash flows is presented overleaf.

Table 10: Kisladag Production Schedule Summary

18. Sensitivity Analysis

The sensitivity of the Kisladag Project to the variables of gold price, gold recovery, capital cost and operating cost is presented in Table 11 below and Figure 13 overleaf. The cashflow model assumes a 100% equity basis with no allowance for inflation.

Table 11: Sensitivity Analysis Summary

Variable	Variable Value	IRR %	NPV@5% US$ millions
Gold Price	US$/oz		
+15%	375	44	211
+8%	350	38	178
Base Case Gold Price	325	33	146
-8%	300	27	114
-15%	275	21	82
Gold Recovery (Oxide Ore)			
+10%	89%	37	159
+5%	85%	35	152
Base Case Oxide Recovery	81%	33	146
-5%	77%	31	140
-10%	73%	29	134
Gold Recovery (Primary Ore)			
+10%	66%	36	176
+5%	63%	34	161
Base Case Primary Recovery	60%	33	146
-5%	57%	31	131
-10%	54%	29	116
Capital Cost	US$ millions		
+20%	65.2	27	139
+10%	59.8	30	142
Base Case Capital Cost	54.4	33	146
-10%	48.9	36	150
-20%	43.5	41	154
Cash Operating Cost*	US$/oz		
+20%	182	26	106
+10%	167	29	126
Base Case Operating Cost	152	33	146
-10%	137	36	166
-20%	121	39	187

Figure 15: Sensitivity Analysis

The Project was found to be most sensitive to changes in gold price and metallurgical gold recovery, followed by operating cost and capital cost. The Project was least sensitive to changes in capital cost.

Should both primary and oxide recoveries decrease to 57% and 77% respectively (down by 5%), the Project NPV@5% will be US$125 million and the IRR will be 29%.

The break even price of gold, defined as the price at which the net Project cash flow is zero, is US$202/oz.

19. Project Risks and Opportunities

19.1 Country Risk

Political

The risk to disruption of business activities in Turkey due to political instability is considered to be moderate. Turkey has oriented its political and economic structure towards the West since the Turkish Republic was formed in 1923. Turkey originally established an industrial base through state intervention and import protection in post World War II period. Policies have since shifted towards liberalisation, reinforced by Turkey s customs union with Europe and pending candidacy for membership in the European Union. The election in 2002 of a majority government has extended a period of political stability which has existed in Turkey for the past two decades. Although the mining industry in Turkey is still dominated by public sector companies there are successful foreign-owned mining operations such as Cayeli (Inmet) and Ovacik (Newmont). The central government continues to promote the privatization of state owned ventures, including its mineral operations, to boost the economy and attract foreign investment. Government support for the mining sector is being promoted through pending state sponsored revisions to the Mining Law, aimed at improving fiscal and regulatory demands on the industry.

Economic

Turkey continues to face the challenges of a sluggish economy affected by external debt, high interest rates and high inflation. Regulatory reforms are needed to implement controls necessary to maintain steady growth and improve confidence in the economy. Continued foreign investment is a key element in stimulating this growth. Projects such as Kisladag have gained support of all levels of government because of the measurable long term impact it will have on the country at large.

19.2 Environmental and Regulatory Risk

The recently submitted Environmental Impact Assessment report on the Kisladag Project has provided an in-depth analysis of the environmental and social impacts which the Project will generate and identified measures to be taken to mitigate these impacts. All aspects of the Project design have considered international best practices followed by the mining industry world wide to protect the environment in the short and long terms and maintain the health and safety of its workers and the community in which it operates. Eldorado believes this approach will lead to acceptance of the EIA report and receipt of an

Environmental Positive Certificate for the Project by mid 2003. There are however a number of ancillary permits and approvals required as the Project moves through development to construction and operation. The remaining permitting steps can be subject to processing delays which have the potential to prolong project implementation.

Completion of the Establishment Permit requires securing rights to surface lands in the immediate Project area. Eldorado is executing a plan to obtain the necessary access rights to private and government controlled lands required for constructing of the Project. Delays to this process will impact the implementation schedule.

Environmental activists and anti-mining lobbyists will still have an opportunity to delay and hinder the process. The use of cyanide is likely to be a focus point, although the Turkish regulators have resolved this issue at Newmont s Ovacik operation and are now knowledgeable about this technology and supportive of the mitigation measures employed for its safe application.

19.3 Financing

Project capital costs for Phase I have been estimated at US $ 54.4 million. Financing for construction of the Project will be provided through a combination of debt and equity. Given the robust nature of the Kisladag Project, the ability of Eldorado to raise the required debt financing is very good. This risk to the Project is considered to be acceptable.

19.4 Construction Costs

Escalation of construction costs for the Project between completion of this Study and project implementation are beyond the scope of the current estimate. Market conditions and available capacity can adversely affect pricing for on site construction work. Provided implementation of the Project is executed within a reasonable time, the cost base used for estimation of construction capital should be maintained with acceptable limits.

19.5 Currency

Due economic pressure the Turkish Lire has continued to weaken against the US dollar over the past several years. Operating costs for Phase II of the Project are made up of 56 % Turkish Lire base and 44 % US Dollar base. Lire based operating costs used in the financial model for the Project however, have not been adjusted over the life of mine, under the assumption that devaluation of the currency in the future will be offset by inflationary pressure, resulting in purchasing price parity remaining constant.

19.6 Exploration Opportunity

Since publication of the resource estimate, Eldorado has completed additional drilling to more accurately define the extent of mineralization around the perimeter of the deposit. A total of 2,650m of shallow holes have been drilled in areas outside the current open pit limits. Preliminary indications are that an additional 180,000 ounces of gold may be available in oxide and primary ore not presently considered in the mine plan.

The mine plan currently extracts approximately 7 million tonnes of inferred resources containing approximately 250,000 ounces of gold. In accordance with NI 43-101 guidelines, this material is classified as sub economic waste and has not been incorporated into the ore schedule. Eldorado has plans to complete a 2,200m drill program to update the classification of this material from inferred to indicated. There is potential for the Kisladag reserve to increase pending receipt of the new data.

19.7 Fuel Price Opportunity

The Project uses a substantial quantity of diesel fuel to operate the mobile mining equipment. In year six of operation, the fuel budget is US$6.6 million, or 18% of total operating cost. The study has assumed a fuel price of US$0.75 /litre. This price level is equivalent to an oil price level of US$30 /barrel. Most oil price projections for the next ten years, forecast a return toward a US$20 /barrel price level once the problems in Iraq are resolved. This study has used a conservative fuel price, a lower fuel price Project will significantly enhance the Project economics.

19.8 Process Gold Recovery Opportunity

Testwork has shown that heap leaching the oxide ore crushed to 80% passing 6.3mm will recover 80% of the contained gold. Initial column testwork indicated that the gold recovery from the oxide ore was not sensitive to particle size and a coarser crush may be viable. Later testwork conducted on lower grade regions of the ore body did not confirm this relationship and a coarse oxide crush size was not included. The Phase I crushing plant rated at 5 mtpa, has the capacity to process 6.5 mtpa at a crush size is 80% passing 12mm. Further testwork will confirm the affect of crush size on gold recovery and determine if higher throughput is viable. Providing gold recovery is not adversely affected, the resulting lower operating costs and increased revenue would improve economic performance of the Project.

19.9 Process Availability Opportunity

The crusher throughput has been calculated assuming that the plant is available for only 70% of the time. The crushing plant will contain a run of mine stockpile, coarse ore stockpile and a fine ore stockpile and it is likely that plant availability will exceed 70%. Provided that the mine can supply the ore to the crusher, it is probable that the tonnage processed will exceed the budget of 5 mtpa in Phase I and 10 mtpa in Phase II.

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