SECTION B: GUINEA

1 LEFA GOLD MINE

1.1 Introduction

1.1.1 Location, Access and Infrastructure

The Lefa Gold Mine is located in the north of the Kankan region, approximately 700km northeast of Conakry (the capital of the Republic of Guinea) at Léro near the Malian border and some 10km south of the village of Siguirini (see Figure 1.1 below).

Figure 1.1: Location of the Lefa Gold Mine

The mine is serviced by its own dedicated air strip that transports the expatriate staff as well as regional local staff who travel from Siguiri (Guinea) 100km to the south-south-east and Bamako (Mali). A further service is run to and from Conakry.

The road network has quadrupled in size since 1996 and several projects are under way to further expand it. Currently about 20% of the country's 30,500km of main routes are paved. Most routes link urban areas to mining areas, and access to the remainder of Guinea is difficult and often impassable during the rainy season. Access to the mine from Conakry is via 450km of paved road, 200km gravel road and a final 150km dirt road, most supplies are conveyed along this route.

The only functioning railway links the ports to several mines (not Lefa) and carries no passengers. The Kamsar to Kankan railway line no longer operates though renovation of the railway system is under consideration.

Conakry port is operating at near saturation levels. Plans are taking place to build an inland container terminal and reactivate Benty port. The country has one international airport, with Air Guinea operating an erratic regional schedule and internal flights to a dozen airstrips around the country.

Guinea has no proven fossil fuel reserves but enormous hydro-electric potential. Nevertheless, firewood accounts for over 80% of domestic energy needs, and petroleum products are imported. Of the 320MW of installed energy production capacity, 40% is privately owned. As little as 10% of the population receive grid electricity, and this group is mainly in the capital though numerous projects are underway to increase electricity production.

The Lefa mine has its own generator plant housing 8 diesel generators, currently being converted to HFO (Heavy Fuel Oil), each developing circa 4MW of electricity (total capacity 32MW).

Water for the plant is supplied by recirculation pumps from the tailings dam recovery pond supplemented by several local boreholes.

1.1.2 Topography and Climate

The Guinean terrain consists of a generally flat coastal plain and a hilly to mountainous interior. Dinguiraye is located in the Upper Guinea region comprising a gently undulating plain of savannah country (grassland, scattered trees and scrubs) with an average elevation of about 300masl.

The climate is hot and humid with two main seasons; a monsoonal type rainy season (June to November) with south-westerly winds and a dry season (December to May) with north-easterly harmattan winds.

1.1.3 History

In January 1984, the Guinean Minister of Mines issued a Ministerial Arrete (No. 1166/SGG/MMG) to create an association for exploring and exploiting gold, diamonds and other minerals. The association was between the Government of Guinea (50%), with FAMA Precious Metals Limited and Norwegian shipping companies, Klaverness Chartering AS and Preco AS (50%). This association was subsequently confirmed by a Protocol d'Accord, providing the partners with a mineral concession of up to 15,000km^2, and the original Dinguiraye concession of 13,791km^2 was subsequently granted.

Following the registration of a joint venture company as Dinguiraye Gold Mines (DGM), the interests of Klaverness Chartering and Preco in DGM were transferred to the Norwegian company Kenor ASA. Limited exploration commenced through until 1985, when the concession area was reduced to 7,300km^2 in the first required retrocession.

In 1986, Societe d'Etudes de Recherches et d'Emploitations Minieres (SEREM), an affiliate of Le Bureau de Recherches Geologiques et Minieres (BRGM), entered into the joint venture by acquiring a 33% interest in DGM, and BRGM was appointed as operator.

In April 1989, a Convention de Base (Basic Agreement) was issued, establishing the operating company, Societe Minieres de Dinguiraye (SMD), as a company owned 50% by DGM and 50% by the Government of Guinea.

In 1989, a further retrocession reduced the concession to 3,554km^2. Notwithstanding this, however, the eastern boundary was subsequently adjusted to include the Fayalala area.

SMD entered into a technical assistance contract with DGM, resulting in an equity adjustment in the latter company, with Kenor and LaSource (BRGM) each retaining a 50% interest. Kenor listed on the Oslo stock exchange in 1994, prior to purchasing a 100% interest in DGM from LaSource in June 1998, and the final equities in SMD (DGM 85% and the Government of Guinea 15%) were therefore confirmed.

BRGM completed an extensive program of regional soil sampling over virtually the whole of the original Dinguiraye Concession. A number of anomalous areas were detected, with that associated with the Lero deposit representing one of the more significant targets identified, generating a peak gold soil response of 3g/t Au.

Extensive field investigation of artisanal workings throughout the Dinguiraye Concession was also completed by BRGM. However, the vast majority of workings were found to be alluvial in origin. Regional geological mapping at 1:200,000 scale, completed under a German Government grant in the 1990's, was used as the basis for geological interpretations, together with remote sensing data, such as Landsat and Radarsat. On the basis of this work, seven regional prospects were Reverse Circulation drilled during the late 1990's. Assessment of two of these prospects resulted in small resources being defined. However, none demonstrated sufficient size potential to justify further exploration until the discovery of the Lero anomaly and peripheral targets.

Following definition of extensive oxide mineralisation at the Lero deposit, feasibility studies were progressed through 1993 and 1994, resulting in a positive outcome. The Lero operation was commissioned in April 1995, at a processing throughput rate of 360,000tpa to exploit a 3.5 year open pit reserve at an average gold grade of 3.7g/t.

In 1998, having acquired the balance of the project, Kenor met with immediate exploration success at the Karta deposit, located immediately adjacent to Lero, and the more distant Fayalala anomaly, with the former having no geochemical expression, and the latter representing one of the subordinate anomalies previously identified by BRGM.

The geochemistry, within what is now referred to as the Lero-Fayalala (Lefa) Corridor, lead to the identification of the majority of other known deposits and prospects. The Lero pit was exhausted of readily accessible oxide ore by the end of 1998, and the Karta open pit was commenced in January 1999.

During 1999 the Fayalala deposit, lying 9km east of the Lero facilities, was brought into production with the mining of principally pisolitic laterite ore, that could be dumped directly onto leach pads without the requirement for crushing and agglomeration. In mid 1999, SMD introduced Moolman Mining, a mining contractor from South Africa, with a larger fleet, including 50t trucks.

During 2000, oxide ore production from the Karta and Fayalala deposits continued. In mid-2000, a trial pit was developed on a near surface, high-grade section of the Tambico resource, and mining continued until October the same year.

In early 2001, Kenor completed, with assistance from independent consultants, a detailed review of the Lero Project, to determine the ultimate exploration and development potential of the concessions. The findings of these studies concluded that the project has the potential to significantly expand and support the development of a large tonnage CIP processing operation, with production from both oxide and primary ore types.

During 2001, the contract mining fleet was expanded to 10 trucks, to enable increased laterite production from the Fayalala deposit, following depletion of the oxide resources at Karta in mid-2001. During the last quarter of 2001, the relatively small saprolite reserve at Kankarta was brought into production.

In 2002, the majority of all production was derived from the Fayalala pit, supplemented by remnant production from Kankarta in the early part of the year. The Fayalala production predominantly comprised pisolitic laterite ore for dump leach processing. However, the first high-fines laterite and saprolite material types were exposed and mined later in the year. The Banko mineral resource was brought into production in mid-2002, with a shallow pit developed on the pisolitic laterite ore. Subsequent optimisation of the remaining laterite and saprolite ore types resulted in a more substantial reserve.

In September 2002, Kenor commenced an exploration strategy, termed the Large Exploration Program (LEP), complimented by a US$9M budget, designed to define sufficient mineral resources to support a large tonnage CIP operation. Mineral resource estimates were progressively updated by RSG Global during the course of the LEP strategy, and subsequent feasibility program.

In 2003, mine production was predominantly comprised of saprolite ore derived, from the Fayalala and Banko deposits, for agglomeration and heap leach processing.

In July 2003, an economic study was undertaken to re-evaluate the viability of establishing an 8Mpta CIP processing operation. The study was based on modelled mineral resources to May 2003, and likely mineralisation projected to through until completion of the LEP strategy in December 2003. The study indicated that the project could generate a healthy return if the assumptions made were realised.

In September 2003, predominantly on the basis of the preliminary economic study, the Kenor board curtailed the LEP strategy in favour of commissioning a Detailed Feasibility Study (DFS) to assess the viability of establishing a large open pit mining and CIP processing operation.

In early 2004, the mining of economic oxide ore associated with the Banko deposit was completed, compensated by the recommencement of mining at the Tambico deposit. Mine production for 2004 was predominately sourced from the Fayalala and Lero-Karta deposits.

On 27 February 2004, Guinor Gold Corporation (Guinor) made an Exchange Offer to acquire all of the issued and outstanding Kenor shares. The offer period expired on 26 March 2004.

On 29 March 2004, Guinor announced all conditions had been fulfilled or waived. By the beginning of April 2004, Guinor became the owner of 133.5M Kenor shares, representing approximately 93.7% of the total 142,544,729 issued and outstanding Kenor shares at that time.

In May 2004, the remainder of the outstanding Kenor shares were compulsorily acquired by Guinor.

In March 2005 RSG submitted a technical report on the Lefa Gold Mine on behalf of Guinor, titled Independent Technical Report filed on SEDAR.

In October 2005, Crew Gold Corporation announced it had offered to purchase 100% of the Guinor Gold Corporation shares in a fully financed all cash transaction.

In December 2005, Crew Gold Corporation announced the conditions offered to acquire all of the issued and outstanding common shares of Guinor Gold Corporation have been complied with or waived.

In June 2006, Crew Gold Corporation purchased the outstanding 15% of the DGM shares owned by the Guinean Government.

Mining in 2005 and 2006 continued to expand, with substantial cutbacks around several of the existing open pits, after approval was given for the development of the Lefa Gold Mine expansion. The majority of material excavated was in the form of waste rock as prestripping.

However, a small amount of ore from the Lero pit was sent to the heap leach pads, prior to de-commissioning in 2006.

Stockpiles for the new CIP plant began accumulating adjacent to the Fayalala pit, where the pre-owned Kelian Gold Mine plant, from Indonesia, was being reassembled. This plant began commissioning in January 2007. The state of the plant and the need for rebuilds for much of the installed equipment, however, meant that despite being partially operational in February 2008, it would only reach full capacity by November 2009.

During 2007, the mining fleet became fully commissioned by earthmoving contractors PMC (Guinea), and full production of the earthmoving equipment of 8m3 backhoes and 40m^3 dump truck combination was realised. Mining was conducted at Lero-Karta with the Lero Cutback, Lero South, Karta, Karta 4 and Camp de Base mineral deposits, all mined in the oxide zone. In addition to this, Kankarta West began to expose the first transitional and fresh ore at the Lefa Gold Mine, and Kankarta East saprolite portion was mined as well. The Bofeko mineral deposit, part of the Fayalala mineral resource, was also mined, with laterite and saprolite ore exposed.

In September 2008, the mining fleet was taken over by SMD due to multiple non compliances in the earthmoving contract by PMC. At the time of the preparation of this report, the mining fleet was owned, operated, and serviced by SMD, and was undergoing full life rebuilds.

In November 2008, the new SGS Lero on site laboratory was commissioned, adjacent to the CIP plant and offices. Since then, all samples have been prepared and assayed at the new laboratory.

At the time of the preparation of this report, mine ore production was approximately 10,000 – 15,000 tonnes per day, at varying gold grades, depending on operational requirements. Historic production at Lefa Gold Mine is presented in Table 1.1 below.

Table 1.1: Lefa Key Performance Indicators for 2008-9m 2010

	Unit	2008	2009	9m 2010
Ore mined	Kt	3,885	4,774	4,491
Ore milled	Kt	3,085	4,422	3,807
Average Au grade	g/t	2.1	1.4	1.3
Gold recovered	kg	6,145	5,523	3,846
Gold recovered	koz	198	178	138.1
Recovery rate	%	93	90	89
Normalised TCC	$/oz	827	809	802

1.1.4 Licences

Along with the original Dinguiraye concession, the Lefa Gold Project comprises a further six contiguous granted Prospecting Permits, covering an aggregate area of 2,552km^2, as presented in Table 1.2 and illustrated in Figure 1.2. The concession boundaries have not been legally surveyed, but are described by latitude and longitude via decree.

Table 1.2: Summary of Licences				
Concession	**Licence**	**Area (km^2)**	**Date Granted**	**Term**
Dinguiraye	Basic Agreement	1,599	21-Mar-94	Initial: 25 years Renewal: 5 years
Kobedara (East)	Prospecting Permit 2003/032	169	15-Sep-03	Initial: 2 years Renewal: 2 years'
Kobedara (West)	Prospecting Permit 2003/030	186	15-Sep-03	Initial: 2 years Renewal: 2 years'
Iroda	Prospecting Permit NoA2002/42	129	08-Nov-02	Initial: 2 years Renewal: 2 years'
Boubere	Prospecting Permit 2003/031	14	15-Sep-03	Initial: 2 years Renewal: 2 years'
Dantinia	Prospecting Permit	230	02-Jun-07	Initial: 2 years
Norum	Prospecting Permit NoA2003/7	224	17-Mar-03	Initial: 2 years Renewal: 2 years'
Total		2,512		

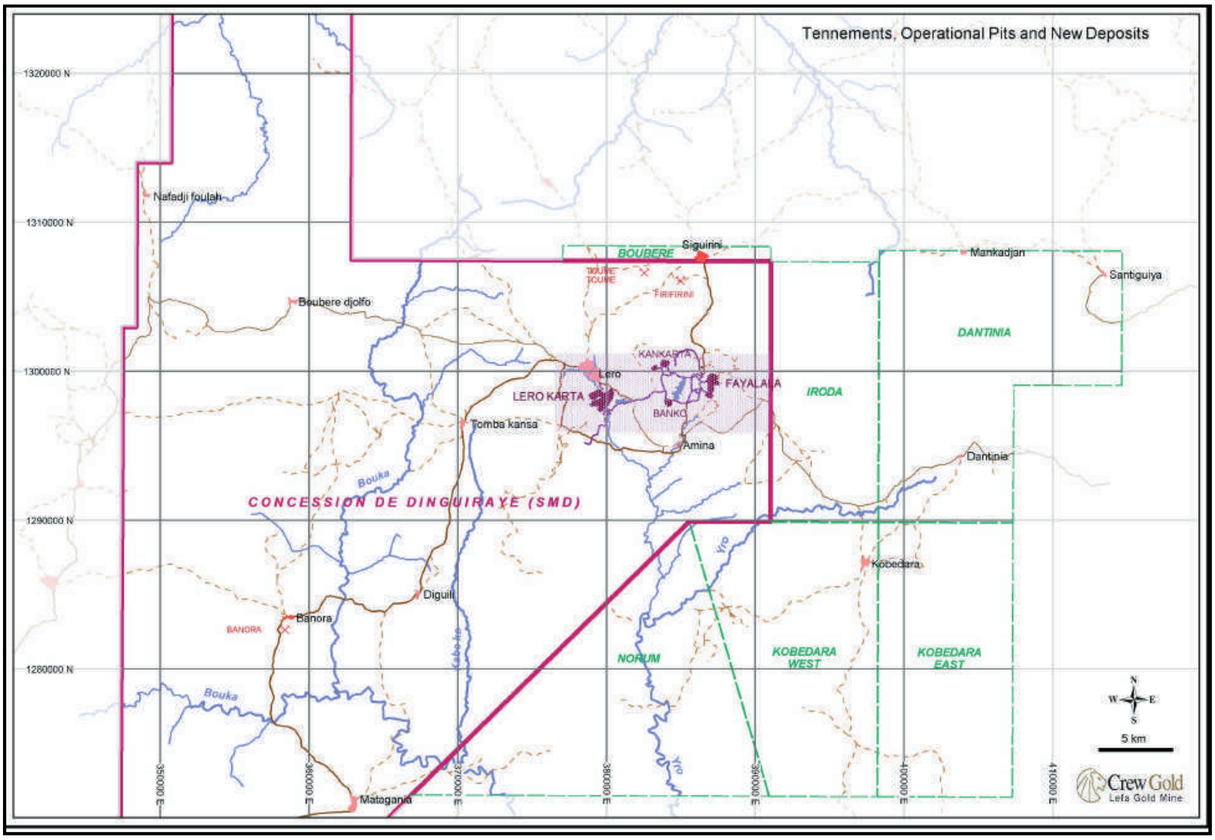


Figure 1.2: Tenement Location

The six granted Prospecting Permits adjoining the Dinguiraye concession are not subject to the Dinguiraye concession's Basic Agreement, but fall under the general mining code of Guinea. These additional Prospecting Permits comprise the granted Kobedara East, Kobedara West, Iroda, Boubere, Dantinia and Norum permits, which require a retrocession of 50% after 2 years. An extension was granted for one year, due to the favourable relationships which exist between SMD and the Mining Ministry of Mines.

The Kobedara East and West concessions comprise two contiguous Licences (2003/032 and 2003/030), both of which were originally granted on 15 May 2001, and have respective areas of $169km^2$ and $186km^2$. The Kobedara concessions lie immediately adjacent to the southeast corner of the Dinguiraye concession within the Prefecture of Siguiri. Renewals have been granted for 2-year periods for the Kobedara East and West concessions since 15 September 2005.

The Dantinia concession was successfully granted on 02 June 2005 for an initial period of 2 years, with renewals granted for 2 year periods since then. This permit lies to the east of Iroda, and forms a contiguous block extending from the Lefa mine area to the East.

The Iroda concession covering some 129km^2 in area, was originally granted on 8 November 2002. The tenement is also located within the Prefecture of Siguiri, to the immediate east of the Fayalala deposit, adjoining the eastern boundary of the Dinguiraye concession.

The Boubere concession covers some 14km^2 in area, and was originally granted on 26 January 2000. The tenement comprises a narrow east-west strip lying along the northern boundary of the Dinguiraye concession, immediately north of the Lefa mine area, within the Prefecture of Siguiri. An extension application for a further 2 years was granted for the Boubere concession on 15 September 2005, and the one year extension was granted before 50% retrocession was required.

The Norum concession, located south of the Lefa mine area and adjoining the southern boundary of the Dinguiraye concession, was granted on 17 March 2005. This tenement forms a contiguous block between the Dinguiraye concession and the Kobedara West permit, covering an area of 224km^2 within the Prefecture of Siguiri.

1.2 Geology

1.2.1 Regional Setting

The Lefa gold project lies within the Siguiri Basin, part of the Birimian volcano-sedimentary series, which dominates the basement geology of the West African Shield. The Birimian Series, including the Firifirini Basin, is under-plated by a cratonised block of Archaean high-grade metamorphic and intrusive rocks termed the Man Shield. A collisional environment resulted in the development of dominantly greenschist facies metamorphism and regional northeast to northwest trending deformation zones, considered to be fundamental to the development of gold mineralisation in the Firifirini Basin.

1.2.2 Local Setting

The Lefa 'corridor' is a zone some 10km wide within the 1,500km^2 Dinguiraye Concession. The oldest rock types, Birimian sediments, comprise an upper clayey formation and a lower coarser arkosic layer, with gold occurrences more common in the latter. Apart from younger dolerites and sandstones there is virtually no fresh outcrop. Extensive weathering and lateritisation have resulted in economic laterite and saprolite gold deposits, with primary gold mineralisation occurring in quartz vein deposits within the Birimian sedimentary units.
There are three major styles of mineralisation within Birimian sedimentary units:

- Quartz stockwork and sheeted quartz-carbonate-albite-sulphide veins (Fayalala);
- Disseminated pyrite-gold mineralisation in haematite-silica-carbonate highly fractured structures (Lero, Karta); and
- Retrograde skarn style mineralisation with magnetite-epidote (Firifirini, formerly Siguirini).

More recently Crew Gold Corporation has discovered additional resources in the form of skarn type deposits some 10km north of the plant location.

1.2.3 Project Setting

The Siguiri Basin can be subdivided into two distinct formations - the upper Matagania Formation and the lower Firifirini Formation. The Matagania Formation is dominated by inter-bedded claystones and siltstones, better represented throughout southern portion of the Dinguiraye Concession. The Firifirini Formation comprises intercalated siltstones and arkosic sandstones or greywackes and occasional conglomerates, which are more prevalent in the northern portion of the Dinguiraye Concession. Deformation and metamorphism appear to have been substantially more subdued within the Firifirini Basin compared with other West African Birimian terrains. Within the Lefa Corridor the basement stratigraphy is essentially sub-horizontal and significant fault offsets are rare. Primary mineral assemblages reflect low-grade regional metamorphism and are characterised by broad monoclinal folding.

The entire stratigraphy has been intruded by massive dolerite dykes and sills during the Jurassic period, associated with the break-up of the Gondwanan continental landmass. Being more resistant to erosion and lateritisation, these intrusions form prominent hills and bluffs within the northern half of the concession, and their magnetic susceptibility can be distinguished from airborne magnetic data.

At the regional scale, gold occurrences are more numerous in the coarser grained arkosic unit (Siguiri Formation). The lithological sequence in the Lefa Corridor area is dominated by sediments and is composed mainly of arkosic siltstones with subordinate horizons of true arkoses.

Mineralisation is preferentially developed in the more permeable, altered, coarser grained sediments, within and adjacent to east-northeast oriented structures and north to north-northwest trending fracture zones. Mineralisation is localised by a combination of lithological and structural controls. The dip and strike of mineralised zones, and to a lesser extent the style of mineralisation, varies considerably between deposits. Gold mineralisation is dominantly associated with stockwork and sheeted quartz-carbonate-sulphide veining, stockwork of albite-carbonate-sulphide veinlets, or as sulphidic hematitic breccia. Pyrite is the dominant sulphide species. Gold is largely developed within fractures in pyrite grains, rarely larger than 50 microns, and is non-refractory.

Extensive weathering and lateritisation of the mineralisation and surrounding host rocks has resulted in the development of economic laterite and saprolite gold deposits. Both transported and residual laterites, up to 15m thick, host economic gold mineralisation, which typically averages 1 to 2g/t Au over extensive lateral areas. Saprolite mineralisation tends to be higher grade (1.5 to 5g/t Au), but is generally developed over more restricted zones being between 2 and 30m wide. The base of oxidation extends to over 100m, but may be locally depressed within zones of fracturing and brecciation. The width and grade of primary mineralised zones appear to be little different from their equivalents within the saprolite profile.

1.3 Mineral Resources

1.3.1 Mineral Resource Estimates

1.3.1.1 Quality Assurance and Quality Control

All current drilling practices are conducted using industry accepted equipment, sampling and techniques including Air Core (AC), Reverse Circulation (RC) and Diamond Drilling (DD). Approved RC samples are dry, taken at 1m intervals and with a recovery of >75% whilst DD core is of NQ or HQ size (48mm and 63.5mm diameter respectively) with a maximum interval length of 1m and a recovery of >95%.

For resource definition drilling, all first pass regional exploration drilling is conducted using AC drilling, with follow up RC drilling. Historically, sampling and assaying of wet RC samples occurred, these data are flagged in the resource database, and a programme of confirmatory DD is ongoing to verify their reliability. Crew Gold Corporation only use RC drilling above the water table (i.e. dry samples) before changing over to DD methods.

All assay results reported have been determined by 50g fire assay, aqua regia digest and atomic absorption spectrometer (AAS) to a detection limit of 0.01g/t Au by independent assay contractors SGS Siguiri. A check assay programme with internationally recognised and certified umpire assay laboratories Genalysis (Perth, Australia) and ALS Chemex (Vancouver, Canada) is also conducted to confirm reliability of assay data. The data are verified on an ongoing basis by Crew Gold Corporation's Qualified Person (QP) and independent resource consultants Helman and Schofield Australia.

A new operational laboratory is located near the main mine office that will house an integrated core logging/cutting, sample preparation and assay facility capable of aqua regia digest, AAS and fire assay. These facilities have been in full operation since August 2008.

1.3.1.2 Bulk Density Measurements

A substantial database of bulk density data was collected using a variety of determination methodologies. The principal methodologies applied include the water displacement, and weight in water/weight in air approaches, as applied to diamond core, along with grab samples of hand specimen-sized samples, collected during mining advance.

SMD's geologists monitored and reviewed the appropriateness of the bulk density data, as part of the mineral resource estimation analysis, which was based on a number of data sources, derived via the following methodologies:

Method A – Water Displacement

During grade control, or ore mining, a fist-sized sample is collected;

- The grab sample is oven dried;
- The grab sample is generally coated with paraffin wax. However, a significant number of determinations were completed without waxing due to a shortage of paraffin; and
- The samples are then collectively weighed when immersed in water, allowing for the determination of their specific gravity.

Method B - Diamond Core Weight in Water and Weight in Air (data to September 2003)

- Two billets of core, representative of the range of lithologies, mineralisation styles, and alteration intensities, are routinely selected from each core tray with hole number and hole depth recorded;
- The core samples are dried in the site laboratory oven, and coated in paraffin wax to preserve any voids; and
- The dried and waxed core billets are weighed in both, air and water, to permit estimation of the bulk density. A spring balance was initially used for weight determination, but has now been replaced with an electronic balance for improved accuracy.

Method C - Half Diamond Core Weight in Water and Weight in Air (data post-September 2003)

- Three billets of half core are selected from every fifth metre downhole and are weighed together in air;
- The three half-core samples are dried in a laboratory oven for 6 hours, at 105° to 110° Celsius;
- The dry samples are collectively weighed again in air;
- The samples are then collectively weighed when immersed in water; and
- The samples are towelled dry, and collectively weighed in air again.

RSG Global completed a thorough review of the available bulk density data, for Methods B and C, during mineral resource evaluation studies, between 1999 and 2003. Limited bulk density data collected via Method B was reviewed, but was reported in various SMD studies. The majority of bulk densities are derived from the various material types within the Lero-Karta and Fayalala mineral deposits, but with some additional determinations also completed on samples from the Kankarta, Banko, Folokadi, Firifirini, Toume Toume and Banora deposits.

RSG Global was concerned that the primary density data applied in the December 2002 mineral resource estimate, superficially appeared to be conservative for the various rock types. Therefore, SMD commissioned an independent external review at Genalysis, a laboratory in Canada, of the density data, and re-determination of some 186 core billets from Fayalala, and 103 from Lero Karta.

Based on the Genalysis derived results, the SMD methodology applied to early bulk density determination was considered flawed. A 6% negative bias was identified in determinations generated in the initial stages of the LEP, primarily due to the wrapping of low porosity primary core billets with plastic film, resulting in the entrapment of air, and hence an over-estimation of the volume. Subsequent density determination work appears to have rectified issues with the cling wrapping of core no longer practiced. SMD considered the adoption of mean bulk density value, grouped by regolith and deposit, as the most appropriate means of density assignment. The dry density assignment on a deposit by deposit basis is presented in Table 1.3 below.

Table 1.3: Dry Density Values

Deposit	Laterite (t/m^3)	Saprolite (t/m^3)	Fresh (t/m^3)
Lero-Karta	2.25	1.70	2.50
Fayalala	2.20	1.70	2.50
Kankarta	2.20	1.70	2.50
Firifirini	2.20	1.70	2.50
Toume Toume	2.20	1.70	2.50
Banora	2.20	1.70	2.50
Banko	2.20	1.70	2.50

1.3.1.3 Statistical Analysis

Helman and Schofield Australia undertook detailed statistical analysis for each of the evaluated mineral deposits, based on composites of the exploration data, and coded by the weathering profile and mineralised domains.

The drillhole database coded with the geology/estimation zone was regularised, or composited, as a mean of achieving a uniform sample support. Appropriate high grade cuts, or caps, were determined, and the statistics pre and post application of the high-grade cuts assessed.

Compositing and Data Coding

Helman and Schofield Australia took a consistent approach to compositing data for each mineral deposit, based on the following factors:

- The vast majority of the data was collected at m sample intervals;
- All mineral resources would be mined via open pit method, using 2.5m flitches;
- Maintenance of some of the short scale variability identified in the drilling, while reducing and stabilising the total sample variance; and
- Run length, or down-the-hole, compositing was appropriate due to the varied drilling orientation present.

Based on the various investigations, typically a regular 2m run length/downhole composite was selected as the most appropriate composite, given the 2.5m flitch height mined at the Lefa Gold Mine. All missing intervals were excluded from the compositing process.

All data relevant to the mineral resource estimation analysis was coded to composites, including:

- Interpreted weathering;
- Mineralisation domains; and
- Where appropriate, lithology, such as dolerites.

Lero-Karta Statistical Analysis

Helman and Schofield Australia completed statistical assessment for the composite data, grouped by regolith. The purpose of the investigation was to check if there was a requirement for segregation of the datasets, for the purpose of estimation. As presented in Table 1.4, Table 1.5 and Table 1.6, Helman and Schofield Australia observed similar statistics for the saprolite, transitional and primary datasets, as with the median gold grades and Coefficients of Variation. The statistical investigation supported the geological interpretation, wherein Helman and Schofield Australia noted little difference in gold mineralisation for saprolite, transitional and primary mineralisation.

Table 1.4: Lero-Karta 2m Composite Statistics, Gold Grade in g/t Au

Statistic	Laterite	Saprolite	Transitional	Primary
Count	1381	13943	2960	12192
Minimum	0.00	0.00	0.00	0.00
Maximum	88.50	143.21	92.04	167.42
Mean	0.76	1.57	1.25	1.30
Median	0.25	0.56	0.52	0.57
Standard Deviation	3.33	4.20	3.18	3.13
Variance	11.08	17.61	10.19	9.78
Coeff Var	4.37	2.67	2.54	2.40

Table 1.5: Lero-Karta 2m Composite Statistics, Gold Grade in g/t Au, by Domain (1)

	Camp de Base				Lero		
Statistic	**Dom1**	**Dom2**	**Dom3**	**Dom10**	**Dom4**	**Dom5**	**Dom6**
Count	7959	2002	150	1564	330	4069	2423
Minimum	0	0	0	0	0.04	0	0
Maximum	143.22	25.52	141.51	85.94	17.2	124.5	88.5
Mean	1.1	0.63	2.96	0.55	1.43	2.1	0.9
Median	0.57	0.38	0.24	0.05	0.6	0.75	0.47
Standard Deviation	2.87	1.52	14.68	3.21	2.15	4.83	2.17
Variance	8.22	2.3	215.51	10.29	4.63	23.3	4.69
Coeff Var	2.62	2.39	5.01	5.85	1.51	2.29	2.4

Table 1.6: Lero-Karta 2m Composite Statistics, Gold Grade in g/t Au, by Domain (2)

	Karta				Lero South		
Statistic	**Dom7**	**Dom8**	**Dom9**	**Dom11**	**Dom12**	**Dom13**	**Dom14**
Count	5025	3875	1387	1265	335	977	115
Minimum	0	0	0	0	0	0	0
Maximum	167.41	62.97	104.21	22.99	8.68	64.65	5.11
Mean	1.55	1.6	1.78	0.55	0.84	3.21	0.8
Median	0.65	0.66	0.66	0.21	0.37	0.78	0.37
Standard Deviation	3.88	3.2	3.89	1.16	1.33	6.22	1.13
Variance	15.03	10.24	15.11	1.35	1.78	38.74	1.27
Coeff Var	2.5	2	2.19	2.11	1.59	1.94	1.42

Summary statistics of the 2m composites located within the domains, excluding missing samples and samples with length less than 1m, are presented in Table 1.5 and Table 1.6. The tightly constrained domains, namely Dom3, Dom5 and Dom13, displayed the highest mean gold grades. This was due to the fact that the domains were 060° structures, which had limited contact with the Karta Fault.

Fayalala Statistical Analysis

Helman and Schofield Australia modelled the mineralisation at Fayalala as eight mineralised structural domains, and one peripheral mineralisation domain. Summary statistics of the 2m composites located within the domains, and excluding missing samples and samples with length less than 1m, are presented in Table 1.7.

The domains of Bofeko, Fayalala South-High Grade, and Fayalala North-High grade, had the highest mean gold grades. These domains had high Coefficients of Variation, suggesting that outliers might have contributed to the distortion of the mean.

However, Helman and Schofield Australia considered if a top cut, or the median, was used, too much metal would have been removed from the mineral resource model. This was a function of the Fayalala mineral deposit, whereby short scale variability was common, with a big contrast between high and low gold grades in a mineralised interval. A top cut of 20g/t Au was applied to Domain 7, where the Coefficient of Variation was some 7.6, which Helman and Schofield Australia considered too high.

Table 1.7: Fayalala 2m Composite Statistics, Gold Grade in g/t Au, by Domain

Statistic	BF Dom1	FY_SLG Dom 2	FY_SHG Dom3	FY_F Dom4	FY_NLG Dom5	FY_NHG Dom6	FY_NMG Dom7	FY FE Dom8
Count	2726	3284	5726	11082	3832	29720	10654	8087
Minimum	0	0	0.01	0.01	0.01	0	0	0
Maximum	38.03	80.85	46.05	58.34	10.51	138.91	378.07	35.44
Mean	0.65	0.34	0.75	0.63	0.34	0.75	0.5	0.32
Median	0.16	0.1	0.16	0.29	0.13	0.19	0.11	0.09
Std. Dev.	1.75	1.6	1.83	1.26	0.57	2.09	3.81	0.98
Variance	3.06	2.55	3.35	1.58	0.32	4.38	14.5	0.96
Coeff Var	2.69	4.65	2.43	1.99	1.68	2.81	7.58	3.11

BF=Bofeko; FY_SLG=Fayalala South-Low Grade; FY_SHG=Fayalala South-High Grade; FY_F=Fayalala Fault; FY_NLG=Fayalala North-Low Grade; FY_NHG=Fayalala North-High Grade; FY_NMG=Fayalala North-Medium Grade; FY_FE=Fayalala Far East

For each domain, Helman and Schofield Australia calculated a series of 14 different cumulative probabilities of the entire domain, allocating a class mean and class median for every cumulative gold grade threshold. The different probability thresholds were 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97 and 0.99. The data was then ranked with each composite given a value from 1 to 14, based on its gold grade within the probability thresholds.

Kankarta Statistical Analysis

Helman and Schofield Australia modelled the mineralisation at the Kankarta deposit as two distinct structural domains, namely Kankarta West and Kankarta East. Both of these domains consisted of sub-domains, which displayed similar geological characteristics, and were included as two distinct mineralised domains.

Sub-horizontally, three regolith sub-domains were modelled, namely laterite, saprolite, and fresh. Nevertheless, considering there was insufficient data to generate meaningful statistics for the variogram analysis and modelling process, as presented in Table 1.8, when combined, the two mineralised domains displayed similar geostatistics. Consequently, the two mineralised domains were collated together.

Table 1.8: Kankarta 2m Composite Statistics, Gold Grade in g/t Au, by Domain

Statistic	Kankarta West	Kankarta East	Combined Domains	Peripheral Domain
Count	1499	3694	5193	22515
Minimum	0	0	0	0
Maximum	24.4	63.4	63.4	13.47
Mean	0.84	1.09	1.02	0.09
Median	0.31	0.4	0.38	0.03
Std. Dev.	1.64	2.33	2.16	0.3
Variance	2.7	5.44	4.66	0.09
Coeff Var	1.95	2.13	2.11	3.19

Firifirini Statistical Analysis

Due to its unique geological and geostatistical characteristics, Helman and Schofield Australia treated the Firifirini mineral deposit a little differently to the other mineral deposits. Because of the random scattering of gold grades found within the dataset, the construction of traditional wireframes for domaining and gold grade estimation was not possible.

All samples within the entire dataset were composited to 2m. In order to generate meaningful statistical conclusions, the domains were broken down into one large mineralised domain, with sub-domains of the different regolith boundaries, namely laterite, saprolite, and fresh, as presented in Table 1.9.

Table 1.9: Firifirini 2m Composite Statistics, Gold Grade in g/t, by Material Type

Statistic	Laterite	Saprolite	Fresh
Count	3935	9782	14440
Minimum	0	0	0
Maximum	22.65	59.2	134
Mean	0.16	0.43	0.25
Median	0.04	0.05	0.03
Standard Deviation	0.85	2.03	1.95
Variance	0.71	4.1	3.82
Coeff Var	5.14	4.72	7.96

Due to the single high grade outlier, Helman and Schofield Australia considered necessary to limit the 99th percentile bin of the statistics for the dataset. Helman and Schofield Australia decided that the value of 30g/t would be suitable as a top cut in this area, and applied it.

Banora Statistical Analysis

Helman and Schofield Australia coded to composites all data relevant to mineral resource estimation analysis, with the mineralised structural domains coded to the database. Helman and Schofield Australia modelled the mineralisation at the Banora mineral deposit as two distinct structural domains.

Sub-horizontally, three regolith sub-domains were modelled, namely laterite, saprolite, and fresh. Nevertheless, and similarly to Kankarta, considering there was insufficient data to generate meaningful statistics for the variogram analysis and modelling process, as presented in Table 1.10, the two mineralised domains were collated together.

Table 1.10: Banora 2m Composite Statistics, Gold Grade in g/t Au, by Domain

Statistic	Combined Domains	Mineralised Domains	Peripheral Domain
Count	8168	1817	6351
Minimum	0	0.01	0
Maximum	42.6	42.6	4.44
Mean	0.25	0.99	0.04
Median	0.03	0.31	0.02
Standard Deviation	1.21	2.4	0.13
Variance	1.45	5.77	0.02
Coeff Var	4.73	2.4	2.94

Helman and Schofield Australia visually inspected log Histogram and Probability plots. The composite data displayed positively skewed behaviour, typical of gold deposits. In general, the mineralisation followed a log normal distribution, except for some variability near the high-grade end of the distribution.

The 2m composites located within the mineralised domains were also coded by weathering, based on a series of modelled weathering horizons, as presented in Table 1.11. The modelled weathering horizons included the base of the laterite, the base of the saprolite, with everything below the base of saprolite considered fresh rock.

Table 1.11: Banora 2m Composite Statistics, Gold Grade in g/t Au, by Material Type			
Statistic	**Laterite**	**Saprolite**	**Fresh**
Count	468	3,576	4,124
Minimum	0	0	0
Maximum	5.59	25.4	42.6
Mean	0.15	0.2	0.31
Median	0.04	0.02	0.03
Standard Deviation	0.47	1.06	1.37
Variance	0.22	1.11	1.88
Coeff Var	3.09	5.27	4.36

The mean gold grades for the fresh was the highest, with the saprolite domain higher than the laterite domain. Helman and Schofield Australia considered estimating domains with a hard boundary between the regolith types. However, the lack of data in the laterite regolith, and the gradational nature of the mineralised intercepts between the mineralised and peripherally mineralised domains, meant that soft boundaries were used.

A review of the histogram and probability plots, along with the mean and standard deviation plots, indicated the lack of outliers in all domains at Banora. As a result, no gold grade top cut was applied during the gold grade estimation process.

Banko Statistical Analysis

The mineralisation at the Banko mineral deposit was relatively simple for domaining, and the mineralisation was modelled as just one structural domain. Sub-horizontally, Helman and Schofield Australia modelled four weathered surfaces into sub-domains, namely laterite, saprolite, transitional, and fresh. However, considering there were insufficient data to generate meaningful statistics for the variogram analysis and modelling process, the saprolite and transitional regolith domains were collated together.

Visual investigation of the mineralised domain showed the significance influence of outliers on the mean, as presented in Table 1.12, significantly higher than the median, with a very high variance and standard deviation. Therefore, a top cut of 40g/t Au was applied, to reduce the influence of these outliers. All sub-domains in the peripheral mineralisation domain were collated together.

Table 1.12: Banko 2m Composite Statistics, Gold Grade in g/t Au, by Domain			
Statistic	**Mineralised**	**Peripheral**	**All Data**
Count	1,181	8,295	9,476
Minimum	0	0	0
Maximum	378.64	16.6	378.64
Mean	2.69	0.11	0.43
Median	0.39	0.03	0.04
Standard Deviation	13.53	0.48	4.87
Variance	182.98	0.23	23.73
Coeff Var	5.03	4.49	11.37

1.3.1.4 Variography

Helman and Schofield Australia used variography to describe the spatial variability, or correlation, of an attribute (gold, silver, sulphur, etc). The spatial variability is traditionally measured by means of a variogram, which is generated by determining the averaged squared difference of data points at a nominated distance (h), or lag. The averaged squared difference (variogram or y(h)) for each lag distance is plotted on a bi-variate plot, where the X-axis represents the lag distance, and the Y-axis the average squared differences (y(h)), for the nominated lag distance.

Therefore, the term variogram is used as a generic word, to designate the function characterising the variability of variables versus the distance between two samples. Both, a traditional measure and a correlogram, were applied for the estimation completed for the Lefa Gold Mine.

Fitted to the determined experimental variography is a series of mathematical models which, when used in the kriging algorithm, recreates the spatial continuity observed in the variography. All variography for the mineral resource estimation was based on the 2m composite gold grade data captured, within the investigated mineralised domains.

Lero-Karta Variography

Helman and Schofield Australia carried out variography using 2m composite assay data. This data was coded by regolith and structural domains, based on gold grade of 0.3g/t Au envelopes, and also geological domains, like dolerite. Helman and Schofield Australia combined the data when performing the variograms for each inclusive domain, due to the similarities in the datasets, relating to regolith between the saprolite, transition and primary material in each domain.

Several mineralisation domains exhibited similar domain geometries, mineralisation styles, and tenors of mineralisation. Therefore, some of the larger domain variography was used, where there was insufficient data in the smaller domains. These smaller size domains tend to be hanging wall structures to the main structure.

Helman and Schofield Australia generated and modelled indicator and grade variograms to allow both, to generate multiple indicator kriging gold grade estimates and to undertake a change of support analysis.

The following summarises the key aspects of the Lero-Karta variogram modelling:

- Relative nugget was modelled at between 0.02 and 0.56 for the indicator variography, and between 0.18 and 0.38 for the grade variograms;
- A reasonable level of directional anisotropy, representing the characteristics of the 060° mineralisation, was present in the variogram models; and
- At lower gold grade indicator thresholds, overall ranges were in excess of the current drill spacing, while for higher gold grade thresholds, ranges were modelled at, or less than, the current drill spacing.

Fayalala Variography

Helman and Schofield Australia carried out variography on 2m composite assay data, grouped as laterite, saprolite, transitional and fresh, and sub-domained into the different regolith layers, similarly as for Lero-Karta. Rank indicator and gold grade variograms were generated, and modelled, to allow for multiple indicator kriging gold grade estimates.

Due to the lack of data in the Fayalala Far East saprolite domain, no meaningful variogram rosette maps were generated, given the small size of the dataset. Subsequently, the modelled variography for the North-South domain was applied for Fayalala Far East.

This was also the case for the laterite domains above Fayalala Far East, the laterite above the Fayalala Fault zone, and the laterite above the Bofeko domain. All of these laterite domains exhibited the same lateral consistency, and were given the same variography as the laterite domain above the North-South domain.

The variography modelled at Fayalala was consistent with observations made in both, the completed exploration programs and the current mining operations. The laterite mineralisation, when considering low gold cutoff grades, represented a significant body of mineralisation that could be adequately correlated both, downhole and between drill sections. However, the saprolite/primary mineralisation exhibited a high degree of short scale variability, wherein significant variation in gold grade was observed at close range, both, down the hole and also in adjacent drillholes. The implication of the modelled variography was that substantial drilling was required to adequately map the gold grade variation at Fayalala.

Kankarta Variography

Helman and Schofield Australia carried out variography on 2m composite assay data. One set of variography was modelled for Kankarta West, while all sub-parallel structures for Kankarta East was also modelled, with the peripheral mineralisation allocated to the same variography as the Kankarta West Dom1 domain. Insufficient data for the saprolite and the fresh sub-domains, and considering that the structures were virtually identical in the regolith zones, meant that the saprolite and fresh zones had the same variography. Typical features of the Kankarta variography were thus:

- Nugget for the gold variograms relatively low at 0.13 and 0.22, indicating reasonable continuity; and
- Indicator ranges were usually at, or greater than, the drill spacing, indicating continuity along strike.

Firifirini Variography

Helman and Schofield Australia carried out variography on indicative thresholds for the moving average of the entire dataset, using 2m composite data. Indicator and gold grade variograms were generated, and modelled, to allow for both, multiple indicator kriging gold grade estimates and change of support analysis, to be determined. Variograms were created for 14 different indicator thresholds, as well as for each sub-domain combination. A single gold variogram of the entire data set was also created.

The following summarises the key aspects of the Firifirini variogram modelling:

- Relative nugget was modelled at between 0.27 and 0.42 for the indicator variography; and
- Short range structures dominated the non-nugget variance, often with a range at, or less than, the average drill spacing, indicating the high gold grade and scattered nature of the deposit.

Toume Toume Variography

Variography for Toume Toume was treated very similarly to Firifirini, reflecting the similarities in their style of deposit and mineralisation. Helman and Schofield Australia also applied the moving average technique to the entire dataset of 2m composites. Nugget value of 0.27 for the gold variogram, and up to 0.42 for the indicator thresholds with short ranges, indicated the high gold grade and scatty nature of the mineralisation.

Banora Variography

Helman and Schofield Australia carried out variography on 2m composite assay data, constrained within the main modelled mineralisation envelope, in preparation for undertaking a Multiple Indicator Kriging estimation. Variography for the mineralised domain encompassed the saprolite and fresh portions of the regolith, as individually there was insufficient data to produce meaningful variography. Therefore, the variography was modelled with a moderately high relative nugget effect, which is typical of the narrow vein high gold grade nature of the deposit. Azimuths typically showed a 060° orientation in plan, with some short-range structures orientated at 340°.

1.3.1.5 Banko Variography

Variography for Banko was relatively simple with a singular, high grade structure modelled. The laterite variograph was typical of laterite style mineralisation with a sub-horizontal spread. Similar variography was applied to the saprolite, transitional and fresh zones of the mineralised structure. The same variography was applied to the peripherally mineralised portion of the deposit. At the upper thresholds, the nugget was high at

0.61, with very high ranges in the Fasting direction, thus reflecting the high grade nature of the deposit on a 060° oriented structure.

1.3.1.6 Block Modelling

Helman and Schofield Australia generated a series of 3-dimensional block models for each mineral deposit, to enable grade estimation via Multiple Indicator Kriging (MIK), as shown in Table 1.13 below. MIK block models are proportional models with each block having a series of varying proportions above a certain cutoff grade. All the models constructed have 11 gold cutoff grades, ranging from 0.3 to 1.0g/t Au, in increments of 0.1, and additional ones at 1.2 and 1.5g/t Au. With an increase in gold cutoff grade, the proportional amount of tonnes in the block above that cutoff decreases. Because MIK block models are proportional models, dilution and ore losses due to mining practices are inherently built into the model. Therefore, these factors do not need to be applied in estimating mineral reserves from the mineral resource models.

Helman and Schofield Australia selected the block model block size in each case to represent the available data, the data characteristics (variability as defined by variography), and the mining practices targeted in the CIL project. Table H presents the block model definition for each mineral deposit.

Helman and Schofield Australia was responsible for building and updating the main Lefa Gold Mine block models: Lero-Karta, Fayalala, Kankarta, Firifirini, Banko, Folokadi, Toume Toume, and Banora. The Sanou Kono block model was built by RSG for the 2005 Technical report. The remaining block models, namely Diguili Bougoufe, Dar Salaam, Diguili North, Banora West, Hansaghere, Sikasso, and Solabe, were built by SMD in 2008.

Table 1.13: Block Model Definition

Deposit	Lower-Left Coordinate			Block Size (m)			Number of Blocks		
	East	North	RL	East	North	RL	East	North	RL
Lefa Corridor									
Lero-Karta	4,712.50	4,325.00	12.5	25	25	5	79	72	107
Fayalala	12,462.50	4,225.00	182.5	25	25	5	93	74	65
Kan karta	9,137.50	6,575.00	267.5	25	25	5	64	32	62
Firifirini	384,062.50	1,305,712.50	397.5	25	25	5	57	51	30
Ban ko	9,762.50	4,125.00	252.5	25	25	5	25	19	41
Folokadi	9,900.00	2,300.00	240.5	20	20	5	51	51	63
Toume Toume	380,563.50	1,306,512.50	402.5	25	25	5	15	18	22
Sanou Kono	9,123.75	8,148.75	601.25	15	15	5	41	27	17
Regional									
Banora	357,737.50	1,282,270.00	200.5	25	15	5	38	49	54
Diguili Bougoufe	365,012.50	1,283,512.50	255	25	25	10	20	20	20
Dar Salaam	389,012.50	1,290,012.50	205	25	25	10	25	120	30
Diguili North	363,012.50	1,288,012.50	205	25	25	10	80	120	30
Banora West	356,012.50	1,283,312.50	255	25	25	10	32	56	20
Hansaghere	364,612.50	1,293,612.50	255	25	25	10	56	32	20
Si kasso	393,012.50	1,296,912.50	305	25	25	10	24	20	23
Solabe	365,012.50	1,283,512.50	255	25	25	10	20	20	20

Block model development was completed in the Surpac software. The raw data was then imported into the GS3 Hellman and Schofield software to allow grade estimation. Helman and Schofield Australia established mineralised domain and weathering domain coding in the block model, based on the modelled wireframe constraints. In addition, sufficient variables were created to enable recording of the results from MIK, selective mining estimates, estimation statistics, density stratification, and resource classification.

Lero-Karta and Fayalala block models were trimmed using topography appended with the pit outlines as of 1 July 2010 by WAI, to allow depletion of the resources for reporting purposes. All of the other deposits have remained dormant since the previous Mineral Resource Estimations so these block models have been cut using pit surfaces dated 31 August 2009.

Lero-Karta Model

Helman and Schofield Australia constructed a block model for mineral resource estimation purposes, based on a 25 x 25 x 5m panel size, which was approximately the drilling spacing, with no sub-blocking occurring. The initial processing was performed in the Surpac software, and it involved creating wireframes based on mineralised envelopes and geology. Helman and Schofield Australia extracted composite data from the database using Surpac, and then imported it into GS3, where statistical analysis and the gold grade estimation were performed. After the creation of the model using GS3, the information was then exported and brought back into Surpac, where verification and subsequent gold grade and tonnage reports were created.

Gold grade estimation was undertaken applying MIK, where interpolation was constrained within the domains, inclusive of the sub-domains of saprolite, transition and primary material, as well as the estimation of peripheral mineralisation. Helman and Schofield Australia carried out gold grade estimation in three passes, to represent the various mineral resource categories, as presented in Table 1.14.

Table 1.14: Lero-Karta Data Search Parameters

	Search			Composites		Octants
Pass	**East**	**North**	**RL**	**Min.**	**Max.**	**No.**
1	25	25	15	16	32	4
2	37.5	37.5	22.5	16	32	4
3	37.5	37.5	22.5	8	32	2

Helman and Schofield Australia used block variance adjustment factors, as determined from the variography, to emulate a 10 x 10 x 2.5 m specialised mining unit (SMU) considered the minimum mining recoverable proportion of the block, as presented in Table 1.15. The shape of the modelling method applied was the Direct–Lognormal, which is typical of gold deposits, and the most conservative estimate of recoverable gold grade.

Table 1.15: Lero-Karta Change of Support Parameters

Domain	Variance Adjustment Factor
0	0.134
1	0.141
2	0.141
3	0.141
4	0.134
5	0.134
6	0.134
7	0.142
8	0.142
9	0.142
10	0.141
11	0.142
12	0.228
13	0.228
14	0.228

Fayalala Model

Helman and Schofield Australia constructed a block model for mineral resource estimation purposes, based on a 25 x 25 x 5m panel size. No sub-blocking occurred due to the spatial variability of the Fayalala data. Helman and Schofield Australia chose the parent block size for its representation of the drillhole spacing, variography, and mineralisation parameters.

The block model, mineralised solids, and the different regolith surfaces, were created in Surpac. This data was then further manipulated so it could be easily incorporated into GS3. Once in GS3, extensive statistical analysis, variogram modelling, and grade estimation were performed. The resultant file was then uploaded back into Surpac for mineral resource reporting, visual assessments and verification.

Gold grade estimation was undertaken applying MIK, where interpolation was constrained within the eight mineralised domains, inclusive of the sub-domains of laterite, saprolite, transition and primary material, as well as the estimation of peripheral mineralisation. Helman and Schofield Australia carried out gold grade estimation in three passes, to represent the various mineral resource categories, as presented in Table 1.16.

Table 1.16: Fayalala Data Search Parameters

	Search			Composites		Octants
Pass	**East**	**North**	**RL**	**Min.**	**Max.**	**No.**
1	25	25	15	16	48	4
2	32.5	32.5	19.5	16	48	4
3	32.5	32.5	19.5	8	48	4

Helman and Schofield Australia applied a minimum of 16 composites, and a maximum of 48, for the first pass, including a minimum of 4 octants from any one drillhole selected. The search radii for the first pass were 25 x 25 x 15m. For the second pass, an expansion factor of 30% was applied. Therefore, the search distance for the minimum number of samples for a successful estimate was 32.5 x 32.5 x 19.5m.

The third search, defining the inferred category, used the same search distances as the second search, but with much reduced data constraints on each panel. All mineralised and peripherally mineralised domain boundaries were hard with each other. Variance adjustment factors, as determined from the variography, were used to emulate a 5 x 5 x 2.5m SMU, as presented in Table 1.17.

Table 1.17: Fayalala Change of Support Parameters

Domain	Variance Adjustment Factor
0	0.196
1	0.311
2	0.243
3	0.243
4	0.233
5	0.196
6	0.196
7	0.196
8	0.214

Kankarta Model

Helman and Schofield Australia constructed in Surpac a block model for mineral resource estimation purposes, based on a 25 x 25 x 5m panel size. This panel size was slightly wider than the drilling spacing, as the Kankarta mineral deposit was essentially overdrilled across strike. All geostatistical analysis, variography modelling, and gold grade estimations was completed in GS3, and then incorporated back into Surpac. Hard boundaries were applied between the mineralised solids and the surrounding peripheral mineralisation.

Data search parameters for the three passes for the mineralised categories are presented in Table 1.18. The greater continuation for the search was along the Fasting axis, as represented by 30m along the Fasting and 20m along the Northing, simulating the continuation of the mineralisation along the 060° – 080° structures.

Table 1.18: Kankarta Data Search Parameters						
	Search			Composites		Octants
Pass	East	North	RL	Min.	Max.	No.
1	30	20	15	16	32	4
2	45	30	22.5	16	32	4
3	45	30	22.5	8	32	2

Firifirini Model

The MIK estimate completed for Firifirini used a regular sample search on a 25 x 25 x 5 m block model. The sample search applied reflects the mineralised domain interpretation, with a minimum of 16 composites, and maximum of 32, used to provide the first and second pass estimates. A third, lower confidence estimate was then completed, which required only 8 composites. Parent blocks were discretised using 5 points in the east dimension, 5 in the north dimension, and 2 in the RL dimension, for a total of 50 discretisation points per block.

Helman and Schofield Australia applied a single domain to the entire dataset for gold grade estimation, with all composites considered, and no boundaries. Based on the modelled variography, variance adjustment factors were generated and applied, to emulate a 5 x 5 x 2.5m SMU, via the indirect log normal change of support. Data search parameters are presented in Table 1.19. Search rotation of -30° in the north direction, and -60° in the RL direction, were also specified in the data search, for all three passes.

Table 1.19: Firifirini Data Search Parameters						
	Search			Composites		Octants
Pass	East	North	RL	Min.	Max.	No.
1	25	25	10	16	32	4
2	32.5	32.5	13	16	32	4
3	32.5	32.5	13	8	32	4

Toume Toume Model

The MIK estimate completed for Toume Toume was virtually identical to Firifirini, reflecting their similarities in styles of mineralisation. The block sizes, search parameters for the different passes, and number of discretisation points were the same as the Firifirini model. Data search parameters for the Toume Toume model are presented in Table 1.20, with the rotation of the searchs as the only major difference with Firifirini, at -30° in the north direction, and 30° in the RL direction.

Table 1.20: Toume Toume Data Search Parameters						
	Search			Composites		Octants
Pass	East	North	RL	Min.	Max.	No.
1	25	25	10	16	32	4
2	32.5	32.5	13	16	32	4
3	32.5	32.5	13	8	32	4

Banora Model

Helman and Schofield Australia constructed a block model for mineral resource estimation purposes, based on a 20 x 15 x 10m panel size. This panel size was approximately half the drilling spacing. No sub-ceiling was applied. The mineral block model, mineralised solids, and the different regolith surfaces, were created in Surpac. All geostatistical analysis, variography modelling, and gold grade estimations, were completed in GS3, and then incorporated back into Surpac, for mineral resource reporting and visual assessments.

Gold grade estimation was undertaken applying MIK, where the interpolation was constrained within two mineralised domains, straddling the sub-domains of laterite, saprolite, and fresh. Helman and Schofield Australia used a three-pass search strategy in gold grade estimation, as presented in Table 1.21 and Table 1.22.

Table 1.21: Banora Data Search Parameters for Mineralised Domain						
	Search			Composites		Octants
Pass	East	North	RL	Min.	Max.	No.
1	35	15	30	16	32	4
2	52.5	22.5	45	16	32	4
3	52.5	22.5	45	8	32	4

Table 1.22: Banora Data Search Parameters for Waste Domain						
	Search			Composites		Octants
Pass	East	North	RL	Min.	Max.	No.
1	50	20	10	16	32	4
2	75	30	15	16	32	4
3	75	30	15	8	32	4

Search rotation of -20° in the east direction, and 20° in the RL direction, were also specified in the data search, for all three passes, for all domains.

For the first pass, a minimum of 16 composites, and a maximum of 32, were applied, including a maximum of 4 octants from any one drillhole selected. Search radii for the first pass were 35 x 15 x 30m. For the second pass, the minimum number of samples for a successful estimate was 52.5 x 22.5 x 45m. This is the first pass estimation with an expansion factor of 50% on the original distances.

The third search, defining the inferred category, used the same search distances as the second search, but with much reduced data constraints on each panel. All domain boundaries were soft with each other with broad mineralised envelopes capturing the majority of the mineralisation, to reduce the effect of overestimating gold ounces on the margins of the mineralised solid. Other variance adjustment factors, as determined from the variography, were used to emulate an 8 x 3 x 2m SMU, as presented in Table 1.23.

Table 1.23: Banora Change of Support Parameters	
Domain	Variance Adjustment Factor
Mineralised	0.175
Waste	0.175

Banko Model

The mineral block model was constructed in Surpac, following the geostatistical, variographic modelling, and gold grade estimation performed in GS3. Block sizes of 25 x 25 x 5m were employed, reflecting drilling spacing and density, with discretisation points of 5 x 5 x 2, making up 50 points per block.

Search parameters for the block model construction utilised a search of 25 x 25 x 15m, with an expansion factor of 50%, as presented in Table 1.24. There was no rotation of the searches. Block variance adjustment factor was 0.219, with hard domains between the mineralised domain and the peripheral mineralisation.

Table 1.24: Banko Data Search Parameters						
	Search			Composites		Octants
Pass	East	North	RL	Min.	Max.	No.
1	25	25	15	16	32	4
2	37.5	37.5	22.5	16	32	4
3	37.5	37.5	22.5	8	32	4

Stockpiles

SMD estimated the tonnages, gold grades, and mineral resource categories, relating to all stockpiles incorporated within the 31 August 2009 mineral resource statement. All stockpile material was either trench sampled or grade control drilled at 10 x 6 m, or closer spacing, with gold grade estimation based around conditional simulation techniques, using Hellman and Schofield's MP3 software. The mineralised rock was then interpreted based on the optimised mineralised outlines, and given to survey to mark out on the bench for extraction by mining. At the completion of every month, the Lefa Gold Mine survey department surveys the stockpiles for volume estimation.

A density was applied to the stockpiles for tonnage estimation, as presented in Table 1.25. SMD believes all stockpile material may be classified in the *Measured* category.

Table 1.25: Stockpile Dry Density Values

Laterite (t/m^3)	Saprolite (t/m^3)	Transitional (t/m^3)	Fresh (t/m^3)
2.025	1.55	2.025	2.25

Additional to the RoM stockpiles, SMD identified the potential of processing mineralised material remaining in the old heap leach pad. Prior to the commissioning of the CIL plant, gold was extracted at the Lefa Gold Mine by cyanide heap leaching, between 1995 and 2006.

Areas of the heap leach pad were drilled to ascertain whether an economic stockpile existed. SMD used an approximate spacing of 15 x 15m, or closer, with vertical holes and samples taken every metre. The samples were assayed at the SGS Laboratory on site, using the same aqua regia methodology for grade control samples. The samples were composited according to their assay on a 2.0m sample interval. Table 1.26 presents the statistics of the heap leach pad's drillhole composite samples.

Table 1.26: Basic Statistics of Heap Leach 2 m Composite Samples

Statistic	Value
No of Samples	14,018
Minimum	0.000g/t
Maximum	43.23g/t
Mean	0.466g/t
Variance	0.61
Standard Deviation	0.781
Co-efficient of Variation	1.678
Skewness	21.01
Kurtosis	858.476
Top Cuts applied	None

SMD block modelled the heap leach pad in Surpac, using the inverse distance squared method, with block dimension of 15 x 15 x 5m, which is the same as the maximum drillhole spacing. The entire Heap Leach stockpile has not been drilled. Therefore, the estimation is based on the drilled areas only.

SMD considered the inversed distance squared method appropriate to model the drilled out portions of the heap leach pad, since there is no geological continuity. That is, due to the lack of geological structures observed within the heap leach pad, as it constitutes an amorphous stockpile. Details of the block model construction are presented in Table 1.27 below.

Table 1.27: Heap Leach Inverse Distance Block Model Construction Parameters	
Parameter	**Value Used**
Data Constraints	Unconstrained
Model Constraints	Unconstrained
Search Radii	35 x 35 x 10m
Major to Semi Major axis anisotropy	01:01
Major to Minor axis anisotropy	01:01
Maximum Search Distance of Major Axis	50m
Maximum Vertical Search Distance	10m
Maximum Number of Informing Samples	16
Minimum Number of Informing Samples	4

The topographic surface of the heap leach stockpile was surveyed by SMD, and with a pre-stacking topographic surface, a total mass of some 13.4Mt was estimated, based on a dry density of 1.5t/m3. SMD believes that, considering the drilling, sampling and surveying techniques to be of sufficient quality, the heap leach stockpile material may be classified in the Indicated category.

1.3.1.7 Mineral Resource Statement

The Updated Mineral Resource Statements, as at 01 November 2010, was prepared and reported in accordance with the guidelines of JORC Code (2004). The Global Mineral Resources are shown in Table 1.28.

Each of the block models provided by Crew Gold Corporation for the Lero-Karta, Fayalala, Kankarta, Firifirini, Banko, Folokadi and Toume Toume deposits were loaded into Datamine Mining Software in order to generate Global Mineral Resources. These were then depleted using topographic surfaces dated 01 November 2010. The global resources are subdivided by mineral deposit and resource category.

Table 1.28: Updated Global Mineral Resources by Classification (WAI, 01 November 2010) (In accordance with the guidelines of the JORC Code (2004))									
	Measured			*Indicated*			*Inferred*		
Deposit	**Tonnage (kt)**	**Grade (g/t Au)**	**Metal (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Metal (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Metal (koz)**
Lefa Corridor									
Lero-Karta	26,038	1.43	1,197	15,627	1.45	729	7,948	1.24	317
Fayalala	44,095	0.99	1,404	8,846	0.89	253	11,534	0.93	345
Kankarta	3,217	1.40	144	1,958	1.23	78	984	1.08	34
Firifirini	4,455	1.49	213	2,715	1.28	112	2,487	1.36	109
Banko	1,134	2.20	80	521	1.58	26	250	0.93	7
Folokadi	545	1.51	27	1,736	1.66	93	691	2.01	45
Toume Toume	239	1.33	10	521	1.26	21	617	1.27	25
Sanou Kono	-	-	-	1,629	1.2	60	-	-	-
Stockpiles	6,985	0.76	170	-	-	-	-	-	-
Heap Leach	-	-	-	2,313	0.8	57	-	-	-
Sub-total	86,708	1.16	3,245	35,866	1.24	1,429	24,511	1.12	882
Regional									
Banora	2,196	1.70	119	598	1.50	29	330	1.60	17
Diguili Bougoufe	-	-	-	-	-	-	273	2.10	18
Dar Salaam	-	-	-	-	-	-	522	1.10	18
Diguili North	-	-	-	-	-	-	1,782	1.40	78
Banora West	-	-	-	-	-	-	432	1.50	21
Hansaghere	-	-	-	-	-	-	511	1.10	18
Sikasso	-	-	-	-	-	-	584	1.40	26
Solabe	-	-	-	-	-	-	371	1.50	18
Sub-Total	2,196	1.70	119	598	1.50	29	4,805	1.40	214
TOTAL	**88,904**	**1.18**	**3,364**	**36,464**	**1.24**	**1,458**	**29,316**	**1.16**	**1,096**
*NB - Resource has been updated by WAI by depleting the July 2010 model with November 2010 topography update.									

The cut-off grades used to report the Lefa Gold Mine Mineral Resource Inventory are presented in Table 1.29.

Table 1.29: Mineral Resource Cut-off Grades by Material Type (Crew Gold Corporation November 2009)			
	Cutoff Grade - g/t Au		
Deposit	**Laterite**	**Saprolite**	**Fresh**
Lero-Karta	0.5	0.4	0.5
Fayalala	0.5	0.4	0.5
Kankarta	0.4	0.4	0.5
Firifirini	0.5	0.5	0.6
Banko	0.4	0.4	0.5
Folokadi	0.4	0.4	0.5
Toume Toume	0.5	0.5	0.6

1.4 Mining

1.4.1 Introduction

Lefa gold mine operates a typical modern open pit operation with drilling and blasting (where required) followed by dig and haul allowing for selective mining of the ore grade material. The target mining rate is 8Mt per year with an overall stripping ratio of approximately 4:1, although this varies from pit to pit. SMD is currently mining in the two main areas of the resource; namely Lero-Karta and Fayalala. The site layout is displayed in Figure 1.3.

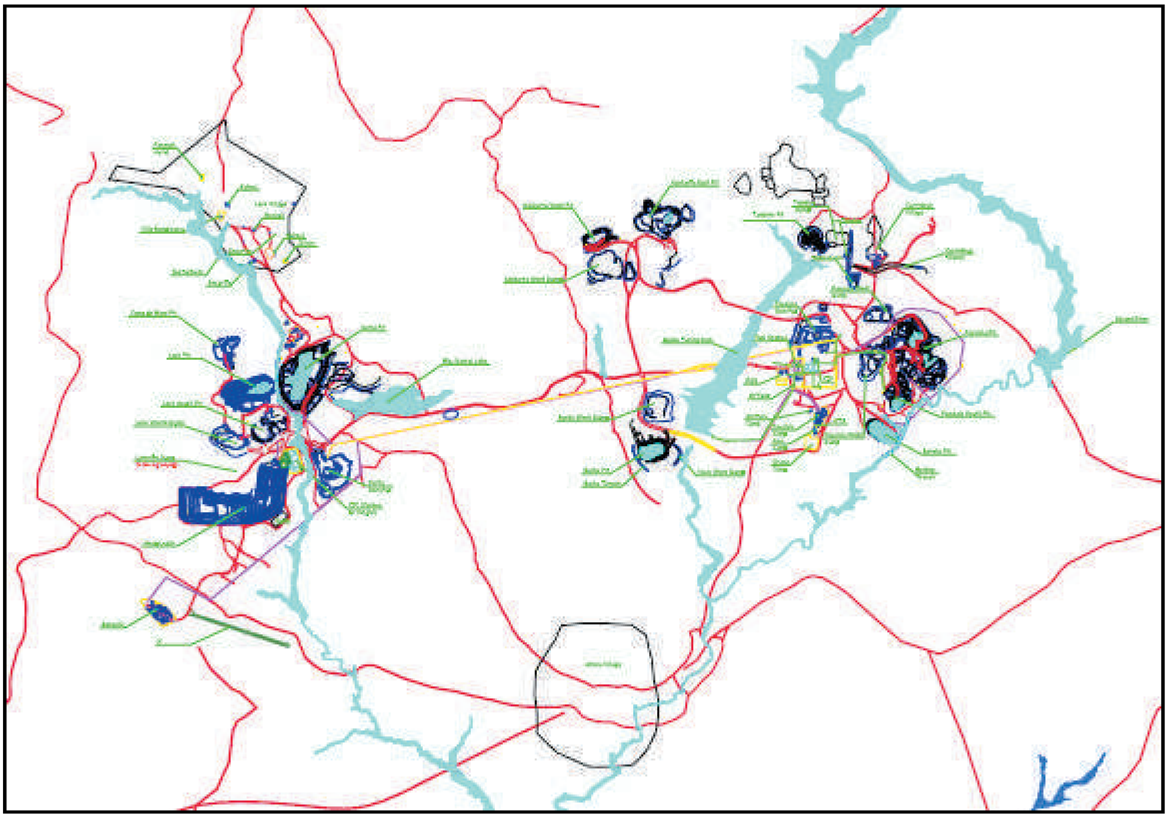


Figure 1.3: Site Layout

1.4.2 Ore Reserve Estimates

The Lefa Gold Mine Ore Reserve estimations were carried out using the Mineral Resource block models generated by WAI using the topographic data as of 01 July 2010, as provided by Crew Gold Corporation.

The Ore Reserves were estimated at a USD 900/oz gold price. This gold price approximated the three year average gold price (London fix) as at 31 August 2009. The methodology applied is summarised as follows:

- Pit shells were estimated using NPV Scheduler, and a set of technical and economic parameters;
- Final pit shells were selected from the NPV pit shell series;
- The final pit shells were smoothed to eliminate non operational areas and for geometry; and
- The smoothed pit shells were converted to practical pit designs, which were considered for mineral reserves reporting.

1.4.3 Pit Optimisation

Based on these mineral resource block models developed in the Datamine software, and a set of technical and economic parameters, NPV Scheduler used a Lerch-Grossmann based algorithm to generate pit shells, where the project operating profit margin, or cash flow, was maximized.

At the economic pit limit, the incremental pit shells were at break even, where the revenue equals the operating costs. This was the economic final pit limit for each mineral deposit. The smaller nested pit shells were useful to help decide where mining was to be started, as these small pit shells were mining the highest profit areas of the mineral resources.

The parameter for generating the incremental pit shells was set to allow the evaluation of the different mineral resource block models at a gold price of US$900/oz. NPV Scheduler reported the results of each incremental nested pit shell on two basis, estimating:

- The undiscounted cash flow, and
- The discounted cash flow.

Typically, the 'optimum' pit shell is then chosen by inspecting these cash flows and selecting the pit shell with the maximum total cash flow. The maximum undiscounted cash flow is the pit shell where the incremental pit is breaking even, and is therefore the maximum economic pit in today's revenue/cost terms. If a discount rate is used, the pit shell with the maximum discounted cash flow is always somewhat smaller. However, this smaller 'final pit' would be more profitable.

Using the discount factor specified, NPV Scheduler produces two cash flows based on different scheduling scenarios. The first case, namely the Best Case, assumes that mining progresses strictly according to a series of incremental nested pit shells. This scenario is optimistic, and is not practical, but it does indicate the highest possible project value that might be achievable. The second case, namely the Worst Case, assumes that mining progresses on a bench by bench basis, mining to the limit of the 'final' pit. This scenario indicates the lowest possible project value. In reality, the pit will operate somewhere between the two cases, where intermediate and practical cutbacks are defined and then mined in sequence.

If the discount factor is 0%, the cash flows for the best and the worst cases will be equal, and therefore the 'optimum' pit shell (maximum cash flow) will be the same.

The Lerch-Grossmann based algorithm is founded on mining whole blocks only. The resulting pit shells are quite irregular and do not incorporate ramps. The pit shell results are used for preliminary project economic analyses, to assess the sensitivity of the project due to changes to input parameters, and to guide intermediate and final pit design.

Pit optimisation inputs were estimated from first principles, and were mainly based on the Lefa Gold Mine productivity models and estimated preliminary budget costs for 2010. The metallurgical parameters were derived from laboratory testwork from representative ore samples.

Table 1.30 presents a summary of the main pit optimisation input parameters per pit/deposit. These parameters include the following:

- Pit slope angles: The recommended inter-ramp slope angles (at a factor of safety of 1.2; As NPV Scheduler requires an overall slope angle, an allowance of some 5° was used to account for access ramps, where appropriate;
- Mining cost: This cost item includes drilling, blasting, loading and hauling. The hauling cost of ore is to the surface ROM and that of waste is to surface waste dumps. A mining cost adjustment factor has been applied to simulate the increase in the mining cost due to longer haul distances as each pit gets deeper;

- Mining dilution and mining recovery: Ore dilution and losses, due to mining activities, are inherently built into the mineral resource models;
- Processing cost: This is the cost per tonne of ore for crushing and processing the ore in the CIL plant;
- Administration cost: It is the cost per tonne of ore of administration and overheads (G&A) for running the mining operation;
- Rehabilitation cost: This cost provides an allowance for rehabilitating the waste dumps, and for funding mine closure;
- Processing recovery: Process/metallurgical recovery factors have been estimated for each ore type, per deposit;
- Selling Costs/Royalty: This factor is based on 5.6% of revenue, plus US$1.05 per gold ounce produced;
- Base Product Price: A gold price of US$900/oz was used;
- Mineral resources: Only measured and indicated mineral resources were considered for ore potential in the pit optimisation exercise. Inferred mineral resources were treated as waste; and
- Material types: Transition and fresh material types were grouped into one material type, namely Fresh.

Table 1.30: Pit Optimisation Parameters

Description	Unit	LK FY	KK BK FL	FF	TT	LK FY	KK BK FL	FF	TT	LK FY	KK BK FL	FF	TT
Ore Type	-	**Laterite**	**Laterite**	**Laterite**	**Laterite**	**Saprolite**	**Saprolite**	**Saprolite**	**Saprolite**	**Fresh**	**Fresh**	**Fresh**	**Fresh**
Gold Price	US$/oz	900	900	900	900	900	900	900	900	900	900	900	900
	US$/g	28.94	28.94	28.94	28.94	28.94	28.94	28.94	28.94	28.94	28.94	28.94	28.94
Royalty	% of revenue	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4
	US$/g	1.56	1.56	1.56	1.56	1.56	1.56	1.56	1.56	1.56	1.56	1.56	1.56
Selling Cost	US$/oz	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05
	US$/g	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03
Total Royalty and Selling Costs	US$/g	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60
Discount Factor	%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%
Waste mining cost	**US$/t**	**1.40**	**1.40**	**1.40**	**1.40**	**1.40**	**1.40**	**1.40**	**1.40**	**1.40**	**1.40**	**1.40**	**1.40**
Ore mining cost	US$/t	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70
G&A	US$/t	3.2	3.2	4.8	5.5	3.2	3.2	4.8	5.5	3.2	3.2	4.8	5.5
Environmental	US$/t	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
Processing	**US$/t**	**7.5**	**5.9**	**5.9**	**5.9**	**5.9**	**5.9**	**5.9**	**5.9**	**8.1**	**8.1**	**8.1**	**8.1**
Grade Control Cost	US$/t	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
Process Mainrtenance Cost	US$/t	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3
Crusher Feed Cost	US$/t	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
Sustaining Capital Cost	US$/t	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Total ore costs	**US$/t ore**	**13.50**	**11.90**	**13.50**	**14.20**	**11.90**	**11.90**	**13.50**	**14.20**	**14.10**	**14.10**	**15.70**	**16.40**
Dilution Factor	%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Mining Recovery Factor	%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
Processing Recovery	%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
CIL Ore Process Rate	Mtpa	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0
Derived Cut-Off Grade	**g/t**	**0.55**	**0.48**	**0.55**	**0.58**	**0.48**	**0.48**	**0.55**	**0.58**	**0.57**	**0.57**	**0.64**	**0.67**

NB -

1) LK=Lero-Karta, Camp de Base, Pharmacie; FY=Fayalala, Bofeko; KK=Kankarta; BK=Banko; FL=Folokadi; FF=Firifirini; TT=Toume Toume

2) Any Transition ore has been assigned the same optimisation parameters as Fresh ore

1.4.3.1 Pit Design Parameters

Following from the pit optimisation exercise, and considering the relative life of each pit, the selected pit shells were used as a template for the design of the practical pits.

The pit shells are based on mining whole ore and waste blocks from the mineral resource block model, which results in the pit shells being quite irregular in shape. The design of practical pits ensures that all material can be mined with a minimum of difficulty, and smoothes out the pit shells.

Practical pit designs endeavour to extract the majority of the economic ore within the pit shells. However, it is not always possible to practically mine all of the ore indicated by the pit shells. For example, this is most evident in the base of the pit shells, where the Whittle program infers that the pits will be mined down to a block. This is typically not possible with earth moving equipment, so the last few meters of the pit shells are often ignored, and the pits are designed with a flat floor large enough for the mining equipment to work in. The smallest area that the equipment can work in is referred to as the "minimum mining width".
The following are the main assumptions for the mining of the mineral deposits:

- Mining equipment with excavators in backhoe configuration;
- Pit wall configuration – as per geotechnical recommendations; and,
- Surface access roads are developed linking the various pits and surface dumps.

In-pit ramps have been designed using the following parameters:

- Ramp width (major) 22m;
- Ramp width (minor) 18.5m and 15m; and
- Ramp gradients 10%.

The pit designs were used to generate triangulated surfaces representing the end mining topography. These surfaces were then used to extract the mineral resource tonnages and grades, on a bench-by-bench basis, from the respective mineral resource block model.

Marginal gold cut-off grades used for estimating in-pit mineral reserves were based on considering all cost beyond the pit rim. Costs that have been included are the extra ore haulage beyond the pit, processing, general site administration, and royalties and selling costs.

The derived gold cut-off grades, as presented in Table 1.31, were estimated using the equation:

$$\text{Cut-off Grade (g/t)} = \frac{\text{(Incremental mining cost + process related cost + G\&A Cost) – Rehab}}{\text{(Gold Price – Sales Cost/Royalty) x Metallurgical Recovery}}$$

The incremental mining cost is the extra cost that would be incurred (for example, grade control and haulage costs) should a block of material be defined as ore rather than waste. Any material with grade greater than, or equal to, the marginal gold cut-off grade was classified as ore. If the marginal gold cut-off grade was not met, then the material was classified as waste, or sub economic material.

Table 1.31: Derived Gold Cut-off Grades (g/t)

Deposit	Laterite	Saprolite	Fresh
Lero-Karta	0.55	0.48	0.57
Fayalala	0.55	0.48	0.57
Kankarta	0.48	0.48	0.57
Firifirini	0.55	0.55	0.64
Banko	0.48	0.48	0.57
Folokadi	0.48	0.48	0.58
Toume Toume	0.58	0.58	0.67

1.4.3.2 Pit Wall Configuration

The following section was extracted from a Scott Wilson report, titled "Lefa Corridor Project – Geotechnical Pit Design Reviews", November 2009.

The following pit slope angle recommendations are only provided for use in planning of mining operations. They are not for use in construction and detailed pit design as they are derived largely from the field observations of the safer elements of the current operational pits and have not been analytically substantiated in respect to a slope design criteria of a FoS of >1.2. Note slope angle comments generally relate to angles within lithological units whereas inter-ramp angles are defined by the slope between the toe of successive benches.

- Laterites (all pits) – in transported laterites where profiles can be accommodated in a single bench, a cut-face slope of 25° and a minimum catch berm width equal to the bench height is recommended. In-situ laterites maybe considered as per saprolites.
- Saprolite slopes for Lero Karta Pits (including Camp de Base and Pharmacie)
  - 37° in saprolites are recommended in all but north west to north east slopes, and
  - 30° inter-ramp slopes are recommended in permanently drained, (bedding controlled) northern, north eastern and north western slopes (i.e. not down gradient from the Lero river diversion), or any undrained slopes.
- Saprolite slopes for Fayalala Pit
  - an inter-ramp slope of 37° is recommended noting that this accords with previous recommendations of an overall slope angle of 40° in saprolite in the east walls of south pit, and field observations of pit west walls indicate the oldest of drained saprolite slopes also appear stable up to 40°, and
  - north walls need to reflect the structural control of bedding day-lighting in the slope an overall slope of 27° is recommended as per previously.
- Saprolite slopes for Firifirini
  - An inter-ramp slope angle in the saprolites of 40° is recommended for the eastern slopes to reflect the structural context established for the pit from the generalised geological model, and an overall slope of 45° is recommended for all other slopes.
- Transition Zone
  - 39° inter-ramp slopes are recommended, as per previously in the transition zone in the Lero Karta pits;
  - a similar inter-ramp slope is also recommended for Fayalala given the performance of the current slopes;
  - a similar slope is also recommended for the proposed pits of Camp De Base, Pharmacie, Fayalala, Bofeco and Banko as they are considered to be located in a similar structural context, and
  - 40° and 42° inter-ramp slopes are as previously recommended in the transition zone for Kankarta West and East pits respectively.
- Fresh Bedrock (all pits) – a 60° inter-ramp has been adopted in all pits as per previous recommendations. However, a 55° inter-ramp slope for all north western, northern and north eastern slopes is recommended and this will ultimately need to be designed taking into account the structures, bedding dip and the condition of the bedding planes. In addition, it is essential that slopes are drained by in pit dewatering and that bench integrity is maintained by the adoption of pre-splitting to reduce face disturbance and maintain the design profile.
- Kankarta East
  - 40° inter ramp slopes above 505mRL and 55° below 505mRL, 60° in fresh rock and 55° in ROM on northern slopes where potentially structurally controlled are recommended and 30° inter-ramp slopes for the north wall pit slopes in the saprolites.
- Kankarta West
  - inter ramp slopes of 42° above 460mRL and 45° below 460mRL in ROAL are recommended and 60° in fresh rock and 55° in ROM on northern slopes where potentially structurally controlled.

- Bofeco
  - 30° inter-ramp slopes in the saprolites are recommended in the south east pit wall as it is assumed it is not drained, given the nearby unlined river diversion, and
  - 37° inter-ramp slopes in the saprolites are recommended for all other pit walls.
- Banko
  - 37° inter-ramp slopes are recommended for the saprolites slopes.
- Folokadi
  - structural information is limited to determine if Folokadi is potentially similar to Fayalala or Lero Karta. 37° inter-ramp slopes and 27° overall slope on the northern pit saprolites slopes are recommended.
- Toume Toume
  - structural information is limited to determine if Toume Toume is potentially similar to Fayalala or Lero Karta. 37° inter-ramp slopes and 30° overall slope on the northern pit saprolites slopes are recommended.
- River sediments – in Camp de Base, Firifirini and Toume Toume pits river sediments need to be stripped back to ensure river diversions pick up surface and near surface interflows. Specific pit slopes are not defined, but allowances for overburden strip and diversion works need to be identified.

A total of over 50 pit slope sections, were inspected for the compliance check of final pit designs with geotechnical slope recommendations. The sections were selected typically at 25m spacing on each pit wall. Within each section, a total of 359 lithological defined slopes were specifically checked with a total height of over 7,650m - the results are summarised in Table 1.32.

Table 1.32: Summary of Compliant and Non-Compliant Slope Sections

NO. OF SECTIONS	LAT	LAT/ SAP	SAP	ROAL	ROM	Totals	%
Compliant Pit Slope Sections	9	0	46	27	130	212	59
Marginal Slope Non-compliance (>+2 deg but <10 m)	38	4	9	16	3	70	24
Marginal Angle Non-compliance (<+2 deg)	0	0	11	3	1	15	
Non-Compliant Pit Slope (> +2 deg and >10 m height)	23	2	30	6	1	62	17
Totals	**70**	**6**	**96**	**52**	**135**	**359**	**100**
Total Height of Compliant Sections (m)							
Total Compliant Pit Slope Height	80	0	1221	402	3753	5457	71
Marginal Slope Non-compliance (>+2 deg but <10 m)	285	40	71	83	14	493	12
Marginal Angle Non-compliance (<+2 deg)	0	0	320	38	70	428	
Non-Compliant Pit Slope (> +2 deg and >10 m height)	384	28	788	99	25	1293	17
Total Height of Pit Slopes by Lithology(m)	**719**	**67**	**2400**	**623**	**3862**	**7671**	**100**

A total of 59% of slopes were compliant with recommended slope angles. Of the non¬compliant slopes, 24% of the slopes are marginal in that they are short inter-ramp slopes of less than 10m in height or within 2° of the recommended slope angle, and therefore, considered within the accuracy of the level of design. Accounting for total slope height for each lithology, the actual length of slopes that are considered compliant is 71% and those that are significantly non-compliant are 17%.

However, a comparison of the significance of the design slopes as compared to the recommendations suggests many compliant slopes are conservative as compared to the recommended slope angles, and many instances of non compliance would result in limited additional excavation. An estimate of these differences suggests the non compliant designs may be reducing the anticipated total pit excavation volumes by 2Mm^3 but, the conservativeness of slopes in compliant areas may be leading to an additional excavation volume of 12Mm^3.

In summary, the pit designs whilst including 17% of lithological slope non-compliances the designs are broadly compliant for the purposes of a mineral reserve estimate and planning.

1.4.4 Mineral Reserve Statement

The mineral reserves for the Lefa Gold Mine concessions are those mineralised materials, within practical engineered pits, using the marginal gold cut-off grades pertaining to the specific pits. Within each pit, the mineral resource category has been used as the guide to the classification of the mineral reserves.

Measured mineral resources were generally converted to proven mineral reserves, and indicated mineral resources converted to probable mineral reserves. The estimated mineral reserves, as at 1 November 2010, by category and ore type are presented in Table 1.33 below.

Table 1.33: Mineral Reserves by Classification (WAI, 01 November 2010)
(In accordance with the guidelines of the JORC Code (2004))

	Proven			*Probable*			*Proven & Probable*		
Deposit	**Tonnage (kt)**	**Grade (g/t Au)**	**Metal (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Metal (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Metal (koz)**
Lero-Karta	23,031	1.51	1,122	12,186	1.61	630	35,217	1.55	1,751
Fayalala	30,419	1.12	1,099	3,231	1.14	119	33,650	1.13	1,218
Kankarta	2,534	1.56	127	1,060	1.44	49	3,595	1.52	176
Firifirini	3,467	1.70	190	1,754	1.52	86	5,221	1.64	275
Banko	788	2.19	56	246	1.62	13	1,034	2.06	68
Folokadi	460	1.69	25	1,389	1.91	85	1,849	1.85	110
Toume Toume	189	1.48	9	411	1.40	19	600	1.43	28
Stockpiles	6,985	0.76	170	-	-	-	6,985	0.76	170
TOTAL	**67,873**	**1.28**	**2,798**	**20,277**	**1.53**	**1,001**	**88,151**	**1.34**	**3,796**

Any *Inferred* Mineral Resources within the pit have been excluded from the Mineral Reserve.

At a gold price of US$900/oz, the Lefa Gold Mine estimated mineral reserves are 88.2Mt, at average gold grade of 1.34g/t, for contained gold of 3.9Moz.

1.4.5 Mine Planning

The production schedule was generated in Microsoft Excel, based on a bench by bench mining approach for each pit. In order to minimise the waste strip and maximise the ore exposed, several criteria needed to be satisfied.

These include:

- Mining fleet flexibility for campaign mining;
- Identification of high grade areas; and
- Minimum number of working faces per period of time.

Applying a truck and excavator mining method for the operation satisfies these requirements. Table 1.34 presents the estimated Life of Mine (LoM) production schedule for the Lefa Gold Mine.

Table 1.34: Life of Mine Production Schedule from 1 November 2010														
	Nov-Dec 2010	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**TOTAL**
Waste Mined (Mt)	3.5	19.9	24.9	26.5	26.5	26.5	26.5	26.5	26.5	26.5	26.5	19.9	12.1	292.6
Ore Mined (Mt)	1.05	6.00	7.50	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	6.00	3.65	88.20
Au (g/t)	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34
Au (kg)	1,403	8022	10,040	10,710	10,710	10,710	10,710	10,710	10,710	10,710	10,710	8,032	4,889	118,063

The current production rate is to deliver ore to the plant at a rate of approximately 6Mtpa for 2010 and 2011 and then to ramp up to 8Mtpa by 2013. It is based on equipment hours and productivities, to ensure a fully utilised fleet is maintained. Only *Measured* and *Indicated* mineral resources were used in the classification of ore tonnes.

In general, the production schedule yields some 88.2Mt of ore, at an average gold grade of 1.34g/t, for the next 12.5 to 13 years, at a waste to ore strip ratio of 3.3:1. On average, about 310koz per annum are delivered to the CIL plant, for a LoM total of some 3.9Moz.

Rock excavating activities are conducted every month. Trucks are filled in 4 passes. Drilling and blasting is required to assist fragmentation and subsequent loading of most rock, waste and ore-bearing. On average, the mine will excavated some 25Mt waste per annum.

5.2Mt of ore, at average gold grade of 1.1g/t, are sent to the stockpiles from the pits. In total, 14.3Mt of ore, at average gold grade of 0.9g/t, are re-handled, including 2.3Mt of ore from the existing heap leach pad material.

Some 293Mt of waste rock from the excavation areas is hauled to external waste dumping areas. The practicality of backfilling any of the final pits was not investigated, even though this could minimise transport costs and visual impact on the local environment.

Some of the stockpiles are joined with waste dumps. However, the contact surface between the two rock categories is relatively small. During mining operations, stockpiled ore may be displaced, or "lost", and waste added to the ore. A small portion of the stockpiled ore is expected to be lost due to vertical and horizontal movement resulting from excavating activities. Therefore, a factor of 5% was assumed for the ore that will be lost to waste.

At the same time that ore is lost, the void left by this ore may be filled with waste material. Therefore, a mining dilution factor of 5% was added. That is, the stockpiled ore tonnes were increased by 5%, after ore losses, but with waste at zero grade.

1.4.6 Mine Operations

1.4.6.1 Mining Method

Lefa gold mine uses conventional surface mining methods with drilling and blasting (where required) followed by dig and haul allowing for selective mining of the ore grade material as shown in Photo 1.1. The current mining rate is 6Mt per year with an overall stripping ratio of approximately 4:1, although this varies from pit to pit. SMD is currently mining in the two main areas of the resource; namely Lero-Karta and Fayalala operating a fleet of Komatsu 785 haul trucks and Komatsu PC1800 excavators along with supplementary plant including dozers, graders and smaller front shovels and excavators.

The two major mining areas are the Fayalala and Lero Karta pits. These pits are approximately 8km apart and will be mined over the entire life of the project. Several smaller, higher grade satellite pits will provide supplementary feed for the mill over the life of the project. Some lateritic cap rock requires drilling and blasting, however the majority of the laterite and saprolite ore types are free digging. Dewatering is undertaken by way of bores and/or trenches and sump pumps. SMD has recently taken the owning and operating of the mining fleet as well as undertaking the mine planning, scheduling, grade control, surveying and quality control.

Photo 1.1: Surface Mining

1.4.6.2 Pit Design

The open pits are generally mined bench by bench with approximate heights of some 2.5m per flitch, leaving a ramp behind for access, with each bench being 5 to 6m. The pit design parameters are generally in keeping with established mining practice for excavator-truck operations. The pit haul ramps are designed with a width of 21m and a gradient of 10%. The bench height for drilling and blasting is 5 to 6m in both ore and waste with excavation of these benches done in two cuts or flitches. Long term plans envisage benches up to 10m in height.

1.4.6.3 Load and Haul

Ore from the Fayalala pit is hauled directly to the Run of Mine (ROM) stockpiles near the process plant by the primary haul fleet. Ore from the Lero-Karta pits is trucked to a ROM pad and primary crusher facility in the vicinity of the Lero-Karta pit entrance for subsequent crushing and conveying the 8km to the main process plant at Fayalala.

It was initially intended that the majority of ore trucked to either of the two ROM pads would be tipped directly into the ROM bin by the haul trucks. However, ore that is stockpiled on either ROM pad for operational reasons is fed into the respective crusher by a dedicated front end loader. This arrangement is currently being modified whereby each ROM pad will operate a ‘skyway’ system with ‘fingers’ of differing grade/quality ore being formed before being selectively loaded by front shovels into the primary crushers. This is believed to facilitate better ore type and grade control for delivery to the plant and to improve efficiency.

1.4.6.4 Grade Control

Grade control drilling takes place independently to the production drilling and is performed by a contractor. There are three ore types that are mined at Lefa mine with only two being treated. A cut-off grade of 0.6g/t Au has recently being applied. High grade ore, 1.2g/t Au and above, is flagged with green tape whilst low grade ore between 0.6g/t and 1.2g/t Au is flagged with blue tape are sent to the ROM pads and sent to the processing plant.

Mineralised waste which is between 0.4g/t and 0.6g/t Au is stockpiled with the view of processing this material at the end of the mine life. The other mining constraint is that a 60:40 ratio of weathered to non-weathered rock is required for the plant.

1.4.6.5 Waste Rock Disposal

Fresh and weathered waste rock materials are, and will continue to be, deposited in dedicated surface waste dumps adjacent to the pits. The waste dumps are typically constructed in 10m lifts with rehabilitated batter slopes of approximately 22. A 10m wide berm is left at every lift, to dissipate the velocity of water run-off from the dump. The required total waste dump capacity is presented in Table 1.35.

However, the Lefa Gold Mine has capacity to deposit waste rock in excess of 150 million cubic metres. As each waste dump reaches its final design limits, the faces will be progressively dozed down to the required angle, and then covered with suitable planting medium to allow the closure and land use plan to be implemented.

Table 1.35: Waste Dump Capacity

Pit	Mass (kt)	Compacted Density (t/m^3)	Volume ('000 m^3)
Lero-Karta			
Main	106,037	1.72	61,777
Pharmacie	3,431	1.58	2,177
Fayalala			
Main	48,167	1.74	27,658
Bofeko	2,829	1.76	1,610
Mid	387	1.62	239
Kankarta			
East	13,582	1.96	6,918
West	3,960	1.85	2,138
Firifirini			
Main	17,762	1.72	10,331
Banko			
Main	1,223	1.5	817
Folokadi			
Main	5,075	1.63	3,107
Toume Toume			
Main	1,491	1.76	847
Total	**203,944**	**1.73**	**117,619**

Backfilling of the pits, to reduce haulage costs, was not fully investigated due to the possibility of some of the pits expanding in all directions, and the long term sustainable development aim to use the pits as water reservoirs for agricultural projects.

1.4.6.6 Pit Dewatering

Pit dewatering is managed by using localised in-pit sumps in strategic locations, with diesel powered mobile pumps lifting the water to the surface, and connecting to a surface collection area with discharge to the receiving environment, if water quality parameters permit.

1.4.6.7 Mining Fleet

The mining operations are equipped with a modern fleet of equipment with the major production units comprising 5 Komatsu PC1800 excavators and 27 Komatsu 785 haul trucks.

Table 1.36 shows the main mining equipment engaged at Lefa mine.

Table 1.36: Summary of Mining Fleet

Item	Make	Model	No.
Excavator	Komatsu	PC1800	5
Excavator	Komatsu	PC300	2
FEL	Komatsu	WA800	2
FEL	Komatsu	WA600	2
Haul truck	Komatsu	HD785-5	27
Blast Hole drill	Tamrock	Pantera - 1500	5
Track Dozer	Komatsu	D275-5	4
Grader	Komatsu	GD825A-2	4
Water Cart	Komatsu	HD465-7wc	3
Wheel Dozer	Komatsu	WD600-3	1

WAI Comment: *WAI has observed the current mining operations and considers that it is achieving positive results. The numbers of equipment and personnel were reviewed were considered fit for purpose for the 6Mtpa production rate but fleet numbers will need to be boosted to achieve 8Mtpa.*

1.5 Processing

1.5.1 Introduction

The Lefa gold processing facility uses conventional crushing, grinding and Carbon-in-Pulp (CIP) technology to produce dore gold. The processing facility is located at Fayalala and is located at some 6.5km to the west of the Lero Karta pit. Ore is delivered to the processing facility by truck from the Fayalala pit and by overland conveyor from the Lero Karta pits.

The processing facility was developed following the results of the November 2004 Definitive Feasibility Study by Lycopodium, for an average annual throughput of some 6.0Mt. The metallurgical testwork for the main mineral deposits at Lero-Karta and Fayalala was carried out by Lycopodium in 2004. In 2008, Wardell Armstrong International Ltd. (WAI) undertook preliminary metallurgical testwork on ore sourced from the Firifirini mineral deposit, although this orebody is not currently mined. Both the Lycopodium and WAI studies showed that the ores were amenable to cyanide leach technologies with recoveries >89% being obtained.

The Lefa processing plant is second hand and was sourced from the Kelian Gold Mine (Kelian), Indonesia. The facility was commissioned in 2007 at the Lero Gold Mine, with some modifications to process the Lefa Corridor ores, as recommended in the February 2005 detailed diligence audit by Mineral Engineering Technical Services Pty. Ltd. (METS). Reportedly, the Kelian plant was examined module by module to determine its fitness in meeting the design criteria, and equipment selection requirements to meet the 6.0 Mtpa of ore throughput.

Since commissioning, the plant has suffered considerable downtime, mainly due to the low availability of the front end loaders, feed conveyors and the milling circuit. These issues are continuing to be remedied and as a consequence downtime related issues continue to improve. The downtime from January 2007 to June 2010 was 48.4%, however in the past six months (January to June 2010) the plant utilization is significantly improved to 76.8%.

Despite these disruptions, the Lefa ore responds favourably to cyanide leaching with an overall recovery of 90.5 % being obtained from January 2007 to June 2010.

1.5.2 Metallurgical Testwork

1.5.2.1 Introduction

A number of metallurgical test work programmes have been undertaken on the Lefa deposits, these are:

- Lycopodium Pty. Ltd. Final Report " Lefa Corridor Project - Definitive Feasibility Study Process Design Criteria", November 2004;
- Orway Mineral Consultants. Final Report " Lefa Corridor Project – Communition Circuit Evaluation", August 2004; and
- Wardell Armstrong International. Final Report "Preliminary Metallurgical tests on a sample of mineralisation taken from the Lefa Gold Mine, Guinea", December 2008.

The metallurgical testwork for the main mineral deposits at Lero-Karta and Fayalala was carried out by Lycopodium in 2004, with the additional study by Wardell Armstrong International Ltd. (WAI) in 2008, on the Firifirini mineral deposit.

1.5.2.2 Mineralogy

Both the Lero Karta and Fayalala ore bodies have significant amounts of oxide ore in addition to primary material. Ore types vary from soft oxides through to carbonate-altered primary ores to albitised and highly silicified rocks.

For Lero Karta, it was shown that gold was present as 30 micron particles in or on the boundary of pyrite. Similarly, the Fayalala ore sample also contained gold located at the boundary of pyrite, although 46% of the gold was coarser than 30 microns and therefore potentially amenable to separation by gravity.

1.5.2.3 Comminution

As part of the Definitive Feasibility Study, comminution testwork and computer modelling was undertaken by Orway Mineral Consultants in August 2004. Testing was undertaken on material sourced from the Lero Karta and Fayalala deposits. In summary it was found that:

- Handling of the oxides would be difficult due to their high moisture content (10 to 17%) and as a consequence of the monsoonal climate;
- No residence time or clay slip formation difficulties were expected when passing oxides through the SAG mill, and minimal energy requirements were forecast. It was suggested that a 55% solids by weight restriction on the oxide density be imposed in the SAG mill as a result of the viscous nature of the oxides. The absence of any rock media suggests that liner damage is possible unless blending is strictly controlled and a variable speed facility is available in the SAG mill; and
- The primary ores appeared to be highly shattered, giving rise to razor-sharp, slabby pieces when broken. Where competent enough to be tested, they were classified as having low strength, with low to moderate resistance to impact breakage. High work indices are ubiquitous, and generally similar to each other. Variability was shown to be very low except where dolerite dykes intrude or unaltered bedrock is encountered. These anomalies are expected to occur rarely.

Bond Ball and Rod Mill determinations undertaken by WAI in 2008 showed that the Firifirini Primary ore was relatively softer than the Lero Karta and Fayalala ore types.

1.5.2.4 Beneficiation

The metallurgical recovery characteristics, from all mineral deposits at Lefa Gold Mine are presented in Table 1.37.

Table 1.37: Summary of Metallurgical Response of Various Ore Types

Ore Lithology	Gravity Recovery %	Leach Recovery %	Total Recovery %
Lero Karta			
Laterite	23.6	72.7	96.2
Oxide	10.3	82.0	92.3
Primary	38.8	51.0	89.7
Fayalala			
Laterite	38.0	59.3	97.3
Oxide	33.6	60.6	94.2
Primary	45.1	44.5	89.6
Minor Deposits			
Pharmacie	44.8	47.2	92.0
Kankarta	36.4	56.2	92.6
Tambico	60.6	34.2	94.8
Banko	25.4	68.8	94.2
Firifirini			
Oxide	10.5	83.4	93.9
Primary	11.9	82.9	94.8

The metallurgical test programme showed that the different ore types (laterite, saprolite and primary) all respond favorably to a flowsheet incorporating gravity and cyanide leach technology.

Direct leach tests also demonstrated that the Lero Karta and Fayalala ore samples were free milling with and without gravity gold removal prior to cyanidation. Preliminarily testwork (by WAI in 2008) on material from Firifirini showed that it was amenable to cyanide leaching with recoveries up to 94.8% being obtained. The Firifirini ore body is reportedly scheduled for mining in 2012.

Orway Mineral Consultants are currently engaged in a plant optimisation study which will be completed in August 2010. As part of the study they are investigating the effect of oxygen addition on the leach kinetics. Orway Mineral Consultants will also be investigating the impact upon the comminution circuit as the proportion of primary ore increases. Following this study they will advise Lefa Mine on measures required to maintain production targets (i.e. if a pebble crusher is required).

1.5.3 Process Description

1.5.3.1 Introduction

The Lefa processing facility uses conventional technology to produce dore gold. Ore from Lero Karta and Fayalala ore bodies are separately crushed and conveyed to two SAG mills, each followed by a ball mill. Further milling is undertaken in a ball circuit to grind the material to 80% passing 150 microns. The milled product is then thickened and leached in a CIP circuit. Gold from the loaded carbon is stripped using the AARL process, electro-won and smelted to produce dore gold assaying typically 92% Au.

1.5.3.2 Stockpile and Lero Karta Conveyor

The product from either crusher can be directed to a common emergency stockpile which can be later fed to either SAG mill. The stockpile currently has capacity for about 24 hours. Stockpiling can only be performed when either of the SAG mills are stopped as the crusher operates in direct line with the mills. The emergency stockpile is shown in Photo 1.2.

Photo 1.2: Emergency Stockpile

At Lero-Karta, ore is crushed and transported to the Fayalala plant site via a 6.5km overland conveyor. The Lero Karta conveyor currently has a capacity of 500tph and is covered to keep the ore as dry as possible during the wet season. The conveyor has been plagued with numerous mechanical failures and has been a significant factor leading to lost production.

> ***WAI Comment****: A conveyor specialist is currently commissioned by SMD to investigate and report on the issues surrounding the Lero Karta conveyor system. Reportedly, the conveyor has a potential capacity of 750tph but currently it is not possible to re-start belt when it is loaded in excess of 500tph.*
>
> *The conveyor has been fitted with an inferior conveyor belt (as SMD suffered from cashflow issues during the early stages of the project). If the same conveyor arrangement is kept, WAI recommends that a quality conveyor belt, such as Goodyear, is installed.*

1.5.3.3 Crushing Plant

Each crushing module at Lero-Karta and Fayalala comprises a grizzly screen and jaw crusher arrangement, to generate a -120mm product, suitable for feeding to the SAG Mills. The crushing, conveying and stockpile area is closely linked to the milling operation. Typically, the crushers will operate on-line with the mills, with the emergency stockpile being used only during unsteady operations.

Once crushed, the material is fed onto a small transfer conveyor. The discharge end of the transfer conveyor can be moved (using a winch pulley system) above either one of two chutes. Each chute is positioned above the SAG Mill conveyor belt feeders. This enables personnel to direct ore types to selected SAG Mills. The transfer belts can be positioned above both chutes to enable a split of material on to both SAG Mill feed conveyors.

A photograph showing the transfer conveyor discharge end for the Lero Karta is shown in Photo 1.3.

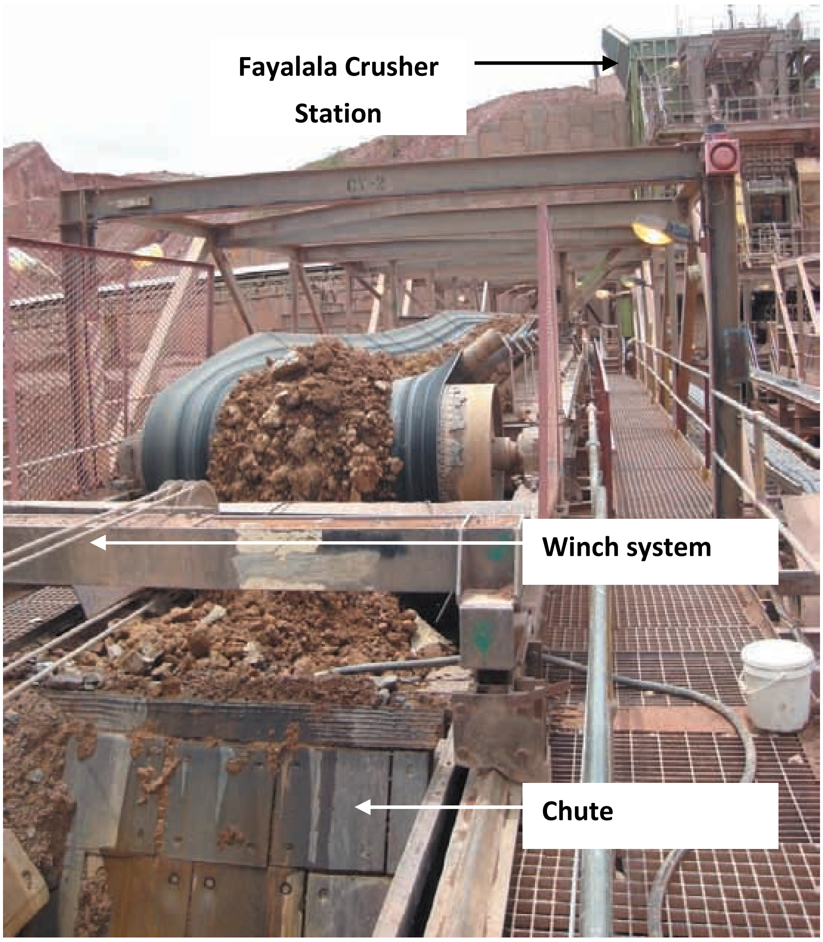


Photo 1.3: Transfer Conveyor Discharge End for the Lero Karta Ore

WAI Comment: *The area beneath the conveyor system and Fayalala had suffered from material overspill, SMD plan to install a concrete floor and sump pumps to enable material to be washed in to the circuit.*

Oversized material frequently enters the Fayalala crusher circuit (which unlike the Lero Karta crusher station is not fitted with a rock hammer) causing crusher stoppage. Improved control of top size entering the crushing circuit is required.

1.5.3.4 Grinding

The ore is ground to 80% passing 150 microns using a grinding circuit consisting of SAG and ball mills. The circuit has two SAG mills (8.4m diameter in size, with twin 1,750kW motors) which are fed by separate conveyors. The ground product from the SAG mills discharges into a common sump along with the discharge from the two ball mills (5.5m diameter in size, with 3,500kW motors).

Slurry from the common sump is pumped to hydrocyclone units and cyclone underflows are fed to either of the two ball mills for further grinding. Material which meets the target grind size is thickened in one of two 26m diameter thickeners with the aid of flucculent. The underflow of the thickener, which has a slurry density of about 50% solids by weight, is pumped to the CIP circuit.

The arrangement between mills and cyclone piping system allows for any one mill to be shut down, without losing all production and adjustment to the feed tonnage. In fact, milling can continue with a number of alternatives, such as 2, 3 or 4 mills, which gives the circuit a degree of flexibility.

The ratio of Transitional and fresh material fed through the plant will increase as the pits developed. Therefore, SMD have commissioned Orway Mineral Consultants to investigate the requirement for a pebble crushing circuit in the grinding section of the plant, to mitigate this potential risk.

1.5.3.5 CIP Plant

The cyanidation of the milled ore is carried out in a CIP circuit (planned to be converted to CIL), which comprises a total of twelve baffled and mechanically agitated tanks (six leach stages and six carbon adsorption stages). The tanks are some 16m in diameter and of varying height, from 20.7m for Tank 1, stepping down to 15.0m for Tank 12, providing a nominal residence time of 42 hours on design flows of 6.0Mtpa, and 35 hours at 7.2Mtpa.

Leaching is undertaken at 50% solids and the slurry is aerated with compressed air. Cyanide levels are monitored (using silver nitrate titration and Rhodanine indicator) in each tank. Cyanide is added to the first tank, while lime is added onto the SAG fed conveyors.

Activated carbon is added to the final tank and moves counter currently again the flow of the slurry. The circuit was originally designed for the removal of carbon from Tank 7 so that the first six tanks were free of carbon (CIP mode). However, carbon is being pumped from Tank 8 and placed into Tank 4. The carbon then travels down with the slurry where it is removed from tank number seven. This configuration makes the circuit more CIP than CIL. This configuration was adopted to improve carbon loadings.

The loaded carbon is treated in an AARL type elution circuit which has a carbon holding capacity of 10-12t. Acid washing prior to elution is not undertaken on every batch as fouling of the carbon is not reportedly problematic. Similarly, the carbon is not regenerated every time following elution as this has been deemed unnecessary.

The gold rich solution from the elution circuit is processed by electrowinning to produce cathode gold. The original eight electrowinning cells are being replaced by two new cells which are more efficient and give better security and health and safety (ventilation) aspects. The electrowinning circuit replacement programme is scheduled for completion in Q3 2010. The steel wool cathodes are washed and smelted to produce dore gold assaying about 90-92% Au.

Detoxification of the CIP tails is not performed as this is not required by law in Guinea.

The CIP tanks are shown in Photo 1.4.

Photo 1.4: CIP Tanks

Photo 1.5: Thickened Feed Entering First CIP Tank

1.5.4 Assay Laboratory

The Assay Laboratory is operated independently by SGS and works a 24 hour, 7 day week service. The laboratory only undertakes gold analysis.

There are 40 workers employed in the Assay Laboratory who prepare and analyse both metallurgical and geological samples. The metallurgical samples are assayed by acid digest and atomic adsorption analysis (units shown in Photo 1.6). Geological samples are analysed using fire assay.

The metallurgical solid and solutions samples are reported to the accuracy of 0.01ppm Au and 0.001ppm Au respectively. For metallurgical and geological samples 50 gram batches are used for analysis. For carbon samples one gram trials are taken. All analysis is undertaken in duplicate.

The laboratory uses standard reference samples for routine checking and calibration purposes. Samples are not routinely sent to other laboratories for checking purposes. During WAI's visit, the laboratory were preparing to send local ore samples of various grades (0.2-1g/t Au, 1-3g/t Au and >3g/t Au) to another SGS laboratory in order to standardize them so they could be returned and consequently used as reference material.

Photo 1.6: Atomic Spectrometry Units

WAI Comment: *SMD have questioned the accuracy of analysis undertaken on some their metallurgical samples. This has forced SMD to submit duplicate samples for checking purposes. WAI recommends that an external consultant is used to provide a detailed audit of the laboratory.*

1.5.5 Technical Control and Plant Sampling

The following samples are taken to determine the daily metallurgical efficiency of the plant:

- SAG feed (via feed belt conveyor): A grab sample is taken every 1 - 2hours over a 24 hour period to make a daily Composite sample. The sample is crushed and split and the moisture content is determined. The sample is also assayed for gold which is used for control purposes rather than being used for metallurgical accounting purposes; and
- Cyclone overflow and CIP tails: A sample of the cyclone overflow and CIP tails is manually taken every hour over a period of 24 hours to produce daily Composites. The samples are assayed for gold.

A weightometer fitted to the SAG mill feed conveyors is used for metallurgical accounting. Reportedly, the weightometers are calibrated on a regular basis.

On a monthly basis a detailed gold inventory is made and includes sampling and analysing all solids, solutions and carbon products and the amount of dore poured.

To maintain efficient stripping, the carbon entering and leaving the elution stage are sampled and assayed daily. The eluant and electrolysis solution samples are sampled every two hours to ensure that the gold remaining in the barren electrolysis solution is <10g/t Au.

> ***WAI Comment:*** *The sampling methods, appear to be adequate, but room for improvement is evident. SMD realise that further improvements in metallurgical accounting are required and plan to introduce automatic sampling devices so that Composite samples are more representative. SMD also plan to install a flowmeter on the CIP feed stream (once the plant feed is more stable) so that the tonnage to the CIP circuit be more accurately determined (considering the lag time between material entering the plant and leaving the thickener circuit).*

1.5.6 Process Water

Water is recovered from the CIP tailings dam and stored in two ponds having a combined capacity of 15,000m3. The process water was reported to be of suitable quality for the processing plant.

The tailings dam is used to collect rain during the wet season so that the plant can operate during the dry months. SMD have commissioned a consulting hydrologist to investigate the water storage capacity of the tailings area.

1.5.7 Tailings Facility

The tailings facility does not have sufficient capacity for storage of all the material that is expected to be generated in the current life-of-mine plan. SMD are considering alternative solutions to the problem and have indicated paste deposition methods as one possibility.

> ***WAI Comment:*** *The issue to tailings capacity needs to be addressed in the near future.*

1.5.8 Reagent Storage

Reagents are stored either outside or within shipping containers. SMD plan to build a reagent storage shed.

1.5.9 Metallurgical Performance

The metallurgical performance of the plant for 2007-2010 is summarised in Table 1.38.

Table 1.38: Lefa Plant Metallurgical Performance

Year	Head Grade (g/t Au)	Tonnage (Mt)	Mill Utilisation* (%)	Plant Recovery (%)
H1 2010	1.27	2.898	76.8	88.8
2009	1.39	4.422	54.2	90.2
2008	2.15	3.085	41.7	92.5
2007	1.37	2.482	35.1	89.0

* Numerical average of months operated in that year

Since the commissioning of the plant in January 2007 the overall recovery of gold has been 90.5%. Up until the end of June 2010 the Lefa processing facility has produced 17,784kg of gold while in 2009, 494.3kg of gold was produced.

Low equipment availability, a high proportion of unplanned maintenance and non-compliance with preventive maintenance programmes have historically been common problems in the maintenance of mining equipment and plant. It is very apparent at Lefa that this subject has previously been a significant issue that has resulted in the poor production record since 2006. Following repair to some parts of the comminution circuit plant availability in the past nine months has significantly improved. SMD are confident that the target of 6Mt of ore will be processed in 2010.

SMD have focused their attention in repairing and fixing problems associated with the construction of the grinding circuit. These issues are planned to be remedied by 2011. SMD have also implemented improved downtime recording and analysis. They also have systems in place to monitor key areas of the milling circuit so that necessary maintenance can be undertaken to prevent unexpected part failure.

1.5.10 Labour

The number of employees in the various sections of the plant are given in Table 1.39.

Table 1.39: Lefa Plant Labour		
Area	No. Employees	
	Expat	National
Management	1	-
Crushing and Milling	3	60
CIP Plant & Tailing		12
Metallurgy Gold Recovery	2	24
Plant General		43
Training	3	2
Maintenance	19	262
Total	**28**	**403**

There are a total of 28 expat and 403 national employees. Expat workers undertake a 6 week on and 4 week roster. The plant runs on a three shift per day system.

Typical monthly salaries for local workers are US$610 with a further US$550 given for allowances and medical benefits.

1.5.11 Processing Cost

A summary of the processing is shown in Table 1.40.

Table 1.40: Cost of Processing from May 2009 to April 2010						
Date	Cost item, US$/t					
	Consumables	Labour	Power	Maintenance	Other overheads	Total
Jun-09	2.78	0.91	6.54	4.00	1.28	15.51
Jul-09	3.20	0.35	4.25	3.11	2.70	13.61
Aug-09	2.59	0.41	7.78	3.25	1.80	15.83
Sep-09	3.64	0.41	8.96	3.71	1.79	18.50
Oct-09	2.50	0.33	6.12	2.41	1.12	12.47
Nov-09	2.78	0.31	6.50	2.30	1.69	13.58
Dec-09	2.56	0.52	6.96	2.88	3.06	15.98
Jan-10	2.68	0.35	5.87	3.07	0.60	12.56
Feb-10	2.76	0.30	6.43	2.33	0.89	12.70
Mar-10	2.41	0.24	5.11	2.06	0.75	10.58
Apr-10	2.34	0.26	6.09	2.75	1.02	12.47
May-10	2.58	0.43	6.69	3.73	4.93	18.35

Power costs have been determined on the estimate that 95% of total power generated on site is used in processing. The cost of consumables includes the costs of transporting the material to site.

The processing cost between June 2009 and May 2010 (at a throughput of 5.18Mt) was US$13.96/t. The cost does not include salaries for expatriate workers (WAI estimates this would be approximately US$0.5/t, making the total processing cost US$14.46/t).

Cyanide and lime consumption during this twelve month period was 0.33kg/t and 1.79kg/t respectively.

1.5.12 Gold Sales

The Doré gold typically contains 90-92% Au with the remaining predominately consisting of silver.

The dore is flown to Conakry national airport ready for air freight to MKS, Switzerland. SMD arrange the security, freight logistics and transportation costs. Prior to the dore disembarking Conakry the gold is weighed for royalty (5%) and taxation (0.4%) purposes. Thirty minutes after departure of the flight from Conakry the responsibility of the shipment transfers to MKS. A day after receiving the dore, MKS make an advance payment of between 80-85% of the predicted value of the gold (based on previous days spot price). The remaining balance is made following the refining stage.

1.5.13 Heap Leach Processing

From its inception until 31st December 2006, Lefa gold mine utilised a heap leach processing operation. This involved the treatment of the finer grained and more clay-rich ‘high-fines’ laterite and saprolite (oxide) ore types. Due to the clayey nature of the material the operation was not particularly efficient as a heap leach pad needs effective permeability to enable the fluids to percolate freely through the material to dissolve the gold. A geotextile matting was placed at the base of each of the ‘lifts’ to improve fluid flow but it is believed that the overall efficiency remained poor (<60% recovery). Consequently an opportunity exists to re-process this material and extract the remaining gold.

SMD consider these leach pads a low priority, with which WAI concurs, but has expressed an interest in undertaking further investigation to determine the available resource and potential to reprocess.

1.5.14 Future Development

SMD have budgeted a capital expenditure for the entire site of $75 million over the next three years. SMD plan to complete the following main areas of work in the processing facility:

- For the comminution circuit:
  - SAG Mill 1 will be fitted with a new trunions in 2010 Q4;
  - New lubrication systems are ordered for both SAG mills and will be fitted either 2010 Q4 or 2011 Q1;
  - SAG Mill 2 to be fitted with girth and pinion gear; (SMD are monitoring the unit and when necessary they will change-this is likely to be in 2011);
  - Slurry is contaminating the SAG Mill liberation systems, and new seals arrangements are required for both mills. These will be fitted in 2010 Q4:
  - Ball Mill 2 requires new pinion and girth gears (two contractors are currently bidding on this work); and
  - Ball Mill 1 will require a replacement of the gear box (the unit is being monitored and will be changed when necessary).
- For the remaining processing facility:
  - Security perimeter fences around processing facility;
  - Concrete flooring and installation of sump pumps surrounding the Fayalala crusher area and conveyors so fallen material can be washed into circuit (particularly problematic in wet season);

- Gold room modifications;
- New tailings pumps (increase capacity);
- New gland water system;
- Carbon regeneration kiln repair/replacement;
- New control room; and
- New reagent storage shed.

SMD are investigating options available for increasing the reliability and throughput of the Lero Karta overland conveyor. Further capital will be required in order to expand the life of the tailing storage facility.

1.5.15 Conclusions

The Lefa gold mine has undoubtedly suffered mechanical and reliability problems since the second hand CIP plant was commissioned, consequently, the mine has been unable to achieve production and predicted gold output. Production in this initial period also suffered from it being the rainy season.

Significant improvements have been made in the past six months to the comminution circuit and overland conveyor which have reduced operational downtime. Repair work to the comminution circuit is scheduled for completion by 2011. Systems are now in place where equipment is routinely monitored so that maintenance schedules can be better managed and unexpected downtime limited. SMD are confident that the throughput of the processing facility can be increased to some 8Mtpa, although this is subject to the findings of Orway Mineral Consultants, study which is due for completion in August 2010.

SMD personnel have been undertaking emergency repairs and have clearly operated in a “fire fighting” role. Consequently, metallurgical issues have not previously been a high priority, although SMD now continue to increase their attention in this area. Regardless, the metallurgical efficiency of the CIP circuit has performed well with overall plant recoveries of 90.5% have been obtained since the plant was commissioned.
SMD are currently undertaking a hydrological study. The ability to increase plant throughput will be subject to local water sources.

The tailings storage facility does not have sufficient capacity for storage of material which will be generated from the life of mine. SMD need to implement alternative storage solutions in the near future.

> ***WAI Comment:*** *despite the issues surrounding mechanical problems associated with the second hand plant, WAI believes that the Lefa mine has potential and is confident that these issues will be resolved.*

1.6 Infrastructure

1.6.1 Administration

SMD has adequate administration facilities. There appears to be good administrative procedures in place and the camps are well run. The company currently has approximately 1,863 employees on site, 75 of whom are expatriates.

Staffing has been a major issue in the past. The Lefa area is fairly isolated with an oppressive climate and expatriate employees were recruited onto a project that had initially been poorly designed. This alone led to an extremely high turnover of expatriate staff, with all the management issues that naturally ensued. Furthermore, it is understood that the salary and related benefits offered by SMD were sub-standard for the global mining industry and accentuated the turnover.

Recently Crew Gold Corporation has essentially recruited a new team of managers and supervisors. The pay scale is now more competitive and the tour of duty on site for non-accompanied staff has been reduced to 6 weeks on and 4 weeks off. This has allowed recruitment of more experienced personnel who appear well motivated. Staff can now see that their individual efforts will ultimately results in an efficient mining operation.

SMD has already recruited many well qualified, enthusiastic national employees. However, there is need to provide additional training and on-going evaluation. A new training centre is under construction and an incentive programme is being developed to run alongside employee evaluation. SMD believes that the number of expatriate staff will decrease and that there will be a progressive trend to nationalise many posts currently held by expatriates.

1.6.2 Mine Facilities

The Lefa project has a well developed infrastructure with direct access to its own airstrip as well as road access to Conakry, where an office is maintained. There are two well appointed camps with comfortable accommodation, messing and leisure facilities. All sites are connected by well maintained roads. There are both land and satellite phone connections and Orange has installed one mobile mast with another planned to improve coverage. The site has internet access and this is currently being upgraded. On site communication is by radio, although it was noted this has yet to be extended to all operating equipment in the pits.

1.6.3 Power Plant

Crew Gold Corporation purchased the Kelian power plant as part of the process plant package. The power plant consisted of 6 x 4MW Wartsila diesel generators and SMD has purchased two similar units giving a nominal installed capacity of 32MW. The total power demand of the site was not ascertained although it was stated the four mills will require 13MW when all are running. At times sufficient power has only been available to operate two mills. The power station construction appears complete and office and maintenance facilities are due for completion shortly.

There are two diesel storage tanks located behind the power station. They have a combined capacity for 16 days operation, a factor that has restricted production when diesel supply was short due to the wet season or when industrial action has been taken in Conakry. To minimise this problem and reduce operating costs the generators are being modified to operate on either Heavy Fuel Oil (HFO) or diesel. To this effect two new HFO storage tanks have been erected that will give sufficient storage capacity for the power station to operate for up to 3 months. The diesel tanks will then have sufficient capacity to allow the mine to operate for up to 9 months. Construction of an HFO filtration and pre-heating plant should be completed in August 2008.

1.7 Maintenance

Although a mine relies on the geological resource, the mining equipment and the process plant to commence operations, it is the ability to maintain that equipment that permits a mine to operate effectively, and at a profit. The maintenance department is the key service activity in the demanding environment of a mine. There are two ways for the department to operate; “fire fighting” and “planned” maintenance. For various reasons SMD has, until now, suffered from too much of the former and a deficiency of the latter.

Low equipment availability, a high proportion of unplanned maintenance and non-compliance with preventive maintenance programmes are common problems in the maintenance of mining equipment and plant. It is very apparent at Lefa that this subject has, and remains, a significant issue that has augmented the poor production record since 2006.

The Maintenance Manager is an experienced and well organised manager. He is responsible for maintenance of the process plant, the power station and electrical reticulation, all mobile equipment (except in the pits), the water reticulation, the camp and other general surface maintenance. SMD is now the responsible for maintenance of the mining equipment and a substantial new workshop and stores building has been provided.

It has been highlighted that a significant improvement can and will be achieved through initiatives designed to reduce the frequency of unplanned failures and the times necessary to repair them. Two initiatives of particular relevance are the elimination of breakdowns through root-cause failure analysis (RCFA) and the development of repair standards that detail the activities, spares, materials, tools and time needed to effect common unplanned repairs.

Allied to maintenance efficiency is the collaboration of a well managed and suitably stocked stores facility. The previous arrangement, numerous shipping containers with no stock accounting or records was wholly unsatisfactory for any operation, particularly one of the scale of Lefa.

> ***WAI Comment:*** *WAI believes the maintenance and supply issues are gradually being resolved and the disruption to production due to the fleet availability and parts has decreased.*

1.8 Environmental and Social

1.8.1 Previous Studies

The following reports were made available to WAI; however a full appraisal of these documents was outside of the scope of this report:

- Baseline Study: commissioned by Societe Miniere de Dinguiraye (SMD) and undertaken by AGEIL (2004); and
- Environmental Impact Assessment (EIA): commissioned by Societe Miniere de Dinguiraye (SMD) and undertaken by Knight Piésold Pty Limited (2004).

On brief appraisal of the Baseline Study, and EIA, the EIA largely fills in many of the gaps left by the baseline study, however no community consultation was taken as part of the EIA, although social data collection and impact assessment was included in the EIA.

The EIA contains an Environmental Management Plan (EMP), including aspects of social management, an emergency plan and restoration plan, further discussed in Section 1.8.3. Given the identification of potential shortcomings in the EIA, there may be areas in these plans which do not effectively mitigate the potential impacts to the environmental and social baseline conditions.

> ***WAI Comment:*** *The EIA has been prepared in compliance with Guinean legislation, but further works would be required to achieve international compliance. The EIA does not comply with International Standards. It is recommended that a gap analysis should be undertaken to identify gaps where further works to achieve international compliance would be required. Should new facilities, changes in project parameters and scope be implemented, these areas could be subsequently identified and included in an updated Environmental and Social Impact Assessment (ESIA), or ESIA addendum. The supporting plans should also be part of this review process. The company recognises that works will be required to achieve international compliance, however it should be noted that the Guinean government has very little regulatory framework developed and this state of affairs has not associated in due cognisance being paid to these standards.*

1.8.2 Environmental and Social Law, Permitting and International Standards

The Guinean legislation under which the project operates is the Environmental Code (1987). In summary, the Code requires the project to mitigate impacts to land, air, water, flora and fauna, and human environmental resources. There is no known law for the management of communities or heritage; however a community fund for the provision of local facilities is a statutory requirement.

The mineral rights of the project (assumed for both the mining and exploration areas) are owned by Lefa; however the surface area of the project is owned by the Guinean government, the case for all land in the country.

The project operates under two permits: a mining licence and an exploration licence. It was reported that the mining licence covers a surface area of approximately 1,500km^2, which includes all facilities associated with the project and their operation. The exploration licence includes a further area to the east of the project adjoining to the mining licence's eastern boundary. The licences are due to expire in approximately 20 to 25 years time (approximately 2030); however the date of expiry and the ability to reapply for expiring licences will be subject to the regulatory situation at that time.

Further to the project itself, three villages are also located within the mining licence area. This is further discussed in Site Access and Security section.

In order to maintain its permitting agreement, Lefa is subject to government inspections. The inspections are held on an approximate quarterly basis; undertaken without prior appointment, and fines for potential non compliances. The nature of a breach is not clearly defined, since the Environmental Code and permits do not prescribe discharge and emission standards or landscaping requirements etc.

The project is not within an environmentally protected area, as designated by Guinean legislation.

> ***WAI Comment:*** *WAI considers the project to have been developed in line with Guinean legislation, however the company recognises that further works would be required to ensure compliance with international standards, as described in the IFC Performance Standard, the Equator Principles and the World Bank Environment, Health and Safety Guidelines.*

1.8.3 Social and Environmental Management Systems and Planning

1.8.3.1 Social and Environmental Plans Held

As outlined in Section 1.8.1, the following management plans have been designed for Lefa as part of the commissioned EIA:

- Environmental (and Social) Management Plan;
- Accident Prevention and Emergency Response (Health and Safety) Plan; and
- Closure Plan.

> ***WAI Comment:*** *The adequacy of the plans in accordance with International Standards has not been substantiated, although they have been developed in line with national requirements. . A review and gap analysis with proposals for further works is recommended in accordance with the current and proposed operations. It is likely that other management plans will also be required in to satisfy international standards, including a framework Environmental, Health, Safety and Community (EHSC) Action Plan to identify key objectives, policies and procedures for management planning.*

1.8.3.2 Implementation of Social and Environmental Management

Lefa has an Environmental, Health and Safety (HSE) Department consisting of an HSE Manager and Officer. Social issues are managed separately by a Community Liaison Officer (CLO) and the Head Mining Engineer. Closure planning falls under the responsibility of the HSE department.

As discussed further below, aspects of the plans are undertaken in line with national requirements, however these have not been formalised via the development of a Social and Environmental Management System (SEMS) and Plan (SEMP). WAI recognises that this is due in part to a legacy of limited environmental and social management by previous owners, and a degree of inexperience of the personnel with regard to international requirements. This has resulted in a fire fighting approach to deal with the most urgent, or easily remedied issues.

***WAI Comment:** Subsequent to undertaking the proposed gap analysis and revision of management plans, it is recommended that the company seeks assistance and support to implement and train personnel in SEMS and SEMP. Lefa has developed and implemented several EHS programmes and procedures which could be included in an SEMS and SEMP, however this has not yet occurred.*

1.8.4 Review of Project Aspects and Impacts

1.8.4.1 Storage Facilities

Explosives are stored in a built facility within a fenced reserve. The reserve is guarded 24 hours a day by a private, military guard. Cyanide for the processing plant is stored in a storage tower. The tower is situated in a concrete bund. Two old cyanide towers are awaiting removal. A specialist contractor is being sought to undertake the work and disposal. Heavy oil for the power station is stored in fuel towers. The towers are situated in concrete bunds. Pipelines run from the towers to intermediary tanks which are raised on stilts, beneath which are concrete bunds. Diesel is stored in above ground tanks located in a concrete bunded compound.

Processing reagents are stored in barrels and in concrete bunded above ground tanks. Lime used for neutralising acidic processing effluents is kept in sacks in an open facility. The storage of lime in particular is poor, with open, damaged sacks left outside, with the lime contents becoming dissolved in surface water runoff during heavy rainfall. Acid barrels storage was not adequate, given that no designated areas or mitigation measure were in place.

***WAI Comment:** WAI would recommend that a hazardous substance handling and storage plan should be developed and consideration should be given to undertaking cyanide management in line with the International Cyanide Management Code (ICMC).*

1.8.4.2 Waste Management, Recycling and Housekeeping

The Tailings Storage Facility (TSF) is described in Tailings Storage Facility section. It was reported that improving general housekeeping at Lefa has been a priority during 2009 and 2010. During this time, many yard and road areas have been subject to waste and litter removal campaigns.

There are numerous waste rock dumps located around the project. The dumps are generally not contained and do not have designated drainage regimes. The dumps contain a mix of oxide and sulphide ore-types. Stability inspections of the dumps are undertaken in areas of Lefa which are in current use. The majority of the dumps are yet to be remediated; however Lefa have begun working on the restoration of the waste rock dumps; this is further discussed in Waste Rock Dumps and Stockpiles (ROM) section.

The domestic landfill is located within a crescent of waste rock dumps. The waste is brought to this location and laid directly onto the ground surface and is not buried. The waste consists of mainly plastics and cardboard. Lefa collects waste from Lero village as a good will gesture. Reprofiling of the landfill occurs on a daily basis using mine machinery. The landfill is not lined and no water quality/leachate monitoring is undertaken. The landfill is subject to continuous trespass by the local community with the purpose of searching for useful items. It was reported that fires are regularly lit by trespassers in order to clear cardboard and plastics to assist with their searching. Lefa does not attempt to stop this activity as this is considered to be futile.

All sewage arising from the project is directed into cess pits, which are regularly pumped out. The sewage is removed offsite by a designated contractor. The cess pits are not functioning adequately; an engineered solution is currently being sought by an external contractor to rectify this.

All packaging arising from the processing plant is burned. Reagents barrels are washed and given to the local community. A proportion of the barrels are used by the community for the storage of water. The methods and reliability of cleaning are not controlled and there are no checks of the hygiene of the containers before they are passed on.

Empty metal oil drums and scrap metal are stored in a scrap yard. A proportion of the drums are collected by their supplier, Total; however Lefa is bound by law to enable the government to reclaim all scrap metals. As such government employees occasionally visit Lefa to remove these materials. It was noted that this is not undertaken as regularly as Lefa would wish and a large amount of scrap metal was evident during the site visit.

It was reported that organic material such as food waste are composted and used for the remediation of waste rock dumps. Wooden pallets are reused; however it is likely that many may be removed by the local community for use as firewood. All electrical waste is burned. Waste oil is either burned (for energy production) or is collected by its supplier, Total.

> ***WAI Comment:*** *WAI would recommend that waste management and recycling should be undertaken in a more structured manner, and that this should form the subject of a waste management and recycling plan. The plan should include all areas above, and should address current landfill management, since there are currently not considered to be appropriate pollution prevention and control measures in place to protect human health and surface and groundwater resources. An alternative use for reagent barrels should be sought.*

1.8.4.3 Water Quality

The hydrology/hydrogeology of the project is not fully understood at present; however a consultant Hydrogeologist has been contracted to ascertain the baseline conditions. The project is located within the watersheds of two rivers: the Karta River, which runs/is diverted through the Karta (western) area of the project and the Salabe River which runs through the Fayalala (eastern) area of the project. The rivers run in an approximate north to south direction and confluence to the south of the project.

The project is located over two aquifers: an unconfined aquifer located within the upper laterite strata and a confined (‘fresh rock’) aquifer approximately 40m below ground level.

As noted in Section 1.1.2 the project is subject to an annual wet season between June and December, causing large fluctuations of surface water flow volumes and infiltration.

Significant interaction between the unconfined aquifer and surface water is evident in the open pits. The presence of water within the open pits is as a result of both direct precipitation and groundwater ingress from the unconfined aquifer during the wet season.

Water quality standards have not been developed in Guinean law. No water discharge permits are held by the project. The project utilises International Standards in the assessment of water quality. Non-compliances are reported to the manager of the determined source facility; however no further action is taken. Water quality samples are analysed in the on site laboratory. It was reported that QA/QC procedures are followed as part of the assessment process.

The following subsections outline areas of potential impact on water quality.

Heap Leach

The old heap leach is not lined and has potential for acid drainage and cyanide contamination of surface water runoff. This area is serviced by a trench which diverts runoff into a settlement pond; however not all runoff is captured by this and as such a proportion continues into the river catchment. It is not clear what happens to the water after settlement. Water and soils monitoring is reported within the vicinity of the heap leach on a quarterly basis. The diversion trench is periodically inspected for blockages.

Waste Rock Dumps and Stockpiles (ROM)

A number of waste rock dumps in the Fayalala area of the project have runoff diversions into an unused open pit. The water collected (along with any groundwater ingress) is subsequently used for dust suppression measures on the haulage roads. With the exception of this location, all other surface runoff within the rock dumps and stockpiles is unmitigated and unmonitored. The ore type (sulphide/oxide) of the rock dumps is mixed.

Landfill and Maintenance Yard

No water quality mitigation or monitoring is undertaken within the facility of the landfill or maintenance yard.

Tailings Storage Facility (TSF)

The TSF has undergone redesign in order to secure increased water supplies for the processing plant during the dry season. The TSF is divided into northern and southern areas, separated by a central waste rock dam with its entirety encapsulated within further northern and southern dams also constructed from waste rock material. It is believed that this area was formerly a river valley.

Effluent from the processing plant is pumped directly into the northern area of the TSF for settlement. The effluent passively passes from the northern area into the southern area through the central dam. The effluent in the southern area is then pumped back to the processing plant for reuse. A spillway is located on the southern dam passively discharges effluent towards the south, where it subsequently joins the southern extent of the Salabe River. The TSF is not lined as such it is likely that the TSF interacts with the groundwater associated with the unconfined aquifer.

It was reported that dams of the TSF are currently being raised following recent studies which have indicated that the capacity of the dam is not sufficient to withstand a once in 100 year rainfall event.

Processing Plant and Power Station Yard

An emergency discharge point is located down gradient of the processing plant for the diversion and capture of accidental discharges from the processing plant. On capture of the effluent, this is directly pumped back to the processing plant. It was reported that this had been used on more than one occasion and had been proven very successful.

Water quality testing for hydrocarbon contamination is undertaken within the vicinity of the power station on a quarterly basis. No waste water management is implemented on the yard surface area.

Open Pits

In order to mitigate surface water ingress of the Karta Pit, the Karta River is diverted around its western perimeter. Nevertheless the pit continues to be subject to precipitation capture and groundwater ingress from the unconfined aquifer; therefore a water extraction pump is also operated in the pit. The pump directly releases the water into the Karta River without mitigation or monitoring. All pits are subject to precipitation capture and groundwater ingress.

Rivers

Water quality monitoring of the Karta and Salabe rivers is undertaken on a quarterly basis. For both rivers, samples are taken upstream, within and downstream of the project. It was discussed that elevated cyanide concentrations have been identified in downstream locations. In the instance of identified contamination, the relevant department is informed. The sampling locations are set out in the EIA.

Village Boreholes

The groundwater from the unconfined aquifer is used for well water supplies in the villages. The wells were dug by Lefa. Basic water quality monitoring is undertaken in all wells on a monthly basis, with a detailed monitoring suite undertaken quarterly. It is not know what contaminants are included in each suite. It was reported that borehole contamination has been identified however the locations and severity of the borehole contamination is not known. It is understood that no action has been undertaken to mitigate the borehole contamination; it is not clear whether the contamination is resultant from poor sanitation, or the project.

> ***WAI Comment:*** *The infrastructure areas above should be incorporated in a site-wide Water Management Plan. WAI considers that there are a number of potential contaminants which may be affecting surface and groundwater supplies, including: Acid Drainage (AD), hydrocarbons, cyanide, turbidity and processing reagents. The water management plan should address current monitoring, and ensure that appropriate water monitoring locations and determinands are selected to reflect potential contamination issues. This should include tests to assess acid generating, and leaching potential, and would be included in part of an overarching SEMS.*

1.8.4.4 Air Quality

No air quality monitoring iscurrently undertaken at any location or facility, since there are no known Guinean air quality standards to be adhered to. The HSE Officer recognises that air quality monitoring should be undertaken as part of an SEMP; it was suggested that the HSE department proposed monitoring in the future. A number of mitigation measures are in place and include dust suppression measures of haulage roads during the dry season (December to May) and equipment and machinery maintenance in order to reduce particulate and gaseous (CO_2, SO_x and NO_x) pollution.

> ***WAI Comment:*** *Whilst there are no formal national requirements for air quality monitoring, the company recognises the need for this to be addressed. WAI would suggest this should be the focus of an air quality and dust management plan, as part of an SEMS.*

1.8.4.5 Noise and Vibration

Noise monitoring is not undertaken at the project; however vibration monitoring of quarry blasting is undertaken as result of several complaints regarding the cracking of building walls in the local villages following blasts. Lefa implemented a programme of monitoring in 2009 as well as introducing vibration mitigation consisting the reduction of charged holes coupled with increasing the rate of charge on detonation. In order to further improve public relations with the local community, the project does not undertake blasting during the night and prior notice is given to the local communities up to two days before each blast.

The results of the vibration monitoring are compared to both Guinean and International limits for blasting; since the implementation of the monitoring programme, no blasts have been recorded to exceed these limits.

> ***WAI Comment:*** *A noise and vibration monitoring plan should be developed in line with national and international requirements.*

1.8.4.6 Flora and Fauna

The management of flora and fauna is not actively undertaken, with the exception of the remediation of the waste rock dumps, as discussed in Section 1.8.4.2.

> ***WAI Comment:*** *Where sensitive species have been identified, monitoring and management should be performed to manage and minimise negative impacts. If protected species are present at the site, they should form the subject of a separate management plan.*

1.8.4.7 Land Quality

Soils analyses and monitoring are undertaken in a number of locations on a quarterly basis, with results reported to the relevant project staff when elevated concentrations are identified. No further action is undertaken. The general house-keeping of the site is considered fair, with the exception of some of the yard areas. To date, much of the area of the project is awaiting remediation.

> ***WAI Comment:*** *It would be recommended that where soil contamination has been identified, this should form the subject of a remediation plan. Similarly, where possible, progressive site rehabilitation should be undertaken as a priority, to limit environmental disturbance, and reduce final rehabilitation costs.*

1.8.4.8 Socio-Economic Setting

Guinea is currently in the process of conducting elections. It was reported by several personnel at Lefa that environmental and social legislative requirements of the project may change following the election of government.

Three villages are located within the mining licence of Lefa. Other villages are also present within the local area. The local population is from a background of subsistence farming, supported by low-level artisanal mining. Most of the population have a Muslim religious background and speak French amongst other Guinean dialects. It is perceived that the local population are generally in support of Lefa as they see the project as an opportunity for employment and to raise the standard of living within the villages. A community plan, differing from proposals outlined in the EIA was discussed; however the plan was not held by the EHS department. It is conjectured that the plan may be currently non-operational. A community fund has been developed, which is contributed to by Lefa, as a government requirement. The fund is to pay for local community facilities.

During the site visit senior staff and the Community Liaison Officer (CLO) were involved in industrial talks with the local unions over proposed strike action by local staff; it was reported that this is common. It was commented that striking became common under the previous management of Lefa as a result of empty promises regarding pay and facilities such as housing and an extension to the medical centre (which should be provided by the community fund). Lefa are trying to resolve these issues at present however this is reported to be a difficult task without the support of the government (due to the elections) and with conflicting requirements of each village. It was reported that a Social Action Plan is being developed by Lefa, however no evidence of this was seen.

> ***WAI Comment:*** *WAI would recommend that a formalised Public Consultation and Disclosure Plan (PCDP) and a Community Development Plan (CDP) should be developed by the company, in line with international requirements. The PCDP would facilitate open, prior, informed and transparent consultation with local communities and other project stakeholders, and the CDP would provide a tool for managing community expectations. WAI is encouraged that a social development budget exists, and these monies could be managed in line with the aims and objectives of the CDP.*

Employment and Training

The majority of senior staff and management at Lefa comprise expatriates from the UK, Canada, USA, Australia and developed African nations. The majority of the non/partially skilled workforce are Guinean, largely from the local community. It was reported that Lefa aim to increase the number of local staff in senior/management positions, decreasing the number of expats employed. It was noted that the majority of senior/management staff converse in English, with the remainder of the workforce conversing in French.

Lefa undertake an induction on employment of staff, and conduct annual targeted (dependent on the facilities worked on/with) training programme for all staff. The programme includes training on equipment use and health and safety. The training programme is not currently effective as a result of the following reasons:

- Language barriers
- Level of local education standards; illiteracy is common
- Lack of acceptance of training from non-Guinean personnel
- Low career expectations.

It was reported that Lefa are in the process of reforming the training programme. No formal Human Resources Manager or policy was identified. Training of the local community in skills including farming and textiles has been implemented.

> ***WAI Comment:*** *In conjunction with the improvement of the training programme it is recommended that steps are taken to unify the language at Lefa. Should Lefa be aiming to shift to the education and training of local senior staff, it is likely that the use of French would be more effective. WAI is encouraged that the company has provided training for communities with the intention of reducing community impacts at mine closure.*

Site Access and Security

At present all haulage roads in and around the project area have been adopted for public use. This includes use by trucks, buses, cars, motorcycles bicycles and pedestrians both relating and not related to the project. As a result of the adoption of the haulage roads, the majority of Lefa's surface area is also readily accessible, although not permitted, with the exception of the explosives compound, processing plant and yard areas, and the mine camps; these are fenced with controlled entrances and exits. Lefa is patrolled by guards, however this is not effective in trespass prevention given that a large number of guards would be required to be effective coupled with the guards comprising local villagers (and know the trespassers). It was reported that theft and artisanal mining (further discussed below) within the project's footprint is common.

Artisanal Mining

Artisanal mining occurs at Lefa on a continuous basis and is considered to be a cultural right of the local population. It is reported that the majority of this activity takes place on the benches of the open pits. Several deaths and injuries have occurred due to the collapse of undermined benches and due to artisans being present during blasts. Security guards are employed to remove artisans but this often proves ineffective. Blasts are also concentrated on Fridays, where practicable, when Muslim custom in the region is not to work.

> ***WAI Comment:*** *It is recommended that the access and security of the project be reviewed, with particular focus on the health and safety of the local communities and their access to haulage roads, the open pits and project facilities. Any curtailment of artisanal mining is likely to bring the mine into conflict with the local community, so this situation and any change in what the local people see as their cultural rights will need to be managed very carefully, as part of a broader social and community development plan. These matters should be addressed in the PCDP and CDP above, and should also be included in the development of a Community Health, Safety and Security Plan, as outlined under international requirements.*

1.8.4.9 Cultural Heritage

On site personnel are not aware of the locations of local cultural and heritage importance and local communities are not consulted when operations are undertaken in new locations or by new practices; however complaints from the local community are considered after the event.

> ***WAI Comment:*** *The management of Cultural Heritage should be undertaken proactively, and should be included in any site monitoring plans. Consultation in this regard should be addressed via the development of a PCDP as described above.*

1.8.4.10 Health and Safety

Health and safety is managed via an alternative plan as set out in the EIA. It was reported that this is more detailed to the EIA; however given the following comments, the contents of the document are not being fully implemented. Health and safety management is coordinated by the HSE department, however delivered by the senior/managing personnel at each facility. During the site visit, Personal Protective Equipment (PPE) was worn by all personnel and signage was commonplace, both in French and English. It was reported that several accidents ending in injury, with one fatality have occurred during the last 12 months. Accidents are resultant from a number of causes. It was discussed that following any incident, this is reported and an investigation undertaken.

Visual stability checks are made at the rock dumps which are in close proximity to current operations. Fire protection measures are implemented in all buildings and plant. All employees with plant interaction are required to learn fire fighting skills. A fire fighting coordinator has been designated. An emergency response plan is implemented in the case of fire.

It was reported that health and safety training is not currently adequate as a result of time and resources constraints. It is conjectured that language barriers and illiteracy are also a contributory factor. It was discussed that health and safety training is improving.

The health and safety of the local population is not considered with regard to trespass and use of the haulage roads; however is planned for with regard to artisans during blasts.

> ***WAI Comment:*** *The management of health and safety was generally good, with appropriate procedures in place. WAI would suggest that training should be further developed, as part of an Occupational Health and Safety Plan (OHSP), and that community health and safety should be the focus of a separate plan. Given the high level of illiteracy amongst the workforce, it should be ensured that OHS signage is provided in a form that is easily understood by all.*

Remediation and Closure

It is reported that a mine closure bond is a requirement of the project by the Guinean government and in place.

The closure plan set out in the EIA is not being used at Lefa; however remediation of the rock waste dumps began in 2009. A spreadsheet was viewed containing budget costs for proposed remediation works during 2010 and 2011; however it is not considered that planning of closure has been undertaken further than the budget in the spreadsheet.

During the site visit, rock dumps at varying stages of remediation were viewed. Remediation is being undertaken through the use of stockpiled soil and organic waste, when available, and the plantation of native floral species and seeds, grown in an on site nursery. Retrofiling of the dumps is not undertaken as part of the remediation process.

> ***WAI Comment:*** *The final fate of the project has not yet not defined. As such it is not currently possible to assess whether the closure bond is sufficient to cover adequate remediation and rehabilitation costs. The work undertaken as part of the remediation of the rock dumps was encouraging however, a draft Mine Closure and Rehabilitation Plan (MCRP) should be developed, outlining all closure tasks, and including post-closure monitoring. It will then be possible to accurately assess costs associated with closure. In addition, progressive rehabilitation should be undertaken wherever possible, to reduce environmental liabilities and final closure costs.*

1.8.5 Conclusions

The project has been developed in compliance with Guinean legislation, and as such fully satisfies national requirements. However, it is identified, both by Lefa personnel and WAI, that Guinean environmental regulatory controls are limited in some areas, and as such national compliance does not automatically translate to international compliance. The company understands that further works will be required to comply with international guidelines and as a first step they have developed the following list of priority initiatives planned for 2010:

- Lefa is actively searching for an EHS Manager to oversee the implementation of a formal EHS Management system and to direct the existing EHS team;
- Standard safety procedures (lock-out procedure, hot works permits, drill-blast procedures) are in place including provision of good PPE to all employees. These procedures will be further reviewed and updated as required;
- Lefa has implemented key programmes for the management of cyanide aligned with the international cyanide Management Code. A review will be conducted to identify additional elements which may be required;
- Safety performance as measured by LTIFR is quite good. In May 2010 the YTD LTIFR was 0.67 with the peak of 1.13 reached for the month of May. An audit of safety and health management systems will be conducted in the 4th quarter of 2010 to identify potential areas for improvement; and
- EHS programs and reporting systems at Lefa and presently being aligned with Nord Gold reporting requirements.

1.8.6 Recommendations

In order for Lefa to move towards international compliance, the following steps are recommended:

- A gap analysis of the Baseline Study and EIA to identify missing data, in accordance with International Standards, current and proposed operations. This document has been drafted but need to be formalised further;
- Preparation of a time-bound framework Environmental, Health, Safety and Community (EHSC) Action Plan for management plans, policies, procedures and actions required, to address the shortfalls identified in the gap analysis. Various supporting documents have been provided but not yet formalised as an Action Plan;
- The collection of missing baseline data in accordance with EHSC Action Plan. Some extra data has been provided but will be further directed by the gap analysis;
- The implementation of management plans and systems. This has been budgeted for and some plans are provided;
- Incorporation of all data into a revised ESIA, including revised EHSC and closure plans. This is planned to be stared in late 2010 and completed by the 1st half of 2011;
- Appropriate implementation of the plans by experienced, qualified personnel and/or independent advisors via an auditing system; and
- The outstanding environmental and social issues highlighted in this report should form the basis of an SEMS and SEMP, associated management and monitoring plans as described and with supporting budgets.

1.8.7 Conclusion

The Lefa gold mine has undoubtedly suffered technical and reliability problems since Crew Gold Corporation acquired the mine in 2006 and installed a second hand CIP plant. Consequently the mine has been unable to achieve production anywhere near Crew Gold Corporation’s predicted gold output and the company has therefore been under some scrutiny. Production in this initial period also suffered during the rainy season, probably a consequence of both an inexperienced management team and mining contractor in place at the time.

However, the introduction of a new management team including Operations Manager, Plant Manager, Maintenance Manager and Stores Manager has assisted in turning the mine productivity around. These new employees have identified issues within their respective departments and have already instigated sweeping improvements and maintenance programmes.

The main threats to the efficiency of the operation are believed to be potential problems with power supply, water, mills and open ore feeders, all of which are currently being addressed with further issues such as fuel, labour and distribution also being tackled. Controls are now being installed to better manage the risks and threats; planning is more 'organic' as progress is being made and unforeseen issues present themselves, whilst previous 'crisis management' is believed to be over.

The existence of a mine planning department means that short and long term plans are being produced to give the production personnel some direction in the sequencing of the pits and the rock types.

WAI concludes, and concurs with the current mine management, that the Lefa mine and plant are capable of achieving the target set by Crew Gold Corporation of 310koz of gold per year. However, this will only be possible once all the current improvements and maintenance have been completed. Furthermore as these programmes are worked through it is likely that additional, unforeseen issues will present themselves that will require rectifying.

It appears to WAI that Crew Gold Corporation has changed the staff ethos and now has a competent senior management team on site, giving the project every opportunity to succeed.

1.9 Life of Mine Model for the Project

1.9.1 Introduction

A life of mine of the Lefa Gold Mine has been prepared in order to demonstrate the potential economical outcome of the project and its robustness. This model was based on reserve figures, described above, capital investment requirement estimates provided by the Client and audited by WAI with a view to similar projects, and other parameters. Input parameters were implemented into a life of mine model. The details of this evaluation are given below.

1.9.2 Operating and Capital Costs

Operating cost estimates were based on the preliminary budget figures for 2010. Since at the time of the preparation of this report the mine site was carrying out work at the CIP plant, has started the upgrading of the mining fleet and some general site improvements and restructuring, the preliminary 2010 budget figures were adjusted, as necessary, to reflect the estimated LOM operating cost.

The following capital cost estimates, shown in Table 1.41 below were used in the preparation of the economic analysis:

- Total required capital expenditure will be US$98.3M of which US$68.3M will be spent during 2010 to 2012;
- The majority of the capital will be to upgrade the fleet and provide critical spares to the plant and machinery;
- This includes an US$8M exploration programme of during the same period; and
- The remaining US$30.0M has been allocated to sustaining capital from 2013 to the end of the mine life.

Table 1.41: Lefa Capex			
ACTIVITY	**2010**	**2011**	**2012**
MINING	**US$K**	**US$K**	**US$K**
Vehicle Spares		1,749	700
Firifirini mining project access haulage road		2,100	
HD 785 Dump Truck x 3		2,800	1,400
D275A Dozers x 2		600	600
GD825 Grader x 1		510	510
WD600 Wheel Dozer x 1		600	600
HD 465 Water Truck			1,400
Excavator Estimate		1,200	1,200
Civil Fleet – Dump Trucks		180	180
Civil Fleet – Loaders		560	
Civil Fleet – Aux Equipment		324	
Heavy Equipment Workshop Wash Pad	500		
REMAN Program	3,000	8,000	3,000
200ton Crane Repairs	75	500	200
150ton Crane Repairs	100	400	200
75 ton Crane Repairs	225	150	100
45 ton Crane Repairs	92	100	
Service Trucks		160	160
PLANT FEED SYSTEM			
CV4 Upgrade to 750 t/h	500	500	
Live Stockpile Including Engineering - Est.	75	4,500	2,500
Apron Feeder Drive – Fayalala	134		
Apron Feeder Drive – Lero		134	
Concrete pad drive in sump and pump tail end CV06 and CV07		25	
Concrete pad tail end CV01		30	
Belt wash station head end CV06 and CV07.	25		
Secondary Crushing system study	300		
MILLING			
SAG MILL SPARES & EQUIPMENT		187	50
FEED PREPARARION AREA SPARES	75	75	
Replace Cyclone Feed Pumps	200	200	
SAG 1 Install SAG Feed Chute Winch		30	
SAG 2 Modify Trunnion Sealing	25		
SAG 2 Install SAG Feed Chute Winch		30	
SAG Mill electric motor		312	180
Ball Mill pinion cart.	135		
Ball Mill Girth Gear	200	500	
Plant De-Bottlenecking	166		
Regrind and Pebble Crusher (Estimates)		1,500	2,000
CIL			
Sump Pumping Systems	111	200	
Thickener Underflow Pumps		150	
CIP Tank Agitators		350	400
Replace inner tank screens	100	236	
Additional Clean up Skid Steer Loaders		233	150
Conversion to Standard CIL	200	600	
Concrete ball scats hopper		10	
Install vibrating screen on Trash		400	
POWER SUPPLY/FUEL/ELECTRICAL			
Powerhouse Critical Spares	400		
Powerhouse Rehabilitation Program	400	900	
Powerhouse and Process Offices			165
Power Station Ventilation	14		

HV Isolators and Arrestor 22kv & 6.6kv		20	
HFO Service Tank - Steel tank Cap 150M3	15		
MAINTENANCE			
Plate Rollers		100	
Long Bed Lathe (Pulley Shafts)		90	
Large Universal Milling Machine		90	
Autotec Spray Welding Kit		10	
Belt Rewinder (300 metre capacity)		130	
GENERAL & ADMINISTRATION			
IT Equipment - Site wide	15		
Lero Staff Village	167	550	250
Community Projects	117	350	350
Communication Equipment		16	
Solid Waste Management Site		30	
Security Building		75	
Aerial Photography unit		12	
Personal Tactical armour protection		15	
Manpower and Equipment hire	646	1,940	1,250
Infarstructure modifications		32	
Upgrade potable water supply		40	
Upgrade Asbuilts		30	
Base vie -new kitchen extension		20	20
ENVIRONMENT			
Dust Monitoring Equipment		8	
Filter Crusher		9	
Water Rain Drainage system		10	
Groundwater Monitoring Equipment		20	
Water waste/Sewerage treatment plants -Base vie /Fayalala	25	110	
EXPLORATION	1,989	3,000	$3,000
CAPITAL COST TOTAL	**10,026**	**37,742**	**20,565**

1.9.2.1 Life of Mine Model Assumptions and Input Data

The assumptions made in the WAI life of mine model for Lefa Gold project were based on:

- Au price of US$900/oz;
- Ore reserves as of 01.11.2010, estimated by WAI in accordance with JORC code guidelines;
- Mining Schedule, Prepared by WAI. The Mining Schedule targeting 8.0Mtpa ore mining rate with appropriate ramp-up of production by 2013; with 6.0Mtpa in 2011 and 7.5Mtpa in 2012;
- Long-term operating costs forecasts based on existing mine costs;
- Overall Au recovery indices obtained from testwork results;
- Annual discount factor of 11.2% (Base case);
- General and administration costs estimated based on Client’s guidance; and
- Overall tax at 20% of net income applied.

A summary of the Lefa Key Performance Indicators for 2008-9m 2010 is given in Table 1.42Table 1.42 below.

Table 1.42: Lefa Key Performance Indicators for 2008-9m 2010

	Unit	2008	2009	9m 2010
Ore mined	Kt	3,885	4,774	4,088
Ore milled	Kt	3,085	4,422	3,365
Average Au grade	g/t	2.2	1.4	1.2
Gold recovered	kg	6,145	5,523	3,846
Gold recovered	koz	198	178	124
Recovery rate	%	93	90	89
Normalised TCC	$/oz	827	809	772

A summary of the Lefa financial model assumptions and input data is given in Table 1.43 below.

Table 1.43: Lefa Financial Model Assumptions and Input Data

Year	TOTAL	Nov-Dec 2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
Gold Price, $/Oz	-	900	900	900	900	900	900	900	900	900	900	900	900	900
Ore production (diluted) kt	88,199	1,048	6,000	7,500	8,000	8,000	8,000	8,000	8,000	8,000	8,000	8,000	6,000	3,651
Waste mined, kt	292,590	3,477	19,904	24,881	26,539	26,539	26,539	26,539	26,539	26,539	26,539	26,539	19,904	12,112
Ore Mining cost, $/t	1.71	1.8	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70	1.70
Waste Mining cost, $/t	1.41	1.5	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40
Mining OPEX, k$	560,019	7,101	38,066	47,583	50,755	50,755	50,755	50,755	50,755	50,755	50,755	50,755	38,066	23,163
Au Grade g/t	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34	1.34
Gold mined, kg	118,186	1,404	8,040	10,050	10,720	10,720	10,720	10,720	10,720	10,720	10,720	10,720	8,040	4,892
Recovery	0.9	0.89	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90
Gold recovered, kg	106,354	1,250	7,236	9,045	9,648	9,648	9,648	9,648	9,648	9,648	9,648	9,648	7,236	4,403
Gold recovered, Oz	3,419,355	40,183	232,643	290,803	310,190	310,190	310,190	310,190	310,190	310,190	310,190	310,190	232,643	141,563
Processing cost, $/Oz	314	365	309	309	309	309	309	309	309	309	309	309	309	309
Processing cost, $/t	12.15	14	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00
Proc and Mining Costs $/Oz	478	542	473	473	473	473	473	473	473	473	473	473	473	473
Depreciation (total) k$	348,000	4,000	25,000	29,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	20,000
Revenue, k$	3,077,419	36,165	209,378	261,723	279,171	279,171	279,171	279,171	279,171	279,171	279,171	279,171	209,378	127,407
Operating Costs, k$	1,620,503	21,773	110,066	137,583	146,755	146,755	146,755	146,755	146,755	146,755	146,755	146,755	110,066	66,975
G&A, Sales	174,600	2,600	15,400	15,400	15,400	15,400	15,400	15,400	15,400	15,400	15,400	15,400	12,000	6,000
Royalty, 5.4%	166,178	1,953	11,306	14,133	15,075	15,075	15,075	15,075	15,075	15,075	15,075	15,075	11,306	6,880
CAPEX, k$	91,649	3,342	37,742	20,565	5,000	5,000	5,000	5,000	3,000	2,000	2,000	2,000	1,000	-

1.9.3 Conclusions

The WAI life of mine results in a positive NPV at various discount rates and at various gold prices, as well as the relatively high internal rate of return at nominal input parameters. It shows that the reserves considered in the financial model are profitable for exploitation in the current economic environment.

The deposit consists of a large gold ore resource, majority of which is represented by weathered ore. This type of ore is relatively easy to mine and process.

The fact that the key financial indices remain reasonably high given the conservative cost input parameters (such as mining and processing costs) used in the models, shows good economic potential for the project.

1.10 Conclusion

The Lefa gold mine has undoubtedly suffered technical and reliability problems since Crew Gold Corporation acquired the mine in 2006 and installed a second hand CIP plant. Consequently the mine has been unable to achieve production anywhere near Crew Gold Corporation's predicted gold output and the company has therefore been under some scrutiny. Production in this initial period also suffered during the rainy season, probably a consequence of both an inexperienced management team and mining contractor in place at the time.

However, the introduction of a new management team including Operations Manager, Plant Manager, Maintenance Manager and Stores Manager has assisted in turning the mine productivity around. These new employees have identified issues within their respective departments and have already instigated sweeping improvements and maintenance programmes.

The main threats to the efficiency of the operation are believed to be potential problems with power supply, water, mills and open ore feeders, all of which are currently being addressed with further issues such as fuel, labour and distribution also being tackled. Controls are now being installed to better manage the risks and threats; planning is more 'organic' as progress is being made and unforeseen issues present themselves, whilst previous 'crisis management' is believed to be over.

The existence of a mine planning department means that short and long term plans are being produced to give the production personnel some direction in the sequencing of the pits and the rock types.

WAI concludes, and concurs with the current mine management, that the Lefa mine and plant are capable of achieving the target set by Crew Gold Corporation of 310koz of gold per year. However, this will only be possible once all the current improvements and maintenance have been completed. Furthermore as these programmes are worked through it is likely that additional, unforeseen issues will present themselves that will require rectifying.

It appears to WAI that Crew Gold Corporation has changed the staff ethos and now has a competent senior management team on site, giving the project every opportunity to succeed.

SECTION C: KAZAKHSTAN ASSETS

1 SUZDAL

1.1 Background

1.1.1 Introduction

Nord Gold N.V. has a 100% interest in the Suzdal gold mine through the acquisition of Celtic Resources Holdings Ltd who in turn had a 100% indirect shareholding in the Joint Stock Company FIC Alel (JSC FIC Alel), the local operator. The licence to the mine was granted to JSC FIC Alel in 1995 and the mine was operated to produce gold rich quartz flux to feed Kazakhstan smelters.

The mine was brought into continuous production in 1999 treating the oxide cap of the orebodies, using open pit heap leaching, although this has now ceased with all production coming from the underground primary ores.

1.1.2 Location & Access

The Suzdal gold mine is located 55km southwest of Semipalatinsk in northern Kazakhstan (Figure 1.1 and Figure 1.2).

Access to the Suzdal site is very straightforward from Semipalatinsk, with the first 32km on asphalt road, followed by 23km on a reasonably well graded dirt road to the south. Total driving time from Semipaltinsk is approximately 60 minutes.

Figure 1.1: Location of Suzdal in Northeastern Kazakhstan Russia

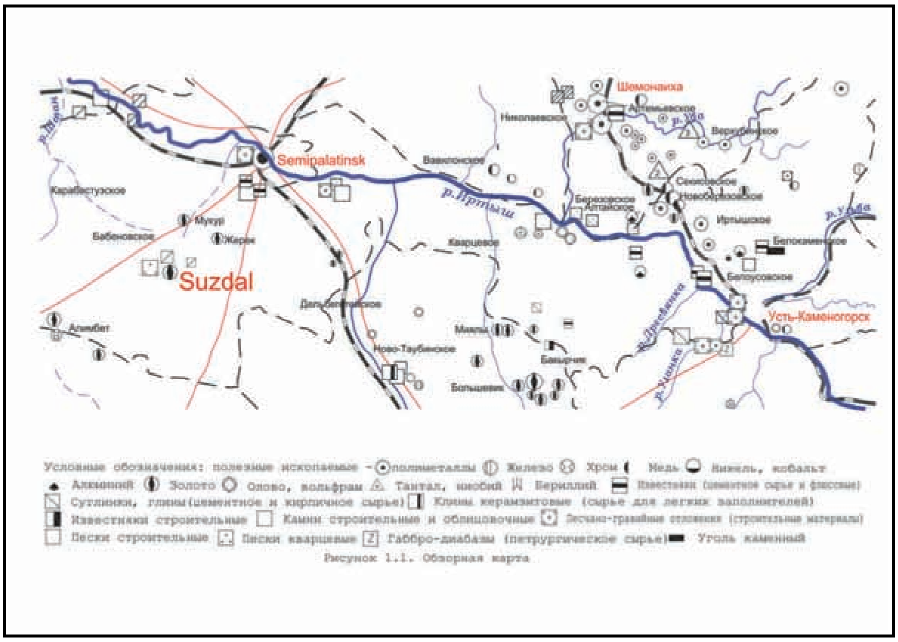


Figure 1.2: Detailed Location of Suzdal

1.1.3 Topography and Climate

The relief of the region is characterised by a gently rolling steppe, with absolute elevation changes from 390 – 500m. There are no rivers, or permanent bodies of water and the mine is located in a quiet seismic zone.

The climate of the region is typical continental, with maximum summer temperatures from +23°-42°C, and minimum winter temperatures from -25°C-40°C. Precipitation is generally light with about 330mm per year. Snow is present from mid November, with the total for the winter of approximately 25-30cm. Freezing depth is 1-1.5m.

> ***WAI Comment:*** *the Suzdal mine is located in an area of unremarkable steppe, and although winter weather conditions can be harsh, the location presents no major obstacles to the continued development of the mine.*

1.1.4 Infrastructure

Power comes from 2 x 6kV power lines, with potable water from an underground aquifer pumped from bores 1.3km to the north of the deposit. Process water is provided from underground water pumped directly from the mine.

The mine comprises a number of disused open pits (one of which contains the portal for the decline), waste dumps, plant, offices, maintenance buildings, accommodation and canteen. In addition, it is serviced by good telephone and email links, and the nearest railway station is at Semipalatinsk, about 55km distant.

> ***WAI Comment:*** *the Suzdal mine is nearing the completion of a major plant expansion which will take the operations to some 500kt per year of ore. The infrastructure to facilitate this expansion is in place and of a high standard.*

1.1.5 Mineral Rights and Permitting

The Mining Licence for Suzdal was approved in June 2004, covers an area of 5.9km^2, and is valid for production down to -820m. The licence co-ordinates are given in Table 1.1 below.

Table 1.1: Licence Co-ordinates		
Point	**Northing**	**Easting**
1	50°01'31.8"	79°43'28.2"
2	50°02'39.1"	79°44'35.9"
3	50°03'17.3"	79°45'44.6"
4	50°02'39.5"	79°46'41.5"
5	50°01'11.6"	79°43'57.7"

WAI Comment: *an inspection of the licence documentation has shown that all are in good order and suitable for the future needs of the Company.*

1.2 Geology and Mineralisation

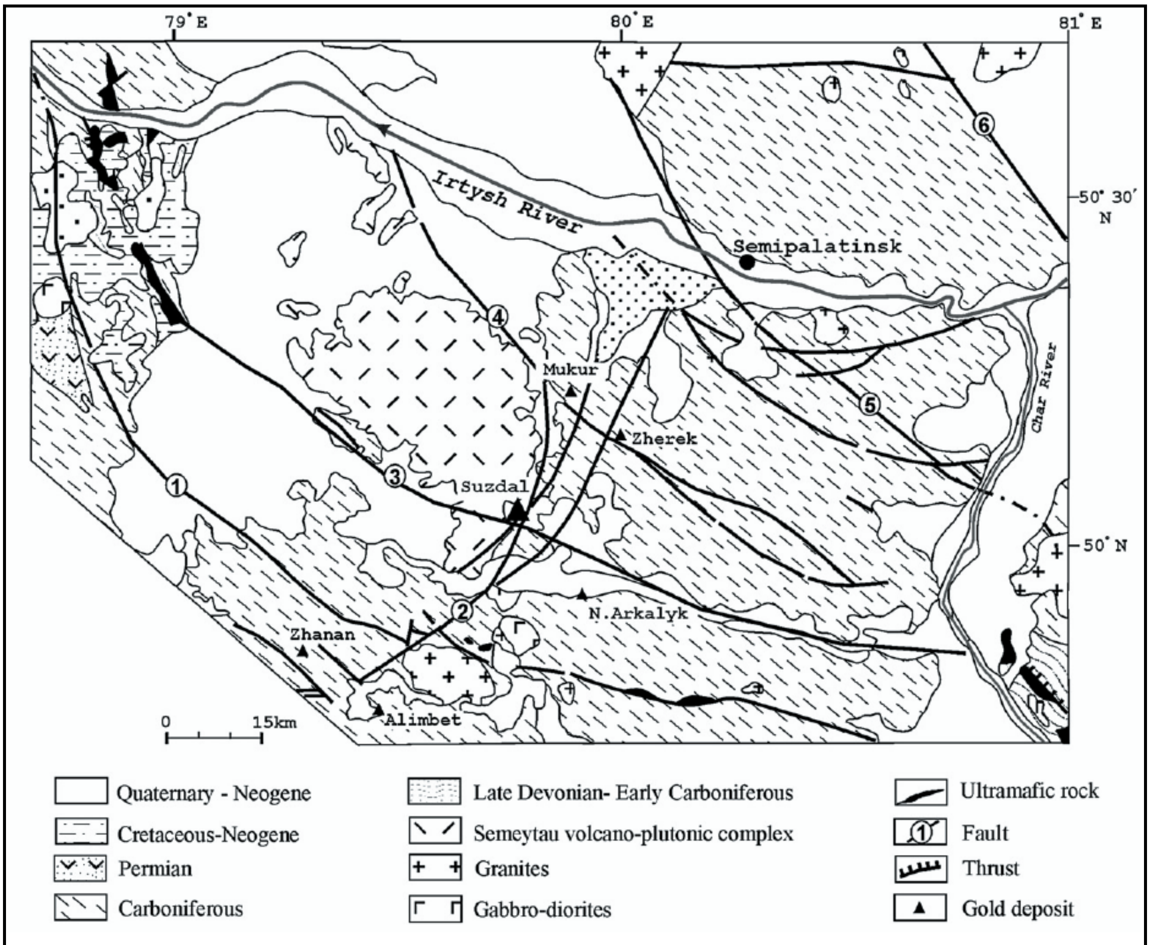
1.2.1 Regional Geology

1.2.1.1 Geology of the Charsk Gold Belt

Eastern Kazakhstan, including the Charsk Gold Belt, is mostly underlain by fragments of Kipchak and Altaid paleo-island-arc systems. The region is characterised by ophiolite slivers separated by marine sedimentary basins.

Between Mesozoic and Eocene times, the region underwent deep surface weathering and chemical erosion producing the current weathering profile. Surface oxidation reaches an average thickness of approximately 30m, except in proximity to fault and alteration zones where it reaches up to 100m.

Quaternary loams, clays and sands cover a very large portion of the Charsk Gold Belt area (approximately 70%). Figure 1.3 shows the detailed geology of the Suzdal area.



Main Fault Zones: 1 Baiguzin-Bulak; 2 Suzdal; 3 Char-Gornostai-Zimunai; 4 Mukur; 5 Western Kalba; 6 Terecty-Ulungu

Figure 1.3: Detailed Geology of the Suzdal Area

Magmatism, hydrothermal alteration and metallic deposits have taken advantage of local disturbances in the dominant northwest-southeast regional structure patterns, when intrusive bodies were generally preferentially emplaced along major faults during the Devonian and Carboniferous. Within the Charsk Gold Belt, most of the known gold deposits are associated with left-lateral fault zones and subsidiary structures and exhibit a strong spatial relationship with felsic and mafic intrusions.

1.2.1.2 Mineralisation of the Charsk Belt

The Charsk Gold Belt is prospective for sedimentary-hosted base metal deposits, volcanogenic massive sulphide (VMS) deposits, magmatic-related copper and copper-gold deposits and orogenic (mesothermal) gold deposits.

The gold deposits invariably exhibit strong structural and lithological controls and comprise quartz veining and/or gold-sulphide dissemination hosted by highly strained rocks; gold is commonly accompanied by elevated silver, arsenic, antimony and, locally, tungsten contents.

In general, sulphide bearing gold mineralisation is refractory, whereas gold in quartz veining is free milling, and the thick oxide zones, amenable to heap leaching, have represented the primary focus of exploration. Refractory sulphide mineralisation has only recently started to draw attention.

1.2.2 Mine Geology & Mineralisation

1.2.2.1 Introduction

Suzdal comprises of a typical shear-hosted mesothermal gold mineralisation distributed within a late Palaeozoic turbidite sequence, along north-east trending shear zones, which contains four distinct mineralised orebodies (Figure 1.4); the most common minerals are pyrite, arsenopyrite and pyrrhotite rendering a significant proportion of the gold refractory. Individual veins have strike lengths ranging from 100-600m with thicknesses ranging from 1- 30m.

Figure 1.4 shows the main orebodies, geology and superimposed infrastructure at Suzdal.

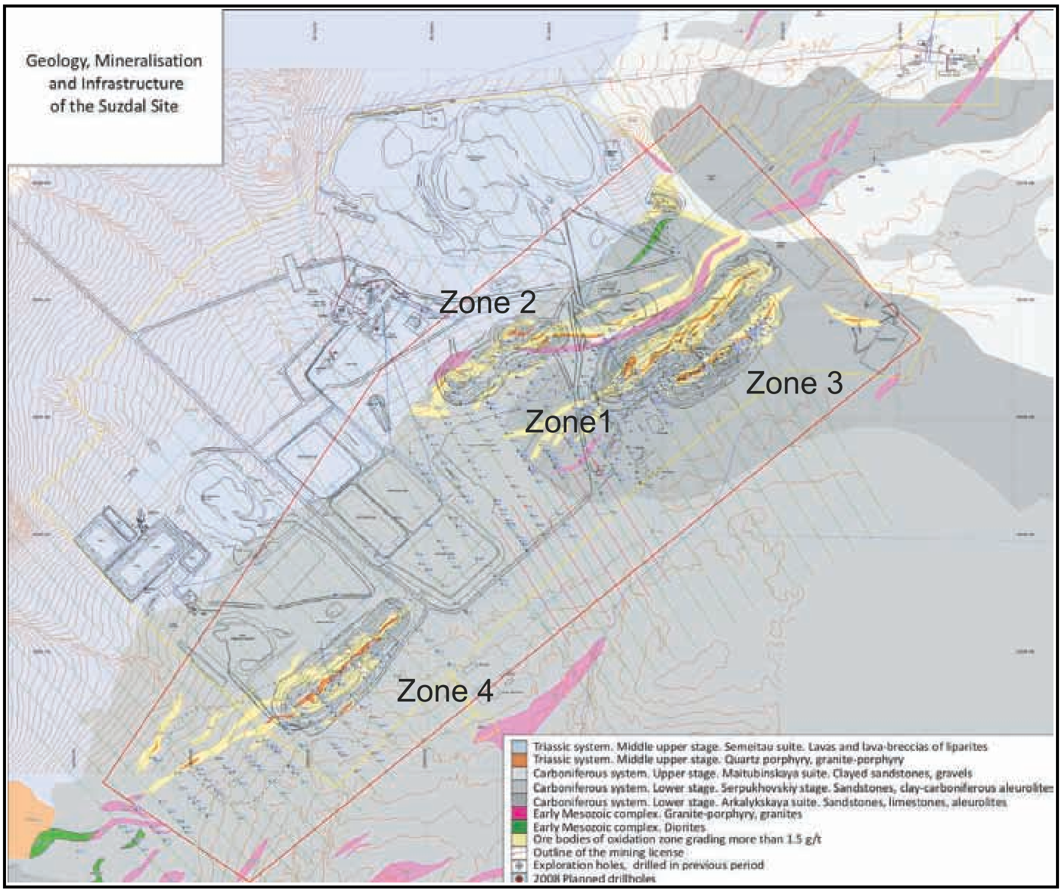


Figure 1.4: Main Orebodies of the Suzdal Deposit

1.2.2.2 Structure

Suzdal is located at the juncture of NW and NE (Suzdal) trending fault zones in common with other gold deposits, such as Zhanan, Zherek, Mukur, Alimbet and Northern Arkalyk, is hosted in carbonaceous carbonate-clastic rocks thrust northeastward over continental deposits, along the NW-trending Gornostai overthrust.

The mineralisation is situated in a 4km by 3-400m zone and is controlled by the NE-trending Suzdal fault.

1.2.2.3 Ore Zones

Four SW-oriented en echelon orebodies are recognised (Figure 1.4), with dips of 40° to 90° SE. Combined orebodies 1/3 (Photo 1.1) and 2 in the northeastern part of the deposit possibly merge at depth and are hosted by a predominantly calcareous facies, whilst orebody 4 to the south west is hosted by sandstone and carbonaceous siltstone. The orebodies intersect or are subconcordant with bedding (Figure 1.5). Individual orebodies extend to a depth of 400m, are 1.5 to 2km in length and 20 to 150m thick, and can be traced for 200 to 800m at the surface, with a thickness of 1 to 40m.

Photo 1.1: Typical Deformed Mineralisation on Orebody 1/3

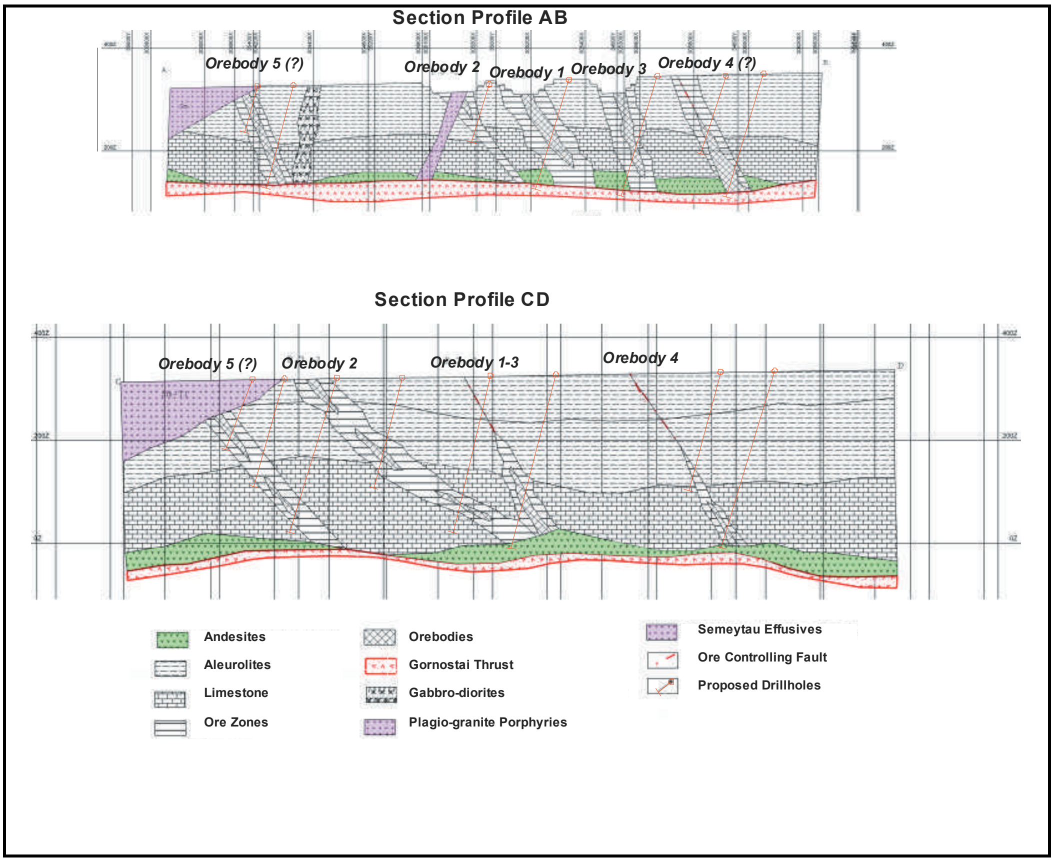


Figure 1.5: Schematic Cross-sections through the Orebodies

Four types of ores are recognised at the Suzdal deposit:

Type A - Stratiform gold-bearing sulphide mineralisation was mined by open pit in orebody 4 and is hosted by turbiditic carbonaceous sandstones and siltstones, which become more calcareous eastwards. The gold content varies from hundreds of ppb to >40g/t Au; however no visible gold has been observed, even where high values are recorded.

Type B - Sulphide impregnations occur in syn-sedimentary, tectonic, and melange breccias. The gold content of bulk samples from calcareous brecciated ores is variable, but on the average it is 9g/t Au.

Type C - Disseminated-stockwork mineralisation is characterised by intensely silicified fragments and breccia cements of different origin. These silicified and mineralised rocks are present is the deeper levels of the orebodies in the eastern part of the deposit. Gold-sulphide mineralisation occurs as impregnations and veinlets with gold contents of up to 54ppm.

Type D - Lode-type antimony-quartz-carbonate mineralisation is of late paragenesis and overprints the above-described types. Antimony mineralisation occurs in veinlets, ranging from several mm to tens of cm in thickness, and overprints sulphidised sandstones and mineralised silicified breccias.

1.2.2.4 Summary

Mineralising processes began with the accumulation of syngenetic mineralisation with the subsequent development of a productive auriferous pyrite-arsenopyrite ore assemblage in fracture zones, associated with hydrothermal processes resulting from tectono-magmatic activity.

Suzdal has been compared to Carlin type deposits, but the mineral paragenesis and the wide occurrence of finely dispersed gold bearing arsenopyrite mineralisation allows a closer comparison with the large Siberian gold deposits.

1.3 Exploration

1.3.1 Historical Work

The Charsk Belt has undergone considerable geological exploration, but it was not until 1980-83 that the oxide part of the deposit was discovered and from 1984-87, limited core drilling and trenching was undertaken. Mining of the oxide zone began in 1985 and continued until 1995 when bankruptcy led to the transfer of the licence to JSC FIC Alel.

Shallow drilling on a grid of 500 x 50m (250 x 25m in mineralised areas) was instigated between 1986 and 1988 and outlined the broad mineralised zones, and between 1987-90, 427,017m of drilling recovered 31,531m of core.

During 1990-93, continued exploration down to 500m was undertaken using core drilling (59,738m) culminating in a C_2 reserve calculation. In 1997, a formal resource evaluation was made.

1.3.2 Current Exploration

1.3.2.1 Introduction

Recent exploration has been undertaken both from surface and as detailed exploration of the deep levels from underground.

The work carried out from 01 January 2009 to 01 July 2010 is shown in Table 1.2 below.

Table 1.2: Samples taken from 01 January 2009 to 01 July 2010			
Trench Samples	Pit Samples	Core Samples	
		Underground	Surface
10,463	4,974	19,233	13,698

1.3.2.2 Drilling

In the period from 01 January 2009 to 01 July 2010, 86 boreholes were drilled totaling 47,688m, including 54 boreholes (26,883m) in orebodies 1/3 and 2 and 32 boreholes (20,805m) in orebody 4.

49 boreholes (71,765m) were drilled from underground into orebodies 1/3 and 2 for operational and detailed excavation. The core storage has been constructed and is operational.

Exploration by surface and underground drilling from mine workings was completed, the former being carried out in orebodies 1/3; 2 and 4 on a grid of 50x50m, required for C_1 category evaluation of State approved reserves (GKZ – Republic of Kazakhstan).

All exploration holes (NQ core 47.6mm) used a Boart Longyear drilling string, with not less than 90% core recovery (and close to 100% where observed) and were subject to geophysical (down-hole) and IP surveys.

> ***WAI Comment:*** *the drilling practices and core handling, logging and storage procedures are generally excellent, although a permanent core storage facility is urgently required.*

1.3.2.3 Sample Preparation

The mine has two sample preparation facilities, one for the geological samples which then go AAS analysis, and one for the plant samples. Both facilities were accredited as of July 2008.

The geologic sample preparation area, although using old equipment, appears clean and well maintained and has a dust suppression system. The preparation laboratory uses standard Soviet protocols of Jaw and Rolls crushers, followed by pulverisation to 0.074mm.

The geological laboratory was also briefly inspected where samples are analysed by AAS. The facility was found to be clean and well maintained.

1.3.2.4 QA/QC

The quality of analysis conducted by the laboratory was regularly controlled by the use of 618 fire assays and 870 atomic-absorption assays for samples collected from mine workings and drill holes during exploration.

External control included 245 fire assays of trench samples, commercial samples and chip samples. Doré gold analyses were controlled by laboratory “Metallor” Switzerland, whilst arbitrary analysis was conducted at “Alex Stewart” laboratory.

The Company reported that the quality of conducted works is within the accepted accuracy.

> ***WAI Comment:*** *overall, the level of quality control was found to be high, and where problems were identified, they were quickly rectified.*

1.3.3 Future Exploration

The mine spends considerable time and money on underground cross-cuts and closely spaced sub-levels in an effort to define the complex mineralised zones and as an alternative WAI recommends that more drilling should be done on possibly 5m pierce points to provide the level of detail necessary for mine planning, providing a more flexible and cheaper approach and, probably, better information. WAI also recommends that, as the degree of deviation has been found to be minimal, surveying all coring holes should be discontinued, with the exception of control holes.

The company plans to complete approximately 25,000m of core drilling before July 2010 (some 3,000m had already been done at the time of the visit), and 5,000m of underground drilling.

The sulphide mineralisation at depth has a considerably larger footprint than the oxide material and there may still be significant opportunities for the discovery of blind orebodies, considering that the original vertical reconnaissance holes were drilled on a 200x40m, and that geochemical surveys were hampered by a thick clay layer, masking any mineral signature. Mineralised zones are <50m wide, and the known Suzdal deposit was only intersected by 3 holes, and hence the original grid may have missed important mineralisation.

WAI recommends that the mine embarks on an extensive RC drilling programme to test the mine and surrounding areas for extensions to the mineralisation.

1.4 Mineral Resources

1.4.1 Introduction

The most recent internal resource statement (01 January 2009) estimated 3.6Mt @ 9.86g/t Au in the combined C_1 and C_2 categories. In addition, the mine has also been developing geostatistical models and the most recent Micromine® model was finalised in February 2009 and approved by GKZ (RK) in March 2009.

Resources for the three orebodies 1/3, 2 and 4 were estimated using deep hole drilling, underground drilling and underground channel samples. However, for the latter two ore bodies, less underground data were available as they are still in the development stage.

1.4.2 WAI Estimate (1st November 2010)

Wardell Armstrong International (WAI) was commissioned by Severstal Resources (Client) to prepare a Mineral Resource estimation in accordance with the guidelines of the JORC Code (2004) for the Suzdal Au deposit, Russian Federation. The work completed during this project is summarised below:

1.4.3 Topography

The pit topography supplied for Suzdal is understood to have been generated from a mine survey carried out on 18th May 2010. Supplied underground stope and development surveys are understood to have been carried out on 1st November 2010.

1.4.4 Database Compilation

All of the sample data collated for the current project are summarised below in Table 1.3.

Table 1.3: Sample Data Summary

	No. of Holes/Lines	No. of Samples	Total Length (m)
Surface Diamond Drillholes	597	68,660	180,798
Underground Drillholes	333	23,696	25,116
Underground Channels/Trenches	5,324	24,683	236,975

A general plan of all these samples types is shown in Figure 1.6. The diamond drillholes have generally been laid out on approximately 50m spaced sections, oriented along the strike of the deposit at approximately 050°.

Underground channel samples have been taken from several underground levels within the area covered by the larger group of surface drill holes. Levels are vertically spaced at approximately 8 to 12m.

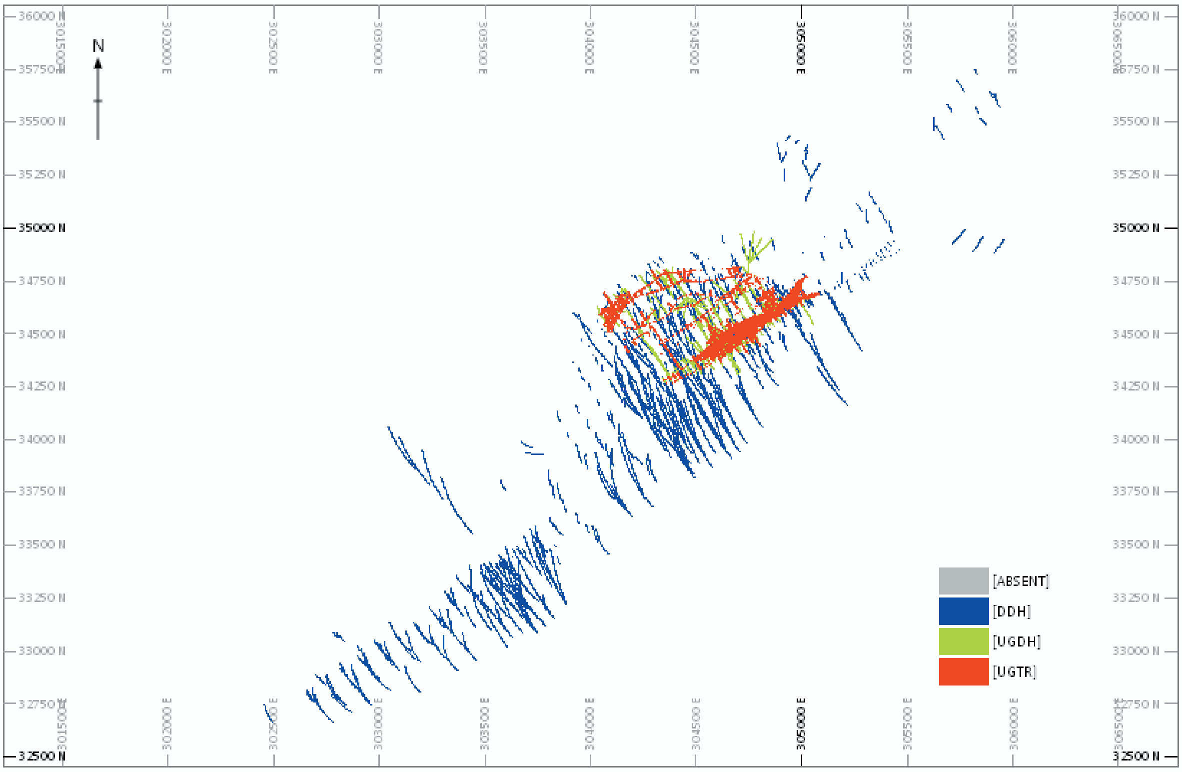


Figure 1.6: Plan View of All Sample Types
(Blue – surface drill hole; Green – underground drill hole; Red – underground channel)

1.4.5 Mineralised Zone Interpretation

Previous interpretations had been based on a gold cut-off grade of 1.5g/t Au and for the purposes of this latest resource estimate, the same value was used, although with current projected forward gold prices, economic cut-off grades may well be lower than this.

Wireframes were provided by the client in dxf format and imported into Datamine®. The wireframes were coded per zone and per sub-zone. Zones were identified by the mineralisation domain, and further sub-zoned by individual discrete areas of mineralisation within the domain. Unique ZONE and SUBZONE ID's were applied to the wireframe to be used in the interpretation and subsequent modelling and estimation phases of work.

A 3D view of the subsequent interpretations is shown in Figure 1.7. As with previous interpretation work, the deposit has been modelled with three main zones (orebodies), Zone 1/3, Zone 2 and Zone 4. Both Zone 1/3 and Zone 2 have been modelled utilising surface drill holes, underground drill holes and underground channel samples. Zone 4 has been modelled using surface drill holes only.

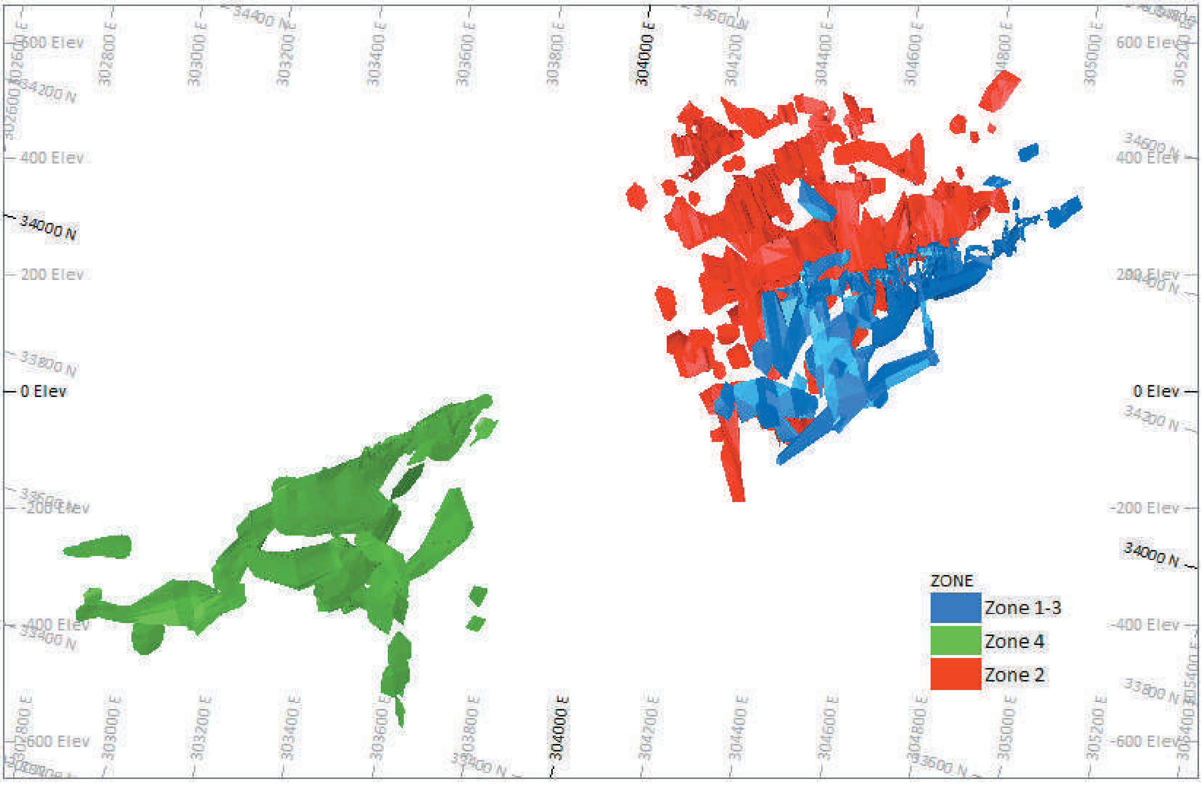


Figure 1.7: 3D View of Mineralised Zones, Looking NorthEast

1.4.6 Geostatistics

The sample data indicates approximately distinct log-normal population of gold grades within the different zones. Figure 1.8 shows log-probablity plots of the mineralised samples by zone and the similarity of the distributions supports the combination of both samples types for grade estimation.

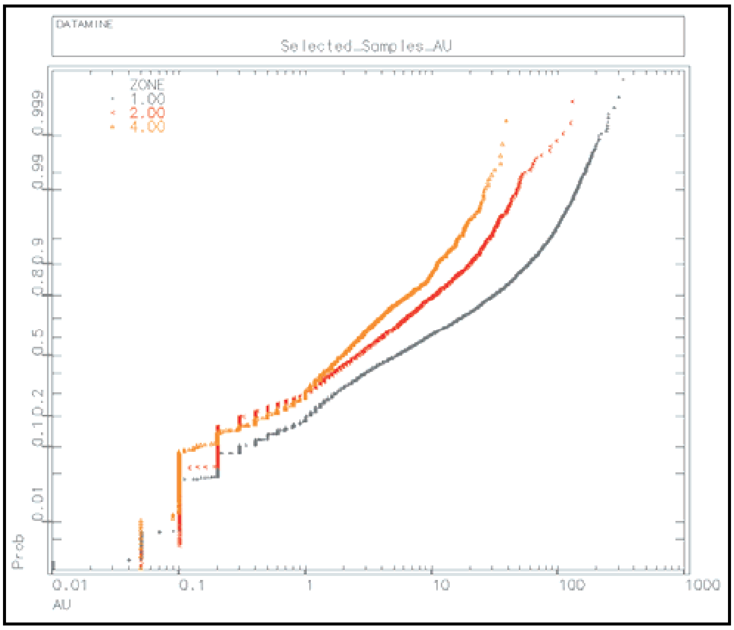


Figure 1.8: Log-Probability Plot of Selected Samples, by Zone (Zone 1/3 – Black; Zone 2 – Red; Zone 4 – Orange)

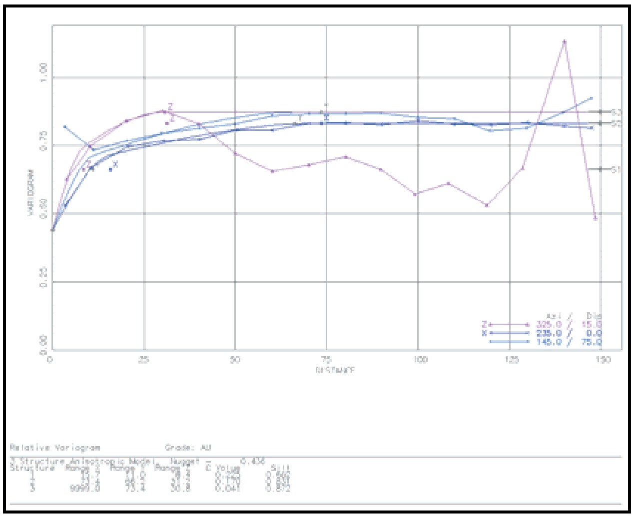
The plot also shows that there are various outlier grades present above 110g/t for Zone 1/3, above 45g/t for Zone 2 and above 25g/t for Zone 4. These outlier grade levels were also verified with decile analysis. These outliers could not be readily separated physically, so top-cuts at these levels were applied, prior to compositing.

The data were then converted into 2m composites. A statistical summary of the composites indicated variogram ranges of approximately 50m and more along-strike, 40m down-dip and 20m cross-strike.

Figure **1.9** illustrates the variogram models for each of the zones.

1.4.7 Block Modelling

A rotated model prototype was set up with a columnar 5m x 5m parent block size. This prototype was then used as the basis to set up a volumetric model, as controlled by the present (30th June 2010) topographical surfaces, the original pre-mining topography, the underground mined topography (1st November 2010) and the interpreted mineralised zones described above.



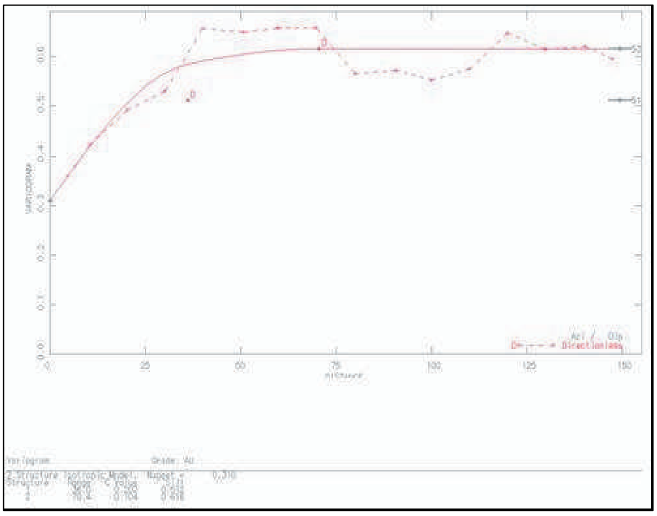


Figure 1.9: Variogram Models for Zones 1-3 and 2 (left) and for Zone 4 (right)

A density value of 2.70t/m^3 was used for sulphide material, as supplied by Nord Gold. An example cross section through the volumetric model is shown below in Figure 1.10.

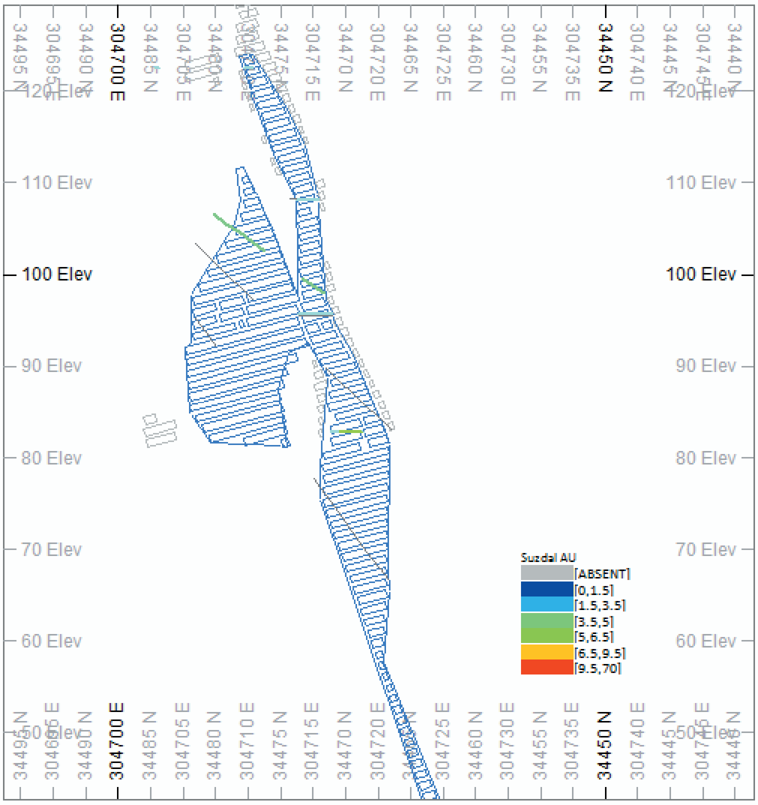


Figure 1.10: Cross-Section through Volumetric Block Model, showing Composites

1.4.8 Grade Estimation

Grade estimation was carried out using Ordinary Kriging (OK) as the principle interpolation method. Inverse Power of Distance Squared (IDW^2) and Nearest Neighbour (NN) were also used for comparative purposes. The OK method used estimation parameters defined by the variography. The estimation was performed only on mineralised material contained within the mineralised zones. A summary of the Kriging Plan is shown below in Table 1.4.

Table 1.4: Summary of Kriging Plan

	All Zones
Search 1 (along strike x down dip x cross strike)	25m x 25m x 5m
Search 2 (along strike x down dip x cross strike)	75m x 75m x 15m
Search 3 (along strike x down dip x cross strike)	150m x 150m x 40m
Discretisation (x/y/z)	3/3/2
Min no. composites (search 1/2/3)	4/4/4
Max no. composites (search 1/2/3)	15/15/15
Minimum Number of Octants	3/3/3
Minimum Composites per Octant	1/1/1
Maximum Composites per Octant	3/3/3

NB –
1) Directional anisotropy has been used during interpolation to orient the search ellipses and reflect local variation in strike and dip

1.4.9 Model Validation

A model validation process included the examination of block model versus composites, and the building up of a model grade profile, to compare average grades on vertical slices, as derived from the composites directly, as well as from interpolated model grades. An example Swath plot from this process is shown below in Figure 1.11.

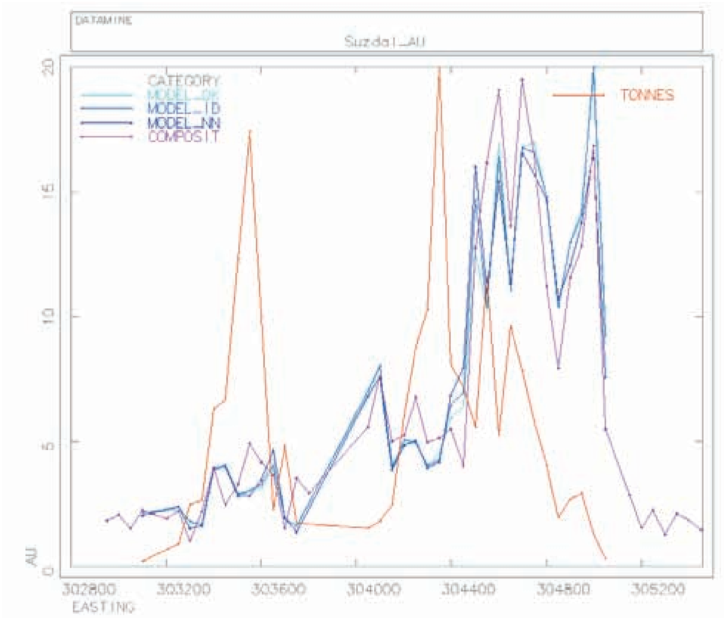


Figure 1.11: Model Grade Profile – Comparison Local Grade Averages

1.4.10 Resource Classification

Criteria for defining resource categories were also derived from the geostatistical studies. Key drillhole spacings for the allocation of *Measured* resources were 25m x 25m, for *Indicated* resources they were 60m x 60m. All of the modelled resources within 60m of a diamond drillhole intersection have been classified as *Inferred*. Figure 1.12 shows the resource classification in Zone 1-3.

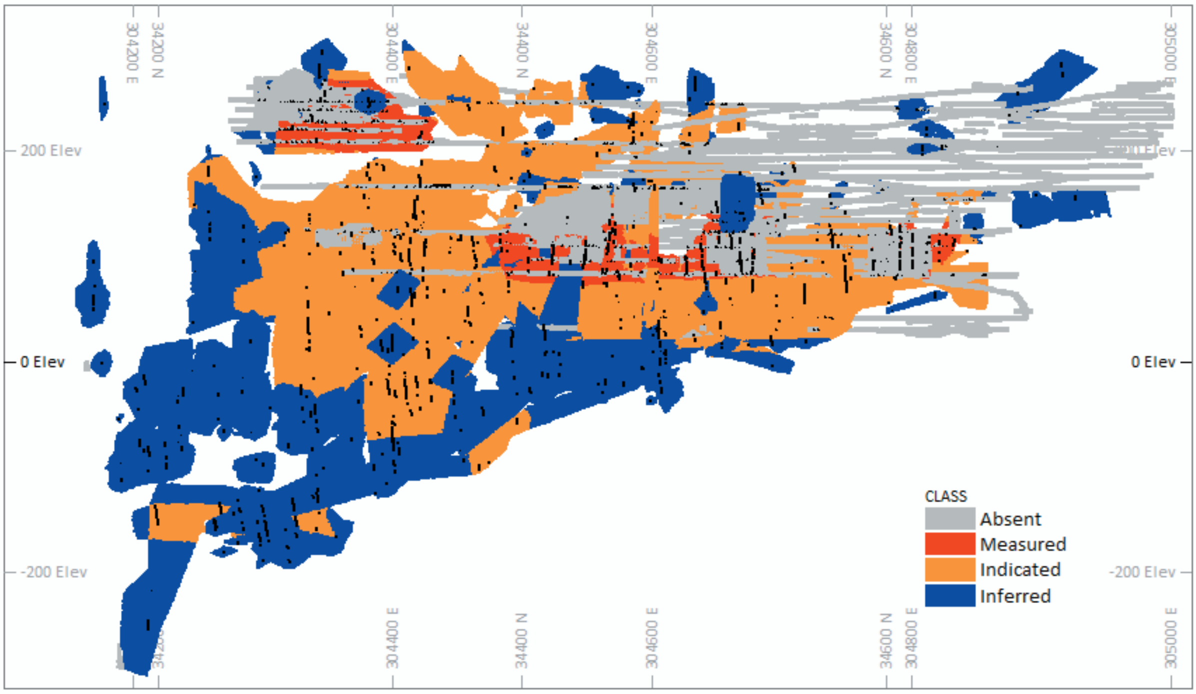


Figure 1.12: Example Long-Section Showing Resource Classification

1.4.11 Resource Evaluation

The final block model was used as the basis for Mineral Resource evaluation. Summary results of the evaluation of the unmined, in-situ resources are shown in Table 1.5 for three different cut-off grade levels: 1.0g/t, 1.5g/t and 2.0g/t Au.

Table 1.5: Suzdal Resource Estimate (WAI, 1st November 2010) with applied minimum mining width of 1.2m (in accordance with the guidelines of the JORC Code (2004)

Ore Type			Sulphide	Sulphide	Sulphide
Cut Off Grade (g/t)			1.5	2.0	4.0
Measured	Tonnage (kt)		128	126	108
	Au (g/t)		8.77	8.88	9.80
	Metal	kg	1,119	1,116	1,062
		koz	35.99	35.89	34.13
Indicated	Tonnage (kt)		4,123	3,995	2,665
	Au (g/t)		6.11	6.25	7.87
	Metal	kg	25,205	24,974	20,973
		koz	810.37	802.94	674.29
Measured + Indicated	Tonnage (kt)		4,250	4,121	2,773
	Au (g/t)		6.19	6.33	7.95
	Metal	kg	26,325	26,090	22,035
		koz	846.36	838.82	708.43
Inferred	Tonnage (kt)		2,893	2,771	1,477
	Au (g/t)		6.11	6.31	9.17
	Metal	kg	17,687	17,472	13,540
		koz	568.65	561.73	435.31

Notes:
1. Mineral Resources are not reserves until they have demonstrated economic viability based on a Feasibility study or pre-feasibility study.
2. Mineral Resources are reported inclusive of any reserves.
3. The contained Au represents estimated contained metal in the ground and has not been adjusted for metallurgical recovery.

WAI Comment: *the latest Measured and Indicated WAI resource estimate for Suzdal has a lower metal content than the most recent GKZ C_1 + C_2 estimate (higher tonnage but lower grade) which is probably a function that WAI has reclassified some of the C_2 resources as Inferred. However, given the past history of the mine, it is likely that much of this Inferred resource will transfer to a higher resource category with additional drilling.*

1.5 Mining

1.5.1 Introduction

The first regular production at Suzdal dates back to 1985, with initial extraction from Orebody 1 and Orebody 3 using open pit methods. JSC FIC ALEL controlled and operated the site between 1995 and 2000, when Celtic Resources Holding Ltd took control over JSC FIC ALEL, with mining moving from open pit to underground methods between 2004-2006.

Mining currently takes place at a rate of some 330ktpa from Orebody 1/3 and by 2014 it is expected that the production rate will be increased to 500Ktpa.

1.5.2 Historical and Current Mining Activities

Prior to the acquisition of the Suzdal deposit by The Client, the deposit was mainly exploited by open pit. The total amount of ore, extracted from the open pit was 1,358Kt at an average grade of 5.18g/t Au (total 7,019 kg

Au/extracted 225,661 oz). With depletion of the ore reserves from the open pit, the deposit has transferred underground.

A summary of production in 2008, 2009 and 9m 2010 is given in Table 1.6 below.

Table 1.6: Summary of Underground Production for Suzdal Mine 2008 to 9m 2010

	Unit	2008	2009	9m 2010
Ore mined	Kt	331	340	245
Ore milled	Kt	302	317	232
Average Au grade	g/t	14.5	13.4	10.3
Gold recovered*	kg	3,201	3,212	1,913
Gold recovered	koz	102.9	103.3	61.5
Recovery rate	%	71.8	74.8	75.9
Full cash cost	US$/oz	372	425	485
Normalised TCC	US$/oz	376	398	434
Ore mining costs	US$/t	38	26	31
Ore processing costs	US$/t	57	50	51
General and administration costs	US$M	2.6	2.8	2.3
CAPEX	US$M	33.4	33.2	26.8
Depreciation	US$M	25.6	25.5	19.4

Note
*** Dore alloy**

During 2009, mining activities have taken place in orebody 1/3, with major ore production focused on the 120m Level.

In 2009, annual production was approximately 340kt whilst the production schedule for 2010 and beyond demonstrates that the mine plans to increase production to 500ktpa by 2011, requiring significant levels of development during 2010 - 2012. In the first half of 2010, 180kt was mined.

1.5.3 WAI Mine Planning and Scheduling

1.5.3.1 Stoping Methods

Sub level caving is the principal mining method utilised at Suzdal. Differing layouts of the sub level caving method will be utilised to mine the various orebody geometries encountered. The main orebody geometric parameters dictating the mining layout are the orebody thickness and dip.

Shallow dipping mining blocks (30-60°) are accessed via cross cuts, driven at 10m intervals along a footwall drive, with an ore drive being driven along the footwall side of the orebody for ring drilling as shown in Figure 1.13.

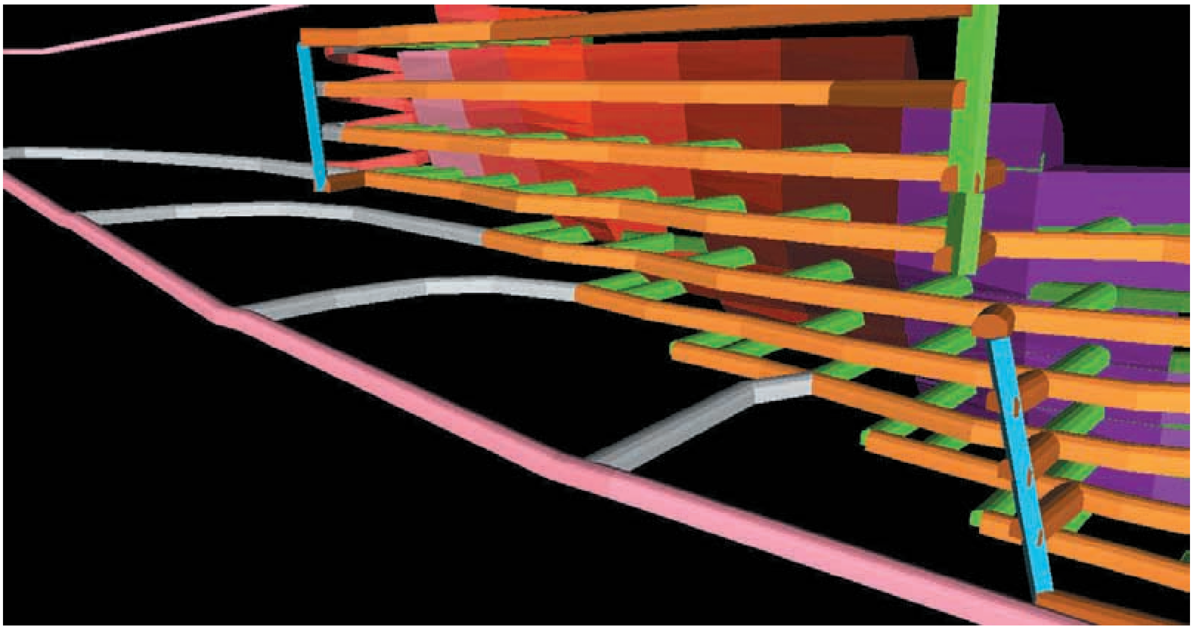


Figure 1.13: Ore zone 4 Showing Footwall Drives (Orange) and Crosscuts (Green)

Steeper mining blocks (45-85°) are currently mined without cross-cuts from a footwall drive, with mucking being undertaken along the ore drive.

A modified sublevel open stoping method with backfill has been utilised for ore extraction in zones which fall within shaft pillars or adjacent to long-term tunnels such as main access ramps, drives or ventilation raises.

All variations of the sub level mining layout are mined by retreating along strike. The stoping sequence is from top to bottom of the orebodies.

Development

Ore drives and sub-level development have dimensions of approximately 3.2m x 2.9m, with 40m intervals between main mining levels, and 10m or 20m intervals between sub levels as dictated by orebody geometry and geotechnical requirements.

Access

The ore and waste is hauled to surface by articulated dump trucks via declines. The dimensions of the declines and haulage drives are 3.6m x 3.4m and 4.4m x 3.4m respectively.

Sub levels are accessed via spiral ramps from the mining levels, with ventilation intakes and second means of egress' connecting with footwall drives on each level as shown in Figure 1.14.

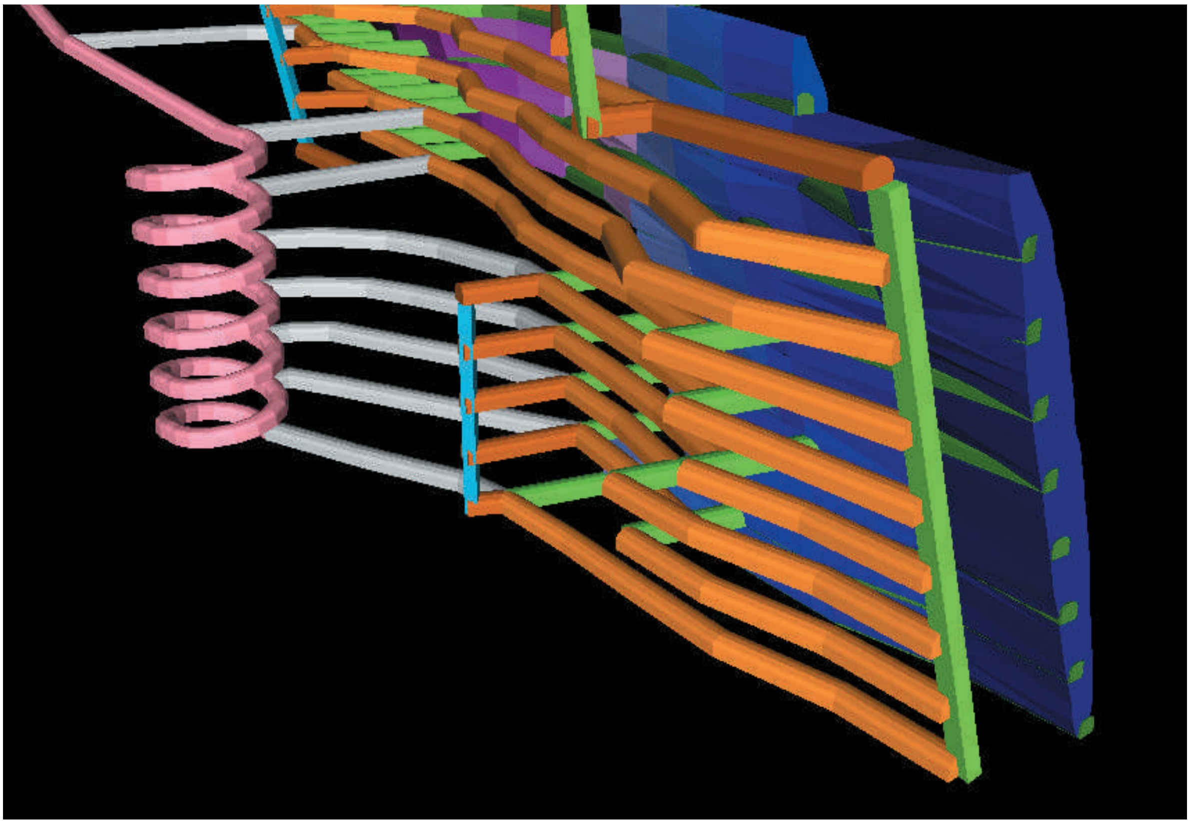


Figure 1.14: Main Access Decline (Pink), Footwall Drives (Orange) and Ventilation Raises Connected to Each Level (Green)

1.5.3.2 Ore Reserve

Based on the Mineral Resource statement prepared by WAI in November 2010, WAI has estimated Suzdal underground *Ore Reserves*. The estimation considered only *Measured* and *Indicated* Mineral Resources.

Global mining factors of 15% dilution and 95% mining recovery have been applied on a basis of historically achieved indices together with forecasted dilution within stopes, planned for extraction in later periods. Detailed results of this estimation are given in Table 1.7 below.

Table 1.7: Suzdal Underground Ore Reserves as of 01.11.2010 (WAI) (in accordance with the guidelines of the JORC Code (2004)			
Block (Stope) Cut Off Grade (g/t)			**4.0**
Proven	**Tonnage(kt)**		266*
	Au (g/t)		7.57
	Metal	**kg**	2,014
		oz	64,752
Probable	**Tonnage (kt)**		2,821
	Au (g/t)		6.83
	Metal	**kg**	19,263
		oz	619,319
Proven + Probable	**Tonnage (kt)**		3,087
	Au (g/t)		6.89
	Metal	**kg**	21,277
		oz	684,070
Note: 15% dilution and 95% mining recovery applied. *Includes 100kt at a grade of 5.4g/t Au in stockpiles			

1.5.3.3 Life of Mine Schedule (LOM)

After completion of the Ore Reserves estimation, WAI has designed a mining schedule. The mining schedule corresponds to the company production plans for the last two months of 2010 to 2012, and the further years of the LOM plan are provided on the basis of remaining Ore Reserves. The Ore Reserve statement was prepared by WAI as of 01.11.2010. Table 1.8 below describes the mining schedule for the Suzdal deposit.

Table 1.8: Suzdal Deposit LOM Mining Schedule (WAI 2010)								
Year	Nov-Dec 2010	2011	2012	2013	2014	2015	2016	Total
Ore (kt)	76	492	519	500	500	500	500	3,087
Au (g/t)	9.3	7.74	6.65	6.65	6.65	6.65	6.65	6.89
Au (kg)	709	3,808	3,453	3,327	3,327	3,327	3,327	21,277

Note: 15% dilution and 95% mining recovery applied
Year 2011 production includes 100kt of ore from stockpiles at an average grade of 5.4 g/t Au

> ***WAI Comment:*** *the LOM schedule is based on Measured & Indicated resources which have been transferred to reserves through the application of dilution and recovery. However, WAI believes that a significant proportion of the existing Inferred resources will ultimately also transfer through to reserves to extend the LOM beyond 2016.*

1.5.4 Mining Procedures and Infrastructure

1.5.4.1 Drilling and Blasting

Drilling at Suzdal is performed using a single-boom Sandvik DD210 drilling jumbo for capital development, and hand held rotary drill and air-legs for other development.

Blasting occurs four times per day and utilises granulated ANFO explosive initiated with non-electric detonators.

1.5.4.2 Ore and Waste Transportation

After blasting has taken place, both ore and waste are transported to the surface using dump trucks (30t capacity), and stockpiled in the open pit, close to the portal of Decline No.1.

From the stockpile in the open pit, ore is transported to the main crusher, and after crushing is re-stockpiled prior to transportation to the processing plant.

> ***WAI Comment:*** *as described above, material is handled in four separate stages and WAI considers that streamlining the process by removing one of the handling stages may be advantageous, although the use of additional stockpiling does provide surge capacity for the system.*

1.5.4.3 Mining Equipment

Maintenance of mining equipment takes place according to a maintenance schedule, which follows manufacturers' recommendations. Workshop facilities are present on the Suzdal site; however these were not examined by WAI during their site visit.

> ***WAI Comment:*** *WAI considers that whilst the type and quantity of mining equipment in use at Suzdal is generally appropriate for the operation, production and development would be greatly improved by the introduction of additional drilling jumbos to negate the necessity of using hand-held drill rigs where possible.*

1.5.4.4 Ventilation

The majority of underground excavations at Suzdal require ventilation. Ventilation varies between $4m^3/s$ and $35m^3/s$ in the main underground workings, with ventilation levels appropriate for development.

> ***WAI Comment:*** *based upon the WAI visit to the underground workings at Suzdal, in addition to review of data and information provided, WAI is of the opinion that ventilation at the site is appropriate with the underground workings provided with sufficient airflow*.

1.5.4.5 Pumping and Water Management

Whilst water inflow to the mine does occur, the hydrological conditions underground are generally acceptable, with appropriate water management and drainage system in place to control mine water.

1.5.5 Infrastructure

1.5.5.1 Power

Power to the Suzdal Mine site is provided by two independent overhead power lines connected to the local power grid at the local village of Znamenka, one acting as a main supply, the other as a reserve. The power supply for the site is 10kV and is distributed to the required areas via overhead cables and substations/transformers.

1.5.5.2 Accommodation & Offices

The principal accommodation and office facilities are divided over two areas at Suzdal. Office facilities are provided in two buildings located close to the entrance of the mine site, with an accommodation and personnel complex located separately, some 3km from the open pit and Decline No. 1 portal.

1.5.5.3 Surface Roads

A simple haul road network is in place on the Suzdal site, providing access between the site entrance and the main operations areas. Roads are generally well delineated, despite snow cover appeared well maintained and suitable for purpose.

1.5.5.4 Maintenance & Stores

In order to maintain production levels at Suzdal, a stock of spares and consumables is maintained for both mining and processing equipment. Spares are now ordered on a month-by-month basis, with a one month supply of spares and equipment maintained as a minimum and procurement of more specific items carried out on an as-required basis.

1.5.6 Fuel

Fuel for vehicles and equipment at Suzdal is supplied on a regular basis and stored in designated petrol and diesel fuel storage tanks. Storage facilities comprise four 50m^3 tanks for diesel and a 100m^3 tank for petrol.

1.5.6.1 Explosives

Explosives at Suzdal are currently stored in a 24t capacity storage facility which includes fencing and security points as required under Kazakhstan regulations, and the entire enclosure is located 500m from the nearest structures.

1.5.6.2 Water Supply

Portable and technical water requirements, supplied to the mine from two boreholes which tap a local underground water source and amounting to 157.6m^3/h technical water and 126m^3/h fresh water, are less than half of what is available in terms of the capacities of the water sources used.

1.5.6.3 Militarised Mine Rescue

Suzdal mine has a service agreement in place with a militarised mine rescue service to provide salvage and rescue in the event of an emergency during operations, as required according to Kazakh regulations. The rescue service also implements controls and monitoring procedures relating to hazardous substances, and ensures compliance with safety management rules.

1.5.7 Conclusions

Suzdal is a mature operation that is well set up with all necessary infrastructure, labour and other facilities. Although small scale hand-held drills are used in some areas of the mine, the operation is still capable of meeting scheduled production, with an additional drill jumbo on order to allow production to be increased as planned.

1.6 Processing

1.6.1 Introduction

The Suzdal ores are refractory to cyanidation and the plant is one of the few in the world that uses the Biox® process (Biox) and one of the few operating the process in extreme cold conditions.

The ore is crushed and ground using a conventional crushing and ball mill circuit and a gold-rich sulphide concentrate is produced by froth flotation. The flotation concentrate is oxidised using the Biox process and the oxidised concentrate is washed and leached using conventional CIL technology.

1.6.2 Laboratory Testwork

There are several orebodies at Suzdal, each with differing ore mineralogy and host rock types. The results of several programmes of diagnostic leaching have been undertaken to quantify the gold associations (Figure 1.15 and Table 1.9).

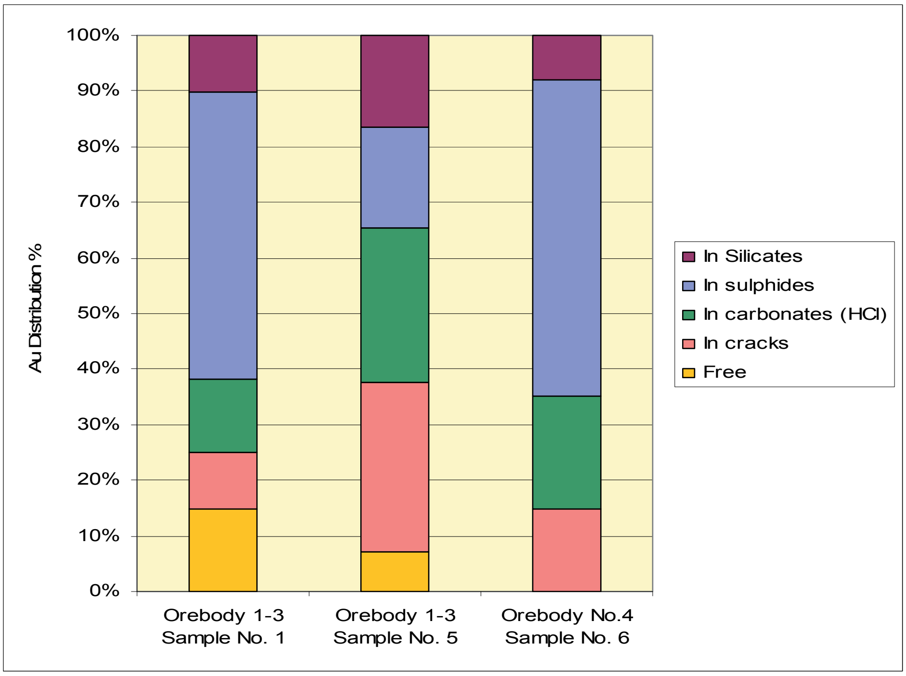


Figure 1.15: VNIITsvetmet Rational Analysis Test Results

Table 1.9: VNIITsvetmet Rational Analysis Test Results			
Gold Associations	**Orebody 1/3 Sample No 1**	**Orebody 1/3 Sample No 5**	**Orebody No.4 Sample No 6**
Free	14.9	7.1	0.0
Cyanidable	10.2	30.6	14.8
In carbonates	13.2	27.7	20.5
In sulphides	51.5	18.2	56.8
In Silicates	10.2	16.5	8.0
Total	100.0	100.0	100.0

The results of a second programme of testing are given in Figure 1.16 and Table 1.10.

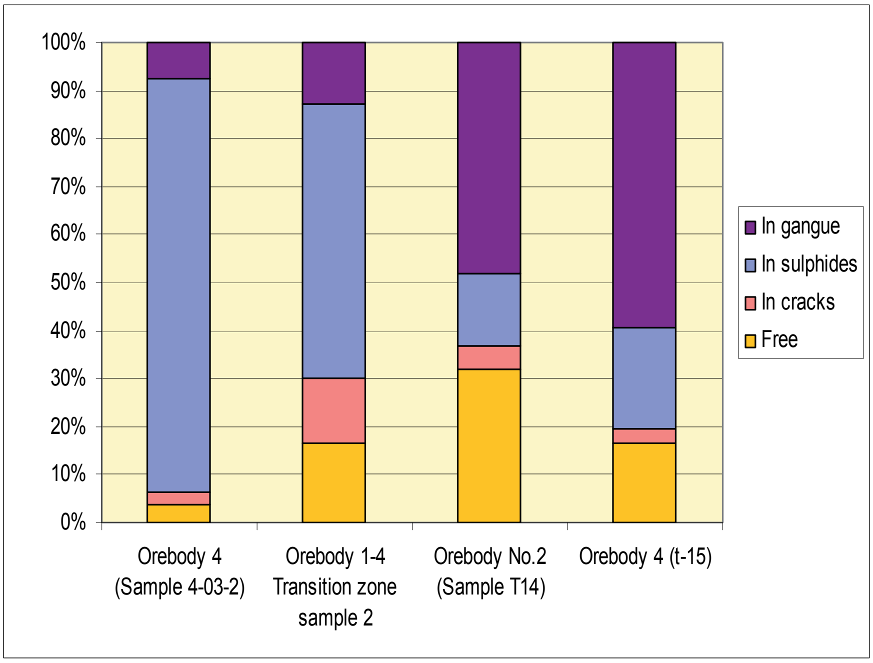


Figure 1.16: Further Diagnostic Leach Test Results

Table 1.10: Further Diagnostic Leach Test Results				
Gold Associations	**Orebody No.4 Sample 4-03-2**	**Orebody 1-4 Transition Zone Sample 2**	**Orebody No.2 Sample T-14**	**Orebody 4 Sample T-15**
Free	3.8	16.4	31.9	16.4
Cyanidable	2.7	13.6	5.0	3.3
In sulphides	85.9	57.1	15.0	20.8
In gangue	7.6	12.9	48.2	59.6
Total	100.0	100.0	100.0	100.0

In this exercise, no differentiation was made between gold in carbonates and gold in silicates, with the total of these two associations being referred to as "gold associated with gangue minerals". The levels of free gold ranged from 3.8% to 31.9%. Recovery of gold associated with gangue minerals by flotation will be difficult or impossible.

1.6.3 Process Description

1.6.3.1 Stockpile and Crushing

Ore is transported to a stockpile and crushed using a three stage crushing plant. The ore is reduced in size from a maximum of 500mm to 100mm. The crushed product is conveyed to a screen fitted with 100mm and 15mm decks. The +15mm passes to a 1,750mm cone crusher and the product is conveyed to a KMD-1200 tertiary crusher. The crushed product is conveyed to a 15mm screen and the oversize is returned to the crusher. The plant is rated at 65tph.

1.6.3.2 Grinding

Ore is ground in one of two single stage ball mill circuits, the discharge is pumped to cyclones and the underflow returned to the mill whilst the overflow passes to flotation. The target grind size is 80% passing 74 microns.

1.6.3.3 Flotation

3-staged flotation is used as follows:

Stage 1: Flotation proceeds in a group of 6x6.3 m^3 cells for the first rougher flotation with the tails going to the SAG mill.

Stage 2: The SAG mill overflow goes to a group of 7x6.3m^3 cells for second rougher flotation with the tails recycled to the SAG mill. Concentrate is produced from the first two cells, whilst concentrate from the other cells goes to cleaner flotation.

Stage 3: The SAG mill overflow goes to the first main 40m^3 cell of the new Outocumpu flotation section, whilst the tails go to two cells of 2nd rougher flotation, each of them is 40m^3. The first rougher flotation concentrate (according to its quality) is either concentrate or goes for cleaner flotation. The second rougher flotation concentrate goes to the cleaner flotation. The second rougher flotation tails go to the first scavenger flotation cell. Concentrate goes to the cleaner flotation. The first scavenger flotation tails go to two 40m^3 cells of the second scavenger flotation. The second scavenger flotation tails go to the tailings dam. Concentrate goes back to the first cell of the second rougher flotation. The cleaner flotation section has two stages. The first stage has 10x1.2m^3 cells. The concentrate from which goes to the second cleaner consisting of 6 x1.2m^3 cells. The first cleaner tails go back to the first cell of the second rougher flotation, whilst the second cleaner tails go back to the first cleaner cells.

1.6.3.4 Biox Plant

The flotation concentrates pass to the Biox Plant via a 242m^3 surge tank. The plant is designed to treat 8tph of concentrate assaying 12% sulphur and 4.2% arsenic, with a 90% availability. This amounts to the oxidation of some 23t of sulphur per day.

The flotation concentrate, at 20% pulp density is inoculated and the pulp is split into three streams which pass to three Primary leach reactors in parallel. There are three inoculum tanks of 0.1m^3, 1.0m^3 and 30m^3 capacity.

The products from the Primary leach reactors are combined and pass to three Secondary reactors in series. The residence time for both Primary and Secondary oxidation is two days giving a total of four days. The volume of each reactor is 643m^3.

The Biox pulp process parameters are: temperature 35-45°C, pH 1.2-1.8 and Eh 560 mV.

The exothermic nature of the sulphide oxidation process requires the use of a three-section cooling tower. Even at temperatures of -45°C, one of the sections needs to be operated to cool the pulp stream.

1.6.3.5 Counter Current Decantation (CCD)

The acid water in the bio-oxidised pulp is removed via a three stage CCD circuit. The acidic water is added to the flotation tailings in the Neutralisation section of the plant and the final washed Biox residue product goes to the carbon in leach (CIL) processing.

1.6.3.6 Neutralisation

The flotation tailings contain carbonate minerals and thus possess a neutralising capacity. The acid water produced from the CCD section is thus mixed with flotation tailings and neutralised in six 98m^3 tanks. Lime can be added if the tailings contain insufficient neutralising minerals. The neutralisation product is pumped to a water recovery thickener and the thickened pulp is pumped to the flotation tailings dam.

The final CCD product is pumped to one of two 50m^3 Ultrasep thickeners. The pH of the thickened pulp is increased to 11 and the pulp passes to the CIL section.

1.6.3.7 Carbon in Leach (CIL)

Cyanidation of the bio-oxidised flotation concentrate residue takes place in seven 85m^3 leach tanks. The free cyanide concentration in the first tank is 1,500ppm and this falls to 600ppm in the last tank. The solids concentration is 33% w/w. Activated carbon is added at a rate of 50g/l and leach residence time is 28 hours.

1.6.3.8 Desorption and Electrowinning

Gold is recovered from the loaded carbon using standard desorption-electrowinning-smelting technologies. The loaded carbon, containing up to 3.5kg/t Au is stripped in batches of 2.5t.

The eluate passes via a heat exchanger to one of two rubber lined electrolytic cells, each of 2.2m^3 capacity. The cathode sludge is smelted to produce a Doré product containing 95% Au and up to 5% Ag.

1.6.3.9 Labour

The plant is operated using a 15 day on-15 day off staff rotation. There are a total of 168 employed in the processing plant, working on 12 hour shifts.

1.6.3.10 Consumables

The complex nature of the Biox process is reflected in the high levels of chemicals required. Sulphuric acid requirement is 216.5kg/t of concentrate which equates to 43kg/t of ore.

1.6.4 Historic Production Data

The historic production data for the Suzdal Plant are given in Table 1.11.

Table 1.11: Suzdal Plant Production

	Ore Treated	Head Grade	Plant Overall
Year	**Tonnes**	**Au g/t**	**Recovery %**
H1 2010*	144,692	11.3	**77.9**
2009	317,448	13.38	**74.8**
2008	302,416	14.5	**71.8**
2007	263,294	15.1	**60.3**
2006	264,827	9.8	**54.76**
2005	151,918	8.1	**50.0**

*as of 01.07.2010

Flotation recovery has increased from 70% (2005) to 84.39% in 2009 and to 87.17% to date in 2010. CIL recovery has also increased over the period from 82.9% in 2005 to 87-86% in 2009 and 87.40% in 2010 year to date.

1.6.5 Plant Expansion

It is planned to increase the throughput of the Suzdal plant to 500ktpa. This will involve the addition of further grinding, Biox, CIL and elution equipment although there will be no expansion of the electrowinning or smelting sections.

The capital cost for the plant expansion was US$18.4M. As of November 2009, a further expenditure of US$3.0M was planned for 2009 and US$7.54M in 2010

1.6.6 Metallurgical Laboratory

There is a small metallurgical laboratory located within the plant. The plant also undertakes ferrous/ferric analysis for Biox plant control.

1.6.7 Assay Laboratory

There are three contract laboratories, accredited to ISO 17025 until 2011 on site, two essentially for plant samples and the other for geological and mining samples. Samples analysed on a daily basis include plant feed, CIL feed, Biox feed, flotation feed and tailings and CIL tails. The limit of detection is 0.1ppm Au.

The laboratory operates a comprehensive QA/QC programme which involves duplicate analysis, the use of blanks and the regular analysis of standard reference materials. These are sourced for "Geostats" and from local institutes. A system of check analyses with external laboratories is also undertaken.

1.6.8 Technical Control Department (OTK)

Sampling of the plant is the responsibility of the Technical Control Department (OTK). The results of the sampling exercise are used to prepare daily metallurgical balances

The OTK also calculates the amount of gold locked within the plant circuit which, due to the long Biox residence times, can amount up to 100kg. The OTK is also responsible for calibrating the truck weigh scales and mill feed weightometers.

1.6.9 Conclusions

The Suzdal plant successfully utilises the Biox process and is one of the very few in the world to do so in conditions of extreme cold. It is interesting to note that, of the flotation, Bio-oxidation and CIL processes, Bio-oxidation is the most efficient operation, but carries a very high operating cost, due primarily to the costs of reagents and power. The flotation process is hindered by highly variable ore mineralogy and an association of a proportion of the gold with carbonate and silicate gangue minerals. CIL recoveries, at 87%, could probably be improved.

1.7 Environmental & Social

1.7.1 Environmental Setting

The site has a 1,000m Sanitary Protection Zone for class 1 hazards, and 300m for the domestic landfill site (class 3). Studies state that there is poor faunal representation at the site, with no Red Book species present, and no archaeological areas of interest. Radioactivity levels at the site are considered to be normal.

1.7.2 Review of Environmental/Social Studies

An OVOS (Russian equivalent to Environmental Impact Assessment (EIA)) produced in 2003 was already in place for the Suzdal mine, to cover the initial mine production and including the BIOX (bio oxidation) plant technology. The planned extension of the mine was not covered by this OVOS, and a further study has recently been completed and submitted for State approval.

1.7.3 Environmental Monitoring and Compliance

Environmental Monitoring plans, prepared by a registered contractor (as stipulated by State requirements) were prepared and approved for 2008-2009, and accompanying discharge and emission limits were also set. The 2010-2011 monitoring plan is currently with the Environmental Ministry for review and approval.

Water quality is also sampled quarterly for a full suite of determinands. Samples do not generally show elevated concentrations; however it was reported to WAI that samples in the vicinity of the TMF have occasionally displayed traces of cyanide, however these have not exceeded permitted levels. No surface water flows are present at the site.

Soils are sampled once per year and are assessed for Cu, Zn, Ni, As, Mn, fluorides, and nitrates. The most recent report has been submitted and is being assessed by State authorities.

Noise monitoring, and vibration and light pollution measurement are carried out annually.

Air monitoring is carried out at sources of emissions on a quarterly basis. Depending on the location, determinands include soot, NO2, SO2, CO and HCN and in some areas, benzene vapour, dusts, hydrocarbons and suspended solids are also included.

Continuous gas monitoring occurs in hazardous chemical storage areas, and in the plant.

1.7.4 Water Management, Sourcing, Consumption and Contamination

There is no water discharge at Suzdal, with a closed loop system. Water from the TMFs and the mine shaft is used for process water in the plant, and supplemented as necessary with water from boreholes on site. Potable water is sourced from boreholes at the site.

All environmental studies produced to date conclude that no potential environmental impacts are likely to arise from the project activities.

1.7.5 Air Emissions

The main air emissions at the site are surface dusts, and during the summer season, water bowsers are provided for dust suppression, however dust generation was noted during the WAI site visit.

1.7.6 Tailings Management

Tailings from the flotation circuit and the CIL circuit are stored in separate, adjacent TMFs, with three storage areas for flotation tails, and two for CIL tails. Structures for the life of the project and an extension are planned, which should provide for the operation up to 2016. It was reported that ultimately tailings are one day intended to be reprocessed, but a closure plan for this facility is also in place.

1.7.7 Liquid Effluent

A large sewage treatment system intended to receive effluent that has passed through a natural filtration system has been constructed at the site, but was due to be commissioned within two months of the site visit. It was reported that sewage has been discharged to surface for the past year under permit.

An emergency pond is also located in the vicinity of the CIL plant, and appropriately fenced.

1.7.8 Waste Management

Scrap metal is stockpiled on rock lined bases, and sold. Mercury bulbs are disposed of via a contractor, since these are a class I hazard. Cyanide drums are neutralised and buried in the old HLPs, which was an officially approved as a process. Industrial waste, such as tyres etc. is sent to the site industrial landfill. Used oil is also stored and sold. Office waste is sent to the site domestic landfill, built to an approved design.

1.7.9 Handling and Storage

The fuel and gas storage station contains two 35m^3 capacity underground petrol tanks and two 50m^3 capacity underground diesel tanks.

There is a hazardous and non-hazardous material store at the site, for various substances including cyanide. The area is alarmed and ventilated, and daily logs are kept of delivery and use. Used containers are decontaminated in the plant, and collected and disposed of by a licensed contractor.

There is an explosives magazine at the site, located in the base of the old open pit and approximately 20t of explosives are stored on the site at one time. The storage licence is applied for annually, and security for the storage facility is provided by State agencies.

1.7.10 Environmental Liability

Since the mine has been operated by JSC FIC Alel since 1995, there are not considered to be any historic liabilities associated with the site. Under Kazakh law, mine closure plans are required to be prepared 2 years before mine closure. Mine closure and remediation plans should be sufficiently rigorous to ensure that potential long-term environmental liabilities are minimised.

1.7.11 Mine Closure and Rehabilitation

It was reported to WAI that a closure plan has been developed, and every year monies are sent to an external fund (held at a bank) to make provision for closure costs. Figures of 0.5% of mining costs per year, rather than actual closure cost estimates are set aside annually. As part of the subsoil use contract, a closure plan is required. Progressive rehabilitation is also to be carried out.

The closure fund for the mine, which is believed to have been set up in 1995/1996 currently contains 31,396,652 Tenge (approx. US$213K or £120K).

It is possible that changes in the regulations will change the closure requirements and planning process during operations, given the longevity of the mine.

1.7.12 Conclusions

The following is a list of WAI's main conclusions with regard to the Suzdal operations:

- Existing environmental studies and environmental management are produced in line with national requirements, but would fall short of international standards;
- The current environmental monitoring plan for the site satisfies national requirements but would require expansion to meet international best practice;
- There is no formalised stakeholder engagement system or grievance mechanism in place, although community initiatives have been developed;
- Housekeeping and maintenance of areas were generally good at the site;
- There is potential for soil and groundwater contamination to occur in the fuel storage area;
- The potential for Acid Rock Drainage from waste rock and tails is not being assessed at the site;

- There is significant dust generation at the site, despite management measures and no erosion control plan is in place;
- The TMFs appear well managed, though some cracks were noted in the plastic liner, and cyanide levels have been reported; however these do not exceed maximum permissible concentration levels;
- The current sewage disposal at the site is not considered acceptable, since a treatment system had not been commissioned at the time of the visit;
- Waste management at the site is generally good although some poor practices were noted;
- The environmental budget for the site is sometimes exceeded due to unplanned expenditure;
- Fire protection and health and safety are being well managed at the site;
- There is asystem for assessing and managing potential environmental liabilities at the site under Kazakh regulations, but this does not come up to international standards;
- The current Mine Closure and Rehabilitation Plan may satisfy national requirements but is not adequately developed to satisfy international standards, and does not include post closure requirements, but it is understood that this will be improved closer to the time of closure;
- While the permit includes certain closure costs, the Company does not fully rely on these figures and will properly address the need for adequate costs, monitoring and treatment requirements, including post-closure monitoring despite the early stages of operations.

1.7.13 Recommendations

The following are WAI's main recommendations with regard to the Suzdal site:

- An ESIA addendum and social needs assessment should be developed;
- The Environmental Manager at the site should be supported in her task by expanding the environmental management team;
- The company should continue the implementation of the ISO 14001 management system;
- An in house monitoring data base should be established, and the company should develop a closer working relationship with the environmental contractors;
- A formal PCDP and grievance mechanism should be established;
- An overall water balance should be produced for the operations;
- The transformer station should be included in monitoring plans, and an energy audit, to identify savings should be produced;
- The fuel storage areas should also be included in the company monitoring plan;
- Waste rock and tailings material should be subject to testing for ARD potential, and appropriate management systems should be developed accordingly;
- Where possible, stockpiles should be covered, and wheel washes should be developed to minimise dust generation, in addition to current measures. An erosion control plan should also be developed, relating to both wind and water erosion;
- The long term stability of the TMF site should be assessed, and any elevated cyanide levels investigated;
- The commissioning of the planned sewage treatment system should be accelerated, and interim disposal options should be sought;
- Alternative waste disposal and recycling options should be sought at the site, and a raw materials and waste audit should be performed;
- Regular meetings should be held with operational managers and the environmental budget should be updated on a 6 monthly basis;
- The OHSAS 18001 accreditation process should be resumed;
- A system to evaluate environmental liabilities should be developed, and the company should seek to minimise the generation of environmental liabilities at the site;

- A detailed MCRP should be developed and regularly updated through the life of operations, to include post closure monitoring and treatment;
- A mine closure cost estimate should be produced in line with actual closure costs and post closure monitoring and treatment requirements, and amended sums should be set aside accordingly; and
- The social development plan should be reviewed in line with international requirements.

WAI recognises that to date the company has only sought to acheive national compliance, which has been done, and hence many of the above recommendations have hitherto not been required. However, WAI recognises the company's commitment to achieving international compliance, and highlights the issues above as areas where further actions will be required.

1.8 Life of Mine Plan

1.8.1 Introduction

A life of mine plan for the Suzdal gold deposit has been prepared in order to demonstrate the potential viability of the project and its robustness. This evaluation was based on the *Ore Reserve* figures described above, capital investment requirements and estimates provided by the Client (fully audited by WAI paying cognisance to similar projects), and other parameters. Parameters were input to the life of mine model, and details of this model are provided below.

1.8.2 Life of Mine Assumptions and Input Data

The assumptions made in the WAI financial model for the Suzdal Gold project were based on:

- Au price of US$900/oz;
- Ore reserves as of 01.11.2010, estimated by WAI;
- Mining Schedule, prepared by WAI. The Mining Schedule targeting a 0.5Mtpa ore mining rate with appropriate ramp-up of production; Nov-Dec 2010 ore tonnage being estimated on the basis of the Company production plan;
- Long-term operating costs forecasts based on year 2008-2010 production results and considering analogous deposits;
- Overall Au recovery indices obtained from actual annual results for 2008-2010;
- Annual discount factor of 11.2% (Base case);
- Capital expenditures, General and Administration costs estimated on the basis of Client's assumption and considering previous periods; and
- Overall tax at 20% of net income applied.

Suzdal is an operating mine, therefore the major capital investments have been put in place. Capital requirements, provided by the Client, were assessed considering the on-going processing plant expansion, which has been highlighted in the LOM schedule and essential expenditures in order to keep the mine operating. Depreciation estimation was based on the Client's assumptions, considering expenditures made in previous periods.

A summary of the Suzdal financial model assumptions and input data is given in Table 1.12 below.

Table 1.12: Suzdal Life of Mine Model Assumptions and Input Data

Year	*TOTAL*	Nov-Dec 2010	2011	2012	2013	2014	2015	2016
Gold Price, US$/Oz		900	900	900	900	900	900	900
Ore production (diluted), kt	***3,087***	76	492	519	500	500	500	500
Ore Mining cost, US$/t	***33***	33	33	33	33	33	33	33
Mining OPEX, k US$	***101,871***	2,508	16,236	17,127	16,500	16,500	16,500	16,500
Au Grade, g/t	***6.89***	9.30	7.74	6.65	6.65	6.65	6.65	6.65
Gold mined, kg	***21,275***	707	3,808	3,453	3,327	3,327	3,327	3,327
Recovery, %	***80.0%***	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
Gold recovered, kg	***17,020***	565	3,046	2,762	2,661	2,661	2,661	2,661
Gold recovered, Oz	***547,200***	18,179	97,946	88,816	85,565	85,565	85,565	85,565
Processing cost, US$/Oz		171	206	240	240	240	240	240
Proc and Mining Costs US$/Oz		309	372	432	432	432	432	432
Revenue, k US$	***492,480***	16,361	88,151	79,935	77,008	77,008	77,008	77,008
Additional Concentrate Processing Earnings	***49,700***	1,700	8,000	8,000	8,000	8,000	8,000	8,000
Operating Costs, k US$	***228,438***	5,624	36,408	38,406	37,000	37,000	37,000	37,000
G&A, Sales, k US$	***62,000***	2,000	12,000	12,000	12,000	12,000	6,000	6,000
Royalty, 5%, k US$	***30,783***	1,023	5,510	4,996	4,814	4,814	4,814	4,814
CAPEX, k US$	***37,816***	7,940	10,876	5,000	5,000	5,000	3,000	1,000

1.8.3 Company Financial Model Assumptions and Input Data

WAI has reviewed the company's Life of Mine (LOM) model (see table below) which assumes a conservative continuing replenishment of reserves of the mine during the next few years as the result of upgrade of *Inferred* Mineral Resources into higher category with subsequent conversion into Ore Reserves. The Company's LOM model excludes any additional mineralised potential that could be identified to further supplement and increase the resource base in the future. This LOM model allows extension of the period of operation until at least 2019.

WAI has reviewed the Company's LOM production plan, and though it is not currently based on Ore Reserves, estimated in accordance with the guidelines of the JORC Code (2004), considers the assumptions used to be reasonable.

Table 1.13: Company's Financial Model Assumptions and Input Data

Year	***TOTAL***	2011	2012	2013	2014	2015	2016	2017	2018	2019
Gold Price, US$/Oz		900	900	900	900	900	900	900	900	900
Ore production (diluted), kt	4,303	500	500	500	500	500	500	500	500	303
Ore Mining cost, US$/t	33	33	33	33	33	33	33	33	33	33
Mining OPEX, k US$	141,895	16,489	16,489	16,489	16,489	16,489	16,489	16,489	16,489	9,983
Au Grade, g/t	7.54	7.74	7.54	7.54	7.54	7.54	7.54	7.54	7.54	7.54
Gold mined, kg	32,527	3,870	3,768	3,768	3,768	3,768	3,768	3,768	3,768	2,281
Recovery, %	80	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
Gold recovered, kg	26,019	3,096	3,014	3,014	3,014	3,014	3,014	3,014	3,014	1,825
Gold recovered, Oz	836,538	99,538	96,905	96,905	96,905	96,905	96,905	96,905	96,905	58,665
Processing cost, US$/Oz	211	206	211	211	211	211	211	211	211	211
Proc and Mining Costs US$/Oz	382	371	382	382	382	382	382	382	382	382
Revenue, k US$	752,880	89,584	87,214	87,214	87,214	87,214	87,214	87,214	87,214	52,798
Concentrate Processing Earnings, k US$	72,000	8,000	8,000	8,000	8,000	8,000	8,000	8,000	8,000	8,000
Operating Costs, k US$	318,193	36,976	36,976	36,976	36,976	36,976	36,976	36,976	36,976	22,385
G&A, Sales, k US$	102,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000	6,000
Royalty, 5%, kUS$	47,057	5,600	5,451	5,451	5,451	5,451	5,451	5,451	5,451	3,300
CAPEX, k US$	42,876	10,876	5,000	5,000	5,000	5,000	5,000	5,000	2,000	-

1.8.4 Conclusions

The WAI life of mine model results in a positive NPV at nominal input parameters. It shows that the reserves considered in the model are profitable for exploitation in the current economic environment.

The deposit consists of large Mineral Resource, although a substantial portion of this falls into Inferred category. Due to differences between the Kazakh resource estimation system and JORC, a considerable portion of this Inferred material has been considered in the Company's production plan.

However, under JORC, Inferred material cannot be transferred to reserves, although WAI believes that much of this material will ultimately be up-graded through additional drilling, and subsequently included in the reserve base and LOM, thus increasing the project lifetime.

2 ZHEREK

2.1 Background

2.1.1 Introduction

Wardell Armstrong International (WAI) was commissioned by LLP "Zherek" ("Zherek") to undertake a technical due diligence of the Zherek open pit gold mine, near Semipalatinsk, northeastern Kazakhstan.

A team of WAI consultants visited the Zherek mine during the period 14 - 15 November 2009, as well as the Zherek office located in Semipalatinsk.

The review of the Zherek mine was based on data, professional opinions and unpublished material submitted by the professional staff of Zherek and their subsidiaries.

2.1.2 History

The Zherek deposit lies in the Semipalatinsk Oblast of northeastern Kazakhstan some 36km to the southwest of Semipalatinsk, Figure 2.1 and Figure 2.2.

The deposit was discovered in 1968 and by 1978, the investigation of the oxidised gold-bearing ores was completed. From 1979 until 1995, investigation of the oxide portion of the deposit by AltaiZoloto culminated in TKZ-approved reserves, at cut-off grades varying from 1-5g/t Au, of some 618kt containing 3.2t Au with an average grade of 5.2g/t Au.

In 1995 mining works stopped, in 1997, Zherek LLP investigated the concept of heap leaching the remaining oxide ore (2.1Mt @ 2.6g/t Au) with full production commencing in 2003 and in December 2007, Zherek took 100% ownership of the company.

2.1.3 Access

Access to the Zherek site is very straightforward from Semipalatinsk, with the first 26km on asphalt road, followed by 10km on a reasonably well graded dirt road to the west. Total driving time from Semipalatinsk is approximately 40 minutes.

Figure 2.1: Location of Zherek in Northeastern Kazakhstan Russia

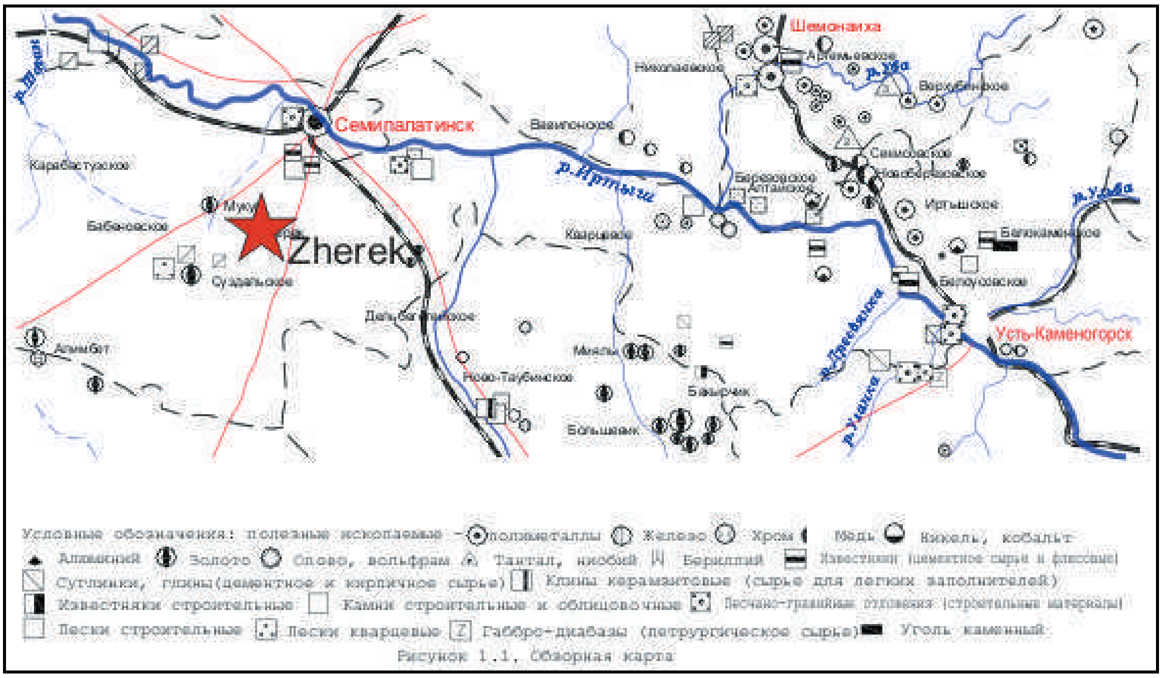


Figure 2.2: Location of the Main with Respect to Semipalatinsk

2.1.4 Topography and Climate

The relief of the region is characterised by a gently rolling steppe which gradually descends to the plain of the Irtysh River to the north. Here, absolute elevations do not exceed 250-260m, whilst the slightly hillier area around the mine has elevations up to 310m on isolated hills.

The area has very poor outcrop as much of it is covered by loose Cenozoic formations and there are few steep or rugged slopes.

The climate of the region is typical continental, with a maximum summer temperature of +23°C to +42°C, and minimum winter temperatures from -25°C to -40°C. Precipitation is generally light at 330mm per year, with snow from mid November, giving total winter accumulations of 25-30cm. The freezing depth is 1-1.5m.

2.1.5 Infrastructure

The infrastructure at Zherek mine is sufficient to operate a relatively small, open pit, heap leach operation, and comprises open pits, small offices, heap leach pads, plant, mechanical workshop, accommodation + gym.

2.1.6 Mineral Rights and Permitting

The original licence for the Zherek property encompassed some 1.031km^2, and allowed exploitation to +120m. Due to the discovery of an extension to the oxide mineralisation in the northwest of the current pit, an application was made to extend the original licence from 1.031km^{2}to some 1.35km^2 in area and to include mineralisation down to +150m. This licence was approved on 10/9/09 and is delineated by the following co-ordinates (Table 2.1).

Table 2.1: Zherek Licence Co-ordinates 10/09/09

Point	Northing	Easting
1	50°09’55.4”	80°00’49.3”
2	50°10’27”	80°01’11”
3	50°09’47.2”	80°02’16”
4	50°09’31.3”	80°02’03.5”

WAI Comment: *an inspection of the licence documentation has shown that it is in good order and suitable for the future needs of the Company.*

2.2 Geology and Mineralisation

2.2.1 Regional Geology

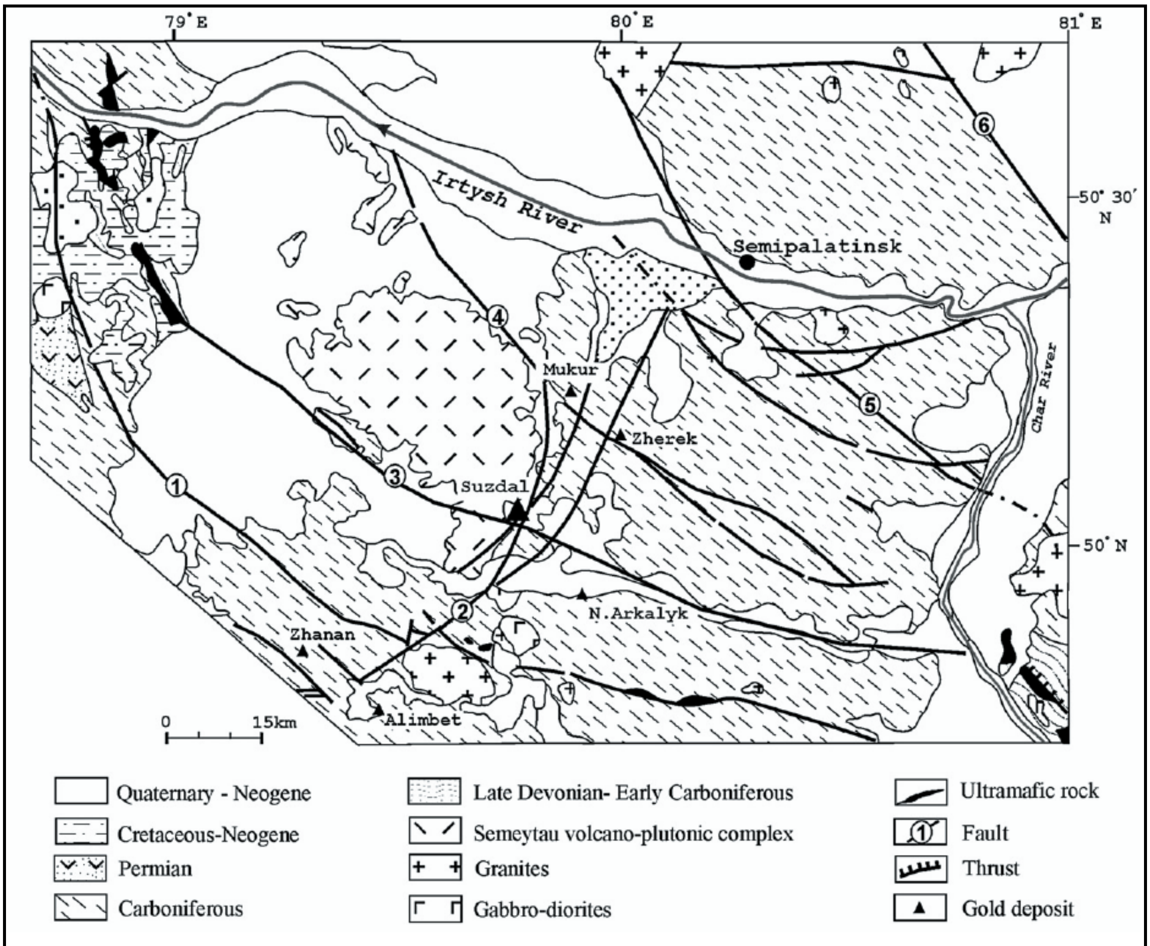
2.2.1.1 Geology of the Charsk Gold Belt

Eastern Kazakhstan, including the Charsk Gold Belt, is mostly underlain by fragments of Kipchak and Altaid paleo-island-arc systems. The region is characterised by ophiolite slivers separated by marine sedimentary basins.

Between Mesozoic and Eocene times, the region underwent deep surface weathering and chemical erosion producing the current weathering profile. Surface oxidation reaches an average thickness of approximately 30m, except in proximity to fault and alteration zones where the surface oxidation reaches up to 100m.

Quaternary loams, clays and sands cover a very large portion of the Charsk Gold Belt area (approximately 70%).

Figure 2.3 shows the detailed geology of the Zherek area.



Main Fault Zones: 1 Baiguzin-Bulak; 2 Suzdal; 3 Char-Gornostai-Zimunai; 4 Mukur; 5 Western Kalba; 6 Terecty-Ulungu

Figure 2.3: Detailed Geology of the Zherek area

Magmatism, hydrothermal alteration and metallic deposits are all controlled by a dominant northwest-southeast regional structure, where intrusive bodies are generally preferentially emplaced along major faults. Within the Charsk Gold Belt, most of the known gold deposits are associated with NW-SE trending fault zones and subsidiary structures and exhibit a strong spatial relationship with felsic and mafic intrusions.

In general, major gold deposits are located within:

- Fragments of fore-arc and back-arc basins, primarily turbidite, calcareous sand carbonaceous sandstones and siltstones (e.g. Bakyrchik, Mukur, Mialy, Kostobe, Zherek); and
- Along the margin of fragments of accretionary prism, hosted by limestone and calcarenite, basalt, cherty rocks, calcareous and carbonaceous sandstone and siltstone (Suzdal, Mirage, North Arkalyk, Zhaima).

2.2.1.2 Mineralisation of the Charsk Belt

The Charsk Gold Belt is prospective for sedimentary-hosted base metal deposits, volcanogenic massive sulphide (VMS) deposits, magmatic-related copper and copper-gold deposits and orogenic (mesothermal) gold deposits.

The gold deposits invariably exhibit strong structural and lithological controls and comprise quartz veining and/or gold-sulphide dissemination hosted by highly strained rocks. Gold is commonly accompanied by elevated silver, arsenic, antimony and locally tungsten contents.

In general, sulphide bearing gold mineralisation is refractory, whereas gold in quartz veining is free milling, the thick oxide zones, amenable to heap leaching, has been primary focus of exploration. Refractory sulphide mineralisation has only recently started to draw attention.

2.2.2 Mine Geology & Mineralisation

2.2.2.1 Introduction

The Zherek deposit comprises a series of lens-like, sub-parallel orebodies and zones of hydrothermal alteration in sediments of Lower Carboniferous age. Orebodies are oxidised at surface where they “mushroom” up to 15-30m in width, and extend into primary mineralisation at depth.

Terrigenous sediments comprise sandstones, siltstones, carbonaceous shales and carbonate sandstones and all the rocks have been subject to chemical weathering, giving a variable thickness of weathering crust from 1-50m, which tends to be thickest in areas of hydrothermal alteration and mineralisation.

The deposit is located within the influence of the northwest trending Mukur fracture system where mineralisation occurs in thin veinlets, stockwork and as disseminations. In the oxide zone, iron and kaolin are prevalent, whilst primary mineralisation occurs at depth.

2.2.2.2 Intrusions

Two stockworks have been identified in the northwest of the deposit, as well as numerous dykes of plagiogranite and porphyrite composition in the central part of the deposit.

2.2.2.3 Alteration and Mineralogy

The main orebodies are associated with metasomatic haloes on the hangingwall side of the gold mineralisation which extend for up to 1,000m. Up to 100-300m away from the orebody, pyrite grades are 1-1.5%, diminishing upwards and the distribution of pyrite in the orebody themselves is not homogeneous, varying from 1–2.5% of the rock, although sporadically up to 5-6%. In addition, irregularly distributed arsenopyrite is also located in the gold zone, and forms a halo from 4-10m wide.

Quartz veins can be massive up 5-8m wide with grades up to 10g/t Au, whilst other thinner quartz veins on the contact of plagiogranites with sediments have lower grades. There are also numerous other complex forms of quartz veins present throughout the deposit.

Graphite in shales appears to show a similar distribution pattern to the pyritisation halo, whilst contact metamorphism is evident around the dykes and small intrusives.

2.2.2.4 Gold Mineralisation

Three types of gold mineralisation have been noted:

1. Gold in sandstones and siltstones with veins of pyrite and arsenopyrite (3.5-5%);
2. Berisitisation of the granites and porphyrites with arsenopyrite, pyrite, and vein mineralisation with higher molybdenum content; and
3. Quartz veins and stockworks with tennatite and sulphide disseminations which have non-homogenous gold distribution.

The highest grade mineralisation seems to be restricted to the sandstones and thin layers of carbonaceous siltstones, although gold mineralisation is also concentrated in fractured zones and particularly on the contacts. Most stockworks and veins have a northwest strike and northeast dip, and tend to be conformable with the strike of the host rocks, although dip is discordant (Figure 2.4).

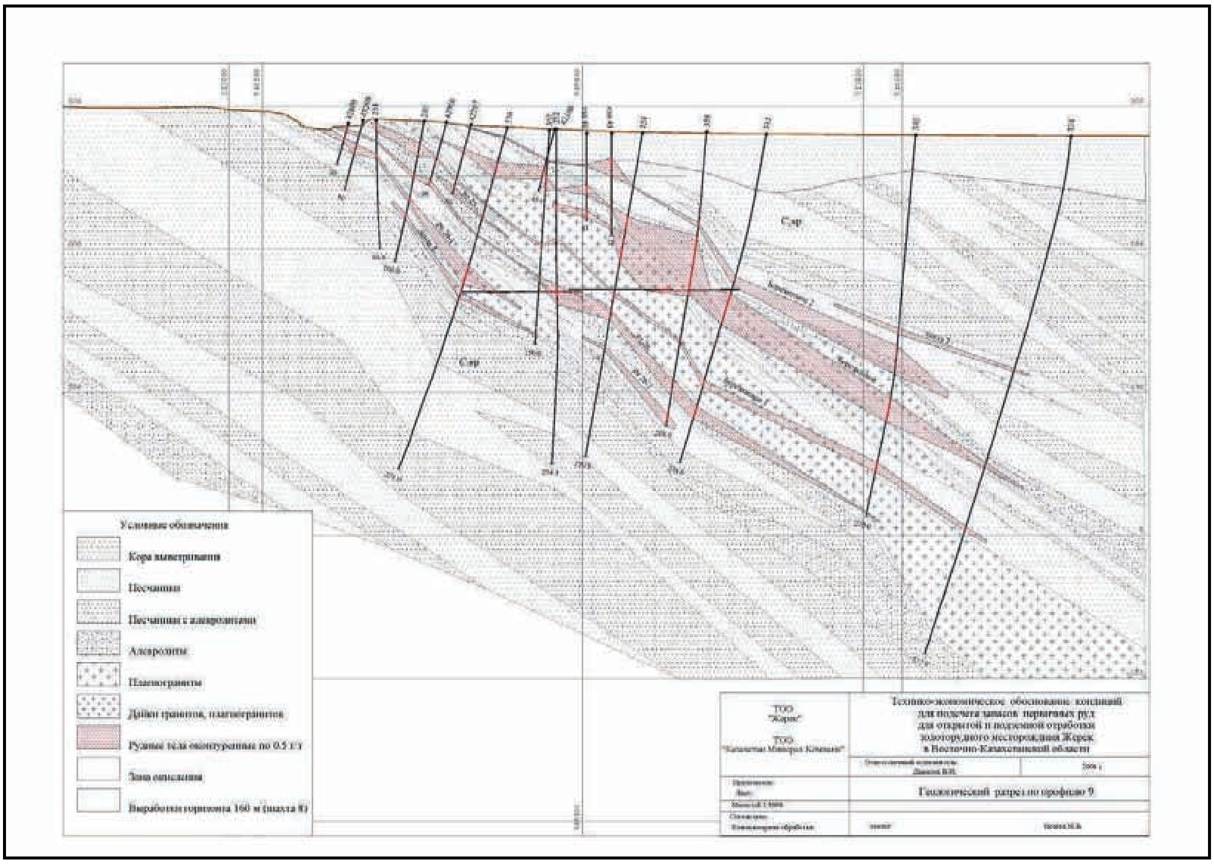


Figure 2.4: Section 9 at the Northwest End of Deposit

Three types of arsenopyrite have been observed:

1. Widely spread acicular needles in hydrothermally altered sediments and stockworks in contact with granite porphyrites;
2. Associated with average and coarse size pyrite, disseminated in beresite porphyries and remote zones of these rocks, and
3. Fine grained, localised in quartz veins.

Two types of gold have been identified:

1. Fine, sub-microscopic associated with the main sulphides of pyrite, arsenopyrite and associated tennantite; and
2. Disseminated and in veins with quartz and tennantite.

2.2.2.5 Orebody Description

Although not exhaustive, the section below presents details of some of the mineralised zones encountered at Zherek.

NW Area - comprises 7 orebodies (10 licences) accounting for 29% of reserves. These strike 140-126° and dip northeast at 30-40°, although individual orebodies pinch and swell:

Berezitovy 2 - 136m along strike, located at the north end of site, with an average thickness of 5m. At 120m depth, the orebody thickens to 17m. It is located 5-20m north of Berezitovy.

Berezitovy - The orebody has been explored to a depth of 260m; the strike is 140° and dips at 30-33° and has a strike length at surface of 258m, and at depth 145-170m. On section line 9, at depths of 220-160m, it becomes an isometric orebody with dimensions 105 x 35-50m, and to the east, it thins. Towards the surface, the orebody splits. The orebody thickness varies from 1.5-34m, and the grade ranges from 1.09-2.3g/t Au.

Berezitovy 1 – This orebody is narrow and contains lower gold grades ranging from 0.5-2.2g/t Au (average 1.3g/t Au), and is located south and parallel to Berezitovy, some 15-25m away. Strike 150°, dipping 33°. The orebody strikes and dips at 33° and shows some splitting.

Orebody 25 – Comprises a lens with a strike lenght of 225m and dip of 40° and is located on section line 9, between orebodies Berezitovy 1 and orebody 26. Further to the southeast, it is parallel to orebody 26. The distance between these orebodies varies from 10-40m. The maximum vertical depth is 95m, with a thickness from 3.5-4m and up to 20m on Profile 10, although further to the southeast, on Profile 39, the grade and thickness decreases and the orebody pinches out.

Orebody 26 – The strike length is 320m, and has been intersected by four grid lines (8 – 11). In Profile 11, the orebody splits into four parts with variable length and thickness. The orebody strikes 140° to 120° on elevation 100m, dipping to the northeast from 30-40°. The thickness varies from 1.5-6m, with an average of 5m. There are two zones of thickening; on Profile 9 at a depth 120m, where the thickness is 12m, and Profile 10, at a depth 180m, where the thickness is 27m.

Orebody 26-1 – Has been explored by five grid lines of deep holes, with a maximum depth of intersection in the central part from 280-350m on grid lines 10 to 11 respectively.

There are many other small, sub-parallel lenses which are not detailed here.

> ***WAI Comment:*** *the geology and mineralisation of the Zherek orebodies is fairly well known, and the oxide zone in particular has been studied in detail and is now almost fully exploited. Knowledge gained from mining has shown that the pinch and swell nature of the mineralised zones at Zherek make true tonnage estimates unpredictable, particularly with regard to the prevalence of sub-parallel lenses which are often not defined at the exploration stage.*

2.3 Exploration Works

2.3.1 Introduction

Details of the exploration work undertaken at Zherek since 1976 are given in Table 2.2 below.

Table 2.2: Historic Exploration Works			
Works	**Volume 1976-78**	**Volume 1979-85**	**Volume 2005-2008**
Trenching, m^3	970	Not Done	
Exploration Shafts, m	169.5	122.8	
Cross-cuts +265, +245, m	1,471.4	2382.0	
Strike Drives, m	1,513.9		
Total Horizontal Development, m	2,985.3	2382.0	
Core Drilling, m	22,018.5	3846.3	3134
Trench Samples	3,591	5200	
Core Samples	7,659	850	2257
Grab Samples	3,263	930	
Composite Samples	142	45	
Spectral Analyses Au	14,513	6980	
Spectral Analyses (12 – 17 elements)	14,513	6980	
AAS			2257
Fire Assay Au	6,458	3350	125
Fire Assay Ag	308	Composite Only	
As Chemical Analysis	142	45	
Geotechnical Tests	32	15	21
Metallurgical Samples	2	1	3
Water Sampling & Analysis		10	5
Internal Control	379	1048	1461
External Control	373	949	948

The sampling methodology was the same as with all GKZ managed projects in that surface trenching was undertaken with internal channel samples, core drilling, and sampling of all underground workings.

From 1976-85, underground development was done at 20 and 40m depths in the oxide zone (265-245masl). A deeper shaft was developed to test the primary mineralisation ($12.6m^2$), the trace of which is currently in the open pit and therefore cannot be used for any future development, although it is likely that the development on the 160m level may still be operative.

2.3.2 Sample Preparation & Analysis

At the time of the visit, no mining activity was taking place, or any further exploration. The small assay lab on site was dealing with leach solutions only.

However, WAI was informed that the standard GKZ-approved techniques were utilised for both sample preparation and analysis which involves crushing using a jaw and rolls crusher, then pulveriser to 0.074mm before being sent for AAS analysis.

2.3.3 QC on Fire Assay

For all the historic exploration drilling and more recent production works, Zherek has followed rigorous State protocols with regard to quality control with both internal and external checks.

Internal control has generally indicated that no significant bias has occurred, although when present, the bias was within tolerance in the results from SGRE and Quartz Laboratory.

Equally, external control showed that again there was no significant bias.

2.4 Mineral Resources

2.4.1 Oxides

The oxide resources at Zherek are now almost depleted comprising 0.346Mt @ 1.86g/t Au in the C_1.category in the GKZ (Republic of Kazakhstan – RK) approved resource statement of 01 January 2008 (Table 2.3); since then a further 0.22Mt @ 1.65g/t has been mined but recent extensions to the north west of the existing pit indicate that 0.2Mt of ore may remain.

Table 2.3: "Reserve" Statement for Zherek as of 01 January 2008 (GKZ (RK) Approved)

Parameters	Units	Oxidised Ore		Sulphide Ore		Total		
		C_1	C_2	C_1	C_2	C_1	C_2	C_1+C_2
Ore	Kt	346	-	-	1,603	346	1,603	1,949
Grade	g/t Au	1.86	-	-	4.17	1.86	4.17	3.76
Gold	Kg	641	-	-	6,647	641	6,647	7,288
Note, not reserves under JORC (2004)								

Areas north and south of the pit indicate the potential for the discovery of additional lenses of oxide (and primary) mineralisation, and the intention is to continue work on these in the near future.

2.4.2 Primary Mineralisation

Deeper drilling has indicated that the primary mineralisation at Zherek is generally between some 1.5-3m in width, although there are areas where the mineralisation thickens and/or has higher grade.

Resources were calculated based on data from the 1976-85 and 2005-08 exploration programmes; Micromine® mining software was used for the resource estimate with six COG's of 0.5, 1.5, 2.0, 2.5, 3.0, 3.5g/t Au.

2.4.3 Database

2.4.3.1 Introduction

The work was undertaken by Kazakhstan Mineral Company (KMC), using Micromine® 10.1, and used both drill hole and development sample assays (Table 2.4).

The database was checked extensively for errors *viz*. wrong coordinates, analytical errors and inclinometry, and corrected accordingly.

Table 2.4: Information Used in the Database

Information Type	Number	Total Samples	Included Samples
Boreholes:			
Before 2002	228	23,171	1,767
2005-06	37	1,204	225
2008	31	1,409	258
Developments (Shaft 8)		8,226	2,406
Operational Pit	1807 Trench	50,329	1,730

2.4.3.2 Statistics

Statistics were run on the database to find the natural cut-off grade (COG) and to define statistical parameters of Au distribution. An analysis of probability plots indicated an inflection point at around 0.5g/t which probably indicates the natural COG.

Statistics reveal a non-normal database and therefore modelling used a lognormal approach; an analysis of the data showed that some 56% of samples had an average length of 1m and composites of 1m were used for the evaluation. Top-cut analysis showed that the database should be cut above 20g/t Au.

2.4.3.3 Wireframing

Each section was interpreted and wireframes constructed using a 0.5g/t Au COG using:

- Minimum orebody thickness 1m;
- Maximum thickness of internal waste 4m; and
- The highest samples (in elevation terms) used were from base of pit.

As the original drill hole profiles were 100m apart, the wireframes were done on 50m sections.

2.4.4 Variography

Variography was done in three directions (Table 2.5). The semi-variogram for the strike direction of the ore zones (120° azimuth) is shown in Figure 2.5 below.

Table 2.5: Variography Parameters

Variogram Axis	Azimuth	Dip	Nugget Effect	Sill	Range м	Lag
Gold						
Down Dip	30	-78	0.31	0.31	32	5
Across Strike	210	-12			23	5
Strike	120	0			37	5

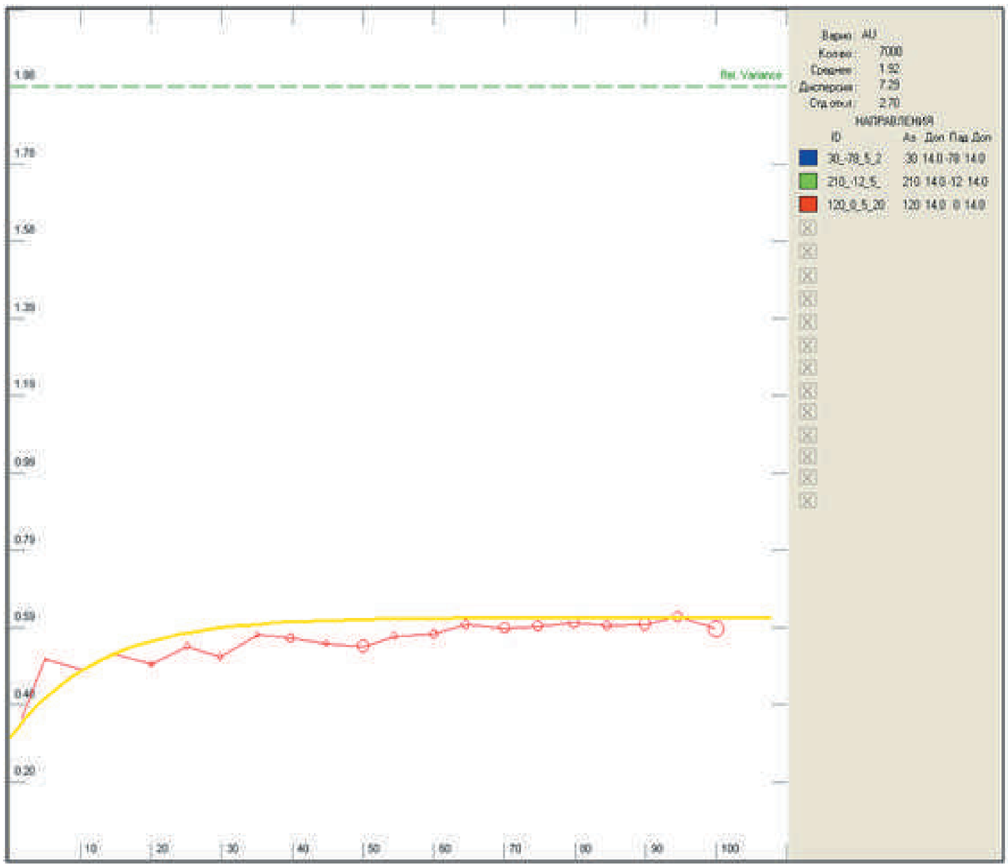


Figure 2.5: Strike Parallel Semi-Variogram

2.4.5 Block Modelling

A parent block size of 10 x 10 x 3m was chosen, with sub-cell splitting to 2 x 2 x 1m to maintain accuracy, giving 390,233 blocks in total. The wireframe model is shown in Figure 2.6 below.

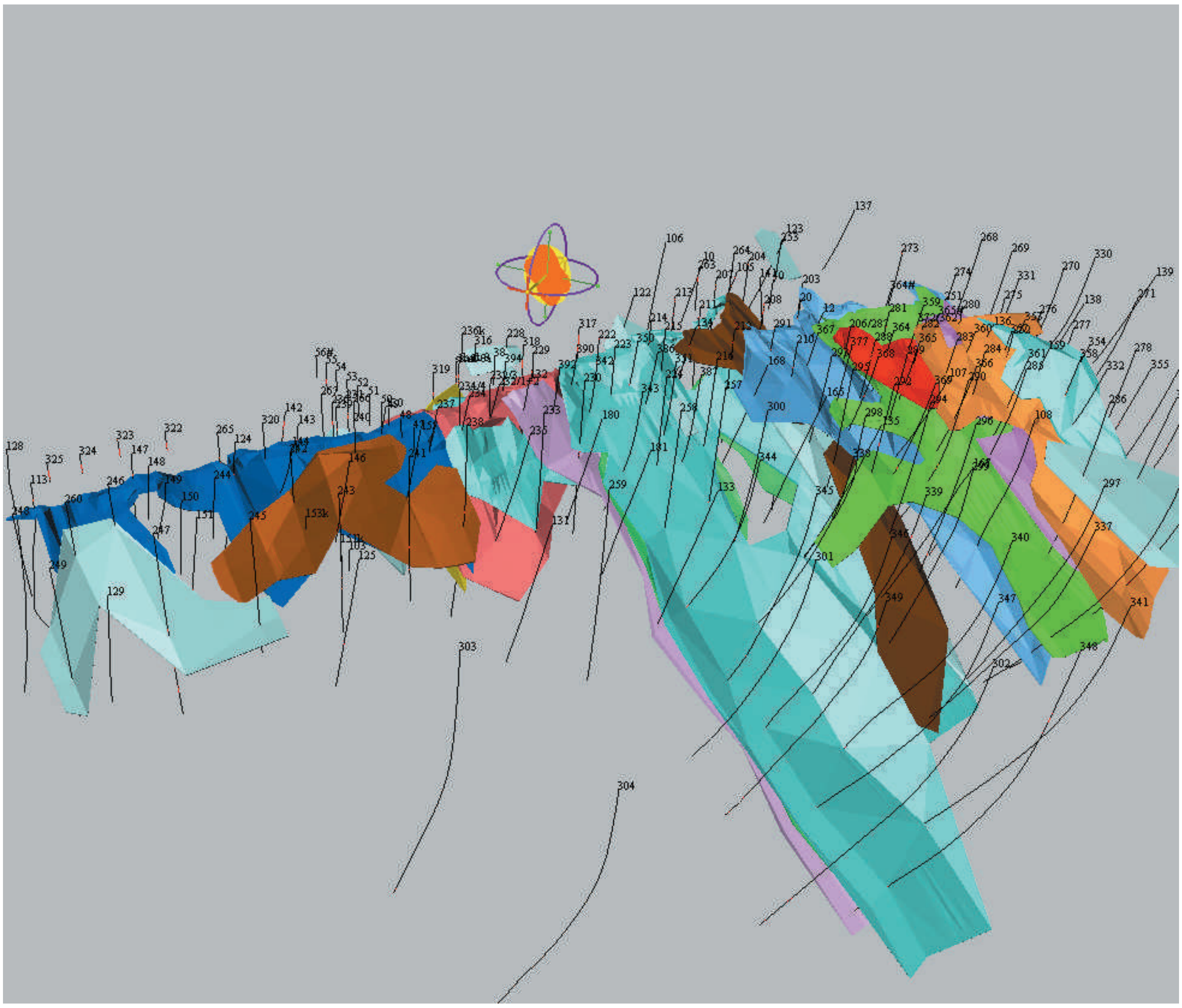


Figure 2.6: Wireframe Model

The composite grades were imported into the block model and interpolated using Ordinary Kriging (Table 2.6) and IDW^3.

The model was interpolated twice – once without top-cut, secondly using 20g/t Au as top-cut.

Table 2.6: Interpolation Parameters

Interpolation Method	Ordinary Kriging		
Interpolation Number	1	2	3
Search Radius	⅔ range	1 range	2 - 5 ranges
Min No. of Samples	3	3	1
Max No. of Samples	12	12	12
Min No. of Developments	2	2	1

2.4.6 Classification

Based on the search radii from the interpolation process, the model was classified in accordance with the guidelines of the JORC Code (2004) into *Measured, Indicated* and *Inferred*.

Figure 2.7 and Figure 2.8 show sections through the block model and the distribution of the classified resources.

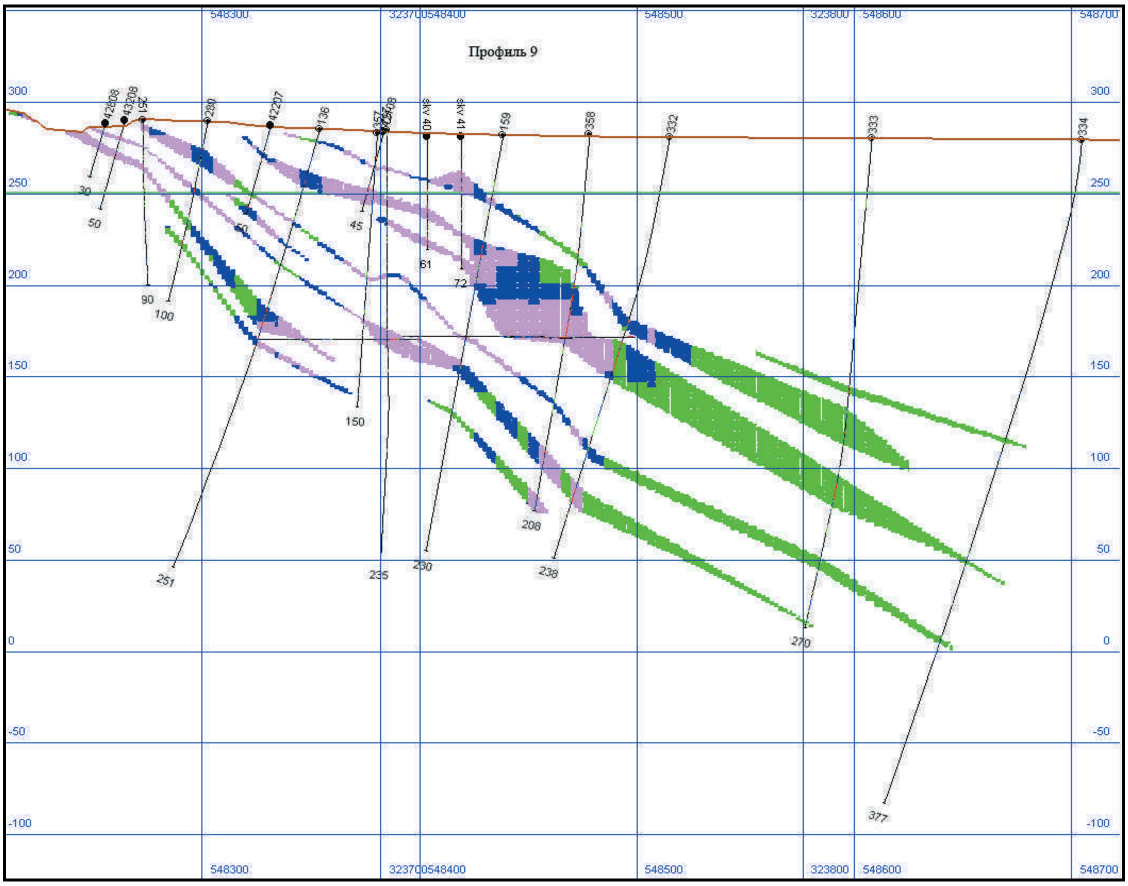


Figure 2.7: Profile 9 Classified Resources
(Pink – *Measured*, Blue – *Indicated*, Green – *Inferred)*

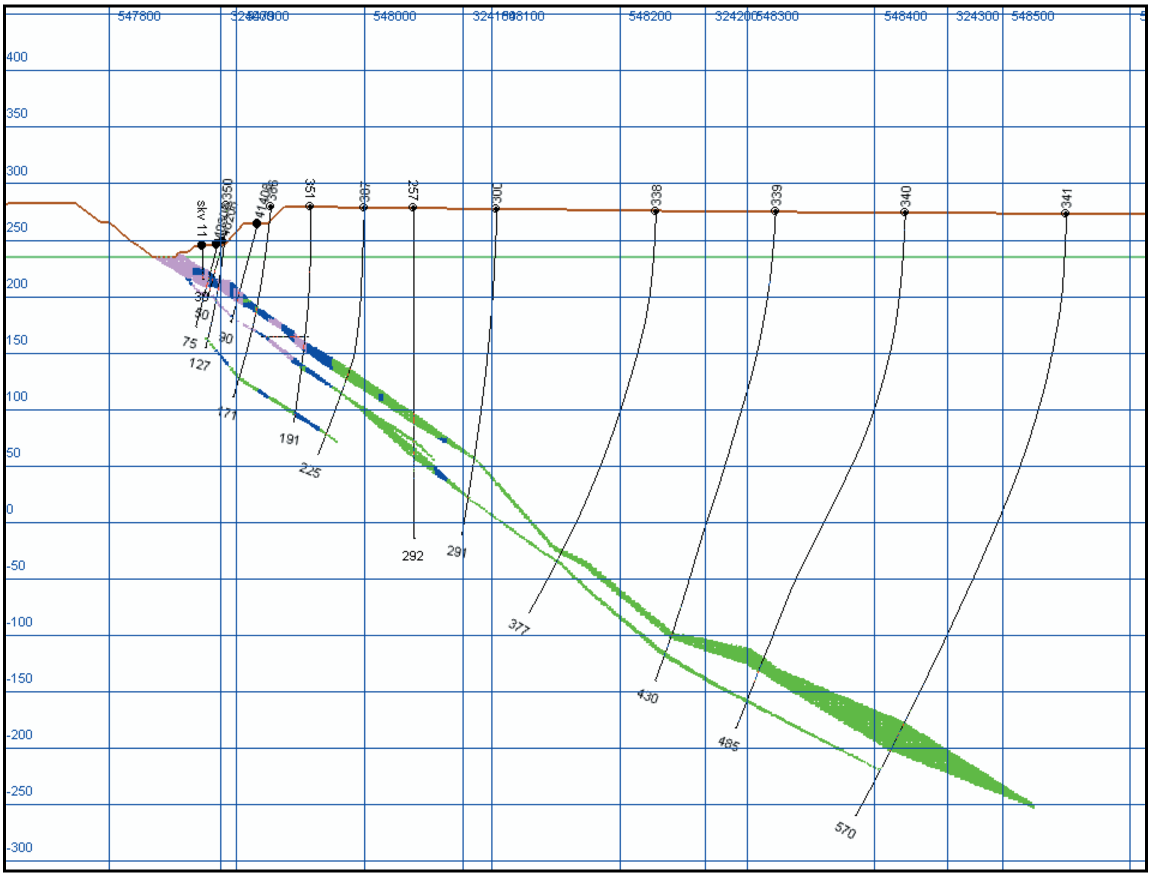


Figure 2.8: Profile 14 Classified Resources
(Pink – *Measured*, Blue – *Indicated*, Green – *Inferred)*

2.4.7 *Mineral Resource Estimate*

The oxide mineralisation is virtually depleted, and therefore only the sulphide resources are summarised in Table 2.7 below (KMC 2008).

Table 2.7: Primary Mineral Resource (KMC 2008)

Au COG	Resources					
	Volume	Ore	Grade (g/t)		Metal (kg)	
	(m^3)	Tonnes (t)	Au	Au (cut)	Au	Au (cut)
3.5	433,616	1,170,763	4.759	4.583	5,571.3	5,366
3	653,820	1,765,314	4.251	4.134	7,504.3	7,298.5
2.5	1,077,200	2,908,440	3.645	3.575	10,602.4	10,396.2
2	1,866,428	5,039,356	3.04	2.999	15,318.6	15,111.5
1.5	3,247,948	8,769,460	2.485	2.462	21,794.7	21,587.8
1	5,508,480	14,872,896	1.974	1.96	29,359.1	29,152.4
0.5	7,245,448	19,562,710	1.688	1.677	33,019.9	32,812.5
0	7,416,500	20,024,550	1.658	1.648	33,202.7	32,996.5

However, not all of this resource is within the existing licence boundary, and the resource directly Net attributable to the Company is shown in Table 2.8 below.

Table 2.8: Primary Mineral Resource Inside Licence (KMC 2008)				
	Resource			
	Volume	Ore	Grade (g/t)	Metal (kg)
Au COG	(m^3)	Tonnes (t)	Au	Au
3.5	250,092	675,248	5.150	3,477.6
3.0	390,268	1,053,724	4.463	4,703.2
2.5	629,376	1,699,315	3.796	6,451.1
2.0	1,077,576	2,909,455	3.138	9,128.7
1.5	1,839,388	4,966,348	2.554	12,681.6
1.0	2,941,920	7,943,184	2.063	16,385.2
0.5	3,677,176	9,928,375	1.810	17,968.4
0.0	3,715,992	10,033,178	1.795	18,013.6

WAI Comment: *the oxide resources at Zherek are virtually depleted, and without further land acquisition and a successful exploration programme, heap leach operations will cease in 2010.*

However, there are primary resources at depth which may be amenable to extraction from an enlarged open pit, but given the fact that the mineralisation is quite thin leading to high strip ratios, partially refractory, and fairly low grade (3-4g/t Au), WAI does not believe that a stand-alone sulphide operation can be established at Zherek. Notwithstanding this, primary ore from Zherek may be suitable as a toll treatable feed at the nearby Suzdal plant, although WAI believes that this is likely to be only in the order of around 1Mt @ 4.5g/t Au at best.

On a positive note, the deeper exploration drilling has intersections up to 150m apart, and given the known structural complexity of the ore zones, particularly with regard to thickness variations, it is possible that thicker zones of mineralisation may be present within the licence that as yet remain untested. Therefore, if there is a will to ascertain the true viability of the primary mineralisation, further drilling will be required to better define the ore zones.

2.5 Mining and Infrastructure

2.5.1 Mining

2.5.1.1 Introduction

The Zherek gold mine is an open pit operation, with mining having commenced in 2003. Currently the mine is producing at an average rate of approximately 200ktpa of oxidised ore. Total ore production since 2003 is approximately 2Mt.

Mining of oxide ore at Zherek is approaching completion, with the majority of the remaining reserves comprising sulphide and transitional ores.

2.5.1.2 Resources, Reserves, Losses and Dilution

Resources and reserves at Zherek are classified according to the Russian classification system, approved by GKZ-RK (01 January 2008) and summarised in Table 2.3 in the resources section of the report above. Reserves are not reported, but rather the "mineable resources", which represent undiluted reserves and do not consider losses during extraction of the material.

Average dilution has increased incrementally since 2006 from 5.8% up to 8.7% in 2009 (year-up to the date of the report). Losses have remained relatively constant, ranging from 2.7% - 2.8% between 2006 and 2009 to date.

The remaining resources will only provide for mining of the oxide ore for a further 6-12 months, as discussed below. After 2010, it will be necessary to mine either transitional or sulphide ore unless a new area of oxide ore is located.

The open pit parameters for mining reserves comprising sulphide ores (and remaining oxide ores) at Zherek, have been produced by Zherek LLP, with no involvement of WAI. WAI does not believe that a stand-alone sulphide operation can be established at Zherek.

2.5.1.3 Historical and Future Mining Operations

Historical production figures since full production commenced in 2003 are given in Table 2.9 below.

Table 2.9: Historical Production Tonnages and Grade at Zherek

Year	ROM Diluted Ore (t)	Average Gold Grade (g/t)	Contained Gold (kg)
2003	293,410	1.97	576.6
2004	320,957	3.37	1,082.1
2005	402,276	2.51	1,009.9
2006	391,254	2.00	782.2
2007	183,338	1.81	331.8
2008	110,980*	0.83	92.4
2009	223,865	1.65	369.5
Total	**1,926,080**	**2.02**	**4,243.4**

*The 111kt shown as mined in 2008 was taken from waste stockpiles mined in previous years with no actual mining from the open pit in that year.

Approximately 120kt of the C_1 oxide ore shown on the reserve statement of 01 January 2008 remains to be mined during 2010, after taking into account the 224kt mined in 2009; additional oxide ore was discovered and extracted during mining operations in 2009, bringing the remaining tonnage of oxide ore up to some 185kt, all of which is planned for extraction in 2010.

2.5.1.4 Mining Activities

Mining at the Zherek deposit takes place only between April and October each year, as the extremely cold weather causes agglomeration of the crushed ore prior to heap leaching.

During 2009, mining activities took place in Orebody 26, and have been concentrated in the northern corner of the open pit (Figure 2.9).

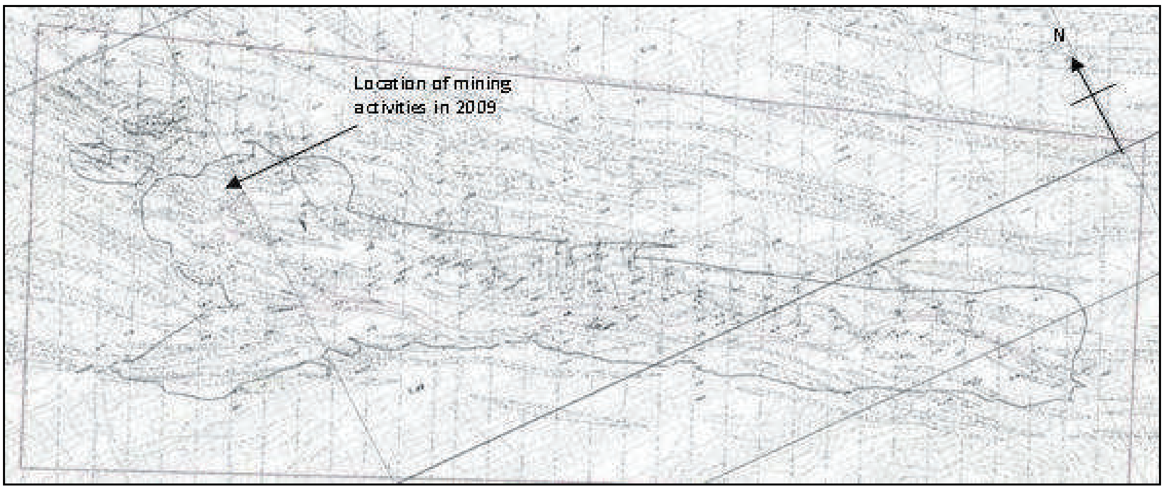


Figure 2.9: Schematic of the Zherek Open Pit

The production schedule for 2009 (updated quarterly) demonstrated scheduled extraction tonnage of 220kt for the year, updated after the first quarter to 226kt, with a final mined tonnage for 2009 of 223.8kt. Mining operations for the year were completed as planned in October. All of the ore mined during that year was oxide ore.

2.5.1.5 Mining Method

Overview

Mining at Zherek is carried out in the open pit using drill and blast methods, followed by transport to the primary crusher or waste dumps *via* truck and shovel.

Currently the open pit has dimensions of 1,400m length, 100m-200m width and an average 50m depth, with the long axis of the pit running approximately southeast to northwest.

The open pit bench heights range from 5m for ore to 10m for waste. Berm widths are 10m, with overall slope angles of 70° and 55° on the northwestern and southeast walls of the open pit respectively.

Ore and Waste Transportation

Ore and waste are both transported by dump truck with the haulage distances from the bottom of the open pit (in 2009) to the crusher station ranging between 700m-800m and the distance to the waste dumps from the pit bottom, on average, 1,000m.

Mining Equipment

Table 2.10 below includes all mobile equipment used for mining at Zherek. It should be noted that no drilling rigs are listed as these are provided by the contractor responsible for drilling and blasting.

Table 2.10: Mining Equipment at Zherek

Equipment	Number
Belaz 27t & 30t Dump Trucks	11
T150 Bulldozer	1
Hyundai 455 Diesel Loader (for selective mining)	1
EKG5 Electric Shovels (5m^3 buckets)	2
RH30 Loader/Backhoe (5 m^3/3.5 m^3 bucket respectively)	1
Total	***16***

> ***WAI Comment:*** *WAI considers that the number of trucks in use at the site is marginally higher than necessary, based upon tonnages of ore and waste mined, and could be reduced by 10-20%.*

Dewatering

Whilst the open pit at Zherek experienced significant water inflow during the early years of production, this has reduced as mining has progressed and currently, water in the open pit drains to a sump (with capacity 500-700m^3) before being pumped to surface.

Pumps run throughout the year, with water from the pit being sent to the plant for use as technical water.

Mine Personnel

Two teams of personnel work on a 15-day rota with two 12 hour shifts per day including 1 hour for breaks and one hour for shift changes.

Given that mining at Zherek, operates for only seven months of the year, the numbers of personnel on site varies during the calendar year *e.g* January - 105, July – 192 and November 122 and appear to be appropriate for the size of operation.

Operational Costs

Summary operational costs have been taken from the Zherek financial data for 2009 to the date of this report, with overall production costs amounting to US$16.11/t (US$1,139/oz), of which mining costs comprise US$9.80/t (US$693/oz). These figures are comparable to similar sized operations (i.e. other small central Asian gold mines). The most significant portion of the mining costs is associated with loading and transportation of blasted ore from the open pit to the waste dumps, crusher and heap leach pads.

2.5.2 *Infrastructure*

2.5.2.1 Power

Power to the Zherek Mine site is provided by overhead power lines connected to the local power grid. The 10kV power supply for the site is distributed to the required areas *via* overhead cables and substations/transformers.

2.5.2.2 Roads

A simple haul road network is in place on the Zherek site, providing access between the site entrance and the main operational areas, and although roads are poorly delineated as a consequence of the layout of the site and to some extent snow cover, they appear generally well maintained and suitable for purpose. Within the open pit, haul roads are approximately 2½ truck widths in dimension.

2.5.2.3 Maintenance & Stores

Given the close proximity of Zherek to both the Suzdal mine site and the city of Semey, both of which are within 40km, approximately one month’s supply of basic spares, equipment and fuel are maintained on site.

2.5.2.4 Explosives

Drilling and blasting operations at Zherek are carried out by a contractor, who is responsible for the maintenance of a suitable supply of blasting equipment and materials at the site. An explosive storage facility is maintained at the mine site by the contractors and is classified as a “mobile explosives magazine” under local regulations.

2.5.2.5 Water Supply

Potable and technical water are supplied to the mine from local water sources, with the supply of technical water to the processing plant supplemented, when/where necessary, by water pumped from the open pit. In general the water requirements of the site are minimal and are well within the capacity of the water sources available.

2.5.3 Conclusions

The Zherek Gold Mine is a relatively mature, small scale open-pit operation, which is well organised and has all necessary infrastructures, labour and facilities appropriate for the operation.

The remaining oxide ore reserves at the mine are sufficient for only 6-12 months of production and any further operations at the mine are dependent on identifying oxide resources; however, WAI believes that it is unlikely that significant oxide resources remain within the current licence area, and that a stand-alone sulphide operation could not be supported at Zherek.

2.6 Processing

2.6.1 Introduction

The orebody at Zherek contains oxide, transition and sulphide ore types. The oxide ores have been treated by heap leaching since 2003 using two stage crushing to -25mm followed by agglomeration. The operation has treated up to 400t per year of ore, with grades ranging between 1.2 and 3.3g/t Au. The leach operations work throughout the year, although agglomeration and stacking only take place between April and October. The crushing plant is rated at 100tph.

2.6.2 Ore Types

The oxide ores are now nearly exhausted with only approximately 200kt remaining. The transition ore type which is only 5m thick and consists of only 30kt, gives poor heap leach recoveries of 10-30%. The sulphide ore type is also refractory with carbon in leach (CIL) recoveries of 0-10% and flotation recovery of up to 80%.

2.6.3 Process Description

2.6.3.1 Crushing and Agglomeration

Ore is fed via a hopper to a jaw crusher. The product is crushed in two more jaw crushers and a cone crusher to pass 25mm and is agglomerated with cement (8kg/t) and transferred to the heaps.

Cyanide is added during the agglomeration stage (1,000ppm) and the cyanide concentration in leaching is 500ppm. It is understood that the heat generated by the cement decomposition in the heap provides sufficient heat energy to keep the temperature of the leach solutions above freezing and spraying takes place under an insulating ice capping formed on the heap over the winter months.

There are a total of six pads, each 200m by 100m, located to the northeast of the site (Figure 2.10); in older heaps, degradation of carbon in the adsorption columns had taken place and some of the heaps were coated with a black carbonaceous layer.

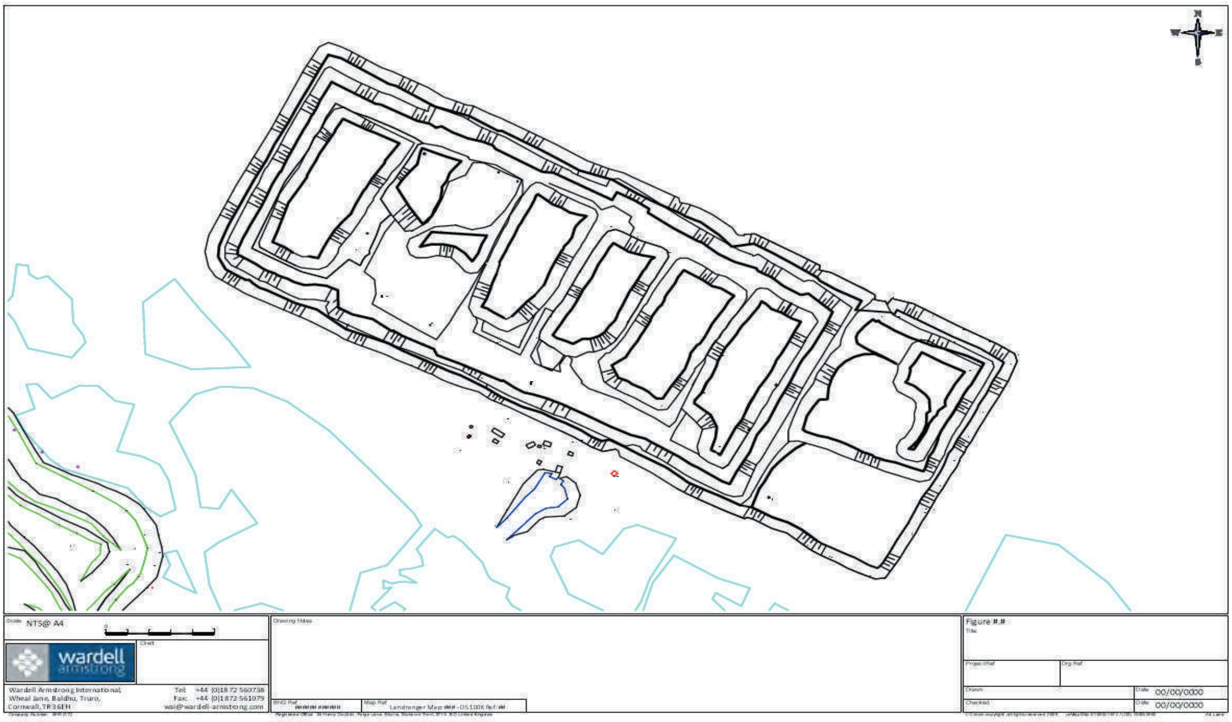


Figure 2.10: Zherek Heap Leach Layout

2.6.3.2 Hydromet Plant

Leach liquors are pumped to the Hydromet Plant where there are two lines of carbon adsorption tanks. The first line consists of six tanks and the second of five tanks, each containing 1.2t of carbon.

Loaded carbon passes to desorption and 2t are stripped after heating to 100°C with 1% NaOH and 4% NaCN. Eluate, at a rate of 3.5m^3/h, passes to electrowinning. Gold levels on the loaded carbon are low at 700g/t Au and these are reduced to <100g/t Au after stripping.

The cathode sludge is smelted to a 98% gold dore product which contains <1% silver.

2.6.3.3 Labour

There is a total of 57 people employed in processing operations, with 32 in crushing, 10 in spray monitoring and 15 in the Hydromet Plant.

2.6.4 Laboratory

There is a small contract laboratory on site which is used for grade control and process control purposes and can also undertake bottle rolls and column leach tests.

2.6.5 Production History

The production of the Zherek heap leach operation is given in Table 2.11 and Figure 2.11 to Figure 2.14.

Table 2.11: Zherek Heap Leach Data

	2003	2004	2005	2006	2007	2008	2009
Tonnes treated	352,891	339,684	402,276	381,254	182,934	120,480	280,846
Ore Au g/t	1.92	3.30	2.51	1.93	1.75	1.18	1.51
Au Recovery %	51.9%	56.1%	53.6%	41.2%	35.9%	49.1%	44.8%
Gold production kg	350.4	628.6	541.4	302.7	114.6	69.9	189.9

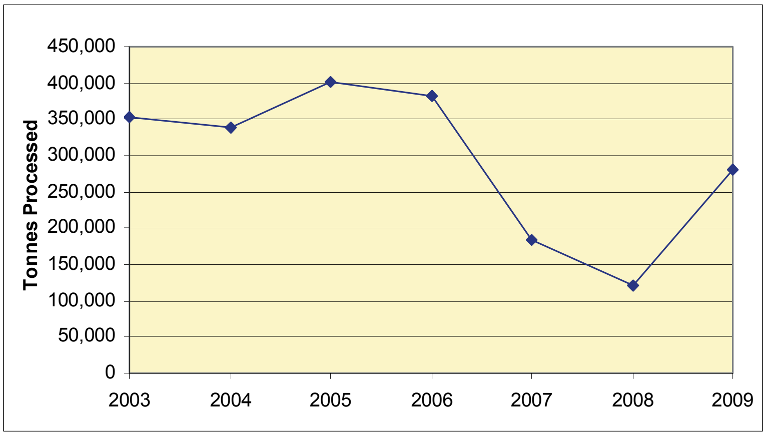


Figure 2.11: Zherek Heap Leach Tonnes Treated

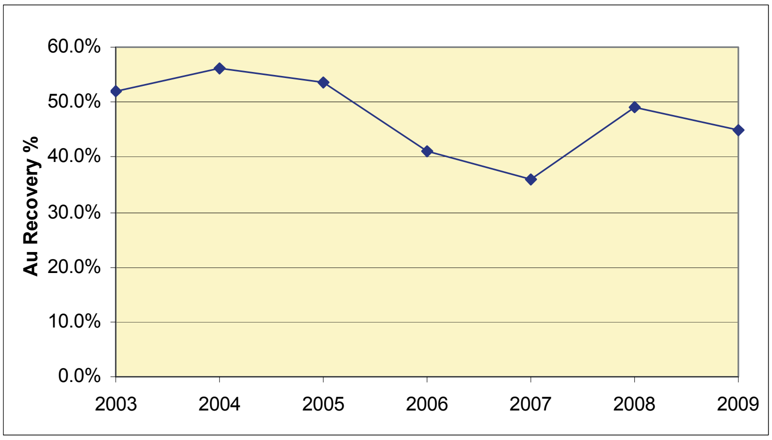


Figure 2.12: Zherek Heap Leach Gold Recovery

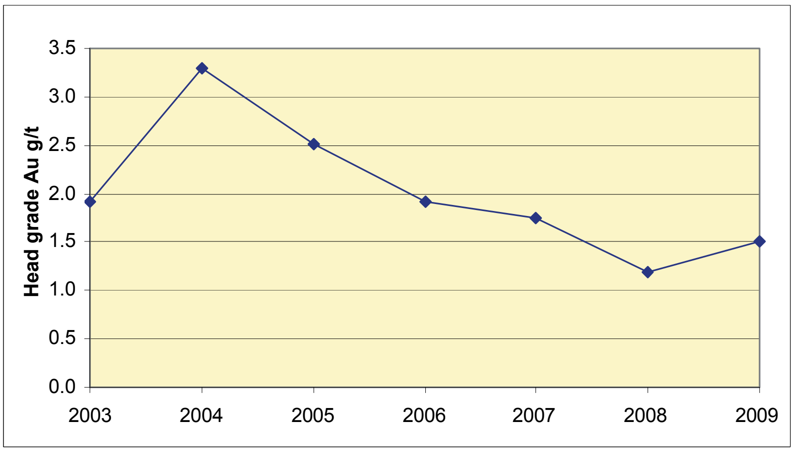


Figure 2.13: Zherek Heap Leach Head Grade Au g/t

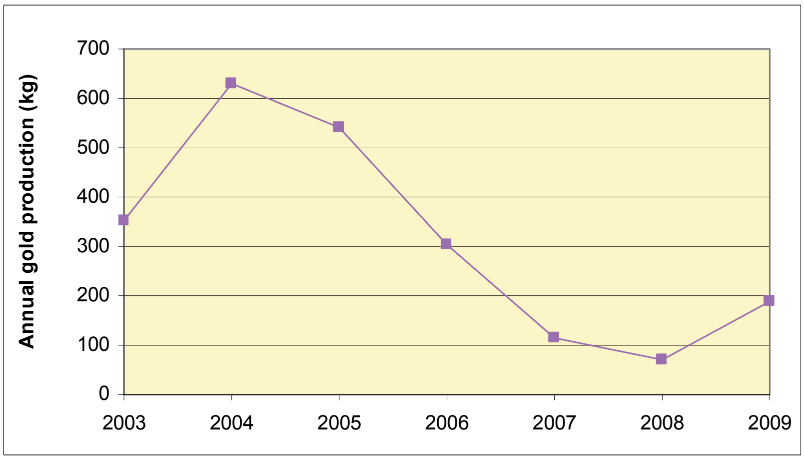


Figure 2.14: Zherek Heap Leach Head Gold Production (kg)

The annual production rate has ranged from 402,276t (2005) to 120,480t (2008). Gold recovery has been generally poor or moderate, ranging from 35.9% (2007) to 56.1% (2004).

The plant head grade reached a peak of 3.30g/t Au in 2004 and has been generally falling since that time. Gold production also reached a peak in 2004 at 628.6kg but fell significantly between 2005 and 2008.

2.6.6 Consumables

The consumables used in the heap leach operation for 2009 are shown in Table 2.12.

Table 2.12: Zherek Heap Leach Consumables

Item	Consumption kg/t
Cement	8.1
Sodium cyanide	0.81
Carbon	0.043
Sodium Hydroxide	0.035
Carbon	0.089

Cement addition would be regarded as high at 8.1kg/t but the previous year the addition rate had peaked at 13.5kg/t. Sodium cyanide consumption of 0.81kg/t is typical for such an operation.

2.7 Operating Costs

The processing operating costs are summarised in Table 2.13.

Table 2.13: Zherek Heap Leach Process Operating Costs

Area	Cost US$/t
Crushing and Agglomeration	1.19
Metallurgy	1.63
Total	2.82

The operating costs are typical for a crush and agglomeration heap leach operation although the rate of cement addition leads to higher than normal cement costs.

2.8 Environmental & Social

2.8.1 Review of Environmental/Social Studies

An OVOS (Russian equivalent to Environmental Impact Assessment (EIA)) produced in 2007 is in place for the HLP and current project at the Zherek mine. This does not include any potential extensions at the Zherek site. The programme for 2010-2011 is currently with the Environmental Ministry, for review and approval.

2.8.2 Land Ownership

Zherek property is leased from the State, with no land ownership by private individuals.

2.8.3 Environmental Monitoring and Compliance

Environmental Monitoring plans, prepared by a registered contractor (as stipulated by State requirements) were prepared and approved for 2008-2009, and accompanying discharge and emission limits were also set. The 2010-2011 monitoring plan is currently with the Environmental Ministry for review and approval.

Water quality is also sampled quarterly for a full suite of determinants via 7 boreholes around the HLP, although only 3 are reported to contain water. Samples do not generally show elevated concentrations, and there have been no exceedences with regard to cyanide. No surface water flows are present at the site.

Soils are sampled once per year and are assessed for Cu, Zn, Ni, As, Mn, fluorides, and nitrates. The company has to report annually on the contamination levels, and compare with the results from previous years

Noise monitoring, and vibration and light pollution measurement are carried out annually.

Air monitoring is carried out at sources of emissions on a quarterly basis. Depending on the location, determinands include soot, NO2, SO2, CO and HCN. In some areas, other determinands such as benzene vapour, dusts, hydrocarbons and suspended solids are also included.

Continuous gas monitoring occurs in hazardous chemical storage areas, and in the plant. The company normally retains paper reports, and does not have an electronic database; however the subcontractors that perform the monitoring are reported to have a database of results.

2.8.4 Water Management, Sourcing and Consumption

There is no water discharge at Zherek, with a closed loop system between the HLP and the plant. Any excess water from the HLP or the plant is stored in a fenced emergency pond at the site with a reported $15km^3$ capacity, and allowed to evaporate. Additional technical and potable water is sourced from boreholes at the site.

2.8.5 Soil, Surface and Groundwater Contamination

There are no water flows at the Zherek site. The ore contains relatively low sulphur (2%) and as such it was reported that the potential for Acid Rock Drainage (ARD) generation was low. ARD potential is not tested for in waste rock or ore, and this is not addressed as part of the programme of environmental monitoring. ARD potential is also not assessed via groundwater monitoring.

If any spills occur in the plant, the liquid is collected via a sump, and sent to the emergency pond. It was reported that approximately 0.1% cyanide is present in the desorption columns.

2.8.6 Air Emissions

The main air emissions at the site are surface dusts, and during the summer season, water bowsers are provided for dust suppression. However, at the time of the visit significant amount of dust were being generated along the haul roads and exposed embankments, however work place air quality has consistently fallen within allowable limits.

2.8.7 Heap Leach Pad Management

The conveyor to the HLP is not covered, and there is no secondary containment beneath it. It was reported that there are 12 monitoring holes around the HLP, which are tested monthly. It was reported that 13m is the average depth to groundwater, but that water is not present in all observation boreholes around the HLPs.

8-10 boreholes are present around an emergency pond near the plant and are assessed monthly for concentrations of gold and cyanide; it was reported that there have been no overflows, or emergency situations at the site.

2.8.8 Liquid Effluent

It was reported that a septic tanks system is present at the site near the plant area, and this is emptied by a registered contractor; however latrines are also present at the site, which are not part of this system.

An emergency pond is also located in the vicinity of the CIL plant, and appropriately fenced.

2.8.9 Waste Management

Scrap metal is stockpiled and sold. Mercury bulbs are disposed of via a contractor, since these are a class I hazard. Cyanide drums are neutralised. Industrial waste, such as tyres etc. is sent to the site industrial landfill. Used oil is also stored and sold. Office waste is sent to the site domestic landfill.

2.8.10 Handling and Storage

Fuel is stored in above ground tanks at the site. Whilst the tanks themselves appeared in good condition, they did not appear to be within a bund.

There is a hazardous material store at the site for various substances including hydrochloric acid and cyanide. The area is alarmed and ventilated, and daily logs are kept of delivery and use.

There is mobile explosives storage at the site, and this process, in addition to blasting, is managed by external contractors.

2.8.11 Environmental Liability

Since the mine has been operated by the company since its inception, there are not considered to be any historic liabilities associated with the site. Mine closure and remediation plans should be sufficiently rigorous to ensure that potential long-term environmental liabilities are minimised.

2.8.12 Mine Closure and Rehabilitation

It was reported that a closure plan has been developed, and every year monies are sent to an external fund (held at a bank) to make provision for closure costs. Figures of 0.5% of mining costs per year, rather than actual closure cost estimates are set aside annually. As part of the subsoil use contract, a closure plan is required, and expected to be prepared 90 days before closure. Progressive rehabilitation is also to be carried out.

The closure fund for the mine currently contains 20,894,273 Tenge (approximately £84K).

2.8.13 Conclusions

The following is a list of WAI's main conclusions with regard to the Zherek operations:

- Existing environmental studies and environmental management have been prepared in line with State requirements, and the operation is considered to be in compliance with national standards, although further works to achieve international compliance would be required
- The lack of an Environmental Manager based at the site limits the level of on site environmental management;
- The current environmental monitoring plan for the site satisfies national requirements but would require expansion to meet international best practice;
- There is no formal stakeholder engagement system or grievance mechanism in place;
- Housekeeping and maintenance of areas was generally good at the site;
- There is potential for soil and groundwater contamination to occur in the fuel storage area;
- The potential for Acid Rock Drainage from waste rock is not being assessed at the site;
- There is significant dust generation at the site despite management measures, and no erosion control is in place;
- The HLP sprinklers are arcing high and the agglomerator conveyor system has no covers or secondary containment, given the high concentrations of cyanide;
- The environmental budget for the site is sometimes exceeded due to unplanned expenditure;
- Fire protection and health and safety are being adequately managed at the site; however there is no full time H&S manager at site;
- The working conditions in the plant appeared to be sub-optimal with regard to accessibility and were not considered to be in line with best practice requirements, however work place conditions meet national requirements;
- There is no system for assessing or managing potential environmental liabilities at the site;
- The Mine Closure and Rehabilitation Plan is not adequately developed to satisfy international requirements, and does not include post closure requirements; and
- The mine closure cost estimate is not considered adequate to meet actual closure costs and does not include post closure monitoring or treatment requirements.

2.8.14 Recommendations

The following are WAI's main recommendations with regard to the Zherek site:

- An ESIA addendum and social needs assessment should be developed;
- An Environmental Manager should be based full time at the site;
- The company should implement an Environmental Management System at the site;
- More proactive community consultation and engagement should be practised at the site;
- An in house monitoring data base should be established, and the company should develop a closer working relationship with the environmental contractors;
- A formal PCDP and grievance mechanism should be established;
- An overall water balance should be produced for the operations;
- An energy audit, to identify savings should be produced;
- The fuel storage areas should also be included in the company monitoring plan;
- Waste rock material should be subject to testing for ARD potential, and appropriate management systems should be developed accordingly;

- Where possible, stockpiles should be covered, and wheel washes should be developed to minimise dust generation, in addition to current control measures. An erosion control plan should also be developed;
- The sprinkler system on the HLPs should be adjusted and secondary containment and covers should be provided for the agglomerator conveyors;
- Options to reduce concentrations of cyanide in the agglomerator should be considered;
- Consideration should be given to managing cyanide in line with the International Cyanide Management Code;
- The emergency pond capacity should be assessed, to ensure all HLP melt water can be safety contained;
- Alternative waste disposal and recycling options should be sought at the site, and a raw materials and waste audit should be performed;
- Regular meetings should be held with operational managers and the environmental budget should be updated on a 6 monthly basis;
- A full time Health and Safety Manager should be based at the site;
- A Health and Safety management system should be developed at the site, in line with best practice;
- A Health and Safety audit of the plant working conditions should be performed;
- A system to evaluate environmental liabilities should be developed, and the company should seek to minimise the generation of environmental liabilities at the site;
- A detailed MCRP should be developed and regularly updated through the life of operations, to include post closure monitoring and treatment as required by international standards;
- A mine closure cost estimate should be produced in line with actual closure costs and post closure monitoring and treatment requirements, and amended sums should be set aside accordingly; and
- The social development plan should be reviewed in line with international requirements.

WAI recognises that to date the company has only sought to comply with State requirements, which has been achieved, and as such many of the above recommendations have not hitherto been required. Nonetheless, WAI notes the company's intention to move towards international compliance and considers that the recommendations above would be a first step in achieving this.

3 BALAZHAL

3.1 Background

3.1.1 Introduction

Wardell Armstrong International (WAI) was commissioned by Semgeo to prepare a Competent Person's Report on the Balazhal open pit gold mine, located south of Ust Kamenogorsk, northeastern Kazakhstan.

A team of WAI consultants visited the Balazhal mine on 12 November 2009, as well as the Semgeo office located in Semipalatinsk.

The review of the Balazhal mine was based on data (some of it supplied by WAI from a previous commission on the project), professional opinions and unpublished material submitted by the professional staff of Semgeo and their subsidiaries.

3.1.2 Location & Access

The Balazhal deposit is located in Kokpektinsk district of the East Kazakhstan region, 100km southeast from the railway station Zhangiz-Tobe, 260km from Semipalatinsk and 65km from Georgievka station (Figure 3.1).

The closest main road to the deposit is the Omsk-¬Maykapchagai highway which lies 20km to the west. There is a well graded dirt road, 18km long, between the deposit and main road, although this road can be difficult to pass in winter and spring. Total driving time from Ust Kamenogorsk is around 2½ hours.

Other former gold mining enterprises are located nearby.

The settlements of Nikolaevka and Chigilek are 25km to the northwest, and 15km to the southeast of the deposit respectively. There are state power supply lines in these settlements, but no fuel resources.

Figure 3.1: Location of Balazhal in Northeastern Kazakhstan Russia

3.1.3 Topography and Climate

The elevation of the Kalbinskiy ridge on which the deposit is located, varies from 750m around Balazhal in the south to 1,050m (Baladzhak Mountain) in the north. Drainage is *via*. the small Baladzhalka River and Berezoviy springs.

The climate of the region is extreme continental with average annual precipitation approaching 250-280mm, with maximums in January (24mm) and in July (32mm) and snow cover is prevalent from end of October until May. Annual average temperature is +1.6°C, the hottest month is July (+21.6°C), the coldest is January (-26.2°C).

3.1.4 Infrastructure

The infrastructure at Balazhal mine is sufficient to operate a relatively small, open pit, heap leach operation, and comprises open pits, small offices, heap leach pads, plant and mechanical workshop.

However, as the mine is currently not operating (other than some heap leaching), some of the infrastructure has been removed to Zherek, or has fallen into disrepair.

3.1.5 Mineral Rights and Permitting

The licence co-ordinates for the Balazhal property are given in Table 3.1 below.

Table 3.1: Licence Co-ordinates for Balazhal

Point	Northing	Easting
1	49° 00’ 36″	82° 14’ 40,8″
2	49° 00’ 58,4″	82° 15’ 00,4″
3	49° 00’ 46,8″	82° 15’ 27,8″
4	49° 00’ 27,5″	82° 15’ 00,9″

WAI Comment: *an inspection of the licence documentation has shown that it is in good order and suitable for the future needs of Semgeo.*

3.2 Geology and Mineralisation

3.2.1 Regional Geology

The Balazhal deposit comprises an auriferous stockwork hosted by carbonate metasomatite (beresite) located at the top of an intrusive gabbro-diorite and lying beneath terrigenous sediments.

The combination of an Early to Middle Hercynian geosynclinal phase, combined with a Permian-Mesozoic orogenic stage led to the development of east-west, north eastern, and north-south tectonic fractures. Early structures and formations comprise zones with a northwestern orientation made up of serpentinites.

The earliest formations comprise volcano-sedimentary units in substantial thicknesses of Middle Devonian rocks of basaltic composition. A change in the sea levels in the Middle Carboniferous led to the development of a molasse facies of sandy siltstone.

Neogene (up to 80m) and Quaternary (15-17m) deposits have been identified.

The Balazhal deposit is crossed by a major fracture zone, although fracturing can be identified through the entire deposit, trending 310°-320°, and dipping to the northeast at 75°- 80°.

Small stocks of dyke shaped bodies of gabbro, dioritic and quartz porphyries are the result of late orogenic magmatism.

3.2.2 Mine Geology & Mineralisation

3.2.2.1 Geology

The Balazhal deposit is located in the south eastern part of the ore field and has been folded within units of Lower Carboniferous age, which in turn are intruded by the Balazhalskij stock.

The Balazhal mineralisation comprises veins and stockwork hosted within beresite. The Balazhalskij intrusive stock is of gabbro-plagiogranite composition and merges at the 863m level with others into one massif. Borax mining and other mine workings exist around the entire perimeter which has assisted in delineating the Balazhalskij intrusive stock which has an ellipsoid shape, being 600m long, with widths ranging from 180m in the north to 300m in the south.

Using drill hole data and magnetic exploration methods, the Balazhalskij intrusive stock has been identified as plunging at a shallow angle ranging between 10°-15° towards the northeast. The southern part of the stock is in contact with the Balzahalskomu fracture zone and country rocks.

3.2.2.2 Mineralisation

The quartz veins of Balazhal form a macro stockwork in beresitised diorite and quartz gabbro which hosts sulphide and gold mineralisation. In plan the stockwork has an ellipsoid shape, 500m on strike and 225m wide, and the depth extent identified through borehole intersections is some 400m from surface. The stockwork and the component veins dip to the west.

Gold-quartz veins, about 40 in number, are surrounded by halos of stockwork beresite mineralisation which develop from the vein footwall for 30-40m in a horizontal direction. Veins lie sub-parallel to the stock, developing zones with a high density of fractures, some of which host dykes of aplite, plagiogranite, porphyry, and in some occurrences, ultrabasite.

Three groups of veins have been identified:

- Veins occurring on the steep western contacts with the intrusive stock, which strike northeast at 25-30° and dip steeply at 60-80° to the west;
- The greatest occurrences of gold-quartz veins can be found in the eastern part of the Balazhalskij intrusive stock, where mineralisation is focused in fractures of a northeastern orientation, striking 55°-70°, dipping at 25°-65°. These veins intersect with the first set of veins, resembling a macro stockwork; and
- A third group of veins, which strike east-west at 100°-110° and dip 40°-45° to the north and have a surface extent of 200-250m and are hosted in the (sedimentary) Kokpektinskoj formation and within the Balazhalskij stock. In areas of the gabbro-diorite intrusive, the veins are accompanied by stockworks.

3.3 Exploration Works

3.3.1 Introduction

Quartz veins were first identified at the beginning of the 19th century, although there was no mining activity until 1902 when exploitation occurred down to the water table; from 1931-1953 2.265t Au was recovered bringing the total to 2.7t. More recent drilling outlined, 3.4t of gold with an average content of 1.5g/t Au in a 500x100 block in beresitised stockworks and from 1985-1990, further reconnaissance work was carried out on the stockwork ores which included drilling to 125m.

A heap leach operation at Balazhal to treat the oxide ores commenced in 2004, although by 2009 gold production was limited to the treatment of 78Kt of material which had been recovered from old stockpiles. The current inactive open pit has been allowed to flood awaiting development of a sulphide project.

3.3.2 Recent Exploration

From December 2008 to May 2009, Semgeo has undertaken a programme of infill core drilling totalling 6,500m, of the sulphide resources with a view to acquiring GKZ (Republic of Kazakhstan) approval in the 4th quarter 2010.

WAI has not seen the results of this work, but has been informed that using a 0.1g/t Au cut-off, a mineralised zone some 50-60m wide comprising veins, small stockworks and beresitic alteration zones was delineated.

3.3.3 Drilling, Sample Collection & Storage

Drilling, using Longyear wireline HQ, produced an average of 95% core recovery. Core samples were usually collected every metre, although occasionally shorter samples were taken where lithology dictated. All cores were photographed. Cores are stored on site in two shipping containers in well labelled, solid core boxes with lids.

3.3.4 Sample Preparation & Assay

Samples were prepared on site in a small sample preparation facility which comprised jaw and rolls crushers and a pulveriser. The simple flowsheet reduced samples to -0.074mm suitable for AAS which was done on site.

In addition, samples were sent to Suzdal for fire assay which quality control work done at TOPAZ Laboratory in Ust Komenogorsk and VNIISVETMET.

> ***WAI Comment:*** *WAI inspected both the sample preparation area and small AAS laboratory. Both were found to be in good condition, clean and suitable for the mine's requirements.*
>
> *In addition, WAI briefly inspected the QA/QC data from the drilling and concluded that the works had been done in a professional and thorough manner. No deviations we*

3.4 Mineral Resources

3.4.1 Historical Estimates

- In 1991 a Soviet style Feasibility study was conducted which established a set of parameters for the deposit, and an estimation was carried out in 1991, presumably for the veins only, as part of an internal audit by Kazzoloto (Table 3.2).

Table 3.2: Balazhal Reserve Summary (as at 27.06.1991)

Resources (Reserves)	Ore ('000 t)	Gold (kg)	Gold (g/t)
Category C_1	200	1111	5.6
Category C_2	24	135	5.6
Category P_1	132	897	6.8

- In the period of 2000-2001, another estimate was carried out in a joint venture by the Semgeo and Geos companies. The estimation work was based on grade control data derived between 1937-1953, as well as the exploration data generated in 1985-1990.

Estimations were carried out at three cut-off grades (COG's) 0.2g/t, 0.4g/t, and 0.8g/t Au. At the 0.4g/t Au COG, a C_2 category was assigned to 10.8Mt of ore, containing 34.8t of gold at an average grade of 3.24g/t Au. Further results placed 11.8t of gold in the P_1 category and 10.9t of gold with an average content of 3.24g/t Au in the P_2 classification.

3.4.2 Current Mineral Resource Estimate

3.4.2.1 Introduction

Work was carried out by the Kazakhstan Mineral Company (KMC) based in Almaty, acting as sub-contractors to WAI, in November 2006. It should be noted that although this estimate was carried out in 2006 no mining or exploration work has been carried out and therefore the estimate may be regarded as current. No account was taken of the oxide-sulphide split in the resources, although at the time of the estimation, little oxide material remained in the pit.

3.4.2.2 Data Management

The initial data, provided by the client in the form of graphs, geological sections, and plans were placed into electronic format and checked comprehensively for any errors which were identified and corrected. The data which were used within this study are shown in Table 3.3.

Table 3.3: Database Composition Summary				
Type	Units	Volume of Geological Works		
		Quantity	Volume	Used
Core drilling from the surface	m	175	14,020	60 boreholes
Underground drilling	m	63		27 boreholes
Underground workings +1 trench	items			25
Gold	assay		4,946	2,134

A number of statistical analyses were run on the samples from which it was ascertained that the gold population is lognormal. Statistics supported the compositing of samples to 1m. Samples that fell within the orebody had additional statistics run on them, identifying outlier high grades which could possibly lead to bias in the estimation process. A top-cut of 24.7g/t Au was applied, and cross checking identified that samples which were not top-cut yielded an overestimation of gold content of 10.7%.

Statistical analysis did show that there is more than one population making up the deposit, most likely a function of the narrow quartz veins. However, due to the lack of geological details the estimation was conducted utilising both vein and stockwork mineralisation and it is likely that the resulting model may be optimistic with regard to mineralised terrain.

3.4.2.3 Modelling

Modelling of Balazhal was conducted using 11 section lines (Figure 3.2), whilst the string generation for the extent of the orebody was conducted with a 0.4g/t Au COG. Sections were then linked to establish a 3-dimensional structure (Figure 3.3).

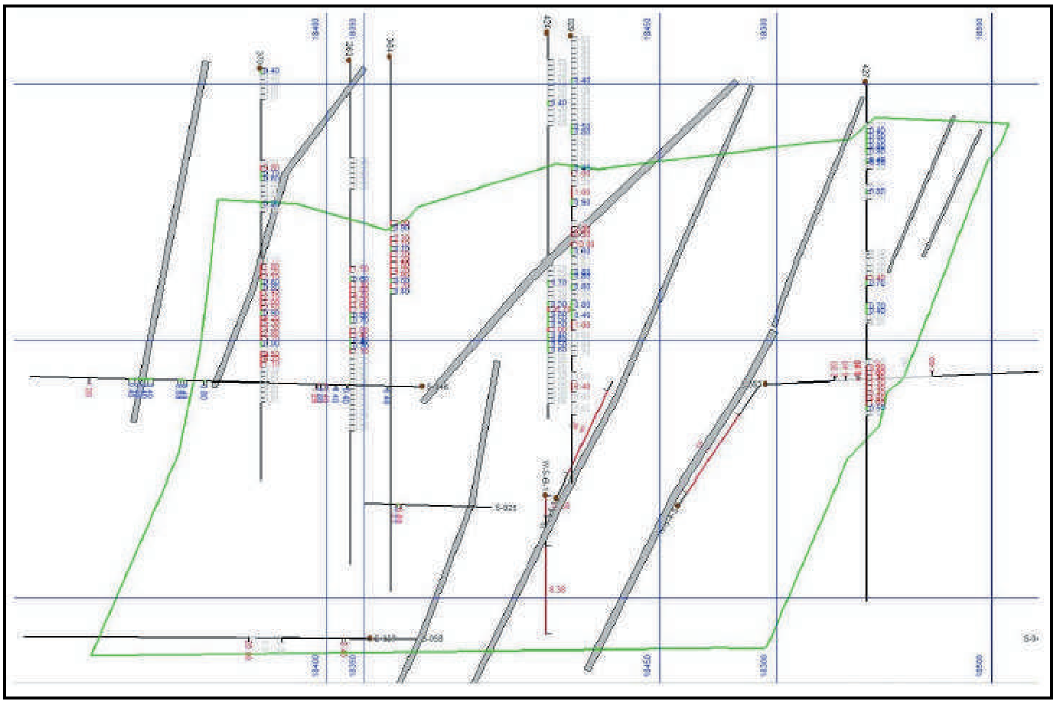


Figure 3.2: Typical Balazhal Interpreted Section

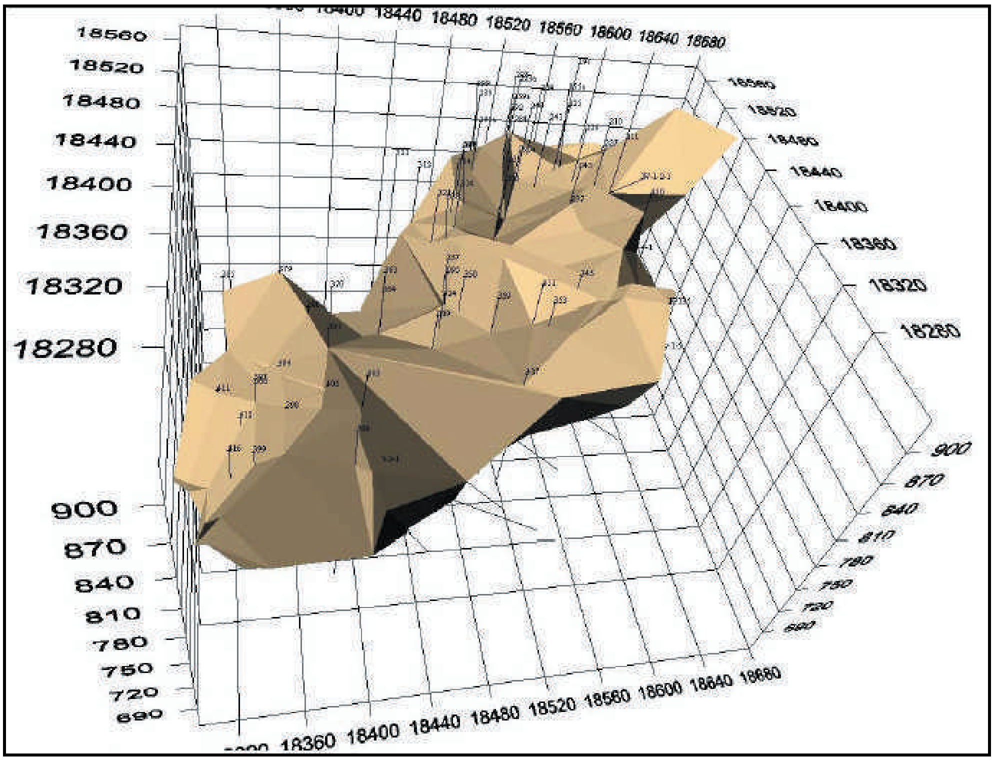


Figure 3.3: 3-D Wireframe Model for Balazhal

The model generated had a depth limitation of 110m from surface. A topographic surface was created in Micromine® using the borehole collars and the surface data provided by the client. A geostatistcal appraisal of the model was carried out with the application of variography, the results of which were not entirely satisfactory for the estimation of the deposit. Block modelling used a prototype of 5 x 5 x 5m parent cells with 0.5 x 0.5 x 0.5m subcells.

3.4.2.4 Interpolation

Whilst estimations were conducted using Ordinary Kriging as well as IDW^3, it was decided that due to the limitations of the variography, IDW^3 was used as the principal estimation method. To remove the effect of clustered samples, octants were applied to the search which limits the number of samples which can occur in a sector, thus removing any bias.

Classification of the deposit has been split into, Measured, Indicated, or Inferred depending on which search radius applies to the blocks. The results of the estimation are shown in Table 3.4 below.

Table 3.4: Balazhal Global Mineral Resources (IDW) (WAI 01 November 2010)						
COG (g/t Au)	Tonnes	Gold Grade (g/t)	Grade with top-cut (g/t)	Category	Gold (kg)	Gold with top-cut (kg)
1	38,329	3.62	3.62	*Measured*	138.6	138.6
0.8	42,684	3.34	3.34		142.6	142.6
0.6	45,798	3.16	3.16		144.7	144.7
0.4	48,544	3.01	3.01		146.0	146.0
0.2	57,141	2.60	2.60		148.7	148.7
0	95,754	1.56	1.56		149.8	149.8
1	129,292	3.74	3.63	*Indicated*	483.6	469.4
0.8	140,900	3.51	3.41		494.3	479.8
0.6	156,993	3.22	3.13		505.6	491.1
0.4	174,317	2.95	2.87		500.0	499.6
0.2	200,082	2.61	2.54		521.7	507.2
0	369,764	1.43	1.39		527.4	512.9
1	3,550,175	4.93	4.30	*Inferred*	17,509	15,265
0.8	3,982,527	4.49	3.93		17,897	15,652
0.6	4,503,317	4.06	3.56		18,262	16,016
0.4	5,081,220	3.65	3.21		18,549	16,301
0.2	5,830,571	3.22	2.83		18,773	16,524
0	12,076,620	1.57	1.38		18,941	16,691
1	3,717,796	4.88	4.27	Total	18,131	15,873
0.8	4,166,111	4.45	3.91		18,534	16,275
0.6	4,706,108	4.02	3.54		18,912	16,652
0.4	5,304,081	3.62	3.19		19,209	16,946
0.2	6,087,794	3.19	2.82		19,443	17,180
0	12,542,138	1.56	1.38		19,618	17,355

Additional work is recommended on the deposit in order to improve the confidence in the classifications. The sample data provided were found to be missing a number of samples, the effect in the estimated blocks being manifest in absent grades. It is important to ascertain whether the missing values represent zones of low core recovery or waste intersections.

As mentioned previously, the preliminary data statistics indicated two populations, most likely explained by quartz veins which comprise a high grade zone and should be modelled accurately, preferably with additional data from between veins to improve the modelling and subsequent estimation.

> ***WAI Comment:*** *the resource estimation detailed above is at best fairly rudimentary as it does not take into account the division between oxide and sulphide (and transition material), and should be treated very much as a global estimate. However, at the time, little oxide material remained in the pit and adjacent areas, and therefore it is likely that a large majority of the estimate consists of primary sulphide mineralisation which is unlikely to be amenable to heap leaching.*
>
> *Without a Feasibility Study, it is not possible at this time to determine whether there is a realistic chance to economically extract the declared resources.*

3.5 Mining and Infrastructure

3.5.1 Mining

3.5.1.1 Introduction

The Balazhal gold mine is an open pit operation which has historically fed oxide ore to a cyanide heap leach operation; however with the depletion of the oxide resources, the mine has recently been mothballed pending the delineation of further resources and sulphide ores have been fully exposed in the base of the pit, which has been left to flood. Currently the mine produces only from stockpiled material which has already been mined.

3.5.1.2 Mining Method

Overview

The Balazhal deposit has been mined by open pit methods using drilling and blasting on a contract basis to break ore, followed by transport to the primary crusher or waste dumps via truck and shovel, before transportation to the heap leach pads.

The open pit has dimensions of 500m length, approximately 100m width and an average depth of 50m, with the base of the pit at absolute elevation of 275masl.

Ore and Waste Transportation

During operations, ore and waste transportation was carried out by dump truck, loaded by a 1.5m^3 diesel shovel. The haulage distance from the bottom of the open pit to the crusher station is approximately 1km.

Mining Equipment

The mobile equipment registered to Balazhal is listed in Table 3.5, not including 3 auxiliary vehicles present on site for the movement of personnel and equipment.

As operations at the site are currently minimal, the majority of this equipment has been moved to the nearby Zherek and Suzdal Mines for storage, maintenance and occasional use when necessary. Maintenance of mining equipment remaining at the site is undertaken on an ad-hoc/as required basis in the basic workshop facilities present on the site.

Table 3.5: Mining Equipment Inventory

Equipment	Number
Belaz 30t Dump Trucks	2
T130 Bulldozer	3
Front-end-loader (model not specified)	1
Diesel Hydraulic Shovels (1.5m^3 bucket)	1
Electrical Excavators (2.5m^3 bucket)	2
Total	***9***

Dewatering

Dewatering of the Balazhal open pit is currently not taking place, water is accumulating and if production is to recommence at the site it would be necessary to re-initialise the pumps.

Mine Personnel

As the Balazhal mine is not currently operational, the majority of personnel on the site are involved with administration and mineral processing, as well as maintenance of the heap leach pads. When the mine was operating, two teams of personnel worked on a 15-day rota with two, 12 hour shifts per day including 1 hour for breaks and one hour for shift changes.

Currently, 63 persons are employed at the site, comprising 13 in administrative work, and 50 operational and maintenance staff.

3.5.2 Infrastructure

3.5.2.1 Power

The power supply to mine site is 35kV, provided by overhead lines connected to the local grid, and is distributed via overhead cables and substations/transformers.

3.5.2.2 Accommodation & Offices

The administrative and accommodation facilities comprise one principal office area, in addition to converted transport containers located around the site which are used as localised office facilities, and basic accommodation and recreational facilities.

3.5.2.3 Roads

A simple haul road network on the Balazhal site, providing access between the site entrance and the main operations areas, is generally in good condition and clearly defined, despite snow-cover.

3.5.2.4 Maintenance & Stores

Relatively few stores are maintained on site, given the current minimal level of operations and the majority of spares at the site are related to mineral processing activities. Approximately one month's supply of basic spares and equipment are maintained, with procurement of more specific items carried out on an ad hoc/as-required basis.

3.5.2.5 Fuel

Fuel for vehicles and equipment at Balazhal is supplied on a regular basis (generally no more than twice per month), and stored in designated petrol (3 x 50m^3) and diesel (1 x 25m^3) fuel storage tanks. At the time of the WAI site visit, consumption was 8m^3/month.

3.5.2.6 Explosives

Drilling and blasting operations at Balazhal are carried out by a contractor, who is responsible for the maintenance of a suitable supply of blasting equipment and materials, including an explosive storage facility.

3.5.2.7 Water Supply

Potable and technical water are supplied to the mine from three local boreholes each with a capacity of 6,000m^3 per annum, and producing an adequate supply for current operations.

> ***WAI Comment:*** *most of the details provided above relates to the previous operations. To move the project forward, a Feasibility Study is required to ascertain the economic viability (or otherwise) of an expanded open pit operation and processing facility.*

3.6 Processing

3.6.1 Introduction

The Balazhal operation, which is presently almost completed, employs standard heap leaching techniques to recover gold from oxide/semi-oxide ore. Run of mine ore is cyanide leached on permanent pads and the dissolved gold is recovered using resin adsorption technology. Gold is sold as a cathode gold product to LANA LLP in Almaty.

3.6.2 Heap Leaching

The heap leach operation at Balazhal commenced in 2004. In 2009 gold production was limited to the treatment of 78kt of material which was recovered from old stockpiles, crushed and used to form “Heap No. 6” between May and August 2009.

Ore was crushed using a jaw crusher and a cone crusher to pass 12mm. The crushing plant can handle 800t of ore per day, although it is currently partially disassembled.

There are six heaps at Balazhal, but only Heap No.3 is still being leached. The total amount of material on Heaps 1-5 is 1.63Mt.

Heap 6 had yielded 17kg of gold by October 2009 and total production for the year was 28kg. It was anticipated that a further 5kg of gold would be produced by the end of 2009.

On 01 December 2009, the Heap 6 solutions were recycled to the heap, by-passing the hydrometallurgical plant, in order to build up solution grades.

3.6.3 Transition Zone Ores

Four samples representing the Transition zone, and weighing 500kg, were subjected to a programme of laboratory testwork at Vniitsvetmet in 2008. The samples assayed 1.3 to 2.8g/t Au and contained 10-15% silica, 65 to 75% feldspars and 3-5% ore minerals. The ore minerals consisted mainly of pyrite and arsenopyrite with minor sphalerite which contained fine grained chalcopyrite.

A single composite sample assaying 1.56g/t Au was formed from the four samples and subjected to a range of processing tests. Rational analysis indicated that the sample contained 54.8% free gold, 35.6% of the gold was locked with other minerals, 2.5% was associated with sulphides and 6.3% was associated with silicate minerals.

Gravity processing gave a gold recovery of 55.6% to a concentrate assaying 112g/t Au. Flotation of gravity tailings gave a flotation concentrate assaying 22g/t Au and the overall recovery to the combined gravity and flotation concentrates was 78.4%.

Flotation testing of the head sample gave a gold recovery of 75.7% to a concentrate assaying 72g/t after two stages of cleaning. The optimum grind size was found to be 91-93% passing 74 microns.

Cyanidation of the head sample gave gold recoveries of 83-85% for samples Nos.1 -3 at a grind size of 80% passing 74 microns. Sample No.4 gave a lower gold recovery of 76.8%. Cyanide consumptions ranged from 0.18 to 0.21kg/t.

Bottle rolls testing at crush sizes of -2mm, -70mm and -1.0mm gave recoveries of 86.7%, 88.3% and 83.1% respectively, indicating that heap leach recoveries in excess of 80% would be achievable when treating Transition zone ore.

3.6.4 Hydrometallurgical Plant

Solution from the heaps is received into four 60m^3 storage tanks and from there is pumped through twelve sorption columns approximately 1m diameter and 6m high (Photo 3.1). The barren solution is recharged with cyanide and the pH adjusted prior to being pumped back to the heap. The loaded resin is stripped of its gold content and reactivated for re-use.

At full production, approximately 18t of resin is processed every month. The gold rich solution from the stripping stage is processed using electrolysis cell. The cathode sediment is filtered and washed prior to being roasted in a 25kW electric furnace. The grade is reportedly 77% Au.

Photo 3.1: Sorption Columns

3.6.5 Historical Metallurgical Performance

The heap is irrigated until the grade of gold in solution from the heap is <0.4g/t Au. In 2006 when the last full data were available, an overall gold recovery of 39.7% was achieved from the heaps and prior to this, recovery has varied between approximately 59% and 25% gold.

A summary of the historical production and recoveries for 1H 2006 are shown in Table 3.6.

Table 3.6: Production Data (1H 2006)

Heap	Ore (kt)	Operating time (days)	Grade, (g/t)	Recovered Au, (kg)	Recovery, (%)
1	100.5	150	0.88	51.7	58.8
1M	18.0	120	1.44	14.0	53.9
2	125.4	300	0.98	61.9	50.3
1/2	37.2	205	0.92	20.6	60.1
3	182.1	350	1.12	107.1	52.6
4	192.1	210	1.05	81.2	40.4
3/4	53.2	290	1.16	26.9	43.7
5	301.6	350	1.26	103.1	27.0
4/5	100.3	200	1.54	38.4	24.9
Total	1110.4	2175	1.15	504.9	39.7

WAI Comment: with relatively low recoveries and diminishing ore supplies, the future of the oxide operation was clearly very limited, to the extent that the operations closed later in 2006. The remaining ore will be predominantly sulphidic and transitional and therefore a different process route will be required to treat this material. A Feasibility Study is required to determine the future of the project.

3.7 Environmental & Social

3.7.1 General Description

It has been reported that during 2010, an effort will be made to recalculate the resources/reserves to meet the Subsoil Use Contract requirements to substantiate the feasibility of further project development the project. Failure to meet this requirement may result in the expropriation of the Licence by the Government.

3.7.2 Licences and Permits

There is an OVOS document (equivalent to EIA) in place for the operations an environmental action plan was prepared for 2009 for the Balazhal operations whilst 2010 was at the drafting stage during WAI’s site visit.

The licences and permits pertaining to the operations appear to be currently compliant with the local legislative requirements in full.

3.7.3 Water Management

There is one well for the potable water supply whilst three more boreholes are used for the technical water purposes.

At the processing stage, the technical water is continuously recycled with some fresh water being added if necessary to compensate for the losses occurring as a result of evaporation. A plastic-lined emergency pond, located near the plant, is intended for collection of any emergency discharge of the liquor.

3.7.4 Air Quality

WAI understands that the main source of air pollution is dust generated by works in the open pit and the crushing complex with the suspended particles being the main pollutant. Overall, the emissions to air from the mining, processing and related activities are relatively small and any effects are localised.

3.7.5 Waste Management

Waste rock is considered to be the main type of waste generated by the Balazhal mining operations and falls under the lowest class (class V) of the waste hazard classification. Scrap metal is accumulated at the storage area and then sold in accordance with the agreement in place; treatment of hazardous waste includes removal of mercury from bulbs whilst cyanide drums are neutralised using hypochlorite, pressed and stacked on to the heap leach pad.

3.7.6 Environmental Monitoring

WAI believes that the environmental reporting, management and monitoring at Balazhal satisfy the State requirements.

It is reported that the monitoring of air, soil, surface and ground water quality for poisonous substances, petroleum products and heavy metals is undertaken on a quarterly basis and has shown that the determinands are within the threshold levels established for the operations.

3.7.7 Mine Closure Planning and Land Rehabilitation

Semgeo has drafted a mine closure plan for the Balazhal operations to meet the regulations and the requirements of the Subsoil Use Contract. There is a financial provision set aside in an escrow account for the final closure costs which, to date amounts to 117,814,000 Tenge (c.US$785,500) and has accrued from the annual transfers of 0.1% of the OPEX; apparently it is the Company's intention to double the existing amount to make substantial provision for closure costs.

3.7.8 Environmental Liability

It is the State policy that the current mine owner is responsible for operations in terms of both current and historical environmental pollution. Up until 2005, the old tailings from a mercury amalgamation route as part of the gravity-amalgamation processing flowsheet during 1938 – 1952, had been stockpiled in the mining licence. Although no hazard had been identified, an incident which resulted in the hospitalisation of personnel and the temporary closure of the operation occurred in 2005.

> ***WAI Comment:*** *it is not known if the modified process flowsheet included neutralisation/de-mercurisation of the toxic waste, if so how the latter has been handled and what actions the Company intends to undertake to mitigate potential negative effects to the surrounding environment and human health as a consequence of migration via natural pathways. The issue needs clarification and provision made for monitoring water courses.*

3.7.9 Environmental, Social and H&S Management

Although personal health and safety awareness at the site has been of high priority for the Company, there has been no formal Environmental/Social Management Plan for the operations; however, standard mechanisms for environmental, health and safety management have been introduced and it is understood that the Health and Safety Plans are in place and being updated with the new policies and procedures introduced by Semgeo.

3.7.10 Conclusions and Recommendations

The mine is a conventional heap leach operation, the main hazard being the use of cyanide and other poisonous substances for the extraction of gold. Any spill or release of such toxic substances into the surrounding environment would be of highest concern and the mine has procedures in place for the transport, storage and handling of cyanide and other hazardous chemicals and is in possession of relevant permits, which lowers the potential risks.

The environmental monitoring programme is comprehensive and sound whilst the monitoring reports show the determinands to be within the established norms.

Implementation of the Environmental and Social Management System should be considered to achieve international best practice techniques. Professional advice should be sought to achieve this aim.

WAI would emphasise that due to the finite nature of the resources and the forthcoming cessation of the Balazhal operations, whether temporarily or not, the Company should address the issues of the mine closure planning, rehabilitation and post-closure monitoring as matter of priority, and, where necessary, financial provisions should be realistically estimated/revised.

In addition, WAI recommends that due cognisance is given to those issues concerned with planning and implementation of procedures for effective decommissioning of the cyanide facilities as well as addressing the possible mercury contamination legacy.

WAI considers that to ensure compliance with national quality standards and for a project to be fully compliant with international best practice in the future, the provisions of the latest IFC performance standards and WB Guidelines will need to be adopted and will become more relevant as and when the Company has proved the feasibility of developing the project further.